TECHNICAL REPORT

ON THE

CARMACKS PROJECT
PRELIMINARY ECONOMIC ASSESSMENT (PEA)
YUKON, CANADA

UTM NAD83 Zone 8N 411900 m E; 6912700 m N
LATITUDE 62° 20' N, LONGITUDE 136° 42' W

Prepared for:

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SGS Project # P2022-14

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1 SUMMARY

1.1 Introduction

Granite Creek Copper Ltd. commissioned SGS Geological Services to complete a Preliminary Economic Assessment (PEA) for the Carmacks Project, located 192 km north of Whitehorse, the capital of the Yukon Territory. The purpose of this study is to develop and document a preliminary project design and economics for recovery of copper, gold, and silver from the oxide and sulphide mineralization using flotation technology.

The structure and content of this report uses National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101) as a guideline.

The current PEA is based on a Mineral Resource Estimate (“MRE”) for the Project previously reported by the Company, for the 147, 2000S and 1213 zones. On March 15, 2022 the Company announced the Carmacks Project MRE to contain:

- In-Pit Oxide – 15.7 million tonnes in Measured and Indicated categories, grading 0.94% Cu, 0.36 g/t Au, 3.23 g/t Ag and 0.01% Mo
- In-Pit Sulphide – 19.2 million tonnes in Measured and Indicated categories, grading 0.71% Cu, 0.18 g/t Au, 2.74 g/t Ag and 0.01% Mo
- Below Pit Sulphide – 1.4 million tonnes in Measured and Indicated categories, grading 0.82% Cu, 0.19 g/t Au, 2.88 g/t Ag and 0.01% Mo

There has been no drilling completed on the Property since the release of the MREs and the MREs are considered current. Details on the MREs are presented in Section 14.

1.2 Accessibility, Climate, Local Resources, Infrastructures, and Physiography

The Carmacks Property (including Carmacks and Carmacks North properties) is located approximately 47 km northeast of the village of Carmacks, and approximately 210 km northwest of Whitehorse; the property occurs in NTS map sheet 115/I07. The Carmacks Project camp is centered at approximately 62°20’N latitude and 136°42’W longitude, or 411900E/6912700N in UTM co-ordinates (NAD83 Zone 8N). The Carmacks Project is within 20 km of grid power and paved highway, and the Minto Mine is located 42 km northwest of the Carmacks Camp.

The combined projects include 974 claims that covers approximately 17,580 hectares (175.80 km^2). All are 100% owned by Granite Creek (subject to certain net smelter return (“NSR”) royalties).

The Company acquired 100% of the Carmacks Project through its acquisition of Copper North. On November 30, 2021, $1.8 million has been paid in advance royalty payments. The Company is required to make an advance royalty payment of $100,000 in any year in which the average daily copper price reported by the London Metal Exchange is US$1.10 per pound or greater (Paid subsequent to November 30, 2021). Any production from the Carmacks Project is subject to either a 15% net profits interest or a 3% net smelter return royalty, at the Company’s election. If the Company elects to pay the net smelter return royalty, it has the right to purchase the royalty for $2.5 million, less any advance royalty payments made to that date. Subsequent to November 30, 2021, the Company paid $100,000 as advance royalty payment.

In January 2019, the Company acquired an undivided 100% interest in the Stu Property in consideration for an aggregate of 3,000,000 units (each, a "Transaction Unit") at $0.075 per Transaction Unit valued at $225,000 and a 3% net smelter return royalty to the vendors on any future production on the Stu Property (the "Royalty"). Granite Creek has the option to purchase up to two-thirds of the Royalty from the vendors.

The Company will also make annual advance Royalty payments of $30,000 to the vendors beginning in May 2022, and in each subsequent year thereafter until the commencement of any commercial production on the Stu Property. Each Transaction Unit was comprised of one common share and one share purchase warrant, with each warrant exercisable into one additional common share at an exercise price of $0.15, with an expiry of January 16, 2022. Subsequently 2,500,000 warrants were exercised and 500,000 expired unexercised.

The Carmacks Project site is currently accessible by way of the Freegold Road that leads northwest of Carmacks for 34 km then by the Carmacks Project access trail for 13 km to the Property. The village of Carmacks, on the Yukon River, is 175 km by paved road north of Whitehorse. The property access road is narrow and rough with steep sections and requires 4x4-vehicle capabilities in inclement weather conditions. A new 13 km access road is proposed to be constructed as part of the Carmacks Project development; brush clearing along the road alignment was completed in 1997. The Freegold Road is maintained by the Yukon Government (YG) and is currently open seasonally, generally from April through September. The road will be kept open year-round by YG once a year-round operation begins.

Beyond the Carmacks Project camp, a 10 km user-maintained gravel road with four creek crossings leads to Hoocheekoo Creek in the middle of the Carmacks North Property area. Bulldozer and ATV trails on the Carmacks North property leads to the various zones on the property. The Carmacks North Property can also be accessed by a 15-20-minute helicopter flight from the Carmacks Property.

Due to its road access, proximity to Carmacks, Whitehorse and the Whitehorse airport, the Property can be efficiently accessed by 4x4-vehicle or helicopter and thus exploration, primarily diamond drilling, can be conducted year-round.

The year-round ports of Anchorage and Skagway, Alaska, and Stewart and Prince Rupert, BC, are accessible by all-weather highway to move overseas–sourced equipment and supplies into the Carmacks Project site and for potential shipment of copper concentrate. Anchorage is 1,133 km west of Whitehorse and Skagway is 180 km south, while Stewart is 1,043 km south, and Prince Rupert is 1,373 km south.

Local commercial resources are limited. The village of Carmacks, with a population of about 500, has some lodging capacity and a few stores and restaurants. Whitehorse has a population of approximately 23,000, which is about two-thirds of the entire Yukon population. Whitehorse has an international airport, serviced by daily commercial flights from British Columbia and Alberta to the south and other northern communities. All-weather paved highways connect Whitehorse to the south and west to Alaska.

There are no permanent facilities currently on the property as all previous work was performed from a tent and trailer camp. Some clearing of brush has been performed in the area of a previously proposed heap leach pad.

The property size and moderate terrain have proven sufficient to accommodate mining facilities, mill processing sites, and waste disposal sites. There is sufficient room for expansion of these facilities. There is sufficient water on the property to supply mining and milling operations, including accommodations and drilling.

Should the Carmacks Copper deposit advance to development, any infrastructure development (roads, power etc.) would benefit the Carmacks North Project. The subdued topography on the Carmacks North Property is suitable for construction of mining operations and there is sufficient water available on the property or nearby for drilling and development.

The climate in the Carmacks Project area is marked by warm summers and cold winters. Average daily temperatures at the Williams Creek Station on the Carmacks Project site range from -30°C in January to 12°C in July. The location close to the Arctic Circle provides 22 hours of daylight at the summer solstice and similarly long nights at the winter solstice.

Precipitation is light with moderate snowfall, the heaviest precipitation being in the summer months. The average annual precipitation is approximately 346.5 mm (water equivalent) with about 30% falling as snow. July is the wettest month. Annual lake evaporation is estimated to be 440 mm to yield a net loss of 93.5 mm. The weather does not impede year-round commercial operations in the Yukon, including outdoor activities in the winter, except in the harshest cold snaps when temperatures may drop to –50°C.

Winter conditions, where daytime maximum temperatures average below zero, occur from November to March. The extreme cold temperatures in the region make outside construction in the winter difficult. In general, the outdoor construction season will be from April to October.

Topography at the Carmacks and Carmacks North property area is subdued. Topographic relief for the entire property is 515 m. In the immediate area of the No. 1 Zone, topographic relief is 230 m. Elevations range from 460 m at the Yukon River to 1,030 m on the western edge of the claim block. The area falls within the Central Yukon Plateau which is characterized by broad valley and rounded ridge crests. Permafrost is discontinuous and scattered as the mean annual ground temperatures exceed -5°C. The permafrost is encountered at depths of 40 to 50 cm on most north-facing slopes where glacial till or colluvium is present.

The claims and leases comprising the Carmacks property are held directly by Granite Creek Copper Ltd., a member of the Metallic group of Companies is listed on the TSX-V under the symbol GCX.

The property is within the Traditional Territories of the Little Salmon/Carmacks First Nation and the Selkirk First Nation but does not lie on any First Nation settlement lands or land selections.

1.3 Geology and Mineralization

The Late Triassic to Early Jurassic magmatism in Yukon resulted from building of a Late Triassic island arc (Lewes River Group and Stikine plutonic suite) and subsequent arc-continent collision, syncollisional magmatism, and exhumation. Volcanic rocks of the Lewes River Group terminate in central Yukon, however their plutonic equivalents, represented by the Stikine and Pyroxene Mountain suites, extend farther northwest into east-central Alaska. The Stikine suite (217–214 Ma) is represented by a series of small plutons that intrude Upper Triassic arc volcanic rocks of Stikinia and Paleozoic metasedimentary and meta-igneous rocks of the Yukon-Tanana terrane in south-central Yukon. The Minto suite (205–194 Ma) occurs as a series of large plutons that intrude the Lewes River Group and the Yukon-Tanana terrane that are interpreted to represent syncollisional magmatism at the onset of arc accretion. The younger Long Lake (188–183 Ma) and Bennett-Bryde (178–168 Ma) plutonic suites represent ongoing syn-collisional magmatism.

The Carmacks Copper deposit is located within the composite Early Jurassic Granite Mountain batholith. The Granite Mountain batholith is the southern extent of a series of Early Jurassic plutons, including the Minto and Yukon River plutons that form part of a single large batholith, ~120 km long by 15 to 25 km wide, segmented by Upper Cretaceous and younger volcanic cover. The eastern Granite Mountain batholith is

assigned to the Minto suite and its western part belongs to the Long Lake suite. The Granite Mountain batholith intrudes and obscures the contact between mid-Paleozoic rocks of the Yukon-Tanana terrane and Upper Triassic rocks of Stikinia. The Yukon-Tanana terrane west of the Granite Mountain batholith is represented mainly by orthogneiss of the Early Mississippian Simpson Range plutonic suite. Stikinia arc rocks east of the Granite Mountain batholith include volcanic and sedimentary rocks and subvolcanic intrusions of the Upper Triassic Povoas Formation of the Lewes River Group. The Povoas Formation in southern Yukon is characterized by variably deformedand subgreenschist to locally amphibolite facies augite porphyritic basalt, volcaniclastic rocks, and hornblende gabbro. These Stikinia units and the Granite Mountain batholith are in fault contact along the dextral-normal oblique-slip Hoocheekoo fault. The Granite

Mountain batholith contains inliers of variably deformed and metamorphosed mafic to intermediate rocks that host Cu-Au-Ag mineralization at the Carmacks Copper deposit, Minto mine, and Stu prospect.

Late Triassic to Early Jurassic batholiths were emplaced into crust that was being exhumed in the Early to Middle Jurassic to form the flanks of the subsiding marine basin of the Whitehourse trough. Exhumation is recorded by regional Early Jurassic metamorphic cooling ages, Al-in-hornblende barometry of Mesozoic plutons, and the Early to Middle Jurassic sedimentologic and detrital zircon record. Exhumation was essentially complete by the mid-Cretaceous, as indicated by the unconformably overlying volcanic rocks of the Mount Nansen Group, which are exposed 40 km to the southwest of the Carmacks Copper deposit. Volcanic rocks of the Upper Cretaceous Carmacks Group are preserved as extensive blankets north and south of the Granite Mountain batholith, and as isolated erosional remnants within the batholith. The Granite Mountain batholith is separated from the Minto pluton to the north, host of the Minto Cu-Ag-Au mine, by a veneer of the Carmacks Group.
The Carmacks Copper deposit area is located near the northwestern limit of Pleistocene glaciation, such that glacial erosion was restricted to subalpine areas and that bedrock below discontinuous till preserves a deep oxidative weathering profile. Paleoweathering profiles that contain copper oxide minerals at the deposit are locally capped by Carmacks Group volcanic rocks, indicating that at least part of the oxidation history is Late Cretaceous or older.

The Carmacks Copper deposit is hosted in a series of elongate, N-NW-trending inliers of amphibolite facies mafic to intermediate meta-igneous rocks and migmatitic derivatives within generally massive granitoids of the Granite Mountain batholith. Mafic rocks include foliated, equigranular amphibolite that locally is texturally transitional with less foliated, hornblende-porphyroblastic amphibolite. Rare augite gabbro is also locally present. Mafic rocks are interlayered with quartz-plagioclase-biotite schist. These metamorphic rocks are texturally transitional with migmatitic rocks, which host the bulk of hypogene copper mineralization. Migmatitic rocks occur preferentially along the eastern flank of the largest, 3-km-long by 20- to 100-m-wide inlier, where they represent a transitional intrusive contact between metamorphic rocks and the Granite Mountain batholith. However, this does not appear to to be the case at depth based on 2021 drilling of Zone 1.

Metamorphic rocks preserve a penetrative transposition fabric (S1), which is defined by a preferred orientation of hornblende and biotite in amphibolite and schist, respectively, or by leucrocratic and melanocratic layering in migmatitic rocks. Felsic plutonic rocks of the Granite Mountain batholith are generally massive but locally exhibit a weak tectonic foliation near contacts with metamorphic inliers.

Several late, E-NE-trending faults cut all previously described rock units and structural elements. Slickenlines on hematitic or pyrolusite-coated fault surfaces show shallow plunge (5°–30°), indicating that the latest movement is dominantly strike-slip.

Faults identified in the deposit area include provide the limits of the resource and opens the recommendation of further exploration beyond these faults:

1. A fault that define the southern end of 2000S zone
2. A north zone 1 fault
3. A N-S trending fault that defines the graben to the east of zone 12
4. A The N-S fault that controls the conglomerates and Carmacks volcanics sitting on the hangingwall above zone 13.

These provide the limits of the resource and opens the recommendation of further exploration beyond these faults.

The Cu-Au-Ag deposits of the Carmacks Project, and the related Minto deposit, are considered rare examples of metamorphosed porphyry Cu systems.

Mineralization of the Carmacks Copper Cu-Au-Ag deposit occurs within a 3-km-long, N-NW-trending belt that includes the northern Main zone, and zones 2000S, 12, and 13 in the south. The Main zone includes zones 1, 4, 7, and 7A. Disseminated, foliaform, and net-textured varieties of hypogene copper sulphide mineralization are recognized where oxidation is partial to absent.

Disseminated chalcopyrite and pyrite are a minor component of the hypogene mineralization, and occur in the undeformed, hornblende porphyroblastic amphibolite, granoblastic quartz-plagioclase-biotite schist, and augite gabbro. Disseminated copper sulphide mineralization also occurs in the leucosome of metatexite, i.e., melt-bearing equivalents of the quartz-plagioclase-biotite schist. Chalcopyrite occurs as fine blebby to skeletal grains intergrown with pyrite and trace pyrrhotite.

Foliaform copper sulphides occur as chalcopyrite-dominant stringers and blebs that parallel foliation in amphibolite and quartz-plagioclase-biotite schist. Chalcopyrite is intergrown with subordinate bornite and rare trace pyrite.

Copper mineralization hosted in diatexite migmatite occurs as net-textured intergrowths of bornite and chalcopyrite that occupy interstitial space between rounded and embayed silicate grains in leucosome. Typical bornite-chalcopyrite ratios are 3/1, and net-textured bornite is especially abundant in melanosome, where it forms higher grade (1–2% Cu) domains. Bornite contains microscopic inclusions of native bismuth, Au-Ag tellurides, and Bi tellurides. Molybdenite is commonly intergrown with net-textured copper sulphides and occurs either as kinked grains separated along cleavage surfaces or as euhedral, undeformed grains. Both chalcopyrite and bornite are commonly replaced by secondary digenite along fractures and grain margins. Gold is principally associated with bornite and occurs as 10-20 μm inclusions of electrum or native gold, or more commonly as gold telluride (calaverite), or solid-solution gold-silver tellurides. Silver is present as hessite inclusions in bornite. As gold and silver are typically associated with bornite, the bornite-chalcopyrite ± digenite zone is precious - metal enriched and the migmatite contains higher copper, gold, and silver grades than the amphibolite and quartz-plagioclase-biotite schist sequence.

Deep oxidation of the deposits has oxidized primary sulphides to copper oxides and copper carbonates, with approximately 15% of the copper in the oxide domain occurring as remnant copper sulphide, in the form of chalcopyrite. This oxidation profile has led to the formation of an oxide cap that can be over 200 metres thick at zone 1 to ~40m thin in zone 12. The majority of the copper found in oxide are in the form of the secondary minerals malachite, cuprite, azurite, tenorite (copper limonite) and crednerite with minor other secondary copper minerals (covellite, digenite, chalcocite). Gold occurs as native grains, most commonly in cavities with limonite or in limonite adjacent to sulphides, but also in malachite, plagioclase, chlorite, and rarely in quartz grains. Gold is rarely greater than five microns in size.

1.4 Exploration History

The first reported copper discovery in this region was made by Dr. G.M. Dawson in 1887 at Hoochekoo Bluff, on the Yukon River, 12 km north of the Property. In 1898, the first claims were staked to cover copper showings that were associated with copper bearing quartz veins located in Williams Creek and Merrice Creek Canyons, east of the present Carmacks Copper deposit.

In the late 1960's, exploration for porphyry copper deposits in the Dawson Range led to the discovery of the Casino porphyry copper deposit, 104 km to the northwest of the Carmacks Copper deposit. This discovery precipitated a staking rush that led to the staking of the Williams Creek property in 1970 by G. Wing and A. Arsenault of Whitehorse. The Dawson Range Joint Venture (Straus Exploration Inc., Great Plains Development of Canada Ltd., Trojan Consolidated Minerals Ltd., and Molybdenum Corporation of America) optioned the property and contracted Archer, Cathro and Associates to conduct reconnaissance prospecting and geochemical sampling. During this program, Zones 1 and 2 were discovered.

Since 1970, the Carmacks Property has been the subject of various exploration campaigns comprising surface trenching, diamond drilling, reverse circulation drilling (RCD), geophysical, and geochemical surveying. The majority of this work focused on the No.1 Zone and was completed before the mid-1990s. From 1972 to 1990 there was no significant work performed on the property.

In early 2006, Glamis Gold Ltd. purchased Western Silver Corporation and spun off a separate firm named Western Copper Corporation. Western Copper retained the rights to the Carmacks Copper Project. In 2006, a new exploration program was initiated on the Carmacks Project. This consisted of diamond drilling and some rapid air blast drilling. A total of 24,100 m in 157 drill holes were completed between 2006 and 2007.

The object of the 2006 program was to examine the down dip extension of Zone 1, with a goal to delineate the oxidation-reduction front at depth on the deposit; confirm historic drill results by twinning two of the previously drilled holes and explore along strike to search for lateral extensions of Zone 1, and to expand the knowledge of some of the other mineralized zones.

In addition, a rotary air blast (RAB) drilling program commenced in August 2006, which was designed to condemn areas of the Property for future plant site development.

In September 2006, Western Copper retained M3 Engineering & Technology Corporation (M3) to revise the earlier studies and to develop a Bankable Level Feasibility Study (FS) fully compliant with NI 43-101 for the heap-leaching recovery of copper. This study was completed in 2007 (M3, 2007). The FS only considered oxide mineralization in Zones 1, 4 and 7.

In 2007, Western Copper continued the exploration and environmental sampling program and conducted geotechnical studies of the proposed heap leach pad, waste rock storage area, processing plant and camp location. The object of the 2007 program was to define the northern and southern limits of Zones 1, 7 and 7A, to delineate Zone 4, to further test and define Zones 12 and 13, expand the exploration of the newly discovered Zone 14, and carry out condemnation drilling in the proposed waste rock storage, heap leach pad and the processing plant areas. The 2007 program consisted of 17,800 m of diamond drilling in 123 holes, 866 m of geotechnical drilling in 36 holes, 31.7 line km of induced polarization surveys and surveying of all drill hole locations including all the historic drill holes, geotechnical holes, and rapid air blast drill holes.

In 2008, Western Copper drilled 12 geotechnical holes (1,923 m) in the pit area, two (2) water wells in the camp area (253.5 m), and one (1) water monitoring well below the heap leach pad (151 m).

In October 2011, Western Copper split into three separate companies, Copper North Mining Corp., which retained the Carmacks Project, NorthIsle Copper & Gold Inc., and Western Copper and Gold Corporation. Copper North continued to manage the Carmacks Project. In 2012, M3 updated the feasibility study for the heap leaching recovery of copper to reflect Carmacks Project design changes made to address environmental concerns.

Copper North carried out limited drilling campaigns in 2014 and 2015 that totaled 4,358 m of drilling in 50 holes. The exploration focused on extending the known mineral resources in an effort to expand the current measured and indicated mineral resources, as a first step in increasing potential mine life.

In 2014, Copper North commissioned Merit Consultant International Inc. (Merit) to prepare a Preliminary Economic Assessment (PEA) on the Carmacks Project (based on the 2007 MRE). The PEA focused again on zones 1, 4 and 7 and specifically examined, at a conceptual level, the potential economic viability of adding gold and silver recovery by cyanidation to the Carmacks Project. The gold and silver was to be recovered from the cyanide leachate using sulfurization, acidification, recycling and thickening (SART) and absorption, desorption and refining (ADR) processes. The PEA concluded that the addition of gold and silver recovery to the Carmacks Project improved the overall Carmacks Project economics with respect to gross and net revenues and the cash cost of copper recovery after deduction of the gold and silver credits.

In 2016, Copper North commissioned JDS Energy & Mining Ltd. (“JDS”) to complete a PEA for the Carmacks Project. The purpose of this study was to develop and document a preliminary project design and economics for recovery of copper, gold, and silver from the oxide mineralization using agitated tank leach technology from Zones 1, 4 and 7. The 2016 PEA included updated MREs for Zones 1, 4, 7, 12, 13 and 2000S. Mineralization for Zones 2000S, 12 and 13 were not considered for the 2016 PEA or any previous FS.

Copper North undertook drilling in September and October 2017 to gather more geotechnical information and exploration in the mineral area that was drilled in 2015. The drilling in the location of the planned deposition of dry stacked tailings was completed as part of preparing for improvement of the environmental report required for submission for new environmental approval and amended permits.

The results of the drilling in zones 2000S, 13 and 12 confirmed the continuity of the zones and copper grades.

The drill results in the south area zones confirmed the continuity of the mineralized zones and extended the mineralized zones to further increase the size of the mineral areas. The Copper North undertook renewing the mineral resource in zones 2000S, 13 and 12 zones to provide a new mineral resource in the south area.

All prior and historic resources are superseded by the MREs for the Carmacks Project reported in the current report.

1.4.1 Carmacks North Property

The Carmacks North Property was worked from 1971 to 1982 by United Keno Hills Mines (UKHM), and again from 1989-2013 by UKHM, Western Copper and other operators. The amount of detailed information and geochemical results from UKHM’s trenching and drilling programs is limited.

UKHM carried out bulldozer trenching programs in 1979 and 1982 over four geochemical and/or geophysical anomalies. Complete assay results are not available, but trench maps with geology and some results were sourced from the UKHM archives. Selected trenches were cleaned and deepened, extended and new trenches were dug in 2015.

There were two programs of drilling on the Stu Property. Approximately 4500 metres of diamond drilling was done by UKHM in 1980 in the A and C Zones. Core from the program is stored near the camp and in 2015 the racks were disassembled and most of the core rehabilitated. Historical drill logs and assay results for the 1980 program are incomplete; the key reports describing the trenching and drilling program were not filed for assessment.

There are reports of three high grade intersections from the 1980 drilling in Zone A:

- 80-09 3.44% Cu, 1.87 g/t Au, 13.37 g/t Ag over 13.5m
- 80-14 3.51% Cu, 2.49 g/t Au, 18.35 g/t Ag over 13.5 m
- 80-18 2.80% Cu, 4.04 g/t Au, 17.42 g/t Ag over 12.5m

There were additional mineralized intercepts with values up to 0.49%. All three intersections were rehabilitated in 2015 but have not been resampled.

1.5 Mineral Processing and Metallurgical Testing

Multiple historical testwork programs have been conducted by various testing labs, based on the anticipated process which including crushing, agglomeration, heap leach, grinding, tank acid leach and cyanide leach. This testwork program implied a complex process flowsheet including comminution, two stages of tank leaching with separate acid and cyanide systems, solvent extraction and electrowinning for copper, ADR (adsorption, desorption and recovery by activated carbon) and a refinery for dore bullion production. This complex flowsheet was not economically viable. In 2021, preliminary flotation testwork was conducted by BV Minerals on the sulfide material, which showed that over 90% copper recovery and a 25% copper concentrate grade is achievable using flotation.

In 2022, Granite Creek sent to the SGS Vancouver Laboratory both sulfide and oxide samples to explore the feasibility of recovering the copper and the precious metals through flotation only. The testwork included both Qemscan mineralogy tests and a series of flotation investigations. The mineralogy test study indicated that the major copper minerals were chalcopyrite and bornite for the sulfide material, and malachite, azurite, and copper bearing iron hydroxide for the oxide material. The major gangue minerals were albite, plagioclase, quartz, k-feldspar, biotite, amphiboles and chlorite.

In addition to the sulfide flotation program, SGS Vancouver also optimized the oxide flotation performance through modifications of the flotation reagents scheme, which led to much-improved metal recoveries on the pure oxide material, as well as oxide-sulfide blended material. The summary of the test results is presented in Table 1-1 below.

Table 1-1 Flotation Test Results Summary

Test No.	Concentrate grade,			Recovery, %		
	Cu, %	Au, g/t	Ag, g/t	Cu	Au	Ag
Sulphide F4	42.7	7.67	117	93.7	69.0	78.4
Oxide F5	26.2	13.6	93	39.8	57.5	37.4
Blend F4	40.8	12.4	120	75.3	65.7	66.2
Blend LCT1	40.1	10.6	104	82.00	70.1	68.6

Since both sulfide and oxide samples as received contained a measurable percentage of acid soluble copper, it is suspected that the quantity of acid soluble copper contained in an individual test sample could have significant impact on the metal recoveries. Regression relationships between the acid soluble copper content and the metal flotation recoveries in a final flotation concentrate were developed. It was found that all payable metals, which include copper, gold, and silver, have a reasonable correlation between their flotation recoveries and the acid soluble copper percentage in the test material. Consequently, these regression relationships were used to estimate the metal recoveries for various blended materials and to optimize the resource model of this project. The detailed regression formulas used to predict the metal recoveries from blended materials are introduced in Section 13 of this report.

The key process parameters from the 2022 Vancouver testwork program are summarized below:

- A primary grind size of 150 um.
- A circuit employing a rougher and two stages of cleaner flotation will be required to ensure a marketable concentrate grade, employing the following retention times,
    - Rougher, 22.5 minutes
    - First cleaner, 12.5 minutes
    - Second cleaner, 10 minutes

- Regrind size approximately 30 um to insure sufficient liberation
- pH around 10 in cleaner flotation
- The reagents as required in flotation
    - Lime for pH adjustment
    - Potassium Amyl Xanthate as a sulfide collector
    - Collector 3418A as a sulfide collector
    - Sodium sulfide as modifier to sulfidize the oxide copper minerals
    - Collector A-OX100 as a collector for copper oxide material
    - Frother

1.6 2022 Carmacks Project Mineral Resource Estimate

Completion of the update MREs for the Carmacks Project involved the assessment of a drill hole database, three-dimensional (3D) mineral resource models, and available written reports. The database provided for the current MREs comprise data for 489 surface drill holes totaling 59,679.07 metres completed on the Carmacks Project area between 1970 and 2021. This includes 36 drill holes (RC and diamond) totaling 9,413.06 m completed by Granite Creek between the fall of 2020 the fall of 2021. The database used for the MREs totals 12,794 drill core assay samples representing 17,233 m of drilling.

All available geological data and geologic information has been reviewed and verified by Author as being accurate to the extent possible. The Author is of the opinion that the database is of sufficient quality to be used for the updated Carmacks Project MREs.

Inverse Distance Squared (“ID2”) restricted to mineralized domains was used to Interpolate grades for Cu_T (total copper in ppm), Cu_X (copper oxide in ppm), Cu_S (copper sulphide in ppm) Au (g/t), Ag (g/t) and Mo (ppm) into block models (blocks 5m x 5m x 5m in size). Composites of 2.0 metre used for the resource estimation procedure have been capped where appropriate. Appropriate interpolation parameters were generated for each deposit based on drill hole spacing, mineralization style and geometry.

The MREs for the Carmacks Project are prepared and disclosed in compliance with all current disclosure requirements for mineral resources set out in the NI 43-101 Standards of Disclosure for Mineral Projects (2016). The classification of the current MREs into Measured, Indicated and Inferred is consistent with current CIM Definition Standards - For Mineral Resources and Mineral Reserves (2014), including the critical requirement that all mineral resources “have reasonable prospects for eventual economic extraction”.

The general requirement that all mineral resources have “reasonable prospects for eventual economic extraction” implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account extraction scenarios and processing recoveries. In order to meet this requirement, the Author considers that the Carmacks Project deposit mineralization is amenable for open pit and underground extraction.

In order to determine the quantities of material offering “reasonable prospects for eventual economic extraction” by an open pit, Whittle™ pit optimization software and reasonable mining and processing assumptions to evaluate the proportions of the block model that could be “reasonably expected” to be mined from an open pit are used. The pit optimization for the Carmacks Project was completed by SGS for the current MREs and the pit optimization parameters used are summarized in Table 1-2. Whittle pit shells at a revenue factor of 1.0 (i.e. 100 % of base case metal prices) were selected as the ultimate pit shells for the purposes of reporting the Carmacks Project MREs. A selected base case cut-off grade of 0.30 % Cu_T is used to determine the in-pit MRE for the Carmacks Project deposits.

The reader is cautioned that the results from the pit optimization are used solely for the purpose of testing the "reasonable prospects for economic extraction" by an open pit and do not represent an attempt to estimate mineral reserves. There are no open pit mineral reserves on the Property. The results are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cut-off grade.

In order to determine the quantities of material offering "reasonable prospects for economic extraction" by underground mining methods, reasonable mining assumptions to evaluate the proportions of the block model that could be "reasonably expected" to be mined from underground are used. A review of the size, geometry and continuity of mineralization of each deposit, and spatial distribution of the three deposits (all within a 1.5 x 1.5 km area), was conducted to determine the underground mineablility of the deposits. It is envisioned that the deposits of the Carmacks Project may be mined using lower cost underground bulk mining methods below the pit shells. The underground parameters used to determine a base case cut-off grade for reporting of underground resources is presented in Table 1-3. Based on these parameters, a selected base case cut-off grade of 0.6 % Cu_T is used to determine the below-pit MREs for the Carmacks Project deposits.

The reader is cautioned that the reporting of the underground resources are presented undiluted and in situ (no minimum thickness), constrained by continuous 3D wireframe models, and are considered to have reasonable prospects for eventual economic extraction. There are no underground mineral reserves reported on the Property.

Table 1-2 Parameters used for Whittle™ Pit Optimization and to Estimate the Open Pit Base and Underground Cut-off Grades for the Carmacks Project MREs

Input Data for Open Pit and Underground Mining Scenarios		
Parameter	Value	Unit
Copper Price	$3.60	US$ per pound
Silver Price	$22.00	US$ per ounce
Gold Price	$1,750.00	US$ per ounce
Molybdenum Price	$14.00	US$ per pound
In-Pit Mining Cost - Overburden	$1.75	US$ per tonne mined
In-Pit Mining Cost - Rock	$2.10	US$ per tonne mined
Underground Mining Cost	$25.00	US$ per tonne mined
Processing Cost	$18.00	US$ per tonne milled
General and Administrative	$5.00	US$ tonne of feed
Overall Pit Slope - Rock	55	Degrees
Overall Pit Slope - Overburden	35	Degrees
Oxide Recoveries		
Copper Recovery	85	Percent (%)
Silver Recovery	65	Percent (%)
Gold Recovery	85	Percent (%)
Molydenum Recovery	70	Percent (%)
Sulphide Recoveries		
Copper Recovery	90	Percent (%)
Silver Recovery	65	Percent (%)
Gold Recovery	76	Percent (%)
Molydenum Recovery	70	Percent (%)
Mining loss / Dilution (open pit)	5 / 2	Percent (%) / Percent (%)
Mining loss/Dilution (underground)	5 / 5	Percent (%) / Percent (%)
Waste Specific Gravity	2.66	
Mineral Zone Specific Gravity		
Oxide	2.64	
Sulphide	2.71 - 2.78	
Block Size	5 x 5 x 5	

The 2022 MREs for the Carmacks Project are presented in Table 1-3 and Table 1-4.

Highlights of the Carmacks Project MRE:

- In-Pit Oxide – 15.7 million tonnes in Measured and Indicated categories, grading 0.94% Cu, 0.36 g/t Au, 3.23 g/t Ag and 0.01% Mo
- In-Pit Sulphide – 19.2 million tonnes in Measured and Indicated categories, grading 0.71% Cu, 0.18 g/t Au, 2.74 g/t Ag and 0.01% Mo

- Below Pit Sulphide – 1.4 million tonnes in Measured and Indicated categories, grading 0.82% Cu, 0.19 g/t Au, 2.88 g/t Ag and 0.01% Mo
- **Combined Measured and Indicated - 36.3 million tonnes, grading 0.81% Cu, 0.26 g/t Au, 3.23 g/t Ag and 0.01% Mo**

Table 1-3 Carmacks Project Mineral Resource Estimates, Effective February 25, 2022

Category	CU_T % Cut-off	Tonnes	CU_T		AG		AU		MO		CuEq	
			(%)	(Mlbs)	(g/t)	Ounces	(g/t)	Ounces	(%)	(Mlbs)	(%)	(Mlbs)
In-Pit Oxide												
Measured	0.30	11,361,000	0.96	239	4.11	1,501,000	0.40	145,000	0.006	1.5	1.30	325
Indicated	0.30	4,330,000	0.91	87	3.37	469,000	0.28	39,000	0.007	0.6	1.16	111
Measured + Indicated	**0.30**	**15,691,000**	**0.94**	**326**	**3.91**	**1,971,000**	**0.36**	**184,000**	**0.006**	**2.1**	**1.26**	**436**
Inferred	0.30	216,000	0.52	2.5	2.44	17,000	0.09	1,000	0.006	0.03	0.63	3
In-Pit Sulphide												
Measured	0.30	5,705,000	0.68	86	2.54	467,000	0.16	28,000	0.016	2.0	0.88	111
Indicated	0.30	13,486,000	0.72	214	2.83	1,226,000	0.19	82,000	0.013	4.0	0.93	277
Measured + Indicated	**0.30**	**19,191,000**	**0.71**	**300**	**2.74**	**1,693,000**	**0.18**	**110,000**	**0.014**	**6.0**	**0.92**	**388**
Inferred	0.30	1,675,000	0.51	19	2.24	120,895	0.13	7,000	0.020	0.7	0.7	26
Below Pit Sulphide												
Measured	0.60	26,000	0.71	0.41	2.54	2,000	0.16	132	0.010	0.0	0.88	0.5
Indicated	0.60	1,341,000	0.82	24	2.88	124,000	0.19	8,000	0.012	0.4	1.03	30
Measured + Indicated	**0.60**	**1,367,000**	**0.82**	**25**	**2.88**	**126,000**	**0.19**	**8,000**	**0.012**	**0.4**	**1.03**	**31**
Inferred	0.60	967,000	0.77	16	2.48	77,000	0.17	5,000	0.012	0.3	0.96	20

(1) The classification of the current Mineral Resource Estimates into Measured, Indicated and Inferred are consistent with current 2014 CIM Definition Standards - For Mineral Resources and Mineral Reserves.

(2) All figures are rounded to reflect the relative accuracy of the estimate.

(3) All Resources are presented undiluted and in situ, constrained by continuous 3D wireframe models, and are considered to have reasonable prospects for eventual economic extraction.

(4) Mineral resources which are not mineral reserves do not have demonstrated economic viability. An Inferred Mineral Resource has a lower level of confidence than that applying to a Measured and Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

(5) It is envisioned that parts of the Carmacks Project deposits may be mined using open pit mining methods. In-pit mineral resources are reported at a base-case cut-off grade of 0.3 % Cu_T within Whittle pit shells. It is envisioned that parts of the Carmacks Project deposits may be mined using lower cost underground bulk mining methods. A selected base-case cut-off grade of 0.6 % Cu_T is used to determine the underground mineral resources.

(6) Cu Eq calculation is based on 100% recovery of all metals using the same metal prices used for the resource calculation.

(7) A pit slope of 55 degrees for rock and 35 degrees for overburden are used for the pit optimization.

(8) The results from the pit optimization are used solely for the purpose of testing the “reasonable prospects for economic extraction” by an open pit and do not represent an attempt to estimate mineral reserves. There are no mineral reserves on the Carmacks Property. The results are used as a guide to assist in the preparation of a Mineral Resource statement and to select an appropriate resource reporting cut-off grade.

(9) Cut-off grades are based on metal prices of $3.60/lb Cu, $22.00/oz Ag, $1,750/oz Au and $14.00/lb for Mo, processing and G&A cost of $US23.00 per tonne milled, and variable mining costs including $US2.10 for open pit and $US25.00 for underground. The cut-off grades should be re-evaluated in light of future prevailing market conditions (metal prices, mining costs, processing costs etc.).

(10) *Metal recoveries used for pit optimization and calculation of base-case cut-off grades include: for oxide material 85% for copper, 65% for Ag, 85% for Au and 70% for Mo; for sulphide material, 90% for copper, 65% for Ag, 76% for Au and 70% for Mo.*

(11) *Composites of 2.0 metre used for the resource estimation procedure have been capped where appropriate. Grades for Cu (oxide, sulphide and total), Ag, Au and Mo for each deposit was interpolated into blocks by the Inverse Distance Squared (ID^2) calculation method.*

(12) *Fixed specific gravity values of 2.64 for oxide material and 2.71 – 2.78 (depending on deposit) were used to estimate the Mineral Resource tonnage from block model volumes. Waste in all areas was given a fixed density of 2.66.*

(13) *The database used for the current MREs comprise data for 489 surface drill holes totaling 56,679 metres completed on the Carmacks Project area between 1970 and 2021. This includes 36 drill holes (RC and diamond) totaling 9,413 m completed by Granite Creek between the fall of 2020 the fall of 2021. Appropriate interpolation parameters were generated for each deposit based on the mineralization style and geometry.*

Table 1-4 Carmacks Project Mineral Resource Estimates, February 25, 2022: Distribution of Cu_X and Cu_S

Category	CU_T % Cut-off	Tonnes	CU_T		CU_S		CU_X	
			(%)	(Mlbs)	(%)	(Mlbs)	(%)	(Mlbs)
In-Pit Oxide								
Measured	0.30	11,361,000	0.96	239	0.18	45	0.78	194
Indicated	0.30	4,330,000	0.91	87	0.19	18	0.72	69
Measured + Indicated	**0.30**	**15,691,000**	**0.94**	**326**	**0.18**	**63**	**0.76**	**263**
Inferred	0.30	216,000	0.52	2.5	0.12	0.6	0.37	1.8
In-Pit Sulphide								
Measured	0.30	5,705,000	0.68	86	0.62	79	0.05	7
Indicated	0.30	13,486,000	0.72	214	0.68	201	0.04	13
Measured + Indicated	**0.30**	**19,191,000**	**0.71**	**300**	**0.66**	**280**	**0.05**	**20**
Inferred	0.30	1,675,000	0.51	19	0.46	17	0.05	2
Below Pit Sulphide								
Measured	0.60	26,000	0.71	0.41	0.68	0.39	0.03	0.02
Indicated	0.60	1,341,000	0.82	24	0.80	24	0.03	0.8
Measured + Indicated	**0.60**	**1,367,000**	**0.82**	**25**	**0.79**	**24**	**0.03**	**0.8**
Inferred	0.60	967,000	0.77	16	0.75	16	0.03	0.1

There is no other relevant data or information available that is necessary to make the technical report understandable and not misleading. The Author is not aware of any known mining, processing, metallurgical, environmental, infrastructure, economic, permitting, legal, title, taxation, socio-political, or marketing issues, or any other relevant factors not reported in this technical report, that could materially affect the updated MRE.

1.7 Mineral Reserve Estimate

No mineral reserves have been estimated for the property.

1.8 Mining Methods

The proposed mining method is conventional open pit mining. Mineralized rock and waste would be drilled, blasted, loaded by hydraulic shovels and hydraulic excavators into off-highway dump trucks, and hauled to the processing plant.

The basis for the pit design work was the mineral resource block model that was developed by SGS as part of a NI 43-101-compliant mineral resource estimate (refer to Section 14).

There are 3 primary deposits currently under consideration. Due to the nature of the deposit, the resultant pits are narrow and deep. Currently no backfilling is contemplated.

The proposed mining method is the development of a slot in front of the mineralized zone at each level. The centralized slot will enable waste mining on one side while mining mineralized material on the other side. This methodology will also facilitate separating mineralized material from waste material.

The target ROM feed to the processing plant is 7,000 tonnes per day or 2,450,000 tonnes/year. The plant feed is mineralized oxide and sulphide containing copper, gold and silver.

The combined Life of Mine of the 3 pits is 9 years.

The processing cost adjustment factor (PCAF) was set at 1, that is all mineralized blocks incur the same processing cost. The mining cost adjustment factor (MCAF) increased with depth. The models were then exported to Whittle using the revenue values as grade values.

The initial step was to design a pit shell without ramps to determine how closely the design could be matched to the Whittle shell while applying batter angles and berm widths. Whittle adds blocks to the pit until the maximum value is reached without consideration for the practicality of mining the resultant pit. This results in drop-cuts of single blocks or small groups of blocks into the pit floor. Consequently, in a narrow deposit such as Carmacks, it is not practical to design a pit as deep as the Whittle shell as the pit bottom becomes too small to deploy equipment. Removing these drop-cuts results in a more practical layout.

These initial designs were then reviewed to determine the number and location of ramps to ensure access for all operating benches. With the inclusion of the ramp system the overall highwall slope in Pit 147 is approximately 52.4°.

At this stage of the mine design (PEA) no optimization of the relevant waste dumps or topsoil stockpile dumps has been attempted. Preliminary dump designs parameters reference used was a geotechnical report completed by Golder in 2008 ("Preliminary Design Waste Rock Storage Area Carmacks Copper Project Western Copper Corporation Yukon", dated April 15, 2008), produced for both 147 and 1213. The nominal lift height assumed is 20 m.

There are numerous water courses with the pits but, no serious dewatering issues are expected. However, maximum pumping capacity has been allowed for water ingress due to rainfall and snow melt will be managed with berms and cut-off drains.

Separate sets of mining equipment are envisaged for waste mining and ROM production. The production schedule was developed using manual scheduling of the resource blocks due to the variance of recoveries based on the proportions of oxide and sulphide material within each block.

1.9 Recovery Methods

The process plant is designed to produce copper concentrate using conventional crushing, grinding, flotation, concentrate dewatering, and tailings handling unit operations. Both sulfide and oxide materials can be fed to the plant at individually or simultaneously with different blend ratios. Sodium sulfide will be added in the conditioner before each stage of flotation depending on the percentage of oxide material in the plant feed.

The nameplate capacity of the process plant is 7,000 metric tons per day. The process plant includes the following individual process steps:

- Primary crushing reducing the ROM material to a P80 of 150 mm or less
- Coarse ore stockpile with a live capacity of 7,000 metric tons
- Grinding circuit comprising of one SAG mill and one ball mill operating in closed circuit using hydrocyclones, producing a final product of 150 um.
- Rougher flotation with a rougher concentrate regrind circuit
- Two stages of cleaner flotation which produces the final concentrate
- Concentrate thickening and filtration
- Tailings handling
- Reagents preparation
- Water and power systems

1.10 Project Infrastructure

Project infrastructure will exist in the vicinity of the three open pits: Zones 147, 1213N and 1213S.

The Carmacks general site includes 3 open pits, waste dumps, mill complex, tailings management facility, electrical distribution, offices, warehouse, maintenance, and effluent treatment.

The concentrate (copper concentrate with gold and silver values) will be produced from mill processing.

1.11 Concentrate Pricing

The output from the process plant is planned at 40% copper, with varying gold and silver grades.

Pricing used in this PEA is $3.75 USD per pound of copper, $1,800 per troy ounce for gold and $22 per troy ounce for silver.

1.12 Socio-Economic and Environmental Impact

Collection of environmental baseline data for the Carmacks Project has been ongoing since 1989. The baseline studies were designed and implemented to support requirements for future planning and permitting purposes. Granite Creek Ltd has taken on active role in communicating and consulting with the local communities.

1.13 Legal and Statutory

For the Project to proceed successfully, several legislative requirements will need to be fulfilled according to the Yukon Energy, Mines and Resources and the Plan Requirement Guidance for Quart Mining Projects. A quartz mining project requires the submission of environmental protection plans and operational plans for the development, operation, and decommissioning of a mine site. These plans will describe how mining activities will be undertaken, and how they will be completed in an environmentally responsible manner. This guide is intended to assist proponents with the development of the required Plans for quartz mining projects. These plans should be considered as living, dynamic documents that will be refined throughout mine planning, development and operation as more information certainty becomes available, and as monitoring, research, analysis and design are advanced.

Mining projects in Yukon usually require both a Quartz Mining License (QML) and a Water Licence (WL). Each license will consider and address mine development and operation plans and environmental protection plans. Mine development and operation activities are inherently integrated across all environmental disciplines and regulatory requirements. Therefore, this Guide describes requirements for the required plans to meet the needs for both QML and WL processes. This guide provides overall guidance about expected contents of environmental and operational plans. This approach is intended to provide flexibility for proponents to identify and optimize plans to suit specific sites and project elements. Granite Creek Ltd. will be responsible for ensuring that the welfare of the local population is not significantly impacted upon due to the mining activities. In addition, Granite Creek Ltd. must ensure that adequate rehabilitation and closure of the mine takes place following the conclusion of the proposed mine.

To ensure that the legislative requirements are met, as well as best practices are implemented, environmental degradation and pollution must be prevented and, where unavoidable, mitigated, and managed. The predominant impacts associated with the mining activities are due to groundwater quantity, groundwater and dust contamination and the potential side effects of chemicals used in processing. Other social ills may result from the project due to the influx of job seekers causing an increase in the population of the town of Carmacks.

There are four active permits covering the Carmacks Project:

1. QML0007 – a Quartz Mining Licence granted in 2009 with an expiry April 1, 2034. The licence authorizes development and production of the Carmacks Copper Mine project as described in project #2006-0050 (assessed by YESAB) and QML application material submitted to the Yukon government. A water licence was applied for but denied by the Yukon Water board in 2010. Since there was no development or production the project went into a state of temporary closure until 2021.
2. LQ00530 Stu Copper – Class 4 Exploration permit expiring April 30, 2030.
3. LQ00433 Hoocheekoo – Class 3 exploration permit expiring Feb 6, 2023.
4. Class 1 Notification Q2021_0259 expiring July 15, 2022.

1.14 Capital Cost Estimate

The total capital costs for the Project are estimated at CAD $349,9M and include direct capital costs for mill site process plant, tailings storage facility; sustaining capital for the mill site process plant, TSF closure costs; indirect costs and contingency. Indirect costs, including EPCM, first fills, spares and a camp allowance have been estimated at 30% of Direct and Indirect plant capital costs. The summary of the initial direct capital and total capital costs are presented in Table 1-5.

Table 1-5 Capital Cost Summary

Area	Initial Capital $CAD	Sustaining Capital $CAD
Mining Directs	$13,961K	$120,202K
Mining Indirects	$7,167K	
Processing Directs	$83,445K	
Processing Indirects	$34,103K	
Non Process Infrastructure	$16,932K	
Tailings	$14,665K	
Pre-Production G&A	$3,370K	
Power Supply	$11,160K	$3,751K
Closure Costs		$5,850K
Contingency	$35,264K	
Total	$220,066K	$129,803K

1.15 Operating Cost Estimate

The overall operating costs for the Mill Site process plant, mining operation and G&A are in Table 1-6.

Table 1-6 Operating Cost Estimates

Summary costs	Life of Mine	Per Tonne Processed	Per Tonne Mined
Mining	$373,459K	$17.56	$3.16
Process	$389,322K	$18.30	
G&A	$104,819K	$4.93	

1.16 Economic Analysis

An engineering economic model was prepared for the Project to estimate annual cash flows and assess sensitivities to certain economic parameters. The economic results of this report are based upon the services performed by SGS and Granite Creek Ltd. The Project includes Three open pits, surface infrastructure to support the mine operations (maintenance and office facilities), water management features, a run-of-mine stockpiling area, processing facility, and a tailings storage facility.

The economic analysis assumes that the Project will be 100% equity financed and uses parameters relevant as of November 2022, under conditions likely to be applicable to project development and operation and analyses the sensitivity of the Project to changes in the key Project parameters. All costs have been presented in Canadian Dollars (CAN$) and wherever applicable conversion from USD has utilized an exchange ratio (CAN$/US$) of 1.33. Mining and treatment data, capital cost estimates and operating cost estimates have been put into a base case financial model to calculate the IRR and NPV based on calculated Project after tax cash flows. The scope of the financial model has been restricted to the Project level and as such, the effects of interest charges and financing have been excluded. For the purposes of the PEA, the evaluation is based on 100% of the Project cash flows before distribution of profits to the equity owners.

Based on the extraction of 2’450’000 of ROM feed from the mine, the project is anticipated to yield a pre-tax IRR of 36% with a pre-tax NPV, at a discount rate of 5% of CAD $324.1M, and an after-tax IRR of 29% with an after-tax NPV, at a discount rate of 5%, of CAD $230.5 M. Cumulative cash flows are CAD $505.9 M pre-tax and CAD $371.2 M after-tax over the nine year LOM.

The project is expected to pay back initial capital in 2 years after production starts.

Table 1-7 Summary Financial Results

	Base Case	Case 1
Pre-Tax NPV @5%	$324.1M	$475.0M
Pre-Tax IRR	36%	48%
Pre-Tax Net Cash Flow	$505.9M	$714.5M
After Tax NPV @5%	$230.5M	$330.1M
After Tax IRR @5%	29%	38%
After Tax Net Cash Flow	$371.2M	$507.4M

1.17 Conclusions

This PEA demonstrate that the Carmacks Project has the potential to be technically and economically viable as a producer of copper concentrates which will contain gold and silver as byproducts.

1.18 Recommendations

- This PEA was based on the Mineral Resource Estimate updated by SGS Geological Services on February 25, 2022. Recommendations include undertaking further exploration to expand the current resource and to upgrade the quality of the resource in order to derive mineable reserves. Significant upside potential exists down dip of the Zone 2000S and Zone 1213 as well as along the several kilometer long strike extensions of the mineralization in both areas. Further exploration could increase the life of mine.
- A SAG mill – ball mill circuit is recommended for project comminution. JK drop weight testing and SAG Mill Comminution (SMC) tests are recommended to size the SAG mill. Bond Ball Mill Work Index and Axb studies are recommended to size the ball mill. In addition, sedimentation and filtration tests are required to size the concentrate thickener and filter and the flotation tailings also require sedimentation testing for tailings thickener sizing.

- The current flotation metal recovery model is based on the acid soluble copper percentage in the material. It is recommended to explore the impact of head grade upon the metal recoveries in future.
- No geotechnical study has yet been undertaken on zones 2013 and 2000S. This should be done if the project advances to PFS as this will have a material impact on the stripping ratio. A preliminary hydrological study should be commissioned to validate the assumption that there is no water related issues at depth.
- Previous reports on the Carmacks Copper project included designs and estimates for a tailings storage facility, based on a conventional tailings dam. This PEA assumes the installation of a dry stack tailings system, in the same location as the previous design. The envisioned concept will be a double liner system with underdrains and progressive reclamation as the stack progresses. It is recommended to be updated and completed at a pre-feasibility level and alternative assessment for the TMF location be considered including overall risk to the operation, waste, and water management practice.
- Carry out a six-month PFS to further develop the engineering design of the plant and recognise value engineering where possible.
- Revisit the capital cost estimates in general for possible savings due to optimising the cost estimates from ± 50% to ± 10% (PFS Level).
- To advance the Carmacks project towards the next stage of Engineering, the proposed budget is estimated at CAD$ 9 million, involves a major upgrade drilling program, water supply studies, geotechnical/hydrogeological, TMF optimization studies, continued environmental and community liaison, mineral processing (metallurgy), power studies. mine access study and an engineering study.

1.19 Opportunities

- The third conceptual pit, 2000S as identified in the Mineral Resource Estimate (“MRE), could be brought into the mine plan if sufficient additional resources were defined by drilling to offset pre-stripping costs.
- Electrification of the mining fleet. Significant cost saving and reduction in greenhouse gas production may be possible through the sourcing of electric vs. diesel haul trucks for the Project. The PEA envisions using a contract mining fleet for the Project and preference will be given to suppliers that can provide either fully electric or hybrid equipment.
- Further discovery. Exploration conducted in 2022 consisting of geophysics, trenching and soil sampling identified four areas proximal to the proposed mine plan that if successfully drilled could enable longer mine life beyond nine years or provide additional sulphide mill feed earlier in the mine’s life. Four targets on the Property require evaluation, all located within 1km of the current deposits. Two of the targets are located beneath the current resource and there is higher geological certainty that these may contain appreciable copper mineralization.
  - Zone 1213 shallow:

    Downward continuation of Zone 12 and 13. Estimated dimensions are 360m long, 15 – 40m wide, starting at approximately 65m below the current drilling.
  - Zone 12 deep:

    Downward continuation of Zone 12. Estimated from geophysics to be continuing for an additional 170m below current resource modelling. Approximated to be 580m long and 15-40m wide.
  - Gap Zone target:

Geophysical anomaly that fits with current geological understanding of the fault offset between 147 and 2000S Zone. Estimated to be 500m long, up to 400m deep, and 30-50m wide.

- Sourtoe target:

  Estimated from geophysics to be a lensoidal body of similar size to known deposits at 370m long x 370m deep with an estimated width of 15-50m. It has been lightly tested at surface by trenching and is weakly mineralized.

In addition, the Carmacks North target area is host to several mineralized zones that have the potential to add resources to the mine plan, all within 15 km of the proposed mill site.

- Additional recovery through metallurgical improvements. The Company has retained Kemetco Laboratories to complete additional leaching and copper precipitating testing to evaluate the processing of tailings. The calculated grade of copper in tailings averages 0.32% with over 140 Mlbs of copper not recovered LOM. Recovery sensitivity show an additional $180M pretax NPV based of a 20% increase in recovery rates. Review of historical metallurgical testing has indicated that copper minerals present in oxidized material respond well to leaching. Once the copper is in solution the copper would be chemically precipitated to produce sulphide minerals that can be added back into the flotation cells.

2 INTRODUCTION

This Report was prepared and compiled by the QPs under employment with SGS and Consultants at the request of Granite Creek Ltd. (the "Company" or "GCX). The purpose of this Report is to provide a Preliminary Economic Assessment ("PEA") for the Carmacks Project ("Carmacks" or the "Project") in Yukon. This PEA aims at a significantly larger annual run-of-mine and plant throughput of 7,000 tonnes per day (2 million tonnes per year) and longer mine life than the historical PEA of 2016 by mining from three sub-deposits namely "147", "2000S" and "1213". Further, the processing flow sheet was simplified to a direct flotation of the run-of-mine material instead of heap leaching. Concentrate copper as final product.

This Technical Report has been prepared to comply with disclosure and reporting requirements set forth in the Toronto Venture Exchange (TSX-V) Corporate Finance Manual, Canadian National Instrument 43-101, Companion Policy 43-101CP, Form 43-101F1, the 'Standards of Disclosure for Mineral Projects' of January 2006 (the Instrument) and the Mineral Resource and Reserve classifications adopted by CIM Council in August 2000.

Granite Creek Ltd (GCX) is a Canadian company listed on the TSX Venture Exchange which holds a diversified portfolio of projects in Canada. The company is a member of the Metallic Group of Companies which is a collaboration of leading precious and base metals exploration companies with a portfolio of large, brownfields assets in established mining districts adjacent to some of the industry's highest-grade copper, silver, and platinum/palladium producers. Member companies include Granite Creek Copper (TSX-V: GCX) in the Yukon's Carmacks copper district, Metallic Minerals (TSX-V: MMG) in the Yukon's Keno Hill Silver District, and Group Ten Metals (TSX-V: PGE) in the Stillwater PGM-Ni-Cu district of Montana. The Company's registered head Office is Suite 904-409 Granville Street, Vancouver, BC Canada V6C 1T2.

The current PEA is based on a Mineral Resource Estimate ("MRE") for the Project previously reported by the Company, for the 147, 2000S and 1213 zones. On March 15, 2022 the Company announced the Carmacks Project MRE to contain:

- In-Pit Oxide – 15.7 million tonnes in Measured and Indicated categories, grading 0.94% Cu, 0.36 g/t Au, 3.23 g/t Ag and 0.01% Mo
- In-Pit Sulphide – 19.2 million tonnes in Measured and Indicated categories, grading 0.71% Cu, 0.18 g/t Au, 2.74 g/t Ag and 0.01% Mo
- Below Pit Sulphide – 1.4 million tonnes in Measured and Indicated categories, grading 0.82% Cu, 0.19 g/t Au, 2.88 g/t Ag and 0.01% Mo

There has been no drilling completed on the Property since the release of the MREs and the MREs are considered current. Details on the MREs are presented in Section 14.

2.1 Purpose of Report

The purpose of this Report is to publish a Technical Report on the Carmacks Project summarizing:

- the land tenures, exploration history, and drilling,
- the mineral resource estimates at Zones 147, 2000S and 1213 open pits;
- a conceptual mine plan at a level to support a Preliminary Economic Assessment;
- the supporting infrastructure including, power, buildings, tailings management facility;
- processing plant, etc. to support the conceptual mine plan;
- the environmental permitting requirements;
- capital expenditure and operating expenditure estimates;

- a financial model and perform an economic analysis and;
- provide recommendations and additional work.

2.2 Terms of Reference

Granite Creek Ltd. engaged the services of SGS and authors on May 11, 2022, to write an independent NI 43-101 Technical Report on the Carmacks Project in Yukon Territory. This Report was prepared in accordance with NI 43-101 and Form NI 43-101F1 and Companion Policy 43 101CP.

2.3 Qualifications of Consultants

SGS and Consultants preparing this technical report are specialists in the fields of geology, exploration, mineral resource estimation, open pit mining, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, civil, mechanical, electrical, capital, and operating cost estimation, and mineral economics.

None of the Consultants or any associates employed in the preparation of this report has any beneficial interest in GCX. The Consultants are not insiders, associates, or affiliates of Granite Creek Ltd. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between GCX, SGS and the Consultants. The Consultants are being paid a fee for their work in accordance with normal professional consulting practice

2.4 Report Responsibility and Qualified Persons

The following individuals, by virtue of their education, experience, and professional association, are considered Qualified Persons (QP) as defined in the NI 43-101 standard, for this report, and are members in good standing of appropriate professional institutions:

- SGS Canada Inc. under the supervision of Allan Armitage P. Geol.

  Sections of the Report dealing with Property Description and Location (Item 4), Accessibility, Climate, Local Resources, Infrastructure and Physiography (Item 5), History (Item 6), Geological Setting and Mineralisation (Item 7), Deposit Types (Item 8), Exploration (Item 9), Drilling (Item 10), Sample Preparation, Analyses and Security (Item 11), Data Verification (Item 12), Mineral Resource Estimate (Item 14), and Adjacent Properties (Item 23).

- SGS Canada under the supervision of William van Breugel (B.Sc. Hons, P.Eng.)

  Sections of the Report dealing, Mining Methods (Item 16.12), Project Infrastructure (Item 18.7), Market Studies and Contracts (Item 19), Environmental Studies, Permitting and Social or Community Impact (Item 20), Capital and Operating Costs (Item 21), and Economic Analysis (Item 22).

- SGS Canada under the supervision of Johnny Canosa (B.Sc., P.Eng.):

  Sections of the Report dealing with Introduction (Item 2), Reliance on Other Experts (Item 3), Mining Methods (Item 16 except for Item 16.12) Project Infrastructure (Item 18 except for Item 18.7), Other Relevant Data and Information (Item 24), Interpretation and Conclusions (Items 25.3 and 25.4), Recommendations (26), and References (27).

- SGS Canada under the supervision of Joseph Keane (Pr Eng):

Sections of the Report dealing with Mineral Processing and Metallurgical Testing (Item 13), and Recovery Methods (Item 17).

The preceding QPs have contributed to the writing of this Report and have provided QP certificates, included at the end of this Report. The information contained in the certificates outlines the sections in this Report for which each QP is responsible. Each QP has also contributed figures, tables, and portions of Sections 1 (Summary), 2, (Introduction), 3 (Reliance on other Experts), 25 (Interpretation and Conclusions), 26 (Recommendations), and 27 (References). Table 2-1 outlines the responsibilities for the various sections of the Report and the name of the corresponding Qualified Person.

2.5 Site Visit

Personal inspections made by the Qualified Persons and their items of responsibility for this report are shown in Table 2-1.

Table 2-1 Details of Site Visits and Responsibilities of the Qualified Persons

Qualified Person	Personal Site Inspection Dates	Items Responsible for
Allan Armitage	November 9, 2021	1.2 to 1.4, 1.6, 4 to 12, 14, 23 and 25.1
Johnny Canosa	June 20, 2022	1.1, 1.7, 1.8, 1.10, 1.13, 1.17, 1.18, 2, 3, 16 except for 16.12, 18 except 18.7, 24, 25.3, 25.4, 26 and 27
William van Breugel	-	1.11, 1.12, 1.14, 1.15, 1.16, 16.12, 18.7, 19, 20, 21, 22, and 25.2
Joseph Keane	-	1.5, 1.9, 13 and 17

2.6 Currency, Units, Abbreviations and Definitions

All units of measurement used in this technical report are International System of Units (SI) or metric, except for Imperial units that are commonly used in industry (e.g., ounces (oz.) and pounds (lb.) for the mass of precious and base metals). All currency is in Canadian dollars (CAD), unless otherwise noted. Frequently used abbreviations and acronyms can be found in Table 2-2. This Report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs consider them immaterial.

Table 2-2 List of Abbreviations

%	Percent sign	kg	Kilograms
°	Degree	km	Kilometres
°C	Degree Celsius	km^2	Square kilometre
°F	Degree Fahrenheit	m	Metres
µm	Micron	m^2	Square metres
AA	Atomic absorption	m^3	Cubic metres
Ag	Silver	mm	Millimetre
Au	Gold	mm^2	square millimetre
Az	Azimuth	mm^3	cubic millimetre
CAD$	Canadian dollar	Mo	Molybdenum
cm	Centimetre	Moz	Million troy ounces
cm^2	square centimetre	MRE	Mineral Resource Estimate
cm^3	cubic centimetre	Mt	Million tonnes
Cu	Copper	NAD 83	North American Datum of 1983
Cu_S	Copper Sulphide	NQ	Drill core size (4.8 cm in diameter)
Cu_T	Copper Total	ozt	Troy ounce (31.1035 grams)
Cu_X	Copper Oxide	ppb	Parts per billion
CuEq	Copper equivalent grade	ppm	Parts per million
DDH	Diamond drill hole	QA	Quality Assurance
ft	Feet	QC	Quality Control
ft^2	Square feet	QP	Qualified Person
ft^3	Cubic feet	RC	Reverse circulation drilling
g	Grams	RQD	Rock quality description
g/t or gpt	Grams per Tonne	SG	Specific Gravity
GPS	Global Positioning System	Tonnes or T	Metric tonnes
Ha	Hectares	US$	US Dollar
ha	Hectare	UTM	Universal Transverse Mercator
HQ	Drill core size (6.3 cm in diameter)	ICP	Induced coupled plasma

2.7 Effective Date

The effective date of this technical report is January 19, 2023.

As of the effective date of this Report, the authors are not aware of any material fact or material change with respect to the subject matter of this Technical Report that is not presented herein, or which the omission to disclose could make this Report misleading.

2.8 Previous Technical Reports

A Preliminary Economic Assessment (PEA) on the Project was completed by JDS Energy & Mining Inc. on October 12, 2016, titled "NI 43-101 Preliminary Economic Assessment Technical Report on The Carmacks Project, Yukon, Canada". Information considered by the QPs to be both current and relevant was sourced from this document. The 2022 Mineral Resource Estimate reported in this current Technical Report is substantially different to that on which the 2017 PEA was completed and therefore the results of the 2017 PEA are not considered current and are no longer relevant.

The sources of information as referenced throughout this report are as follows:

- Optimized Resource model based on the metal recovery curve as provided by William Van Breugel.
- Metallurgical Test report dated December 12, 2022, by SGS Vancouver including the mineralogy and flotation testwork
- Data supplied by Granite Creek Ltd.;
- Technical report by SGS Geological Services on mineral resource estimate and associated geological background information dated April 29, 2022, titled Updated Mineral Resources Estimates for the Carmacks Cu-Au-Ag Project;
- Information provided by MDA Consulting Ltd. March 2000 Environmental Assessment, Western Coppermine/Williams Creek, Yukon Region-Final Report, Report No.:5 MDA Project no. 99-09-03. This report was used as reference by SGS to prepare Section 20;
- Information provided by Golder Associates on Tailings Management Facility titled Tailings Storage Conceptual Design dated May 3, 2016. Report Number: 1417183-4000 Distribution;
- Information provided by Golder Associates "Draft Report on Open Pit Slope Design Carmacks Copper project", dated October 22, 2008;
- Information provided by Golder Associates "Preliminary Design Waste Rock Storage Area Carmacks Copper Project Western Copper Corporation Yukon", dated April 15, 2008.

3 RELIANCE ON OTHER EXPERTS

Final information concerning claim status and ownership of the Carmacks Property, which is presented in Section 4 below, has been provided to Armitage by Debbie James for Granite Creek on April 18, 2022, by way of e-mail.

Armitage only reviewed the land tenure in a preliminary fashion (location and number of claims and leases, total area and expiry dates) and has not independently verified the legal status or ownership of the Carmacks Property or any underlying agreements. However, Armitage has no reason to doubt that the title situation is other than what is presented in this technical report. Armitage is not qualified to express any legal opinion with respect to Carmacks Property titles or current ownership.

The QPs have reviewed and analyzed data and reports provided by Granite Creek Copper, together with publicly available data, drawing its own conclusions augmented by direct field examination.

The QPs who prepared this report relied on information provided by experts who are not QPs. The QP believes that it is reasonable to rely on these experts, based on the assumption that the experts have the necessary education, professional designations, and relevant experience on matters relevant to the technical report.

Final information concerning claim status and ownership of the Carmacks Property, which is presented in Section 4 below, has been provided to the Author by Granite Creek Copper on April 18, 2022, by way of e-mail.

SGS has relied upon Environmental Assessment, Western Copper/Williams Creek, Yukon Region-Final Report by MDA Consulting Ltd Environmental Solutions and JDS Energy Mining (NI 43-101 PEA Report on the Carmacks Project) matters pertaining to the Summary of Environmental Impact Assessment Report and special Studies for the Carmacks Mining Project dated November 25, 2016, as disclosed in Section 20.

Johnny Canosa, P. Eng. (SGS) has relied upon Golder Associates, SGS Bateman who completed an independent analysis on the TMF and Process Plant for the infrastructures summarized in Section 18.

William van Breugel, P. Eng. (SGS) has relied upon Granite Creek Copper who supplied pricing forecast for this PEA and derived from recent market analysis and other published NI 43-101 complaint resource reports on selling prices for Copper Concentrate (with Gold and Silver by product), as summarized in Section 19.

William van Breugel, P. Eng. (SGS) has relied upon the Tailings Storage Conceptual Design provided by Golder Associates who completed an independent analysis on the Tailings Storage Facility for the data used in the Capital and Operating Expenses estimate as summarized in Sections 21.

William van Breugel has relied upon previously published reports with respect to Environmental Studies, Permitting And Social Or Community Impact as shown in Section 20.

William van Breugel, P. Eng. (SGS) has relied upon Shaohai Yu and SGS Bateman, who completed an independent analysis on the Process Plant quantities and costs for the data used in the Capital and Operating Expenses estimate, and Economic Analysis as summarized in Sections 21 and 22.

The QPs have assumed, and relied on the fact, that all the information and existing technical documents listed in the References Section 27 of this report are accurate and complete in all material aspects. While the QPs reviewed all the available information presented, we cannot guarantee its accuracy and completeness. The QPs reserve the right, but will not be obligated, to revise the report and conclusions, if additional information becomes known subsequent to the date of this report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Carmacks Property (including Carmacks and Carmacks North properties) is located approximately 47 km northeast of the village of Carmacks, and approximately 210 km northwest of Whitehorse (Figure 4-1). The Carmacks Property occurs in NTS map sheet 115/I07. The Carmacks Project camp is centered at approximately 62°20'N latitude and 136°42'W longitude, or 411900E/6912700N in UTM co-ordinates (NAD83 Zone 8N). The Carmacks Project is within 20 km of grid power and paved highway (Figure 4-2), and the Minto Mine is located 42 km northwest of the Carmacks Camp.

4.2 Property Description, Ownership and Royalty

The combined projects includes 974 claims that covers approximately 17,580 hectares (175.80 square km) (Figure 4-3, Figure 4-4, Figure 4-5). All are 100% owned by Granite Creek (subject to certain net smelter return ("NSR") royalties) (Table 4-1).

4.2.1 Carmacks Property

The Company acquired 100% of the Carmacks Project, an oxide copper, gold, and silver deposit located in Yukon, Canada, through its acquisition of Copper North. At November 30, 2021, $1.8 million has been paid in advance royalty payments. The Company is required to make an advance royalty payment of $100,000 in any year in which the average daily copper price reported by the London Metal Exchange is US$1.10 per pound or greater (Paid subsequent to November 30, 2021). Any production from the Carmacks Project is subject to either a 15% net profits interest or a 3% net smelter return royalty, at the Company's election. If the Company elects to pay the net smelter return royalty, it has the right to purchase the royalty for $2.5 million, less any advance royalty payments made to that date. Subsequent to November 30, 2021, the Company paid $100,000 as advance royalty payment.

4.2.2 Carmacks North Property (formerly the Stu Property)

In January 2019, the Company acquired an undivided 100% interest in the Stu Property in consideration for an aggregate of 3,000,000 units (each, a "Transaction Unit") at $0.075 per Transaction Unit valued at $225,000 and a 3% net smelter return royalty to the vendors on any future production on the Stu Property (the "Royalty"). Granite Creek has the option to purchase up to two-thirds of the Royalty from the vendors. The Company will also make annual advance Royalty payments of $30,000 to the vendors beginning in May 2022, and in each subsequent year thereafter until the commencement of any commercial production on the Stu Property.

Each Transaction Unit was comprised of one common share and one share purchase warrant, with each warrant exercisable into one additional common share at an exercise price of $0.15, with an expiry of January 16, 2022. Subsequently 2,500,000 warrants were exercised and 500,000 expired unexercised.

Figure 4-1 Carmacks Property Location Map

Figure 4-2 Carmacks Property Location with respect to a Major Highway and Power

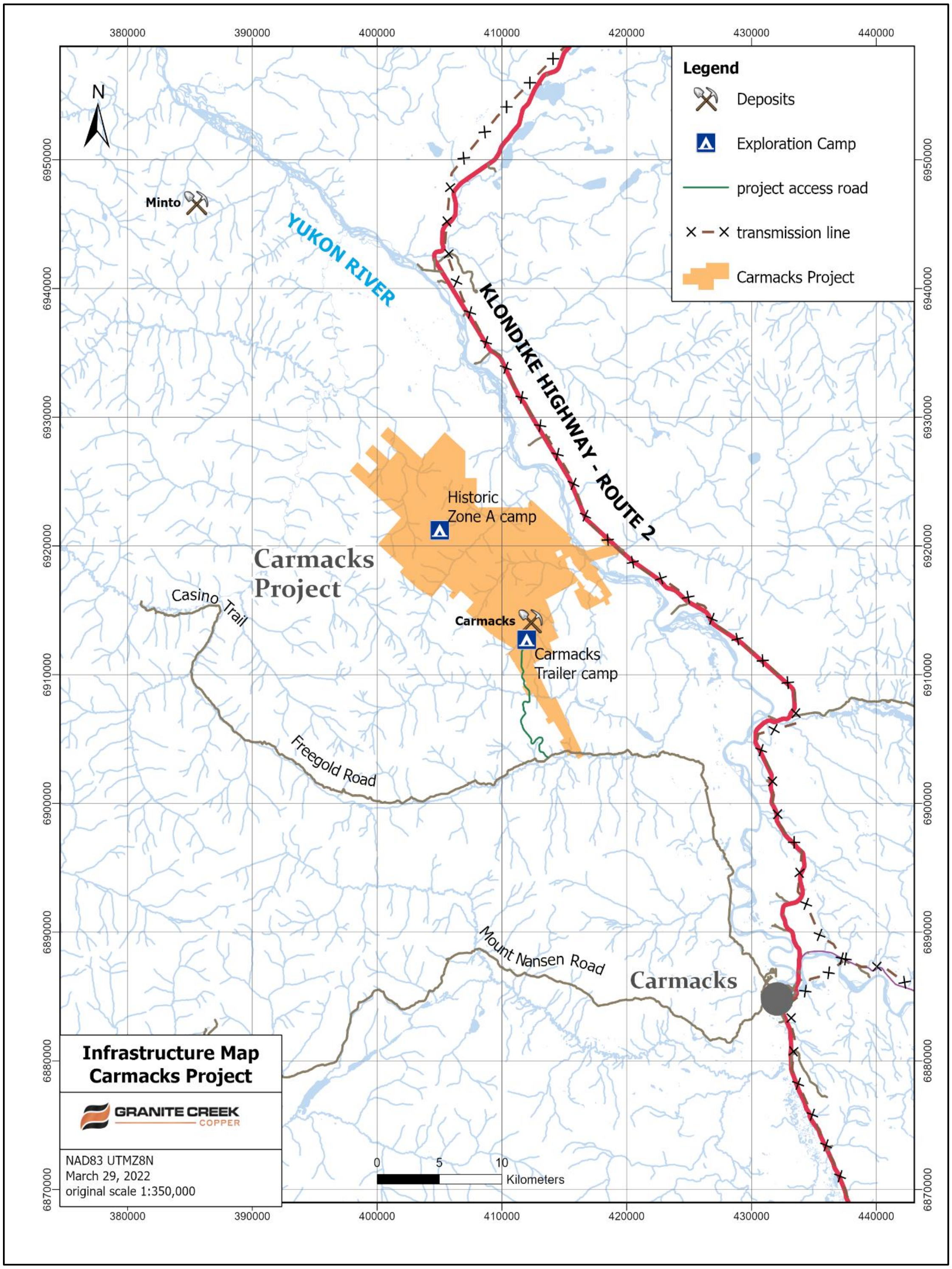


Figure 4-3 Carmacks Project Property Map

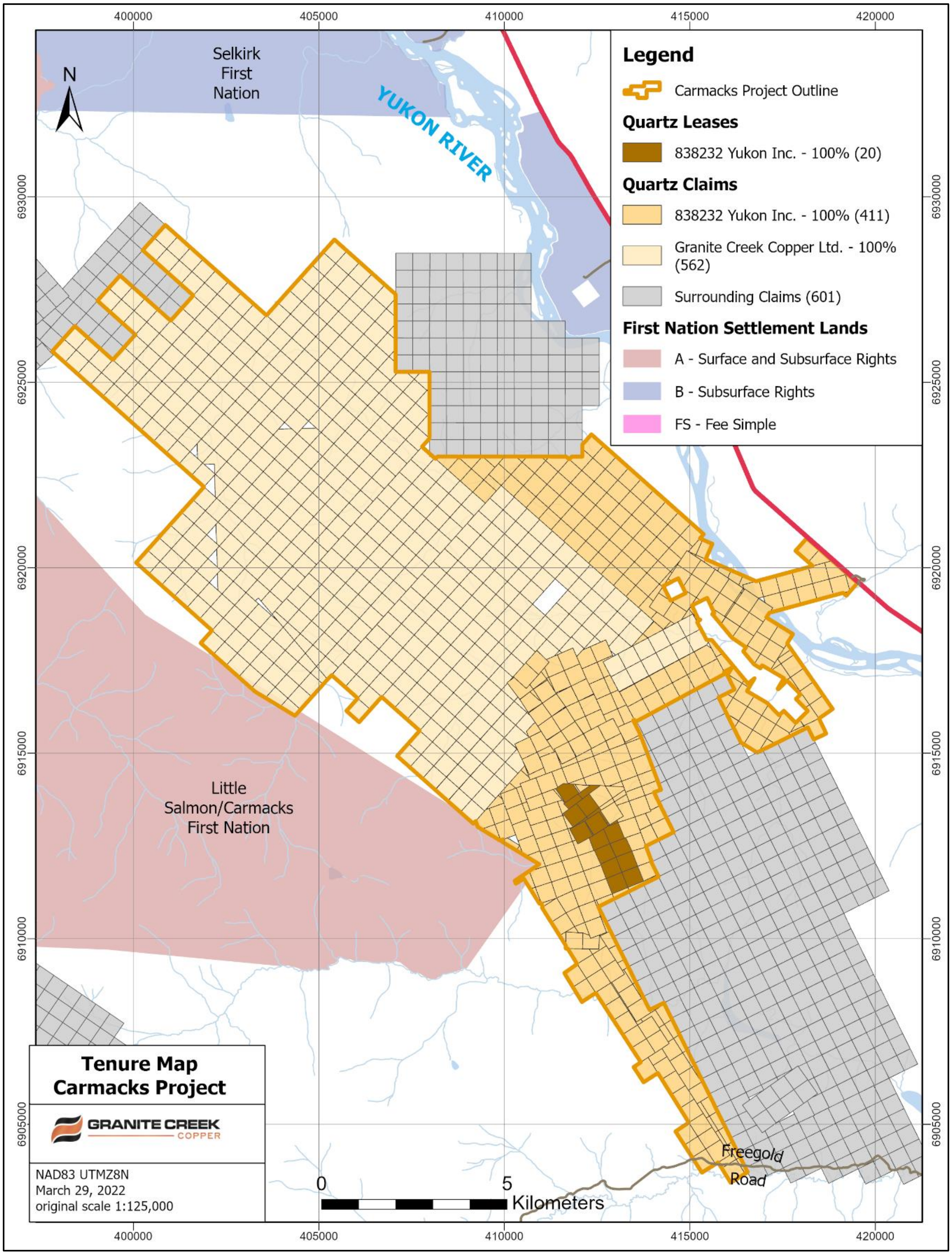


Figure 4-4 Carmacks and Carmacks North Project Tenure Map – South Half

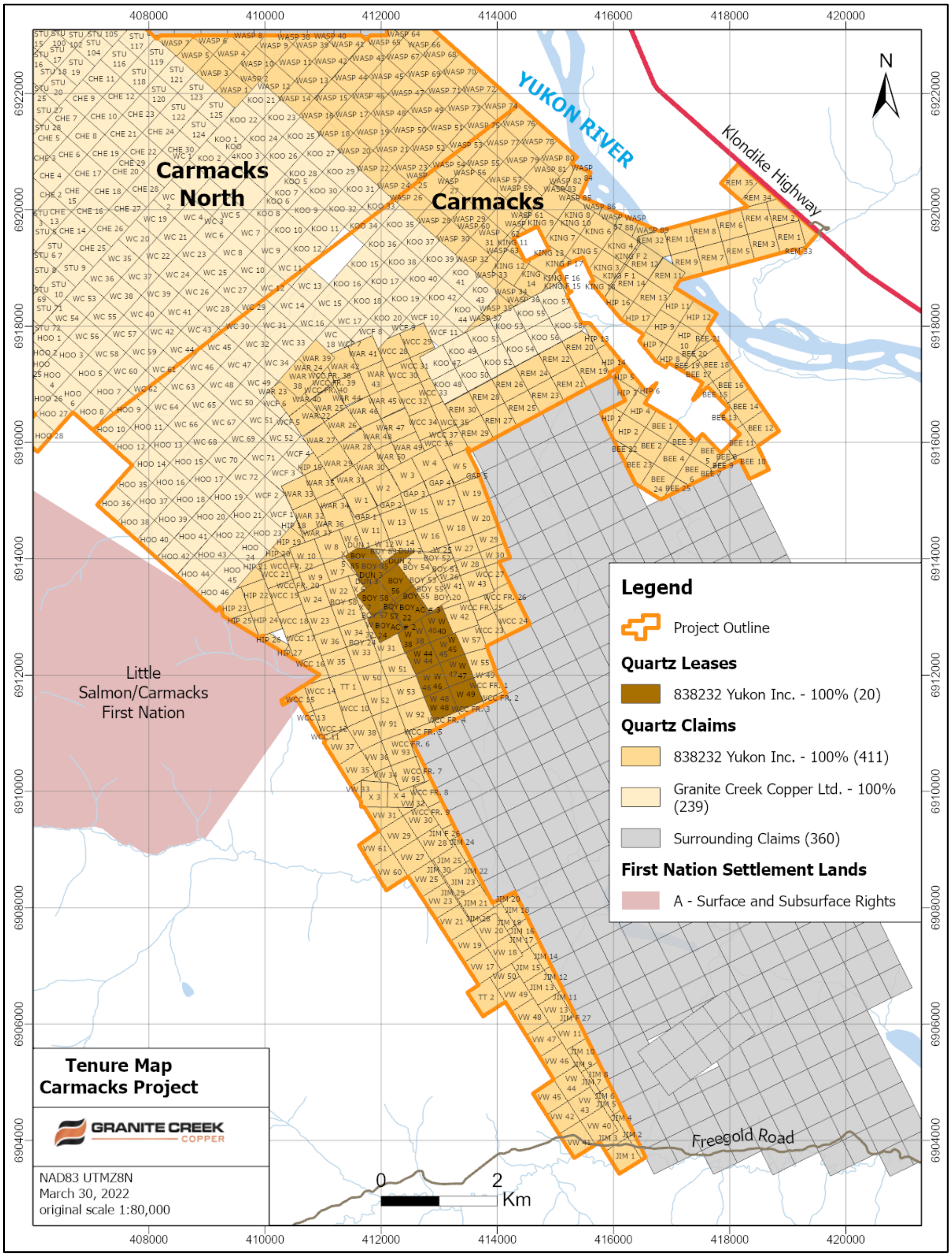


Figure 4-5 Carmacks and Carmacks North Project Tenure Map – North Half

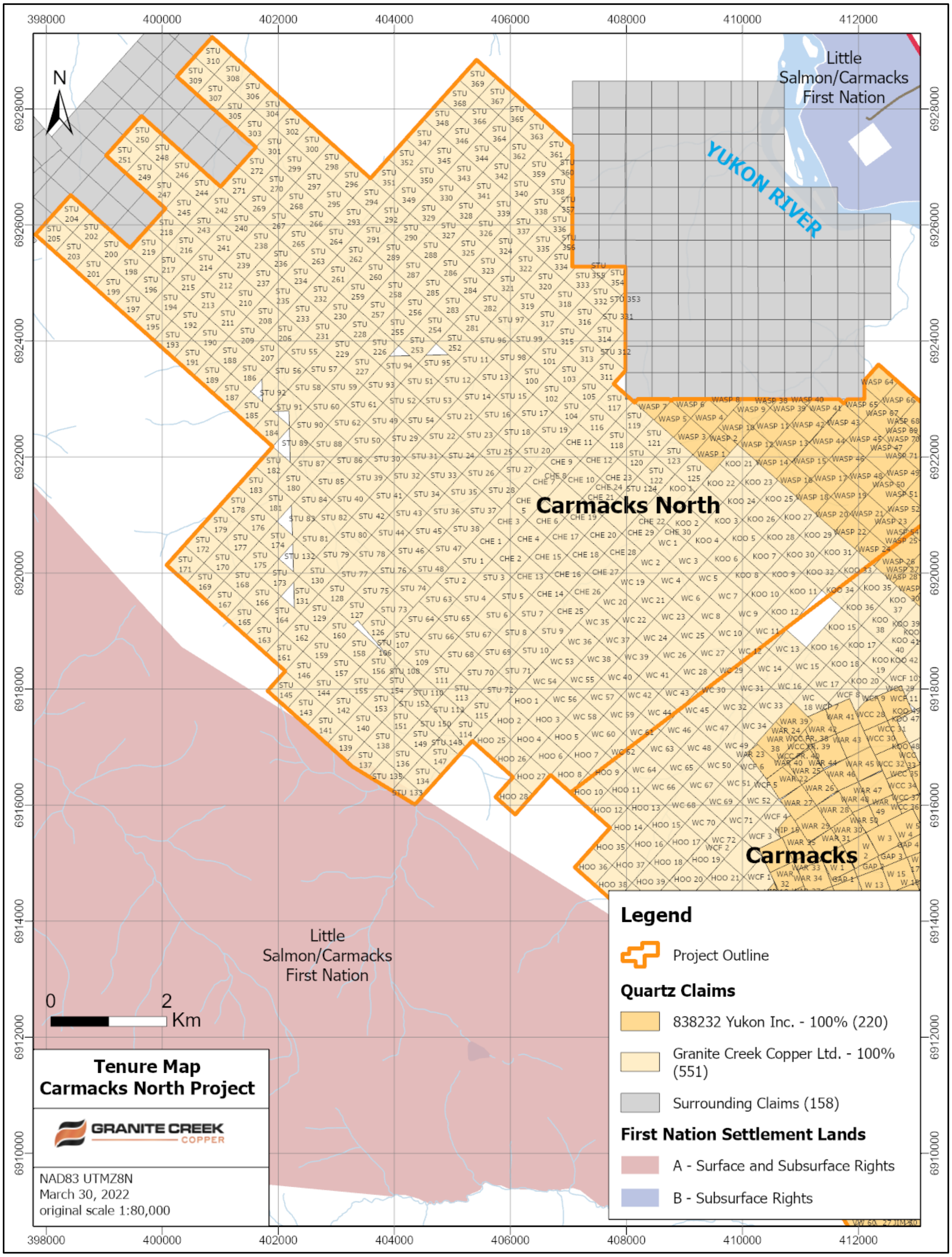


Table 4-1 Carmacks and Carmacks North Listing of Claims and Leases

Grant Number	Owner	Claim label	No. of claims	Date staked	Expiry date
YC37770-779	Granite Creek Copper Ltd.	STU 1-10	10	Dec 11, 2004	Dec 13, 2030
YC40249-258	Granite Creek Copper Ltd.	STU 11-20	10	Sep 6, 2005	Dec 13, 2030
YC37788-795	Granite Creek Copper Ltd.	STU 21-28	8	Dec 11, 2004	Dec 13, 2030
YC40259-260	Granite Creek Copper Ltd.	STU 29-30	2	Sep 6, 2005	Dec 13, 2030
YC37780-787	Granite Creek Copper Ltd.	STU 31-38	8	Dec 11, 2004	Dec 13, 2030
YC40261-276	Granite Creek Copper Ltd.	STU 39-54	16	Sep 6, 2005	Dec 13, 2030
YC40201-218	Granite Creek Copper Ltd.	STU 55-72	18	July 30, 2005	Dec 13, 2030
YC65256-315	Granite Creek Copper Ltd.	STU 73-132	60	July 6, 2007	Dec 13, 2029
YE91341-427	Granite Creek Copper Ltd.	STU 133-219	87	Dec 11-12, 2017	Dec 13 and 21, 2028
YE91434-480	Granite Creek Copper Ltd.	STU 226-272	47	Dec 11-12, 2017	Dec 13 and 21, 2028
YE91489-556	Granite Creek Copper Ltd.	STU 281-348	68	Dec 11-12, 2017	Dec 13 and 21, 2028
YF29049-069	Granite Creek Copper Ltd.	STU 349-369	21	June 18, 2019	Dec 26, 2027
YF20773-800	Granite Creek Copper Ltd.	HOO 1-28	28	July 4-5, 2014	Dec 13, 2029
YF46387-398	Granite Creek Copper Ltd.	HOO 35-46	12	July 5, 2014	Dec 13, 2029
YF46357-380	Granite Creek Copper Ltd.	CHE 1-24	24	July 5, 2014	Dec 13, 2029
YF46401-406	Granite Creek Copper Ltd.	CHE 25-30	6	July 27, 2014	Dec 13, 2029
YF46501-512	Granite Creek Copper Ltd.	KOO 1-12	12	July 5, 2014	Dec 13, 2029
YF46515-544	Granite Creek Copper Ltd.	KOO 15-44	30	July 5-6, 2014	Dec 13, 2029
YF46547-552	Granite Creek Copper Ltd.	KOO 47-52	6	July 6, 2014	Dec 13, 2029
YF46553-556	Granite Creek Copper Ltd.	KOO 53-56	4	July 6, 2014	Dec 13, 2030
YF46399-400	Granite Creek Copper Ltd.	KOO 57-58	2	July 6, 2014	Dec 13, 2030
YF20701-772	Granite Creek Copper Ltd.	WC 1-72	72	Jul 3-4, 2014	Dec 13, 2029
YF46407-417	Granite Creek Copper Ltd.	WCF 1-11	11	Jul 29, 2014	Dec 13, 2029
		TOTAL	**562**		

Grant Number	Owner	Lease	Claim label	No.	Date staked	Expiry date
Y 91722-723	838232 Yukon Inc.	OW00375-376	AC #2-3	2	Nov 4, 1974	Oct 28, 2040
YF50001-025	838232 Yukon Inc.		BEE 1-25	25	June 5, 2016	Mar 9, 2029
Y 51118	838232 Yukon Inc.		BOY 20	1	Feb 22, 1970	Mar 9, 2030
Y 51120, 122	838232 Yukon Inc.	OW00366, 367	BOY 22, 24	2	Feb 22, 1970	Oct 28, 2040

Grant Number	Owner	Lease	Claim label	No.	Date staked	Expiry date
Y 51149-152	838232 Yukon Inc.		BOY 51-54	4	Feb 23, 1970	Mar 9, 2030
Y 51153-156	838232 Yukon Inc.	OW00368-371	BOY 55-58	4	Feb 23, 1970	Oct 28, 2040
Y 51181	838232 Yukon Inc.		BOY 83	1	Feb 23, 1970	Mar 9, 2030
Y 51183	838232 Yukon Inc.	OW00372	BOY 85	1	Feb 23, 1970	Oct 28, 2040
Y 59382	838232 Yukon Inc.		DUN 1	1	Oct 17, 1970	Mar 9, 2030
Y 59383-384	838232 Yukon Inc.		DUN 2-3	2	Oct 17, 1970	Oct 28, 2040
YC65320-324	838232 Yukon Inc.		GAP 1-5	5	July 6, 2007	Mar 9, 2030
YC65554-559	838232 Yukon Inc.		HIP 1-6	6	Aug 7, 2007	Mar 9, 2030
YC65560-565	838232 Yukon Inc.		HIP 7-12	6	Aug 7, 2007	Mar 9, 2028
YC65566-567	838232 Yukon Inc.		HIP 13-14	2	Aug 7, 2007	Mar 9, 2031
YC65568	838232 Yukon Inc.		HIP 15	1	Aug 7, 2007	Mar 9, 2030
YC65569	838232 Yukon Inc.		HIP 16	1	Aug 7, 2007	Mar 9, 2027
YC65570	838232 Yukon Inc.		HIP 17	1	Aug 7, 2007	Mar 9, 2028
YC65571-580	838232 Yukon Inc.		HIP 18-27	10	Aug 7, 2007	Mar 9, 2030
YC66844-873	838232 Yukon Inc.		JIM 1-25, 28-30	28	Mar 4, 2008	Mar 9, 2030
YC66869-870	838232 Yukon Inc.		JIM F 26-27	2	Mar 4, 2008	Mar 9, 2030
YF57282-293,297	838232 Yukon Inc.		KING 3-14, 18	13	Mar 29, 2017	Mar 31, 2027
YF57280-281	838232 Yukon Inc.		KING F 1-2	2	Mar 29, 2017	Mar 31, 2027
YF57294-296	838232 Yukon Inc.		KING F 15-17	3	Mar 29, 2017	Mar 31, 2027
YC39221-234	838232 Yukon Inc.		REM 1-14	14	Apr 8, 2005	Apr 11, 2028
YC39239-250	838232 Yukon Inc.		REM 19-30	12	Apr 7, 2005	Mar 9, 2031
YC39251	838232 Yukon Inc.		REM 32	1	Apr 8, 2005	Apr 11, 2027
YC39252-254	838232 Yukon Inc.		REM 33-35	3	Apr 8, 2005	Apr 11, 2028
YB97068	838232 Yukon Inc.		TT 1	1	Dec 20, 1996	Mar 9, 2030
YB97251	838232 Yukon Inc.		TT2	1	Jan 10, 1997	Mar 9, 2030
YB96620, 622,626-630, 632, 634, 636-647	838232 Yukon Inc.		VW 11,13, 17-21, 23, 25, 27-38	21	Oct 6, 1996	Mar 9, 2030
YB96986-998	838232 Yukon Inc.		VW 40-50, 60-61	13	Dec 4, 1996	Mar 9, 2030
YB26708-744	838232 Yukon Inc.		W 1-37	38	Aug 21, 24, 25, 1989	Mar 9, 2030
YB26745-747	838232 Yukon Inc.	OW00377-379	W 38-40	3	Aug 25, 1989	Oct 28, 2040
YB26248-750	838232 Yukon Inc.		W 41-43	3	Aug 28, 1989	Mar 9, 2030
YB26751-755	838232 Yukon Inc.	OW00380-384	W 44-48	5	Aug 25, 1989	Oct 28, 2040
YB26756	838232 Yukon Inc.	OW00080	W49	1	Aug 25, 1989	Mar 9, 2025
YB36249-252,254,256	838232 Yukon Inc.		W 50-53,55,57	6	July 27-28, 1991	Mar 9, 2030

Grant Number	Owner	Lease	Claim label	No.	Date staked	Expiry date
YB36929-933	838232 Yukon Inc.		W 91-93, 95	4	July 2, 1992	Mar 9, 2030
Y 59373	838232 Yukon Inc.		WAR 22	1	Oct 16, 1970	Mar 9, 2032
YB36240-248	838232 Yukon Inc.		WAR 23-31	9	July 28, 1991	Mar 9, 2030
YB36446-451	838232 Yukon Inc.		WAR 32-37	6	Sept 10, 1991	Mar 9, 2032
YB36765-777	838232 Yukon Inc.		WAR 38-50	13	Feb 22, 1992	Mar 9, 2030
YF50879-967	838232 Yukon Inc.		WASP 1-89	89	July 16-20, 2016	Mar 9, 2027
YC60390-401,403-404,407-413	838232 Yukon Inc.		WCC 10-19,21,23-24,27-33	20	Apr 29, May 1, 8, 2007	Mar 9, 2030
YC60414-417	838232 Yukon Inc.		WCC 34-37	4	May 8, 2007	Mar 9, 2031
YC60381-389,400,402,405-406,418-420	838232 Yukon Inc.		WCC FR. 1-9,20,22,25-26,38-40	16	Apr 27-30 & May 1,7,2007	Mar 9, 2030
YB36898-899	838232 Yukon Inc.		X 3-4	2	June 12, 1992	Mar 9, 2030
YB36962-964	838232 Yukon Inc.		X 5-7	3	Aug 1, 1992	Mar 9, 2032
			TOTAL	**412**		

4.3 Mineral Rights in Yukon Territory

In the Yukon there is no requirement to obtain a prospecting license from the Mining Recorder for the right to prospect for the purposes of staking a claim, or to undertake the staking of a claim. Any individual who is 18 years of age or older, or an individual authorized by any corporation authorized to carry out business in the Yukon, or anyone on the behalf of someone else who is at least 18 years of age may prospect on available lands for mining purposes to locate, prospect, and mine for gold and other precious minerals or gemstones.

Regulations for hard rock mineral claims are set out by the Yukon Government and are outlined within the Quartz Mining Act (QMA). As defined within the mining act, a claim is a rectangular plot of land which must not exceed 1, 500’ X 1, 500’ in size. All claims must be formed of right angles except where a boundary line of a previously located claim is incorporated as common to both locations, as per section 18 of the QMA. In some situations, a parcel of land which measures less than 1, 500’ X 1, 500’ may be staked. This type of claim is referred to as a fractional claim and occurs when a plot of land lies between and is bounded by on opposite sides by a previously located mineral claim. In this situation the claim does not have to be rectangular in form and the angles need not be right angles. According to section 19 of the QMA, lines of previously located mineral claims, between which the fractional mineral claims are located, may be adopted as the boundaries of the fractional mineral claim.

Grounds open for staking should be referenced prior to staking to ensure their availability, the relevant maps are available at the Mining Recorder Office and online at www.yukonminingrecorder.ca. Areas where staking is prohibited include: areas over active mineral claims, First Nation Category “A” Settlement Land, curtilage (yard) of a dwelling house, parks, special management areas, cemeteries, burial grounds or other church property, lands withdrawn for the settlement of land claims, agricultural land currently under active cultivation, and any land removed from staking by Order in Council.

Prior to staking in the field, claim tags must be acquired from the Mining Recorder at a cost of $2.00 for a set of two. Two tags are required for each claim (Post #1 and Post #2) as per the Yukon’s two post system whereby the claim lies to one side of the line joining the two posts. Once the claims have been properly staked they must be recorded with the Mining Recorder responsible for the Mining District where the claim is located within 30 days of staking. An application to record a claim must be submitted with all fees ($10 per claim) and a sketch of the claim. The date that the “Application to Record” form and fees are received is deemed to be the recording date (anniversary date) as per sections 41-47 of the QMA.

Once claims have been issued by the Mining Recorder there is a minimum work requirement (“representation work” or “assessment work”) of $100 per claim/per year based on the Schedule of Representation Work outlined in the QMA. Where work is not performed, or insufficient work has been performed, the claimant may choose to make a payment in lieu of work. In this case a payment of $100 per claim per year plus a $5 fee for the certificate of work per claim/per year may be paid as per sections 53-60 of the QMA and Schedule 2 Fee Section 104. Work requirements apply to every claim unless groupings are filed.

Groupings consist of groups of adjoining claims (up to a maximum of 750 claims) where work performed may be applied to any or all of the claims within in the group to satisfy annual work requirements, provided the work performed is sufficient to renew claims for that period. Work performed must not be filed later than 14 days following the expiry date of the claims or the claims will be deemed to have lapsed. Work requirements on claims may be still be filed after this 14-day grace period, but no later than 6 months after the expiry date of the claims. Work requirements filed during this period will be subject to penalty fees. Work filed within three months of the expiry date is subject to a $15 fee/per claim and work filed between three and six months will incur late penalties of $25 per claim for the work certificates.

Claims may be converted into a Quartz Lease once a vein or lode is confirmed within the claim boundaries. This type of mining lease is effective for a 21-year term and may be renewed for an additional 21-year term provided all conditions of the lease and provisions of the legislation were adhered to during the first 21-year term. Claims may be converted into a lease provided various conditions are met some of which include: a vein or lode has been found within the claim boundary and have been confirmed by the Yukon Government’s Chief Geologist, applicant must do or cause to have done $500 of work per claim, and the claim must be surveyed by a Canada Lands Surveyor. The holder of the lease has the exclusive rights to explore for minerals in, on, or under the area of land described in the lease; however, it does not include surface rights.

The following outlines the costs required to maintain a claim for one year and the cost required to maintain a lease for one term:

Annual work requirements per claim

Claims Anniversary Years	Work Requirements
1	$100/Claim

Schedule of fees related to staking, work requirements and leases

Recording mineral claim	$10/claim
Application for a lease	$10/claim
Certificate of work	$5/per claim/per year
Grouping certificate	$5/claim
Lease rent for 21-year term	$50/claim
Add for each acre over 51.65 acres	$5/claim

4.4 Permits and Environmental Liabilities

There are four active permits covering the Carmacks Project:

1. QML0007 – a Quartz Mining Licence granted in 2009 with an expiry April 1, 2034. The licence authorizes development and production of the Carmacks Copper Mine project as described in project #2006-0050 (assessed by YESAB) and QML application material submitted to the the Yukon government. A water licence was applied for but denied by the Yukon Water board in 2010. Since there was no development or production the project went into a state of temporary closure until 2021.
2. LQ00530 Stu Copper – Class 4 Exploration permit expiring April 30, 2030
3. LQ00433 Hoocheekoo – Class 3 exploration permit expiring Feb 6, 2023
4. Class 1 Notification Q2021_0259 expiring July 15, 2022

Granite Creek is in the process of acquiring a new Class 4 permit combining claims and leases currently covered by QML007, LQ00433, and Q2021_0259 and closing QML007.

To the Authors knowledge there are no environmental liabilities accruing to Carmacks Copper on the Carmacks or Carmacks North properties.

The Author is unaware of any other significant factors and risks that may affect access, title, or the right, or ability to perform the exploration work recommended for the Carmacks or Carmacks North Property.

4.5 Surface Rights in Yukon Territory

Surface rights are not included with mineral claims in Yukon Territory.

4.6 Permitting

Exploration Mining Land Use permits within the Yukon are divided into four classes whereby specific mining activities of varying levels are categorized in order of increasing potential to cause adverse environmental impacts. Each permit, from Class 1 through to Class 4, confers specific and exclusive rights to its holder. A detailed list of acceptable mining activities permissible within each class of permit is available online at: https://yukon.ca/en/mining. Permits are attached to a specific property and can be transferred to new operators.

A Class 1 permit is defined as a "grassroots" exploration program that has low potential to cause adverse environmental effects and where activities and reclamation are completed within one year. This type of program does not require government approval provided the conditions of the Class 1 permit are adhered to. In identified areas, operators are required to submit a notice to the Yukon Government to carry out Class 1 quartz exploration if the work is not covered under another mining land use approval. There is no charge to apply, and a response is usually received within 25 days. These programs may be subject to random inspections by a Natural Resources Officer to ensure all exploration activities fall within the scope of a Class 1 permit.

A Class 2 permit is the upper level of grassroots mining exploration activities. These types of activities have moderate potential to cause adverse environmental effects and therefore require prior assessment through the Yukon Environmental & Socio-economic Assessment Act (YESEA) along with a $100 fee. Programs carried out under a Class 2 permit must be completed within 12 months of the program start date and includes any reclamation requirements and camp removals.

Class 3 and 4 permits require the submission of a detailed Operating Plan to the Mining Lands Officer with prior approval before any mining activities may commence and assessment through the Yukon Environmental & Socio-economic Assessment Act (YESEA). The Operating Plan should outline all proposed mining activities up to a ten-year timeframe, which may be approved or altered by the Chief of Mining Land Use. Permitting fees vary depending on the timeframe outlined in the Operating Plan. For programs of no more than five years a $250 fee applies, and a $500 fee for programs between five and ten years in duration.

To develop a major quartz (hard rock) mine in Yukon, proponents need to:

1. Complete an environmental and socioeconomic assessment through YESAB, which includes a public comment period and consultations with affected First Nations. Larger mine proposals require an executive committee screening or a panel of the board review. These are multi-year processes.
2. Determine if a Water Licence is needed; and if necessary, apply for one through the Yukon Water Board
3. Apply for a Quartz Mining Licence complete with plans for mine development, operations, environmental monitoring and mitigation; and decommissioning.

Submit financial security to YG to cover the full outstanding mine reclamation and closure liability.

4.7 Traditional territory

The project lies within the traditional territory of the Little Salmon-Carmacks First nation and there is overlap in the northern portion of the project with the traditional territory of the Selkirk First nation. Within traditional territories, First Nations maintain rights to traditional uses and are an integral part of the permitting and development process. In addition to traditional territories, there exist settled lands adjacent to the project where surface rights have been confirmed known as category B settlement lands and settled lands where surface and mineral rights have been confirmed known as category A settlement lands.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Carmacks Project site is currently accessible by way of the Freegold Road that leads northwest of Carmacks for 34 km then by the Carmacks Project access trail for 13 km to the Property. The village of Carmacks, on the Yukon River, is 175 km by paved road north of Whitehorse. The property access road is narrow and rough with steep sections and requires 4x4-vehicle capabilities in inclement weather conditions. A new 13 km access road is proposed to be constructed as part of the Carmacks Project development; brush clearing along the road alignment was completed in 1997. The Freegold Road is maintained by the Yukon Government (YG) and is currently open seasonally, generally from April through September. The road will be kept open year-round by YG once a year-round operation begins.

Beyond the Carmacks Project camp, a 10 km user-maintained gravel road with four creek crossings leads to Hoocheekoo Creek in the middle of the Carmacks North Property area. Bulldozer and ATV trails on the Carmacks North property leads to the various zones on the property. The Carmacks North Property can also be accessed by a 15-20-minute helicopter flight from the Carmacks Property.

Due to its road access, proximity to Carmacks, Whitehorse and the Whitehorse airport, the Property can be efficiently accessed by 4x4-vehicle or helicopter and thus exploration, primarily diamond drilling, can be conducted year-round.

The year-round ports of Anchorage and Skagway, Alaska, and Stewart and Prince Rupert, BC, are accessible by all-weather highway to move overseas–sourced equipment and supplies into the Carmacks Project site and for potential shipment of copper concentrate. Anchorage is 1,133 km west of Whitehorse and Skagway is 180 km south, while Stewart is 1,043 km south, and Prince Rupert is 1,373 km south.

5.2 Local Resources and Infrastructure

5.2.1 Local Resources

Local commercial resources are limited. The village of Carmacks, with a population of about 500, has some lodging capacity and a few stores and restaurants.

Services in the village include:

- Nursing station with doctors' consultations by appointment.
- Tantalus School offering classes for K-12. Yukon College provides GED, academic upgrading, computer training and occupational courses.
- Recreation Centre with attached, covered skating rink.
- Airport and helicopter pad within city limits, No scheduled flights.
- Landfill site at south end of town. Recycling services once a week at landfill.
- A community water system, although some residents have private wells, and there is a water delivery service.
- Electricity from the Yukon electrical grid.

- Cell service, internet and telephone available.
- RCMP station, volunteer ambulance and for protection
- Government of Yukon – Lands and Forestry
- Little Salmon Carmacks First Nation government offices

Commercial services are limited, but include:

- 2 service stations
- Restaurants
- Grocery store
- Hotel and rental cabins
- Campground

Human resources are as such limited. A large part of the workforce will be drawn from other areas, probably from Whitehorse. The Tantalus School serves the village of Carmacks and provides education for grades K-12. Yukon College operates a satellite school in Carmacks, providing academic upgrading courses, GED, computer training, and various occupation-related courses.

There are plenty of outdoor recreational opportunities that have proven to be popular within this area. These include fishing, hunting, and trapping. These activities are basic to the Yukon way of life and central to the sustenance of many people. In addition, another significant activity during summertime is canoeing in the Yukon River. This activity brings many people from outside the area.

5.2.2 Infrastructure

The Carmacks Project is approximately 220 km from Whitehorse, the capital of Yukon Territory. Whitehorse has a population of approximately 23,000, which is about two-thirds of the entire Yukon population. Whitehorse has an international airport, serviced by daily commercial flights from British Columbia and Alberta to the south and other northern communities. All-weather paved highways connect Whitehorse to the south and west to Alaska.

In the past, the Yukon & White Pass Route (Y&WPR) railroad provided rail service from Whitehorse to the port at Skagway Alaska, approximately 180 km south. Concentrate from the Faro mine was trucked from the mine to Whitehorse, and then shipped to Skagway by Y&WPR. Operations at Faro were suspended in 1982, and the railroad operation was reduced to tourist excursions. When the Faro mine reopened for a short period in 1985, the railroad was not available, and the concentrate was trucked to Skagway for overseas shipment. Skagway currently provides port facilities for cruise ships taking tourists to Yukon and Alaska. The nearest operational rail head is at Fort Nelson, BC, approximately 1,200 km by paved road from Carmacks.

The village of Carmacks can provide a location for support and administrative services during construction and during mine operations. Carmacks has full communications services available including cell phone service. Permanent power for the Project could be provided by Yukon Energy Corp. (YEC) by means of a 138/34.5 kV tap-off from the existing power grid at McGregor Creek and an 11 km overhead 34.5 kV power line to a main substation at the site.

There are no permanent facilities currently on the property as all previous work was performed from a tent and trailer camp. Some clearing of brush has been performed in the area of a previously proposed heap leach pad.

The property size and moderate terrain have proven sufficient to accommodate mining facilities, mill processing sites, and waste disposal sites. There is sufficient room for expansion of these facilities. There is sufficient water on the property to supply mining and milling operations, including accommodations and drilling.

Should the Carmacks Copper deposit advance to development, any infrastructure development (roads, power etc.) would benefit the Carmacks North Project. The subdued topography on the Carmacks North Property is suitable for construction of mining operations and there is sufficient water available on the property or nearby for drilling and development.

5.3 Climate

The climate in the Carmacks Project area is marked by warm summers and cold winters. Average daily temperatures at the Williams Creek Station on the Carmacks Project site range from -30°C in January to 12°C in July. The location close to the Arctic Circle provides 22 hours of daylight at the summer solstice and similarly long nights at the winter solstice.

Precipitation is light with moderate snowfall, the heaviest precipitation being in the summer months. The average annual precipitation is approximately 346.5 mm (water equivalent) with about 30% falling as snow. July is the wettest month. Annual lake evaporation is estimated to be 440 mm to yield a net loss of 93.5 mm. The weather does not impede year-round commercial operations in the Yukon, including outdoor activities in the winter, except in the harshest cold snaps when temperatures may drop to –50°C. The Cyprus Anvil open pit lead/zinc mine at Faro and the Brewery Creek open pit/heap leach gold mine, both located in proximity to the Carmacks Project have both been successfully operated year-round for many years in this climate.

Winter conditions, where daytime maximum temperatures average below zero, occur from November to March. The extreme cold temperatures in the region make outside construction in the winter difficult. In general, the outdoor construction season will be from April to October.

5.4 Physiography

Topography at the Carmacks and Carmacks North property area is subdued. Topographic relief for the entire property is 515 m. In the immediate area of the No. 1 Zone, topographic relief is 230 m. Elevations range from 460 m at the Yukon River to 1,030 m on the western edge of the claim block. The area falls within the Central Yukon Plateau which is characterized by broad valley and rounded ridge crests. Permafrost is discontinuous and scattered as the mean annual ground temperatures exceed -5°C. The permafrost is encountered at depths of 40 to 50 cm on most north-facing slopes where glacial till or colluvium is present.

Outcrop is uncommon because of the subdued topography and recent glaciation. The major portion of the claim block lying north of Williams Creek is unglaciated above the 760 m elevation line. The claim block area south of the Williams Creek valley and peripheral portions of the claim block, especially to the east, are covered by a veneer of ablation and lodgment boulder till with a sandy to silty matrix, generally less than 1 m thick. Valley bottoms and north-facing slopes have moderate to thick surficial cover that include far travelled sediments such as till, loess and glaciolacustrine sand. Valley bottoms contain thick Quaternary fill.

6 HISTORY

6.1 Carmacks Property Exploration History

The following description of the exploration history of the Carmacks Property has been extracted from Arseneau (2016). A considerable amount of historical exploration and drilling has been carried out on the property leading up to and during the discovery and definition of the Carmacks deposit. In addition to drilling, the main method of exploration has been surface trenching. Zones 1, 4, 7 and 7A zones have been trenched at 200-foot spacing. All trenches across Zone 1 were channel sampled with 5 or 10 foot (1.52 m or 3.05 m) sample lengths.

6.1.1 Exploration History

The first reported copper discovery in this region was made by Dr. G.M. Dawson in 1887 at Hoochekoo Bluff, on the Yukon River, 12 km north of the Property. In 1898, the first claims were staked to cover copper showings that were associated with copper bearing quartz veins located in Williams Creek and Merrice Creek Canyons, east of the present Carmacks Copper deposit (Arseneau, 2016).

In the late 1960's, exploration for porphyry copper deposits in the Dawson Range led to the discovery of the Casino porphyry copper deposit, 104 km to the northwest of the Carmacks Copper deposit. This discovery precipitated a staking rush that led to the staking of the Williams Creek property in 1970 by G. Wing and A. Arsenault of Whitehorse. The Dawson Range Joint Venture (Straus Exploration Inc., Great Plains Development of Canada Ltd., Trojan Consolidated Minerals Ltd., and Molybdenum Corporation of America) optioned the property and contracted Archer, Cathro and Associates to conduct reconnaissance prospecting and geochemical sampling. During this program, Zones 1 and 2 were discovered.

Extensive drilling campaigns were undertaken on the property in 1971 (5,583 m of diamond drilling in 25 holes) and 1972 (1,531 m of diamond drilling in 8 holes) along with other exploration activities such as trenching, access road construction, ground magnetic surveys, ground VLF-EM surveys, airborne geophysical surveying, geological mapping, soil and rock sampling. From 1972 to 1990 there was no significant work performed on the property.

The property was purchased by Western Copper Holdings and Thermal Exploration Ltd in 1991. Later that year, they performed 3,464 m of diamond drilling in 36 holes and initiated a baseline environmental study. Ground geophysics was carried out in 1991 by Interpretex Ltd, over the Zone 1 area and continued north and south over a total of 6,096 m (20,000-foot) strike length. The survey was done at 61 m (200-foot) line spacing for a total of 84.3-line km (52.4 miles). The VLF-EM and magnetometer survey identified numerous structures assumed to be faults as well as the main zone style mineralization.

In 1992, they drilled 1,164 m in 8 diamond drill holes and 856 m in 11 reverse circulation holes. The companies also conducted additional metallurgical test work, baseline environmental testing, a biophysical assessment of the area and contracted Knight Piésold Ltd to conduct geotechnical studies on the deposit consisting of test pit excavation, overburden sampling, oriented diamond drill core logging and geologic mapping.

In 1993, Sander Geophysics Ltd. conducted an airborne magnetic, radiometric, and VLF-EM survey over an even larger grid. Two hundred and fifteen-line km were flown at 100-metre line spacing.

In 1994, Kilborn Engineering Pacific Ltd. was contracted to perform a Feasibility Study (FS). The study indicated that, based on the copper price at the time the Carmacks Project was viable using open pit mining methods and solvent extraction-electrowinning.

In 1995, Western Copper Holdings and Thermal Exploration Ltd. merged to become Western Copper Holdings Ltd. The company contracted Knight and Piésold Ltd. to initiate a preliminary mine design and also initiated clearing and grubbing of a site access road and leach pad area. The company submitted a mine permit application later that year.

While the company was awaiting a mine permit, they contracted Kilborn Engineering to produce a basic engineering report, in 1997. The permit was not forthcoming and, due to changing market conditions the company withdrew the permit application. The property sat dormant until the re-initiation of permitting in 2004.

The Carmacks area in general was also covered by a regional fixed wing airborne geophysical survey conducted by Fugro Airborne Surveys for the YG in 2001 (Shives et al, 2002).

In February 2003, Western Copper Holdings Ltd. changed their name to Western Silver Corporation as a result of a corporate redirection toward silver mining.

In late 2004, based in part on renewed optimism in the price of copper, Western Silver agreed with the Yukon Territorial Government to re-enter the permitting process and re-engaged in the environmental review process under the YEA process and more recently the newly enacted Yukon Environmental and Socio-economic Assessment Act (YESAA) process.

In early 2006, Glamis Gold Ltd. purchased Western Silver Corporation and spun off a separate firm named Western Copper Corporation. Western Copper retained the rights to the Carmacks Copper Project. In September 2006, Western Copper retained M3 Engineering & Technology Corporation (M3) to revise the earlier studies and to develop a Bankable Level Feasibility Study (FS) fully compliant with NI 43-101 for the heap-leaching recovery of copper. This study was completed in 2007 (M3, 2007). The FS only considered oxide mineralization in Zones 1, 4 and 7.

In October 2011, Western Copper split into three separate companies, Copper North Mining Corp., which retained the Carmacks Project, NorthIsle Copper & Gold Inc., and Western Copper and Gold Corporation. Copper North continued to manage the Carmacks Project. In 2012, M3 updated the feasibility study for the heap leaching recovery of copper to reflect Carmacks Project design changes made to address environmental concerns (M3, 2012).

In 2014, Copper North commissioned Merit Consultant International Inc. (Merit) to prepare a Preliminary Economic Assessment (PEA) on the Carmacks Project (Merit Consultants International Inc., 2014) (based on the 2007 MRE). The PEA focused again on zones 1, 4 and 7 and specifically examined, at a conceptual level, the potential economic viability of adding gold and silver recovery by cyanidation to the Carmacks Project. The gold and silver was to be recovered from the cyanide leachate using sulfidization, acidification, recycling and thickening (SART) and absorption, desorption and refining (ADR) processes. The PEA concluded that the addition of gold and silver recovery to the Carmacks Project improved the overall Carmacks Project economics with respect to gross and net revenues and the cash cost of copper recovery after deduction of the gold and silver credits.

In 2016, Copper North commissioned JDS Energy & Mining Ltd. ("JDS") to complete a PEA for the Carmacks Project (JDS, 2016). The purpose of this study was to develop and document a preliminary project design and economics for recovery of copper, gold, and silver from the oxide mineralization using agitated tank leach technology from Zones 1, 4 and 7. The 2016 PEA included updated MREs for Zones 1, 4, 7, 12, 13 and 2000S (Arseneau, 2016). Mineralization for Zones 2000S, 12 and 13 were not considered for the 2016 PEA or any previous FS.

In 2020, Granite Creek acquired 100% of the Carmacks Project through its acquisition of Copper North.

6.1.2 Historical Drill Programs

Prior to 2006, a total of 75 DDH and 11 reverse circulation holes, amounting to approximately 11,900 m of drilling, were drilled in the exploration of the Property (Figure 6-1 and Table 6-1). The DDH prior to 2006 are numbered by zone, so hole 101 would be the first hole drilled on Zone 1 and hole, 1302 would be the second hole in Zone 13.

Core drilling of Zone 1 utilized BQ size (36.5 mm) in 1971, NQ size (47.5 mm) in 1990, and HQ size (63.5 mm) in 1991 and 1992. Three NQ size holes drilled in 1990 had variable recoveries. Hole 118 recovered virtually 100% of the core, hole 119 averaged in the high 80% range, and the third hole, hole 120, averaged in the low 90% range. Core recovery for the HQ size holes averaged in the mid to high 90% range. In 1992, an NQ size hole, number 158, was drilled using the triple (split) tube system. Except for rare instances where the core tube failed to latch, core recovery was 100%. Friable or broken sections were more completely recovered using larger diameter core (HQ) and the triple tube system.

Three reverse circulation downhole hammer holes were drilled on Zone 1 in 1992. They were drilled to twin DDH 119 (NQ), 125 (HQ) and 126 (HQ). The purpose of these holes was to determine if significant quantities of copper mineralization were lost through water circulation during diamond drilling and to determine if the expected higher recovery of friable or broken mineralized gneiss in large diameter holes would improve the grade.

The three reverse circulation holes, RC-4, RC-5, and RC-6 were drilled dry through the mineralized section so that no losses to washing could take place. Hole RC-4 twinned HQ core hole 125 and was similar in grade and width, 39.62 m averaging 1.40% Cu versus 48.16 m averaging 1.36% Cu, respectively. Hole RC-5 twinned HQ- core hole 126 and improved the grade, 48.77 m averaging 1.07% Cu versus 44.50 m averaging 0.83% Cu, respectively. Hole RC-6 twinned NQ-core hole 119 and also improved the grade, with 44.20 m averaging 1.11% Cu versus 49.68 m averaging 0.96% Cu, respectively. Hole 125 recoveries averaged in the mid-90% range while holes 126 and 119 both averaged in the high-80% range. The improved grades in RC-5 and RC-6 suggest that when core recoveries were below the mid-90% range, grades are possibly understated by diamond drill results; however, a statistical analysis (t-test comparison) of reverse circulation holes versus DDH indicated there was no statistical difference in the results.

For the 2006 and 2007 drill programs, each hole started with HQ core (63.5 mm) and most holes reduced to NTW (56.0 mm) with the occasional hole having to reduce down to BTW (42.0 mm) at greater depths. In general, core recovery for the 2006 and 2007 programs was greater than 97%.

The object of the 2006 program was to examine the down dip extension of Zone 1, with a goal to delineate the oxidation-reduction front at depth on the deposit; confirm historic drill results by twinning two of the previously drilled holes and explore along strike to search for lateral extensions of Zone 1, and to expand the knowledge of some of the other mineralized zones (Figure 6-2).

In addition, a rotary air blast (RAB) drilling program commenced in August 2006, which was designed to condemn areas of the Property for future plant site development.

In 2007, Western Copper continued the exploration and environmental sampling program and conducted geotechnical studies of the proposed heap leach pad, waste rock storage area, processing plant and camp location.The object of the 2007 program was to define the northern and southern limits of Zones 1, 7 and 7A, to delineate Zone 4, to further test and define Zones 12 and 13, expand the exploration of the newly discovered Zone 14, and carry out condemnation drilling in the proposed waste rock storage, heap leach pad and the processing plant areas. The 2007 program consisted of 17,800 m of diamond drilling in 123 holes, 866 m of geotechnical drilling in 36 holes, 31.7 line km of induced polarization surveys and surveying of all drill hole locations including all the historic drill holes, geotechnical holes, and rapid air blast drill holes.

In 2008, Western Copper drilled 12 geotechnical holes (1,923 m) in the pit area, two (2) water wells in the camp area (253.5 m), and one (1) water monitoring well below the heap leach pad (151 m).

Table 6-1 summarizes the historical drilling on the Carmacks project.

Table 6-1 Summary of Historical Drilling Carmacks Project

Year	Hole Type	no holes	metres	Company
1970/1971	DD	20	4,725	Historic
1972	DD	8	1,531	Historic
1991	DD	38	3,854	Western Copper
1992	DD	9	938	Western Copper
1992	RC	11	857	Western Copper
1995	GEOT	10	185	Western Copper
1996	GEOT	18	972	Western Copper
2006	DD	34	7,103	Western Copper
2006	RAB	61	1,235	Western Copper
2007	DD	123	17,845	Western Copper
2007	GEOT	36	923	Western Copper
2008	GEOT	12	1,923	Western Copper
2008	Water	4	460	Western Copper
	Total	**384**	**42,551**	

Figure 6-1 Isometric View looking Northeast: Pre-2006 Drilling in the Carmacks Oxide (brown) and Sulphide Deposit Areas

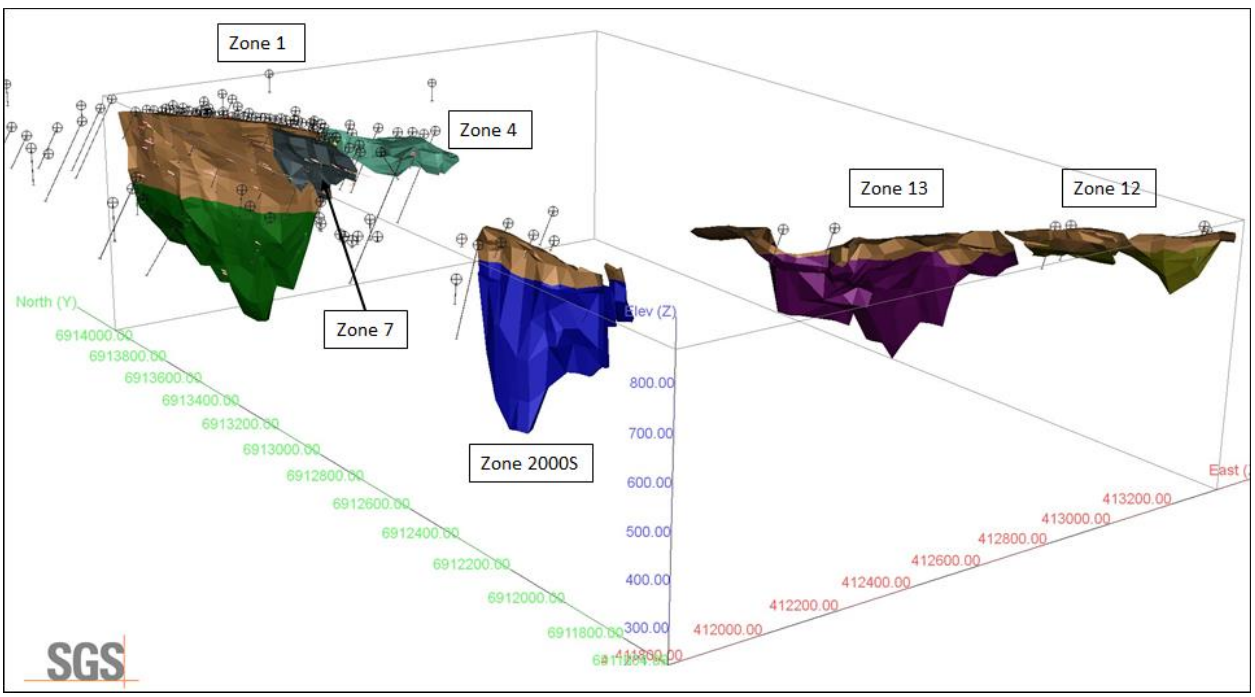
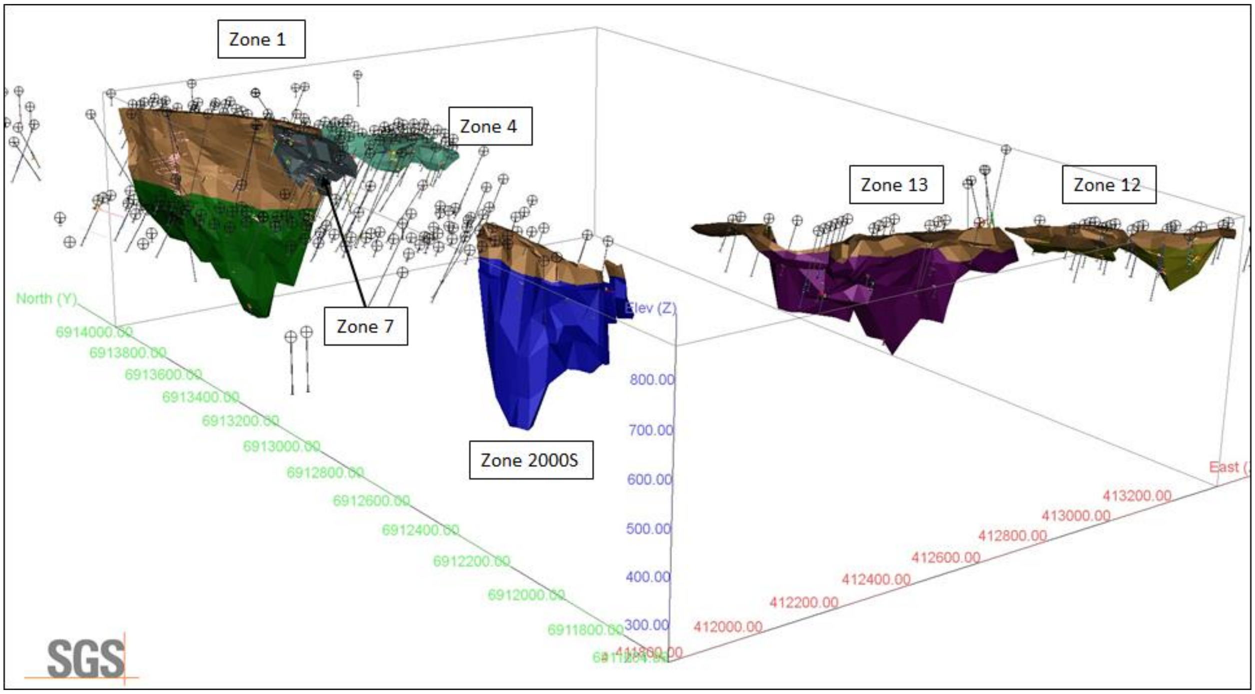


Figure 6-2 Isometric View looking Northeast: 2006 – 2008 Drilling in the Carmacks Carmacks Oxide (brown) and Sulphide Deposit Areas



6.2 Carmacks Property Historical and recent Mineral Resource Estimates

The Carmacks Project has been subject to several historical tonnage and grade estimations over the years as summarized in Table 6-2 (Arseneau, 2016). The historical Mineral Resources are presented here to show the progression of development of the Mineral Resources on the Property.

The mineral resources presented in Table 6-2 are considered historical in nature. The historical mineral resources were not prepared and disclosed in compliance with all current disclosure requirements for mineral resources or reserves set out in the NI 43-101 Standards of Disclosure for Mineral Projects (2016). A qualified person has not done sufficient work to classify the historical mineral resources as current mineral resources and Granite Creek is not treating the historical mineral resources as current mineral resources. These historical mineral resources have been superseded by the MRE for the Carmacks Project reported in Section 14 of this report.

Table 6-2 Historical Mineral Resource Estimates for the Carmacks Project (from Arseneau, 2016)

Year	Source	Tons	CuOx %	Cu %	Au oz/t	Comments
1991	MPH Consulting Ltd. (Zone 1)	14,564,600	0.9	1.05	-	Conventional by section 76% proven, 13% probable
1991	MPH Consulting Ltd. (Zone 1)	14,564,600	0.88	1	-	IDS block model 78% proven, 10% probable
1993	Western Copper Audited by Kilborn	12,984,240	0.911	1.195	0.016	Measured and indicated at cutoff of 0.8% total copper
1993	Western Copper Audited by Kilborn	15,867,140	0.829	1.096	0.014	Measured and indicated at cutoff of 0.5% total copper
1993	Western Copper Audited by Kilborn	19,062,390	0.725	0.972	0.013	Measured and indicated at cutoff of 0.01% total copper
1997	Western est. Audit by Kilborn/SNC	13,300,000	-	0.97	-	Cutoff grade 0.29%T Cu Mine use 4.6:1 strip ratio
2007	Wardrop (Zones 1, 4 and 7)	10,000,000	0.96	1.13	0.017	Oxide Resource, Measured and indicated at cutoff of 0.25% total copper

6.2.1 2014 Mineral Resource Estimate for Preliminary Economic Assessment

The MRE presented in the 2014 PEA was originally calculated in 2007 (Arseneau, 2007) and remained unchanged (Merit Consultants International Inc., 2014).

Arseneau (2007) constructed a block model of the No. 1, 4, and 7 Zones using historical data and data derived from the 2006 drilling campaign. Three mineralized zones (zone 1, 4, 7, and 7a) were interpreted on the basis of total copper grade. Surfaces were generated to represent the hanging wall and foot wall contacts with the mineralized zones. The surfaces honour the drill hole intersections in 3D. The solids were extended laterally approximately 15 m beyond the outermost drill hole intersections. The solids were generated by stitching the two non-intersecting surfaces together and then clipping the solids against the topographic surface.

The oxide-sulphide boundary was modeled using a minimum 20% ratio of oxide copper to total copper. All assays that contained at least 20% of the total copper value as oxide copper were coded as oxide in the model. A polyline was generated on an inclined longitudinal section to represent the oxide-sulphide boundary. The polyline was snapped to the assays on the down dip drill holes, honouring the 3D points. A clipping solid was generated by extruding the polyline 100 m on either side of the section. The three mineralized zones were then clipped and intersected with the oxide clipping solid to create final oxide and sulphide solids for all three mineralized zones.

Mineral resources were estimated with 3-dimentional software provided by Gemcom. Grades were interpolated for total copper, oxide copper, gold and silver into 5 by 5 by 5 m blocks. The block model was rotated 24.2 degrees anti-clockwise around the origin, aligning it parallel to the strike of the deposit and the surface exploration grid.

Copper grades (total copper percent and oxide copper percent) were interpolated into blocks using ordinary kriging with weighting parameters based on correlogram data. Grade interpolation search ellipses were designed from the correlogram information, trend of mineralization and sample data distribution. The grades were interpolated in three separate passes with differing sample support and search ellipses.

Grades were only interpolated if at least three samples, no more than one sample per hole, were found within the search ellipse, and a maximum of twelve samples were used to interpolate any block for the first pass. The second pass only estimated grades in blocks that were un-interpreted in pass one. Blocks were assigned a grade in pass two if at least two samples, no more than one per hole, were found within the search ellipse. The third pass only estimated grades in blocks that were un-interpreted in pass one and two. Blocks were assigned a grade in pass three if at least two samples, no more than one per hole, were found within the larger search ellipse. Sample selections for grade interpolations were restricted by oxidation zones.

Sulphide copper grades were calculated using a simple manipulation block model edit according to the following formula:

Cu Sulphide% = Cu Total% – Cu Oxide%

During the estimation, approximately 2,500 blocks estimated slightly higher Oxide Copper grades than Total Copper grades resulting in a negative Copper Sulphide grade after running the simple manipulation. The negative blocks were selected and the copper oxide grade was set to the total copper grade. An oxide copper proportion was calculated to determine the percentage of the total copper grade attributable to oxide or soluble copper. The oxide copper proportion was calculated by using a simple manipulation of the block model using the following formula:

Cu Oxide Proportion = Cu Oxide / Cu Total * 100%

Gold and silver grades were interpolated into blocks using inverse distance weighted to the second power. The same search ellipse from pass 3 for copper grades was used to interpolate gold and silver grades. The grades were interpolated in one pass.

The classification model was based on the average distance of the samples used to interpolate grade within a block. For classification purpose only, both holes in the sulphide and oxide mineralization were used to estimate the average distance of points used. All blocks that were interpolated during pass one and had an average distance of samples used less than 50 m were assigned to the Measured category. Blocks interpolated with an average distance of points used greater than 50 m were assigned to the Indicated category. Blocks that had not been interpolated during pass one were assigned to the Inferred category.

Arseneau (2007) estimated that the combined Zones 1, 4 and 7 contained approximately 12 million tonnes of oxide resource in the Measured plus Indicated categories grading 1.07 TCu, 0.86 CuX, 0.21% CuS, 0.46 g/t Au, and 4.58 g/t Ag at a 0.25% total copper (TCu) cut-off grade.

Zone 1 also contained an additional 4.3 million tonnes of sulphide resource in the Measured plus Indicated categories grading 0.75% TCu, 0.03% CuX, 0.73% CuS, 0.22 g/t Au, and 2.37 g/t Ag. In addition to the measured and indicated resource, the deposit contains 90,000 tonnes of oxide inferred resource grading 0.73% TCu, 0.53% CuX, 0.20 CuS, 0.12 g/t Au and 1.8 g/t Ag and 4 million tonnes of sulphide inferred resources grading 0.71 TCu, 0.01 CuX, 0.70 CuS, 0.18 g/t Au and 1.9 g/t Ag.

6.2.2 2016 Mineral Resource Estimate

Arseneau Consulting Services Inc. (ACS) was retained by Copper North Mining Corp. (Copper North) to update the mineral resources (Arseneau, 2016). The effective date of the revised MRE was January 25, 2016.

The Updated Mineral Resource consisted of Maiden Resources on zones 12, 13 and 2000S combined with the previously defined mineral resource for the Carmacks Project as set out in the 2014 PEA (Merit Consultants International Inc., 2014).

Maiden Mineral Resource Estimate (zones 12, 13, and 2000S)

Oxide and transition mineral resources:

- Measured and Indicated of 3.7 Mt grading 0.50% Cu, 0.35% acid-soluble Cu, 0.132 g/t Au and 2.011 g/t Ag
- Inferred of 0.8 Mt grading 0.42% Cu, 0.28% acid-soluble Cu, 0.119 g/t Au and 1.910 g/t Ag

Sulphide mineral resources:

- Measured and Indicated of 3.7 Mt grading 0.60% Cu, 0.128 g/t Au and 2.288 g/t Ag
- Inferred of 4.4 Mt grading 0.55% Cu, 0.123 g/t Au and 2.081 g/t Ag

2016 Mineral Resource Estimate

Oxide and transition mineral resources:

- Measured and Indicated of 15.7 Mt grading 0.94% Cu, 0.74% acid-soluble Cu, 0.379 g/t Au and 3.971 g/t Ag; an increase of 31%.
- Inferred Resources of 0.9 Mt grading 0.45% Cu, 0.30% acid-soluble Cu, 0.119 g/t Au and 1.900 g/t Ag; a tenfold increase.

Sulphide mineral resources:

- Measured and Indicated of 8.1 Mt grading 0.68% Cu, 0.178 g/t Au and 2.332 g/t Ag; an increase of 86%.
- Inferred resource of 8.4 Mt grading 0.63% Cu, 0.150 g/t Au and 1.994 g/t Ag; an increase of 108%.

The mineral resource estimate for zones 12, 13 and 2000S, was based on drilling carried out in 2006 - 2007 by Western Copper and Gold Corporation and additional drilling in 2014-2015 by Copper North. Copper minerals in the oxide resources largely comprise the acid-soluble minerals malachite, azurite and tenorite. The sulphide mineral resources are located at depth and comprise chalcopyrite-bornite mineralization. In zone 13, a transition mineral resource has been estimated, where chalcocite-native copper mineralization is developed between the sulphide and oxide zones. Oxide, transition and sulphide zones were assessed visually during core logging and validated by the ratio of acid-soluble to total copper assays. Sulphide zones largely comprise material with less than 20% of total copper as acid-soluble.

Wireframes for the mineralized zone were built in 3D from the geological interpretation along and between cross sections made up of fences of drill holes. Inverse distance to the second power was used to estimate grade in zones 2000S and 12. Ordinary kriging was used to estimate grade in zone 13. The influence of anomalously high copper and silver assays was restricted by the capping of high values. The estimates were run in two passes: firstly at 50 x 50 x 20 m, then at 100 x 100 x 45 m. A minimum of two drill holes with a minimum of three composites and a maximum of twelve composites were used to estimate resources. Blocks were classified as measured mineral resources if they were estimated with four drill holes during pass one. Blocks that were estimated with two or three drill holes during pass one were classified as indicated mineral resources; all other blocks were classified as inferred mineral resources. A cut-off grade of 0.25% total Cu was used for the sulphide mineral resources. A cut-off grade of 0.15% acid-soluble Cu was used to estimate oxide and transition mineral resources. An average density of 2.74 t/m^3 was used to estimate tonnage for sulphide mineral resources. An average density of 2.70 t/m^3 was used to estimate tonnage for Zones 12, 13 and 2000S oxide mineral resources. An average density of 2.68 t/m^3 was used to estimate tonnage for the Zone 13 transition mineral resource.

The previous mineral resource for zones 1, 4 and 7 was estimated in 2007 ("Previous Mineral Resource") and the total mineral resources for these zones remains unchanged. The Updated Mineral Resource comprises the Previous Mineral Resource in addition to the Maiden resource estimate on zones 12, 13 and 2000S.

6.2.3 2018 Mineral Resource Update, Zones 2000S, 12 and 13

Copper North undertook drilling in September and October, 2017 to gather more geotechnical information and exploration in the mineral area that was drilled in 2015. The drilling in the location of the planned deposition of dry stacked tailings was completed as part of preparing for improvement of the environmental report required for submission for new environmental approval and amended permits.

The results of the drilling in zones 2000S, 13 and 12 confirmed the continuity of these zones and their copper grades.

The drill results in the south area zones confirmed the continuity of the mineralized zones and extended the mineralized zones to further increase the size of the mineral areas. The Copper North undertook renewing the mineral resource in zones 2000S, 13 and 12 zones to provide a new mineral resource in the south area. The update mineral resource was presented in a news release on April 9, 2018 and available on SEDAR under Copper North's profile.

The Updated Mineral Resource estimate for Zones 2000S, 13, and 12 was prepared in 2018 by Independent Qualified Person, Dr. Gilles Arseneau, P.Geo., but was not supported by a NI 43-101 technical report as Copper North did not consider the magnitude of change in total project mineral resources to be material. It should be noted that neither the 2016 MREs nor the 2018 update were pit constrained and no economit parameters were used to justify the cut-off grades for the resources. Neither the 2016 nor the 2018 resource estimates comply with the CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019). The resources are superseded by the MREs for the Carmacks Project reported in Section 14 of this report.

The highlights of the 2018 Mineral Resource in Zones 2000S, 13, and 12:

- Step-out and infill drilling in fall 2017 was successful in confirming continuity of the mineral zones and grades of the oxide and sulphide mineralization in the southern extension of the Carmacks mineral deposit.
- Updated Oxide Measured and Indicated Mineral Resource increased to approximately 4,300,000 tonnes - grading 0.47% copper, 0.13 g/t gold and 1.92 g/t silver:
- The new Oxide Measured and Indicated Mineral Resource increased 40% over the 2016 Mineral
- Resource, primarily from lateral expansion of the oxide mineralized zone – upgrading of the already minor inferred resource category was a small contributor;

- The new Oxide Measured and Indicated Mineral Resource in Zones 2000S, 13, and 12 has the potential to provide an additional 2.4 years of mill feed at the planned processing rate of 1.775 million tonnes per year, subject to economic confirmation by future mine development planning; and,

- The Updated Sulphide Measured and Indicated Mineral Resource totaled 4,416,000 tonnes grading 0.62% copper, 0.13 g/t gold and 2.3 g/t silver, and an equal tonnage of mineral resource in Inferred category.

Zones 2000S, 13, and 12 are located 400 to 2,000 m to the south of the proposed open-pit, defined in the 2016 PEA based on the mineral resources in Zones 1, 4 and 7. The oxide mineral resources occur from surface and extend to depths of 80 to 100 m that may be amenable to open pit mining with a modest strip ratio. The Measured category represents 79% of the total Measured and Indicated resource.

The Updated Sulphide Measured and Indicated Mineral Resource in Zones, 2000S, 13, and 12 is 4,416,000 tonnes, of which 26% is Measured Resource and 74% is Indicated Resource. The Measured Resource is 1,136,000 tonnes, grading 0.59% copper, 0.13 g/t gold and 2.3 g/t silver. The Indicated Resource is 3,280 tonnes, grading 0.63% copper, 0.13 g/t gold and 2.3 g/t silver. The Inferred Mineral Resource is 4,281,000 tonnes, grading 0.54% copper, 0.12 g/t gold and 1.9 g/t silver.

The Updated Mineral Resource for Zones 2000S, 13, and 12 is based on drilling conducted in 2006 and 2007 by Western Copper Corporation and additional drilling in 2014, 2015, and 2017 by Copper North. Copper minerals in the oxide resource are largely comprised of the acid-soluble minerals malachite, azurite, and tenorite. The sulphide resources are located at depth and are comprised of chalcopyrite bornite mineralization. In zone 13, a transition resource has been estimated, where chalcocite-native copper mineralization developed between the sulphide and oxide zones. The oxide, transition, and sulphide zones were assessed visually during core logging and validated by the ratio of acid-soluble to total copper assays. Sulphide zones largely comprise material with less than 20% of total copper as acid-soluble.

Wireframes for the mineralized zone were built in 3D from the geological interpretation along and between cross-sections made up of fences of drill holes. Inverse distance to the second power was used to estimate grade in zones 12 and 2000S. Ordinary kriging was used to estimate grade in zone 13. The influence of anomalously high copper and silver assays was restricted by the capping of high values. The estimates were run in two passes: firstly, at 50x50x20 m, then at 100x100x45 m. A minimum of two drill holes with a minimum of three composites and a maximum of twelve composites were used to estimate resources. Blocks were classified as Measured Resources if they were estimated with four drill holes during pass one. Blocks that were estimated with two or three drill holes during pass one were classified as Indicated Resources; all other blocks were classified as Inferred Resources. A cut-off grade of 0.25% total Cu was used for the sulphide mineral resources. A cut-off grade of 0.15% acid-soluble Cu was used to estimate oxide and transition mineral resources. Bulk density was estimated by inverse distance squared using a minimum of three density values from the geological unit, any blocks that had insufficient density data were assigned average density values determined by averaging all readings taken from the geological unit.

The previous mineral resource for Zones 2000S, 13, and 12 was estimated in 2016 (“2016 Resource”). The Updated Mineral Resource includes the 2007 Resource, in addition to a revised estimate on Zones 2000S, 13, and 12. Further details of the 2007 Resource and the 2016 Resource can be found in the NI 43-101 technical report for the Carmacks property, filed on www.sedar.com, with a report date of 25 November 2016.

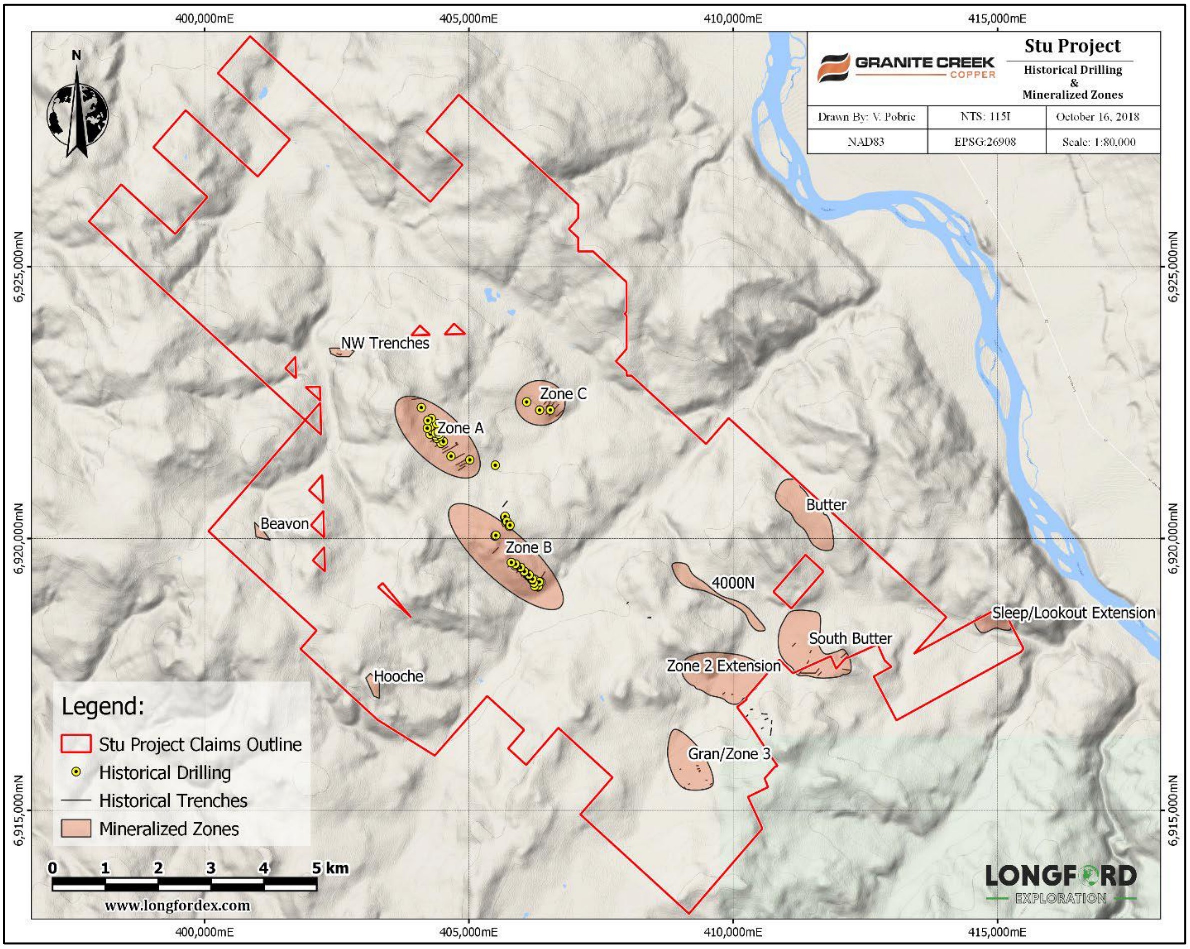
6.3 Carmacks North Property Exploration History (formerly the Stu Property)

The following summary of description of the exploration history of the Carmacks North Property has been extracted from James and Davidson (2018) (and references therein). For simplicity, the reference to the Stu Property will be maintained for this section.

Intensive exploration near the Stu Copper property started in the late 1960s following discovery of the Casino porphyry copper deposit in the Dawson Range, 100 km northwest of the property. Prior to this time, copper showings had been staked close to the Yukon River in the late 1890s. Following the Casino discovery, a staking rush in the area unearthed the Carmacks deposit and Minto mine properties in the early 1970s. The Stu property was worked from 1971 to 1982 by United Keno Hills Mines (UKHM), and again from 1989-2013 by UKHM, Western Copper and other operators. The amount of detailed information and geochemical results from UKHM's trenching and drilling programs is limited.

While under the ownership of the vendor, short programs consisting of examination and inspection of the property, rock sampling, surveying of trenches and drill holes, petrography, data compilation, collection of magnetic susceptibility measurements, claim staking and a limited amount of chip sampling of trenches were undertaken between 2005 and 2014. The information and results from these programs partially confirmed missing surface information from the UKHM work. In 2015, the vendor undertook a larger program of excavator trenching, systematic sampling, rehabilitation of old core and selected relogging and re assaying of core.

Figure 6-3 Location of Mineralized Zones, Drillholes and Trenches on the Stu Property (from James and Davidson, 2018)



6.3.1 Mapping and Prospecting

Most of the current Stu Property configuration was mapped between 1977 and 1981 at 1" = 400' (1:5000) scale using a cutline grid for survey control. The author has field checked mapping from this era and found it to be reliable and accurate, other than displacement of outcrops due to scanning and georeferencing errors.

The record of samples is sparse, but it appears that most of the samples were collected from Zones A, B and C in the central part of the property. Since 2005, only small mapping and prospecting programs were undertaken.

6.3.2 Soil Geochemistry

The bulk of soil sampling over the Stu Property was in the 1970s and early 1980s. Some of the grids overlap and provide a useful check on each other (Figure 6-4).

The first recorded soil sampling was in 1970, when the Dawson Range Joint Venture carried out reconnaissance geochemical sampling and prospecting over the Carmacks Property which located two mineralized outcrops – Zones 1 and 2. Additional claims were staked north towards Hoocheekoo Creek covering parts of the present day Stu property. Grid soil samples were then collected, and reconnaissance geological mapping undertaken over an 800' by 400' (244 m by 122 m) grid covering 14 square miles (3,626 ha).

In 1971 Hudson's Bay soil sampled the Bay claims on a property wide grid which covered the southern part of the Stu Property. Later workers criticized the quality of the sampling, suggesting the samplers did not consistently sample below the volcanic ash layer so prevalent over the property. This survey is worth further inspection before it is rejected, because it detected Zone C on the STU claims. The survey also picked up soil anomalies southwest of the Butter showing and southwest of the 4000N anomaly.

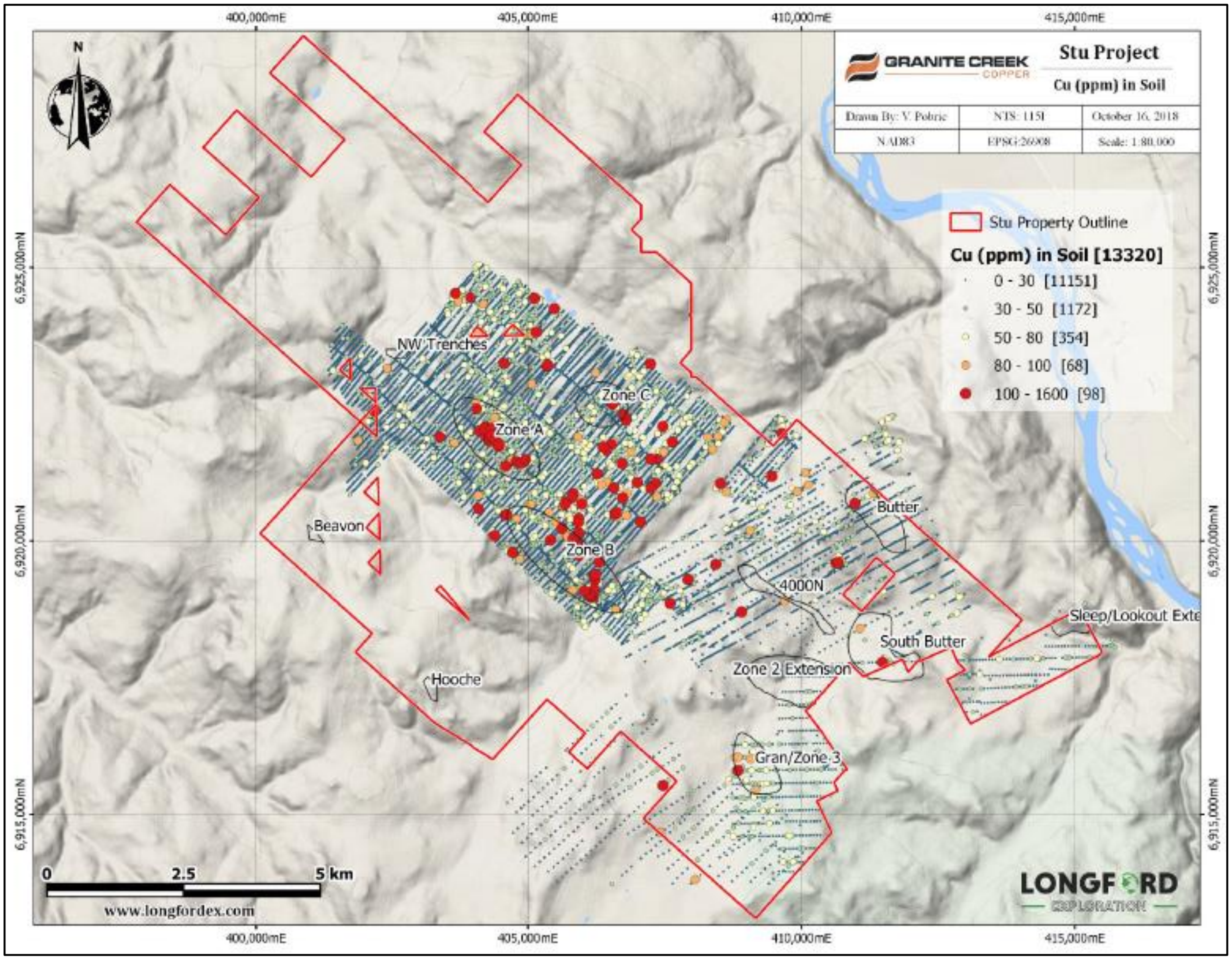
A series of large property-wide soil sampling programs were undertaken by UKHM in 1977-1981. Samples were collected along cutline grids at 30 m intervals along lines 100m apart. Zones A, B and C were outlined along with other northwest trending anomalies to the south and east. In the southern part of the Stu property sampling in 1981 delineated five separate northwest trending, moderate to strong copper anomalies at the headwaters of Nancy Lee Creek. The area covered was 500m long by 230-330m wide (Coughlan and Joy, 1981) and is in the same location as Gran/Zone 3. The other significant anomaly covers the South Butter showing and there are spot anomalies around the Butter showing. The programs were not documented to current standards; there are no laboratory analysis certificates, and no documentation of QAQC. However; the reported methodology is sound, samplers were aware of the detrimental effects of volcanic ash and collected samples in B horizon soils below the ash layer (0.9m deep on average). The value of a survey can be judged on whether it locates mineralization, and under this criterion the surveys was successful. In 1977, a stream sediment survey was carried out along Stu, Camp and Hoocheekoo Creeks. 362 active inorganic and quiet water organic samples were taken at 100m intervals.

In 1994, Western Copper cleared a baseline through part of the southern Stu Property. Survey lines were put in at 500m intervals and stations were spaced along each line at 25m intervals. Soil samples were collected at each station, with every other sample sent in for analysis. Moderate to highly anomalous copper in soil values were found northwest of Gran/Zone 3 and spotty soil geochemical values up to 323 ppm Cu over the 4000N anomaly.

Minimal soil sampling has been done since Bill Harris staked claims in 2005. Other that a small grid over the Nic showing in 2014 and two short lines along roads, most soil samples were collected to supplement rock samples in trenches or areas with no outcrop.

In 2008, BC Gold collected MMI soil samples over grids around the edge of the Stu Property. The strongest anomalies were south of the Gran/Zone 3 and to the northwest of Zone A. Their surveys confirmed or extended anomalies previously delineated in historic work. The Stu Property has been extended to cover some of the BC gold soil anomalies.

Figure 6-4 Historic Soil Sample Results from the 1970 Dawson Range Joint Venture Program in the South and the 1977 UKHM Soil Program in the North (from James and Davidson, 2018)



6.3.3 Trenching

UKHM carried out bulldozer trenching programs in 1979 and 1982 over four geochemical and/or geophysical anomalies. Complete assay results are not available, but trench maps with geology and some results were sourced from the UKHM archives. Selected trenches were cleaned and deepened, extended and new trenches were dug in 2015.

In 1979, nine bulldozer trenches were dug in Zone A to expose 900m of strike length. No results are available, but the best trench intersection was 0.19% copper over 15m.

In Zone B, 14 bulldozer trenches were excavated in 1979 and 1982 and up to 2% malachite over 0.5m in gneiss was observed. Recent trench work has revealed similar narrow zones of malachite.

Three trenches over 350m of strike length were excavated in Zone C in 1979, and no further trenching has been done since. There are 3 short trenches in the Northwest Zone, exposing mostly glacial till. No information from this period is available but a sample of clay altered granodiorite with limonite fractures and manganese staining collected in 2010 was not anomalous.

In Gran/Zone there are 8 or 9 trenches, either from work by the Dawson Range Joint Venture in the 1970s or UKHM in the 1980s. No results or mapping are available from this time. There is some exposure of weakly altered granodiorite but no mineralization was encountered. The remainder of the trenches are sloughed and overgrown.

In the South Butter Zone, bulldozer trenching has exposed mafic intrusive rocks but no mineralization was observed. The data of trenching is not known but probably occurred in the early 1980s as a follow-up to the soil anomaly from 1981.

Between 1982 and 2014 no mechanized trenching was done on the STU property. Some older trenches were partly cleared by hand and 50 grab samples collected between 2005 and 2014 from trenches in Zones A, B and C. No consistent sampling along trenches was done due to poorly exposed bedrock in sloughed and overgrown trenches.

In 2013, 38 chip samples were collected from three Zone B trenches where bedrock was exposed. Chip samples were taken between 0.5 and 2.0m long on good bedrock exposures. Where exposure was poor samples were either taken at a single location or pieces of rock were collected over a length.

In 2014, systematic hand trenching was done over the Nic showing 200m along the eastern side of Zone A. Four 2-8m long northeast trending hand trenches were dug about 10m apart and 19 rock samples were collected. Significant results were obtained from 3 of the 4 trenches. The northernmost trench (14-03) intersected a 5m zone of unmineralized granodiorite cut by a 1m wide diorite dyke.

- Trench 14-01 returned 0.55% Cu, 1.9 g/t Ag and 0.27 g/t Au over 6m
- Trench 14-02 returned 0.49% Cu, 2.2 g/t Ag, 0.33 g/t Au over 3.5m
- Trench 14-03 no significant results, 3 samples all under 100 ppm Cu
- Trench 14-04 returned 0.36% Cu, 1.3 g/t Ag, 0.16 g/t Au over 4.0m

Between July 23 and 31st, 2015 a Hitachi 33-ton excavator was used to dig 385m in 5 new trenches, and to clean and deepen 630m in 7 old trenches in Zones A and B. Mineralized zones in trenches were chip sampled, and XRF readings were taken at 5m intervals along the length of the trench. In all, 97 samples were collected, 6 grab samples and 91 chip samples between 0.5-3m long, averaging 1.8 m long.

6.3.4 Drilling

There were two programs of drilling on the Stu Property. The first was in 1980 on Zones A and C, and the second in 1989 on Zone B.

Diamond Drilling

Approximately 4500 metres of diamond drilling was done by UKHM in 1980 in the A and C Zones. Core from the program is stored near the camp and in 2015 the racks were disassembled and most of the core rehabilitated. Historical drill logs and assay results for the 1980 program are incomplete; the key reports describing the trenching and drilling program were not filed for assessment. Results are only available for one hole (80-17) and high grade composite intersections were reported for 3 holes in a Yukon Government publication. Following the diamond drilling in Zone A in 1980, no reserves were calculated.

High grade intersections:

- 80-09 3.44% Cu, 1.87 g/t Au, 13.37 g/t Ag over 13.5m
- 80-14 3.51% Cu, 2.49 g/t Au, 18.35 g/t Ag over 13.5 m
- 80-18 2.80% Cu, 4.04 g/t Au, 17.42 g/t Ag over 12.5m.

The lengths of the intersections are based on composite sample lengths and their relation to true width is unknown. The mineralization in Zone A appears to dip moderately to steeply to the northeast and the hole collar information indicates that the holes were drilled perpendicular to mineralization. All three high grade intersections were rehabilitated in 2015 but have not been resampled.

Drillhole 80-17 was a deep hole (426m), drilled behind and beneath hole 80-14, presumably as a follow-up beneath the high grade intersection. From 376-401m the hole intersected 25m of 0.155% copper, 6.2 g/t silver and trace gold (UKHM, 1981), at 380m below surface. Similar to previously reported intersections the relation between true width and sample length is not known, but the hole was drilled perpendicular to mineralization.

In 2015, drillhole 80-6 was relogged and reassayed by geologists from the Yukon Geological Survey. Sampling from 11.58 - 35.66 m (24.08 m sample length not true width) ranged from 0.03% to 0.34% Cu, averaging 0.18% Cu over the entire interval. A second interval from 52.43-55.78 m averaged 0.46% Cu over 3.35 m (sample width not true width). Diamond drill sections with geology, alteration, mineralization and structure (but no assay results) were recovered from the UKHM archives in 2013. The information from these sections has been entered into a drill database and converted into metric. Further relogging and sampling of old holes will improve the database, but it was used by the YGS to create sections and a simple 3D model of Zone A.

Three holes were drilled in the C Zone; drill logs are available for 2 of them. No mineralization was logged. There are 2-3 drill pads are in Gran/Zone 3 the general area, either from the 1960s or the 1980s but no information has been found for these holes. The rehabilitated core is valuable for its geological information, and although reassaying will assist greatly in understanding the mineralization geometry there are a few points that should be considered before sampling:

- Not all the collars have been located so there is some doubt on the exact hole locations.
- Historic core is BQ and was split when sampled previously so in some intersections there is not much material remaining to sample.
- If all the remaining half intersections are completely sampled there will be no physical record of the intersections. Partially resampling the intersections is a workaround but is not completely satisfactory.
- The rehabilitated core is in good condition and pieces appear to be in order, but there will always be some question especially in intersections where the core has been split allowing it to shift around.
- Assay values from the historic holes alone may not be reliable enough for a resource calculation, mainly for the reasons stated in the previous points, should the project reach that point.

It is noted that sections of core contain tenorite that had not been sampled, and it is possible that UKHM geologists focused on the visible copper oxides such as malachite and azurite when choosing sample intervals.

Percussion Drilling

In 1989, 30 percussion drill holes were drilled along trenches in the B Zone. Most holes were oriented at 225° azimuth, with dips ranging from -49° to -63°. Three holes were oriented at 45°. Two to three holes were drilled 3 to 20 m apart in each trench. Hole depths are 27 to 88 m and the entire length of each hole was sampled in 5 foot (1.5m) intervals. Copper results were plotted onto sections, and copies of assay certificates are available. Most holes intersected multiple zones with anomalous copper values ranging from 100-500 ppm.

The zones can be traced from hole to hole in about half of the sections, but they do not always coincide with malachite occurrences in the trenches. The best results are:

- hole SB-4 in trench 7600E 10 feet (3m) of 0.135% Cu
- hole SB-6 in trench 7400E 5 feet (1.5m) of 0.71% Cu
- hole SB-8 in trench B-1 5 feet (1.5m) of 0.11% Cu
- hole SB-9 in trench B-1 5 feet (1.5m) of 0.23% Cu
- Hole SB-10 in trench B-1 5 feet (1.5 m) of 0.16% Cu

All lengths in the list above are sample lengths. The relation to true width is unknown but all holes were drilled perpendicular to mineralization.

6.3.5 Geophysical Surveys

Hudson's Bay carried out a magnetometer survey in 1971 over the Bay claims. Prominent magnetic highs were mapped over the granodiorite-volcanics contact, prominent narrow highs were mapped over dykes in the granodiorite and less prominent highs occurred over increased magnetite in the porphyritic granodiorite.

Further magnetometer and electromagnetic surveys by UKHM outlined five zones of which the best four were followed up with an IP survey prior to trenching. It was concluded that there was little or no direct correlation between geochemical anomalies and IP anomalies over Zones A, B and C. IP anomalies were generally very weak and poorly defined, tending to complexity, caused by variations in resistivity, a response expected over weathered sulphides.

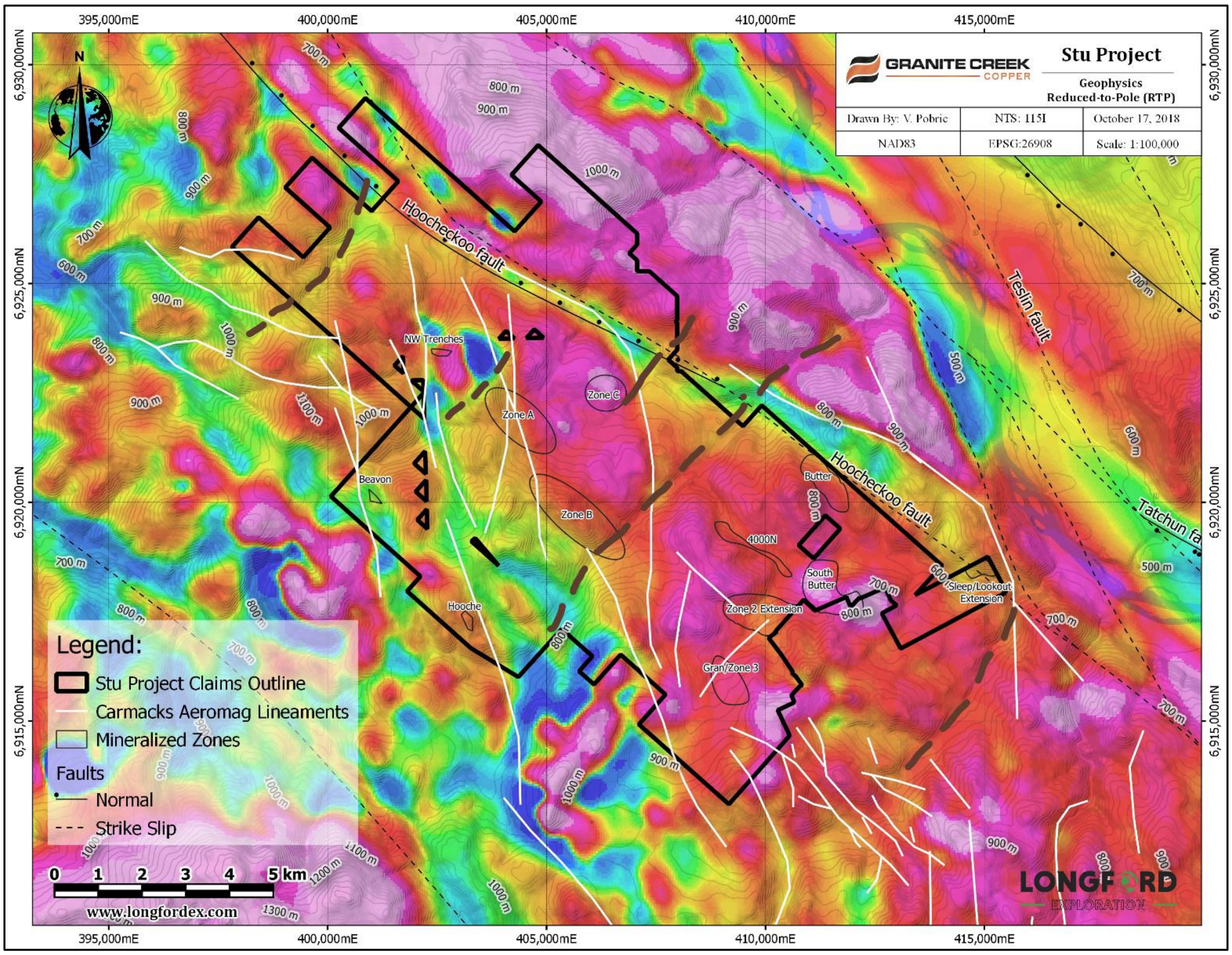
The 1974 Bay claims VLF-EM and IP geophysical surveys found linear geophysical anomalies were found between Hoocheekoo and Nancy Lee Creeks, over the Butter showing and southwest towards the 4000N zone.

In 1993, Western Copper flew an airborne electromagnetic survey and found the 4000N anomaly which stretched from lines 3000E to 4000E, hence the name “4000N.” The next year, Western Copper cleared a baseline through the centre of the WC claims with cross lines at 500m spacing and stations along each line at 25 m intervals. The entire grid was surveyed for total field magnetics, magnetic gradient and VLF-EM. The northwest extension of Gran/Zone 3 occurrence showed up as a weak magnetic anomaly associated with moderate to highly anomalous copper in soil values. It averaged 300-500 m in width and 1500 m in length. The 4000N zone showed possible narrow extensions onto lines 2500E and 4500E. A 2007 assessment report by Casselman contains a compilation of historic geophysics from a 1991 ground total magnetic and VLF-EM survey and the 1993 airborne total magnetic and VLF-EM survey over the Carmacks Copper property, reaching to north of Zone A.

In 2007, BCGold carried out a 3295 km airborne magnetic and radiometric survey over an area extending from south of the Carmacks Project to north of the Stu Property. Lineaments interpreted from the survey are overlain on regional scale reduced to pole (RTP) magnetic data from the YGS in Figure 6-5.

In 2008, BC Gold carried out 12.8 line km of IP surveying on the Copper claims (close to the Gran/Zone 3) and 18 line km over the Hooche Zone. Anomalous apparent resistivity and apparent chargeability correlate well on the Copper and may be caused by changes in lithology.

Figure 6-5 Airborne RTP Magnetic Survey from YGS Open File 2017-38 for NTS 115I White lines are interpreted magnetic lineaments from a 2008 magnetic survey flown by BC Gold. The brown dashed lines are NE trending interpreted faults from the RTP data (from James and Davidson, 2018).



7 GEOLOGICAL SETTING AND MINERALIZATION

The following description regarding the Carmacks Project Geology and Mineralization has been extracted from a paper written by Kovacs et.al., 2020 (and references therein), titled "Carmacks Copper Cu-Au-Ag Deposit: Mineralization and Postore Migmatization of a Stikine Arc Porphyry Copper System in Yukon, Canada", which includes information extracted from a 2018 M.Sc. thesis by Nikolett Kovacs (Kovacs, 2018) titled "Genesis and Post-ore Modification of the Migmatized Carmacks Copper Cu-Au-Ag Porphyry Deposit, Yukon, Canada".

7.1 Regional Geology

The Late Triassic to Early Jurassic magmatism in Yukon resulted from building of a Late Triassic island arc (Lewes River Group and Stikine plutonic suite) and subsequent arc-continent collision, syncollisional magmatism, and exhumation (Kovacs et.al., 2020). Volcanic rocks of the Lewes River Group terminate in central Yukon, however their plutonic equivalents, represented by the Stikine and Pyroxene Mountain suites, extend farther northwest into east-central Alaska (Figure 7-1). The Stikine suite (217–214 Ma) is represented by a series of small plutons that intrude Upper Triassic arc volcanic rocks of Stikinia and Paleozoic metasedimentary and meta-igneous rocks of the Yukon-Tanana terrane in south-central Yukon. The Minto suite (205–194 Ma) occurs as a series of large plutons that intrude the Lewes River Group and the Yukon-Tanana terrane that are interpreted to represent syncollisional magmatism at the onset of arc accretion. The younger Long Lake (188–183 Ma) and Bennett-Bryde (178–168 Ma) plutonic suites represent ongoing syn-collisional magmatism.

The Carmacks Copper deposit is located within the composite Early Jurassic Granite Mountain batholith (Figure 7-1). The Granite Mountain batholith is the southern extent of a series of Early Jurassic plutons, including the Minto and Yukon River plutons that form part of a single large batholith, ~120 km long by 15 to 25 km wide, segmented by Upper Cretaceous and younger volcanic cover. The eastern Granite Mountain batholith is assigned to the Minto suite and its western part belongs to the Long Lake suite (Figure 7-2). The Granite Mountain batholith intrudes and obscures the contact between mid-Paleozoic rocks of the Yukon-Tanana terrane and Upper Triassic rocks of Stikinia. The Yukon-Tanana terrane west of the Granite Mountain batholith is represented mainly by orthogneiss of the Early Mississippian Simpson Range plutonic suite. Stikinia arc rocks east of the Granite Mountain batholith include volcanic and sedimentary rocks and subvolcanic intrusions of the Upper Triassic Povoas Formation of the Lewes River Group. The Povoas Formation in southern Yukon is characterized by variably deformedand subgreenschist to locally amphibolite facies augite porphyritic basalt, volcaniclastic rocks, and hornblende gabbro. These Stikinia units and the Granite Mountain batholith are in fault contact along the dextral-normal oblique-slip Hoocheekoo fault. The Granite Mountain batholith contains inliers of variably deformed and metamorphosed mafic to intermediate rocks that host Cu-Au-Ag mineralization at the Carmacks Copper deposit, Minto mine, and Stu prospect.

Late Triassic to Early Jurassic batholiths were emplaced into crust that was being exhumed in the Early to Middle Jurassic to form the flanks of the subsiding marine basin of the Whitehourse trough. Exhumation is recorded by regional Early Jurassic metamorphic cooling ages, Al-in-hornblende barometry of Mesozoic plutons, and the Early to Middle Jurassic sedimentologic and detrital zircon record. Exhumation was essentially complete by the mid-Cretaceous, as indicated by the unconformably overlying volcanic rocks of the Mount Nansen Group, which are exposed 40 km to the southwest of the Carmacks Copper deposit. Volcanic rocks of the Upper Cretaceous Carmacks Group are preserved as extensive blankets north and south of the Granite Mountain batholith, and as isolated erosional remnants within the batholith. The Granite Mountain batholith is separated from the Minto pluton to the north, host of the Minto Cu-Ag-Au mine, by a veneer of the Carmacks Group.

The Carmacks Copper deposit area is located near the northwestern limit of Pleistocene glaciation, such that glacial erosion was restricted to subalpine areas and that bedrock below discontinuous till preserves a deep oxidative weathering profile. Paleoweathering profiles that contain copper oxide minerals at the deposit are locally capped by Carmacks Group volcanic rocks, indicating that at least part of the oxidation history is Late Cretaceous or older.

Figure 7-1 Simplified geologic map of south-central Yukon, showing distribution of Late Triassic-Early Jurassic plutons and locations of the Carmacks and Minto Cu-Au-Ag deposits (from Kovacs, 2020)

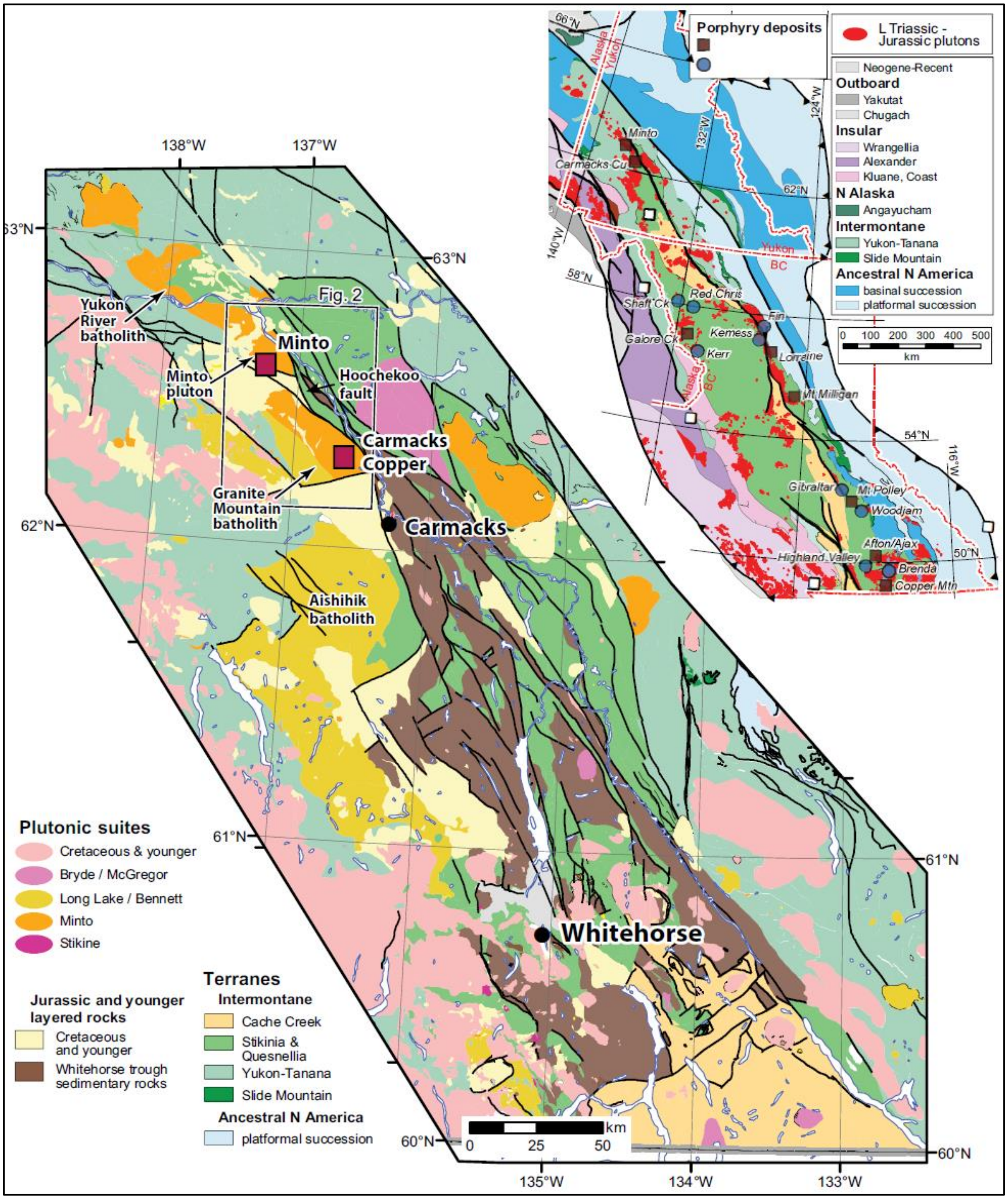
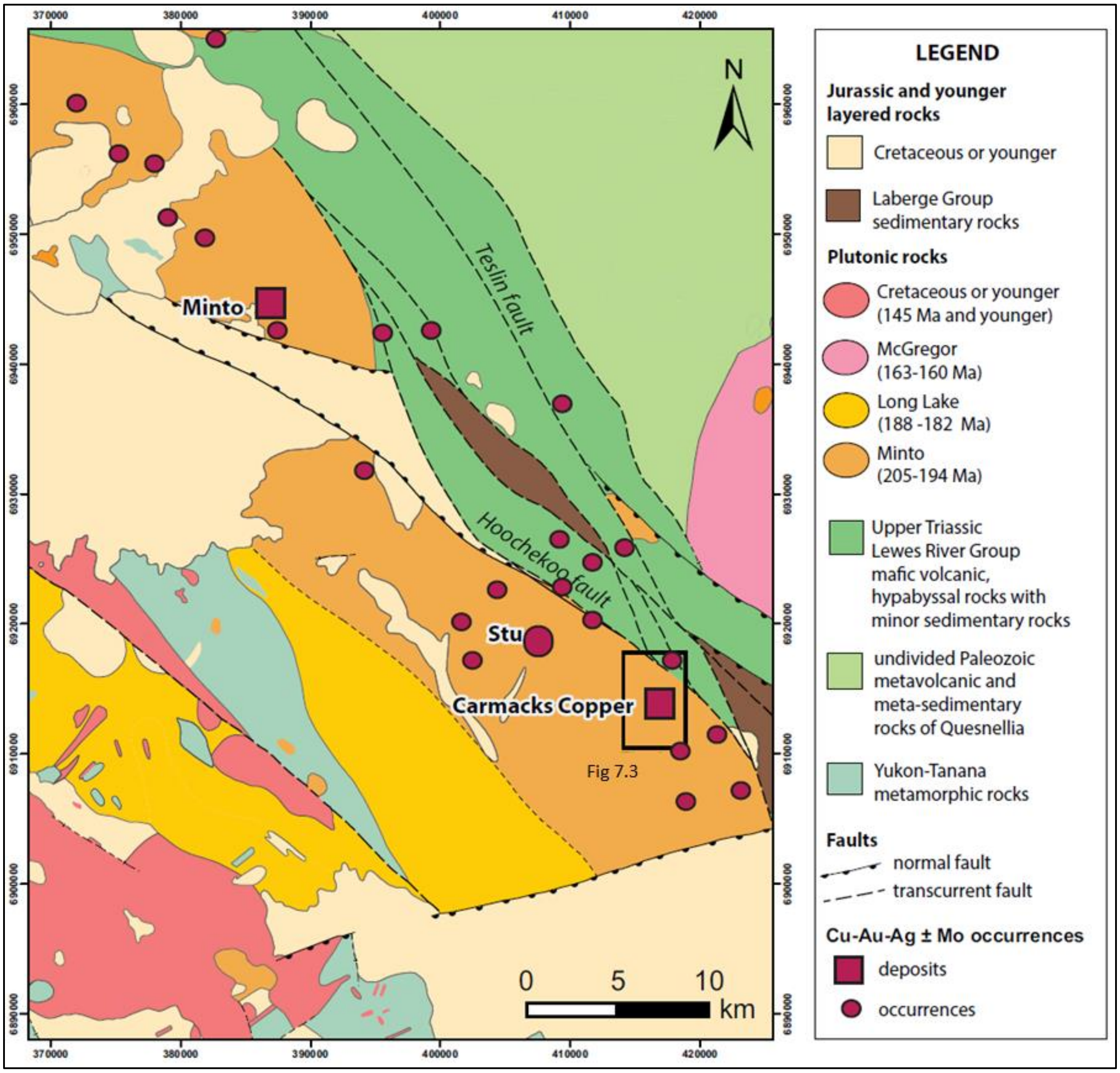


Figure 7-2 Geology of the Granite Mountain Batholith Area (Grid in UTM Projection, Zone 8N, NAD 83) (from Kovacs, 2020)



7.2 Deposit Geology

The Carmacks Copper deposit is hosted in a series of elongate, N-NW-trending inliers of amphibolite facies mafic to intermediate meta-igneous rocks and migmatitic derivatives within generally massive granitoids of the Granite Mountain batholith (Figure 7-3) (Kovacs et.al., 2020). Mafic rocks include foliated, equigranular amphibolite that locally is texturally transitional with less foliated, hornblende-porphyroblastic amphibolite. Rare augite gabbro is also locally present. Mafic rocks are interlayered with quartz-plagioclase-biotite schist. These metamorphic rocks are texturally transitional with migmatitic rocks, which host the bulk of hypogene copper mineralization. Migmatitic rocks occur preferentially along the eastern flank of the largest, 3-km-long by 20- to 100-m-wide inlier, where they represent a transitional intrusive contact between metamorphic rocks and the Granite Mountain batholith (Figure 7-3). However, this does not appear to be the case at depth based on 2021 drilling of Zone 1 (Jacob Longridge, pers. comm.).

Plutonic rocks

Metamorphic inliers in the main zone of the Carmacks Copper deposit are intruded to the east by a mainly monzodioritic phase of the Granite Mountain batholith. West of the main zone metamorphic inlier, the Granite Mountain batholith includes K-feldspar megacrystic granodiorite and quartz diorite. These Granite Mountain batholith phases are typically undeformed, although a weak magmatic foliation is locally defined by the alignment of phenocrysts. Dikes of quartz monzonite, quartz monzodiorite, granite pegmatite, and aplite (LTrEJM3) crosscut the metamorphic host rocks and other massive intrusive phases and are variably overprinted in the metamorphic rocks by folding and boudinage.

These late pegmatite dykes exist, but the supposed folding and "boudinage" interpreted by Granite Creek as reflective of the temperature of dyke emplacement and not a ductile folding event (as neither the batholith nor the metamorphic inliers show this proposed later stage deformation). This later deformation event is likely over-emphasised Jacob Longridge, pers. comm.).

The eastern phase is massive, medium grained, locally plagioclase-porphyritic, and ranges compositionally from diorite to monzodiorite to monzonite, has SiO2 contents ranging from 53 to 60%, and is alkaline and metaluminous to weakly peraluminous. The western phase is dominantly medium to coarse grained, K-feldspar megacrystic granodiorite, which is in gradational contact with subordinate quartz diorite. Collectively, the western phase ranges from quartz diorite to granodiorite to granite, contains 63 to 76% SiO_2, is subalkaline and peraluminous to weakly metaluminous, and has a slightly more evolved geochemistry than the eastern phase, as indicated by elevated Zr/Ti ratios. The geochemistry of all Granite Mountain batholith phases has negative Nb and Ti anomalies and Ba, K, Sr, Zr, and Hf enrichment in MORB-normalized diagrams and are LREE-enriched relative to HREE. All phases of the Granite Mountain batholith at the Carmacks Copper deposit are therefore consistent with I-type, magmatic arc affinity.

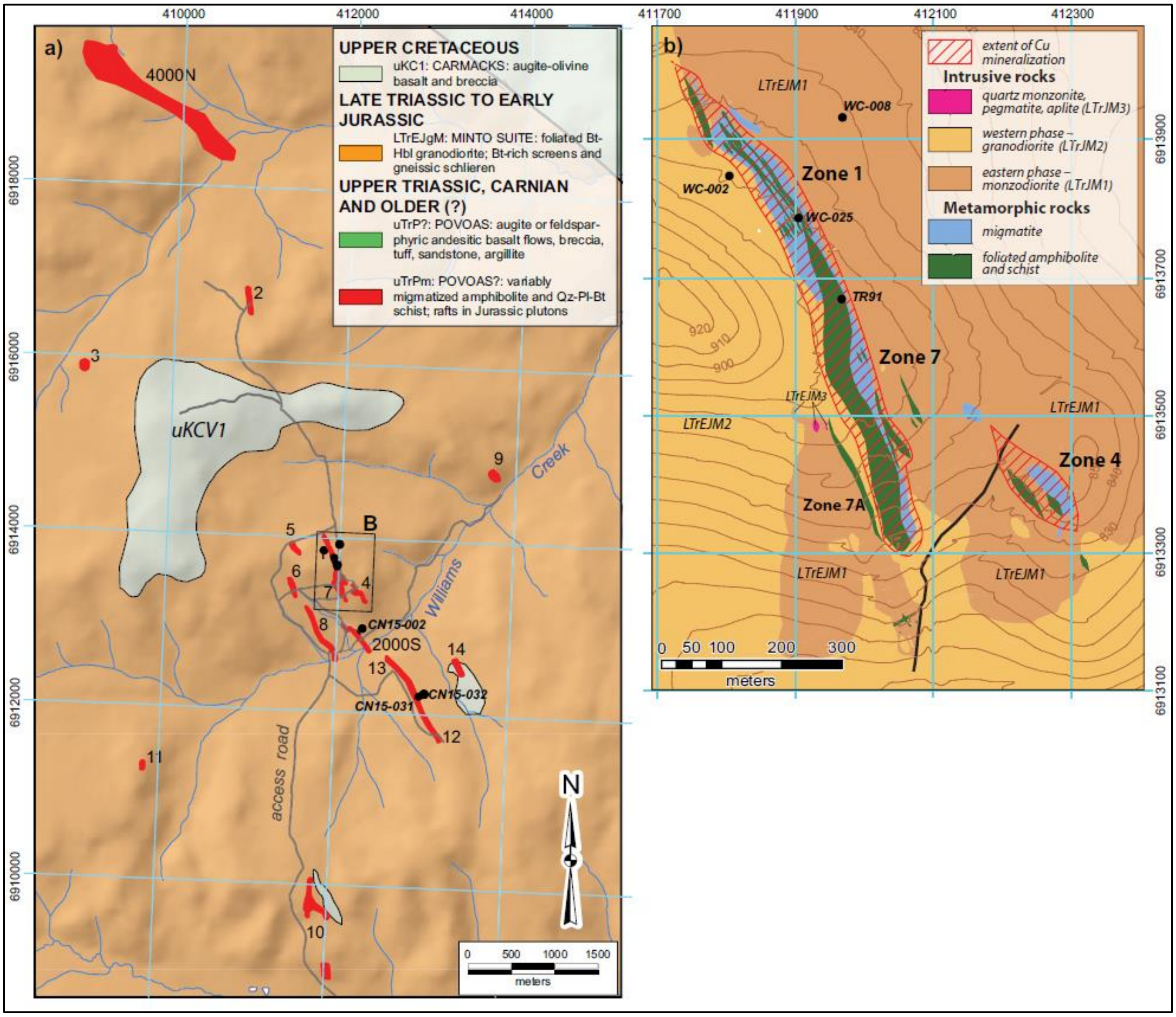
Metamorphic rocks

Inliers of metamorphic rock within the Granite Mountain batholith contain various meta-igneous rock types. Augite gabbro is massive to locally foliated and consists of 40 to 45% plagioclase, 30 to 35% clinopyroxene, 20 to 25% hornblende, and accessory titanite. The unit is transitional with hornblende porphyroblastic amphibolite, which contains 60% plagioclase, 30 to 35% hornblende porphyroblasts that replaced clinopyroxene, 5% biotite, and accessory titanite. Melanocratic varieties of amphibolite contain up to 75% hornblende. Foliated, equigranular amphibolite typically contains 60 to 65% plagioclase feldspar, 25 to 35% hornblende, 10 to 15% biotite, minor quartz (5%), and accessory titanite. A penetrative foliation is defined by biotite.

Amphibolitic rocks are interlayered with brown-weathering schist, which contains 10 to 12% quartz, 70 to 80% plagioclase, 20 to 25% biotite, ~1% titanite, and accessory zircon. The schist is locally recrystallized to a granoblastic texture.

Interlayered amphibolite and schist are in gradational contact with texturally and compositionally variable migmatitic rocks, including metatexite and diatexite. The metatexite is paleosome dominant and contains interlayed equivalents of the amphibolite and quartz-plagioclase-biotite schist units with local leucosome. In contrast, the diatexite is dominated by neosome and contains a large proportion of massive leucosome, with lesser proportions of melanosome. Contacts between metatexite and diatexite are gradational over 10s of metres and structurally chaotic.

Figure 7-3 a) Detailed Geologic Map Showing Distribution of Mineralized Rafts of Amphibolite and Intermediate Schist within Granodiorite at the Carmacks Copper Deposit and b) Detailed Map of the Main Zones at the Carmacks Copper Deposit. This Map Shows the Various Phases of the Granite Mountain Batholith in This Area, and the Relationship between Migmatized and Relatively Intact Metamorphic Rafts in the Main Deposit Area (Grid in UTM Projection, Zone 8N, NAD 83) (from Kovacs, 2020)



Metatexite: Metatexite is predominantly composed of fine-grained, foliated to granoblastic and stromatically layered quartz-plagioclase-biotite paleosome, and subordinate quartzofeldspathic leucosome, which occurs as irregular pods and as foliation-parallel layers that locally trace mesoscopic folds. Biotite-dominant, 1- to 2-cm-wide melanocratic selvages are typically well-developed at leucosome margins. The unit typically has a bulk mineralogy consisting of plagioclase (50–70%), quartz (7–10%), biotite (7–15%), and K-feldspar (~2–5%) and is interpreted as a melt-bearing equivalent of the quartz-plagioclase-biotite schist described above. Amphibolite locally contains pods, thin foliation-parallel layers or discordant veinlets of leucocratic melt.

Partial melting in quartz-plagioclase-biotite paleosome is indicated by (1) lobate to cuspate plagioclase grain boundaries, (2) accumulations of fine-grained quartz at triple junctions of other silicate grains, and (3) interconnected quartz and plagioclase subgrains. Partial melt in amphibolite is recognized by quartzofeldspatic pods with lobate to cuspate grain boundaries, by plagioclase that forms cuspate tapering extensions between hornblende grains, by monomineralllic films of biotite and plagioclase along grain boundaries, or by intergrowths of plagioclase, biotite, and fine-grained quartz at triple junctions.

Medium-grained, euhedral garnet and subhedral orthopyroxene occur locally at the margins of leucosome, although this mineral assemblage is generally rare.

Diatexite: Diatexite is characterized by predominantly massive and compositionally and mineralogically homogenous neosome with mm- to m-scale melanosome interpreted as schlieren of residual metamorphic protolith. The leucosome component is medium grained and composed of 45 to 60% plagioclase, 15 to 25% hornblende, 5 to 10% biotite, 3 to 5% clinopyroxene, 1 to 2% quartz, 1% epidote, and accessory titanite and apatite. Plagioclase is typically albitic and forms medium (1.0–2.5 mm), anhedral grains that are typically mantled by intergrowths of quartz, feldspar, and biotite that represent late-stage melt. Hornblende and clinopyroxene are medium grained (1.5–2.5 mm) and embayed, which is indicative of partial consumption by melt-forming reactions. Melanosome typically contains 80% hornblende, 7 to 10% biotite, 10% calcite, 2 to 4% K-feldspar, and 1 to 2% quartz. Similarly, hornblende occurs as embayed oikocrysts with chadacrysts of biotite and quartz. Fine-grained quartzofeldspathic intergrowths occupy interstitial space between hornblende grains and represent late-stage melt.

Structure

Metamorphic rocks preserve a penetrative transposition fabric (S1), which is defined by a preferred orientation of hornblende and biotite in amphibolite and schist, respectively, or by leucrocratic and melanocratic layering in migmatitic rocks. Felsic plutonic rocks of the Granite Mountain batholith are generally massive but locally exhibit a weak tectonic foliation near contacts with metamorphic inliers.

Several late, E-NE-trending faults cut all previously described rock units and structural elements (Figure 7-3). Slickenlines on hematitic or pyrolusite-coated fault surfaces show shallow plunge (5°–30°), indicating that the latest movement is dominantly strike-slip.

Faults identified in the deposit area include provide the limits of the resource and opens the recommendation of further exploration beyond these faults:

1. A fault that define the southern end of 2000S zone
2. A north zone 1 fault
3. A N-S trending fault that defines the graben to the east of zone 12
4. A The N-S fault that controls the conglomerates and Carmacks volcanics sitting on the hangingwall above zone 13.

These provide the limits of the resource and opens the recommendation of further exploration beyond these faults.

7.3 Mineralization

The following section regrading the Carmacks Project mineralization has been modified from Kovacs (2018). Modifications have been made based on recent work by Granite Creek.

Mineralization of the Carmacks Copper Cu-Au-Ag deposit occurs within a 3 km–long, north-northwest trending belt that is separated into a northern Zone 1, 4 & 7, central Zones 2000S and southern zones 13 and 12.

Three different varieties of hypogene copper mineralization are recognized, with distinct mineralogical and textural characteristics: disseminated, foliaform, and net textured.

Disseminated copper sulphide mineralization is interpreted to be the least modified by metamorphism because it typically occurs in the undeformed metamorphic lithologies and augite gabbro. Disseminated chalcopyrite and pyrite comprise a minor portion of the hypogene mineralization and is typical in the undeformed, hornblende porphyroblastic amphibolite, granoblastic quartz-plagioclase-biotite schist, and augite gabbro (Figure 7-4). Disseminated copper sulphide minerals also occur in leucosome within the quartz-plagioclase-biotite schist unit. Chalcopyrite is typically fine-grained (0.3-0.7 mm) and occurs as anhedral to wavy, skeletal grains intergrown with also fine-grained (0.1 mm), minor subhedral pyrite.

Foliaform copper sulphides are restricted to the amphibolite and the quartz-plagioclase-biotite schist and occur as chalcopyrite-dominant stringers that parallel the dominant foliation (). Chalcopyrite is fine-grained (0.3-0.5 mm) and forms elongated anhedral grains. Bornite is less common in the foliaform mineralization, but where present forms fine-grained (0.2-0.5 mm), anhedral, blebby intergrowth with foliaform chalcopyrite. Pyrite intergrown with chalcopyrite is rare, but it is euhedral where present. The foliaform nature of copper sulphides is interpreted as the result of Late Triassic deformation and temporally associated upper amphibolite facies metamorphism.

In contrast, mineralization hosted by the migmatite occurs as net-textured intergrowths of bornite and chalcopyrite, which comprises as much as 20%-40% of the rock by volume, with typical bornite-chalcopyrite ratios of 3:1. Pyrite is absent in the migmatite. Bornite and chalcopyrite occur together in irregular net-textured domains up to 2–4 mm across forming low interfacial angles that are clearly interstitial with respect to silicate grains (Figure 7-4). Both chalcopyrite and bornite are commonly replaced by digenite along 50 mm-wide fractures and grain margins and is interpreted as the latest copper phase due to secondary oxidation. Net-textured bornite is especially abundant in melanosome, where it typically forms higher-grade (1-2% Cu) domains. The net-textured domains contain numerous inclusions of fine-grained native bismuth, Au-Ag tellurides, and bismuth tellurides. Molybdenite is commonly intergrown with net-textured copper sulphides and occurs as kinked anhedral grains separated along cleavage surfaces or as euhedral undeformed grains.

The pyrite content of the deposit is notably low (~1%). Gold is principally associated with bornite and occurs as 10-20 µm inclusions of electrum or native gold, or more commonly as gold telluride (calaverite), or solid-solution gold-silver tellurides, as determined by SEM-EDS spectrometry (Figure 7-5 and Figure 7-6). Silver is present as hessite inclusions in bornite. Because gold and silver are typically associated with bornite, the bornite-chalcopyrite ± digenite zone is precious - metal enriched and the migmatite contains higher copper, gold, and silver grades than the amphibolite and quartz-plagioclase-biotite schist sequence.

Copper grades increase progressively northwards from the lower grade material found in the southern deposit Zone 12 and 13 deposits through to the highest-grade material at the Zone 1 deposit. This change in grade is interpreted to be caused by the increasing northward metamorphic gradient of the inliers which is also reflected in the general depth of emplacement of the batholiths.

Deep oxidation of the deposits has oxidized primary sulphides to copper oxides and copper carbonates with approximately 15% of the copper in the oxide domain occurring as remnant copper sulphide, in the form of chalcopyrite. This oxidation profile has led to the formation of an oxide cap that can be over 200 metres thick at zone 1 to ~40m thin in Zone 12. The majority of the copper found in oxide are in the form of the secondary minerals malachite, cuprite, azurite, tenorite (copper limonite) and crednerite with minor other secondary copper minerals (covellite, digenite, chalcocite). Native copper occurs as dendritic secondary precipitates on fractures, disseminated grains or thin veinlets. Other secondary minerals include limonite, goethite, specular hematite and gypsum.

Within the oxidized portion of the deposit, pyrite is virtually absent and pyrrhotite is absent. Oxidation has resulted in 1% to 3% pore space and the rock is quite permeable. Secondary copper and iron minerals line and in-fill cavities, form both irregular and coliform masses, and fill fractures and rim sulphides (Figure 7-7).

Gold occurs as native grains, most commonly in cavities with limonite or in limonite adjacent to sulphides, but also in malachite, plagioclase, chlorite, and rarely in quartz grains. Gold is rarely greater than five microns in size.

Figure 7-4 Examples of Copper Sulphide Mineralization Styles at Carmacks Copper

Top left image: disseminated chalcopyrite mineralization in undeformed amphibolite (WC-025 317.90m); top right image: disseminated chalcopyrite mineralization in the quartz-plagioclase-biotite schist (CN15-024 44.50m); bottom left image: foliaform chalcopyrite mineralization in the foliated amphibolite (WC-005 254.4m); bottom right image: net-textured bornite-chalcopyrite-molybdenite mineralization in the diatexite migmatite (WC-002, 147.31m). All scale bars are 1cm.

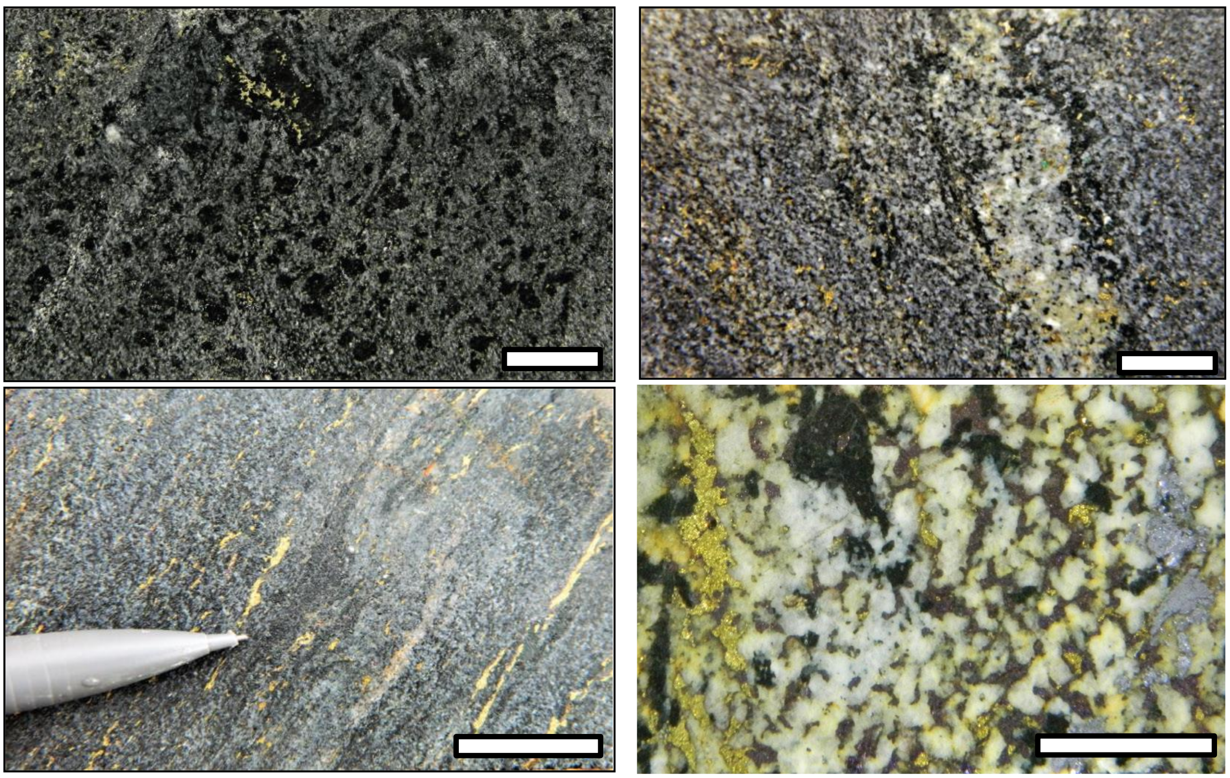


Figure 7-5 Reflected Light Petrography of Copper Mineralization at Carmacks Copper

Left image: Sample from zone 1 shows metamorphosed Hornblende-Biotite diorite with blebby bornite with an inclusion of chalcopyrite (both altered to chalcocite and malachite along fractures and borders of the sulphides against silicates); intergrown with plagioclase, hornblende, biotite and minor sphene. Right image: sample from Zone 2000S. Bornite and chalcopyrite with no oxidation, representative of weakly foliated hornblende diorite that contains blebs of chalcopyrite-bornite, typical of sulphide domain mineralization in this zone.

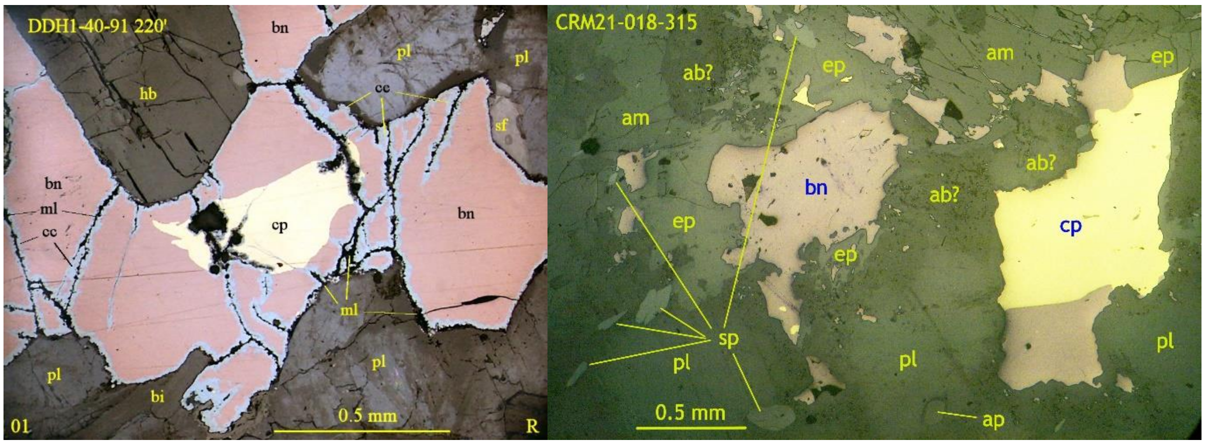


Figure 7-6 Gold Telluride Inclusion in Bornite with in Partial Melted Amphibolite (WC-002 194.25 m) and Gold-silver Telluride Inclusions in Net-textured Bornite within Diatextite Migmatite (WC-008 174m)

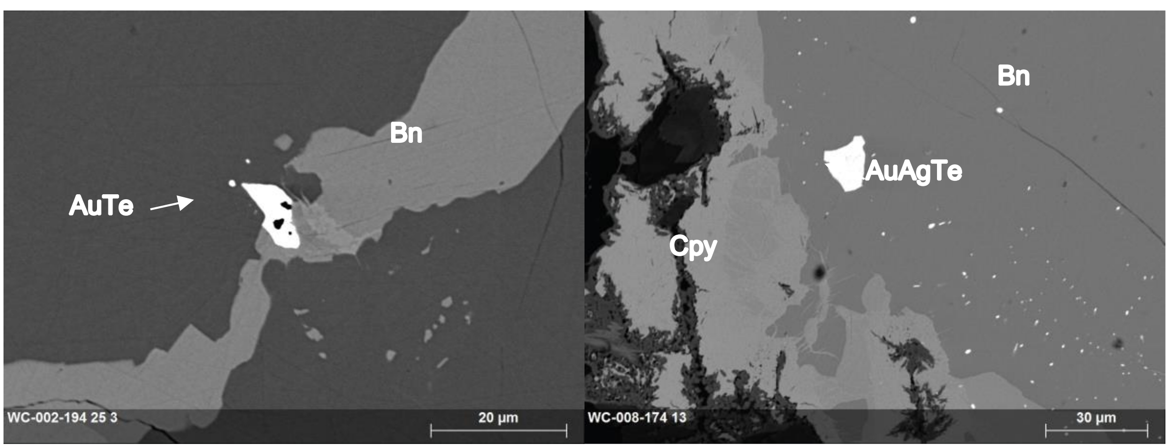
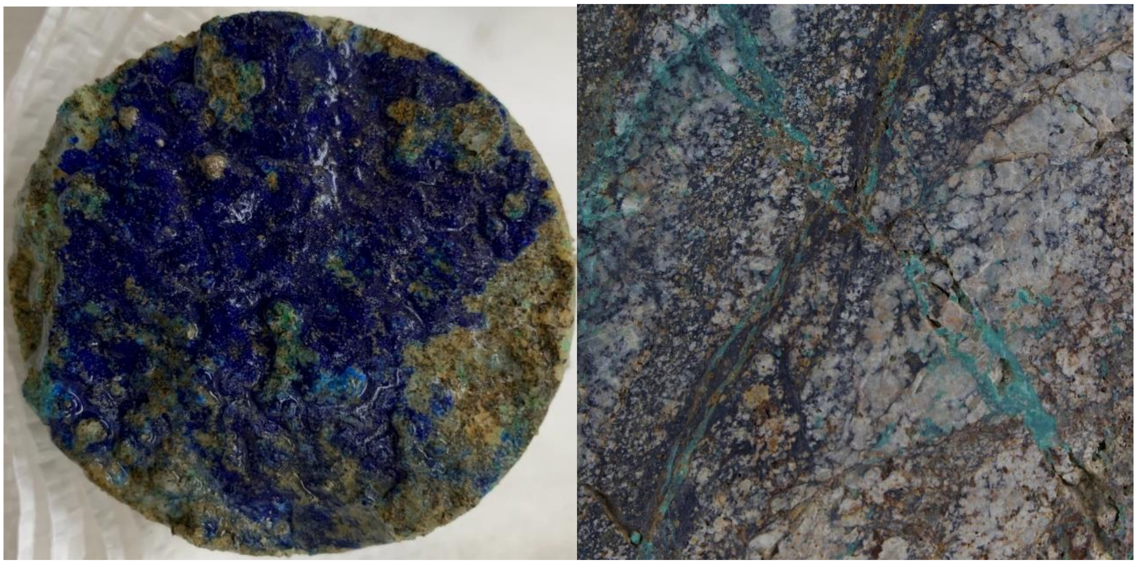


Figure 7-7 Copper Oxide Mineralization in the Form of Malachite and Azurite with Limonite Staining (Left) and As Fracture Network of Malachite and Chrysocolla (Right Image)



8 DEPOSIT TYPES

The following description of the Deposit Model of the Cu-Au-Ag deposits of the Carmacks Project is extracted from a recent paper written by Kovacs et.al., 2020 (and references therein), titled "Carmacks Copper Cu-Au-Ag Deposit: Mineralization and Postore Migmatization of a Stikine Arc Porphyry Copper System in Yukon, Canada". The Cu-Au-Ag deposits of the Carmacks Project, and the related Minto deposit, are considered rare examples of metamorphosed porphyry Cu systems (Kovacs et al., 2020).

8.1 Deposit Model

Since discovery of the Carmacks Copper and Minto deposits in the 1970s, several models have been proposed for their genesis, including:

- copper mineralization in digested Triassic volcanic rocks,
- metamorphosed red-bed copper,
- deformed and metamorphosed porphyry copper-gold,
- iron-oxide copper (IOCG), and
- shear-hosted hydrothermal system generated in the ductile root zones of a porphyry system.

The most current geologic and geochronologic constraints indicate that mineralization was an inherited feature of a Late Triassic protolith, which was subsequently metamorphosed in the latest Triassic and texturally modified prior to subsequent magmatism. For this reason, a synmetamorphic or syn-Granite Mountain batholith model for ore formation is unsupported. A viable deposit model must therefore recognize those mineralogical, textural, or geochemical features related to postore modification versus those inherited from the protolith. A deposit model for the mineralized protolith should also be permissible within established tectonic and geodynamic constraints.

The recognition that the least deformed and migmatized host rocks at the Carmacks Copper deposit contain low-grade, disseminated Cu as a chalcopyrite ± pyrite assemblage hosted in biotite-bearing and K-enriched host rocks is consistent with a porphyry copper deposit model. Hypogene grades from ~0.2 to 1% Cu and ~0.1 to 1 g/t Au at the Carmacks Copper, Minto, and Stu systems are within the range of typical porphyry copper grades globally, with the caveat that post-ore processes may have affected grade. Copper to gold ratios of 23,000 to 34,000 are also typical of gold-bearing porphyry copper deposits.

Although no intrusive phases related to the premetamorphic hydrothermal system are recognized at Carmacks Copper, it is permissible that the population of 217.53 ± 0.16 Ma igneous zircons represents magmatic activity temporally and genetically related to >212.5 ± 1.0 Ma copper mineralization. Hydrothermal features such as veins, alteration halos, or hydrothermal breccias are not recognized through the overprinting effects of metamorphism, penetrative deformation, and partial melting. However, the general lack of quartz rich domains within metamorphic rocks at Carmacks Copper suggests that quartz-sulphide veins were likely absent from the protolith. It is therefore likely that protolith mineralization was introduced as disseminations or as sulphide-dominant veinlets in conjunction with widespread biotite ± magnetite alteration.

Together, these observations suggest that the Carmacks Copper and Minto deposits each preserve the high temperature potassic core of a porphyry copper system. Several features listed above are also consistent with alkalic porphyry affinity:

- low abundance of pyrite,

- association with alkaline intrusions,
- low volume or absence of hydrothermal quartz, and
- Cu-Au metal tenor (compared to Cu-Au ± Mo in calcalkalic porphyry systems).

The interpretation of the Carmacks Copper and Minto deposits as metamorphosed porphyry copper systems is further supported by their temporal and lithotectonic affinity with porphyry belts in British Columbia. First, correlation of metavolcanic host rocks at Carmacks Copper with Stikinia arc equivalents in Yukon supports a similar tectonic and geodynamic setting to porphyry systems in British Columbia. Second, the ~217 to 213 Ma age of mineralization at Carmacks Copper constrains the system to within the prolific 227 to 178 Ma epoch of porphyry Cu mineralization in the Stikinia and Quesnellia arcs of British Columbia, and broadly coincident with peak productivity in Stikinia (e.g., Schaft Creek ~222 Ma, Galore Creek ~210–205 Ma, Red Chris ~204 Ma).

8.2 Comparison with Other Examples of Metamorphosed Porphyry Cu Systems

Although relatively rare, examples of metamorphosed porphyry copper systems have been documented on all continents (Kovacs et al., 2020). All examples documented in the literature are Precambrian and have the large tonnage and low grades typical of many calc-alkaline porphyry copper systems. In contrast, the Carmacks Copper and Minto deposits are distinctly younger (Mesozoic), an order of magnitude smaller than the Precambrian systems, but with generally higher grades.

In all examples of metamorphosed porphyry copper systems, the hypogene sulphide mineralization occurs as disseminations and foliaform stringers in foliated biotite (±magnetite)-rich schist and gneiss, with biotite enrichment interpreted to reflect the metamorphosed potassic core of the porphyry system. The Precambrian examples are also characterized by quartz veins and stockworks; features that are not typical of the Carmacks Copper and Minto deposits. The Calingiri (Western Australia), Chapada (Brazil), Carmacks Copper, and Minto deposits have in common that host lithology and porphyry copper mineralization were deformed and metamorphosed to amphibolite facies conditions before intrusion by younger granitoids. However, evidence of partial melting and development of migmatitic rocks is only documented at the Calginri, Carmacks Copper, and Minto deposits. The degree of migmatization appears more advanced at Carmacks Copper and Minto, where the mineralized rafts are engulfed in the Granite Mountain batholith and where net-textured sulphides in migmatitic rocks suggest the presence of a sulphide melt.

Deformation, metamorphism, and tectonic burial of the Carmacks Copper and Minto deposits occurred during imbrication of the Intermontane terranes in latest Triassic and earliest Jurassic (~205–195 Ma), shortly after formation of the deposits at ca. 217 to 213 Ma. Syncollisional magmatism of the Minto plutonic suite (ca. 205–194 Ma, including the Granite Mountain batholith) at midcrustal depths, is partly responsible for the destruction, or lower preservation potential of, preaccretionary, Stikinia-affinity copper systems in Yukon compared to British Columbia. Carmacks Copper and Minto are thus the surviving remnants of a preaccretionary porphyry belt in the northern Cordillera that originally may have been exceptionally endowed.

8.3 Conclusions

The Carmacks Copper deposit represents a Late Triassic (ca. 217–213 Ma) alkalic porphyry system that is hosted in metavolcanic rocks of the Stikinia arc terrane (Kovacs et al., 2020). The original porphyry deposit was deformed and metamorphosed at amphibolite facies in the latest Triassic (ca. 206 Ma), leading to mobilization of a chalcopyrite-dominant sulphide assemblage along foliation. Subsequent intrusion of the Granite Mountain batholith in the Early Jurassic (ca. 200–194 Ma) resulted in anatectic melting of the mineralized xenoliths at midcrustal depths ca. 197 to 196 Ma, predominantly through a water-fluxed melting mechanism, but locally through biotite dehydration melting. Sulphide melting took place under the same ~600° to 750°C conditions of silicate melt formation, whereby an immiscible Cu-Fe-S ± Bi-Au-Ag-Te melt phase preferentially accumulated in neosome. Two distinct types of molybdenite with $^{187}Re/^{187}Os$ dates ranging from 212.5 to 198.5 Ma occur in the Carmacks Copper ore: Mol-1 comprises kinked and deformed grains and Mol-2 consists of euhedral grains intergrown with net-textured sulphides. The range of $^{187}Re/^{187}Os$ dates is interpreted to reflect mixing of these two molybdenite types with Mol-1 retaining radiogenic Os from protolith ore (>212.5 Ma) and Mol-2 reflecting the age of recrystallization of net-textured Cu mineralization at <198.5 Ma. In comparison with Carmacks Copper, the Minto deposit is geologically similar but host rocks are at a more advanced state of migmatization.

The preservation of the Carmacks Copper and Minto deposits results from tectonic burial of porphyry copper systems soon after their formation, but prior to profound exhumation of the Intermontane terranes in Yukon, compared to similar deposits in British Columbia, which escaped both tectonic burial and exhumation. The Carmacks Copper and Minto deposits share similarities with global examples of metamorphosed porphyry copper systems but are distinguished by their young, Mesozoic age, their advanced state of migmatization, and occurrence as rafts within a younger, post-mineralization granitoid batholith.

9 EXPLORATION

Project) in January 2019. Following acquisition of the Carmacks North project, Granite Creek secured and reviewed an extensive, privately held exploration database and, based on those findings, launched the 2019 exploration program which was designed to refine drill targets for the 2020 drill campaign through the use of soil sampling and IP surveys.

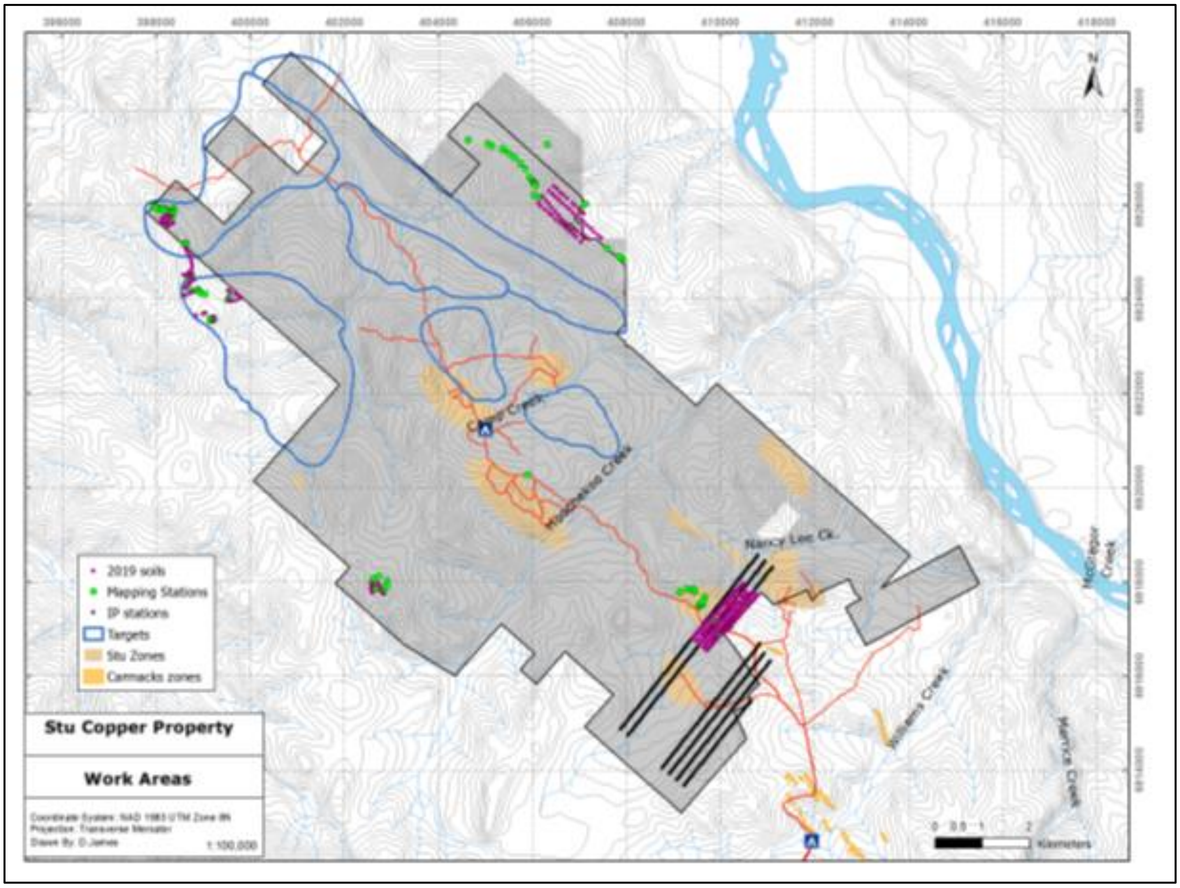
In August 2020 the Company acquired the Carmacks deposit through its acquisition of Copper North, which altered the exploration plans for the Company. The 2020 Exploration season comprised of re-logging and re-sampling historic core, drilling and soil sampling at Carmacks North's Zone A and a small diamond drill program at Carmacks Zone 13 deposit.

The 2021 Exploration season was primarily focused on resource expansion drilling at the Carmacks deposits (see section 10 below) with limited trenching at Carmacks North's Zone A. An IP survey was conducted across Zone A and extended eastward over areas with anomalous soils. An RC drilling program was run to test geophysical targets at Zone A but technical drilling difficulties resulted in the RC drill being moved to Zones 2, Zone 5 and West of Zone 12.

9.1 2019 Field Program – Carmacks North Property

The 2019, the Carmacks North exploration program included reconnaissance mapping and soil surveys along with an IP survey (Figure 9-1).

Figure 9-1 Location of 2019 Exploration Activities: IP survey – black lines, soil sample locations – purple dots, mapping stations – green dots reflect mapping stations



9.1.1 South Target Area

In addition to geological mapping and access rehabilitation, an Induced Polarization (IP) survey consisting of seven lines totaling 24-line kilometres was completed in the South Target area (Figure 9-1). Designed to confirm the projected northern continuation of Copper North mineralized zones hidden by cover, the survey partially covered the Gran Zone, Zone 2 (north) extension and the northern extension of South Butter. Granite Creek worked with geophysicists to design a survey that would detect potential mineralization along strike from the adjacent Carmacks project and, importantly, penetrate deep enough to detect sulphide mineralization beneath the shallow copper oxide layer.

The results of the IP program in Zone 2 extension showed deep broad chargeability response with localized shallower pods of elevated chargeability. The discrete higher chargeability shallower responses had somewhat moderate correlation with elevated copper soils collected in the season, however as the strength of the soil response and size of the chargeability did not provide sufficient merit to promote the exploration ranking of this target area.

The Gran Zone area is defined by a weak magnetic anomaly associated with moderate to highly anomalous copper in soils in an area of deep glacial till. The IP over the area showed elevated chargeability over the broad area with some more distinctive higher chargeable zones. Trenching in the general area of Gran

Zone in 1981 identified weakly foliated granodiorite, but no anomalous copper was found in rock samples. The source of the anomalous copper in soils may be coincident with the zone of higher chargeability but remains untested. Further work should be done to evaluate the glacial direction thickness and potentially test the zones of higher chargeability in this area.

The area northwest of the south butter target area showed some of the highest chargeability zones on the geophysical line, occurring 70m below surface and approximately 150 m wide. Further IP is recommended, as the chargeable zone appears near the margin of this survey and appears to weaken in the northern lines.

The four IP lines that occurred southwest of Zone 2 and south of the Gran Zone show erratic chargeability over the Cretaceous Carmacks volcaniclastics. Zones of elevated chargeability occur south of the interpreted Carmacks volcanics and extend to surface. However, the soil geochemistry does not appear to reflect anomalous copper over these areas (Figure 9-2).

9.1.2 East Target Area

Crews also focused on the ‘East Target’ and the historic ‘Zone D’ which lie within a regional scale, northwest trending structure hosted in Povoas Formation volcanics. Zone D was initially discovered in 2012 and was added to the Carmacks North Project during a 2017 staking program. Compilation work completed by the Granite Creek brought the zone into focus and an additional 21 new claims were staked on the northeast side of the property (see news release dated July 15t^{h}, 2019). Follow-up prospecting revealed a fracture zone with malachite and iron oxides within the larger northwest trending structure. Anomalous copper values in soils suggest mineralization continues along the structure in both directions from Zone D. The presence of copper mineralization is confirmed in this area by a rock sample collected at Zone D which returned 0.74% Cu and 0.4 g/t Au, and sample 1481712 which was collected 1200m SE of Zone D and carried anomalous copper (180 ppm).

Figure 9-2 Zone 2 Extension (north) Showing Cu in Soils Collected in 2019

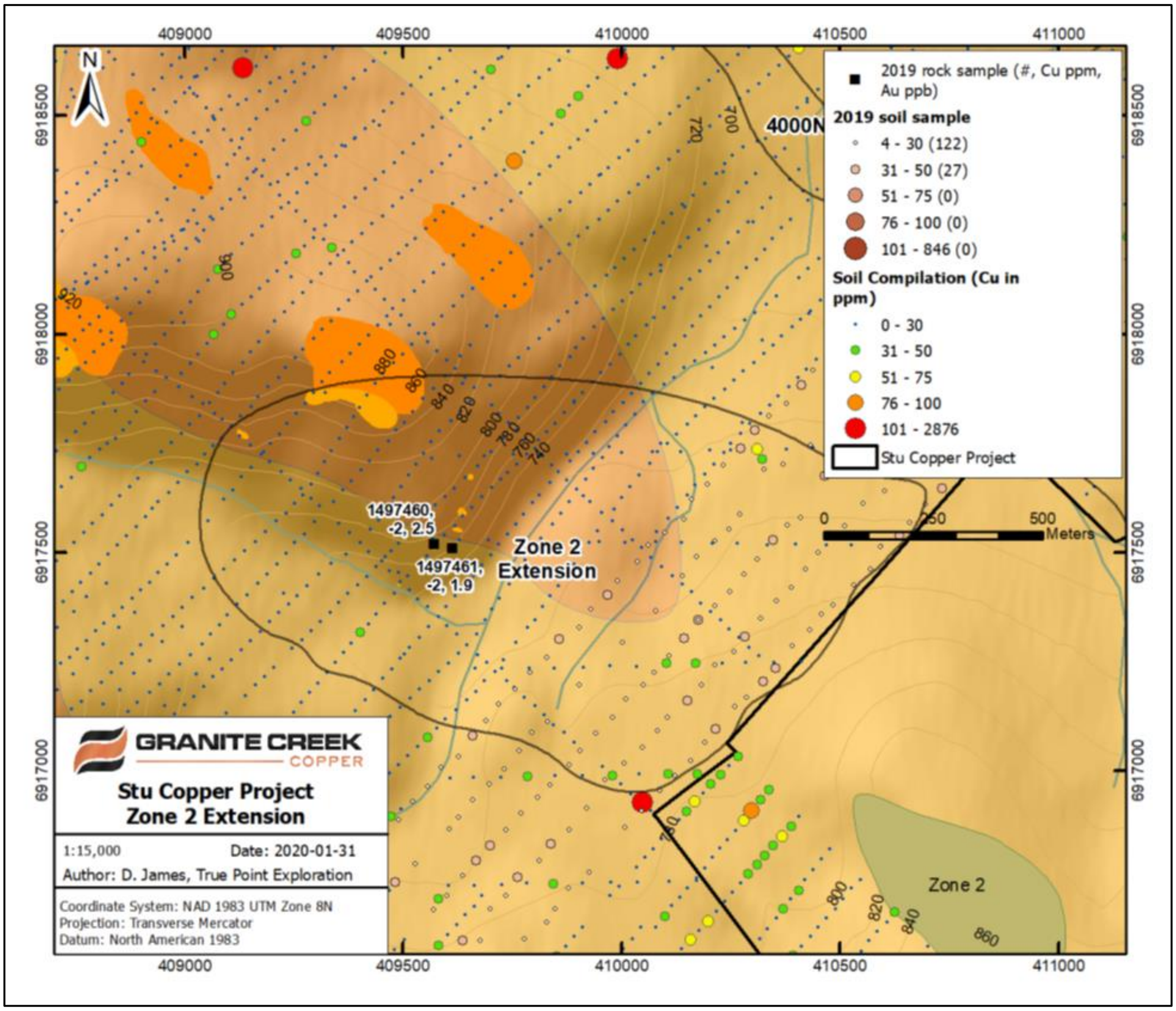
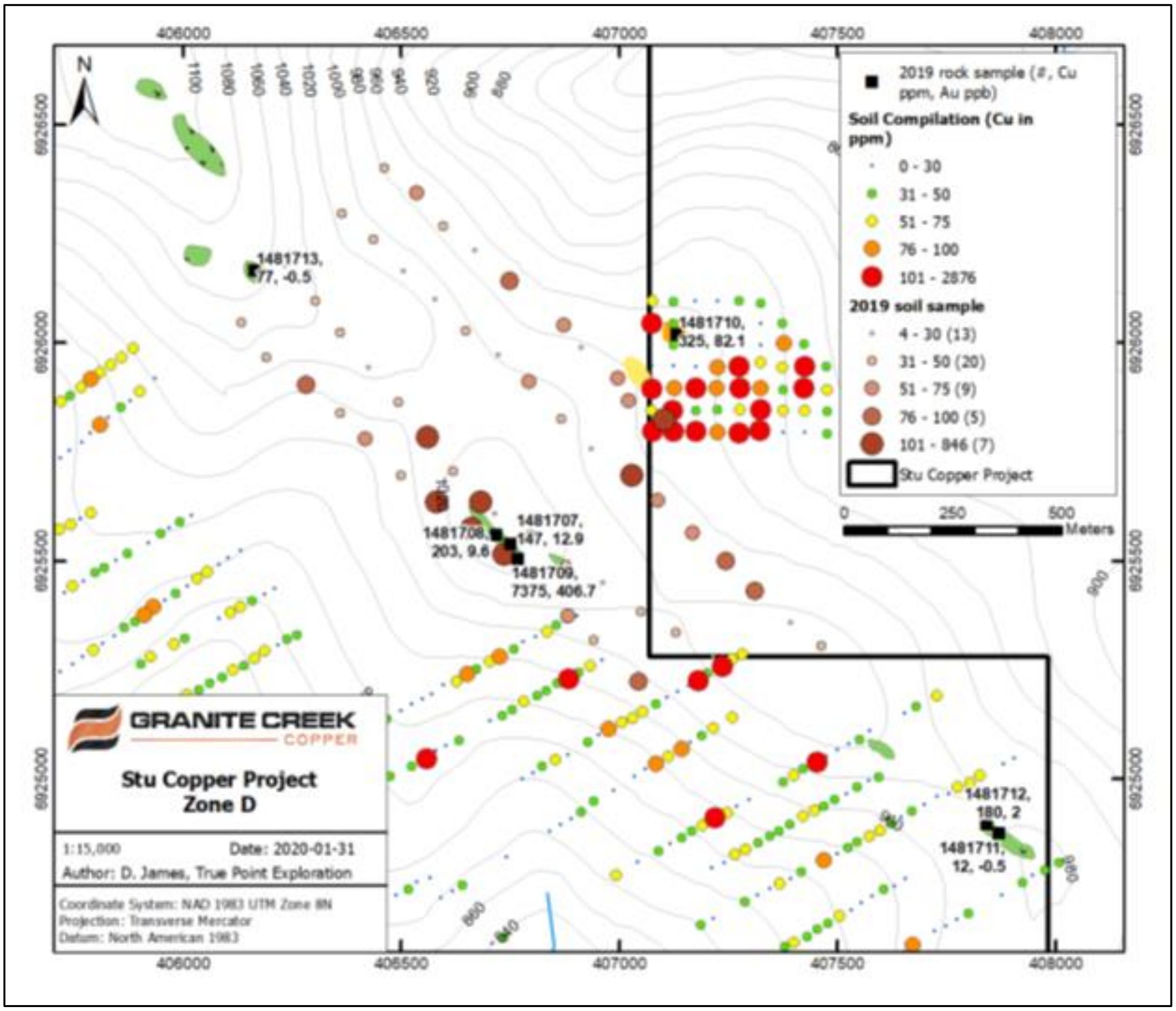


Figure 9-3 Zone D Soils and Rock Sample Locations

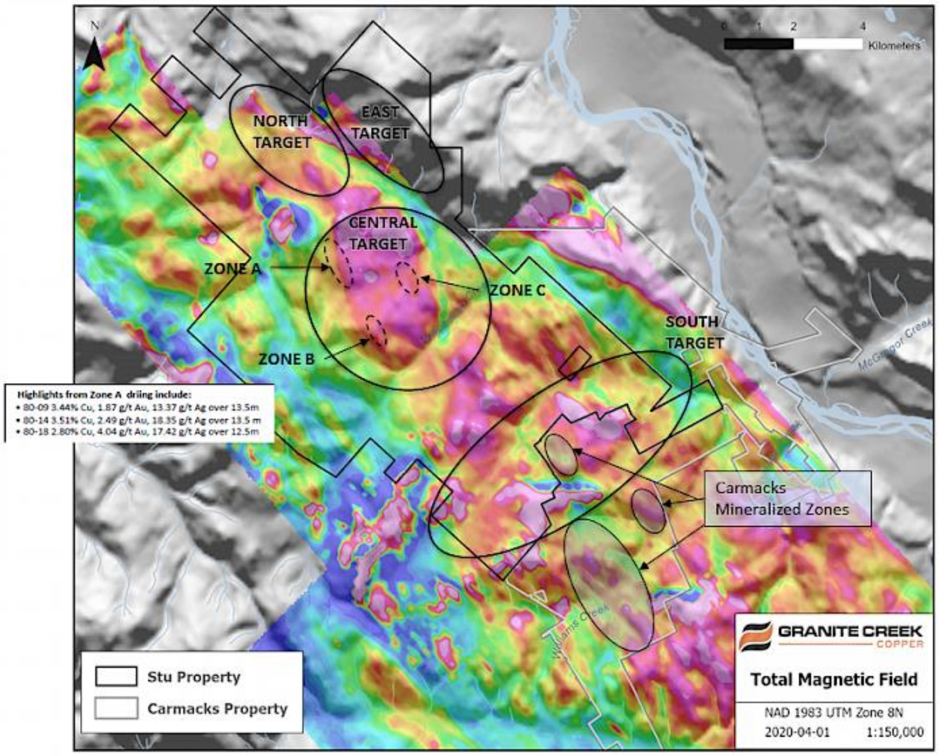


9.2 2020 Acquisition of Airborne Geophysical Data

On April 7, 2020, Granite Creek announced the acquisition of airborne geophysical data, including magnetic and radiometric surveys, covering the Carmacks North Property.

The addition of these airborne magnetic (Figure 9-4) and radiometric surveys, which show distinctive signatures over areas of known mineralization, along with new exploration results from 2019 and compilation of historic data, enabled Granite Creek to develop a predictive model of the geologic and structural controls for Minto-style copper and gold mineralization to guide exploration in 2020.

Figure 9-4 Carmacks North Property Total Magnetic Field with Surface Geochemical Target Areas and Mineralized Zones



9.3 2020 Historic Core Re-logging and Re-sampling Program

In July of 2020, Granite Creek launched a re-logging and re-sampling program on core that was drilled in 1980 by previous operators on the property's A Zone. The goal of the program was to provide a full multi-element assay of the historic core and for company geologists to get a better understanding of alteration, and various styles of mineralization in order to assist in modeling of the zone as well as targeting for future infill drilling programs. This resulted in 712 total drill core samples collected and assayed across 9 historic drill holes.

In addition to the core re-sampling limited soil sampling was conducted over known mineralization at Zone A to determine the precious metal and indicator mineral response because the historic soil sampling was analyzed only for copper.

The results of the soil survey showed anomalous results near the known mineralized zones, with some dispersion occurring downslope of the zone. A second copper anomaly appears to occur east of Zone A with a similar trend to mineralization. This was followed up in 2021 with additional IP survey and some trenching.

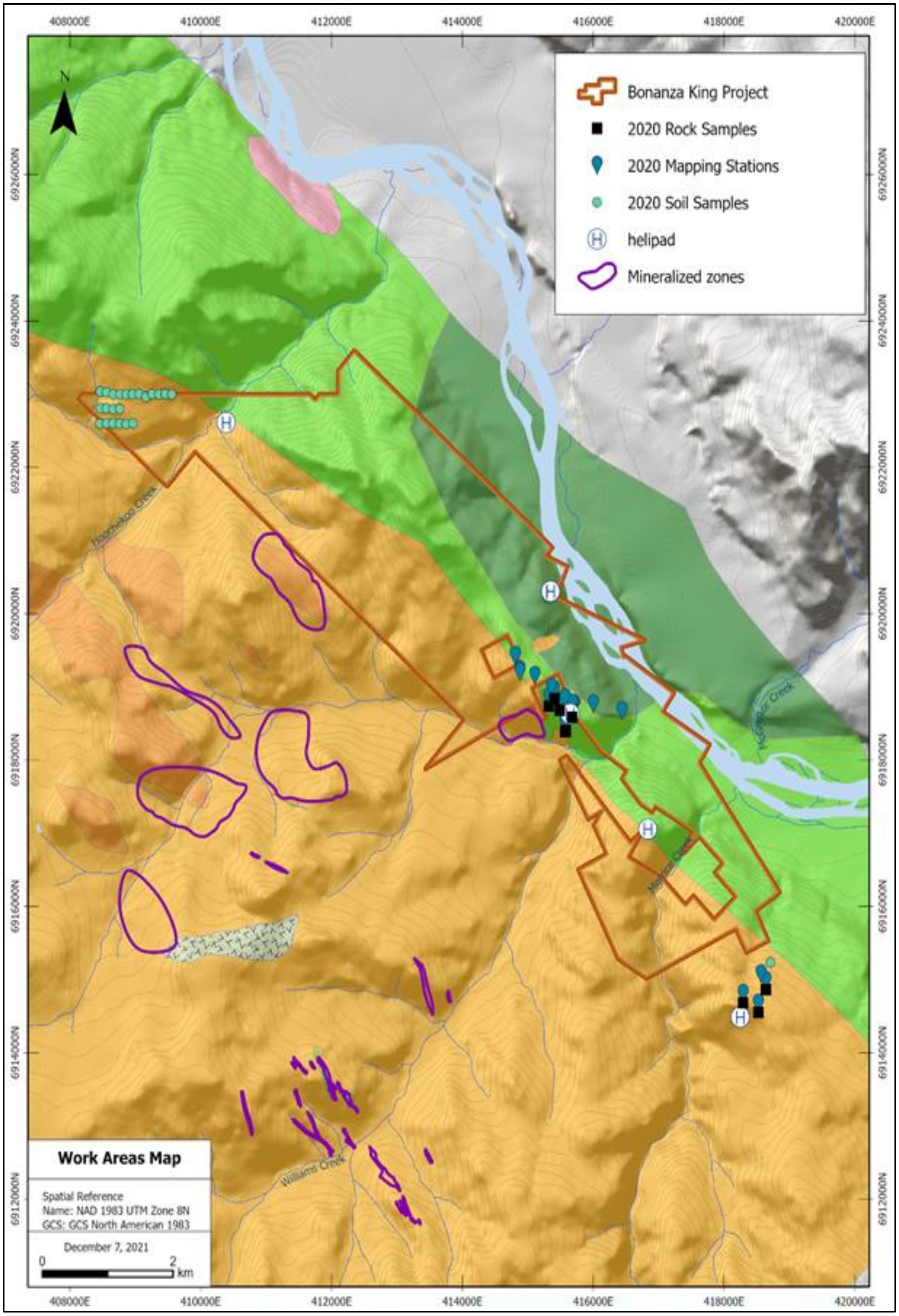
9.4 2020 Trenching Program

During the Phase 2 program, a single 23.5m trench (TRSTU20-001) was dug (E-W) on the eastern-side of Zone A, intersecting foliated granodiorite (trending NW-SE) with both oxide ± sulphide mineralization.

9.5 2020 Soil and prospecting Program

In 2020 a mapping, prospecting and soil sampling program was conducted over the Bonanza King area along the eastern flank of the property following a large-scale Hoocheekoo Fault that transposes the Povoas Formation adjacent to the Granite Mountain Batholith (Figure 9-5). The results of this showed that there was reasonable agreement with historic soil samples and the location of the fault, but vegetation and lack of outcrop made access challenging and resulted in incomplete program.

Figure 9-5 2020 Soil and Mapping at Bonanza King



9.6 2021 IP Survey – Carmacks North Property

Simcoe Geophysics completed nine profile lines for a total of 21.7 line km induced polarization (IP) survey on the Company's Carmacks North target area (Figure 9-6 and Figure 9-7). A total of 43 anomalous IP responses were identified in the IP survey. The general trend of the strongest IP responses outline two 1.2 km long northwest trending corridors, spaced approximately 900 m apart. These two zones are also underlain by zones of magnetic vector intensity, interpreted to reflect areas of change in batholith composition. The western anomaly is in general agreement with historic and current drilling of Zone A, suggesting these are an offset of a deeper, stronger zone with higher chargeability (Figure 9-7) with the eastern trend being untested. These IP anomalies are generally coincidental with elevated copper in soil collected in 2020.

Figure 9-6 Oblique view of the UBC 2D DCIP inversion sections of the chargeability from 20.8 km of IP collected by Simcoe Geophysics over Carmacks North (from news release dated July 14, 2021)

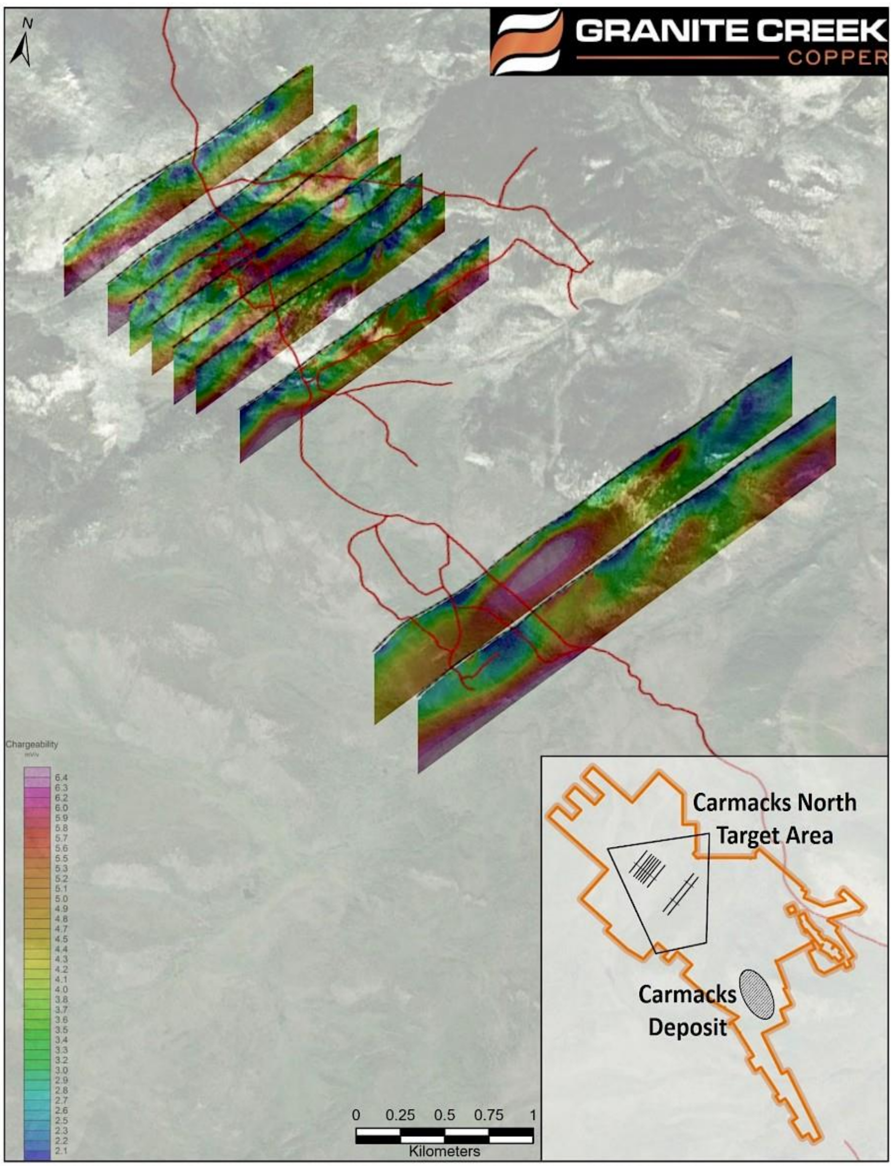
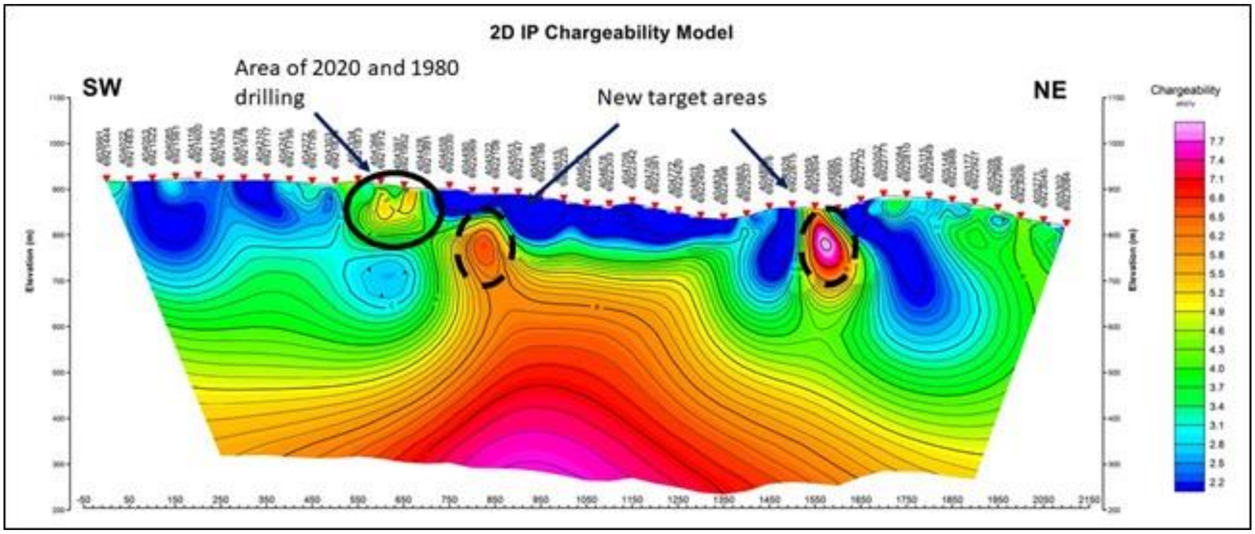


Figure 9-7 2D IP Chargeability Model of Line 4 from the Left (see Figure 9-6 above) (from news release dated July 14, 2021)



9.7 2022 IP Survey – Carmacks Property

From May 25 to June 22, 2022 Simcoe Geoscience Limited (Simcoe) completed an Alpha IP – Wireless Time Domain Induced Polarization Survey over the Carmacks property (Simcoe Geoscience Project # SGL-22118). Six (6) profiles, totalling 20.0-line km of IP data were acquired using 'pole-dipole' configuration with 100 m dipole and injection spacings (Figure 9-8). Line lengths are variable from 2.4 Km up to 4.0 Km and line spacings was variant based on the area of interest. Current injections at every 100m were made by adopting "reverse and forward" pattern and "off-end" for maximum depth penetration and highest resolution.

The exploration objectives of the Alpha IPTM survey at Carmacks Copper Grid were to map high chargeability and low resistivity geophysical responses associated with disseminated and sub-massive sulphides for copper mineralisation.

The useful geophysical data for targeting the potential Copper mineralization are the magnetic data for structural mapping and the IP chargeability for drill targets.

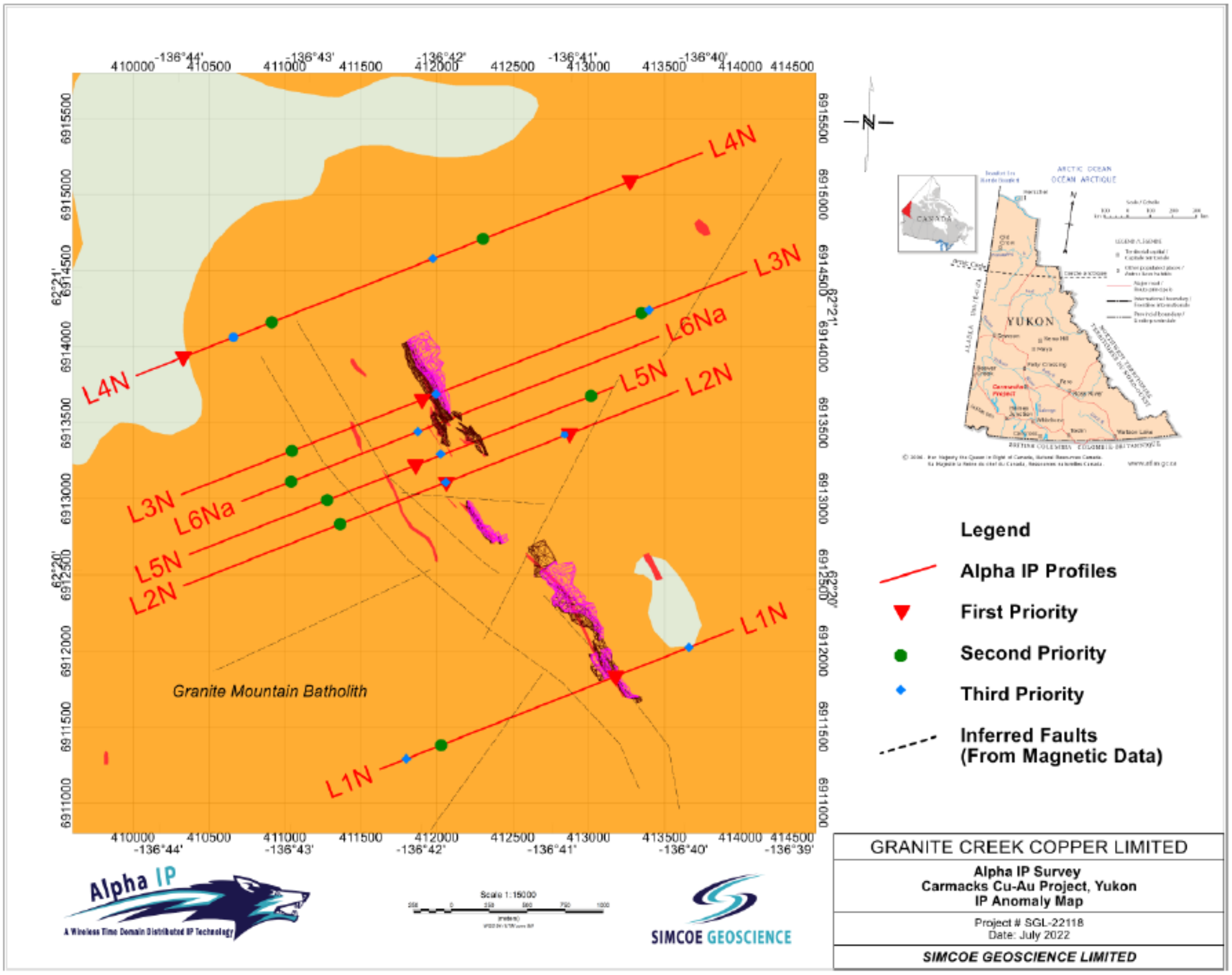
Over the Carmacks grid, at least twenty-six (26) chargeability anomalous zones are interpreted along six (6) profiles as significant targets to follow up. Out of the twenty-six (26) anomalous chargeable zones, seven (7) are considered first priority, nine (9) second priority and, ten (10) third priority targets. Alpha IP Anomaly Map and Interpreted First, Second and Third priority targets along with Inferred Faults, interpreted from airborne magnetic data are shown in Figure below.

The chargeability ranges from 0mV/V – 6 mV/v and resistivity range of 400Ωm - 25000Ωm. In general, this area comprised of moderate to high resistivity rocks units and 6% to 12% sulphides, confirmed with chargeability.

The interpreted anomalous zones were ranked according to the chargeability amplitude, size, possible profile to profile continuation and multi-parameter (resistivity and chargeability) association. The anomalous zones consist of high chargeability and low to moderate resistivity values. The selected anomalies are presented in table below.

The first priority targets with high chargeability and low to moderate resistivities, could be associated with disseminated sulfides, i.e., pyrite or arsenopyrite, in silicification alteration zones.

Figure 9-8 Alpha IP Anomaly Map and Interpreted Targets



9.7.1 Results

Line 1N

One (1) first priority target S1 is selected along Line 1N, (Figure 9-9). It is located in the moderate to low resistivity zone and has strong to moderate chargeabilities. Other anomalies which are selected along this line are one (1) second priority and two third priority target. P1 is potentially the extension of W1 at depth.

Line 2N

Two (2) first priority targets S1 and S2 are selected in Line 2N (Figure 9-10). They are located in the moderate to high resistivity zones and has strong to moderate chargeabilities. One (1) second and two (2) third priority targets are selected from this line as well. First priority targets are located in the central and northeastern parts of the profile and are fault/contact bounded. The P1 and P2 are potentially the extensions of S1and S2, respectively, at depth.

Figure 9-9 Line 1N Chargeability and Resistivity Section Interpretations

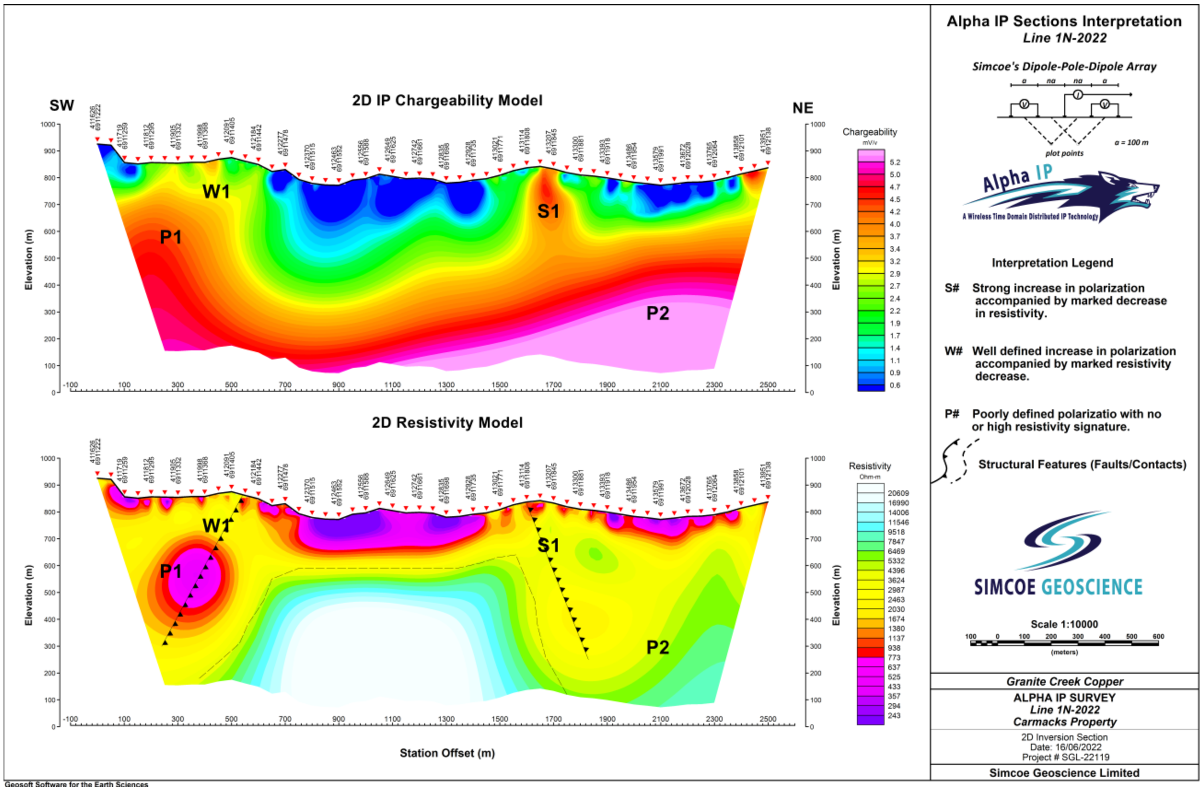
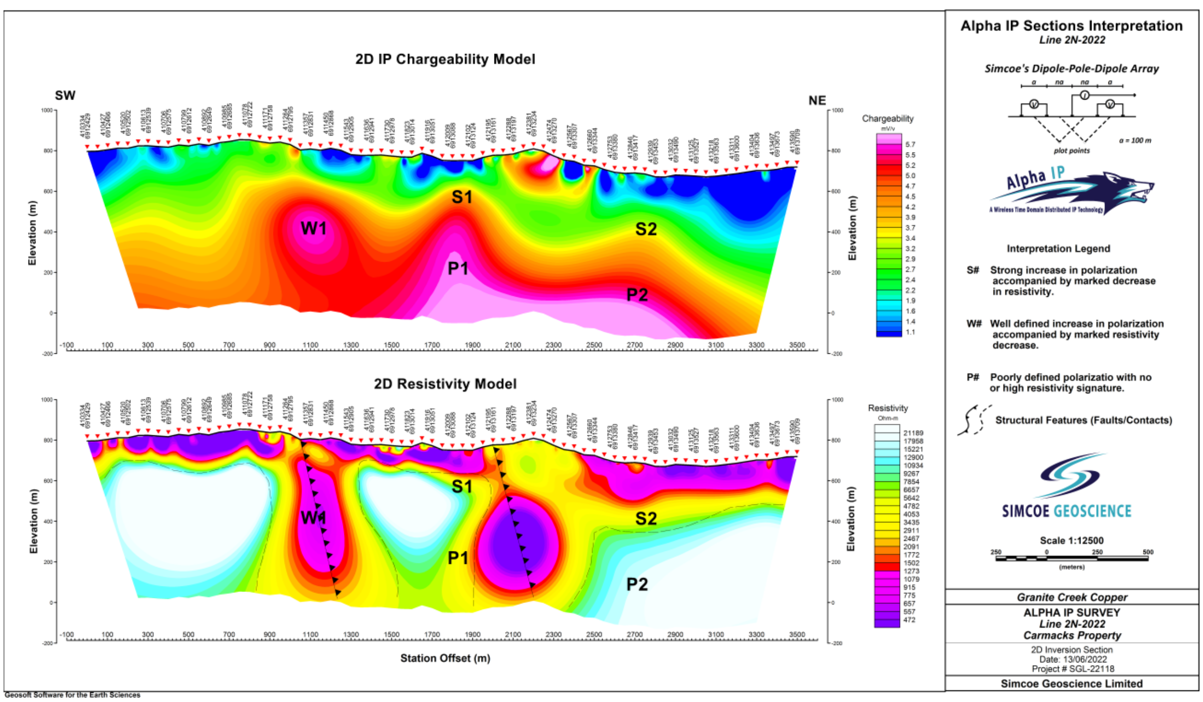


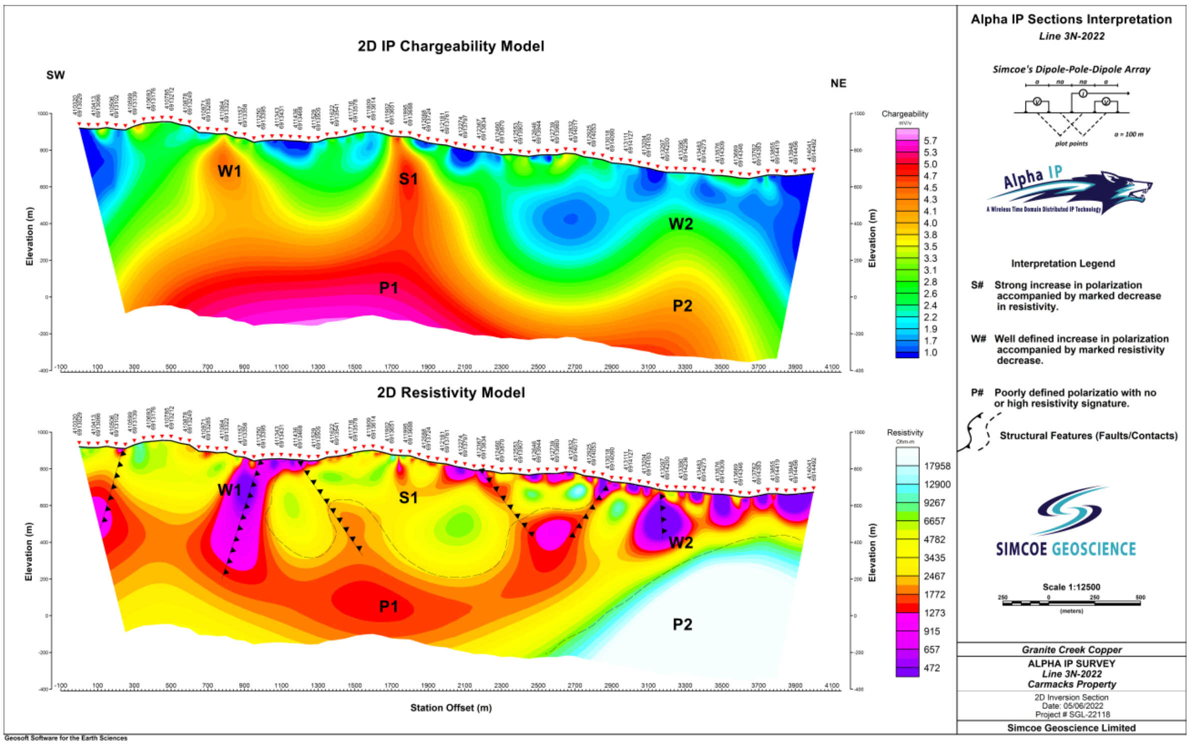
Figure 9-10 Line 2N Chargeability and Resistivity Section Interpretations



Line 5N

One (1) first priority target S1 is selected along Line 5N (Figure 9-11). It is located in the moderate resistivity zone and has strong to moderate chargeabilities. Two (2) second and one third priority targets are selected from this line as well. S1 is situated at the central part of the profile. P1 is potentially the extension of S1 at depth. W1 is the continuation of the same trend as of W1 at line 2N, they are both located in a low resistivity zone as well.

Figure 9-11 Line 5N Chargeability and Resistivity Section Interpretations



Line 6aN

One (1) second priority target W1 is selected along Line 5N (Figure 9-12). It is located in the low to moderate resistivity zone and has strong to moderate chargeability responses. One (1) third priority target is selected from this line as well. W1 is located in the southwest part of the profile on the same trend as of W1 on lines 5N and 2N. But it reaches to the surface at this line and extending to depths of about 400 m.

Line 3N

One (1) first priority target S1 is selected in Line 3N, (Figure 9-13). It is located in the low to moderate resistivity zone and has moderate chargeability. Two (2) second and two (2) third priority targets are selected from this line as well. W1 is located in the southwestern part of the profile and its on the same trend as of W1 anomalies at lines 2N, 5N and 6aN. The P1 is potentially the extension of S1 at depth. P2 is also extension of W2 at depth and potentially formational response and in the resistive intrusive unit.

Figure 9-12 Line 6ane Chargeability and Resistivity Section Interpretations

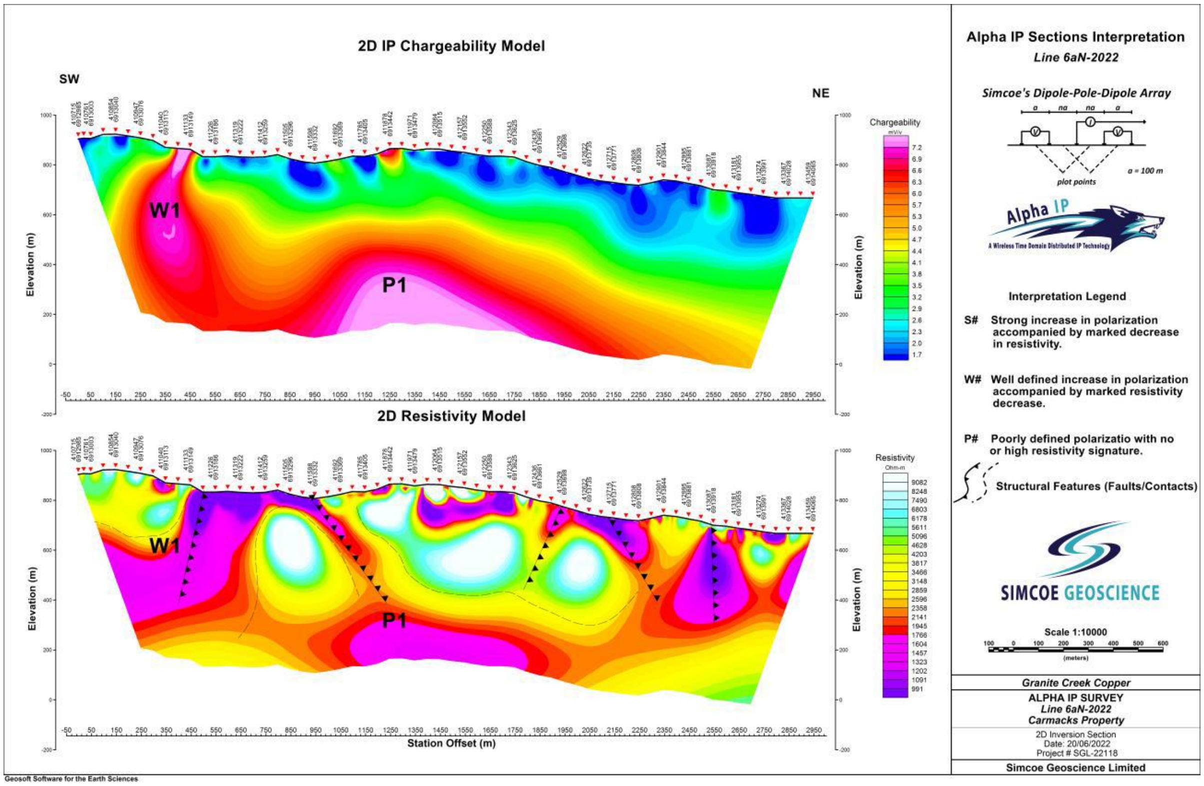
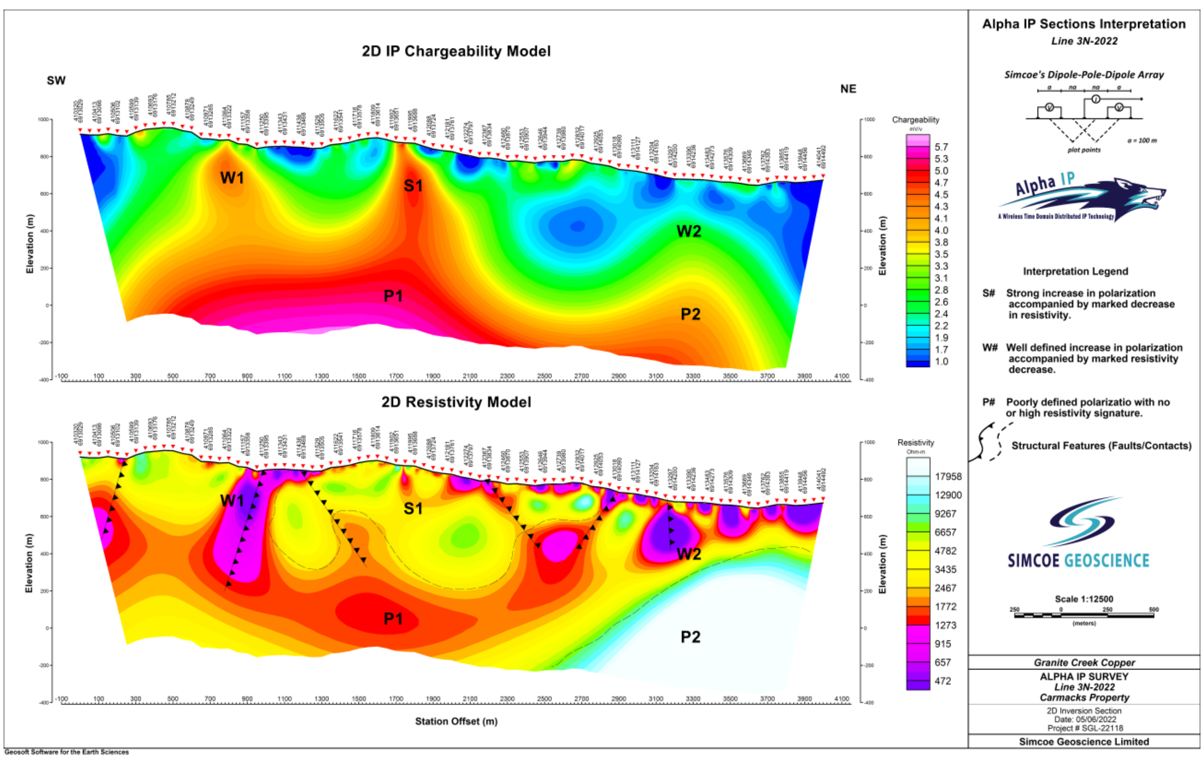


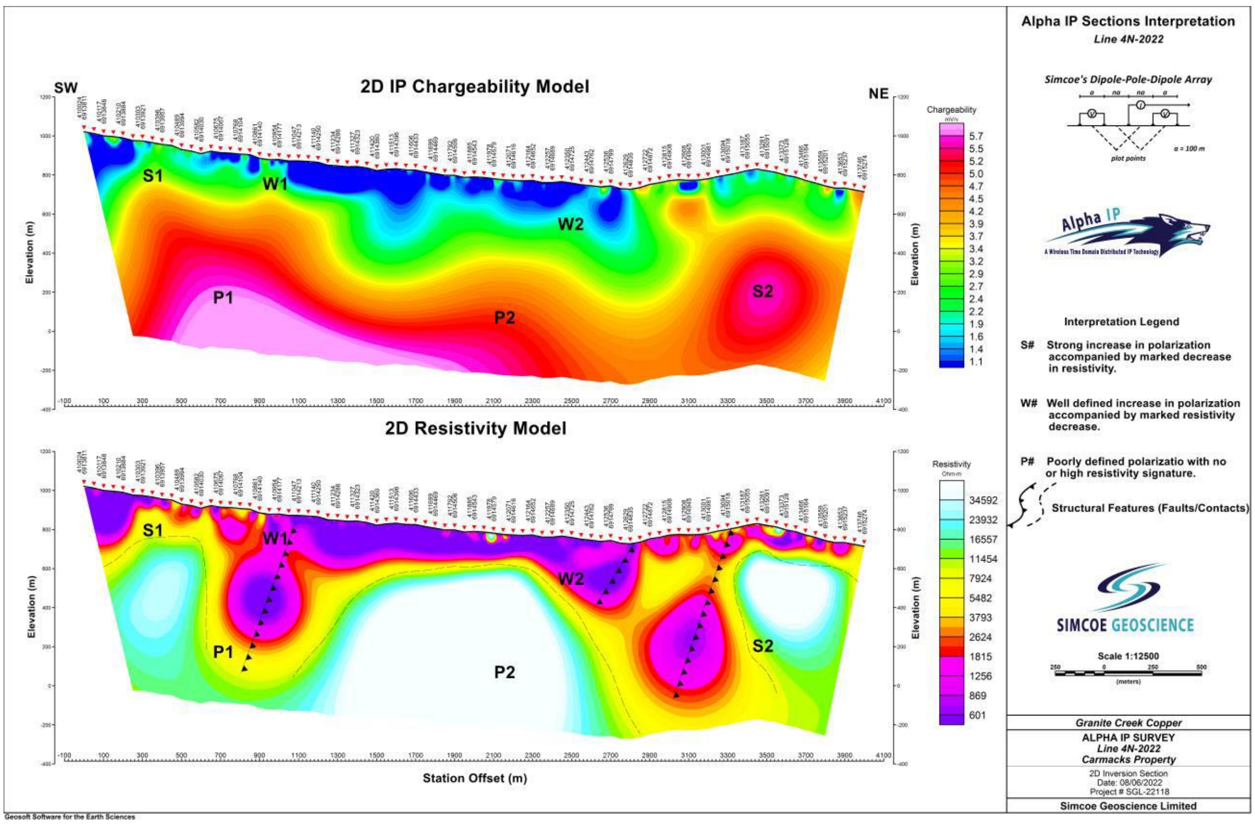
Figure 9-13 Line 3N Chargeability and Resistivity Section Interpretations



Line 4N

Two (2) first priority targets S1 and S2 are selected in Line 4N, (Figure 9-14). Both located in the moderate resistivity zones, S1 has moderate while S2 has strong chargeability reponses. Two (2) second and two (2) third priority targets are selected from this line as well. First priority targets are located in the southwest (S1) and northeast (S2) parts of the profile. The P1 is potentially the extension of S1 at depth. P2 is potentially formational response and in the resistive intrusive unit.

Figure 9-14 Line 4N Chargeability and Resistivity Section Interpretations

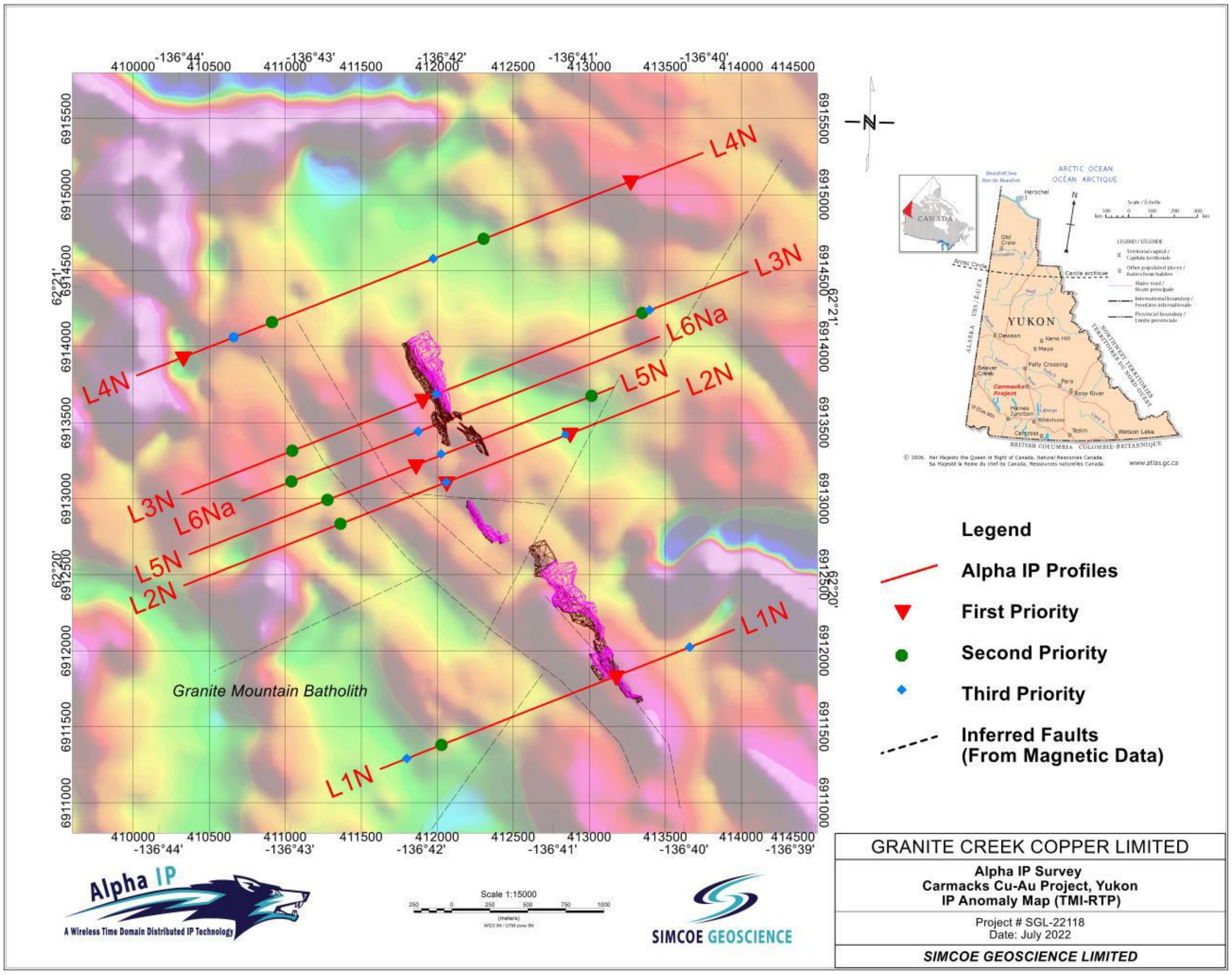


9.7.2 PLAN MAP INTERPRETATION

The magnetic data over the Carmacks project are provided by Granite Creek Copper. The magnetic data are reduced to pole (RTP). The first vertical derivative of the RTP is computed. The structural interpretation of the data is carried out and the results are shown in Figure 9-15. The inferred faults are trending mainly in the SW-NE direction. A few are trending in the near NW-SE direction. An intrusion is inferred in the NW part of the Project. The known mineralization (Oxide & Sulphide) is also displayed in relation to the inferred structures.

The selected targets over the RTP are shown in Figure 9-15. The inferred faults are displayed. Most of the first priority targets are located in and around the NW-SE trending structures and well correlated with known mineralization (Oxide & Sulphide). The helicopter magnetic data are provided by Granite Creek Copper.

Figure 9-15 IP Anomalies and Structural Interpretations Over The RTP



10 DRILLING

10.1 2014-2015 by Copper North

Copper North carried out limited drilling campaigns in 2014 and 2015 that totaled 4,358 m of drilling in 50 holes (Table 10-1). The exploration focused on extending the known mineral resources in an effort to expand the current measured and indicated mineral resources, as a first step in increasing potential mine life.

In 2014, Copper North initiated a diamond drilling program aimed at defining additional mineralization in Zones 2, 2000S, 12 and 13. The Zone 2 area is located approximately 2,500 metres to the north of the north end of Zone 1 (Figure 7-3); both Zone 1 and 2 were discovered by prospecting in 1971. Little work was done on Zone 2 following geochemical and geophysical surveys, and trenching. Evaluation of a trench on the Zone 2 discovery outcrop indicates a steep dipping mineralized structure trending southeast. Historic sampling of the discovery trench returned 1.0% copper over 45.7 metres within the sheared granite that hosts almost all mineralization at Carmacks. Zone 2000S is located immediately south of Zone 1 and was defined by previous drill holes and a distinct anomaly of low magnetic susceptibility caused by alteration associated with oxide mineralization.

Drilling was carried out by Kluane Drilling of Whitehorse, Yukon using a custom designed drill rig. Core size was NQ for the 2014-2015 drilling program, and a combination of HQ and NQ for the 2015 program. Table 10-2 summarizes the Copper North 2014-2015 drilling programs.

Table 10-1 Summary of Copper North 2014-2015 Drilling Programs

Zone Targeted	Number of Holes	Total Metres
12	6	394.52
13	20	1932.26
1, 4, 7	1	88.39
2	10	619.57
2000S	12	1195.07
Exploration	1	128.02
Grand Total	50	4,357.83

Generally, all drilling at Carmacks has been oriented to intersect the mineralized intervals at right angles, which means that most holes were drilled toward azimuth 245 to 248°. Most drill holes were drilled at a -50° dip, giving nearly true thickness intersections for most holes. Core was generally sampled in 1.0 m lengths for the 2014 drilling campaign but sample intervals were changed to 1.5 m for the 2015 drilling program. Core recovery was generally excellent both in wall rock as well as within the mineralization. Poor core recovery was encountered only where the drill hole intersected fault structures. Sample lengths do vary slightly depending on the lithology and mineralization style. The samples lengths were determined during logging by the geologist.

To test the mineralization to depth on Zone 2, ten drill holes (CN14-01 to 10) were undertaken and defined the mineralized structure over a distance of 450 metres. The drill holes intersected the mineral zone (Table 10-2) at depths between 9.6 metres and 81.0 metres. All holes intercepted the mineral zone and yielded an average of 10.5 metres grading a weighted average of 0.36% copper, 0.069 g/t gold and 4.37 g/t silver.

The trenching and drill hole intercepts in Zone 2 confirm the continuity of the mineralization to the south. Additional exploration is warranted to determine if the Zone 2 mineralization is an extension of Zone 1. Of interest, part of the target area is covered by near flat lying Cretaceous volcanic rocks, which may well provide an erosional cover that may preserve the oxidized mineralization. The preservation of oxide mineralization to depth is key to developing substantial oxide copper resources.

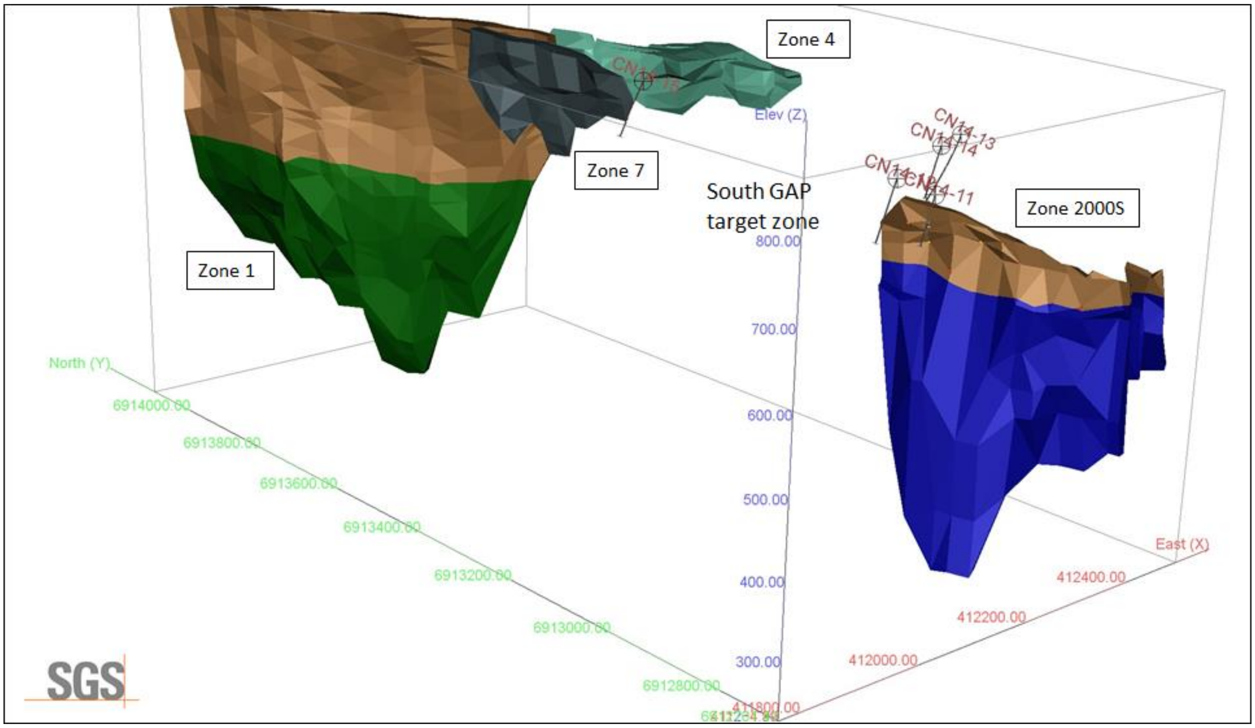
The South Gap target zone consists of an approximate 300 metre gap between the main proposed mining area of Zone 1 and the 2000S zone (Figure 10-1). The 2000S zone was intersected in 6 drill holes, with both oxide and sulphide mineralization. Drill hole WC92-01 intersected 10.67 m grading 0.67% copper at a depth of 30 to 41 metres. To follow up on this intercept, drill hole CN14-11 was collared 22 metres to the north northwest of WC92-01, and intersected 18.74 metres grading 0.58% copper, 0.189 g/t gold and 2.46 g/t silver at a depth 21 to 40 metres. Three other drill holes failed to intercept the mineralized zone (Figure 10-1).

Additional drilling is warranted in defining the continuation of the mineralization to the northwest and location of the cross faults that are displacing the mineralization in the GAP zone. Any expansion of the 2000S zone has the potential to define a mineral resource amenable to open pit mining and could coalesce with the main proposed pit area in Zone 1, 4 and 7.

Table 10-2 Carmacks 2014 Drill Hole Results

Drill Hole	Interval (m)	Drill width (m)	True Width (m)	Total Cu (%)	Au (g/t)	Ag (g/t)
Zone 2 Extension						
CN14-01	9.60 - 19.78	10.18	7.74	0.54	0.061	5.97
CN14-02	14.10 - 21.60	7.50	3.4	0.48	0.152	22.34
CN14-03	21.06 – 28.95	7.89	5.92	0.45	0.037	2.9
CN14-04	28.90 – 42.20	13.30	8.42	0.51	0.055	5.89
CN14-05	22.00 – 34.00	12.00	6.04	0.24	0.033	1.02
CN14-06	49.70 – 60.70	11.00	4.99	0.45	0.079	2.55
CN14-07	24.20 – 33.30	9.10	6.92	0.13	0.04	0.071
CN14-08	49.80 – 57.80	8.00	3.63	0.21	0.043	2.35
CN14-09	22.80 -24.80	2.00	1.52	0.29	0.044	1.45
CN14-10	57.00 - 81.00	24.00	10.9	0.38	0.103	2.54
GAP Zone						
CN14-11	21.26 – 40.00	18.74	13.25	0.58	0.189	2.46

Figure 10-1 Isometric View looking Northeast: 2014 Drilling in the Zone 1 and 2000S Zone Oxide (brown) and Sulphide Deposit Areas



The 2015 fill-in drilling program has confirmed continuity of both oxide and sulphide mineralization in Zones 2000S, 12, and 13, covering a strike length of 2,000 metres (Figure 10-2). Zone 2000S extends along strike for approximately 300 metres. It is open to the south and has a variable width: the zone widens to the south and widens with depth. There is a fault that follows the trend of Williams creek that offsets mineralization south of Zone 2000S.The average vertical thickness of the oxide zone is approximately 95 metres and the sulphide zone is open to depth.

The highlights of 2015 drilling include an estimated true width of 18.32 metres of oxide mineralization grading 0.72% total copper, 0.47% soluble copper in hole CN15-02, and an estimated true width of 24.01 metres of sulphide mineralization grading 1.01% total copper from hole CN15-07.

The 2015 drilling in Zone 13 focused on a 300 metre long section where the widest and thickest amount of near-surface oxide copper mineralization is evident (Figure 10-2). Thirteen of fourteen holes drilled in 2015 intersected oxide copper, native copper or copper sulphide mineralization. The fourteenth hole was lost due to poor ground conditions. All mineralized intercepts from the 13 holes are within 120 metres of surface. Highlights include:

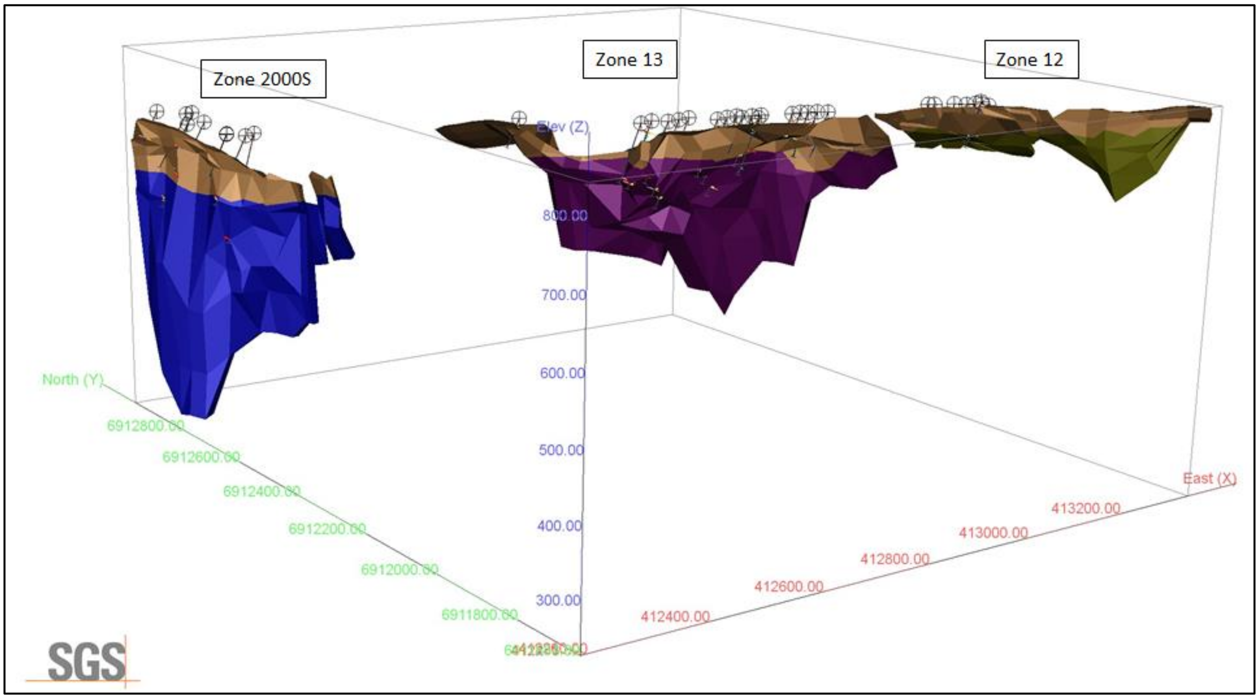
- CN15-13: 31.3 metres of 0.80% Total Cu, 0.18 g/t Au, 2.57 g/t Ag
- CN15-19: 102.1 metres of 0.37% Total Cu, 0.09 g/t Au, and 1.21 g/t Ag
- CN15-20: 82.6 metres of 0.52% Total Cu, 0.14 g/t Au, and 1.85 g/t Ag
- CN15 -21 119.6 metres of 0.35 % Total Cu, 0.08 g/t Au, and 0.98 g/t Ag

Intercepted widths of over 100 metres demonstrate that Zone 13 is a wide zone of near-surface copper mineralization that has the potential to be mined with a low strip-ratio. Zone 13 is open to the north, to the south and at depth. Zone 13 mineralization was not included in the Measured and Indicated resource used in the 2014 PEA. However, the drill density is sufficient to define a mineral resource and is included in the current MREs.

The Company undertook drilling in September and October to gather more geotechnical information and exploration in the mineral area that was drilled in 2015. The drilling in the location of the planned deposition of dry stacked tailings was completed as part of preparing for improvement of the environmental report required for submission for new environmental approval and amended permits.

The results of the drilling in zones 2000S, 13 and 12 confirmed the continuity of these zones and their copper grades.

Figure 10-2 Isometric View looking Northeast: 2015 Drilling in the 2000S Zone and Zones 12 and 13 Oxide (brown) and Sulphide Deposit Areas



10.2 2017 Diamond Drilling by Copper North

Copper North undertook drilling in September and October of 2017 to gather more geotechnical information and exploration in the mineral area that was drilled in 2015. The drilling in the location of the planned deposition of dry stacked tailings was completed as part of preparing for improvement of the environmental report required for submission for new environmental approval and amended permits.

The results of the 2017 drilling in zones 2000S, 13 and 12 zones confirmed the continuity of these zones and their grades. The drill holes information was released in news releases on January 8 and 18th 2018 (see Copper North SEDAR profile). The drill results in the south area zones confirmed the continuity of the mineralized zones and extended the mineralized zones to further increase the size of the mineral areas. A total of 36 holes were completed for 4,175 m (Figure 10-3) (Table 10-3).

Seven holes were drilled within the zone 2000S as infill and small step-outs to better define the mineralized zone and the boundary between the sulphide and oxide domains. Copper mineralization was intersected in all holes within the zone. The completion of seven drill holes has successfully confirmed the presence of visible malachite, azurite, and tenorite in the zone up to approximately 100 metres vertically from surface. The additional drilling has provided better constraints on the location of the oxide to sulphide interface, confirming the presence of copper oxide mineralization at depth.

Highlights of the assays included in the extension area:

- CN17-21: 34.9 m true width of 0.65% Cu, 0.14 g/t Au and 2.47 g/t Ag
- CN17-15: 21.5 m true width of 0.65% Cu, 0.14 g/t Au and 2.93 g/t Ag
- CN17-24: 49.5 m true width of 0.44% Cu, 0.13 g/t Au and 2.14 g/t Ag

Thirteen holes were drilled within Zone 13 and two holes within Zone 12 as infill and small step-outs to better define the mineralized zone and the boundary between the sulphide and oxide domains. Copper mineralization was intersected in all fifteen holes (Table 10-3).

Highlights of the assays in Zone 13 include:

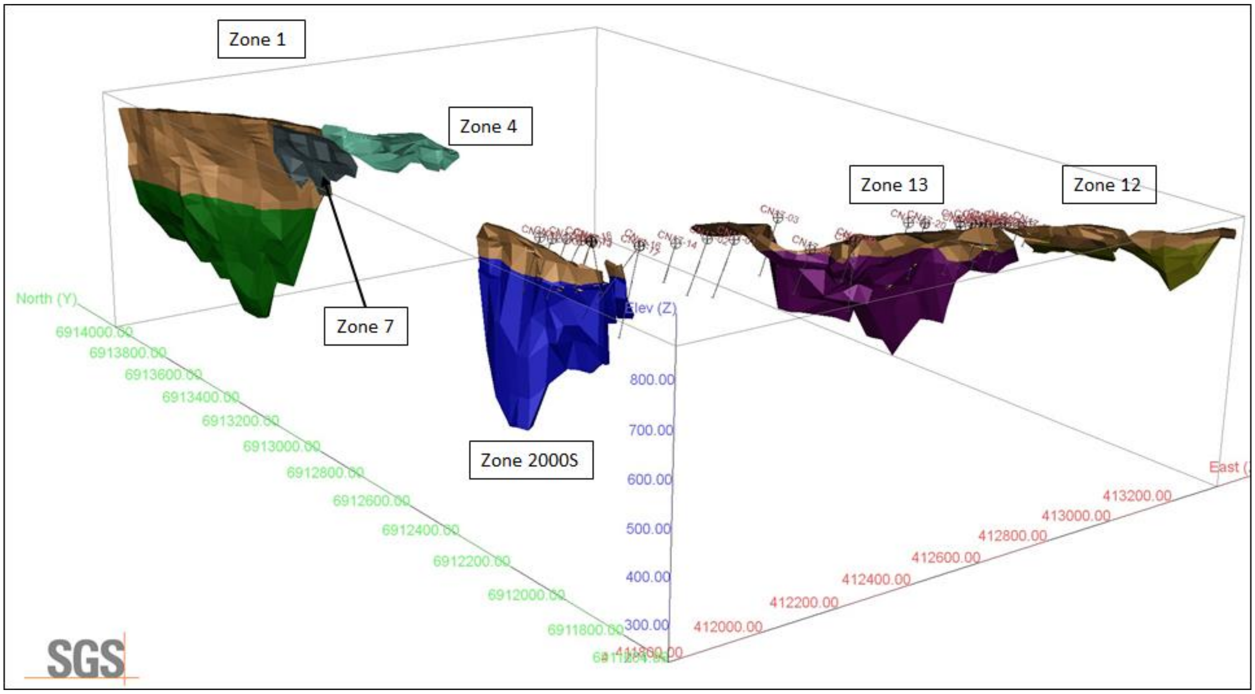
- CN17-19: 50.2 m true width of 0.68% Cu, 0.13 g/t Au and 1.95 g/t Ag (sulphide)
- CN17-20: 55.4 m true width of 0.61% Cu, 0.13 g/t Au and 1.84 g/t Ag (oxide transition to sulphide)
- CN17-32: 32.3 m true width of 0.68% Cu, 0.18 g/t Au and 2.46 g/t Ag (oxide)

Zone 13 is located 1100 metres south of the proposed open pit encompassing zones 1, 4 and 7. The mineralized zone has now been demonstrated by drilling to extend over a strike length of approximately 380 m. The 2017 step-out drilling has increased the known strike-length of mineralization by approximately 70 metres beyond the limit of the 2015 drilling. In-fill drilling has confirmed that the zone can reach widths of over 100 metres and that grade continuity is excellent between drill sections. The oxide cap of zone 13 shows a variable level of preservation but can extend vertically up to 85 metres. The 50.19 metres of sulphide mineralization in drill hole CN17-19, grading 0.68% copper, 0.13 g/t gold, and 1.95 g/t silver, indicates the potential for sulphides to depth.

Zone 12 is located approximately 120 metres south of Zone 13. Two step-out holes were drilled in Zone 12 and a thin body of copper oxide mineralization was discovered. The gap between zones 12 and 13 has now been closed-off. Both Zone 12 and Zone 13 have sulphide zones that are open to depth. From zones 2000S, 13 and 12, there are 22 new drill holes that were integrated in to a new geological models for each zone and included in the updated 2018 resource estimate.

Holes CN-01 to CN-02, CN17-04 to CN17-11, CN17-16, and CN17-17 were drilled away from known mineral zones considered as wildcat holes. No significant mineralization was intersected in these drill holes.

Figure 10-3 Isometric View looking Northeast: 2017 Drilling in the 2000S Zone and Zones 12 and 13 Oxide (brown) and Sulphide Deposit Areas



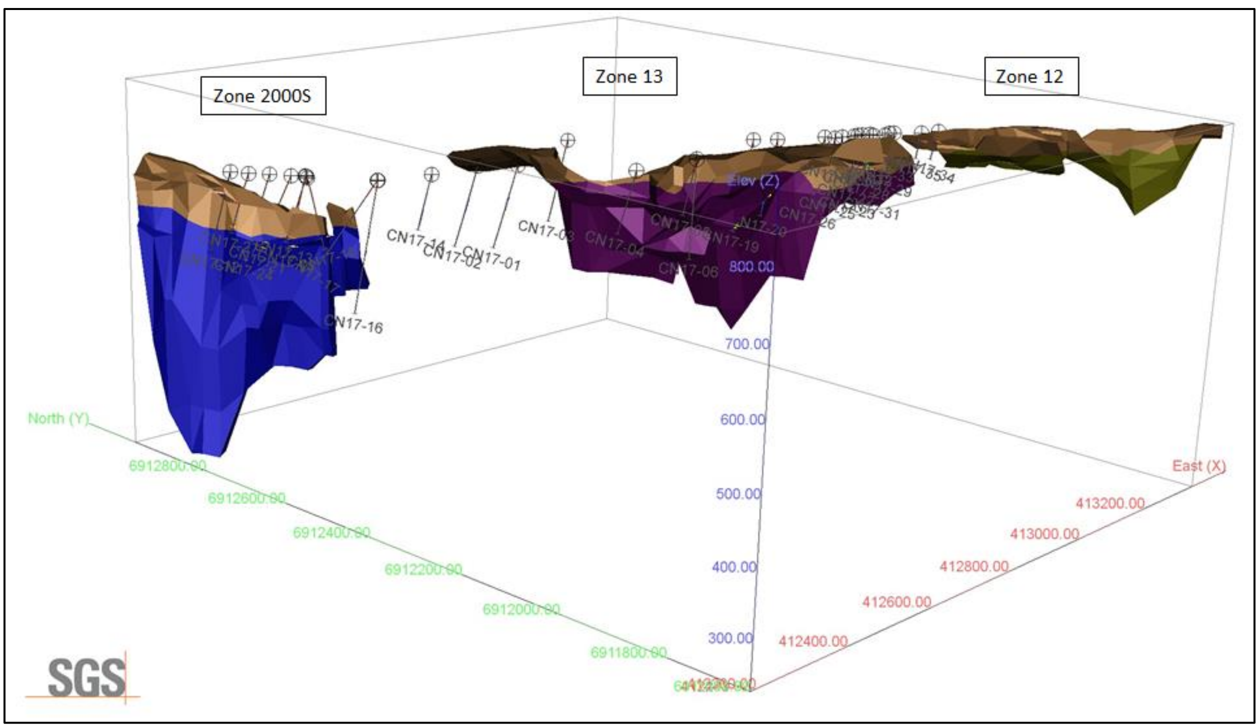


Table 10-3 Highlights from of 2017 Diamond Drill Program

Hole	From (m)	To (m)	Intersected Width (m)	Estimated True Width (m)	Au (g/t)	Ag g/t	Acid-soluble Cu (%)	Total Cu (%)	Style
Zone 2000S									
CN17-12	18.43	24.77	6.34	5.1	0.09	1.36	0.23	0.25	Oxide
CN17-12	37.93	40.6	2.67	2.25	0.76	12.62	1.82	1.96	Oxide
CN17-13	79.38	111	31.66	23.95	0.1	1.83	0.25	0.34	Oxide
CN17-15	97.31	121.9	24.61	21.53	0.14	2.93	0.17	0.65	Oxide to Sulphide
including	97.31	109.6	12.28	10.74	0.11	2.15	0.26	0.54	Oxide
and	110.68	121.9	11.24	9.83	0.19	4.05	0.08	0.83	Sulphide
CN17-18	100.85	107.9	7.05	5.09	0.23	3.98	0.04	0.78	Sulphide
CN17-21	76.74	123.4	46.7	34.91	0.14	2.47	0.34	0.65	Oxide to Sulphide
including	76.74	106.2	29.46	22.02	0.13	2.09	0.47	0.6	Oxide
and	106.2	123.4	17.24	12.89	0.16	3.12	0.12	0.73	Sulphide
CN17-24	59.26	120.4	61.14	49.5	0.13	2.14	0.31	0.44	Oxide
CN17-27	28.9	31.23	2.33	1.86	0.1	1.76	0.32	0.42	Oxide
CN17-27	38.02	41.91	3.89	3.11	0.12	1.67	0.57	0.63	Oxide
CN17-27	53.62	55.43	1.81	1.45	0.46	8.29	0.81	0.92	Oxide
CN17-27	94.49	97.54	3.05	2.44	0.07	1.55	0.22	0.36	Oxide
Zone 13									
CN17-19	42.67	67	24.33	20.83	0.01	0.42	0.15	0.21	Oxide
CN17-19	85.35	147.8	62.45	50.19	0.13	1.95	0.04	0.68	Sulphide
CN17-19	160	167.9	7.85	6.63	0.17	2.04	0.04	0.57	Sulphide
CN17-20	57.04	118.9	61.83	55.39	0.13	1.84	0.22	0.61	Oxide to Sulphide
including	80.75	105.1	24.38	22.01	0.18	2.52	0.22	0.86	Oxide to Sulphide
and	57.04	88.39	31.35	30.36	0.11	1.9	0.37	0.64	Oxide
and	88.39	118.9	30.48	27.45	0.14	1.79	0.07	0.59	Sulphide
CN17-22	41.35	62.44	21.09	20.01	0.11	1.66	0.46	0.54	Oxide
CN17-23	66.25	103.4	37.15	32.12	0.12	2.3	0.33	0.48	Oxide
CN17-25	70.29	99.53	29.24	23.92	0.18	2.67	0.42	0.56	Oxide
CN17-26	72.25	86.22	13.97	12.04	0.12	1.75	0.36	0.49	Oxide
CN17-26	92.22	100.6	8.36	7.2	0.15	1.63	0.22	0.46	Oxide
CN17-26	100.58	109.6	9.04	7.78	0.17	1.67	0.08	0.43	Sulphide
CN17-28	29.87	51.41	21.54	19.8	0.1	1.89	0.38	0.44	Oxide
CN17-28	57.3	65.19	7.89	7.25	0.17	3.51	0.57	0.62	Oxide
CN17-29	56.68	59.3	2.62	2.6	0.16	1.88	0.32	0.41	Oxide
CN17-30	11	45.72	34.72	21.78	0.09	1.52	0.43	0.5	Oxide
CN17-31	31.4	32.8	1.4	1.27	0.03	1.4	0.19	0.25	Oxide
CN17-31	61.67	68	6.33	5.76	0.1	1.71	0.29	0.43	Oxide
CN17-31	71.63	86.87	15.24	13.86	0.14	1.93	0.07	0.54	Sulphide

Hole	From (m)	To (m)	Intersected Width (m)	Estimated True Width (m)	Au (g/t)	Ag g/t	Acid-soluble Cu (%)	Total Cu (%)	Style
CN17-32	10.67	46.18	35.51	32.33	0.18	2.46	0.61	0.68	Oxide
including	24.14	30.14	6	5.46	0.36	3.63	0.93	1.03	Oxide
CN17-33	35.59	38.31	2.72	2.39	0.1	1.3	0.32	0.38	Oxide
CN17-36	28.5	34.58	6.08	6.03	0.09	1.23	0.31	0.39	Oxide
Zone 12									
CN17-34	12.19	13.21	1.02	0.96	0.13	1.1	0.41	0.51	Oxide
CN17-34	16.17	18.29	2.12	2.05	0.21	2.5	0.59	0.7	Oxide
CN17-35	10.35	13.87	3.52	3.43	0.24	3.85	0.73	0.88	Oxide

10.3 2020 Diamond Drilling – Carmacks and Carmacks North Property

Granite Creek completed 1,067 m of drilling in five holes on the combined Carmacks and Carmacks North projects in October and November of 2020. Highlights of the 2020 drill program are presented in Table 10-4.

Highlights:

- 127 metres of continuous copper mineralization, in drillhole CRM20-001, grading 0.85% copper equivalent ("CuEq") including 28.65m of 1.74% CuEq and 19.2m of 1.19% CuEq (see table below) from Zone 13 in the Carmacks deposit.
- High-grade oxide mineralization at Carmacks North Zone A with STU20-003 intercepting 10.28 m of 1.93% copper equivalent within a broader intercept of 25 m of 1.27% CuEq (see Table 1 below).
- A 19.2-metre intersection of 0.104% Mo in CRM20-001 suggests potential for molybdenum to increase contained metal value.

Table 10-4 Highlights from of 2020 Diamond Drill Program

Drillhole	From (m)	To (m)	Length* (m)	Cu (%)	Mo (%)	Au (g/t)	Ag (g/t)	CuEq** (%)
CRM20-001	102.85	230.12	127.27	0.61	0.028	0.13	2.14	0.85
including	104.85	133.5	28.65	1.03	0.014	0.2	3.09	1.28
including	157.75	176.95	19.2	0.71	0.104	0.14	2.32	1.19
CRM20-002	174	187.8	13.8	0.37	0.009	0.08	1.45	0.49
STU20-001	151.6	172.52	20.92	0.31		0.14	2.87	0.46
STU20-002	27.37	32	4.63	4.63		3.41	23.8	7.51
STU20-003	11.3	36.3	25	0.97		0.32	2.91	1.27
including	11.3	21.58	10.28	1.21		0.77	5.78	1.93

***Copper equivalent (CuEq) values assume Cu $3/lb, Au $1800/oz, Ag $18/oz, Mo $10/lb and 100% recovery. *Weighted average intercepts shown. Estimated true widths vary but, based on geological interpretation of cross-sections, are estimated to be typically 60-70% of the intersected widths.*

10.4 2021 Diamond and RC Drilling – Carmacks Property

In May of 2021, Grainite Creek launched a 2-phase (Phase 1 and Phase 3), two rig diamond drill program with the goal of increasing the confidence in and growing sulphide resources at Carmacks Project. In addition, the Company completed an RC drill program (Phase 2 drill program) to advance targets at Carmacks North.

Between May and September, 2021, Granite Creek completed 7,742 m of diamond drilling in 23 holes on the Carmacks Property (Figure 10-4 and Figure 10-5). Highlights of the 2021 Phase 1 and Phase 3 drill programs are presented in Table 10-5 and Table 10-6. Diamond drilling focused on the existing resource area with the goals of strengthening confidence in the resource model, evaluating opportunities for resource expansion and/or upgrading the sulphide portion of Zones 1 and 2000S from an inferred to indicated resource, and evaluating continuity of mineralization in Zone 13.

The objective of the Phase 2 RC program was early-stage evaluation of additional targets adjacent to known zones as well as step-out drilling at Carmacks Norths' Zone A area. The program was successful in identifying mineralization in 13 of 20 holes with several areas prioritized for follow-up diamond drilling in the 2022 field season. The phase 2 RC drilling was conducted in Zones 2, 5, 12 at Carmacks and Zone A at Carmacks North. The purpose of the program was to test zones peripheral to the deposit and find targets for follow up with the diamond drill. The program was most successful in Zone 5 where 4 of 4 holes intersected mineralization and, consequently, Zone 5 will be a priority for drilling in 2022. The program was also successful in Zone 12 where it was used to trace and test mineralization on the west side of the zone that is not included in the 2016 resource estimate. Six of seven holes in zone 12 intersected mineralization. At Zone 2, mineralization was intersected in 2 of 6 holes.

At Zone A, the drill was used to test geophysical targets, but the program was cut short due to drilling difficulties. One of the 3 holes intersected mineralization but the other two did not reach target depth due to difficult drilling conditions.

Figure 10-4 Isometric View looking Northeast: 2021 Drill Locations

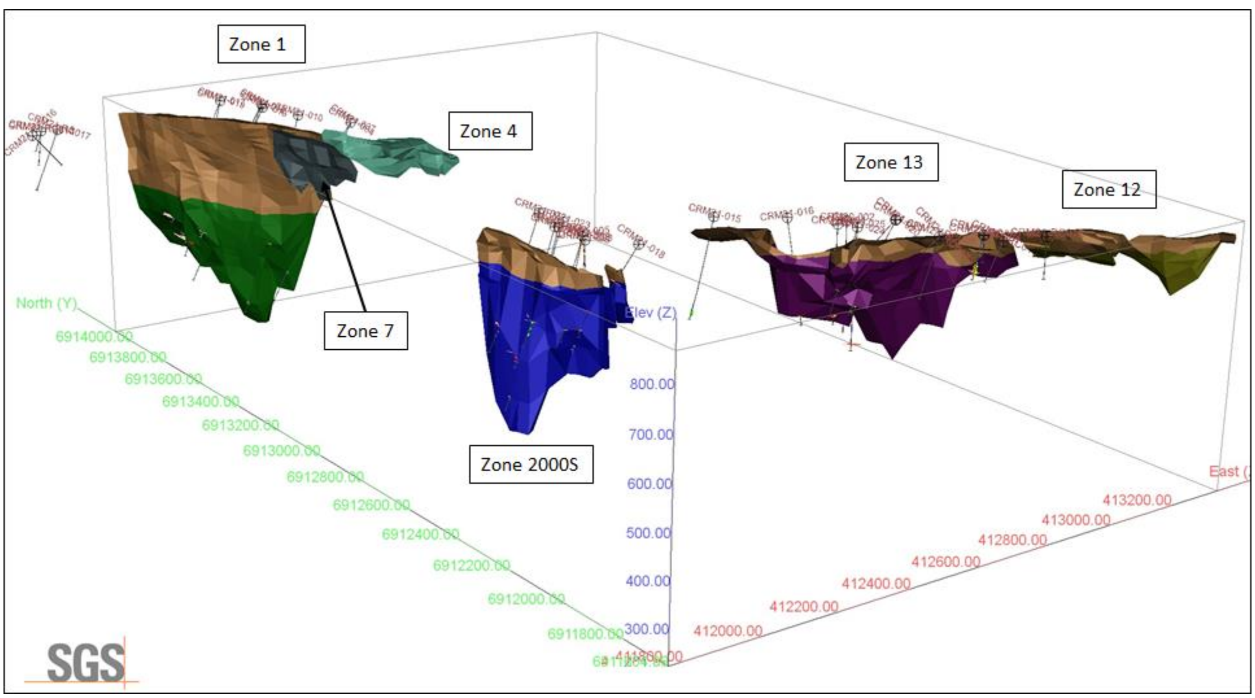


Figure 10-5 Location of 2021 Carmacks Drill Holes

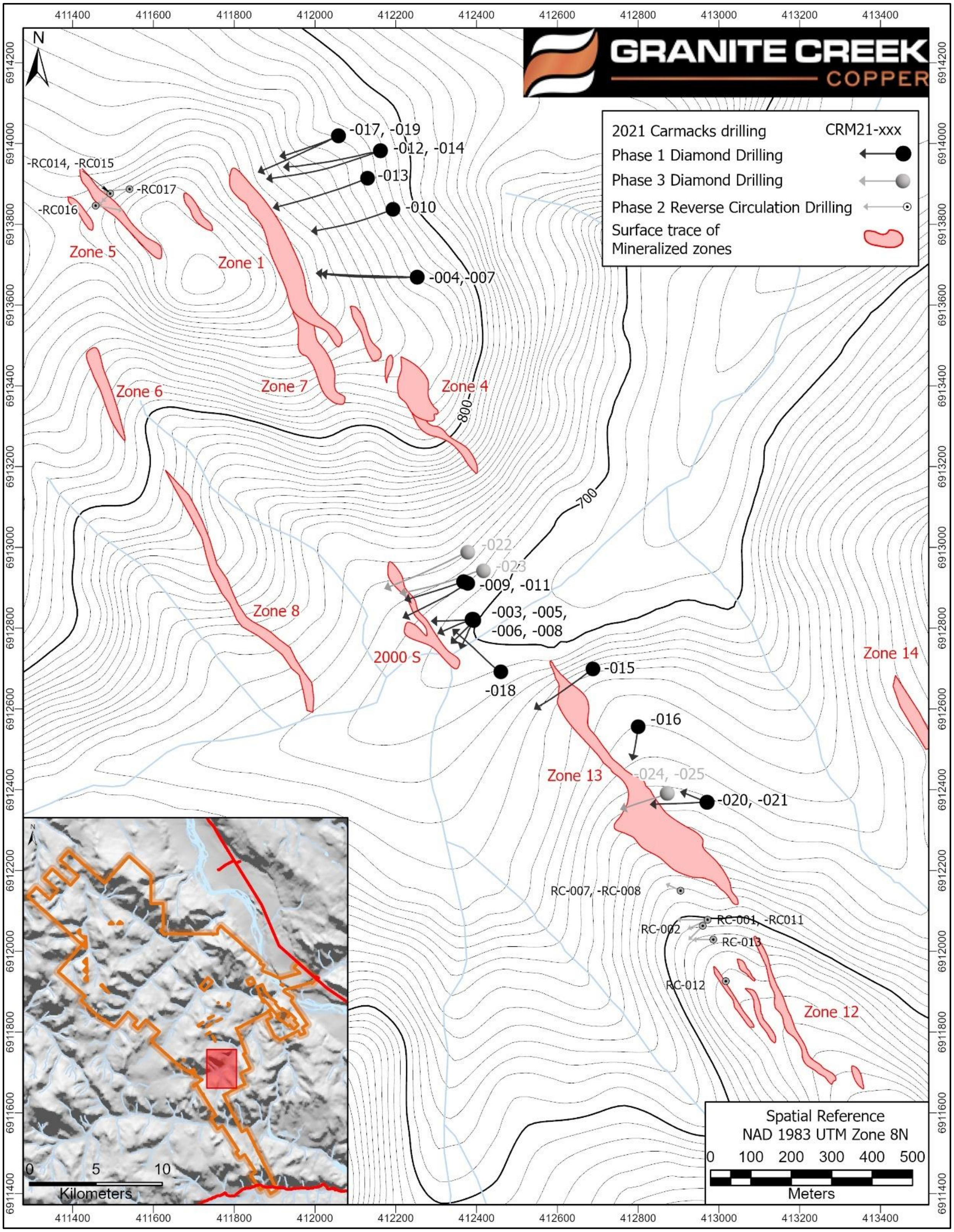


Table 10-5 Highlights from of 2021 Phase 1 Diamond Drill Program

Drillhole	From (m)	To (m)	Length* (m)	Cu (%)	Mo (%)	Au (g/t)	Ag (g/t)	CuEq** (%)	Target
CRM21-004	323.5	367	43.5	1.12	0.028	0.2	3.41	1.4	Zone 1
Including	338.5	367	28.5	1.57	0.042	0.29	4.53	1.96	
and including	352.00†	367	15	1.8	0.066	0.33	4.81	2.31	
CRM21-007	222.52	226.6	4.08	0.91	0.006	0.19	6.32	1.13	
CRM21-010	450	513.4	63.4	0.27	0.003	0.08	1.31	0.35	
Including	450	482.25	32.25	0.3	0.004	0.08	1.41	0.39	
Including	488.9	513.4	24.5	0.3	0.003	0.09	1.47	0.39	
CRM21-012	400.65	415.75	15.1	0.34	0.006	0.11	2.13	0.47	
Including	405.85	411.2	5.35	0.55	0.016	0.15	3.01	0.75	
CRM21-013	311	378.9	67.9	0.73	0.005	0.18	2.69	0.9	
Including	324.75	343.63	22.88	0.92	0.006	0.23	3.76	1.14	
and including	355.09	368	12.91	1.39	0.006	0.37	5.29	1.73	
CRM21-014	355.7	423.45	67.75	0.93	0.009	0.26	5.16	1.2	
Including	398	423.45	24.45	1.53	0.009	0.41	6.21	1.91	
CRM21-017	317.42	363.2	45.78	0.42	0.001	0.15	2.41	0.55	
Including	323.5	335.85	12.35	0.67	0.002	0.28	3.9	0.92	
CRM21-019	277.95	345.3	67.35	0.93	0.011	0.31	4.23	1.23	
Including	322	345.3	23.3	1.7	0.016	0.57	7.51	2.27	
CRM21-003	146.35†	214.5	68.15	0.59	0.028	0.14	3.69	0.83	Zone 2000S
Including	161.4	179.8	18.04	0.81	0.033	0.21	4.8	1.13	
CRM21-005	137.05	179.8	43.24	0.74	0.047	0.16	3.82	1.06	
Including	142.05	158.4	16.35	1.2	0.036	0.26	6.11	1.58	
CRM21-006	194.4	278.2	83.8	0.64	0.012	0.13	3.23	0.81	
Including	229.2	278.2	49	0.87	0.018	0.17	3.88	1.1	
Including	248.76	266.2	17.44	1.21	0.033	0.22	5.11	1.53	
CRM21-008	195.8	228.4	32.6	0.8	0.019	0.17	3.88	1.02	
Including	201.55	215.55	14	1.1	0.023	0.24	4.86	1.4	
CRM21-009	190.5	243.85	53.35	0.59	0.012	0.14	2.71	0.75	
Including	191.3	201.7	10.4	0.87	0.004	0.25	3.7	1.09	
and including	209	225.95	16.95	0.62	0.009	0.13	2.76	0.77	
and including	229.9	235.25	5.35	1.21	0.064	0.28	4.88	1.68	
CRM21-011	223.98	329.5	105.52	0.96	0.013	0.18	4.06	1.18	
Including	223.98	245.2	21.22	2.17	0.01	0.36	9.13	2.56	
CRM21-018	92.4	110.4	18	0.91	0.008	0.17	6.79	1.12	
and	158.8	170	11.2	0.72	0.013	0.14	4.27	0.91	
and	233.6	249	15.4	0.39	0.024	0.09	2.09	0.56	
and	263	298.9	35.9	0.35	0.008	0.1	2.62	0.48	
CRM21-015	36.69	49.38	12.69	0.23	0.003	0.04	0.96	0.27	Zone 13

Drillhole	From (m)	To (m)	Length* (m)	Cu (%)	Mo (%)	Au (g/t)	Ag (g/t)	CuEq** (%)	Target
CRM21-016	91.3	238.5	147.2	0.38	0.025	0.1	2.28	0.56	
CRM21-021	132.15	229	96.85	0.62	0.014	0.2	3.04	0.84	
Including	132.15	168	35.85	0.82	0.013	0.2	3.8	1.04	
and including	207.65	229	21.35	0.8	0.021 `	0.43	3.51	1.21	

*** Copper equivalent (Cu Eq) values assume Cu $3.35/lb, Au $1600/oz, Ag $24/oz, Mo $12/lb and 100% recovery. *Weighted average intercepts shown. Estimated true widths vary but, based on geological interpretation of cross-sections, are estimated to be typically 40-60% of the intersected widths.*

Table 10-6 Highlights from of 2021 Phase 3 Diamond Drill Program

Drillhole	From (m)	To (m)	Length* (m)	Cu (%)	Mo (%)	Au (g/t)	Ag (g/t)	CuEq** (%)	Target
CRM21-022	233.7	302	68.3	0.51	0.009	0.13	2.3	0.66	2000S
CRM21-023	324.23	446	121.77	0.39	0.007	0.13	1.76	0.52	
Including	330	382.75	52.75	0.63	0.009	0.21	2.74	0.84	
CRM21-024	54.8	93	38.2	0.79	0.005	0.16	3.27	0.95	13
Including	64	77	13	1.47	0.006	0.23	5.85	1.71	
CRM21-024	106.5	158.7	52.2	0.26	0.01	0.06	1.01	0.34	
Including	134	149	15	0.36	0.021	0.08	1.28	0.51	
CRM21-025	88.65	209.3	120.65	0.76	0.016	0.14	2.53	0.94	
Including	106	155.4	49.4	1.08	0.015	0.2	3.41	1.31	
CRM21-025	283.75	287.85	4.1	1.76	0.014	0.14	7.99	1.99	

*** Copper equivalent (Cu Eq) values assume Cu $3.35/lb, Au $1600/oz, Ag $24/oz, Mo $12/lb and 100% recovery. *Weighted average intercepts shown. Estimated true widths vary but, based on geological interpretation of cross-sections, are estimated to be typically 40-60% of the intersected widths.*

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Sample preparation, analyses and security for the Carmacks Project completed prior to Granite Creeks acquisition of Copper North is described in previous technical reports on the Carmacks Project, with the exception of the 2017 drilling by Copper North. The Author assumes that the sample preparation, analyses, and security for drilling completed prior to the Granite Creek acquisition has been reviewed and validated by previous authors of resource estimates including Aseneau (2007), MERIT (2014), Arseneau (2016), JDS (2016) and has been reviewed by the Author for the current resource estimate. The Author assumes that sample preparation, analysis and security by previous operators was completed in a manner consistent with industry standard sampling techniques at the time. Arseneau (2016) was of the opinion that the sample preparation, analytical procedures and sample security followed by Copper North, Western Copper, and previous operators (prior to 2017) were adequate for inclusion in resource estimation.

It should be noted that the details of the sample preparation, analyses and security for drill core from the 2017 drill program were not available at the time of the preparation of the current MREs or report and were not reviewed by the Author. However, the 2017 drill data was used for the 2018 MRE updates for Zones 2000S, 13, and 12, prepared by Independent Qualified Person, Dr. Gilles Arseneau, P.Geo. The 2018 resource updates were not supported by a NI 43-101 technical report and the results of the 2017 drill program sample preparation, analyses and security was not documented. The Author has no reason to believe that the sample preparation, analyses and security for the 2017 drill program was not conducted to industry standards. As mentioned above, the results of the 2017 drilling in 2000S, 13 and 12 zones in the south area, confirmed the continuity and grades of mineralization.

Basically the results of prior QA/QC programs to date on the Project indicate there are no significant issues with the drill core assay data. The data verification programs undertaken on the data collected from the Project support the geological interpretations, and the analytical and database quality, and therefore data can support a mineral resource estimation.

The following sections include a summary of previous sample preparation, analyses and security, and sample preparation, analyses and security for drilling completed by Copper North in 2014-2015 and Carmacks Copper in 2020 and 2021.

11.1 Drill Core Sampling and Security

11.1.1 Historical

Drill core in 1971 was sampled in 10-foot (3.05 m) intervals. Reverse circulation holes were sampled over five foot (1.52 m) intervals within Zone 1 and at 10- foot intervals for 25 to 50 feet (7.62 m to 15.24 m) on either side of the mineralization.
Western Copper

In 1991 and 1992, drill core was sampled by rock type for geological information but sampling was largely within 10-foot intervals to facilitate later statistical analysis of assay data. Reverse circulation holes were sampled over five foot (1.52 m) intervals within Zone 1 and at 10-foot intervals for 25 to 50 feet (7.62 m to 15.24 m) on either side of the mineralization.

For the 2006 and 2007 programs, all drill core sample intervals were marked at 1.0 m intervals by a qualified geologist. All samples were cut using a diamond core saw to obtain the best possible representative sample. Samples were packaged and shipped using industry standard secure packaging and were sent to Chemex for processing.

All older core samples were split with a manual core splitter.

The 2006 and 2007 sampling and shipping procedure was handled in a secure manner. The sampling procedure was set up by Scott Casselman, P. Geo. and all shipments were supervised by a representative of Aurora Geosciences Ltd. to the point that they were delivered to the trucking company in Whitehorse for trucking to Chemex or Acme in Vancouver. There has been no indication from either of the labs that samples or shipments had been tampered with.

11.1.2 Copper North

For the 2014 drilling program, sampling was generally done at a one-metre interval with samples being interrupted at geological contact. The shortest sample collected was 0.25 m and the longest interval was 1.65 m. Sample lengths were increased to 1.5 m for the 2015 drilling. As for the 2014 drilling, sample lengths were interrupted at geological contact. A total of 1,079 samples were collected with the shortest sample being 0.25 m and the longest being 3.5 m in length. All core sampled by Copper North was cut with a diamond saw and half was shipped for assays and half was retained in core boxes stored at the Carmacks site. All samples were bagged and delivered by Copper North personnel to ALS Minerals Laboratory in Whitehorse for preparation.

For the 2017 drilling program, a combination of HTW and NTW sized core were drilled by diamond-drilling. Drill core samples were sawn in half, labelled, placed in sealed bags, and were shipped straight to the preparatory laboratory of ALS Minerals in Whitehorse. The other half of core was retained in core boxes stored at the Carmacks site. Sampling was generally done at a 1.4 to 1.6 m interval with samples being interrupted at geological contact. The shortest sample collected was 0.11 m and the longest interval was 2.15 m.

True widths were determined by construction of geological cross sections along drill fences, interpretation of the bounding surfaces of the mineralized zones, and the width of the mineralized zone was measured perpendicular to the zone boundaries at the centre point of the intersection.

11.1.3 Granite Creek

During the 2020 and 2021 drilling program diamond drill core samples were taken of mineralized intervals at 2.00 m with a minimum 2.00 m shoulder sample being taken to ensure entire widths of mineralization was sampled. The shortest sample width was 0.30 m and the longest was 3.84m. All core samples were cut along the drill core axis with a diamond saw and half of the sample was shipped for assays, with the remaining half retained in the core boxes, stored at site. The sampling procedure was set up by Debbie James, P. Geo. and all shipments were supervised by a representative of TruePoint Exploration. There has been no indication that samples or shipments had been tampered with. All samples were bagged in rice bags and delivered to BV Laboratory in Whitehorse for sample preparation.

Reverse circulation holes were sampled over five foot (1.52 m) intervals. The RC chips were split twice by riffle splitter and samples were bagged, with duplicate spilts remaining at site.

11.2 Analytical Procedures

11.2.1 Historical

In 1971, rock assays were performed by Whitehorse Assay Office in Whitehorse. Two batches of sample rejects were sent to ALS Chemex Labs Ltd. (Chemex) in North Vancouver, BC for check assays. The first batch results from Chemex were 5.9% higher than the originals but the second batch returned values 5.7%

lower on average. In the 1990s programs, trench and drilling samples were sent to Chemex for analysis. All samples were dried and crushed to better than 60% minus 10 mesh. An appropriate size split then underwent chrome-steel ring pulverization until >90% was minus 150 mesh size.

Total copper was assayed by HClO4 – HNO3 digestion followed by Atomic Absorption Spectrometry (AAS) with a 0.01% detection limit. Non-sulphide copper was assayed by dilute H2SO4 digestion followed by AAS with a 0.01% detection limit. Gold was assayed by 1/2 assay ton fire assay followed by AAS with a 0.002 oz/t (0.0686 g/t) detection limit and an upper limit of 20 oz/t (685.71 g/t). Silver was assayed by aqua regia digestion followed by AAS with a 0.01 oz/t (0.34 g/t) detection limit and an upper limit of 20 oz/t (685.71 g/t).

11.2.2 Western Copper

All 1990 to 1992 drill samples were assayed for total copper, non-sulphide copper, gold, and silver. Most trench samples were assayed for the same elements but a few peripheral trench samples were not assayed for non-sulphide copper, gold, or silver. In 1971, any drill sample without obvious copper oxides or carbonates was not assayed for non-sulphide copper and deeper intercepts were generally not assayed for gold or silver.

For the 2006 program, all drill core sample intervals were marked at 1.0 m intervals by a qualified geologist. All samples were cut using a diamond core saw to obtain the best quality split core sample. Samples were packaged and shipped using industry standard secure packaging and were sent to ALS Chemex Laboratories in North Vancouver for processing. Samples were processed by crushing to >70% <2 mm and pulverizing a 250-g split to >85% -75 mm according to Chemex's Prep 31 procedure. The samples were then analyzed for 27 elements by "Near Total" digestion and Inductively Couple Plasma Emission Spectroscopy (ICP-ES) by Chemex's ME-ICP61 or ME-ICP61a procedures. As well, each sample was analyzed for gold by fire assay and AAS on a 30-g sample by procedure Au-AA23, total copper content by four-acid (HF-HNO3-HClO4-HCl) digestion and atomic absorption according to procedure Cu-AA62 non-sulphide copper by sulphuric acid leach and AAS according to procedure Cu-AA05.

Duplicate samples were collected regularly, nominally every 20th sample, and were given unique sample numbers. For the first portion of the program, the duplicates were sent along with the original samples to Chemex for processing and were processed as described below. For the latter portion of the 2006 program the duplicates were sent to Acme Analytical Laboratories (Acme) in Vancouver for analysis.

The samples sent to Acme were processed by crushing to >70% <-10 mesh and pulverizing a 250 g split to >95% -150 mesh according to the Acme R1 50 procedure. The samples were then analyzed for 43 elements by "Four-Acid" digestion and Inductively Couple Plasma Mass Spectroscopy (ICP-MS) by Acme's 1T-MS procedure. As well, all samples were analyzed for gold by fire assay and Inductively Coupled Plasma Emission Spectroscopy (ICP-ES) on a 30 gm sample by procedure 3B ICP-ES. Total copper content was determined by four-acid (HF-HNO3-HClO4-HCl) digestion and ICP-ES according to procedure 7TD and, for non-sulphide copper, by sulphuric acid leach and AAS according to procedure 8.

11.2.3 Copper North

Core samples collected by Copper North were shipped to ALS Minerals Laboratory in Whitehorse for preparation. In Whitehorse, the samples were dried and then crushed to 70% passing a 2 mm screen. The samples were then split with a riffle splitter and a 250 g portion was pulverized using a ring and puck pulverizer so that 85% or more was less than 75 microns.
The samples were then shipped to ALS Minerals in North Vancouver for analysis. In Vancouver, the samples were analyzed by inductively coupled plasma atomic emission spectrometry (ICP-AES) for a suite of 33 trace elements. For ICP-AES method, the sample is digested in a mixture of nitric, perchloric and

hydrofluoric acids. Perchloric acid is added to assist oxidation of the sample and to reduce the possibility of mechanical loss of sample as the solution is evaporated to moist salts. Elements are determined by ICP.

For samples that returned values in excess of the limits of the ICP-AES, these were treated with a four-acid digestion followed by ICP-AES analysis. For this method, the sample is digested with nitric, perchloric, hydrofluoric, and hydrochloric acids, and then evaporated to incipient dryness. Hydrochloric acid and de-ionized water is added for further digestion, and the sample is heated for an additional allotted time. The sample is cooled to room temperature and transferred to a volumetric flask (100 mL). The resulting solution is diluted to volume with de-ionized water, homogenized and the solution is analyzed by inductively coupled plasma - atomic emission spectroscopy or by AAS.

Copper oxide values were determined using method Cu-AA05. The procedure uses sulphuric acid to leach the acid soluble copper oxide minerals. The cyanide leach dissolves the oxides (with the exception of chrysocolla, which is only partially digested), secondary sulphides like chalcocite and covellite, and bornite. The chalcopyrite content remains largely undissolved by either sulphuric acid or cyanide leach. Dissolved copper is then analyzed by AAS methods.

Gold was determined by AAS method with fire assay finish. Procedures include fusing a 30-gram sub-sample with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents as required, inquarted with 6 mg of gold-free silver and then cupelled to yield a precious metal bead. The bead is digested in 0.5 ml dilute nitric acid in the microwave oven, 0.5 ml concentrated hydrochloric acid is then added and the bead is further digested in the microwave at a lower power setting. The digested solution is cooled, diluted to a total volume of 4 ml with de-mineralized water, and analyzed by atomic absorption spectroscopy against matrix-matched standards.

ALS has developed and implemented at each of its locations a Quality Management System (QMS) designed to ensure the production of consistently reliable data. The system covers all laboratory activities and takes into consideration the requirements of ISO standards.

The QMS operates under global and regional Quality Control (QC) teams responsible for the execution and monitoring of the Quality Assurance (QA) and Quality Control programs in each department, on a regular basis. Audited both internally and by outside parties, these programs include, but are not limited to, proficiency testing of a variety of parameters, ensuring that all key methods have standard operating procedures (SOPs) that are in place and being followed properly, and ensuring that quality control standards are producing consistent results.

For the 2017 drill program, similar to the 2014-2015 programs, all geochemical analyses were performed by ALS Minerals in North Vancouver. Total copper assays were performed by four-acid digestion with an AAS finish. Soluble copper assays were carried out by sulphuric acid digestion with an AAS finish. Gold was analysed by a 30 g charge fire assay with an AAS finish. Silver was analyzed by four-acid digestion and ICP-AES finish.

11.2.4 Granite Creek

Core samples collected by Granite Creek were shipped to Bureau Veritas Laboratory in Whitehorse for sample preparation. In Whitehorse, received samples are entered into the Laboratory Information Management System (LIMS), weighed, dried, and crushed to ensure that greater than 70% pass a 2mm sieve. A 250g split of the crushed material is then pulverized to greater than 85% passing a 75µm sieve.

At random intervals and at the start of each shift QC testing is completed on both crushed and pulverized material to ensure that the above specifications are met.

The sample pulps are then shipped to Bureau Veritas Minerals in Vancouver for analysis. In Vancouver, the samples were analyzed by inductively coupled plasma emission spectrometry (ICP-ES) for a suite of 35 elements (MA300). A 0.25g split of the sample is heated in a mixture of nitric, perchloric and hydrofluoric acids to fuming and taken to dryness. The residue is dissolved in hydrochloric acid. Elements are determined by ICP-ES. Most minerals are digested using this multi-acid digestion method. Copper oxide values were determined using method LH402. The procedure uses a 5% concentration of sulphuric acid to digest a 1gram sample split, the acid soluble copper oxide minerals are digested, and the dissolved copper is then analyzed by AAS methods.

For high-grade copper samples in which copper exceeding the upper detection limit (10 000ppm) of MA300 procedure, the sample was analysed using MA401 method. This is an aqua regia and multi-acid digestion with AAS analysis are optimized for moderate to high grade ore samples with a detection range of 0.001%-10% Cu. A single sample (3821431) exceed this 10% Cu threshold and was analysed using the MA404 method, in which the ICP-AAS detection limit ranges 0.01%-30% Cu.

Gold was analyzed by igniting a 15 g sample followed by an aqua regia digestion with an ICP-MS finish (AQ115-IGN).

Bureau Veritas has a proactive Quality Management System (QMS) designed to ensure the production of consistently reliable data. The system covers all laboratory activities and takes into consideration the requirements of ISO standards. It is an ISO 9001 certified laboratory.

11.3 Quality Control Protocols

11.3.1 Historical

For the 1970 and 1990 drilling duplicate 12.5% splits were collected with one sample for assay and one sample kept at the core storage area. No other information is available on the quality control procedures followed during the 1970 and 1990 drilling programs.

11.3.2 Western Copper

Duplicate samples were collected regularly, nominally every 20th sample, and were given unique sample numbers. For the first portion of the program, the duplicates were sent along with the original samples to Chemex for processing and were processed as described below. For the latter portion of the 2006 program the duplicates were sent to Acme Analytical Laboratories (Acme) in Vancouver for analysis.

For the 2007 program a set of three standards reference material (SRM) were included with each sample shipment to Chemex and Acme. These standards were collected from the property and represented “high grade” oxide mineralized material (AGL-1), “low-grade” oxide mineralized material (AGL-2) and a blank which was comprised of un-mineralized granodiorite (AGL-3). The standard samples were collected in the 2006 exploration season and prepared by CDN Resource Laboratories in Vancouver with assay certification by Smee and Associates of Vancouver, BC. The processed standards were received in February 2007, hence were not available in time to be included in the 2006 sample shipments.

No special sample handling practices were used for the pre-2006 work. No special security precautions were noted in the sampling, shipping, and analysis of the mineralization from the deposit. No irregularities were found in the historical data, and some check assays were performed. ALS Chemex and Acme Labs are independent of Western Copper. Both labs were ISO 9001 accredited at the time the assays were carried out.

The twin holes, WC-003 and WC-004, were drilled to test historical holes 140 and 141 respectively, drilled in 1991. The hanging wall and footwall contacts were well defined in all four drill holes. The lengths of the mineralized intercepts from the hanging wall contact to the footwall compared well. There were well mineralized intersections below the footwall contact in all four holes, but these were not used in the mineralization comparison.

The historical grade and geological interpretations are repeatable using modern drilling, core handling and sampling methods, and assay procedures. The differences in section widths are a function of the fact that the historical drill results were sampled on a 10-foot (3.05 m) interval while the 2006 drilling was sampled on a three-metre interval. The small discrepancy between total copper values in hole 141 and WC-004 are caused by a short intersection of anomalously high grade copper (6.5% Cu) over a length of 9 feet (2.74 m) in hole 141 that was not present in hole WC-004.

A number of check samples were also collected from selected portions of 1991 drill core stored on the property. The samples were collected by quartering remaining split core with a rock saw. The samples were collected at one-metre intervals falling within 1991 sample intervals for comparison purposes. The sample handling, shipping, and preparation control procedures followed were the same as those employed for the 2006 diamond drill program.

It was not possible to sample exactly the same intervals of drill core as were sampled in 1991; nonetheless, the results are consistent with the previous sampling. On average, the new assay values are close to, and in most cases, are higher than, the historic values. In fact, the average values of the re-assays are substantially higher than the historic assay results.

11.3.3 Copper North

Copper North collected a total of 1,349 samples as part of the 2014-2015 drilling programs. They inserted standard reference material, blanks and assayed field duplicates as part of their quality control program. The protocol was to insert either a standard, blank or duplicate sample with every 20 samples submitted. The procedure resulted in standards (AGL-1 or AGL-2) being inserted at a rate of approximately one in 30, blanks and duplicates were inserted at about one for every 60 samples submitted.

Duplicate samples were collected from quartered core and shipped to Acme in Vancouver for assay. Acme laboratory is now part of Bureau Veritas Mineral Laboratories (Bureau Veritas). Bureau Veritas is a world recognized laboratory and is ISO9000:2008 certified.

The review of the duplicate sampling indicated that there is no significant bias associated with the assay data provided by ALS. It was noted that the very low grade gold values were slightly higher at Acme than at ALS, but this difference is not indicative of any significant bias. Both the soluble copper and total copper values show very comparable results for both laboratories with ALS retuning slightly higher total copper than Acme for values less than 0.4% copper.

Copper North also ran a native copper screen assay on one drill hole (CN15-09), that contained much native copper to evaluate if native copper was not passing the pulverizing process and not making it through to the digestion stage, thereby underrepresenting the sample total copper grade. A total of 62 samples were assayed for total copper by screen assay and compared with the 4-acid digestion total copper for the same samples. The results were very similar indicating that copper was being properly represented by the 4-acid digestion method.

For the 2017 drill program, quality assurance and quality control procedures include the systematic insertion of duplicate and standard samples into the sample stream. No other information is available on the quality control procedures or results were available as of the effective date of the current report.

11.3.4 Granite Creek

For the 2020-2021 drill programs, quality assurance and quality control procedures include the systematic insertion of duplicate, blank and standard samples, making up 11.5% of the sample stream.

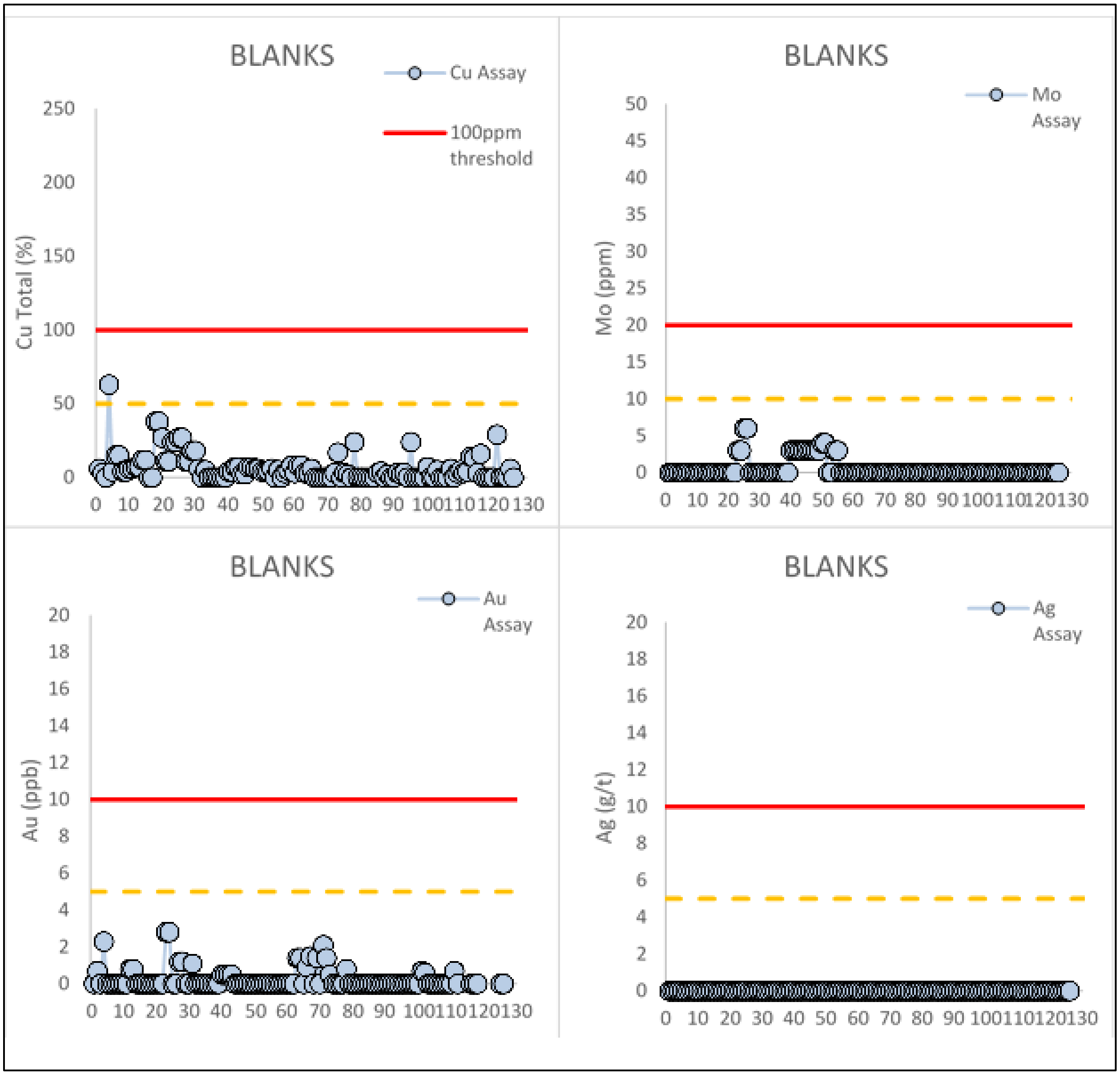
Blanks Samples

Limestone blanks were inserted approximately every 25th sample and after each mineralized interval. The threshold for these blanks was determined to be 100 ppm Cu, 10 ppb Au, 1.0 g/t Ag and 20 ppm Mo.

Of the 129 blanks inserted into the sample stream there were two failures. Sample 3821068 of certificate WHI21000067 failed on Cu, Au, Mo as well as multiple other elements having anomalous values. The blank along with the two preceeding and three succeeding samples were re-assayed. The results passed, with the blank reporting below threshold for all three elements and appearing to have more typical blank signature. See certificate WHI21000067P.

Sample number 3836477 also appeared to have been a mislabelled standard, reflecting the composition of AGL-2 rather than a blank. The sample was re-classified in the final database.
These two blank failures were resolved according to best practices and the final database reflects the accurate assay certificate data (Figure 11-1).

Figure 11-1 Results of Blank Assays for the 2020-2021 Drill Programs

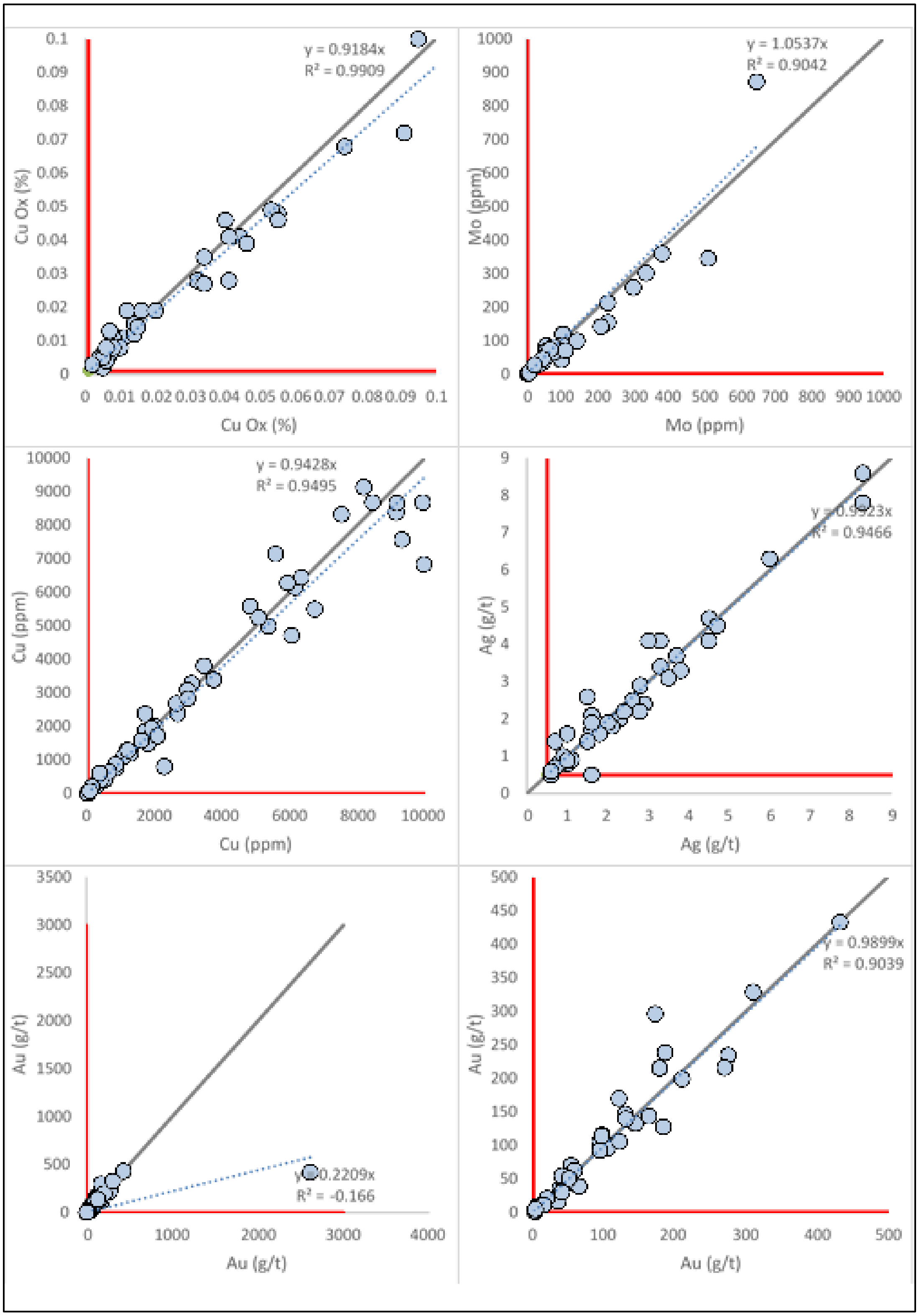


Duplicate Samples

Duplicates were done approximately every 40 samples, typically within a mineralized interval. The duplicates were done to ensure proper sampling procedures of the drill-core and thus to illustrate repeatable results.

Duplicates showed good correlation between the original sample and the duplicate assay for all elements (Figure 11-2). Note that Au had one failure (sample 3821626 and duplicate 3821619) which when re-assayed returned similar results and could not be reconciled (ccertificate WHI21000168). The sample was from a high-grade intercept with 1.90% Cu.

Figure 11-2 Results of Duplicate Samples for the 2020-2021 Drill Programs (red line is the detection limit and solid grey line is 1:1 ratio)



Standard Reference Material Samples

The standards are used to determine the accuracy of the laboratory results for elements of interest. The standards are inserted within the sample sequence to ensure that results from the laboratory are accurate. If a standard fails, the batch that is covered by the standard reference material is re-run.

The Carmacks deposit has typically used AGL-1 and AGL-2 standards as representative high-grade and low copper values, respectively. These standards were produced from Carmacks material. The standard samples were collected in the 2006 exploration season and prepared by CDN Resource Laboratories in Vancouver with assay certification by Smee and Associates of Vancouver, BC. The processed standards were received in February of 2007. These standards were used at the start of the season and continued. As standard reference material was in short supply and noting that the non-sulphide material was reporting consistently low, it was decided to incorporate standards CDN-CM-41 and CDN-CM-47 starting mid way through the 2021 drilling program.

The CDN-CM-41 and CDN-CM-47 standards are prepared using ore from the Minto Mine. CDN-CM-41 was supplied to CDN Resource Laboratories as coarse reject from diamond drilling. Mineralization is primary chalcopyrite and bornite pervasively disseminated and as stringers within foliated granodiorite units rich in secondary biotite. Sulphide mineralization is typically accompanied by magnetite. Gold is intimately associated with the bornite mineralization and rarely observed as free gold. The CDN-CM-47 standard was prepared using ore from the Minto Mine and blended with 70 kg of Hecla’s Greens Creek deposit and 240kg of Molybdenum concentrate. The Greens Creek deposit is a polymetallic, stratiform, massive sulphide deposit. The host rock consists of predominantly marine sedimentary, and mafic to ultramafic volcanic and plutonic rocks, which have been subjected to multiple periods of deformation. Mineralization occurs discontinuously along the contact between a structural hanging wall of quartz mica carbonate phyllites, and a structural footwall of graphitic and calcareous argillite.

These two additional standards provide high-grade and medium-grade values and are representative standards with similar matrix to that of the Carmacks deposit.

AGL-1- the non-sulphide assays report consistently below the certified value, with the total copper and gold reporting within 2 SD of the certified value (Figure 11-3). Although, total copper is is reporting within the 2 SD range, the majority of the values are below the mean.

AGL-2 standard- showed similar results with nonsulphide copper consistently reporting below the certified value, but the majority of the copper total and gold reporting within 2 standard deviations of the certified reference material (Figure 11-4). Again, although total copper and gold are generally reporting within the 2 SD range, the majority of the values are below the mean.

The issue with standards AGL-1 and AGL-2 needs to be investigated moving forward and the use of these standards needs to be reconsidered moving forward. The issue with the Cu_T values may be in how Bureau Veritas Laboratory is able to analyze the Cu_X. the The Author strongly recommends that Granite Creek send 10-20 % of the 2020-2021 samples to a second lap for Cu_X and Cu_T analysis as a check on the Bureau Veritas Laboratory results. It should be noted that the issue with the oxide value was also an issue with values in assays from ALS and is likely an issue with the standard. In the Authors opinion there is no risk to the MREs. It should also be noted that the majority of the 2021 drilling was in the sulphide zones. Granite Creek will no longer be using the AGL-1 and AGL-2 standards moving forward.

Note tha Granite Creek has selected a number of samples to be sent to an umpire lab. However, the results were not available as of the date of the current report.

CDN-CM-41

Three failures occurred with this standard (Figure 11-5). Sample 1808271 from certificate WHI21000260 was a CDN-CM-47 standard, but was mislabelled in the database. This was rectified.

Sample 1808271 of certificate WHI21000260 was re-run along with two preceeding and six succeeding samples. The results came back within range of the certified value for silver.

Sample 1808245 of certificate WHI21000261 exceeded the certified value for gold. The sample was re-run with the preceeding six samples and succeeding 12 samples. This included sample 1808258 which was a FCDN-CM-41 standard. The results of this showed the standards were in range of the certified value.
All re-run assays were inserted in the final database.

CDN-CM-47

Sample 3821913 of certificate WHI21000230 was re-run with two preceeding samples and ten succeeding samples (Figure 11-6). The Samples 3838038 and 3838044 were mislabelled CDN-CM-41 standards

Sample 3838334 of certificate WHI21000590 reported low Ag. As the sample stream was preceeded and succeeded by low grade (i.e. <1000ppm Cu, <0.5 ppm Ag and below detection Au) the batch was not deemed of material significance to warrant re-assay.

Figure 11-3 Standard AGL-1 Assay Results versus Certified Value with respect to the Mean (red line) and 2 Standard Deviations of the Mean (orange dashed line)

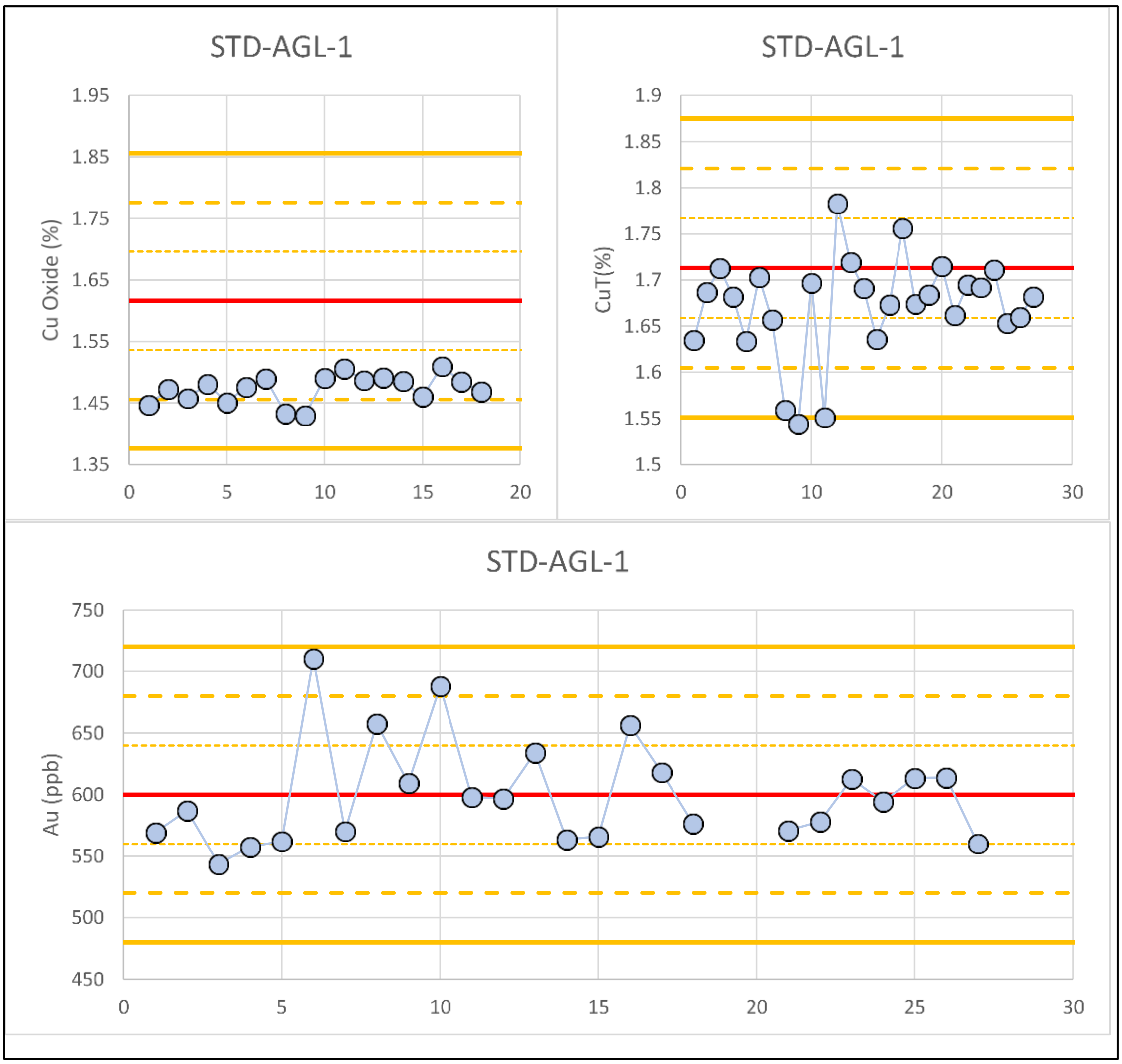


Figure 11-4 Standard AGL-2 Assay Results versus Certified Value with respect to the Mean (red line) and 2 Standard Deviations of the Mean (orange dashed line)

Figure 11-5 Standard CDN-CM-41 Assay Results versus Certified Value with respect to the Mean (red line) and 2 Standard Deviations of the Mean (orange dashed line)

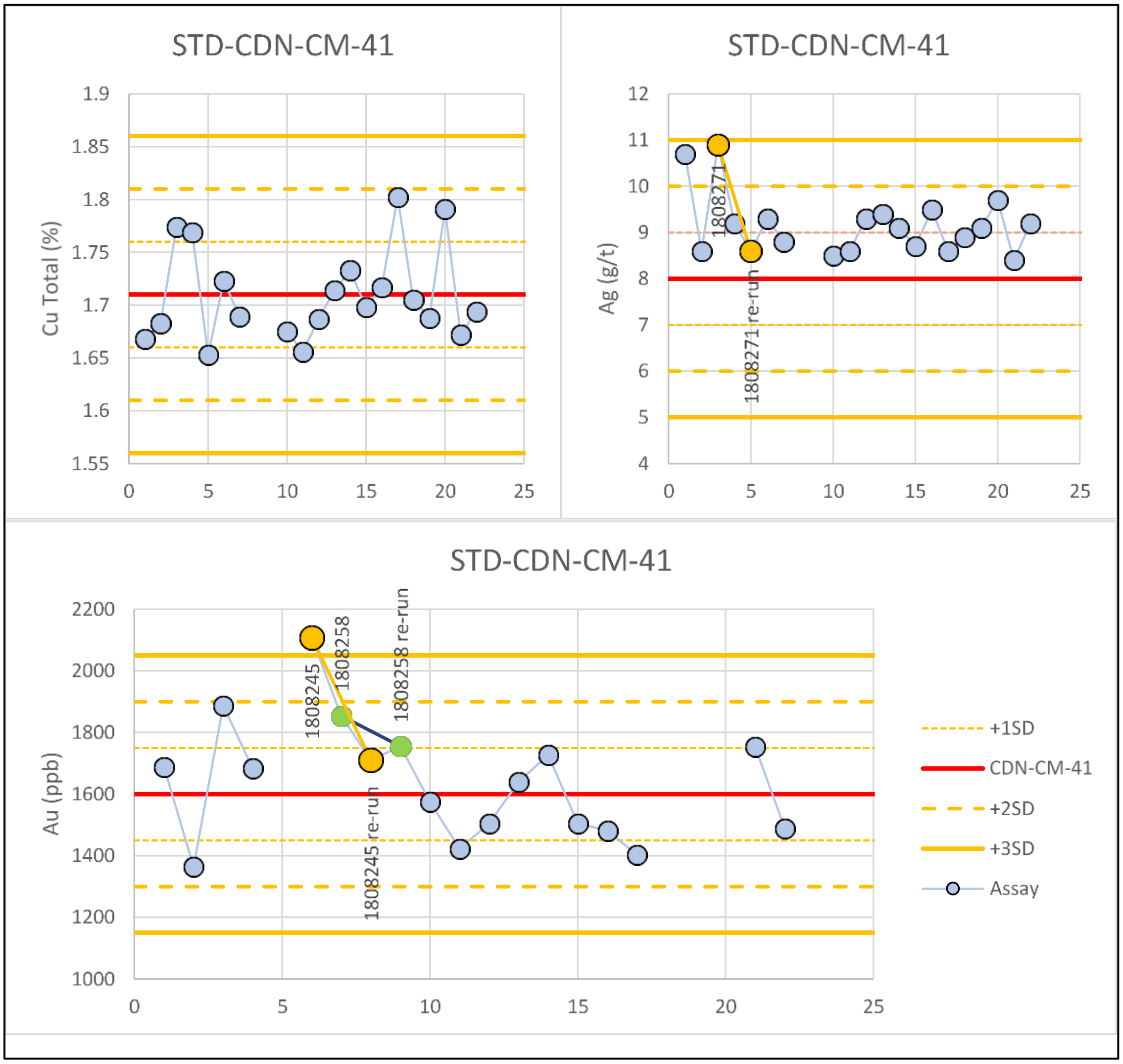
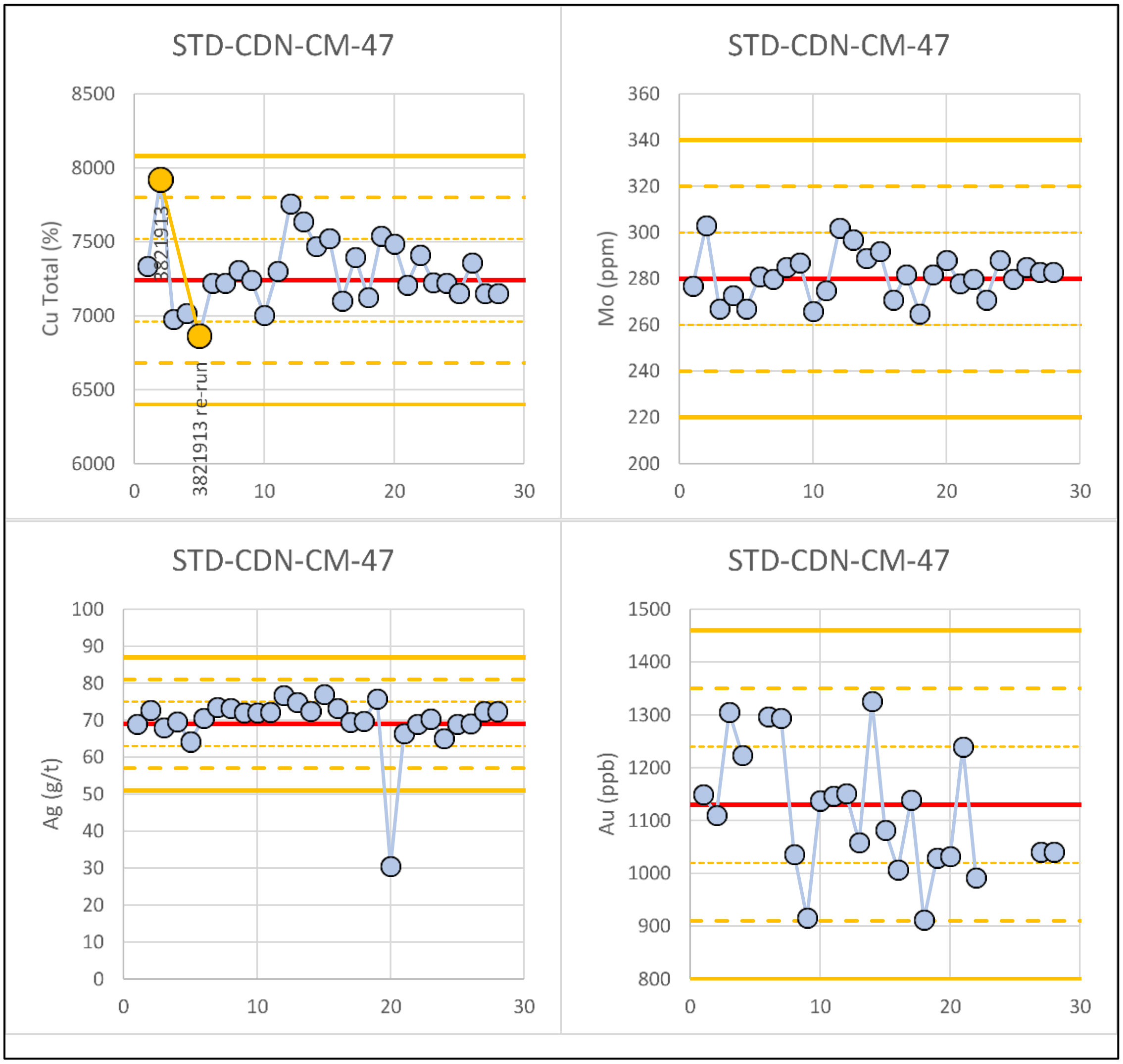


Figure 11-6 Standard CDN-CM-47 Assay Results versus Certified Value with respect to the Mean (red line) and 2 Standard Deviations of the Mean (orange dashed line)



12 DATA VERIFICATION

The following section summarise the data verification procedures that were carried out and completed and documented by the Author for this technical report, including verification of data collected during 2021 drill program by Granite Creek.

As part of the verification process, the Author reviewed all geological data and databases, past public and technical reports (as referenced within the report and listed in Section 27), and reviewed procedures and protocols as practiced by the Granite Creek field and technical team. The Granite Creek technical team provided all relevant data, explanations, and interpretations. To the Authors knowledge, there was full and open access to all the information and materials necessary to enable the Author to prepare the current Technical Report, and there were no limitations imposed upon the scope of the Authors investigation by Granite Creek.

Allan Armitage conducted verification of the laboratories analytical certificates and validation of the Project digital database supplied by Granite Creek for errors or discrepancies. A minimum of 20% of the digital assay records (including the 2021 data) were randomly selected and checked against the laboratory assay certificates. Verifications were carried out on drill hole locations (i.e. collar coordinates), down hole surveys, lithology, specific gravity, trench data, and topography information. Minor errors were noted and corrected during the validation process but have no material impact on the 2022 MREs presented for the Carmacks Project in the current report. The database is considered to be of sufficient quality to be used for the current and future MREs.

In addition, as described below, the Author has conducted a site visit to the Carmacks Project to better evaluate the veracity of the data.

12.1 Site Visits

Allan Armitage P. Geo. conducted a site visit to the Carmacks Project on November 9, 2021, accompanied by Debbie James, P. Geo, consulting geologist and qualified person for the purposes of National Instrument 43-101 for Granite Creek. Due to winter weather conditions and snow coverage during the time of the site visit, the property had to be accessed via helicopter from Whitehorse. Drilling was not underway during the site visit and the exploration camp was shut down for the season. However, the Author was able to examine drill core, and was able to visit a number of drill sites.

During the site visit, Allan examined a number of selected mineralized core intervals from recent diamond drill holes from Zones 1, 4, 7, 2000S, 12 and 13. The Author examined accompanying drill logs and assay certificates and assays were examined against the drill core mineralized zones. All core boxes were accessible, labelled (with metal tags) and properly stored outside in core racks. Sample tags were still present in the boxes, and it was possible to validate sample numbers and confirm the presence of mineralization in witness half-core samples from the mineralized zones.

Allan had the opportunity to inspect the offices, core logging and sampling facilities and core storage areas, and had discussions with Debbie James regarding the core sampling, QA/QC and core security procedures. The Author participated in a field tour, via helicopter, of the deposit areas and was able to visit a number of recent and historical drill sites (identified by casing, metal tags and flagging), and view the overall property access from the air. However, due to snow cover, the Author was not able to view outcrops or channel sample locations.

Johnny Canosa P. Eng. conducted the site visit to the Carmacks Project on June 20 -21, 2022 with Jacob Longridge (Consulting Senior Geologist-TruePoint). Upon arrival, existing facilities were inspected such as the offices and camp. The tour started with a drive to a look out location and view the exploration cut line, the proposed mining areas (zones) and the view of the engineered access road. The following were discussed:

- kilometers engineered mine access road
- waste dumps
- source of power
- water source for potable and process supply
- map of treed areas
- reclamation material stockpile
- source of gravel for construction (haul road and road surfacing)
- camp (location)

12.2 Conclusion

All geological data has been reviewed and verified by Allan Armitage as being accurate to the extent possible and to the extent possible all geologic information was reviewed and confirmed. Minor errors were noted and corrected during the validation process but have no material impact on the 2022 MRE's presented in the current report. The Author is of the opinion that the database is of sufficient quality to be used for the current and future mineral resource estimates for the Carmacks Project.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

The Carmacks project consists of a copper deposit in The Yukon, Canada that also includes minor portions of the precious metals, gold, and silver. The copper mineralization contains both sulfide minerals and oxide minerals. Previous testwork had been focused on the oxide material and the recovery of acid soluble copper through leaching. However, the economic evaluation developed for this process option indicated it was not economical. In 2022, SGS conducted a series of laboratory tests to investigate the potential to recover the contained valuable metals using flotation technology. The test work was completed in October 2022 and indicated that flotation will provide enhanced project economics. This section summarizes the flotation testwork conducted in 2022. The test results, together with industry standard engineering design practices, were used to develop the process design criteria in Section 17.

13.1 Historical Test Work

Since 1989, various ore grade samples were collected at the project site and then sent for metallurgical testwork. The initial test program was focused on the recovery of acid soluble copper mineralization in oxide material, and the emphasis of the work was to develop the process design criteria and operational parameters for a copper heap leach operation. This process configuration was to include crushing, agglomeration, and heap leaching followed by solvent extraction and electrowinning. This work was carried out by M3 Engineering and was summarized in the 2012 FS report.

In 2014, Copper North examined the value of adding precious metals recovery to the project using a two stage heap leach option. Between 2014 and 2015, Bureau Veritas Commodities Canada (BV Minerals) completed a full suite of metallurgical tests to evaluate an alternative to heap leaching, namely, a procedure that was characterized as a VAT leach. The procedure consisted of grinding the material to a P80 size of 664 µm and agitation leaching with sulfuric acid to recover the copper. The acid leach residue will be dewatered, repulped, the pH will be adjusted, and then leached by cyanide to recover gold and silver. The term VAT leach is a misnomer as the procedure described is an agitation tank leach.

In 2016, BV Minerals conducted additional copper leach optimization tests using the 2014 Composite Sample. Based on the testwork, a copper and gold leach circuit were selected as the preferred recovery method. In the envisioned flowsheet, the mineralized material will be reduced to a P80 of 664 µm using a jaw crusher followed by a SAG mill in closed circuit with hydrocyclones. Copper will be recovered using a sulfuric acid leach and solvent extraction. The copper leach residue will be neutralized, and gold and silver will be leached with cyanide in a CIL (carbon in leach) circuit and recovered by ADR circuit.

In 2021, BV Minerals conducted preliminary flotation tests on the sample to evaluate how amenable the material is to concentration and recovery by flotation. A preliminary copper flotation recovery model was generated with test results summarized below:

- Recoveries of greater than 95% for copper into a 25% copper concentrate are possible for the sulfide material.
- Copper sulfide minerals are well-liberated for rougher recovery via flotation at a P80 of 150 µm.
- A secondary regrind size at a P80 of 25 µm can achieve a 25% copper concentrate grade with a reasonable cleaner stage recovery.
- Gold is associated primarily with copper sulphide minerals and only minor amounts are associated with pyrite. Flotation of gold into the copper concentrate is likely the most economical way to recover gold.
- The chalcopyrite content as the most abundant copper mineral and the low pyrite content within the samples tested indicates utilization of a simple reagent scheme will result in relatively easy copper flotation upgrade.

The flotation testwork listed above was preliminary but it did indicate that well-established flotation methods with known reagents will likely be the preferred processing method for sulfide material at Carmacks. The next stage of metallurgical test work involved greater variability of samples to validate copper and gold recoveries as well as assessing potential flotation recovery of the oxide material.

13.2 Sample Selection and Preparation

In May 2022, Granite Creek sent to the SGS Lakefield laboratory three pails of sulfide material and five pails of oxide material for additional testing. The sample compositions are listed below for both sulfide and oxide material.

Table 13-1 Sulfide Sample Identification and Weight

Sulphide Sampl ID		Wt, g
WHI 21000599	RJ 16 (1/3)	2987.5
WHI 21000599	RJ 16 (2/3)	3729
WHI 21000599	RJ16 (3/3)	5536
WHI 21000599	RJ17 (1/3)	3775.5
WHI 21000599	RJ18 (3/3)	4777
WHI 21000599	RJ 19 (1/3)	1049.5
WHI 21000599	RJ 19 (2/3)	6179
WHI 21000599	RJ20 (1/3)	4066
WHI 21000260	RJ22 (3/3)	6132
WHI 21000260	RJ23 (2/3)	5217.5
WHI 21000260	RJ23 (3/3)	4188
WHI 21000260	RJ24 (1/3)	4637
WHI 21000260	RJ24 (2/3)	2565
Total		54839

Table 13-2 Oxyde Sample Identification and Weight

Oxide Sample ID	Wt, g
C490889	4293.5
C490890	2337.5
C490891	4073
C490946	4065.5
C490947	2566
C490948	2266.5
C490505	2105.5
C490506	3212
C490507	3583.5
C490538	2793.5
C490539	2719.5
C490540	2533.5
C490541	2962
C490542	2781
C490543	2665
C490544	2783
C490545	2849
C490546	2775
C490547	2908
C490548	2839.5
TOTAL	59112

For both the sulfide and oxide samples, the material was blended, screened, and crushed to minus 10 mesh, and then evenly split to 2 kg or 10 kg of sample charge for testing.

13.3 Mineralogy

SGS conducted mineralogy studies on the samples with both the Tescan Integrated Mineral Analyzer (TIMA) and X-ray diffraction analysis (XRD) technology. Based on the TIMA test, the major components of the material are feldspar/quartz, followed by chlorite/mica, clays, and other silicate minerals. For the sulfide mineralization, the main copper bearing minerals are chalcopyrite and bornite, comprising roughly 98% of the copper deportment. For the oxide mineralization, the main copper bearing minerals are malachite and azurite, with minor amounts of sulfide copper minerals including chalcopyrite, chalcocite, bornite, and the remaining copper is associated with iron hydroxide and silicate.

For the sulfide mineralization, the copper sulfide mineral has grain size p80 of 78um and a median size of 30 um. For the oxide mineralization, the non-sulfide copper has a grain size p80 of 46 um and a median size of 18 um. The liberation analysis indicated that the sulfide copper mineral has better liberation in the sulfide minerals compared to in the oxide minerals, with over 90% exposed in the sulfide minerals compared to around 80% exposed in the oxide minerals.

Oxide sample has a higher percentage of non sulfide copper mineral exposed compared to the sulfide sample. However, the percentage of non sulfide copper exposed in oxide sample is still much lower than the sulfide copper contained in sulfide sample. This may imply that oxide material could require a finer grind size to optimize the overall copper recovery or to achieve the maximum copper concentrate grade.

XRD tests give similar results to TIMA, with the major components of the material being albite with portions of quartz, chlorite, orthoclase and actinolite in the sample material. For the copper bearing minerals, the sulfide sample mainly contains chalcopyrite and bornite, and the oxide sample mainly contains malachite and cuprite.

13.4 Comminution

No additional comminution tests were conducted during the beneficiation campaign by SGS during 2022. The historical comminution test studies indicated the material would have an average ball mill Bond Working Index (Bwi) of 15.2 kwh/mt and an average abrasion index (AI) of 0.09 g. This material is of medium hardness with respect to ball mill grinding.

Additional comminution testing is recommended, including the SAG Milling Comminution (SMC) test and the full scale JK drop weight test, especially since the preferred comminution unit operation includes a SAG mill installation.

13.5 Flotation

SGS conducted a series of flotation tests on both the sulfide sample and the oxide sample as received from Carmacks project. According to the head assays of the sulfide material and the oxide material, the sulfide sample contains 10.8% acid soluble copper whereas the oxide contains 77.3% acid soluble copper. Besides copper, this deposit also has minor amounts of precious metals including both gold and silver.

13.5.1 Sulfide Flotation

The sulfide material responds quite well to the froth flotation process with the copper recovery in the rougher stage varying between 96% to 98.5% in the laboratory studies. The precious metals also respond relatively well to the flotation process, with gold recoveries in the rougher stage between 74.3% to 82.2%, and silver recovery approximately 86%.

There appears to be a trend in that copper recovery increases slightly when the primary grind decreases from 200 um to 100 um. However, the gold and silver extractions seem to be insensitive to the grind size. Based upon an evaluation of the test data and considering project economics, a primary P80 grind size of 150 microns is recommended.

The optimized laboratory rougher concentrate had a copper content of approximately 11% Cu. To determine marketable copper concentrate parameters, cleaner flotation tests which included a rougher concentrate regrind step were conducted on the sulfide material. With the regrind size between 30 and 33 um, and an elevated pH above 10, the cleaner concentrate flotation procedure achieved a copper concentrate grade above 40% Cu with the copper recovery being between 90.4% and 93.7%. The test program indicated that a regrind size that is too fine may lose copper recovery in the cleaner stages. Copper concentrate grades above 40% Cu were realized in the test program and this content is readily marketable. It is recommended to control the regrind size at about 40 microns.

13.5.2 Oxyde Flotation

The oxide material requires sulfidization prior to flotation. Therefore, an appropriate amount of sodium sulfide was added to a conditioning step prior to oxide flotation. With the addition of supplemental sodium sulfide and other oxide flotation reagents in both the rougher stage and cleaner flotation stages, the realized copper recovery was 55.5% to 61.7% Cu, gold and silver recoveries varied between 59.2% - 72.4% and 56.7% - 68.2%, respectively. This level of recovery is considered reasonable for the oxide copper material, and significantly higher than 35% - 40% copper recovery achieved in the previous testwork conducted in 2021.

The oxide rougher concentrate contained approximately 4% copper, which indicated the necessity of investigating a rougher concentrate regrind step and multiple cleaner flotation stages. With test regrind sizes at 23 and 29 um and two stages of cleaner flotation, the final concentrate grades assayed 17.6% and 26.2% copper, respectively. The lower concentrate grade implies poor liberation of copper minerals in the oxide material compared to that of sulfide material. However, from the testwork the finer regrind size at 23 um did not improve the final concentrate grade compared with the regrind size at 29 um. It was deduced that the adjustment of reagent schedule was the reason for the improved the final concentrate grade.

From the testwork, it was revealed that for the oxide material alone, the copper recovery can approximate 40% with the concentrate grade reaching 25% copper, after regrinding the rougher concentrate to 25-30 um and employing two stages of cleaner flotation.

13.5.3 Flotation with Blended Material

Communications with Granite Creek personnel proposed that additional flotation tests to be conducted on the blended material. Two flotation tests were conducted employing the rougher stage only, another two flotation tests were conducted with both the rougher concentrate regrind and cleaner steps, and the last investigation was a locked cycle test. The first four flotation tests were conducted on the composite sample having a 50:50 blend between sulfide and oxide material, and the last locked cycle test utilized a 60:40 blend of sulfide and oxide material.

For each flotation test, the blended material was sulfidized with sodium sulfide. A measured amount of A-OX100 reagent was added to maximize the oxide copper recovery. For the 50:50 sulfide and oxide material blend, the average copper recovery in the rougher step was between 82.4% to 84.4%, with the gold recovery being between 73% to 83.8% and the silver recovery being between 78.9% to 83.8%. The rougher concentrate contains approximately 7% copper, so additional regrinding and cleaner flotation steps were tested.

Due to the presence of oxide minerals which has poorer liberation and tend to produce a lower concentrate grade, two stages of cleaner flotation were employed during the tests. After two stages of cleaner flotation, the 50:50 sulfide and oxide blend material produced concentrate grades above 25% copper. It was found that reagent A-OX is not required during cleaner flotation since this reagent tends to significantly increase gangue entrainment into the concentrate. With a regrind step having a P80 of about 30 microns and two stages of cleaner flotation, the final concentrate will have a content of 40% copper. With the proper reagent schedule, the blended material can produce a concentrate grade above 25% copper with single stage of cleaning. However, to be conservative, two stages of cleaner flotation is recommended in the process design.

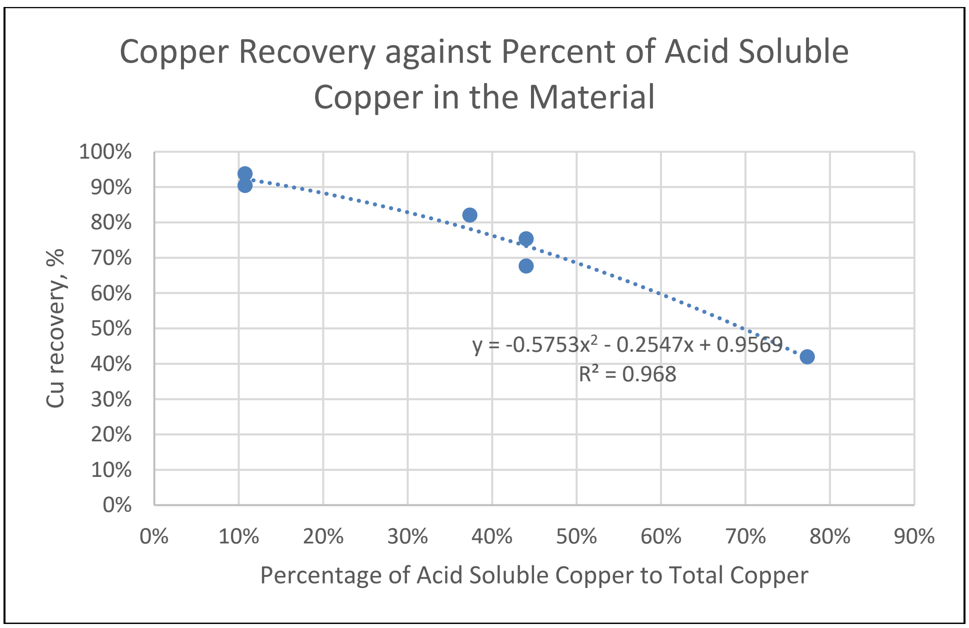
The final locked cycle test was conducted on the blended sample containing 60% sulfide and 40% oxide employing the optimum conditions tested from the previous four flotation tests. These studies employed two stages of cleaner flotation with a regrind size at 31 um and a minor amount of A-OX100 used in the rougher stage. This locked cycle test achieved 82% copper recovery, 70% gold recovery and 68.6% silver recovery. The final concentrate grade assayed 40% copper.

13.6 Metals Recovery Model

The major flotation performance differences in the grade and recovery results between sulfide material and oxide material implies that the blend ratio of two material types has a significant impact on the metal recoveries and produced concentrate grades. As both sulfide and oxide material contain a certain amount of acid soluble copper, a logical process optimization approach is to explore the relationship between the percentage of acid soluble copper in the material and the final copper recoveries. The test data used for regression analysis includes test sulfide F3, F4, oxide F3 and F4, blend test F3 and F4, and locked cycle test.

Based on the head assays of the test samples, the sulfide material contains 10.8% of acid soluble copper, and the oxide material contains 77.3% of acid soluble copper. The percentage of acid soluble copper of the blended material can be calculated based on the blend ratios between sulfide and oxide material. Figure 13-1 plots the copper recovery versus the percent of acid soluble copper in the flotation feed.

Figure 13-1 Copper Recovery vs Percent of Acid Soluble Copper in Feed



In above figure, the regression equation has a quite high R-square value of 0.968, which confirmed the correlation between percent of acid soluble copper to the final copper recovery. To further explore the relationship between the percent of acid soluble copper to gold and silver recovery, the relationships between gold and silver recovery against the percentage of acid soluble copper in the material are also plotted in Figure 13-2 and Figure 13-3 as below.

Figure 13-2 Gold Recovery vs Percent of Acid Soluble Copper in Feed

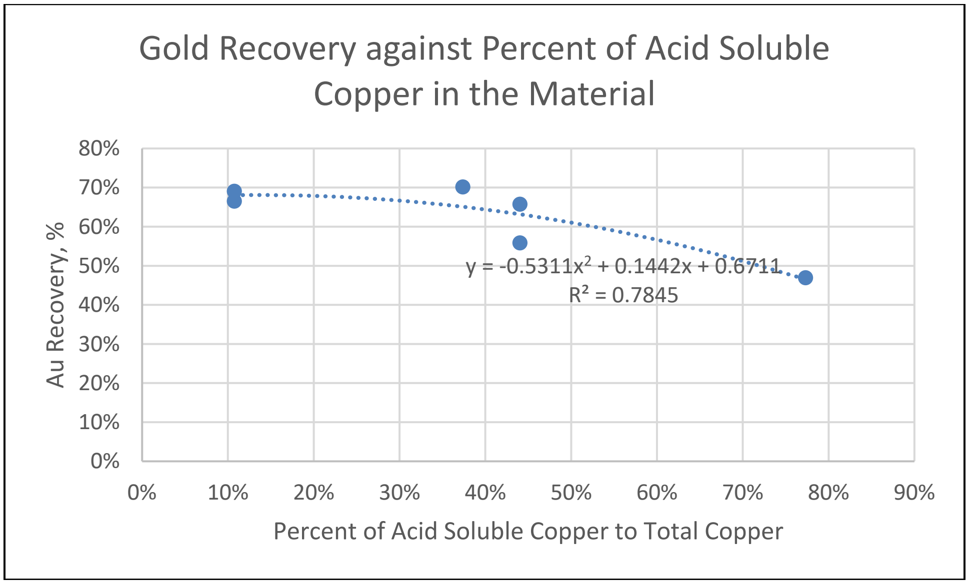
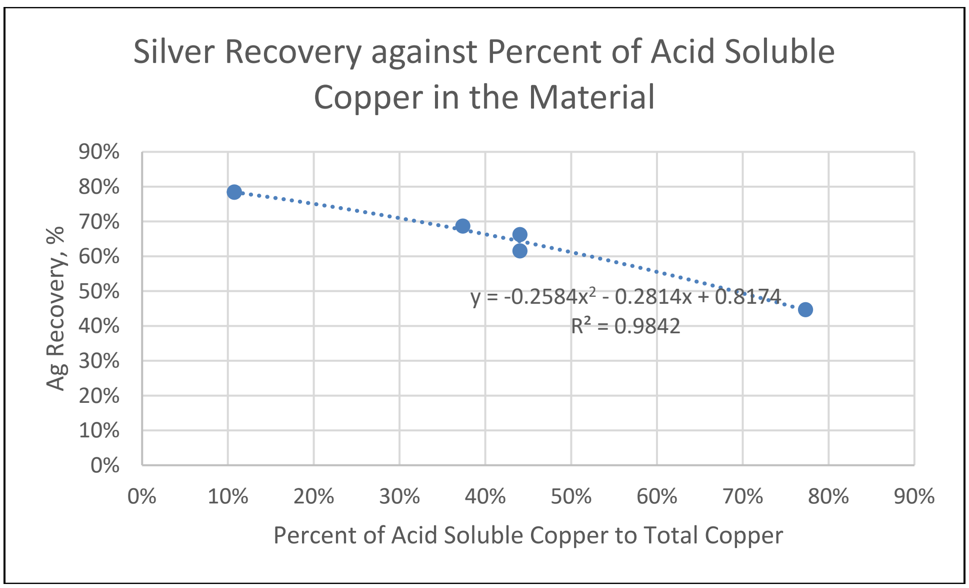


Figure 13-3 Silver Recovery vs Percent of Acid Soluble Copper in Feed



Above figures indicated that the precious metal recovery does have a reasonable correlation with the percent of acid soluble copper in the feed. Based on the regression equations, the metal recoveries can be predicted as indicated below:

For copper recovery,
When acid soluble copper is more than 80%, Recovery % = -0.475*(percent of acid soluble copper) + 0.76

When acid soluble copper is less than 80%, Recovery % = -0.5753*(percent of acid soluble copper)^2 - 0.2547*(percent of acid soluble copper) + 0.9569

For gold recovery, Recovery % = Recovery % = -0.5311*(percent of acid soluble copper)^2 +0.1442*(percent of acid soluble copper) + 0.6711

For silver recovery, Recovery % = Recovery % = -0.2584*(percent of acid soluble copper)^2 -0.2814*(percent of acid soluble copper) + 0.8147

These metal recovery predictions were also used for updating the resource model and the optimization of mine plan in this PEA study.

13.7 Conclusions and Recommendations

The previous Preliminary Economic Assessment (PEA) study by JDS proposed a process utilizing tank acid leaching for copper recovery, followed by countercurrent decantation, solvent extraction, and electrowinning for copper. The residue after the acid leach was neutralized and then went to a cyanide leach process for recovery of gold and silver, followed by ADR circuit. For a small deposit at a remote location, this process requires extensive capital cost and may involve operational challenges and risks. This process was not economical based on the previous PEA study and is not recommended.

Based on the testwork conducted by SGS Lakefield in 2022, samples from the subject deposit demonstrated a better performance by flotation process than previous test work indicated. The sulfide material demonstrated a final copper recovery in the low 90% range, with gold recovery in the high 60% range and silver recovery in the high 70% range. For oxide material, the metal recovery by flotation can realize around 40% for copper, approximately 50% for gold and 40% for silver. Flotation tests using a 50%/50% blend of sulfide and oxide material can achieve around 70% copper recovery, and 60% extraction for both gold and silver recovery. The final locked cycle flotation test with 60% sulfide and 40% oxide produced a copper recovery of 82%, gold and silver recoveries around 70%. This level of recovery is reasonable considering that significant amount of oxide mineralization in the orebody. Based on the test results and the economic evaluation, a flotation process is recommended for this material, which requires much less capital and operational cost expenditures compared to the previous process configurations considered.

Based on the test work, a P80 of 150 microns is recommended as the primary grind size for flotation. To make sure that the final flotation concentrate is marketable, two stages of cleaner flotation including a rougher concentrate regrind will be required. The regrind size is recommended between 30-35 microns based on the testwork. When the plant feed contains oxide material, the material will be sulfidized with either sodium sulfide or sodium hydrosulfide added to the conditioning tank before flotation.

In the next stage of the study, the following test work efforts and considerations are recommended.

- Since the process requires SAG milling, a SAG Milling Comminution (SMC) test or full scale JK drop weight tests are recommended.
- For both flotation concentrate and final tailings, it is recommended to conduct sedimentation test and filtration test for equipment sizing.
- Currently the test samples are composited into a sulfide sample and an oxide sample, each with the one head grade. It is recommended to conduct variability testing in the next stage, which can further explore the impact of head grade upon the final metal recovery.
- The current resource model study indicated a limited amount of mineable resources; additional drilling and exploration are recommended to increase the total resource for enhanced project economics.

- Based on the current resource model and plant throughput suggested by Granite Creek, the effective mining life is 10 years. If no additional resources are found, it is recommended to further optimize the mine life ore extraction methodology to maximize the project economics.

14 MINERAL RESOURCE ESTIMATE

14.1 Introduction

Completion of the update MRE's for the Carmacks Project involved the assessment of a drill hole database, which included all data for surface drilling completed through the end of 2021, as well as three-dimensional (3D) mineral resource models, and available written reports.

Inverse Distance Squared ("ID^2") restricted to mineralized domains was used to Interpolate grades for Cu_T (total copper in ppm), Cu_X (copper oxide in ppm), Cu_S (copper sulphide in ppm) Au (g/t), Ag (g/t) and Mo (ppm) into block models.

Measured, Indicated and Inferred mineral resources are reported in the summary tables in Section 14.11. The current MREs take into consideration that the Projects deposits may be mined by open pit and underground mining methods.

14.2 Drill Hole Database

In order to complete MREs for the Carmacks Project, a database comprising a series of comma delimited spreadsheets containing drill hole information was provided by Granite Creek. The database included diamond drill hole location information (NAD83 / UTM Zone 08), survey data, assay data, lithology data, and specific gravity data for Zones 1, 4, 7, 2000S, 12 and 13. The data in the assay table included assays for Cu_T (ppm), Cu_X (ppm), Au (g/t), Ag (g/t) and Mo (ppm); Cu_S was calculated (Cu_S = Cu_T – Cu_X). The data was then imported into GEOVIA GEMS version 6.8.3 software ("GEMS") for statistical analysis, block modeling and resource estimation.

The database provided for the current MREs comprise data for 489 surface drill holes totaling 59,679.07 metres completed on the Carmacks Project area between 1970 and 2021 (Table 14-1, Figure 14-1 and Figure 14-2). This includes 36 drill holes (RC and diamond) totaling 9,413.06 m completed by Granite Creek between the fall of 2020 the fall of 2021. The database used for the MREs totals 12,794 drill core assay samples representing 17,233 m of drilling.

The database was checked for typographical errors in drill hole locations, down hole surveys, lithology, assay values and supporting information on source of assay values. Overlaps and gapping in survey, lithology and assay values in intervals were checked. No errors have been noted. The database is of sufficient quality to be used for the current resource estimates.

Table 14-1 Drill Holes in the Carmacks Project Database

Company	Year	No. Holes	Metres	Hold Type
Historic	1970-72	30	6,782.44	DDH
Western Copper	1990-92	47	4,792.24	DDH
Western Copper	1990-92	11	856.79	RC
Western Copper	1995-96	28	1,106.39	RC
Western Copper	2006-08	263	29,001.93	DDH
Copper North	2014-15	40	3,738.26	DDH
Copper North	2017	34	3,987.96	DDH
Granite Creek	2020-21	25	8,269.97	DDH
Granite Creek	2021	11	1,143.09	RC
Total		489	59,679.07	

Figure 14-1 Plan View of the Distribution of Drilling in the Carmacks Deposit Area

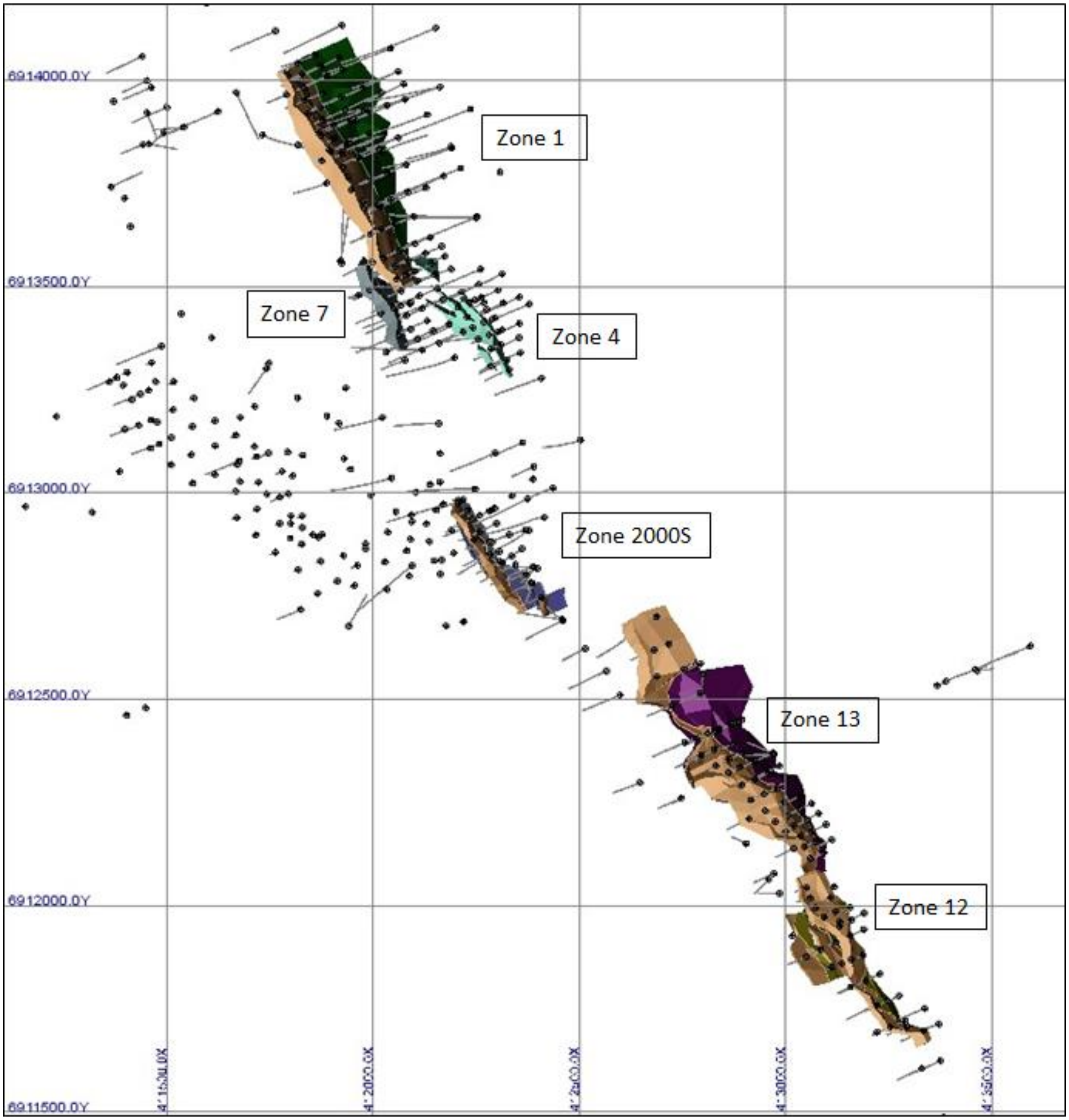
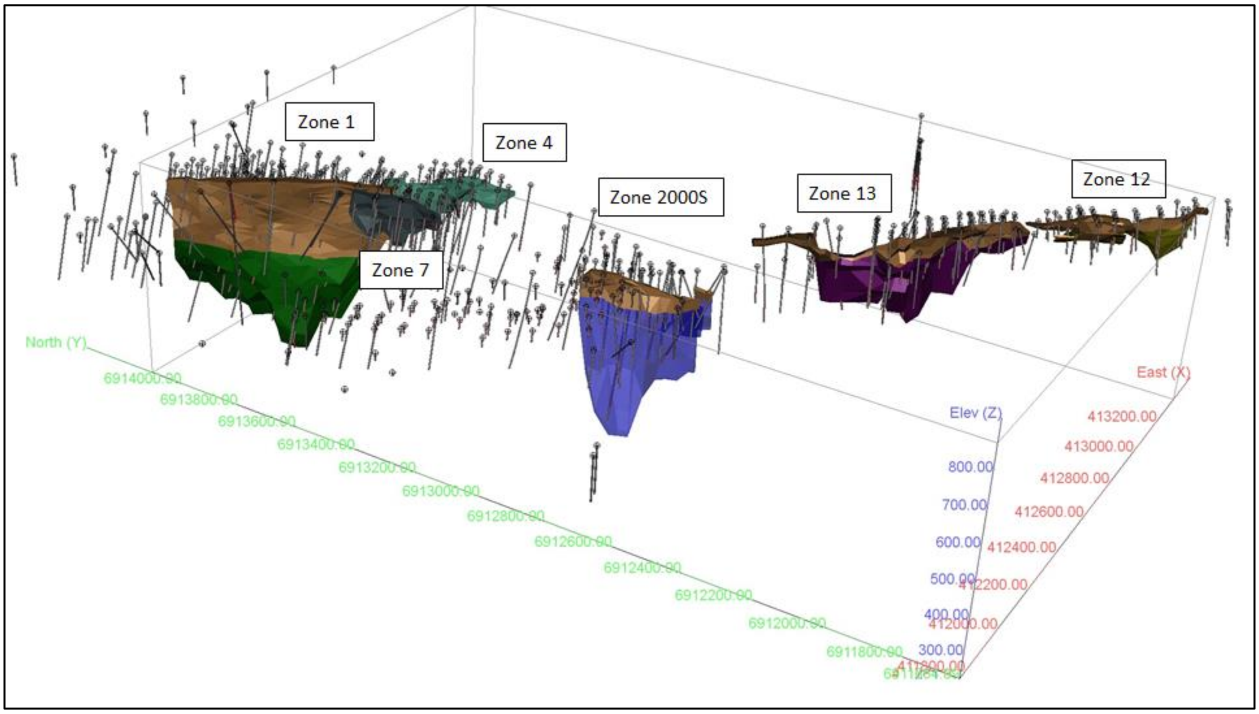


Figure 14-2 Isometric View Looking Northwest of the Distribution of Drilling in the Carmacks Deposit Area



14.3 Mineral Resource Modelling and Wireframing

Three-dimensional (3D) grade controlled wireframe models, representing separate mineralized structures for Zones 1, 4, 7, 2000S, 12 and 13 were originally constructed in GEMS (6.7) by Gilles Arseneau (Arseneau, 2016) for the 2016 MREs for the Carmacks Project. The 2016 wireframe models incorporated date for historical drilling, most of which has been carried out by Western Copper as well as 50 drill holes drilled by Copper North in 2014-15.

The 2016 wireframe models (in DXF format) were imported into GEMS, reviewed by the Author and revised based on data from drill holes completed by Copper North in 2017 and Granite Creek in 2020 and 2021. The revised 3D grade-controlled models were built in GEMS by visually interpreting mineralized intercepts on northeast-trending cross sections, perpendicular to the general strike of the mineralization. Polylines (3D rings) of mineral intersections (snapped to drill holes), representing a ~0.2% total copper cut-off, were made on each section and these were wireframed together to create continuous resource wireframe models. Polylines of mineral intersections were constructed on 30 m spaced cross sections with a 15 m sectional influence. The 3D grade-controlled wireframe models are summarized in Table 14-2. The modeling exercise provided broad controls of the dominant mineralizing direction for each deposit. Deposits of the Carmacks Project extend for approximately 2.8 km along strike (NNW-SSE) (Figure 14-3).

In addition, the Author was provided with a 3D DXF surface of topography (Figure 14-4) as well as a DXF surface of the base of overburden (Figure 14-5) for the deposit areas. The base of overburden surface was revised to fit the drill lithology information for the 2017 and 2020 – 2021 drill holes. All resource wireframe models were then clipped against the overburden-bedrock surface for each deposit area (Figure 14-3).

Surfaces representing the boundary between the upper oxide and lower sulphide mineralization was interpreted based on drill hole intersections (Arseneau, 2016). The transition between oxide and sulphide mineralization occurs over a few metres for most zones with the exception of Zone 13 where a larger volume of transitional material may be present. The boundary between oxide and fresh rock was interpreted as occurring where the proportion of oxide copper to total copper dropped below 20%. 3D surfaces were generated by connecting all drill hole points to form the oxide/sulphide interface (Figure 14-1). These surfaces were provided to the Author and were revised to incorporate data from the 2017 and 2020-2021 drilling.

Zones 1, 4, 7 models define a steep east-northeast dipping structure which extends for 900 m along strike and reaches a maximum depth of approximately 550 m below surface in Zone 1 (Figure 14-6); Zone 2000S model defines a near vertical structure which extends for 350 m along strike and reaches a maximum depth of approximately 450 below surface (Figure 14-7); Zones 12 and 13 models define moderate northeast dipping structures which extend for 1,250 m along strike and reaches a maximum depth of approximately 325 m below surface in Zone 13 (Figure 14-8). All deposits are open down dip.

Table 14-2 Carmacks Project Deposit Domain Descriptions

Domain	Rock Code	Density	Domain Volume	Domain Tonnage
Zone 1 - Oxide	1	2.64	3,990,311	10,534,421
Zone 1 - Sulphide	11	2.78	3,989,397	11,090,524
Zone 4 - Oxide	4	2.64	405,648	1,070,911
Zone 7 - Oxide	7	2.64	399,415	1,054,456
Zone 2000S - Oxide	20001	2.64	380,593	1,004,766
Zone 2000S - Sulphide	20002	2.75	2,755,798	7,578,445
Zone 12 Oxide	121	2.64	481,101	1,270,107
Zone 12 Sulphide	122	2.71	377,834	1,023,930
Zone 13 Oxide	131	2.64	1,670,626	4,410,453
Zone 13 Sulphide	132	2.71	5,350,157	14,498,925
Total:			**19,800,880**	**53,536,936**

Figure 14-3 Plan View of the Distribution of Drill holes and Carmacks Deposit Grade Controlled Wireframe Models

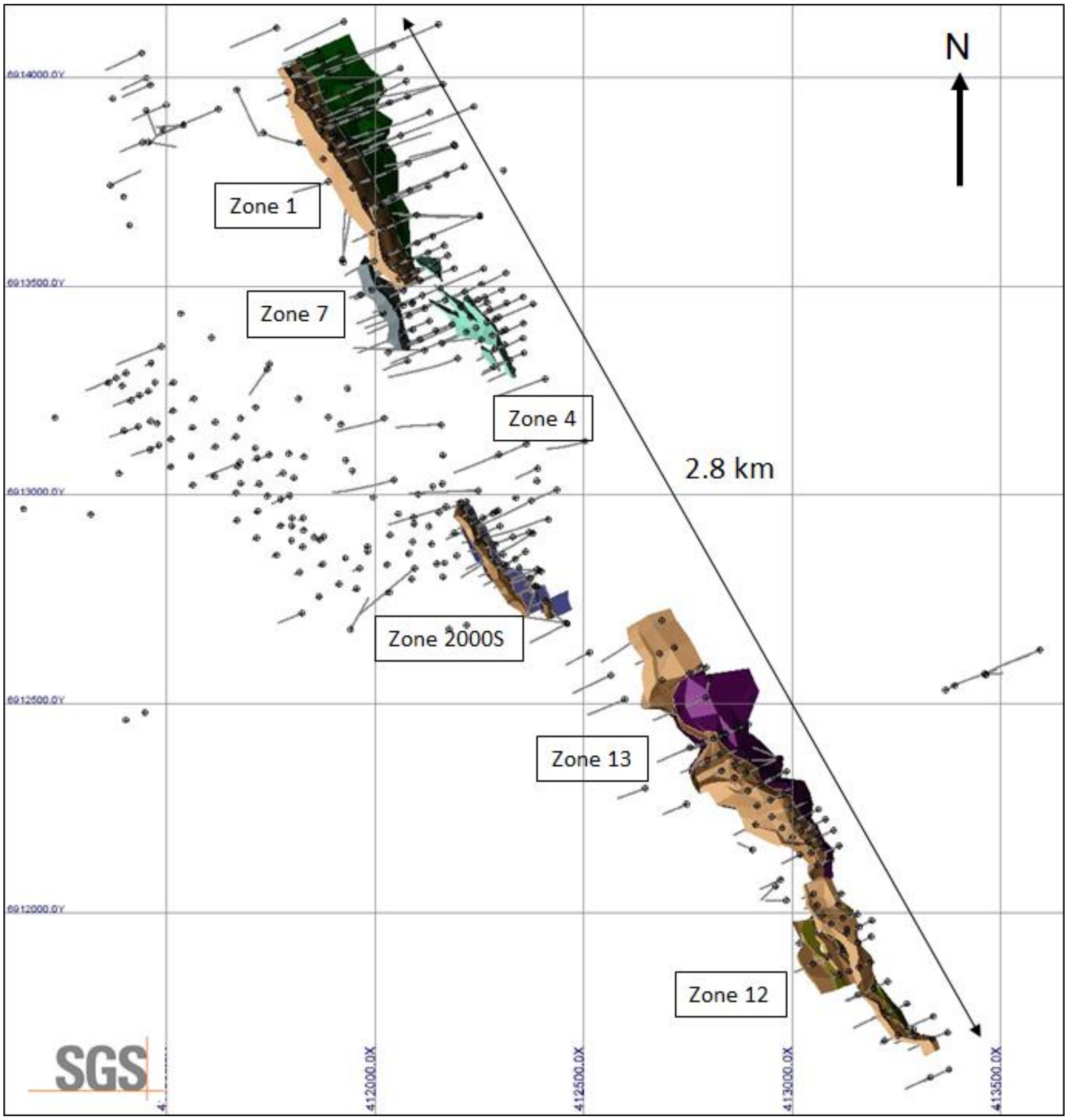


Figure 14-4 Isometric View Looking Northwest of the Topographic Surface

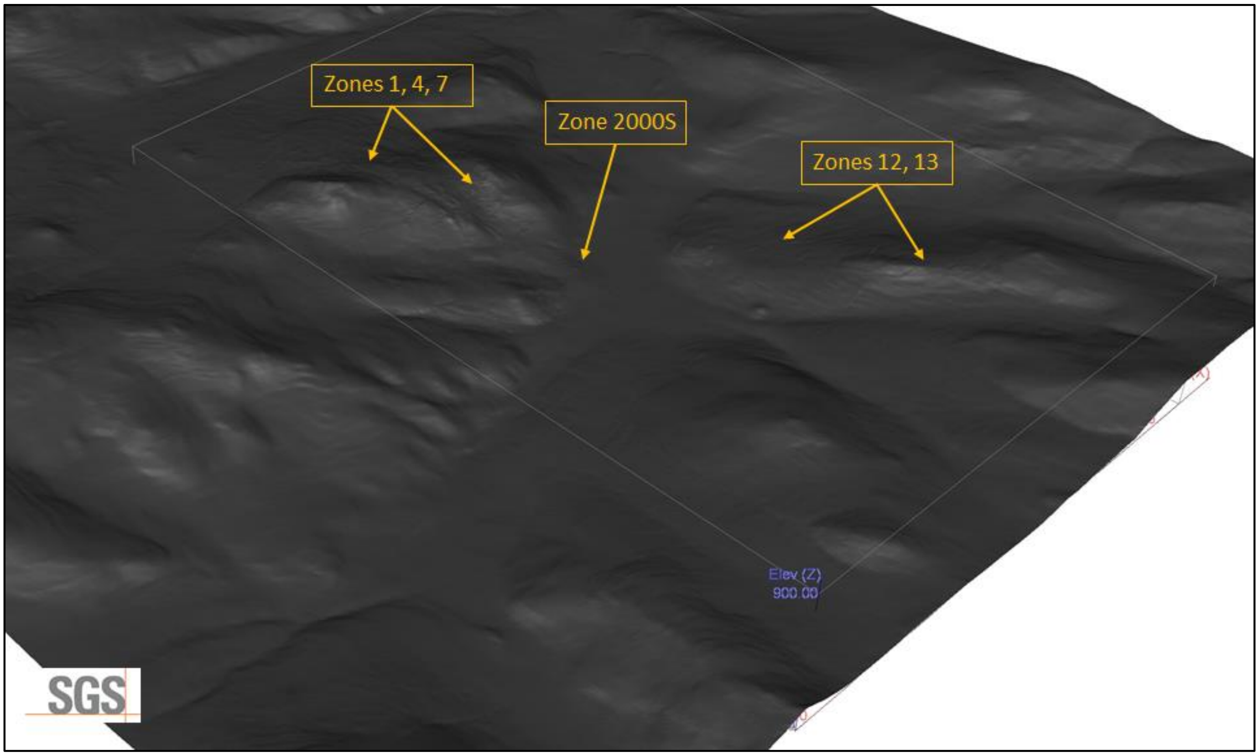


Figure 14-5 Isometric View Looking Northwest of the Overburden Surface

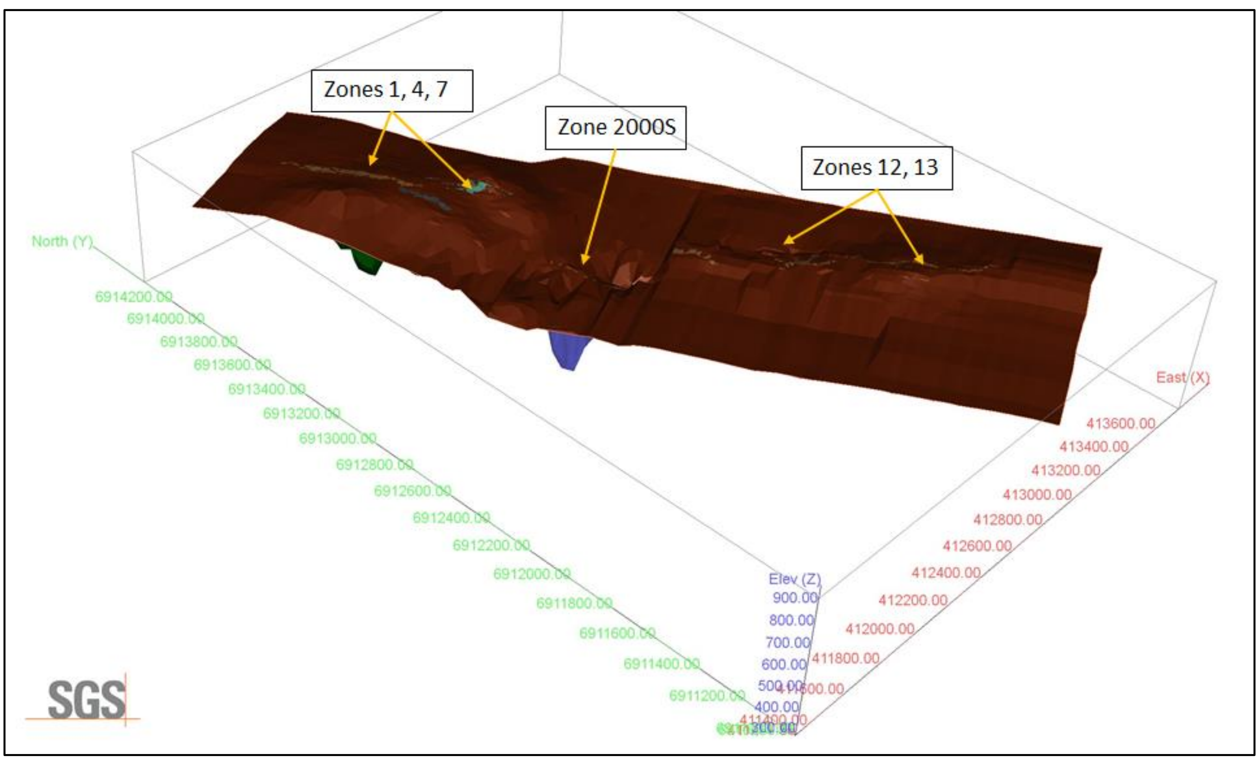


Figure 14-6 Isometric View Looking Northwest of the Zones 1, 4, 7 Oxide and Sulphide Zones and Distribution of the Drill holes

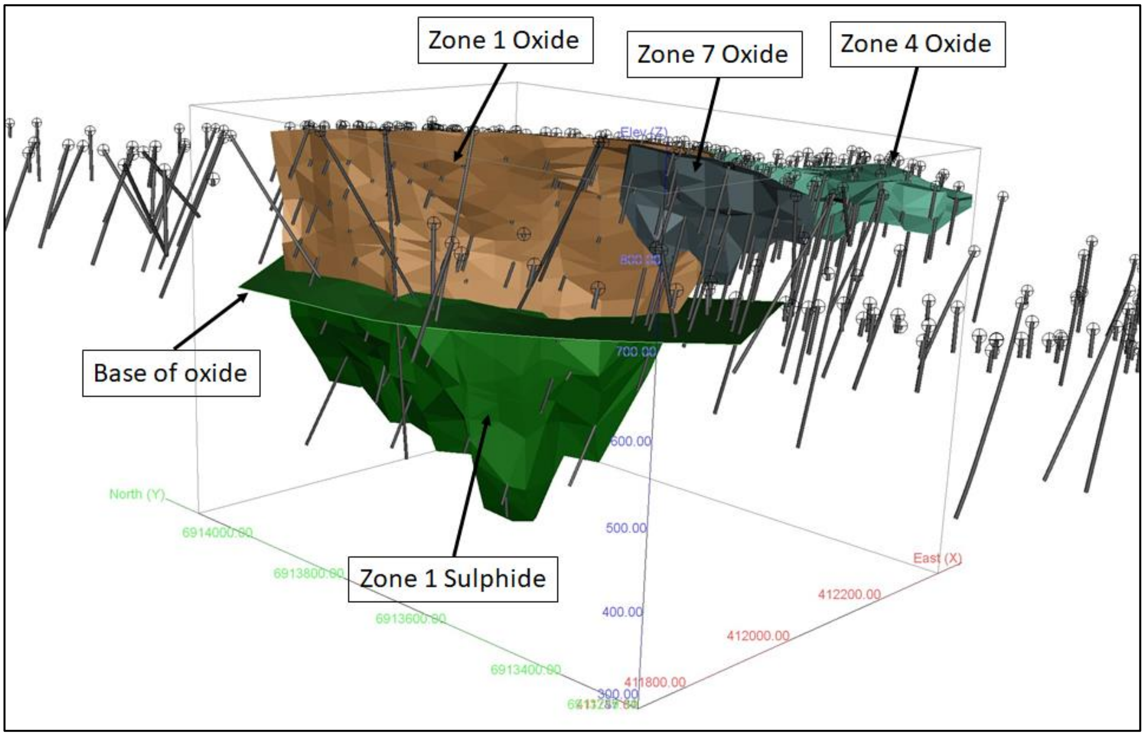


Figure 14-7 Isometric View Looking Northwest of the Zone 2000S Oxide and Sulphide Zones and Distribution of the Drill hole

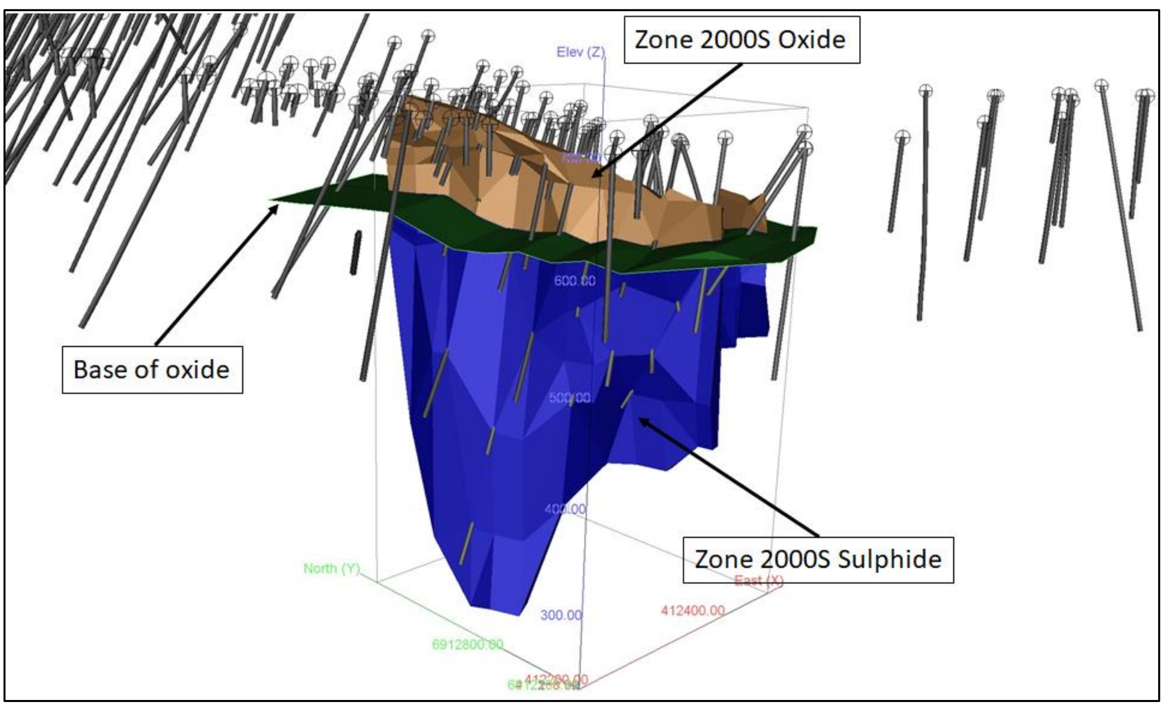
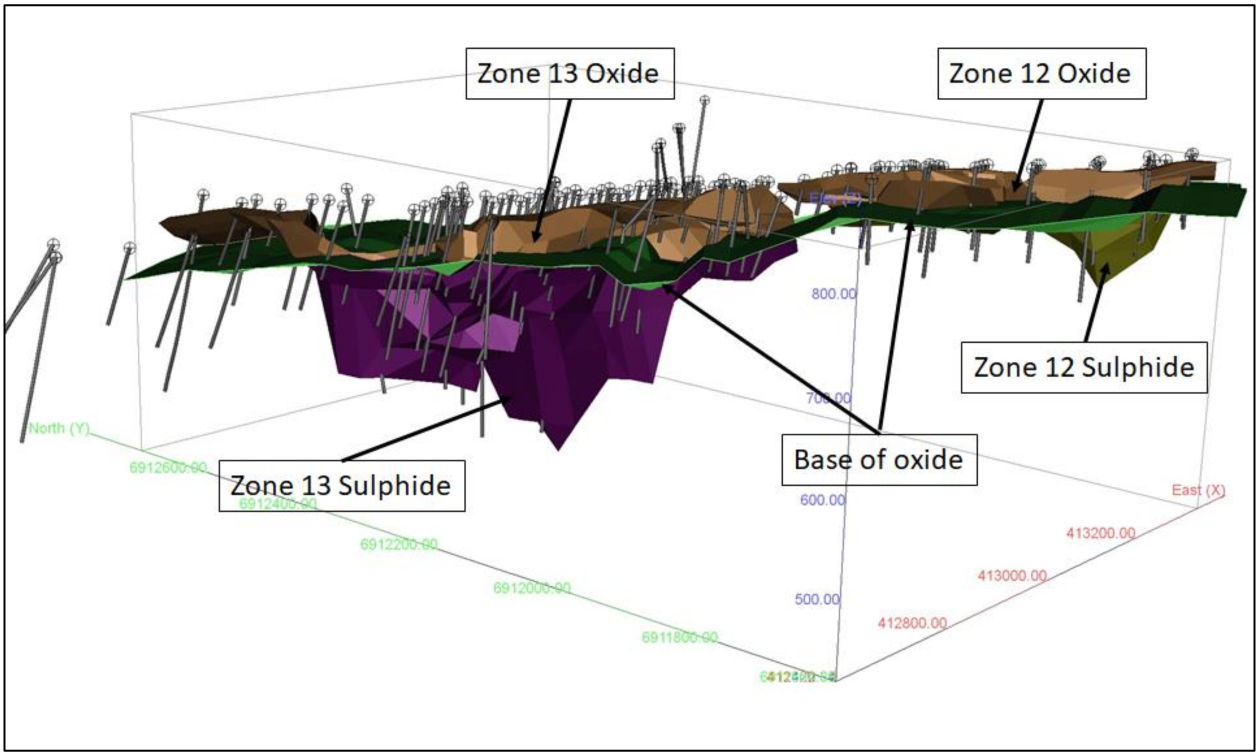


Figure 14-8 Isometric View Looking Northwest of the Zones 12, 13 Oxide and Sulphide Zones and Distribution of the Drill holes



14.4 Compositing

The assay sample database available for the current resource modelling totals 12,794 drill core assay samples representing 17,233 metres of drilling. Of these assays, 7,281 from 239 drill holes occur within the Carmacks Project mineral domains. A statistical analysis of the drill core assay data from within the mineralized domains is presented in Table 14-3. Average width of the drill core sample intervals within the models is 1.33, within a range of 0.18 m to 4.27 m. Of the total assay population approximately 73% are 1.5 m or less; an additional 19 % of samples are between 1.5 and 2.0 m. To minimize the dilution and over smoothing due to compositing, a composite length of ~2.0 m was chosen as an appropriate composite length for the resource estimation of all deposits.

For the Carmacks Project resource estimates, composites were generated starting from the collar of each drill hole, and un-assayed intervals were given a value of 0.0001 for all elements. The composites were extracted to point files for statistical analysis and capping studies. The composites were grouped based on the mineral domain (rock code) of the constraining wireframe model. Each wireframe model was considered a hard boundary and only those 2 m composites constrained by each wireframe were used to to estimate the mineral resource for that wireframe.

A total of 4,860 composite sample points occur within the resource grade-controlled models (Table 14-4); the average grade of all composites varies based on deposit. The cumulative composite sample points for each deposit was used to interpolate grade into resource blocks for each deposit.

14.5 Grade Capping

A statistical analysis of the cumulative composite database within the Carmacks Project wireframe models (the “resource” population) was conducted to investigate the presence of high-grade outliers, which can have a disproportionately large influence on the average grade of a mineral deposit. High grade outliers in the composite data were investigated using statistical data (Table 14-4), histogram plots, and cumulative probability plots of the composite data. The statistical analysis was completed using GEMS.

Analysis of the composite data for all zones indicate very few outliers within the database. It is the Author’s opinion that minimal capping of high-grade composites to limit their influence during the grade estimation is necessary. Capping values are based on a review of combined composites from all zones rather than by zone. The Author believes that the impact of capping composites is negligible to the overall resource estimate for the Carmacks deposits. Capping values are as follows:

- Silver – capped at 100 gms; only 1 sample in Zone 2000S sulphide zone
- Cu_T – capped at 5.5 % (55,000 ppm): 6 samples from Zones 1, 4, 7 Oxide
- Cu_S – capped at 3.0 % (30,000 ppm): 14 Samples from Zones 13 sulphide, 2000S sulphide, Zones 147 oxide and Zone 1 sulphide
- Cu_X – capped at 4.2 % (42,000 ppm): 9 samples from Zones 1, 4, 7 Oxide
- Mo – 0.10 % (1,000 ppm): 18 samples from Zones 13 sulphide, 2000S sulphide, Zones 147 oxide and Zone 1 sulphide

Table 14-3 Statistical Analysis of the Drill Core Assay Data from Within the Carmacks Project Mineral Resource Models

Variable	Zones							
	1, 4, 7 Oxide				1, 4, 7 Sulphide			
	Cu_T (%)	Cu_X (%)	Au (g/t)	Ag (g/t)	Cu_T (%)	Cu_X (%)	Au (g/t)	Ag (g/t)
Total # Assay Samples	2,994				947			
Average Sample Length (m)	1.33				1.25			
Minimum Grade	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum Grade	10.95	7.82	23.70	60.00	3.78	0.91	3.54	27.40
Mean	1.04	0.83	0.49	5.04	0.68	0.04	0.19	2.61
Standard Deviation	0.98	0.80	0.94	6.89	0.61	0.06	0.21	2.55
Coefficient of variation	0.94	0.96	1.92	1.37	0.89	1.58	1.07	0.98
97.5 Percentile	3.52	2.89	2.70	26.20	2.40	0.17	0.70	9.70
Variable	2000S Oxide				2000S Sulphide			
	Cu_T (%)	Cu_X (%)	Au (g/t)	Ag (g/t)	Cu_T (%)	Cu_X (%)	Au (g/t)	Ag (g/t)
Total # Assay Samples	265				572			
Average Sample Length (m)	1.45				1.53			
Minimum Grade	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum Grade	2.39	2.19	1.16	13.90	18.97	0.39	0.85	1,775
Mean	0.51	0.35	0.15	2.32	0.66	0.03	0.15	6.24
Standard Deviation	0.41	0.34	0.18	2.37	0.96	0.04	0.12	74.10
Coefficient of variation	0.80	0.96	1.20	1.02	1.45	1.63	0.83	11.88
97.5 Percentile	1.56	1.03	0.64	7.85	1.92	0.15	0.45	10.80
Variable	12, 13 Oxide				12, 13 Sulphide			
	Cu_T (%)	Cu_X (%)	Au (g/t)	Ag (g/t)	Cu_T (%)	Cu_X (%)	Au (g/t)	Ag (g/t)
Total # Assay Samples	1,050				1,453			
Average Sample Length (m)	1.31				1.31			
Minimum Grade	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum Grade	4.25	3.76	2.65	18.50	5.60	0.68	2.92	28.60
Mean	0.44	0.31	0.11	1.83	0.53	0.05	0.12	2.00
Standard Deviation	0.38	0.31	0.13	2.00	0.47	0.06	0.13	1.95
Coefficient of variation	0.87	0.99	1.25	1.09	0.89	1.22	1.07	0.97
97.5 Percentile	1.24	1.03	0.36	7.05	1.63	0.23	0.37	6.90

Table 14-4 Summary of the 2.0 metre Composite Data Constrained by the Carmacks Project Mineral Resource Models

Variable	**Zones**							
	1, 4, 7 Oxide				**1, 4, 7 Sulphide**			
	Cu_T (%)	Cu_X (%)	Au (g/t)	Ag (g/t)	Cu_T (%)	Cu_X (%)	Au (g/t)	Ag (g/t)
Total # Assay Samples	2,014				592			
Average Sample Length (m)	2.00				2.00			
Minimum Grade	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum Grade	7.07	5.68	12.17	53.00	3.64	0.91	2.00	17.00
Mean	1.04	0.84	0.48	4.82	0.71	0.04	0.20	2.60
Standard Deviation	0.87	0.73	0.76	5.99	0.55	0.07	0.19	2.34
Coefficient of variation	0.83	0.87	1.59	1.24	0.77	1.72	0.93	0.90
97.5 Percentile	3.30	2.74	2.50	23.00	2.16	0.16	0.70	9.41

Variable	2000S Oxide				2000S Sulphide			
	Cu_T (%)	Cu_X (%)	Au (g/t)	Ag (g/t)	Cu_T (%)	Cu_X (%)	Au (g/t)	Ag (g/t)
Total # Assay Samples	185							
Average Sample Length (m)	2.00				2.00			
Minimum Grade	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum Grade	1.84	1.72	0.81	12.28	5.22	0.27	0.62	900.00
Mean	0.46	0.33	0.14	2.29	0.63	0.03	0.15	5.05
Standard Deviation	0.33	0.27	0.14	1.95	0.55	0.04	0.10	42.88
Coefficient of variation	0.70	0.81	0.98	0.85	0.87	1.45	0.72	8.49
97.5 Percentile	1.10	0.87	0.52	6.92	1.79	0.14	0.38	9.00

Variable	12, 13 Oxide				12, 13 Sulphide			
	Cu_T (%)	Cu_X (%)	Au (g/t)	Ag (g/t)	Cu_T (%)	Cu_X (%)	Au (g/t)	Ag (g/t)
Total # Assay Samples	685				946			
Average Sample Length (m)	2.00				2.00			
Minimum Grade	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum Grade	3.85	3.27	1.68	18.50	4.11	0.52	1.57	19.05
Mean	0.44	0.31	0.10	1.76	0.52	0.05	0.12	1.93
Standard Deviation	0.35	0.27	0.11	1.78	0.40	0.06	0.10	1.60
Coefficient of variation	0.79	0.87	1.06	1.01	0.76	1.10	0.85	0.83
97.5 Percentile	1.17	0.93	0.35	6.36	1.51	0.23	0.32	5.60

14.6 Specific Gravity

In 1991, bulk densities were estimated by Chemex on 21 drill core samples (Arseneau, 2016). The samples submitted comprised of five granodiorites, two pegmatites and fourteen gneiss samples. The specific gravity ("SG") of granodiorite samples surrounding the mineralization ranged between 2.69 to 2.71 for an average of 2.70. The specific gravity of gneissic material hosting the mineralization ranged from 2.59 to 2.97 although only one sample was greater than 2.73. In 2006 and 2007, bulk density was measured by Aurora in the field on 1,358 drill core samples. An average specific gravity of 2.64 was determined for samples collected within Zone 1 oxide and 2.75 within the Zone 1 sulphide. During the 2015 drill program, Copper North collected an additional 1,225 bulk density measurements from zones 12, 13 and 2000S. The average density of 90 mineralized samples collected in 2015 was 2.74 t/m^3.

For the current MREs, the Author was provided with an SG database of 6,101 samples (217 drill holes) from mineralized (1,779) and unmineralized rocks for all zones (Table 14-5). The database includes 2,269 samples collected in 2017 by Copper North and 185 samples collected by Granite Creek in 2021.

SG measurements were predominately completed on core by the Weight in Air/Weight in Water method using the following formula:

SG = [sample weight dry (g) / (dry weight (g) - wet weight (g))]

Samples from the 2021 drill program were done on core in the lab by pychnometer.

Based on the results of the SG measurements a fixed SG value of 2.64 is used for all oxide zones, fixed SG values of 2.71 to 2.78 are used for the sulphide zones and a fixed SG of 2.66 is used for waste (Table 14-5).

Table 14-5 Summary of Specific Gravity Measurements for the Carmacks Project Deposits

Zone	Total # of Drill Holes	Total # of SG Values	Range	Average SG Values	Used for MREs
Complete Data Set	217	6,101	2.01 – 3.35	2.67	
1, 4, 7 Oxide		206	2.24 – 2.93	2.65	2.64
1 Sulphide		108	2.41 – 2.95	2.78	2.78
2000S Oxide		324	2.30 – 2.99	2.62	2.64
2000S Sulphide		202	2.35 – 3.35	2.70	2.75*
12, 13 Oxide		517	2.36 – 2.97	2.66	2.64
12, 13 Sulphide		422	2.30 – 3.18	2.71	2.71
Waste		4,322	2.01 – 3.26	2.66	2.66

**Predominantly based on 70 samples collected in 2021. Additional data recommended.*

14.7 Block Model Parameters

The Carmacks Project deposit wireframe grade controlled models are used to constrain composite values chosen for interpolation, and the mineral blocks reported in the estimate of the mineral resource. Block models (Table 14-6; Figure 14-9 and Figure 14-10) within NAD83 / UTM Zone 8 North space were placed over the wireframe models with only that portion of each block inside the wireframe models recorded (as a percentage of the block) as part of the MRE's (% Block Model). Block sizes were selected based on drillhole spacing, composite assay length, the geometry of the mineralized structures, and the selected starting mining method (open pit and underground). At the scale of the Carmacks Project Deposits this provides a reasonable block size for discerning grade distribution, while still being large enough not to mislead when looking at higher cut-off grade distribution within the model. The model was intersected with a topographic surface models and overburden surface models to exclude blocks, or portions of blocks, that extend above theses surfaces.

Table 14-6 Carmacks Deposits Block Model Geometry

Model Name	X (East; Columns)	Y (North; Rows)	Z (Level)
Zones 1, 4, 7 Block Model			
Origin (NAD83 / UTM Zone 8N)	411883.512	6913000.36	935
Extent	145	226	132
Block Size	5	5	5
Rotation (counter clockwise)	24.2°		
Zone 2000s Block Model			
Origin (NAD83 / UTM Zone 8N)	412180	6912400	820
Extent	125	135	120
Block Size	5	5	5
Rotation (counter clockwise)	25°		
Zones 12, 13 Block Model			
Origin (NAD83 / UTM Zone 8N)	412970.657	6911294.72	920
Extent	140	325	105
Block Size	5	5	5
Rotation (counter clockwise)	25°		

Figure 14-9 Isometric View Looking Northeast Showing the Carmacks Project Deposit Mineral Resource Block Model and Wireframe Grade-Controlled Models

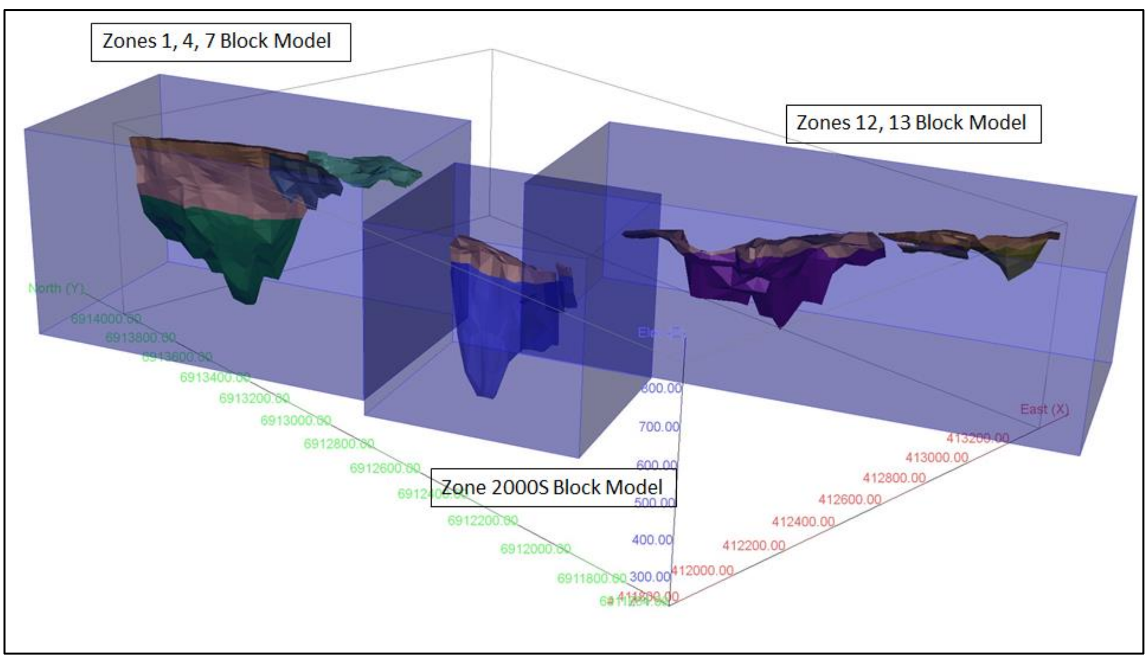
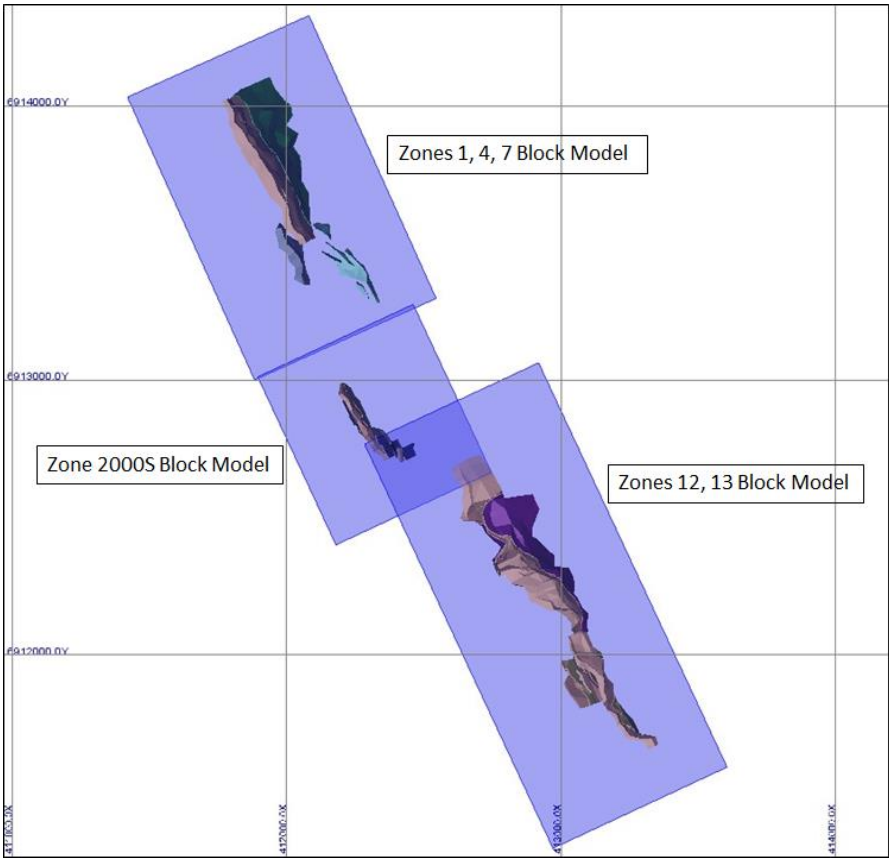


Figure 14-10 Plan View: Carmacks Project Mineral Resource Block Models and Wireframe Grade-Controlled Models



14.8 Grade Interpolation

Grades for Cu_T, Cu_X, Cu_S, Ag, Au and Mo for each deposit mineralized structure was interpolated into blocks by the Inverse Distance Squared (ID^2) calculation method. Search ellipses for each of the mineral domains was interpreted based on drill hole (Data) spacing, and orientation and size of the resource wireframe models (Table 14-7). The search ellipse axes are generally oriented to reflect the observed preferential long axis (geological trend) of the mineral structures and the observed trend of the mineralization down dip/down plunge.

Three passes were used to interpolate grade into all of the blocks in the mineral domains (Table 14-7). Blocks were classified as Measured if they were populated with grade during Pass 1, Indicated if they were populated with grade during Pass 2. All remaining blocks were classified as Inferred if they were populated with grade during Pass 3. In addition, it was decided that inferred blocks that occurred at the oxide-sulphide interface be re-classified as Indicated as the classification at the contact was more a reflection of domaining rather than availability of data. Otherwise the oxide-sulphide boundary was considered a hard boundary and metal grades were restricted by composites by domain.

Grades were interpolated into blocks using a minimum and maximum number of composites based on available data in each mineral domain, to generate block grades during Pass 1 -3 (Table 14-7). During Pass 1, a maximum of 3 samples per drill hole (or minimum of 3 drill holes) is used to generate block grades; during Pass 2, a maximum of 3 samples per drill hole (or minimum of 2 drill holes) is used to generate block grades; during Pass 3, a maximum of 3 samples per drill hole or minimum of 1 drill hole is used to generate block grades.

Table 14-7 Grade Interpolation Parameters by Deposit

Parameter	Zones 1, 4, 7		
	Pass 1	Pass 2	Pass 3
	Measured	Indicated	Inferred
Calculation Method	ID2		
Search Type	Ellipsoid		
Principle Azimuth	50°		
Principle Dip	-70°		
Intermediate Azimuth	150°		
Anisotropy X	50	90	130
Anisotropy Y	50	90	130
Anisotropy Z	10	15	30
Min. Samples	7	5	3
Max. Samples	12	10	10
Samples/drill hole	3	3	3
Parameter	Zone 2000S		
	Pass 1	Pass 2	Pass 3
	Measured	Indicated	Inferred
Calculation Method	ID2		
Search Type	Ellipsoid		
Principle Azimuth	45°		
Principle Dip	-85°		
Intermediate Azimuth	145°		
Anisotropy X	45	90	120
Anisotropy Y	45	90	120
Anisotropy Z	10	15	20
Min. Samples	7	5	3
Max. Samples	12	10	10
Samples/drill hole	3	3	3
Parameter	Zone 12		
	Pass 1	Pass 2	Pass 3
	Measured	Indicated	Inferred
Calculation Method	ID2		
Search Type	Ellipsoid		
Principle Azimuth	55°		
Principle Dip	-55°		
Intermediate Azimuth	150°		
Anisotropy X	45	90	120
Anisotropy Y	45	90	120
Anisotropy Z	10	15	20
Min. Samples	7	5	3
Max. Samples	12	10	10
Samples/drill hole	3	3	3
Parameter	Zone 13		

	Pass 1	Pass 2	Pass 3
	Measured	Indicated	Inferred
Calculation Method	ID2		
Search Type	Ellipsoid		
Principle Azimuth	50°		
Principle Dip	-40°		
Intermediate Azimuth	145°		
Anisotropy X	45	90	120
Anisotropy Y	45	90	120
Anisotropy Z	10	15	20
Min. Samples	7	5	3
Max. Samples	12	10	10
Samples/drill hole	3	3	3

14.9 Mineral Resource Classification Parameters

The MREs for the Carmacks Project are prepared and disclosed in compliance with all current disclosure requirements for mineral resources set out in the NI 43-101 Standards of Disclosure for Mineral Projects (2016). The classification of the current MREs into Measured, Indicated and Inferred is consistent with current CIM Definition Standards - For Mineral Resources and Mineral Reserves (2014), including the critical requirement that all mineral resources “have reasonable prospects for eventual economic extraction”.

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth’s crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

Interpretation of the word ‘eventual’ in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage ‘eventual economic extraction’ as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

Measured Mineral Resource

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

Indicated Mineral Resource

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Pre-Feasibility Study which can serve as the basis for major development decisions.

Inferred Mineral Resource

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.

There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource.

14.10 Mineral Resource Statement

The general requirement that all mineral resources have “reasonable prospects for eventual economic extraction” implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account extraction scenarios and processing recoveries. In order to meet this requirement, the Author considers that the Carmacks Project deposit mineralization is amenable for open pit and underground extraction.

In order to determine the quantities of material offering “reasonable prospects for eventual economic extraction” by an open pit, Whittle™ pit optimization software and reasonable mining and processing assumptions to evaluate the proportions of the block model that could be “reasonably expected” to be mined from an open pit are used. The pit optimization for the Carmacks Project was completed by SGS for the current MREs and the pit optimization parameters used are summarized in Table 14-8. Whittle pit shells at a revenue factor of 1.0 (i.e. 100 % of base case metal prices) were selected as the ultimate pit shells for the purposes of reporting the Carmacks Project MREs. A selected base case cut-off grade of 0.30 % Cu_T is used to determine the in-pit MRE for the Carmacks Project deposits. The pit optimization tended to take in > 90% of the oxide material from all deposits. As result, based on the shallow nature of the oxide mineralization, it was decided that the remaining oxide mineralization be included in the in-pit resources. It is the Authors opinion that this material will be extracted by open pit methods.

The reader is cautioned that the results from the pit optimization are used solely for the purpose of testing the “reasonable prospects for economic extraction” by an open pit and do not represent an attempt to estimate mineral reserves. There are no open pit mineral reserves on the Property. The results are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cut-off grade.

In order to determine the quantities of material offering “reasonable prospects for economic extraction” by underground mining methods, reasonable mining assumptions to evaluate the proportions of the block model that could be “reasonably expected” to be mined from underground are used. A review of the size, geometry and continuity of mineralization of each deposit, and spatial distribution of the three deposits (all within a 1.5 x 1.5 km area), was conducted to determine the underground mineablility of the deposits. It is envisioned that the deposits of the Carmacks Project may be mined using low cost underground bulk mining methods below the pit shells. The underground parameters used to determine a base case cut-off grade for reporting of underground resources is presented in Table 14-8. Based on these parameters, a selected base case cut-off grade of 0.6 % Cu_T is used to determine the below-pit MREs for the Carmacks Project deposits.

The reader is cautioned that the reporting of the underground resources are presented undiluted and in situ (no minimum thickness), constrained by continuous 3D wireframe models, and are considered to have reasonable prospects for eventual economic extraction. There are no underground mineral reserves reported on the Property.

The 2022 MREs for the Carmacks Project are presented in Table 14-9 and Table 14-10 (Figure 14-11 to Figure 14-14).

Highlights of the Carmacks Project MRE:

- In-Pit Oxide – 15.7 million tonnes in Measured and Indicated categories, grading 0.94% Cu, 0.36 g/t Au, 3.23 g/t Ag and 0.01% Mo
- In-Pit Sulphide – 19.2 million tonnes in Measured and Indicated categories, grading 0.71% Cu, 0.18 g/t Au, 2.74 g/t Ag and 0.01% Mo
- Below Pit Sulphide – 1.4 million tonnes in Measured and Indicated categories, grading 0.82% Cu, 0.19 g/t Au, 2.88 g/t Ag and 0.01% Mo

- **Combined Measured and Indicated - 36.3 million tonnes, grading 0.81% Cu, 0.26 g/t Au, 3.23 g/t Ag and 0.01% Mo**

Table 14-8 Parameters used for Whittle™ Pit Optimization and to Estimate the Open Pit and Underground Base Case Cut-off Grades for the Carmacks Project MREs

Input Data for Open Pit and Underground Mining Scenarios		
Parameter	Value	Unit
Copper Price	$3.60	US$ per pound
Silver Price	$22.00	US$ per ounce
Gold Price	$1,750.00	US$ per ounce
Molybdenum Price	$14.00	US$ per pound
In-Pit Mining Cost - Overburden	$1.75	US$ per tonne mined
In-Pit Mining Cost - Rock	$2.10	US$ per tonne mined
Underground Mining Cost	$25.00	US$ per tonne mined
Processing Cost	$18.00	US$ per tonne milled
General and Administrative	$5.00	US$ tonne of feed
Overall Pit Slope - Rock	55	Degrees
Overall Pit Slope - Overburden	35	Degrees
Oxide Recoveries		
Copper Recovery	85	Percent (%)
Silver Recovery	65	Percent (%)
Gold Recovery	85	Percent (%)
Molydenum Recovery	70	Percent (%)
Sulphide Recoveries		
Copper Recovery	90	Percent (%)
Silver Recovery	65	Percent (%)
Gold Recovery	76	Percent (%)
Molydenum Recovery	70	Percent (%)
Mining loss / Dilution (open pit)	5 / 2	Percent (%) / Percent (%)
Mining loss/Dilution (underground)	5 / 5	Percent (%) / Percent (%)
Waste Specific Gravity	2.66	
Mineral Zone Specific Gravity		
Oxide	2.64	
Sulphide	2.71 - 2.78	
Block Size	5 x 5 x 5	

Table 14-9 Carmacks Project Mineral Resource Estimates, February 25, 2022

Category	CU_T % Cut-off	Tonnes	CU_T		AG		AU		MO		CuEq	
			(%)	(Mlbs)	(g/t)	Ounces	(g/t)	Ounces	(%)	(Mlbs)	(%)	(Mlbs)
In-Pit Oxide												
Measured	0.30	11,361,000	0.96	239.32	4.11	1,501,000	0.40	145,000	0.006	1.5	1.30	324.93
Indicated	0.30	4,330,000	0.91	86.85	3.37	469,000	0.28	39,000	0.007	0.6	1.16	110.99
Measured + Indicated	**0.30**	**15,691,000**	**0.94**	**326.17**	**3.91**	**1,971,000**	**0.36**	**184,000**	**0.006**	**2.1**	**1.26**	**435.93**
Inferred	0.30	216,000	0.52	2.47	2.44	17,000	0.09	1,000	0.006	0.03	0.63	3.01
In-Pit Sulphide												
Measured	0.30	5,705,000	0.68	86.05	2.54	467,000	0.16	28,000	0.016	2.0	0.88	111.53
Indicated	0.30	13,486,000	0.72	214.32	2.83	1,226,000	0.19	82,000	0.013	4.0	0.93	277.23
Measured + Indicated	**0.30**	**19,191,000**	**0.71**	**300.37**	**2.74**	**1,693,000**	**0.18**	**110,000**	**0.014**	**6.0**	**0.92**	**387.76**
Inferred	0.30	1,675,000	0.51	18.92	2.24	120,895	0.13	7,000	0.020	0.7	0.7	25.95
Below Pit Sulphide												
Measured	0.60	26,000	0.71	0.41	2.54	2,000	0.16	132	0.010	0.0	0.88	0.51
Indicated	0.60	1,341,000	0.82	24.33	2.88	124,000	0.19	8,000	0.012	0.4	1.03	30.42
Measured + Indicated	**0.60**	**1,367,000**	**0.82**	**25.74**	**2.88**	**126,000**	**0.19**	**8,000**	**0.012**	**0.4**	**1.03**	**30.92**
Inferred	0.60	967,000	0.77	16.46	2.48	77,000	0.17	5,000	0.012	0.3	0.96	20.44

(1) The classification of the current Mineral Resource Estimates into Measured, Indicated and Inferred are consistent with current 2014 CIM Definition Standards - For Mineral Resources and Mineral Reserves.

(2) All figures are rounded to reflect the relative accuracy of the estimate.

(3) All Resources are presented undiluted and in situ, constrained by continuous 3D wireframe models, and are considered to have reasonable prospects for eventual economic extraction.

(4) Mineral resources which are not mineral reserves do not have demonstrated economic viability. An Inferred Mineral Resource has a lower level of confidence than that applying to a Measured and Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

(5) It is envisioned that parts of the Carmacks Project deposits may be mined using open pit mining methods. In-pit mineral resources are reported at a base-case cut-off grade of 0.3 % Cu_T within Whittle™ pit shells. It is envisioned that parts of the Carmacks Project deposits may be mined using low cost underground bulk mining methods. A selected base-case cut-off grade of 0.6 % Cu_T is used to determine the underground mineral resources.

(6) Cu Eq calculation is based on 100% recovery of all metals using the same metal prices used for the resource calculation.

(7) A pit slope of 55 degrees for rock and 35 degrees for overburden are used for the pit optimization.

(8) The results from the pit optimization are used solely for the purpose of testing the "reasonable prospects for economic extraction" by an open pit and do not represent an attempt to estimate mineral reserves. There are no mineral reserves on the Carmacks Property. The results are used as a guide to assist in the preparation of a Mineral Resource statement and to select an appropriate resource reporting cut-off grade.

(9) Cut-off grades are based on metal prices of \$3.60/lb Cu, \$22.00/oz Ag, \$1,750/oz Au and \$14.00/lb for Mo, processing and G&A cost of \$US23.00 per tonne milled, and variable mining costs including \$US2.10 for open pit and \$US25.00 for underground. The cut-off grades should be re-evaluated in light of future prevailing market conditions (metal prices, mining costs, processing costs etc.).

(10) *Metal recoveries used for pit optimization and calculation of base-case cut-off grades include: for oxide material 85% for copper, 65% for Ag, 85% for Au and 70% for Mo; for sulphide material, 90% for copper, 65% for Ag, 76% for Au and 70% for Mo.*

(11) *Composites of 2.0 metre used for the resource estimation procedure have been capped where appropriate. Grades for Cu (oxide, sulphide and total), Ag, Au and Mo for each deposit was interpolated into blocks by the Inverse Distance Squared (ID2) calculation method.*

(12) *Fixed specific gravity values of 2.64 for oxide material and 2.71 – 2.78 (depending on deposit) were used to estimate the Mineral Resource tonnage from block model volumes. Waste in all areas was given a fixed density of 2.66.*

(13) *The database used for the current MREs comprise data for 489 surface drill holes totaling 56,679 metres completed on the Carmacks Project area between 1970 and 2021. This includes 36 drill holes (RC and diamond) totaling 9,413 m completed by Granite Creek between the fall of 2020 the fall of 2021. Appropriate interpolation parameters were generated for each deposit based on the mineralization style and geometry.*

Table 14-10 Carmacks Project Mineral Resource Estimates, February 25, 2022: Distribution of Cu_X and Cu_S

Category	CU_T % Cut-off	Tonnes	CU_T		CU_S		CU_X	
			(%)	(Mlbs)	(%)	(Mlbs)	(%)	(Mlbs)
In-Pit Oxide								
Measured	0.30	11,361,000	0.96	239	0.18	45	0.78	194
Indicated	0.30	4,330,000	0.91	87	0.19	18	0.72	69
Measured + Indicated	**0.30**	**15,691,000**	**0.94**	**326**	**0.18**	**63**	**0.76**	**263**
Inferred	0.30	216,000	0.52	2.5	0.12	0.6	0.37	1.8
In-Pit Sulphide								
Measured	0.30	5,705,000	0.68	86	0.62	79	0.05	7
Indicated	0.30	13,486,000	0.72	214	0.68	201	0.04	13
Measured + Indicated	**0.30**	**19,191,000**	**0.71**	**300**	**0.66**	**280**	**0.05**	**20**
Inferred	0.30	1,675,000	0.51	19	0.46	17	0.05	2
Below Pit Sulphide								
Measured	0.60	26,000	0.71	0.41	0.68	0.39	0.03	0.02
Indicated	0.60	1,341,000	0.82	24	0.80	24	0.03	0.8
Measured + Indicated	**0.60**	**1,367,000**	**0.82**	**25**	**0.79**	**24**	**0.03**	**0.8**
Inferred	0.60	967,000	0.77	16	0.75	16	0.03	0.1

Figure 14-11 Isometric View Looking Northeast of the Carmacks Project Deposit Mineral Resource Block Grades and Revenue Factor 1.0 Pits

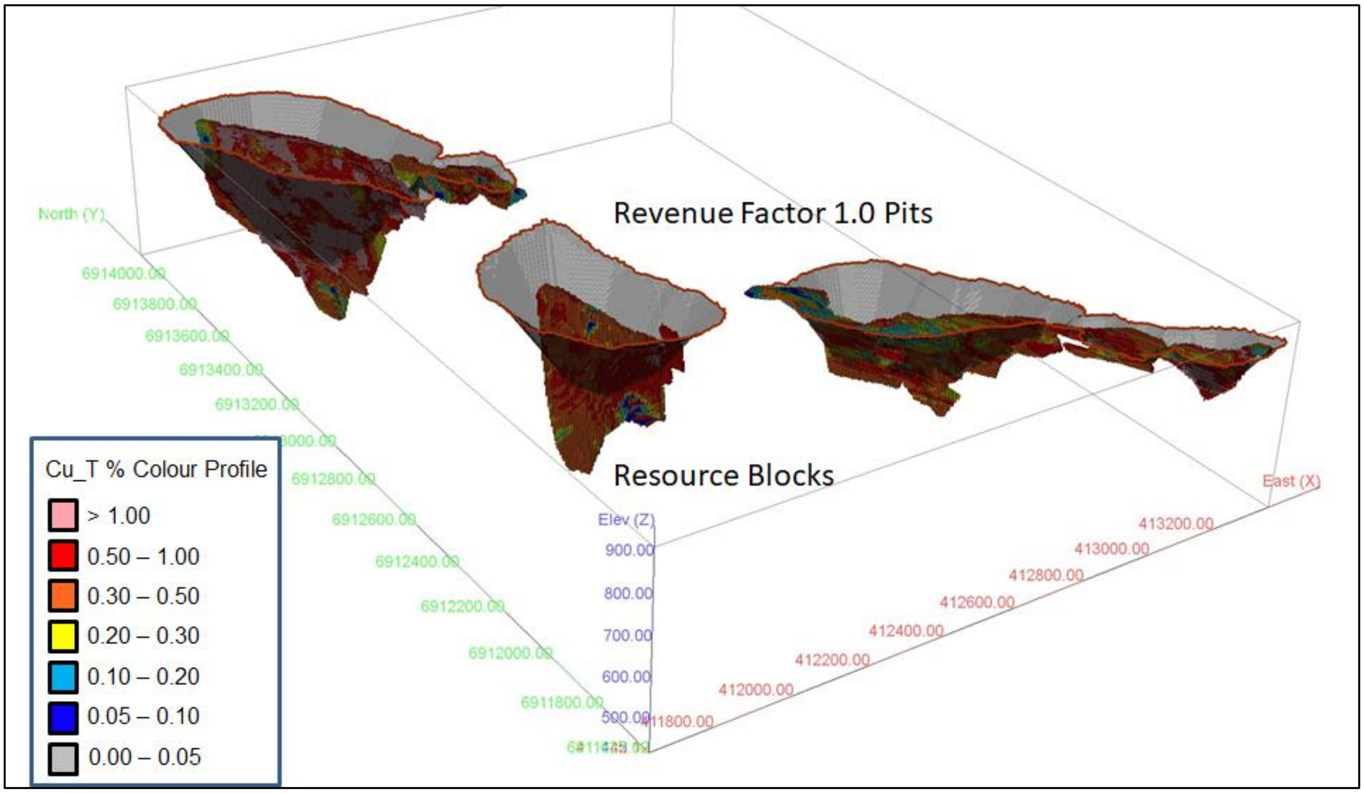
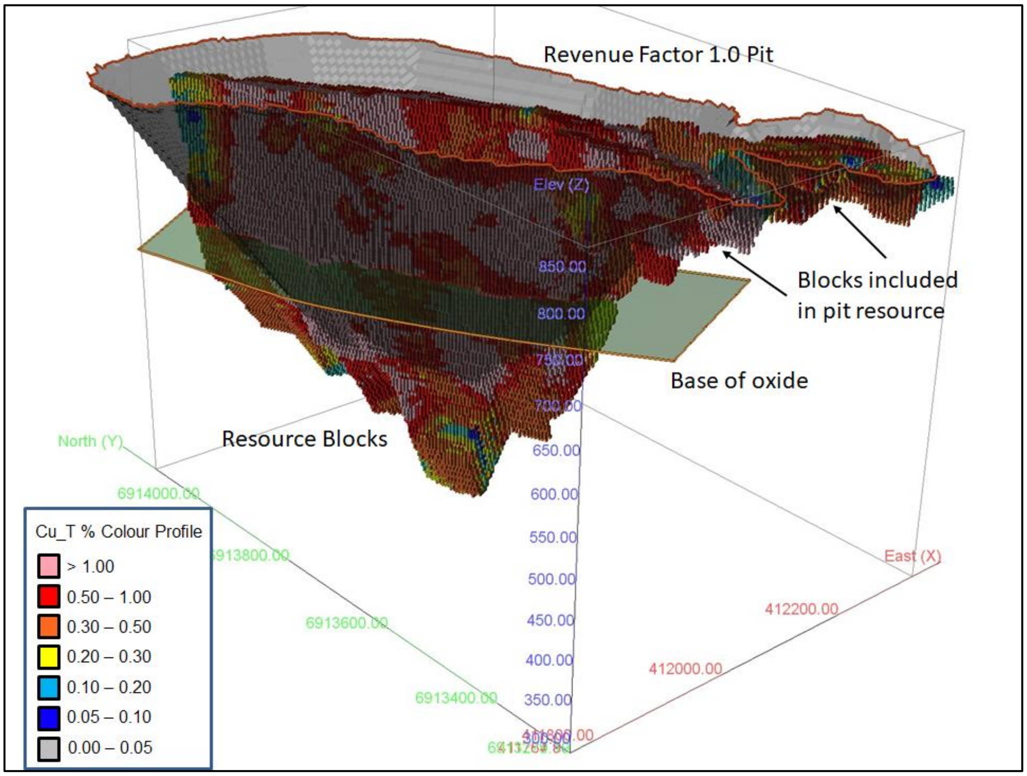


Figure 14-12 Isometric View Looking Northeast of the Zone 1, 4, 7 Deposit Mineral Resource Block Grades and Classification, and Revenue Factor 1.0 Pit



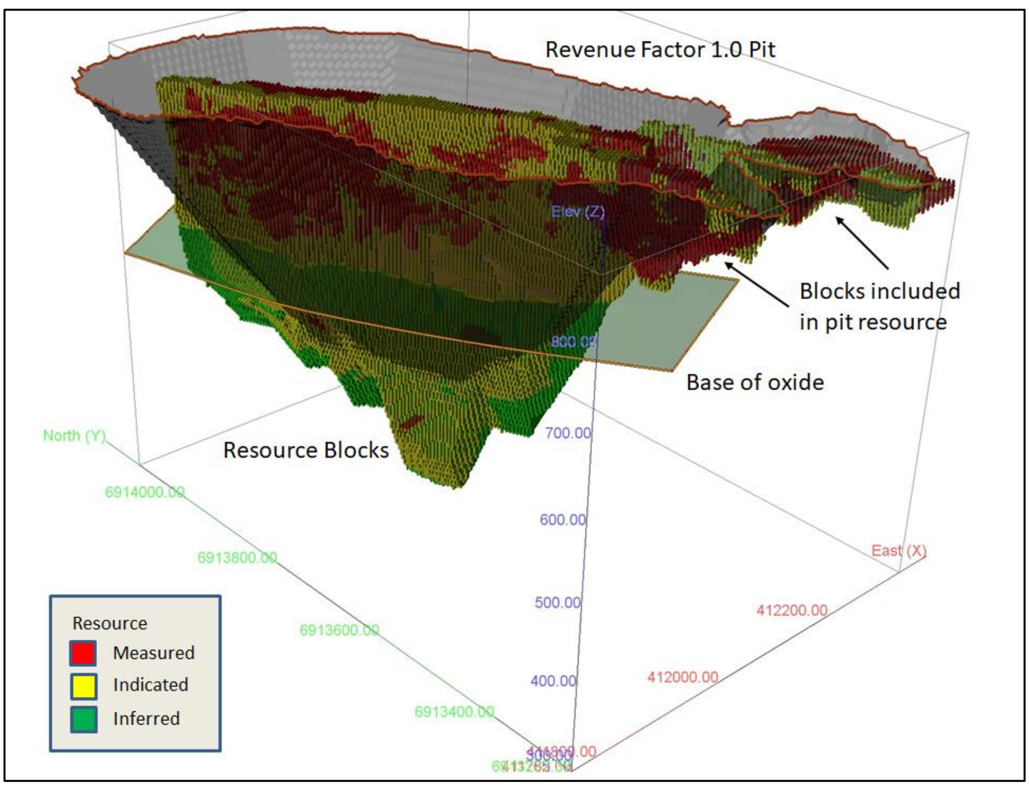
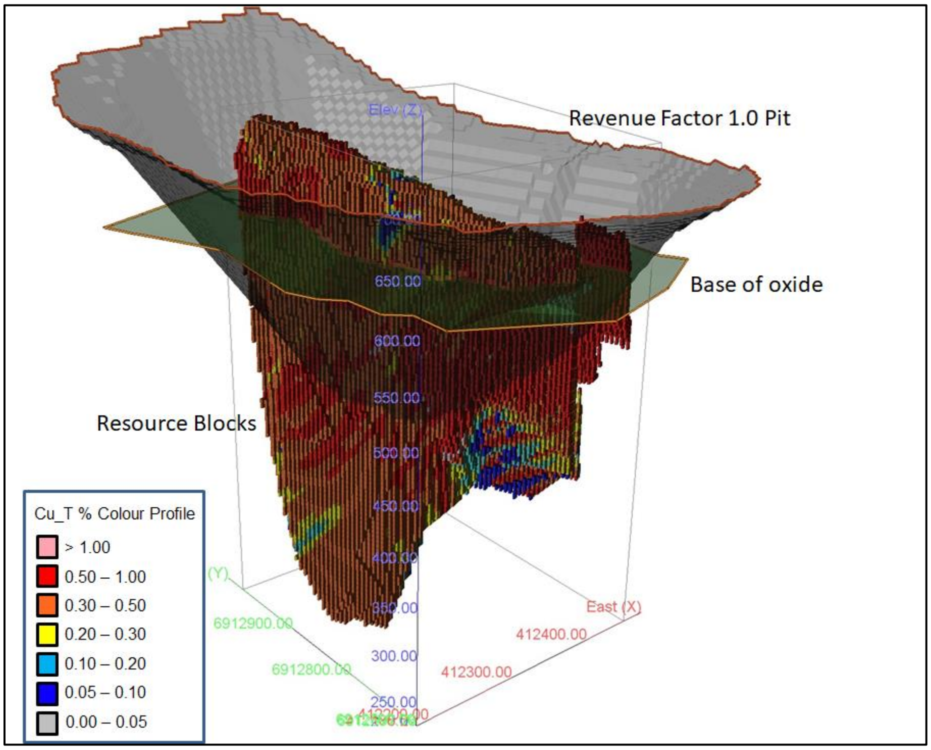


Figure 14-13 Isometric View Looking Northeast of the Zone 2000S Deposit Resource Block Grades and Classification, and Revenue Factor 1.0 Pit



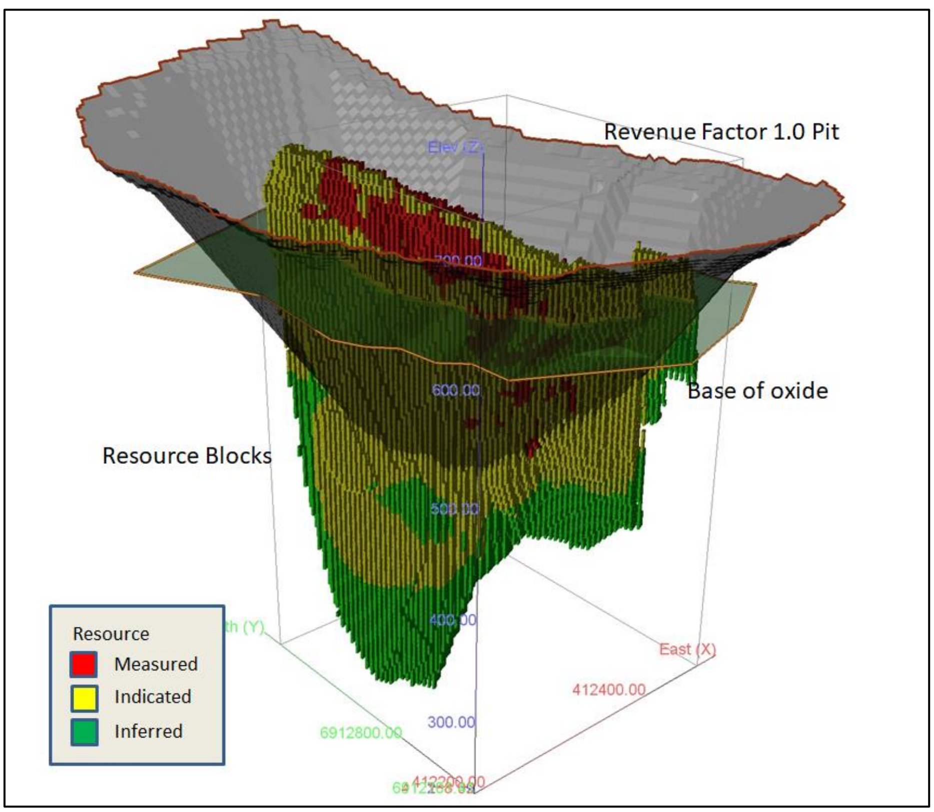
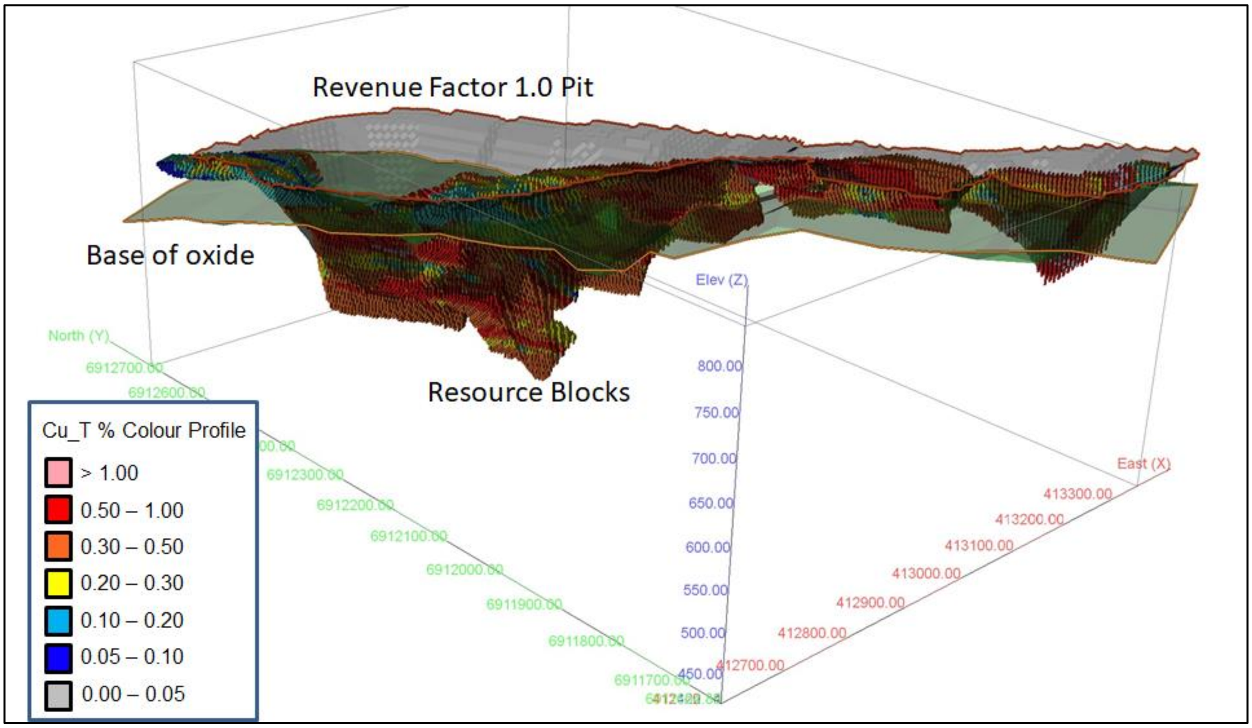
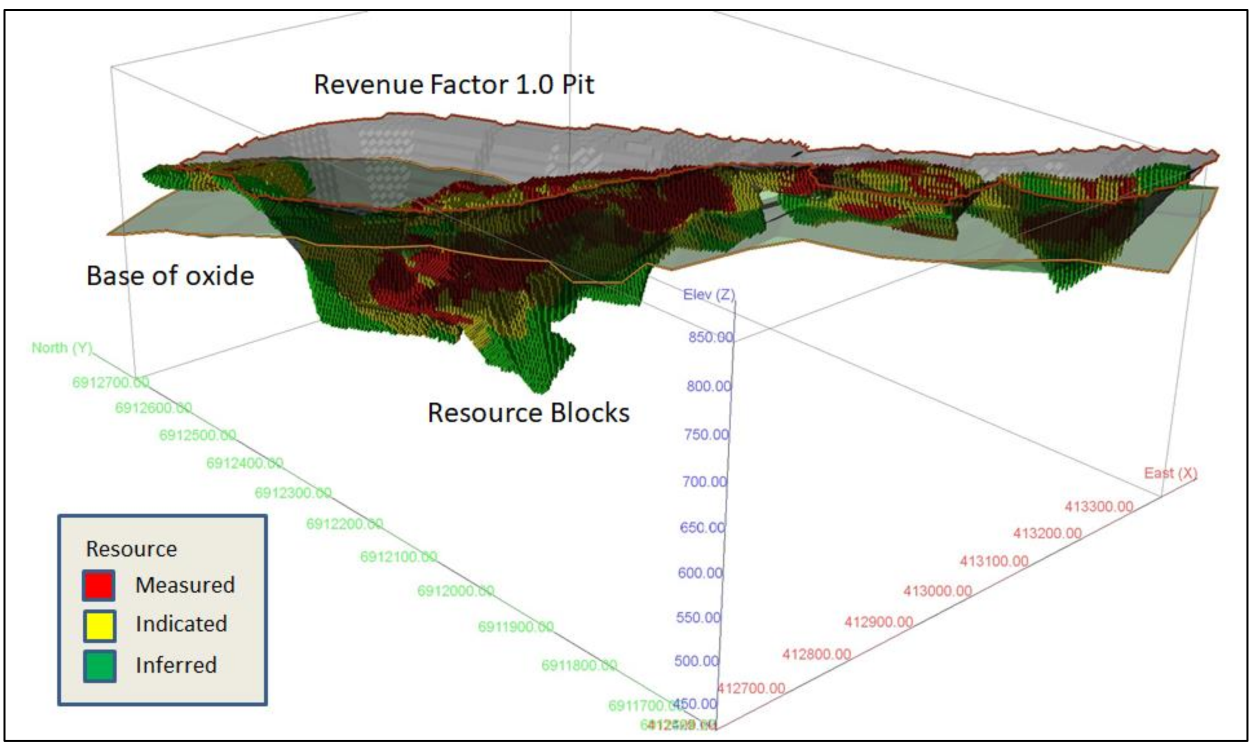


Figure 14-14 Isometric View Looking Northwest of Zones 12 and 13 Deposit Resource Block Grades and Classification, and Revenue Factor 1.0 Pit





14.11 Model Validation and Sensitivity Analysis

The total volume of the Carmacks Project deposit resource blocks in the mineral resource models at a 0.0 % Cu_T cut-off grade value (global) compared well to the total volume of the mineralized structures (Table 14-11). Differences in models vs block madels is mainly due to models being marginally larger than the search ellipse search distance. As a result, not all the wireframe models were completely populated with grade blocks.

Visual checks of block copper, gold, silver and molybdenum grades against the composite data on vertical sections showed good spatial correlation between block grades, composite grades and assay grades.

A comparison of the average composite grades for Cu_T %, Au g/t and Ag g/t with the average block grades 0.0 Cu_T % cut-off grade was completed and is presented in Table 14-12. The average grade of the block model compares well with the average grade of the capped composites used for the resource estimate. Block model grades are generally lower than the capped composites grades demonstrating a level of smoothing during the interpolation procedure.

For comparison purposes, additional grade models were generated using a varied inverse distance weighting (ID^2 or ID^3) and nearest neighbour (NN) interpolation methods. The results of these models are compared to the chosen models at various cut-off grades in a series of grade/tonnage graphs shown in Figure 14-15. In general, the ID^2 and ID^3 models show similar results and both are more conservative and smoother than the NN model. For models well-constrained by wireframes and well-sampled (close spacing of data), ID^2 should yield very similar results to other interpolation methods such as ID^3 or Ordinary Kriging.

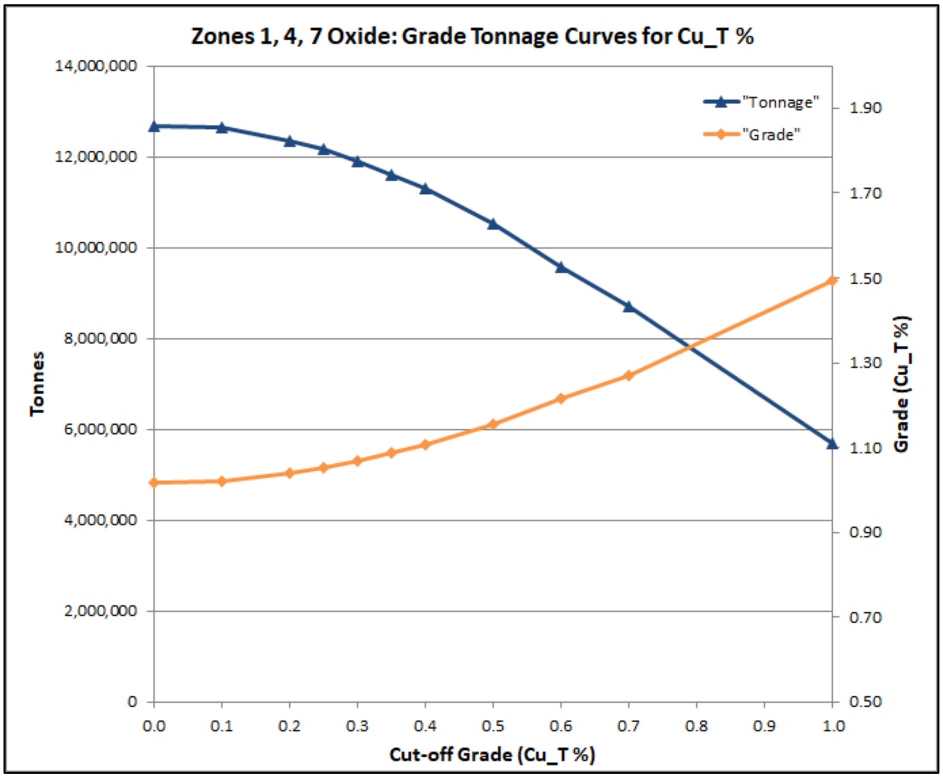
Table 14-11 Comparison of Block Model Volume with Total Volume of the Mineralized Structures

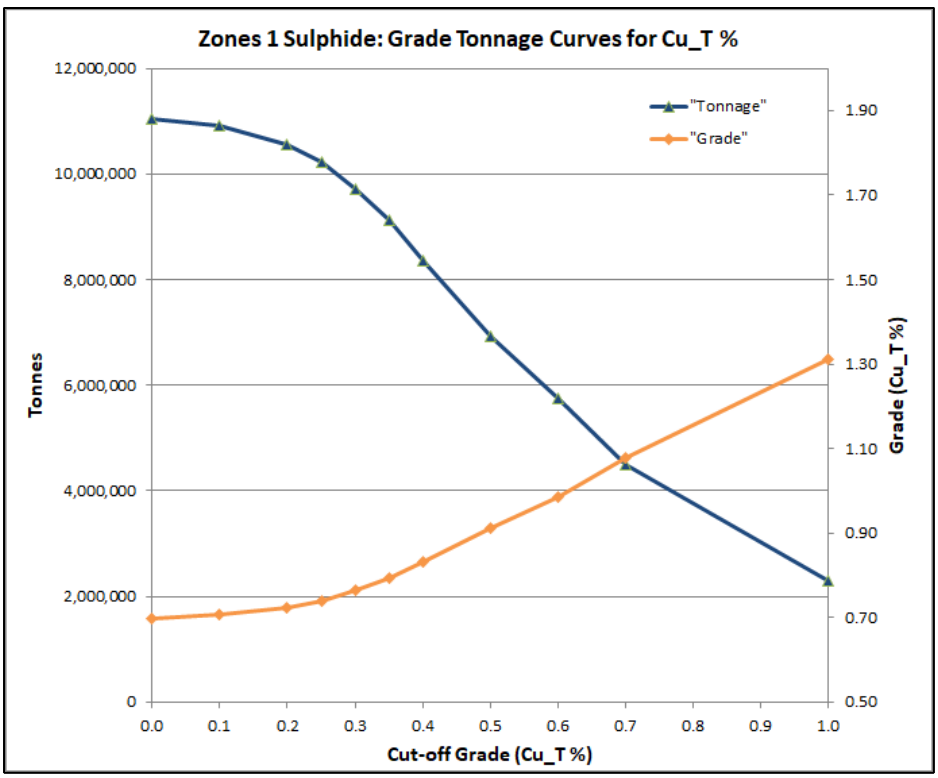
Zone	Wireframe Model Volume	Block Model Volume	Difference %
Zones 1, 4, 7 - Oxide	4,795,374	4,795,707	0.01
Zone 1 - Sulphide	3,989,397	3,971,576	0.45
Zone 2000S - Oxide	380,593	380,955	0.10
Zone 2000S - Sulphide	2,755,798	2,726,921	1.06
Zone 12 Oxide	481,101	480,835	0.06
Zone 12 Sulphide	377,834	370,860	1.88
Zone 13 Oxide	1,670,626	1,642,487	1.71
Zone 13 Sulphide	5,350,157	5,198,758	2.91
Total:	**19,800,880**	**19,568,099**	

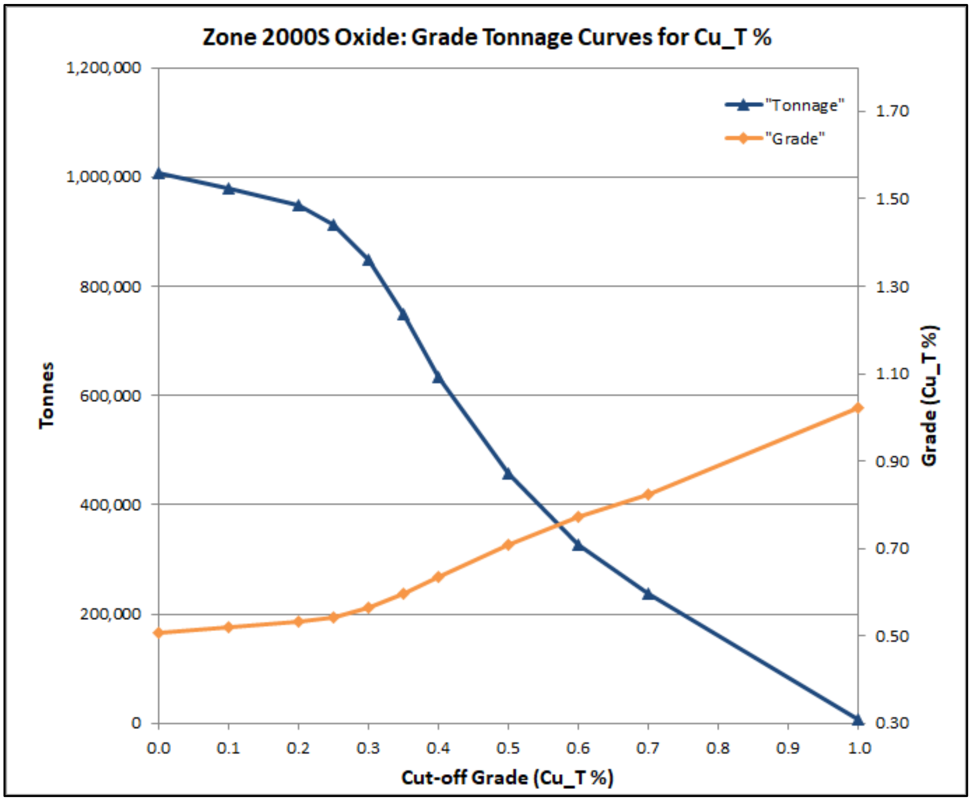
Table 14-12 Comparison of Average Composite Grades with Block Model Grades

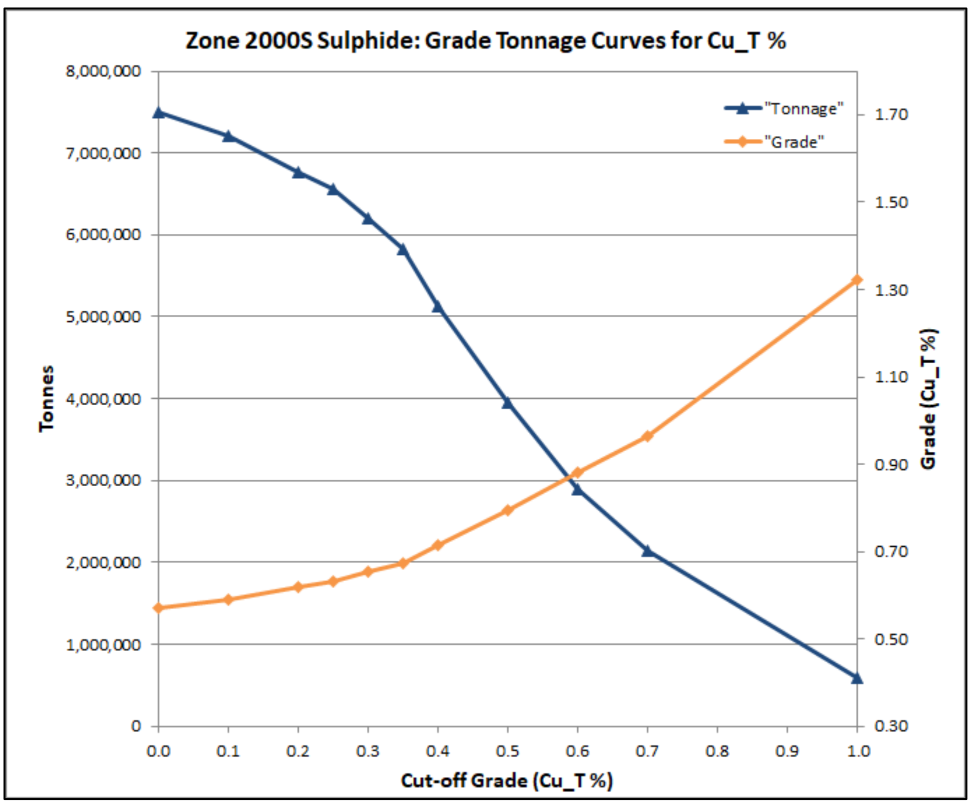
Zone	Composite Average Grade		Block Average Grade	Difference %
Zones 1, 4, 7 - Oxide	Cu_T (%)	1.04	1.02	1.96
	Au (g/t)	0.48	0.42	14.29
	Ag (g/t)	4.82	4.20	14.76
Zone 1 - Sulphide	Cu_T (%)	0.71	0.70	1.43
	Au (g/t)	0.20	0.19	5.26
	Ag (g/t)	2.60	2.40	8.33
Zone 2000S - Oxide	Cu_T (%)	0.46	0.51	-9.80
	Au (g/t)	0.14	0.16	-12.50
	Ag (g/t)	2.29	2.55	-10.20
Zone 2000S - Sulphide	Cu_T (%)	0.63	0.57	10.53
	Au (g/t)	0.15	0.13	15.38
	Ag (g/t)	5.05	2.73	84.98
Zone 12, 13 Oxide	Cu_T (%)	0.44	0.39	12.82
	Au (g/t)	0.11	0.10	10.00
	Ag (g/t)	1.76	1.62	8.64
Zone 12, 13 Sulphide	Cu_T (%)	0.52	0.51	1.96
	Au (g/t)	0.12	0.12	0.00
	Ag (g/t)	1.93	2.00	-3.50

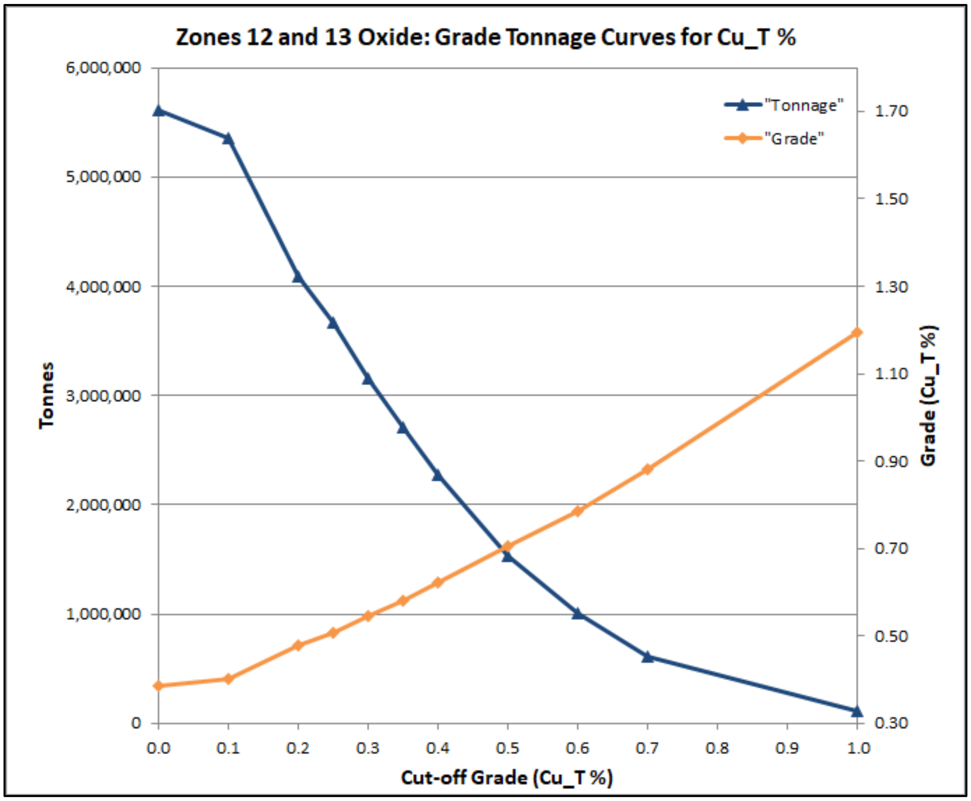
Figure 14-15 Grade Tonnage Plots to show sensitivity to cut-off for Oxide and Sulphide Mineralization

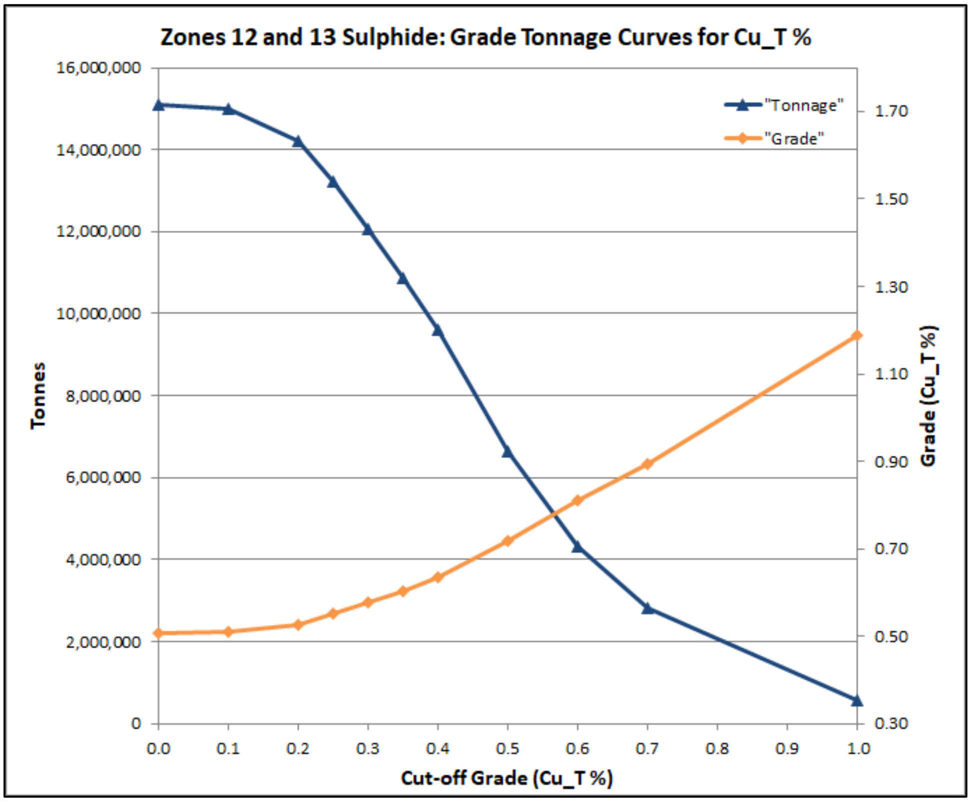





Zone 2000S Oxide: Grade Tonnage Curves for Cu_T %
"Tonnage"
"Grade"
Tonnes
Grade (Cu_T %)
Cut-off Grade (Cu_T %)
1,200,000
1,000,000
800,000
600,000
400,000
200,000
0
1.70
1.50
1.30
1.10
0.90
0.70
0.50
0.30
0.0
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9
1.0


Zone 2000S Sulphide: Grade Tonnage Curves for Cu_T %
"Tonnage"
"Grade"
Tonnes
Grade (Cu_T %)
Cut-off Grade (Cu_T %)
8,000,000
7,000,000
6,000,000
5,000,000
4,000,000
3,000,000
2,000,000
1,000,000
0
1.70
1.50
1.30
1.10
0.90
0.70
0.50
0.30
0.0
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9
1.0


Zones 12 and 13 Oxide: Grade Tonnage Curves for Cu_T %
"Tonnage"
"Grade"
Tonnes
Grade (Cu_T %)
Cut-off Grade (Cu_T %)
6,000,000
5,000,000
4,000,000
3,000,000
2,000,000
1,000,000
0
1.70
1.50
1.30
1.10
0.90
0.70
0.50
0.30
0.0
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9
1.0


Zones 12 and 13 Sulphide: Grade Tonnage Curves for Cu_T %
"Tonnage"
"Grade"
Tonnes
Grade (Cu_T %)
Cut-off Grade (Cu_T %)
16,000,000
14,000,000
12,000,000
10,000,000
8,000,000
6,000,000
4,000,000
2,000,000
0
1.70
1.50
1.30
1.10
0.90
0.70
0.50
0.30
0.0
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9
1.0

14.12 Sensitivity to Cut-off Grade

A copper and copper equivalent grade sensitivity analysis for both oxide and sulphide resources contained in the proposed pits is provided in Table 14-13 below, which demonstrates the variation in grade and tonnage in the deposit at these various cut-off grades. This sensitivity analysis is reflective of the discrete nature of the mineralized bodies. Comparing the cut-off grade of 0.30% Cu_T with a 0.25% Cu_T and a 0.35% Cu_T cut-off, show a <3% variation in contained copper and a ~5% variation in the tonnage.

Table 14-13 Carmacks Project Mineral Resource Estimate Grade Sensitivity

Category	CU_T % Cut-off	Tonnes	CU_T		AG		AU		MO		CuEq	
			(%)	(Mlbs)	(g/t)	Ounces	(g/t)	Ounces	(%)	(Mlbs)	(%)	(Mlbs)
In-Pit Oxide												
Measured + Indicated	0.25	16,459,000	0.91	330.85	3.78	2,000,000	0.35	186,000	0.006	2.19	1.22	442.18
Measured + Indicated	**0.30**	**15,691,000**	**0.94**	**326.17**	**3.91**	**1,971,000**	**0.36**	**184,000**	**0.006**	**2.15**	**1.26**	**435.93**
Measured + Indicated	0.35	14,886,000	0.98	320.41	4.04	1,931,000	0.38	181,000	0.006	2.08	1.30	428.25
In-Pit Sulphide												
Measured + Indicated	0.25	20,102,000	0.69	305.92	2.68	1,729,000	0.17	113,000	0.014	6.24	0.89	395.47
Measured + Indicated	**0.30**	**19,191,000**	**0.71**	**300.37**	**2.74**	**1,693,000**	**0.18**	**110,000**	**0.014**	**6.00**	**0.92**	**387.76**
Measured + Indicated	0.35	18,028,000	0.73	292.00	2.83	1,640,000	0.19	107,000	0.014	5.67	0.95	376.25

(1) Values in these tables are reported above and below a base case cut-off grade (highlighted) for pit constrained and underground and should not be misconstrued with a Mineral Resource Statement. The values are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade.

(2) All figures are rounded to reflect the relative accuracy of the estimate. Composites have been capped where appropriate.

14.13 Disclosure

All relevant data and information regarding the Carmacks Project Deposits is included in other sections of this Technical Report. There is no other relevant data or information available that is necessary to make the technical report understandable and not misleading.

The Author is not aware of any known mining, processing, metallurgical, environmental, infrastructure, economic, permitting, legal, title, taxation, socio-political, or marketing issues, or any other relevant factors not reported in this technical report, that could materially affect the Mineral Resource Estimate.

15 MINERAL RESERVE ESTIMATES

There are no Mineral Reserve estimates stated on this Property. This section does not apply to the Technical Report.

16 MINING METHODS

16.1 Caution to the Reader

The reader is cautioned that this PEA uses Inferred Mineral Resources. NI 43-101 Part 2, Section 2.3(1)(b) and Companion Policy 43-101 CP, Part 2, Section 2.3(1) Restricted Disclosure, prohibits the disclosure of the results of an economic analysis that includes or is based on Inferred Mineral Resources, an historical estimate, or an exploration target.

However, under NI 43-101, Part 2, Section 2.3(3) and Companion Policy 43-101CP, Part 2 Section 2.3(3), a PEA is allowed to use inferred mineral resources and to carry out an economic assessment in order to inform investors of the potential of the property. Investors must be informed that the preliminary economic assessment is preliminary in nature, that it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised. Mineral resources that are not mineral reserves do not have demonstrated economic viability. To emphasise, no consideration was given to the resource classification in the pit optimisation and mine design.

16.2 Overview

The proposed mining method is conventional open pit mining. Mineralised rock and waste would be drilled, blasted, loaded by hydraulic shovels and hydraulic excavators into off-highway dump trucks, and hauled to the processing plant.

The basis for the pit design work was the mineral resource block model that was developed by MSA as part of a NI 43-101-compliant mineral resource estimate (refer to Section 14).

There are two primary Carmacks deposits currently under consideration (Zones 147 and 1213). These are to be mined as open pits, with the normal sequence of drilling, blasting and hauling. Due to the nature of the deposit, the resultant pits are narrow and deep. Currently no backfilling is contemplated.

The target ROM feed to the processing plant is 2,450,000 tonnes/year. The plant feed is mineralised material with a copper % >= 0.3%.

The combined Life of Mine of the two pits is 9 years and note that all prices and costs are in $CAN.

16.3 Geotechnical Evaluation

Geotechnical or hydrological studies have been undertaken but are limited to Zone 147. The geotechnical reports used for the geotechnical parameters in design of the pits are the following:

1. Open Pit Slope Design Carmacks Copper Project Dated October 22, 2008
2. Report on Pit Slope Stability (REF. No. 1782/3) dated January 1993 and
3. Report on 1996 Geotechnical and Hydrogeological Site Investigations (REF No. 1784/1).

An assumed maximum pit slope angle (IRA) of 52.6° is applied to all slopes other than the low wall side, which follows the orebody and is at approximately 50°.

With the inclusion of the ramp system the overall highwall slope in Pit 147 is approximately 55°. For Zone1213, the pit design assumptions were derived from the zone 147 geotechnical parameters.

The pit design parameters are shown in Table 16-1.

Table 16-1 Pit Design Parameters

Pit Design Parameters	
Bench Height	10 m
Face Batter Angle	70°
Berm Width	4 m
Ramp Width	m
Single	12.11-15.6
Double	16.3-21.3
Ramp Gradient	1:10

16.3.1 Geotechnical Summary

The proposed 147 open pit will have an elongated northwest-southeast configuration about 983 metres long and 458 metres wide with a total surface area of 34.2 hectares. The ultimate pit floor will be located at the 630-metre elevation. The west wall is planned to be the highest wall of this pit, with a maximum height of approximately 270 metres. The crest of the west wall will be located at about the 900-metre elevation. The east wall will have a maximum height of about 217 metres, with the crest located at about the 820-metre elevation. The ultimate pit floor will be accessed by haul road ramp located on the east wall. The proposed 1213 pit is located 818.34 meters from the 147 pit. 1213 pit is composed of the north and south pits covering an area of 25 hectares. The final elevations when completely mined will be 560 m at the north pit while the south pit at elevation 700 m. The proposed operating life of the open pits is approximately 9 years.

The main mineralized zone, ore Zone 1, is a relatively narrow tabular zone of gneissic rocks striking northwest-southeast to an overall azimuth of about 336 degrees and dipping about 70 degrees to 80 degrees to the northeast, approximately 30 meter wide. The proposed pit walls are anticipated to expose mostly granodiorite rocks.

Geotechnical and hydrogeological investigation programs were carried out as part of the present assessment. In this report, the proposed 147 open pit plan and the geology of pit area are described. The existing data that has been reviewed and, the field investigations and laboratory testing that have been carried out as part of this assessment are described. The results of the geotechnical and hydrogeological site investigation and laboratory testing are discussed. The engineering geology of the proposed 147 open pit is discussed. The assessment of pit slope stability aspects is presented and discussed. The stability analyses of the proposed pit walls are presented, and the results discussed. Finally, recommendations are provided regarding the pit wall design of the proposed147 open pit.

The geotechnical investigation program comprised mostly of investigation of the granodiorite rocks that will be exposed on the two main pit walls, i.e. west (footwall zone) and east (hanging wall zone) walls. The site investigation program was carried out from April 2 to May 24, 2008 and included the following components.

Drilling of six inclined NTW size diamond coring geotechnical holes (denoted GA-01 to GA-06) with core orientation using the EZY-MARK system.

Geotechnical core logging including description and orientation of discontinuities.

Groundwater level measurements and hydraulic conductivity testing (packer testing) at varying intervals.

Point Load Testing (PLT) of intact rock core samples.

Core sample collection and laboratory rock strength testing including Uniaxial Compressive Strength testing (UCS) of intact rock and Direct Shear testing of natural discontinuities.

Mineralization within the proposed pit is hosted by the gneissic rocks occurring along the central portion of the pit. The current interpretation of ore zones has been provided by the company as “envelopes” of ore Zones 1, 4 and 7, and are interpreted to be mostly representative of gneissic rock occurrence within the proposed pit area. At the current stage, the limits of gneissic rocks have not been interpreted geologically, and those envelops represent the best approximation of the occurrence of gneissic rocks within the pit. Therefore, for the purpose of the present geotechnical slope stability assessment has considered the following main rock type domains.

Hanging Wall, i.e. east wall: granodiorite of the Granite Mountain Batholith.

Footwall Wall, i.e. west wall: granodiorite of the Granite Mountain Batholith.

End Walls, i.e. northwest and southeast walls: mostly granodiorite of the Granite Mountain Batholith, and to a lesser degree some possible extension of feldspathic- mafic gneisses of the “gneissic ore zones” intersecting the walls at both ends of the proposed pit.

In general terms, the regional structural setting is mostly described to be poorly understood as a result of limited outcrop exposures. Limited information on the regional geological features is provided on the available documentation of previous works carried out in the area of the Carmacks Copper Project. A few geological structures as faults and probable faults are interpreted to occur within the planned pit vicinity, as presented in previous geological reports. However, the understanding on the nature and dip orientation of these faults is uncertain at the current stage. Consequently, the impact of these faults to the stability of the proposed pit walls is uncertain.

The review of geotechnical core holes indicated that major wide fault zones were not intersected. Faults that were intersected appear to be limited to short intervals of drilled core. The majority of structures described as faults in geotechnical core logging of the core holes GA-01 to GA-06, appear to be thin, discrete features and are likely minor faults or sheared joints.

An assessment of structural data from the geotechnical core holes was carried out to estimate the major and minor discontinuity sets that are anticipated to represent the structural fabric of the rock masses to be exposed on the proposed pit walls.

The understanding of the rock fabric in terms of discontinuity sets is an important step in the stability assessment of the proposed pit walls for structurally controlled failure mechanisms. Structurally controlled failure mechanisms in rock occur as the result of sliding along pre-existing geologic discontinuities. The three basic mechanisms of structurally controlled failure in rock slopes are plane failures, wedge failures, and toppling failures.

The geologic structures observed at the Carmacks Project and anticipated to influence the slope stability of the proposed the pit walls, are faults and joints sets. Pervasive foliation has not been observed along the geotechnical core holes.

Review of structural data from core holes has indicated a relative complex data collection in terms of separation of major structural trends. The complexity of the structural data can be considered in the context of the development of natural discontinuities at the Carmacks Project area. The development of primary structures in igneous rocks can usually be related to the mode of emplacement of an intrusive mass, such as the Granite Mountain Batholith. In addition to primary structures and cooling joints, discontinuities in the Carmacks area are also expected to have resulted from regional tectonic deformation. Finally, joints can also develop as the result of uplift and unloading of the igneous rocks, as well “sheet-jointing” related to relief development.

Consequently, it is anticipated that discontinuities resulting of different mechanisms, but most likely related to the emplacement of the igneous rocks and regional tectonic deformation, occur within the rock masses at the Carmacks open pit area. However, at the current stage, it is not feasible to distinguish those features in terms of genetic mechanisms. Furthermore, the limited information on the regional geological features constrains the identification of discontinuity sets on the basis of tectonic deformation pattern.

The definition of structural sets has been based essentially on the frequency analysis of discontinuity orientation, and discontinuity sets are interpreted accordingly to hanging wall and footwall zones. In addition, the orientation of the interpreted structural sets has also been compared to the faults and probable fault trends of Aurora (2007). Some structural sets of the footwall and hanging wall zones were found to be parallel to the alignments presented by Aurora (2007). However, absence of information on the dip orientation of those faults and probable faults has limited further interpretation at this stage.

Finally, the geotechnical core hole orientation data has been compared with previous existing data presented by Knight Piésold (1993), which included data collected from one oriented core hole (92-158) and data gathered from geological trenching programs. The previous existing data appears to represent mostly the structural fabric within the gneissic ore zone (Zone 1), and therefore it is possibly dominated by steep northeast dipping gneissic foliation. Consequently, the previous data collection has limited representation of the granodiorite rocks anticipated to dominate the proposed pit walls on both footwall and hanging wall zones.

The strength of the intact rock of the granodiorite rocks has been estimated based on the assessment of Rock Hardness as part of the core logging, the results of Point Load Testing and Unconfined Compressive Strength (UCS) laboratory testing. The results of intact rock strength of the granodiorite rocks yielded average estimates ranging between 60 MPa to 70 MPa, therefore mostly R4 – strong rock (ISRM –intact rock strength classification).

Review of the intact rock strength results per rock mass zones, i.e. hanging wall and footwall zones, did not indicate significant differences between these two zones in terms of intact rock strength based on the tested rock core samples.

In general, the results of intact rock strength indicate that the tested rocks might exhibit some weak anisotropy. However, for the purpose of the present assessment, it appears reasonable to assume that on average those rocks are close to isotropic in terms of intact rock. This understanding is consistent with the core logging data indicating that those rock masses do not exhibit strong development of tectonic foliation or schistosity, which in general results on stronger anisotropy.
The results of intact rock strength from the present assessment were compared to the testing results that were carried out previously in 2006 on rock core samples collected by WCC of exploration core holes. Results of the 2006 UCS testing program yielded intact rock strength of granodiorite rocks that are significantly lower than the results of the

testing program carried out as part of the present assessment. The discrepancy in results has been discussed in this report, and the review of the 2006 UCS testing samples indicate that these results might not be representative of the intact rock strength of the granodiorite rocks.

An assessment of the overall quality of the rock masses that will comprise the pit walls of the Carmacks Copper open pit has been prepared using the RMR (Bieniawski, 1976) rock mass classification system. The characterization of rock mass quality yielded from geotechnical logging of core holes GA-01 to GA-06 resulted in the following suggested rock mass quality zoning.

Granodiorite rocks in the footwall and hanging wall zones closer to surface, i.e. above about 40 to 45 metres depth within the footwall zone and about 45 to 80 metres depth within the hanging wall zone, mostly Fair Rock mass quality with higher degree of fracturing closer to surface.

Granodiorite rocks deeper in the footwall and hanging wall zones, mostly Good Rock mass quality with some alternating intervals of Fair Rock.

Gneissic rocks within the ore zone likely to exhibit lower rock mass quality than observed on the granodiorite rocks. It is anticipated that rock mass quality within the narrow gneissic rocks could range within Poor and Fair Rock mass quality. Note that these rocks have a strong gneissic-texture controlled discontinuity set.

In general, it is expected that most faults within the granodiorite rocks might not have an extensive deleterious impact on the rock mass quality other than some lower rock mass quality limited to localized intervals. It is also anticipated that some zones with higher degree of fracturing might occur at depth within the granodiorite rocks in both footwall and hanging wall zones and yield more continuous interval zones of Fair Rock mass quality.

It is possible that faulting within the gneissic rocks might result on further lower rock mass quality zones within the ore zone, and eventually exposed at the proposed end walls. However, it is also expected that the gneissic rocks zones are relatively narrow.

A comparative review was carried out of the data from the geotechnical core holes drilled as part of the present assessment and geotechnical data provided by WCC from previous exploration core holes. In summary, the comparison between geotechnical data collected from exploration and geotechnical core holes, has shown that data from the geotechnical investigation yielded overall better rock quality in terms of degree of fracturing and intact rock strength than was anticipated by the exploration data.

Based on the results of the geotechnical investigation program, the main consideration for rock slope failure mechanisms at the Carmacks Copper Project open pit are structural controlled mechanisms (kinematics) at either a small scale (i.e., benches), or at a larger scale (i.e. slope failure along persistent shear or fault). Kinematic stability analyses were carried out to address these potential mechanisms. The structural interpretation of discontinuity sets within the footwall and hanging wall zones has been used to support the kinematic stability assessment of the proposed pit walls.

In addition, as a result of the assessment of rock mass quality at the Carmacks Copper Project open pit based on the data from the geotechnical investigation, circular failure due to rock mass strength failure is not anticipated as a critical mechanism for the rock masses encountered in the geotechnical investigation. Overall rock mass stability analyses were carried out for confirmation purposes.

Regarding the hydrogeology of the pit area, measurements of groundwater levels taken routinely prior to the start of the hydrogeologic testing of core holes GA-01 to GA-06 indicated that groundwater levels were observed to be relatively consistent and deep, i.e. vertical depth ranging from about 70 to 155 metres. The groundwater levels appear to be generally at similar elevation in the west and east zones to the ore zone. The results of the packer testing have yielded mostly low values of hydraulic conductivity coefficient ranging within 10-8 to 10-9 m/s. Based on these results, it appears that the majority of the rock mass within the proposed open pit is 'tight' showing low hydraulic conductivity values.

For the purpose of the overall rock mass stability analyses, the pit walls were conservatively assumed fully saturated below the interpreted water level elevations. A water level at about 725 metre elevation was assumed for both hanging wall and footwall zones based on the results of the hydrogeology investigation. However, this stability analysis scenario is somewhat conservative, and not expected to represent the groundwater conditions that will exist in the pit walls. As the rock mass is fractured, it is anticipated that groundwater flow is likely to occur toward the excavated pit walls. Consequently, it is expected that portions of the rock mass could be “naturally dewatered” during the excavation, and therefore resulting in partial saturated conditions with a “dry zone” close to the pit walls.

Design Issues

The results of the overall rock mass stability analyses indicate that the proposed design pit walls of the 147 pit are expected to exhibit adequate stability conditions with respect to circular type overall rock mass failure mechanisms.

Rather, open pit slope design is anticipated to be controlled by the structural fabric of the granodiorite at the bench scale and multi-bench scale. Further, the need to control potential for rockfall hazard to personnel and equipment must also be addressed, due to the anticipated blocky nature, particularly on near-surface benches, but also expected elsewhere in the pit.

Slope designs were based on kinematic assessments, discussed below. Rock fall hazard control, particularly on walls with no planned ramps such as the footwall, can be enhanced by the placement of extra-wide berms (geotechnical berms) at strategic intervals along the slope height, effectively breaking the slope. This is also discussed below. Overall, kinematic assessment suggested that planar and wedge failures may occur, and that slope design will be controlled by the potential for kinematic instability.

Kinematic Assessment

Structural populations from the oriented core were divided into two domains, footwall and hanging wall. Peak orientations of the discontinuity populations, and sub-peaks, were selected from the lower hemisphere equal area projections (stereonets).

Slope designs were selected by first performing a deterministic review of the potential kinematic controls on slope design, using the peak orientation determined from oriented core. This was compared to statistical assessments of potential wedges and planar failures using all data, with consideration weighted towards the 30% percent occurrence point on the cumulative frequency curves, selected based on engineering judgment.

For the statistical kinematic analyses, the discontinuity data of the geotechnical core holes has been grouped into sets. For this analysis, the shallower dipping structures, i.e. dipping less than 25 degrees, were not included in the analyses data, since those shallow dipping structures do not appear to be a concern in terms of kinematic stability, given the joint surface characteristics and the shear strengths determined from laboratory testing.

For the purpose of the kinematic analyses, the following design shear strength parameters were adopted:

- Friction Angle – 35º
- Cohesion – 0 KPa

The results of laboratory testing of natural discontinuities carried out as part of the present assessment have yielded friction angles of 35º and 45º for cohesionless residual and peak shear strength, respectively. For comparison, the shear strength of faults in similar geotechnical setting could yield friction angle in the range of 20º to 30º, while friction angle of joints could be in the range of 30º to 50º.

Geotechnical core hole logging has indicated several discontinuities with slickensided surfaces. Review of the core photographs indicated that the majority of those features might likely be sheared joints, and therefore not major faults. Only a few possible faults were identified in the review of the geotechnical core hole logging data. Therefore, the lower bound friction angle of 35º has been adopted for the kinematic stability analyses, which appears to be a reasonable assumption for the more continuous discontinuities features possibly represented by those “sheared” joints. Locally, planes and wedges with lower strength than cohesion of 0 KPa and friction angle of 35 degrees may occur.

In addition to kinematic analyses with above assumptions, sensitivity analyses of the two main pit wall orientations, i.e. design sectors 1 and 2, were carried out adopting a higher friction angle of 40º for the natural discontinuities.

Results of the deterministic and statistic kinematic analyses as well as the results of the sensitivity analyses of design sectors 1 and 2 with friction angle of 40º are summarized in the Golder report ‘’Preliminary Design Waste Rock Storage Area Carmacks Copper Project Western Copper Corporation Yukon’’.

The results of the statistical kinematic analyses have indicated the following.

High percent occurrence of wedge and planar failures with F.O.S. less than 1.0 undercut by the proposed BFA. Note that many of these have dips or plunges between 35 degrees and 45 degrees.

The wedge analyses of the different design sectors indicated that with a BFA of 70º, about 13% to 17% of the total number of kinematically admissible wedges could potentially fail.

On the highwall or footwall of the proposed open pit, the BFA of 70 degrees allows control of potential wedge combinations with peak plunges in the mid to high sixty-degree range.

The planar analyses of the different design sectors indicated that with a BFA of 70º, about 28% to 46% of the total number of kinematically admissible planes could potentially fail.

For the design sector with the proposed IRA of 52.6º, the kinematic analyses indicated that about 10% of the total number of kinematically admissible wedges in those sectors could potentially fail, i.e. F.O.S. less than 1.0 and the 52.6º inter-ramp slope could potentially undercut those wedges. The analyses also indicated that this inter-ramp angle controls all but about 21% to 37% of the total number of kinematically admissible planes in the same design sectors.

For the design sector with the proposed IRA of 55º, the kinematic analyses indicated that about 7% to 10% of the total number of kinematically admissible wedges in those sectors could potentially fail, i.e. F.O.S. less than 1.0 and the 55º inter-ramp slope could potentially undercut those wedges. The analyses also indicated that this inter- ramp angle controls all but about 15% to 32% of the total number of kinematically admissible planes in the same design sectors.

Adopting an IRA of 52.6º for the design sectors originally proposed as IRA of 55º, the kinematic analyses indicated that about 6% to 8% of the total number of kinematically admissible wedges in those sectors could potentially fail, i.e. F.O.S. less than 1.0 and the 52.6º inter-ramp slope could potentially undercut those wedges. The analyses also indicated that this inter-ramp angle controls all but about 13% to 30% of the total number of kinematically admissible planes in the same design sectors.

In general, the combined structural assessments indicate that in order to achieve adequate stability conditions and catchment on the proposed pit walls, somewhat shallower slopes would be required in most of design sectors in comparison to the currently proposed designs for the Carmacks pit. Based solely on the most limiting results of the statistical kinematic assessment of potential wedges and planar failures using all data, with preference to the 30% cumulative frequency of undercutting wedges and planes with FOS of less than 1.0, it would require adopting slope angles mostly in the range of 42º to 48º depending on the design sector, as presented in the Golder report ''Preliminary Design Waste Rock Storage Area Carmacks Copper Project Western Copper Corporation Yukon''.

The results of kinematic analyses of design sectors 1 and 2 with a friction angle of 40º yielded slope angles of 49º and 53º respectively, with reference to the 30% cumulative frequency of undercutting wedges and planes with FOS of less than 1.0, as presented in the Golder report ''Preliminary Design Waste Rock Storage Area Carmacks Copper Project Western Copper Corporation Yukon''. A significant reduction of the frequency of undercutting failure structures has been demonstrated. These results could be argued as a better representation of the less continuous structures with rougher surfaces. At the bench or inter-ramp scale, first order waviness can be expected to add 5 degrees or more to the typical discontinuity.

In addition, it is important to emphasize that the kinematic analysis does not take in account the actual spatial occurrence of the geologic structures. It does not consider persistence, spacing and if actually those structures will occur together to combine forming any potential wedges. Consequently, the use of kinematic analysis results must be considered in the context of those limitations.

Based on the above results, the following aspects have been considered in order to provide the final recommendations for pit wall configuration.

BFA steeper than the proposed 70º bench faces are not likely to be feasible on the main footwall and highwall orientations, as will increase the risk of bench scale instability and loss of catchment berms. Steeper bench face angles on a double bench configuration will not be an option.

In order to significantly reduce the likelihood of undercutting wedges and planar failures at bench scale, significantly flatter BFA would be required. However, modern mining operations with large scale excavation equipment constrain the ability of successfully excavate flatter bench slopes, and in practice slopes are excavated at higher BFA and will eventually break back to flatter bench slopes.

Therefore, it is considered that the proposed BFA of 70º be maintained for the design of the Carmacks open pit, and the inter-ramp slope configuration adjusted by increasing the catch-bench width as required to account for the eventual loss of catchment due to kinematic controls.

It is recommended that a minimum bench width of 8 metres to be adopted for the design of the Carmacks open pit, which in conjunction with the 70º BFA results in a maximum IRA of 52.6º. This slope configuration is recommended as the default slope design in kinematically favourable rock slopes.

This slope configuration is modified where it appears that slopes would break-back excessively due to kinematic controls on potentially continuous structures that may adversely affect the slope at the multiple bench or inter-ramp scale.

The generalised rock mass model based on the rock mass classification assessment indicated that granodiorite rocks in the footwall and hanging wall zones closer to surface, i.e. above about 40 to 45 metres depth within the footwall zone and about 45 to 80 metres depth within the hanging wall zone, could be expected to exhibit higher degree of fracturing closer to surface. Therefore, it is anticipated that a higher degree of ravelling can be expected from the upper bench slopes of the Carmacks open pit.

Similarly, it is also anticipated that gneissic rocks within the ore zone will likely exhibit lower rock mass quality than observed on the granodiorite rocks. Therefore, it is also anticipated that a higher degree of ravelling can be expected from endwalls of the Carmacks open pit intersecting the gneissic rocks.

For these reasons and the likelihood of excessive undercutting of planes and wedges at bench scale, it is possible that intense ravelling might result from the excavation of the pit walls of the Carmacks open pit. Since double-bench configuration and relatively steep walls are been recommended, it is possible that ravelling and rock-fall hazards might be critical as the catchment bench will become partially filled with debris and the pit wall height increases as mining progress. Therefore, it is also recommended to provide additional catchment-benches at certain elevations to provide additional safety catchment width at the west wall and north end wall, i.e. design sectors 1, 3 and 4.

The recommendations for the pit wall design of the proposed 147 open pit are summarized in Table 16-2.

Assumption was carried out with the open pit design for 1213 following the parameters that were established for the 147 pit. It is recommended that on the next phase of study, a geotechnical assessment shall be carried out for the proposed 1213 pit.

Table 16-2 Summary of Design Sectors and Pit Wall Design Recommendations

Design Sector	Pit Wall	Principal Wall Dip Direction	Pit Wall Design Azimuth	Bench face Angle (BFA)	Bench Height	Bench Width	Inter-ramp Angle (IRA)	12-metre wide Catchment Benches
Units		degrees	degrees	degrees	meters	meters	degrees	meters
	West side of the pit.							840
1	Footwall Zone.	055º	235º	70º	20	8	52.6º	700 & 760
2	East side of the pit. Hanging wall Zone.	235º	055º	70º	20	8	52.6º	None
3	North end of the pit.	105º	285º	70º	20	8	52.6º	700 & 760
4	North end of the pit.	180º	360º	70º	20	8	52.6º	760
5	South end of the pit.	280º	100º	70º	20	8	52.6º	None
6	South end of the pit.	330º	150º	70º	20	8	52.6º	None

16.4 Hydrogeological Evaluation

No hydrogeological inputs are available currently.

16.5 Open Pit Optimisation

SGS Geological Services had undertaken pit optimization calculations for the PEA using the mineral resource block models prepared by the QP from Section 14.

16.5.1 Optimization Parameters

The optimization parameters used for the open are summarised in Table 14-8. The parameters were used for Whittle™ Pit Optimization and to estimate the open pit and underground base case cut-off grades for the Carmacks Project MREs. For the section of this report, only the open pit optimized pit shell results were used for the design. Completion of the update MRE's for the Carmacks Project involved the assessment of a drill hole database, which included all data for surface drilling completed through the end of 2021, as well as three-dimensional (3D) mineral resource models, and available written reports.

16.5.2 Geological Block Model Input to Whittle

Variably blocked geological models were provided by the SGS Geological Services as inputs. These are:

- BM Zone 147 Combined.csv
- BM Zone 1213 Combined.csv
- BM Zone 2000S Combined.csv

16.5.3 Block Model Parameters

The Carmacks Project deposit wireframe grade-controlled models are used to constrain composite values chosen for interpolation, and the mineral blocks reported in the estimate of the mineral resource. Block sizes were selected based on drillhole spacing, composite assay length, the geometry of the mineralized structures, and the selected starting mining method (open pit and underground). At the scale of the Carmacks Project Deposits this provides a reasonable block size for discerning grade distribution, while still being large enough not to mislead when looking at higher cut-off grade distribution within the model. The model was intersected with a topographic surface models and overburden surface models to exclude blocks, or portions of blocks, that extend above theses surfaces. The block size that was used for each block model is 5 m X 5 m X 5 m. The Carmacks deposits block model geometry is provided in Table 14-6.

16.5.4 Pit Shell Selection

In order to determine the quantities of material offering "reasonable prospects for eventual economic extraction" by an open pit, Whittle™ pit optimization software and reasonable mining and processing assumptions to evaluate the proportions of the block model that could be "reasonably expected" to be mined from an open pit are used. The pit optimization for the Carmacks Project was completed by SGS for the current MREs and the pit optimization parameters used are summarized in Table 14-8. Whittle pit shells at a revenue factor of 1.0 (i.e. 100 % of base case metal prices) were selected as the ultimate pit shells for the purposes of reporting the Carmacks Project MREs. A selected base case cut-off grade of 0.30 % Cu_T is used to determine the in-pit MRE for the Carmacks Project deposits. The pit optimization tended to take in > 90% of the oxide material from all deposits.

16.5.4.1 Optimization Results

The pit-by-pit graph for the base case optimization for zone 147 and 1213 is shown in Figure 16-1 and Figure 16-2 respectively below, and the full Whittle Phase results are tabulated in Table 16-3 and Table 16-4.

Figure 16-1 Zone 147 Pit by Pit Phase Graph for Base Case Optimization

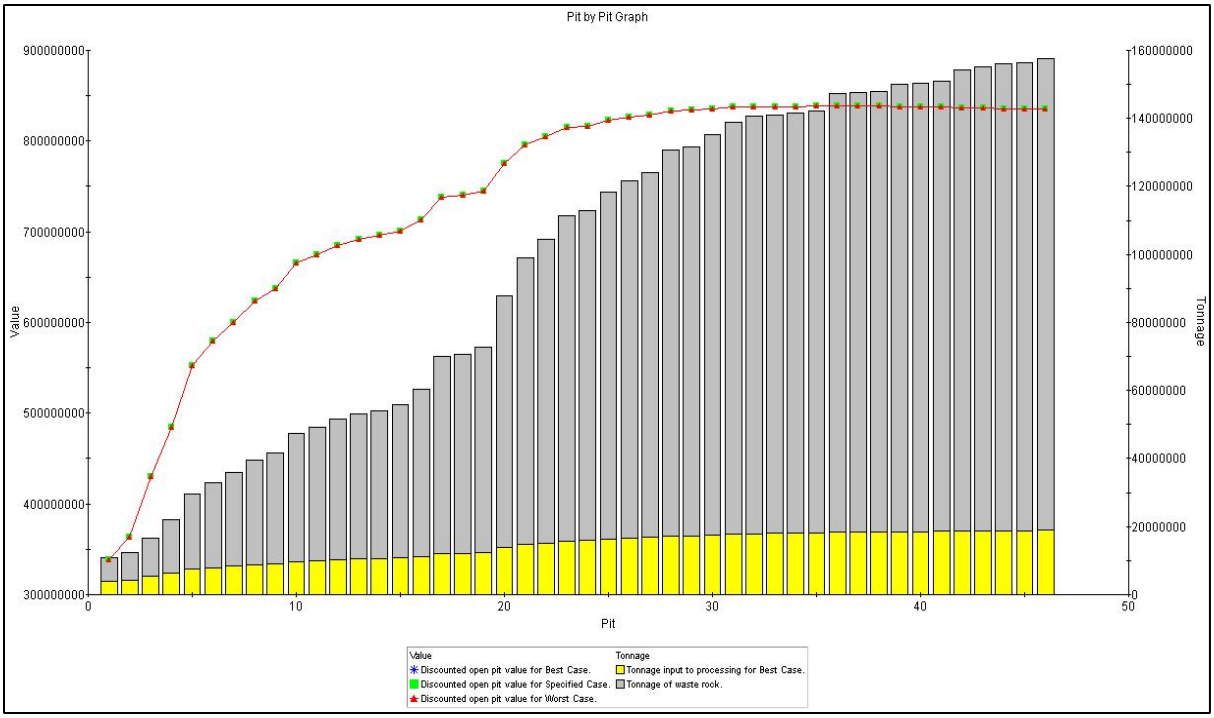
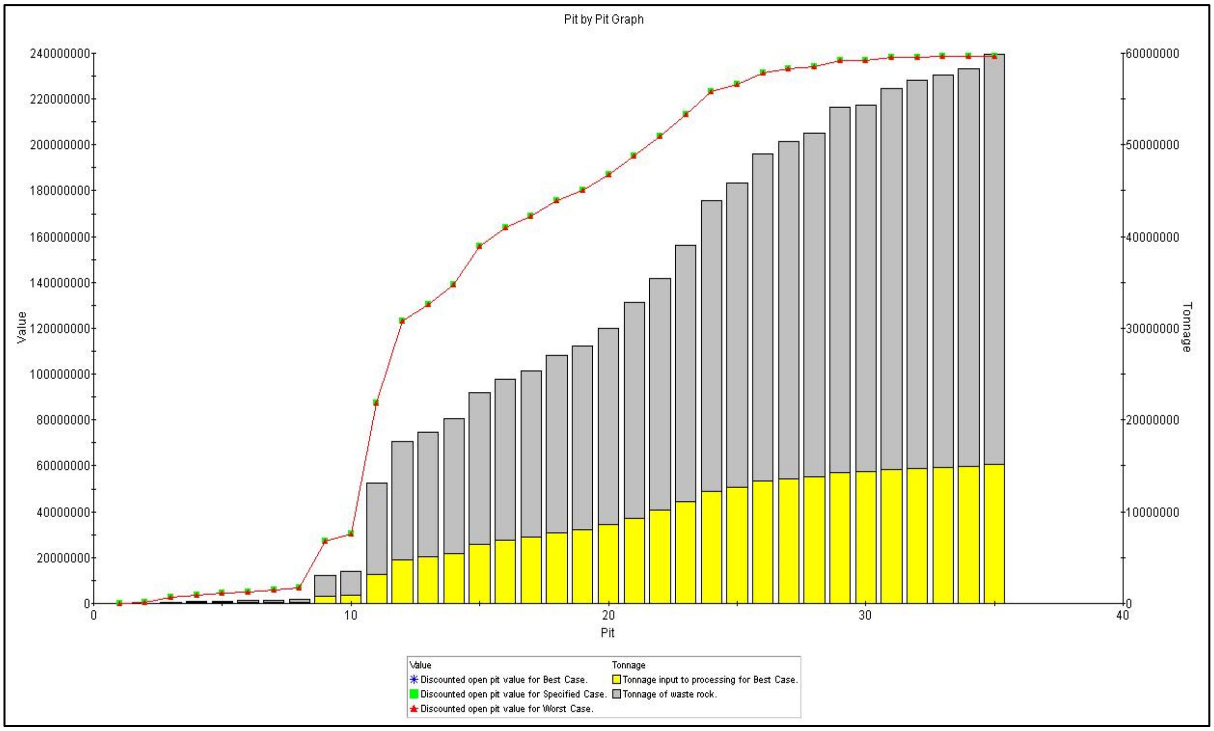


Figure 16-2 Zone 1213 Pit by Pit Phase Graph for Base Case Optimization



16.5.4.2 Final Pit Selection Consideration

When identifying a pit shell to be taken forward to open pit design and pushback selection; financial, operational, and corporate requirements should be considered.

Poniewierski, 2016, notes that the pit limits that maximize the undiscounted cashflow for a given project will not necessarily maximize the NPV of the project, and that when the time value of money is considered, the outer shells of the Revenue Factor 1 pit can be shown to reduce value, due to the fact that the cost of waste stripping precedes the margins derived from ore ultimately obtained. The effect of discounted cash flow means the discounted costs outweigh the more heavily discounted revenues.

As such, the optimal pit from a Net Present Value (NPV) viewpoint may fall anywhere between the RF=0.65 and RF=0.95. For the Carmacks Mine optimization, this is evident as the maximum cash flow of the specified case occurs at a lower overall ore tonnage than the peak of the best-case curve.

Despite this, the selection of the RF=1.00 shell is still commonly seen in the industry for ore reserves work and project feasibility studies, even though the curve cashflow versus tonnage tends to flatten as RF approaches 1.00. This is the case for the Carmacks deposit, which shows that both the specified and best case cashflow curves flatten.

Whittle, 2009, observes that whilst it may be beneficial to maintain operations during this period in case additional resources become available, or prices, costs, or technology improve; this part of the resource should not be regarded as a core driver of the project.

SGS considers that the selection of pit shell to take forward to mine design should be driven both by the results of the optimization; and the industry standards discussed above. Further metallurgical test work determining final recoveries may change these selections to lower revenue factors.

As such, SGS recommends that the following pit shells (pit 35 for zone 147, and pit 24 for 1213) be considered as the “base case” pit shell to be take forward to mine design, managing both economic viability and achieving corporate requirements to maximizing both mine life and extracted gold ounces.

The Whittle pit shells selected as the templates for the pit designs for zone 147, zones 2000S and are those with a revenue factor of 1, corresponding to shell 36 in each case, with the tonnages shown in Table 16-3 and Table 16-4.

Table 16-3 Zone 147 Whittle Results

Zone 147 Whittle shell 35		
	Units	
Resource	tonnes	18,012,865
Waste	tonnes	123,900,606
Total	tonnes	141,913,471
SR		6.88
Copper	(Mlbs)	382.40

Table 16-4 Zone 1213 Whittle Results

Zone 1213 Whittle shell 24		
	Units	
Resource	tonnes	10,626,251
Waste	tonnes	33,307,848
Total	tonnes	43,934,099
SR		3.13
Copper	(Mlbs)	138.9

Please take note that zone 2000s was initially designed as a pit based on the pit optimization but was eventually excluded from the PEA study as the deposit is not economically viable after the tonnes and grade were included in the mining schedule. The strip ratio in the 2000S pit was high with the deep location of the mineralized zone (sulphide).

16.5.5 Processing Plant Capacity

The planned processing plant capacity for scheduling purposes is determined by applying several industry standard formulae based on overall resources. A cursory economic trade off of process efficiencies was conducted with a range between 5,400 tonnes per day up to 9,700 tonnes per day. The final value of 7,000 tonnes per day was chosen based on capital and operating rade off and client guidance to maximize mine life. This results in a nominal plant capacity of 2.45 Million tonnes per year

16.5.6 Processing Recovery

The applied processing recovery that was used for Whittle optimization is 85% for copper oxides and 90% for copper sulphides. During the course of the study, recovery regresssion curves between metal recoveries and acid soluble copper percentages determined that the average recovery is 58% from the supplied metallurgical test material. When applied to the mining blocks in the resource model this blended recovery averages 64% over the life of mine.

(Note – this value is 57% in the plant processing section)

16.5.7 Mining and Transportation Costs

16.5.7.1 Mining Costs

The average mining cost of $3.16/tonne, for both mineralised material and waste, was derived from preliminary contractor estimates for the designed pits.

16.5.7.2 Selling Costs

The cost of transporting the concentrate to Port of Skagway is estimated at $102 CAD per tonne. This is derived from previous studies and escalated by annual inflation since those studies were conducted.

- Other selling costs include treatment and refining charges, and total $0.14 per pound for copper, $4.26 per ounce for gold and $0.31 per ounce for silver.

16.5.7.3 Royalties

Yukon royalties are applied on a sliding scale after allowances for exploration, capital development and operating costs are deducted. A landowner's royalty exists, however this has a buyout clause, which is included in the pre-production capital.

As the revenue is calculated within the block model the price less royalty in Whittle is applied as a factor of 0.97.

16.5.8 Processing Costs

The applied processing cost of $28.00/ ROM tonne was supplied by the client.

16.5.9 Open Pit Constraints and Mining Limits

There are no pit constraints applicable for either of the two pit areas. However, the model frameworks were extended in Whittle to ensure that the pit shells did not hit the edge of the model. The blocks in the extended areas are populated as waste.

16.5.10 Mining Recovery and Dilution

The assumed values applied for geological loss and mining loss are 5% and 2.5% respectively.

A dilution value of 2% was also applied in Whittle.

16.5.11 Applied Revenue

The revenues for the copper con were calculated in the mining model and exported to Whittle as grade values. The price in Whittle was therefore set at a value of 1.

16.6 Open Pit Design

As standard practice in mine design, the Whittle pit shells are used as templates to guide the pit design process. The following wire frames were considered in the pit design. Several pit design iterations were completed based on the value per block for copper values and prices for each bench to satisfy economics.

The initial step was to design a pit shell without ramps to determine how closely the design could be matched to the Whittle shell while applying batter angles and berm widths.

Whittle adds blocks to the pit until the maximum value is reached without consideration for the practicality of mining the resultant pit. This results in drop-cuts of single blocks or small groups of blocks into the pit floor. Consequently, in a narrow deposit such Carmacks, not practical to design a pit as deep as the Whittle shell as the pit bottom becomes too small to deploy equipment. Removing these drop-cuts results in a more practical layout.

These initial designs were then reviewed to determine the number and location of ramps to ensure access for all operating benches.

Final designs were then developed incorporating the ramp systems as shown in Figure 16-3 and Figure 16-4.

Figure 16-3 Zone 147 Pit Design

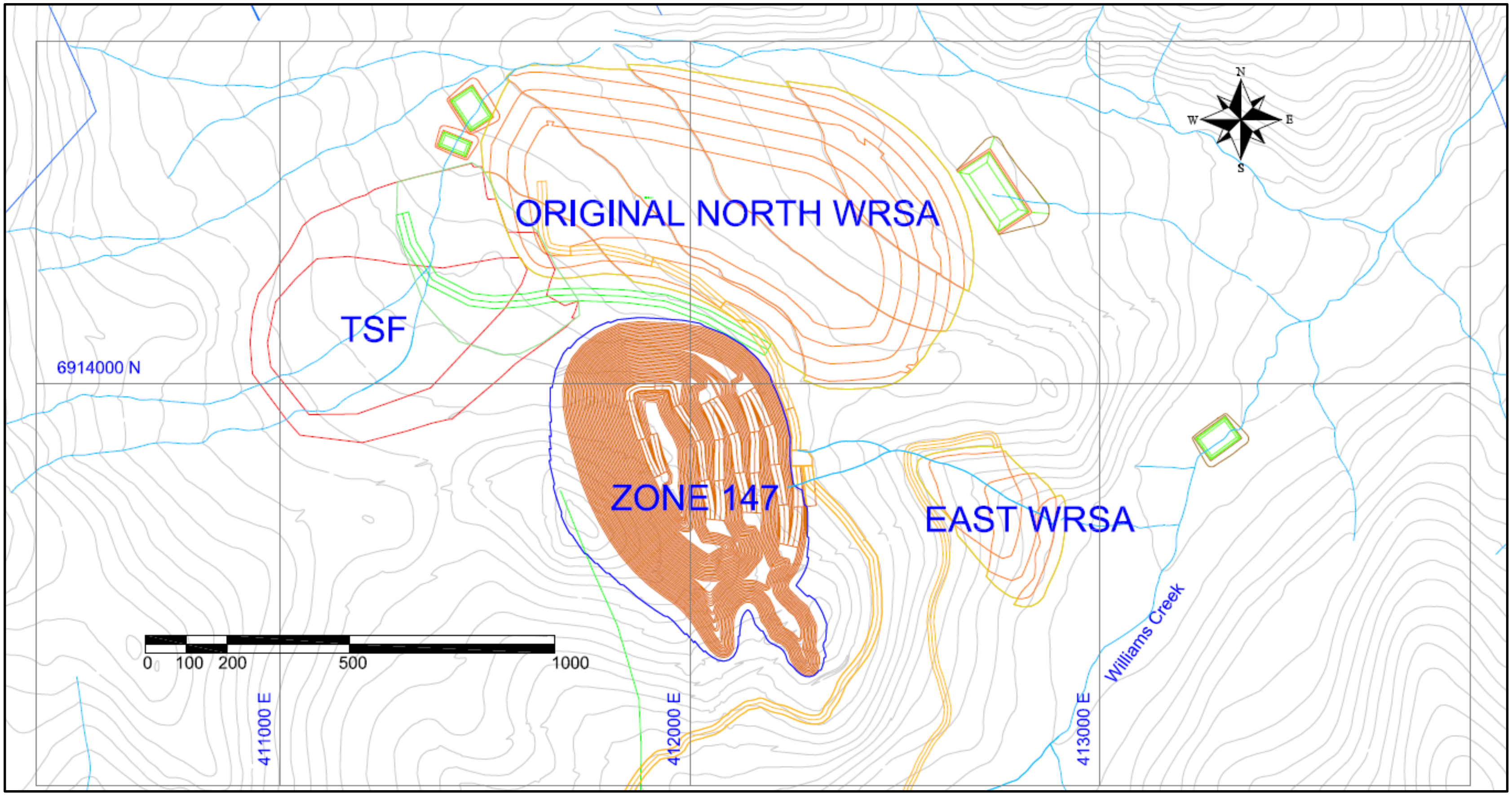
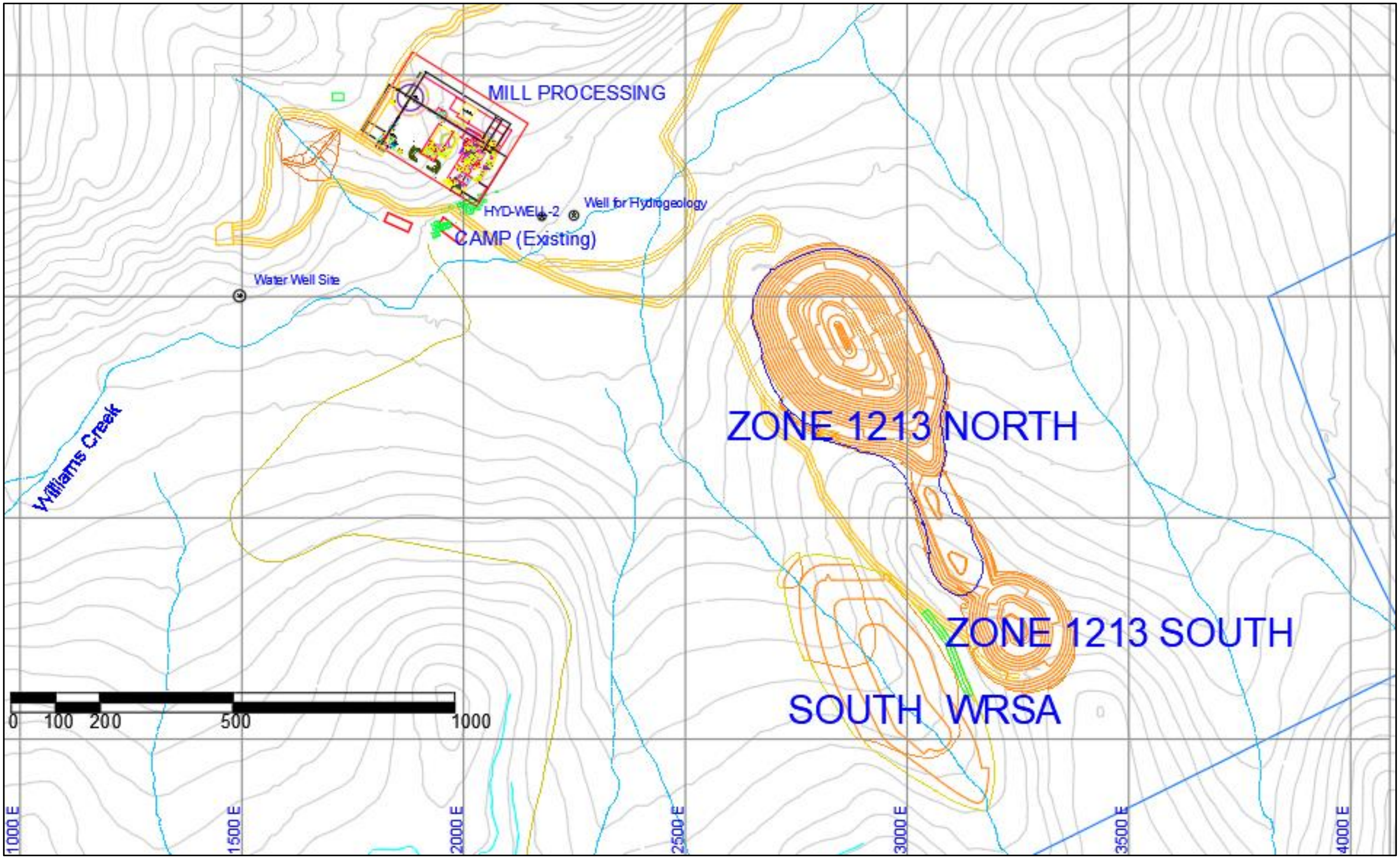


Figure 16-4 Zone 1213 Pit Design



16.6.1 Haul Road Design (Yukon Government Regulations)

A haulage road shall be designed, constructed, and maintained to provide

a) a travel width
   i. not less than three times the widest haulage vehicle used where dual lane traffic exists, or
   ii. not less than two times the widest haulage vehicle used where only single lane traffic exists, and

b) a shoulder barrier or berm
   i. at least three quarters of the height of the largest tire on any vehicle hauling on road,
   ii. located and maintained along the edge of the haulage road wherever a drop-off greater than 3 m (10 ft.) exists, and
   iii. incorporating breaks that do not exceed the width of the blade of the equipment constructing and maintaining the breaks to allow for drainage and snow clearance.

The width of the shoulder barrier referred to a haulage road is not included in the width required in the travel width.

Clearly marked emergency runaway lanes or retardation barriers shall be provided and maintained at suitable locations and be capable of safety bringing a runaway vehicle to a stope, where the road grade exceed 5%.

16.7 Dump Design

The reference for the dump design used in this report is contained in the Golder Associates Report on the Preliminary Design Waste Rock Storage Area Carmacks Copper Project Western Copper Corporation Yukon dated April 15, 2008. At this stage of the mine design (PEA) no optimization of the relevant waste dumps or topsoil stockpile dumps has been completed for 1213 proposed open pit. The only criteria applied was that waste could not be placed between the pits for environmental reasons and should not be too close to the pit area where it may restrict further pit expansion. Two additional waste dumps were planned and designed in proximity to 147 and 1213 proposed open pit mainly to reduce haulage distances. The design basis for 1213 adopted the design parameters recommended in the Golder Associates Report of the Preliminary Design Waste Rock Storage area. The mine layout is shown in Figure 16-5.

The following sections presents the basis of design and the design parameters recommended for the Waste Rock Storage area.

16.7.1 Introduction

16.7.1.1 Site Conditions

The area proposed for the WRSA has a generally subdued topography with an elevation of approximately 730 m at the east or lower end of the WRSA. The ground surface rises gradually to 860 m near or at the west end of the WRSA. The area is naturally covered by the local forest vegetation and the forest cover is very heavy. Several small creeks cross the proposed site and there are no open meadows. The north limit of the WRSA is defined by North Williams Creek which drains the general area and flows to the east towards

Williams Creek. The south limit of the storage area would be defined by a 50 m wide buffer zone planned between the WRSA and north limit of the mine open pit area.

16.7.1.2 Geologic Conditions

The project site lies within the Yukon Cataclastic Terrane geological area. The copper deposit is in a feldspathic mafic gneiss that is underlain by Upper Triassic deposits of hornblende – biotite granodiorite. The bedrock is overlain by overburden deposits of gravelly sand, silty sand, silt and silty clay. These deposits vary from 1 to 2 m thick on topographical high points on the southeast edge of the WRSA to over 90 m thick on the north side of the WRSA site in the North Williams Creek valley. The site is underlain by continuous permafrost which is known to extend to a depth of at least 50 m under the WRSA. The underlying bedrock at depth is not frozen. Preliminary results from site investigations suggest the active layer is 1 to 2 m thick depending on site cover (trees or open cleared areas) and/or slope direction (north or south facing slope).

The groundwater table appears to be located at depth below the continuous permafrost that underlies the site. The groundwater regime appears to be connected to the local creeks and to be flowing towards Williams Creek east of the WRSA.

16.7.1.3 Design and Operations

The WRSA has been designed based on the guidelines set out in the B.C. Ministry of Energy, Mines and Petroleum Resources document for the 'Investigation and Design of Mine Dumps, Interim Guidelines, May 1991'. The design is based on a projected capacity of 60 million tonnes of waste rock assuming a unit density of 2 tonnes per cubic meter. Testing to date suggests that the rock that would be placed in the WRSA is not acid generating or metal leaching. Additional testing is planned and will be continued during operations to confirm this trend. The waste rock is a durable granodiorite or biotite gneiss and would be placed by end-dumping starting near the center of the site and progressing to the east limit of the WRSA before progressing to the west side of the storage area in lifts up to 20 meters thick. The design anticipates the WRSA would have sufficient operating space on the working lifts so the slope stability and settlement should not be a concern.

The WRSA was sited at the present location based on a general sitting study completed by Knight Piésold in 1995. Several sites to the north and northwest were investigated, and the selected location ranked the most reasonable for stability, minimizing haul distance and minimizing the overall mine footprint.

The WRSA has been sited to the north of the open pit in an area that has a thick overburden layer and is understood to be beyond the area that would be considered for mining with the open pit operation. The north limit of the WRSA was determined by the local drainage and would stay south of North Williams Creek, the first major creek north of the mine area.

16.7.1.4 Previous Work

This design was based on field work previously carried out by: Knight and Piésold in 1992, 1995, and 1996, EBA Engineering (EBA) in 1997, Clearwater Consultants Ltd. (CCL), Access Consulting Group (Access) in 2006, Western Copper Corporation (WCC) in 2006 and Golder Associates in 2007. Borehole and thermistor installations from the 2007 Investigation are shown on Figure 3A . All borehole and test pit locations completed in the area of the Waste Rock Storage Area are shown on Figure 3B . The results of this work was documented in reports prepared by Knight Piésold (1993, 1995, 1996, 1996a, and 1997), Hallam Knight Piésold (1995), EBA (2006, 2006a), and Access (2006). Golder has relied on the work in 2007 and on the previous work and assumes there are no major omissions in the previous work. Where available information is limited, or where additional detailed field investigation work may affect the design, attempts have been made to identify the interpretations and assumptions made in this report.

16.7.1.5 Yukon Regulatory Requirements

The Waste Rock Storage Area or waste rock dump is to be developed and designed based on the criteria set out in 2005 agreement between 'Western Silver Corporation' (now Western Copper Corporation) and the Government of the Yukon. In particular the agreement sets out for the WRSA, the performance objectives and regulations/guidelines as specified by the Yukon Government (Yukon Government 2005a) and has additionally considered the following design and operational objectives: • Permanent stability of the WRSA under both operation and closure conditions; • Physical stability as defined by the guidelines in the B.C. Ministry of Energy, Mines and Petroleum Resources document 'Investigation and Design of Mine Dumps, Interim Guidelines, May 1991'; and • The waste rock will be managed in a manner to prevent significant impacts downstream based on the 'Guidelines for Metal Leaching and Acid Rock Drainage at Mine sites in British Columbia' and 'Guidelines and Recommended Methods for Predictions of Metal Leaching and Acid Rock Drainage at Mine sites in British Columbia'. The WRSA will also be operated in a manner to maintain efficient collection and management of surface water around and on the site of the WRSA. The surface waters run off from the WRSA would be directed to a surface water sedimentation pond east and down slope of the WRSA.

16.7.1.6 Design Overview and Proposed Operations

The design is to be carried out in accordance with the current industry design standards and performance objectives and in accordance to regulations/guidelines specified by the Yukon Government. The WRSA is to be located north of the open pit at the mine site to minimize haul distance. The design will accommodate potential changes in the mine plan and will be designed to manage surface water from the storage area to a water management and sedimentation pond east of the facility.

The WRSA has been designed and will be operated to minimize the effort needed at the end of the mine operation for closure. The WRSA will cover some 70 hectares and would contain up to 60 million tonnes of waste rock. The waste rock would be placed in the storage area while the mine is operating for approximately 330 days a year. The WRSA would be developed in a series bench which for the current design are proposed at 20 meters. The benches would be developed with internal or interim slopes and final or ultimate bench slopes that would be developed at 1.4 or 1.5 horizontal to 1 vertical (1.4 or 1.5 H: 1 V). The actual slope angle for the benches or lifts of the WRSA will be at or consistent with the angle of repose of the rock material that is placed in the facility. The design anticipates that for the ultimate or external WRSA slopes, 10- to 20-meter-wide benches would be developed to create an overall slope of between 2 and 2.25 H: 1 V (bench slopes at 1.4 H to 1 V with set back between benches of 20 m) with total WRSA height of 90 m. The current closure plan does not include re-sloping of these benches.

The design and operation of the WRSA will leave a facility at the end of the mine life or at closure that is physically stable in the long term (static and seismic stability issues to be considered) and that has been reclaimed to manage any metal leaching considerations and to address issues with surface runoff and erosion. The operational ditches around the facility would be infilled or removed and the water retaining sedimentation pond to the east of the WRSA would be re-contoured so as not to retain water. The flat areas of each bench set back and the flat top area of the WRSA and the surrounding area disturbed by the operation would be vegetated to minimize long term erosion. Natural vegetation of the slopes of the WRSA would be encouraged.

16.7.2 Design Considerations

16.7.2.1 Design Requirements and Criteria

The WRSA will contain up to 60 million tonnes of waste rock. The WRSA has been sited with a 50 m wide buffer zone from the north of the open pit in an area that has a thick overburden layer and is understood to be beyond the area to be mined by the open pit operation. The bedrock increases in elevation from the west to the east along the north edge of the pit and at the northwest corner of the pit, the bedrock is at a depth of 20 m while on the northeast corner of the pit, bedrock is within 5 m of surface. The 50 m set back will be re-evaluated once the pit slope design is completed and the evaluation confirms the pit slopes are stable under the loading of the WRSA. The north limit of the storage area was determined by the local drainage and the WRSA would be operated to stay south of the first major creek north of the mine area.

It is planned that the storage area would be developed in a series of 20-meter-high lifts. The 20 m lifts would be developed with interim slopes that would be developed at 1.4 or 1.5 horizontal to 1 vertical (1.4 or 1.5 H: 1 V). The actual slope angle for the face of the intermediate slopes of the WRSA will be defined by the angle of repose of the material as it is dumped. The rock will be placed starting from near the center of the site and progressing to the east limit of the storage area or in a down slope direction. The WRSA has been laid out to have Factors of Slope Safety for the interim and final bench slopes of 1.0 or 1.1 during operation (and considering local seismic events). At the end of mine life and after reclamation the Factor of Safety for the local or bench slope would be above 1.3. At present it is not anticipated that re-contouring of the external bench slopes will be required. The overall or total slope should have a Factor of Safety of 1.3 to 1.5 (static) during operation and at closure a Factor of Safety of greater than 1.5 (static). The Factor of Safety long term for the maximum design earthquake or seismic events should be in the range of 1.1 to 1.2.

16.7.2.2 Operation Sequence

The eastern half of the WRSA footprint would be cleared at least 1 year before pre-production stripping starts in the pit to allow the permafrost under the WRSA footprint to thaw. The remainder of the area would be stripped as or before production mining starts. The thawing of the permafrost is important, as the stability of the intermediate slopes of the WRSA is impacted by the thawing ground and this impacts the slope stability of the initial lift of the WRSA. The ground would be monitored to confirm the permafrost retreats under the WRSA. If local pockets of the permafrost remain in the ground, the interim operations slopes would be flattened locally to less than the angle of repose or a ‘small catch berm’ would be developed on the east side of the WRSA to ‘catch’ potential small slope slumps or creep failures that may occur in the initial phase of the operation. As the WRSA expands and the upper lifts of the facility are developed, the permafrost will disappear, or the permafrost zone will stabilize under the WRSA. This condition would increase the bench slope stability. The stability of the intermediate slopes would then be defined by the strength of the waste rock and the drained unfrozen soils which underlie the site. This sequencing may allow for a steepening of the slopes as mining progresses. Geotechnical monitoring will occur during operation (Section 9) to predict if in closure the permafrost will move up into the WRSA.

A perimeter surface water ditch system would be developed before and as the WRSA is developed. The series of ditches would surround the WRSA and direct surface water to the main ditch on the north side of the WRSA which would then flow into the WRSA sediment pond to the east of the storage area. It is proposed to develop the initial lift waste rock starting at elevation 760 m (760 m bench) and filling to the east limit of the storage area. Then, a second lift would be developed starting at the elevation 780 m and this bench would be developed over the eastern half of the WRSA. As the second lift nears completion, the southeastern portion of the WRSA would be developed to an elevation of 800 m as shown on Figure 4 . The 800 m bench would then be extended all the way to the western limit of the WRSA at elevation 800 m. The west portion would then be developed to elevation 860 m in three 20 m thick lifts starting from the southwest edge of the storage area as shown on Figure 5 . The access ramp on the south side of the WRSA would be developed to maintain access to the top level of the WRSA. The ramp would ‘climb’ on the south slope of the WRSA to the top elevation of the WRSA at 860 m. Finally, the waste rock would be placed on the east side of the WRSA area to complete the development of the east half of the footprint as shown on Figure 6 The storage area would be finished out at an elevation of 860 m area as shown on Figure 7 . It is anticipated based on the sequencing of the WRSA that there would be adequate space to adjust to minor slope movements and shift dumping to stable portions of the facility. It is proposed that as the operations guidelines or manuals for the WRSA are prepared, criteria would be developed to identify trigger levels when dumping should be slowed or moved to new areas on the dump face.

16.7.2.3 Subsurface Conditions

Geotechnical site investigations have previously been carried out by others across the planned area of the WRSA (Knight Piésold, 1992, 1995, and 1996) and by Golder Associates in 2007. The investigations included a total of 13 boreholes, four large trenches, and 17 test pits.

Based on the site investigation results, the subsurface soil stratigraphy across the area generally, consists of the following soil layers:

- Organic peat and/or ash layer;
- Glacio-fluvial/Glacio-lacustrine silts and clays;
- Well-graded compact to dense silty sand and gravelly sand (glacial deposits);
- Weathered bedrock; and
- Unweathered bedrock.

The thickness of soil above the weathered and unweathered bedrock varies substantially over the area from approximately 1 to 2 meters under the southeast corner of the WRSA to over 90 m on the north side of the site and to 70 m to the east of the WRSA. In general, the overburden thickness decreases towards the south or towards the open pit area. The depth to bedrock on the east and north side is such that the WRSA stability will be governed by the overburden soils at the site. The results of all of the site investigations to date indicated that most of the area planned for the WRSA is underlain by shallow isolated pockets of peat and organic silt. These deposits are typically less than 1.0 m thick and are underlain by compact sand or silty sand and sand deposits. These deposits range from 8 to 10 m thick and are underlain by sandy silt or silt which may vary in thickness from 2 to 9 m thick. In several boreholes, thin silt layers were encountered in the sand layer within 5 m of the ground surface. The silt layers were discontinuous and appeared to be localized. The upper sand deposits also have varying amounts of silt and/or clay, but generally the upper sandy materials appear well graded. At depth under the east portion of the WRSA site, the sands are underlain at depths ranging from 13 to 25 m by a low plasticity silt or silt with trace sand and clay.

On the north side of the WRSA, the silt and clayey silt layers are at depths of 7 and 14 m. The silt layer under the west and southwest side of the WRSA were encountered at depths of 10 to 16 m and are typically interlayered with sandy zones which are generally 0.5 to 1 m thick. The interlayered clay, silt and sand deposits then extend to depths of some 70 m at the east limit of the facility, while on the north side the depth to rock appears to vary from 60 m to 90 m. The investigations also indicated that the site is underlain by permafrost. Typically, the permafrost was encountered at depths of about 1 to 2 meters. Further, the boreholes drilled over the WSRA footprint indicated that the groundwater table was located at depths of up to 10 meters and typically at or just above the bedrock that underlies the site at depth. The deep ground water system appears to follow the site topography and flow to the east or down slope. A limited number of permeability tests have been completed in the bedrock that underlies the WRSA and the tests indicate the bedrock has permeability in the order of 1x10-4 to 1x10-5 cm/sec. The results of the site investigation and the review of the site conditions indicate that the sandy deposits which would thaw under the initial loading of waste rock will control the slope stability of the WRSA. The silt zones at depth which are frozen will also influence the slope stability of the WRSA but it is anticipated that the silts are deep enough and should remain frozen. Thus, the results of the investigations indicate that the general foundation conditions under the WRSA which consider the impact of permafrost, bearing capacity and long-term settlement are acceptable for the proposed development. The key to the operation of the WRSA will be the impact that the operation will have on the permafrost and the impact the waste rock loading from 20 to 80 m of waste rock will have on the silt strata under the WRSA. These factors are considered in design and would be monitored in the operation of the facility.

16.7.2.4 Seismic Criteria

The seismic risk in the Carmacks Copper Project area has previously been characterized by a seismic hazard assessment carried out for the project site (Knight Piésold, 1995), providing probabilistic and deterministic values for the maximum ground acceleration. The evaluation characterized the site as a low-risk site due the low level seismic activity recorded in this area of the Yukon. The evaluation for this phase of the project was based on the more recent 2005 National Building Code Seismic Hazard Calculations. The site remains a low risk site with the peak ground acceleration for the site at 0.055g for the 475-yr return period and 0.076g for the 1000-yr return period. The seismic loadings or conditions on site were set by the design criteria for the heap leach facility. The design earthquake for the heap leach pad and the associated events pond was selected from the greater or larger event of 50% of the MCE or the 1 in 1,000-yr earthquake. The peak ground acceleration for 50% of the MCE was determined to be 0.055g with a return period of 475 years. The Maximum Design Earthquake (MDE) which was determined to be the 1 in 1,000-year event has a local firm ground acceleration of 0.076g.

16.7.3 Design

16.7.3.1 Slope-Benches and Overall Slope

The Waste Rock Storage Area was evaluated, and the design is based on the BC Guidelines for Mine Waste Dumps, May 1991. In order to set out the design for the WRSA, it was determined based on the above guidelines that the facility would be a large facility (at low end of large dumps rating based on volumes) with a moderate overall slope height on a moderate foundation slope of approximately 10% to 14% to the northeast. The storage area is defined as an ‘unconfined facility’ with no confining gullies or side slopes to act to confine the facility on an intermediate foundation situation with the presence of permafrost being the major foundation stability item. The bench lifts are generally considered favourable and are less than 25 m in height with an ascending construction sequence and a large enough dumping area that the rate of advancement of the front face of the WRSA should be considered slow to moderate. Settlement is not considered a critical design issue as long as settlement does not impact the slope stability. Settlement would be monitored during operation and as it is in the estimated range of approximately 2 % to 3 % of the overall facility height should not be a concern. The settlement will however impact the planning and sequencing for the closure activities of the facility at the end of the mine life. The settlement would be monitored to make sure that surface water ditches during operations direct run off to the WRSA perimeter ditches on each level or lift, so water flows off the WRSA and does not seep or flow through the waste rock, causing potential stability issues during operations. The stability evaluation for the WRSA considered the results of the Golder 2007 site investigation and the results from the previous field programs in defining the parameters for the stability evaluation. The strength parameters for the silty sand and silt stratum were assigned based on the results of the recent laboratory work and are summarized on Table 16-5. The analysis considered that while the silty sand is currently frozen, the construction sequence would allow the silty sand time to thaw and based on the rate of development or the rate of advance of the front / active face of the WRSA, there would be time to allow drainage of the silty sand material so that the drained thaw stable strength parameters were used in design of the slopes of the WRSA facility.

Table 16-5 Material Properties for Waste Rock and Foundation Soils

Material Type	Bulk Unit Weight (kN/m^3)	Cohesion (kN/m^3)	Phi (Degrees)
Bedrock	20	0	40
Silty Sand (Thawing)	22.8	33	10
Silty Sand (Frozen	19.6	0	34
Silty Sand (Thawed)	19.6	0	28
Waste Rock (Surcharge < 200 kPa)	19.6	0	36
Waste Rock (Surcharge > 200 kPa)	19.6	0	38
Clay or Silt (Thawing)	18	33	0
Clay or Silt (Frozen)	18	30	10

The waste rock was assigned a conservative friction angle for confining pressures of less than 200 kPa corresponding to slope heights less than 10 m. The friction angle was set at 36 degrees with a unit weight of 19.6 kN/m^3. The values were increased with increasing confining pressure above 200 kPa. The field investigations indicated that in the north and east directions, bedrock was at a great enough depth that it would not impact the stability of the WRSA facility (i.e. deep failures of overall WRSA slope would not pass through bedrock).

16.7.3.2 Slope Stability Reports

The stability of the WRSA was modeled using the computer program SLOPE/W by Geo Studio produced by Geo-SLOPE International Ltd. The geotechnical criteria used for the design included a bench slope angle of 1.4 H: 1V with a bench set back of from 15 to 20 meters and bench height of 20 meters.

The minimum Factors of Safety for deep seated failure used in the design were:

F.S. ≥ 1.3 to 1.5 for static during the operations period;
F.S. ≥ 1.5 for static at closure; and
F.S. ≥ 1.0 for seismic during the operation period and in closure.

The WRSA was split into the following zones or material types for the foundation and waste rock:

Material 1: Bedrock
Material 2: Silty Sand (Thawing)
Material 3: Silty Sand (Frozen)
Material 4: Silty Sand (Thawed)
Material 5: Waste Rock (Surcharge < 200 kPa)
Material 6: Waste Rock (Surcharge > 200 kPa)
Material 7: Clay Layer

For the purpose of this analysis, two conditions were considered:

- Case 1: Thawing of the upper silty sand layer which results in a weakening of the foundation soils resulting from permafrost degradation and the development of excess pore water pressures in the thawing silty / clay layer and in the thin silt layers within the silty sand strata; and
- Case 2: The silty sand layer is thawed, and the permafrost (frozen foundation soil conditions) stabilizes within the silty sand layer and the base or bottom of the new or re-established active layer is located above the regional phreatic surface. The soil within the active zone consists of silty sand with only thin thawed silt layers and the clay layer at depth remains frozen.

The thawing of the silty sand and deep clay layer was modeled with a thaw weakened silty sand layer (Material 3 - thawing silty sand) and a thawing clay (Material 7 – clay layer).

The results of this evaluation for these foundation conditions resulted in a global failure of the overall slope with factors of safety less than 1.0 for both the static and seismic cases (FoS = 0.9 static and FoS = 0.8 seismic). This case was not considered further as thawing to the depth of the clay layer would not be expected.

The second condition evaluated assumed that the shallow soils below the WRSA were thawed and stable and the soils at depth remained frozen. The frozen conditions were applied to the silty sand layer (Material 3) and the clay layer (Material 7), with the upper silty sand (Material 2) thawed and stable in the active zone. The phreatic surface was assumed to be below the active layer.

Three failure modes were analyzed for this condition for both static and seismic conditions - single bench failure, double bench failure and global failure. Global failure modes were analyzed for both a single 20 m lift and the complete WRSA configuration. The results of the analysis are presented in Table 16-6 and Table 16-7. The results indicate that if the silty sand and clay remain frozen and the upper silty sand thaws and is thaw stable, the WRSA has Factors of Safety that are above the minimums suggested by the BC Guidelines for Design of Waste Dumps. Thus, the design and successful operation of the WRSA will depend on the permafrost conditions at the site. The monitoring planned for the WRSA will be important to confirm a safe operation of the WRSA. Further, the results suggest that the WRSA slopes may be steepened in later stages of the operation as the behavior of the permafrost is understood.

Table 16-6 Factors of Safety for Static Condition

Failure Mode	FOS
Global Failure – Single Lift	1.7
Single Bench Failure	1.2
Double Bench Failure	1.7
Global Failure – Overall Slope	1.5

Table 16-7 Factors of Safety for Seismic Condition

Failure Mode	FOS
Global Failure – Single Lift	1.3
Single Bench Failure	1.1
Double Bench Failure	1.5
Global Failure	1.2

The evaluation indicated that if the operational sequence is such that the silty sand remains frozen, the intermediate bench slope will be stable, and the Factors of Safety will be in the same range as for the thawed conditions.

It is suggested that the WRSA be developed with an overall slope of 2.0 H or 2.25 H to 1 V. This would allow for adjustments to the slope configuration depending on the monitoring to be placed under the WRSA and on the local dump benches. The proposed WRSA layout allows for 20 m wide bench set back at present to achieve the above overall WRSA slope. The evaluation for this configuration indicates that the overall Factor of Safety is at the upper end of the range suggested by the BC guidelines for Mine Waste Dumps. This conservative approach is recommended until the behaviour of the permafrost is understood under the WRSA.

The key to the design of the WRSA is the fill sequencing that will fill initially into the east area of the WRSA with two 20 m high benches and then filling will shift to the west end of the WRSA footprint. The silty sand layer in the east area is 16 to 20 m thick and is underlain by a silty clay or clayey silt. In the event that the clayey silt thaws, it will not be thaw stable and would develop excess pore water pressures which will reduce the above factors of safety and local instability may occur. While it is not anticipated the permafrost would thaw to these depths, it is important to monitor the behaviour of the material as it is loaded by the waste rock. If the clayey silt does not thaw, it is still anticipated that the material would have some minor loss of strength and there would be minor creep of the foundation soils. Thus, the monitoring will determine if the clayey silt thaws. The development sequence is such that there will be a year or two after the initial filling, before the final lifts are placed in the east portion of the WRSA. This will allow time to respond if the slope configuration needs to be modified. The filling sequence on the north slope is also sequenced to allow modification if needed.

16.7.4 Construction Sequence

Based on the WRSA design, proposed dumping sequence and the frozen silty sand beneath the WRSA, it is anticipated that during the pre-production pit stripping phase the following construction activities would be carried out across the WRSA footprint and sediment pond areas:

- general site development and site preparation removing all trees and stripping the area down to the mineral soil over the area to average depths of 30 to 50 cm;
- stockpiling the material in a selected area north or west of the WRSA;
- site grading with construction of the surface water perimeter ditch and the internal ditches and french drains;
- excavation and construction of surface water sediment pond east of the WRSA;
- installation of instrumentation;
- construction of required section of main mine haul road in WRSA footprint; and
- start of the stripping and grading the west portion of the WRSA before the start of production of the mine operation.

The clearing of the east portion of the WRSA as noted above is proposed for at least 1 year before pre-production stripping for the open pit starts and this will allow time to install several ditches or french type drains to drain the upper silty sand that underlies the overall footprint of the WRSA.

The site drains or ditches would direct water or run off to the main perimeter ditch system on the north side of the WRSA. The north perimeter ditch will ultimately direct surface water flows to the WRSA sediment pond east of the site. It is anticipated that to ensure the silty sand under the WRSA drains some of the french drains will be installed at depths of 3 to 5 m. The french drains would be placed below grade with a coarse drain rock core and a sandy surround to allow the drains to function as the waste rock is placed. During the production phase (year 1 and on), it is anticipated that the following construction activities would be carried out:

- Placement of the waste rock in the first bench to completion and the start of the 780 m bench on the east site; and
- Completion of the installation of the monitoring instrumentation on the benches as the WRSA is developed.

16.7.5 Surface Water Management

A Preliminary Surface Water Management Plan (Golder, November 2007) and a Construction Surface Water Management Plan (Golder, March 2008a) were developed for the project and site. The plan included design recommendations for the WRSA and the area around the perimeter of the WRSA. The plan minimizes the project related impacts to surface water and minimizes the quantity of contact water across the site. The plan details surface water management structures that collect or divert surface water to storage and treatment facilities. The surface water management plan sets out the objectives for the surface water management strategy for the WRSA. The features (ditches etc.) in the plan direct non-contact surface water from the area around the perimeter of the facility away from the disturbed areas towards North Williams Creek on the north side of the WRSA. Contact surface water from the WRSA or the immediate disturbed areas would be collected and conveyed through a series of perimeter ditches into the WRSA sedimentation pond for treatment (removal of suspended solids), prior to discharge to the environment. There will also be the option of reusing the water as part of mine operations (e.g. dust control or make-up water for operation of the heap leach facility), or, if necessary, treatment of the water in the on-site treatment plant, prior to discharge to the environment.

The plan sets out that unlined diversion ditches will be used where expected flow velocities are less than 1 m/s, and riprap and geotextile lined ditches will be constructed anywhere expected flow velocities are in excess of 1 m/s. In addition, any life of mine or long-term surface water diversion drains will be lined with riprap.

The surface water management plan has provided the criteria for the design of the sediment pond which is to capture runoff for the 1 in 10 year, 24-hour, rainfall event, plus snowmelt. In addition, the sediment pond has been designed with a dead storage capacity equal to 50 per cent of the runoff storage. The proposed storage capacity of the sediment pond is 45,000 m3 based on the current management strategy for the north portion of the project area.

For normal operations, it is proposed that the sediment pond will have a riser decant structure that will be used to slowly draw down the stored water allowing sufficient time for the settling of suspended sediment. For extreme events, a riprap lined spillway will convey the 1 in 200-year return flood event, plus snowmelt, with a minimum freeboard of 1 m, to prevent overtopping of the embankment. The decant structure and spillway will discharge downstream into a plunge pool, then to North Williams Creek and finally to Williams Creek.

16.7.6 Operational Considerations

The pre-production construction and the initial stages of the operation of the WRSA has been set out with the anticipation that pre-production site work will result in thawing of the permafrost in the upper silty sand strata and any silt layers in the upper soils beneath the WRSA. The sequence and pace of the placement of the waste rock is set to allow an opportunity to monitor the waste rock slopes and the performance of the site. The monitoring planned will enable the operations group to collect information on the re-stabilizing permafrost and to provide changes, if needed, in the operation as required.

The sequencing of waste rock on the west and north portion also allows time for modifications to the procedures to complete the WRSA in a successful manner. During operation, a series of samples of the waste rock will be collected to complete an operational set of kinetic tests to confirm the initial test data which suggests that acid rock drainage and metal leaching will not be a concern with the rock placed in the WRSA during operation and in closure. The testing would be set up to allow the operation to monitor where waste rock is placed in the storage area so if required, measures can be implemented to address any potential concerns. Based on testing to date this should not be an issue and no changes to the operation are anticipated and therefore would be no impact on the currently proposed closure plan for the WRSA.

16.7.7 Monitoring and Long-Term Performance

The construction or fill sequence of the WRSA is set to develop an initial 20 m thick bench starting at the 760 m contour in the middle of the overall WRSA footprint with the dumping progressing east and down slope. The waste rock material would be placed over a larger bench area and the slope would be monitored. The second bench would be developed starting further west at the 780 m contour and would progress east and stop 20 m short of the end point of the initial bench. The initial bench would have been instrumented with survey settlement and movement hubs. At present the dump sequence has not been finalized and trigger levels to move dumping from one area of the WRSA face to another area has not been set. The trigger levels which would set alarms would be based on the guidelines in the Operations and Monitoring Manual developed by BC Mine Waste Rock Pile Research Committee, May 1991. The guidelines indicate that if movements greater than 50 cm per day are recorded in an area, the mine would be required to move operations to alternative dump areas within the WRSA. It is anticipated that once the fill sequence is finalized by WCC operations personnel, a detailed set of guidelines would be prepared for the facility based on the BC guidelines. It is anticipated that the detailed plan would include the installation and monitoring of survey hubs and wire line extensometers to provide an indication of the bench movement and along with monitoring of data from the piezometers and thermistors under the WRSA and around the WRSA. The overall performance of the facility would be reviewed daily by mine staff. Geotechnical monitoring of the WRSA throughout operations will evaluate the performance of the facility and would determine if the operating conditions are as assumed in the design. Further, due to the continuing development or continuous dumping at the WRSA, observations on the performance of the initial stages may provide useful information for optimizing subsequent stages of development.

It is anticipated that instrumentation installed within, and beneath the WRSA will consist of the following:

- Thermistors: Placed beneath the WRSA in boreholes up to 20 m deep under the east and north portions of the site. These instruments will provide a profile of ground temperature with depth for use in assessing changes to frozen ground and permafrost conditions beneath the WRSA, and the resulting impact to geotechnical stability and groundwater seepage conditions and thus the stability of the bench slopes.
- Vibrating Wire Piezometers: These are to be placed within the foundation drains and in the boreholes with the thermistors beneath the east and north portions of the WRSA. These instruments will provide information regarding pore pressure levels within the foundation soils beneath the structures, for use in monitoring geotechnical stability.

- Survey Monuments: Installed on the final surface of each intermediate bench. These relatively simple reflectors will provide a means by which to detect settlements, deformations, or slope movements.
- Wireline Extensometers: Wireline extensometers will be used to monitor movements of the active dump crests during dumping.

A detailed monitoring plan would be developed by the mine to establish monitoring frequencies and reporting responsibilities. The plan would set out trigger levels and action items if movement or the measurements approach 50% of the trigger level and then 80% of the trigger level. The plan would identify staff that would be responsible to respond to the warnings provided by the monitoring system. The final plans for the WRSA slope monitoring would be developed in conjunction with the preparation of the dumping and monitoring plans to be developed by the mine.

The monitoring of the surface water quality and quantity would be set out in the final surface water management plan. It is anticipated that as a minimum, the surface water quality would be monitored on a regular basis during spring runoff and after major storms or when there is active discharge of water to the surrounding environment.

16.7.8 Closure Plan

Revegetation of the general site and the WRSA would follow the general guidelines for reclamation in the Yukon. The plan would be to provide fall seeding to flat areas on the WRSA and to encourage lodgepole pine on south and east facing slopes and white spruce on the north slopes of the WRSA. The flat surfaces of the WRSA would be covered with 0.3 to 0.5 m of the organic material that was stripped from the area before mining. This material will have been stockpiled around the area and would be moved onto the site at closure to encourage re-vegetation. The initial seeding would include native seed mixtures to minimize erosion and provide time for the natural trees to re-vegetate the area. While revegetation would be encouraged on the WRSA slopes, it is not proposed at this time to reseed these areas. If areas of erosion are noted, the slopes may be seeded.

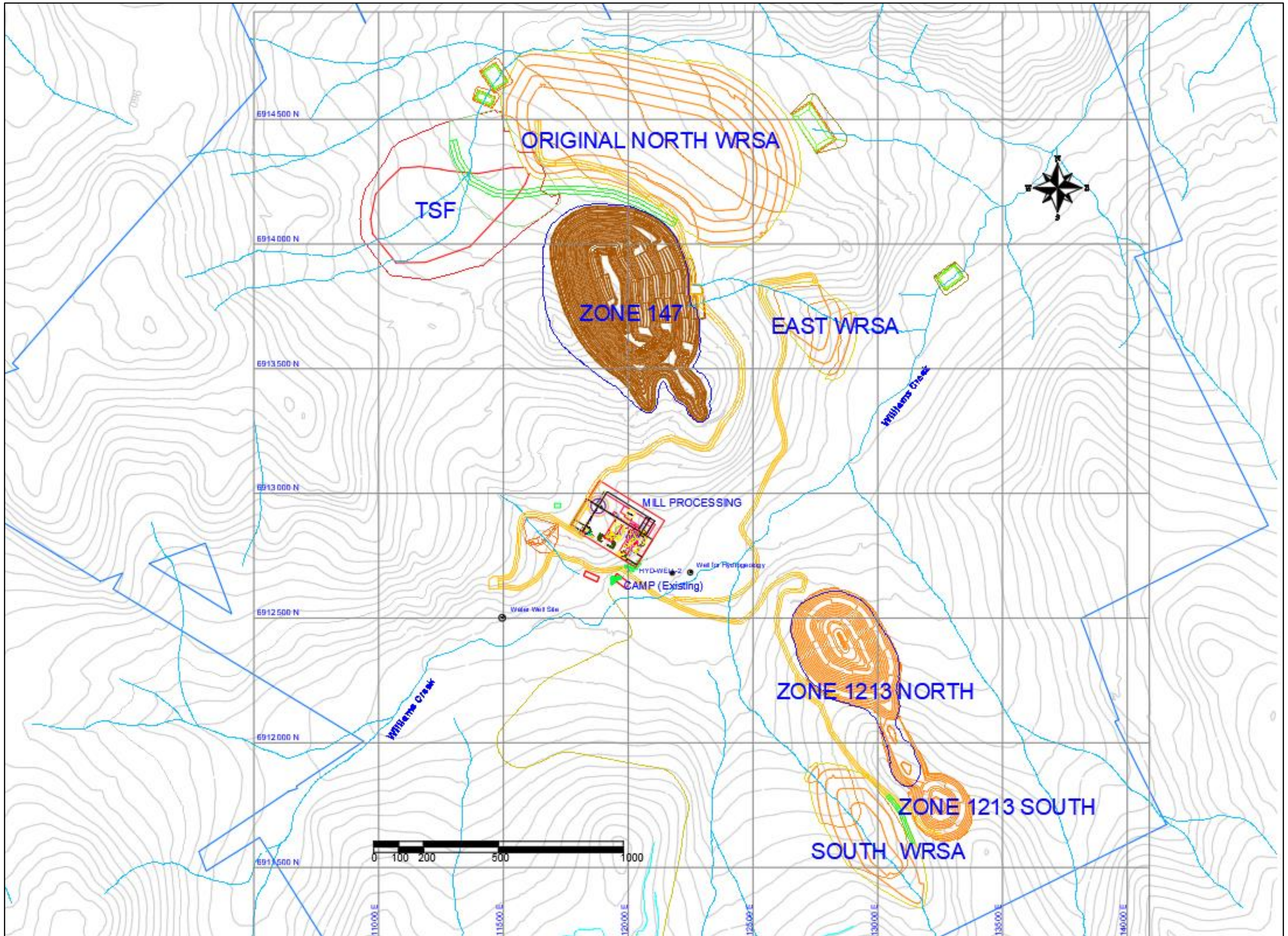
The objective of the cover would be to minimize erosion and to return the area to a vegetative cover similar to the current tree cover in the area. Since the testing to date indicates that neither acid generation nor metal leaching will occur in the WRSA, the cover will not be designed as an evapo-transpiration type cover.

Instead, the objective of the cover would be to minimize erosion and to return the area to a vegetative cover similar to the current tree cover on the area. Establishment of the cover would commence during the later stages of mine operation in areas of the WRSA where the final design elevations have been achieved. This will allow additional time to monitor the success of the initial cover application and if necessary, refine the soil and seed application techniques before completing the application of the cover. Surface water (snow melt and rainwater) will be managed off the west side of the WRSA down the main access ramp, and on the east and north sides of the WRSA in lined rock engineered channels. The drainage ditches and the sediment pond to the east of the WRSA would be removed from service or de-built once the above revegetation is established and is successful and suspended solids are not an issue in surface run off flows.

There are several tasks and research projects that will be carried out during the operation in order to improve the potential success of the revegetation. The work would be to classifying the soils which create the optimum growing conditions and which fertilizers provide the best initial boost to growth and how long should the fertilizer be placed as the long-term goal is to have the vegetation self sustaining without the need for on-going care or maintenance. This work may require small test plots managed by the mine that would be reported in the annual mine environmental summary reports.

The mine layout is shown in Figure 16-5.

Figure 16-5 Mine Site Layout



The nominal lift height is 15 m. For both waste dumps the first lift fills low areas of the topography. The waste dump volumes are shown in Table 16-9 and Table 16-10.

Table 16-8 Zone 147 Waste Dump Design Parameter

15-Apr-08	Preliminary Design WRSA -Golder Associates		
	Carmacks Project Waste Dump Design		
Criteria	Slope		Units
Bench Slope Face	1.4H:1V	36	degrees
Bench Set Back		15-20	meters
Bench Height		20	meters
OSA	2.0H:1V	26.6	degrees
Total Height	90	90	degrees

Figure 16-6 Waste Dump Design Parameters

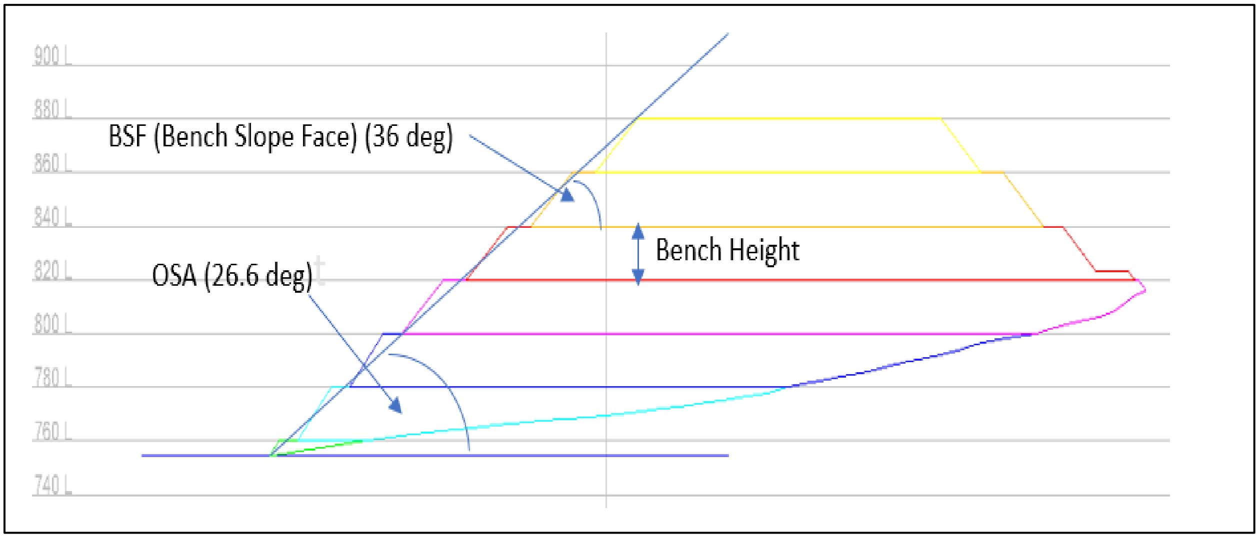


Table 16-9 Zone 147 Waste Dump Volume

Zone 147 Waste Dump		
Toe Elevation	Crest Elevation	Volume
754	760	700,315
760	780	3,060,412
780	800	6,159,439
800	820	8,446,365
820	840	8,131,847
840	860	6,141,525
860	880	4,110,986
	Total	36,750,889

Table 16-10 Zone 1213 Waste Dump Volume

Zone 1213 Waste Dump		
Toe Elevation	Crest Elevation	Volume
770	775	5,713
775	795	580,529
795	815	2,396,792
815	835	3,767,765
835	855	2,162,021
855	875	601,015
	Total	9,513,835

Provision has been made to stockpile topsoil stripped from the pit and waste dump areas. However, topsoil is understood to be minimal and an assumed thickness of 0.3m has been used to estimate the stockpile area. A swell of 15% and a maximum height of 3m have been assumed.

16.8 Dewatering

The climate observed at the project is defined by distinct seasons. In winter (October to April), precipitation is accumulated as snow. Peak flows occur during the freshet month corresponding to the snowmelt in May. Steady state flows are then established during the remaining months (June to September). No serious dewatering issues are expected. However, pumping capacity has been allowed for Water ingress due to rainfall will be managed with berms, cut-off drains and in-pit sumps.

16.9 Operating Hours

Total yearly hours used was 8760, with 730 hrs / month.

Other factors that were included were mobilise and start-up checks, lunch and other breaks, end of shift, blasting and demobilising time. Adding Utilisation and Availability together with overall job efficiency results in 5500 actual working hours per year. This was then used in the equipment calculations.

16.10 Mining Equipment

Due to the nature of the deposit where selective mining is envisioned, separate sets of mining equipment are envisaged for both ore and waste mining and ROM production.

- Waste mining: CAT 6015B with 65t CAT trucks.
- ROM mining; CAT 374FL with 40t CAT trucks.
- Drilling CAT MD6250

Pre-split drilling has been allowed to facilitate steeper pit slopes. Note that geotechnical work was not done and therefore the slope parameters are based on what is termed a “reasonable” assumption. This will be verified during following studies.

Pit in ramp width single road was set at 15.62 m for Zone 147 and 12 m for Zone 1213 which is in line with the physical size of the selected fleet.

16.11 Personnel

Contract mining personnel are calculated using a 20 days on/10 days off shift rotation. There are 3 operators allowed for each piece of production equipment excluding annual and sick leave.

16.12 Production Schedule

The production schedule was developed using gross block values and waste stripping costs, based on providing a minimum of 2.45 million tonnes per year to the process plant.

The pits are scheduled to ensure positive cash flow in every period throughout the life of mine. Several iterations were evaluated, with the final sequence being 147 Stage 1 – 1213N – 1213S – 147 Stage 2. This results in higher recovery sulphide material processed during the stripping of Stage 2 of the 147 pit.

The resource block model provides grades for both oxide and sulphide resources. The block model was interrogated to provide plant feed by resource type (sulphide or oxide) to satisfy the recovery calculation developed by SGS Bateman, thus mined resources are presented with the breakdown of sulphide and oxide resources.

The schedule by year is presented in Table 16-11.

Table 16-11 Schedule by Year

Mine Production Table	Year	-1	1	2	3	4	5	6	7	8	9	Totals
147 Stage 1												
Overburden Stripping	bcm	768K										768K
Waste Mining	t	909K	11,067K	8,742K	2,349K							23,067K
Resource Mining	t	406K	2,592K	2,558K	2,180K							7,736K
Sulphide Resource Tonnes	t	64K	469K	438K	316K							1,287K
Oxide Resource Tonnes	t	342K	2,123K	2,120K	1,864K							6,449K
Total Copper Grades	%	0.98%	0.98%	1.03%	1.25%							1.07%
Sulphide Copper Grades	%	0.94%	0.91%	0.99%	1.25%							1.02%
Oxide Copper Grades	%	0.99%	0.99%	1.04%	1.25%							1.08%
Gold Grade	gpt	0.38	0.41	0.45	0.57							0.46
Silver Grade	gpt	4.12	4.34	4.27	5.25							4.56
1213 North												
Overburden Stripping	bcm				2,729K							2,729K
Waste Mining	t				1,471K	11,998K	2,698K	611K				16,778K
Resource Mining	t				878K	2,555K	2,555K	2,204K				8,192K
Sulphide Resource Tonnes	t				297K	1,220K	2,304K	2,029K				5,849K
Oxide Resource Tonnes	t				581K	1,335K	251K	175K				2,342K
Total Copper Grades	%				0.57%	0.52%	0.59%	0.63%				0.58%
Sulphide Copper Grades	%				0.56%	0.52%	0.59%	0.63%				0.59%
Oxide Copper Grades	%				0.57%	0.52%	0.54%	0.59%				0.54%
Gold Grade	gpt				0.11	0.13	0.12	0.14				0.13
Silver Grade	gpt				2.62	1.76	1.96	2.23				2.04
1213 South												
Overburden Stripping	bcm							336K				336K

Mine Production Table	Year	-1	1	2	3	4	5	6	7	8	9	Totals
Waste Mining	t							1,816K	2,496K			4,311K
Resource Mining	t							192K	570K			762K
Sulphide Resource Tonnes	t							94K	520K			615K
Oxide Resource Tonnes	t							97K	49K			147K
Total Copper Grades	%							0.58%	0.80%			0.75%
Sulphide Copper Grades	%							0.56%	0.82%			0.78%
Oxide Copper Grades	%							0.61%	0.58%			0.60%
Gold Grade	gpt							0.09	0.14			0.13
Silver Grade	gpt							3.36	3.99			3.83
	gpt											
147 Stage 2												
Overburden Stripping	bcm						688K					688K
Waste Mining	t						14,241K	18,169K	17,045K	3,346K	31K	52,833K
Resource Mining	t						10K	28K	1,712K	2,556K	275K	4,581K
Sulphide Resource Tonnes	t						2K	5K	300K	910K	210K	1,429K
Oxide Resource Tonnes	t						8K	22K	1,412K	1,646K	65K	3,152K
Total Copper Grades	%						0.39%	0.34%	1.03%	1.09%	0.98%	1.05%
Sulphide Copper Grades	%						0.37%	0.34%	0.89%	0.96%	0.82%	0.92%
Oxide Copper Grades	%						0.38%	0.34%	1.06%	1.15%	0.67%	1.08%
Gold Grade	gpt						0.16	0.14	0.42	0.36	0.25	0.37
Silver Grade	gpt						2.12	1.66	4.11	4.03	2.99	3.98
	gpt											
Total Mine Production												
Overburden Stripping	bcm	768K			2,729K		688K	336K				4,521K
Waste Mining	t	909K	11,067K	8,742K	3,820K	11,998K	16,939K	20,595K	19,541K	3,346K	31K	96,989K
Resource Mining	t		2,998K	2,558K	3,058K	2,555K	2,565K	2,424K	2,282K	2,556K	275K	21,271K
Sulphide Resource Tonnes	t		533K	438K	613K	1,220K	2,306K	2,129K	821K	910K	210K	9,180K
Oxide Resource Tonnes	t		2,466K	2,120K	2,445K	1,335K	259K	294K	1,461K	1,646K	65K	12,090K

Mine Production Table	Year	-1	1	2	3	4	5	6	7	8	9	Totals
Total Copper Grades	%		0.98%	1.03%	1.05%	0.52%	0.59%	0.62%	0.98%	1.09%	0.86%	0.86%
Sulphide Copper Grades	%		0.91%	0.99%	0.92%	0.52%	0.59%	0.63%	0.85%	0.96%	0.85%	0.72%
Oxide Copper Grades	%		0.99%	1.04%	1.08%	0.52%	0.54%	0.58%	1.05%	1.15%	0.90%	0.98%
Gold Grade	gpt		0.40	0.45	0.44	0.13	0.12	0.13	0.35	0.36	0.25	0.30
Silver Grade	gpt		4.31	4.27	4.50	1.76	1.96	2.32	4.08	4.03	2.99	3.44
Total Copper Metal Mined	Tonnes		29K	26K	32K	13K	15K	15K	22K	28K	2K	184K
Total Gold Ounces Mined	Ozt		39K	37K	43K	10K	10K	10K	26K	29K	2K	207K
Total Silver Ounces Mined	Ozt		415K	351K	442K	144K	162K	181K	299K	331K	26K	2,351K

Note: Numbers may not sum due to rounding.

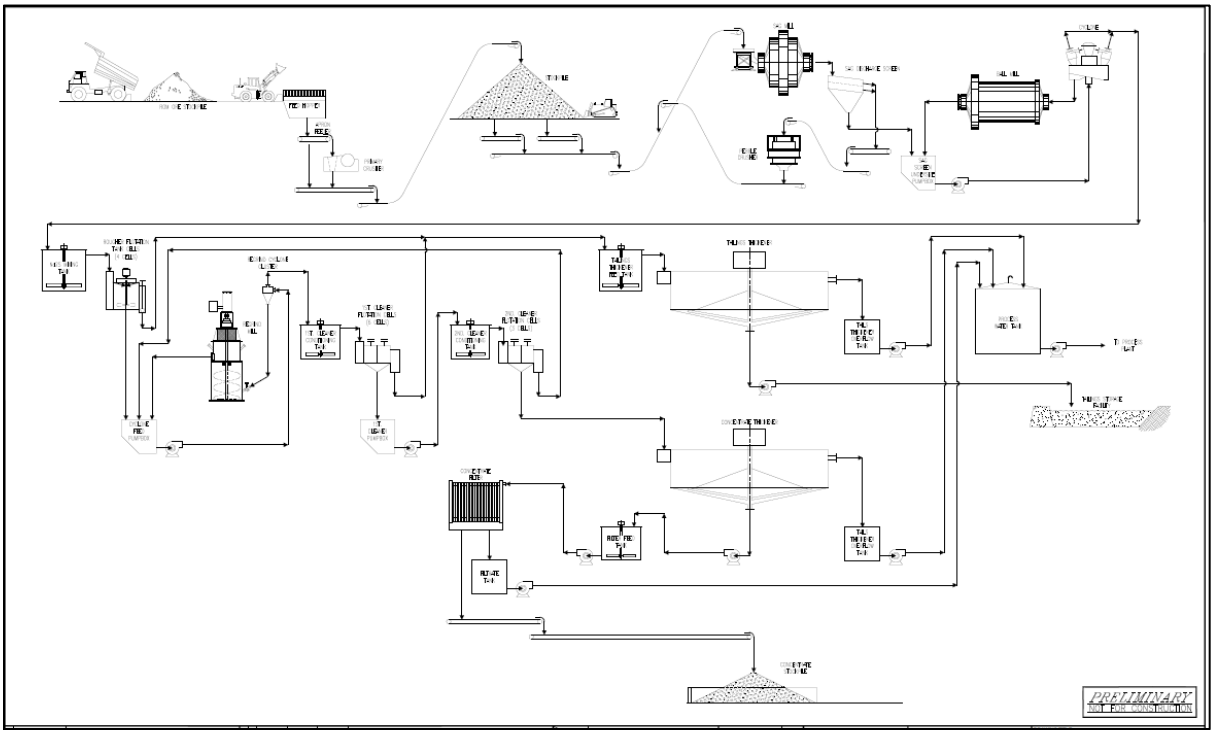
17 RECOVERY METHODS

17.1 General Description

The results of the metallurgical test work as described in Section 13 were used to develop the process design criteria and flowsheet, which in turn were employed to design the process facility as described in this report section. As specified by Granite Creek Copper, the process plant will be designed based upon a daily throughput of 7,000 metric tons. The mine is assumed to operate 350 days per year, corresponding to an annual flotation plant throughput of 2.45 million metric tons. The process includes crushing, grinding, flotation, concentrate dewatering and tailings handling unit operations, as depicted by Figure 17-1.

The Run of Mine (ROM) material will be reduced to a P80 size of approximately 150 microns by a comminution system consisting of primary crushing, SAG milling, pebble crushing and ball milling circuits. Optimal economic considerations dictate that the plant feed will usually be a blend of sulfide and oxide ores. The proportions of oxide and sulfide ore types in the blend fed to the flotation plant is driven by the mine plan which defines the availability of each ore type as the mining effort progresses. The flotation circuit includes rougher flotation, rougher concentrate regrinding, and two stages of cleaner flotation. Whenever there is oxide material in the plant feed, sodium sulfide will be added in the rougher conditioning tank so the oxide copper can be sulfidized and floated. The final concentrate will be thickened, filtered, and then shipped to a broker of refiner. The tailings will be thickened, and the tailings thickener underflow will be pumped to the tailings pond.

Figure 17-1 Overall Carmacks Process Flow Diagram



The METSIM mass balance, process design criteria and equipment sizing are based on the final locked cycle test which utilized blended material which contained 60% sulphide and 40% oxide in the flotation feed.

17.2 Process Design Criteria

The Process Design Criteria (PDC) and mass balance details the annual mineralized material to be processed, production capabilities, major mass flow quantities, key operating parameters, and plant availability estimates. The major process design criteria parameters are listed in Table 17-1. The reagent consumption and consumable supplies estimates can be found in the operating cost estimate in Section 21.

Table 17-1 Carmacks Major Process Design Criteria

Plant Throughput	mtpd	7,000	Granite Creek
Operating days per year		350	Assumed
Annual Ore Feed Rate	mtpy	2,450,000	Calculated
Resource Average Specific Gravity		2.70	Test
Bond Ball Mill Work Index	kwh/mt	15.1	Test
Crushing Utilization & Availability	%	75%	Assumed
Crushing Throughput	mtph	429	Calculated
Grinding and Downstream Utilization and Availability	%	92%	Assumed
Grinding Throughput	mtph	350	Calculated
Stockpile Live Capacity	mt	7,000	Recommended
Primary Grind Size	um	150	Test
Rougher Flotation Slurry Density	% w/w	30.5%	Simulation
Rougher Flotation Retention Time	minute	22.5	Test
First Cleaner Retention Time	minute	12.5	Test
Second Cleaner Retention Time	minute	10	Test
Regrind Size	um	30	Test
Tailings Thickener Settling Rate	m^2/mtpd	0.08	Assumed
Tailings Thickener Underflow Density	%, w/w	60%	Assumed
Flotation Recovery - Copper, LOM	%	78%	Calculated
Flotation Recovery - Gold, LOM	%	65%	Calculated
Flotation Recovery - Silver, LOM	%	67%	Calculated

17.3 Crushing Circuit

For the crushing circuit, the total utilization and availability factor is assumed to be 75% which equates to a nominal crusher feed rate of 373 mtph and a design rate of 429 mtph. Based on this throughput, a Metso C150 jaw crusher or equivalent is recommended. The ROM ore will be trucked from the mine to the ROM stockpile near the jaw crusher. A front-end loader will transfer the material from the ROM stockpile to the jaw crusher dump pocket. The dump pocket will be covered with a stationary grizzly to prevent any oversize material from entering the jaw crusher. The dump pocket has a live capacity of 140 metric tons, which is twice capacity that the mining truck recommended in this study can carry. A hydraulic rock breaker is installed near the dump pocket to break the large oversize boulders retained by the Grizzly.

The material is conveyed from the dump pocket to the jaw crusher using an apron feeder. The jaw crusher will have an open side setting (OSS) of 150 mm (6 inches) or less. The jaw crusher product is expected to have a p80 of approximately 150 mm and top size of 250 mm. The jaw crusher product will be conveyed to a Coarse Ore Stockpile feed conveyor.

The Coarse Ore Stockpile is designed to have a live volume of 24 hours, corresponding to 7,000 metric tons of material. The total stockpile capacity will be about 28,000 metric tons. This translates to stockpile dimensions of 56.5 m in diameter and 21.3 m in height. The crushed ore will be reclaimed by two belt feeders, one operating and one standby. The belt feeders transfer the material to a semiautogenous grinding (SAG) feed conveyor which reports to the grinding circuit.

17.4 Grinding Circuit

The grinding circuit and ancillaries are assumed to have a 92% total utilization and availability factor, corresponding to a nominal plant throughput of 304 mtpd or a design throughput of 350 mtph. There were no JK drop weight tests or SAG Mill Comminution (SMC) tests performed during this test campaign. These studies are typically used for the purpose of sizing the SAG mill. However, based upon the previous JDS PEA study in 2016, the JK Axb value was estimated between 50-55 indicating a medium hard ore. Using this Axb value, a SAG mill of 20 feet diameter and 10 feet effective grinding length (6.1m x 3.05m) is recommended for this process having installed power of 1.75 MW. The SAG mill will be equipped with a variable frequency drive, so the mill speed can be adjusted between 70% to 80% of the critical speed.

Dilution water will be added into the SAG feed chute, so the slurry percent solids in the SAG mill can be maintained at approximately 75% solids by weight. The ball charge is typically controlled between 8% and 15% by volume. The mill discharge will be screened with a vibrating screen. The screen aperture will be 10mm and the screen oversize will be recycled through a series of conveyors to a pebble crusher. The screen undersize will report to a pump box feeding the ball mill cyclone cluster.
The pebble recycle rate is typically between 15% and 30% of the SAG fresh feed rate. An HP 100 short head cone crusher or equivalent will be sufficient to reduce the size of this recycle stream. The closed side setting (CSS) of the pebble crusher can be set to 10 mm or smaller and the crushed product will be recycled using the SAG feed conveyor back to the SAG mill. If the pebble recycle rate falls below 10%, the pebble crusher can be taken offline.

The ball mill circuit is configured and operated in closed circuit with cyclones. The SAG mill product from pebble screen undersize reports to a pump box which receives both the ball mill discharge and the SAG mill discharge. The slurry is pumped from the pump box to a cyclone cluster. The cyclone underflow is fed to the ball mill and the hydrocyclone overflow is the grinding circuit slurry product having a P80 size of approximately 150 um. The Bond Ball Mill Work Index of this material is 15.1 kwh/mt. Based on the design rate of 350 mtph, a 16 foot diameter by 26 foot effective grinding length ball mill is recommended. The installed power for this ball mill will be 3 MW. The ball mill will be rubber lined and will be operated at a fixed speed of 76% of the mill critical speed. The ball charge load is assumed to be a maximum of 33% by volume. The slurry density in the ball mill will be about 70% solids which is approximately equal to the slurry density of the cyclone underflow. The ball mill circulating load is assumed to be 250%. The cyclone feed typically has a solids concentration of 55%. Dilution water will be added into the pump box to maintain the desired slurry density. Two cyclone feed pumps will be installed, with one operating and one standby.

The grinding balls will be added to the SAG mill and ball mill using ball bins. Air compressors will provide instrument air for the mill operational and maintenance requirements. Both the SAG mill and the ball mill will have dedicated mill liner handling machines. An overhead crane will be installed for the maintenance of the grinding circuit.

17.5 Flotation Circuit

The flotation feed can be either sulphide material, oxide material, or a blend of the two ore types. Whenever the flotation feed contains oxide material, the slurry will be first sulfurized with sodium sulfide in a conditioning tank, otherwise the flotation feed slurry can bypass directly to the rougher flotation cells. The slurry density of the flotation feed will be around 30% solids based on the METSIM simulation. This flow rate corresponds to a nominal slurry flow rate of 807 m^3/hr. The design slurry flow rate is 928 m^3/hr based on a scale up factor of 1.15. The sodium sulfide conditioning tank is 4.7 m in diameter and 5.2 m in height, which provides approximately 5 minutes of retention time.

A commonly used scale up factor of 2.5 is used for flotation cell sizing. Based on the slurry design flowrate and required rougher retention time, a bank of six 70 m3 flotation cells is recommended, which provides a total of 23 minutes of retention time. The rougher flotation will be conducted at the natural pH of about 8. Two Collectors, Potassium Amyl Xanthate (PAX) and Reagent 3418A, will initially be added to the rougher flotation feed slurry. When oxide material is present in the flotation feed, a small amount of A-OX100 will be added, which functions to optimize the recovery of oxidized copper. The flotation air will be provided by an air compressor. The total air requirement for the rougher cells is estimated to be 6,000 m_3/hr.

The rougher concentrate will be reground to 30 um before cleaner flotation. A Verti-mill (Metso HIG 700/3000) or equivalent is recommended for this duty. The regrind system is a closed grinding circuit using regrind cyclones. The circulating load of the regrind circuit is assumed to be 150%. The regrind cyclone feed pump box also receives the tailings from the second cleaner flotation circuit. The ball charge in the regrind mill is typically 0.5" diameter steel balls and which are added using ball charge bin.

The rougher concentrate slurry after regrinding will report to the first cleaner circuit. There will be a sodium sulfide conditioning tank before the first cleaner cells. When the flotation feed contains oxide material, the slurry will be first conditioned with sodium sulfide to sulfidize the oxidized copper minerals. The first cleaner consists of a series of four 10 m3 flotation cells. Based on a scale up factor of 2.5, this installed flotation cell capacity allows more than 12.5 minutes of flotation retention time and is sufficient to recover the metals as indicated by the laboratory tests. To increase the selectivity and reject pyrite, the pH is recommended to be maintained around 10 or above. Reagents PAX and 3418A will be required in the cleaner flotation circuits. Reagent A-OX100 proved to lower the cleaner concentrate grade in the laboratory tests and may not be required in the cleaner circuit. If the concentrate grade from this first cleaner is marketable, the concentrate can bypass the second cleaner to the concentrate thickener.

When there is more oxide in the flotation feed, the concentrate grade from the first cleaner will be lower and may require additional cleaner stages. The second cleaner flotation consists of two of 5 m3 flotation cells, which provide a flotation retention time of more than 10 minutes which is deemed to be sufficient to achieve the targeted metal recovery as indicated by laboratory testing. If the flotation feed contains oxide material, the slurry will first report to the second cleaner conditioning tank using sodium sulfide for sulfidizing the oxide copper minerals. The concentrate from the second cleaner will be the final concentrate product, and the cleaner tailings will be recycled back to the regrind circuit. Based upon the locked cycle test, the final concentrate can be expected to have a grade of 40% copper after two stages of cleaner flotation.

17.6 Concentrate Dewatering

The final concentrate from the second cleaner flotation circuit will be pumped to a concentrate thickener. Assuming a typical copper concentrate sedimentation rate, a thickener with a diameter of 5 m is recommended. The concentrate thickener overflow will be recycled to the process water tank. A thickener spray bar will be installed to control the froth on the thickener surface. The thickener underflow will be pumped to an agitated storage tank, which serves as the filter feed tank. The slurry density of the concentrate thickener underflow is approximately 60% solids by weight. An agitated slurry tank with a diameter of 5.5 meters and a height of 6.5 meters will provide a slurry retention time around 12 hours.

The concentrate filter is a plate and frame filter press. Assuming filtration a utilization and availability factor of 83%, a total filtration area of 20.5 m^2 will be required assuming a typical copper concentrate filtration rate. The filtrate will be recycled to the process water tank and the final concentrate cake will be collected to a covered concentrate stockpile using a series of concentrate conveyors. A total of two weeks of storage time for the concentrate is recommended.

17.7 Tailings

The final tailings product consists of the tailings from both the rougher and the first cleaner circuits. The tailings slurry will flow by gravity to the tailings thickener. Assuming a typical copper tailings sedimentation rate, a 31 m diameter thickener will be recommended. The tailings thickener overflow reports to the process water tank, and the tailings thickener underflow will gravity flow to the tailings pond. The tailings thickener underflow will have a solids concentration of 60% by weight. The supernatant from the tailings pond can be pumped back to the process water tank.

Conceptually, the process plant will employ the use of thickened tailings and dry stack disposal, outlined in Section 18.7.

17.8 Reagents

The following reagents will be utilized in the process:

- Lime
- Potassium Amyl Xanthate (PAX)
- Collector 3418A
- Collector A-OX100
- Modifier sodium sulphide
- Frother MIBC or similar
- Flocculant

Pebble lime will be added onto the conveyor feeding the grinding circuit. This pebble lime addition establishes an alkaline grinding circuit environment. A portion of the delivered pebble lime will be converted to Milk of Lime (MOL) by using an installed lime slaker, MOL is used to adjust the flotation pH to the optimum level in cleaner flotation to insure flotation selectivity. Based on laboratory test work, the lime consumption is assumed to be 86 grams per metric tons of plant material. The lime will be delivered onsite in the form of pebble lime and is stored in a lime silo of 25 metric tons of capacity. MOL will be produced by an onsite lime slaker and is stored in an agitated MOL tank. MOL will be pumped through a MOL pipe loop to the cleaner flotation circuit. The daily lime consumption is estimated to be 670 kg/day assuming 90% lime activity.

PAX is the major collector for sulphide copper minerals. PAX will be delivered as solid briquettes contained in supersacks. The PAX briquettes will be dissolved in a mixing tank to produce a PAX solution of 20% concentration and then is stored in a day tank. PAX solution will be added to the flotation rougher, cleaner, and second cleaner flotation sections. The total PAX consumption is estimated to be 41 gram per metric ton of flotation feed. The consumption of PAX calculates to 320 kg/day.

Collector 3418A also assists in the recovery of sulphide copper minerals and is added with PAX together in the rougher and both stages of the two cleaner circuits. This reagent is in liquid form and is delivered in totes. The tote can be placed near each stage of flotation and will be metered to the flotation circuits through metering pumps. The consumption of 3418A is approximately 340 kg/day.

Reagent A-OX100 is an effective collector for oxide copper minerals. This reagent is only added in the rougher stage and usually in the last several cells. The dosage is around 5 grams per metric ton of flotation feed. This reagent is also in liquid form and is delivered onsite in totes. Individual totes are placed in specific a metering pump. The consumption of A-OX100 is 37 kg/day.

Sodium sulfide is used to sulfidize the oxide copper material. This reagent will be added in the conditioners before each stage of flotation. The total dosage is 69.5 grams per metric ton of flotation feed. The reagent will be in solid form and is delivered onsite in the supersacks. The solid briquettes will be dissolved in a mixing tank to prepare a sodium sulfide solution of 20% concentration for storage in a day tank. The sodium sulfide solution will be pumped to conditioning tanks located in front of each stage of flotation cells. The consumption will be approximately 540 kg/day.

MIBC or other appropriate frother is a liquid and is delivered onsite in totes. The tote is positioned near the flotation cells and frother is dosed to the flotation cells through metering pumps. The frother dosage is estimated to be 30 grams per metric ton of flotation feed based on the laboratory test work. The consumption is estimated around 220 kg/day.

No new sedimentation tests were performed on the test samples during the most recent test campaign. However, based upon a previous PEA report, 35 g/t of flocculant dosage to the concentrate thickener and 45 g/t of flocculant dosage to the tailings thickener is assumed. A total of 44.8 grams of flocculant per metric ton of flotation feed is calculated based upon the final concentrate mass pull. The flocculant is shipped in solid form and is delivered in supersacks. The flocculant will be dissolved in a mixing tank producing a flocculant concentration in solution of 0.05% and then is stored in a day tank. The daily consumption will be approximately 320 kg.

17.9 Water and Power

The total fresh water required for the beneficiation process is estimated at 152 m^3/hr., which includes usage for dust control, reagent preparation, and process water makeup. A freshwater tank with a diameter of 16 meters and a height of 20 meters is recommended to provide a total retention time in excess of 24 hours. The fresh water will flow by gravity to the process area for various application needs.

The process water flow is around 750 m^3/hr, and a process water tank of 10 meters in diameter and 12 meters in height will provide a retention time over one hour.

The power will be provided by grid from a local source, and the power line and accessories will be built by Granite Creek.

18 PROJECT INFRASTRUCTURE

The mine, mill processing plant and major mine site related infrastructures will be located at the Carmacks mine site approximately 38 km west of town of Carmacks Town are described in Sections 18.1 to 18.14.2. The Project infrastructure is designed to support an operation with two (2) open pit mines supplying 7,000 tonnes per day tpa processing plant, operating on 350 days per year. The overall site layout showing location of the open pits, processing plant and waste management is provided on Figure 18-1.

18.1 Summary

The infrastructure required for the Carmacks Project will include:

- Mill complex;
- Site development and access;
- Overall water management plan;
- A tailings storage facility (TSF) and associated water management structures;
- Electrical site reticulation and generated power;
- A 100 person camp
- Warehouse, offices, facilities, and other services.

The proposed layout of the Carmacks Project site is shown on Figure 18-1 and the 3D mill plant site layout on Figure 18-2.

Figure 18-1 Carmacks Project Conceptual Site Layout

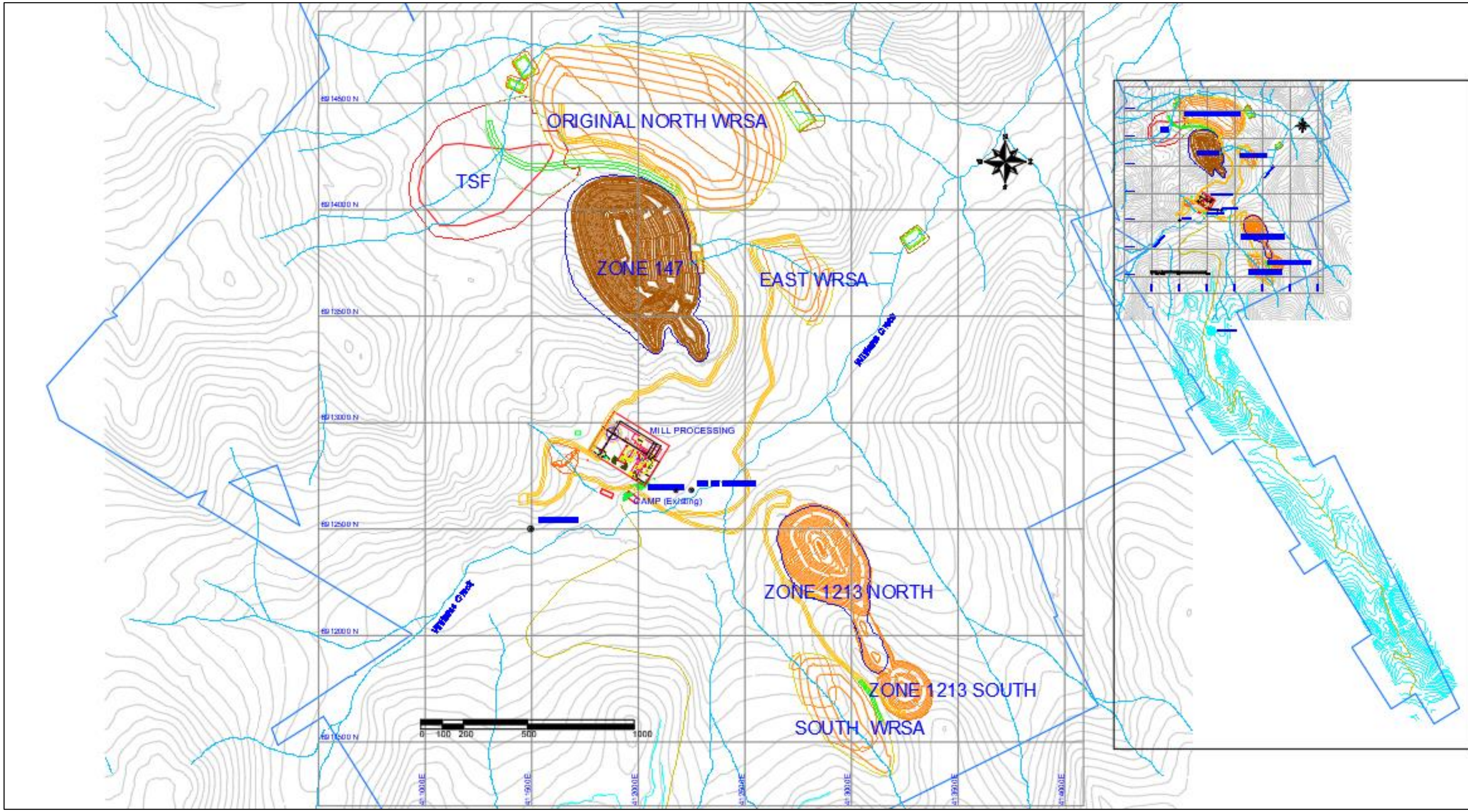
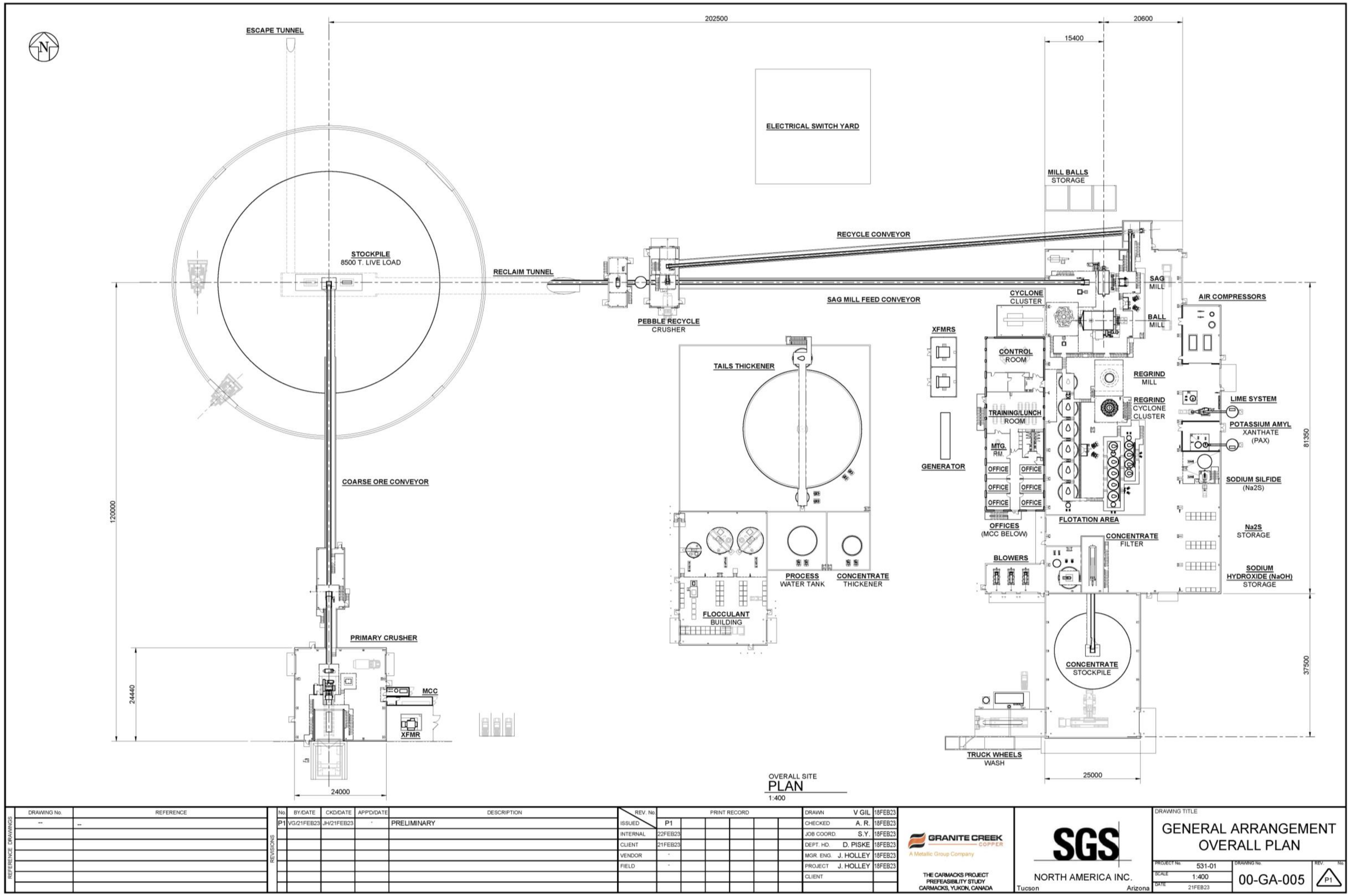


Figure 18-2 Mill Complex Layout



18.2 Carmacks General Site Plan

A Site Layout scale plan has been produced for the project: 2022 Site Layout for the PEA represents the overall area where the mining project is to be constructed. The drawing shows the haul roads, open pits, tailings storage facility (TSF), and processing plant with offices, maintenance shop, and related services for waste rock areas, and water treatment, Mill Complex process plant.

The mill complex site is centrally located, approximately 1,326 m haul road distance southwest of 147 pit, 2,645 m southeast of 1213 north pit and 3.5 km south of the tailing storage facility. It will contain most of the offices, warehouse, maintenance, process plant, and safety office/first aid.

The layout of mill facilities has been optimized to take advantage of topography, and to reduce the earthworks. The entire process plant will cover an area of approximately 21,000 m². The complex will include the primary crusher, ore sorting, grinding via milling, magnetic separation, flotation, E-House, fuel storage, warehouse, office and dry.

18.3 Site Development and Access

The main access road to the Mill Complex and Camp from the Village of Carmacks from the turn out starts at Klondike Highway. A bypass road is presently being built to connect to the Freegold road.

The existing Freegold Road is a two-way gravel road with a finished road width of approximately four to six meters. The posted speed limit is 40 km/h; however, actual operating speeds are lower through many sections in order to safely navigate the horizontal and vertical curves. The road is used to access a number of traditional First Nation fishing camps along the Yukon River and provides access for recreational activities such as fishing, hunting snowmobiling, and hiking. Current resource and industrial users of the Freegold Road include placer mine operators and exploration mining companies active in the Seymour Creek and Big Creek valleys.

The road is seasonally maintained by Government of Yukon up to km 61.8 at Seymour Creek. There are a number of culvert stream crossings along the Freegold Road and a single lane bridge is located at Crossing Creek.

More specifically, to meet the current and future demands along the Freegold Road Corridor for both public and industrial road users, this component includes:

- reconstruction of the Freegold Road from km 0 to km 82;
- construction of a 4.8 km Carmacks By-Pass Road including construction of a new bridge across the Nordenskiold River which is in progress:
- replacement of bridges at Crossing Creek, Seymour Creek, Big Creek and Bow Creek with single lane bridges; and
- construction of stream crossings to meet current fish passage and environmental requirements.

The main access road to the mill complex from the Village of Carmacks starts at Highway N0. 2 turn-offs to Freegold road and it is a constructed gravel district road and regularly maintained by the Yukon Department of Highways and Public Works. The new engineered access road previously built by Copper North Mining Corp shall be surfaced with gravel starts with the coordinates X=415659.181, Y=6903934.002 & X=705.25 masl to the Process Plant area and will be a 12.60 km, two-lane gravel road with a main gate to be manned 24/7.

There are a few minor water courses to cross identified by Granite Creek as areas to consider road construction. The new gravel road will be constructed as per Yukon Government standards to support two-way traffic and to be utilized to transport materials, supplies, and concentrate product for shipment to the port of Skagway.

The road will be designed to Low Volume Road (LVR) 80 Standard Roads will be gravel surfaced at the conclusion of construction. The reconstructed road will follow the existing alignment and no major revisions are anticipated.

All bridges will be built to meet L100 vehicular loading to facilitate industrial ore hauling and facilitate transport of overweight and oversized construction components. Most stream crossings on this portion of the project are small and bridge structures will be a standard design.

The Freegold Road component of the YRGP will also involve construction of a bypass route around Carmacks to take industrial traffic out of the downtown core of this small community.

The Carmacks Bypass Road will provide an alternative route for industrial traffic to avoid travelling through Carmacks. The new 4.8 km route will connect directly to the Klondike Highway near km 354 at the Garvice Industrial Subdivision. During the YESAB review of the Carmacks Copper Project in 2008, comments were received from Carmacks that indicated community members preferred the Carmacks Bypass Road as the route for mine related traffic. The YESAB Executive Committee Screening Report and Recommendation for the Carmacks Copper Project also identified the bypass route as the community's preference. The Village of Carmacks and the Little Salmon Carmacks First Nation have also written letters of support for a bypass to Yukon government on several occasions over the past eight years. Construction of the Carmacks bypass will necessitate a new bridge over the Nordenskiold River just south of the community. This will be the largest bridge on the project (at an estimated bridge length of 72.477 m and total width of 11.35 m) and will require additional design considerations

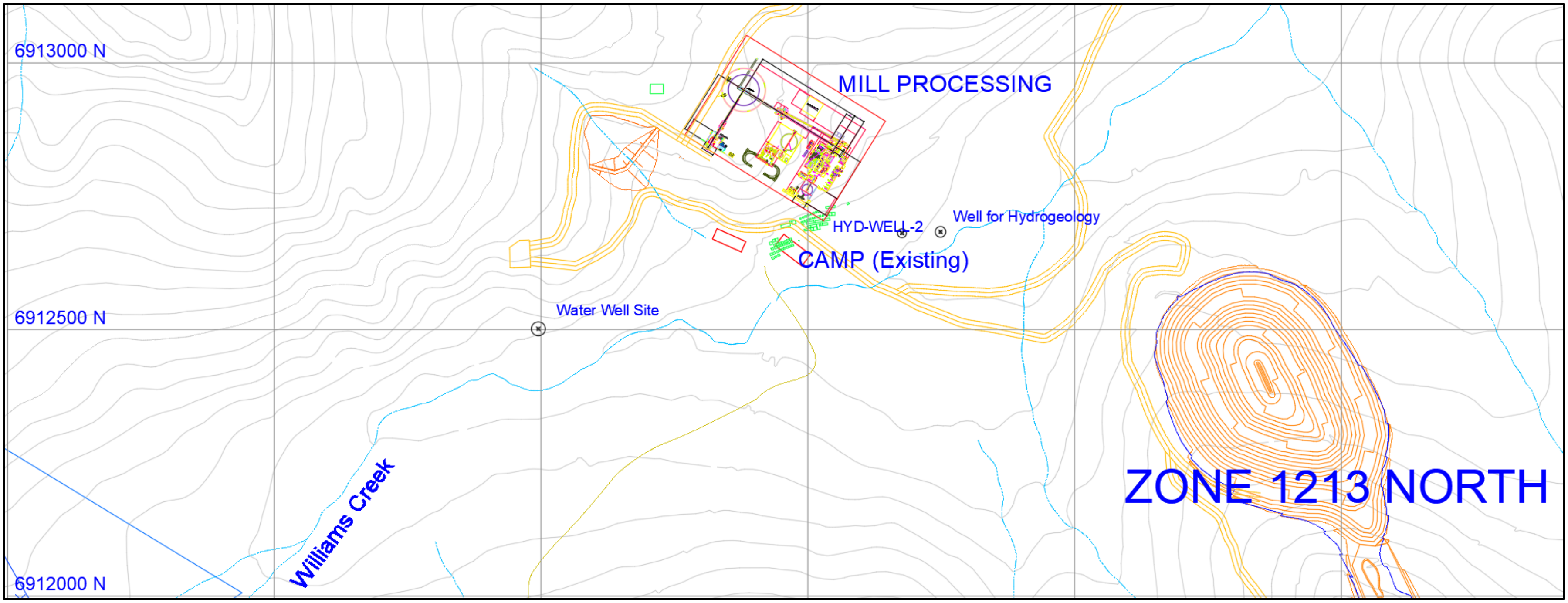
The site haul roads are planned to be constructed on site for transporting ore and waste from the open pits to their designated destinations. Mine haul road and service roads are planned to be constructed to accommodate 40 and 70 nominal tonne trucks carrying ore and waste from the pit to the crusher, and waste to the waste rock area and to the TMF. The waste haul road will connect with the aggregate pit and will also serve as the access to the contractor aggregate primary crusher.

18.4 Overall Water Management Plan

Fresh water to the mill complex site will be supplied by pipeline from two water wells that will be developed to supply water to storage tanks located at the mill. Potential well locations are taken from previous studies as shown in Figure 18-3.

Surface runoff and return water from the TSF and 147 open pit will be collected in a settling pond located at the east side of the North WRSA in a natural low area to enable gravity feed where possible and to minimize earthworks. Run-off collected from areas that are not able to gravity feed to the pond will be pumped. The pond will be lined with a 60 mil HDPE liner and will be equipped with a subdrain system to anticipate groundwater in this area. The pond water will be pumped to either the process plant or the water treatment plant as required. Treated water will be discharged to a 6-inch diameter HDPE pipes.

Figure 18-3 Potential Groundwater Supply Wells



In general, utilities at the plant site will be buried for vehicular access, while outside the process plant area they will run at grade on gravel pads, with culvert / casing protection at road crossings as required.

Yard utilities comprise potable water, sanitary sewage, and fire main, complete with yard hydrants and building connections. Due to the presence of bedrock, these lines will have a relatively shallow bury for mechanical protection.

Spill containment systems will be provided for the fuel storage facility and the oil-filled transformers in the electrical substation.

Aggregate required for site development can be obtained from the mine site quarry area equipped with a mobile crusher.

A water management plan has been developed and includes the following elements:

The mine's facilities are planned to be designed to minimize the effects on the environment that need to be mitigated to the maximum extent practicable.
Natural runoff is planned to be away from, and around, areas disturbed by the mining and processing activities.

Detention storage is planned to be provided for runoff from disturbed areas to allow suspended particles to settle out. The water will then be discharged under a permit, recycled to the process plant, or pumped to the TSF for re-use. Compliant water may also be used for dust control and progressive reclamation.

Waters that contain, or potentially contain, elevated dissolved metals when precipitation meets mined materials are planned to be collected in water quality control ponds and recycled for reuse in the process plant.

Sufficient water storage is planned to be provided in the TSF and water quality control ponds to prevent discharges during extreme wet periods.

Recycling of all mine waters is planned to be maximized, thereby minimizing the need for make-up from surface and/or groundwater.

Provide treatment of stored waters requiring discharge to meet applicable discharge and receiving water standards, as necessary.

18.4.1 Water Management

18.4.1.1 Climate

The Carmacks Project is located approximately 47 km northeast of the village of Carmacks. The climate in Carmacks be described as the summers are long and comfortable, the winters are frigid and snowy and it is mostly cloudy year-round. Over the course of the year, the temperature typically varies from - 26°C to 22°C and is rarely below -43°C or above 27 °C. Based on the tourism score, the best time of year to visit Carmacks CS, Y. T. for warm-weather activities is from late June to late July.

The project is approximately 210 km northwest of Whitehorse, Yukon. The property occurs in NTS map sheet 115/I07. The climate in Yukon is highly variable and one of the coldest regions in Canada with an average high temperature of only 5 degrees centigrade. The climate is predominantly frosty cold and in the winter months the mercury column does not even reach the positive range. In the cold season even, the maximum temperature is well below zero.

18.4.1.2 Annual Precipitation Trend

Daily rainfall data was received from the Meteowhitehorse services for the active Weather Stations in Yukon for a record period of 56 years from the year 2005 until 2012.

The monthly rainfall distribution as obtained from the Meteowhitehorse station summarized in Table 18-1. The mean annual Pan evaporation is 2 850 mm (DWA, 1988) and the mean annual precipitation of 223 mm (NMET, 2021). The months with the highest evaporation are December and January and rainfall are February and March.

Table 18-1 Mean Monthly Rainfall

Annual Precipitation Trend								
Month/Year	Normals Mayo Rd 1981-2010	Normals Mayo Rd 1971-2000	Average since September 2009	Current Month/Year	Wettest Month (year)	Driest Month (year)	Difference Actuals-Average	Difference Actuals-Normals 1981-2010
January	21.7 mm	20.6 mm	23.6 mm	3.3 mm (2023)	31 mm (2018)	11 mm (2014)	-20.30 mm ↓	-18.40 mm ↓
February	13.8 mm	14.0 mm	17.1 mm	49.5 mm (2022)	36 mm (2018)	3 mm (2014)	32.40 mm ↑	35.70 mm ↑
March	11.8 mm	11.6 mm	14.2 mm	22.2 mm (2022)	31 mm (2018)	2 mm (2013)	8.00 mm ↑	10.40 mm ↑
April	5.2 mm	5.3 mm	10.1 mm	13.6 mm (2022)	28 mm (2013)	3 mm (2011)	3.50 mm ↑	8.40 mm ↑
May	25.3 mm	26.6 mm	20.0 mm	19.7 mm (2022)	35 mm (2014)	6 mm (2013)	-0.30 mm ↓	-5.60 mm ↓
June	39.0 mm	37.4 mm	35.9 mm	42.3 mm (2022)	64 mm (2012)	13 mm (2015)	6.40 mm ↑	3.30 mm ↑
July	51.0 mm	55.4 mm	43.7 mm	59.0 mm (2022)	83 mm (2013)	17 mm (2010)	15.30 mm ↑	8.00 mm ↑
August	47.9 mm	50.5 mm	42.8 mm	31.6 mm (2022)	70 mm (2012)	27 mm (2013)	-11.20 mm ↓	-16.30 mm ↓
September	35.9 mm	36.5 mm	24.6 mm	15.5 mm (2022)	69 mm (2010)	12 mm (2011)	-9.10 mm ↓	-20.40 mm ↓
October	23.3 mm	24.0 mm	33.5 mm	64.5 mm (2022)	36 mm (2009)	13 mm (2012)	31.00 mm ↑	41.20 mm ↑
November	18.9 mm	19.3 mm	22.2 mm	13.8 mm (2022)	43 mm (2010)	15 mm (2017)	-8.40 mm ↓	-5.10 mm ↓
December	19.7 mm	20.8 mm	22.1 mm	27.7 mm (2022)	56 mm (2013)	5 mm (2017)	5.60 mm ↑	8.00 mm ↑
Year	313.4 mm	321.9 mm	310 mm	3.3 mm	387 mm (2013)	307 mm (2011)	-306.70 mm ↓	-310.10 mm ↓

12 Month Average Differentials
Average 12 Month Precipitation since September 2009: 309.80 mm
Precipitation for last 12 months: 363.10 mm
Current 12 Month Precipitation difference: 53.30 mm ↑
Year Precipitation Differentials
Average Year to Date Precipitation based on historical data since 2009: 17.35 mm
Current Year to Date Precipitation (2023): 3.30 mm
Current Year to Date Departure from Normal (2023): -14.05 mm
Month Precipitation Differentials
Average Month to Date Precipitation based on historical data since 2009: 17.35 mm
Current Month to Date Precipitation (January): 3.34 mm
Current Month to Date Departure from Normal (January): -14.01 mm
This data is reflective of 12 month trending at the Meteowhitehorse Weather Station location. This is not official scientific data. The weather data presented here is based on data collected since September 2009. Script courtesy of **Michael Holden** from Relay Weather. Script calculations courtesy of **Murry Conarroe** of Wildwood Weather. **Scrift modified by Meteowhitehorse March 3, 2014**

18.4.1.3 Stormwater Management

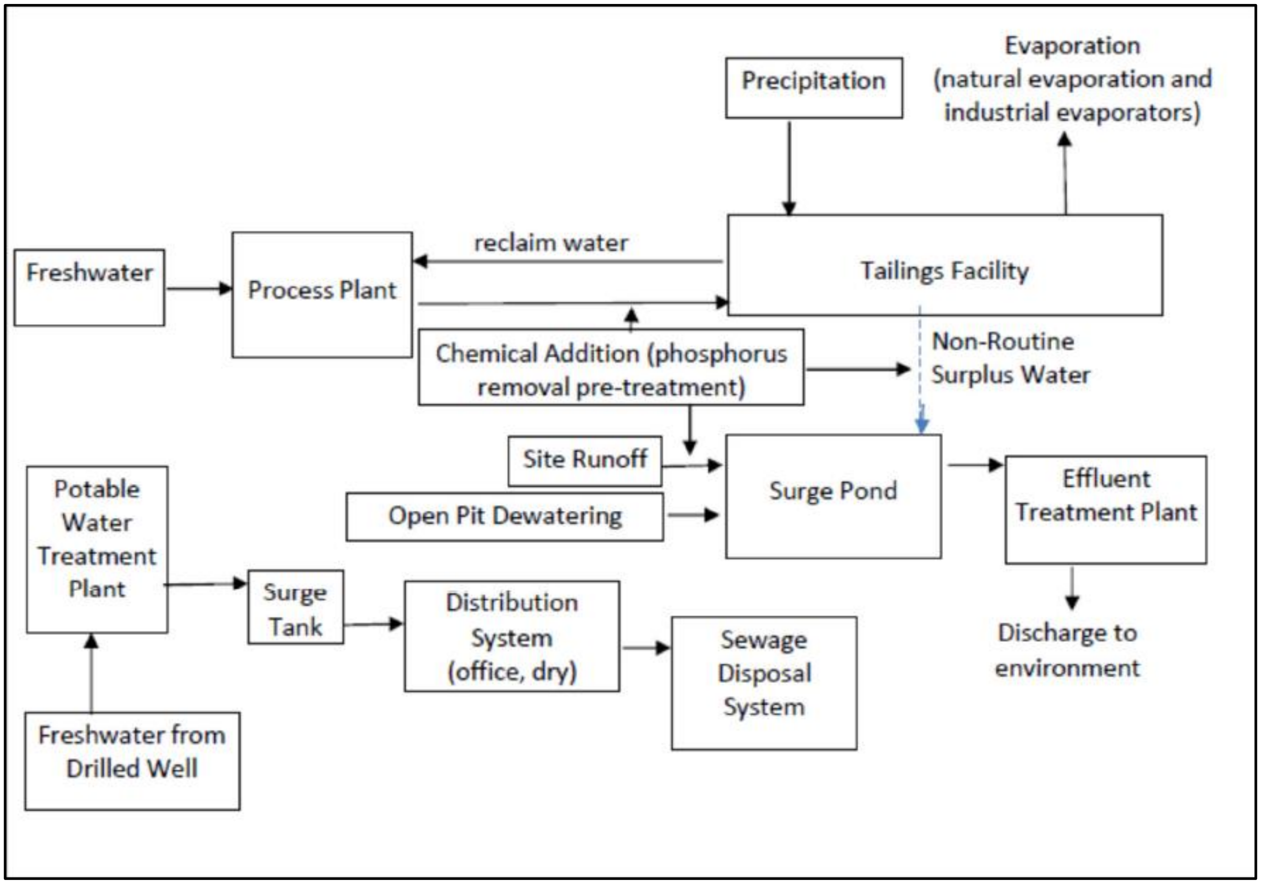
A high-level stormwater management assessment was conducted on the surrounding catchments, and they were found to be small in size, as they form the upper reaches of the North Wiiliams Creek to the north and the Wiiliams Creek to the south. It was found that no storm water diversion system is required upstream of the TSF. It is unlikely that the runoff generated in the catchment adjacent to the toe of the TSF will have an impact on the TSF, as there is sufficient distance between the main watercourse, the Sediment Ponds, and the toe of the TSF. There are proposed settling/catchment ponds across the entire mine as presented in the overall site plan.

18.5 Effluent Treatment

18.5.1 Conceptual Water Management Plan

The conceptual water management plan prepared by SGS (Reference: Past Projects) is presented in Figure 18-4 which is expected to minimize surplus water from the tailings storage facility that will require treatment and discharge.

Figure 18-4 Conceptual Water Management Plan



It is assumed that the TSF will be operated as a stand-alone system and will generally have a negative water balance due to water loss to tailings solids pore space. Further detailed studies are required in the next level of study to ascertain the proper treatment of water for discharge to the environment.

18.6 Waste Rock Pile

Waste rock will be stored in three (3) areas and are called North WRSA, EAST WRSA and South WRSA. The North WRSA is located west side of the TSF. Contact water, generated from water-rock interactions, will contain soluble constituents (i.e., major ions, metals and nitrogen species) from mineral weathering by-products and from residual explosives from blasting, which can persist in the waste rock and are water soluble and provide a source of ammonia and nitrate.

The contact water from the waste rock areas will report to the TSF.

18.7 Conceptual Tailings Storage Facility (TSF)

Previous reports on the Carmacks project indicated a design of a conceptual Tailings Storage Facility. The data used in this section of the report was based on the Tailings Storage Conceptual Design dated May 3, 2016 completed by Golder Associates and there were no updates thereafter. Previous studies carried out by others included Knight Piésold (1995, 1996), EBA Engineering (197), and Aurora Geosciences (2006). These studies were used to inform more recent work carried out by Golder including:

- Geotechnical site investigation (Golder 2008b)
- Site Water Balance

SGS updated the preliminary open pit design for zone 147 and the design for the Waste Rock Storage Area (WRSA). The TSF location was planned to be constructed at the west tip of the WRSA.

In addition to design work, Golder has conducted annual site inspections from 2008 to 2015; including reading and reporting thermistor data collected form the site (Golder 2015)

This PEA envisions the use of a dry stack tailings facility in the same footprint as the tailings impoundment identified by Golder. The TSF will be a valley impoundment with a starter embankment constructed from rockfill and thereafter the tailings will be filtered and stacked within the TSF until final elevation. The TSF is designed as an integrated waste landform constructed of filtered tailings surrounded by an HDPE linedwaste rock shell. The tailings will be delivered to the TSF via trucking and thereafter distributed around the TSF through a dozing equipment for compaction. The processing facility will operate year-round at a nominal rate of 7000 tonnes per day (tpd) resulting in approximately 2.2 to 2.5 million tonnes per year (Mtpa) of tailings.

Contact water, generated from interaction of runoff water with the tailings, will contain soluble constituents from by-products of tailings oxidation and process water. The contact water from the tailings area will report to a sediment pond located at the immediate East of the TMF. The sediment pond will have a storage of 40 000 m^3 to continuously absorb decant water, consisting of production water and floods, it is recommended to split the sediment pond into two compartments, for operational purposes.

Progressive reclamation of the TSF will be conducted as the facility is constructed with lower berms sloped, covered and vegetated as part of the operations cost of the facility. Final site closure contains an overall allowance for the closure of the site including the final containment of the TSF in Year 9.

18.7.1 Concept Design Assumptions

- Tailings throughput: Average 2.4 million tonnes per annum (Mtpa), maximum 2.5 Mtpa, at 76% solids content by mass.
- Total storage requirement: 22 million tonnes (Mt).
- Life of Mine: 9 years.
- Maximum rate of rise: 2.5 meter per year for upstream raises (m/yr.).
- Tailings geochemistry and classification assumed to require lining system.

In general, the TSF will be designed to:

Satisfy the regulatory requirements specified by the Yukon Government.

Comply with the following guidelines:

- Canadian Dam Association: Dam Safety Guidelines (2007, and revised in 2013);
- Canadian Dam Association: Technical Bulletin -Application of Dam Safety Guidelines to Mining Dams (2014);

- ‘British Columbia Mined Rock and Overburden Investigation and Design Manual, Interim Guidelines, May 1991’ and Mined Rock and Overburden Piles (Runout Characteristics of Debris from Dump Failures in Mountainous Terrain Stage 2: Analysis, Modelling and Prediction -Interim Report February, 1995)
- ‘Draft Acid Rock Drainage Technical Guide’ (British Columbia Acid Mine Drainage Task Force 1989);
- ‘Draft Guidelines and Recommended Methods for Predictions of Metal Leaching and Acid Rock Drainage at Mine sites in British Columbia’ (Price, 1997);
- Global Acid Rock Drainage Guide (INAP 2009); and
- Prediction Manual for Drainage Chemistry from Sulphidic Geologic Materials (Mend 2009).

18.8 Open Pit

The excavation of mine rock and the development of the open pits will result in the rock face of the pit walls being exposed to atmospheric conditions. The blasting of the rock typically results in a “damaged zone” of rock that consists of shallow fractures that extend into the bedrock from the face of the pit wall. The surfaces of the fractures in the damaged zone are also exposed to atmospheric conditions. Contact water, generated from water-rock interactions (i.e., from direct precipitation, groundwater inflow and runoff from the open pit catchment area) at the pit wall surface, will contain constituents of the exposed rock and explosive residues. Pit-wall contact water will report to the pit sump and affect the quality of the sump water.

The contact water from the open pits will report to the Settling/Sediment Ponds.

18.9 Sediment Pond

The Sediment Pond was sized to continuously absorb decant water, consisting of supernatant water and floods. It is envisaged to split the pond into two compartments for operational purposes, namely an operational compartment and a stormwater management compartment.

The water quality in the sediment ponds will therefore consist of contact water from the TSF and open pits, together with natural runoff from the catchment areas.

Excess water in the Sediment Ponds will be discharged to the environment in accordance with the Department of Environment/Government of Yukon effluent water discharge regulatory/permitting requirements and effluent discharge restrictions (i.e., effluent concentration limits). The surface water receiver of the effluent will need to have an appropriate assimilative capacity to allow for rapid mixing of the effluent, such that Water Quality Objectives are achieved downstream of the outfall. If the effluent has concentrations greater than the effluent concentration limits, and/or the surface water receiver does not have the necessary assimilative capacity, the effluent will need to be treated prior to discharge to the environment.

18.9.1.1 Subsurface Conditions

The project site lies within the Yukon Cataclastic Terrane geological area. The deposit is in a feldspathic mafic gneiss that is underlain by Upper Triassic deposits of hornblende – biotite granodiorite. The seismicity at the site is considered low to moderate.

A total of two boreholes and 11 test pits have been excavated within the TSF footprint. Review of the test pit and borehole records for the TSF footprint indicate that subsurface conditions are similar to those found elsewhere at the project site. In general, the subsurface conditions consist of:

- surficial organic silt/volcanic ash layer, typically 0.3 m thick, ranging from 0.1 m up to 1.0 m thick;
- varying overburden consisting primarily of till (silty sand and gravelly sand) frequently interlayered with glacio-fluvial and glacio-lacustrine silts and clays;
- weathered granodiorite bedrock at varying depths (3 m to 60 m below ground surface); and
- decreased weathering of bedrock with increasing depth.

The regional groundwater occurs as an unconfined deep system (recharged at high elevation and discharged at low elevation), following a subdued replica of the topography. Seasonal perched flow on the valley floor results in shallow groundwater conditions in local swampy areas (Golder 2012b). No piezometers have been installed within the TSF footprint. Piezometers at other locations within the project site measured groundwater occurring 12 to 60 meters below ground surface.

The project is located in a zone classed as ‘widespread’ discontinuous permafrost (Brown 1970). One thermistor was installed at the eastern edge of the TSF footprint in BH-12-07. This thermistor most recently recorded an active layer of 7 m, although previous measurements indicated an active layer of 5 m (Golder 2015). An active layer of 5 m to 7 m is typical for the project site based on measurements from the other thermistors at the site.

18.9.1.2 Climate

The site is located in a sub-arctic climate, characterized by cold winters, and short, cool summers. Snow cover is present at the site from October to May. The average annual average air temperature is approximately -1º C with monthly average air temperatures above zero from April to September. The site has an annual average precipitation of 346 mm (Golder 2012a), comprising approximately 190 mm rainfall and 157 mm of snowfall water equivalent. The highest average rainfall is in July, with an average of 54 mm.

Climate data for the site were previously derived from correlating the nearby Williams Creek climate station (data from 1994 to 2011) and the Pelly Ranch climate station (data from 1952 to 2011) (Golder 2012a). The Williams Creek climate station is located approximately 1 km from the project site and the Pelly Ranch station approximately 35 km from the project site.

Pelly Ranch climate station has records of climate data to the end of 2014. Based on a comparison of average precipitation and temperature at Pelly Ranch from 1952 to 2014 and average precipitation at Pelly Ranch from 1952 to 2011, there has been a recent trend of slightly increasing precipitation (in both snow and rain) and average temperatures. As only an additional three years of data are available and the difference in precipitation is small, the precipitation data and storm events have not been re-calculated for the current conceptual-level analyses. Table 18-2 provides the design storm events assumed for the purpose of the conceptual design.

Table 18-2 Design Storm Events

Return Period (years)	24-hour Rainfall (mm)
2	27
5	36
10	42
20	47
50	53
100	57
200	61

Source: Frequency analysis results from 3 parameter log normal distribution for Pelly Ranch (1952-2011) adjusted to Williams Creek. (Golder 2012a)

18.9.1.3 Tailings Production

Tailings will be produced at an average rate of 7000 tpd operating year-round for an annual production of 2.45 Mtpa, inclusive of shutdowns for maintenance. The project has an operational life of 9 years over which time it will process 21.5 million dry tonnes of ore and generate approximately 20 million dry tonnes of tailings.

18.9.2 Cost Estimate

The TSF will be constructed during the pre-production capital period. Progressive lifts of the facility will occur under operations.

Golder provided a Class D cost estimate in 2016. This cost estimate was factored by inflation and estimated on site gravel supply costs. Fuel prices were updated to reflect increased fuel costs. Capital and operating costs are presented in Chapter 21 of this PEA.

The cost estimate is based on 'typical' unit rates for similar projects and conceptual estimates of quantities.

18.10 Stockpiles

Three waste rock dump (WRD) areas will be constructed. One north of 147 open pit, and one south, South WRD, 1213 open pit to store mine rock from the open pit excavations. The rock piles will be built in 15 m lifts to provide an overall safe slope of 35 degrees. The inter-bench slopes will be at the angle of repose of the rock. Details of both WRD specifications are found in Section 16.8.

Collection ditches and contact water collection ponds/sumps will be built at topographical low points around both WRD's perimeters to collect runoff and seepage, which will then be pumped to the RW pond.

A topsoil and overburden stockpile will be established to contain stripped materials from all excavations for the project development. Sedimentation ponds will be built to settle out solids before release to the environment. A perimeter ditch will be constructed at the toe of the topsoil and overburden to keep the material intact.

The stockpiled/reclamation materials will be utilized for rehabilitation applications upon open pit closure.

18.11 Electrical Site Reticulation and Diesel Power Generation

18.11.1 Electrical Load

The predicted electrical demand load is approximately 10 MW during open pit mining operations.

This estimated load is based on the current mill process-mechanical load, mill utility load, tailings management facility load, and auxiliary building load, open pit mine load, ancillary loads, and an allowance for future nominal growth / changes of auxiliary loads over time.

18.11.2 Power Generation

Diesel power generation is to be used as the primary power source during construction of the mill and mine site. Diesel power generation during pre-production will accommodate the camp and construction facilities.

Grid power will not be available until after Production year 2 of the project. It is envisioned that generator power will be used in stages for the first 4 years of the project starting at 1.8 MW, ramping up to 9.6 MW during production year 1 and 2. These generators will be rented for this period and decommissioned after the installation of grid power.

18.11.3 Main Substation & Site Power Distribution

The main substation will include the following equipment:

- Substation transformer skid 10 MVA 115 kV/5 kV and accessories;
- A substation E-house (modularized, assembled and tested off site) complete with 2000A power distribution center, protective relaying, generator synchronization and load shedding equipment;
- Substation P&C, synchronization and network cabinets
- 4160v switchgear

The power distribution center will distribute power to the site. Power to the mill substation will route through underground trenches, while power to the remote gate house, ancillary buildings and to the open pit mine will route through kV O/H line. The mill substation will be equipped with pad mounted step-down transformers, while power to remote loads will be stepped-down with pole mounted transformers.

18.11.4 Mill Substation

The mill substation will include two 4160/600V 1.5 MVA outdoor oil filled step-down transformers with secondary 600V 2000A power distribution centers feeding motor control centers to provide utilization voltages for mill process and utility equipment.

The power distribution centre and motor control centers will be housed in a E-house (modularized, assembled and tested off site) and will provide power to:

- Crushing, conveying loads;
- Grinding area, and ball mills;

- Floatation area;
- Reagents, thickening and filter areas;
- Tailings, reclaim water, fresh water and ancillary services.

Plant equipment utilization voltages are provided in Table 18-3.

Table 18-3 Power Utilization Voltages

Plant Equipment	Voltages
All motors / VFDs including ball mills	600 volt three-phase
Small drives below 0.5 HP	120 V one-phase
Electrical heaters over 2 kW	600 V three-phase
Electrical heaters up to 1.8 kW	120 V one-phase
Lighting – LED	120 V one-phase
Small power & instrumentation	120 V one-phase
Welding receptacles	600 V three-phase

18.12 Site-Wide Communications

The mine site will employ a site-wide communications system based on a single mode fiber optic backbone. VOIP telephones, intranet/internet access, and control system network connectivity will be integrated into this fiber backbone so that these systems can be accessible anywhere on site. Broadband internet access will be purchased from a satellite internet service provider. The corporate network (intranet) will be isolated from the control system network via a firewalled DMZ (de-militarized zone) network.

18.13 Warehouse, Offices, Facilities, and Services

Warehouse, offices, facilities, and services will include the following:

- Gate house;
- 100 person modular camp
- Four-bay heavy and light vehicle truck shop;
- Truck wash and lube;
- Emergency vehicle and first-aid centre;
- Warehouse/cold storage;
- Assay laboratory;
- Administration office;
- Mine dry;
- Fuel storage and dispensing;
- Process water system;
- Potable water system;

- Sanitary system;
- Water treatment;
- Fire protection;
- Waste management and disposal;
- Auxiliary equipment fleet;
- Emulsion plant and storage areas;
- Sea container;
- Generator set housing; and,
- Network of site access roads.

18.14 Concentrate Process Facility

18.14.1 Location

At the PEA level study management of Granite Creek decided to construct the process plant at the proposed processing complex.

18.14.2 Site Services

There are multiple possible locations at the Port of Skagway area that are existing brownfield sites with all required services including shipping access. The exact location for the facility is not determined as further study is required to select the optimal location.

18.14.3 Buildings

The Concentrate Process facility will consist of:

- Ore Storage (Stockpile)
  - This area will receive the ore from the mine and stockpile it for feed into the processing plant.

- Process Plant
  - The plant will process the ore to produce the concentrate (copper, gold, and silver)
  - All reagents required will be stored within the plant facility.
  - All offices, laboratories, lunch/washrooms, and warehouse areas will be located within the plant facility.

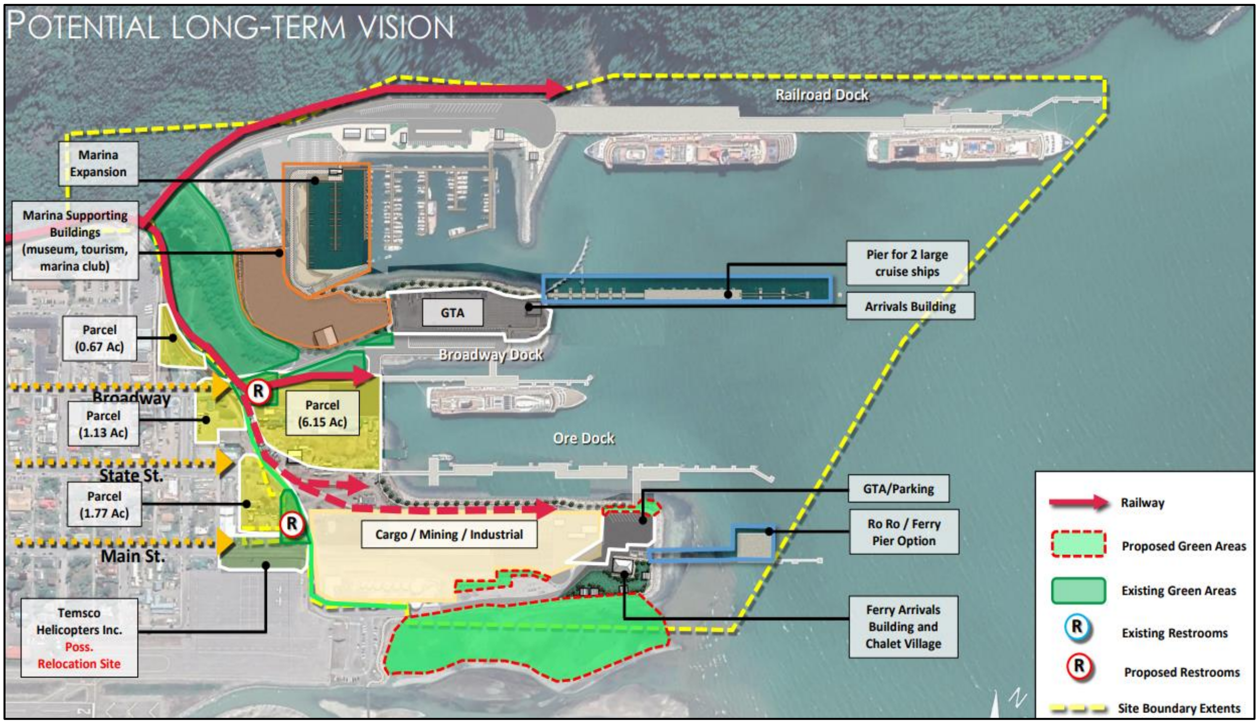
- Finished Product Storage Area
  - This area will store the finished concentrate final product in bags or drums prior to shipment for further processing.

- Marine shipping

- Access to the port is available through gravel and well paved road with a total distance of 395.94 km from the mine to Skagway's port
- Long term expansion plan by the Municipality of Skagway to provide flexibility for growth of diverse set of waterfront businesses-cargo, mining, fuel, commercial etc. (Figure 18-5) shows the mining cargo berth for concentrate shipment abroad for further processing.

- Communications
  - Telephone and internet services will be available from local suppliers in the area.

Figure 18-5 Potential Long-Term Vision



19 MARKET STUDIES AND CONTRACTS

19.1 Market Study

Granite Creek Copper or its consultants have conducted no market study on the sale of copper concentrate. Therefore, the market terms for this study are based on a review of current market conditions and discussions with Granite Creek Copper and recently published terms from other similar studies. The QP is of the opinion that the marketing and commodity price information is suitable to be used in cashflow analyses for a PEA level of study.

The concentrate will be trucked from the project site to a port storage facility in Skagway, Alaska, where it will be loaded onto bulk material carriers. The concentrate will be transported by sea to clients. Concentrates will be sold into the general market to North American, European, or Asian smelters and refineries.

19.2 Commodity Price Projections

For this technical report, the metal prices presented below in Table 19-1 were used for financial modelling. The metal prices are long-term forecasts over three years provided by an analyst consensus long-term forecast and as agreed by NorthWest Copper.

Table 19-1 Metal Price Assumptions

Copper	Pound (lb)	USD $3.75
Gold	Troy ounce (ozt)	USD $1,800
Silver	Troy ounce (ozt)	USD $22.00

For this PEA, it is assumed all metals are 100% payable. No studies were conducted to determine effects of any deleterious material, nor is any indicated in current testwork.

Overall shipping costs were assessed as $102 CAD per tonne.

Treatment and refining costs for copper were assessed from recently published studies. Copper treatment and refining is assessed at CAD $0.14 per pound copper, gold refining at CAD$4.26 per ozt and silver at CAD$0.31 per ozt.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

The Carmacks Project area lies within the Klondike Plateau and is part of the Pelly River Ecoregion (Oswald and Senyk 1977), which is comprised of portions of the Stewart, Macmillan, Lewes, and Klondike Plateaus and Tintina Valley physiographic subdivisions (Bostock, 1970). Surface drainage flows both north and east from the study area. A number of valley streams, of which Williams Creek is the largest, drain northeastward to the Yukon River.

Environmental baseline conditions on and around the Carmacks Project site have been documented in numerous surveys conducted since 1989. The terrestrial and aquatic resources, current land uses, and heritage resources potentially affected by the project are summarized below, followed by an overview of project effects on the natural and human environments.

20.1.1 Terrestrial Resources

20.1.1.1 Vegetation

The project area is predominantly forested with approximately 97% of the project area in forest cover. Black spruce is the dominant forest community type (58%), with Lodgepole pine (20%) and White spruce (17%) each approximately equally represented (Access 2007a). Trembling aspen forest (2%), willow fen (2%), and grassland (1%) are minor vegetation community types.

20.1.1.2 Wildlife

Environment Yukon has identified key wildlife areas for important wildlife species occurring in the territory. These areas may be important in one or more stages of a species' life history, such as winter range, calving/lambing, salt licks, or summer nesting habitat and are considered important to the long-term management of the species. The mine site is located well outside any key wildlife area. Key wildlife areas occurring in the general vicinity of the project site include summer breeding habitat for golden eagles located in the northern portion of the project study area, overlapping lower Williams Creek and the adjacent Yukon River; and winter range for moose south of the main project development area that includes the area of the mine access road corridor but does not include the mine site or immediate surrounding area (Access 2007b). The powerline corridor crosses the Yukon River and passes through the golden eagle habitat. The corridor route was surveyed for golden eagle and other raptor nesting sites during the 2013 nesting season so the final powerline alignment will avoid any raptor habitat.

In general, the project is located in a low-density moose survey block in which moose occur year-round in low numbers (PAH 1993, HKP 1995, Access 2007b, Markel and Larsen 1988, O'Donoghue et al. 2008a and 2008b).

The project area occurs outside the known range of wood bison with no known permanent occupancy in the area (PAH 1993, Access 2007b). The project area also is well west and north of key habitat areas for wood bison. Black bears are common in the project area.

Grizzly bears are much less abundant, but grizzlies have been observed along the Freegold Road and the Yukon Quest trail (PAH 1993, Access 2007b).

The Little Salmon/Carmacks First Nation (LSCFN 2011) has identified the Yukon River between Tatchun Creek and Minto as important habitat for moose, salmon, and other wildlife. This reach of the river includes several sloughs and islands and provides important calving, summer range, and winter range habitat for moose. Moose were commonly observed through this reach in the 1960s, but are less commonly observed currently, perhaps due to increased river travel traffic during summer. Hunting does not appear to be responsible for the reduced frequency of moose observance. Few people are hunting along the river and licensed harvests are low. Dog Salmon Slough is another important habitat area located approximately 2.5 km downstream of the confluence of Williams Creek with the Yukon River. Bears use this area for fishing. None of the project elements interact with these areas.

20.1.2 Aquatic Resources

20.1.2.1 Surface Water Hydrology

The project site is located in the Williams Creek watershed, a local tributary of the Yukon River. The watershed is comprised of two sub-basins, Nancy Lee Creek and Williams Creek. The entire project site is located in the Williams Creek sub-basin.

Nancy Lee Creek drains approximately 44 km^2, flowing into Williams Creek approximately 1 km from the Yukon River. Williams Creek drains approximately 42 km^2 upstream of the confluence with Nancy Lee Creek, with approximately 2 km^2 contributing to flows below the confluence. Williams Creek discharges to the Yukon River approximately 40 km northwest of the village of Carmacks. The Yukon River above the mouth of Williams Creek drains approximately 90,600 km^2. Based on the ratio of drainage areas, the Williams Creek watershed accounts for approximately 0.1% of the total Yukon River flow below the confluence.

Flows were monitored on Williams and Nancy Lee creeks periodically between 1991 and 1994 (HKP 1995), and from 2006 through the 2012 open water season (Access 2013). Flows are highly seasonal, typically peaking in May during freshet and then dropping to steady state flow maintained by a combination of baseflow and precipitation runoff in June through September, and finally dropping to baseflow only in October through to the next freshet. Baseflow in upper Williams Creek above the confluence with Nancy Lee Creek is estimated at <0.02 m^3/sec (1,500 m^3/day) compared to average steady state flow of 0.3 to 0.6 m^3/sec (25,000 to 50,000 m^3/day) and average freshet flow of 3.4 m^3/sec (293,000 m^3/day) (Golder 2012b).

20.1.2.2 Hydrogeology

The understanding of the groundwater system on and around the project site has been developed through the monitoring of numerous wells on the project site, and particularly in the vicinities of the planned open pit, WRSA, and formerly considered the heap leach facility (HLF). Pump tests, determinations of hydraulic conductivity, and monitoring of piezometric levels informed the development of an updated FEFLOW groundwater model for the project site (Golder 2012c). Additional investigation of hydrogeology conditions in the area of the proposed TSF is planned for spring 2017 and this information will be incorporated into an updated groundwater model for the site.

The general project area is characterized by a regional groundwater flow system within bedrock. Groundwater is recharged by precipitation at higher elevations in the upland areas and flows toward the valleys of Nancy Lee Creek, Williams Creek, Merrice Creek, and the Yukon River. Overall, the water table mimics ground surface topography and the depth to groundwater generally increases with increasing ground surface elevation. Based on groundwater levels in the monitoring wells, the depth to groundwater in the previously proposed WRSA ranges from 2 m (near Williams Creek) to 50 m. In the vicinity of the proposed open pit, the depth to groundwater exceeds 91 m. The presence of permafrost may have resulted in the development of perched water tables in some areas; however, these are assumed to be isolated and discontinuous. The permafrost likely acts as a barrier to infiltration in some areas, thereby reducing recharge and potentially resulting in the overall depression of the regional water table. In the vicinity of the mine site, groundwater flow direction is toward Williams Creek and maintains baseflow in the creek. Mine site development, operation, and closure therefore only have the potential to affect groundwater reporting to, and affecting baseflow in upper Williams Creek (Golder 2012c). Peak flows are not expected to be measurably affected.

20.1.2.3 Surface Water Quality

Background surface water quality on and around the Carmacks project site has been extensively characterized, with monitoring conducted periodically beginning in 1989. The most recent monitoring program operated from 2005 through the 2012 open water season, with 11 monitoring stations located on Williams Creek and its tributaries and two stations on the Yukon River, 100 m upstream and 300 m downstream of the mouth of Williams Creek (Access 2013). Yukon typically manages water quality through application of the CCME water quality guidelines appropriate to the water use being protected (aquatic life, drinking, recreation, or agriculture) with protection of aquatic life being the focus for the local streams and the adjacent Yukon River. The applicable British Columbia guideline may be used in place of the CCME guideline in cases where the BC criterion is considered more appropriate to the local conditions. Site Specific Water Quality Objectives (SSWQO) are developed for locations where background concentrations of one or more parameters typically exceed the established guideline.

The local surface waters are slightly alkaline, with mean Ph at all monitoring stations between 7.6 and 8.1. Consistent with the alkaline Ph, the waters of the Williams Creek watershed are well buffered.

Mean total alkalinity is highest on the Williams Creek mainstem and on Nancy Lee Creek. Similar patterns of spatial variation are evident in hardness, sulphate concentration, specific conductance, and total dissolved solids. The spatial patterns of alkalinity and hardness are likely a reflection of the relative contribution of groundwater to baseflow, with the higher concentrations indicating a greater groundwater influence. In the Williams Creek watershed and in the Yukon River near the mouth of Williams Creek, most parameters consistently occur at concentrations below the applicable CCME or BC Guideline for the Protection of Freshwater Aquatic Life. Several parameters occasionally exceeded the applicable guidelines in the 393 samples collected between 2005 and 2012, including: cadmium (in three samples), lead (in two samples), silver (in six samples), and zinc (in one sample). Aluminum, copper, and iron concentrations frequently (Al in 28% of samples, Cu in 11%, and Fe in 36%) exceeded the applicable guidelines and SSWQOs have been proposed for these parameters using the Background Concentration Procedure (CNMC 2016).

20.1.2.4 Sediment Quality

Stream sediment quality was initially sampled in July 1992 and then in each of the 2005, 2006, and 2007 open water seasons (Access 2008). Parameters analyzed included: Ph, Al, As, Cd, Cu, Fe, Pb, Ni, Se, and Zn. Sediment Ph was circumneutral at all locations. Arsenic, cadmium, lead, and zinc concentrations were below the respective CCME Interim Sediment Quality Guidelines (ISQG) at all locations in the watershed. Mean Al concentrations ranged between 6,525 and 9,191 µg/g, with no evident trend regarding the location in the watershed. Mean Fe concentrations ranged between 14,355 and 23,725 µg/g, also with no evident spatial trend in the watershed. Mean Ni concentrations ranged between 2.6 and 18.5 µg/g, with the highest concentrations occurring near and below the confluence with Nancy Lee Creek. Selenium concentrations were typically below the reportable detection limit throughout the watershed. No CCME guidelines have been set for Al, Fe, Ni, or Se in sediment.

20.1.2.5 Fish and Fish Habitat

Several fish surveys have been conducted in Williams Creek, Nancy Lee Creek, Merrice Creek near the access road crossing, and at the mouth of Williams Creek at the Yukon River (August 1991 and August 1992 (PAH 1993); Oct 2005; June 2006; July/August 2006; and, September 2006 (Access 2007c); July and

September 2009 (Access 2010b)). Fish have consistently been found in the section of Williams Creek below the confluence with Nancy Lee Creek (i.e., at stations W10 and W12) but have not been found at any other location in Williams Creek or Nancy Lee Creek. Fish captures or observations upstream of station W12 have been limited to the capture of a single slimy sculpin at W13 in October 2005 and the observation of a single adult grayling in the pool at W11 in July 2009. No fish have ever been captured upstream of station W13 and no fish have been found in the reach of Merrice Creek where the access road crossing is located, consistent with the absence of fish from the upper reaches of Williams Creek. Fish are thought to avoid moving up Williams Creek because of the consistently colder water than in the lower creek and in the Yukon River. Benthic invertebrate standing stocks also are much lower in upper Williams Creek. Lower Williams Creek, below the confluence with Nancy Lee Creek, provides rearing habitat for fish during the open water season. Species found in lower Williams Creek include juvenile Chinook Salmon, Arctic Grayling, Slimy Sculpin, Longnose Sucker, Burbot, and Northern Pike.

20.1.2.6 Species at Risk

Species at risk with the potential to occur in the project area include:

- Threatened – wood bison, peregrine falcon (Anatum subspecies);
- Special Concern – grizzly bear, wolverine, short-eared owl; and,
- At Risk in Yukon but not elsewhere – mule deer, elk, cougar.

The project area does not provide critical habitat to any life stage of these species and is not expected to adversely affect any of these species (Access 2006).

20.2 Current Land Uses

20.2.1 Commercial and Industrial

The Carmacks copper property is comprised of 373 mineral claims and 20 leases, all of which are 100% owned by Granite Creek. Site activities to date have included access road and exploration camp construction, exploration drilling and trenching, environmental baseline studies, and limited site preparation in the form of forest clearing from portions of the area previously designated for a HLF. The Granite Creek exploration camp is operated seasonally under a Class 3 Quartz Mining Exploration Permit. There is no commercial forest harvest activity in the project area due primarily to the low timber values and distance from markets. The mine site, access road, and powerline corridor west of the Yukon River are located within Registered Outfitting Concession #13, and the powerline corridor on the east side of the Yukon River is in Registered Outfitting Concession #14. The holder of Concession #13 has indicated the project area is not generally hunted.

20.2.2 Traditional and Cultural Land and Resource Use

The property is located within the Traditional Territories of the LSCFN and the Selkirk First Nation (SFN).

The late summer/fall Chinook and Chum salmon spawning runs on the Yukon River support aboriginal food and commercial fisheries. Members of the LSCFN fish at many sites along the Yukon River between Carmacks and Fort Selkirk as well as at sites upstream of Carmacks. Fishing locations vary annually depending upon flow conditions on the river. Most fishing is along the mainstem of the Yukon River, although traditional fishing may at times occur at the mouth of Williams Creek. Some sport fishing may also occur at the mouth of Williams Creek as recreational canoeists make their way down the river to Dawson City.

The mine site, access road, and powerline corridor west of the Yukon River are located within Registered Trapline #147. The powerline corridor east of the Yukon River is located in Registered Trapline #143. Trapline production statistics are not publicly available. Expected harvest includes mink, beaver, fox, marten, squirrel, lynx, coyote, and wolverine. The property is part of the LSCFN traditional hunting grounds. The LSCFN collects native plants for medicinal and traditional purposes throughout the region. The property does not provide a unique source of any plants used by LSCFN.

20.2.3 Settlement Land and Land Claim

None of the project components or activities is located on settlement lands and any nearby settlement lands are held by LSCFN. The closest settlement lands downstream of the project, LSC S-30B1, are located approximately 4.5 km downstream of the mouth of Williams Creek. Settlement lands LSC R17-B are situated on the east bank of the Yukon River approximately 4.8 km downstream of the mouth of Williams Creek. Six LSCFN settlement land parcels occur adjacent or near to the Freegold Road.

There is one land claim selection located near the project. LSCFN has selected parcel R-9A west of the project site. This parcel extends into the project environmental assessment study area but does not include any of the mineral claims or leases or any of the areas in which project activities are proposed. The land selection is upstream of any project components or activities and is not expected to be affected by the project.

20.2.4 Heritage Resources

Archaeological impact assessments were conducted in the Williams Creek valley for the proposed project in August 1992 (Antiquus 1993) and along potential powerline routings in 1994 (Antiquus 1995) and in 2013 (EcoFor 2015). In 1998, Hare (1999) conducted archaeological assessments at several locations in southern Yukon in connection with various development proposals and community requests. Thomas (2006) also examined heritage resources in the area of McGregor Creek as part of the Heritage Resources Impact Assessment conducted in preparation for the Carmacks Stewart/Minto Spur Transmission line. The collective findings of these studies are that no archaeological sites occur on the proposed mine site; five sites occur along the powerline corridor (three sites near the confluence of McGregor Creek and the Yukon River and two sites adjacent to Williams Creek between Nancy Lee Creek and the Yukon River); and there are two locations with medium heritage resource potential on the planned access road corridor, at the crossings of Merrice and Williams creeks. All of the sites along the powerline can be avoided through minor adjustment to the line alignment within the corridor. Site specific surveys of the stream crossing locations will be conducted during detailed design and the final crossing locations adjusted as necessary on the basis of these survey findings to avoid interaction

20.3 Environmental and Social Effects

20.3.1 Terrestrial Resources

The project is not expected to have significant adverse effects on terrestrial resources due to the small footprint, absence of critical wildlife habitat and the absence of vegetation species at risk.

20.3.2 Aquatic Resources

The project site and all infrastructure in the Williams Creek watershed are located well upstream of any waterbodies that directly provide habitat for any fish species. The closest fish habitat on Williams Creek is located more than 3.5 km downstream of any site development, below the confluence with Nancy Lee Creek. Similarly, the existing and proposed bridge crossings of Merrice Creek are located upstream of any fish bearing waters. No direct interaction with fish habitat is part of any phase of the project plan and no

Fisheries Act authorizations are required. Potential effects of the project on aquatic resources therefore are related to how the project will affect the quantity and quality of water leaving the site. Effects on quantity are limited to the Williams Creek sub-watershed, and arise from: surface runoff management on the mine site, groundwater withdrawals from the water supply wells, open pit dewatering for mining, and then open pit filling after mining has been completed.

A FEFLOW groundwater model has been developed for the site to examine the effects of project plans on groundwater conditions (Golder 2012c) but the model has not yet been updated to incorporate the project changes since the 2012 FS. The model indicated local groundwater flows maintain baseflow in the adjacent upper Williams Creek and that project-related effects on groundwater flows would reduce baseflows in upper Williams Creek. The model will be updated during subsequent project planning phases prior to finalizing the YESAB Project Proposal. Four sources of contact water will be managed during operations: groundwater seepage and precipitation pumped from the open pit sump; seepage and surface runoff from the WRSA; surface runoff from the TSF; and excess process water from the process plant.

Water inflow to the open pit will originate from precipitation, runoff from the local contributing watershed, and groundwater seepage.

20.4 Socio-Economic Effects

An assessment of the socio-economic effects of the project was completed in 2007 for the previous project proposal to YESAB (Vector Research and Research Northwest 2007). This assessment indicated the project as then proposed would not have any adverse socio-economic effects on local communities or Yukon as a whole, and there were several identified significant positive effects associated with the project. The changes to the project detailed in the present study are not expected to alter these findings. Specific issues are examined below.

20.4.1 Commercial Land Use

There is limited commercial land use activity in the project area, currently amounting to occasional commercial hunting. The only concern related to the project is the potential of increased bear control actions related to site management. Granite Creek plans to manage the mine site and overall project operation to minimize the attraction of bears, and therefore the requirement for bear control actions. This approach is both for protection of site personnel and the bears. No adverse effects on commercial land use is expected to occur during project development, operations, closure, or post-closure.

20.4.2 Traditional Resource Use

The primary traditional resource use in the project area is trapping. The mine site, access road, and powerline west of the Yukon River are located on Registered Trapline #147. The powerline and substation east of the Yukon River are located on Registered Trapline #143. Granite Creek will work with the RTL holders to ensure access to lines is maintained, portions of trapline trails that are disturbed by project elements are relocated, overall effects are minimized, and non-mitigable effects appropriately compensated. No effects on fishing success are expected to occur during project development, operations, closure, or post-closure. Similarly, no effects on hunting success are expected to occur during project development, operations, closure, or post-closure.

20.4.3 Recreational Land Use

The project is not expected to affect recreational land use. No recreational uses will be displaced by the project and the mine site will not be visible from the Yukon River, an important recreational waterway.

20.4.4 Community Engagement

Engagement with the local communities related to the Carmacks Copper Project has been undertaken in several periods since interest in developing the deposit was first expressed in 1991. The local stakeholder communities include the LSCFN, the SFN, and the village of Carmacks. The project is located on the traditional resource areas of both First Nations and primary project access passes through the Village.

The first period of community engagement extended from 1991 to at least 1997, and included public meetings, as well as exchanges of technical documents and correspondence. The project did not fully complete environmental permitting at that time, instead being put on hold due to market conditions. Interest in project development returned in 2004, when then owner Western Silver.

initiated enquiries into the permitting process in consideration of legislative changes since the initial project submission. The communities were again engaged using a combination of meetings, information sessions, and exchanges of technical documentation and correspondence in 2005, 2006, and 2007, and the public process of both the Yukon Environmental and Socio-Economic Assessment review in 2007/2008 and the Yukon Water Board review in 2009/2010. Concerns expressed by the communities were primarily related to the potential environmental effects of the project and, in particular, the post-closure effects. Since formation in October 2011, Granite Creek has been working on project design changes to address the environmental concerns of the local communities and to communicate these changes to the communities. All technical documentation submitted to the YG agencies is also provided to the communities; a public open house information session was held in the village of Carmacks in August 2012 and an information sharing meeting was held with LSCFN administration in August 2012. Granite Creek signed a letter of intent with LSCFN in December 2012, which initiated consultation on the project and its potential environmental and socio-economic effects. Funding was provided to LSCFN by Granite Creek to enable the First Nation to conduct an independent technical review of the project plans, which will continue through the project re-engineering and subsequent permitting.

Granite Creek has continued community engagement throughout the project, and is continually addressing concerns through the Yukon Environmental and Socio-Economic Assessment Act. Communication is ongoing via personal communications with the communities, written responses to concerns and postings to the Yukon Environmental and Socio-Economic Assessment Board. Details of the engagement may be found at www.yesab.ca.

20.5 Permits

Major hard rock mining projects in Yukon are required to satisfy a two-step regulatory review and approval process before mining activity may commence. The first step is an environmental and socio- economic assessment conducted in accordance with the Yukon Environmental and Socio- Economic Assessment Act (YESAA) which is administered by the Yukon Environmental and Socio- Economic Assessment Board (YESAB). The YESAA review typically takes from 9 to 18 months to complete, depending on the project, the issues, and the need for supplementary information beyond that initially submitted by the proponent.

The second step is the regulatory phase involving two enabling licenses, the Quartz Mining License (QML) and the Water Use License (WUL). The QML process is administered by Yukon Energy, Mines, and Resources (EMR) and the QML regulates the following mining related activities:

- The area and mineral deposits to be mined;
- Allowable mining and milling rates;
- Pre-construction plans and drawings;
- Post-construction as-built drawings;
- Monitoring programs;

- Design of mine workings, including underground and open pit development and production, and
- waste dumps;
- Site infrastructure, including buildings, roads, fuel storage, etc.;
- Solid waste disposal;
- Reclamation, including slope stability, erosion control, and re-vegetation;
- Financial security; and,
- Annual reporting requirements.

The WUL process is administered by the Yukon Water Board and regulates the use of water, the deposit of waste into water, receiving water quality, and all water conveyance and retention structures associated with a development. Any WUL issued for the project will set limits on the quality and quantity of discharges to water and on the quantities of any surface or groundwater takings. The WUL also will set monitoring and reporting requirements for surface and ground waters, for water discharges, and for water management structures such as dams, dykes, and ponds. A Type B WUL would be necessary for the project construction phase, in order to provide the necessary water to supply the construction camp and other construction activities as well as for the sanitary septic system. A Type A WUL will be required for project operation, involving the taking of process water, pit dewatering, and the discharge of any treated water.

The environmental assessment phase must be completed and a positive decision (i.e., an approval) issued by YESAB before the regulatory phase of permitting can be completed. Yukon EMR will review a QML submission in advance of a YESAB decision but cannot issue a QML until the decision document for the YESAA review has been issued. With a QML application developed and submitted in advance of a YESAB decision, the QML decision can proceed quickly following a positive decision by YESAB. The Yukon Water Board does not review a WUL license application until the YESAA process is complete and a decision document issued, and the WUL review process can take several months, particularly for a Type A license review which also requires a public hearing.

The project, as it was previously proposed in 2007, received a positive environmental and socioeconomic assessment determination from YESAB in 2008 and a Quartz Mining License in 2009. The nature of the project changes proposed in the present plan are such that that the project proposal must again pass an Executive Committee Level Environmental Screening Assessment before the regulatory phase, in which the QML is either amended or a new QML is issued and a WUL issued. Much of the project information and potential environmental effects have already been reviewed by YESAB, as part of the previous copper-only, project submission which should assist in expediting the next YESAB review.

20.6 Schedule

The project schedule has been developed on the basis of the expected schedules for the project environmental assessment and permitting and the time required for project engineering and ordering of long lead capital equipment. The schedule assumes that some construction can proceed on the basis of the existing and valid QML, along with a Type B WUL that would allow operation of the construction camp. Other factors that may potentially affect the project schedule are project financing and metal prices.

Accordingly, Granite Creek plans to initiate pre-feasibility engineering with an emphasis on metallurgical testing and related engineering costing studies for refinement of the copper and gold/silver recovery processes.

Development of the updated YESAB Project Proposal would occur in parallel with the feasibility study work, leading to a submission to YESAB. Based on discussions with YG and YESAB, the YESAB review is expected to take approximately 9 to 12 months to complete given that much of the information was assessed in the previous YESAB review.

The construction schedule is based on starting construction of project elements covered by the existing QML

20.7 Water Management Plan

The climate observed at the project is defined by distinct seasons. In winter (October to April), precipitation is accumulated as snow. Peak flows occur during the freshet month corresponding to the snowmelt in May. Steady state flows are then established during the remaining months (June to September). Net water production from the site can either be used as process make-up water or must be managed for off-site discharge to local receiving waters.

The conceptual water management plan has been developed to manage water from the following site facilities:

- Open pit;
- TSF;
- WRSA; and,
- The process plant site.

The current water management plan builds on the plan developed by Golder (2012b) for the copper leach project as it was previously detailed in the 2012 FS. Water will be managed to minimize discharges to Williams Creek by supplementing freshwater requirements in the process plant with site water. The water management strategy is summarized below by project phase.

20.7.1 Phase 1: Operations

The operations period extends from the start of mining until the completion of metal recovery. Mineralized material will be mined for approximately 10 years, and copper, gold, and silver recovery will continue for a short period thereafter, for a total project life of approximately eleven years.

The process plant contains the copper and gold flotation, recovery, and concentrating facilities. The metallurgical process will initially be supplied by fresh water drawn from groundwater, but ongoing operation will largely be supplied by water reclaimed from the process streams or from the tailings management area sediment pond (TSFSP) and the WRSASP. Fresh make-up water will only be necessary for reagent make-up and gland water.

Seepage from TSF underdrains and surface runoff diverted around the TSF will be managed as non-contact water and will be directed through an unlined settling pond for discharge to North Williams Creek. Runoff from the TSF will be managed as contact water and will be collected in the TSFSP. The collected water can be reclaimed to the process or directed to the WRSASP for management and discharge.

Runoff and seepage originating from the WRSA is collected in the WRSASP. Precipitation, local runoff, and groundwater seepage will collect in the pit sump. The pit sump refers to the combined water contained in the pit bottom sump as well as in the satellite pit once that becomes available for water storage. Water from the pit sump will be pumped to the WRSASP. Water may be reclaimed to the process from the WRSASP or will be discharged to North Williams Creek. Water quality in the WRSASP will be monitored during operations and will be released if the quality is acceptable for discharge, if water quality is not acceptable for discharge the water will be directed to the HDS water treatment plant for treatment before release.

The following additional contingencies have also been built into the operations water management strategy:

- If an operational process plant shutdown occurs, water can be pumped from the WRSASP to the open pit to prevent discharges from these facilities; and
- The SSP provides a third level of containment for the copper and gold leach circuits. In the event of a major leak in a leach circuit the leakage would be contained by the underlying liner. The collected leakage could then be pumped to the SSP to allow access to the leach circuit for repairs. The leakage transferred to the SSP could then be returned to the process stream.

Following the cessation of mining, pit dewatering will be discontinued, and pit flooding will commence. The site-wide water balance developed for the previous project plan will be updated and optimized during subsequent project design phases.

20.7.2 Phase 2: Closure

For purposes of this PEA, progressive reclamation is envisioned throughout operations, primarily for the TSF. As pits 1213N and 1213S are mined out, reclamation activities will be conducted during operation of the 147 Stage 2 pit. The initiation of the project closure period corresponds to the shutdown of the process plant. Some decommissioning activities will be initiated earlier as part of the progressive reclamation plan (e.g., WRSA and TSF), while the remainder of site facilities decommissioning will commence at this time. Water management during closure is similar to Phase 1, with the exception that pumping from the open pit will cease and no water will be reclaimed to the process plant. Surface runoff and seepage from tailings in the TSF will be directed through the TSFSP to the WRSASP. Underdrain seepage and water diverted around the TSF will continue to be discharged directly to North Williams Creek. Runoff and seepage from the WRSA will report to the WRSASP. Water collected in the WRSASP will be monitored for quality and will either be discharged directly to North Williams Creek or will be treated prior to discharge. Previous water quality modelling determined that runoff and seepage from the WRSA would not require treatment following closure (Golder 2008b). Further geochemical testing and modelling of the TSF runoff and seepage is in progress and, until otherwise determined, active treatment has remained part of the post-closure water management plan. A key component of the closure plan for the project is the transition of the active treatment plant into a passive treatment system. For the current assessment, it was assumed only active water treatment would occur during the closure phase while the passive treatment system is implemented. This phase was considered to have a duration of two to three years.

20.7.3 Phase 3: Post-Closure

Post-closure will begin when the passive treatment system becomes fully operational, receiving all water from the WRSASP. During the post-closure period, no active water management will occur. Outflows from the passive treatment system will drain to North Williams Creek and from there to the Williams Creek mainstream.

20.8 Closure and Reclamation

All quartz mines in Yukon are required to have an approved Closure and Reclamation Plan and agreed upon financial security in place prior to starting operations. There is a current approved plan in place for the project based on an earlier project design, but this plan will need to be modified to reflect the project design changes described in this report. Closure plans are then reviewed and updated at two-year intervals through construction and operation to ensure the plan reflects the project as it is developed and to account for progressive reclamation measures that would reduce the final overall closure cost.

The updated conceptual closure plan for the project as described in this PEA is summarized below by major project component mine reclamation.

20.8.1 Open Pit

Closure of the open pit will involve the removal of all equipment and installations, blocking of access to the pit ramp with boulders, placement of a boulder fence along accessible sections of the pit rim, and erection of signage to warn of the open pit hazard. Once mining is complete dewatering will be terminated and the pit will be allowed to gradually fill, creating a pit lake over a period of 200 yearsTailings Management Area

Closure planning for the TSF is at the conceptual stage, consistent with the design stage of the facility. The closure concept involves creating a stable re-vegetated landform from the TSF and the abutting WRSA. The slopes of the TSF are built to 4H:1V during operation to allow progressive reclamation of the landform. Therefore, the closure landform is expected to have the same approximate footprint and size as the ultimate TSF landform. Post-decommissioning earthworks are expected to be limited to the construction of swales on the upper TSF surface and integration of the drainage with the WRSA. Drainage paths will be designed to limit down-slope flow distances and reduce surface erosion.

Progressive reclamation during operations reduces future reclamation costs and enhances environmental protection. Progressive reclamation is also valuable in establishing which closure measures will be effective during permanent closure. This can reduce the length of the active care closure phase.

Testing to date indicates that acid generation and metal leaching are not a concern for the waste rock. It has been assumed that this will also be true of the filtered tailings. Therefore, an evapo-transpirative type cover may not be required. Instead, the objective of the cover would be to minimize erosion, and facilitate establishment of a vegetative cover that is consistent with the final land use and harmonious with the surrounding environment.

Organic material stripped from the area before mining and stockpiled will be re-spread on the TSF surface. The organic material will be initially seeded with native seed mixtures to minimize erosion. In closure, revegetation of the general site, TSF and WRSA would follow the general guidelines for reclamation in the Yukon.

20.8.2 Water Treatment

Water treatment during operations and closure will be carried out in the water treatment plant. The plant will incorporate a HDS treatment circuit for management of metal concentrations, The HDS circuit may be comprised of more than one treatment train to allow the effective treatment of the variable water flows expected over the course of project operations. This will be examined in detail in later design stages. A passive treatment facility (PTS) will be constructed and commissioned during closure and will be progressively brought online during the closure period, with all water treatment carried out in the passive facility by the end of the closure period.

The PTS will handle surface runoff and seepage from the closed TSF and WRSA. The design for the PTS will be developed during subsequent design phases of the project.

20.8.3 Waste Rock Storage Area

The WRSA will be developed so that a minimum of slope re- contouring is necessary for closure; slope grading on bench surfaces will be maintained and operational slopes will be established and maintained at a stable 2.25H:1V slope. Closure will involve placement of 0.3 to 0.5 m of organic soils on the flat bench areas. Soil will be sourced from the overburden stockpiles. Lodgepole pine will be seeded on areas facing south and west and white spruce will be seeded on areas facing north and all soil placement areas will have an initial seeding of native grasses to control erosion while the seeded and native trees become established. Slopes will not be seeded. Surface runoff collection ditches and the sediment control pond (WRSASP) will be maintained as long as necessary to control sediment in WRSA runoff – typically until vegetation is well-established on the WRSA.

Geochemical testing to date has indicated the rock to be placed in the WRSA is not acid generating and is not a metal leaching source concern, so a cover to control infiltration is not necessary. In consideration of the expected runoff quality determined in humidity cell tests, the WRSASP overflow will be directed to Williams Creek at closure. The expected WRSASP overflow quality will be verified by monitoring prior to directing the discharge to surface waters. The GoldSim water quality model results indicate that treatment of this discharge source is not expected to be necessary in order to protect receiving water quality (Golder 2012d). The mine plan, waste rock quantities and properties have not changed since the GoldSim model was developed. However, the model will need to be updated to incorporate the TSF.

20.8.4 Other Mine Site Facilities

The general approach to closure and reclamation of the other site facilities and infrastructure is to:

- Remove equipment from the site that is no longer required, typically for sale or salvage;
- Remove supplies form the site that are no longer needed – either returned to the supplier for credit or sold;
- Remove, dismantle, or demolish (as appropriate) buildings and structures – for sale, salvage, recycling of key components, or disposal, either on-site or off-site;
- Survey and remediation of all areas of soil contamination;
- Demolition of foundations to grade;
- Grading to stabilize slopes, maintain natural drainage patterns, and fit with the natural local topography;
- Cover of pads, and other disturbed areas as needed, with overburden to support vegetation; and
- Scarification of other areas and seeding of all disturbed areas to locally appropriate vegetation.

All facilities not required for reclamation and water treatment purposes will be dismantled and removed during the closure period.

Final closure of the landfill and solid waste facility will require the filing of a final closure plan to the Yukon Government (YG) documenting the contained materials and the conditions of the facility. Prior to final closure, any hazardous materials will be removed to a licensed handling facility and salvageable materials (metal, tires) may be recovered for salvage/recycling. Final closure will involve coverage with two compacted lifts (each 200 mm thickness) of soil, grading for drainage, and seeding.

Closure of the land treatment facility also is subject to the submission of a formal closure plan to the YG, including sampling results to document the final concentrations of contaminants in the soils being treated. Once contaminant levels have been reduced to regulated concentrations, the treated soil can be removed from the facility and used for site reclamation. The land treatment facility will be one of the last facilities to be closed on the site to ensure there is the capacity to properly manage any soil contamination identified in the course of site closure.

The power line will be removed, and the right-of-way reclaimed once line power to the site is no longer required. Reclamation will be limited to contouring and vegetation of disturbed areas. These costs are included in the capital cost estimate.

20.8.5 Roads

Roads used for exploration, for access to the site (access road), and access around the site (site roads) will be decommissioned and reclaimed once they are no longer required. Granite Creek expects that the final disposition of the access road will be determined in consultation with the local communities and the Yukon Government. For the purpose of this study the closure cost estimate includes costs for reclamation of the 13 km site access road. The general closure approach for roads is to ensure physical stabilization of the surface, natural drainage is not impeded (i.e., culverts removed, and adjacent banks are stable), and locally appropriate vegetation is established along the cleared right-of-way. Site roads will be reclaimed during closure. Culverts will be removed, and slope surfaces re-contoured for stability and to reflect the natural local topography.

Surfaces will be scarified and re-vegetated. Exploration trails typically require minimal contouring and stabilization. Any side-cast material will be recovered, trenches backfilled, and the trail left to natural re-vegetation.

The exploration trail currently used to access the project site is not under company authority and consequently, it is not a Granite Creek closure responsibility. Costs for access trail closure have not been included in the closure cost estimate.

20.8.6 Closure Costs

Reclamation costs are estimated to be $5.6M. Any closure plan filed with and accepted by regulatory agencies would include costing based on third party contracting of the works, as required by Yukon closure regulations. Typical activities include:

- Removal of all surface infrastructure and buildings;
- Closure and capping of the TSF (performed progressively through the mine life until final;
- closure post-operations);
- Access road closure;
- Power transmission line and substation removal;
- Re-vegetation and seeding; and
- Ongoing site monitoring.

21 CAPITAL AND OPERATING COSTS

The capital costs for this PEA are estimated by SGS, using a combination of vendor quotes, recent project costs from the SGS Bateman database and published costs from industry reports. Operating costs are calculated from first principal calculations and vendor quotes (mining). The estimate base is December, 2022 and all costs are shown in $CAD.

21.1 Capital Cost Estimate

For this Preliminary Economic Assessment, the accuracy of the capital and operating cost estimates is considered to be ± 40%.

Cost components are provided in a combination of USD and CAD. Final totals are presented in CAD for financial analyses.

The exchange rate used for this study is $0.75 USD = $1.00 CAD.

No provision has been made for cost escalation or exchange rate fluctuations.

Fuel costs used for this PEA are $1.30 CAD.

Capital cost estimates for the identified disciplines have been estimated as per the following areas.

Mining

Mining contractors provided quotes for equipment supply and operation. Management and technical services are provided by the company. Benchmarking was also conducted to compare the quality of the quotes received and found that the quotes are suitable for this level of study.

Mining indirect capital includes equipment for site services for clearing, road construction and maintenance, and company support for mining and construction not covered under the process plant estimate.

Mining indirect labour for the construction period includes technical services and contract management support for mining.

Sustaining capital captures the costs of developing the 1213 pits and the 147 Stage 2 pits after production on 147 Stage 1 pit commences.

Processing

SGS Bateman provided capital and operating cost estimates for the process plant based on:

- Design criteria
- Equipment lists
- Process Flowsheet
- SGS engineering equipment database for recent similar projects
- Budget quotations from vendors for major equipment
- Engineering drawings performed by SGS

Sustaining capital for the process plant was not factored as the current envisioned life of mine doesn't warrant capital replacement of large equipment. Operating costs (discussed later) capture replacement of wear items in the process plant.

Site Power

Site power will be provided initially by generator sets with grid power available by production Year 3. Grid power costs and associated site distribution are placed under sustaining capital due to the longer lead times for supply.

Generator power costs include budgetary estimates for equipment rental and fuel consumption.

Non process infrastructure power is estimated from generator rentals and fuel consumption for camps and offices, until grid power is available.

Earthworks

Earthworks for clearing and developing the plant site are included in process plant capital. Earthworks for other site facilities are included as allowances in the cost estimates for each area (camps, offices, fuel storage, warehousing, mine maintenance and warehousing).

Tailings

Initial capital estimates for tailings management are taken from previous studies and escalated with inflation to 2022. An allowance for a tails filter system is included to allow for a dry stack facility.

General and Administration

Pre-production G&A is an estimated cost based on camp operation during construction, company management support, site services.

External Infrastructure

The following surface facilities are estimated under non process infrastructure:

- 100 person camp
- Site Water and Fuel Supply
- Tailings Deposition (dry stack assumed)
- Non Process Site Buildings and Infrastructure
- Access Roads and Transportation
- Grid Power and Site distribution
- Concentrate Storage Skagway Alaska
- Site Communications / IT

Table 21-1 presents the total capital costs for each area.

Table 21-1 Summary of Project Capital Costs

Area	Initial Capital $CAD	Sustaining Capital $CAD
Mining Directs	$13,961K	$120,202K
Mining Indirects	$7,167K	
Processing Directs	$83,445K	
Processing Indirects	$34,103K	
Non Process Infrastructure	$16,932K	
Tailings	$14,665K	
Pre-Production G&A	$3,370K	
Power Supply	$11,160K	$3,751K
Closure Costs		$5,850K
Contingency	$35,264K	
Total	$220,066K	$129,803K

21.1.1 Processing Capital Cost Estimate

A summary of the initial pre-production capital costs for the process plant and associated infrastructure is shown in Table 21-2 This table includes direct costs, indirect costs, and a 30% contingency. This capital cost was based on equipment cost and multiplied by factors for installation.

Table 21-2 Process Capital Estimate

Carmacks Project			
SUMMARY FACTORED COST ESTIMATE WORKSHEET		**Project No:**	**531-01**
Factored Cost Estimate $CAD		**Revision:**	**P2**
		Date:	**February 2, 2023**
Description	Purchased Equipment Costs	Method / Factor	**Installed Cost**
DIRECT COSTS			
Process Plant :	-		
Area 10 - Primary Crushing	*$5,017K*	Equipment Cost x 1.60	*$8,026K*
Area 15 - Grinding	*$14,492K*	Equipment Cost x 1.60	*$23,187K*
Area 20 - Cu Flotation and Regrind	*$7,435K*	Equipment Cost x 1.60	*$11,896K*
Area 30 - Concentrate Dewatering Thickening and Filtration	*$1,518K*	Equipment Cost x 1.60	*$2,429K*
Area 40 - Tailing Thickening Dewatering	*$1,467K*	Equipment Cost x 1.60	*$2,347K*
Area 50 - Reagents	*$1,533K*	Equipment Cost x 1.60	*$2,453K*
Area 55 - Water System	*$805K*	Equipment Cost x 1.60	*$1,288K*
Area 60 - Plant Air	*$380K*	Equipment Cost x 1.60	*$608K*

Purchased Equipment Cost	**$32,646K**	***Installed Equipment Cost***	**$52,234K**
General Site Development		(5% of Purchased Equipment Costs)	*$1,632K*
Process and overland piping on site		(15% of Purchased Equipment Costs)	*$4,897K*
Buildings (process and non-process)		Estimate	*$11,623K*
Electrical power distribution on site		(40% of Purchased Equipment Costs)	*$13,058K*
Site Development Cost			*$31,211K*
TOTAL DIRECT COSTS			***$83,445K***
PLANT INDIRECT COSTS			
EPCM		(17% of Direct Cost)	*$14,186K*
Construction Indirect Costs incl:		(12% of Direct Cost)	*$10,013K*
Construction Supervision			
Equipment Rental			
Field Office Expense			
Mobilization / Demobilization			
Owner's Costs		(5% of Direct Cost)	*$4,172K*
Spare Parts		(5% of Purchased Equipment Cost)	*$1,632K*
Initial Fill & Reagents		(1% of Direct Cost)	*$834K*
Equipment Insurance & Freight Cost		(10% of Purchased Equipment Cost)	*$3,265K*
TOTAL INDIRECT COSTS			***$34,103K***
TOTAL DIRECT AND INDIRECT			***$83,445K***
CONTINGENCY 30%		(30% of Direct and Indirect)	***$35,264K***
TOTAL CAPITAL			***$152,812K***
Exclusions			
Geotechnical	Environmental Costs		
Mining / Heap leach	Permits, Royalties and Licenses		
Reclamation and Closure Costs	Local Sales & Import Taxes		
Metallurgical Testing	Hazardous Waste Removal		
Property Acquisition Cost	Other Consultant Costs		
Permitting Costs	Off-Site Costs		
Access Road Costs			

21.1.1.1 Direct Costs

The direct capital costs were based on the following list of documents prepared by SGS:

- Design criteria
- Equipment list
- Process Flowsheet
- SGS engineering equipment database for recent similar projects
- Budget quotations from vendors for major equipment
- Engineering drawings performed by SGS

The direct costs exhibited in this estimate include, but are not limited to, labor, equipment and materials for the detailed construction activities set forth below:

21.1.1.2 Equipment Costs

An equipment list was developed and incorporated into the cost estimate. The estimate for equipment was developed from the following sources:

- Written or e-mailed budgetary estimates from vendors for major equipment.
- Historical data and budget costs from recent similar projects for miscellaneous equipment.

The cost for "Installed equipment" was estimated using a factor of fifty percent (50%) of purchased equipment costs. This factor reflects typical local costs to install equipment and covers labor, concrete foundations, steel, and other services and construction materials associated with equipment foundations, erection, and placement.

21.1.1.3 Site Development

General site development costs include excavations, backfills, grading, roads, and fencing within the process plant area. The initial construction site development cost was estimated using a ten percent (10%) factor of the purchased plant equipment cost. The factor was selected based on the nature of the proposed project site, and the type of native soils in the area. The project will require development at the following major locations:

- On site access roads
- Primary crusher area
- Stockpile area
- Process plant areas

21.1.1.4 Buildings (Process And Non-Process)

Building costs include materials, labor, and other miscellaneous costs associated with erecting covered structures within the project site. The initial construction building cost was estimated using current costs per unit area This method was selected to reflect the projected costs of the buildings based on building type. The project will require the following buildings:

- Primary crusher building
- Pebble crusher building
- Grinding building
- Floatation and filter building
- Product storage building
- Control rooms and offices

21.1.1.5 Electrical Power Distribution At Process Plant

Electrical distribution costs include transformation, wiring, cable tray, instrumentation, lighting, and grounding within the process plant and on-site power distribution to ancillary facilities and mining surface facilities. The initial electrical cost was estimated using forty percent (40%) of purchased process equipment costs. The factors were selected based on preliminary equipment power requirements of about 12.7 MW for the process plant. Allowances were made for power to ancillary facilities for about 1.7 MW and to mining surface facilities for about 1.5 MW.

The project will require the following electrical power distribution items:

- 12.5 kV distribution on site (switchgear, distribution lines)
- Pad-mounted distribution transformers
- Medium voltage (4 kV) switchgear
- Low voltage (575 V) motor control centers (MCC)
- Back-up diesel generators

21.1.1.6 Main Electrical Power Supply

The main electrical power supply will be from the Canadian grid system. The costs required by the utility are included in infrastructure below.

21.1.1.7 Off Site Costs

Offsite costs for concentrate storage at Skagway Alaska are included in infrastructure below

21.1.1.8 Water Source

The cost of the delivery of water from the water source is included in infrastructure below. Distribution within the process plant is included in the process plant estimate.

21.1.1.9 Indirect Costs

Certain indirect costs exhibited in this estimate include, but are not limited to, labor, equipment and materials for the detailed activities set forth below:

EPCM for the process facilities and associated infrastructure was estimated using 17% of the direct costs and includes the following:

- Feasibility study
- Detailed engineering
- Procurement Construction management
- Construction indirect costs for the initial construction and mill expansion were estimated using a twelve percent (12%) factor of the total direct costs and includes:
- Construction supervision
- Equipment rental
- Field office expenses
- Mobilization / demobilization
- Consumables

Owner's costs for the plant are estimated at five percent (5%) of direct costs.

Spare parts costs were estimated using a five percent (5%) factor of the purchased plant equipment cost. Initial fill and reagents costs were estimated using a one percent (1%) factor of the total direct costs.

Equipment insurance and freight costs were estimated using a ten (10%) percent factor of the purchased plant equipment costs.

21.1.1.10 Process Plant Contingency and Accuracy

The SGS crushing and process plant portion of the cost estimate includes a 30% contingency for project unknowns and identified risks. Contingency is a necessary part of the cost estimate since less than three percent (< 3%) of the engineering has been completed to date. SGS believes the estimated contingency amount will be spent during the construction period of the plant site and associated infrastructure for identified risks and unknown items.

While SGS has not performed a statistical analysis of the process plant accuracy of the capital cost estimate. SGS has confidence that the accuracy of the process portion of the estimate will end up between minus twenty-five percent and plus thirty-five percent (-25 / +35%) of the SGS capital cost estimate.

21.1.1.11 Exclusions from the Process Plant Cost Estimate

SGS has excluded the following cost items from the process plant estimate and assumed these are included in other sections of the report:

- Geotechnical
- Utility connections and metering to site

- Tailing filtration
- Off-site costs
- Mining
- Downstream processing
- Reclamation and closure
- Metallurgical testing
- Property acquisition
- Permitting
- Environmental
- Permits, royalties and licenses
- Taxes, duties, and import fees
- Local sales and import taxes
- Hazardous waste removal
- Other consultants

21.1.2 Tailings Storage Facility Cost

The estimated initial capital cost of the TSF is $14.665M CAD (refer to Table 21-3 for details). The estimate includes site investigation, dam construction cost, and construction personnel.

An allowance is of CAD $7M is used for a tailings filtration and conveying system to provide dry stack material.

Rental of a portable crusher is covered under mining indirects. Aggregates are available from mined waste rock crushed on site. Among other uses on site, the aggregates will be used for placement of under drains on the tailings storage facility.

Table 21-3 Tailings Storage Capital Costs

Description	Capital Cost
Mobilization/Demob (Lump sump)	$397 K
EPCM (Lump Sum)	$1,268 K
Earthworks	$1,884 K
Granular Material / Rip Rap	$224 K
Drainage and Piping	$1,756 K
TSF Liners	$2,137 K
Tailings Filtration Allowance	$7,000 K
Total Tailings Storage Costs	**$14,665 K**

21.1.3 Site Closure Costs

An allowance of $5.00M is made for site closure, assuming progressive closure is conducted by the mining operations and site services throughout the mine life.

A further allowance of $850K is provided for decommissioning of the power line to site.

21.1.4 Mine Capital Costs

21.1.4.1 Mine Direct Capital Costs

Direct mining capital costs are shown in Table 21-4.

Direct Mining costs include contractor mobilization, direct pit grubbing and clearing, pit waste development and pre-production ore (stockpiled) prior to processing production. Direct support equipment for dewatering is also included.

Table 21-4 Mine Direct Capital Costs

Area	Capital Cost
Contractor Mobilization	$500K
Pit Clearing and Overburden Removal	$2,220K
Pit Waste Stripping	$3,224K
Haul Road Construction	$870K
Pit Pre-Production Fuel	$3,753K
Pit Dewatering	$1,200K
Mobile Crusher Rental	$969K
Fuel Storage	$724K
NSR Buyout	$500K
Total Pre-Production Direct Mine Capital	**$13,961K**

21.1.4.2 Mine Indirect Capital Costs

Mine Indirect Capital costs include pre-production costs of company labour, including pre-production management and technical services. Pre-production indirect labour is estimated at 25% of annual total operational indirects, at $979K.

Equipment purchased for pit and site operations is also included in the mine indirects. This equipment is shown in Table 21-5

Table 21-5 Mine Support Equipment (Indirect Mine Capital)

Equipment	Quantity	Cost Each	Total Cost
Grader	1	$2,450 K	$2,450 K
Water Truck	1	$1,100 K	$1,100 K
Fuel Lube Truck	1	$966 K	$966 K
Mechanic Truck	1	$160 K	$160 K
Welding/Crane Truck	1	$135 K	$135 K
Pickup Truck	5	$53 K	$267 K
Light Plant	6	$17 K	$101 K
Forklift for Warehouse	1	$375 K	$375 K
Backhoe Loader	1	$250 K	$250 K
Ambulance	1	$250 K	$250 K
Skid Loader	1	$135 K	$135 K
	Total Mine Support Equipment Capital		**$6,188 K**

21.1.5 Initial Infrastructure Capital Costs

Site Infrastructure capital includes costs for a 100 person camp, mine buildings, maintenance facility, power generation, propane storage, site preparation, access road, site power distribution, discharge water treatment, site landfill / waste management, site communications / IT and offsite storage concentrate.

Grid Power is under sustaining capital due to availability in production year 3.

Table 21-6 Site Infrastructure Initial Cost

Site Infrastructure	Cost
Site Preparation	$4,477K
Site Access Road	$4,777K
Site Buildings (Non Process)	$3,642K
Water Treatment	$2,249K
Landfill / Waste Management	$619K
Propane Storage	$550K
Site IT / Communications	$182K
Site Power Supply	$11,160K
Off Site Concentrate Storage	$436K
Total Initial Infrastructure Capital	**$28,091K**

21.2 Basis of Operating Cost Estimate

The accuracy of the operating costs estimates is considered to be ± 40%.

Costs are presented in $CAD. Operating costs provided by SGS Bateman are converted from USD at an exchange rate of $0.75:$1.00 USD:CAD.

Fuel pricing used in this PEA is $1.30 per litre.

No provision has been made for cost escalation or exchange rate fluctuations.

21.2.1 Summary of Total Operating Cost Estimate

The overall operating costs for the Mill Site process plant, mining operation and G&A are in Table 21-7. This table shows the overall operating costs per tonne of material processed and cost per tonne mined (all tonnes).

Table 21-7 Site Infrastructure Initial Cost

Summary Costs	Life of Mine	Per Tonne Processed	Per Tonne Mined
Mining	$373,459K	$17.56	$3.16
Process	$389,322K	$18.30	
G&A	$104,819K	$4.93	

21.2.2 Mine Operating Cost Estimate

The mine operating costs average $29.1M per year for direct mining costs and $2.6M per year for indirect mining costs. Indirect mining costs include the company labour required for mine operations.

Quotes were obtained from contractors based on the initial pit shell designs, and average costs are provided below in Table 21-8 (Fuel excluded):

Table 21-8 Overall Operating Costs (CAD)

Overall Operating Costs (CAD)	Cost per Tonne Mined
Overall Operating Costs (Capital Excluded)	$3.37
147 Stage 1 & 2	$2.97
1213 N & S	$3.23
Weighted Average Operating Cost (including Capital Stripping Cost)	$3.16

The contractor quotes included:

- Equipment and labour for loading, hauling, drilling and blasting
- Contractor equipment rental rates
- Contractor capital recovery on the rentals
- Fuel consumption based on 5,600 operating hours per year
- Factors for maintenance, ground engaging tools, lubricants and tire consumption

Averages of the quotes were used to determine costs per tonne mined for load, haul, drill and blast. These values were used to determine annual costs. The contractor provided estimated fuel consumption. This was converted to a value per tonne and then factored against the price per litre supplied.

The contractor provided average fuel usage per year based on initial designs. This was converted to litres per tonne mined, and costed using $1.30 per litre. The overall fuel consumption is estimated at 0.17 litres per tonne, or $0.22 per tonne.

A suggested fleet is provided in Table 21-9. Direct and indirect personnel are provided in Table 21-10. Contractor personnel are based on a 20 days on – 10 days off schedule, while mine indirect personnel are shown on a 5 day on 2 day off schedule.

Table 21-9 Projected Mining Fleet

Fleet	Year									
	-1	1	2	3	4	5	6	7	8	9
Cat 6015 Shovel		2	2	1	2	2	2	2	1	
Cat 375 excavator	1	1	1	1	1	1	1	1	1	1
Cat 775 Haul Truck		9	7	3	8	8	9	9	3	
Cat 740B Articulated Truck	2			2		5	5	3		3
Cat 992 Wheel Loader	1	1	1	1	1	1	1	1	1	1
Cat MD 6250 Blasthole Drill	1	1	1	1	1	1	1	1	1	1
Cat D9 Dozer	1	1	1	1	1	1	1	1	1	1
Cat 14M Grader	1	1	1	1	1	1	1	1	1	1
Cat 775 Fuel Truck	1	1	1	1	1	1	1	1	1	1

Projected Personnel Complements are shown in Table 21-10.

Table 21-10 Mine Personnel Complement

	Year									
Mining Personnel	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**
Contractor Based										
Contractor Supervision	3	6	6	6	6	6	6	6	6	3
Shovel Operators	0	6	6	3	6	6	6	6	3	0
Excavator Operators	3	3	3	3	3	3	3	3	3	3
Truck Operators	6	27	21	15	24	39	42	36	9	9
Loader Operators	3	3	3	3	3	3	3	3	3	3
Drill and Blast Personnel	6	6	6	6	6	6	6	6	6	6
Dozer Operators	3	3	3	3	3	3	3	3	3	3
Grader Operators	3	3	3	3	3	3	3	3	3	3
Fuel Truck Drivers	3	3	3	3	3	3	3	3	3	3

Maintenance Personnel	9	12	12	10	15	15	15	15	9	6
Total Contractor Personnel	**39**	**72**	**66**	**55**	**72**	**87**	**90**	**84**	**48**	**39**
Mining Indirects (Company)										
Mine Superintendent	1	1	1	1	1	1	1	1	1	1
Mine Captains / General Foreman	2	2	2	2	2	2	2	2	2	2
Technical Services Superintendent	1	1	1	1	1	1	1	1	1	1
Senior Geologist	1	1	1	1	1	1	1	1	1	1
Field Geologists	2	2	2	2	2	2	2	2	2	2
Sampling Technicians	4	4	4	4	4	4	4	4	4	4
Mine Planning Engineer	1	1	1	1	1	1	1	1	1	1
Mine Surveyors	2	2	2	2	2	2	2	2	2	2
Total Mine Indirects (Compmany)	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**
Total Mine Complement	53	86	80	69	86	101	104	98	62	53

Total mining costs are provided in Table 21-11.

Table 21-11 Summary of Mine Operating Costs

		2026	2027	2028	2029	2030	2031	2032	2033	2034	
	Year	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**Totals**
Mining											
Waste Mining	tonnes	10,804K	8,742K	2,888K	11,998K	2,698K	2,324K	13,308K	3,346K	31K	56,139K
Mined Resource Tonnes	tonnes	2,998K	2,558K	3,058K	2,555K	2,565K	2,424K	2,282K	2,556K	275K	21,271K
Total Tonnes Mined		13,802K	11,300K	5,946K	14,553K	5,262K	4,748K	15,590K	5,903K	307K	77,409K
Costs											
Mining Direct Costs		$39,807K	$33,579K	$18,041K	$47,074K	$17,020K	$15,350K	$47,132K	$17,540K	$2,975K	$239,724K
Mining Indirects		$2,558K	$2,558K	$2,558K	$2,558K	$2,558K	$2,558K	$2,558K	$2,558K	$639K	$21,102K
Total Cost		**$43,571K**	**$36,137K**	**$20,598K**	**$49,631K**	**$19,578K**	**$17,908K**	**$49,690K**	**$20,098K**	**$3,614K**	**$260,826K**

The average mining cost per tonne during operations is $3.37 CAD. Contractor quotes indicated lower costs for the larger 147 Pit. The overall cost per tonne including capital stripping for all pits averages $3.16 per tonne CAD.

Costs for Ore mined during the pre-stripping stage in Year -1 are included in Year 1 operating costs. This material goes to stockpile and is processed at the end of life of mine.

21.2.3 Process Plant Operating Cost Estimate

General Information

Process operating costs are built up from estimated reagent consumption based on metallurgical testwork, power consumption and labour costs. Plant Maintenance is estimated at 10% of capital cost of installed capital equipment. Plant labour also includes operations personnel, supervision, maintenance personnel and Plant Management.

Overall operating costs are provided in Table 21-12 below:

Table 21-12 Process Costs by Year and Per Tonne

By Discipline	Year 1-2			Year 3+		
	CAD $/yr	CAD$/tonne	% of OPEX	CAD $/yr	CAD$/tonne	% of OPEX
Reagents	$4,587K	$1.87	12.4%	$4,587K	$1.87	13.10%
Power	$5,142K	$2.10	13.9%	$3,231K	$1.32	9.23%
Maintenance & Spare	$2,747K	$1.12	7.4%	$2,747K	$1.12	7.84%
Grinding media and liner wear	$13,995K	$5.71	37.9%	$13,995K	$5.71	39.96%
Process Labor	$9,390K	$3.83	25.4%	$9,390K	$3.83	26.81%
Mobile equipment	$1,073K	$0.44	2.9%	$1,073K	$0.44	3.06%
Total	**$36,934K**	**$15.08**	**100%**	**$35,023K**	**$14.30**	**100%**

Processing costs were developed by SGS Bateman and are based on first principles. Consumption rates for diesel, power, reagents, and mill consumables were estimated. Contingency amounts were added in each area based on values from the SGS database and published literature.

Power consumption is estimated from equipment listed on the conceptual flowsheet. The power rate for the first two years of operation is $0.39 per kw-hr for the first two years of operation, including generator rental and diesel costs at $1.30 per litre.

When grid power becomes available, this rate drops to $0.25 per kw-hr, including consumption and demand charges.

Maintenance and spares are factored from the original equipment supply cost at a factor of 7.5%. A further contingency amount of 15% is added onto this.

Griding media and liner wear is estimated from test work and values in the SGS database. A further contingency of 30% is added onto the calculated values.

Annual operation of plant mobile equipment is an allowance of$930k per year with a contingency of 15% for a total of $1,073k per year.

Treatment charges are based on benchmark costs from other published reports and applied in Table 21-3. The treatment charges and concentrate shipping are based on concentrate containing 40% copper.

Table 21-13 Treatment and Refining Charges

Treatment Charges	Per Pound Copper	$0.08
Refining Charges Copper	Per Pound Copper	$0.06
Refining Charges Gold	Per Ounce Gold	$4.26
Refining Charges Silver	Per Ounce Silver	$0.31

Concentrate shipping charges of $102 CAD per tonne concentrate are applied. This cost is based on previously published reports, factored up to date by inflation.

Plant Labour

The labour rates used for the process plant OPEX estimate have been based on labour rates from the SGS Database and other current published reports.

The annual salary includes payroll burdens of 22%. Payroll burdens include benefits, workers compensation, employment insurance and Canada Pension Plan amounts.

Table 21-14 Process Plant Complement and Payroll Costs

Position	Complement	Annual Salary	Total Costs Per Year
Plant Superintendent	1	$248K	$248K
Maintenance Superintendent	1	$214K	$214K
Shift Foreman	4	$160K	$641K
Metallurgist	1	$183K	$183K
Crushing Operator	8	$130K	$1,037K
Crushing helper	4	$92K	$366K
Grinding operator	8	$130K	$1,037K
Flotation operator	8	$130K	$1,037K
Concentrate and tailing dewatering	4	$130K	$519K
Reagent operator	4	$130K	$519K
Shift labor	8	$92K	$732K
Lab supervisor	1	$160K	$160K
lab technician/sampler	8	$122K	$976K
Mechanic	4	$130K	$519K
Mechanic Helper	4	$114K	$458K
Electrician	2	$130K	$259K
Electrician Helper	2	$114K	$229K
Instrumentation Technician	2	$130K	$259K
Total Staff	**74**	**Annual Cost**	**$9,390K**

21.2.4 Tailings Handling Costs

The processed tails facility is assumed to be a dry stack system, with tailings delivered, placed and compacted. Previous studies identified annual operating costs, which have been factored with inflation to an average of $1,808K per year.

21.2.5 General and Administration Expenses

General and Administration expenses are comprised of the following components:

- Company management and Administrative Personnel
- Non Process Facilities
- Security and First Responders
- Warehousing
- Site Services

Non Process facilities include camp operation for a 100 person camp, water supply and treatment, generator rental and fuel (years 1-2) and power (Year 3+).

An allowance of $750k per year is made for site services (Site maintenance, licence fees, insurance, office supplies).

Table 21-15 provides the breakdown of General and Administrative Costs.

Table 21-15 General & Administrative Cost Breakdown

G&A Components	Year 1-2	Year 3+
Labour	$4,043K	$4,043K
Travel (Camp Based Personnel)	$2,880K	$2,880K
Non Process Facilities	$5,638K	$3,961K
Propane Supply	$550K	$550K
Site Services / Office Supplies / Misc. Expenses	$750K	$750K
Annual Costs Total	**$13,860K**	**$12,184K**
Costs per Tonne Processed	**$5.42**	**$4.77**

Company personnel are shown in Table 21-16.

Table 21-16 General & Administrative Complement and Payroll Costs

Position	Complement	Annual Salary	Total Costs Per Year
General Manager	1	$264K	$264K
Administration Assistant	1	$92K	$92K
Controller	1	$198K	$198K
HR manager	1	$198K	$198K
SHE Manager	1	$198K	$198K
Accounts Payable	2	$92K	$183K
Payroll	1	$92K	$92K
Contract Administration	1	$92K	$92K
IT Support	1	$92K	$92K
HR Assistant	1	$92K	$92K
Purchasing Manager	1	$159K	$159K
Purchasing assistant	1	$92K	$92K
Warehouse Shipping Receiving	6	$85K	$512K
Enviro Techs	2	$110K	$220K
Security / First Responders	8	$92K	$732K
Site Services Operators	8	$104K	$830K
Total Staff	**37**	**Annual Cost**	**$4,043K**

22 ECONOMIC ANALYSIS

22.1 Introduction

The economic analysis contained in this report is based on Measured, Indicated and Inferred

Mineral Resources. The economic analysis of Project therefore is based on Inferred Mineral Resources and is preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have mining and economic considerations applied to them and cannot be categorized as Mineral Reserves.

The economic evaluation of the Carmacks Copper Project as presented in this PEA and prepared SGS assumes the project will be 100% equity financed.

This economic evaluation uses parameters relevant as of January, 2023.

Mining and treatment data, capital cost estimates and operating cost estimates have been put into a base case financial model to calculate the internal rate of return (IRR) and net present value (NPV) based on calculated project after tax cash flows. The scope of the financial model has been restricted to the project level and as such, the effects of interest charges and financing have been excluded.

The model includes sensitivity analyses to demonstrate the effect of variations in key parameters on the economic returns from the project.

Estimated project returns and the key financial statistics are summarized and discussed in this section and are supported by tables and charts. A summary of the financial model results for the project is included in Table 22-1.

22.2 Basis of Economic Analysis

The analysis has been conducted on a pre-debt financing basis. Escalation and inflation have been excluded.

The currency adopted for this analysis is the Canadian dollar ($CAD) unless otherwise stated.

In calculating the returns from the project, the following fundamental assumptions have been made:

- The operating life of the project will be approximately 9 years;
- The design throughput for the project is 2,45,000 t/a,
- The economic returns are assessed at the project level on a pre-financing basis;
- The evaluation includes a 24-month project development period prior to the commencement of production, a total of approximately 11 years;
- The exchange rate used to convert CAD into USD is 16.07 to 1.33 ($0.75USD = $1.00CAD

All assumptions made as part of the economic evaluation are detailed in Section 22.5.

Base case economics are developed based on metal prices of USD$3.75 per pound for copper, USD$1,800 per ounce for gold and USD$22 per ounce for silver.

A second case (Case 1) of economics used metal prices of USD$4.25 per pound for copper, USD$2,000 per ounce for gold and USD$25 per ounce for silver.

22.3 Summary of Results

The key project statistics for the life of the project are summarized in Table 22-1 below.

Table 22-1 PEA Key Parameters

Parameter	Unit	Base Case[1]	Case 1
Metal Prices			
Copper Price	US$/pound	$3.75	$4.25
Gold Price	US$/troy ounce	$1800.00	$2000
Silver Price	US/troy ounce	$22.00	$25
Recovery to Cu Concentrate			
Copper Recovery - Sulfide	%	92%	
Copper Recovery – Blend	%	64%	
Copper Recovery – Oxide	%	55 %	
Gold Recovery	%	58%	
Silver Recovery	%	60%	
Concentrate Grade			
Copper %	40%		
Gold g/t	11.0 g/t		
Silver g/t	134.4 g/t		
Production Data			
Resource Tonnes	21,270,518		
Copper Equiv. Grade	1.10%		
Daily Mill Throughput	Tonnes / day	7000 t	7000 t
Annual Processing Rate	Mtonnes/ year	2,45 Mt	2,495 Mt
LOM Strip ratio	Waste: Ore	4.6:1	4.6:1
Mine Life		9 years	9 years
Average annual production			
Copper EQ production[5]	Million Pounds / year	33.9 M	
Copper in concentrate	Million Pounds / year	27 M	
Gold in copper concentrate	Troy ounces / year	12,385	
Silver in copper concentrate	Troy ounces / year	151,584	
Operating Costs (LOM avg)[2]			
Mining	C$/t mined	$3.16	
Milling	C$/t processed	$18.30	

G&A	C$/t processed	$ 4.93	
All in Sustaining[3,4]	US$/lb CUEQ	$ 2.57	
Capital Costs			
Initial Capital	C$220M		
LOM Sustaining Capital	C$130M		
Financial Analysis			
Pre Tax NPV 5%		C$324M	C$475M
Pre Tax IRR		36%	48%
After Tax NPV 5%		C$230M	C$330M
After Tax IRR		29%	38%
Pay back period (Years)		2.0	1.5

1. Base case metal prices based on 36-month trailing average from January 15, 2023
2. Total operating costs include mining, processing, tailings, surface infrastructures, transport, and G&A costs.
3. AISC includes cash operating costs, sustaining capital expenses to support the on-going operations, concentrate transport and treatment charges, royalties and closure and rehabilitation costs divided by copper equivalent pounds produced.
4. AISC is a non-IFRS financial performance measures with no standardized definition under IFRS. Refer to note at end of this news release.
5. Copper equivalents are calculated by total gross revenue divided by the copper price or in detail from the following formula: ((copper lbs x copper price)+(Au ozt x Au price) + (Ag ozt x Ag price))/(copper price)

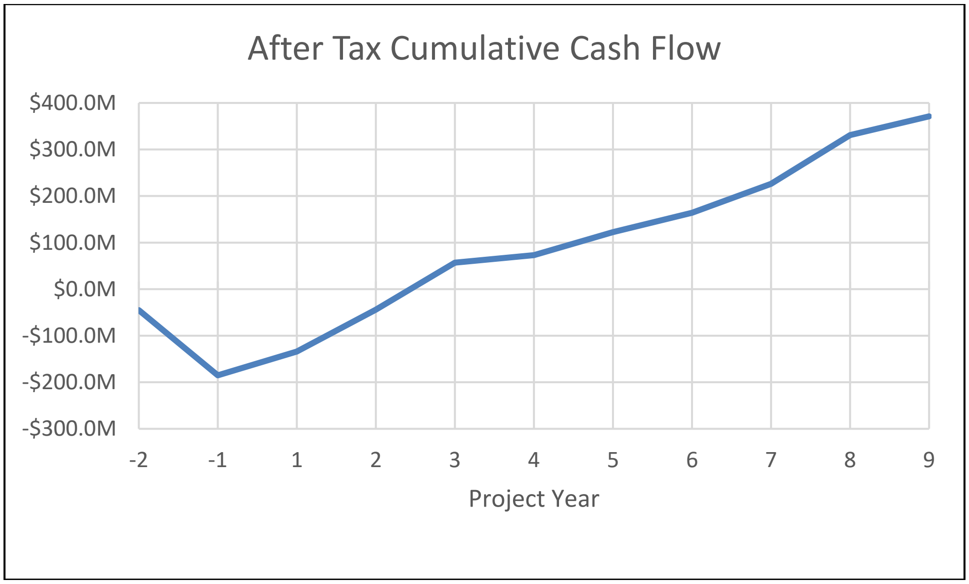
22.4 Project Economics

Based on the nominal extraction of 2,450,000 t/a of ROM feed from the mine, the project is anticipated to yield a pre-tax IRR of 36% with a pre-tax NPV, at a discount rate of 5% CAD $324.1M, and an after-tax IRR of 29% with an after-tax NPV, at a discount rate of 5%, of CAD $230.5 M. Cumulative cash flows are CAD $505.8 M pre-tax and CAD $371.2 M after-tax over the nine year LOM.

The project is expected to pay back initial capital in 2 years after production starts.

Figure 22-1 shows the cumulative after-tax cash flow over the total project life.

Figure 22-1 Cumulative Cash Flow (After Tax)



22.5 Clarification and Assumptions

Economic analysis has been carried out on the basis detailed within this section.

22.5.1 Analysis Period

The period of analysis is from the commencement of Process plant construction during quarter one, Year -2 until the end of mining through to the last sale of product in Year 9.
Revenue

The calculation of production tonnages is based on the mining and treatment schedules. The recovery factors for copper, gold and silver are based on available testwork and the mined proportions of oxide and sulphide copper. The overall average copper recovery for the life of mine is 64%, average gold recovery is 58% and average silver recovery is 60%.

22.5.2 Operating Costs

Operating costs have been estimated as per the following functional headings:

- Mining;
- Mill Site Process Plant;
- General and Administration.

A full description of operating costs can be found in Section 21 of the Report.

22.5.3 Capital Costs

Capital costs include the direct capital costs the process plant, non process infrastructure and tailings storage facility; sustaining capital for pre-stripping of pits after the initial capital strip at 147 Stage 1; mine and tailings facility closure costs; indirect costs (including EPCM, first fills, spares and a construction camp allowance) and a contingency. Sustaining capital for the process plant was not factored due to the project life, however replacement equipment is factored into the process operating costs.
A full description of capital costs can be found in Section 0 of the Report.

22.5.4 Funding

The financial model for the project is presented on the basis of pre-financing cash flows and as such excludes the impact of both debt funding and equity finance.

22.5.5 After Tax Free Cash Flow

After-tax free cash flow is calculated by deducting operating costs, royalties, taxes and sustaining capital expenditures from revenue.

A third party royalty currently exists, with a payout option of $500,000 prior to production. This payout option is captured in mining pre-production capital.

Yukon royalties are calculated on a sliding scale, with allowable deductions for operating costs, site development and depreciation.

Depreciation for federal and territorial taxes are based on Canadian Development Expenses, Canadian Exploration Expenses and Undepreciated Capital Costs as defined by Canada Revenue Agency. Taxation rates applied are a federal rate of 15% of free cash flow and 12% for the territorial tax.

22.5.6 Net Present Value

Pre-tax NPV’s are calculated from cash flows before taxes, depreciation and royalties.

After-tax NPVs are calculated from the annual free cash flows. The financial model is capable of applying a range of discount factors. The use of various discount rates in the base case and sensitivity analysis of this report should not be taken as an endorsement of those discount rates as appropriate rates of return for this project.

22.5.7 After Tax Internal Rate of Return

The pre-tax IRR is calculated fromcash flows before taxes, depreciation and royalties.

The after-tax IRR is calculated from the annual after-taxes and royalties free cash flows.

22.5.8 Payback Period

The payback period is identified as the period in which the cumulative undiscounted cash flow becomes positive, having paid back the development costs.

22.5.9 Financial Model

A summary of the financial model results is presented below. The base case financial analysis is presented at a 5% discount rate.

Table 22-2 Summary Financial Results

	Base Case	Case 1
Pre-Tax NPV @5%	$324.1M	$475.0M
Pre-Tax IRR	36%	48%
Pre-Tax Net Cash Flow	$505.9M	$714.5M
After Tax NPV @5%	$230.5M	$330.1M
After Tax IRR @5%	29%	38%
After Tax Net Cash Flow	$371.2M	$507.4M

Table 22-3 Base Case Financial Model

	Year	-2	-1	1	2	3	4	5	6	7	8	9	Totals
Mining													
Waste Mining	tonnes		909K	11,067K	8,742K	3,820K	11,998K	16,939K	20,595K	19,541K	3,346K	31K	96,989K
Mined Resource Tonnes	tonnes			2,998K	2,558K	3,058K	2,555K	2,565K	2,424K	2,282K	2,556K	275K	21,271K
Sulphide Tonnes	tonnes			533K	438K	613K	1,220K	2,306K	2,129K	821K	910K	210K	9,180K
Oxide Tonnes	tonnes			2,466K	2,120K	2,445K	1,335K	259K	294K	1,461K	1,646K	65K	12,090K
Total Copper	tonnes			29K	26K	32K	13K	15K	15K	22K	28K	2K	184K
Sulphide Copper	tonnes			5K	4K	6K	6K	14K	13K	7K	9K	2K	66K
Oxide Copper	tonnes			24K	22K	27K	7K	1K	2K	15K	19K	1K	118K
Gold Grams	grams			1,206K	1,144K	1,334K	324K	317K	320K	799K	913K	68K	6,425K
Silver Grams	grams			12,911K	10,909K	13,749K	4,489K	5,032K	5,614K	9,315K	10,293K	823K	73,136K
Total Copper Grade	%			0.98%	1.03%	1.05%	0.52%	0.59%	0.62%	0.98%	1.09%	0.86%	0.86%
Sulphide Copper Grades	%			0.91%	0.99%	0.92%	0.52%	0.59%	0.63%	0.85%	0.96%	0.85%	0.72%
Oxide Copper Grades	%			0.99%	1.04%	1.08%	0.52%	0.54%	0.58%	1.05%	1.15%	0.90%	0.98%
Gold Grade	g/t			0.40	0.45	0.44	0.13	0.12	0.13	0.35	0.36	0.25	0.30
Silver Grade	g/t			4.31	4.27	4.50	1.76	1.96	2.32	4.08	4.03	2.99	3.44
Processing													
Tonnes Processed	tonnes			2,551K	2,558K	2,566K	2,555K	2,555K	2,381K	2,241K	2,556K	1,307K	21,271K
Sulphide Tonnes	tonnes			463K	438K	417K	1,220K	2,304K	2,118K	811K	910K	499K	9,180K
Oxide Tonnes	tonnes			2,089K	2,120K	2,149K	1,335K	251K	263K	1,431K	1,646K	808K	12,090K
Total Copper	tonnes			25K	26K	29K	13K	15K	15K	22K	28K	10K	184K
Sulphide Copper	tonnes			4K	4K	5K	6K	14K	13K	7K	9K	4K	66K
Oxide Copper	tonnes			21K	22K	25K	7K	1K	2K	15K	19K	6K	118K
Gold Grams	grams			1,052K	1,144K	1,334K	324K	317K	320K	799K	913K	223K	6,425K
Silver Grams	grams			11,239K	10,909K	13,749K	4,489K	5,032K	5,614K	9,315K	10,293K	2,495K	73,136K
Total Copper Grade	%			0.98%	1.03%	1.14%	0.52%	0.59%	0.63%	0.98%	1.09%	0.76%	0.86%
Sulphide Copper Grades	%			0.91%	0.99%	1.08%	0.52%	0.59%	0.63%	0.85%	0.96%	0.74%	0.72%
Oxide Copper Grades	%			0.99%	1.04%	1.15%	0.52%	0.54%	0.60%	1.06%	1.15%	0.78%	0.98%
Gold Grade	g/t			0.41	0.45	0.50	0.13	0.12	0.13	0.35	0.36	0.29	0.31
Silver Grade	g/t			4.35	4.27	4.78	1.76	1.96	2.32	4.12	4.03	4.24	3.50
Copper Recovered	tonnes			13K	14K	15K	9K	14K	13K	14K	17K	7K	117K
Gold Recovered g	grams			555K	609K	678K	196K	214K	214K	444K	523K	225K	3,659K
Silver Recovered g	grams			6,027K	5,904K	6,584K	2,925K	3,944K	4,315K	5,533K	6,114K	3,444K	44,791K
Copper Recovered	lbs			29,548K	30,979K	34,074K	20,756K	29,827K	29,348K	29,898K	38,238K	14,552K	257,220K
Gold Recovered	ozt			18K	20K	22K	6K	7K	7K	14K	17K	7K	118K
Silver Recovered	ozt			194K	190K	212K	94K	127K	139K	178K	197K	111K	1,440K
Costs													
Mining													

Mining Directs All tonnes			$1,207K	$39,807K	$33,579K	$18,041K	$47,074K	$17,020K	$15,350K	$47,132K	$17,540K	$2,975K	$239,724K
Mining Indirects				$2,558K	$2,558K	$2,558K	$2,558K	$2,558K	$2,558K	$2,558K	$2,558K	$639K	$21,102K
Total Mining Costs			$1,207K	$42,365K	$36,137K	$20,598K	$49,631K	$19,578K	$17,908K	$49,690K	$20,098K	$3,614K	$260,826K
Processing													
Direct Process Charges				$38,459K	$38,560K	$36,686K	$36,521K	$36,524K	$34,034K	$32,041K	$36,541K	$18,683K	$308,049K
Treatment Charges				$2,281K	$2,391K	$2,630K	$1,602K	$2,302K	$2,265K	$2,308K	$2,952K	$1,123K	$19,856K
Refining Charges (all metals)				$1,847K	$1,936K	$2,131K	$1,258K	$1,796K	$1,771K	$1,847K	$2,347K	$908K	$15,840K
Shipping Charges				$3,418K	$3,583K	$3,941K	$2,401K	$3,450K	$3,395K	$3,458K	$4,423K	$1,683K	$29,752K
Tailings Costs				$1,809K	$1,809K	$1,809K	$1,809K	$1,809K	$1,809K	$1,809K	$1,809K	$1,356K	$15,825K
Total Process Costs				$47,813K	$48,279K	$47,197K	$43,590K	$45,880K	$43,274K	$41,463K	$48,071K	$23,754K	$389,322K
General and Administration				$13,860K	$13,860K	$12,184K	$12,184K	$12,184K	$12,184K	$12,184K	$12,184K	$3,998K	$104,819K
Total Operating Costs			**$1,207K**	**$104,038K**	**$98,276K**	**$79,979K**	**$105,405K**	**$77,642K**	**$73,366K**	**$103,337K**	**$80,352K**	**$31,366K**	**$754,967K**
Gross Revenue													
Copper Revenue				$147,742K	$154,897K	$170,371K	$103,778K	$149,135K	$146,740K	$149,491K	$191,189K	$72,758K	$1,286,101K
Gold Revenue				$42,848K	$47,016K	$52,347K	$15,125K	$16,544K	$16,498K	$34,232K	$40,386K	$17,371K	$282,367K
Silver Revenue				$5,684K	$5,568K	$6,210K	$2,758K	$3,720K	$4,070K	$5,218K	$5,766K	$3,248K	$42,241K
Total Gross Revenue				**$196,275K**	**$207,480K**	**$228,928K**	**$121,661K**	**$169,399K**	**$167,307K**	**$188,942K**	**$237,341K**	**$93,377K**	**$1,610,710K**
Revenue Before Taxes, Royalties, Depreciation			**-$1,207K**	**$92,237K**	**$109,205K**	**$148,948K**	**$16,256K**	**$91,757K**	**$93,941K**	**$85,605K**	**$156,989K**	**$62,011K**	**$855,744K**
Capital Costs													
Initial Mining Capital		$314K	$23,336K	$849K									$24,499K
Process Capital		$42,461K	$75,522K	$35,264K									$153,248K
Tailings Capital		$0K	$14,665K										$14,665K
Non Process Infrastructure Capital		$2,473K	$25,182K										$27,655K
Sustaining Capital						$14,400K		$41,153K	$51,255K	$17,145K		$5,850K	$129,803K
Total Capital Costs		**$45,248K**	**$138,705K**	**$36,113K**	**$0K**	**$14,400K**	**$0K**	**$41,153K**	**$51,255K**	**$17,145K**	**$0K**	**$5,850K**	**$349,869K**
Total Capital and Operating Costs		**$45,248K**	**$139,912K**	**$140,150K**	**$98,276K**	**$94,379K**	**$105,405K**	**$118,795K**	**$124,621K**	**$120,482K**	**$80,352K**	**$37,216K**	**$1,104,836K**
Net Cash Flow Before Taxes, Royalties, Depreciation		**-$45,248K**	**-$139,912K**	**$56,124K**	**$109,205K**	**$134,548K**	**$16,256K**	**$50,604K**	**$42,686K**	**$68,460K**	**$156,989K**	**$56,161K**	**$505,874K**
Royalties, Depreciation, Taxation													
Depreciation Allowances		$0K	$0K	$51,006K	$60,572K	$41,527K	$16,256K	$49,324K	$41,807K	$54,455K	$9,155K	$11,991K	$336,093K
Yukon Royalty		$0K	$0K	$5,118K	$8,185K	$11,733K	$0K	$1,280K	$880K	$3,766K	$16,549K	$5,738K	$53,249K
Taxable Income		$0K	$0K	$0K	$40,448K	$81,288K	$0K	$0K	$0K	$10,238K	$131,285K	$38,432K	$301,692K
Federal Tax		$0K	$0K	$0K	$6,067K	$12,193K	$0K	$0K	$0K	$1,536K	$19,693K	$5,765K	$45,254K
Yukon Tax		$0K	$0K	$0K	$4,854K	$9,755K	$0K	$0K	$0K	$1,229K	$15,754K	$4,612K	$36,203K
Net Earnings After Tax and Royalties		**-$45,248K**	**-$139,912K**	**$51,006K**	**$90,099K**	**$100,867K**	**$16,256K**	**$49,324K**	**$41,807K**	**$61,929K**	**$104,993K**	**$40,046K**	**$371,169K**
Net Present Value @ 5% discount Rate		$230,492K											
Internal Rate of Return		29%											

Table 22-4 Case 1 Financial Model

	Year	-2	-1	1	2	3	4	5	6	7	8	9	Totals
Mining													
Waste Mining	tonnes		909K	11,067K	8,742K	3,820K	11,998K	16,939K	20,595K	19,541K	3,346K	31K	96,989K
Mined Resource Tonnes	tonnes			2,998K	2,558K	3,058K	2,555K	2,565K	2,424K	2,282K	2,556K	275K	21,271K
Sulphide Tonnes	tonnes			533K	438K	613K	1,220K	2,306K	2,129K	821K	910K	210K	9,180K
Oxide Tonnes	tonnes			2,466K	2,120K	2,445K	1,335K	259K	294K	1,461K	1,646K	65K	12,090K
Total Copper	tonnes			29K	26K	32K	13K	15K	15K	22K	28K	2K	184K
Sulphide Copper	tonnes			5K	4K	6K	6K	14K	13K	7K	9K	2K	66K
Oxide Copper	tonnes			24K	22K	27K	7K	1K	2K	15K	19K	1K	118K
Gold Grams	grams			1,206K	1,144K	1,334K	324K	317K	320K	799K	913K	68K	6,425K
Silver Grams	grams			12,911K	10,909K	13,749K	4,489K	5,032K	5,614K	9,315K	10,293K	823K	73,136K
Total Copper Grade	%			0.98%	1.03%	1.05%	0.52%	0.59%	0.62%	0.98%	1.09%	0.86%	0.86%
Sulphide Copper Grades	%			0.91%	0.99%	0.92%	0.52%	0.59%	0.63%	0.85%	0.96%	0.85%	0.72%
Oxide Copper Grades	%			0.99%	1.04%	1.08%	0.52%	0.54%	0.58%	1.05%	1.15%	0.90%	0.98%
Gold Grade	g/t			0.40	0.45	0.44	0.13	0.12	0.13	0.35	0.36	0.25	0.30
Silver Grade	g/t			4.31	4.27	4.50	1.76	1.96	2.32	4.08	4.03	2.99	3.44
Processing													
Tonnes Processed	tonnes			2,551K	2,558K	2,566K	2,555K	2,555K	2,381K	2,241K	2,556K	1,307K	21,271K
Sulphide Tonnes	tonnes			463K	438K	417K	1,220K	2,304K	2,118K	811K	910K	499K	9,180K
Oxide Tonnes	tonnes			2,089K	2,120K	2,149K	1,335K	251K	263K	1,431K	1,646K	808K	12,090K
Total Copper	tonnes			25K	26K	29K	13K	15K	15K	22K	28K	10K	184K
Sulphide Copper	tonnes			4K	4K	5K	6K	14K	13K	7K	9K	4K	66K
Oxide Copper	tonnes			21K	22K	25K	7K	1K	2K	15K	19K	6K	118K
Gold Grams	grams			1,052K	1,144K	1,334K	324K	317K	320K	799K	913K	223K	6,425K
Silver Grams	grams			11,239K	10,909K	13,749K	4,489K	5,032K	5,614K	9,315K	10,293K	2,495K	73,136K
Total Copper Grade	%			0.98%	1.03%	1.14%	0.52%	0.59%	0.63%	0.98%	1.09%	0.76%	0.86%
Sulphide Copper Grades	%			0.91%	0.99%	1.08%	0.52%	0.59%	0.63%	0.85%	0.96%	0.74%	0.72%
Oxide Copper Grades	%			0.99%	1.04%	1.15%	0.52%	0.54%	0.60%	1.06%	1.15%	0.78%	0.98%
Gold Grade	g/t			0.41	0.45	0.50	0.13	0.12	0.13	0.35	0.36	0.29	0.31
Silver Grade	g/t			4.35	4.27	4.78	1.76	1.96	2.32	4.12	4.03	4.24	3.50
Copper Recovered	tonnes			13K	14K	15K	9K	14K	13K	14K	17K	7K	117K
Gold Recovered g	grams			555K	609K	678K	196K	214K	214K	444K	523K	225K	3,659K
Silver Recovered g	grams			6,027K	5,904K	6,584K	2,925K	3,944K	4,315K	5,533K	6,114K	3,444K	44,791K
Copper Recovered	lbs			29,548K	30,979K	34,074K	20,756K	29,827K	29,348K	29,898K	38,238K	14,552K	257,220K
Gold Recovered	ozt			18K	20K	22K	6K	7K	7K	14K	17K	7K	118K
Silver Recovered	ozt			194K	190K	212K	94K	127K	139K	178K	197K	111K	1,440K
Costs													
Mining													

Mining Directs All tonnes			$1,207K	$39,807K	$33,579K	$18,041K	$47,074K	$17,020K	$15,350K	$47,132K	$17,540K	$2,975K	$239,724K
Mining Indirects				$2,558K	$2,558K	$2,558K	$2,558K	$2,558K	$2,558K	$2,558K	$2,558K	$639K	$21,102K
Total Mining Costs			$1,207K	$42,365K	$36,137K	$20,598K	$49,631K	$19,578K	$17,908K	$49,690K	$20,098K	$3,614K	$260,826K
Processing													
Direct Process Charges				$38,459K	$38,560K	$36,686K	$36,521K	$36,524K	$34,034K	$32,041K	$36,541K	$18,683K	$308,049K
Treatment Charges				$2,281K	$2,391K	$2,630K	$1,602K	$2,302K	$2,265K	$2,308K	$2,952K	$1,123K	$19,856K
Refining Charges (all metals)				$1,847K	$1,936K	$2,131K	$1,258K	$1,796K	$1,771K	$1,847K	$2,347K	$908K	$15,840K
Shipping Charges				$3,418K	$3,583K	$3,941K	$2,401K	$3,450K	$3,395K	$3,458K	$4,423K	$1,683K	$29,752K
Tailings Costs				$1,809K	$1,809K	$1,809K	$1,809K	$1,809K	$1,809K	$1,809K	$1,809K	$1,356K	$15,825K
Total Process Costs				$47,813K	$48,279K	$47,197K	$43,590K	$45,880K	$43,274K	$41,463K	$48,071K	$23,754K	$389,322K
General and Administration				$13,860K	$13,860K	$12,184K	$12,184K	$12,184K	$12,184K	$12,184K	$12,184K	$3,998K	$104,819K
Total Operating Costs			**$1,207K**	**$104,038K**	**$98,276K**	**$79,979K**	**$105,405K**	**$77,642K**	**$73,366K**	**$103,337K**	**$80,352K**	**$31,366K**	**$754,967K**
Gross Revenue													
Copper Revenue				$167,441K	$175,550K	$193,088K	$117,615K	$169,020K	$166,305K	$169,424K	$216,681K	$82,459K	$1,457,582K
Gold Revenue				$47,609K	$52,240K	$58,163K	$16,806K	$18,382K	$18,331K	$38,035K	$44,874K	$19,301K	$313,742K
Silver Revenue				$6,459K	$6,327K	$7,056K	$3,134K	$4,227K	$4,625K	$5,930K	$6,552K	$3,691K	$48,002K
Total Gross Revenue				**$221,510K**	**$234,117K**	**$258,307K**	**$137,555K**	**$191,629K**	**$189,261K**	**$213,389K**	**$268,107K**	**$105,451K**	**$1,819,325K**
Revenue Before Taxes, Royalties, Depreciation			**-$1,207K**	**$117,472K**	**$135,841K**	**$178,328K**	**$32,150K**	**$113,987K**	**$115,894K**	**$110,052K**	**$187,755K**	**$74,085K**	**$1,064,358K**
Capital Costs													
Initial Mining Capital		$314K	$23,336K	$849K									$24,499K
Process Capital		$42,461K	$75,522K	$35,264K									$153,248K
Tailings Capital		$0K	$14,665K										$14,665K
Non Process Infrastructure Capital		$2,473K	$25,182K										$27,655K
Sustaining Capital						$14,400K		$41,153K	$51,255K	$17,145K		$5,850K	$129,803K
Total Capital Costs		**$45,248K**	**$138,705K**	**$36,113K**	**$0K**	**$14,400K**	**$0K**	**$41,153K**	**$51,255K**	**$17,145K**	**$0K**	**$5,850K**	**$349,869K**
Total Capital and Operating Costs		**$45,248K**	**$139,912K**	**$140,150K**	**$98,276K**	**$94,379K**	**$105,405K**	**$118,795K**	**$124,621K**	**$120,482K**	**$80,352K**	**$37,216K**	**$1,104,836K**
Net Cash Flow Before Taxes, Royalties, Depreciation		**-$45,248K**	**-$139,912K**	**$81,359K**	**$135,841K**	**$163,928K**	**$32,150K**	**$72,834K**	**$64,640K**	**$92,907K**	**$187,755K**	**$68,235K**	**$714,489K**
Royalties, Depreciation, Taxation													
Depreciation Allowances		$0K	$0K	$73,213K	$46,802K	$39,312K	$21,161K	$56,790K	$61,401K	$26,925K	$6,387K	$9,946K	$341,937K
Yukon Royalty		$0K	$0K	$8,146K	$11,381K	$15,258K	$30K	$4,092K	$3,239K	$7,428K	$20,606K	$7,200K	$77,380K
Taxable Income		$0K	$0K	$0K	$77,658K	$109,358K	$10,959K	$11,952K	$0K	$58,553K	$160,762K	$51,089K	$480,331K
Federal Tax		$0K	$0K	$0K	$11,649K	$16,404K	$1,644K	$1,793K	$0K	$8,783K	$24,114K	$7,663K	$72,050K
Yukon Tax		$0K	$0K	$0K	$9,319K	$13,123K	$1,315K	$1,434K	$0K	$7,026K	$19,291K	$6,131K	$57,640K
Net Earnings After Tax and Royalties		**-$45,248K**	**-$139,912K**	**$73,213K**	**$103,492K**	**$119,143K**	**$29,162K**	**$65,515K**	**$61,401K**	**$69,669K**	**$123,743K**	**$47,241K**	**$507,419K**
Net Present Value @ 5% discount Rate		$330,119K											
Internal Rate of Return		38%											

22.6 Sensitivity Analysis

For the purposes of the PEA, the evaluation is based on 100% of the Project cash flows before distribution of profits to equity owners. Economic sensitivities are presented for various scenarios:

- Discount rates of 5%, 6%, 7%, 8%, 9% and 10%
- Sensitivity ranges for operating and capital costs between +/- 40% of base case values
- Sensitivity ranges for revenues (Copper Pricing) of +/- 40%
- Sensitivity Ranges for Foreign Exchange ranging from USD:CAD of \$0.56, \$0.60, \$0.75, \$0.90, \$0.94 (75% to 125% of Basis of Estimate)

Table 22-5 to Table 22-8 present the various sensitivity results.

Table 22-5 Sensitivity to Capital Costs

Capital Sensitivity						
Pre-Tax NPV						
		60%	80%	100%	120%	140%
	5%	$439.8M	$381.9M	$324.1M	$266.2M	$208.3M
	6%	$408.4M	$352.5M	$296.6M	$240.7M	$184.8M
	7%	$379.6M	$325.5M	$271.5M	$217.4M	$163.4M
	8%	$353.1M	$300.8M	$248.4M	$196.1M	$143.8M
	9%	$328.7M	$278.0M	$227.3M	$176.6M	$125.9M
	10%	$306.1M	$257.0M	$207.8M	$158.7M	$109.5M
After Tax NPV		60%	80%	100%	120%	140%
	5%	$251.6M	$243.5M	$230.5M	$215.6M	$199.6M
	6%	$228.9M	$221.4M	$209.2M	$195.3M	$180.4M
	7%	$208.1M	$201.1M	$189.8M	$176.8M	$162.8M
	8%	$189.1M	$182.6M	$171.9M	$159.7M	$146.6M
	9%	$171.7M	$165.5M	$155.5M	$144.1M	$131.8M
	10%	$155.6M	$149.9M	$140.5M	$129.8M	$118.2M

Table 22-6 Sensitivity to Operating Costs

Operating Sensitivity						
Pre-Tax NPV						
		60%	80%	100%	120%	140%
	5%	$580.3M	$452.2M	$324.1M	$196.0M	$67.9M
	6%	$537.8M	$417.2M	$296.6M	$176.0M	$55.4M
	7%	$498.8M	$385.1M	$271.5M	$157.8M	$44.2M
	8%	$462.9M	$355.7M	$248.4M	$141.2M	$34.0M
	9%	$429.8M	$328.5M	$227.3M	$126.0M	$24.7M
	10%	$399.3M	$303.6M	$207.8M	$112.1M	$16.4M
After Tax NPV		60%	80%	100%	120%	140%
	5%	$397.8M	$318.9M	$230.5M	$131.2M	$23.6M
	6%	$366.9M	$292.5M	$209.2M	$115.8M	$14.5M
	7%	$338.4M	$268.2M	$189.8M	$101.8M	$6.3M
	8%	$312.3M	$246.0M	$171.9M	$89.0M	-$1.1M
	9%	$288.2M	$225.5M	$155.5M	$77.3M	-$7.8M
	10%	$266.0M	$206.6M	$140.5M	$66.6M	-$13.9M

Table 22-7 Sensitivity to Copper Price

Copper Price Sensitivity						
Pre-Tax NPV		$2.25	$3.00	$3.75	$4.50	$5.25
		60%	80%	100%	120%	140%
	5%	-$47.0M	$138.6M	$324.1M	$509.6M	$695.1M
	6%	-$52.2M	$122.2M	$296.6M	$471.0M	$645.4M
	7%	-$56.8M	$107.3M	$271.5M	$435.6M	$599.8M
	8%	-$60.9M	$93.8M	$248.4M	$403.1M	$557.8M
	9%	-$64.5M	$81.4M	$227.3M	$373.2M	$519.0M
	10%	-$67.7M	$70.1M	$207.8M	$345.6M	$483.3M
After Tax NPV						
		60%	80%	100%	120%	140%
	5%	-$69.4M	$88.3M	$230.5M	$361.0M	$482.8M
	6%	-$72.9M	$75.5M	$209.2M	$332.0M	$446.6M
	7%	-$76.0M	$63.9M	$189.8M	$305.4M	$413.3M
	8%	-$78.7M	$53.3M	$171.9M	$280.9M	$382.6M
	9%	-$81.0M	$43.7M	$155.5M	$258.4M	$354.4M
	10%	-$83.0M	$34.9M	$140.5M	$237.7M	$328.3M

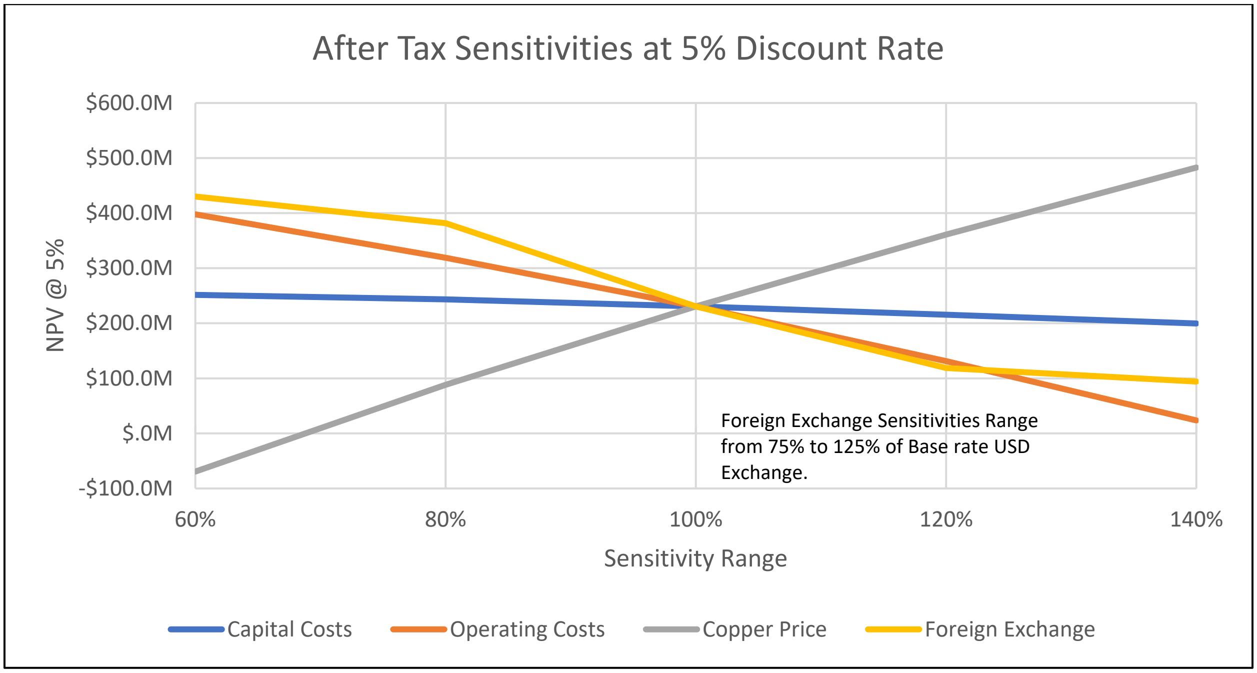
Table 22-8 Sensitivity to Foreign Exchange

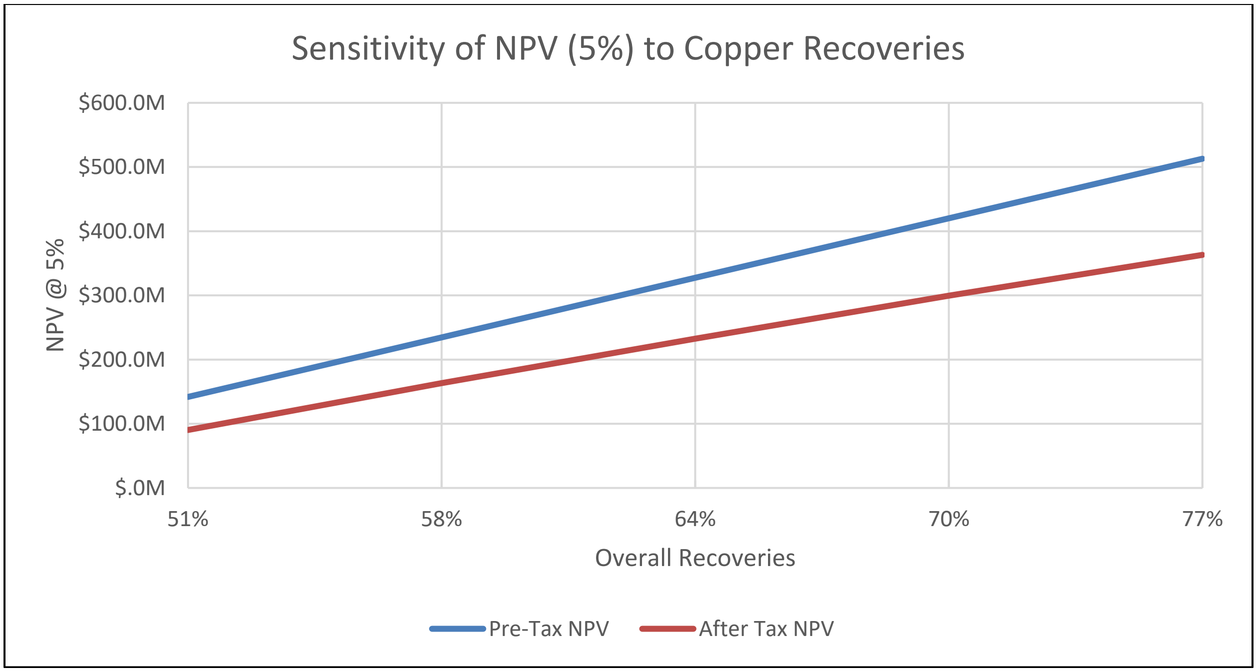
Foreign Exchange Sensitivity					
		Canadian Dollar Increasing>>>			
USD:CAD	$0.56	$0.60	$0.75	$0.90	$0.94
Pre-Tax NPV	75%	80%	100%	120%	125%
5%	$712.8M	$615.6M	$324.1M	$129.7M	$90.9M
6%	$662.3M	$570.9M	$296.6M	$113.8M	$77.2M
7%	$616.0M	$529.8M	$271.5M	$99.2M	$64.8M
8%	$573.3M	$492.1M	$248.4M	$86.0M	$53.5M
9%	$533.9M	$457.3M	$227.3M	$73.9M	$43.3M
10%	$497.6M	$425.2M	$207.8M	$62.9M	$34.0M
After Tax NPV	75%	80%	100%	120%	125%
5%	$430.1M	$381.8M	$230.5M	$118.6M	$94.1M
6%	$394.7M	$349.9M	$209.2M	$104.8M	$81.9M
7%	$362.2M	$320.6M	$189.8M	$92.2M	$70.7M
8%	$332.4M	$293.7M	$171.9M	$80.8M	$60.6M
9%	$304.9M	$268.9M	$155.5M	$70.3M	$51.4M
10%	$279.6M	$246.2M	$140.5M	$60.7M	$42.9M

Table 22-9 Sensitivity to Recovery

Copper Recovery Sensitivity	Overall Recovery				
Pre-Tax NPV	51%	58%	64%	70%	77%
Sensitivity Range	80%	90%	100%	110%	120%
5%	$138.6M	$231.3M	$324.1M	$416.8M	$509.6M
6%	$122.2M	$209.4M	$296.6M	$383.8M	$471.0M
7%	$107.3M	$189.4M	$271.5M	$353.6M	$435.6M
8%	$93.8M	$171.1M	$248.4M	$325.8M	$403.1M
9%	$81.4M	$154.3M	$227.3M	$300.2M	$373.2M
10%	$70.1M	$139.0M	$207.8M	$276.7M	$345.6M
After Tax NPV					
Sensitivity Range	80%	90%	100%	110%	120%
5%	$88.3M	$161.0M	$230.5M	$297.4M	$361.0M
6%	$75.5M	$143.8M	$209.2M	$272.2M	$332.0M
7%	$63.9M	$128.2M	$189.8M	$249.0M	$305.4M
8%	$53.3M	$113.9M	$171.9M	$227.8M	$280.9M
9%	$43.7M	$100.8M	$155.5M	$208.3M	$258.4M
10%	$34.9M	$88.8M	$140.5M	$190.3M	$237.7M

Figure 22-2 After Tax Sensitivities at 5% Discount Rate





The sensitivity tables and charts indicate that the project, at a PEA stage, is relatively insensitive to capital costs.

Operating costs show greater sensitivity with negative NPV's at higher discount rates in an after tax scenario.

Foreign exchange exhibits higher sensitivity with a decreasing Canadian dollar value, and mirrors the operating sensitivity at higher Canadian dollar values up to USD:CAD exchange of $0.90. Higher Canadian exchange rates show minimal change in NPV at rates higher than $0.90.

The project is most sensitive to copper prices, showing a negative NPV at 60% of base case value, and a doubling of NPV at 140% of base case copper prices. Copper recoveries also show a similar trend, as recovered copper revenues are in a direct relationship with copper prices.

23 ADJACENT PROPERTIES

There is no information on properties adjacent to the Carmacks Project necessary to make this technical report understandable and not misleading.

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information available that is necessary to make the current technical report understandable and not misleading. To SGSs' knowledge, there are no significant risks and uncertainties that could reasonably be expected to affect the reliability or confidence in the exploration information or Mineral Resource estimate.

25 INTERPRETATION AND CONCLUSIONS

25.1 Mineral Resource Estimate

On behalf of GXC, SGS has completed a Mineral Resource estimate for Zones 147& 2000S and 1213 Area of the Carmacks Project.

The Mineral Resource was reported as Measured, Indicated and Inferred Mineral Resources as shown in Table 25-1 and Table 25-2. The Mineral Resource was estimated using The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019) and is reported in accordance with the 2014 CIM Definition Standards, which have been incorporated by reference into National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101).

In the QP's opinion, the Mineral Resources reported herein at the selected cut-off grade have "reasonable prospects for eventual economic extraction", taking into consideration mining and processing assumptions (refer to Table 14-8). The Mineral Resource was reported from within Whittle optimised pit shells.

Table 25-1 Carmacks Project Mineral Resource Estimates, Effective February 25, 2022

Category	CU_T % Cut-off	Tonnes	CU_T		AG		AU		MO		CuEq	
			(%)	(Mlbs)	(g/t)	Ounces	(g/t)	Ounces	(%)	(Mlbs)	(%)	(Mlbs)
In-Pit Oxide												
Measured	0.30	11,361,000	0.96	239	4.11	1,501,000	0.40	145,000	0.006	1.5	1.30	325
Indicated	0.30	4,330,000	0.91	87	3.37	469,000	0.28	39,000	0.007	0.6	1.16	111
Measured + Indicated	**0.30**	**15,691,000**	**0.94**	**326**	**3.91**	**1,971,000**	**0.36**	**184,000**	**0.006**	**2.1**	**1.26**	**436**
Inferred	0.30	216,000	0.52	2.5	2.44	17,000	0.09	1,000	0.006	0.03	0.63	3
In-Pit Sulphide												
Measured	0.30	5,705,000	0.68	86	2.54	467,000	0.16	28,000	0.016	2.0	0.88	111
Indicated	0.30	13,486,000	0.72	214	2.83	1,226,000	0.19	82,000	0.013	4.0	0.93	277
Measured + Indicated	**0.30**	**19,191,000**	**0.71**	**300**	**2.74**	**1,693,000**	**0.18**	**110,000**	**0.014**	**6.0**	**0.92**	**388**
Inferred	0.30	1,675,000	0.51	19	2.24	120,895	0.13	7,000	0.020	0.7	0.7	26
Below Pit Sulphide												
Measured	0.60	26,000	0.71	0.41	2.54	2,000	0.16	132	0.010	0.0	0.88	0.5
Indicated	0.60	1,341,000	0.82	24	2.88	124,000	0.19	8,000	0.012	0.4	1.03	30
Measured + Indicated	**0.60**	**1,367,000**	**0.82**	**25**	**2.88**	**126,000**	**0.19**	**8,000**	**0.012**	**0.4**	**1.03**	**31**
Inferred	0.60	967,000	0.77	16	2.48	77,000	0.17	5,000	0.012	0.3	0.96	20

(1) The classification of the current Mineral Resource Estimates into Measured, Indicated and Inferred are consistent with current 2014 CIM Definition Standards - For Mineral Resources and Mineral Reserves.

(2) All figures are rounded to reflect the relative accuracy of the estimate.

(3) All Resources are presented undiluted and in situ, constrained by continuous 3D wireframe models, and are considered to have reasonable prospects for eventual economic extraction.

(4) Mineral resources which are not mineral reserves do not have demonstrated economic viability. An Inferred Mineral Resource has a lower level of confidence than that applying to a Measured and Indicated Mineral

Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

(5) *It is envisioned that parts of the Carmacks Project deposits may be mined using open pit mining methods. In-pit mineral resources are reported at a base-case cut-off grade of 0.3 % Cu_T within Whittle pit shells. It is envisioned that parts of the Carmacks Project deposits may be mined using lower cost underground bulk mining methods. A selected base-case cut-off grade of 0.6 % Cu_T is used to determine the underground mineral resources.*

(6) *Cu Eq calculation is based on 100% recovery of all metals using the same metal prices used for the resource calculation.*

(7) *A pit slope of 55 degrees for rock and 35 degrees for overburden are used for the pit optimization.*

(8) *The results from the pit optimization are used solely for the purpose of testing the "reasonable prospects for economic extraction" by an open pit and do not represent an attempt to estimate mineral reserves. There are no mineral reserves on the Carmacks Property. The results are used as a guide to assist in the preparation of a Mineral Resource statement and to select an appropriate resource reporting cut-off grade.*

(9) *Cut-off grades are based on metal prices of $3.60/lb Cu, $22.00/oz Ag, $1,750/oz Au and $14.00/lb for Mo, processing and G&A cost of $US23.00 per tonne milled, and variable mining costs including $US2.10 for open pit and $US25.00 for underground. The cut-off grades should be re-evaluated in light of future prevailing market conditions (metal prices, mining costs, processing costs etc.).*

(10) *Metal recoveries used for pit optimization and calculation of base-case cut-off grades include: for oxide material 85% for copper, 65% for Ag, 85% for Au and 70% for Mo; for sulphide material, 90% for copper, 65% for Ag, 76% for Au and 70% for Mo.*

(11) *Composites of 2.0 metre used for the resource estimation procedure have been capped where appropriate. Grades for Cu (oxide, sulphide and total), Ag, Au and Mo for each deposit was interpolated into blocks by the Inverse Distance Squared (ID^2) calculation method.*

(12) *Fixed specific gravity values of 2.64 for oxide material and 2.71 – 2.78 (depending on deposit) were used to estimate the Mineral Resource tonnage from block model volumes. Waste in all areas was given a fixed density of 2.66.*

(13) *The database used for the current MREs comprise data for 489 surface drill holes totaling 56,679 metres completed on the Carmacks Project area between 1970 and 2021. This includes 36 drill holes (RC and diamond) totaling 9,413 m completed by Granite Creek between the fall of 2020 the fall of 2021. Appropriate interpolation parameters were generated for each deposit based on the mineralization style and geometry.*

Table 25-2 Carmacks Project Mineral Resource Estimates, February 25, 2022: Distribution of Cu_X and Cu_S

Category	CU_T % Cut-off	Tonnes	CU_T		CU_S		CU_X	
			(%)	(Mlbs)	(%)	(Mlbs)	(%)	(Mlbs)
In-Pit Oxide								
Measured	0.30	11,361,000	0.96	239	0.18	45	0.78	194
Indicated	0.30	4,330,000	0.91	87	0.19	18	0.72	69
Measured + Indicated	**0.30**	**15,691,000**	**0.94**	**326**	**0.18**	**63**	**0.76**	**263**
Inferred	0.30	216,000	0.52	2.5	0.12	0.6	0.37	1.8
In-Pit Sulphide								
Measured	0.30	5,705,000	0.68	86	0.62	79	0.05	7
Indicated	0.30	13,486,000	0.72	214	0.68	201	0.04	13
Measured + Indicated	**0.30**	**19,191,000**	**0.71**	**300**	**0.66**	**280**	**0.05**	**20**
Inferred	0.30	1,675,000	0.51	19	0.46	17	0.05	2

Below Pit Sulphide								
Measured	0.60	26,000	0.71	0.41	0.68	0.39	0.03	0.02
Indicated	0.60	1,341,000	0.82	24	0.80	24	0.03	0.8
Measured + Indicated	**0.60**	**1,367,000**	**0.82**	**25**	**0.79**	**24**	**0.03**	**0.8**
Inferred	0.60	967,000	0.77	16	0.75	16	0.03	0.1

There is no other relevant data or information available that is necessary to make the technical report understandable and not misleading. The Author is not aware of any known mining, processing, metallurgical, environmental, infrastructure, economic, permitting, legal, title, taxation, socio-political, or marketing issues, or any other relevant factors not reported in this technical report, that could materially affect the updated MRE.

25.2 Capital and Operating Costs

The PEA includes open pits 147 and 1213N. Open pit production is delivered to a central mill complex on the property. The mill complex produces a copper concentrate final product.

Table 25-3 summarizes the estimated operating costs developed for the PEA.

Table 25-3 Total Operating Cost Summary

Description	Cost per Tonne Processed (CAD/t)
Mining	$17.56
Mill Site Process Plant	$18.30
General and Administration	$4.93
Total Operating Cost	$40.79

The plans and costs presented in this PEA indicates a robust project with favourable economics. The mine plan and copper recoveries honour the mineral resource.

Financial analysis yields positive economic returns for the project with an Initial Direct Capital Costs US$ 220.1 million and US$ 129.8 million of sustaining Capital investment including mine closure cost, indirect and contingencies. Table 25-4 shows the capital requirements for the Carmacks Copper Project

Table 25-4 Total Capital Cost Summary

Area	Initial Capital $CAD	Sustaining Capital $CAD
Mining Directs	$13,961K	$120,202K
Mining Indirects	$7,167K	
Processing Directs	$83,445K	
Processing Indirects	$34,103K	
Non Process Infrastructure	$16,932K	
Tailings	$14,665K	
Pre-Production G&A	$3,370K	
Power Supply	$11,160K	$3,751K
Closure Costs		$5,850K

Contingency	$35,264K	
Total	$220,066K	$129,803K

Based on the results of the PEA (Table 25-5), the following conclusions can be made:

- The Carmacks Project has significant potential to provide positive and robust returns.
- Additional exploration and expansion of the current resource, including the potential conversion of Inferred resources to Measured and Indicated.

Table 25-5 Summary Financial Results

	Base Case	Case 1
Pre-Tax NPV @5%	$324.1M	$475.0M
Pre-Tax IRR	36%	48%
Pre-Tax Net Cash Flow	$505.9M	$714.5M
After Tax NPV @5%	$230.5M	$330.1M
After Tax IRR @5%	29%	38%
After Tax Net Cash Flow	$371.2M	$507.4M

25.3 Opportunities

The PEA demonstrates that the Carmacks Copper Project has the potential to be technically and economically viable. The Project is technically uncomplicated because of the near surface nature of the deposit and relatively simple access. Several opportunities for the project are available to further enhance the project:

- The third conceptual pit, 2000S as identified in the Mineral Resource Estimate ("MRE), could be brought into the mine plan if sufficient additional resources were defined by drilling to offset pre-stripping costs.
- Electrification of the mining fleet. Significant cost saving and reduction in greenhouse gas production may be possible through the sourcing of electric vs. diesel haul trucks for the Project. The PEA envisions using a contract mining fleet for the Project and preference will be given to suppliers that can provide either fully electric or hybrid equipment.
- Further discovery. Exploration conducted in 2022 consisting of geophysics, trenching and soil sampling identified four areas proximal to the proposed mine plan that if successfully drilled could enable longer mine life beyond nine years or provide additional sulphide mill feed earlier in the mine's life. Four targets on the Property require evaluation, all located within 1km of the current deposits. Two of the targets are located beneath the current resource and there is higher geological certainty that these may contain appreciable copper mineralization.
  - Zone 1213 shallow:

    Downward continuation of Zone 12 and 13. Estimated dimensions are 360m long, 15 – 40m wide, starting at approximately 65m below the current drilling.
  - Zone 12 deep:

Downward continuation of Zone 12. Estimated from geophysics to be continuing for an additional 170m below current resource modelling. Approximated to be 580m long and 15-40m wide.

- Gap Zone target:

  Geophysical anomaly that fits with current geological understanding of the fault offset between 147 and 2000S Zone. Estimated to be 500m long, up to 400m deep, and 30-50m wide.

- Sourtoe target:

  Estimated from geophysics to be a lensoidal body of similar size to known deposits at 370m long x 370m deep with an estimated width of 15-50m. It has been lightly tested at surface by trenching and is weakly mineralized.

In addition, the Carmacks North target area is host to several mineralized zones that have the potential to add resources to the mine plan, all within 15 km of the proposed mill site.

- Additional recovery through metallurgical improvements. The Company has retained Kemetco Laboratories to complete additional leaching and copper precipitating testing to evaluate the processing of tailings. The calculated grade of copper in tailings averages 0.32% with over 140 Mlbs of copper not recovered LOM. Recovery sensitivity show an additional $180M pretax NPV based of a 20% increase in recovery rates. Review of historical metallurgical testing has indicated that copper minerals present in oxidized material respond well to leaching. Once the copper is in solution the copper would be chemically precipitated to produce sulphide minerals that can be added back into the flotation cells.

25.4 Risks

There are risks that have been identified within the recommendations.

- Main risks to project success would be:
- Changes in environmental regulations;
- The potential for additional area requirement for Tailings Storage facility;
- Availability of skilled labour during the construction phase.

26 RECOMMENDATIONS

It is recommended that Granite Creek Ltd complete the following:

Mineral Resources

- Undertake additional drilling to expand the current resource and to upgrade the quality of the resource to derive mineable reserves and to extend the life of mine.

Mining

- No geotechnical study has yet been undertaken on Zones 2000S and 1213. This should be done if the project advances to PFS as this will have a material impact on the stripping ratio.
- The inferred resources category should be minimized (by additional drilling) to ensure enough reserves for scheduling.
- A preliminary hydrological study should be commissioned to validate the assumption that there is no water related issues at depth.
- A detailed re-design of the tailings storage facility and engineering of a dry stack tailings system.

Mineral Processing

- Current process flowsheet requires a SAG mill in the grinding circuit. JK drop weight test or SMC test is recommended to size the SAG mill more accurately. Bond Ball Mill Work Index and Axb determinations should be developed to solidify the ball mill sizing criteria.
- For the flotation concentrate, sedimentation and filtration tests are recommended to better size the concentrate thickener and concentrate filter.
- For the flotation tailings, sedimentation testing is recommended to better size the tailings thickener and better estimate the water balance around the tailings storage facility.
- Current metal recoveries are predicted based on the acid soluble copper percentage in the plant feed. However, the impact of head grade of the flotation feed is also recommended to be explored in the next phase of the project.
- Based on the sample material received at SGS Lakefield laboratory, the sulfide material contains 10.8% of acid soluble copper, and the oxide material contains 77.3% acid soluble copper. It is uncertain that all sulfide material or oxide material has the same percentage of acid soluble copper. Therefore, it is recommended to measure the acid soluble copper for every major resource block in the next phase of study.
- Further metallurgical work focused on improving recoveries in the oxide domain.

- **Water Supply**

- As stipulated by Granite Creek, the sources for water are boreholes with a depth of 74 m. Further development of the water supply is recommended to include quality/quantity testing. Assessment of the groundwater potential in the area around this borehole is highly recommended.

Electrical Supply

- Details for the completion of the power supply connection from Yukon Power Energy which will provide the optimum megawatts onto the company's electricity system should be finalized.

Mine Access Road

- For the access road construction, topographic survey of the road alignment is essential to accurately quantify the cut and fill volumes including a geotechnical assessment.

Concentrate Pricing

- The marketing study and concentrate price should be updated to reflect current market conditions when the pre-feasibility study is conducted.

Economic Analysis

- This PEA provides suitable economics to progress to the next stage of project development via a Pre-Feasibility study, with updated costs.

Tailings Management Facility

Additional studies will be required to advance the conceptual design to pre-feasibility and feasibility level. Golder recommend that future studies should include the tasks described below.

- Definition of the environmental performance criteria for the design.
- Documentation of a methodical approach to the site and options selection that will support funding decisions and permit applications.
- Development of the design as an integrated waste landform. This should include plans for the staged development of waste rock and tailings over the life of mine considering the mine plan, management of surface and sub-surface drainage and closure objectives.
- Additional geotechnical and hydrogeological investigations to provide supporting information for design of the TMF, associated ponds, and surface water diversions. At a feasibility level, Golder anticipate this is like to include drilling of five additional boreholes and 20 test pits at the proposed TMF site.
- Probabilistic water balance model to estimate discharges and changes to surface water quantity in the receiving environment and sizing of return water infrastructure. Hydrological modelling and design, including updates to the rainfall runoff design events and more detailed analysis of surface water drainage on and around the TMF. Hydrogeological study of the TMF performance and its groundwater impacts. This could be coupled with water quality modelling, if required. Review of previous waste rock geochemical and geotechnical testing to confirm appropriateness of assumptions in the context of the TMF waste rock shell design. Analysis of geochemical testing completed on the tailings to support design.
- Further geotechnical testing of the tailings to provide shear strength parameters.
- Stability, deformation and seepage analysis for the landform and ponds.
- Thermal modelling of the TMF to assess the permafrost response and the development of frozen layers within the tailings. Air dispersion modelling incorporating the new site configuration to assess effects on air quality.
- Development of a conceptual closure plan for the integrated waste landform.

Overall

- Carry out a six-month PFS to further develop the engineering design of the plant and recognise value engineering where possible.
- Revisit the capital cost estimates in general for possible savings due to optimising the cost estimates from ± 50% to ± 10% (PFS Level).
- The budget for the recommendations provided is designed to collect the data required to complete a pre-feasibility study is estimated at $USD 9.0M with the details provided in the following Table 26-1.

Table 26-1 Budget for Future Work

Tasks	Estimated Cost (US$)
Geology (Major Upgrade Drilling Program)	1,000,000
Water Supply (Alternative Source)	50,000
Geotechnical and Hydrogeology	750,000
TSF Optimization/Studies	500,000
Environmental, Permitting and Community Relations	450,000
Mineral Processing (Metallurgy and Hydrometallurgy)	750,000
Mine Access PFS (Road Studies)	100,000
Engineering and Prefeasibility	5,400,000
Total	**9,000,000**

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28 DATE AND SIGNATURE PAGE

This report titled "Technical Report on the Carmacks Project Preliminary Economic Assessment (PEA) Yukon, Canada" dated March 6, 2023 (the "Technical Report") for Granite Creek Inc. was prepared and signed by the following authors:

The effective date of the report is January 19, 2023.

The date of the report is March 6, 2023.

Signed by:

"Original Signed and Sealed"

Qualified Person Allan Armitage, P.Geo March 6, 2023	Company SGS Canada Inc.

Signed by:

"Original Signed and Sealed"

Qualified Person William van Breugel, P.Eng. March 6, 2023	Company SGS Canada Inc.

Signed by:

"Original Signed and Sealed"

Qualified Person Johnny Canosa, P.Eng. March 6, 2023	Company SGS Canada Inc.

Signed by:

"Original Signed and Sealed"

Qualified Person	Company
Joseph M. Keane, P.E.	Consultant to SGS Bateman S.A.
March 6, 2023	

29 CERTIFICATES OF QUALIFIED PERSONS

QP CERTIFICATE – ALLAN ARMITAGE

To Accompany the Report titled "Technical Report on the Carmacks Project Preliminary Economic Assessment (PEA) Yukon, Canada" dated March 6, 2023 (the "Technical Report") prepared for Granite Creek Copper Ltd.

I, Allan E. Armitage, Ph. D., P. Geo. of 62 River Front Way, Fredericton, New Brunswick, hereby certify that:

a) I am a Senior Resource Geologist with SGS Geological Services, 10 de la Seigneurie E blvd., Unit 203 Blainville, QC, Canada, J7C 3V5 (www.geostat.com).

b) I am a graduate of Acadia University having obtained the degree of Bachelor of Science - Honours in Geology in 1989, a graduate of Laurentian University having obtained the degree of Master of Science in Geology in 1992 and a graduate of the University of Western Ontario having obtained a Doctor of Philosophy in Geology in 1998.

c) I have been employed as a geologist for every field season (May - October) from 1987 to 1996. I have been continuously employed as a geologist since March of 1997.

d) I have been involved in mineral exploration and resource modeling at the grass roots to advanced exploration stage, including producing mines, since 1991, including mineral resource estimation and mineral resource and mineral reserve auditing since 2006 in Canada and internationally. I have extensive experience in Archean and Proterozoic load gold deposits, volcanic and sediment hosted base metal massive sulphide deposits, porphyry copper-gold-silver deposits, low and intermediate sulphidation epithermal gold and silver deposits, magmatic Ni-Cu-PGE deposits, and unconformity- and sandstone-hosted uranium deposits.

e) I am a member of the Association of Professional Engineers, Geologists and Geophysicists of Alberta (P.Geol.) (License No. 64456; 1999), the Association of Professional Engineers and Geoscientists of British Columbia (P.Geo.) (Licence No. 38144; 2012), and the Professional Geoscientists Ontario (P.Geo.) (Licence No. 2829; 2017).

f) I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation of my professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person".

g) I am an author of this report and responsible for sections 1.2 to 1.4, 1.6, 4 to 12, 14, 23 and 25.1 of the Technical Report and accept professional responsibility for those sections of the Technical Report.

h) I conducted a site visit to the Carmacks Property on November 9, 2021.

i) I have had prior involvement in the Carmacks Property. I was the author of a previous NI 43-1-1 Technical Repot for the Property, dated April 29, 2022 for Granite Creek.

j) I am independent of Granite Creek Copper Ltd. and the Carmacks Property as defined by Section 1.5 of NI 43-101.

k) As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

l) I have read NI 43-101 and Form 43-101F1 (the "Form"), and the Technical Report has been prepared in compliance with NI 43-101 and the Form.

Signed and dated this 6th day of March 2023 at Fredericton, New Brunswick.

"Original Signed and Sealed"

Allan Armitage, Ph. D., P. Geo.

QP CERTIFICATE – WILLIAM VAN BREUGEL, P.Eng.

To Accompany the Report titled "Technical Report on the Carmacks Project Preliminary Economic Assessment (PEA) Yukon, Canada" dated March 6, 2023 (the "Technical Report") prepared for Granite Creek Copper Ltd.

a) I, William van Breugel, P. Eng. of Saskatoon, hereby certify that:

b) I am an Associate Mining Engineer for SGS Canada Inc, with an office located at 235 Ajawan Street, Christopher Lake, Saskatchewan, Canada.

c) I graduated from the University of Waterloo in 1990 (BaSc (Hons). Geological Engineering). I am a member of good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License #53342). I have worked as a mining engineer for over 31 years since my graduation from University. I have worked on precious metals, base metals, industrial commodities and diamond projects including mine operations and property evaluations. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").

d) I have not conducted a site visit of the property.

e) I am an author of this report and responsible for sections 1.11, 1.12, 1.14, 1.15, 1.16, 16.12, 18.7, 19, 20, 21, 22, and 25.2. I have reviewed these sections and accept professional responsibility for these sections of this technical report.

f) I am independent of Granite Creek Copper as defined in Section 1.5 of National Instrument 43-101.

g) I have had no prior involvement with the subject property.

h) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.

i) As at the effective date of the technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

j) I have read National Instrument 43-101, Form 43-101F1 and confirm that this technical report has been prepared in accordance therewith.

Signed and dated this 6th day of March 2023 at Saskatoon, Saskatchewan.

"Original Signed and Sealed"

William van Breugel, P.Eng.

QP CERTIFICATE – JOHNNY CANOSA, P. Eng.

To Accompany the Report titled “Technical Report on the Carmacks Project Preliminary Economic Assessment (PEA) Yukon, Canada” dated March 6, 2023 (the “Technical Report”) prepared for Granite Creek Copper Ltd.

I, Johnny Canosa, P. Eng. of Alberta, and Ontario, hereby certify that:

a) I am a Senior Mining Engineer for SGS Canada Inc, with an office located at 3260 Production Way, Burnaby, BC V5A 4W4, Canada.

b) I am a graduate of Bachelor of Science in Mining Engineering from Saint Louis University, Baguio City, Benguet, Philippines with diploma issue date on March 23, 1980. I am a member in good standing of the Association of Professional Engineers of Ontario (licence # 100509964) and the Association of Professional and Geoscientist of Alberta (licence #93946). My relevant experience includes more than 20 years of experience in mine engineering, mine planning and mining operation, including mine optimization, projects, open pit planning and scheduling, and mining consultancy.

c) I conducted a site visit to the Carmacks Property on June 20, 2022.

d) I am an author of this report and responsible for the preparation of sections 1.1, 1.7, 1.8, 1.10, 1.13, 1.17, 1.18, 1.19, 2, 3, 16 except for 16.12, 18 except for 18.7, 24, 25.3, 25.4, 26 and 27. I have reviewed these sections and accept professional responsibility for these sections of this technical report.

e) I am independent of Granite Creek Copper as defined in Section 1.5 of National Instrument 43-101.

f) I have had no prior involvement with the subject property.

g) I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.

h) As at the effective date of the technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

i) I have read National Instrument 43-101, Form 43-101F1 and confirm that this technical report has been prepared in accordance therewith.

Signed and dated this this 6th day of March 2023 at Surrey, British Columbia, Canada.

"Original Signed and Sealed"

Johnny Canosa, P.Eng.

QP CERTIFICATE – Joseph Keane, Pr Eng

To Accompany the Report titled “Technical Report on the Carmacks Project Preliminary Economic Assessment (PEA) Yukon, Canada” dated March 6, 2023 (the “Technical Report”) prepared for Granite Creek Copper Ltd.

I, Joseph M. Keane, P.E. do hereby certify that:

a) I am an independent mineral processing engineer consultant currently residing at 1061 W. Calle Santiago, Sahuarita, Arizona 85629 and am an associate of SGS North America Inc., 3845 North Business Center Drive, Suite 115, Tucson, Arizona 85705

b) I graduated with a degree of Bachelor of Science in Metallurgical Engineering from the Montana School of Mines in 1962. I obtained a Master of Science degree in Mineral Processing Engineering in 1966 from the Montana College of Mineral Science and Technology. In 1989, I received a Distinguished Alumni Award from that institution. I have worked as a mineral processing engineer for a total of 55 years since my graduation from university. I am a member of the Society for Mining, Metallurgy, and Exploration, Inc. (SME #1682600) and am a registered professional metallurgical engineer in Arizona (#12979) and Nevada #5462).

c) I have not visited the property.

d) I am an author of this report and am responsible for Sections 1.5, 1.9, 13, and 17. I. have reviewed these sections and accept professional responsibility for these sections of this technical report.

e) I am independent of Granite Creek Copper Ltd. as defined in Section 1.5 of National Instrument 43-101.

f) I have had no prior involvement with the property.

g) I have read the definition of "qualified person" set out in N 43-101 and certify that by reason of my education, affiliation with professional associations (as noted above) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

h) As of the effective date of this certificate, to the best of my knowledge, information, and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

i) I have read National Instrument 43-101, Form 43-101F1 and confirm that this technical report has been prepared in accordance therewith

Signed this this 6th day of March 2023 at Tucson, Arizona USA.

"Original Signed and Sealed"

Joseph M. Keane, P.E.
Qualified Person, Consultant to SGS Bateman