TECHNICAL REPORT

ON THE

UPDATED MINERAL RESOURCE ESTIMATES FOR THE CARMACKS CU-AU-AG PROJECT

NEAR CARMACKS, 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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Company
SGS Geological Services (“SGS”)

SGS Project # P2021-34

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1 SUMMARY

SGS Geological Services ("SGS") was contracted by Granite Creek Copper Ltd. ("Granite Creek" or the "Company") to complete updated Mineral Resource Estimates ("MREs") for several copper deposits of their Carmacks Cu-Au-Ag Project ("Carmacks Project" or the "Property") and to prepare a technical report written in support of the current MREs. The reporting of the MREs comply with all disclosure requirements for Mineral Resources set out in the NI 43-101 Standards of Disclosure for Mineral Projects (2016). The classification of the MREs are consistent with current CIM Definition Standards - For Mineral Resources and Mineral Reserves (2014). Granite Creek acquired 100% of the Carmacks Project in 2020 through its acquisition of Copper North Mining Corp ("Copper North").

The current report also includes information regarding the adjacent Carmacks North Copper Property ("Carmacks North"), formerly the Stu Copper Property ("Stu Property"). On September 13, 2018 Granite Creek entered into an agreement with an arm's length vendor (William G. "Bill" Harris), under which the Company acquired an undivided 100% interest in the Stu Property, resulting in a consolidation the Carmacks and Carmacks North properties.

The Carmacks and Carmacks North Properties are located in the Minto Copper Belt, an area of well-known copper-gold-silver mineralization in Canada's Yukon Territory. Situated approximately 47 km northeast of the village of Carmacks, and approximately 175 km northwest of Whitehorse, the capital city of the Yukon Territoryhe project is within 20 km of grid power and paved highway. The combined projects cover approximately 17,580 hectares (176 square km) and are on trend with the Minto copper mine approximately 35 km north of the center of the project.

Granite Creek is a growth stage exploration company, focused on the acquisition and development of exploration properties that host copper, gold and silver. Granite Creek was originally incorporated on May 10, 2007 under the British Columbia Business Corporations Act. The Company is a reporting issuer and trades on the TSX Venture Exchange ("TSX-V") in Canada under the symbol "GCX", in the United States on the OTC Markets under the symbol "GCXXF" and the Frankfurt Stock Exchange under the symbol "A2PFE0". The Company's principal business is the acquisition, exploration and development of mineral properties with the goal of establishing a mineable mineral resource. Granite Creek is a member of the Metallic Group of Companies. Their current business address is Suite 904-409 Granville Street, Vancouver, BC Canada V6C 1T2.

This technical report will be used by Granite Creek in fulfillment of their continuing disclosure requirements under Canadian securities laws, including National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101"). This technical report is written in support of updated resource estimates for several copper deposits on the Carmacks Project released by the Company on March 15, 2022.

The current report is authored by Allan Armitage, Ph.D., P. Geo., ("Armitage" or the "Author") of SGS. Armitage is an independent Qualified Person as defined by NI 43-101 and is responsible for all sections of this report.

1.1 Property Description, Location, Access, 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 includes 974 claims that covers approximately 17,580 hectares (175.80 square km). 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. 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.

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 yearround 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 discontinueous 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.

1.2 **History, Exploration and Drilling**

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 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. 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.

1.2.1 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.

1.3 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 (

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.

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 and Drilling

Since 2014, exploration work on the Carmacks Project has been predominantly drilling. 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 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; 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.

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 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.

The South Gap target zone consists of an approximate 300 metre gap between Zone 1 and the 2000S zone. The 2000S zone was intersected by 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.

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. 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. 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

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 drilling in zones 2000S, 13 and 12 confirmed the continuity of these 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. A total of 36 holes were completed for 4,175 m.

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.

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)

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.

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:

- 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.

Between May and September, 2021, Granite Creek completed 7,742 m of diamond drilling in 23 holes on the Carmacks Propert. Highlights of the 2021 drill program are presented in Table 1-1. 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.

Table 1-1 2021 Drill Results

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	

Drillhole	From (m)	To (m)	Length* (m)	Cu (%)	Mo (%)	Au (g/t)	Ag (g/t)	CuEq** (%)	Target
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	
CRM21-022	233.7	302	68.3	0.51	0.009	0.13	2.3	0.66	
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	
CRM21-015	36.69	49.38	12.69	0.23	0.003	0.04	0.96	0.27	Zone 13
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	
CRM21-024	54.8	93	38.2	0.79	0.005	0.16	3.27	0.95	
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	

1.5 Mineral Processing and Metallurgical Testing

Metallurgical testwork on various ore samples started in 1989 and has been ongoing since that time. These tests have included:

- 27 bottle roll tests;
- 45 column tests;
- One crib test near site; and,
- SX/EW testing by a manufacturer.

The metallurgical testing program on the Carmacks Project focused on the recovery of acid soluble copper mineralization in the oxide cap of the Zone 1 deposit. The primary emphasis of the work conducted up to 2012, has been on development of design criteria and optimal operating parameters for heap leaching the crushed and agglomerated mineralized material, followed by solvent extraction for solution concentration and purification and electrowinning for recovery of cathode copper metal. Some limited testing has been performed on heap leaching using run of mine (ROM) mineralized material, examining leaching of the sulphide mineralization, and recovering gold following copper recovery.

The recovery of gold and silver from copper leach residues was examined as part of the previous testwork but was not reported. Specifically, the copper leach metallurgical work undertaken in 2009 by PRA Metallurgical Division of Inspectorate America Corporation included preliminary testing of gold and silver leaching from copper leach residues by cyanidation. The testwork was conducted in two columns, with the results indicating a gold recovery of 78% and a silver recovery of 75%.

The testwork also indicated that a portion of the copper remaining after the copper leach would be extracted during gold/silver cyanidation and that this dissolved copper could be recovered in the SART circuit associated with the gold and silver ADR plant, with concurrent regeneration of cyanide for recirculation to the process. For the purpose of the present study, the cyanide consumption as NaCN has been estimated at 0.5 kg/tonne.

In 2014, Copper North examined the value of adding precious metals recovery to the project plan using a two-stage heap leaching approach. Results were reported in the PEA prepared by Merit International Consultants. This study indicated the value of recovering gold and silver, leading to further metallurgical test work and the present study.

During 2014-2015, Bureau Veritas Commodities Canada (BV Minerals) completed a full suite of metallurgical testing to evaluate an alternative to heap leaching. This new vat leach recovery method focused on grinding the samples to a P_{80} of 664 µm, and leaching with sulphuric acid to recover the copper. The leach residue was then leached with cyanide to recover the gold and silver. Metallurgical testing included flowsheet parameter finalization, a full locked cycle Cu/Au leach test, cyanide destruction, and variability comminution and batch leach testing.

Thirteen different composites were used in the 2014 test program. In Phase 1, four trench composites (BS-1, BS-2, BS-3 and BS-4) were created and a Master Composite was constructed from core samples. In Phase 2, eight variability composites were created from the remaining core samples kept in storage. In February 2016, BV Minerals conducted additional copper leach optimization test work using 2014 Master Composite.

Based on the test work, a copper/gold leach circuit was selected as the preferred recovery method. The criteria and recoveries from CALT2 were selected for design due to the low copper grade reporting to the gold/silver leach circuit, eliminating the need for a sulphidization, acidification, recycling and thickening (SART) process. Mineralized material will be reduced to a P_{80} of 664 µm using a jaw crusher followed by a SAG mill in closed circuit with hydrocyclones. Copper will be recovered using a sulphuric acid leach and solvent extraction / electrowinning (SX-EW). Normally high silver grades and low gold grades dictate the use of Merrill Crowe; however ferric sulphate addition substantially reduces silver recovery, allowing for a smaller footprint with a CIL circuit. Copper leach residue will be neutralized and gold/silver will be leached into solution using cyanide while simultaneously being adsorbed onto activated carbon. An adsorption, desorption and refining (ADR) circuit will be implemented to concentrate the gold/silver into doré bars. The resulting tailings residue is then passed through an Inco SO2-Air circuit for destruction of the residual cyanide, followed by filtration and placement in the TMA as dry stack tailings.

In 2021, metallurgy testing was completed on sulphide and oxide mineralized core samples from the Carmacks Project. Two representative samples of copper sulphide material and one sample of copper oxide material were delivered to Bureau Veritas Commodities, Metallurgy Division, for rougher flotation kinetic testing and open cleaner flotation testing. The purpose of the testing was to determine how amenable the sulphide mineralization present at the Carmacks deposit was to concentration by flotation, what recoveries could be expected, and to lay the groundwork for further testing. A preliminary copper flotation recovery model was generated at a fixed 25% copper concentrate grade with test results as summarized below:

- Recoveries of greater than 95% for copper into a 25% copper concentrate are possible.
- Copper sulphide minerals are well-liberated for rougher recovery via flotation at P_{80} of 150 µm.
- A secondary regrind size at P_{80} of 25 µm can achieve 25% copper grade and high cleaner stage recovery.
- Gold is associated primarily with copper sulphide minerals and minor pyrite. Flotation of gold with copper concentrate is likely the most economical way to recover gold.
- High chalcopyrite content as the copper mineral and low pyrite content within the samples indicate a simple reagent scheme and relatively easy copper flotation upgrade.

The preliminary tests indicate that well-established flotation methods with known reagents will likely be the preferred processing method for sulphide material at Carmacks. The next stage of metallurgical test work will involve greater variability of samples to validate copper and gold recoveries as well as assessing potential levels of silver and molybdenum recoveries.

1.6 2022 Carmacks Project Mineral Resource Statement

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 1-2. 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 Recommendations

The copper deposits of the Carmacks Project contain within-pit and underground Measured, Indicated and Inferred Mineral Resources that are associated with well-defined mineralized trends and models. All deposits are open along strike and at depth. The Author considers that the Project has potential for delineation of additional oxide and sulphide Mineral Resources along strike, down dip and elsewhere on the Carmacks Property and that further exploration is warranted.

Granite Creek is planning on continuing exploration on the Carmacks Property in 2022. The exploration program is to include geophysics, trenching, soil sampling and diamond drilling (~ 7,500 m) in several areas, as well as re-logging of old core and reclamation and baseline environmental studies. The total cost of the recommended work program is estimated at C$3,533,500 million.

The 2022 exploration program at the Carmacks Property will be in completed in two phases. The first phase centres on completion of a PEA study and report, in conjunction with field work including IP geophysics. While the camp is operating to support the geophysics survey, activities such as trenching, soil sampling, relogging core, sampling for additional metallurgical testwork, reclamation and baseline environmental studies will be conducted.

The budget for the second phase is preliminary and based on an all-in cost of $325 per metre of diamond drilling. This dollar amount is based on previous road-based diamond drill programs at Carmacks and does not include other exploration activities. Targetting of the Phase 2 drill program is contingent on the results of the Phase 1 program.

Given the prospective nature of the Carmacks Property, it is the Author's opinion that the Carmacks Property merits further exploration and that the proposed 2022 plan for further work by Granite Creek is justified. A proposed work program by Granite will help advance the deposits and will continue to provide key inputs required to further evaluate the economic viability of the Carmacks Project.

The Author is recommending Granite Creek conduct further exploration, subject to funding and any other matters which may cause the proposed exploration program to be altered in the normal course of its business activities or alterations which may affect the program as a result of exploration activities themselves.

2 INTRODUCTION

SGS Geological Services ("SGS") was contracted by Granite Creek Copper Ltd. ("Granite Creek" or the "Company") to complete updated Mineral Resource Estimates ("MREs") for several copper deposits of their Carmacks Cu-Au-Ag Project ("Carmacks Project" or the "Property") and to prepare a technical report written in support of the current MREs. The reporting of the MREs comply with all disclosure requirements for Mineral Resources set out in the NI 43-101 Standards of Disclosure for Mineral Projects (2016). The classification of the MREs are consistent with current CIM Definition Standards - For Mineral Resources and Mineral Reserves (2014). Granite Creek acquired 100% of the Carmacks Project in 2020 through its acquisition of Copper North Mining Corp ("Copper North").

The current report also includes information regarding the adjacent Carmacks North Copper Property ("Carmacks North"), formerly the Stu Copper Property ("Stu Property"). On September 13, 2018 Granite Creek entered into an agreement with an arm's length vendor (William G. "Bill" Harris), under which the Company acquired an undivided 100% interest in the Stu Property, resulting in a consolidation the Carmacks and Carmacks North properties.

The Carmacks and Carmacks North Properties are located in the Minto Copper Belt, an area of well-known copper-gold-silver mineralization in Canada's Yukon Territory. Situated approximately 47 km northeast of the village of Carmacks, and approximately 175 km northwest of Whitehorse, the capital city of the Yukon Territory, the project is within 20 km of grid power and paved highway. The combined projects cover approximately 17,580 hectares (176 square km) and are on trend with the Minto copper mine approximately 35 km north of the center of the project.

Granite Creek is a growth stage exploration company, focused on the acquisition and development of exploration properties that host copper, gold and silver. Granite Creek was originally incorporated on May 10, 2007 under the British Columbia Business Corporations Act. The Company is a reporting issuer and trades on the TSX Venture Exchange ("TSX-V") in Canada under the symbol "GCX", in the United States on the OTC Markets under the symbol "GCXXF" and the Frankfurt Stock Exchange under the symbol "A2PFE0". The Company's principal business is the acquisition, exploration and development of mineral properties with the goal of establishing a mineable mineral resource. Granite Creek is a member of the Metallic Group of Companies. Their current business address is Suite 904-409 Granville Street, Vancouver, BC Canada V6C 1T2.

This technical report will be used by Granite Creek in fulfillment of their continuing disclosure requirements under Canadian securities laws, including National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101"). This technical report is written in support of updated resource estimates for several copper deposits on the Carmacks Project released by the Company on March 15, 2022. Granite Creek reported that the deposits of the Carmacks Project contains 36.2 Mt (M&I), grading 1.07% CuEq (0.81% Cu, 0.26g/t Au, 3.23g/t Ag and 0.011% Mo) for a total of 651 Mlbs contained M&I copper with an additional 38 Mlbs Cu Inferred (0.3 % Cu Total cut-off grade). The effective date of the resource estimate is February 25, 2022. Details of the MRE is presented in Section 14.

The current report is authored by Allan Armitage, Ph.D., P. Geo., ("Armitage" or the "Author") of SGS. Armitage is an independent Qualified Person as defined by NI 43-101 and is responsible for all sections of this report.

2.1 Sources of Information

The data used in the estimation of the update MREs and the development of this report was provided to SGS by Granite Creek. Some information including the property exploration history and regional and property geology (Sections 5 to 9) have been sourced from previous Technical Reports and revised or updated as required. The current Technical Report also benefits from extensive discussions with Granite Creek personnel regarding the geology of the deposits and results of recent exploration programs completed by Granite Creek.

The Carmacks Project was the subject of several recent NI 43-101 Technical Reports for Copper North including:

- NI 43-101 Preliminary Economic Assessment Technical Report on the Carmacks Project, Whitehorse, Yukon, Canada, dated November 25, 2016, with an effective date of October 12, 2016, prepared for Copper North Mining Corp.
- Independent Technical Report on the Carmacks Copper Project, Yukon, Canada, dated March 10, 2016, with an effective date of January 25, 2016, prepared for Copper North Mining Corp.
- Carmacks Project - Preliminary Economic Assessment of Copper, Gold, and Silver Recovery, dated July 10, 2014, with an effective date of July 10, 2014, prepared for Copper North Mining Corp.

The Carmacks North Project was was the subject of a recent NI 43-101 Technical Report for Granite Creek including:

- NI 43-101 Technical Report on the Stu Copper Property near Carmacks, Yukon, Canada, dated, January 16, 2019, with an effective date of November 15, 2018, prepared for Granite Creek Copper Ltd.

In addition, the Author has reviewed company news releases and Management's Discussions and Analysis ("MD&A") which are posted on SEDAR (www.sedar.com).

SEDAR, "The System for Electronic Document Analysis and Retrieval", is a filing system developed for the Canadian Securities Administrators to:

- facilitate the electronic filing of securities information as required by Canadian Securities Administrator;
- allow for the public dissemination of Canadian securities information collected in the securities filing process; and
- provide electronic communication between electronic filers, agents and the Canadian Securities Administrator

The Author has carefully reviewed all of the Carmacks Project information and assumes that all of the information and technical documents reviewed and listed in the "References" are accurate and complete in all material aspects.

The Author believes the information used to prepare the current Technical Report is valid and appropriate considering the status of the Carmacks Project and the purpose of the Technical Report. By virtue of the Authors' technical review of the Carmacks Project, the Author affirms that the work program and recommendations presented herein are in accordance with NI 43-101 requirements.

2.2 Site Visit

The Author 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 was accessed via helicopter from Whitehorse. Drilling was not underway during the site visit and the exploration camp was shut down for the season.

During the site visit, the Author 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 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.

The Author 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, 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.

2.3 Units and Abbreviations

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 US dollars, unless otherwise noted. Frequently used abbreviations and acronyms can be found in Table 2-1.

Table 2-1 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	oz	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
ft2	Square feet	QP	Qualified Person
ft3	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		

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 the Author by Granite Creek on April 18, 2022, by way of e-mail.

Author 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, the Author has no reason to doubt that the title situation is other than what is presented in this technical report. The Author is not qualified to express any legal opinion with respect to Carmacks Property titles or current ownership.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 **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 (**Error! Reference source not found.**). 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

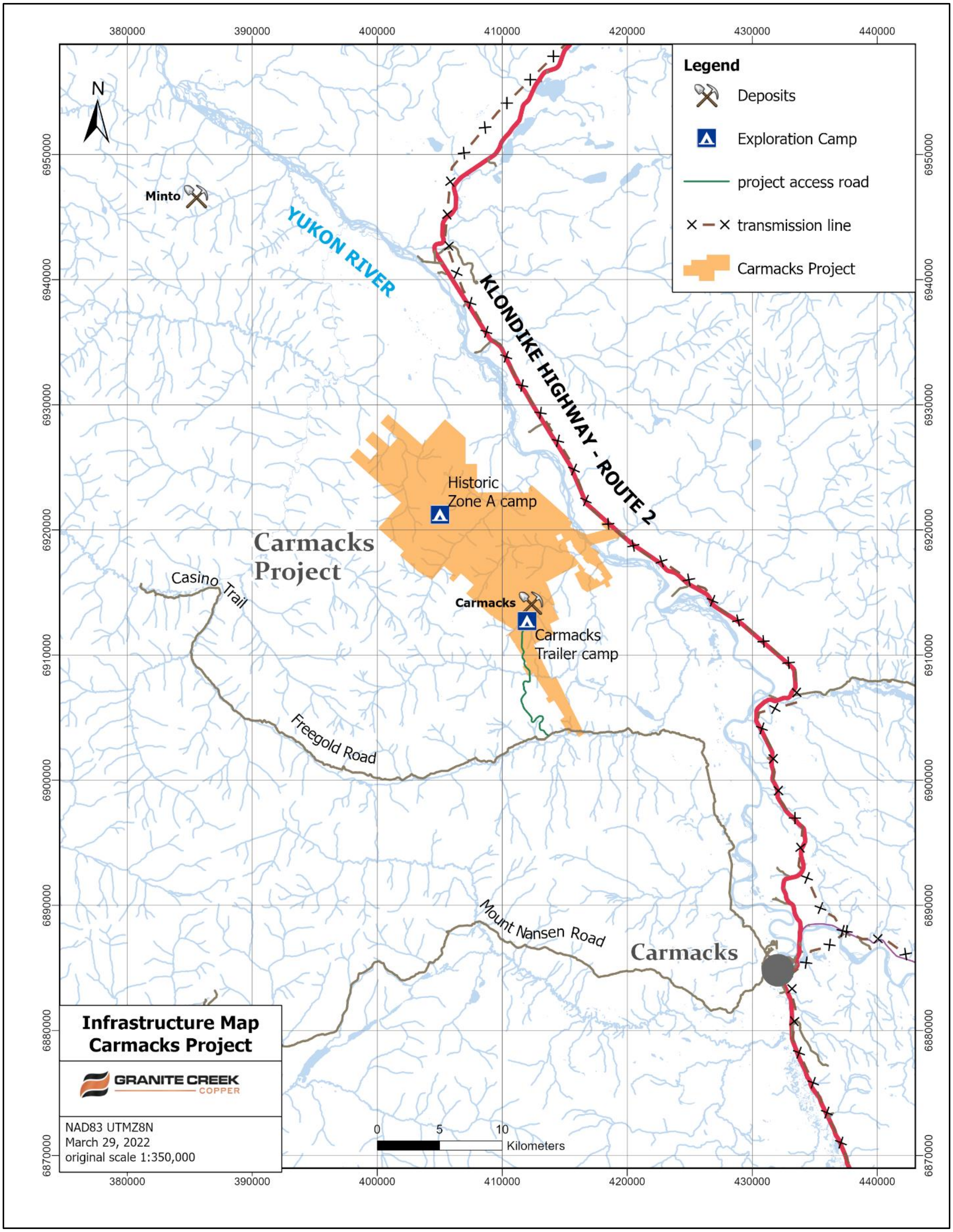


Figure 4-3 Carmacks Project Property Map

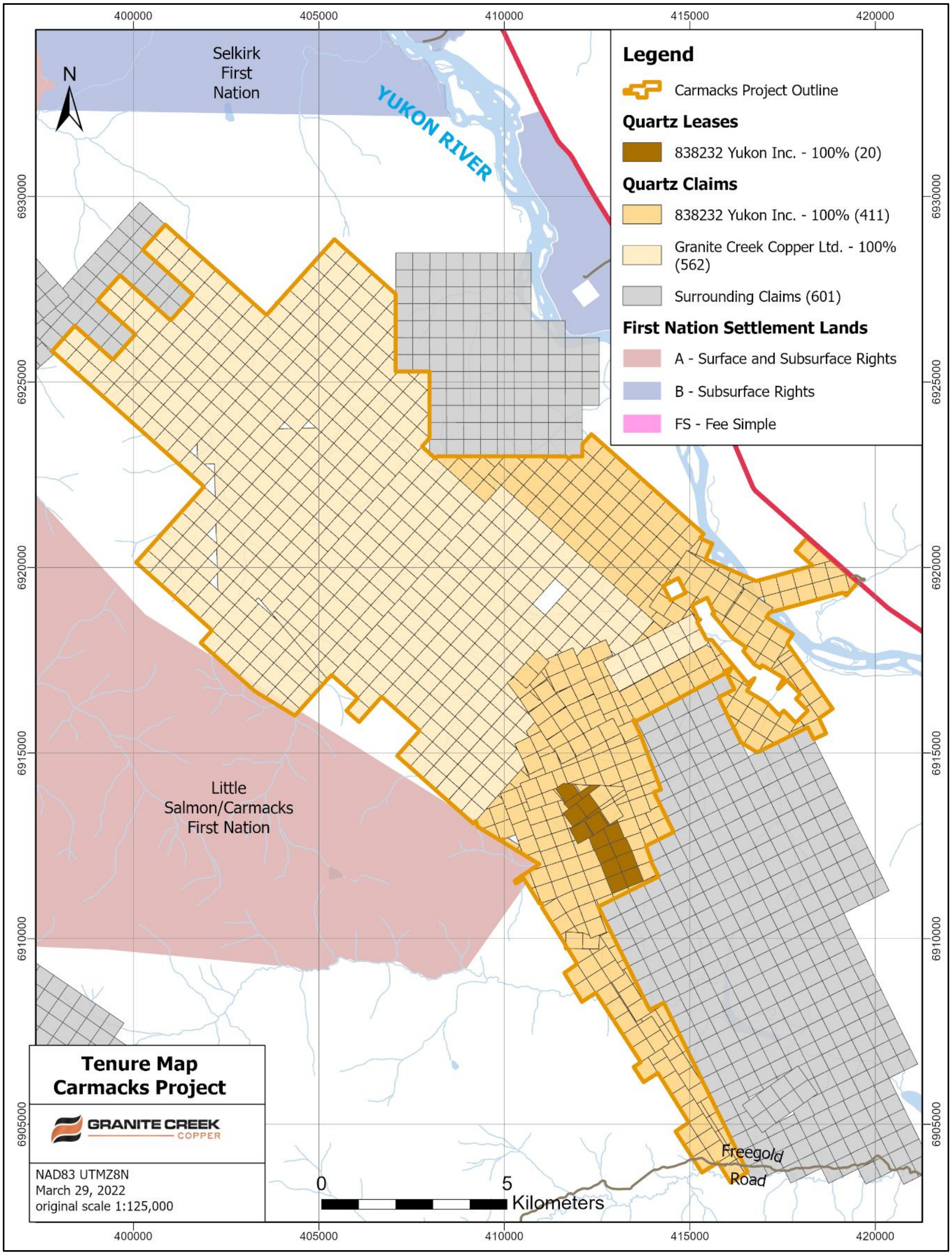


Figure 4-4 Carmacks and Carmacks North Project Tenure Map – South Half

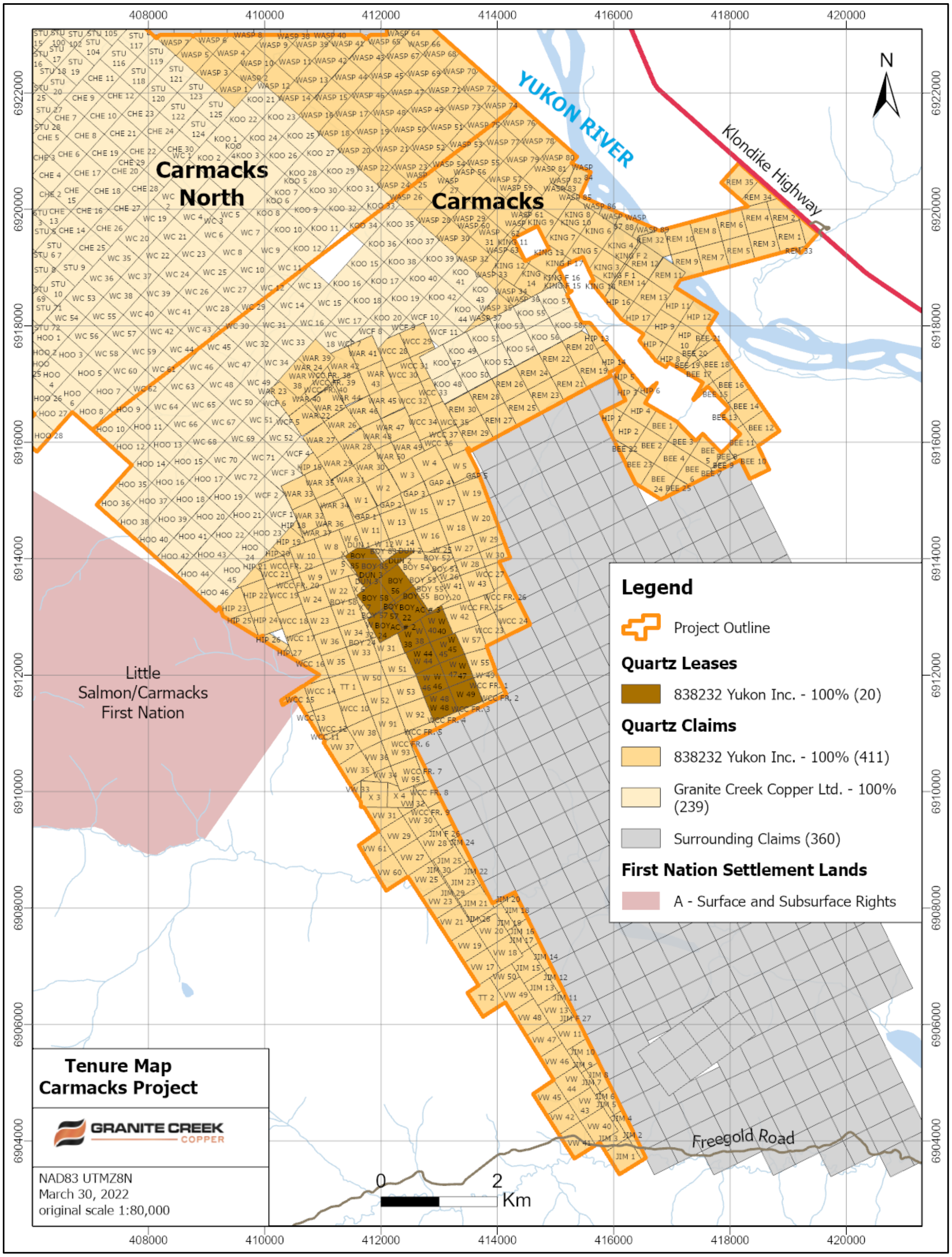


Figure 4-5 Carmacks and Carmacks North Project Tenure Map – North Half

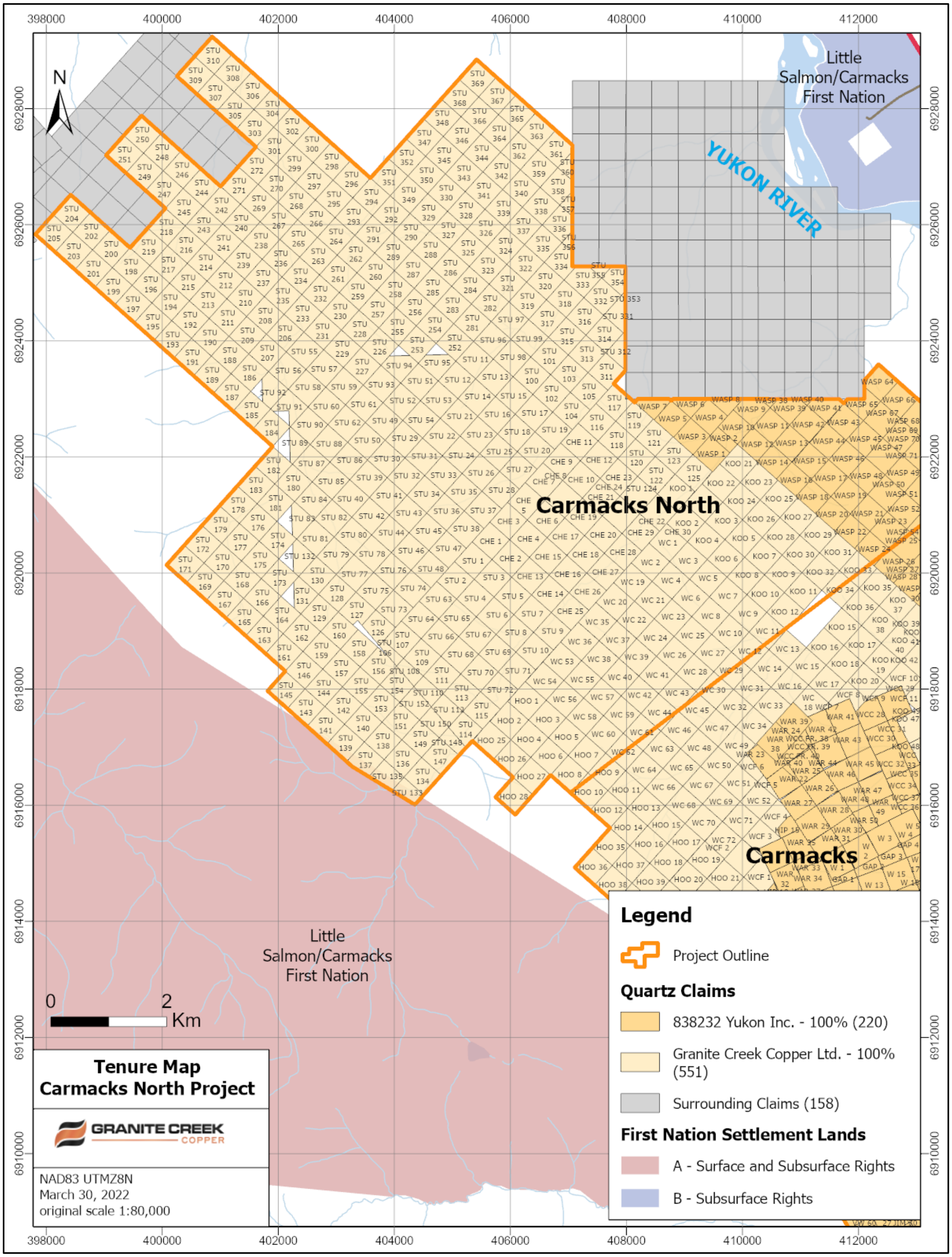


Table 4-1 Carmacks and Carmacks North Listing of Claims and Leases

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
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

Grant Number	Owner	Lease	Claim label	No.	Date staked	Expiry date
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
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

Grant Number	Owner	Lease	Claim label	No.	Date staked	Expiry date
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 gem stones.

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 propertties.

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 decomissioning.

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

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 yearround 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 discontinueous 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 Piesold 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 Piesold 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

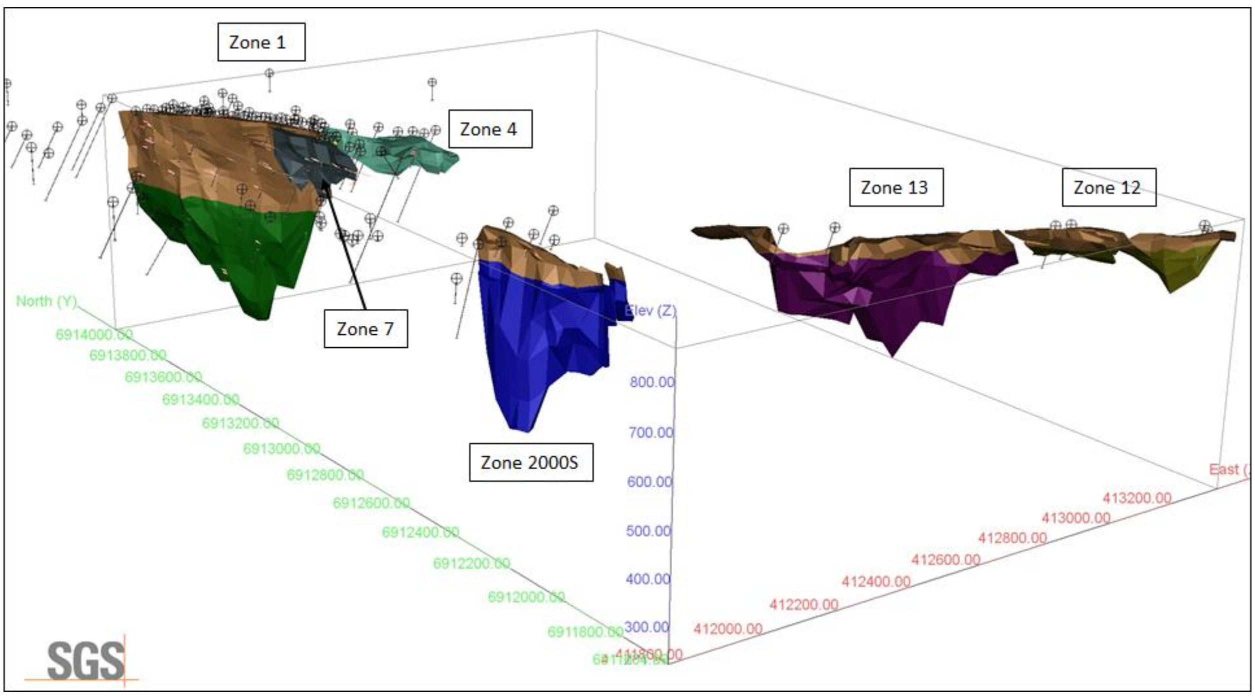
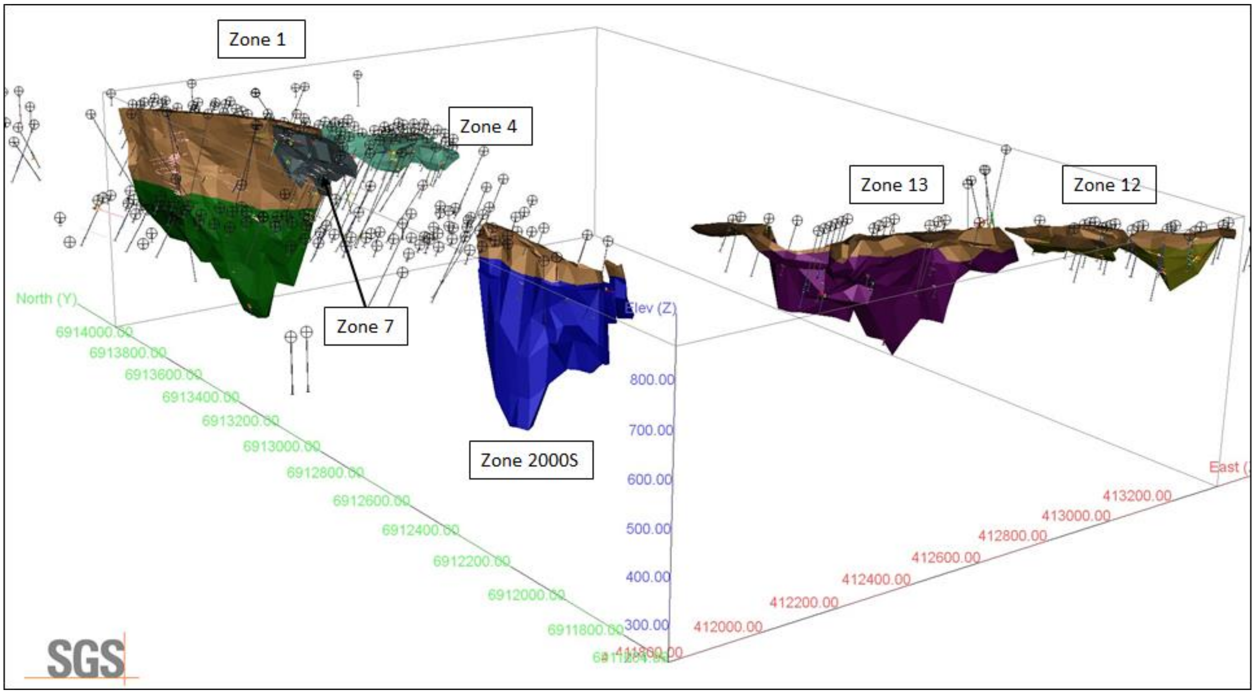


Figure 6-2 Isometric View looking Northeast: 2006 – 2008 Drilling in the Carmacks Carmacks Oxide (brown) and Sulphide Deposit Areas



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:

$$\text{Cu Sulphide\%} = \text{Cu Total\%} - \text{Cu Oxide\%}$$

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:

$$\text{Cu Oxide Proportion} = \text{Cu Oxide} / \text{Cu Total} * 100\%$$

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.

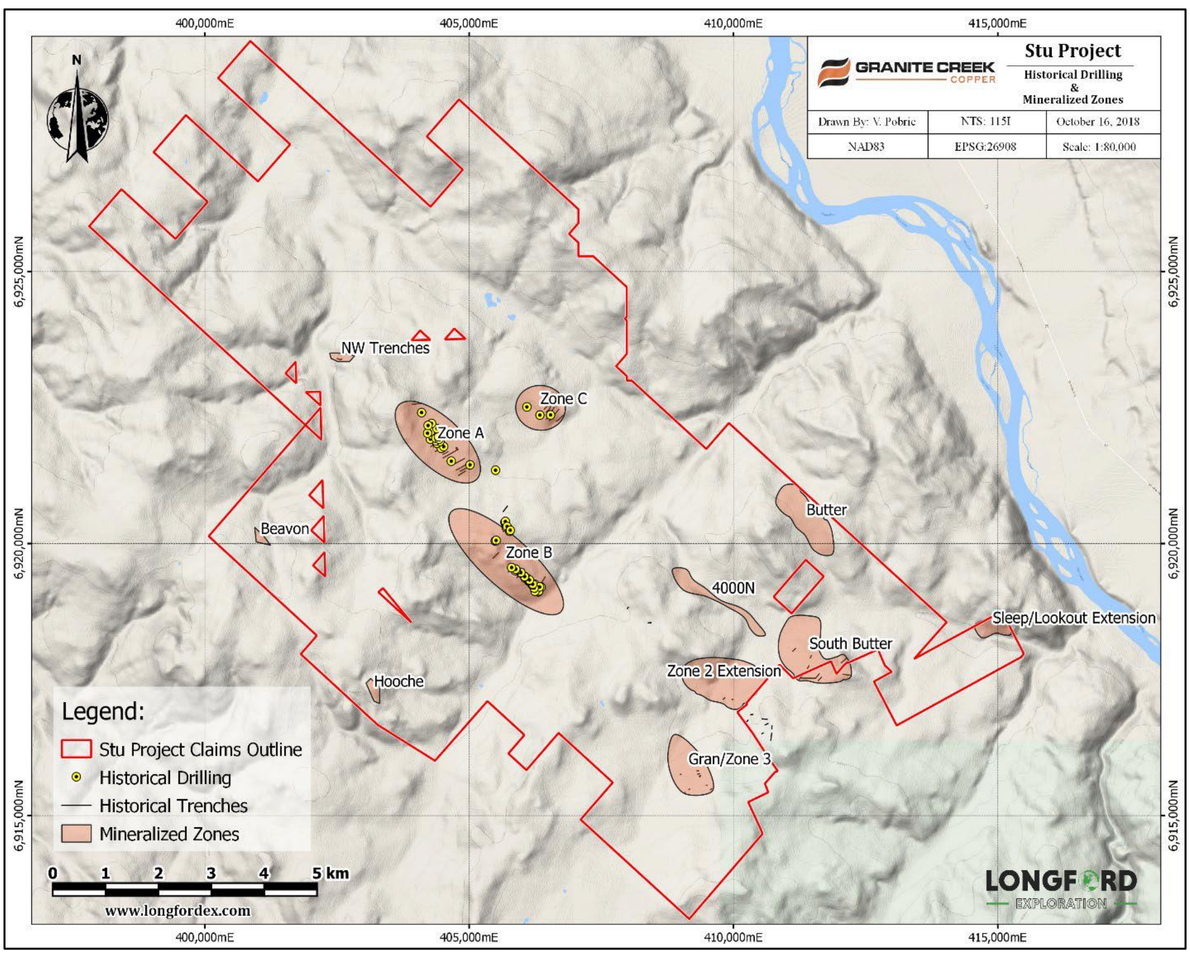
6.3 Carmacks North Property Exploration History (formerly the Stu Property)

The following summary of description 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 reassaying 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.

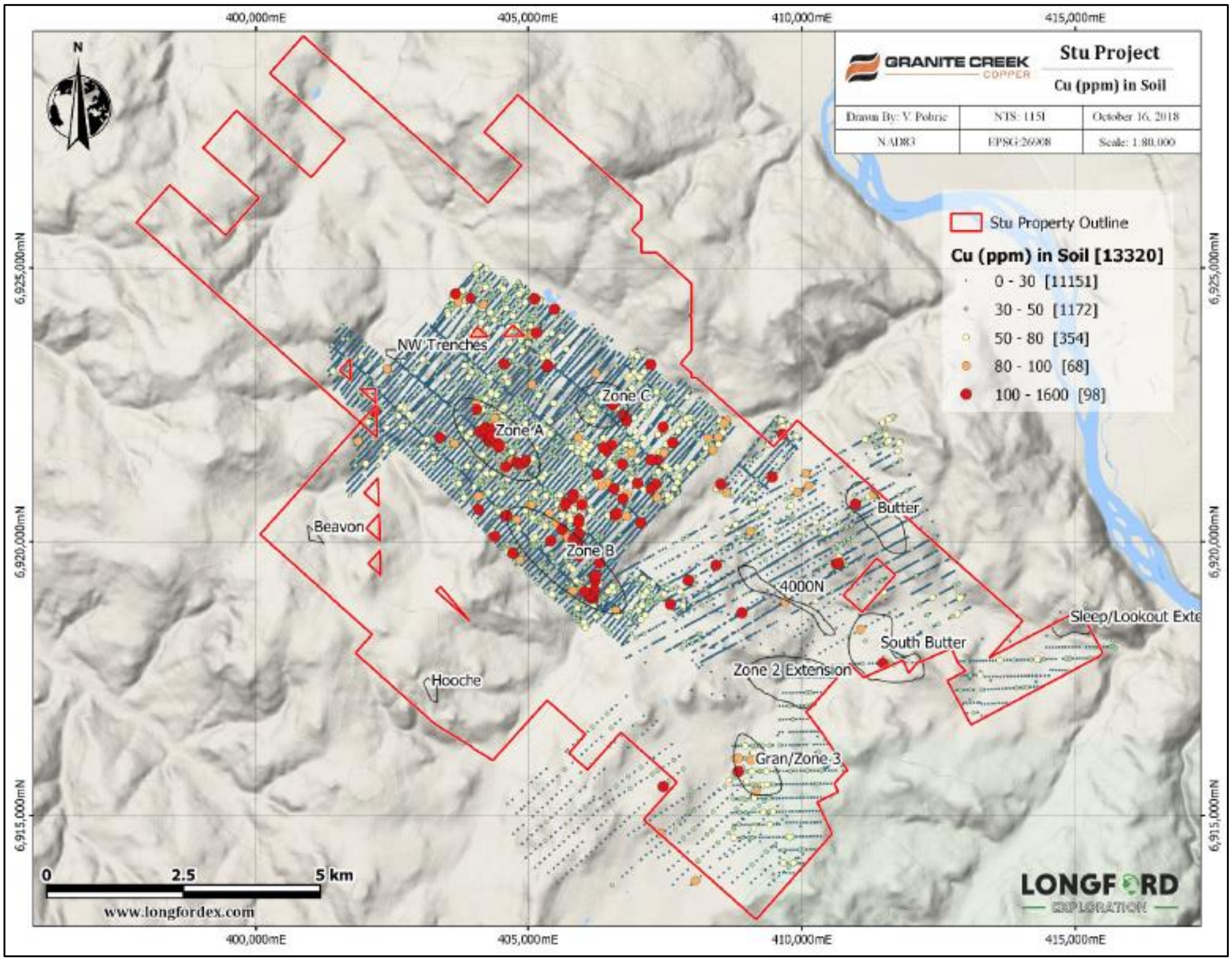
A series of large property-wide soil sampling programs were undertaken by UKHM in 1977-1981. Samples were collected along cutline grids at 30 m intervals along lines 100m apart. Zones A, B and C were outlined along with other northwest trending anomalies to the south and east. In the southern part of 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.

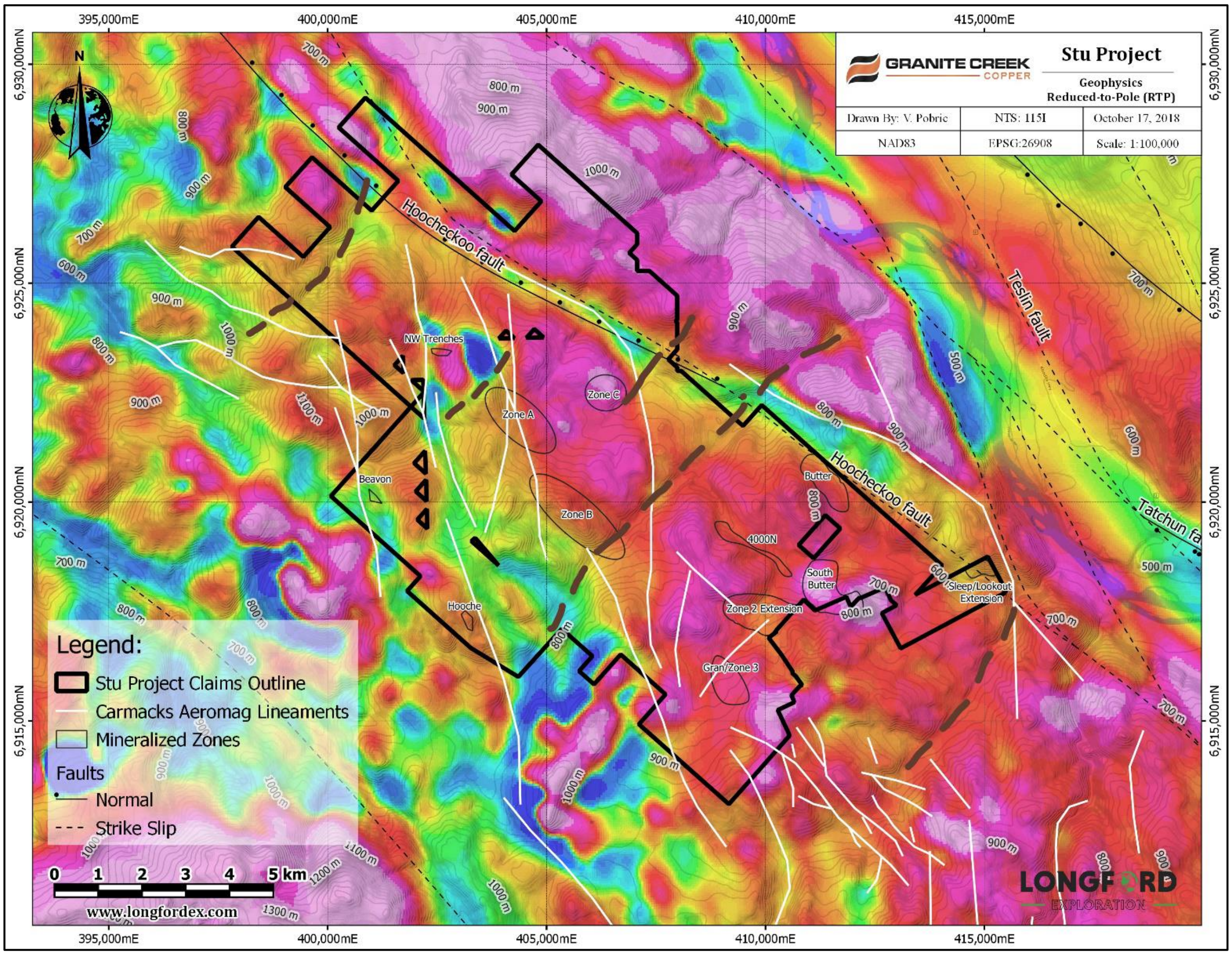
The 1974 Bay claims VLF-EM and IP geophysical surveys found linear geophysical anomalies were found between Hoocheekoo and Nancy Lee Creeks, over the Butter showing and southwest towards the 4000N zone.

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)

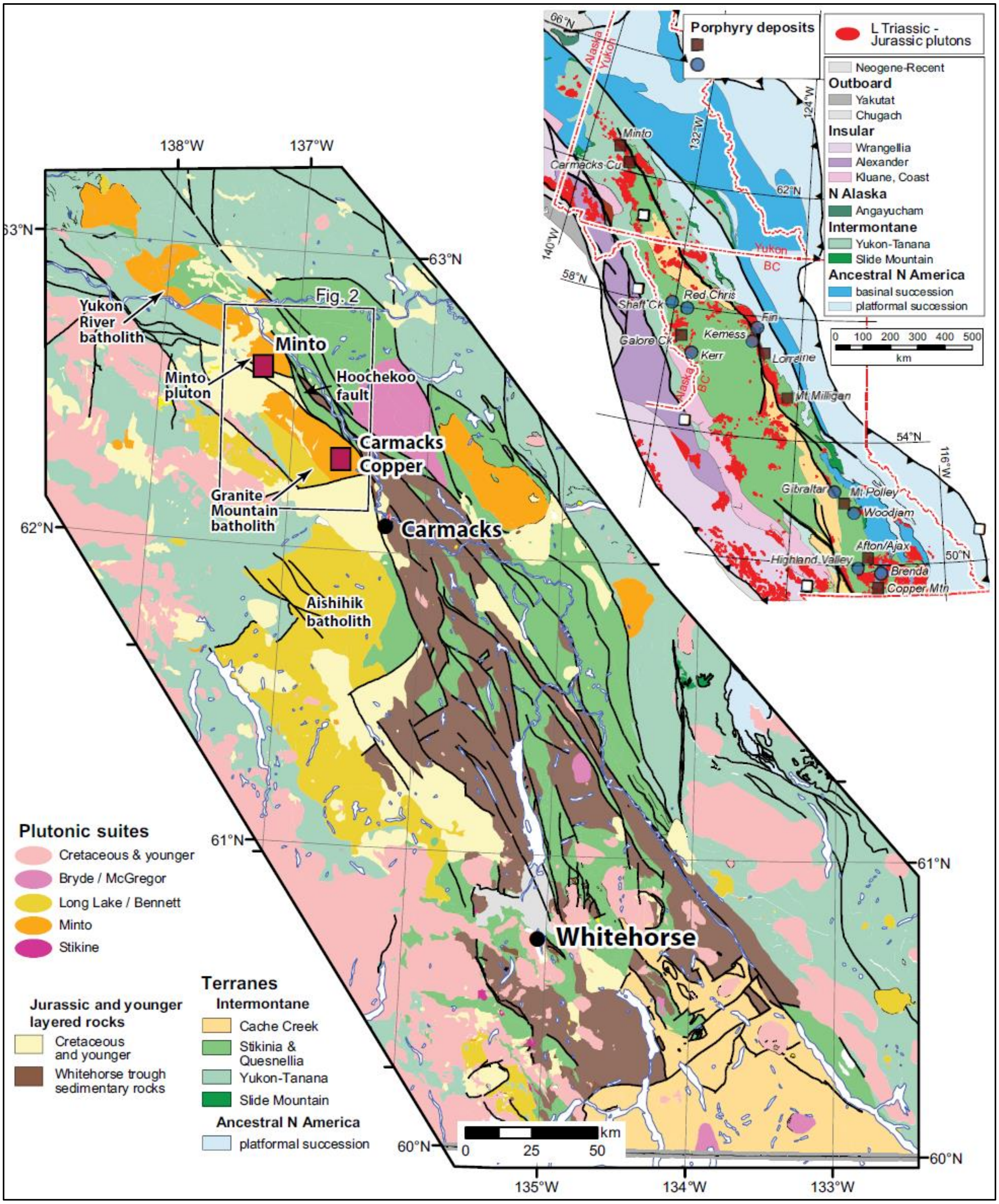
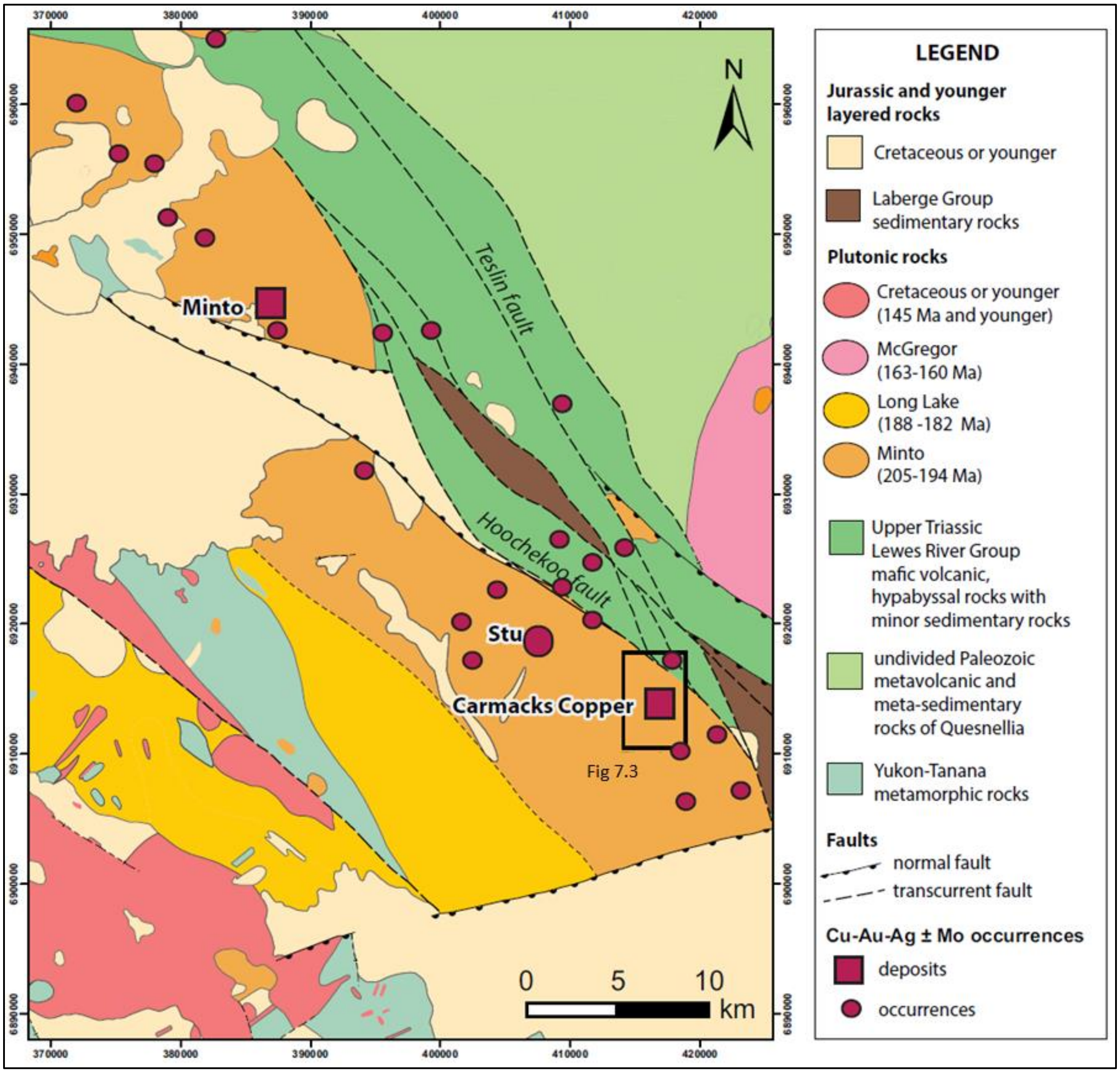


Figure 7-2 Geology of the Granite Mountain Batholith Area (Grid in UTM Projection, Zone 8N, NAD 83) (from Kovacs, 2020)



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 7-3b). However, this does not appear to 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 SiO_2 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.

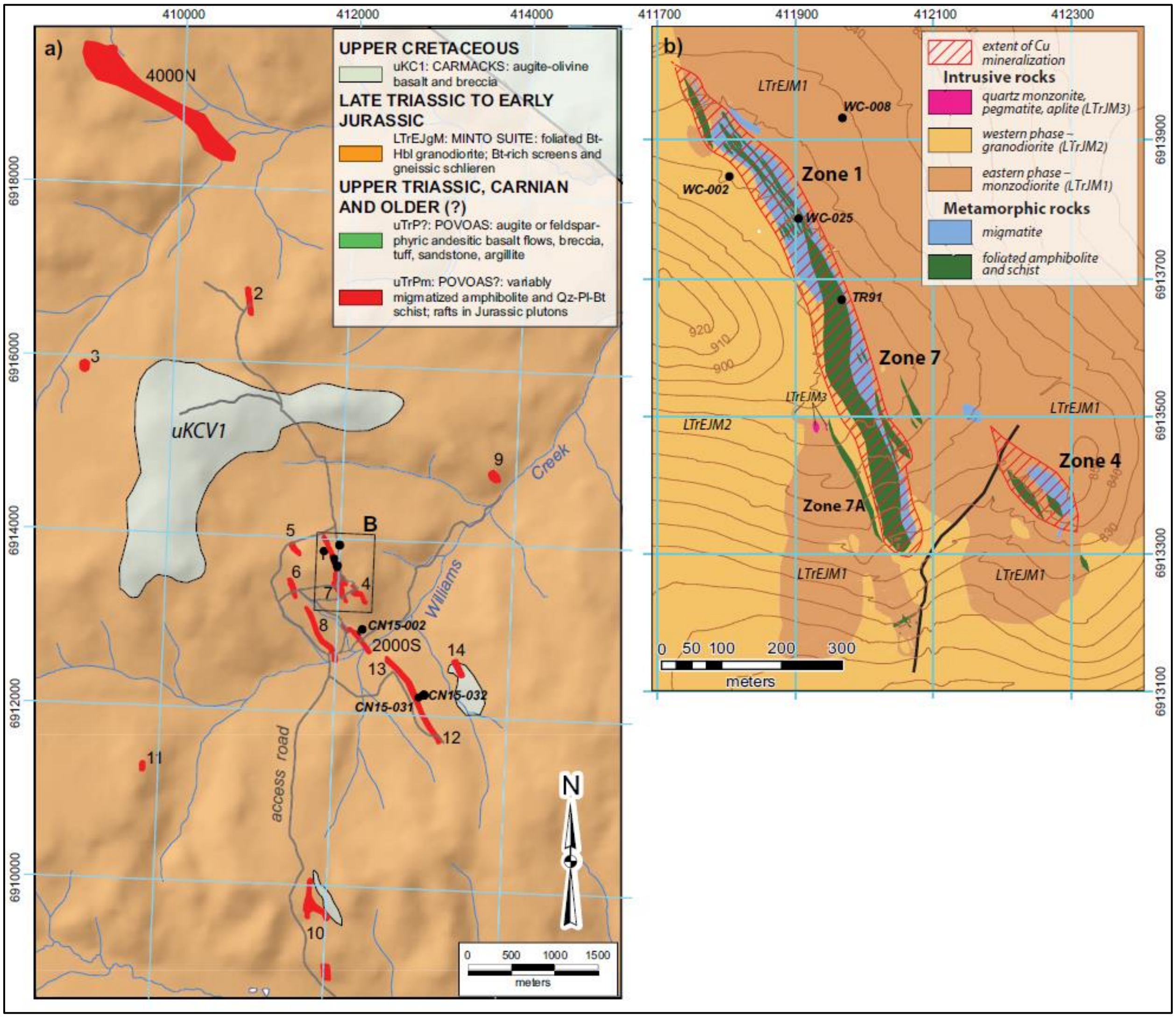
Metamorphic rocks

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 monominerallic 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:

5. A fault that define the southern end of 2000S zone
6. A north zone 1 fault
7. A N-S trending fault that defines the graben to the east of zone 12
8. 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.

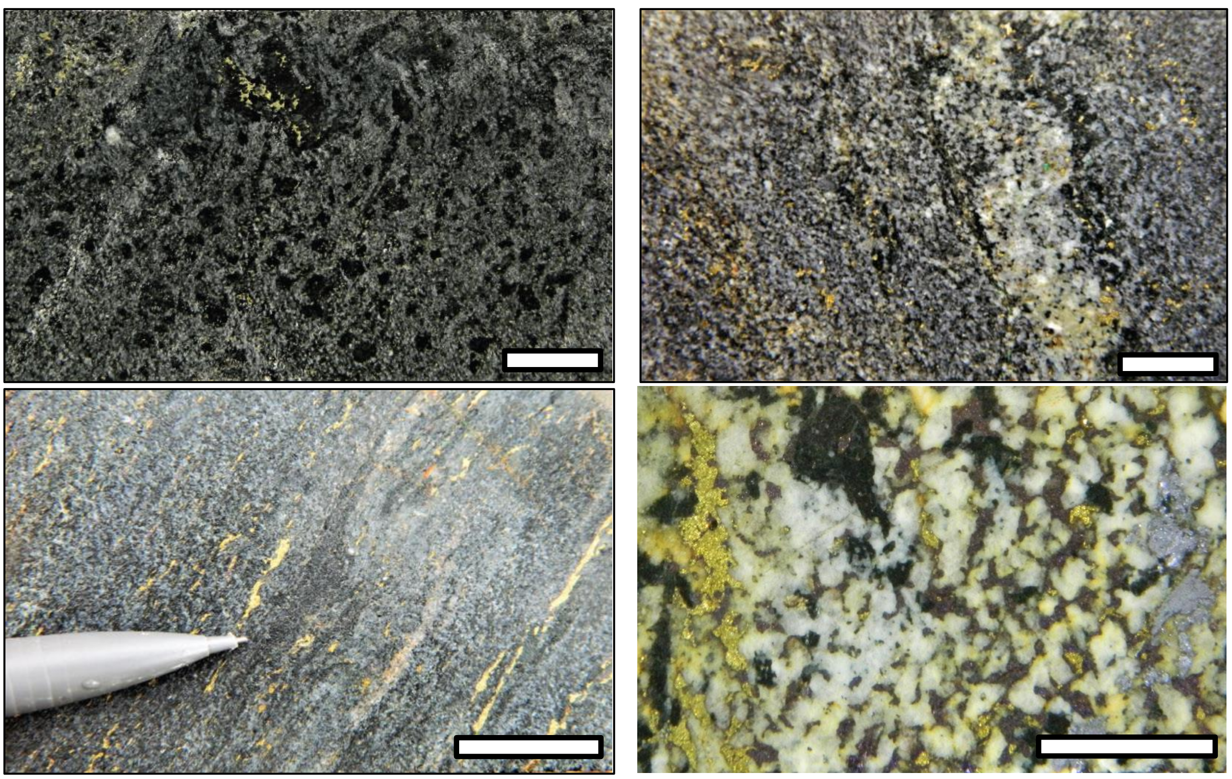


Figure 7-5 Reflected Light Petrography of Copper Mineralization at Carmacks Copper –

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.

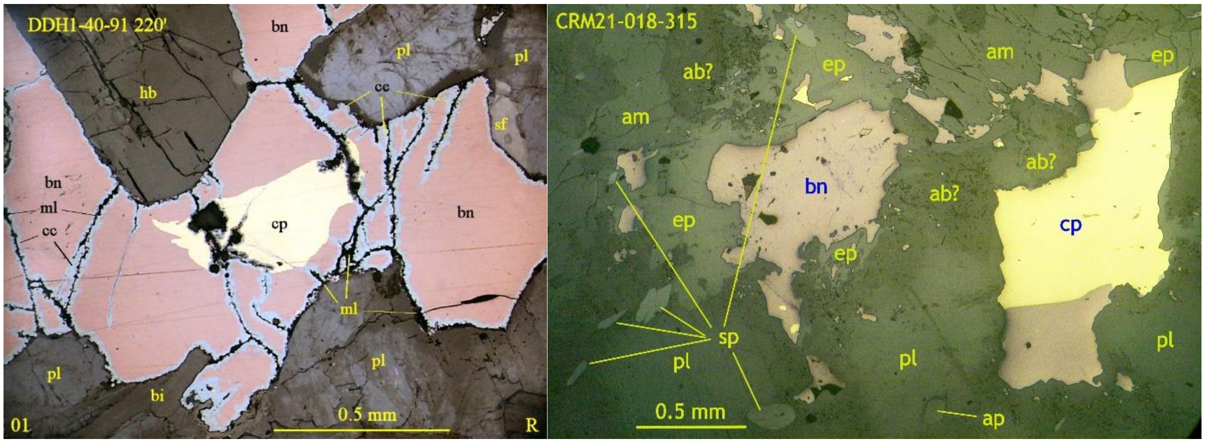


Figure 7-6 Gold Telluride Inclusion in Bornite with in Partial Melted Amphibolite (WC-002 194.25 m) and Gold-silver Telluride Inclusions in Net-textured Bornite within Diatextite Migmatite (WC-008 174m)

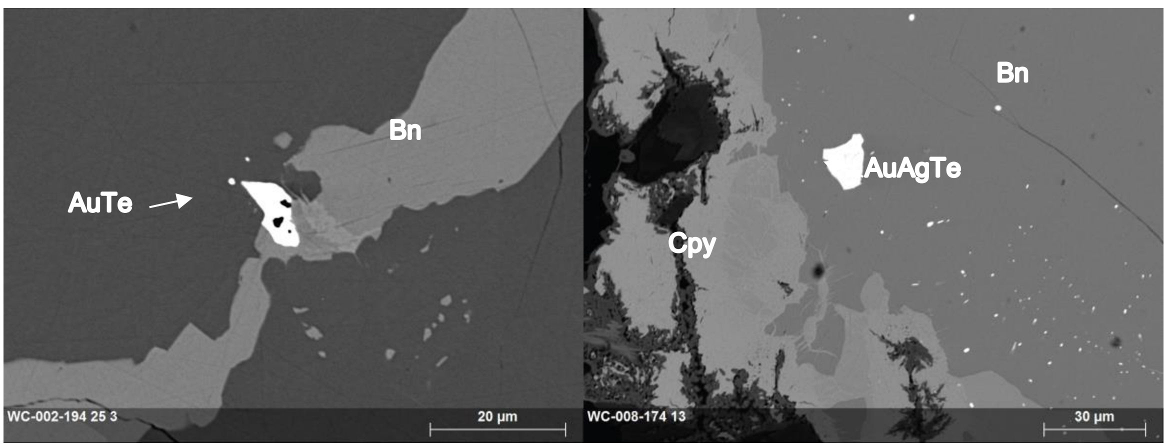
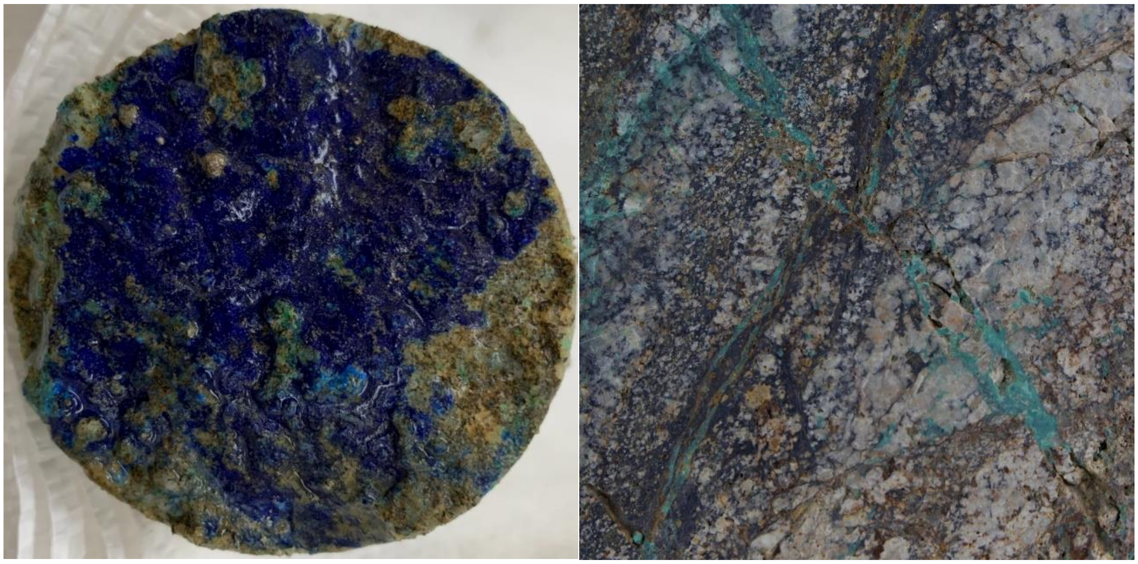


Figure 7-7 Copper Oxide Mineralization in the Form of Malachite and Azurite with Limonite Staining (Left) and As Fracture Network of Malachite and Chrysocolla (Right Image)



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

As discussed above, Granite Creek acquired the Carmacks North Property (formerly the Stu Copper-Gold 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.

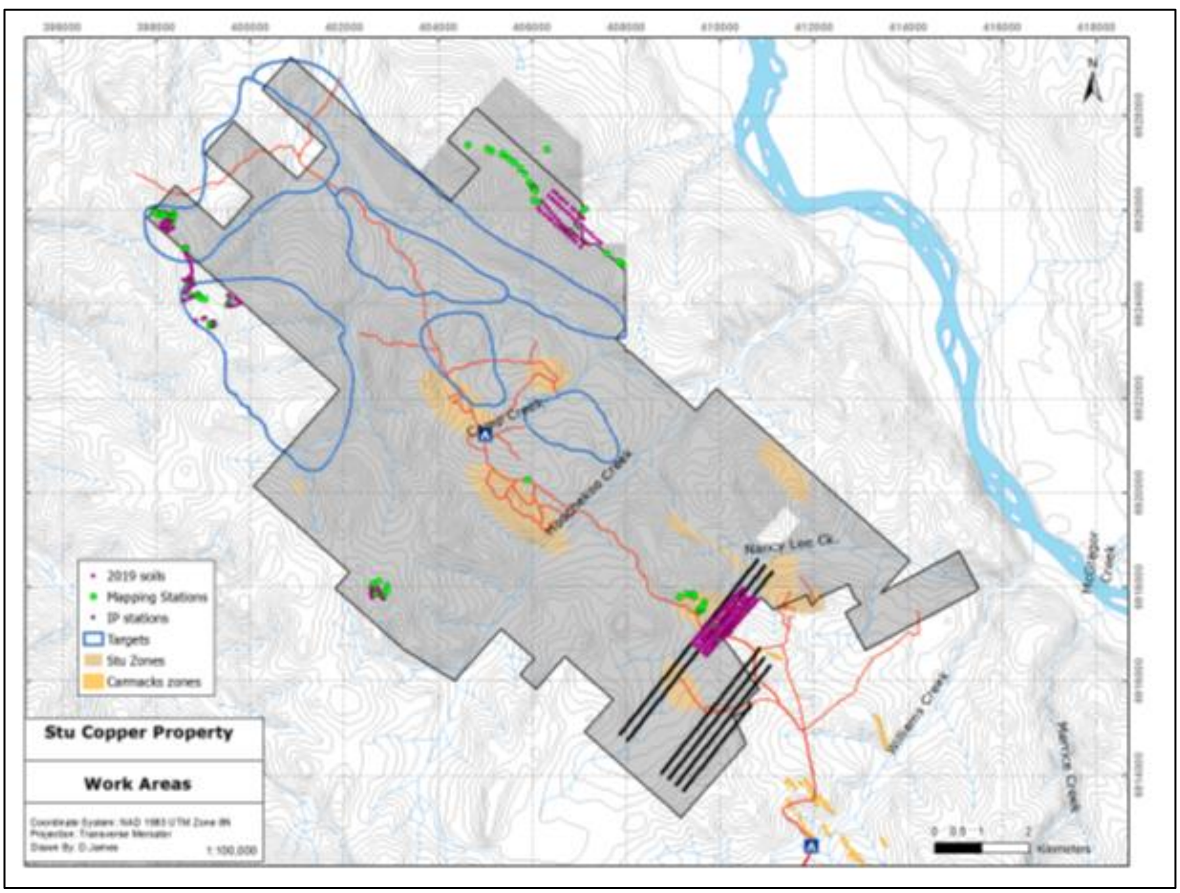
In August 2020 the Company acquired the Carmacks deposit through its acquisition of Copper North, which altered the exploration plans for the Company. The 2020 Exploration season comprised of re-logging and re-sampling historic core, drilling and soil sampling at Carmacks North's Zone A and a small diamond 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 (**Error! Reference source not found.**). 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 150m 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 (**Error! Reference source not found.**).

9.1.1 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 15th, 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

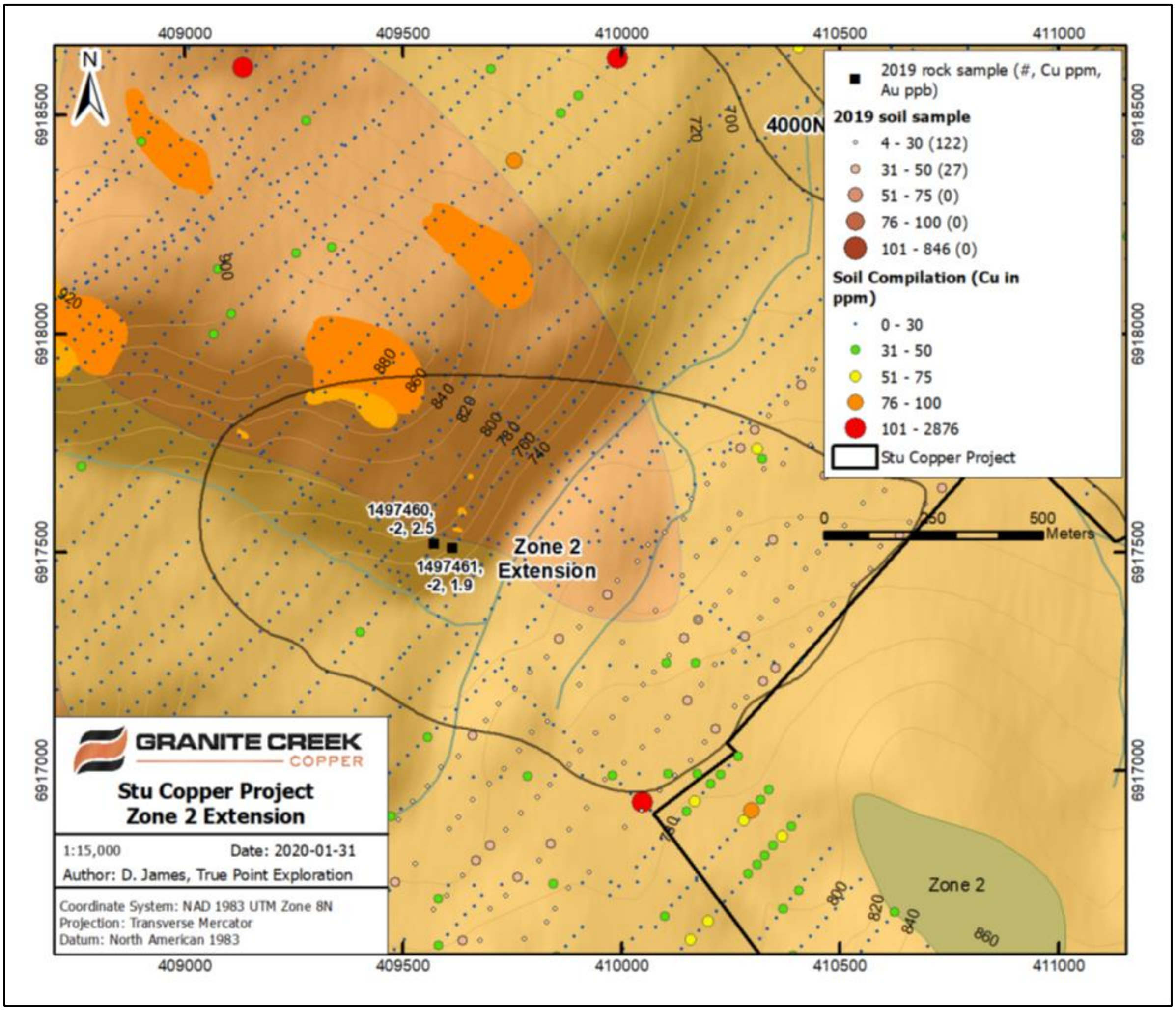
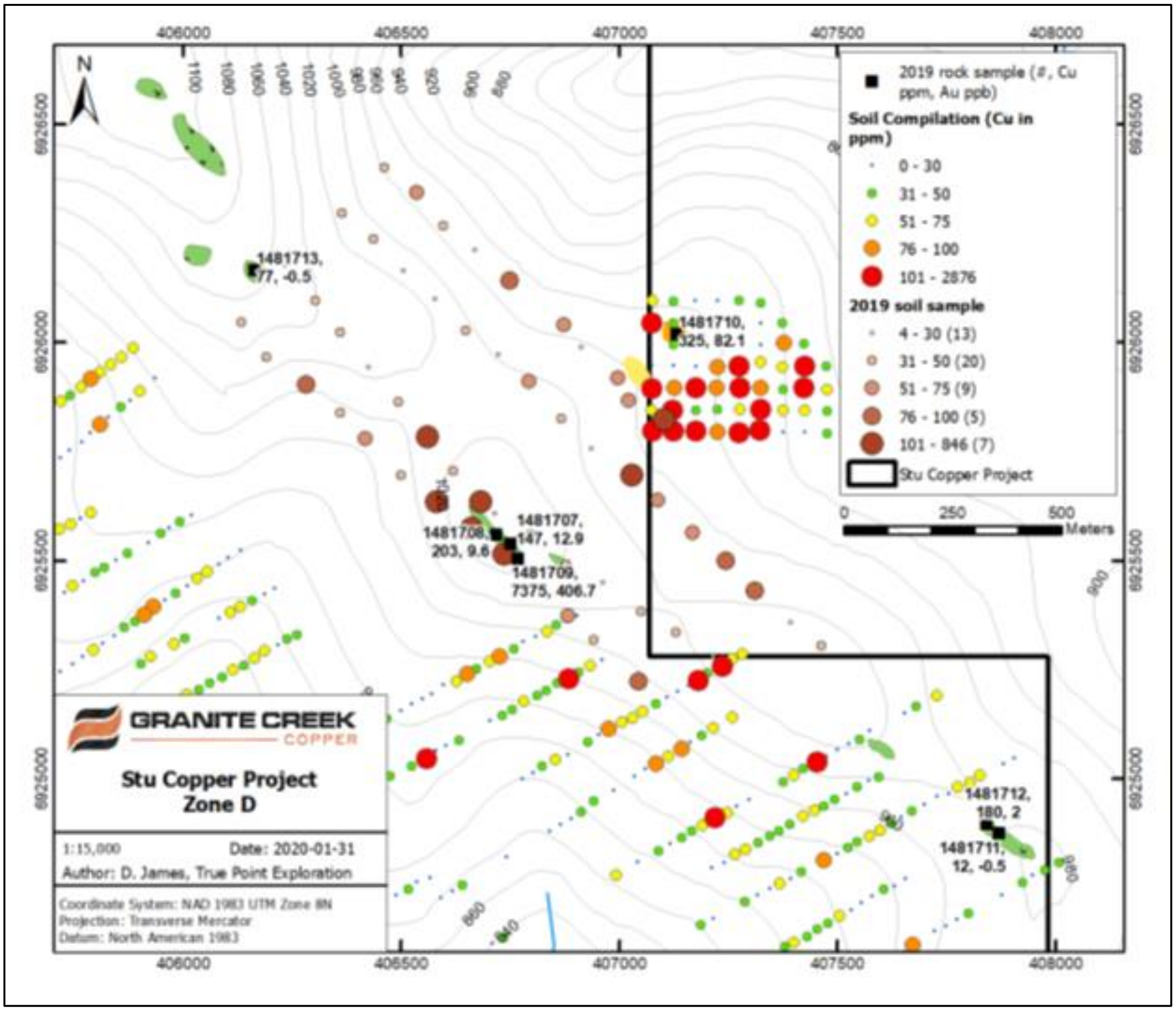


Figure 9-3 Zone D Soils and Rock Sample Locations

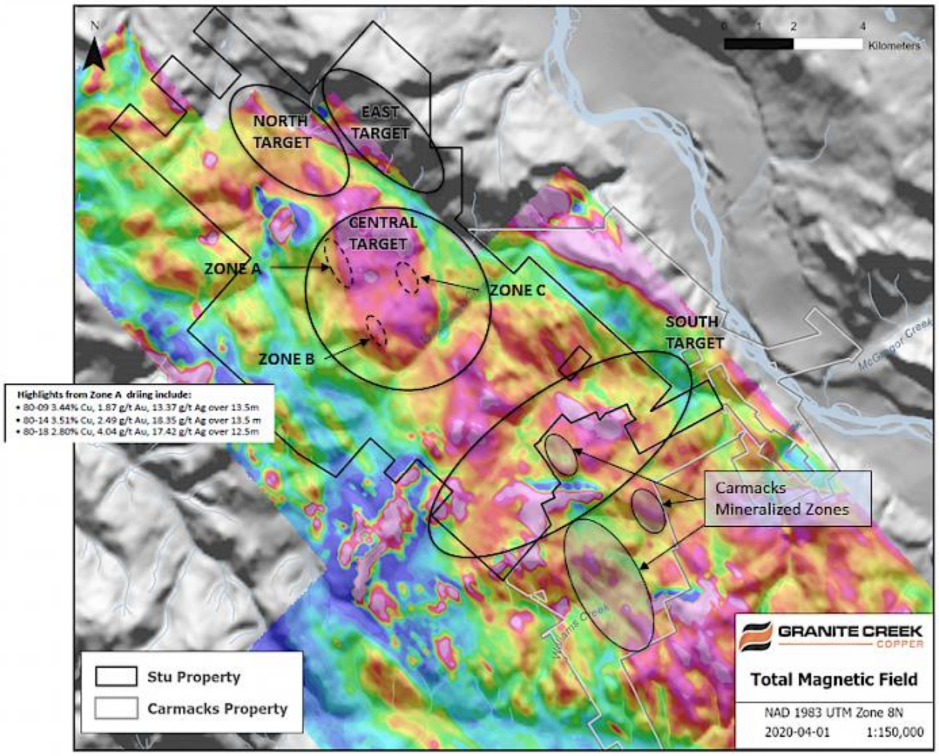


9.2 2020 Acquisition of Airborne Geophysical Data

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.

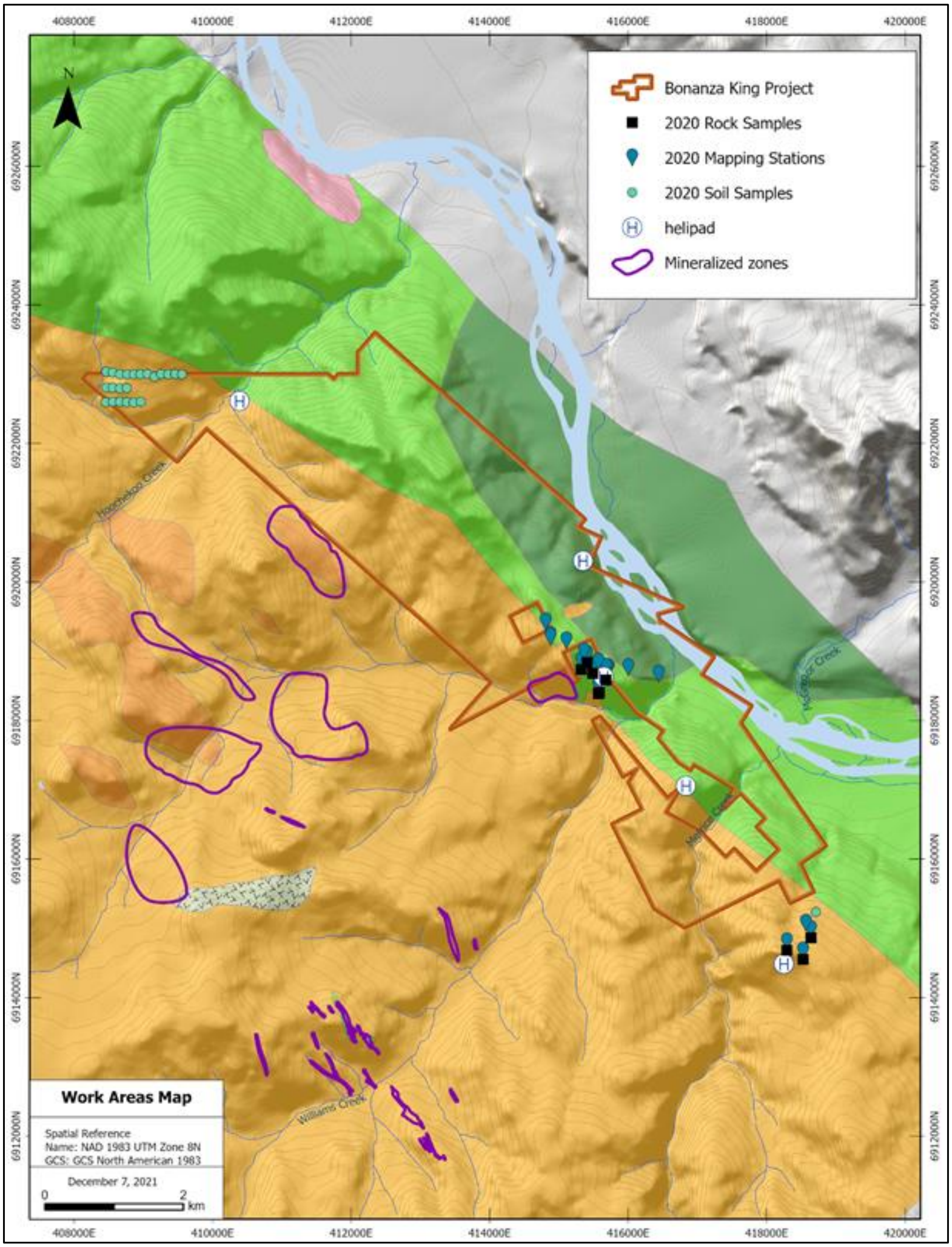
9.4 2020 Trenching Program

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 1.2km long northwest trending corridors, spaced approximately 900m 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)

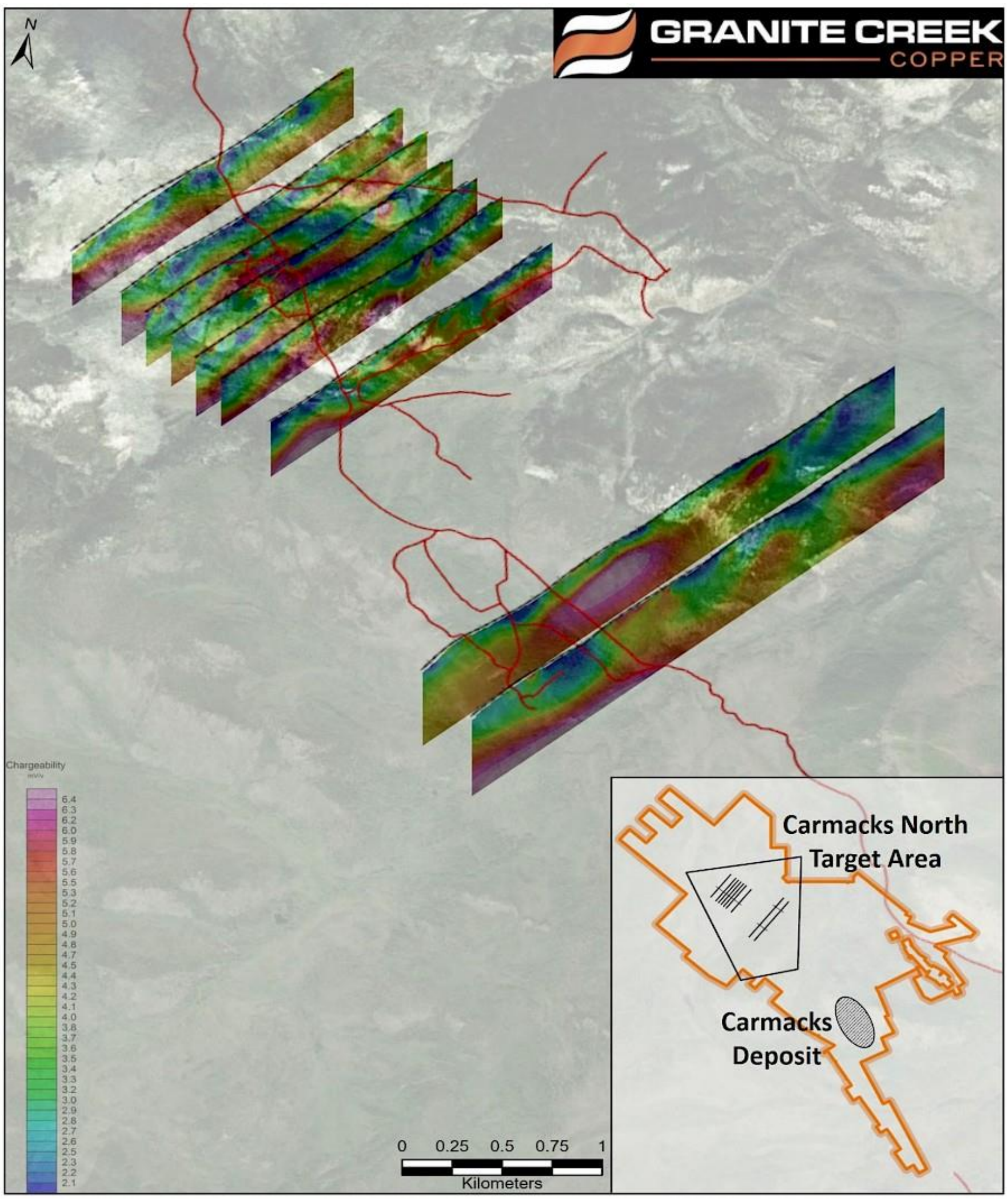
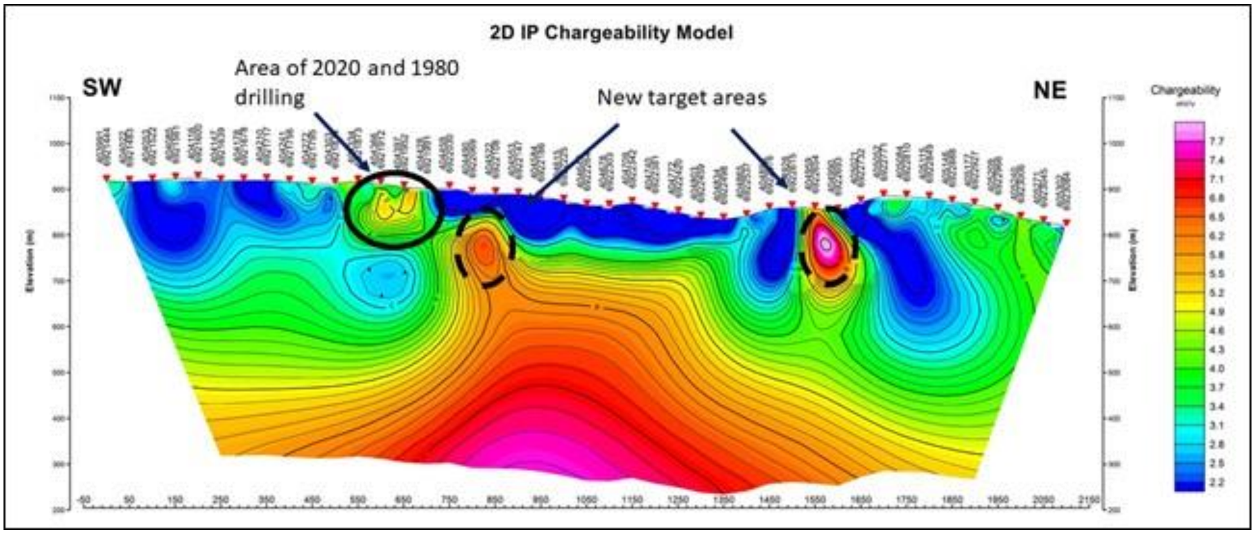


Figure 9-7 2D IP Chargeability Model of Line 4 from the Left (see Figure 9-6 above) (from news release dated July 14, 2021)



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.

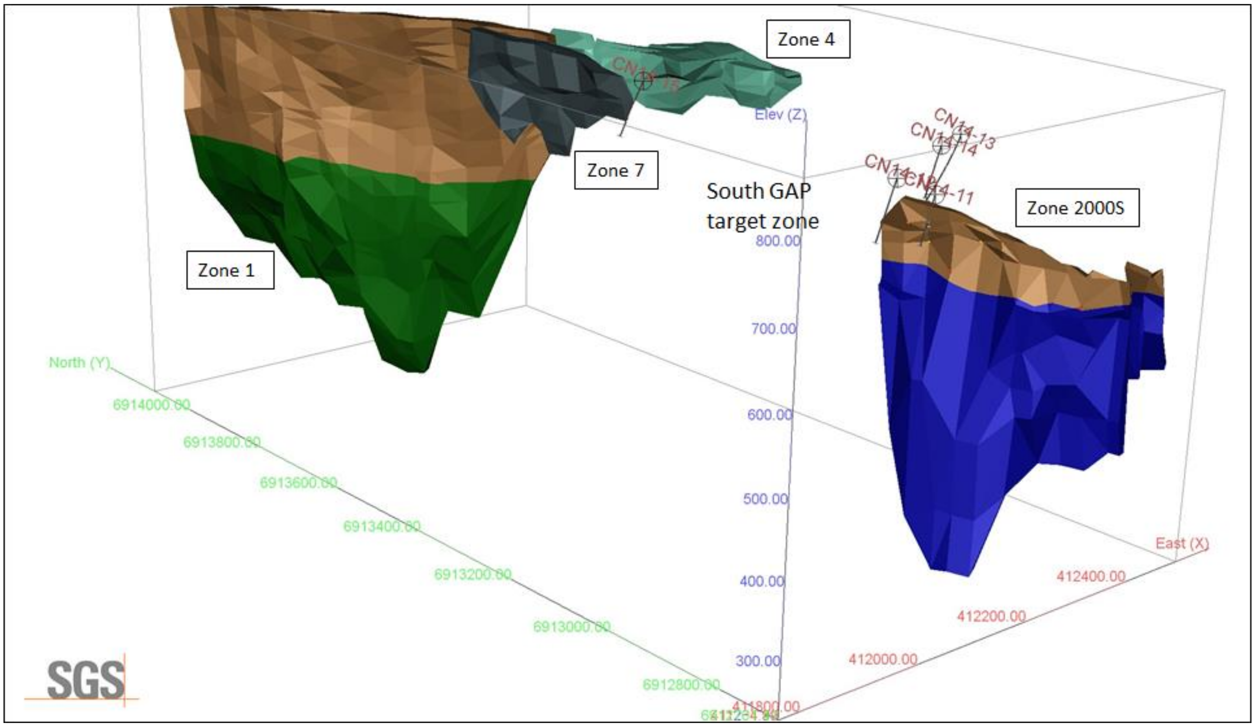
The South Gap target zone consists of an approximate 300 metre gap between the main proposed mining area of Zone 1 and the 2000S zone (Figure 10-1). The 2000S zone was intersected in 6 drill holes, with both oxide and sulphide mineralization. Drill hole WC92-01 intersected 10.67 m grading 0.67% copper 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:

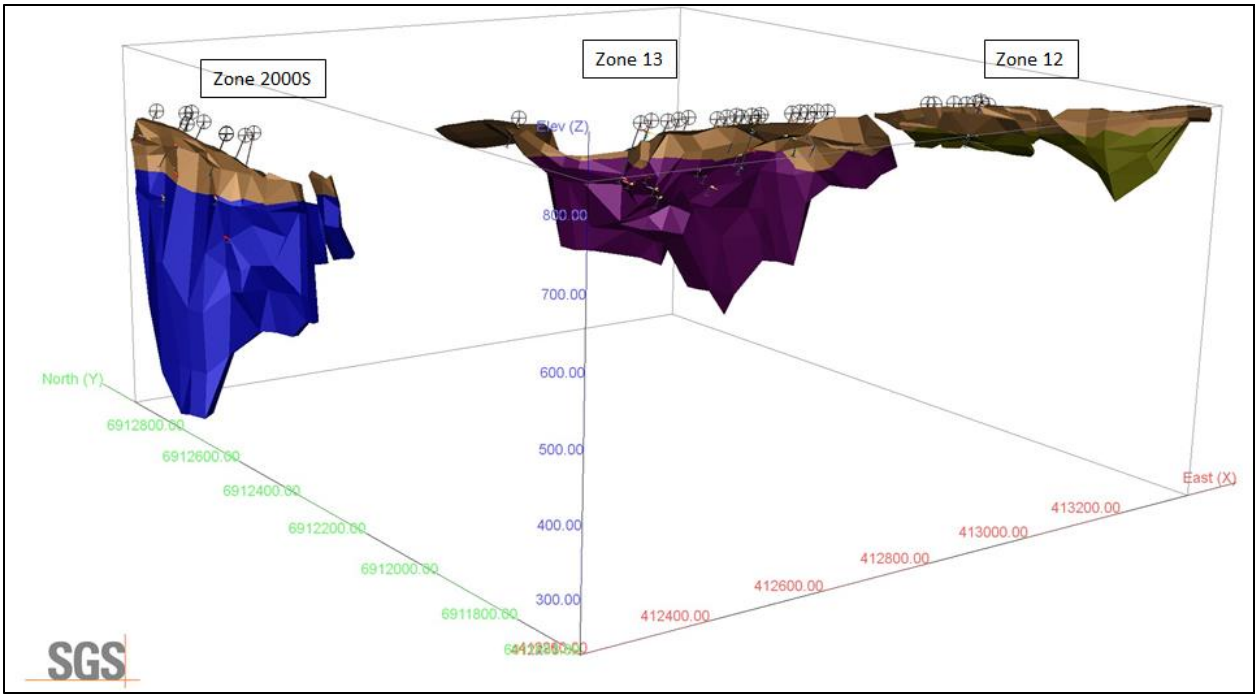
- CN15-13: 31.3 metres of 0.80% Total Cu, 0.18 g/t Au, 2.57 g/t Ag
- CN15-19: 102.1 metres of 0.37% Total Cu, 0.09 g/t Au, and 1.21 g/t Ag
- CN15-20: 82.6 metres of 0.52% Total Cu, 0.14 g/t Au, and 1.85 g/t Ag
- CN15 -21 119.6 metres of 0.35 % Total Cu, 0.08 g/t Au, and 0.98 g/t Ag

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:

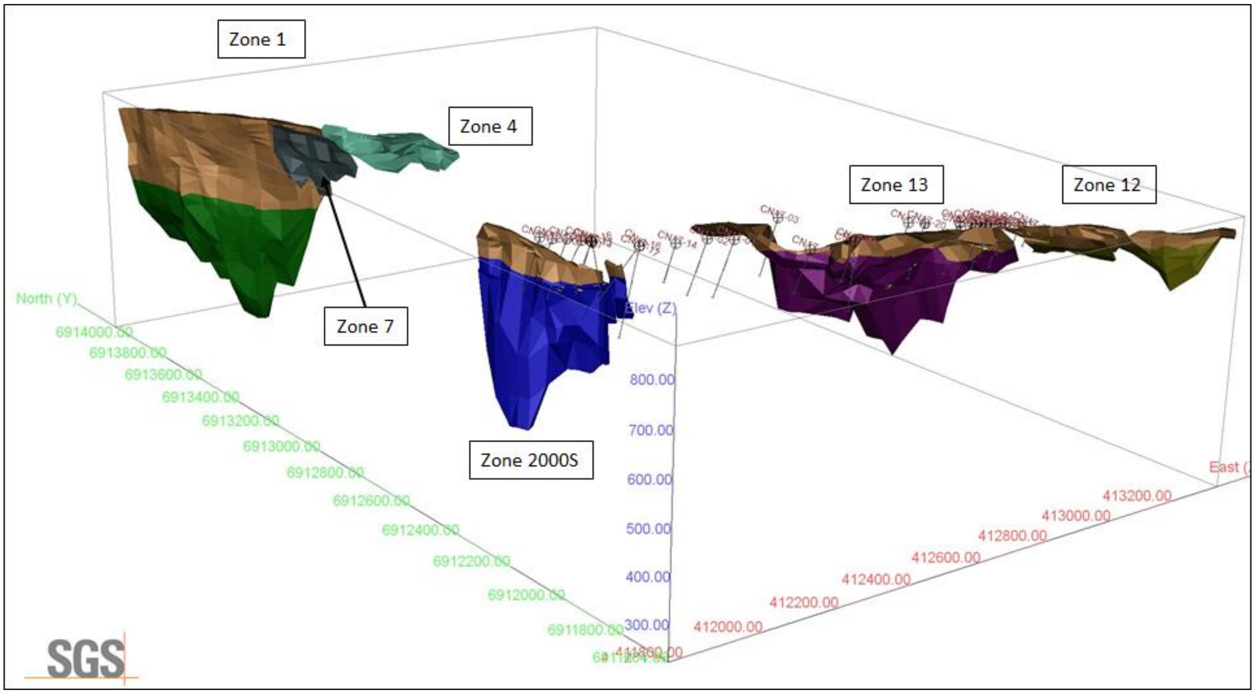
- CN17-19: 50.2 m true width of 0.68% Cu, 0.13 g/t Au and 1.95 g/t Ag (sulphide)
- CN17-20: 55.4 m true width of 0.61% Cu, 0.13 g/t Au and 1.84 g/t Ag (oxide transition to sulphide)
- CN17-32: 32.3 m true width of 0.68% Cu, 0.18 g/t Au and 2.46 g/t Ag (oxide)

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



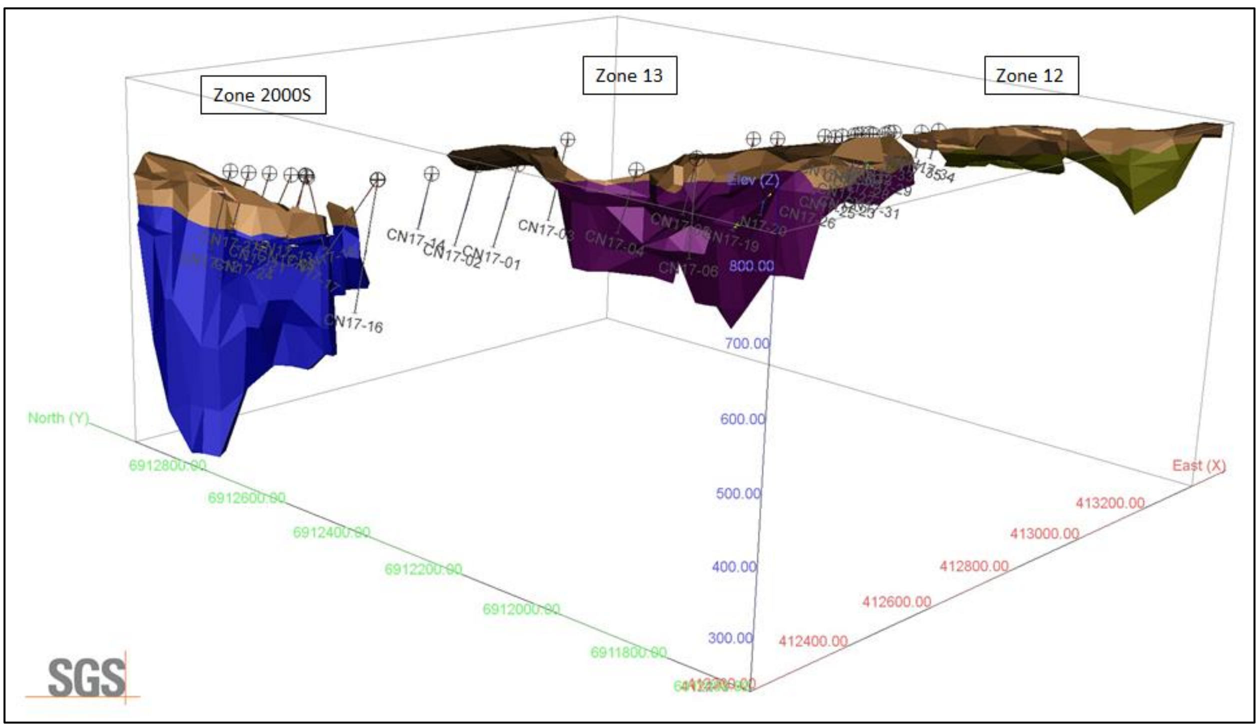


Table 10-3 Highlights from of 2017 Diamond Drill Program

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
CN17-32	10.67	46.18	35.51	32.33	0.18	2.46	0.61	0.68	Oxide

Hole	From (m)	To (m)	Intersected Width (m)	Estimated True Width (m)	Au (g/t)	Ag g/t	Acid-soluble Cu (%)	Total Cu (%)	Style
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

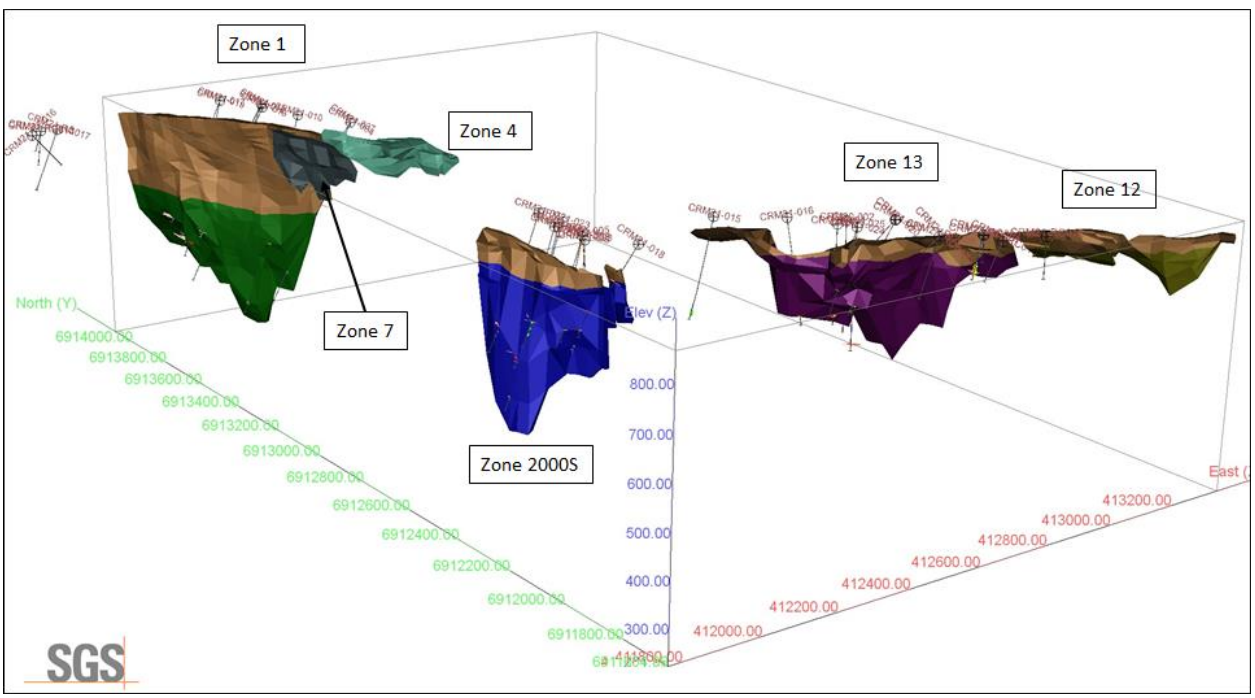


Figure 10-5 Location of 2021 Carmacks Drill Holes

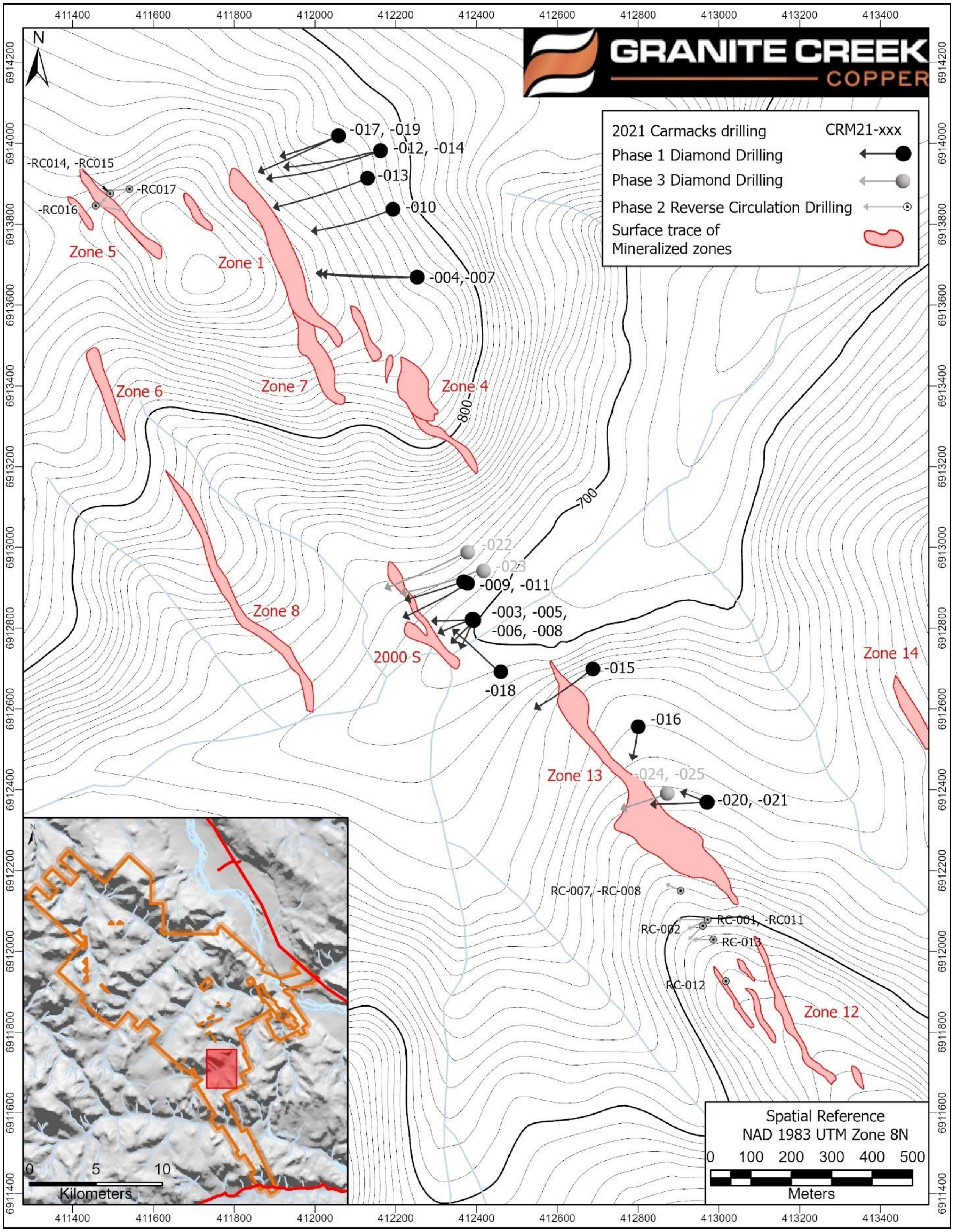


Table 10-5 Highlights from of 2021 Phase 1 Diamond Drill Program

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
CRM21-016	91.3	238.5	147.2	0.38	0.025	0.1	2.28	0.56	

Drillhole	From (m)	To (m)	Length* (m)	Cu (%)	Mo (%)	Au (g/t)	Ag (g/t)	CuEq** (%)	Target
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	
CRM21-023	324.23	446	121.77	0.39	0.007	0.13	1.76	0.52	2000S
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	
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	13
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.

11.1.2 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.

11.1.3 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.

11.1.4 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 HCIO4 – 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) excced 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

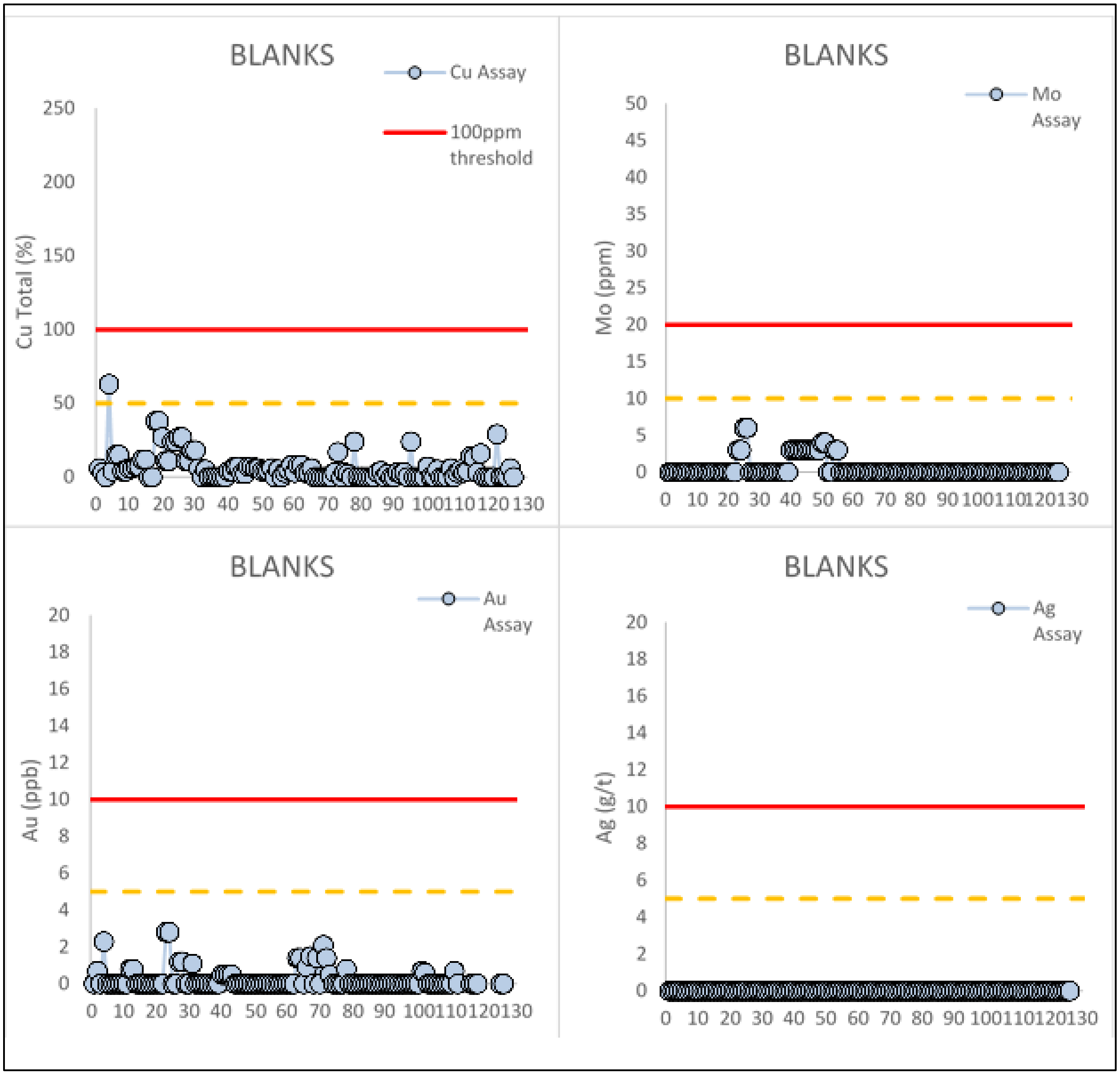
Limestone blanks were inserted approximately every 25th sample and after each mineralized interval. The threshold for these blanks was determined to be 100 ppm Cu, 10 ppb Au, 1.0 g/t Ag and 20 ppm Mo.

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

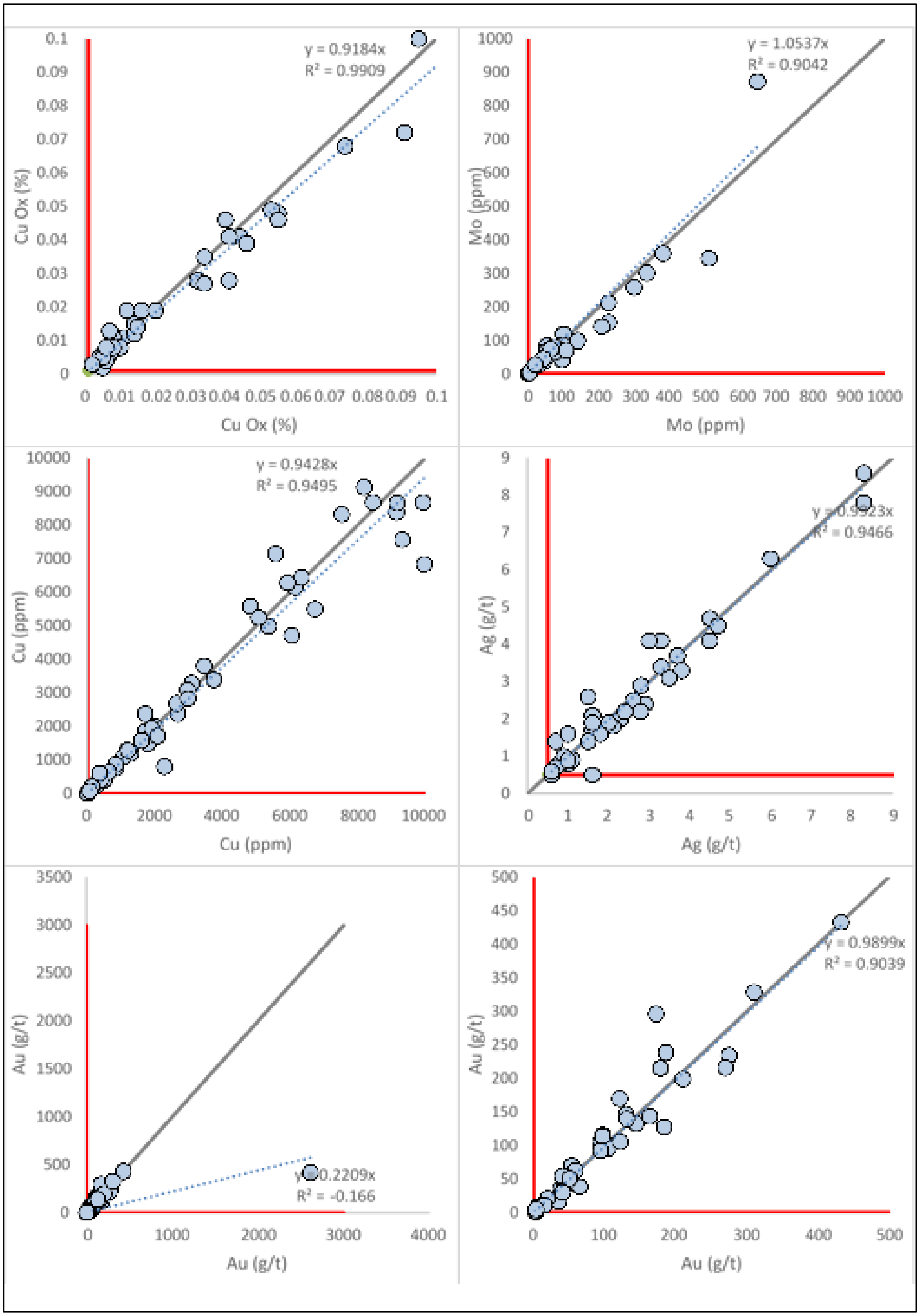


Duplicate Samples

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)

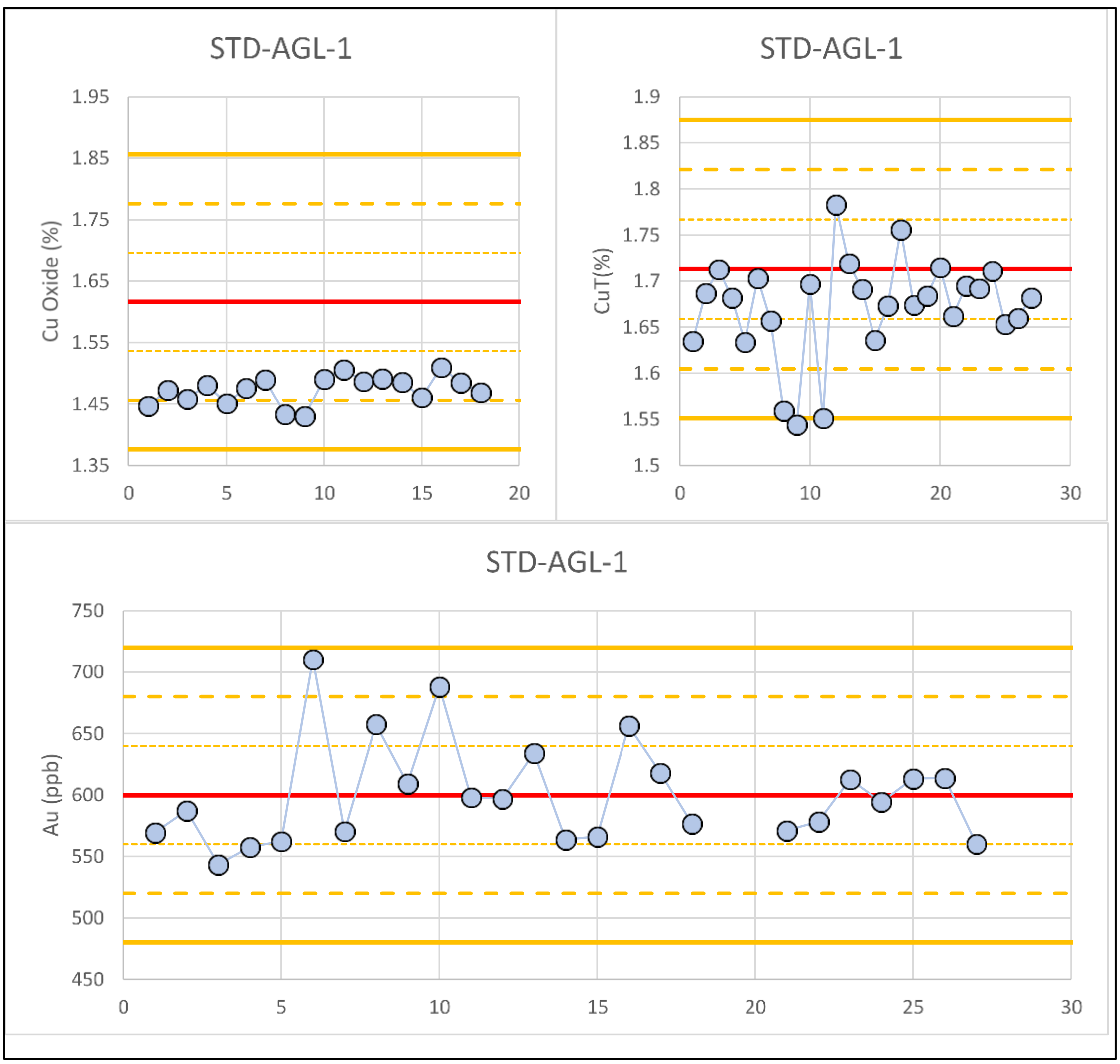


Figure 11-4 Standard AGL-2 Assay Results versus Certified Value with respect to the Mean (red line) and 2 Standard Deviations of the Mean (orange dashed line)

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)

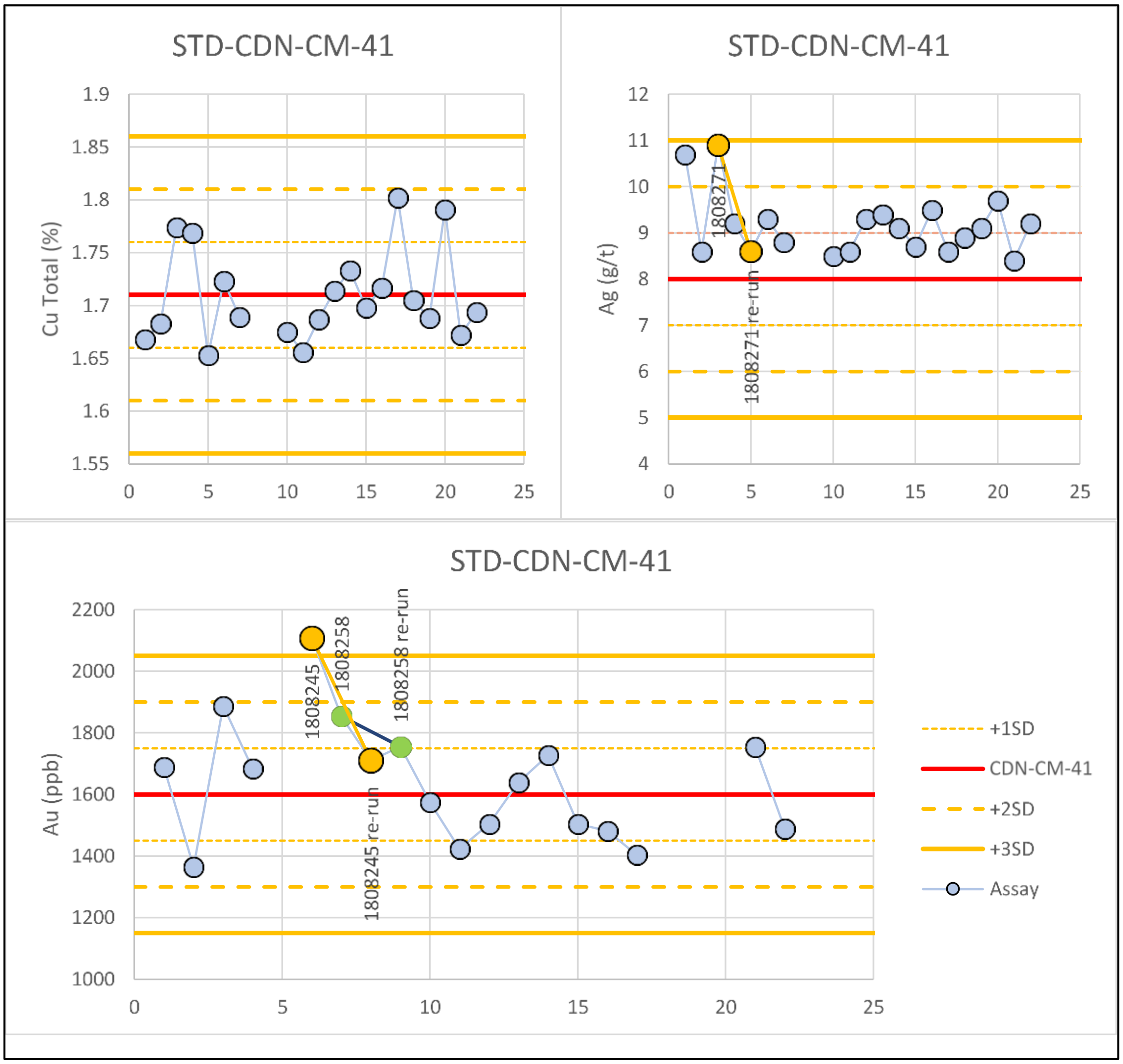
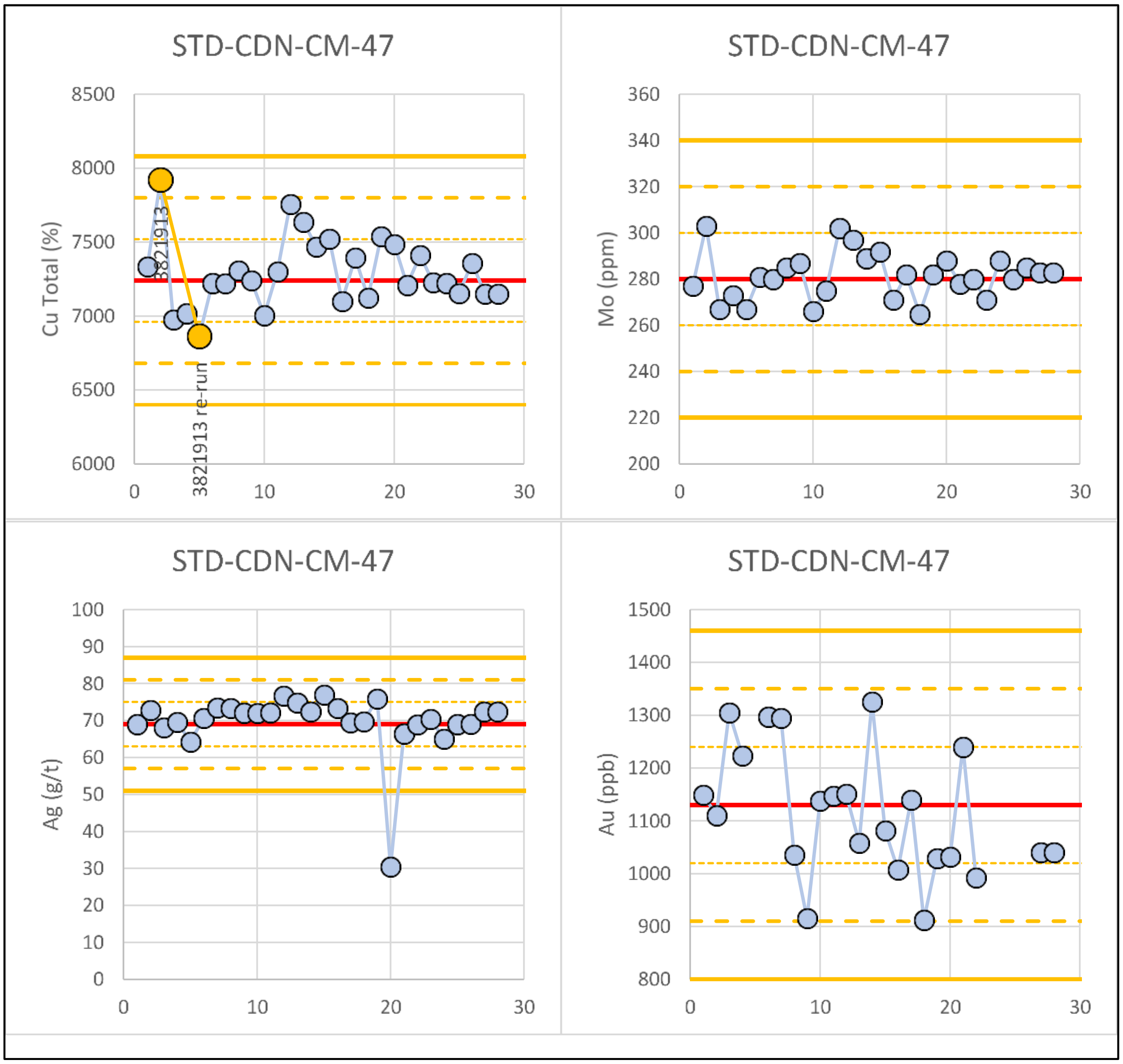


Figure 11-6 Standard CDN-CM-47 Assay Results versus Certified Value with respect to the Mean (red line) and 2 Standard Deviations of the Mean (orange dashed line)



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.

The Author 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

2022 Site Visit

The Author 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, the Author 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.

The Author 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.

12.2 Conclusion

All geological data has been reviewed and verified by Author 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 following description of Mineral Processing and Metallurgical Testing for the Carmacks Project has been extracted and summarised from a PEA for the Carmacks Project, completed by JDS Energy & Mining Ltd. (JDS, 2016) for Copper North. This work is supplemented by metallurgical test work completed by Granite Creek in 2021.

13.1 Introduction

The metallurgical testing program on the Carmacks Project focused on the recovery of acid soluble copper mineralization in the oxide cap of the Zone 1 deposit (JDS, 2016). The primary emphasis of the work conducted up to 2012, has been on development of design criteria and optimal operating parameters for heap leaching the crushed and agglomerated mineralized material, followed by solvent extraction for solution concentration and purification and electrowinning for recovery of cathode copper metal. Some limited testing has been performed on heap leaching using run of mine (ROM) mineralized material, examining leaching of the sulphide mineralization, and recovering gold following copper recovery.

Table 13-1 Historical Metallurgical Test Programs (from Arseneau, 2016)

Test Date	Company	Test by	Sample Classification	Test type
9/1971	Treadwell Corp.	Goodwin, J	Unknown	B. Roll
1989				Bulk samples from Z1 (2x1350 from trenches 0, 400N, 800N, 1200N and 1500N)
10/1989	Coastech Research	Lawrence, R	Unknown	Reactor and column
6/1990	Bacon, Donaldson & Associates (BD&A)	Unknown	Ore Composite	Bottle Roll and column leach
5/1992	BD&A	Beattie, M	Drill Core Composite	B. Roll
6/1992	Lakefield Research Association Ltd.	Webster, S.	Drill Core Composite	B. Roll
4/1994	Brown & Root, Inc.	Schlitt, W.J.	Ore Composite	Crib
5/1994	Beattie Consulting, PRA	Beattie, M	Bulk sample from trenches and composite from drill core	Column
1993-1994	Brown & Root, Inc.	Schlitt J.W., Henderson, W.	Bulk sample (250 tonne)	Pilot heap leach testing over winter
1995	Kilborn Engineering		Bulk sample (350 tonnes) Sample from 3 trenches (T91-20, T91-21, T91-02). Tests done at Carmacks.	Heap leach in arctic conditions, Leaching at large scale and public awareness. (From 1995 Kilborn FS)
2/1/1996	Beattie Consulting, PRA	Beattie, M	Drill Core Composite	Column
09/10/1997	Kilborn Engineering		Bulk Sample	Run-of-Mine Bulk Sample Collection
2/28/2001	Beattie Consulting, PRA	Beattie, M	Ore Composite	Column
4/20/2005	Westcoast Biotech	Bruynesteyn, A.	Ore Composite	Bottle Roll and Column test
3/1/2006	Westcoast Biotech	Bruynesteyn, A.	Ore Composite	Column
06/2006	Canadian Environmental & Metallurgical Inc		Ore composite	Neutralization Testwork
01/05/2007	G&T METALLURGICAL SERVICES LTD.	Shouldice, T.	Drill core composite	Mineralogical assessment- mostly cpy, transition zone 50% sulphide 50% non-sulphide. Oxide: 15% of Cu is cpy. Non-sulphide mostly malachite and unidentified minerals

20/06/2007	Process Research Associates Ltd.	Tan, G.	Drill samples	Sequential leaching and other characterization of drill ore samples
18/07/2007	Aurora Geosciences	Casselman, S.	Bulk Met sample	6 x 210l metal drums from six trenches in zone 1.
4/15/2009	PRA Metallurgical Division	Tan, G.	Ore Composite	Column
02/12/2009	PRA Metallurgical Division	Tan, G.	Ore composite	Column- cyanide leaching of gold in residues column 1&2
15/04/2009	PRA Metallurgical Division	Tan, G.	Ore composite	Column (one metre copper column Acid-leach test results)
2/28/2011	PRA Metallurgical Division (Inspectorate Exploration and Mining Services)	Tan, G.	Ore Composite	Column
29/08/2014	Casslelman Geological Services Ltd.	Casselman	Metallurgical bulk sample	4x400kg samples from 4 different trenches in Zone 1\7
09/2014	Halle Geological Services Ltd.	Halle, J.R.	Vat leach test samples	VAT leach test samples. 4 trench samples and 4 drill core composites (oxide-Zone 4 & 7and sulphide Zone 1&12&13)
10/02/2015	Bureau Veritas Commodities Canada Ltd.	Shi, A; Grcic, B. & Redfearn M.	2014 bulk sample	Vat leach (acid leach followed by cyanidation)
18/02/2015	Bureau Veritas Commodities Canada Ltd.	Shi, A; Grcic, B. & Redfearn M.	2014 bulk sample	As above
28/05/2015	Beattie Consulting Ltd.	Beattie, M		Vat leach
28/04/2016	JDS Energy & Mining Inc.	JDS		Simplified flow sheet for acid leach followed by cyanidation
01/10/2021	Sedgeman	Way, D., Pope, J., Cho, S., Wilkin, M., Caldbick, J.	Drill core composite (51kg) -WC-021A	Carmacks Oxide Metallurgical Testwork
01/10/2021	Sedgeman	Way, D., Pope, J., Cho, S., Wilkin, M., Caldbick, J.	2 composites from two intervals in drill core Zone 2000S (CRM21-003)	Carmacks Sulphide Metallurgical Testwork

In 2014, Copper North examined the value of adding precious metals recovery to the project plan using a two-stage heap leaching approach. Results were reported in the PEA prepared by Merit International Consultants (Merit 2014). This study indicated the value of recovering gold and silver, leading to further metallurgical test work and the present study.

In 2016, the test work program was revised from a heap leach to vat or tank leach testing for copper, gold and silver recovery. This current PEA is based on crushing and grinding the mineralized material to a P_{80} of 664 µm and leaching the copper into solution with sulphuric acid. The copper is then recovered using solvent extraction and electrowinning. Tailings from the copper leach circuit, after neutralization with lime, will be processed through a carbon-in-leach (CIL) circuit, where gold and silver are leached into solution with sodium cyanide and adsorbed onto activated carbon. The precious metals are then recovered through elution, electrowinning and refining.

The 2021 metallurgical testwork evaluated the extraction of copper sulphide component through conventional floatation circuit. Both oxide and sulphide domain samples were tested.

The oxide-domain material had a maximum copper rougher recovery of approximately 41%, at 2.8% Cu concentrate grade. Partical Mineral Analysis (PMA) indicated that the majority, approximately 61%, of the copper content is within credhneite, iron oxide and chlorite minerals and is not floatable due to the minerals being hydrophilic. The remaining 39%, where majority are copper oxide minerals (malachite/azurite/cuprite) can be recovered via flotation with sulphurdisation and/or special oxide collectors, which presents a very low cap on achievable metallurgical performance of ores represented by this sample. An acid leach test of the oxide sample were found to be comparable to the 2016 metallurgical studies, despite the primary grind sizes being significantly different. This suggests that the oxide domain tested here likely contain a similar composition of minerals to the 2016 oxide sample and there is no separate sub-domain of Carmacks oxide.

Sulphide process-floatation. Liberation in grinding is excellent, copper sulphide floatation and recoveries. The Carmacks primary sulphide samples selected for flotation test work provided high flotation copper recoveries at P_{80} 150 µm primary grind and P_{80} 25 µm regrind sizes. Preliminary copper flotation recovery model was generated based on two open circuit cleaner flotation test results at fixed 25% copper concentrate grade. This preliminary recovery estimation should only apply to copper feed grade of 0.46% to 1.0%, which is the typical range for Carmacks Deposit sulphide domain. PMA indicated that most of the sulphide mineral consists of chalcopyrite and minor covellite/chalcocite, with minor pyrite. Copper sulphide minerals are well liberated for rougher recovery via flotation at P_{80} of 150 µm. Gold is associated with mainly copper sulphide minerals and some pyrite. The majority of gold particles are <10 µm and are categorized as refractory and flotation of gold with copper concentrate would be the most economical way to recover this gold.

13.2 **Historical Heap Leach Test Work**

Previous metallurgical test programs for the Carmacks Project, including process development and studies have focused on heap leach technology for copper and gold/silver extraction. The most recent heap leach study was completed by Merit Consultants International Inc. in 2014 (Merit Consultants International Inc., 2014).

13.2.1 Copper Extraction and Recoveries

The copper extraction from previous column tests, operated with the optimal crush size, acid addition, and leach time, was remarkably similar. The column tests that were operated under conditions that most closely mimic those being considered commercially were those done by Beattie Consulting and Process Research Associates Ltd. (PRA) between 1996 and 2001. These were tests where ore was crushed to -20 mm and agglomerated, columns were greater than 5 m in height, and where the columns were irrigated with solution at a pH of 1 – 1.5.

Copper extraction for all of these tests exceeded 80%, and columns that were leached for longer periods of time reached 85% or greater. 80% recovery with 85% recovery after an extended leach time was observed in several other tests. The 1990 composite columns both achieved greater than 85% copper extraction. Bottle rolls on assay rejects performed by Beattie in 1992 all achieved greater than 85% extraction except for the lower grade (<0.5% copper).

The best indication of copper recovery for the resource comes from sequential leaching tests run by PRA Labs in 2007. The sequential leaching results were reduced to the following equations:

- If Cu (oxide)/Cu (total) > 0.79, Leachable Copper = 85%
- If Cu (oxide)/Cu (total) < 0.79, Leachable Copper = 95% x Cu(oxide)/Cu(total) + 10%

13.2.2 Sulphuric Acid Consumption

The acid consumption rates calculated during the sequential leaching tests are a good indication of the acid consumption over the range of material expected from the mine. These tests, correlated with column recoveries on the same material, had an average acid consumption of 20kg/t.

The test results indicate that acid consumption during leaching of the Carmacks project ore increases with the level of acid addition and with a decrease in particle size for the various ore types. Excess acid provided is readily consumed by the constituents of the rock. The testwork indicates that a favourable operating strategy is to agglomerate the ore with at least 5 kg/t H_2SO_4 and to apply leach solution at approximately pH of 1.5. Addition of high concentrations of acid should be limited to overcoming the initial neutralization potential of the ore. Under these conditions, it is evident that a total acid consumption of no greater than 20 kg/t H_2SO_4 can be achieved.

13.2.3 Other Reagent Requirements

Organic Reagents

The organic phase of the SX process will be composed of 16% Cytec Industry's Acorga M5774 and 84% diluent (kerosene). Consumption of the organic reagent is mostly due to entrainment in the raffinate with subsequent loss on the heap. Consumption rates are expected to be 30,039 kg/yr for the reagent and 155,496 kg/yr for the diluent.

Other

Other reagents include:

- Guartec will be used as a plating aid in the electrowinning process at a rate of 3,534 kg/yr.
- Cobalt sulphate also assists the plating process. Consumption is estimated at 10,602 kg/yr.

13.2.4 Ore Hydrodynamic Characterization

Ore hydrodynamic characterization was conducted by HydroGeoSense, Inc. of Tucson, AZ in 2012 and the results are summarized here. Overall, the copper oxide sample received from the Carmacks project ore body is quite competent and permeable. The hydrodynamic characterizations of these samples indicate that optimal agglomeration (a Level 4 out of 5) can be attained with an average moisture addition of 6.8% (with respect to the dry ore mass) and 80% of the Net Acid Consumption. Although the head sample, even under partly-agglomerated conditions, was sufficiently permeable under a lift height of 10 m, the hydrodynamic tests show that full agglomeration will significantly improve the physical behavior of the ore and hence would effectively reduce the potential for high liquid saturation on a heap leach operation at an industrial scale. Notwithstanding this good hydraulic performance, the results from the stacking tests on the "leached" ore indicate some decrepitation which could reduce the maximum height of a multi-lift heap design.

The hydrodynamic evaluation summarized in the following paragraphs indicates that the sample of the Carmacks ore tested under this program would perform well under percolation leaching at a lift height of 12 m and a heap height of up to 32 m. The results from the ongoing Hydrodynamic Column Tests on the "leached" sample would be used to further verify this conclusion.

Stacking Test Results

Agglomeration trials indicated that the optimal moisture addition to obtain full agglomeration of this Carmacks sample is about 6.5% for an acid addition equal to 80% the Net Acid Consumption. Six stacking tests (ST's) were conducted on Fresh (head) samples while a single sample from the "leached" residue from one of the Hydrodynamic Column Tests (discussed below) was undertaken.

The tests on the head samples include conditions to represent partly-agglomerated (level 1), level 4 (at the optimal moisture content ranging from 6.4% to 7.3% depending on the amount of acid addition), and various levels of acid during agglomeration. The residue ore from one of the Hydrodynamic Column Tests (HCTs) was used to obtain a preliminary assessment of the potential impact of a leached cycle on the physical and hydraulic properties of the Carmacks ore. The acid addition was determined as a percent of the Net Acid Consumption (NAC estimated at 22.5 kg/ton) as indicated by Copper North personnel.

Bulk density of the ore has a significant effect on the metallurgical performance of a sample and should be carefully considered during the planning and interpretation of metallurgical column tests. Ample empirical evidence shows the performance of a metallurgical test is strongly correlated to the ore density and thus metallurgical columns should be built to represent a realistic bulk density value. Without this information, an accurate scale-up of the results from the metallurgical column tests to the industrial scale is not possible.

From the practical point of view the results indicate the following:

- The shape of the density profiles for the fresh ore indicates a relative competent porous structure and rock fragments which, as discussed below, lead to a good percolation capacity. This percolation capacity likely arises from the minimal content of fines (1.5% minus 105 µm and 0.5% minus 74 µm).
- Although the shape of the density profiles for all the fresh-ore samples agglomerated at a Level 4 is similar, agglomeration with an optimal level of moisture and 65% and 50% of the NAC produces a slightly higher as-placed density (1.48 ton/m^3) and a slightly steeper density profile than observed for the sample prepared with 80% if the NAC.
- A comparison of the density profiles indicates optimal agglomeration results from an acid addition of 80% of the NAC and a moisture content of 6.4%. These conditions achieve the maximum level of agglomeration possible for this sample (L4) and a reduced bulk density throughout the range of heap heights investigated by the STs (0.3 m to 30 m).
- The less-resilient nature of the "leached" sample results in density values which are larger than those of the fresh ore agglomerated at Level 4 once the heap height exceeds 3 m. For a heap height of 10 m, the density of the "leached" sample is 1.63 ton/m^3 and reaches 1.84 ton/m^3 once the heap height is 28 m which correspond to about 5% and 7% increase with respect to the density values obtained for the fresh ore.
- Industrial experience shows that as long as the total porosity is larger than or equal to 30% an ore sample can still support heap leaching. Porosity values below this threshold result on too low saturated hydraulic conductivity and too high moisture retention capacity. Based on the specific gravity (SG) for the ore at 2.7 it is estimated that the maximum bulk density for this fresh Carmacks ore is 1.89 ton/m^3. Extrapolation from the density profile from the 80%NACL4mH sample suggest that this threshold density value will be reached for a heap height of about 70 m. For the "leached" sample, the heap height corresponding to this density threshold is about 35 m.
- The density profiles of the fresh ore indicate that this sample from the Carmacks ore would be a good candidate for a multi-lift heap design even after one leach cycle as long as the ore properly agglomerated.

Another key parameter for the design of a heap obtained from the Stacking Test is the reduction of the ore percolation capacity as the density (heap height) increases.

Inspection of the conductivity profiles indicates that:

- All the samples subjected to the Stacking Tests procedure show reasonable as-placed saturated hydraulic conductivity (Ks > 1 x 10^{-2} cm/s). More importantly, the conductivity profiles show only a slight reduction in the conductivity of these samples as the heap height increases. The maximum reduction in conductivity seems to occur as the heap height increases beyond 8 m.
- Experimental evidence from a large number samples from a variety of ore types indicates that as long as the saturated hydraulic conductivity is larger than or equal to 10^{-2} cm/s; the ore should be a good candidate for heap leaching. All these samples (even the partly- agglomerated 80%NAC-L1) clearly satisfy this minimal requirement.
- The reduction in conductivity observed on these samples is relatively minor over a heap profile of 10 m. In fact, for the optimally agglomerated sample loaded to represent a 30-m heap the conductivity decreases only by a factor of 5.6. The minimum conductivity value (1.4 x 10-1 cm/s) obtained for a heap height of 30 m is still adequate to keep this sample as a candidate for heap leaching.

- Overall, the conductivity curves resulting from the Stacking Tests on this Carmacks sample is relatively flat suggesting that the fresh ore is relatively resilient. Typically, an increase in bulk density over the range of heap heights tested during this study (up to 30 m) results in a reduction of conductivity of one order of magnitude or larger.
- Comparison among the conductivity profiles for the various samples indicates that agglomeration to a minimum level of 4 has a positive impact on the percolation capacity of the ore. For instance, agglomeration at a level 4 with moisture content of agglomeration of 6.5% results in an increase of the saturated hydraulic conductivity between a factor of 5 (for either the 50% or 65% NAC) and a factor of 12 (for the 80% NAC) with respect to a level 1 agglomerated under a 10-m lift.
- Optimal agglomeration will result in minimal bulk density, maximum total porosity and maximum saturated hydraulic conductivity. All these characteristics will provide an opportunity to reduce the risk of high liquid saturation along the heap profile and overall improvement in the metallurgical performance of the process.
- Significant reduction in hydraulic conductivity (two orders of magnitude) was observed on the re-agglomerated “leached” sample over a heap height of 28 m. Similar to the change observed on the density profile, the higher level of agglomeration produces an initially large hydraulic conductivity but the effect of decrepitation (likely due to increase level of fines) produces a sharper reduction in conductivity than that observed for the fresh ore.
- The slope of the hydraulic conductivity profile of the re-agglomerated “leached” ore is steeper than that of any of the fresh samples such that over the first 10 m the conductivity decreases from 3.5 x 100 cm/s to 1.8 x 10^{-1} cm/s (a reduction of over an order of magnitude). As the heap height increases to 28 m, the ore conductivity further decreases to 2.0 x 10^{-2} cm/s (one more order of magnitude reduction).
- It is noted that at a heap height of about 30 m, the percolation capacity of the “leached” sample (2.0 x 10^{-2} cm/s) is about one order of magnitude (a factor of 10) smaller than that of the fresh ore (1.4 x 10^{-1} cm/s).
- Another important observation from the comparison between the conductivity profiles of the fresh ore with that of the “leached” sample is that the slope of the former is relatively flat while that for “leached” sample is steeper as the heap height increases beyond 8 m.
- From an operational point of view, the moisture content of the leached ore after drainage will be higher than that used for re-agglomerating the “leached” sample so over stacking of the first lift during the construction of the second lift may produce additional compaction and loss of conductivity than inferred from the results of the STs.
- Comparison among the conductivity profiles for the various samples indicates that agglomeration to a minimum Level of 4 will have a positive impact on the percolation capacity of the ore. Note that minimal agglomeration of the fresh ore at a Level 1 results in reduction of the saturated hydraulic conductivity of at least a factor of 10 with respect to the optimally agglomerated fresh ore (80%NAC-L1 versus 80%NAC- L4).
- Although testing of the “leached” residue indicates an increase in bulk density and reduction of saturated hydraulic conductivity (likely related to an increase in the content of fines), the samples tested during this program retain a good level of porosity and percolation capacity so they can be considered good candidates for percolation leaching.
- The best performing ore from the point of view of the physical and hydraulic behavior is the fresh ore agglomerated at a Level 4.
- Clearly, the minimally-agglomerated ore is the worst performing sample (maximum bulk density and minimum saturated hydraulic conductivity).

The Stacking Test results presented in this section show that optimally agglomerated Carmacks fresh ore and a residue sample simulating a single leach cycle are good candidates for a percolation leaching process.

Hydrodynamic Column Test

Two tests were completed on the fresh ore to represent lift-heights of 8 m and 32 m. A Hydrodynamic Column Test (HCT) is performed by placing the ore sample into six-inch diameter columns. The diameter of the columns is selected based on the top size of the ore (~19 mm) to minimize wall effects on the hydrodynamic parameters of the ore.

As indicated on the Test Matrix, the tests on the fresh ore were conducted on optimally agglomerated samples (Level 4) to simulate the response of 8m lift and 32m heap.

One of the key pieces of information derived from a HCT is the hydraulic conductivity curve; the relationship between solution application and degree of saturation. In general, the shape of the hydraulic conductivity curve is influenced by the particle size distribution, level of ore conditioning and moisture content, the blending ratio of the ore type, the type of solution used during the agglomeration process, and the bulk density.

From the operational point of view, the results from the HCTs on these Carmacks samples indicate that:

- For a typical range of solution application rates (5 L/h/m^2 to 15 L/h/m^2), the ore near the bottom of a 12-m lift would operate at a degree of saturation below 38%. As the heap height increases to 32 m, it is anticipated that the degree of saturation would increase to about 45%;
- Additional loading of the leached ore, as that resulting from a multi-lift heap, will result in higher liquid saturation near the bottom of the heap.
- Given that by design, the samples tested on the HCT represent the bottom of the lift/heap, the material higher on the profile of the lift/heap will in theory operate at a lower degree of saturation.
- Therefore, the results from the HCTs on the fresh sample confirm the preliminary determination obtained from the STs; the particular Carmacks samples tested during this characterization effort are sufficiently competent to support percolation leaching on either a dynamic heap with a minimum lift height of 12 m or a multi-lift heap with a total height of up to 32 m.

13.2.5 Gold and Silver Extraction and Recoveries

Initial cyanidation bottle roll tests on residues for copper leaching were carried out at the University of British Columbia in 2008. Copper extraction from the feed material to this program was incomplete at 63% but cyanidation successfully extracted 76% and 81.7% of the gold in two tests. Silver extraction in the tests was 55% and 100%. Cyanide consumption in these two tests was high at 1.51 and 2.43 kg NaCN/t due to the dissolution of copper remaining in the cyanidation feed. Additional testwork was carried out to evaluate the use the Sulfidization, Acidification, Recycling, Thickening (SART) process for the precipitation of copper from cyanide solution with concurrent regeneration of cyanide. This testwork demonstrated that the SART process was applicable to Carmacks process solutions for the removal of dissolved copper and the regeneration of cyanide for recycle within the overall process.

Several scoping-level gold/silver recovery tests were completed by PRA Metallurgical Division on copper leach residues from the April 2009 copper column leach test program. The focus of this work was to demonstrate the recovery of gold and silver from copper leach residues by cyanidation in a heap leach setting. The size distribution of material used for this testwork was considerably coarser than is predicted for a commercial operation so that copper, gold and silver extraction may be understated and required leach times may be overstated. The testwork does however demonstrate the viability of precious metal leaching from copper leach residues, achieving gold extractions of 79.80% and 80.05% in two column tests. The silver recovery ranged from 74.6% to 96.2% in the two tests.

The PRA testwork utilized carbon to demonstrate the removal of gold and silver from process solutions before recirculation. It was noted that although copper is dissolved during the cyanidation of the copper leach residues, very little of the copper adsorbed on the carbon.

The testwork completed to date is preliminary in nature and detailed testwork is required to optimize the operating conditions and provide a basis for accurate predictions of metal recoveries.

13.2.6 Lime and Cyanide Consumption

Following copper leaching and rinsing lime is added to the gold/silver feed material to increase the pH to an alkaline condition. Testwork completed by both UBC and PRA demonstrated that a lime $Ca(OH)_2$ addition of 10 kg/t would be adequate to control the pH to the desired level.

Cyanide consumption in testwork was shown to be relatively high at 1 to 2 kg NaCN/t due to the presence of cyanide-soluble copper. Testwork completed at UBC demonstrated that through the use of the SART process a significant portion of the cyanide that was associated with soluble copper could be recovered with concurrent precipitation of a copper sulphide product. Tests that included SART resulted in an overall cyanide consumption of 0.4 to 0.9 kg NaCN/t, depending on cyanidation conditions. The test results were derived from tests on material in which copper had only been partially leached and the low copper extraction is expected to have contributed to the overall results. For the current analysis a cyanide consumption of 0.5 kg NaCN/t has been used.

13.2.7 Other Reagent Requirements

Process testing for gold/silver recovery has not progressed to the point where minor reagent consumptions have been demonstrated. The following consumptions are taken from normal industry practice.

In addition to lime and cyanide, the process requires carbon for gold/silver recovery from solution and hydrochloric acid for regeneration of the carbon. The final precious metals precipitate will be smelted into a dore bar with the addition of a flux. The consumption of reagents is summarized as follows:

- Carbon 0.02 kg/t
- HCl 0.003 kg/t
- Flux 0.01 kg/t

The composition of the flux is as follows: 30% SiO2, 40% Borax, 10% niter, 20% soda ash.

For the SART process the reagent consumptions based on estimated solution compositions and testing completed to date are:

- 0.40 kg H2SO4 / m^3
- 0.37 kg NaOH / m^3
- 0.2 kg/NaSH / m^3

13.3 Metallurgical Testing (2014 to 2016) – Agitated Tank Leach Process

During 2014-2015, Bureau Veritas Commodities Canada (BV Minerals) completed a full suite of metallurgical testing to evaluate an alternative to heap leaching (JDS, 2016). This new vat leach recovery method focused on grinding the samples to a P_{80} of 664 µm, and leaching with sulphuric acid to recover the copper. The leach residue was then leached with cyanide to recover the gold and silver. Metallurgical testing included flowsheet parameter finalization, a full locked cycle Cu/Au leach test, cyanide destruction, and variability comminution and batch leach testing.

- BV Minerals, Project No. 1500602 “Metallurgical Testing of Samples from the Copper North Mining Corp., Carmacks Project, Yukon Phase 1”, February 18, 2015 (BV Minerals 2015a); and
- BV Minerals, Project No. 1500602 “Metallurgical Testing of Samples from the Copper North Mining Corp., Carmacks Project, Yukon Phase 2”, July 21, 2015 (BV Minerals 2015b).

Sample Selection

Thirteen different composites were used in the 2014 test program. In Phase 1, four trench composites (BS-1, BS-2, BS-3 and BS-4) were created and a Master Composite was constructed from core samples. In Phase 2, eight variability composites were created from the remaining core samples kept in storage. The head assays for all 13 samples are summarized in Table 13-2.

Comminution Testing

Variability comminution testing was carried out on all 13 composite samples. Bond ball mill work index (BWI), Bond rod mill work index (RWI) and Bond abrasion index (AI) testing were completed to assist in designing the comminution circuit. Based on the results, a RWI of 9.1 kWh/t, a BWI of 15.2 kWh/t and an AI of 0.09 g were selected for design.

Table 13-2 BV Minerals’ Test Program Head Assays (from JDS, 2016)

Sample ID	Cu (%)	Au (g/t)	Ag (g/t)	S (%)	Total Carbon (%)	Organic Carbon (%)
Master Composite	1.10	0.51	5.00	0.07	0.11	0.09
Trench Composite BS-1	1.39	0.68	6.00	0.07	0.10	0.10
Trench Composite BS-2	1.18	0.72	7.00	0.07	0.08	0.07
Trench Composite BS-3	0.71	0.14	2.00	0.04	0.12	0.07
Trench Composite BS-4	0.79	0.49	4.00	0.05	0.07	0.06
Composite 1(CAR Z1S-OX50)	0.73	0.18	2.00	0.04	0.07	0.04
Composite 2 (CAR Z1S-OX150)	0.54	0.08	2.00	0.00	0.04	0.03
Composite 3 (CAR Z1N-OX50)	1.00	0.61	7.00	0.11	0.09	0.06
Composite 4 (CAR Z1N-OX150)	0.91	0.44	4.00	0.05	0.07	0.06
Composite 5 (CAR Z4-OX)	0.45	0.15	2.00	0.01	0.14	0.06
Composite 6 (CAR Z7-OX)	1.09	0.36	4.00	0.02	0.05	0.04
Composite 7 (CAR Z1-SX)	0.92	0.19	4.00	0.93	0.09	0.07
Composite 8 (CAR Z1213-SX)	0.69	0.14	3.00	0.87	0.29	0.16

Batch Copper Leach Testing

Fifteen preliminary copper acid leach tests were completed on Master Composite in Phase 1. An additional six tests were completed in Phase 2 to confirm operating parameters for variability and locked cycle testing. At a P_{80} of 664 µm and a solids density of 50%, temperatures in the range of 20 – 50ºC and leach times of eight and 12 hours were tested. As temperature increases, so does copper extraction.

Based on preliminary testing, the following operating parameters were selected for variability testing on the remaining composites:

- P_{80} Grind Size = 664 µm;
- Slurry Temperature = 40ºC;
- Pulp Density = 50% Solids;

- Leach Time = 6 hours; and
- Acid Addition = 30 g/L raffinate to start at 23 g/L H_2SO_4 (5 g/L added at 1hr mark).

Copper recovery ranged from 76.5 to 88.8%.

Batch Gold Cyanidation Testing

The residue from each preliminary Master Composite copper leach test was subjected to a bottle roll leach test using cyanide. The recovery curves flatten out after 12 hours.

Bottle roll leach tests, at 1.0 g/L NaCN, were also conducted on the residues from copper leach variability testing. Gold recovery ranged from 38 to 83.4% and silver recovery ranged from 6.8 to 83.1%.

Locked Cycle Testing

A single locked cycle test was done on 6 kg of Master Composite to assess circuit stability and potential copper and gold recoveries. The copper circuit remained very stable throughout the test and resulted in an average Cu recovery of 87.9% at a PLS Cu grade of 8,083 mg/L.

As the recycle products increased and the overall circuit settled into a constant operating mode, the PLS solution grade stabilized at 0.45 mg/L Au. Since an arbitrary volume equivalent to 20% was removed following the first 8-hour cyanide leach stage, it was not possible to accurately calculate a cycle by cycle recovery. An overall recovery of 80.1% Au and 61.8% Ag was determined based on an equal volume of PLS being removed in each cycle. The report noted that overall recovery would be expected to increase as PLS remained constant at 0.45 mg/L in subsequent cycles.

Cyanide Destruction

Scoping SO_2/Air cyanide destruction tests were performed on the cyanidation residue from each cycle of the locked cycle test. The test was carried out in batches using a 20 L agitated reactor. The source of the SO_2 came from a concentrated solution of sodium metabisulphite (SMBS), with copper in the form of copper sulphate ($CuSO_4$) added as needed. Oxygen was provided by compressed air aeration and the reaction lasted for three hours. The pH was maintained at 8.7 by adding hydrated lime slurry.

Settling and Filtration Test Work

Settling tests were conducted on cyanide leach residue from test C8 (Phase 1) and CLT 3 (Phase 2).

Vacuum filtration tests were conducted on the Master Composite to estimate filter cake moisture content and evaluate various filter cloths. The sample was subjected to pour-on-leaf vacuum filtration tests.

Copper Leach Testing at Elevated Temperature

In February 2016, BV Minerals conducted additional copper leach optimization test work using 2014 Master Composite.

- BV Minerals, Project No. 1500602 “Additional Metallurgical Testing of Master Composite from the Copper North Mining Corp., Carmacks Project, Yukon”, April 8, 2016 (BV Minerals, 2016).

Three leach tests evaluated the effect of increased temperature and ferric addition on copper recovery. Operating the copper leach at 80ºC improved copper extraction by 2.2% over the 40ºC baseline, while ferric addition increased copper extraction slightly. Further testing is strongly recommended to confirm the effect of ferric addition at different concentrations and leach temperatures.

The copper leach residue was then leached with cyanide to investigate the effect of the noted improved copper extraction on subsequent gold recovery. A reduction in copper reporting to cyanidation as a result of the higher extraction at elevated temperature, appeared to improve gold recovery by 1.7 – 2.7%, while ferric addition appears to hinder silver recovery.

13.3.1 Relevant Results

Based on the test work summarized above, a copper / gold leach circuit was selected as the preferred recovery method. The criteria and recoveries from CALT2 were selected for design due to the low copper grade reporting to the gold/silver leach circuit, eliminating need for the sulphidization, acidification, recycling and thickening (SART) process that was part of the previous process plan. Mineralized material will be reduced to a P_{80} of 664 µm using a jaw crusher followed by a SAG mill in closed circuit with hydrocyclones. Copper will be recovered using a sulphuric acid leach and solvent extraction / electrowinning (SX-EW). Normally high silver grades and low gold grades dictate the use of Merrill Crowe; however ferric sulphate addition substantially reduces silver recovery, allowing for a smaller footprint with a CIL circuit. Copper leach residue will be neutralized and gold/silver will be leached into solution using cyanide while simultaneously being adsorbed onto activated carbon. An ADR circuit will be implemented to concentrate the gold/silver into doré bars. Preliminary design criteria and reagent requirements are summarized in
Table 13-3 and
Table 13-4.

Table 13-3 Process Design Criteria Derived from Test Work (from JDS, 2016)

Description	Units	Value
Ore Characteristics		
Ore Solids Density	t/m^3	2.64
Ore Moisture	% w/w	7.6
Crushed mineralized material Bulk Density	t/m^3	1.64
Bond Rod Mill Work Index	kWh/t	9.1
Bond Ball Mill Work Index	kWh/t	15.2
Abrasion Index	g	0.09
Thickener Loading Rate	t/h/m_2	0.25
Filter Filtration Rate	kg/m^2/hr	230
Head Grade		
Head Grade (Average LOM)	% Cu	0.98
Head Grade (Average LOM)	g/t Au	0.435
Head Grade (Average LOM)	g/t Ag	4.34
Metal Recovery		
Copper Leach Recovery	% Cu	86.3
Copper Overall Recovery	% Cu	85.2
Gold Leach Recovery	% Au	85.5
Gold Overall Recovery	% Au	84.4
Silver Leach Recovery	% Ag	9.5
Silver Overall Recovery	% Ag	9.4
Copper Leach		
Leach Time	hr	6
Leach Density	%	50
Leach Temperature	ºC	80
Gold/Silver Neutralization		
Neutralization Time	hr	1

Description	Units	Value
Neutralization Density	% Solids	40
Neutralization Ph	-	10.5
Gold/Silver Leach		
Process Selected	-	CIL
Leach Time	hr	12
Leach Density	% Solids	40
Gold/Silver Recovery		
Process Selected	-	ADR
Circuit Capacity	t Carbon	2
Carbon Loading	g Au/t Carbon	3,000*
Cyanide Destruction		
Destruction Retention Time	hr	3
Total Cyanide Concentration	mg/l TCN	245
SMBS Consumption	g SMBS / g TCN	9.89
Lime Consumption	g Lime / g TCN	3.46
Copper Sulphate Consumption	g $CuSO_4 \cdot 5H_2O$ / g TCN	0.49
Filtration		
Filter Filtration Rate	kg/m2/hr	230
Filter Cake Density	%	83
Filter Cake Moisture Content	%	17

Table 13-4 Reagent Consumption Derived from Test Work (from JDS, 2016)

Description	Units	Value
Lime	g/t	3,917
Elemental Sulphur	g/t	5,393
Sodium Cyanide	g/t	630
Flocculant (Magnafloc 333)	g/t	120
Sodium Metabisulphite	g/t	4,552
Copper Sulphate	g/t	226
Acorga M5774	g/t	17
Kerosene	g/t	85
Guartec	g/t	2
Cobalt Sulphate	g/t	6
Ferric Sulphate	g/t	90
Carbon	g/t	33

13.4 Metallurgical Testing 2021

In 2021, metallurgy testing was completed on sulphide and oxide mineralized core samples from the Carmacks Project (Sedgman, 2021 a, b).

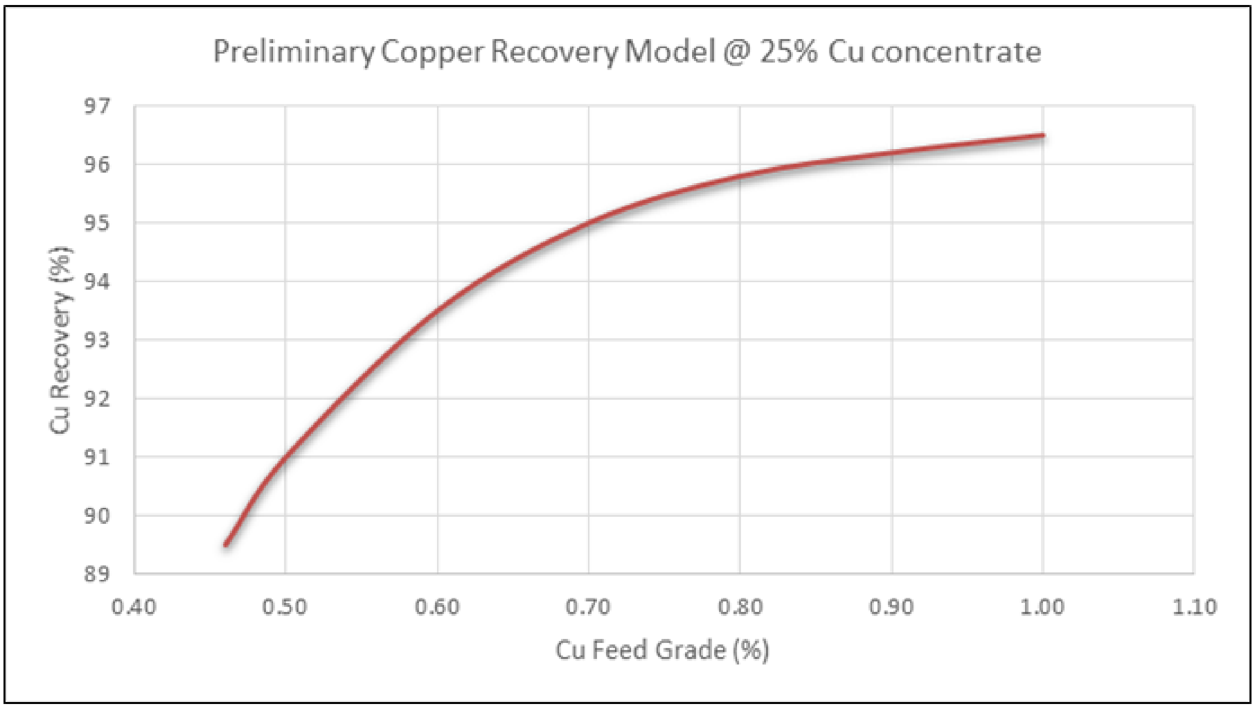
Two representative samples of copper sulphide material and one sample of copper oxide material were delivered to Bureau Veritas Commodities, Metallurgy Division, for rougher flotation kinetic testing and open cleaner flotation testing. The purpose of the testing was to determine how amenable the sulphide mineralization present at the Carmacks deposit was to concentration by flotation, what recoveries could be expected, and to lay the groundwork for further testing.

13.4.1 Carmacks Sulphide Metallurgical Testwork

Rougher flotation kinetic tests and open cleaner flotation tests were conducted on the Var 2 and Var 4 variability samples at Bureau Veritas Commodities, Metallurgy – Mineralogy Division, (BV). The Carmacks primary sulphide samples selected for flotation test work provided high flotation copper recoveries at P_{80} 150 µm primary grind and P_{80} 25 µm regrind sizes. Preliminary copper flotation recovery model is generated based on two open circuit cleaner flotation test results at fixed 25% copper concentrate grade as shown in Figure 13-1. This preliminary recovery estimation should only apply to copper feed grade of 0.46% to 1.0%, which is the typical grade of sulphide domain at Carmacks.

Running parallel to this flotation test work program was a process mineralogy assessment to determine the mineral composition and mineral liberation and association for the sample at the defined primary grind size of P_{80} 150 µm. Particle Mineral Analysis (PMA) indicated that most of the sulphide mineral consists of chalcopyrite and minor covellite/chalcocite, with minor pyrite. Copper sulphide minerals are well liberated for rougher recovery via flotation at P_{80} of 150 µm. Gold is associated with mainly copper sulphide minerals and some pyrite. The majority of gold particles are <10 µm and are categorized as refractory and flotation of gold with copper concentrate would be the most economical way to recover this gold.

Figure 13-1 Preliminary Copper recovery estimate @ 25% Cu concentrate grade.



Sulphide Sample ID and Composition

The primary sulphide variability samples (Var 2 and 4) selected for flotation test work are from Zone 2000S hole CRM21-003 and continuous intersection from 152 m to 168 m and 188m to 202m. A total mass of 43 kg and 42 kg was provided for Var 2 and Var 4, respectively.

The chemical composition for the two sulphide samples and the oxide sample sent to BV for process mineralogy and flotation test work assessment is provided in Table 13-5. The sulphide samples contain 0.46 and 0.76% Cu and 0.18 and 0.17 g/t Au, respectively.

The mineral compositions for the two sulphide samples and the oxide sample are provided in Table 13-6.

Chalcopyrite is main copper sulphide mineral with relatively low pyrite content. Both Var 2 and 4 sulphide samples are expected to float easily with no or very little difficulties.

Over 90% of the sulphide copper content is made up with chalcopyrite and the rest as covellite/chalcocite. When liberated, chalcopyrite is expected to exhibit fast flotation kinetic and recover it in relatively short period of time.

Table 13-5 Chemical Composition of Two Sulphide Samples and an Oxide Sample used for the 2021 Metallurgical Test Work as Determined by BV (from Sedgman, 2021a)

Element	Symbol	Chemical Compositions (percent or grams/tonne)		
		Var 2 Sulphide Composite	Var 4 Sulphide Composite	Oxide Composite
Copper	Cu	0.46	0.76	1.01
Iron	Fe	2.39	4.75	3.56
Molybdenum	Mo	0.01	0.04	0.01
Sulphur	S	0.78	1.19	0.03
Carbon	C	0.05	0.06	0.24
Gold	Au	0.18	0.17	0.18

Notes: 1) Gold w as measured in grams/tonne. All other elements w ere measured in percent.

Table 13-6 Mineral Compositions for Two Sulphide Samples and the Oxide Sample as Determined By BV (from Sedgman, 2021a)

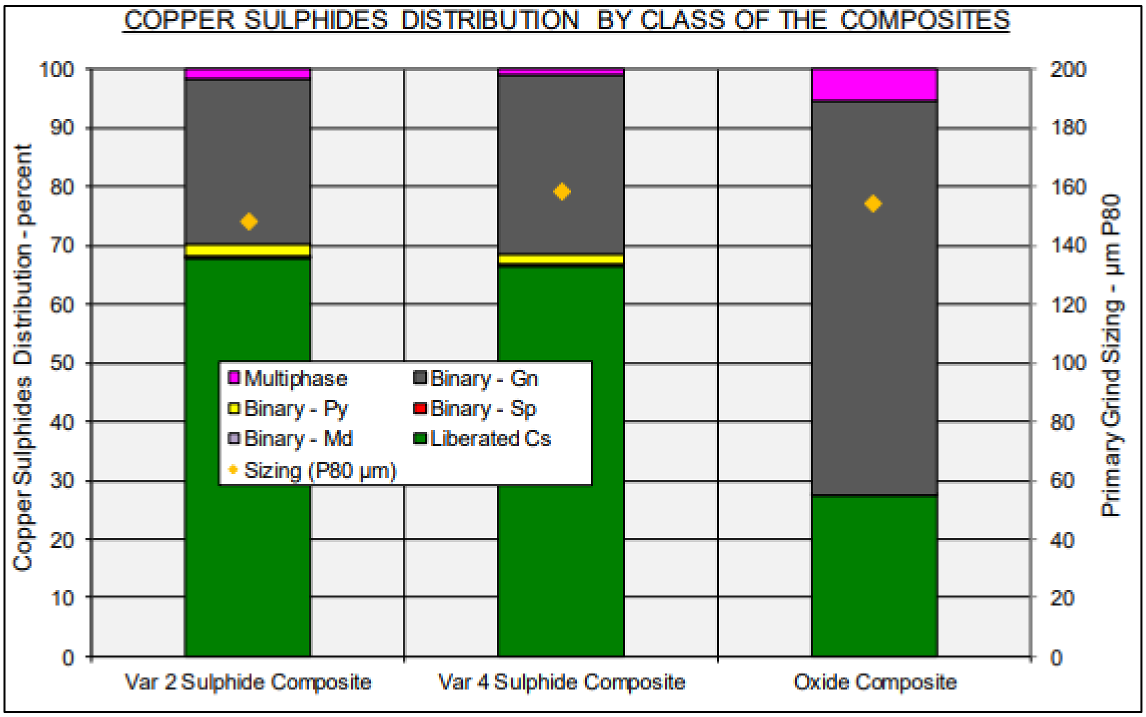
Mineral	Mineral Compositions (Mass percent)		
	Var 2 Sulphide Composite	Var 4 Sulphide Composite	Oxide Composite
Chalcopyrite	1.23	2.14	0.10
Molybdenite	0.04	0.12	<0.01
Pyrite	0.64	0.44	0.02
Other Sulphides	0.02	0.02	0.01
Sulphide Total	**1.93**	**2.71**	**0.13**
Malachite/Azurite	<0.01	0.00	0.59
Credhneite (CuMnO2)	0.00	0.00	0.15
Iron Oxides	0.37	0.33	2.49
Quartz	21.6	10.5	6.60
Plagioclase Feldspar	32.0	42.1	59.9
K-Feldspars	30.7	9.85	4.11
Biotite/Phlogopite	4.65	16.9	5.21
Amphibole (Actinolite)	0.49	4.13	6.10
Chlorite	4.61	4.34	10.6
Muscovite	0.90	1.53	0.75
Epidote	0.88	4.24	0.26
Calcite	0.30	0.38	1.06
Sphene/Titanite	0.92	1.15	0.92
Apatite	0.41	0.70	0.68
Ca-sulphate (Gypsum)	0.01	0.88	<0.01
Others	0.20	0.18	0.44
Total	100.0	100.0	100.0

Notes: 1) Chalcopyrite includes Bornite, Chalcocite/Covellite, Enargite/Tennantite and Tetrahedrite.
Other sulphides include Sphaleirte and Galena.
2) Sphene/Titanite includes trace amounts of Rutile/Anatase and Ilmenite.
3) Others include Kaolinite (Clay), Zircon, Fluorite, Barite, Corumdum, Cassiterite and Al-Phosphate

Sulphide Sample Process Mineralogy

The copper sulphide minerals have high level of liberation (~67%) as illustrated in Figure 25. Most two-product copper concentrators (one concentrate + one tailing) operate with 50 – 60% copper mineral liberation in flotation feed according to an operational benchmark database. The non-sulphide gangue liberation is equally important and for Var 2 and Var 4 the two-dimensional liberation was measured to be 97.4% and 96.5%, respectively. This level of liberation at the primary grind size of 154 µm P_{80} means that coarser primary grind sizes should be explored in test work because the samples are expected to maintain sufficient liberation of copper sulphide and non-sulphide gangue minerals for effective rougher flotation separation.

Figure 13-2 Copper Mineral Liberation for the two Sulphide Samples and the Oxide Sample



BV plotted the limiting grade-recovery curves for the two sulphide samples and one oxide sample. Limiting grade-recovery curves present the maximum theoretical grade and recovery performance if all copper minerals reported to concentrate without any gangue entrapment. They are best applied as a guide to determine whether metallurgical performance in plant operation or test work development programs is achieving efficient mineral separation in respect to the liberation profile of the fragmented minerals in the feed.

The limiting grade-recovery plot for the Carmacks sulphide sample identifies that a suitable copper grade of 6% - 12% can be achieved to rougher concentrate at high recovery.

Sulphide Sample Flotation Performance

BV completed three open circuit batch rougher flotation tests for both sulphide samples. Different combination and dosages of PAX and 3418A collectors are used to float all sulphide minerals. Modifiers, such as lime to regulate pH, were not needed to depress iron sulphides (pyrite) because the composition of these gangue sulphides is low. Rougher concentrate was collected and assayed in 5-minute intervals for total of 25 minutes flotation. Notable characteristics of the rougher flotation are:

1. Var 4 (0.76% Cu) showed faster flotation kinetic than the lower grade Var 2 (0.46% Cu) sample. Both samples resulted in typical porphyry copper rougher flotation performance of ~ 10 concentration ratio (feed weight/concentrate weight) and enrichment ratio (concentrate grade/feed grade) of ~0.1. Var 4 reached 98% Cu rougher flotation recovery with 11% mass pull and 7.9% Cu grade in 15 minutes and Var 2 reached ~98% Cu rougher recovery with 9.0% mass pull and 5.6% Cu grade in 25 minutes. Rougher flotation was very simple with no copper mineral separation difficulties.

2. Rougher flotation kinetics for both Var 2 and 4 are relatively slow (25 minutes) compared to our benchmark kinetics for porphyry copper ores with chalcopyrite as the main copper mineral. Most porphyry copper ores with chalcopyrite as the main copper mineral and low pyrite/clay content would exhibit fast flotation kinetics and likely complete rougher flotation in 5 to 10 minutes. Flotation kinetics is usually determined by particle size, liberation, and hydrophobicity. More testing is required to optimize the flotation performance in the next stage of study.

3. Overall selectivity of Var 4 sample was far superior to that of Var 2 sample. More kinetic flotation test work with a shorter time frame (less than current 5 minutes) analysis should be conducted in the future works to fully understand the kinetic spectrum.

4. Gold recovery is closely related to Cu recovery. As the copper recovery increases, the gold recovery increases but at a slower rate, indicating that some gold is associated with gangue minerals and/or liberated gold particles that are slower floating. Much of the gold associated with gangue is likely in pyrite. Rougher flotation was performed to float all sulphide minerals including pyrite. The BV gold deportation (QEM Scan PMA) analysis confirms the gold detected is associated with copper minerals and pyrite. However, gold content in non-sulphide gangue is not conclusive from the gold deportation analysis.

Sulphide Cleaner Flotation Performance

BV completed three open circuit cleaner flotation tests for both sulphide samples. PAX was used as the only collector and MIBC as the frother with lime to adjust the pH in the cleaners only. pH of the cleaner flotation slurry was increased as the cleaner stages advanced. BV applied pH of 9.5 and a flotation time of 9 minutes for the first cleaner and 3 minutes for the cleaner scavenger, 10.5 pH and 5 minutes for the second cleaner, and 11.5 pH and 3 minutes for the third cleaner. Three different regrind sizes (no regrind, P80 of 36 µm, and P80 of 23 µm) were studied and the 23 µm test provided the best copper selectivity (best grade and recovery). Cleaner test results at P80 of 23 µm will only be analysed in the following discussion.

Notable characteristics of the cleaner flotation tests include:

1. Overall copper selectivity of Var 4 sample was superior to that of Var 2 sample. 25% copper concentrate grade is likely to be achieved with one (1) stage cleaning with ~96% overall recovery when Var 4 (0.76% Cu & low pyrite) or greater feed grade is introduced to the plant. Current cleaner flotation condition; using PAX as the only collector and MIBC as the frother with incremental pH adjustment over three (3) stage cleaning at P80 of 23 µm regrind; did not achieve 25% copper concentrate grade with satisfactory recovery for Var 2 (0.46% Cu & low pyrite). Further variability samples need to be selected to determine whether the liberation characteristics of Var 2 is typical or not for the sulphide deposit. If Var 2 is determined to have typical liberation characteristics, then regrinding to 15mm P80 should be completed.

2. Gold recovery is closely related to Cu recovery. For Var 2 sample, gold recovery changes proportional to copper recovery in cleaner circuit flotation testing, indicating that most of the gold in this sample is associated with copper minerals (mainly chalcopyrite). Var 4 sample however shows a drastic change of gold recovery as the copper recovery varies at different stages of cleaner

flotation, indicating some gold is associated with gangue minerals (mostly pyrite in this case) that is depressed by increasing pH.

3. Pyrite associated gold in Var 4 is hard to recover once depressed in the cleaner circuit. Further flotation test work will focus on decreasing the pH in cleaning to reduce costs and increase gold recovery to copper concentrate. Gold deportation QEMScan PMA analysis confirms that most of the gold is refractory (<10 µm size gold) in sulphide minerals. It is therefore very important to reject the minimum amount of pyrite by adjusting the flotation conditions to minimize gold loss.

Recovery Estimation

Two open circuit rougher-cleaner flotation test results were used to generate the preliminary copper recovery estimation at 25% copper concentrate grade. Open circuit cleaner flotation tests resulted in 24% Cu grade (maximum slightly above this value) and 85% Cu recovery for Var 2 and 25% Cu grade and 95.5% Cu recovery for Var 4. Sedgman experience was applied to scale up the open circuit flotation results to actual likely process plant recovery values of 88% for Var 2 and 96% for Var 4. Based on these two estimated recovery numbers, copper recovery vs copper feed grade plot was further extrapolated using first order equation as shown below. This preliminary recovery estimation should only apply to Cu feed grade of 0.46% to 1.0%.

Preliminary Copper Recovery = 97*(1-exp(-5.6 x Cu feed grade))

Gold recovery modelling is difficult with two samples containing almost equal contents of gold 0.16g/t (Var 2) and 0.18g/t (Var 4) and therefore a flat gold recovery at 76% gold recovery is recommended as a preliminary estimate until the project advances with more testing to support an improved model.

As the project advances, more batch and locked cycle tests (LCT) are required on a larger pool of representative samples to increase the confidence level of the recovery estimation.

Conclusion and Recommendation

The Carmack sulphide samples tested in 2021 contain chalcopyrite and minor covellite/chalcocite as the copper containing minerals, with low levels of pyrite (Sedgman, 2021a). A series of flotation kinetic and open cleaner circuit tests demonstrated these two samples were easy to concentrate using PAX and MIBC in the flotation process at P80 of 150 µm for rougher flotation and 23 µm regrind size for the cleaner flotation stages. However, rougher flotation time was unusually long compared to typical porphyry copper ore and further optimization test work needs to be conducted as the project advances. Two initial flotation test results indicated that the main Carmacks ore (represented by Var 4 sample, 0.76% Cu) can be recovered at >95% into a 25% copper concentrate grade.

As the project moves forward, further metallurgical test work and variability samples are required to evaluate process equipment design parameters as well as validate copper and gold recoveries.

13.4.2 Carmacks Oxide Metallurgical Testwork

The Carmacks oxide sample selected for flotation testwork provided a maximum copper rougher recovery of approximately 41%, at 2.8% Cu concentrate grade (Sedgman, 2021b). Running parallel to this flotation testwork program was a process mineralogy assessment to determine the mineral composition, and mineral liberation and association for the sample at the defined primary grind size of P^{80} 150 µm.

Particle Mineral Analysis (PMA) indicated that the majority, approximately 61%, of the copper content is locked in credhneite, iron oxide and chlorite minerals and is not floatable due to the minerals being hydrophilic and no known processes exist to selectively increase the hydrophobicity of these minerals. The remaining 39%, where majority are copper oxide minerals (malachite/azurite/cuprite) can be recovered via flotation with sulphurdisation and/or special oxide collectors, which presents a very low cap on achievable metallurgical performance of ores represented by this sample. This PMA finding is backed up by batch rougher flotation testwork.

The liberation of the copper bearing minerals in the oxide sample is extremely poor, wherein an uneconomic primary grind to sub-20 μm P^{80} is required to overcome this issue.

An acid leach test of the oxide sample was undertaken to determine whether the high extraction rates previously reported (PEA 2016) are achievable. Acid (for copper extraction) and cyanide (for gold extraction) leaching results were found to be comparable to the PEA 2016 work, even though the primary grind sizes are significantly different. This is an important result as it suggests that the PEA samples likely contain a similar composition of minerals to the oxide sample and there is no separate sub-domain of Carmacks oxide. Therefore, all oxides at Carmacks Copper are likely to be similar.

This rules out process pathways via oxide flotation, and even glycine and ammonia leaching, which have not been demonstrated to leach the most abundant copper containing minerals identified in the oxide sample. It is recommended that any samples available from the 2016 PEA program be identified and submitted for mineral composition analysis to confirm that the oxide zone is similar across the Carmacks Copper resource before selecting further variability samples.

Oxide Sample ID and Composition

The oxide sample selected for flotation testwork is from Zone 1 hole WC-021A and continuous intersection from 14 m to 28 m. A total mass of 51 kg was provided. The oxide content of the sample was estimated to be 90%.

The chemical composition for the two sulphide samples and the oxide sample sent to BV for process mineralogy and flotation testwork assessment is provided in Table 13-5. The oxide sample contains 1.01% Cu and low levels of sulphur (0.03% S). It is important to run flotation and mineralogy programs in parallel because the process mineralogy can provide important details about the composition of the sample and estimates for primary grind and regrind size targets that should be feed into the flotation testwork program for guidance on flowsheet development.

The mineral compositions for the two sulphide samples and the oxide sample are provided in Table 13-6. As expected, the copper sulphide content of the oxide sample is very low at 0.10%. The oxide sample also contains the copper bearing minerals Malachite/Azurite at 0.59% and Credhneite at 0.15%. Other minerals found to contain copper in the oxide sample are iron oxides (2.49%), Chlorite (10.6%) and biotite (5.21%).

Copper deportment by copper bearing minerals for the two sulphide samples and the oxide sample is as follows:

61% of the copper content in the oxide composite sample is from non-sulphide gangue minerals, including:

- credhneite (6.5%);
- iron oxides (mostly goethite and limonite) (25.7%);
- chlorite (28.6%); and
- minor Cu-biotite.

The presence of these minerals is most important because they are not expected to be recovered via the flotation process.

It is possible that the remaining 39% of the copper minerals can be recovered via flotation, which presents a very low cap on achievable metallurgical performance of ores represented by this sample. These minerals include:

- malachite/azurite (30.3%);
- cuprite (4.6%);
- chalcocite/covellite (1.3%);
- bornite (0.1%); and
- chalcopyrite (2.5%).

Oxide Sample Process Mineralogy

The two-dimensional liberation of the dominant copper bearing minerals malachite/azurite (including cuprite) in the oxide composite (at P_{80} of 154 µm) is 40%. In addition, the copper sulphides (including chalcopyrite, bornite, chalcocite/covellite and enargite/tennantite) have a very low level of liberation (28%). Most two-product copper concentrators (one concentrate + one tailing) operate with 50 – 60% copper mineral liberation in flotation feed according to an operational benchmark database. This poor level of liberation at the primary grind size of 154 µm P_{80} means that a finer primary grind is required to achieve sufficient liberation of copper sulphide minerals for efficient flotation recovery.

Interpretation of the liberation profile by size fractions of the copper bearing minerals Malachite/Azurite and Cu Sulphides in the oxide sample, suggest that a primary grind to sub-20 µm P_{80} is required to achieve the copper mineral liberation target of typical two-product copper concentrators. Clearly, it is not economically feasible to primary grind the entire ore to sub-20 µm P_{80}.

Oxide Sample Flotation Performance

BV completed three open circuit batch rougher flotation tests for the oxide sample. Notable characteristics of these plots are:

1. The limit for the oxide sample copper rougher recovery to 35% – 40% as predicted by the mineralogy data; and
2. The first rougher concentrate grade and shape of the grade-recovery curves indicate a process struggling to achieve high concentrate grade targets because of poor copper mineral liberation or copper minerals competing with gangue minerals for recovery or both. Interpretation of the liberation data certainly supports the earlier case that there is poor copper mineral liberation.

The rougher mass recoveries after 25-minute laboratory flotation times are:

1. 1150g/t AM28 = 15%
2. 550g/t AM28 + 50g/t PAX = 10%
3. 300g/t NaSH + 50g/t PAX = 5%

Even with slow flotation kinetics and low rougher mass recovery the best rougher stage grade achieved is 9.1% copper grade at 10% recovery. Similarly, gold recovery appears limited via flotation to less than 50% Au in total.

BV plotted the limiting grade-recovery curves for the two sulphide samples and one oxide sample. Limiting grade-recovery curves are very important that they present the maximum theoretical performance that can be expected if all copper minerals reported to concentrate at a target primary grind size. The limiting curves are never achieved but approached. They are best applied as a guide to determine whether metallurgical performance in plant operation or testwork development programs is achieving efficient mineral separation in respect to the liberation profile of the fragmented minerals in the feed.

The limiting grade-recovery plot for the Carmacks oxide sample at P_{80} of 154 µm identifies the maximum copper recovery at just under 40% (the “cap on recovery” via flotation), if all the copper sulphides and malachite/azurite where recovered. It also highlights that if all copper sulphides and malachite/azurite were recovered to the copper concentrate then the best copper concentrate grade achievable is 26.5% copper. Any increase in copper recovery beyond 40% will coincide with a significant decrease in copper concentrate grade achievable.

The oxide flotation testwork on this sample was halted due to the poor rougher flotation recovery of the oxide sample and details about the mineral composition which support the flotation results that the majority of copper minerals present should not recover via the flotation process.

Oxide Sample Leach Performance

BV completed an acid leach test for copper and cyanide leach for gold using the same oxide sample. The leach conditions were kept consistent with the 2015 test work program, except for the primary grind size is 252 µm P_{80} (as-received oxide sample). The 2015 test work was undertaken at a primary grind size of approximately 660 µm P_{80}.
Note: No detailed mineral composition analysis can be found for the 2015 test work.

Leach extraction comparisons are provided below:

1. Copper extraction after 6 hours of acid leaching = 83%
   - 2015 results after 6 hours of acid leaching ranged from 76.5% to 88.8%
2. Gold extraction after 12 hours of cyanide leaching = 77%
   - 2015 results after 12 hours of cyanide leaching typically ranged from 71% to 83%, apart from a few outliners of lower gold extraction

The oxide sample leaches similarly to the 2014 Master composite under similar leach conditions. Therefore, the mineral composition of these two samples is likely to be similar.

It is recommended that any samples available from the 2015 program be submitted for mineral composition analysis to confirm that the oxide zone at Carmacks Copper is similar across the resource and no significant variations in mineral compositions exist. The intention of this work is to link knowledge gained across the current and previous testwork programs. This result simplifies the development program forward for this project because options such as oxide flotation can be confidently ruled out and options such as glycine and ammonia leaching haven’t yet been reported to extract copper from credhneite, goethite, limonite and chlorite which accounts for 61% of the copper content in the oxide sample.

Conclusion and Recommendation

The Carmack oxide sample presented for testwork in 2021, contains 61% of the copper in mineral forms not expected to recover by flotation (from Sedgman, 2021b). This limits the copper concentrate recovery to 30% - 39% for this sample and laboratory flotation testwork confirmed the mineralogy assessment.

These concerns were substantial enough to recommend halting the flotation testwork program for this oxide sample. Sedgman then recommended that an acid leach test of the sample be undertaken to determine whether the high extraction rates previously reported (JDS, 2016) are achievable. Acid (copper) and cyanide (gold) leaching results were found to be comparable to the 2015 work, even though the primary grind sizes are substantially different. This is an important result as it suggests that the 2015 samples likelycontain a similar composition of minerals to the oxide sample and there is no separate sub-domain of Carmacks oxide. No detailed mineral composition analysis can be found for the 2015 testwork.

This rules out process pathways via oxide flotation, and even glycine and ammonia leaching which have not been demonstrated to leach many of the copper containing minerals identified in the oxide sample. It is recommended that any samples available from the 2016 PEA program be identified and submitted for mineral composition analysis to confirm that the oxide zone is similar across the Carmacks Copper resource, as well as perform further variability testwork and mineral composition assessments.

14 MINERAL RESOURCE ESTIMATES

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 Carmack 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 Carnacks 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: Distribution of Drilling in the Carmacks Deposit Area

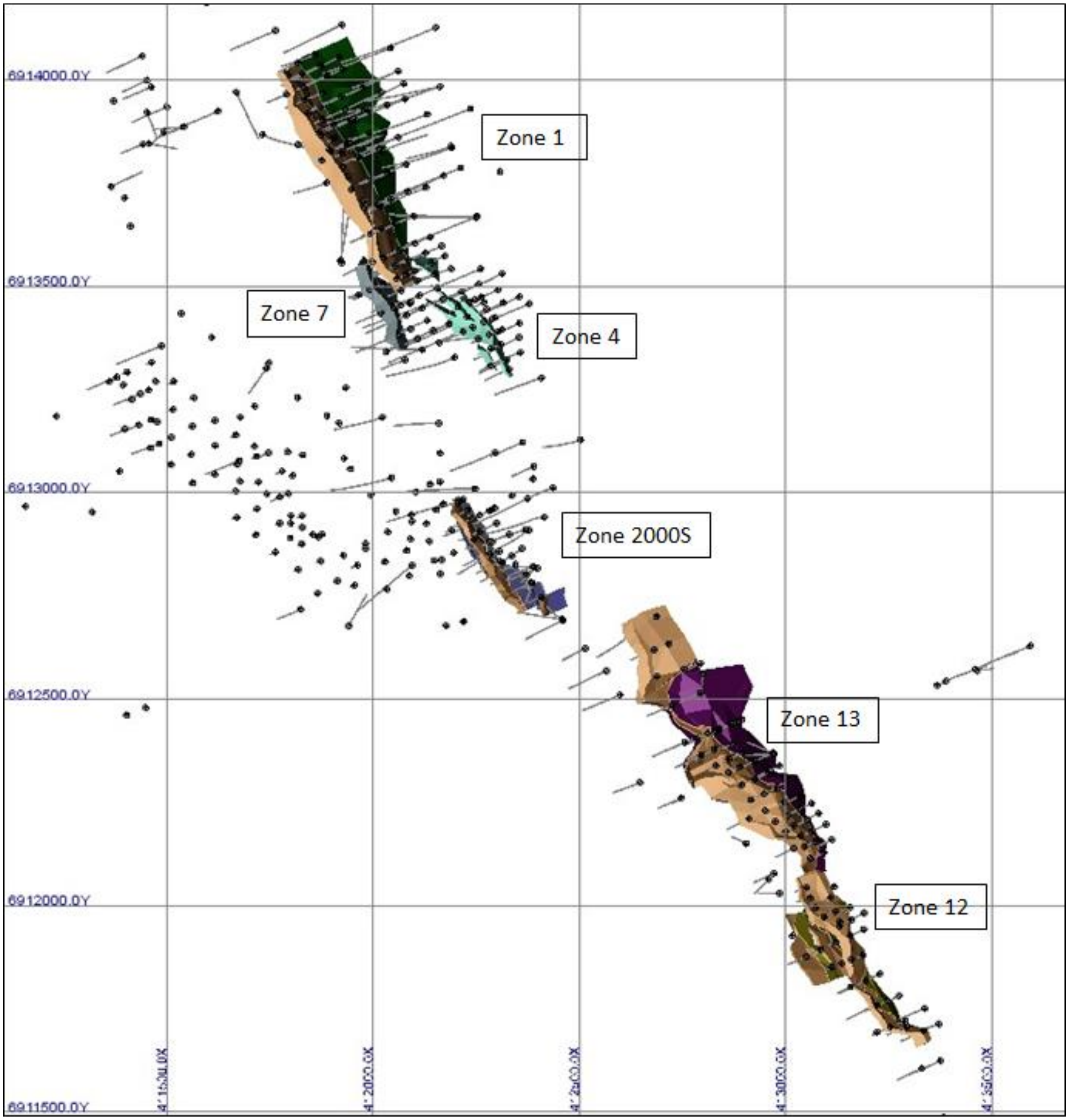
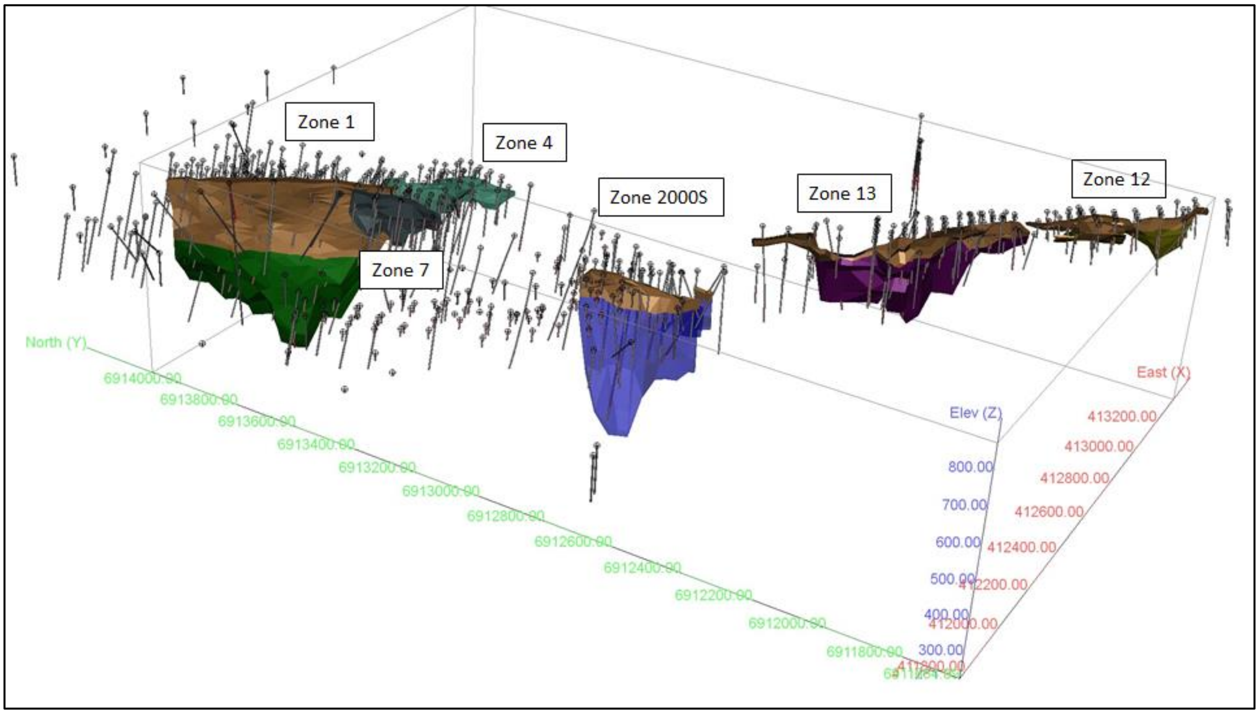


Figure 14-2 Isometric View Looking Northwest: Distribution of Drilling in the Carmacks Deposit Area



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 interpeted 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: Distribution of Drill holes and Carmacks Deposit Grade Controlled Wireframe Models

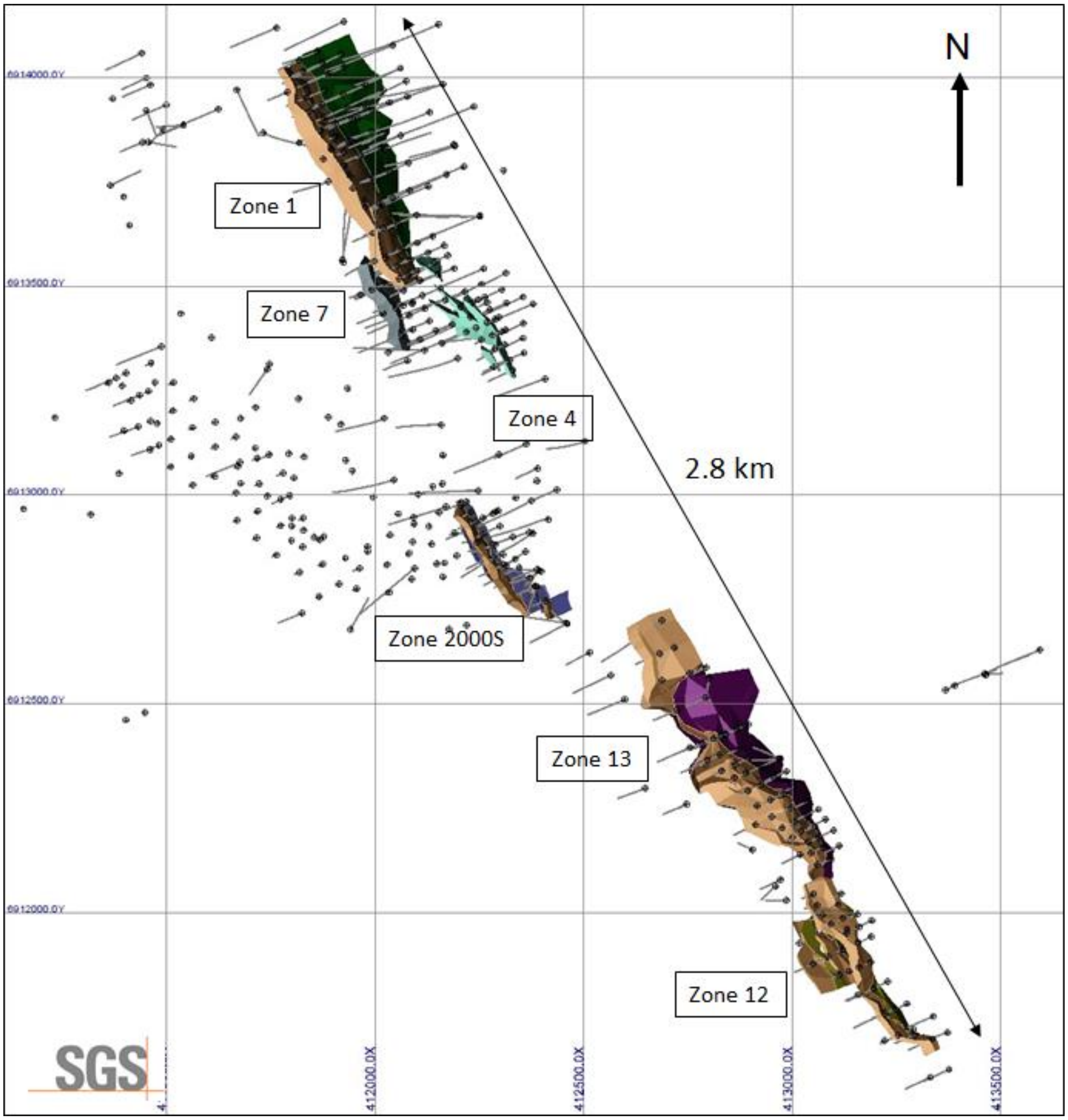


Figure 14-4 Isometric View Looking Northwest: Topographic Surface

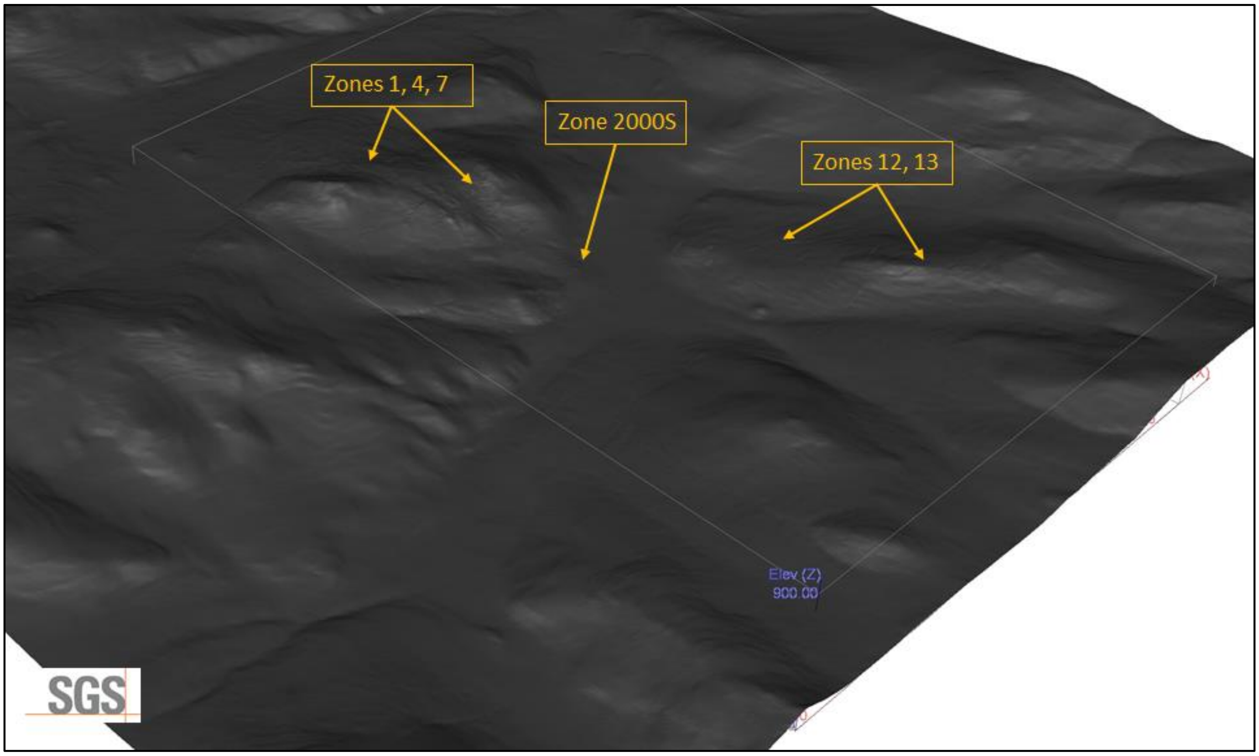


Figure 14-5 Isometric View Looking Northwest: Overburden Surface

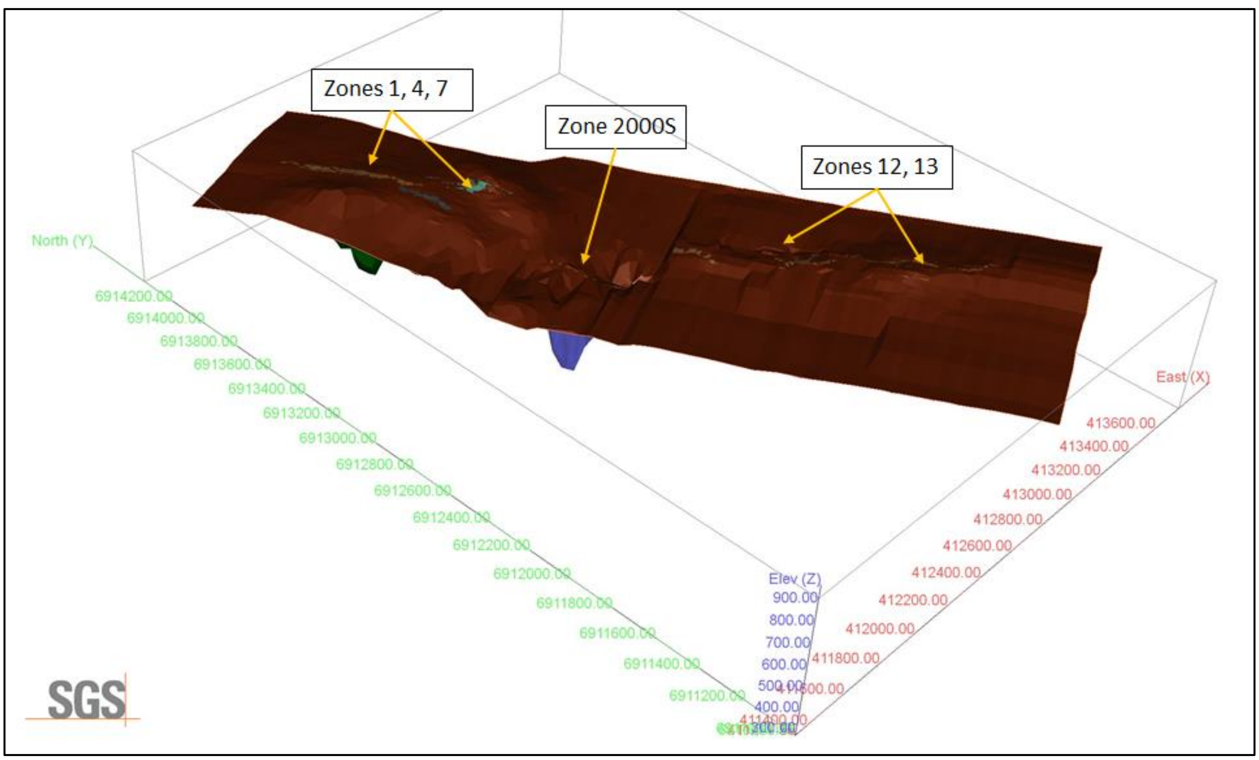


Figure 14-6 Isometric View Looking Northwest: Zones 1, 4, 7 Oxide and Sulphide Zones and Distribution of the Drill holes

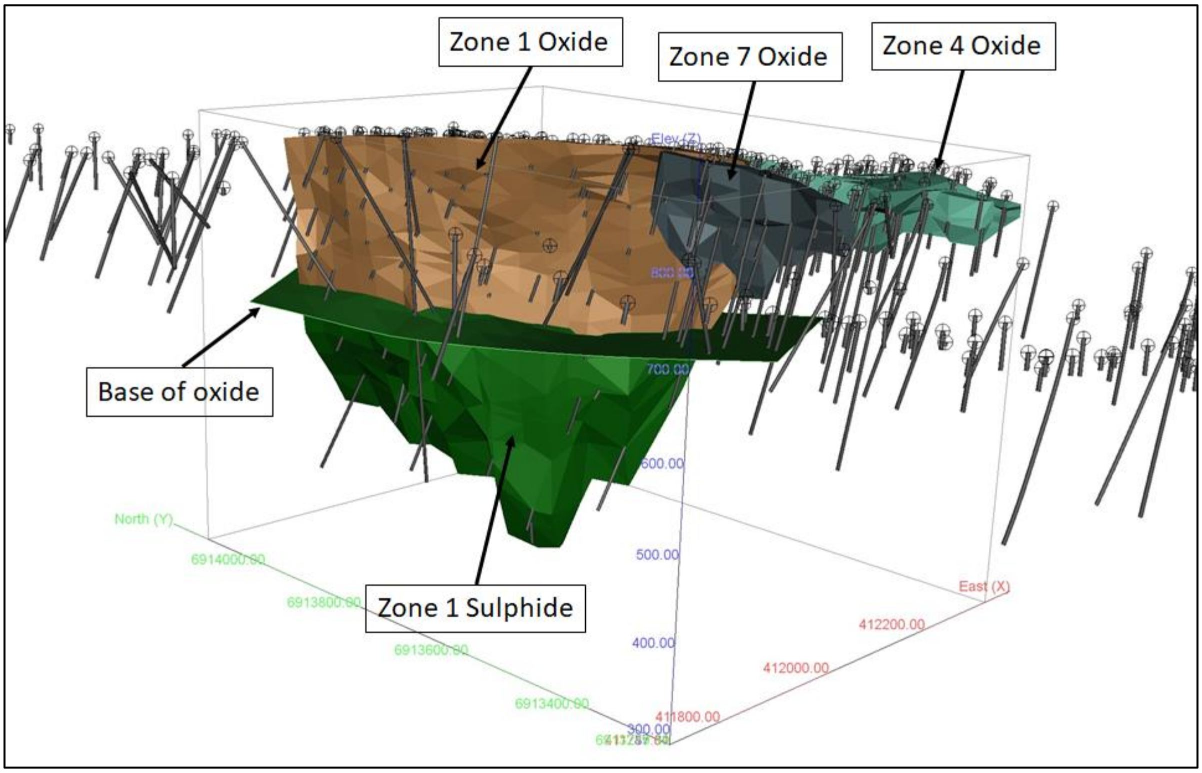


Figure 14-7 Isometric View Looking Northwest: Zone 2000S Oxide and Sulphide Zones and Distribution of the Drill hole

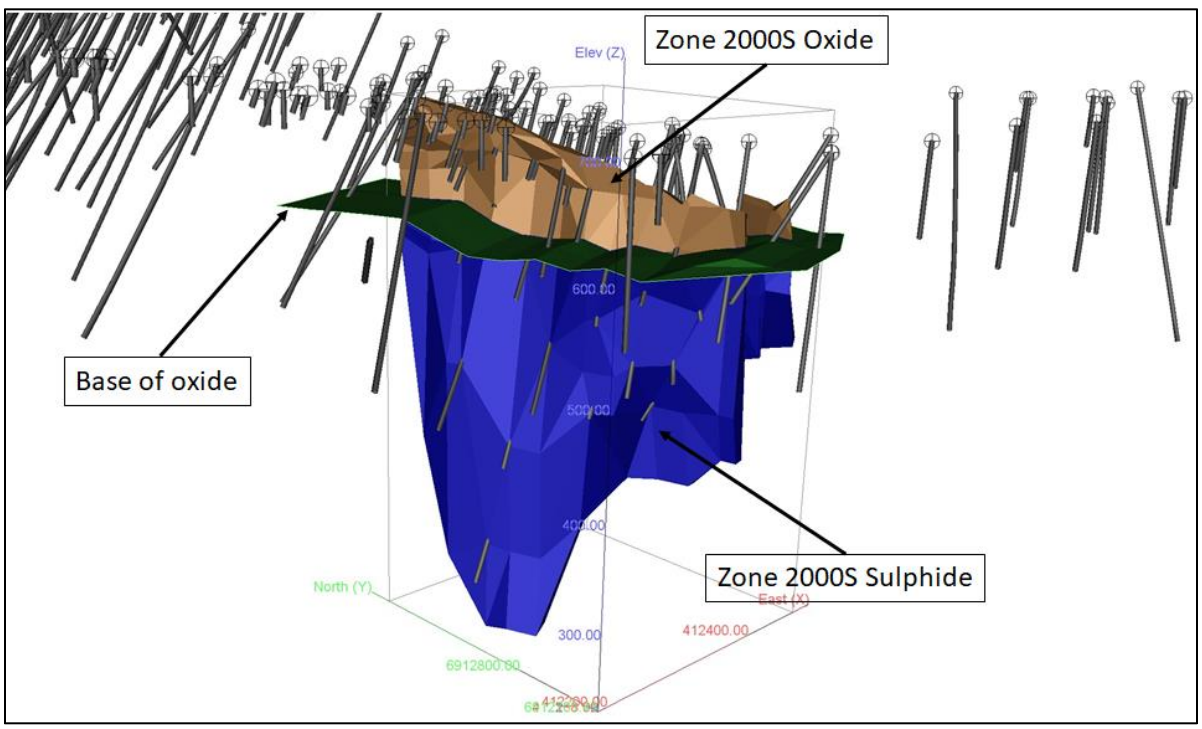
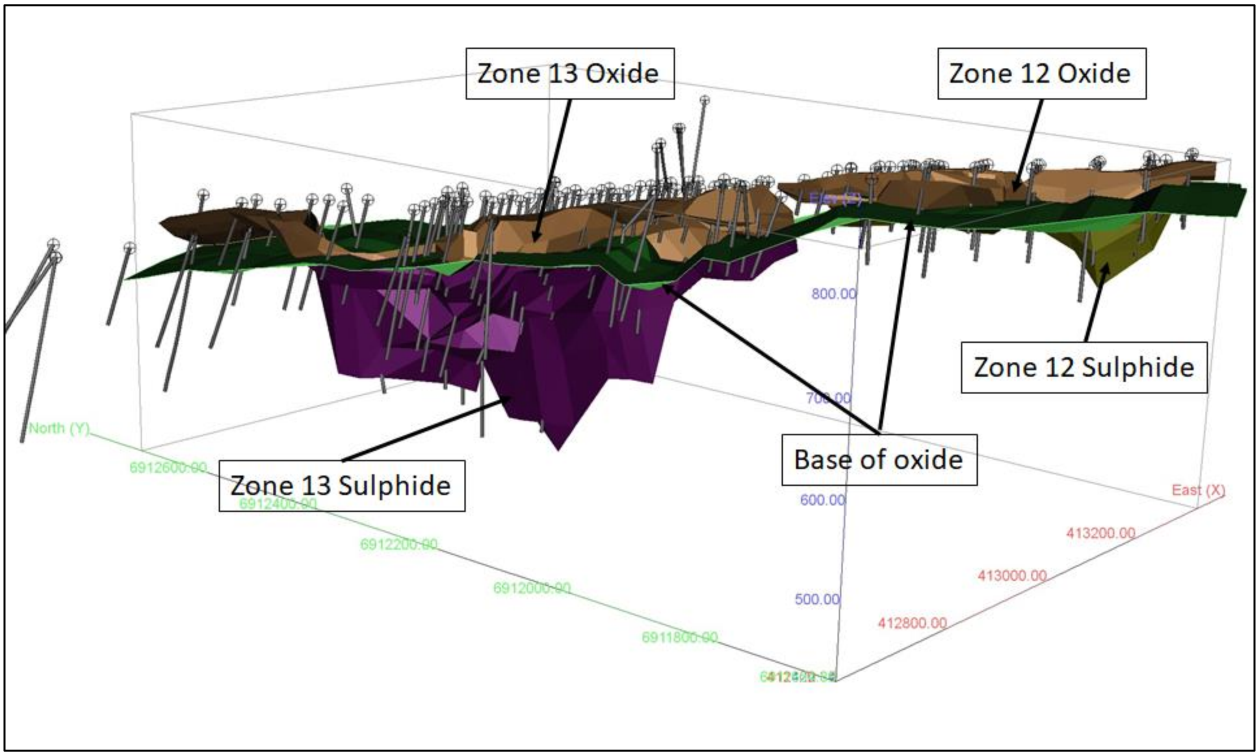


Figure 14-8 Isometric View Looking Northwest: Zones 12, 13 Oxide and Sulphide Zones and Distribution of the Drill holes



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 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 controled 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

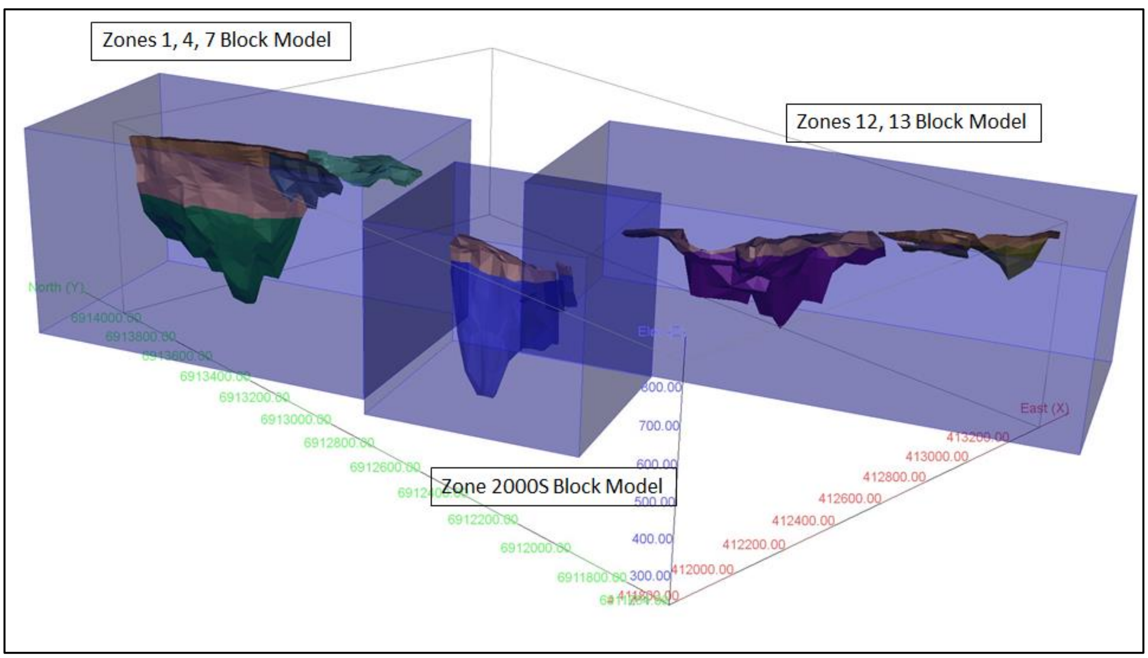
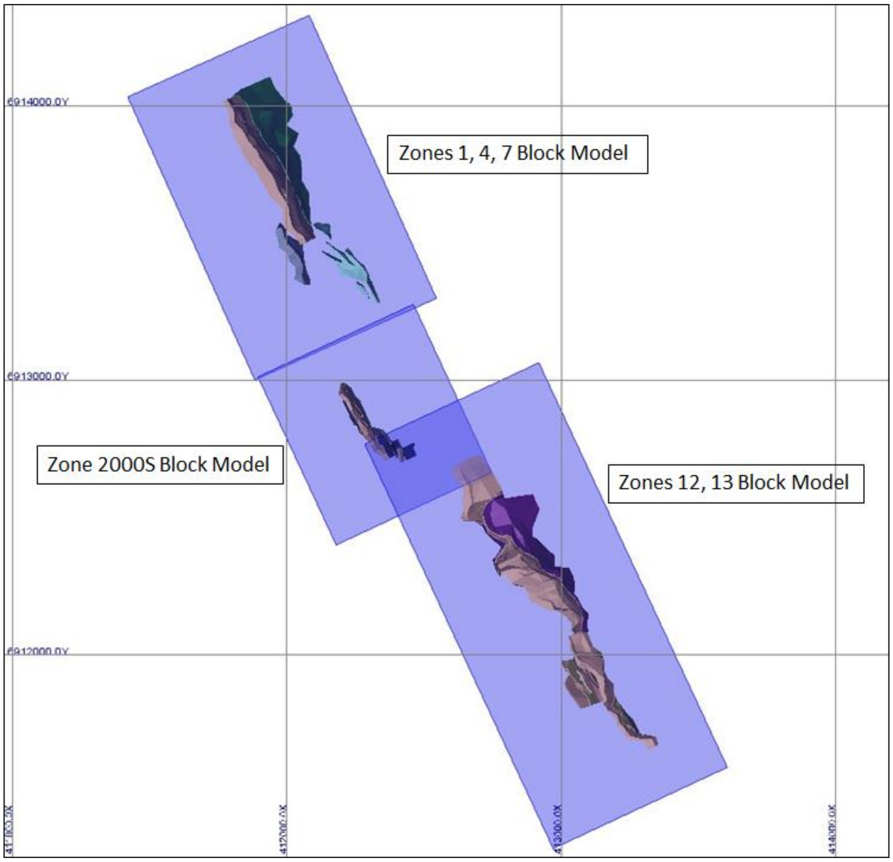


Figure 14-10 Plan View: Carmacks Project Mineral Resource Block Models and Wireframe Grade-Controlled Models



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 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

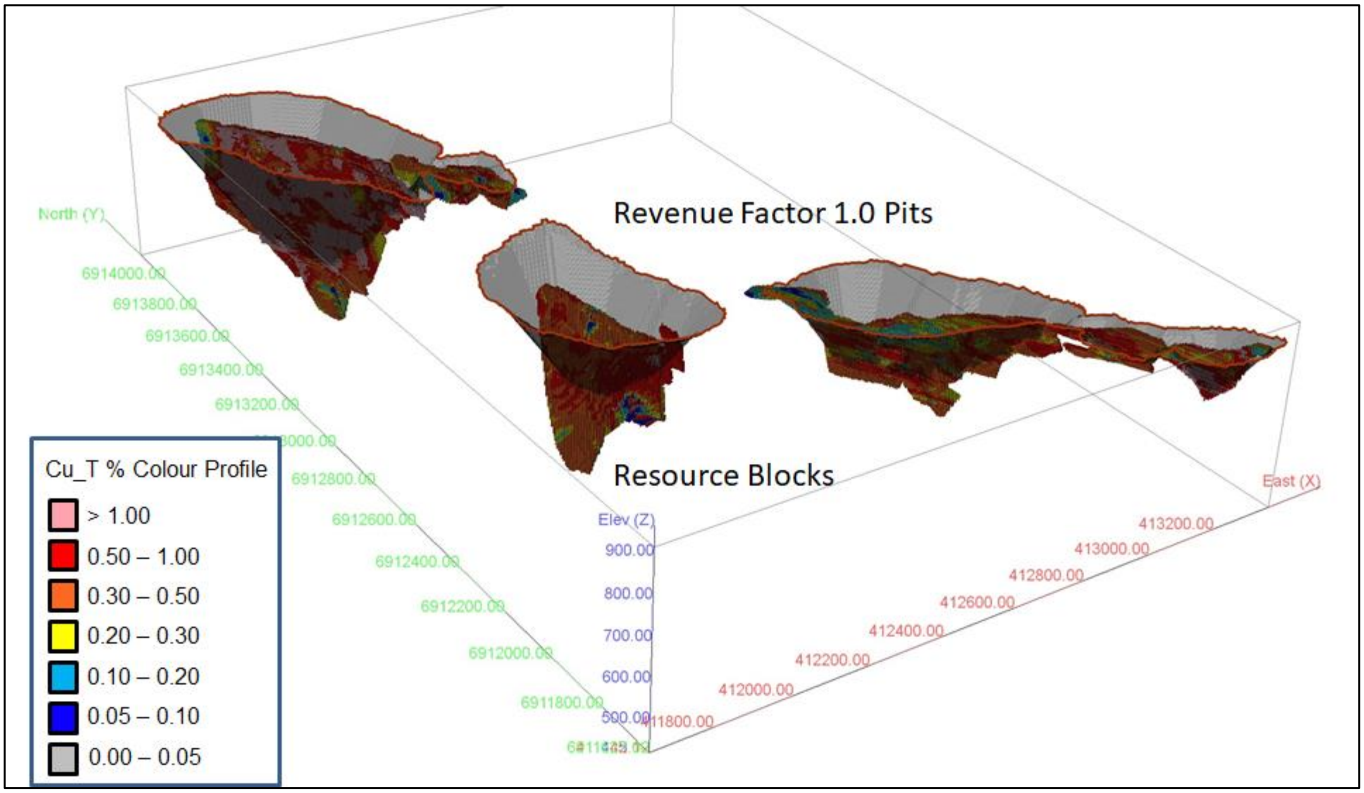


Figure 14-12 Isometric View Looking Northeast of the Zone 1, 4, 7 Deposit Mineral Resource Block Grades and Classification, and Revenue Factor 1.0 Pit

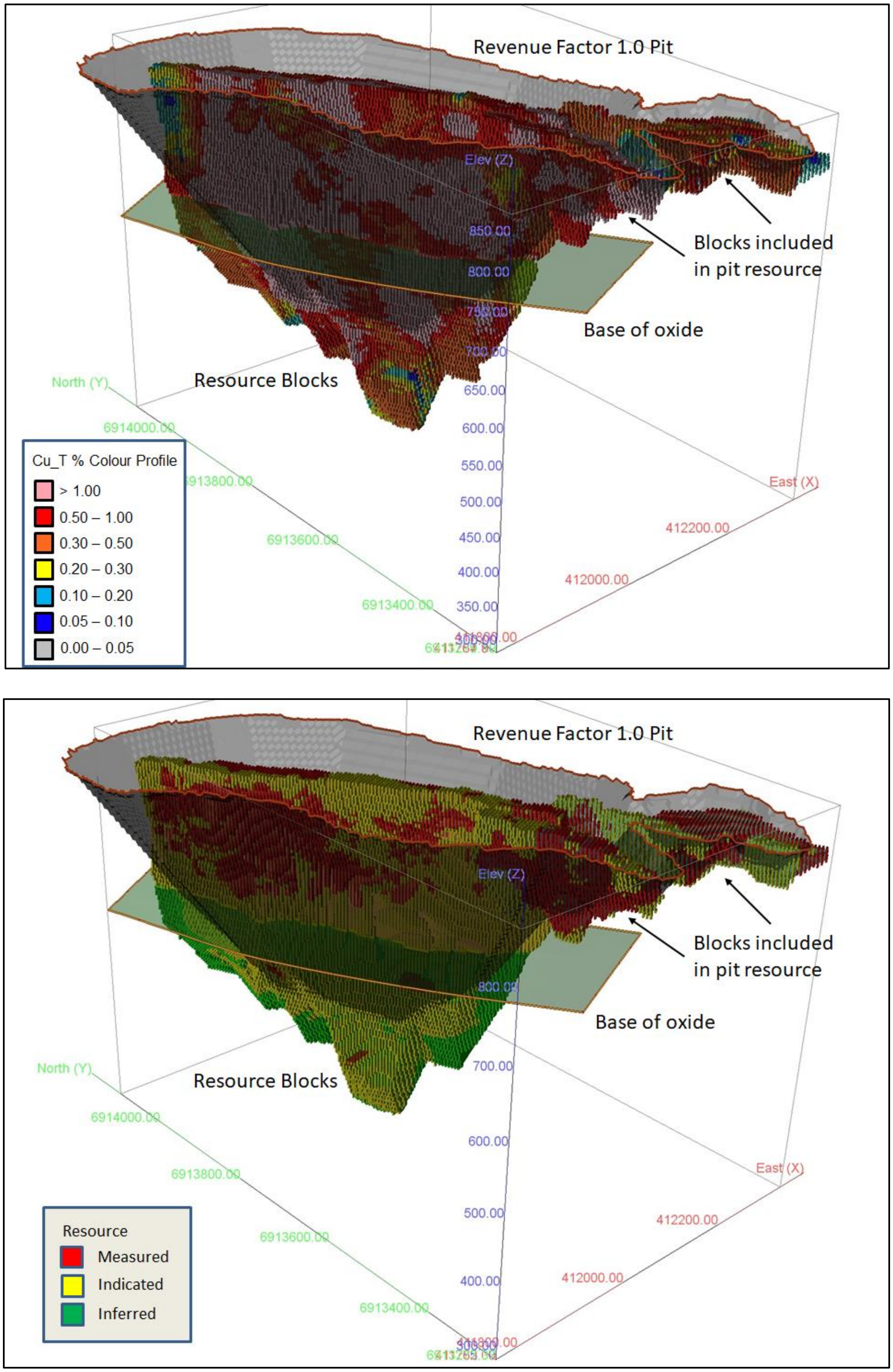
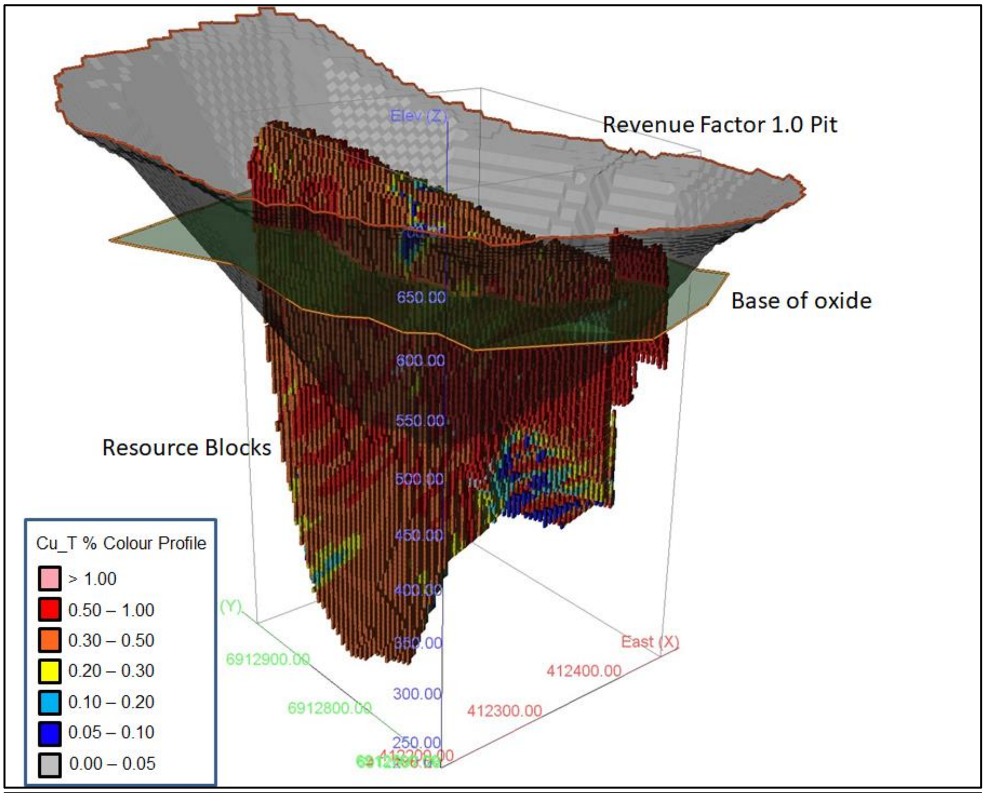


Figure 14-13 Isometric View Looking Northeast of the Zone 2000S Deposit Resource Block Grades and Classification, and Revenue Factor 1.0 Pit



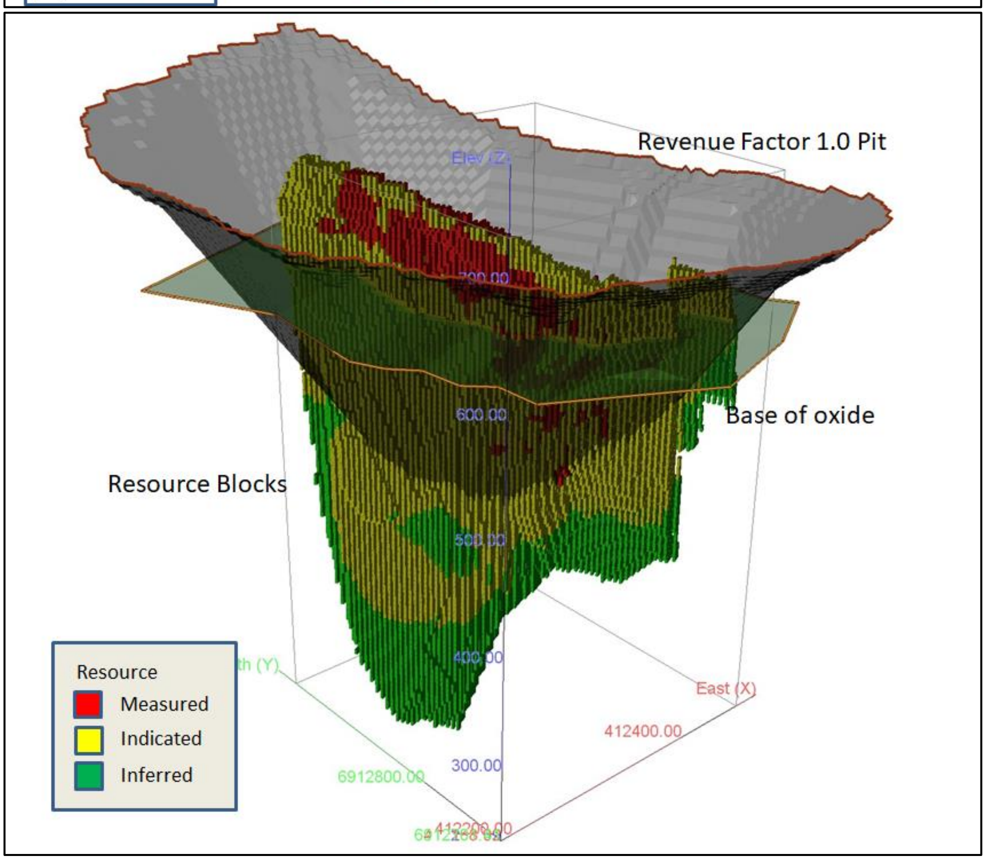
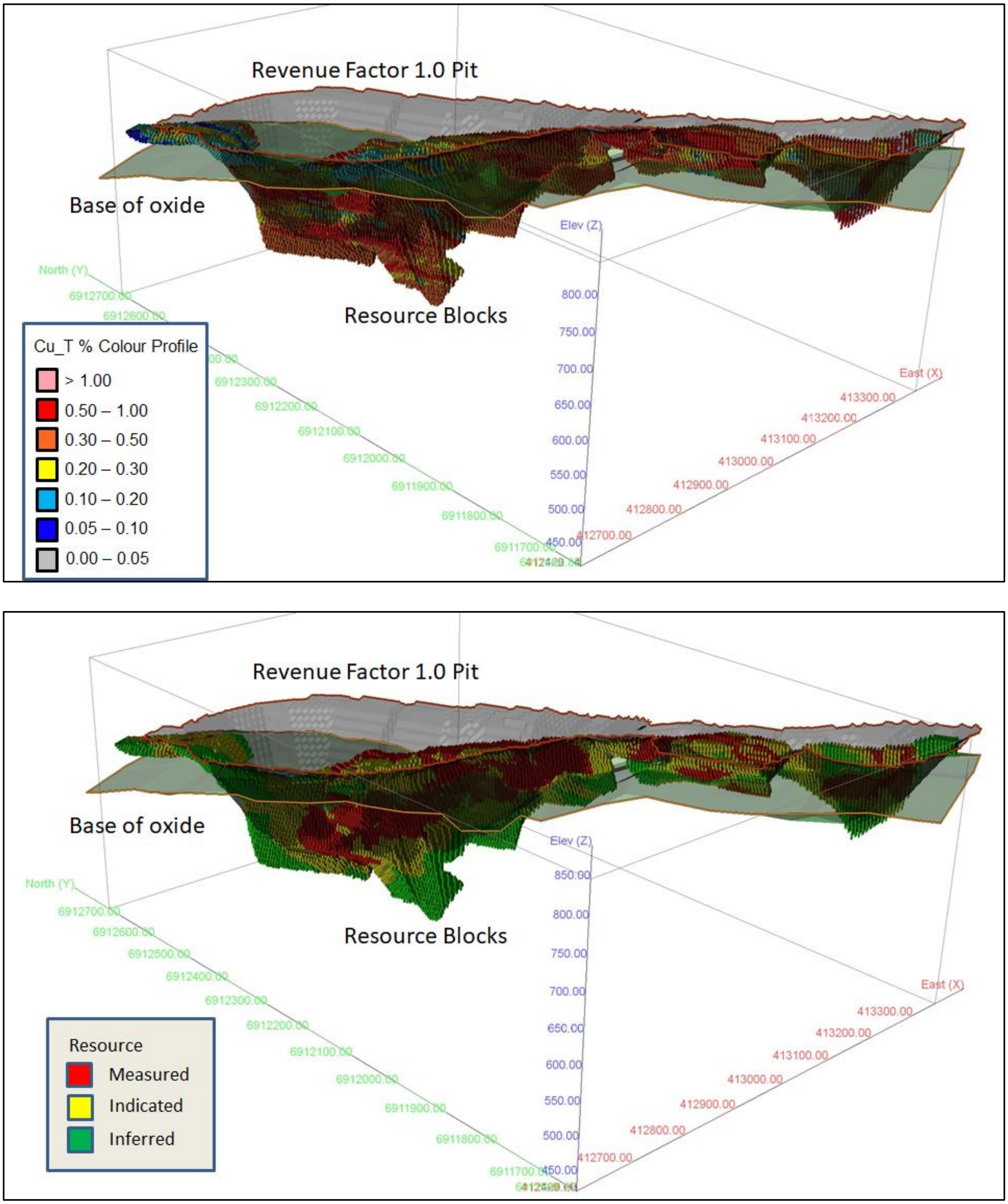


Figure 14-14 Isometric View Looking Northwest of Zones 12 and 13 Deposit Resource Block Grades and Classification, and Revenue Factor 1.0 Pit



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.

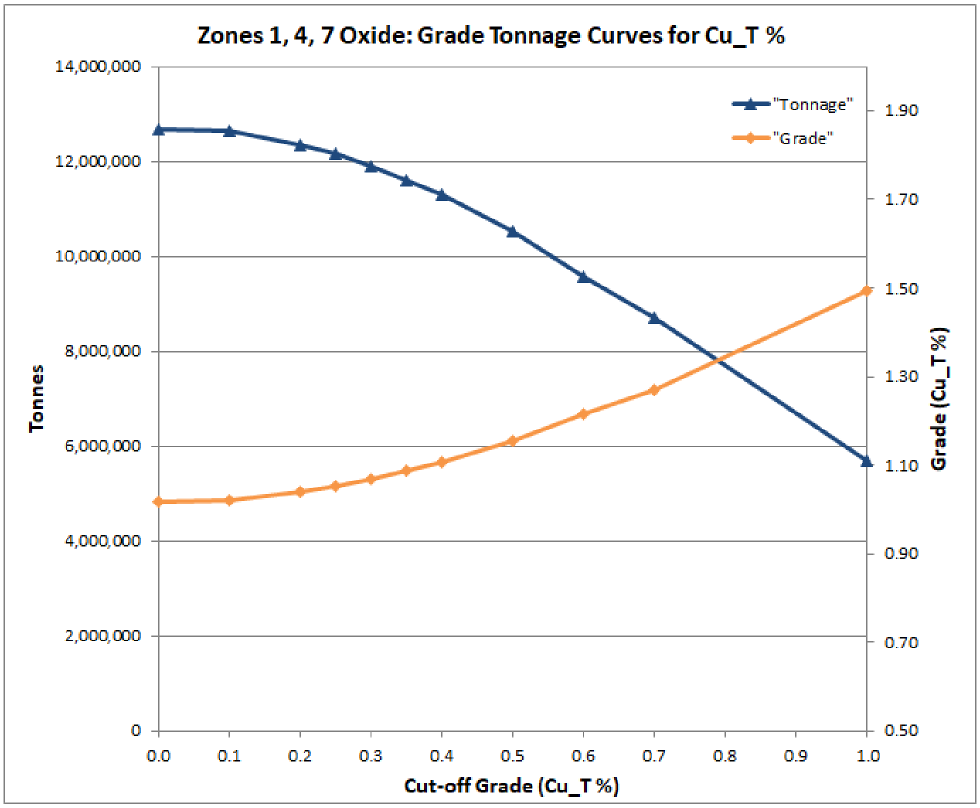
Table 14-11 Comparison of Block Model Volume with Total Volume of the Mineralized Structures

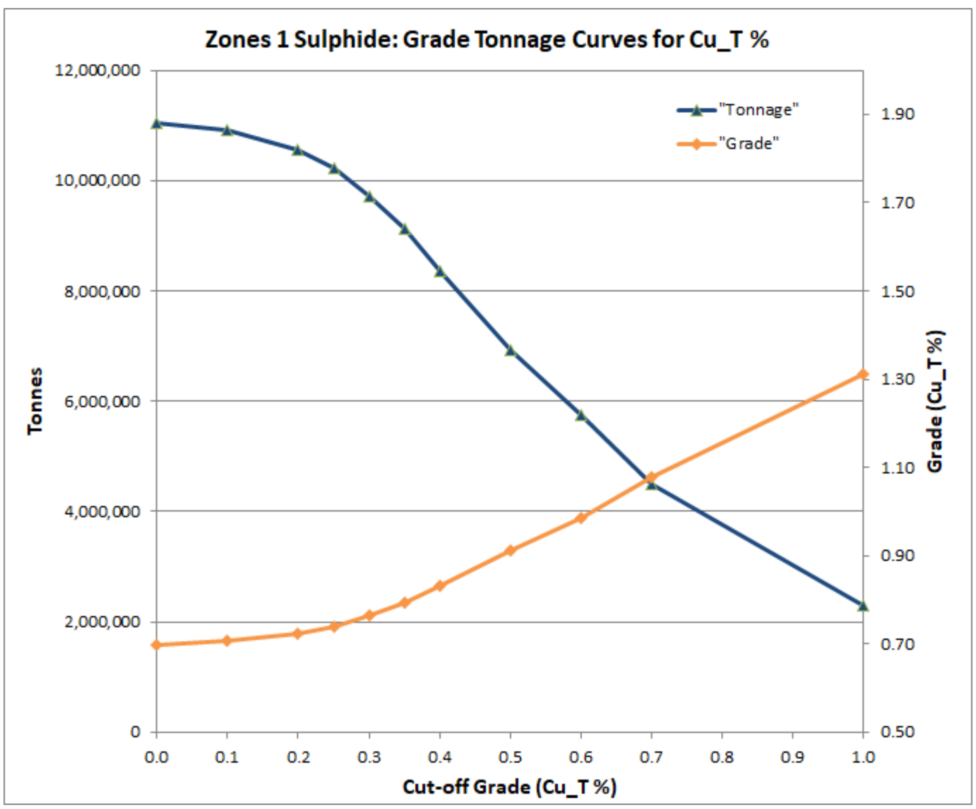
Zone	Wireframe Model Volume	Block Model Volume	Difference %
Zones 1, 4, 7 - Oxide	4,795,374	4,795,707	0.01
Zone 1 - Sulphide	3,989,397	3,971,576	0.45
Zone 2000S - Oxide	380,593	380,955	0.10
Zone 2000S - Sulphide	2,755,798	2,726,921	1.06
Zone 12 Oxide	481,101	480,835	0.06
Zone 12 Sulphide	377,834	370,860	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**	

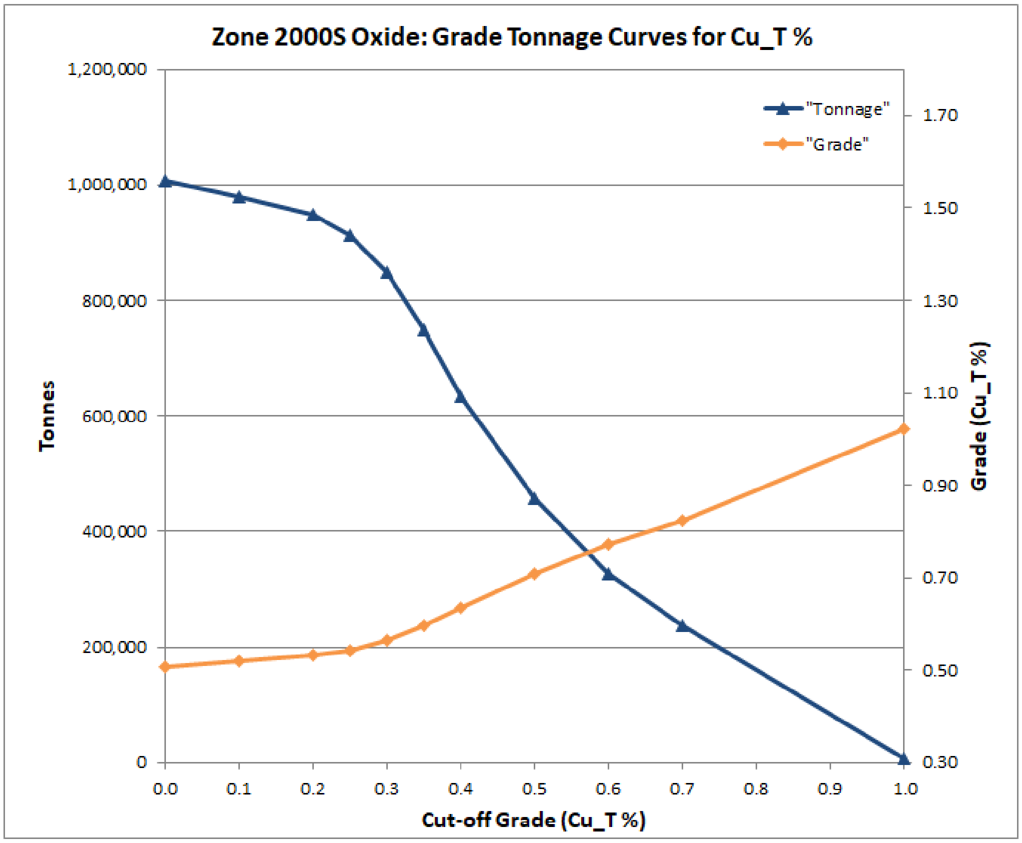
Table 14-12 Comparison of Average Composite Grades with Block Model Grades

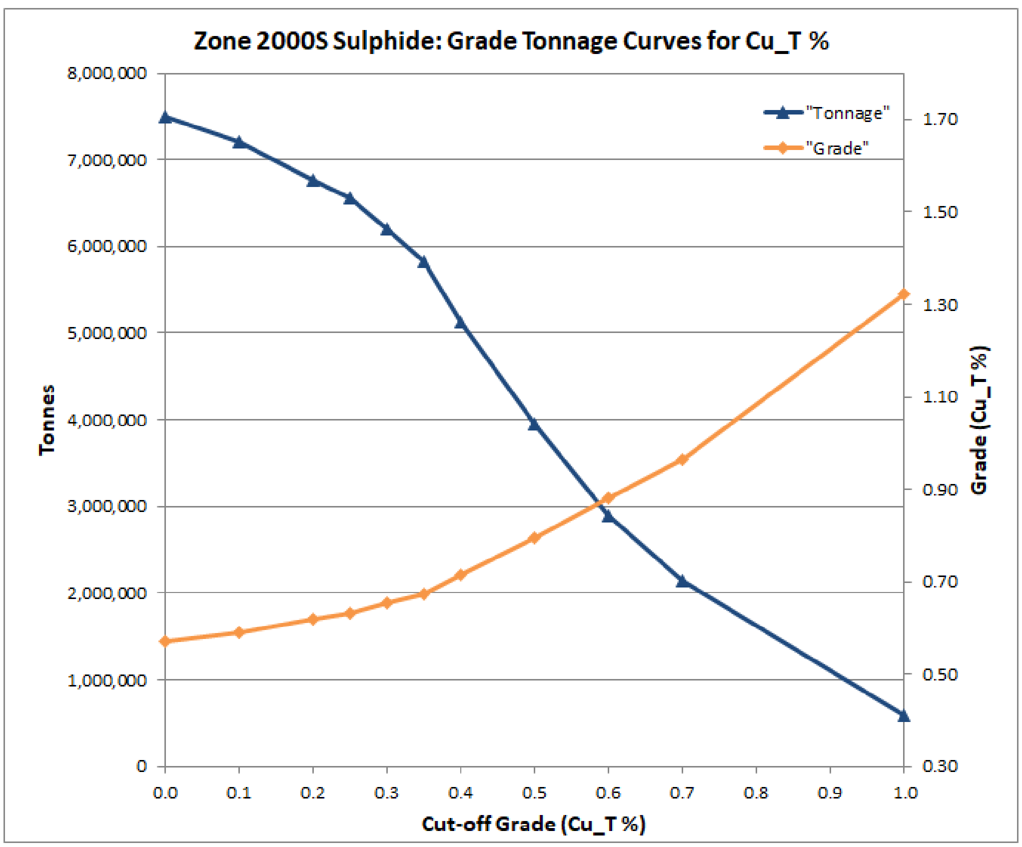
Zone	Composite Average Grade		Block Average Grade	Difference %
Zones 1, 4, 7 - Oxide	Cu_T (%)	1.04	1.02	1.96
	Au (g/t)	0.48	0.42	14.29
	Ag (g/t)	4.82	4.20	14.76
Zone 1 - Sulphide	Cu_T (%)	0.71	0.70	1.43
	Au (g/t)	0.20	0.19	5.26
	Ag (g/t)	2.60	2.40	8.33
Zone 2000S - Oxide	Cu_T (%)	0.46	0.51	-9.80
	Au (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

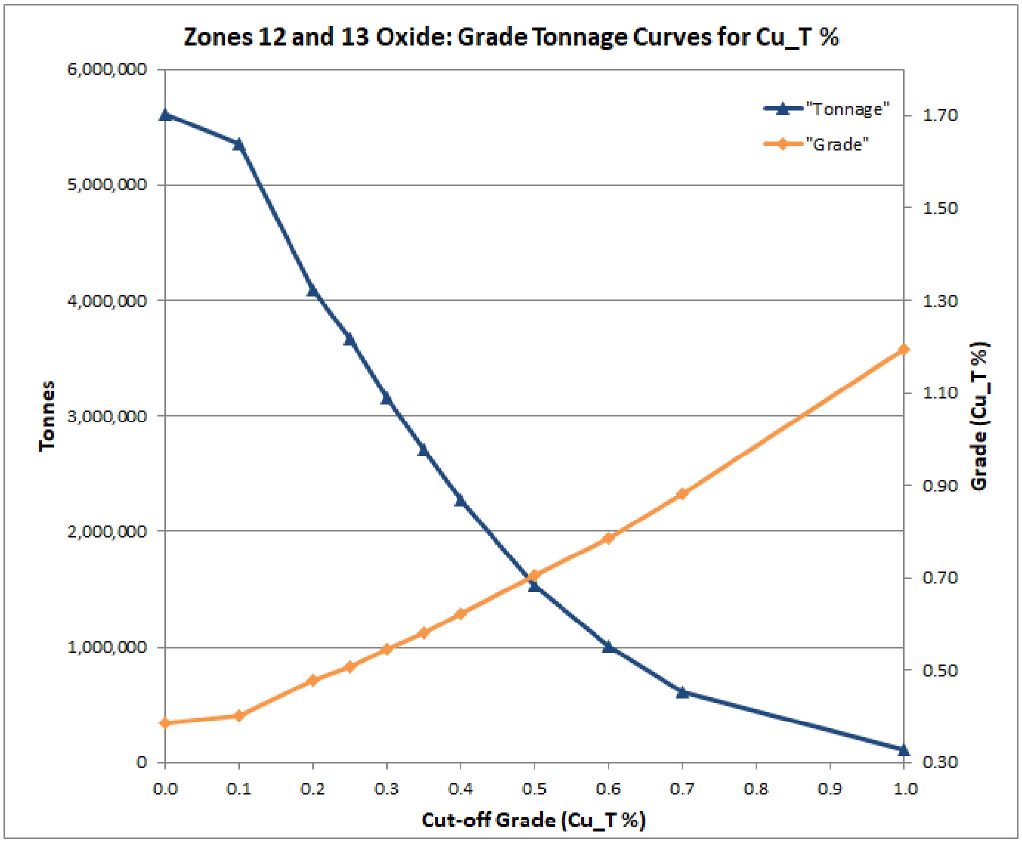
Figure 14-15 Grade Tonnage Plots to show sensitivity to cut-off for Oxide and Sulphide Mineralization

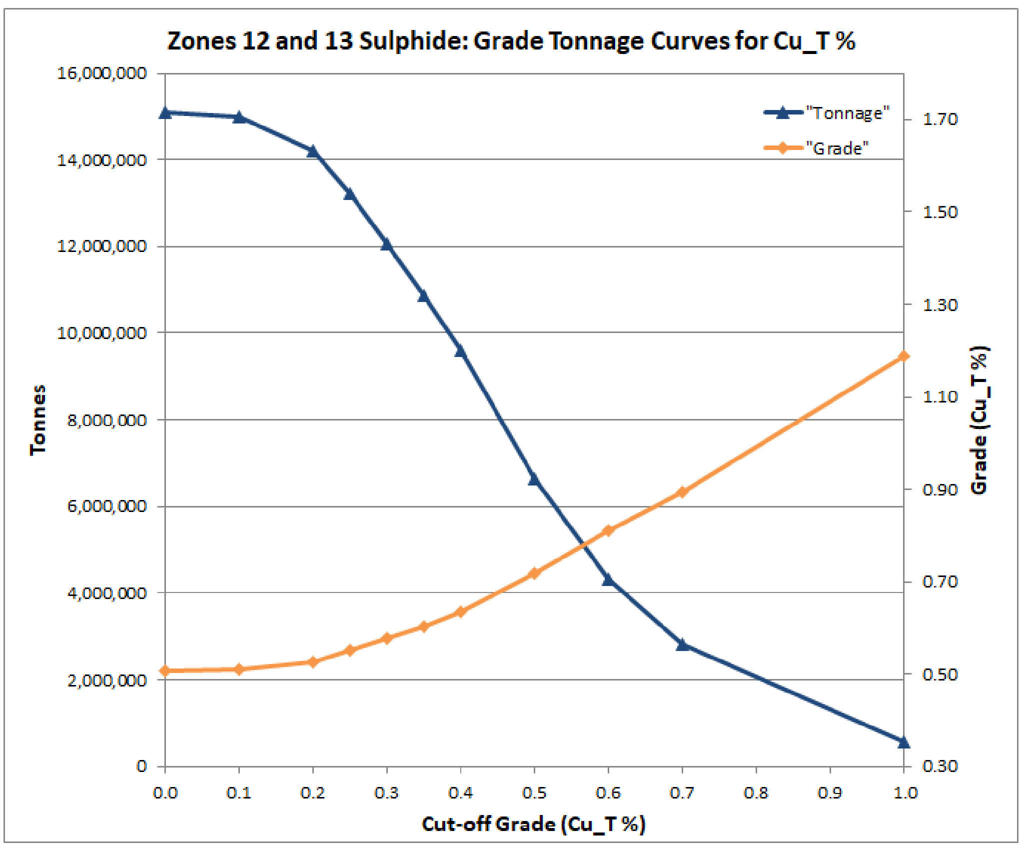





Zone 2000S Oxide: Grade Tonnage Curves for Cu_T %
Tonnes
1,200,000
1,000,000
800,000
600,000
400,000
200,000
0
"Tonnage"
"Grade"
Grade (Cu_T %)
1.70
1.50
1.30
1.10
0.90
0.70
0.50
0.30
0.0
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9
1.0
Cut-off Grade (Cu_T %)


Zone 2000S Sulphide: Grade Tonnage Curves for Cu_T %
Tonnes
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
"Tonnage"
"Grade"
Grade (Cu_T %)
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
Cut-off Grade (Cu_T %)


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 project. This section does not apply to the Technical Report.

16 MINING METHODS

This section does not apply to the Technical Report.

17 RECOVERY METHODS

This section does not apply to the Technical Report.

18 PROJECT INFRASTRUCTURE

This section does not apply to the Technical Report.

19 MARKET STUDIES AND CONTRACTS

This section does not apply to the Technical Report.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This section does not apply to the Technical Report.

21 CAPITAL AND OPERATING COSTS

This section does not apply to the Technical Report.

22 ECONOMIC ANALYSIS

This section does not apply to the Technical Report.

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

All relevant data and information regarding the Carmacks Project has been disclosed under the relevant sections of this report.

25 CONCLUSIONS

SGS was contracted by Granite Creek to complete updated MREs for several copper deposits of their Carmacks Project and to prepare a technical report written in support of the current MREs. The reporting of the MREs comply with all disclosure requirements for Mineral Resources set out in the NI 43-101 Standards of Disclosure for Mineral Projects (2016). The classification of the MREs are consistent with current CIM Definition Standards - For Mineral Resources and Mineral Reserves (2014). Granite Creek acquired 100% of the Carmacks Project in 2020 through its acquisition of Copper North.

The current report also includes information regarding the adjacent Carmacks North Property, formerly the Stu Property. On September 13, 2018 Granite Creek entered into an agreement with an arm's length vendor, under which the Company acquired an undivided 100% interest in the Stu Property, resulting in a consolidation the Carmacks and Carmacks North properties.

The Carmacks and Carmacks North Properties are located in the Minto Copper Belt, an area of well-known copper-gold-silver mineralization in Canada's Yukon Territory. Situated approximately 47 km northeast of the village of Carmacks, and approximately 175 km northwest of Whitehorse, the capital city of the Yukon Territory, the project is within 20 km of grid power and paved highway. The combined projects cover approximately 17,580 hectares (176 square km) and are on trend with the Minto copper mine approximately 35 km north of the center of the project.

Granite Creek is a growth stage exploration company, focused on the acquisition and development of exploration properties that host copper, gold and silver. Granite Creek was originally incorporated on May 10, 2007 under the British Columbia Business Corporations Act. The Company is a reporting issuer and trades on the TSX-V in Canada under the symbol “GCX”, in the United States on the OTC Markets under the symbol “GCXXF” and the Frankfurt Stock Exchange under the symbol “A2PFE0”. The Company’s principal business is the acquisition, exploration and development of mineral properties with the goal of establishing a mineable mineral resource. Granite Creek is a member of the Metallic Group of Companies. Their current business address is Suite 904-409 Granville Street, Vancouver, BC Canada V6C 1T2.

This technical report will be used by Granite Creek in fulfillment of their continuing disclosure requirements under Canadian securities laws, including NI 43-101 – Standards of Disclosure for Mineral Projects. This technical report is written in support of updated resource estimates for several copper deposits on the Carmacks Project released by the Company on March 15, 2022.

The current report is authored by Allan Armitage, Ph.D., P. Geo., of SGS. Armitage is an independent Qualified Person as defined by NI 43-101 and is responsible for all sections of this report.

25.1 Recent Drilling

Since 2014, exploration work on the Carmacks Project has been predominantly drilling. 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 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; 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.

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 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.

The South Gap target zone consists of an approximate 300 metre gap between Zone 1 and the 2000S zone. The 2000S zone was intersected by 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.

The 2015 fill-in drilling program confirmed continuity of both oxide and sulphide mineralization in Zones 2000S, 12, and 13, covering a strike length of 2,000 metres. 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. 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

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 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. A total of 36 holes were completed for 4,175 m.

Seven holes were drilled within 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 successfully confirmed the presence of visible malachite, azurite, and tenorite 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.

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)

The 2017 step-out drilling 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 into a geological model of each zone and included in the updated 2018 resource estimate.

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:

- 127 metres of continuous copper mineralization, in drillhole CRM20-001, grading 0.85% copper equivalent ("CuEq") including 28.65m of 1.74% CuEq and 19.20m 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 me 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.

Between May and September, 2021, Granite Creek completed 7,742 m of diamond drilling in 23 holes on the Carmacks Propert. Highlights of the 2021 drill program are presented in Table 25-1. 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.

Table 25-1 2021 Drill Results

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	
CRM21-022	233.7	302	68.3	0.51	0.009	0.13	2.3	0.66	
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	

Drillhole	From (m)	To (m)	Length* (m)	Cu (%)	Mo (%)	Au (g/t)	Ag (g/t)	CuEq** (%)	Target
CRM21-015	36.69	49.38	12.69	0.23	0.003	0.04	0.96	0.27	Zone 13
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	
CRM21-024	54.8	93	38.2	0.79	0.005	0.16	3.27	0.95	
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	

25.2 2022 Carmacks Project Mineral Resource Statement

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. 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 has been reviewed and verified by Author as being accurate to the extent possible and to the extent possible all geologic information was reviewed and confirmed. 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 25-2. 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 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 25-2. 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 25-2 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 25-3 and Table 25-4.

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 - 36.2 million tonnes, grading 0.81% Cu, 0.26 g/t Au, 3.23 g/t Ag and 0.01% Mo**

Table 25-3 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	0.70	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 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 25-4 Carmacks Project Mineral Resource Estimates, February 25, 2022: Distribution of Cu_X and Cu_S with respect to Cu_T

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.

25.3 **Recent Metallurgical Testwork**

The metallurgical testing program on the Carmacks Project focused on the recovery of acid soluble copper mineralization in the oxide cap of the Zone 1 deposit. The primary emphasis of the work conducted up to 2012, has been on development of design criteria and optimal operating parameters for heap leaching the crushed and agglomerated mineralized material, followed by solvent extraction for solution concentration and purification and electrowinning for recovery of cathode copper metal. Some limited testing has been performed on heap leaching using run of mine (ROM) mineralized material, examining leaching of the sulphide mineralization, and recovering gold following copper recovery.

The recovery of gold and silver from copper leach residues was examined as part of the previous testwork but was not reported. Specifically, the copper leach metallurgical work undertaken in 2009 by PRA Metallurgical Division of Inspectorate America Corporation included preliminary testing of gold and silver leaching from copper leach residues by cyanidation. The testwork was conducted in two columns, with the results indicating a gold recovery of 78% and a silver recovery of 75%.

The testwork also indicated that a portion of the copper remaining after the copper leach would be extracted during gold/silver cyanidation and that this dissolved copper could be recovered in the SART circuit associated with the gold and silver ADR plant, with concurrent regeneration of cyanide for recirculation to the process. The cyanide consumption as NaCN has been estimated at 0.5 kg/tonne.

In 2014, Copper North examined the value of adding precious metals recovery to the project plan using a two-stage heap leaching approach. Results were reported in the PEA prepared by Merit International Consultants. This study indicated the value of recovering gold and silver, leading to further metallurgical test work.

During 2014-2015, Bureau Veritas Commodities Canada (BV Minerals) completed a full suite of metallurgical testing to evaluate an alternative to heap leaching. This new vat leach recovery method focused on grinding the samples to a P_{80} of 664 µm, and leaching with sulphuric acid to recover the copper. The leach residue was then neutralized and leached with cyanide to recover the gold and silver. Metallurgical testing included flowsheet parameter finalization, a full locked cycle Cu/Au leach test, cyanide destruction, and variability comminution and batch leach testing.

Thirteen different composites were used in the 2014 test program. In Phase 1, four trench composites (BS-1, BS-2, BS-3 and BS-4) were created and a Master Composite was constructed from core samples. In Phase 2, eight variability composites were created from the remaining core samples kept in storage. In February 2016, BV Minerals conducted additional copper leach optimization test work using 2014 Master Composite.

Based on the test work, a copper/gold leach circuit was selected as the preferred recovery method. The criteria and recoveries from CALT2 were selected for design due to the low copper grade reporting to the gold/silver leach circuit, eliminating the need for a sulphidization, acidification, recycling and thickening (SART) process. Mineralized material will be reduced to a P_{80} of 664 µm using a jaw crusher followed by a SAG mill in closed circuit with hydrocyclones. Copper will be recovered using a sulphuric acid leach and solvent extraction / electrowinning (SX-EW). Normally high silver grades and low gold grades dictate the use of Merrill Crowe; however ferric sulphate addition substantially reduces silver recovery, allowing for a smaller footprint with a CIL circuit. Copper leach residue will be neutralized and gold/silver will be leached into solution using cyanide while simultaneously being adsorbed onto activated carbon. An adsorption, desorption and refining (ADR) circuit will be implemented to concentrate the gold/silver into doré bars. The resulting tailings residue is then passed through an Inco SO_2-Air circuit for destruction of the residual cyanide, followed by filtration and placement in the TMA as dry stack tailings.

In 2021, metallurgy testing was completed on sulphide and oxide mineralized core samples from the Carmacks Project. Two representative samples of copper sulphide material and one sample of copper oxide material were delivered to Bureau Veritas Commodities, Metallurgy Division, for rougher flotation kinetic testing and open cleaner flotation testing. The purpose of the testing was to determine how amenable the sulphide mineralization present at the Carmacks deposit was to concentration by flotation, what recoveries could be expected, and to lay the groundwork for further testing. A preliminary copper flotation recovery model was generated at a fixed 25% copper concentrate grade with test results as summarized below:

- Recoveries of greater than 95% for copper into a 25% copper concentrate are possible.
- Copper sulphide minerals are well-liberated for rougher recovery via flotation at P_{80} of 150 µm.
- A secondary regrind size at P_{80} of 25 µm can achieve 25% copper grade and high cleaner stage recovery.
- Gold is associated primarily with copper sulphide minerals and minor pyrite. Flotation of gold with copper concentrate is likely the most economical way to recover gold.
- High chalcopyrite content as the copper mineral and low pyrite content within the samples indicate a simple reagent scheme and relatively easy copper flotation upgrade.

The preliminary tests indicate that well-established flotation methods with known reagents will likely be the preferred processing method for sulphide material at Carmacks. The next stage of metallurgical test work will involve greater variability of samples to validate copper and gold recoveries as well as assessing potential levels of silver and molybdenum recoveries.

25.4 **Risk and Opportunities**

The following risks and opportunities were identified that could affect the future economic outcome of the project. The following does not include external risks that apply to all exploration and development projects (e.g., changes in metal prices, exchange rates, availability of investment capital, change in government regulations, etc.).

There is no other relevant data or information available that is necessary to make the technical report understandable and not misleading. To the Authors knowledge, there are no additional risks or uncertainties that could reasonably be expected to affect the reliability or confidence in the exploration information or mineral resource estimate.

25.4.1 Risks

25.4.1.1 Mineral Resource Estimate

In the Authors opinion the risks to the current mineral resources is limited. Roughly 7.3 % of the MRE of the Carmacks Project, at the reported cut-off grades, are in the Inferred Mineral Resource classification, which is that part of the mineral resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. However, it is reasonably expected that the majority of Inferred Mineral resources could be upgraded to Indicated Minerals Resources with continued exploration.

The mineralized structures (mineralized domains) in all zones are relatively well understood and the mineralization grade distribution in each domain is generally consistant. However, all mineralization zones might be of slightly variable shapes from what have been modeled. A different interpretation from the current mineralization models may adversely affect the current MREs. Continued drilling will help define with more precision the shapes of the zones and confirm the geological and grade continuities of all mineralized zones.

25.4.2 Opportunities

25.4.2.1 Mineral Resource Estimate

There is an opportunity on all deposits to extend known mineralization at depth, on strike and elsewhere on the Carmacks Property and to potentially convert Inferred Mineral Resources to Indicated Mineral Resources. Granite Creeks's intentions are to direct their exploration efforts towards resource growth in 2022 with a focus on extending the limits of known mineralization and testing other targets on the greater Carmacks Property.

26 RECOMMENDATIONS

The copper deposits of the Carmacks Project contain within-pit and underground Measured, Indicated and Inferred Mineral Resources that are associated with well-defined mineralized trends and models. All deposits are open along strike and at depth. The Author considers that the Project has potential for delineation of additional oxide and sulphide Mineral Resources along strike, down dip and elsewhere on the Carmacks Property and that further exploration is warranted.

Granite Creek is planning on continuing exploration on the Carmacks Property in 2022. The exploration program is to include geophysics, trenching, soil sampling and diamond drilling (~ 7,500 m) in several areas, as well as re-logging of old core and reclamation and baseline environmental studies. The total cost of the recommended work program is estimated at C$3,533,500 million (Table 26-1Table 26-1 Recommended 2022 Work Program for the Carmacks Project).

The 2022 exploration program at the Carmacks Property will be in completed in two phases. The first phase centres on completion of a PEA study and report, in conjunction with field work including IP geophysics. While the camp is operating to support the geophysics survey, activities such as trenching, soil sampling, re-logging core, sampling for additional metallurgical testwork, reclamation and baseline environmental studies will be conducted.

The budget for the second phase is preliminary and based on an all-in cost of $325 per metre of diamond drilling. This dollar amount is based on previous road-based diamond drill programs at Carmacks and does not include other exploration activities. Targetting of the Phase 2 drill program is contingent on the results of the Phase 1 program.

Given the prospective nature of the Carmacks Property, it is the Author's opinion that the Carmacks Property merits further exploration and that the proposed 2022 plan for further work by Granite Creek is justified. A proposed work program by Granite will help advance the deposits and will continue to provide key inputs required to further evaluate the economic viability of the Carmacks Project.

The Author is recommending Granite Creek conduct further exploration, subject to funding and any other matters which may cause the proposed exploration program to be altered in the normal course of its business activities or alterations which may affect the program as a result of exploration activities themselves.

Table 26-1 Recommended 2022 Work Program for the Carmacks Project

Phase 1 – June start - 30-35 days

Item	Cost	Details
Camp costs	$52,000	Camp rental, cook, expediting, groceries
Heavy Equipment	$50,000	Roadwork, trenching, reclamation
Geophysics Survey	$210,000	20 line km IP survey
Geological	$160,000	Geologists, geotechs, project manager, includes office work pre and post field
PEA report	$300,000	
Metallurgical testwork	$100,000	
Lidar survey	$73,000	
Environmental	$50,000	Baseline, gap analysis
Sample analysis	$20,000	Core, rock, soil, check samples
Equipment & communications	$17,000	Vehicles, satellite communications
Fuel	$22,000	Diesel, gas, JetA, propane
Transport and travel	$35,000	Airline, helicopter,
Lands and permitting	$2000	
Community consultation	$5000	Event in Carmacks, site tour
Total	**$1,096,000.00**	

Phase 2 – 7,500m of diamond drilling

Item	Cost	Details
7,500 m diamond drilling	$2,437,500.00	All in cost for drilling $325/m. Approx 5,000m near deposit expansion drilling, 1000 m geotechnical drilling and 1500m exploration drilling Inclusive of sampling cost, assaying, logging, geotechnical, drill management, core storage, travel accommodation, logging facilities, consumables, and data reporting
Total	**$2,437,500.00**	

Total – both programs	**$3,533,500.00**	

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28 DATE AND SIGNATURE PAGE

This report titled “Technical Report on the Updated Mineral Resource Estimates for the Carmacks Cu-Au-Ag Project near Carmacks, Yukon, Canada” dated April 29, 2022 (the “Technical Report”) prepared for Granite Creek Copper Ltd. was prepared and signed by the following authors:

The effective date of the report is Febuuary 25, 2022.
The date of the report is April 29, 2022.

Signed by:

Qualified Persons	Company
Allan Armitage, Ph. D., P. Geo.	SGS Geological Services (“SGS”)
April 29, 2022	

29 CERTIFICATES OF QUALIFIED PERSONS

QP CERTIFICATE – ALLAN ARMITAGE

To Accompany the Report titled "Technical Report on the Updated Mineral Resource Estimates for the Carmacks Cu-Au-Ag Project near Carmacks, Yukon, Canada" dated April 29, 2022 (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:

1. 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).

2. 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.

3. 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.

4. 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.

5. I am a member of the Association of Professional Engineers, Geologists and Geophysicists of Alberta and use the title of Professional Geologist (P.Geol.) (License No. 64456; 1999), I am a member of the Association of Professional Engineers and Geoscientists of British Columbia and use the designation (P.Geo.) (Licence No. 38144; 2012), and I am a member of Professional Geoscientists Ontario (PGO) and use the designation (P.Geo.) (Licence No. 2829; 2017), I am a member of the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (NAPEG) and use the designation (P.Geo.) (Licence No. L4375, 2019),

6. 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".

7. I am the author of this report and responsible for all sections. I have reviewed all sections and accept professional responsibility for all sections of this technical report.

8. I conducted a site visit to the Carmacks Property on November 9, 2021.

9. I have had no prior involvement in the Carmacks Property.

10. I am independent of Granite Creek Copper Ltd. and the Carmacks Property as defined by Section 1.5 of NI 43-101.

11. 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.

12. 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 29th day of April, 2022 at Fredericton, New Brunswick.

"Original Signed and Sealed"

Allan Armitage, Ph. D., P. Geo., SGS Geological Services.

APPENDIX A

Drilling Collar Coordinates, Azimuth, Dip, and Hole Depth

1970 – 2021 Carmacks Project

HOLE-ID	LOCATIONX	LOCATIONY	LOCATIONZ	LENGTH	AZIMUTH	DIP	YEAR
DDH-1-01	412049.35	6913448.08	857.05	22.86	246.00	-45.00	1970
DDH-1-02	412018.39	6913435.96	855.41	8.53	66.00	-45.00	1970
DDH-1-03	412090.89	6913461.94	855.16	125.90	253.00	-50.00	1971
DDH-1-04	412018.22	6913694.74	866.32	85.04	246.00	-50.00	1971
DDH-1-05	411945.25	6913903.17	853.40	153.32	246.00	-50.00	1971
DDH-1-06	412127.71	6913740.43	837.30	234.70	247.00	-52.00	1971
DDH-1-07	412078.11	6913954.78	836.09	349.91	248.00	-55.00	1971
DDH-1-08	411763.89	6914119.76	838.95	185.62	247.00	-50.00	1971
DDH-1-09	412157.12	6913496.29	848.51	240.49	247.00	-50.00	1971
DDH-1-10	412015.43	6913833.30	848.38	205.13	247.00	-55.00	1971
DDH-1-11	412236.78	6913931.17	808.90	505.36	248.00	-55.00	1971
DDH-1-12	411926.25	6914026.65	845.41	258.47	247.00	-55.00	1971
DDH-1-13	412042.27	6914078.22	830.97	345.95	248.00	-55.00	1971
DDH-1-14	412152.39	6914128.19	813.03	544.98	246.00	-55.00	1971
DDH-1-15	412212.87	6913787.74	814.95	416.97	247.00	-55.00	1971
DDH-1-16	412138.49	6913620.36	850.26	262.74	248.00	-55.00	1971
DDH-1-17	412251.51	6913669.72	822.90	365.15	247.00	-55.00	1971
DDH-5-01	411541.34	6913887.03	864.62	115.82	247.00	-50.00	1971
DDH-5-02	411461.98	6913983.22	856.73	115.52	247.00	-50.00	1971
DDH-6-01	411487.87	6913356.14	836.24	182.88	248.00	-50.00	1971
DDH-1-18A	411883.61	6914308.03	811.78	243.84	247.00	-50.00	1972
DDH-13-01	412828.36	6912420.38	754.36	126.49	247.00	-50.00	1972
DDH-13-02	412724.88	6912484.65	752.50	99.06	247.00	-50.00	1972
DDH-4-01	412260.62	6913542.79	839.64	187.45	247.00	-50.00	1972
DDH-4-02	412378.99	6913458.17	833.88	265.18	247.00	-50.00	1972
DDH-4-03	412160.41	6913364.06	828.34	112.78	67.00	-50.00	1972
DDH-4-04	412289.00	6913422.00	850.00	426.73	247.00	-50.00	1972
DDH-5-03	411624.48	6913924.82	862.11	228.60	247.00	-50.00	1972
DDH-8-01	412095.00	6912825.00	733.00	244.14	230.00	-45.00	1972
DDH-8-02	411941.00	6912678.00	715.00	122.83	30.00	-45.00	1972
DDH-1-18	411905.23	6913891.64	859.84	109.42	247.00	-50.00	1990
DDH-1-19	411944.36	6913847.09	858.32	105.16	247.00	-50.00	1990
DDH-1-20	411964.87	6913814.88	857.54	106.98	247.00	-50.00	1990
DDH-1-21	411790.21	6913965.79	857.61	79.25	247.00	-50.00	1991
DDH-1-22	411843.14	6913920.09	860.96	59.74	247.00	-50.00	1991

HOLE-ID	LOCATIONX	LOCATIONY	LOCATIONZ	LENGTH	AZIMUTH	DIP	YEAR
DDH-1-23	411872.34	6913935.22	857.62	116.13	247.00	-50.00	1991
DDH-1-24	411923.15	6913961.14	851.41	172.51	247.00	-50.00	1991
DDH-1-25	411870.96	6913874.94	862.92	56.69	247.00	-50.00	1991
DDH-1-26	411893.87	6913831.27	864.00	44.50	247.00	-50.00	1991
DDH-1-27	411926.94	6913790.19	863.98	56.69	247.00	-50.00	1991
DDH-1-28	411946.96	6913735.58	870.16	46.02	247.00	-50.00	1991
DDH-1-29	411994.30	6913761.51	859.82	88.70	247.00	-50.00	1991
DDH-1-30	412035.23	6913774.91	849.39	137.46	247.00	-50.00	1991
DDH-1-31	411977.87	6913690.28	870.86	56.69	247.00	-50.00	1991
DDH-1-32	412010.84	6913697.56	866.97	84.12	247.00	-50.00	1991
DDH-1-33	412042.61	6913711.84	858.02	108.51	247.00	-50.00	1991
DDH-1-34	412019.60	6913640.71	871.50	78.03	247.00	-50.00	1991
DDH-1-35	412054.63	6913650.38	864.68	96.32	247.00	-50.00	1991
DDH-1-36	412025.70	6913569.57	874.71	81.08	247.00	-50.00	1991
DDH-1-37	412065.13	6913587.99	868.63	106.98	247.00	-50.00	1991
DDH-1-38	411878.23	6913970.26	853.85	129.84	247.00	-50.00	1991
DDH-1-39	411834.65	6913952.06	858.18	77.11	247.00	-50.00	1991
DDH-1-40	411878.56	6913904.88	859.52	78.03	247.00	-50.00	1991
DDH-1-41	411908.52	6913860.87	861.20	84.12	247.00	-50.00	1991
DDH-1-42	411925.65	6913822.05	861.67	78.03	247.00	-50.00	1991
DDH-1-43	412101.37	6913604.30	860.73	138.68	247.00	-50.00	1991
DDH-1-44	412099.83	6913670.73	853.25	44.80	248.50	-50.00	1991
DDH-1-45	412083.70	6913730.22	849.46	166.42	247.00	-50.00	1991
DDH-1-46	412061.41	6913861.75	840.20	233.48	247.00	-51.00	1991
DDH-1-47	412099.83	6913670.73	853.25	161.24	247.00	-50.00	1991
DDH-1-48	412057.96	6913518.83	867.24	75.59	247.00	-50.00	1991
DDH-1-49	412101.04	6913538.38	861.90	71.93	247.00	-50.00	1991
DDH-1-50	412067.60	6913453.86	858.15	78.03	247.00	-50.00	1991
DDH-1-51	411831.21	6913983.85	853.86	62.79	247.00	-50.00	1991
DDH-1-52	411992.03	6913872.67	849.93	178.00	247.00	-50.00	1991
DDH-1-53	411979.84	6913928.87	850.00	209.09	247.00	-50.00	1991
DDH-1-54	412090.67	6913397.19	841.62	68.88	247.00	-50.00	1991
DDH-1-55	412130.97	6913417.36	839.20	126.80	248.50	-50.00	1991
DDH-12-01	413294.25	6911706.58	846.00	82.30	247.00	-50.00	1992
DDH-12-02	413093.50	6911972.47	823.68	106.68	247.00	-50.00	1992
DDH-12-03	413131.28	6911960.15	823.69	87.78	247.00	-50.00	1992
DDH-12-04	413338.12	6911750.84	839.66	137.16	247.00	-50.00	1992
DDH-1-56	412108.47	6913373.07	833.97	68.58	247.00	-50.00	1992
DDH-1-57	412068.62	6913490.11	862.46	117.35	247.00	-50.00	1992
DDH-1-58	412086.31	6913564.87	867.66	157.19	247.00	-50.00	1992
DDH-4-05	412304.00	6913392.00	849.00	97.54	247.00	-50.00	1992
DDH-4-06	412269.00	6913461.00	844.00	83.82	247.00	-50.00	1992

HOLE-ID	LOCATIONX	LOCATIONY	LOCATIONZ	LENGTH	AZIMUTH	DIP	YEAR
RC92-01	412258.97	6912945.39	730.20	91.44	247.00	-50.00	1992
RC92-02	412314.01	6912832.45	705.80	25.91	247.00	-50.00	1992
RC92-03	412436.51	6913013.08	716.36	92.35	247.00	-50.00	1992
RC92-04	411875.02	6913866.65	863.20	61.87	247.00	-50.00	1992
RC92-05	411896.56	6913828.52	864.12	60.96	247.00	-50.00	1992
RC92-06	411936.14	6913846.50	858.56	121.92	247.00	-50.00	1992
RC92-07	411745.75	6914199.16	833.10	36.58	0.00	-90.00	1992
RC92-08	411442.60	6913845.35	866.10	91.44	247.00	-50.00	1992
RC92-09	411741.67	6913301.44	808.00	106.68	215.00	-45.00	1992
RC92-10	412288.59	6912881.49	712.39	60.96	247.00	-50.00	1992
RC92-11	411170.60	6914256.50	851.76	106.68	247.00	-50.00	1992
DH95-1	413892.03	6914584.52	649.16	30.02	0.00	-90.00	1995
DH95-2	413955.66	6914675.08	644.87	28.60	0.00	-90.00	1995
DH95-A	411411.66	6913646.43	867.18	7.62	0.00	-90.00	1995
DH95-B	411472.71	6913269.85	831.89	6.10	0.00	-90.00	1995
DH95-C	411713.73	6913209.80	793.46	25.91	0.00	-90.00	1995
DH95-D	411888.47	6913186.93	783.85	7.62	0.00	-90.00	1995
DH95-E	411712.79	6913112.56	789.57	18.29	0.00	-90.00	1995
DH95-F	412090.69	6912887.93	732.85	19.81	0.00	-90.00	1995
DH95-G	411867.51	6912893.28	772.49	10.67	0.00	-90.00	1995
DH95-H	411829.90	6913091.91	776.35	30.05	0.00	-90.00	1995
DH96-11	411799.76	6912890.79	774.61	15.50	0.00	-90.00	1996
DH96-12	411828.82	6912915.45	774.38	18.40	0.00	-90.00	1996
DH96-13	411800.21	6912926.30	775.51	18.30	0.00	-90.00	1996
DH96-14	411800.28	6912926.25	776.12	14.33	0.00	-90.00	1996
DH96-15	411800.74	6912945.19	776.09	16.76	0.00	-90.00	1996
DH96-16	411805.87	6913041.81	777.17	18.30	0.00	-90.00	1996
DH96-2	411371.27	6913950.07	859.70	22.90	0.00	-90.00	1996
MW96-A	411397.51	6913714.97	868.76	91.44	0.00	-90.00	1996
MW96-B	411534.99	6913435.01	838.92	91.44	0.00	-90.00	1996
MW96-C	411945.76	6913057.86	762.39	50.00	0.00	-90.00	1996
MW96-D	412166.73	6912837.46	723.82	41.15	0.00	-90.00	1996
MW96-E	411394.54	6913261.54	838.63	91.44	0.00	-90.00	1996
MW96-F	411750.81	6914708.77	792.40	62.50	0.00	-90.00	1996
MW96-G	412216.96	6914311.58	784.59	74.70	0.00	-90.00	1996
MW96-H	412534.67	6914627.86	744.18	55.20	0.00	-90.00	1996
MW96-I	412931.76	6914355.11	720.66	54.90	0.00	-90.00	1996
MW96-J	411959.92	6913902.16	853.31	91.44	0.00	-90.00	1996
MW96-K	412089.39	6913458.81	855.27	93.00	0.00	-90.00	1996
WC-001	411818.51	6913844.68	868.14	483.61	65.00	-80.00	2006
WC-002	411818.51	6913844.68	868.14	448.17	65.00	-72.00	2006
WC-003	411878.56	6913904.88	859.52	86.89	245.00	-50.00	2006

HOLE-ID	LOCATIONX	LOCATIONY	LOCATIONZ	LENGTH	AZIMUTH	DIP	YEAR
WC-004	411904.88	6913858.79	861.42	85.36	245.00	-50.00	2006
WC-005	412034.29	6913940.66	842.22	288.05	245.00	-50.00	2006
WC-006	412019.83	6913970.01	841.41	160.06	245.00	-50.00	2006
WC-007	411933.29	6913929.37	852.89	163.50	245.00	-50.00	2006
WC-008	411993.11	6913923.11	846.68	227.66	245.00	-50.00	2006
WC-009	411976.02	6913982.40	845.88	234.70	245.00	-50.00	2006
WC-010	412027.69	6913870.36	844.39	219.50	245.00	-50.00	2006
WC-011	411451.75	6913921.74	860.27	112.78	156.00	-55.00	2006
WC-012	411676.27	6913078.40	797.77	153.93	246.00	-55.00	2006
WC-013	411365.91	6913742.66	873.29	140.21	65.00	-55.00	2006
WC-014	411668.78	6913971.37	855.89	213.36	155.00	-55.00	2006
WC-015	411500.11	6913935.56	858.96	157.01	245.00	-55.00	2006
WC-016A	411452.39	6913999.70	856.76	34.45	245.00	-55.00	2006
WC-016B	411452.39	6913999.70	856.76	132.62	245.00	-60.00	2006
WC-017	411440.94	6914058.82	853.24	164.63	245.00	-55.00	2006
WC-018	412171.60	6913768.54	821.57	305.62	246.00	-50.00	2006
WC-019	412080.18	6913796.37	839.91	232.87	246.00	-50.00	2006
WC-020	412793.33	6912515.18	747.61	139.63	246.00	-55.00	2006
WC-021A	411875.85	6913805.11	868.98	68.58	66.00	-74.00	2006
WC-021B	411875.85	6913805.11	868.98	420.63	66.00	-74.00	2006
WC-022	412895.23	6912450.80	751.16	207.30	246.00	-55.00	2006
WC-023	412969.11	6912369.03	758.58	193.90	246.00	-55.00	2006
WC-024	412969.11	6912369.03	758.58	307.85	246.00	-85.00	2006
WC-025	411939.42	6913767.11	866.54	567.85	66.00	-82.00	2006
WC-026	412951.09	6912229.63	779.73	165.20	246.00	-70.00	2006
WC-027	413034.95	6912264.94	770.94	139.17	246.00	-55.00	2006
WC-028	411976.45	6913708.30	869.65	288.31	66.00	-83.00	2006
WC-029	413034.95	6912264.94	770.94	182.88	246.00	-76.00	2006
WC-030	412682.10	6912620.10	742.57	89.92	246.00	-45.00	2006
WC-031	412717.47	6912634.44	740.40	138.69	246.00	-60.00	2006
WC-032	412753.77	6912572.60	745.13	147.83	246.00	-55.00	2006
WCR-001	412245.00	6912887.00	718.00	15.25	246.00	-57.00	2006
WCR-002	412196.00	6912855.00	723.50	15.25	246.00	-56.00	2006
WCR-003	412148.16	6912835.96	724.17	15.25	246.00	-58.00	2006
WCR-004	412088.29	6912798.92	729.16	15.25	246.00	-57.00	2006
WCR-005	412034.00	6912767.00	737.00	33.55	246.00	-58.00	2006
WCR-006	411956.00	6912776.00	754.80	21.35	246.00	-57.00	2006
WCR-008	412388.00	6913034.00	738.70	18.30	246.00	-57.00	2006
WCR-009	412338.00	6912994.00	726.00	18.30	246.00	-58.00	2006
WCR-010	412290.00	6912960.00	729.00	36.60	246.00	-60.00	2006
WCR-011	412191.00	6912910.00	730.00	27.45	246.00	-58.00	2006
WCR-012	412136.00	6912883.00	738.70	15.25	246.00	-59.00	2006

HOLE-ID	LOCATIONX	LOCATIONY	LOCATIONZ	LENGTH	AZIMUTH	DIP	YEAR
WCR-013	412082.00	6912860.00	738.90	15.25	246.00	-50.00	2006
WCR-014	412029.00	6912833.00	748.00	15.25	246.00	-57.00	2006
WCR-015	411963.00	6912825.00	759.80	33.55	246.00	-58.00	2006
WCR-016	411914.00	6912787.00	764.60	18.30	246.00	-56.00	2006
WCR-017	411866.00	6912757.00	759.10	18.30	246.00	-60.00	2006
WCR-018	412206.00	6912975.00	745.00	24.40	246.00	-58.00	2006
WCR-019	412154.00	6912959.00	743.00	18.30	246.00	-60.00	2006
WCR-020	412094.00	6912931.00	740.60	15.25	246.00	-58.00	2006
WCR-021	412036.00	6912905.00	742.30	15.25	246.00	-55.00	2006
WCR-022	411982.00	6912877.00	750.40	14.63	246.00	-55.00	2006
WCR-023	411929.00	6912849.00	762.90	12.20	246.00	-55.00	2006
WCR-024	411874.00	6912835.00	772.50	24.40	246.00	-58.00	2006
WCR-025	411819.00	6912815.00	770.90	25.91	246.00	-60.00	2006
WCR-027	412162.00	6913027.00	755.00	15.25	246.00	-58.00	2006
WCR-028	412104.00	6913002.00	758.00	15.85	246.00	-60.00	2006
WCR-032	411877.08	6912900.45	772.16	27.45	246.00	-54.00	2006
WCR-033	411827.22	6912876.67	772.46	21.03	246.00	-60.00	2006
WCR-034	411763.69	6912856.48	767.45	25.91	246.00	-60.00	2006
WCR-035	412163.00	6913096.00	744.00	17.07	246.00	-60.00	2006
WCR-041	411828.00	6912945.00	757.00	22.86	246.00	-55.00	2006
WCR-042	411774.29	6912927.18	776.74	19.81	246.00	-54.00	2006
WCR-043	411716.72	6912897.65	772.93	19.81	246.00	-60.00	2006
WCR-048	411774.00	6912990.00	766.00	18.29	246.00	-54.00	2006
WCR-049	411719.00	6912962.00	767.00	19.81	246.00	-55.00	2006
WCR-050	411670.00	6912940.00	763.00	21.34	246.00	-58.00	2006
WCR-054	411779.65	6913052.64	779.88	30.48	246.00	-54.00	2006
WCR-055	411722.40	6913026.93	788.75	18.29	246.00	-51.00	2006
WCR-056	411667.00	6913004.74	792.59	21.34	246.00	-55.00	2006
WCR-059	411717.72	6913088.58	788.52	24.38	246.00	-55.00	2006
WCR-060	411671.30	6913068.63	798.76	18.29	246.00	-56.00	2006
WCR-061	411615.91	6913045.28	801.83	18.29	246.00	-56.00	2006
WCR-061A	411563.32	6913023.81	803.43	19.81	246.00	-58.00	2006
WCR-064	411667.35	6913139.85	795.57	22.86	246.00	-52.00	2006
WCR-065	411617.75	6913115.45	810.13	18.29	246.00	-50.00	2006
WCR-066	411560.00	6913093.00	795.00	16.76	246.00	-58.00	2006
WCR-067	411510.66	6913068.24	815.54	16.76	246.00	-60.00	2006
WCR-070	411617.37	6913174.98	804.26	18.29	246.00	-58.00	2006
WCR-071	411563.00	6913162.00	800.00	18.29	246.00	-54.00	2006
WCR-072	411511.28	6913134.19	825.11	16.76	246.00	-55.00	2006
WCR-073	411461.15	6913109.09	823.85	19.81	246.00	-60.00	2006
WCR-075	411566.00	6913231.00	803.00	21.34	246.00	-57.00	2006
WCR-076	411515.84	6913202.47	826.80	19.81	246.00	-56.00	2006

HOLE-ID	LOCATIONX	LOCATIONY	LOCATIONZ	LENGTH	AZIMUTH	DIP	YEAR
WCR-077	411462.04	6913177.32	831.46	16.76	246.00	-55.00	2006
WCR-078	411398.40	6913154.54	828.41	19.81	246.00	-58.00	2006
WCR-081	411516.56	6913270.08	825.63	24.38	246.00	-55.00	2006
WCR-082	411457.00	6913249.00	818.00	18.29	246.00	-55.00	2006
WCR-083	411415.23	6913226.86	834.35	18.29	246.00	-55.00	2006
WCR-086	411463.65	6913316.66	835.23	27.43	246.00	-55.00	2006
WCR-087	411404.00	6913293.00	839.00	21.34	246.00	-55.00	2006
WCR-088	411361.91	6913269.59	839.23	21.34	246.00	-55.00	2006
BH-01-07	411386.44	6913051.94	810.64	20.40	0.00	-90.00	2007
BH-02-07	411318.74	6912953.26	822.31	8.40	0.00	-90.00	2007
BH-03A-07	411232.77	6913185.75	838.79	10.97	0.00	-90.00	2007
BH-03B-07	411232.77	6913185.75	838.79	18.59	0.00	-90.00	2007
BH-04-07	411098.12	6913323.80	875.30	7.62	0.00	-90.00	2007
BH-05-07	411816.82	6913230.98	794.15	10.63	0.00	-90.00	2007
BH-06-07	412162.77	6912804.17	721.23	43.00	0.00	-90.00	2007
BH-07-07	411904.22	6914360.55	804.80	33.86	0.00	-90.00	2007
BH-08-07	411653.42	6914499.08	812.24	21.37	0.00	-90.00	2007
BH-09-07	411854.04	6912899.56	772.86	12.50	0.00	-90.00	2007
BH-10-07	412055.54	6912955.01	741.91	31.42	0.00	-90.00	2007
BH-11-07	411995.69	6912995.62	750.98	39.75	0.00	-90.00	2007
BH-12-07	411597.43	6914187.78	841.45	67.37	0.00	-90.00	2007
BH-13-07	412138.44	6913020.35	753.78	20.10	0.00	-90.00	2007
BH-14-07	411698.14	6914499.88	810.12	88.41	0.00	-90.00	2007
BH-15-07	412129.25	6912925.36	737.28	23.20	0.00	-90.00	2007
BH-15A-07	412093.00	6912947.00	740.55	15.40	0.00	-90.00	2007
BH-16-07	411982.54	6912865.50	750.34	24.71	0.00	-90.00	2007
BH-17-07	411793.36	6913099.20	780.37	29.28	0.00	-90.00	2007
BH-18-07	412208.66	6914484.84	771.09	68.90	0.00	-90.00	2007
BH-19-07	411477.96	6913171.95	829.97	5.00	0.00	-90.00	2007
BH-20-07	411747.13	6913314.45	808.05	0.20	0.00	-90.00	2007
BH-21-07	411608.59	6913377.21	828.75	3.70	0.00	-90.00	2007
BH-22-07	411918.24	6913169.75	781.08	6.05	0.00	-90.00	2007
BH-23-07	411929.53	6913083.57	767.11	21.60	0.00	-90.00	2007
BH-24-07	412744.56	6914471.64	735.10	95.69	0.00	-90.00	2007
BH-25-07	411934.00	6913255.27	807.07	3.05	0.00	-90.00	2007
BH-26-07	411677.97	6913183.90	795.60	26.23	0.00	-90.00	2007
BH-27-07	411742.21	6912997.64	784.45	3.40	0.00	-90.00	2007
BH-28-07	411678.13	6913028.03	796.02	2.10	0.00	-90.00	2007
BH-29-07	412436.76	6914293.28	768.02	27.76	0.00	-90.00	2007
BH-30-07	412257.59	6914195.86	793.35	4.88	0.00	-90.00	2007
BH-31-07	411157.71	6912967.27	848.87	7.53	0.00	-90.00	2007
BH-32-07	412308.06	6913778.00	798.36	9.46	0.00	-90.00	2007

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HYD WELL-1	412219.38	6912687.94	709.24	55.20	0.00	-90.00	2007
MW-01-07	412220.00	6912688.00	709.00	55.20	0.00	-90.00	2007
WC-033	412330.21	6912899.76	709.76	182.88	249.00	-55.00	2007
WC-034A	412398.43	6912816.72	702.04	67.06	246.00	-55.00	2007
WC-034B	412398.43	6912816.72	702.04	173.73	246.00	-60.00	2007
WC-035	412296.78	6913097.65	769.96	179.83	245.00	-50.00	2007
WC-036	412159.05	6913168.50	779.55	167.03	270.00	-50.00	2007
WC-037	412408.24	6912747.00	702.36	184.40	248.00	-72.00	2007
WC-038	412362.78	6913122.59	775.55	129.92	246.00	-50.00	2007
WC-039	412282.68	6912881.68	712.41	60.96	246.00	-50.00	2007
WC-040	412022.28	6913183.72	793.96	182.88	246.00	-50.00	2007
WC-041	412170.68	6912972.81	744.06	227.69	247.00	-50.00	2007
WC-042	412198.54	6913328.08	820.46	264.57	246.00	-50.00	2007
WC-043	412247.34	6913010.19	748.21	257.56	272.00	-50.00	2007
WC-044	412118.95	6913347.17	825.95	263.66	270.00	-50.00	2007
WC-045	412045.18	6913036.81	756.65	256.95	252.00	-54.00	2007
WC-046	412313.11	6913395.84	851.00	10.67	280.00	-50.00	2007
WC-047	412313.11	6913395.84	851.00	245.37	247.00	-50.00	2007
WC-048	412408.07	6913279.01	805.46	190.50	250.00	-50.00	2007
WC-049	412315.71	6913460.68	850.75	192.03	246.00	-50.00	2007
WC-050	412081.99	6913430.69	849.02	140.06	246.00	-52.00	2007
WC-051	412275.10	6913445.24	844.31	335.28	244.00	-50.00	2007
WC-052	412119.66	6913447.79	846.26	235.31	246.00	-50.00	2007
WC-053	412169.17	6913468.72	844.41	182.88	246.00	-50.00	2007
WC-054	412229.29	6913426.58	840.63	306.00	245.00	-50.00	2007
WC-055	412127.24	6913581.09	856.12	268.23	245.00	-50.00	2007
WC-056	412185.03	6913409.50	835.74	251.46	249.10	-48.50	2007
WC-057	412108.16	6913514.18	859.45	179.83	245.00	-51.00	2007
WC-058	412145.88	6913394.37	833.19	182.88	245.00	-50.00	2007
WC-059	412150.57	6913529.20	851.54	249.94	245.00	-50.00	2007
WC-060	412215.59	6913487.55	841.53	381.00	250.00	-48.00	2007
WC-061	412166.59	6913598.27	846.65	281.94	245.00	-50.00	2007
WC-062	412255.62	6913505.61	842.12	159.11	245.00	-50.00	2007
WC-063	412313.78	6913531.85	839.06	97.84	245.00	-50.00	2007
WC-064	412031.87	6913341.64	833.13	163.07	62.00	-75.00	2007
WC-065	412054.48	6913550.48	870.46	137.16	243.00	-50.00	2007
WC-066	411888.13	6914010.20	849.27	175.26	245.00	-48.00	2007
WC-067	412141.44	6913556.45	853.59	158.50	240.00	-52.00	2007
WC-068	411872.95	6914036.17	847.44	161.54	243.00	-50.00	2007
WC-069	412112.93	6913478.59	853.89	190.50	243.00	-49.00	2007
WC-070	411861.18	6914063.89	844.63	138.68	243.00	-50.00	2007
WC-071	412025.25	6913505.95	869.73	143.26	246.00	-50.00	2007

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WC-072	411918.75	6914056.35	842.91	222.50	243.00	-48.00	2007
WC-073	412189.92	6913543.33	843.74	292.60	251.00	-50.00	2007
WC-074	411924.93	6914135.00	833.46	236.06	244.00	-50.00	2007
WC-075	411828.23	6914016.42	851.51	80.77	245.90	-49.60	2007
WC-076	411818.53	6914042.77	848.50	87.17	246.00	-50.00	2007
WC-077	412297.15	6913425.86	847.81	135.64	245.00	-50.00	2007
WC-078	411482.28	6913119.43	825.37	111.25	246.00	-50.00	2007
WC-079	412268.94	6913413.73	843.38	102.11	240.00	-50.00	2007
WC-080	411432.49	6913164.20	831.45	109.40	247.00	-50.00	2007
WC-081	412281.47	6913383.68	844.98	73.15	243.00	-50.00	2007
WC-082	411435.96	6913239.53	836.10	105.16	245.00	-50.00	2007
WC-083	412263.59	6913474.36	842.95	123.45	245.00	-50.00	2007
WC-084	411379.76	6913280.46	840.01	122.22	247.00	-50.00	2007
WC-085	412303.70	6913492.32	844.56	157.58	245.00	-50.00	2007
WC-086	411825.11	6912717.25	732.18	96.01	246.00	-50.00	2007
WC-087	413195.10	6911817.75	837.02	105.16	245.00	-50.00	2007
WC-088	412233.68	6913462.62	841.58	110.34	245.00	-49.00	2007
WC-089	412354.77	6913475.13	843.20	179.83	245.00	-50.00	2007
WC-090	413335.28	6911696.39	847.70	97.54	245.00	-50.00	2007
WC-091	413330.79	6911604.81	861.50	103.63	245.00	-50.00	2007
WC-092	412355.70	6913411.69	840.47	120.40	245.00	-49.00	2007
WC-093	413376.10	6911623.71	863.65	106.68	245.00	-50.00	2007
WC-094	412311.08	6913360.35	845.72	91.44	246.00	-48.00	2007
WC-095	413383.78	6911485.85	870.95	60.96	245.00	-50.00	2007
WC-096	412354.07	6913376.10	837.80	185.93	244.00	-49.00	2007
WC-097	413050.03	6911875.97	842.77	76.20	246.00	-50.00	2007
WC-098	412324.19	6913323.82	832.39	121.92	246.00	-49.00	2007
WC-099	413157.89	6911927.66	825.96	149.35	246.00	-51.00	2007
WC-100	413189.40	6911941.91	819.15	115.83	245.00	-49.00	2007
WC-101	412357.02	6913340.82	831.28	137.16	249.00	-51.00	2007
WC-102	412285.81	6913307.45	826.36	48.77	245.00	-50.00	2007
WC-103	412329.53	6913299.84	822.88	91.44	245.00	-45.00	2007
WC-104	412329.53	6913299.84	822.88	32.00	245.00	-75.00	2007
WC-105	412242.57	6913401.33	840.05	54.86	245.00	-61.00	2007
WC-106	412218.97	6913390.68	833.89	62.48	250.00	-61.50	2007
WC-107	412255.49	6913372.77	836.02	18.29	0.00	-90.00	2007
WC-108	412286.58	6913350.81	839.55	15.24	245.00	-50.00	2007
WC-109	412247.67	6913468.33	842.25	100.07	245.00	-50.00	2007
WC-110	412206.01	6913450.71	840.40	18.29	245.00	-50.00	2007
WC-111	412174.41	6913574.49	846.84	160.04	245.00	-48.50	2007
WC-112	413117.95	6912044.80	809.49	86.87	245.00	-50.00	2007
WC-113	412997.39	6912250.07	776.53	170.06	248.00	-49.00	2007

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WC-114	413018.97	6912138.61	797.33	19.20	242.00	-50.00	2007
WC-115	413018.97	6912138.61	797.33	97.55	242.00	-60.00	2007
WC-116	413158.30	6911802.56	839.10	54.86	241.00	-51.00	2007
WC-117	413229.02	6911833.97	831.16	121.35	244.00	-60.50	2007
WC-118	413372.51	6911712.66	844.25	86.87	249.00	-58.00	2007
WC-119	413290.82	6911720.08	844.79	103.63	245.00	-59.00	2007
WC-120	413253.90	6911705.53	844.79	92.96	245.00	-59.00	2007
WC-121	413222.00	6911693.18	843.92	22.86	245.00	-60.00	2007
WC-122	413275.05	6911781.17	836.66	127.41	245.00	-59.00	2007
WC-123	413247.94	6911768.91	839.38	117.35	245.00	-60.00	2007
WC-124	413222.84	6911758.25	841.05	165.51	246.00	-60.00	2007
WC-125	413160.00	6911868.54	833.81	103.63	246.00	-60.00	2007
WC-126	413188.34	6911880.23	829.77	118.87	246.00	-60.00	2007
WC-127	413136.76	6911858.92	836.00	80.77	245.00	-60.50	2007
WC-128	413113.18	6911849.93	838.95	64.92	246.00	-60.00	2007
WC-129	413121.48	6911911.57	831.09	112.87	245.00	-50.00	2007
WC-130	413464.13	6912569.22	762.70	250.09	76.00	-80.00	2007
WC-131	412925.95	6912350.14	762.18	166.12	245.00	-60.00	2007
WC-132	412889.24	6912334.66	763.58	152.40	245.00	-61.00	2007
WC-133	412861.76	6912322.47	764.62	53.34	245.00	-60.50	2007
WC-134	412867.89	6912439.43	752.40	224.03	245.00	-60.00	2007
WC-135	412840.06	6912428.70	753.34	190.50	245.00	-60.00	2007
WC-136	412811.51	6912416.84	754.73	128.02	246.00	-60.50	2007
WC-137	412783.68	6912406.13	756.29	152.40	247.00	-60.00	2007
WC-138	412755.60	6912395.40	757.32	182.88	246.00	-62.00	2007
WC-139	413367.18	6912532.60	748.40	213.66	64.00	-80.00	2007
WC-140	413389.16	6912543.54	751.57	218.50	62.00	-60.00	2007
WC-141	413461.15	6912572.11	762.58	242.32	65.00	-60.00	2007
WC-142	413593.74	6912628.79	784.95	230.12	245.00	-68.00	2007
WC-143	411998.44	6913559.96	875.83	121.92	245.00	-50.00	2007
WC-144	411992.41	6913492.35	868.45	74.80	245.00	-50.00	2007
WC-145	411994.55	6913627.89	872.94	82.29	245.00	-50.00	2007
WC-146	412079.99	6913528.66	864.90	213.36	245.00	-50.00	2007
WC-147	411965.72	6913480.28	870.89	30.48	245.00	-49.50	2007
WC-148	412019.20	6913435.81	856.50	108.20	247.00	-51.50	2007
WC-149	412063.88	6913382.42	842.16	170.68	245.00	-50.00	2007
WC-150	412472.17	6914303.12	764.76	240.79	245.00	-50.00	2007
WC-151	411795.24	6912998.81	776.84	73.15	245.00	-50.00	2007
WC-152	411746.06	6913097.05	784.38	53.34	245.00	-50.00	2007
WC-153	412266.64	6914213.32	790.67	198.12	65.00	-50.00	2007
WC-154	412074.81	6913992.02	833.94	149.35	245.00	-50.00	2007
DW-08-01	411449.00	6912479.00	731.00	130.14	0.00	-90.00	2008

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DW-08-02	411403.00	6912461.00	734.00	123.36	0.00	-90.00	2008
GA-01-08	411923.58	6913557.43	878.94	280.42	0.00	-60.00	2008
GA-02-08	411887.80	6913751.63	873.17	250.00	246.00	-70.00	2008
GA-03-08	411731.92	6913868.53	869.24	265.18	118.00	-60.00	2008
GA-04-08	412022.67	6913828.23	847.91	240.79	66.00	-60.00	2008
GA-05-08	411921.83	6913564.02	879.02	227.69	10.00	-60.00	2008
GA-06-08	412188.94	6913840.28	815.70	228.32	246.00	-60.00	2008
HYD WELL-2	412177.03	6912678.51	710.05	150.88	0.00	-90.00	2008
CN14-11	412241.00	6912937.00	730.00	73.15	247.00	-50.00	2014
CN14-12	412219.00	6912984.00	745.00	93.57	247.00	-45.00	2014
CN14-13	412503.00	6913129.00	734.00	128.02	247.00	-45.00	2014
CN14-14	412390.00	6913064.00	747.00	83.82	247.00	-45.00	2014
CN14-15	412078.00	6913322.00	828.00	88.39	247.00	-45.00	2014
CN15-01	412260.00	6912907.00	722.90	48.77	245.00	-50.00	2015
CN15-02	412300.00	6912926.00	720.30	108.21	245.00	-50.00	2015
CN15-03A	412307.00	6912858.00	709.60	38.71	240.00	-50.00	2015
CN15-03B	412307.00	6912858.00	710.60	83.06	240.00	-50.00	2015
CN15-04	412282.00	6912956.00	731.40	106.68	245.00	-50.00	2015
CN15-05	412296.00	6912963.00	731.50	170.47	245.00	-60.00	2015
CN15-06	412336.00	6912847.00	706.50	134.12	240.00	-50.00	2015
CN15-07	412361.00	6912865.00	705.70	193.55	235.00	-60.00	2015
CN15-08	412230.00	6912959.00	739.20	60.96	240.00	-50.00	2015
CN15-09	412926.00	6912306.00	768.70	123.45	245.00	-50.00	2015
CN15-10	412948.00	6912270.00	774.70	93.98	245.00	-50.00	2015
CN15-11	412916.00	6912256.00	776.60	68.58	245.00	-50.00	2015
CN15-12	412894.00	6912291.00	770.50	106.68	245.00	-50.00	2015
CN15-13	412864.00	6912354.00	762.90	143.26	245.00	-50.00	2015
CN15-14	412827.00	6912378.00	760.40	128.02	245.00	-50.00	2015
CN15-15	412831.00	6912339.00	765.00	115.83	245.00	-50.00	2015
CN15-16	412796.00	6912365.00	761.00	111.26	245.00	-50.00	2015
CN15-17	413005.40	6912217.40	783.40	56.39	245.00	-50.00	2015
CN15-18	412688.00	6912556.00	749.60	56.70	245.00	-50.00	2015
CN15-19	412958.00	6912321.00	766.40	152.40	245.00	-50.00	2015
CN15-20	412980.00	6912285.00	771.30	114.30	245.00	-50.00	2015
CN15-21	412896.00	6912369.00	761.20	172.21	245.00	-50.00	2015
CN15-22	412912.00	6912210.00	783.10	91.44	245.00	-50.00	2015
CN15-23	413086.00	6911894.00	839.90	80.77	245.00	-50.00	2015
CN15-24	413131.00	6911950.00	826.10	64.10	245.00	-50.00	2015
CN15-25	413161.00	6911965.00	820.60	64.01	245.00	-50.00	2015
CN15-26	413121.00	6911986.00	821.40	47.25	245.00	-50.00	2015
CN15-27	413191.00	6911981.00	813.50	71.02	246.00	-50.00	2015
CN15-28	413073.00	6911994.00	822.80	53.34	245.00	-50.00	2015

HOLE-ID	LOCATIONX	LOCATIONY	LOCATIONZ	LENGTH	AZIMUTH	DIP	YEAR
CN15-29	413157.00	6911996.00	815.80	67.37	245.00	-50.00	2015
CN15-30	413110.00	6912019.00	816.40	68.58	245.00	-50.00	2015
CN15-31	412976.00	6912203.00	786.00	50.29	245.00	-50.00	2015
CN15-32	413036.00	6912234.00	778.40	83.82	245.00	-50.00	2015
CN15-33	413054.00	6912206.00	781.80	86.87	245.00	-50.00	2015
CN15-34	413021.00	6912194.00	786.60	54.86	245.00	-50.00	2015
CN17-01	412598.47	6912510.54	733.55	150.88	245.00	-50.00	2017
CN17-02	412565.05	6912568.47	726.44	149.35	245.00	-50.00	2017
CN17-03	412794.12	6912586.58	737.24	153.92	245.00	-50.00	2017
CN17-04	412647.23	6912297.72	754.25	111.25	245.00	-50.00	2017
CN17-05	412747.41	6912259.78	766.17	100.58	245.00	-50.00	2017
CN17-06	412747.41	6912259.78	766.17	160.02	245.00	-80.00	2017
CN17-09	412923.59	6915694.13	788.39	152.40	170.00	-50.00	2017
CN17-10	413348.27	6914988.63	811.69	103.67	236.00	-50.00	2017
CN17-11	413358.97	6915263.99	783.71	117.35	216.00	-50.00	2017
CN17-12	412284.65	6912850.54	706.27	145.00	239.00	-50.00	2017
CN17-13	412385.41	6912781.92	696.86	120.70	245.00	-50.00	2017
CN17-14	412514.49	6912622.85	709.94	100.58	245.00	-50.00	2017
CN17-15	412384.09	6912782.96	697.61	129.54	245.00	-60.00	2017
CN17-16	412460.17	6912691.92	695.37	222.50	243.00	-62.00	2017
CN17-17	412459.57	6912692.71	695.63	170.00	280.00	-50.00	2017
CN17-18	412386.95	6912781.44	693.86	126.49	197.00	-51.00	2017
CN17-19	412986.10	6912338.17	754.88	173.74	245.00	-50.00	2017
CN17-20	413002.89	6912300.17	759.23	144.78	245.00	-50.00	2017
CN17-21	412370.71	6912802.37	696.26	138.42	255.00	-50.00	2017
CN17-22	413064.45	6912179.76	778.88	94.49	245.00	-50.00	2017
CN17-23	413098.83	6912196.48	772.44	120.40	245.00	-50.00	2017
CN17-24	412345.69	6912826.54	698.07	153.93	245.00	-50.00	2017
CN17-25	413079.83	6912222.95	767.12	117.35	245.00	-50.00	2017
CN17-26	413062.77	6912246.90	765.02	135.64	245.00	-50.00	2017
CN17-27	412308.31	6912833.65	703.06	106.68	245.00	-50.00	2017
CN17-28	413074.57	6912156.10	780.98	82.30	245.00	-50.00	2017
CN17-29	413093.32	6912127.86	787.07	80.77	245.00	-50.00	2017
CN17-30	413037.97	6912168.08	782.22	53.34	235.00	-75.00	2017
CN17-31	413112.70	6912159.46	776.55	118.91	245.00	-50.00	2017
CN17-32	413045.70	6912142.25	788.52	53.34	255.00	-75.00	2017
CN17-33	413060.00	6912114.12	793.02	50.29	245.00	-75.00	2017
CN17-34	413060.08	6912017.38	811.31	51.82	255.00	-50.00	2017
CN17-35	413050.72	6912044.04	805.80	47.24	245.00	-50.00	2017
CN17-36	413001.06	6912179.63	781.77	50.29	245.00	-50.00	2017
CRM20-001	412840.21	6912424.83	764.13	247.80	0.00	-90.00	2020
CRM20-002	412887.17	6912445.46	760.85	278.89	0.00	-90.00	2020

HOLE-ID	LOCATIONX	LOCATIONY	LOCATIONZ	LENGTH	AZIMUTH	DIP	YEAR
CRM21-003	412393.00	6912819.00	701.00	231.65	220.00	-70.00	2021
CRM21-004	412251.39	6913667.62	832.15	387.10	270.00	-52.00	2021
CRM21-005	412388.14	6912822.00	711.37	185.93	205.00	-65.00	2021
CRM21-006	412388.14	6912822.00	711.37	283.46	265.00	-70.00	2021
CRM21-007	412251.39	6913667.62	832.15	509.02	270.00	-63.00	2021
CRM21-008	412388.14	6912822.00	711.37	257.56	244.50	-70.00	2021
CRM21-009	412378.93	6912908.56	714.45	309.37	241.00	-57.00	2021
CRM21-010	412191.15	6913836.94	824.95	535.41	246.00	-68.00	2021
CRM21-011	412369.62	6912911.12	714.90	350.52	252.50	-64.00	2021
CRM21-012	412162.75	6913985.09	827.94	501.40	250.00	-63.00	2021
CRM21-013	412131.01	6913917.58	837.38	387.10	245.00	-50.00	2021
CRM21-014	412162.75	6913985.09	827.94	440.44	250.00	-49.00	2021
CRM21-015	412687.43	6912700.00	741.15	271.27	235.00	-50.00	2021
CRM21-016	412800.54	6912561.22	754.07	258.50	190.00	-70.00	2021
CRM21-017	412061.26	6914023.03	843.27	420.00	245.00	-69.00	2021
CRM21-018	412458.36	6912693.48	710.94	315.50	300.00	-60.00	2021
CRM21-019	412061.26	6914023.03	843.27	356.62	245.00	-52.09	2021
CRM21-020	412971.54	6912369.02	768.71	106.68	292.00	-48.00	2021
CRM21-021	412971.14	6912366.78	768.18	249.94	269.00	-55.00	2021
CRM21-022	412375.05	6912986.11	731.44	367.28	245.00	-55.00	2021
CRM21-023	412415.21	6912941.28	714.10	482.36	269.20	-65.00	2021
CRM21-024	412872.00	6912391.00	760.00	233.17	245.00	-60.00	2021
CRM21-025	412872.00	6912391.00	760.00	303.00	260.00	-87.71	2021
CRM21-RC001	412972.00	6912078.00	802.00	108.20	245.00	-60.00	2021
CRM21-RC002	412960.00	6912063.00	806.00	70.10	225.00	-45.00	2021
CRM21-RC007	412905.00	6912150.00	791.00	56.40	295.00	-45.00	2021
CRM21-RC008	412905.00	6912150.00	791.00	152.40	295.00	-75.00	2021
CRM21-RC011	410795.75	6916603.75	872.24	150.90	270.00	-60.00	2021
CRM21-RC012	413017.00	6911926.00	827.00	99.10	0.00	-90.00	2021
CRM21-RC013	412986.00	6912029.00	796.00	100.58	270.00	-60.00	2021
CRM21-RC014	411493.00	6913876.00	867.00	57.91	225.00	-45.00	2021
CRM21-RC015	411493.00	6913876.00	867.00	83.82	225.00	-70.00	2021
CRM21-RC016	411457.00	6913846.00	867.00	105.18	100.00	-50.00	2021
CRM21-RC017	411541.00	6913887.00	862.00	158.50	265.00	-65.00	2021