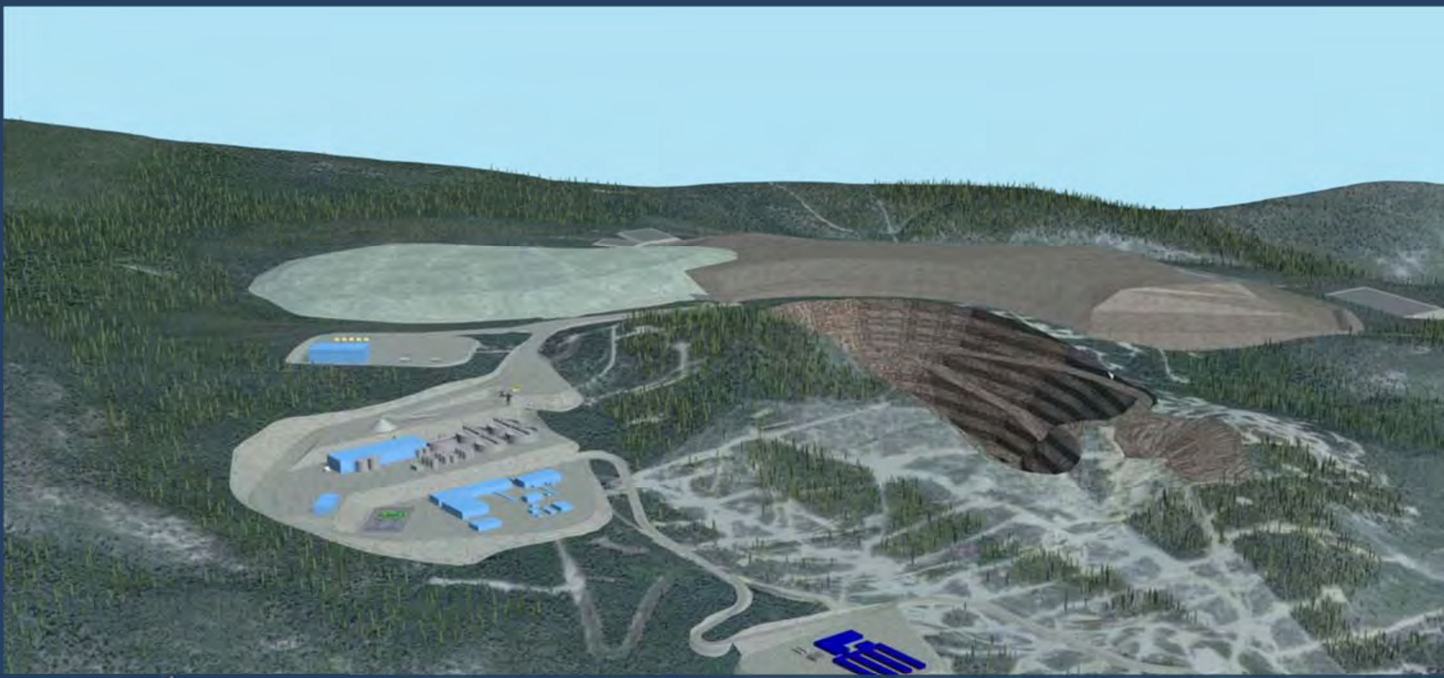

NI 43-101 PRELIMINARY ECONOMIC ASSESSMENT TECHNICAL REPORT ON THE CARMACKS PROJECT, YUKON, CANADA

JDS Energy & Mining Inc.

Prepared for:

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NOTICE

JDS Energy & Mining, Inc. prepared this National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Copper North Mining Corp. The quality of information, conclusions and estimates contained herein is based on: (i) information available at the time of preparation; (ii) data supplied by outside sources, and (iii) the assumptions, conditions, and qualifications set forth in this report.

Copper North Mining Corp. filed this Technical Report with the Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities law, any other use of this report by any third party is at that party's sole risk.

Contents

Tables and Figures

1 Executive Summary

1.1 Introduction

Copper North Mining Corp. (CNMC) commissioned JDS Energy & Mining Ltd. (JDS) to complete a Preliminary Economic Assessment (PEA) for the Carmacks Project, located 192 km north of Whitehorse, the capital of the Yukon Territory. The purpose of this study is to develop and document a preliminary project design and economics for recovery of copper, gold, and silver from the oxide mineralization using agitated tank leach technology.

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

1.2 Project Description

The Carmacks project is located in the Dawson Range at latitude 62°21'N and longitude 136° 41'W. The project site is located on Williams Creek, 8 km west of the Yukon River and 38 km northwest of the town of Carmacks. Figure 1.1 shows the general project location on a territorial scale. The project site is located in the Whitehorse mining division of the Yukon and consists of 373 quartz claims and 20 quartz leases. The claims and leases comprising the Carmacks property are held directly by Carmacks Mining Corp., a wholly-owned subsidiary of Copper North Mining Corp. (CNMC). CNMC is listed on the TSX-V under the symbol COL.

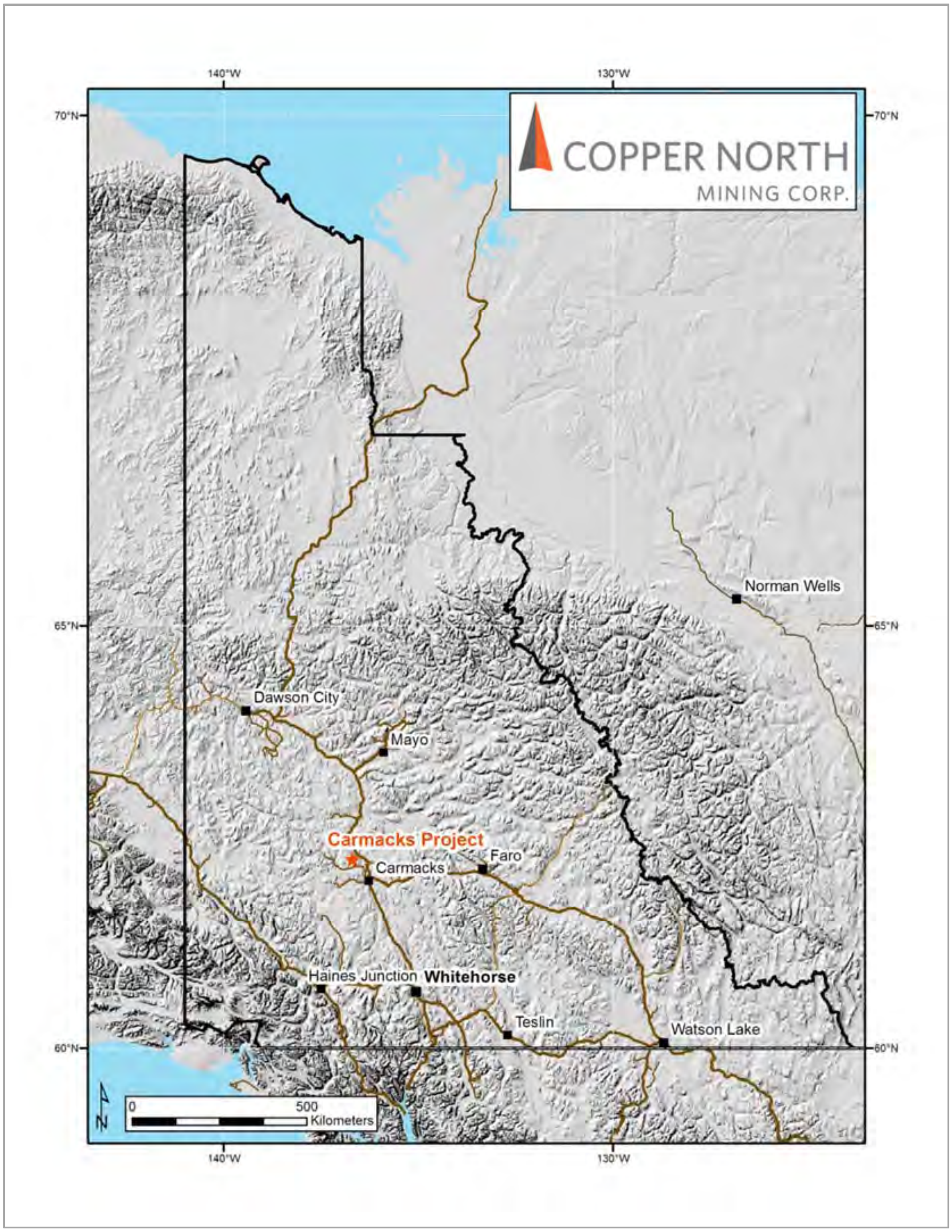
The climate in the Carmacks area is marked by warm summers and cold winters. Average daily mean temperatures range from -30°C for the month of January to 12°C for the month of July. Precipitation is light with moderate snowfall, the heaviest precipitation being in the summer months. The average annual precipitation is approximately 346.5 mm (water equivalent) with one-third falling as snow. July is the wettest month. Mean annual lake evaporation is estimated to be 440 mm with the maximum evaporation occurring in July.

Topographic relief for the entire property is 515 m. In the immediate project area the topographic relief is 230 m. Elevations range from 485 m at the Yukon River to 1,000 m on the western edge of the claim block. Discontinuous permafrost is present at varying depths in most north-facing slope locations and at depth in other areas.

The Quartz Mining Act and Quartz Mining Land Use Regulations in the Yukon provide for the holder of mineral claims to obtain surface rights of Crown land covered by mineral claims for the purpose of developing a mining property. This attracts a minor fee of $1.00 per acre per year. All the mineral claims held by CNMC on this project are overlain by Crown land.

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

Figure 1.1: Project Location within Yukon Territory



Source: CNMC (2012)

1.3 History, Exploration and Drilling

The property was first staked in 1970 and since that time has been the subject of various exploration campaigns comprising trenching, diamond drilling, reverse circulation drilling (RCD), geophysical, and geochemical surveying. Prior to 2006, a total of 80 diamond drill holes (DDH) and 11 RCD, totalling 12,900 m of drilling, had been completed in exploration of the property. In addition, over 8,000 m of surface trenching were completed. The majority of this work focused on the No.1 Zone and was completed before the mid-1990s.

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.

In 2014, Copper North commissioned Merit Consultants International Inc. to prepare a PEA on the Carmacks Project. The PEA focused 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.

Following the completion of the PEA, Copper North initiated a drilling program to explore Zones 2000S, 12 and 13. A total of 50 core holes totalling 4,358 m were completed during the 2014 and 2015 drill programs.

1.4 Geology & Mineralization

The Carmacks copper-gold deposit lies within the Yukon Cataclastic Terrane. The deposit is hosted by amphibolite and mafic gneisses (generally quartz deficient) that form a roof pendant or rafts within Upper Triassic hornblende-biotite granodiorite of the Granite Mountain Batholith.

The No. 1, 4 and 7 Zones, as presently defined, extend over a 700 m strike length and at least 450 m down dip. The deposit is open at depth. These zones are oxidized to an approximate depth of 250 m below surface. Within the oxidized area, pyrite is virtually absent and pyrrhotite is absent. Weathering 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, fill fractures and rim sulphides. Primary sulphide minerals and magnetite have been disseminated and form narrow massive bands or heavy disseminations in bands.

The character of the deposit changes along strike, leading to a division into northern and southern halves. The northern half is more regular in thickness, dip angle, width, and down dip characteristics. The southern half splays into irregular intercalations, terminating against sub-parallel faults down dip. Both the north and south ends of the deposit are offset by cross- cutting faults. The No. 4 Zone is interpreted as the southern offset extension of the No. 1 Zone. The northern offset has not been identified yet.

The majority of the copper found in the oxide portion of the No. 1 Zone is in the form of the secondary minerals malachite, cuprite, azurite and tenorite (copper limonite), with very minor other secondary copper minerals (covellite, digenite, djurlite). Other secondary minerals include limonite, goethite, specular hematite, and gypsum. Primary copper mineralization is restricted to bornite and chalcopyrite. Other primary minerals include magnetite, gold, molybdenite, native bismuth, bismuthinite, arsenopyrite, pyrite, pyrrhotite and carbonate. Molybdenite, visible gold, native bismuth, bismuthinite and arsenopyrite occur rarely.

1.5 Metallurgical Testing and Mineral Processing

The initial 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, as documented in the 2012 FS report, 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.

In 2016, test work has switched from a heap leach to tank leach of the copper, gold and silver. 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 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.

Based on the test work, a copper/gold leach circuit was selected as the preferred recovery method. The criteria and recoveries from CALT2 (BV Minerals 2016) 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.

1.6 Mineral Resource Estimates

The Mineral Resources reported for the Carmacks Project were prepared by ACS of Vancouver, British Columbia (Arseneau, 2016).

Grades were estimated by ordinary kriging and inverse distance to the second power (ID^2) constrained within individually identified geological units using sample data composited to 5 m for Zones 1, 4 and 7 and to 2.5 m for Zones 12, 13 and 2000S. All Mineral Resources were estimated in model blocks measuring 5 m by 5 m by 5 m vertically.

Grade interpolation strategies were based on zone orientations, drill hole distances and parameters derived from variographic analysis. Grade interpolations were carried out in two passes with successive passes only interpolating block grades for blocks that had not been interpolated by the previous passes.

ACS is satisfied that the geological modelling reflects the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support resource evaluation. ACS considers that blocks that were estimated during the first pass and had an average distance of samples used less than 50 m could be assigned to the Measured category. Blocks interpolated during the first pass and had an average distance of points used greater than 50 m were assigned to the Indicated category. All other interpolated blocks were assigned to the Inferred category.

ACS considers that the blocks with grades above the cut-off grade satisfy the criteria for "reasonable prospects for economic extraction" and can be reported as a Mineral Resource. Mineral resources for the Carmacks Project are summarized in Table 1.1. There are no known legal, political, environmental, or other risks that could materially affect the potential development of the Mineral Resources.

Table 1.1: Carmacks Project Mineral Resource Statement January 25, 2016

	Class	Tonnes (000)	Total Cu (%)	Soluble Cu (%)	Au (g/t)	Ag (g/t)	Sulphide Cu (%)
Oxide and Transition mineralization	Measured	6,484	0.86	0.69	0.41	4.24	0.17
	Indicated	9,206	0.97	0.77	0.36	3.8	0.2
	Measured + Indicated	15,690	0.94	0.74	0.38	3.97	0.2
	Inferred	913	0.45	0.3	0.12	1.9	0.15
Sulphide mineralization	Measured	1,381	0.64	0.05	0.19	2.17	0.59
	Indicated	6,687	0.69	0.04	0.17	2.34	0.65
	Measured + Indicated	8,068	0.68	0.05	0.18	2.33	0.65
	Inferred	8,407	0.63	0.03	0.15	1.99	0.61

Source: ACS (2016)

In the opinion of ACS, the resource evaluation reported herein is a reasonable representation of the copper, gold and silver Mineral Resources found in the Carmacks Project at the current level of sampling. Mineral resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into mineral reserve.

1.7 Mineral Reserve

This PEA does not state a mineral reserve.

1.8 Mining

The mine plan for the Carmacks project remains unchanged from that developed by IMC (2012) and detailed in the 2012 FS. Mineralized material production will be 1.775 million tonnes per annum (Mt/a), at an average rate of 4,860 tonnes per day (t/d). The peak total material rate will be 13.5 Mt/a. Mining will be conducted on two 12-hour shifts per day for 335 days per year with three mining crews working a 20-day on/10-day off rotation. With the current mine production schedule, the commercial project life is estimated to be about 6.5 years, after a brief pre-production period.

Four mining phases were designed for the Carmacks Project. Inter-ramp slope angles are projected to be 52.6°. The design is also based on proposed 10 m mining benches in a double bench configuration for final walls. The main road will be 25 m wide, at a maximum grade of 10%. This will accommodate trucks of approximately 90 t, such as Caterpillar 777 class trucks.

A mine production schedule was developed to estimate annual mineralized material and waste movements from the pit. Table 1.2 shows the mine production schedule.

The total material will be 70.0 Mt for a waste to mineralized material ratio (strip ratio) of 5.1 to 1. Pre-production will be minimal at 1.0 Mt. The total material movement will be 9.5 Mt during Year 1 and will peak at 13.5 Mt for Years 2 through 4. The strip ratio will be 6.6 to 1 during these peak years. Estimated mining dilution is incorporated into the production schedule.

Table 1.2: Mine Production Schedule

		PP	1	2	3	4	5	6	7	Total
Mill Feed	Mt	0.15	1.63	1.78	1.78	1.78	1.78	1.78	0.90	11.55
Recovered Copper Grade	%	0.701	0.792	0.752	0.828	0.774	0.709	0.859	0.957	0.793
Total Copper Grade	%	0.867	0.965	0.932	1.019	0.907	0.896	1.065	1.204	0.977
Soluble Copper grade	%	0.691	0.802	0.756	0.839	0.777	0.773	0.864	0.943	0.805
Sulphide Copper Grade	%	0.176	0.164	0.176	0.18	0.13	0.136	0.201	0.261	0.172
Gold Grade	g/t	0.306	0.472	0.411	0.49	0.343	0.412	0.462	0.497	0.435
Silver Grade	g/t	2.99	4.43	4.26	4.84	3.59	3.91	4.59	5.4	4.34
Total Tonnes	MT	0.95	9.50	13.50	13.50	13.50	11.78	5.82	1.41	69.96
Waste Tonnes	Mt	0.80	7.88	11.73	11.73	11.73	10.00	4.05	0.50	58.41
Strip Ratio	None	5.4	4.8	6.6	6.6	6.6	5.6	2.3	0.6	5.1

PP=pre-production
Source: IMC (2016)

1.9 Recovery Methods

Mineralized material will be crushed using a jaw crusher followed by a SAG mill operated in closed circuit with hydrocyclones. Copper will be recovered using a sulphuric acid leach and solvent extraction / electrowinning (SX-EW). Copper leach residue slurry will be neutralized with lime and gold/silver will be leached into solution using cyanide and adsorbed onto activated carbon. The processing facilities will include an ADR circuit to concentrate the gold/silver into doré.

The copper and gold recovery process was designed on the basis of 4,860 t/d with average head grades of 0.98% Cu and 0.435 g/t Au.

A jaw crushing plant will operate at a nominal crushing rate of 312 t/h, 16 hours a day for 365 days per year. The process plant will operate 24 hours per day for 365 days per year with a plant availability of 92% and a processing rate of 220 t/h. The copper will be leached with sulphuric acid, recovered in a solvent extraction/electrowinning circuit (SX-EW) and shipped as cathode copper. The copper circuit tailings will be leached in CIL tanks and the carbon is processed in a 2 t/d carbon ADR plant for gold extraction and the production of gold doré. This process will achieve an estimated recovery of 85.2% Cu and 84.4% Au.

The mineralized material processing facilities will include the following unit operations:

- Crushing and material handling:
    - Primary crushing: A vibrating grizzly screen and jaw crusher in open circuit, producing a final product P_{80} of approximately 114 mm; and
    - Fine mineralized material stockpile: 5,000 t fine mineralized material stockpile and reclaim feeders.
- Process plant:
    - Primary grinding: A SAG mill operating in closed circuit with a cyclone cluster, producing a final product P_{80} of approximately 664 µm;
    - Copper leaching and recovery: A pre-leach thickener, six copper leach tanks, four counter current decantation (CCD) thickeners and an SX-EW circuit; and
    - Gold leaching: 6 - CIL tanks, an ADR plant and cyanide destruction.
- Tailings management facility
    - Tailings filtration, load-out and dry stacks at the tailings management facility.

1.10 Infrastructure

The project envisions the upgrading or construction of the following key infrastructure items:

- Approximately 12 km of all-seasonal access road from Freegold Road to the project site;
- Dry stack tailings management area (TMA);
- Waste rock storage area (WRSA);
- Crushing and grinding circuits;

- Gold extraction plant;
- Solvent extraction / electrowinning plant;
- Electrical connection to Yukon Energy, transmission line and on-site substation and distribution network;
- Process and fire water storage and distribution;
- Sewage collection system;
- Truck shop and warehouse building;
- Administration building; and
- Mine dry and camp facility.

1.11 Environment and Permitting

Major hard rock mining projects in Yukon are required to satisfy a two-step regulatory review and approval process before mining activity may commence. The first step is an environmental and socio-economic assessment conducted in accordance with the Yukon Environmental and Socio-Economic Assessment Act (YESAA) which is administered by the Yukon Environmental and Socio-Economic Assessment Board (YESAB). The YESAA review typically takes from nine to 18 months to complete, depending on the project, the issues, and the need for supplementary information beyond that initially submitted by the proponent.

The second step is the regulatory phase involving two enabling licenses, the Quartz Mining License (QML) and the Water Use License (WUL). The QML process is administered by Yukon Energy, Mines, and Resources (EMR) and the QML regulates the following mining related activities:

- The area and mineral deposits to be mined;
- Allowable mining and milling rates;
- Pre-construction plans and drawings;
- Post-construction as-built drawings;
- Monitoring programs;
- Design of mine workings, including underground and open pit development and production, and waste dumps;
- Site infrastructure, including buildings, roads, fuel storage, etc.;
- Solid waste disposal;
- Reclamation, including slope stability, erosion control, and re-vegetation;
- Financial security; and,
- Annual reporting requirements.

The WUL process is administered by the Yukon Water Board and regulates the use of water, the deposit of waste into water, receiving water quality, and all water conveyance and retention structures associated with a development. Any WUL issued for the project will set limits on the quality and quantity of discharges to water and on the quantities of any surface or groundwater takings. The WUL also will set monitoring and reporting requirements for surface and ground waters, for water discharges, and for water management structures such as dams, dykes, and ponds. A Type B WUL would be necessary for the project construction phase, in order to provide the necessary water to supply the construction camp and other construction activities as well as for the sanitary septic system. A Type A WUL will be required for project operation, involving the sourcing of process water, pit dewatering, and the discharge of any treated water.

The environmental assessment phase must be completed and a positive decision (i.e., an approval) issued by YESAB before the regulatory phase of permitting can be completed. Yukon EMR will review a QML submission in advance of a YESAB decision but cannot issue a QML until the decision document for the YESAA review has been issued. With a QML application developed and submitted in advance of a YESAB decision, the QML decision can proceed quickly following a positive decision by YESAB. The Yukon Water Board does not review a WUL license application until the YESAA process is complete and a decision document issued, and the WUL review process can take several months, particularly for a Type A licence review which also requires a public hearing.

The Carmacks project, as it was previously proposed in 2007, received a positive environmental and socio- economic assessment determination from YESAB in 2008 and a Quartz Mining License in 2009. The nature of the project changes proposed in the present plan are such that the project proposal must again pass an Executive Committee Level Environmental Screening Assessment before the regulatory phase, in which the QML is either amended or a new QML is issued and a new WUL issued. Much of the project information and potential environmental effects have already been reviewed by YESAB, as part of the previous copper-only, project submission which should assist in expediting the next YESAB review.

1.12 Operating and Capital Cost Estimates

1.12.1 Operating Costs

The operating cost estimate (OPEX) is based on a combination of experience, reference project, budgetary quotes and factors as appropriate with a preliminary study.

Preparation of the operating cost estimate is based on the JDS philosophy that emphasizes accuracy over contingency and utilizes defined and proven project execution strategies.

The operating cost estimate in this study includes the costs to mine and process the mineralized material to produce copper cathode and doré and general and administrative expenses (G&A). These items total the mine operating costs and are summarized in Table 1.3.

The estimate is based on leasing equipment and owner operating the mining and services fleet. The target accuracy of the operating cost is -25/+30%.

The operating cost estimate is divided into three major sections:

- Open Pit Mining;
- Processing; and
- General & Administrative.

The total operating unit cost is estimated to be $45.45/t processed. Average annual, total LOM and unit operating cost estimates are summarized in Table 1.3. The mine operating costs also include the lease payments for mining equipment.

Table 1.3: Breakdown of Estimated Operating Costs

Operating Costs	Avg Annual (M$)	$/t processed	LOM (M$)
Mining*	26	15.73	182
Processing	38	23.27	269
G&A	11	6.45	75
Total	**75**	**45.45**	**525**

*Average LOM Mining cost amounts to $2.63/t mined at a 5.1:1 strip ratio (excluding pre-production tonnes mined).
Numbers may not add due to rounding
Source: JDS (2016)

Operating costs are expressed in Canadian dollars with a fixed exchange rate of US$:C$ = 0.78. No allowance for inflation has been applied.

The main OPEX component assumptions are outlined in Table 1.4.

Table 1.4: Main OPEX Component Assumptions

Item	Unit	Value
Electrical Power Cost	$/kWh	0.105
Overall Power Consumption (all facilities)	kWh/t processed	45.75
Diesel Cost (delivered)	$/litre	0.762
LOM Average Manpower	employees	260

Source: JDS (2016)

1.12.2 Capital Costs

LOM project capital costs total $264M, consisting of the following distinct phases:

- Pre-production Capital Costs – includes all costs to develop the property to a 4,850 t/d production. Initial capital costs total $241M (including $26M contingency) and are expended over a 23-month pre-production construction and commissioning period; and
- Sustaining & Closure Capital Costs – includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations. Sustaining capital costs total $23M (including $3M in contingency) are expended in operating Years 1 through 10.

The capital cost estimate (CAPEX) was compiled using a combination of quotations, database costs, and database factors.

Table 1.5 presents the capital estimate summary for initial, sustaining, and closure capital costs in Q3 2016 dollars with no allowance for escalation. The estimate is also based on the assumption that the mining equipment will be leased.

Table 1.5: Capital Cost Summary

WBS	Area	Pre-Production (M$)	Sustaining/ Closure (M$)	Total (M$)
1000	Mining	9.9	3.0	12.9
2000	Site Development	10.6	6.9	17.5
3000	Ore Crushing & Handling	3.0	-	3.0
4000	Process Plant	129.2	1.9	131.1
5000	On-Site Infrastructure	15.0	1.8	16.8
6000	Off-Site Infrastructure	7.3	-	7.3
7000	Indirect Costs	8.3	1.3	9.6
8000	EPCM	16.8	-	16.8
9000	Owners Costs	14.4	-	14.4
C100	Closure Costs	-	5.6	5.6
	Subtotal Pre-Contingency	**214.7**	**20.5**	**235.2**
9900	Contingency	25.9	2.5	28.4
	Total Capital Costs	**240.6**	**23.0**	**263.6**

Source: JDS (2016)

1.13 Economic Analysis

1.13.1 Main Assumptions

Detailed market studies on the potential sale of copper cathode and doré from the Carmacks Project were not completed. The terms were reviewed and found to be acceptable by QP Gord Doerksen, P.Eng.

No contractual arrangements for shipping, port usage, or refining exist at this time. Table 1.6 outlines the terms used in the economic analysis.

Table 1.6: Net Smelter Return (NSR) Assumptions Used in the Economic Analysis

Assumptions	Unit	Value
Payable Copper Cathode	%	100
Payable Gold	%	100
Payable Silver	%	100
Copper Cathode Shipping Charge	US$/lb	0.015
Au Refining Charge	US$/payable oz	4.00
Ag Refining Charge	US$/payable oz	0.40

Source: JDS (2016)

Table 1.7 outlines the metal price and exchange rate used in the economic analysis.

Table 1.7: Metal Price and Exchange Rate used in the Economic Analysis

Assumptions	Unit	Value
Cu Price	US$/lb	2.50
Au Price	US$/oz	1,300
Ag Price	US$/oz	17.50
Exchange Rate	US$:C$	0.78

Source: JDS (2016)

1.13.2 Results

The economic results for the project based on the assumptions made are shown in Table 1.8.

Table 1.8: Economic Results

Summary of Results	Unit	Value
Cash Cost (Net of Byproduct)	US$/lb Cu	1.08
Cash Cost (incl. Sustaining and Closure CAPEX)	US$/lb Cu	1.16
Capital Costs		
Pre-Production Capital	M$	215
Pre-Production Contingency	M$	26
Total Pre-Production Capital	**M$**	**241**
Sustaining & Closure Capital	M$	21
Sustaining & Closure Contingency	M$	3
Total Sustaining & Closure Capital	M$	23
Total Capital Costs Incl. Contingency	**M$**	**264**
Working Capital	M$	10
Pre-Tax Cash Flow	LOM M$	118
	M$/a	17
Taxes	LOM M$	43
After-Tax Cash Flow	LOM M$	75
	M$/a	11
Economic Results		
Pre-Tax NPV$_{8\%}$	**M$**	**12**
Pre-Tax IRR	**%**	**9.4**
Pre-Tax Payback	**Years**	**5.2**
After-Tax NPV$_{8\%}$	**M$**	**-11**
After-Tax IRR	**%**	**6.6**
After-Tax Payback	**Years**	**5.3**

Source: JDS (2016)

1.13.3 Sensitivities

A simplistic sensitivity analysis was performed to determine which factors most affect the project economics and is discussed in Section 23. Each variable evaluated was tested using the same sensitivity values, although some may be more likely to experience significantly more fluctuation in value over the LOM (i.e. CAPEX versus exchange (FX) rate). The confidence attributed to each variable in this study does not factor into the sensitivity analysis, the inter-correlation between certain variables, and for this reason is considered a simplistic approach to determine which variable would most affect the economic results of the project.

Sensitivity analyses were performed on copper and gold prices, mill head grade, CAPEX, and OPEX as variables. The value of each variable was changed, plus and minus 15% independently, while all other variables were held constant. The results of the sensitivity analyses are shown in Table 1.9 and Table 1.10.

Table 1.9: Sensitivity Results (Pre-and After-Tax $NPV_{8\%}$)

Variable	Pre-Tax $NPV_{8\%}$ (M$)			After-Tax $NPV_{8\%}$ (M$)		
	-15% Variance	0% Variance	15% Variance	-15% Variance	0% Variance	15% Variance
Copper Price	-54	12	78	-58	-11	32
FX Rate	115	12	-65	57	-11	-66
Head Grade	-76	12	100	-77	-11	47
CAPEX	46	12	-22	23	-11	-46
OPEX	64	12	-40	24	-11	-48

Source: JDS (2016)

Table 1.10: Metal Price Sensitivity Analysis (M$ Pre-Tax $NPV_{8\%}$)

	Copper Price US$/lb						
Au US$/oz	$1.75	$2.00	$2.25	$2.50	$2.75/lb	$3.00	$3.25
1,000	-154	-110	-66	-22	22	66	110
1,100	-143	-99	-55	-11	33	77	121
1,200	-131	-87	-43	1	45	89	133
1,250	-126	-82	-38	6	50	94	138
1,300	-120	-76	-32	12	56	100	144
1,400	-109	-65	-21	23	67	111	155
1,500	-97	-53	-9	35	79	123	166

Source: JDS (2016)

1.14 Conclusions

It is the conclusion of the QPs that the PEA summarized in this technical report contains adequate detail and information to support the economic result herein contained. The PEA proposes the use of industry standard equipment and operating practices. To date, the QPs are not aware of any fatal flaws for the project.

Using the assumptions highlighted in this report, the Carmacks project would require an extension of the mine life through resource expansion, improvements in silver metal recoveries, capital and/or operating cost reduction or modest increases to gold and copper metal prices to be advanced to the next stage of study (Preliminary Feasibility Study).

1.14.1 Risks

As with any proposed mining project, there are risks. The most significant potential risks associated with the Carmacks project are the level of Mineral Resource estimate, level of metallurgical testing and process design, operating and capital cost escalation, permitting, unforeseen schedule delays, changes in regulatory requirements, retention of mining personnel due to the remote location, the ability to raise financing, and commodity price variability.

These risks are common to most mining projects, many of which may be mitigated, at least to some degree, with adequate engineering, planning and pro-active project management.

1.14.2 Opportunities

There are several opportunities identified at this time that could improve the economics of the project and are summarized in the following sections. Further information and assessments are required before these opportunities could be included in the project economics.

1.14.2.1 Process

The metallurgical process detailed in this PEA has effectively maximized the recovery of copper and gold from the oxide mineralization. However, there remains an opportunity to address improvement in the silver recovery, which is only 9.4% based on the metallurgical test work completed to date. This low Ag recovery appears related to changes in the metallurgical process given that earlier process configurations achieved much higher Ag recovery (61 to 72%, with a mean of 67.7%; (Beattie, 2015)). Increasing silver recovery to 67% has the potential to add another $18.8 M in gross LOM revenue.

There are several opportunities related to further refinement of the metallurgical process to be examined that have the potential to reduce CAPEX and/or OPEX, including:

- Examination of alternative solid/liquid separation technology in the copper circuit and for tailings filtration, with the potential for both CAPEX (equipment cost) and OPEX (energy and reagent consumption) reductions;
- Optimization of leach temperature and reagent additions for copper leaching (potential for energy and reagent consumption reductions); and,
- Reagents account for 54% of processing OPEX, and the cyanide destruction reagents account for approximately 28% of that reagent cost. Consideration of alternative methods of cyanide destruction in the final tailings slurry, with a focus on reduction of reagent costs, represents a potentially significant OPEX reduction.

1.14.2.2 Extend Mine Life

Extension of the mine life beyond seven years has the potential to provide the single largest increase in NPV of all the opportunities to be examined. Recent exploration drilling in 2014 and 2015 identified additional near-surface oxide Mineral Resources, in Zones 2000S, 12, and 13 (ACS 2016), that remain to be brought into the project plan. These additional resources remain open along strike, indicating a potential to add further oxide resources to the project.

Additional drilling, metallurgical testing, and mine planning need to be completed to bring these resources into the project plan.

If these additional Mineral Resources were to be brought into the resource model with an extension of the mine life by four years at the LOM average throughput, grade and recoveries, there is potential to increase the NPV by approximately $90 M and the IRR by approximately 6%.

1.14.2.3 Other Potential Opportunities

Other potential opportunities include:

- Mine and plant construction efficiency and timelines;
- Global sourcing of used equipment for operations; and
- Evaluation of processing sulphide Mineral Resource at Carmacks, for mine life extension.

1.15 Recommendations

JDS recommends a staged approached to future work on the Carmacks project due to the risks of the project and the current economic results. Due to the marginal nature of the project at the base case metal prices, it is recommended that CNMC initially investigate improving the project economics through the improvement and opportunities listed in Section 1.14.2.

The first stage of the recommended work should focus on process and metallurgical improvements. Table 1.11 provides a breakdown of the recommended task and their associated costs.

Table 1.11: Process and Metallurgical Improvement Costs

Item	Cost ($)
Improve Ag recovery, test work to confirm CIL flowsheet, leach testing and carbon loading	75,000
Optimization of leach temperature and reagent additions(variability and confirmation samples)	50,000
Examination of alternative solid/liquid separation technology	50,000
Reagent Optimization	Included Above
Total Estimate	**175,000**

Source: JDS (2016)

2 Introduction

2.1 Basis of Technical Report

CNMC commissioned JDS to complete the PEA for the Carmacks Project. The purpose of this study is to complete a review and compilation of the resources, mining designs and preliminary economics following the reporting requirements stipulated by Canadian National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101).

The Carmacks deposit has been the subject of several prior studies, including:

Kilborn Engineering Pacific Ltd. (Kilborn)

- 1995 - Carmacks Copper Project Feasibility Study
- 1997 - Carmacks Copper Project, Yukon, Canada, Basic Design Report and Definitive Cost Estimate

M3 Engineering & Technology Corp. (M3)

- 2007 - Carmacks Copper Project, Copper Mine and Process Plant, NI 43-101 Technical Report, Feasibility Study
- 2012 - Carmacks Copper Project, Copper Mine and Process Plant, NI 43-101 Technical Report, Feasibility Study (2012 FS)

Merit Consultants International Inc. (Merit)

- 2014 – Carmacks Copper Project, Yukon, Canada, Preliminary Economic Assessment of Copper, Gold, and Silver Recovery.

Arseneau Consulting Services Inc. (ACS)

- 2016 – Carmacks Copper Project, Yukon, Canada, Independent Technical Report on the Carmacks Copper Project

The Kilborn studies examined the development of the copper oxide mineral occurrence as an open pit mine with valley fill heap leaching followed by solvent extraction and electrowinning. The 2007 and 2012 M3 studies updated the development plan, capital and operating cost estimates, and financial analysis for the project as a copper heap leach operation. The Merit study was a PEA of the recovery of copper from the deposit by acid heap leaching using an on/off pad and recovery of gold and silver by heap leaching using cyanide.

The present report is a PEA of the recovery of copper, gold, and silver, using a two-stage agitated tank leach process and dry stack tailings storage. This PEA is based only on the oxide copper-gold-silver Mineral Resource in Zones 1, 4, and 7 of the Carmacks deposit defined by Arseneau (2016).

The remaining sulphide Mineral Resource in Zones 1, 4, and 7, and the combined oxide and sulphide Mineral Resource defined in Zones 2000S, 12, and 13 by Arseneau (2016) are not included in this PEA.

This PEA does not include Inferred Mineral Resources in the economic calculations; only Measured and Indicated Mineral Resources are included. However, the PEA is preliminary in nature and there is no guarantee that any of the Mineral Resources will be converted to Mineral Reserves or that the PEA will be realized

2.2 Scope of Work

This report summarizes the work carried out by several consultants and the scope of work for each company is listed below, and combined, makes up the total project scope.

JDS Energy & Mining Inc. (**JDS**) scope of work included:

- Compiling the technical report including information provided by other consulting companies;
- Establishing an economic framework for potentially mineable resources;
- Selecting mining equipment;
- Develop a conceptual flowsheet, specifications and selection of process equipment;
- Designing required site infrastructure, identify suitable sites for plant facilities and other ancillary facilities;
- Estimating mining, process plant and infrastructure OPEX and CAPEX for the project;
- Preparing a financial model and conducting an economic evaluation including sensitivity and project risk analysis; and
- Interpreting the results and make conclusions that lead to recommendations to improve value and reduce risks.

ARSENEAU Consulting Services Inc. (ACS) scope of work included:

- Data verification of geology, sample collection and sample processing; and
- Completing a Mineral Resource estimate.

Golder Associates Ltd. (Golder) scope of work included:

- Scoping and selection of appropriate waste rock storage facilities;
- Scoping and selection of TMA;
- Scoping and selection of appropriate mine water management facilities related to the waste rock and tailings facilities.

Independent Mining Consultants Inc. (IMC) scope of work included:

- Open pit mine planning and scheduling.

Dreisinger Consulting Inc. scope of work included:

- Mineral processing and metallurgical testing.

2.3 Qualifications and Responsibilities

The results of this PEA are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between CNMC and the respective QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of appropriate professional institutions/associations. The QPs are responsible for the specific report sections as follows in Table 2.1.

Table 2.1: QP Responsibilities

QP	Company	QP Responsibility/Role	Report Section(s)	Site Visit
Gord Doerksen, P.Eng.	JDS Energy & Mining Inc.	Overall responsibility, Costs and Economics	1 to 3, 15, 18, 19, 20, 21, 22, 23, 24, 25, 26, 27, 28, 29	July 26, 2016
Kelly McLeod, P. Eng.	JDS Energy & Mining Inc.	Processing	17	Did not visit site
Gilles Arseneau, P.Geo	ARSENEAU Consulting Services Inc.	Geology and Mineral Resource Estimate	4 to 12, 14	October 14, 2015
David Dreisinger, Ph.D., P.Eng	Dreisinger Consulting Inc.	Metallurgy	13	Did not visit site
David Anstey, P. Eng.	Golder Associates Ltd.	Dry Stack Tailings Management Area	18.5.1	July 12, 2016
Fiona Esford, P.Eng.	Golder Associates Ltd.	Waste Rock Storage Area	18.5.2	July 8, 2010
Michael G. Hester, M.S., FAusIMM	Independent Mining Consultants, Inc.	Mining Method	16	May 16 & 17, 2007

2.4 Units, Currency and Rounding

The units of measure used in this report are as per the 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 dollar figures quoted in this report refer to Canadian dollars (C$ or $) unless otherwise noted.

Frequently used abbreviations and acronyms can be found in Section 29. This report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

2.5 Sources of Information

This report is based on information collected by JDS during a site visit performed on July 26, 2016 and on additional information provided by CNMC throughout the course of JDS's investigations. Other information was obtained from the public domain. JDS has no reason to doubt the reliability of the information provided by CNMC. This technical report is based on the following sources of information:

- Discussions with CNMC personnel;
    - Harlan Meade, Ph.D., President and CEO; and
    - Doug Ramsey, M.Sc., R.P.Bio, Vice President, Sustainability and Environmental Affairs.
- Inspection of the Carmacks Project area;
- Review of exploration data collected by CNMC;
- Additional information from public domain sources including;
    - Kilborn Engineering Pacific Ltd.
        - 1995, "Carmacks Copper Project Feasibility Study"
        - 1997, "Carmacks Copper Project, Yukon, Canada, Basic Design Report and Definitive Cost Estimate"
        - M3 Engineering & Technology Corp.
        - 2007, "Carmacks Copper Project, Copper Mine and Process Plant, NI 43-101 Technical Report, Feasibility Study Volume I, Executive Summary"
        - 2012, "Carmacks Copper Project, Copper Mine and Process Plant, NI 43-101 Technical Report, Feasibility Study" (2012 FS)

The documentation received and the sources of information are listed in Section 28.

3 Reliance on Other Experts

The QPs have assumed that all the information and technical documents listed in the Reference section (Section 28) of this report are accurate and complete in all material aspects. While the QPs have carefully reviewed all the available information, they have not audited this work and cannot guarantee its accuracy and completeness. However, as a result of their review of the work, the QPs believe the work has been performed diligently by other QPs and that the conclusions derived are reasonable. If any new information becomes known or available that would have a material effect on the findings and conclusions contained in this report, the QPs will revise this report.

The QPs did not review any licenses, permits, work contracts, or perform an independent verification of land title and tenure. The QPs have not verified the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s), such as royalty agreements, between third parties.

Baseline surface water quality, hydrology, fish and fish habitat, vegetation, wildlife, and groundwater studies were conducted by P.A. Harder and Associates Ltd., Hallam Knight Piésold Ltd, Access Consulting Group, and Ecological Logistics and Research Ltd., and EcoFor Consulting Ltd.. Baseline heritage resources surveys were conducted by Antiquus Archaeological Consultants Ltd. and EcoFor Consulting Ltd. The site hydrogeology model, site-wide water balance model, and water quality model were developed by Golder Associates Ltd. The preliminary design for the WRSA and the conceptual design for the dry stack TMA were developed by Golder. All studies were conducted under the direction of CNMC and its predecessors.

While the QPs have relied largely on the documents listed in Section 28 for the information in this report, the conclusions and recommendations belong exclusively to the QPs. The results and opinions outlined in this report are dependent on the aforementioned information being current, accurate, and complete as of the date of this report. The QPs assume no information has been withheld which would impact the conclusions or recommendations made herein. Should the QPs become aware of facts or information that could materially alter the conclusions and recommendations of the report, the QPs will make necessary revisions so the report is correct and accurate.

Non-QP specialists relied upon for specific advice are:

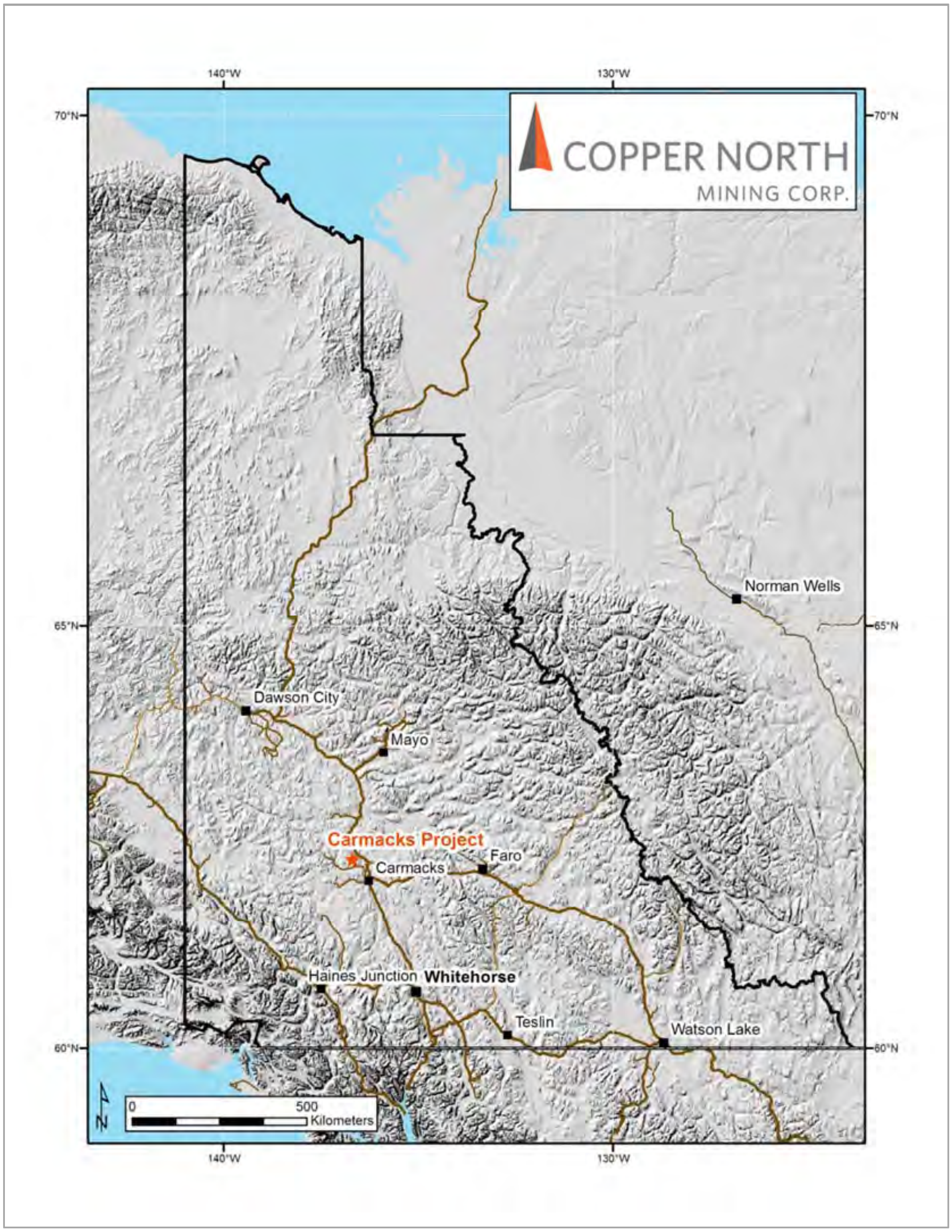
- Wentworth Taylor, CPA for taxation guidance

4 Property Description and Location

4.1 Location

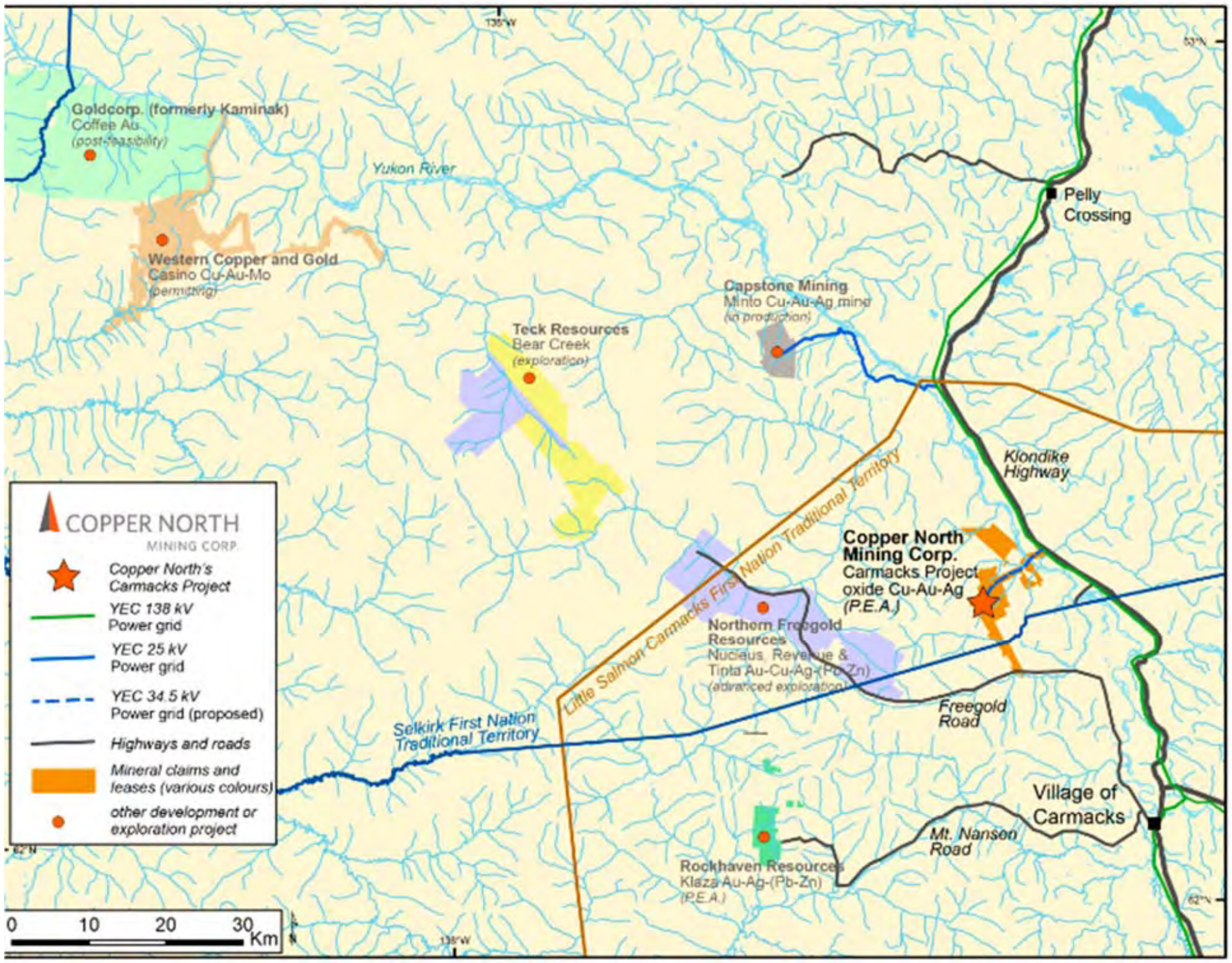
The Carmacks Project is located in the Dawson Range at latitude 62°-21’N and longitude 136° - 41’W, approximately 220 km north of Whitehorse, Yukon Territory. The project site is located on Williams Creek, 8 km west of the Yukon River and some 38 km northwest of the town of Carmacks. Figure 4.1 shows the general project location on a territorial scale. Figure 4.2 shows the location on a smaller scale, proximate to the village of Carmacks and the Yukon River.

Figure 4.1: Project Location in Yukon



Source: CNMC (2016)

Figure 4.2 : Location Map



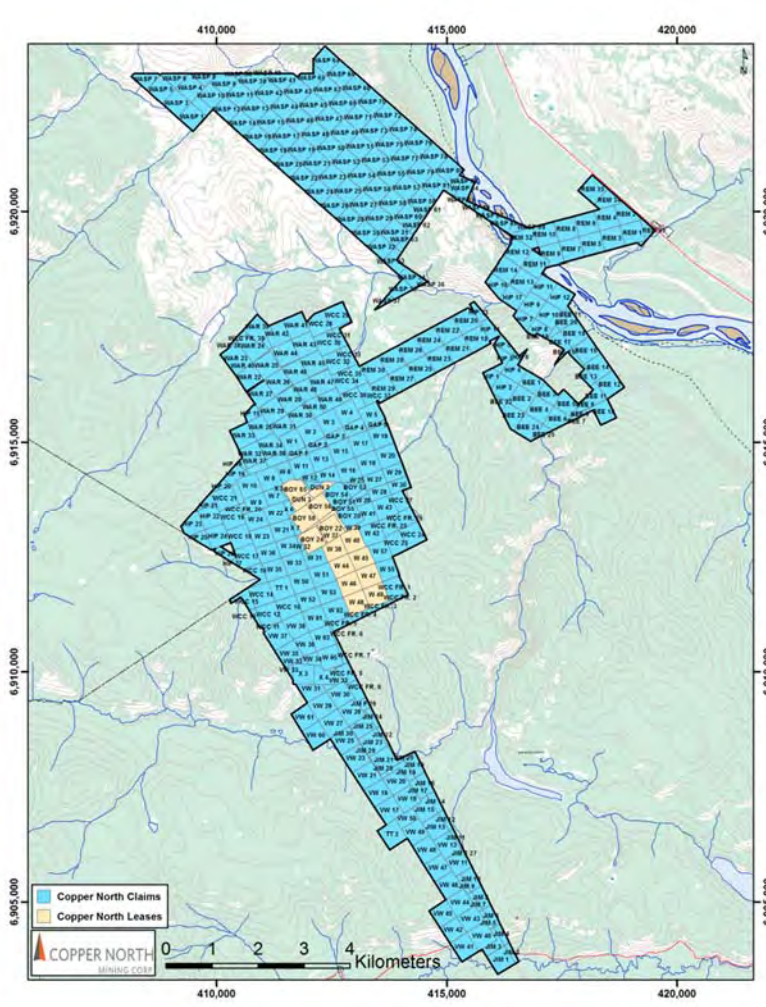
4.2 Mineral Tenure

The Carmacks project site, located in the Whitehorse mining division, consists of 373 quartz mineral claims and 20 quartz leases, as shown on Figure 4.3. The term ‘quartz’ for a claim in the Yukon is the nomenclature used to distinguish between a claim for bedrock or lode mineral rights, in contrast to a ‘placer’ claim for placer mineral rights. The registered owner of the claims is Carmacks Mining Corp.(CNMC), a 100% owned subsidiary of Copper North Mining Corp. Archer Cathro & Associates (1981) Limited retains, at the election of CNMC, either a 15% net profits interest or a 3% net smelter royalty. If CNMC elects to pay the net smelter royalty, it has the right to purchase the royalty for $2.5M, less any advance royalty payments made to that date. CNMC is required to make an advance royalty payment of C$100,000 in any year in which the average daily copper price reported by the London Metal Exchange is US$1.10 or more per pound. To date C$1.3M in advance royalty has been paid. As a result, the maximum amount of royalty that remains payable as of the date of this report is C$1.2M.

Claims in the Yukon are valid for one year and may be renewed yearly provided annual assessment work of $100 per claim is carried out or a payment of $100 per claim in lieu of work is made. A fee of $5 for a certificate of work on each claim to record the assessment work is also applicable. Assessment work on a full-size fraction (greater than 25 acres) is the same as a claim but on a small-size fraction (less than 25 acres) only $50 per year assessment work is required. Quartz leases have a term of 20 years and may be renewed. Work done on the leases may not be transferred to the claims by ‘grouping’ and therefore does not qualify for assessment work on claims.

In 2007, the majority of the claims in the centre part of the claim block, covering the No. 1, 4, 7, 7A, 12,13, and 14 zones were legally surveyed.

Figure 4.3: Carmacks Claim Location



5 Accessibility, Climate, Local Resources, Infrastructure & Physiography

5.1 Accessibility

The 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 project development; brush clearing along the road alignment was completed in 1997. The Freegold Road is maintained by the Yukon Government (YG) and is currently open seasonally, generally from April through September. The road will be kept open year-round by YG once a year-round operation begins.

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 project site and for shipment of copper cathode. 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 Climate

The climate in the project area is marked by warm summers and cold winters. Average daily temperatures at the Williams Creek Station on the 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.

5.3 Local Resources and Infrastructure

5.3.1 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. Table 5.1 lists businesses currently based in Carmacks.

Table 5.1: Carmacks Based Business

Business	Type of Service
Barrack Office and Retail Services	Canada Post Outlet, Propane Service
Berdoe Enterprises	General contracting
Busy B	Cleaning and general services
Canadian Wilderness Travel Ltd	Tourism Tour Operator
Carmacks Hotel Ltd	Hotel, RV, guest services
Carmacks Towing	Vehicle Towing, Service & storage and vehicle repair
Carmacks Yukon Gems and Things	Making and selling crafts and art
Charlie Rose Contracting	Janitorial and Wood haul
Coalmine Campground	Food, Camping, Housing Rentals
Domingo Cleaning Services	Cleaning Services
Dunena Zra Sanchi Ku Daycare	Child Care
Ghost Lake General Contracting	General contracting
Gold Panner Restaurant	Licensed Restaurant
Graceland Construction	Construction and Maintenance
Highwind Construction	Excavation and Construction
Hub Towing	Towing, Service and storage and vehicle repair
Kando Enterprises	General contracting
Mukluk Manor	Bed and Breakfast
PS Sidhu Trucking	General Contracting
Precision Builders	Construction, carpentry and building
Sunset Ridge Ranch	Breeding horses, contract work and farming
Tatchun Centre	General store and gas

Source: Yukon Business Directory (2016)

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.

A recent community recreation centre with video games, table games, and other activities is a focal point for local youth. The centre also offers a gymnasium with fitness equipment and an outdoor covered skating rink.

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 summer time is canoeing in the Yukon River. This activity brings many people from outside the area.

5.3.2 Infrastructure

The 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 will provide a location for support and administrative services during construction and during mine operations. Permanent power for the project will 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 the proposed pit. Proposed areas sufficient for the WRSA, tailings management area, and other mine facilities have been located as part of the present PEA and previous feasibility studies. Carmacks has full communications services available including cell phone service.

5.4 Physiography

Topography at the 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 485 m at the Yukon River to 1,000 m on the western edge of the claim block.

Outcrop is uncommon because of the subdued topography and lack of 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.

Overburden is generally thin; a few centimetres of moss and organic material overlie 5 to 20 cm of white felsic volcanic ash (White River ash approximately 1,250 years old). In unglaciated areas, the white ash is underlain by 10 cm of organics or peat, and 15 to 50 cm of soil. Bedrock is extensively weathered, particularly the gneissic units. At the eastern end of Trench 91-6, for example, bedrock is 7 m below surface, the deepest recorded in the unglaciated area. In the glaciated areas, the white ash is underlain by tills, generally 1 m thick, except along Williams Creek valley where an undetermined depth of till and colluvium has collected. Permafrost is present at varying depths in most north-facing slope locations and at depth in other areas. Vegetation in wet areas, especially along the William Creek valley, consists of willows and alders. Drier areas are covered by spruce trees. The property as a whole is below the tree line.

6 History

The first report of copper in this region was made by Dr. G.M. Dawson in 1887 concerning occurrences at Hoochekoo Bluff, located 12 km north of the property on the Yukon River. 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. 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 conducted reconnaissance prospecting and geochemical sampling. Archer, Cathro & Associates Limited acted as manager. During the site examination by the Dawson Range Joint Venture, G. Abbott and D. Eaton located the present No. 1 and No. 2 Zones. The property was purchased by Western Copper Holdings Ltd. and Thermal Exploration Ltd. in 1989. The two companies merged in1996 to become Western Copper Holdings Ltd.

Kilborn Engineering Pacific Inc. (Kilborn) completed the first full feasibility study for the project in 1994, as an open pit mine and heap leach to recover copper metal by solvent extraction/electrowinning (Kilborn, 1994). Kilborn updated that study in 1995 (Kilborn, 1995). Based on positive results reported by Kilborn, Western Copper Holdings Ltd. made the decision to proceed with project development and filed for an environmental review, together with Quartz Mining and Water License applications. In December 1997, Kilborn issued a basic engineering study and a definitive capital cost estimate (Kilborn, 1997). Western Copper Holdings Ltd. then began the process of obtaining proposals for the construction of the project. In 1998, after completing some early pre-construction preparatory work on the site, the company suspended the project indefinitely due to low copper prices.

In February 2003, Western Copper Holdings Ltd. changed its 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 environmental assessment and permitting process.

In early 2006, Glamis Gold Ltd. purchased Western Silver Corporation and spun off a separate company named Western Copper Corporation (Western Copper). Western Copper retained the rights to the Williams Creek property (the Carmacks project as referred herein). In September 2006, Western Copper retained M3 Engineering & Technology Corporation (M3) to revise the earlier feasibility and engineering studies and to develop a Bankable Level Feasibility Study in accordance with NI 43-101 guidelines for the heap leaching recovery of copper. This study was completed in 2007 (M3, 2007).

The project Proposal for an Executive Committee level environmental assessment under the Yukon Environmental and Socio-Economic Assessment Act (YESAA) was filed in 2007. Western Copper received a positive determination from the Yukon Environmental and Socio-economic Assessment Board (YESAB) in 2008. Western Copper applied to Yukon Energy and Mines for a Quartz Mining license for the project in 2008 and received QML-0007 in 2009, and this license remains in good standing.

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

In 2014, Copper North commissioned Merit Consultant International Inc. (Merit) to prepare a PEA on the Carmacks Project. The PEA focused 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 an ADR process and included SART (sulphidization, acidification, recycling and thickening) technology. The PEA concluded that the addition of gold and silver recovery to the project improved the overall project economics with respect to gross and net revenues and the cash cost of copper recovery after deduction of the gold and silver credits.

The present PEA study has been developed to examine concepts for the sequential agitated tank leaching of copper, using sulphuric acid, followed by neutralization of the leached mineralized material with lime, and leaching of gold and silver using cyanide. Copper would be recovered from the acid leachate, using solvent extraction and electrowinning to produce an LME Grade A cathode copper. The gold and silver would be recovered from the cyanide leachate using CIL and ADR processes.

6.1 Historical Mineral Resource Estimate

The Carmacks deposit has been subject to several historical tonnage and grade estimations over the years as summarized in Table 6.1. The historical Mineral Resources are presented here to show the progression of development of the Mineral Resources on the property.

Table 6.1: Historical Mineral Resource Estimates for the Carmacks Project

Year	Source	Tonnes	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 cut-off of 0.8% total copper
1993	Western Copper Audited by Kilborn	15,867,140	0.829	1.096	0.014	Measured and Indicated at cut-off of 0.5% total copper
1993	Western Copper Audited by Kilborn	19,062,390	0.725	0.972	0.013	Measured and Indicated at cut-off of 0.01% total copper
1997	Western Copper Audited by Kilborn/SNC	13,300,000	-	0.97	-	Cut-off 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 cut-off of 0.25% total copper

Source: ACS (2016)

The Mineral Resource estimate presented in Table 6.1 have not all been classified in accordance with the CIM approved standards as required in NI 43-101. These estimates have been obtained from sources believed reliable and conform to disclosure standards in use at the time of their publication, but have not all been independently verified. These resource estimates are no longer relevant and should not be relied upon, as they are replaced by the estimate which is presented in Section 14 of this report.

7 Geological Setting and Mineralization

7.1 Regional Geology

The regional geology was described by Bostock in 1936, by Tempelman- Kluit in 1981 and 1985 and in 2013 by Allan et al. The regional geological map based on Yukon Geological Survey data is shown in Figure 7.1. The Carmacks region lies within the Intermontane Belt, which in the Carmacks map-area is divisible into the Yukon Tanana Terrane, Yukon Crystalline Terrane and Whitehorse Trough.

The Whitehorse Trough lies to the east of the Hoochekoo Fault, east of the Carmacks Project. The Whitehorse Trough comprises Upper Triassic intermediate to basic volcanic capped by carbonate reefs (Povoas Formation) and Lower Jurassic greywacke, shale and conglomerate, derived from the underlying Upper Triassic granitic rocks (Laberge Group).

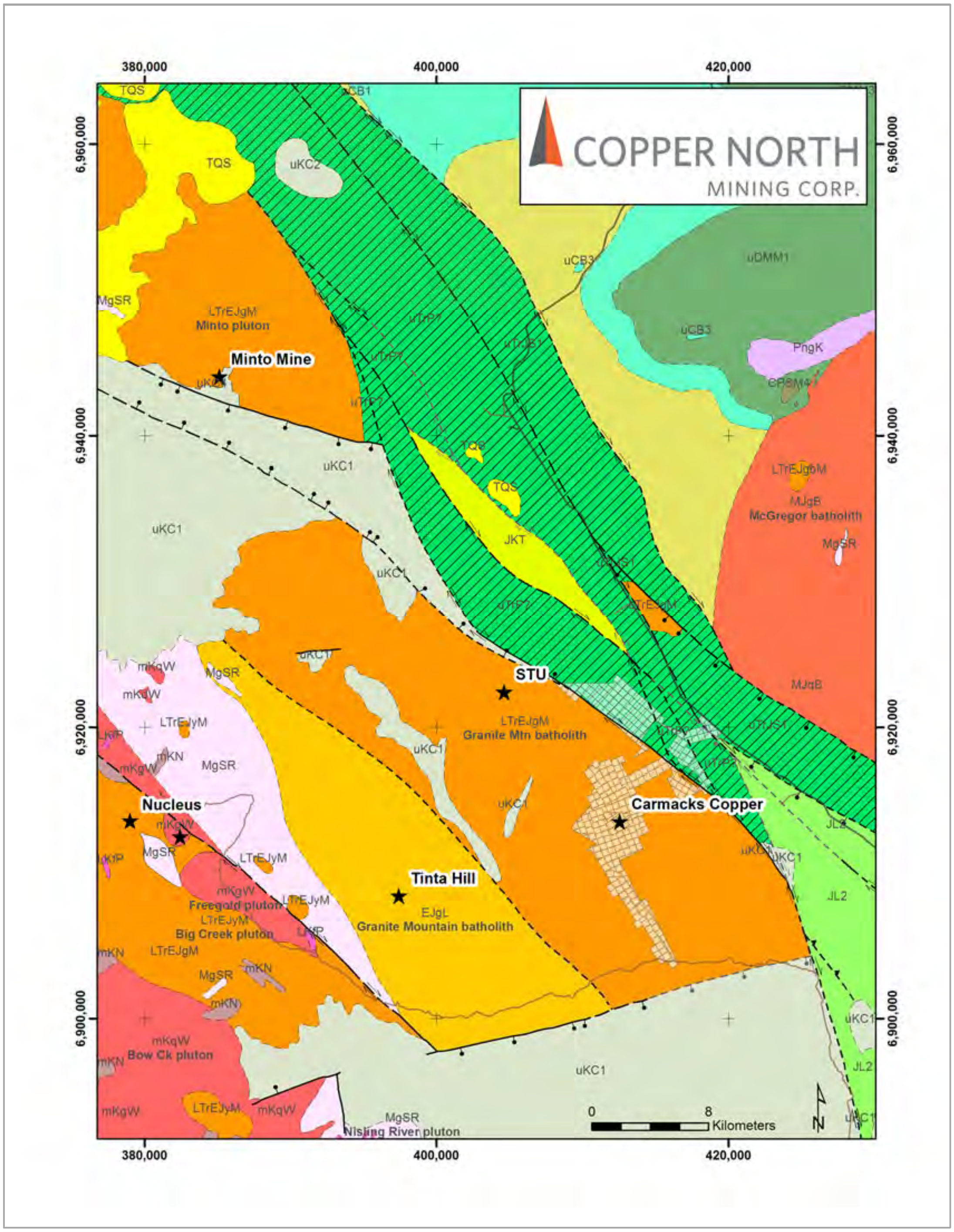
The Yukon Tanana Terrane includes hornblende-biotite-chlorite gneiss with interfoliated biotite granite gneiss, Permian Selwyn Gneiss and is intruded by Early Jurassic Aishihik Suite Granite Mountain Batholith. Weakly foliated, mesocratic, biotite-hornblende, Granite Mountain granodiorite contains screens or pendants of strongly foliated feldspar-biotite-hornblende-quartz amphibolite-gneisses that host the Carmacks Project deposit.

Younger plutonic rocks intrude all three divisions of the Intermontane Belt and the contacts between them. Carmacks Group and Mount Nansen volcanic rocks overlie portions of all older rocks, suggesting that they should not be classified in the Yukon Crystalline Terrane, but are younger rocks that obscure relationships between the older terrane rocks. Tempelman-Kluit (1985) has included the Carmacks Group in the Yukon Crystalline Terrane.

Mesozoic strata of the Whitehorse Trough are only exposed in fault contact with the Yukon Crystalline Terrane and Yukon Tanana Terrane, but may rest depositionally on them or certain of their strata.

The predominant northwest structural trend is represented by the major Hoochekoo, Tatchun and Teslin faults to the east of the Carmacks Project and the Big Creek Fault to the west. East to northeast younger faulting is represented by the major Miller Fault to the south of the Carmacks Project.

Figure 7.1: Regional Geologic Map



Source: CNMC (2016)

Bedrock Geology

TERTIARY(?) AND QUATERNARY

TQS: SELKIRK: resistant, brown weathering, columnar jointed, vesicular to massive basalt flows; minor pillow basalt; basaltic tuff and breccia (Selkirk Volcanics)

LATE CRETACEOUS TO TERTIARY

LKfP: PROSPECTOR MOUNTAIN SUITE: quartz-feldspar porphyry

MID-CRETACEOUS

mKdW: WHITEHORSE SUITE: hornblende diorite, biotite-hornblende quartz diorite and mesocratic, often strongly magnetic, hypersthene-hornblende diorite, quartz diorite and gabbro (Whitehorse Suite, Coast Intrusions)

mKgW: WHITEHORSE SUITE: biotite-hornblende granodiorite, hornblende quartz diorite and hornblende diorite; leucocratic, biotite hornblende granodiorite locally with sparse grey and pink potassium feldspar phenocrysts (Whitehorse Suite, Casino granodiorite, McClintock granodiodrite, Nisling Range granodiorite)

mKqW: WHITEHORSE SUITE: biotite quartz-monzonite, biotite granite and leucogranite, pink granophyric quartz monzonite, porphyritic biotite leucogranite, locally porphyritic (K-feldspar) hornblende monzonite to syenite, and locally porphyritic leucocratic quartz monzonite (Mt. McIntyre Suite, Whitehorse Suite, Casino Intrusions, Mt. Ward Granite, Coffee Creek Granite)

mKN: MOUNT NANSEN: massive aphyric or feldspar-phyric andesite to dacite flows, breccia and tuff; massive, heterolithic, quartz- and feldspar-phyric, felsic lapilli tuff; flow-banded quartz-phyric rhyolite and quartz-feldspar porphyry plugs, dykes, sills and breccia (Mount Nansen Gp., Byng Creek Volcanics, Hutshi Gp.)

UPPER CRETACEOUS

uKC1: CARMACKS: augite olivine basalt and breccia; hornblende feldspar porphyry andesite and dacite flows; vesicular, augite phyric andesite and trachyte; minor sandy tuff, granite boulder conglomerate, agglomerate and associated epiclastic rocks (Carmacks Gp., Little Ridge Volcanics, Casino Volcanics)

uKC2: CARMACKS: andesite

uKC4: CARMACKS: medium-bedded, poorly sorted, coarse- to fine-grained sandstone, pebble conglomerate, shale, tuff, and coal; massive to thick bedded locally derived granite or quartzite pebble to boulder conglomerate (Carmacks Gp.)

UPPER JURASSIC AND LOWER CRETACEOUS

JKT: TANTALUS: massive to thickly bedded chert pebble conglomerate and gritty quartz-chert-feldspar sandstone; interbedded dark grey shale, argillite, siltstone, arkose and coal; at one locality includes red-weathering dacite to andesite flows at base (Tantalus)

MID-JURASSIC

MJqB: BRYDE SUITE: medium to fine grained, equigranular, leucocratic monzonite, syenite and granite and related dykes of dacite to andesite porphyry with euhedral andesine, hornblende and locally quartz in aphanitic greenish, or grey groundmass (Teslin Crossing Stock)

MJgB: BRYDE SUITE: medium grained, hornblende monzodiorite, hornblende-biotite quartz monzodiorite and minor hornblendite; pink, potassium feldspar megacrystic, hornblende granite to granodiorite and associated easterly trending mafic dyke swarms (Mt. Bryde Pluton; Bennett Granite)

EARLY JURASSIC

LTrEJyM: MINTO SUITE: syenite

LTrEJgM: MINTO SUITE: medium- to coarse- grained, variably foliated to massive biotite-hornblende granodiorite; biotite-rich screens and gneissic schlieren; foliated hornblende diorite to monzodiorite with local K-feldspar megacrysts (Minto Suite)

LTrEJgbM: MINTO SUITE: gabbro

EJgL: LONG LAKE SUITE: massive to weakly foliated, fine to coarse grained biotite, biotite-muscovite and biotite-hornblende quartz monzonite to granite, including abundant pegmatite and aplite phases; commonly K-feldspar megacrystic (Long Lake Suite)

LOWER AND MIDDLE JURASSIC, HETTANGIAN TO BAJOCIAN

JL2: TANGLEFOOT:

UPPER TRIASSIC, CARNIAN AND OLDER (?)

uTrP?: POVOAS: augite or feldspar phyric, locally pillowed andesitic basalt flows, breccia, tuff, sandstone and argillite; local dacitic breccia and tuff with minor limestone; greenschist, chlorite schist, chlorite-augite-feldspar gneiss, amphibolite (Povoas)

UPPER TRIASSIC TO LOWER JURASSIC

uTrJS1: SEMENOF:

LATE DEVONIAN TO MISSISSIPPIAN

MgSR: Simpson Range - tonalite, diorite

UPPER CARBONIFEROUS, LOWER AND MIDDLE PENNSYLVANIAN

PngK: KELLY STOCK: tonalite orthogneiss

uCB1: BOSWELL: recessive, dark weathering, slate, phyllite, greywacke chert, chert conglomerate and breccia, volcanic breccia, greenstone and limestone (Boswell)

uCB3: BOSWELL: massive, dark weathering, coarse to medium grained, hornblendite-gabbro (Boswell)

uDMM3: Moose - interm. volc.

uDMM1: MOOSE: basalt, greenstone

CARBONIFEROUS TO PERMIAN

CPSM4: SLIDE MOUNTAIN: ultramafic

7.2 Property Geology

Most of the geological information for the project comes from geophysics, drill core and trenches, as there is only limited outcrop on the property found along spines on the ridges and hill tops. Float, derived locally because the area was not glaciated by continental glaciation, can be seen in the old trenches on the property and along the cuts of the drill roads.

The Carmacks copper-gold-silver deposit is enclosed within the Early Jurassic Granite Mountain Batholith. The copper mineralization is hosted by amphibolite, gneisses, and intrusive rocks that range from granodiorite to diorite. Copper mineralization occurs along a linear trend, following a brittle-ductile deformation zone.

The deposit is subdivided into several zones (Figure 7.2), each comprising a tabular raft of amphibolite-gneisses that dip steeply to the east and are up to 100 metres wide, strike up to 700 m and persist down dip to at least 450 m, being open at depth. Exploration has identified at least 14 mineralized zones comprising steep easterly dipping zones that occur along a strike length of at least 5 km. The discoveries also include local zones of mineralization that appear sub-parallel to the main mineralized structure. The rafts of copper bearing amphibolite-gneisses are enclosed within a younger granodiorite batholith as roof pendants or partially digested rafts. The copper mineralization at depth comprises copper sulphides bornite and chalcopyrite. Gold and silver accompany the copper mineralization; higher gold grades are associated with the more bornite-rich areas.

The typical host rock for the hypogene mineralization is a dark grey to black hornblende-biotite amphibolite with a pervasive foliation. The amphibolite varies from massive to bearing relict hornblende phenocrysts (or hornblende after pyroxene) and may represent variation in the, possibly volcanic, protolith. Locally, the amphibolite becomes more gneissic where mineralogical and colour segregation occurs. The content of mafic minerals is variable from ~50% to ~100%. Locally, the amphibolite lacks a penetrative fabric and appears to have recrystallized to microdiorite from the heat of the adjacent granodiorite intrusions. Sulphide mineralization in the amphibolite is typically foliaform with some discordant sulphide veinlets. Diorite is also host to sulphide mineralization, where chalcopyrite and bornite occur interstitially between hornblende crystals as a net-texture. Alteration phases include proximal potassic (K- spar-Bt) alteration and hematization.

Deformation is seen to increase toward the mineralized zones, suggesting that an underlying structure may be a control on the mineralization. There is a complex magmatic-deformation history involving multiple phases of granitoid intrusions, boudinage and faulting. There are at least two stages of pegmatite-aplite intrusions, each associated with epidote alteration.

The mineralization is cross-cut by barren late phases of the Granite Mountain Batholith including K-feldspar porphyritic granodiorite, aplite and pegmatite. The porphyritic phases contain phenocrysts of K-(potassium) feldspar, plagioclase and/or quartz. In some instances, the K-feldspar phenocrysts range up to 3 cm long. Post-mineralization granitic pegmatite and aplite dykes are widespread in the area and range from a few centimetres to approximately three metres in thickness. Hornblende is present in dioritic intrusive rocks and locally in the granodioritic phases. Quartz, K-feldspar and plagioclase are present in all intrusive phases. Plagioclase is subhedral and very locally displays growth zoning.

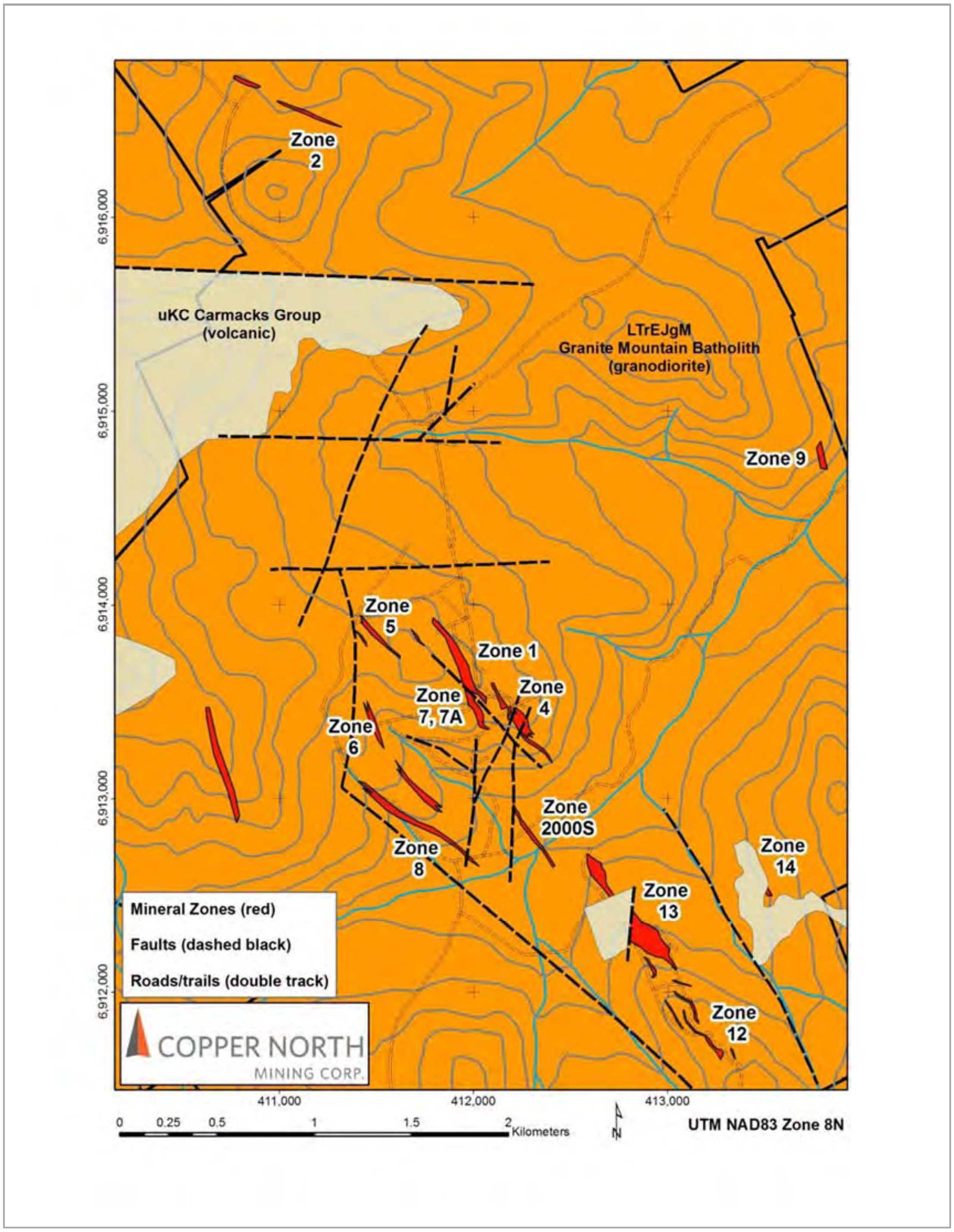
Petrographic examination indicates Granite Mountain granodiorites have a varied mineralogical content with areas of silica under-saturation and plagioclase oversaturation. These variations may be the result of the assimilation of precursor rock to the amphibolite-gneiss units.

The combined strike length from the northern end of Zone 1 to the southern tip of Zone 12 is just over 2 km. The character of the deposit changes along strike leading to a division into northern and southern halves. The northern half is more regular in thickness, dip angle, width and down dip characteristics. The southern half splays into irregular intercalations, in Zones 7 and 7A, terminating against sub-parallel faults down dip.

Zones 12 and 13 are located 1.2 km south of Zone 1 and occur over a strike length of1.2 km and up to 100 m in width. The mineralization in Zones 12 and 13 is hosted by less mafic amphibolite and gneisses than those found in Zone 1. The gneisses are highly silicified and K-feldspar altered; the gneissic texture may be the result of alteration along closely spaced parallel planes, rather than the product of high strain. The gap between Zones 12 and 13 has not been drill tested and it is unclear as to whether mineralization is continuous between the two zones. In Zone 12, the mineral zones bifurcate and split into several parallel zones and are affected by post-mineralization faulting.

The Carmacks Group is a late Cretaceous, post-mineralization sequence of andesitic- basaltic volcanic rocks and basal conglomerates and sandstones. The Carmacks Group is present in across the property in several areas, but most prominently affects mineralization in Zones 13 and 14 where it forms a fault-bounded segment of cover rocks. Thin mafic dykes that were feeders for Carmacks Group volcanic are uncommon.

Figure 7.2: Property Geologic Map



Source: CNMC (2016)

7.3 Mineralization

Deep oxidation of the deposit has led to the formation of an oxide cap that can be over 200 m thick. The majority of the copper found in oxide are in the forms of the secondary minerals malachite, cuprite, azurite and tenorite (copper limonite) with very 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. Primary copper mineralization, occurring below the oxidized level, is present as bornite and or chalcopyrite. Other primary minerals include magnetite, gold, molybdenite, native bismuth, bismuthinite, arsenopyrite, pyrite, pyrrhotite, and carbonate. Molybdenite, visible gold, native bismuth, bismuthinite, and arsenopyrite occur rarely.

The copper in the upper 200 m in Zones 1, 4, 7, and 7A is oxidized; whereas in Zones 13 and 12 the oxide cap can be as thin as ~40 m. Within the oxidized area, pyrite is virtually absent and pyrrhotite is absent. Weathering 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. Primary sulphide minerals and magnetite are disseminated and form narrow massive bands or heavy disseminations in bands. Gypsum occurs as microveinlets. Carbonate occurs as pervasive matter, irregular patches, or microveinlets, not commonly but on the order of 1% where present. 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.

Primary (hypogene) copper mineralization appears to be associated primarily with the amphibolite-gneiss units and the early-formed diorites, whereas secondary copper mineralization does not appear to be preferential to a particular rock type. This is owing to the remobilization of copper during supergene processes. In the north half of Zone 1, copper mineralization forms high and low grade zones that are reasonably consistent, both along strike and down dip, and these zones are broadly constrained to the deformed rocks and diorites, but transcend local lithological boundaries. Higher grades tend to form a footwall zone, while lower grades form a hanging wall zone. Primary mineralization, below the zone of oxidation comprises mainly of chalcopyrite, bornite, molybdenite, magnetite, pyrite and pyrrhotite. Primary copper mineralization appears to be zoned from bornite on the north to chalcopyrite, and finally to minor pyrite-pyrrhotite in the south. Narrow veinlets of anhydride were found in the deepest drill hole.

Alteration minerals associated with the mineralizing include K-feldspar and biotite. Epidotization and some K-feldspar are related to pegmatite dyke intrusion, which is a post-mineralization event. Clay (montmorillonite type) and sericite development are clearly weathering products. Silica introduction, usually as narrow veinlets, is not common and may be related to aplite dyking or metasomatism. Chloritization of mafics, biotitization of hornblende, rare garnets, carbonate, and possibly anhydrite all appear related to metasomatism and assimilation of precursor rocks to the gneissic units.

In Zone 1, oxide copper grades increase with depth in both the footwall and hanging wall. There is no association of copper values with mafic mineral content, or grain size. Gold values are higher in the north half of the deposit. They average 0.75 g/t compared with 0.27 g/t in the south half. There is no apparent increase in values with depth and the highest grade gold values are not associated with the highest copper values; however, gold values in the northern half are higher in the footwall section. This lack of increase in gold values with depth suggests that the gold distribution reflects a primary distribution rather than a secondary distribution such as oxide copper values. As with oxide copper, the gold content does not correlate with rock type, mafic constituents or grain size.

8 Deposit Types

The Carmacks copper-gold deposit is similar to the Minto deposit, located 50 km to the northwest (Sinclair, 1976; Pearson, 1977), except that the Minto deposit is flat lying and primarily a sulphide deposit. A number of theories for the genesis of the Carmacks deposit have been postulated over the years and by different operators. The Cu-Au- Ag metal tenor and association with Late Triassic-Early Jurassic granodiorites would appear to suggest a link between mineralization at Carmacks and the porphyry copper deposits of the same age that occur across British Columbia. However, the linear deposit shape, lack of mineralized material-stage veining and lack of porphyry alteration show clearly that the Carmacks copper deposit is not a classic porphyry system. Evidence from the drilling campaigns suggests the deposit was formed by assimilation of older, copper bearing volcano-sedimentary rocks into the Jurassic Granite Batholith. These “rafts” of mineralized rock would have been variably metamorphosed, and in places completely assimilated into the granodiorite. The volcano-sedimentary rafts would tend to pull apart along bedding planes forming large tabular sheets as observed in Zones 1, 4, 7, 7A, 8, 12, 13 and 2000S. Evidence suggests the sulphide mineralization has been remobilized out of the rafts into the surrounding diorite. At a later time, when the upper parts of the batholith where eroded and the rocks were exposed to the atmosphere and meteoric waters, the sulphide mineralization began to oxidize and precipitate as the oxide minerals.

The Minto deposit is owned by Capstone Mining Corporation and began production in June 2007. The Minto deposit has been interpreted as either a metamorphosed stratiform sedimentary copper deposit or a metamorphosed porphyry copper deposit.

9 Exploration

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 mode of exploration has been trenching. Zones 1, 4, 7 and 7A Zones have been trenched at 61m (200-foot) spacing. All trenches across Zone 1 were channel sampled with 1.52 m or 3.05m (5 or 10 foot) sample lengths. Trenches parallel to the zone were not sampled.

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 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.

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 2006, Western Copper Corp. conducted 7,100 m of diamond drilling in 34 holes, 1,235 m of rotary air blast drilling in 61 holes, and re-initiated environmental baseline studies. In 2007, Western Copper Corp. continued the exploration and environmental sampling program and conducted geotechnical studies of the proposed heap leach pad, WRSA, processing plant and camp location. The 2007 program consisted of 17,845 m of diamond drilling in 123 holes, 866 m of geotechnical drilling in 36 holes, 32-line km of induced polarization surveys and the surveying of all drill hole locations including all the historic drill holes, geotechnical holes, and rapid air blast drill holes.

The surveying was conducted by Lamerton and Associates of Whitehorse, Yukon and was performed by Differential GPS. The hole markers at a few of the historic drill holes were destroyed during later road building, trenching or drill pad construction and these sites were located approximately. The accuracy of the post processed survey points is estimated at approximately 20 mm.

In 2008, Western Copper drilled 12 geotechnical holes (1,923 m) in the pit area, two water wells in the camp area (253.5 m), two water monitoring well below the heap leach pad (206 m), and conducted a small soil geochemical sampling program.

10 Drilling

Extensive drilling has been carried out on the Carmacks Project prior to the involvement by Copper North. The majority of the drilling on the property was carried out in the 1970s and in 2006-2007. CNMC carried out drilling campaigns in 2014 and 2015.

10.1 Historical Drill Programs

Prior to 2006, a total of 77 DDH and 11 reverse circulation holes, amounting to approximately 12,400 m of drilling, were drilled in the exploration of the property. 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.

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.

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 2008 program was designed to complete the geotechnical studies initiated in 2007. Table 10.1 summarizes the historical drilling on the Carmacks project.

Table 10.1: Summary of Historical Drilling Carmacks Project

Year	Hole Type	No. holes	metres	Company
1971	DD	25	5,290	Historic
1972	DD	8	631	Historic
1991	DD	36	4,461	Western Copper
1992	DD	8	1,191	Western Copper
1992	RC	11	856	Western Copper
1995	GEOT	10	185	Western Copper
1996	GEOT	12	489	Western Copper
2006	DD	34	7,101	Western Copper
2006	RAB	61	1,235	Western Copper
2007	DD	123	17,845	Western Copper
2007	GEOT	36	866	Western Copper
2008	GEOT	12	1,923	Western Copper
2008	Water	4	460	Western Copper
	Total	**380**	**42,533**	

Source: ACS (2016)

10.2 Copper North Drilling

In 2014, Copper North initiated a diamond drilling program aimed at defining additional mineralization in Zones 2, 2000S, 12 and 13. 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 (Figure 7.2 above).

Drilling was carried out by Kluane Drilling of Whitehorse, Yukon using a custom designed drill rig. Core size was NQ for the 2014 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 10.2: Summary of Copper North 2014-2015 Drilling Programs

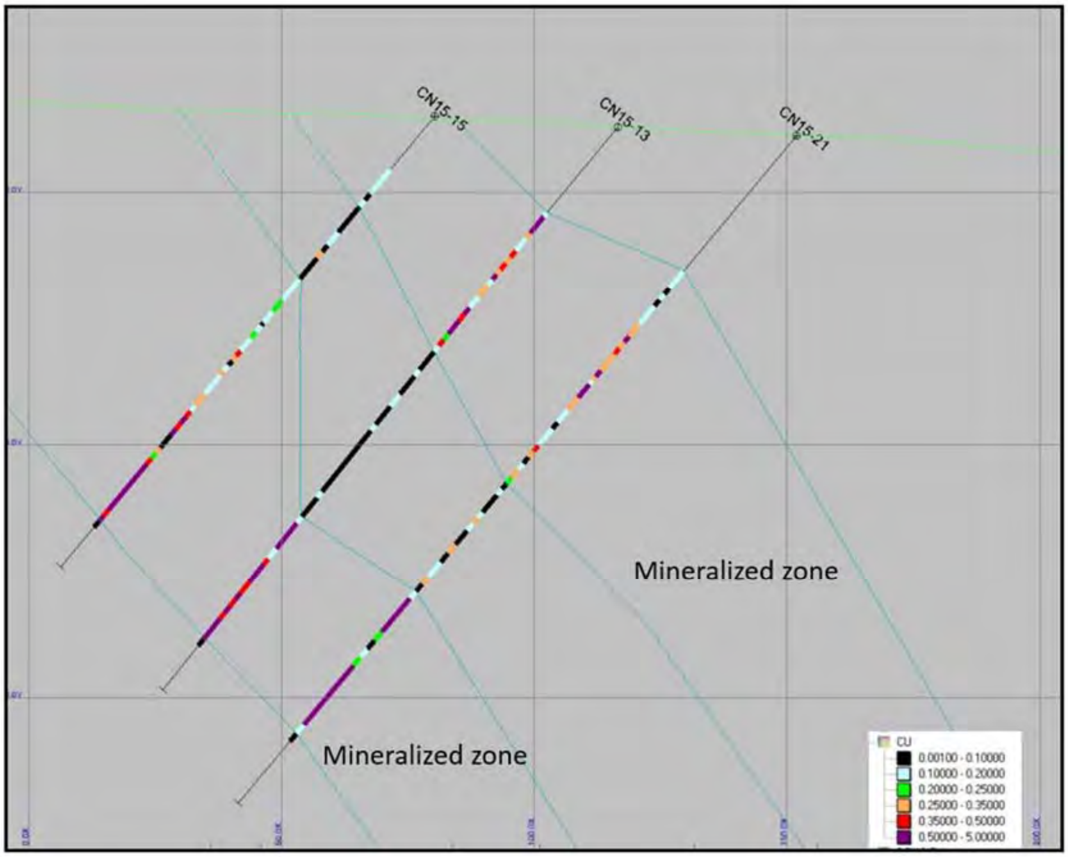
Zone Targeted	Number Of Holes	Total Metres
12	6	394.52
13	20	1932.26
1/4/2007	1	88.39
2	10	619.57
2000S	12	1195.07
Exploration	1	128.02
Grand Total	**50**	**4357.83**

Source: ACS (2016)

10.3 Sample Length/True Thickness

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. Figure 10.1 shows a typical drill section of the Copper North drilling in Zone 13.

Figure 10.1: Cross Section of Copper North Drilling in Zone 13 looking Northwest



Note: Grid lines are 50 m apart
Source: ACS (2016)

11 Sample Preparation, Analyses and Security

11.1 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.

11.1.1 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 (Casselman, 2007). There has been no indication from either of the labs that samples or shipments had been tampered with.

11.1.2 Copper North

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

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 $HClO_4$ – HNO_3 digestion followed by Atomic Absorption Spectrometry (AAS) with a 0.01% detection limit. Non-sulphide copper was assayed by dilute H_2SO_4 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.

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-HNO_3-$HClO_4$-HCl) digestion and atomic absorption according to procedure Cu-AA62 non-sulphide copper by sulphuric acid leach and AAS according to procedure Cu-AA05.

11.2.3 Copper North Mining Company (CNMC)

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.

ALS laboratories are registered or are pending registration to ISO 9001:2008. ALS Whitehorse and Vancouver analytical facilities have received ISO 17025 accreditations.

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.

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-HNO_3-$HClO_4$-HCl) digestion and ICP-ES according to procedure 7TD and, for non-sulphide copper, by sulphuric acid leach and AAS.

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) (Table 11.1). 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.

Table 11.1: Expected Value and Standard Deviation for SRM used in 2006-2007 Drilling Programs

SRM	Au (g/t)	CuOx (%)	CuT (%)
AGL-1	0.6	1.616	1.713
AGL-2	0.45	0.885	0.913
AGL-3	0.05	0.04	0.05
Standard deviation	Au (g/t)	CuOx (%)	CuT (%)
AGL-1	0.041	0.061	0.06
AGL-2	0.02	0.039	0.03
AGL-3	0.003	0.003	0

Source: ACS (2016)

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 locations and orientations of the holes are listed in Table 11.2 below.

Table 11.2: Coordinates of Twin Drill Holes

Hole	NAD83UTM E (m)	NAD83UTM N (m)	Az True (°)	Dip (°)
140	411878	6913907	248.5	-50
WC-003	411875	6913902	245	-50
141	411902	6913855	248.5	-50
WC-004	411905	6913857	245	-50

Source: ACS (2016)

A comparison between the historical and current assay results can be found in Table 11.3 below. The hanging wall and footwall contacts were well defined in all four drill holes. The lengths of the intercepts listed in the table are from the hanging wall contact to the footwall. There were well-mineralized intersections below the footwall contact in all four holes, but these were not used in the comparison below.

Table 11.3: Comparison of Check Drilling and Historical Drilling

	140		WC-003		Difference (%) (new-old)	141		WC-004		Difference (%) (old-new)
	Total Cu	CuOx	Total Cu	CuOx		Total Cu	CuOx	Total Cu	CuOx	
Length (m)	39.6	39.6	39	39	-1.54%	48.8	48.8	48	48	-1.67%
Average (%Cu)	1.24	0.84	1.67	0.97	+15.77% (CuOx)	1.23	0.98	1.13	0.99	1% (CuOx)
SD (%)	0.7	0.5	0.87	0.44		1.45	1.05	0.94	0.87	
Var (%)	0.59	0.41	0.7	0.34		0.91	0.66	0.65	0.59	

Source: ACS (2016)

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. Results are presented in Table 11.4. The sample handling, shipping, and preparation control procedures followed were the same as those employed for the 2006 diamond drill program.

Table 11.4: Comparison of 2006 Twin Drilling with Historical Drilling

Hole No.	1991 Sample Intervals						2006 One-Metre Re-Assays					
	From (m)	To (m)	Length (m)	CuOx (%)	Total Cu (%)	Au (ppm)	From (m)	To (m)	Length (m)	CuOx (%)	Total Cu (%)	Au (ppm)
122	38.4	42.06	3.66	0.77	1.6	1.1	39.92	40.84	0.92	0.51	1.32	0.748
127	34.75	37.8	3.05	2.95	3.11	0.34	36.88	37.79	0.91	2.43	2.8	0.289
128	23.77	26.82	3.05	1.61	1.72	0.41	24.68	25.6	0.92	3	3.34	1.925
132	50.9	53.95	3.05	1.81	2.02	0.07	51.81	52.7	0.89	2.93	3.25	0.25
135	77.42	80.47	3.05	1.82	1.96	0.27	77.41	78.33	0.92	3.14	3.54	0.296
138	117.81	119.18	1.37	1.12	1.2	0.55	118.56	119.48	0.92	0.93	1.04	0.399
150	64.53	67	2.47	0.9	1	0.07	64.31	65.22	0.91	0.9	1.14	0.454
156	54.86	57.91	3.05	1.86	1.9	0.45	54.86	55.77	0.91	1.28	1.39	0.944
157	79.25	81.69	2.44	1.2	1.33	3.63	78.94	79.85	0.91	0.81	1.03	0.181
158	88.39	91.44	3.05	0.18	0.19	0	88.39	89.3	0.91	0.37	0.42	0.013
Average				**1.42**	**1.6**	**0.689**	**Average**			**1.63**	**1.93**	**0.55**

Source: ACS (2016)

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.

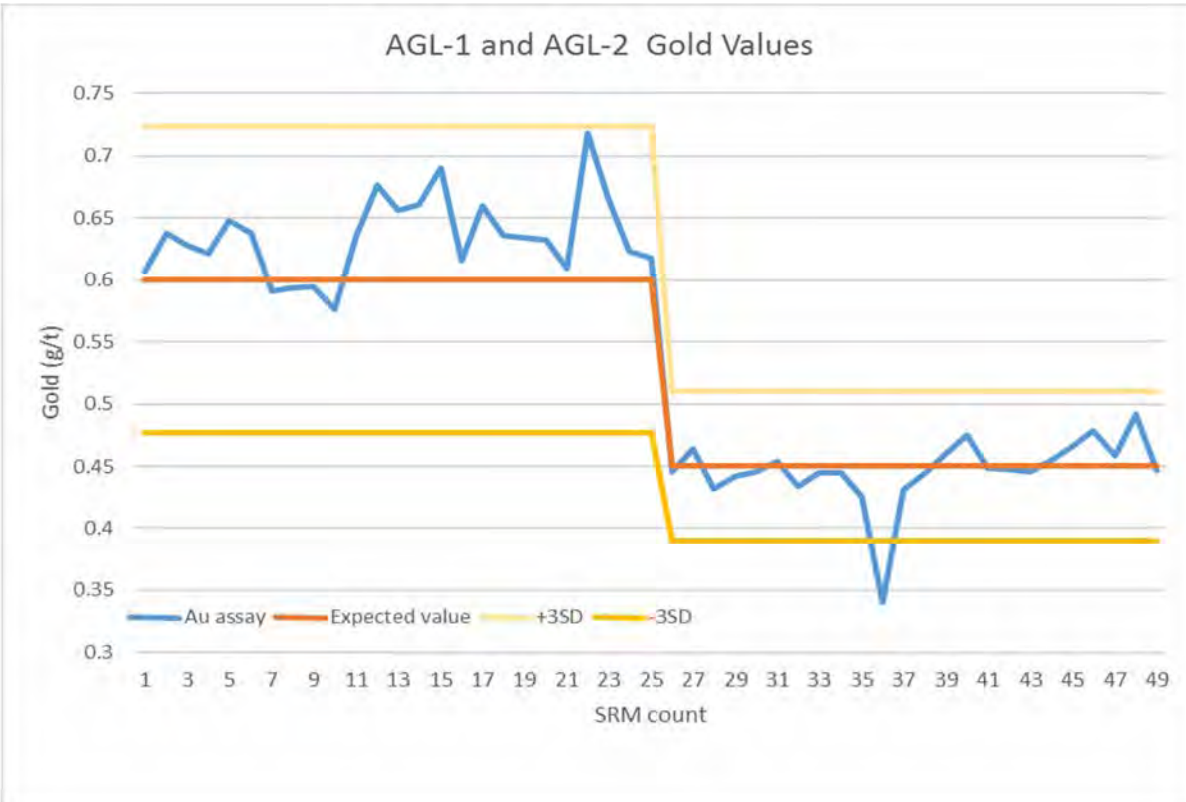
11.3.3 Copper North

CNMC collected a total of 1,349 samples as part of the 2014-2015 drilling programs. CNMC 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.

ACS reviewed the SRM, blanks, and duplicate sampling and found them to be acceptable. Figure 11.1 to Figure 11.3 show the results of AGL-1 and ALG-2 for gold, soluble copper and total copper assays. Only one gold value falls slightly outside of the standard deviation line, all other assays are well within the acceptable limits. ACS did note that the values for soluble copper all seem to report lower than the expected value for both standards, possibly indicating a degradation of the SRM material.

Figure 11.1: Gold Assay Results for SRM AGL-1 and AGL-2



Source: ACS (2016)

Figure 11.2: Soluble Copper Assay Results for SRM AGL-1 and AGL-2

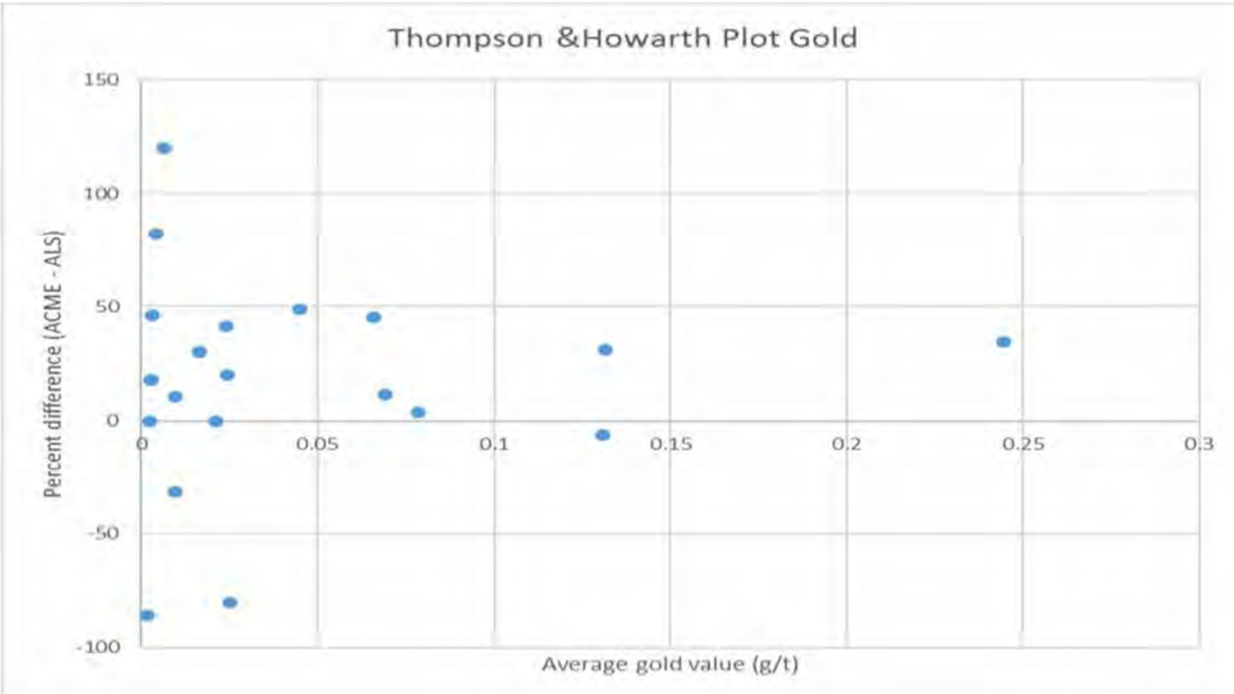
Source: ACS (2016)

Figure 11.3: Total Copper Assay Results for SRM AGL-1 and AGL-2

Source: ACS (2016)

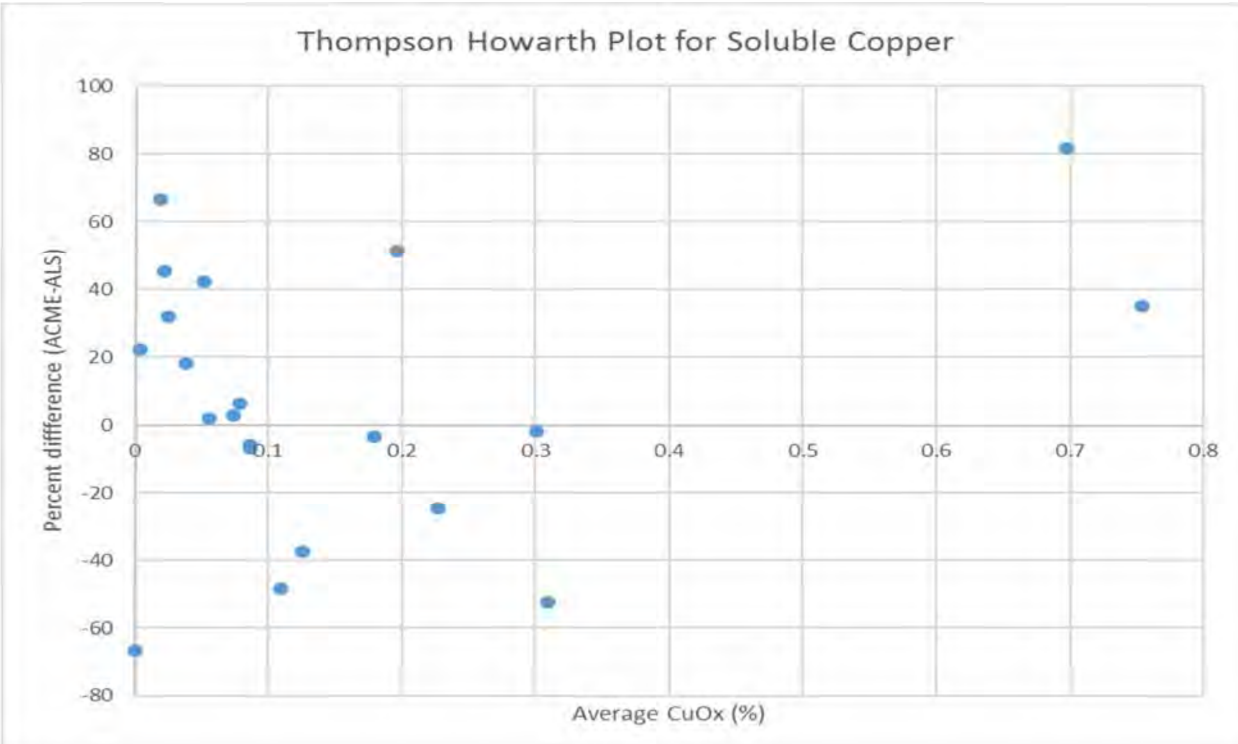
The review of the duplicate sampling indicated that there is no significant bias associated with the assay data provided by ALS. ACS did note that the very low grade gold values were slightly higher at Acme than at ALS, but this difference is not indicative of any significant bias (Figure 11.4). Both 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 (Figure 11.5 and Figure 11.6).

Figure 11.4: Thompson Howarth Plot for Gold Duplicate Samples



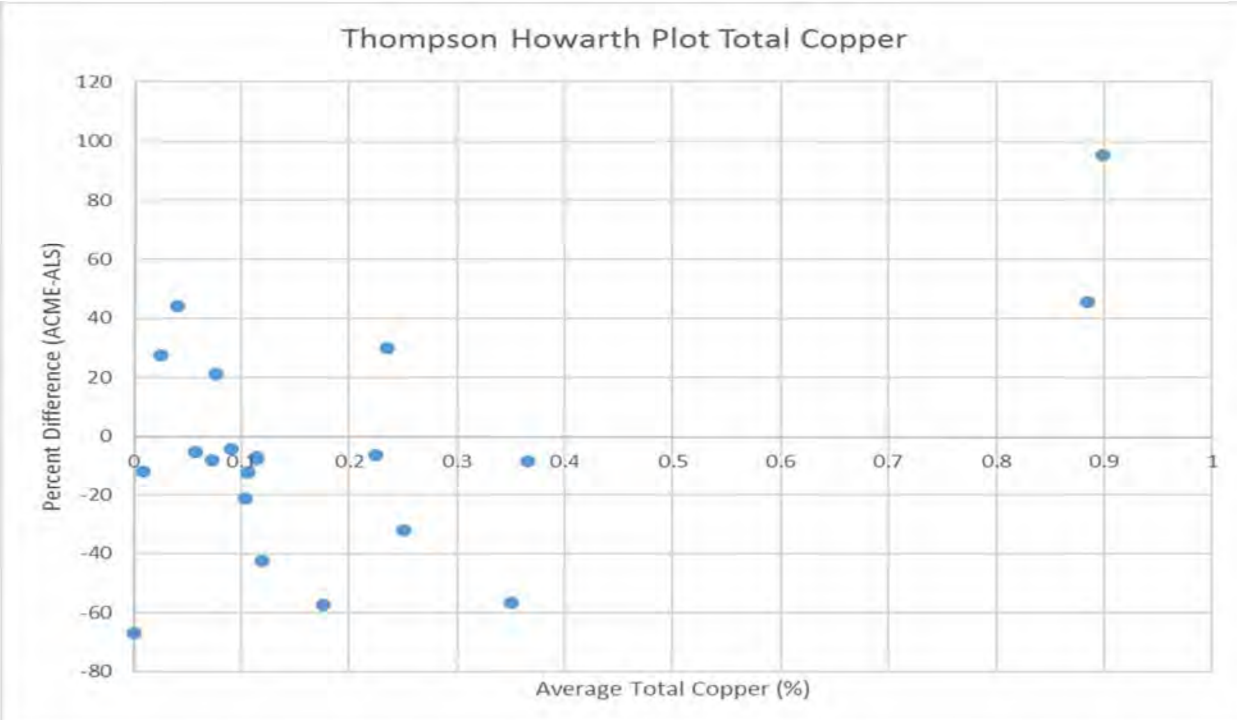
Source: ACS (2016)

Figure 11.5: Thompson Howarth Plot for Soluble Copper Duplicate Samples



Source: ACS (2016)

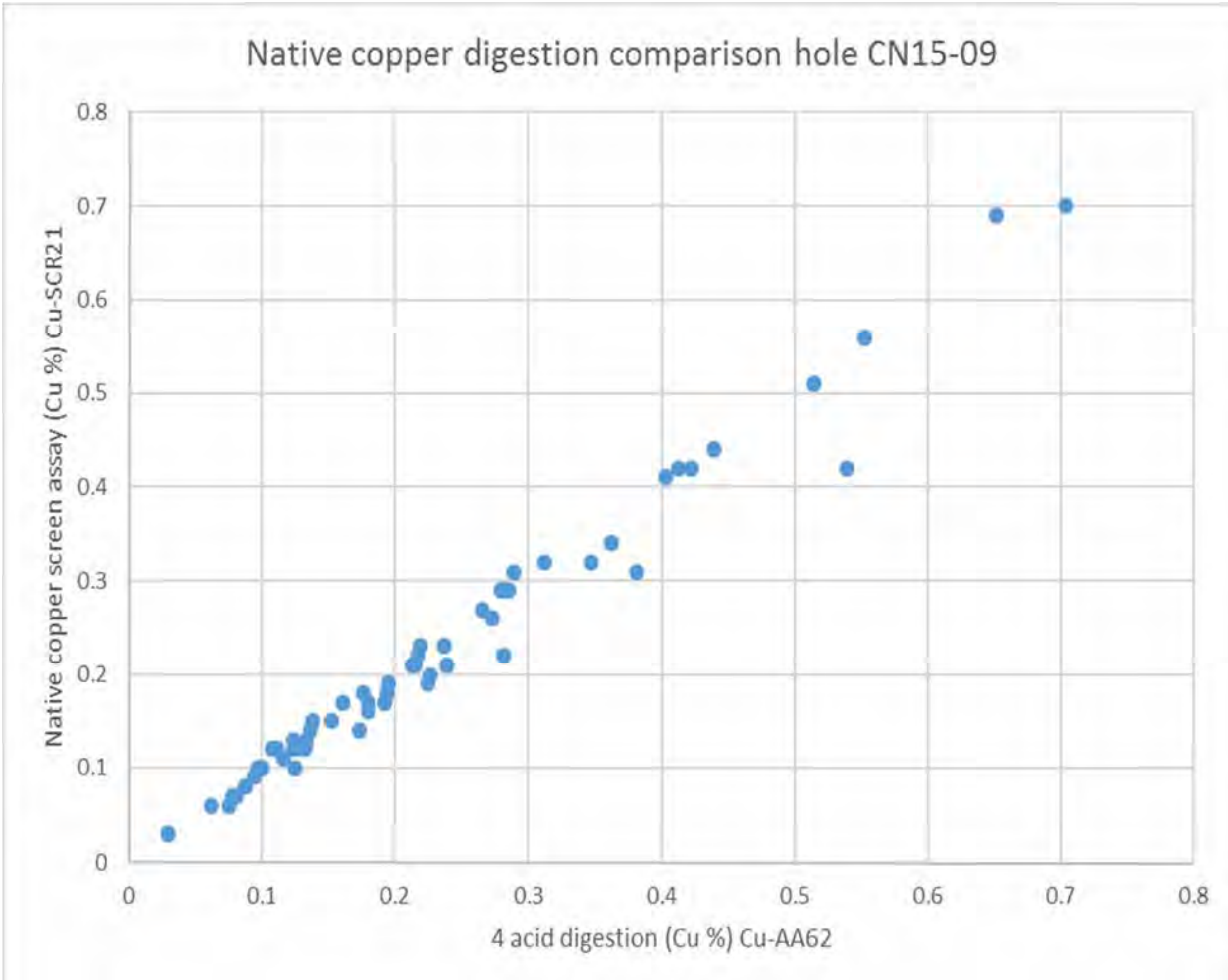
Figure 11.6: Thompson Howarth Plot for Total Copper Duplicate Samples



Source: ACS (2016)

CNMC 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 (Figure 11.7).

Figure 11.7: Comparison of Screen Copper Assays With Four-Acid Digestion Results



Source: ACS (2016)

11.4 ACS Comments

ACS is of the opinion that the sample preparation, analytical procedures and sample security followed by CNMC, Western Copper, and previous operators were adequate for inclusion in the resource estimation.

12 Data Verification

Gilles Arseneau of ACS carried out two visits to the Carmacks Project, the first visit was on the 16th and 17th of May 2007 as part of a Mineral Resource estimation prepared for Zones 1, 4 and 7. As part of this earlier Mineral Resource estimate, Dr. Arseneau carried out the following data verifications.

The integrity of the digital assay data was verified by checking 69% of the database records against the original electronic assay certificates. Assay records from 53 drill holes were verified and a total of eight data entry errors were found as a result of the check. All of the discrepancies found were negligible based on their low-grade values. All errors were corrected in the digital database. Collar coordinates were checked against the database entries. No discrepancies were observed. It was concluded that the assay and survey database was sufficiently free of error to be adequate for resource estimation of the Carmacks deposit.

In 2007, Dr. Arseneau also collected three representative samples from surface trenches. The samples contain visible copper oxide mineralization and appeared representative of the oxide mineralization of Zone 1 oxide at Carmacks. Results of the samples collected are shown in Table 12.1.

Table 12.1: Assay Results of Representative Samples of Zone 1

Sample No	Description	Total Cu (%)
C048024	Trench 1 grab sample	2.09
C048025	Trench 1 grab sample	1.08
C048026	Trench 1 grab sample	2.16

Source: ACS (2016)

These samples were assayed by ICP at ALS Chemex in North Vancouver. The purpose of the sample was to demonstrate that copper mineralization was present on the property in the range of values that had been previously reported by past exploration programs.

As part of the Mineral Resource estimate prepared in 2007, Dr. Arseneau compiled the standards submitted with each batch of samples for the 2007 program. The review showed that the greatest variability occurred with the gold and copper values in the high grade standard, which can be expected due to the potential for the nugget effect from such a high grade sample. These results are considered acceptable. Table 12.2 lists the statistical results from the standards analyses from both Chemex and Acme.

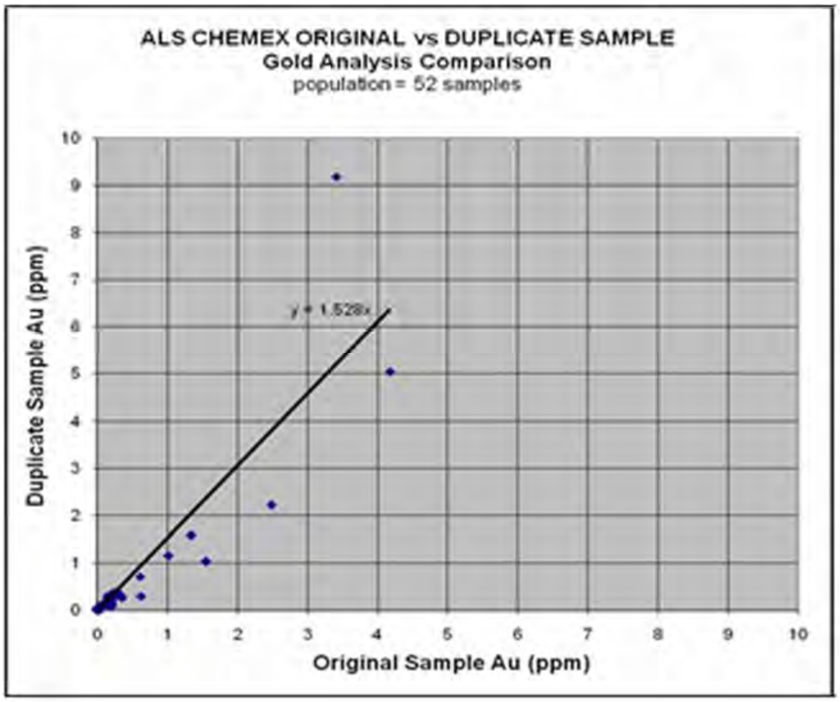
Table 12.2: Analytical Results of 2007 Standard Reference Material

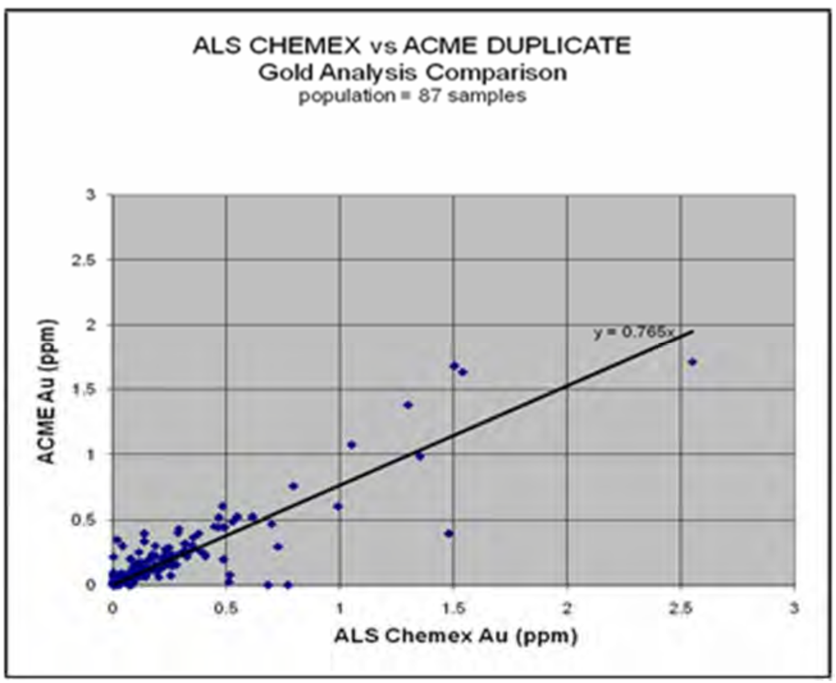
	AGL-1 (high grade Cu)			AGL-2 (moderate grade Cu)			AGL-3 (blank)		
	Au (ppm)	CuOx (%)	Total Cu (%)	Au (ppm)	CuOx (%)	Total Cu (%)	Au (ppm)	CuOx (%)	Total Cu (%)
Certification Value	0.60	1.616	1.713	0.45	0.885	0.913	0.05	0.04	0.05
Maximum	0.713	1.711	1.96	0.495	0.935	0.98	0.021	0.025	0.02
Minimum	0.531	1.430	1.64	0.391	0.754	0.82	0.004	0.006	0.01
Standard Deviation	0.041	0.061	0.06	0.020	0.039	0.03	0.003	0.003	0.00

Source: ACS (2016)

The duplicate samples submitted in 2007 returned generally acceptable values. Figure 12.1 to Figure 12.5 show the results of the comparisons between original samples and duplicate samples, submitted to ALS Chemex and between original samples submitted to ALS Chemex and duplicate samples submitted to Acme for gold, oxide copper, and total copper analyses.

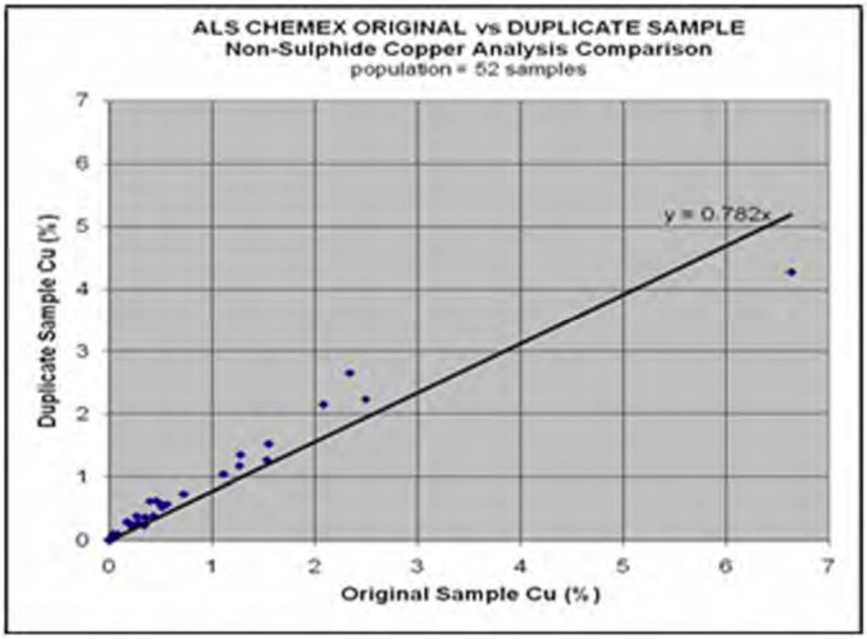
Figure 12.1: Original ALS Chemex versus ALS Chemex Duplicate Gold Analysis





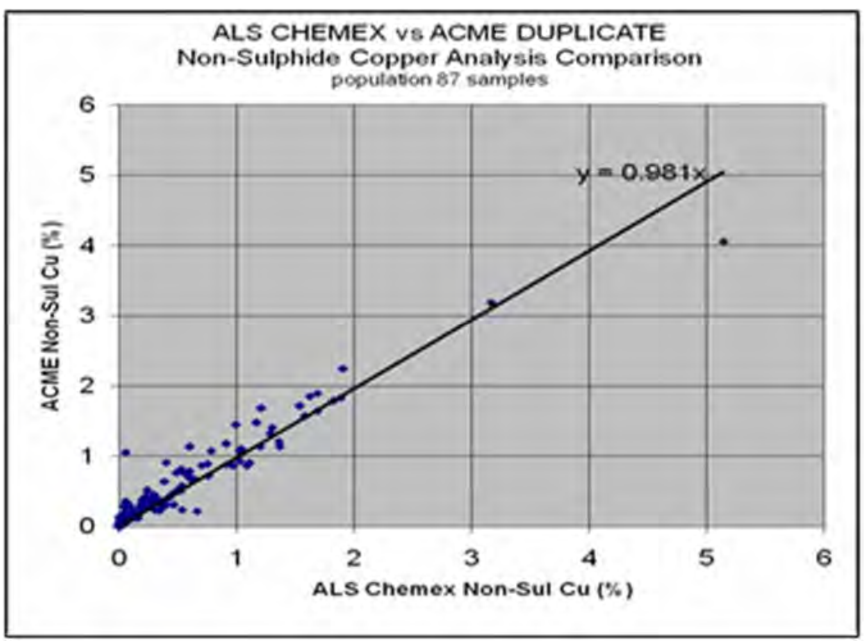
Source: ACS (2016)

Figure 12.2: Original ALS Chemex versus ALS Chemex Duplicate Oxide Copper Analysis



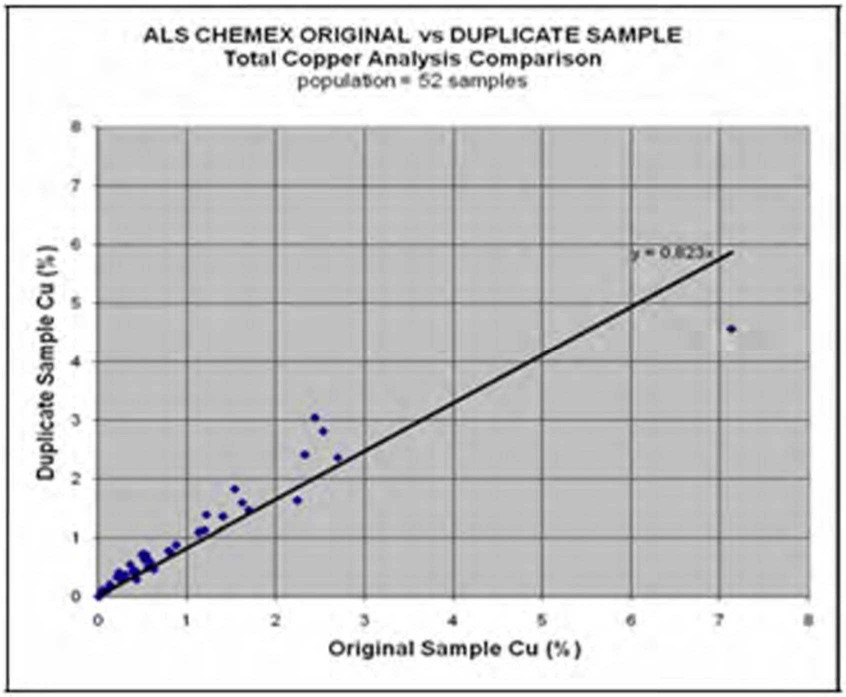
Source: ACS (2016)

Figure 12.3: Original ALS Chemex versus Acme Duplicate Oxide Copper Analysis



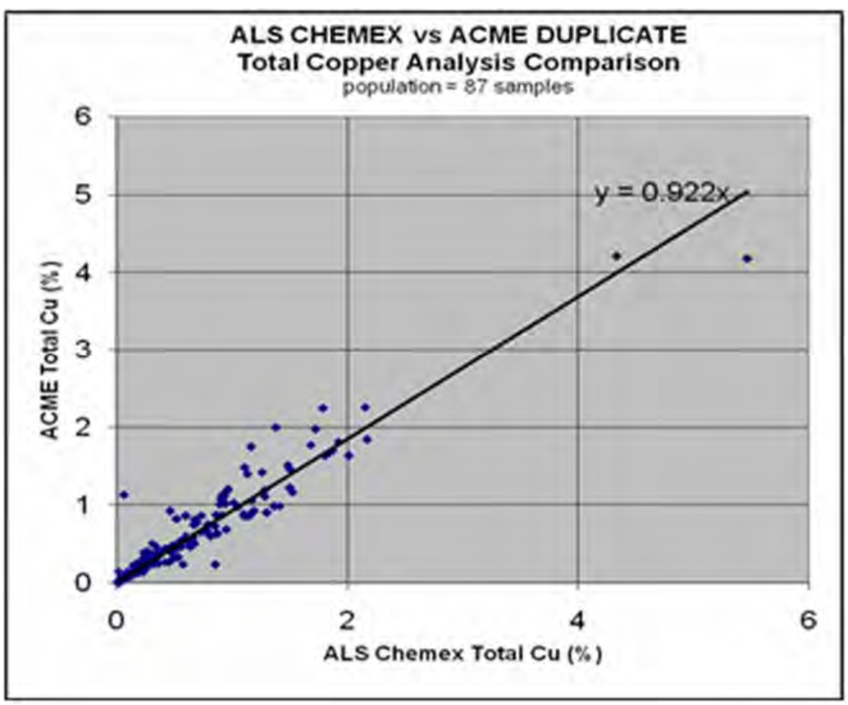
Source: ACS (2016)

Figure 12.4: Original ALS Chemex versus ALS Chemex Duplicate Total Copper Analysis



Source: ACS (2016)

Figure 12.5: Original ALS Chemex versus Acme Duplicate Total Copper Analysis



Source: ACS (2016)

The greatest variability is seen in the gold analyses, which can be expected due to the coarse free-gold that has been observed from petrographic work on the core and due to the nugget effect of gold. The copper analyses show acceptable correlation.

12.1 2015 Data Verification

Dr. Arseneau carried out a second site visit on October 14, 2015 to verify the data used in the resource estimate for Zones 2000S, 12, and 13. During this site visit, 22 drill sites were observed. All drill collars appear to be well marked and easy to locate (Figure 12.7). Drill holes were located with hand-held GPS and all locations agreed well with the data entered in the digital database received from Copper North. The surface geology was observed in surface trenches on Zones 12 and 2000S. Drill core was examined at the Carmacks camp. The core is in excellent condition and both oxide and sulphide copper minerals were evident in drill core. Copper mineralization was observed in all drill holes examined.

Figure 12.6: Collar marking for Drill Hole CN14-11

Source: ACS (2016)

There were no limitations placed on ACS during the site visit and ACS acknowledges the full support of Copper North personnel, specifically Dr. Jack Milton and Mr. Doug Ramsey who facilitated the property visit and supplied details of the last exploration programs carried out by Copper North.

ACS collected four mineralized rock samples during the site visit. All samples contained visible copper mineralization. The samples were dropped off to ALS Minerals in North Vancouver for sample preparation and assay. Table 12.3 summarizes the results of the surface samples collected during the site visit. The samples were collected to confirm that soluble copper occurred on the property in the relative concentrations previously reported. As can be seen from Table 12.3 soluble copper is present in amounts similar to what has been reported for the property.

Table 12.3: Assay Results for Surface Samples Collected by ACS

Sample	Total Cu (%)	Original Total Cu (%)	CuOx (%)	Original CuOx (%)	DDH	Zone
I951062	1.068	0.75	0.978	0.698	CN14-11	2000S
I951063	0.192	0.269	0.144	0.20	CN15-11	13
I951064	0.143	0.357	0.098	0.104	CN15-20	13
I951065	0.949	0.848	0.045	0.028	CN15-07	2000S

Source: ACS (2016)

In addition to the data validation carried out in 2007, all of the Copper North assay data, 1,349 assay records were checked against original assay certificates and no errors were noted.

12.2 ACS Comments

ACS is of the opinion that the drill hole data are adequate for inclusion in resource estimation.

13 Mineral Processing and Metallurgical Testing

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. The primary emphasis of the work conducted up to 2012, as documented in the 2012 FS report (M3, 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.

In 2014, CNMC 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 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 QP of this section confirms that, to the extent known, the test samples are representative of the various types and styles of mineralization and the mineral deposit as a whole; and that no extraordinary processing factors or deleterious elements exist that could have a significant effect on potential economic extraction.

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 (Ref: Merit Consultants International Inc. (2014) Carmacks Copper Project, Yukon, Canada, Preliminary Economic Assessment of Copper, Gold, and Silver Recovery).

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. 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. The test program was divided into two phases and summarized in the following two reports:

- 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).

13.3.1 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 13.1.

Table 13.1: BV Minerals' Test Program Head Assays

Sample ID	Copper (%)	Gold (g/t)	Silver (g/t)	Sulphur (%)	Total Carbon (%)	Organic Carbon (%)
Master Composite	1.10	0.51	5	0.07	0.11	0.09
Trench Composite BS-1	1.39	0.68	6	0.07	0.1	0.1
Trench Composite BS-2	1.18	0.72	7	0.07	0.08	0.07
Trench Composite BS-3	0.71	0.14	2	0.04	0.12	0.07
Trench Composite BS-4	0.79	0.49	4	0.05	0.07	0.06
Composite 1(CAR Z1S-OX50)	0.73	0.18	2	0.04	0.07	0.04
Composite 2 (CAR Z1S-OX150)	0.54	0.08	2	0	0.04	0.03
Composite 3 (CAR Z1N-OX50)	1.00	0.61	7	0.11	0.09	0.06
Composite 4 (CAR Z1N-OX150)	0.91	0.44	4	0.05	0.07	0.06
Composite 5 (CAR Z4-OX)	0.45	0.15	2	0.01	0.14	0.06
Composite 6 (CAR Z7-OX)	1.09	0.36	4	0.02	0.05	0.04
Composite 7 (CAR Z1-SX)	0.92	0.19	4	0.93	0.09	0.07
Composite 8 (CAR Z1213-SX)	0.69	0.14	3	0.87	0.29	0.16

Source: BV Minerals 2015b

13.3.2 Comminution Testing

Variability comminution testing was carried out on all 13 composite samples discussed in Section 13.1. 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. A summary of results is presented in Table 13.2. Based on these 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: Comminution Testing Results

Sample ID	Bond RWI (kWh/t)	Bond BWI Closing Screen (µm)	Bond Ball Mill Work Index (kWh/t)	Bond AI (g)
Master Composite	9.1	105	15.2	0.0901
Trench Composite BS-1	10.1	105	15.0	0.1345
Trench Composite BS-2	9.1	105	14.8	0.0876
Trench Composite BS-3	8.7	105	15.5	0.0982
Trench Composite BS-4	8.6	105	14.8	0.0480
Composite 1 (CAR Z1S-OX50)	10.4	-	-	0.1394
Composite 2 (CAR Z1S-OX150)	11.8	-	-	0.1296
Composite 3 (CAR Z1N-OX50)	10.4	-	-	0.1275
Composite 4 (CAR Z1N-OX150)	11.6	-	-	0.1838
Composite 5 (CAR Z4-OX)	11.6	-	-	0.1277
Composite 6 (CAR Z7-OX)	11.7	-	-	0.1119
Composite 7 (CAR Z1-SX)	9.6	-	-	0.1224
Composite 8 (CAR Z1213-SX)	9.7	-	-	0.1439

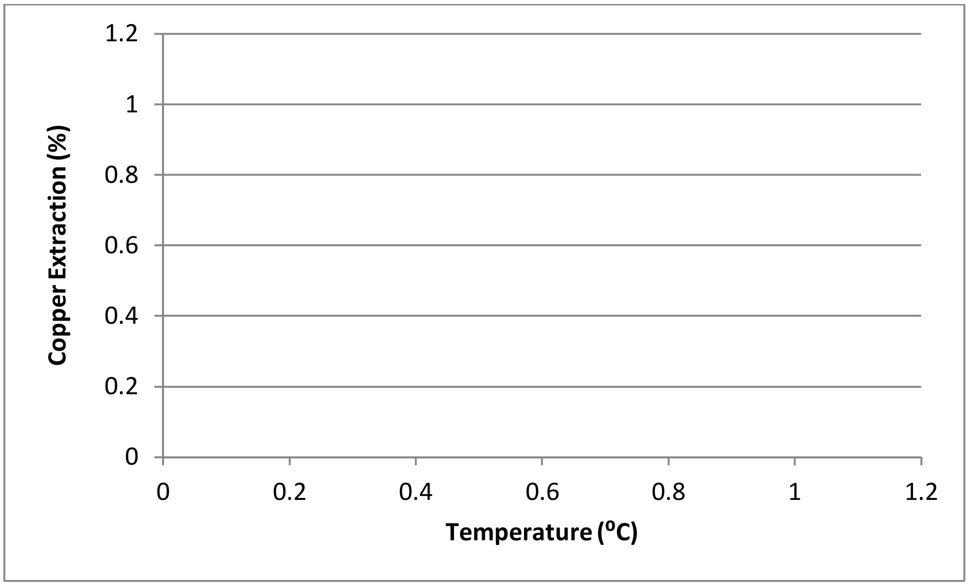
Source: BV Minerals 2015b

13.3.3 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 – 50^{o}C and leach times of eight and 12 hours were tested.

The effect of temperature on copper extraction is shown in Figure 13.1. As temperature increases, so does copper extraction.

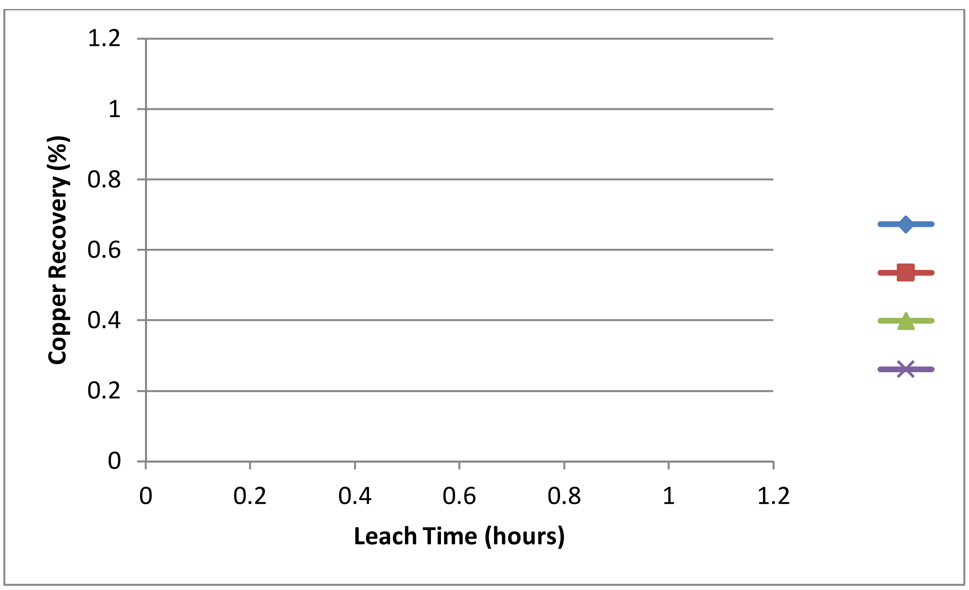
Figure 13.1: Slurry Temperature vs Copper Leach Extraction Rate



Source: BV Minerals 2015b

Copper leach kinetic curves at various temperatures are presented in Figure 13.2. The curves tend to level off after six hours.

Figure 13.2: Copper Leach Kinetics at Varying Temperature



Source: BV Minerals (2015b)

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 = 40^{0}C;
- 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).

The results are shown in Figure 13.3. Copper recovery ranged from 76.5 to 88.8%.

Table 13.3: Copper Leach Variability Test Results

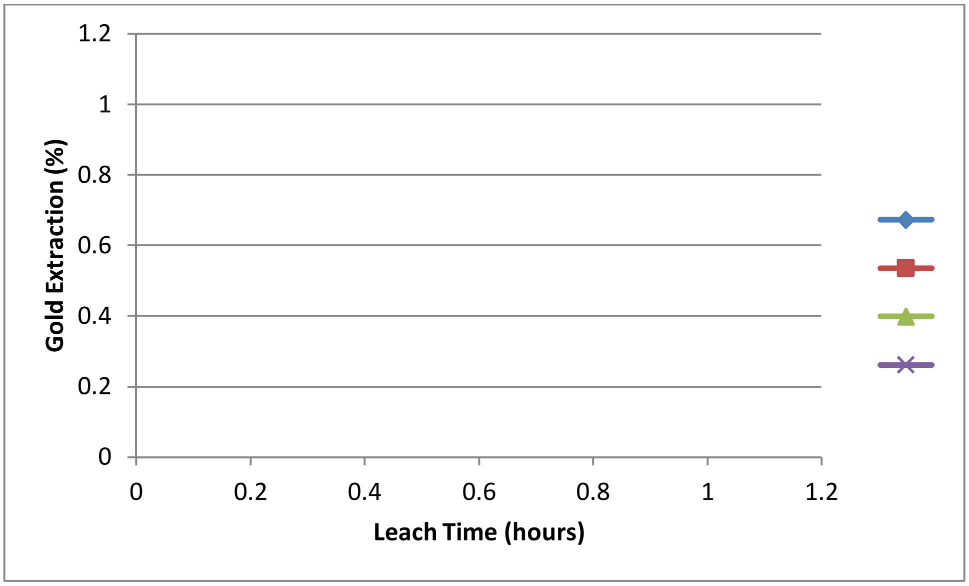
Test ID	Composite	Leach Time (hours)	Final PLS* (Cu, g/L)	Final PLS (Fe, g/L)	Copper Extraction (%)	H_2SO_4 Consumption (kg/t)
LT 5	Master Composite	8	8.79	7.32	**85.5**	28.5
LT 7	Trench Composite BS-1	6	11.23	6.07	**86.6**	26.2
LT 8	Trench Composite BS-2	6	10.33	7.37	**88.8**	24.6
LT 9	Trench Composite BS-3	6	5.55	11.60	**79.0**	27.0
LT 10	Trench Composite BS-4	6	6.04	6.88	**82.0**	27.4
LT 11	Composite 1	6	6.04	6.36	**84.7**	24.5
LT 12	Composite 2	6	4.28	5.75	**83.1**	24.4
LT 13	Composite 3	6	7.26	7.07	**76.5**	24.7
LT 14	Composite 4	6	7.41	6.70	**81.4**	25.3
LT 15	Composite 5	6	3.54	6.32	**84.1**	26.5
LT 16	Composite 6	6	8.18	5.60	**79.5**	27.1

*{LS = Pregnant Leach Solution
Source: BV Minerals (2015b)

13.3.4 Batch Gold Cyanidation Testing

The residue from each preliminary Master Composite copper leach test was subjected to a bottle roll leach test using cyanide. Gold leach kinetic curves at varying copper leach slurry temperatures are presented in Figure 13.3. The recovery curves flatten out after 12 hours.

Figure 13.3: Gold Leach Kinetics at Varying Copper Leach Temperatures



Source: BV Minerals (2015b)

Bottle roll leach tests, at 1.0 g/L NaCN, were also conducted on the residues from copper leach variability testing. The results are summarized in Table 13.7.

Gold recovery ranged from 38 to 83.4% and silver recovery ranged from 6.8 to 83.1%.

Table 13.4: Gold Cyanidation Test Results

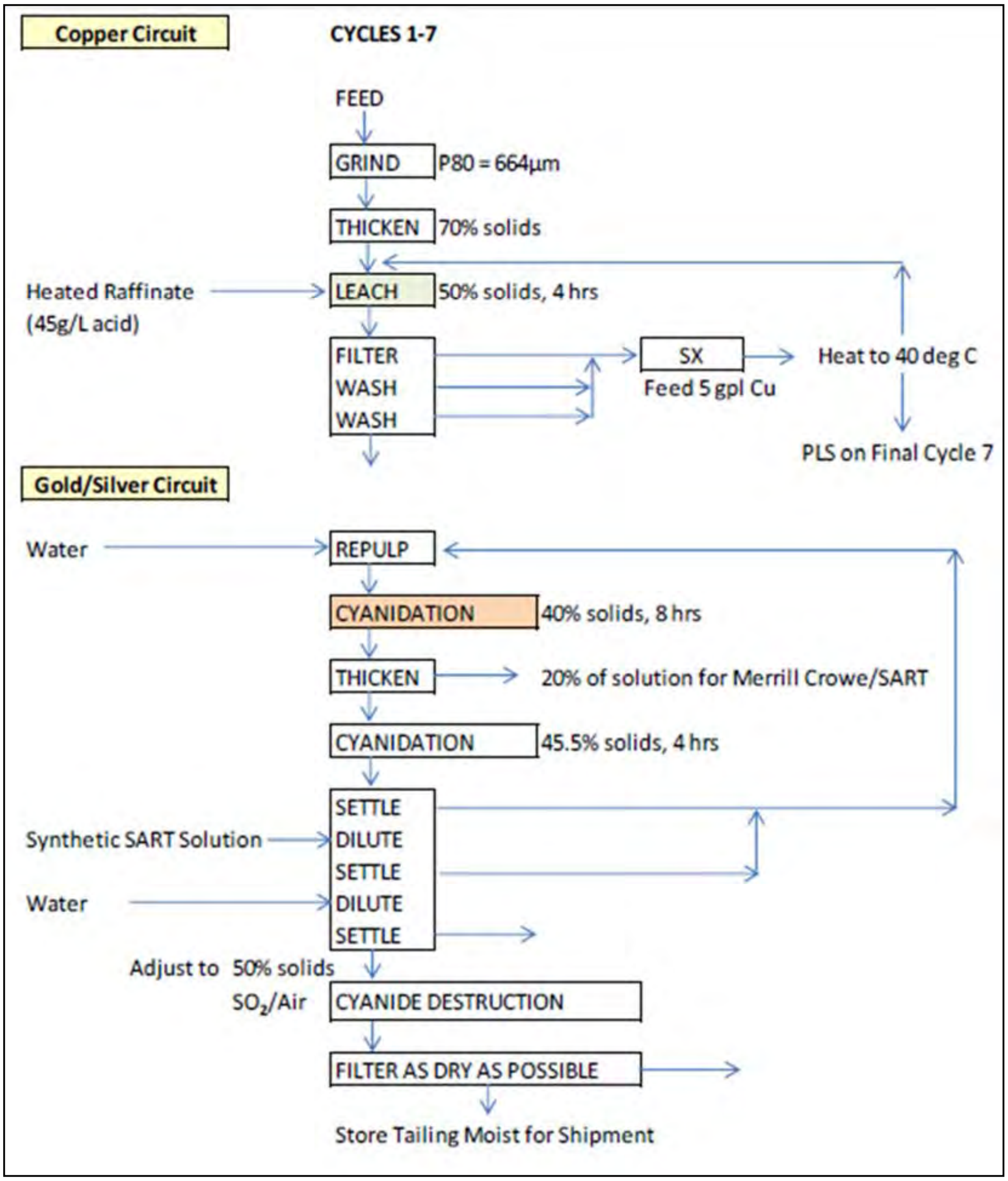
Test ID	Calculated Head			Consumption		Leach Residue			Recovery		
	Au (g/t)	Ag (g/t)	Cu (%)	NaCN (kg/t)	Lime (kg/t)	Au (g/t)	Ag (g/t)	Cu (%)	Au (%)	Ag (%)	Cu (%)
CLT 5: Master Composite	0.48	6.2	0.18	0.8	4.4	0.08	2	0.15	**83.2**	**67.9**	**14.9**
CLT 7: Trench Composite BS-1	0.7	6.16	0.2	0.93	3.75	0.33	3.1	0.18	**53.7**	**50**	**11.3**
CLT 8: Trench Composite BS-2	0.96	7.28	0.17	0.87	4.64	0.16	1.6	0.14	**83.4**	**77.6**	**16.4**
CLT 9: Trench Composite BS-3	0.17	2.25	0.17	0.83	3.59	0.05	0.6	0.14	**71.3**	**75.5**	**15.4**
CLT 10: Trench Composite BS-4	0.28	3.44	0.16	0.86	4.3	0.08	0.6	0.14	**71.4**	**83.1**	**12.6**
CLT 11: Composite 1	0.19	2.46	0.13	0.95	4.05	0.04	1	0.12	**79.4**	**59.3**	**10.8**
CLT 12: Composite 2	0.09	1.55	0.12	1.06	4.1	0.02	0.7	0.11	**78.7**	**54.5**	**9.8**
CLT 13: Composite 3	0.63	6.4	0.25	1.01	4.03	0.39	6	0.21	**38**	**6.8**	**15.8**
CLT 14: Composite 4	0.38	4.65	0.2	0.97	4.14	0.15	2.8	0.18	**60.8**	**39.5**	**11.7**
CLT 15: Composite 5	0.17	2.18	0.09	0.85	4	0.04	1	0.08	**76.3**	**54.2**	**6.4**
CLT 16: Composite 6	0.28	4.86	0.25	1.08	4.79	0.06	2.2	0.23	**78.6**	**55**	**8.6**

Source: BV Minerals (2015b)

13.3.5 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. An overall flowsheet for the locked cycle test is presented in Figure 13.4.

Figure 13.4: Locked Cycle Test Flowsheet



Source: BV Minerals (2015b)

The copper leach test results are presented in Table 13.5. 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.

Table 13.5: Locked Cycle Copper Leach Results

Cycle	PLS (Cu mg/L)	Wash 1 (Cu mg/L)	Wash 2 (Cu mg/L)	Residue (% Cu)	Recovery (% Cu)	Calculated Feed Grade (% Cu)	Total Acid Consumption (kg/t)
1	7890	2403	536	0.12	**87.8**	0.96	30.0
2	7900	2509	558	0.12	**87.7**	0.95	30.8
3	7957	2620	753	0.12	**88.0**	0.97	30.3
4	7955	2678	767	0.13	**87.3**	0.99	30.8
5	8080	2878	484	0.11	**89.1**	0.98	29.6
6	8420	2555	580	0.12	**88.1**	0.97	30.6
7	8379	2508	594	0.13	**87.1**	0.96	30.8
Average	**8083**	**2593**	**610**	**0.12**	**87.9**	**0.97**	**30.4**

Source: BV Minerals (2015b)

The gold cyanidation results are shown in Table 13.6 and Table 13.7.

. 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.

Table 13.6: Locked Cycle Gold Leach Results

Cyanide Leach Product	Solution Metal Grade			Metal Distribution		
	Au (mg)	Ag (mg)	Cu (mg)	Au (%)	Ag (%)	Cu (%)
Cycle 1-7 Bleed to Merrill Crowe	4.37	46.0	2,131	21.8	17.3	3.7
Cycle 1-7 12 hour PLS Solution	11.72	118.2	5,837	58.3	44.5	10.2
Total in Solution	**16.09**	**164.2**	**7,968**	**80.1**	**61.8**	**13.9**
Cycle 1-7 Residue	4.00	101.5	49,481	19.9	38.2	86.1

Source: BV Minerals (2015b)

Table 13.7: Cycle by Cycle Gold Solution Grades

Cycle	8 hr PLS Product to Merrill Crowe		12 hr PLS Solution to Recycle		Tailings Assays (CN Leach Residue)		Reagent Consumption	
	(Au mg/L)	(Ag mg/L)	(Au mg/L)	(Ag mg/L)	(Au g/t)	(Ag g/t)	NaCN (kg/t)	Lime (kg/t)
1	0.22	2.5	0.24	2.6	0.10	**2.4**	0.91	7.6
2	0.32	3.8	0.35	3.9	0.10	**2.4**	0.86	6.8
3	0.34	3.7	0.36	3.8	0.10	**2.6**	0.61	7.0
4	0.35	3.8	0.36	3.8	0.10	**2.5**	0.79	7.8
5	0.39	4.0	0.42	4.0	0.11	**2.8**	0.71	7.1
6	0.44	4.1	0.48	4.2	0.09	**2.5**	0.62	7.8
7	0.44	4.3	0.47	4.5	0.10	**2.5**	0.87	8.1
Average	**0.36**	**3.7**	**0.38**	**3.8**	**0.10**	**2.53**	**0.77**	**7.5**

Source: BV Minerals (2015b)

13.3.6 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. The cyanide destruction results per cycle are shown in Table 13.8 and the reagent consumption rates are summarized in Table 13.9.

Table 13.8: Cyanide Destruction Results

Analysis	Stage	Unit	DCN-1 (Cycle 1)	DCN-2 (Cycle 2)	DCN-3 (Cycle 3)	DCN-4 (Cycle 4)	DCN-5 (Cycle 5)	DCN-6 (Cycle 6)
Total CN^-	Feed	mg/L	247	307	302	357	312	342
	Final	mg/L	<0.05	0.06	<0.05	0.15	<0.05	0.09
WAD CN^-	Feed	mg/L	217	267	277	282	287	297
	Final	mg/L	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Free CN^-	Feed	mg/L	212	262	224	229	273	245
	Final	mg/L	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
SCN^-	Feed	mg/L	21	26	28	32	27	34
	Final	mg/L	28	29	25	32	22	31
CNO^-	Feed	mg/L	40	48	71	35	41	38
	Final	mg/L	142	164	403	409	335	344

Source: BV Minerals (2015b)

Table 13.9: Cyanide Destruction Reagent Requirements

Test Number	Total Cyanide (TCN) Analysis (mg/L)		Reagent Usage (g/g TCN)			
	Before	After	SO_2	$Na_2S_2O_5$ (SMBS)	Lime	$CuSO_4$
DCN-1 – Cycle 1	247	<0.05	5.62	8.34	1.75	0.40
DCN-2 – Cycle 2	307	0.06	5.96	8.84	2.41	0.44
DCN-3 – Cycle 3	302	<0.05	7.16	10.62	3.39	0.59
DCN-4 – Cycle 4	357	0.15	6.06	8.99	2.59	0.48
DCN-5 – Cycle 5	312	<0.05	8.39	12.45	3.88	0.59
DCN-6 – Cycle 6	342	0.09	6.81	10.10	3.21	0.45
Average	**311**	**<0.05**	**6.67**	**9.89**	**2.87**	**0.49**

Source: BV Minerals (2015b)

13.3.7 Settling and Filtration Test Work

Settling tests were conducted on cyanide leach residue from test C8 (Phase 1) and CLT 3 (Phase 2). The results are shown in Table 13.10.

Table 13.10: Settling Test Results

Test ID	Sample	P_{80} Grind Size (µm)	Flocculant		Sludge Density (% Solids)		Initial Settling Rate (m/h)	Unit Thickener Area ($m^2/t/d$)
			Type	Dosage (g/t)	Initial	Final		
ST 11	C8 CN Residue	1,000	Magnafloc 351	30	47.8	61.0	0.12	0.23
ST 1	CLT 3 CN Residue	664	Magnafloc 351	30	49.8	63.9	0.04	0.17

Source: BV Minerals (2015b)

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. The results are summarized in Table 13.11.

Table 13.11: Filtration Test Results

Test No.	Filter Cloth	Filtrate Volume (ml)	Filtration Time		Filter Cake		Cake Capacity ($kg/m^2/h$)	Filtrate Capacity ($L/m^2/h$)
			Form (sec)	Dry (sec)	Thickness (mm)	Moisture (%)		
VF 1	NY330	120	568	142	12	15.8	86.5	65
VF 2	NY330	280	1,432	38	27	21.4	97.9	73
VF 3	POPR 901F	120	517	98	9	14.9	98.1	75
VF 4	POPR 901F	65	96	60	3	15.7	194.6	159
VF 5	Needle6420	114	195	78	7.5	20.4	189.9	160
VF 6	NY330	70	230	140	17	18.8	229.7	72
VF 7	NY330	98	382	205	25	21.0	208.9	64

Source: BV Minerals (2015b)

13.3.8 Copper Leach Testing at Elevated Temperature

In February 2016, BV Minerals conducted additional copper leach optimization test work using 2014 Master Composite, and summarized the results in the following report:

- 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. A summary of results is presented in Table 13.12. Operating the copper leach at 80^{0}C improved copper extraction by 2.2% over the 40^{0}C 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.

Table 13.12: Copper Acid Leach Test Results

Test ID	P_{80} Grind Size (µm)	Temperature (ºC)	Initial Leach Solution (g/L H_2SO_4)	Ferric Addition (g/L)	Pulp Density (%)	Leach Duration (h)	Final Cu PLS (g/L)	Cu Extraction (%)
ALT 1	664	80	24.7	0	50	6	10.75	**86.3**
ALT 2	664	80	24.7	10	50	6	11.18	**86.6**
ALT 3	664	40	24.7	0	50	6	8.94	**84.1**

Source: BV Minerals (2016)

The copper leach residue was then leached with cyanide to investigate the effect of the noted improved copper extraction on subsequent gold recovery. A summary of results is shown in Table 13.13. 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.

Table 13.13: Gold Cyanidation Test Results

Test ID	Calculated Head			Consumption		Leach Residue			Recovery		
	Au (g/t)	Ag (g/t)	Cu (%)	NaCN (kg/t)	Lime (kg/t)	Au (g/t)	Ag (g/t)	Cu (%)	Au (%)	Ag (%)	Cu (%)
CALT 1	0.45	4.88	0.13	0.50	3.00	0.07	3.4	0.12	**84.5**	**30.3**	**7.8**
CALT 2	0.45	4.97	0.13	0.63	3.63	0.07	4.5	0.13	**85.5**	**9.5**	**6.8**
CALT 3	0.47	5.05	0.16	0.72	2.31	0.08	1.6	0.14	**82.8**	**68.3**	**11.4**

Source: BV Minerals (2016)

13.4 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 (BV Minerals 2016) 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 detailed by Merit (2014). 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 13.14 and Table 13.15.

Table 13.14: Process Design Criteria Derived from Test Work

Description	Units	Value
Ore Characteristics		
Ore Solids Density	t/m^3	2.64
Ore Moisture	% w/w	7.60
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
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	2.0
Carbon Loading	g Au / t Carbon	3,000*
Cyanide Destruction		

Description	Units	Value
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	$kg/m^2/hr$	230
Filter Cake Density	%	83
Filter Cake Moisture Content	%	17

Note: Vendor Recommended, no test work available

Table 13.15: Reagent Consumption Derived from Test Work

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

Source: JDS (2016)

14 Mineral Resource Estimate

The Mineral Resource estimation work for the Carmacks project was completed with the use of 3D introduction by Dr. Gilles Arseneau, P. Geo. (APEGBC) of Arseneau Consulting Services Ltd. (ACS), an appropriate independent “qualified person” within the meaning of NI 43-101. The effective date of the Mineral Resource statement is January 25, 2016.

The Mineral Resources reported in this section combines work done by Dr. Arseneau in 2007 (ACS, 2007), when Mineral Resources were estimated for Zones 1, 4, 7, and 7A, and with work done more recently on Zones 12, 13, and 2000S.

The Mineral Resource model prepared by ACS utilized a total of 246 drill holes, 39 of which were drilled by Copper North in the 2014-2015 drilling program.

This section describes the resource estimation methodology and summarizes the key assumptions considered by ACS. In the opinion of ACS, the resource evaluation reported herein is a reasonable representation of the copper, gold and silver Mineral Resources found at the Carmacks Project at the current level of sampling. The Mineral Resources have been estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practices” guidelines (2003) and are reported in accordance with the Canadian Securities Administrators’ NI 43-101. Mineral resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into mineral reserve.

The database used to estimate the Carmacks Mineral Resources was audited by ACS. ACS is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries of the copper mineralization and that the assay data are sufficiently reliable to support Mineral Resource estimation.

14.1 Resource Estimation Procedures

The resource evaluation methodology involved the following procedures:

- Database compilation and verification;
- Construction of wireframe models for the boundaries of the copper mineralization;
- Definition of resource domains;
- Data conditioning (compositing and capping) for geostatistical analysis and variography;
- Block modelling and grade interpolation;
- Resource classification and validation;
- Assessment of “reasonable prospects for economic extraction” and selection of appropriate cut-off grades; and
- Preparation of Mineral Resource statement.

14.2 Drill Hole Database

The drilling database consists of historical drilling most of which has been carried out by Western Copper. The database also includes 50 drill holes drilled by Copper North in 2014-2015. Table 14.1 summarizes the drill holes used for each mineralized zone estimated.

Table 14.1: Drill holes used in the Block Model Estimation

Company	Year	No. Holes	Metres	Hole Type	Zone
Historic	1970-71	33	5,806	DD	1, 4, 7
Western Copper	1990-92	25	2,501	DD	12
Western Copper	1990-92	24	3,561	DD	13
Western Copper	2006-07	116	18,811	DD	1, 4, 7
Western Copper	1990-92	9	1,463	DD	2000S
Western Copper	2006-07	27	4,377	DD	Exploration
Western Copper	1992	3	244	RC	1, 4, 7
Western Copper	1992	4	271	RC	2000S
Western Copper	1992	4	341	RC	Exploration
Copper North	2014-15	6+	395	DD	12
Copper North	2014-15	20	1,932	DD	13
Copper North	2014-15	1	88	DD	1, 4, 7
Copper North	2015-15	12	1,195	DD	2000S
Copper North	2014-15	11	748	DD	Exploration
Total		**295**	**41,733**		

Source: ACS (2016)

There are a total of 10,577 records in the assay database, of these 5,661 represent samples taken from the mineralized horizons. Table 14.2 summarizes the basic statistical data for all the assays in the database and Table 14.3 summarizes the assays contained within the mineralized zones only.

Table 14.2: Basic Statistical Properties of All Assay Data

Property	CuT (%)	CuOx (%)	Au (g/t)	Ag (g/t)
Valid Cases	10,577	10,577	10,577	10,577
Mean	0.47	0.29	0.18	2.45
Standard Deviation	0.72	0.57	0.51	17.82
Variation Coefficient	1.5	1.94	2.75	7.27
Minimum	10.95	7.82	17.14	1775
Maximum	0	0	0	0
5th percentile	0	0	0	0
10th percentile	0.01	0	0	0.25
25th percentile	0.02	0	0	0.25
Median	0.2	0.04	0.04	0.7
75th percentile	0.67	0.34	0.17	2.5
90th percentile	1.29	0.92	0.41	5.49
95th percentile	1.81	1.37	0.75	9
99th percentile	3.23	2.56	2.47	23.02

Source: ACS (2016)

Table 14.3: Basic Statistical Properties of Assays Within The Mineralized Units

Property	CuT (%)	CuOx (%)	Au (g/t)	Ag (g/t)
Valid Cases	5,661	5,661	5,661	5,661
Mean	0.8	0.49	0.32	3.91
Std. Deviation	0.82	0.68	0.65	24.15
Variation Coefficient	1.03	1.39	2.05	6.17
Minimum	0	0	0	0
Maximum	10.95	7.82	17.14	1775
5th percentile	0.08	0.01	0.01	0.25
10th percentile	0.13	0.01	0.01	0.25
25th percentile	0.27	0.04	0.06	0.8
Median	0.57	0.23	0.14	2
75th percentile	1.03	0.7	0.32	4
90th percentile	1.69	1.28	0.69	8
95th percentile	2.26	1.81	1.17	12.8
99th percentile	3.91	3.06	3.06	29.12

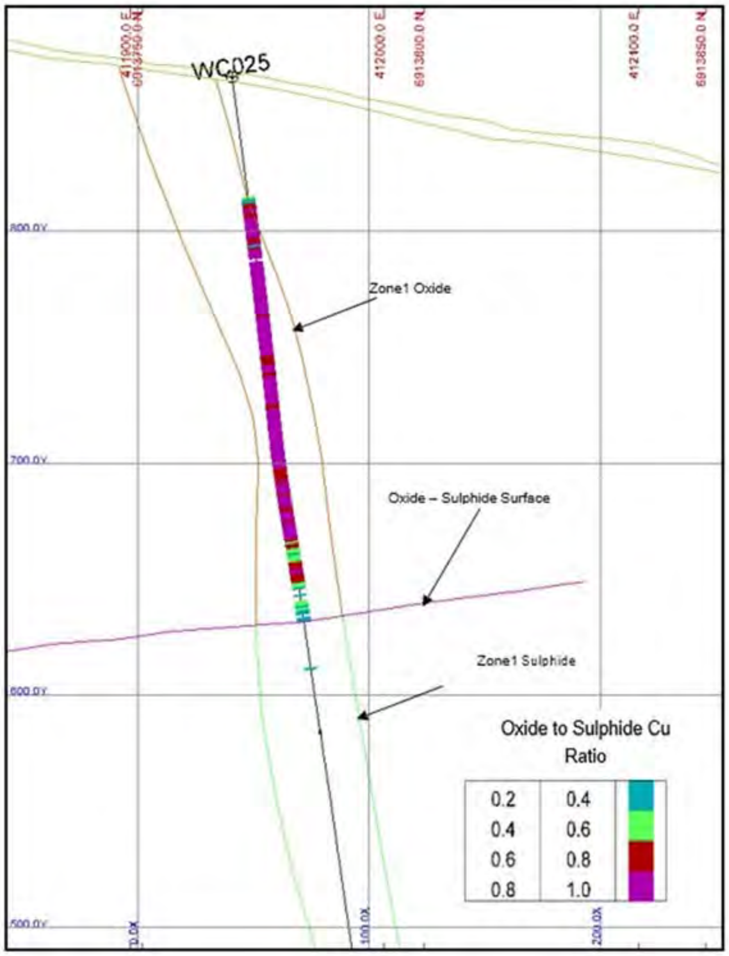
Source: ACS (2016)

14.3 Geological Model

Three mineralized zones were interpreted on the basis of total copper grade and geological cross-sections spaced at 25 m spacing. Polylines representing a 0.2% total copper cut-off were generated on northeast sections perpendicular to the mineralized zones. The polylines honoured the drill hole intersections in 3D. Wireframes were created from the lines and then clipped against the overburden-bedrock surface.

A surface representing the boundary between the upper oxide and lower sulphide mineralization was interpolated based on drill hole intersections. The transition between oxide and sulphide mineralization occurs over a few metres for most zones with the exception of Zone 13 where a large volume of transitional material seems to 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%. A 3D surface was then generated by connecting all drill hole points to form the oxide-sulphide interface (Figure 14.1).

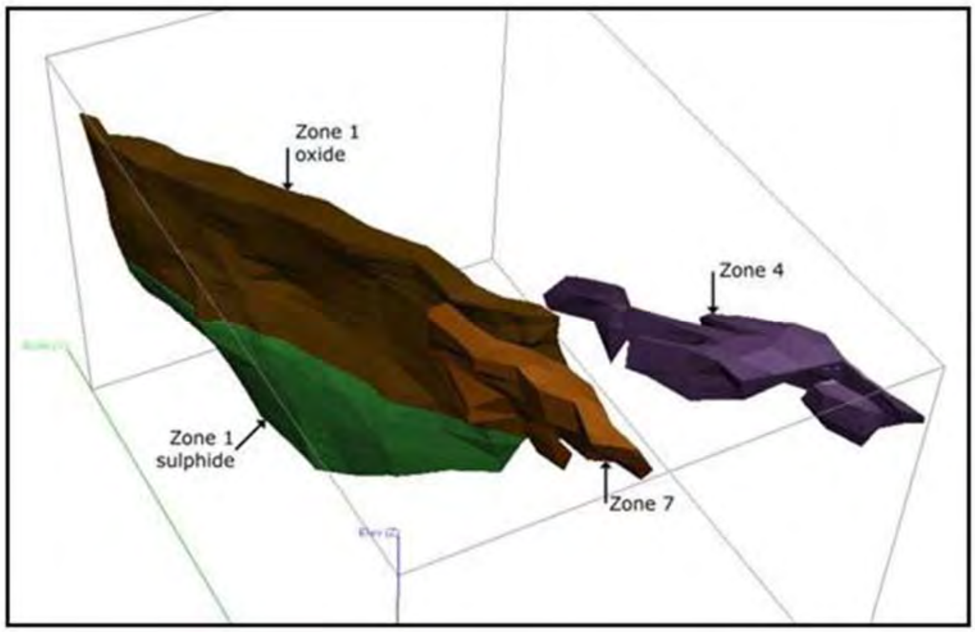
Figure 14.1: Cross Section 1100N showing Total Copper and Oxide Copper Proportion and Oxide/Sulphide Surface



Source: ACS (2016)

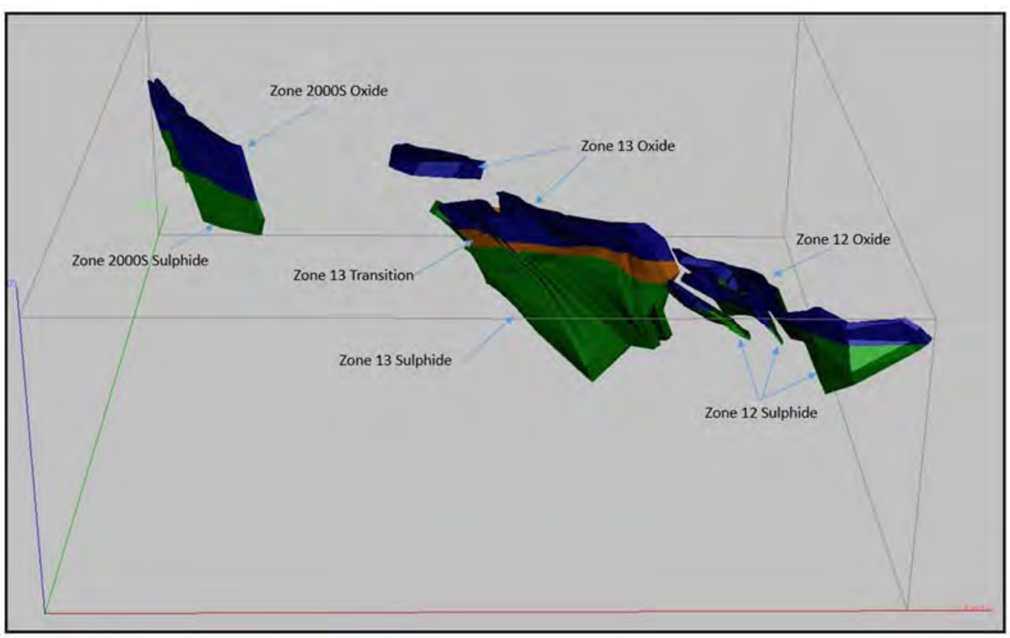
The wireframes were then clipped above and below the oxide boundary to create final oxide and sulphide wireframes (Figure 14.2 and Figure 14.3).

Figure 14.2: Perspective View Of Zones 1, 4 and 7 Looking Northeast



Source: ACS (2016)

Figure 14.3: Perspective View Of Zones 12, 13 and 2000S Looking North



Source: ACS (2016)

14.4 Compositing

All assay data were composited to a fixed length prior to estimation. ACS evaluated the assay lengths for the various deposits and found that most samples had an average length of three metres or less with 91% of samples lengths being less than 2.5 m. For the 2007 resource estimation of Zones 1, 4 and 7, ACS decided to composite all assay data to 5 metres prior to estimation. For the 2016 resource estimation of Zones 12, 13 and 2000S, ACS decided to composite the assays to 2.5 m to better define the grade variability within these zones. Table 14.4 summarizes the basic statistical data for uncapped composites used in the resource estimates.

Table 14.4: Descriptive Statistics of Composite Within all Mineralized Zones

Property	CuT (%)	CuOx (%)	Au (g/t)	Ag (g/t)
Valid Cases	2,002	2,002	2,002	2,002
Mean	0.72	0.42	0.26	3.4
Std. Deviation	0.61	0.55	0.43	11.98
Variation Coefficient	0.86	1.29	1.65	3.52
Minimum	0	0	0	0
Maximum	5.51	4.73	5.7	368
5th percentile	0.11	0.01	0	0.17
10th percentile	0.16	0.02	0.02	0.31
25th percentile	0.29	0.04	0.06	0.86
Median	0.54	0.2	0.14	1.97
75th percentile	0.95	0.62	0.26	3.67
90th percentile	1.48	1.15	0.58	6.28
95th percentile	1.91	1.57	1	9.77
99th percentile	2.84	2.38	2.22	20.19

Source: ACS (2016)

14.5 Evaluation of Outliers

Block grade estimates may be unduly affected by high grade outliers. Therefore, assay data were evaluated for high grade outliers. Based on the analysis of the assay distribution, ACS decided that capping of high grade assays was not warranted for Zones 1, 4, and 7. Capping levels for Zones 12, 13, and 2000S are summarized in Table 14.5.

Table 14.5: Capping Levels For Zones 12, 13 and 2000S

Zone 12				
	CuT (%)	CuOx (%)	Au (g/t)	Ag (g/t)
Cap Level	2.2	no cap	no cap	20
No Capped	1	0		1
CoV Uncap	0.79	1.26	1.13	1.14
CoV Cap	0.76	1.26	1.13	1.09
Metal Loss (%)	0.4	0	0	0.7
Zone 13				
	CuT (%)	CuOx (%)	Au (g/t)	Ag (g/t)
Cap Level	no cap	no cap	no cap	10
No Capped	0	0	0	2
CoV Uncap	0.99	1.62	1.14	0.99
CoV Cap	0.99	1.62	1.14	0.97
Metal Loss (%)	0	0.3	0	0.3
Zone 2000S				
	CuT (%)	CuOx (%)	Au (g/t)	Ag (g/t)
Cap Level	2	1.1	no cap	30
Number Capped	2	2	0	1
CoV Uncap	0.69	1.26	0.83	10.6
CoV Cap	0.65	1.24	0.83	1
Metal Loss (%)	1.1	0.9	0	226

Source: ACS (2016)

14.6 Spatial Analysis

Spatial continuity of copper was evaluated with correlograms developed using SAGE 2001 version 1.08. The correlogram measures the correlation between data values as a function of their separation distance and direction. The distance at which the correlogram is close to zero is called the "range of correlation" or simply the range. The range of the correlogram corresponds roughly to the more qualitative notion of the "range of influence" of a sample or composite.

Directional correlograms were generated for composited data at 30° increments along horizontal azimuths. For each azimuth, correlograms were calculated at dips of 0, 30 and 60°. A vertical correlogram was also calculated, using the information from these 37 correlograms. SAGE will then determine the best fit model using the least squares fit method. The correlogram model is described by the nugget (C0), the variance contribution of the two nested structure (C1, C2) and the range of each of the structures.

Variographic analysis was evaluated for total copper for all mineralized zones but robust correlograms could only be constructed for Zones 1 and 13. No correlograms could be developed for any of the other zones estimated. Table 14.6 summarizes the correlogram parameters used to interpolate grades in Zones 1, 4, 7, and 13.

The correlogram models applied in the Mineral Resource estimates in each domain derived from drill hole composites are presented in Table 14.6. Model rotations follow the right hand rule and nugget effects were established from downhole variogram analysis.

Table 14.6: Correlogram Parameters Used for Grade Estimation

Domains	Metal	Model Type	Nugget (C0)	C1 & C2	Rotation			Range		
					(Z)	(Y)	(Z)	Rot X	Rot Y	Rot Z
Zone 1, 4, and 7	Cu	Exponential	0.025	0.64	-24.8	39	-49	17.4	127	16
				0.335	-35	-23	48	268	381	27
Zone 13	Cu	Exponential	0.23	0.495	33	66	-11	12	11	21
				0.275	33	66	-11	146	180	60

Source: ACS (2016)

14.7 Block Model

Because of the distance between Zones 1 and 12, ACS decided to construct two separate block models to estimate the Mineral Resources at Carmacks. Both models were with Geovia GEMs version 6.7 block modelling software. The models included parameters for rock code, density, total copper, copper oxide and copper sulphide grades. Other parameters such as distance to the nearest drill hole, number of composites used, the average distance of the composite used and the number of drill holes used to interpolate block grades were also recorded in the model.

The block models are set in UTM NAD 83 coordinates and rotated 24.2° counter clockwise to line up with the mineralized zones. The model parameters are defined in Table 14.7.

Table 14.7 Block Model Parameters

Zones	Coordinate	Minimum (UTM)	Block Size (m)	No. of Blocks
1, 4, and 7	Easting	412,050	5	70
	Northing	6,913,130	5	195
	Elevation	350	5	110
12, 13, and 2000S	Easting	412,900	5	155
	Northing	6,911,150	5	416
	Elevation	350	5	110

Source: ACS (2016)

14.7.1 Grade Estimation

Grades were estimated by ordinary kriging and by inverse distance weighted to the second power for all other zones (ID2). Grades were constrained within individually identified geological units using sample data composited to 2.5 metre intervals into model blocks measuring 5 m by 5 m by 5 m vertically.

Grade interpolation strategies were based on zone orientations, drill hole distances and parameters derived from variographic analysis. Grade interpolations were carried out in two passes with successive pass only interpolating block grades for blocks that had not been interpolated by the previous pass. Search ellipse orientation and number of samples used to interpolate a block are listed in Table 14.8 for Zones 1, 4, and 7 and in Table 14.9 for Zones 12, 13, and 2000S.

Table 14.8: Copper Search Interpolation Parameters for Zones 1, 4, And 7

Zones	Pass	Rotation			Search Ellipse Size			No. of Composites		Max no. per hole
		X (m)	Y (m)	Z (m)	X (m)	Y (m)	Z (m)	Min	Max	
1, 4, and 7	1	0	70	0	100	100	15	3	10	1
1, 4, and 7	2	0	70	0	150	150	50	2	12	1

Source: ACS (2016)

Table 14.9: Interpolation Parameters for Zones 12, 13, and 2000S for All Metals

Zones	Pass	Rotation			Search Ellipse Size			No. of Composites		Max no. per hole
		Azm. (°)	Dip (°)	Azm. (°)	X (°)	Y (°)	Z (°)	Min	Max	
12	1	45	-45	60	50	20	50	3	10	2
12	2	45	-45	60	100	45	100	3	12	2
13	1	45	-40	30	50	20	50	3	10	2
13	2	45	-40	30	100	45	100	3	12	2
2000S	1	-30	10	20	50	20	50	3	10	2
2000S	2	-30	10	20	100	45	100	3	12	2

Source: ACS (2016)

For Zones 1, 4, and 7 blocks were only interpolated in the first pass if at least three samples, no more than one sample per hole, were found within the search ellipse and no more than 10 samples were used to interpolate grade within a block. The second pass only estimated grades in blocks that were un-assigned during 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 radius and no more than 12 samples were used to interpolate grade in a block.

For Zones 12, 13 and 2000S, blocks were only interpolated in the first pass, if at least three samples, no more than two sample per hole, were found within the search ellipse and no more than 10 samples were used to interpolate grade within a block.

The second pass only estimated grades in blocks that were un-assigned during pass one. Blocks were assigned a grade in pass two if at least three samples, no more than two per hole, were found within the search radius and no more than 12 samples were used to interpolate grade in a block.

The same estimation parameters were used for gold and silver for Zones 12, 13, and 2000S, but for Zones 1, 4 and 7 gold and silver were interpolated using the same parameters as for the pass two copper search ellipse.

Sulphide copper grades were calculated into the model after grade estimation was completed using a simple manipulation of the block model parameters according to the following formula:

Cu Sulphide% = Cu Total% – Cu Oxide%

Any blocks found with negative sulphide copper grades were reset to 0.0%.

14.8 Density

In 1991, bulk densities were estimated by ALS Chemex on 21 drill core samples. The samples submitted comprised of five granodiorites, two pegmatites and 14 gneiss samples.

The density of granodiorite samples surrounding the mineralization ranged between 2.69 t/m^3 to 2.71 t/m^3 for an average of 2.70 t/m^3. The bulk density of gneissic material hosting the mineralization ranged from 2.59 to 2.97 t/m^3 although only one sample was greater than 2.73 t/m^3.

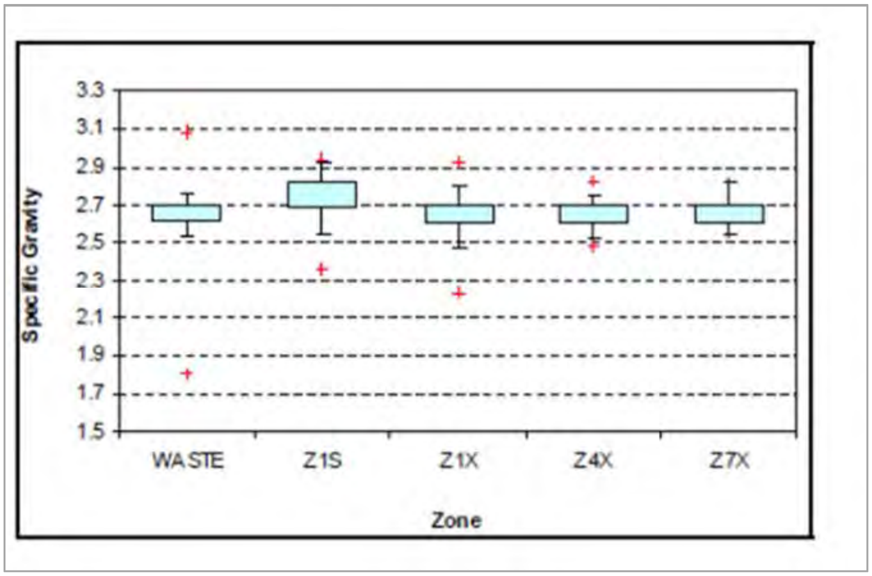
In 2006 and 2007, bulk density was measured by Aurora in the field on 1,358 drill core samples (Aurora Geoscience, 2007). An average bulk density of 2.64 t/m^3 was determined for samples collected within Zone 1 oxide and 2.75 t/m^3 within the Zone 1 sulphide (Table 14.10 and Figure 14.4).

Table 14.10: Summary of Bulk Density Measurements (t/m^3)

	Zone 1	Zone 4	Zone 7	Zone 1	Granodiorite
Valid cases	132	50	22	59	1095
Mean	2.643	2.646	2.663	2.749	2.661
Std. Deviation	0.100	0.068	0.074	0.110	0.088
Minimum	2.24	2.48	2.55	2.37	1.80
25th percentile	2.60	2.60	2.60	2.69	2.62
Median	2.64	2.65	2.66	2.76	2.66
75th percentile	2.70	2.70	2.70	2.82	2.70
Maximum	2.93	2.83	2.82	2.95	3.08

Source: ACS (2016)

Figure 14.4: Box Plot of Bulk Density (Specific Gravity) for Zones 1, 4 and 7



Source: ACS (2016)

During the 2015 drill program, Copper North collected an additional 215 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.

Density was interpolated into blocks in two passes using isotropic inverse distance weighted to the second power for Zones 1, 4 and 7. Interpolation occurred in two passes with sample support summarized in Table 14.11.

Table 14.11: Interpolation Parameters for Density Model

Pass	Axes Rotation	Ranges (m)	Occurrence per Hole	Minimum Samples	Maximum Samples
1	Z=0	X=50	Not limited	3	8
	X=70	Y=50			
	Z=0	Z=50			
2	Z=0	X=20	Not limited	1	8
	X=70	Y=20			
	Z=0	Z=20			

Source: ACS (2016)

After the estimation process, any mineralized blocks that had a bulk density value less than 2.5 t/m^3 were re-initialized to an average value of 2.64 t/m^3.

Because of the limited density data from Zones 12, 13 and 2000S, ACS decided to use average density values in the 2016 block model for Zones 12, 13 and 2000S as outlined in Table 14.12.

Table 14.12: Bulk Density Values Used for Zones 12, 13 and 2000S

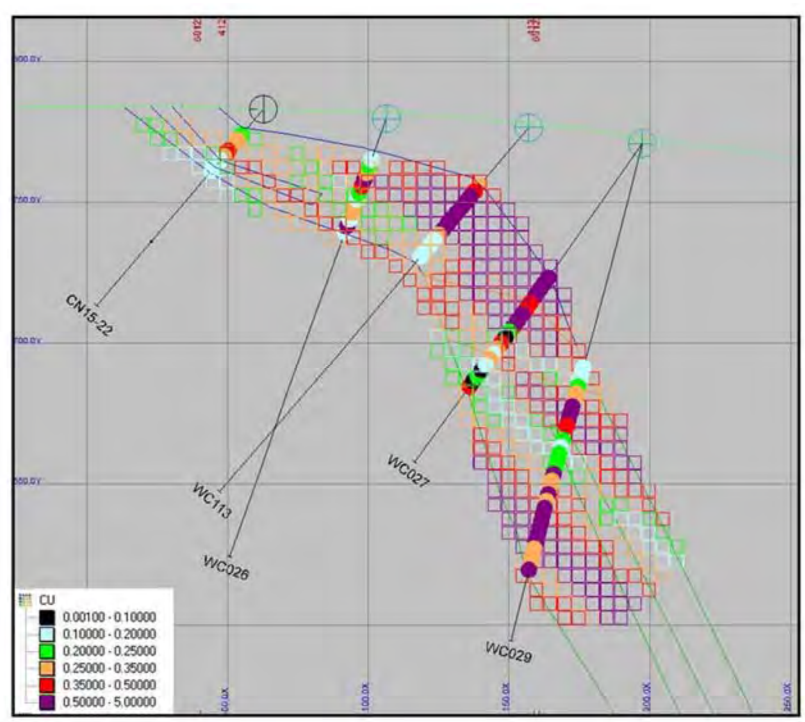
Rock type	Density (t/m^3)
Overburden	2.00
Waste rock	2.65
Oxide	2.70
Sulphide	2.74
Transition material	2.68

Source: ACS (2016)

14.9 Model Validation

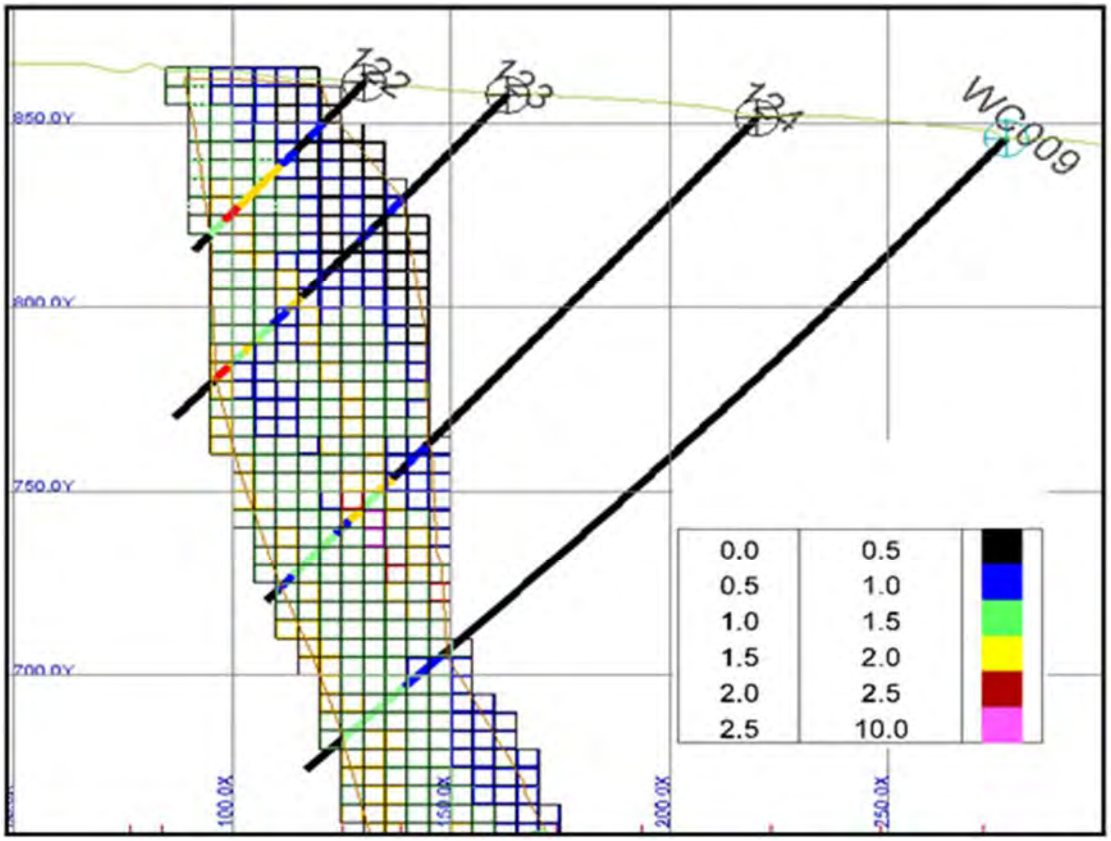
The block model was validated through a detailed visual validation on section and plan views. The model was checked for proper coding of drill hole intervals and block model cells. Coding was found to be properly done. Grade interpolation was examined relative to drill hole composite values by inspecting sections and plans. The checks showed good agreement between drill hole composite values and model cell values (Figure 14.5 and Figure 14.6).

Figure 14.5: Cross Section 4800 S showing Drill Hole Composites and Total Copper Block Model Grades for Zone 13



Note: Grid lines are 50 m apart.
Source: ACS (2016)

Figure 14.6: Cross Section 1700 N showing Drill Hole Composites and Total Copper Block Model Grades for Zone 1



Note: grid is 50 by 50 m
Source: ACS (2016)

14.10 Model Classification

Block model quantities and grade estimates for the Carmacks Project were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (the CIM Definition Standards, May 2014) by Dr. Gilles Arseneau, P. Geo. (APEGBC), of ACS, an independent "qualified person" for the purpose of NI 43-101.

Mineral Resource classification is typically a subjective concept, however, industry best practices suggest that resource classification should consider the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating these concepts to delineate regular areas at similar resource classification.

ACS is satisfied that the geological modelling reflects the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support resource evaluation. The sampling information was acquired primarily by core drill holes. Drilling samples were from sections spaced at 30 to 60 m.

ACS considers that blocks estimated 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. All other estimated blocks can be classified in the Inferred Mineral Resource category within the meaning of the CIM Definition Standards.

14.11 Mineral Resource Statement

CIM Definition Standards defines a Mineral Resource as:

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

The “reasonable prospects for eventual economic extraction” requirement generally 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, ACS considers that major portions of the Carmacks deposits are amenable for open pit extraction.

In order to determine the quantities of material satisfying “reasonable prospects for economic extraction”, ACS assumed a minimum mining cut-off of 0.25% total copper for Zones 1, 4 and 7. Because Zones 12, 13 and 2000S are on average shallower than the other deposits, ACS decided to use a 0.15% soluble copper to estimate the oxide and transition Mineral Resources in these zones and a 0.25% total copper for the sulphide mineralization. The reader is cautioned that there are no Mineral Reserves at the Carmacks deposits.

ACS is unaware of any known environmental, permitting, legal, title, taxation, socio- economic, marketing, political issues that may adversely affect the Mineral Resources presented in this report.

ACS considers that the blocks with grades above the cut-off grade satisfy the criteria for “reasonable prospects for economic extraction” and can be reported as a Mineral Resource. Oxide and transition Mineral Resources for each deposit at the Carmacks Project are summarized in Table 14.13. Sulphide Mineral Resources are summarized in Table 14.14 and total Mineral Resources are summarized in Table 14.15.

Table 14.13: Mineral Resource Statement for Oxide and Transition Mineralization at the Carmacks Project January 25, 2016

Deposit	Class	Cut-off	Tonnes	Total Cu	Soluble Cu	Au	Ag	Sulphide
Zone 1 oxide	Measured	0.25% CuT	2985	1.25	1.02	0.70	6.5	0.23
Zone 4 oxide	Measured	0.25% CuT	614	0.48	0.37	0.21	2.4	0.11
Zone 7 oxide	Measured	0.25% CuT	432	0.97	0.82	0.38	4.4	0.15
Zone 12 oxide	Measured	0.15% CuOx	522	0.50	0.37	0.10	2.4	0.13
Zone 13 oxide	Measured	0.15% CuOx	1501	0.44	0.35	0.12	1.5	0.09
Zone 13 transition	Measured	0.15% CuOx	286	0.48	0.23	0.13	1.7	0.28
Zone 2000S oxide	Measured	0.15% CuOx	144	0.74	0.55	0.30	3.7	0.17
Total Measured	**Measured**		**6484**	**0.86**	**0.69**	**0.41**	**4.2**	**0.17**
Zone 1 oxide	Indicated	0.25% CuT	7058	1.07	0.86	0.41	4.1	0.21
Zone 4 oxide	Indicated	0.25% CuT	257	0.51	0.35	0.18	2.2	0.16
Zone 7 oxide	Indicated	0.25% CuT	634	0.90	0.74	0.32	4.2	0.16
Zone 12 oxide	Indicated	0.15% CuOx	317	0.54	0.4	0.09	2.7	0.14
Zone 13 oxide	Indicated	0.15% CuOx	315	0.38	0.30	0.12	1.3	0.08
Zone 13 transition	Indicated	0.15% CuOx	359	0.70	0.30	0.16	2.3	0.41
Zone 2000S oxide	Indicated	0.15% CuOx	267	0.60	0.46	0.19	2.9	0.13
Total Indicated	**Indicated**		**9206**	**0.97**	**0.77**	**0.36**	**3.8**	**0.20**
Zone 1 oxide	Inferred	0.25% CuT	64	0.84	0.62	0.12	1.8	0.22
Zone 4 oxide	Inferred	0.25% CuT	23	0.41	0.25	0.14	1.9	0.16
Zone 7 oxide	Inferred	0.25% CuT	3	0.81	0.64	0.18	1.6	0.18
Zone 12 oxide	Inferred	0.15% CuOx	36	0.55	0.40	0.11	3.7	0.16
Zone 13 oxide	Inferred	0.15% CuOx	413	0.28	0.23	0.11	1.3	0.05
Zone 13 transition	Inferred	0.15% CuOx	106	0.52	0.24	0.12	1.8	0.28
Zone 2000S oxide	Inferred	0.15% CuOx	267	0.57	0.34	0.14	2.7	0.24
Total Inferred	**Inferred**		**913**	**0.45**	**0.30**	**0.11**	**1.9**	**0.15**

Source: ACS (2016)

Table 14.14: Mineral Resource Statement for Sulphide Mineralization at the Carmacks Project January 25, 2016

Deposit	Class	Cut-off (% CuT)	Tonnes (000)	Total Cu (%)	Soluble Cu (%)	Au (g/t)	Ag (g/t)	Sulphide Cu (%)
Zone 1 sulphide	Measured	0.25	695	0.8	0.02	0.26	2.5	0.77
Zone 12 sulphide	Measured	0.25	178	0.49	0.12	0.07	2.3	0.37
Zone 13 sulphide	Measured	0.25	485	0.46	0.04	0.11	1.5	0.43
Zone 2000S sulphide	Measured	0.25	24	0.75	0.4	0.31	4.1	0.35
Total sulphide Measured	**Measured**	**0.25**	**1,381**	**0.64**	**0.05**	**0.19**	**2.2**	**0.59**
Zone 1 sulphide	Indicated	0.25	3,645	0.74	0.03	0.21	2.3	0.71
Zone 12 sulphide	Indicated	0.25	639	0.69	0.08	0.11	2.9	0.63
Zone 13 sulphide	Indicated	0.25	1,804	0.57	0.04	0.13	1.9	0.55
Zone 2000S sulphide	Indicated	0.25	599	0.73	0.11	0.18	3.4	0.62
Total sulphide Indicated	**Indicated**	**0.25**	**6,687**	**0.69**	**0.04**	**0.17**	**2.3**	**0.65**
Zone 1 sulphide	Inferred	0.25	4,031	0.71	0.01	0.18	1.9	0.7
Zone 12 sulphide	Inferred	0.25	263	0.52	0.06	0.08	1.9	0.46
Zone 13 sulphide	Inferred	0.25	3,552	0.5	0.04	0.12	1.7	0.48
Zone 2000S sulphide	Inferred	0.25	561	0.88	0.07	0.2	4.6	0.85
Total sulphide Inferred	**Inferred**	**0.25**	**8,407**	**0.63**	**0.03**	**0.15**	**2**	**0.61**

Source: ACS (2016)

Table 14.15: Carmacks Project Mineral Resource Statement January 25, 2016

	Class	Tonnes (000)	Total Cu (%)	Soluble Cu	Au	Ag	Sulphide Cu
Oxide and Transition mineralization	Measured	6,484	0.86	0.69	0.41	4.24	0.17
	Indicated	9,206	0.97	0.77	0.36	3.80	0.20
	Measured +	15,690	0.94	0.74	0.38	3.97	0.20
	Inferred	913	0.45	0.30	0.12	1.90	0.15
Sulphide mineralization	Measured	1,381	0.64	0.05	0.19	2.17	0.59
	Indicated	6,687	0.69	0.04	0.17	2.34	0.65
	Measured +	8,068	0.68	0.05	0.18	2.33	0.65
	Inferred	8,407	0.63	0.03	0.15	1.99	0.61

Source: ACS (2016)

Mineral resources were estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserve Best Practices” Guidelines. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. The Mineral Resources may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors. There is insufficient information in this early stage of study to assess the extent to which the Mineral Resources will be affected by these factors that are more suitably assessed in a conceptual study.

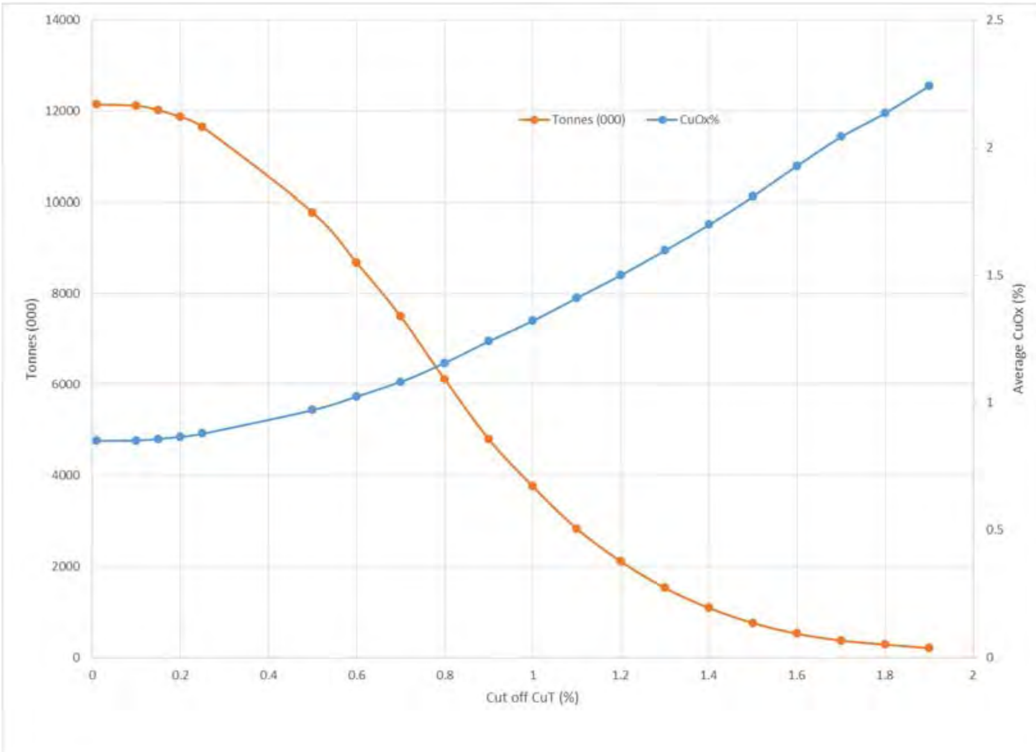
Mineral reserves can only be estimated based on the results of an economic evaluation as part of a Preliminary Feasibility Study or Feasibility Study. As such, no Mineral Reserves have been estimated by ACS. There is no certainty that all or any part of the Mineral Resources will be converted into a mineral reserve.

Inferred Mineral Resources have a great amount of uncertainty as to their existence and as to whether they can be mined legally or economically. It cannot be assumed that all or any part of the Inferred Mineral Resources will ever be upgraded to a higher category. Mineral resources that are not Mineral Reserves have no demonstrated economic viability.

14.12 Grade Sensitivity Analysis

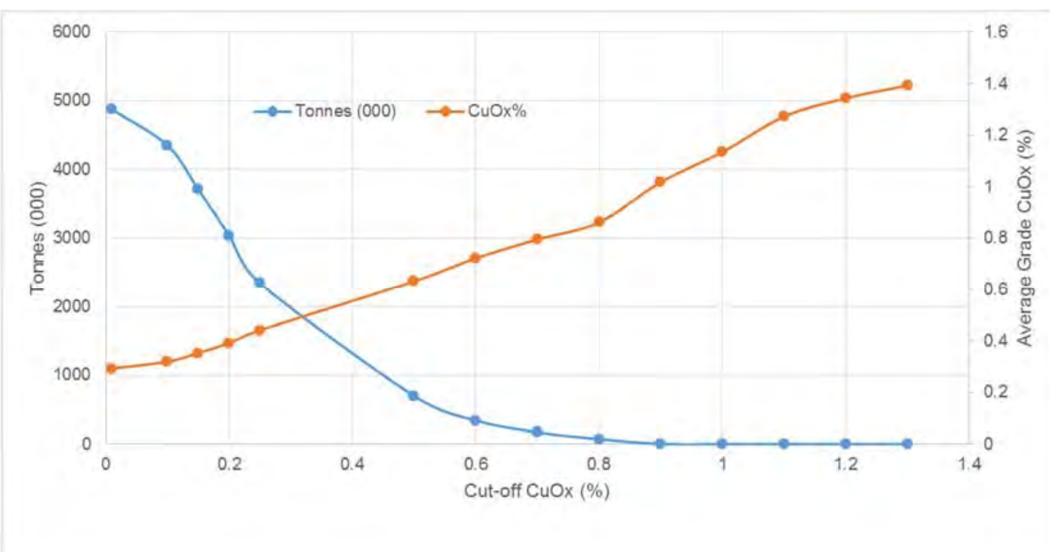
The Mineral Resources at the Carmacks are sensitive to the selection of the reporting cut-off grade. To illustrate this sensitivity, the global model quantities and grade estimates of the Measured and Indicated oxide resource for Zones 1, 4, and 7 are presented in Figure 14.7 and the Measured and Indicated oxide resources for Zones 12, 13 and 2000S are presented in Figure 14.8. As can be seen, the Mineral Resources for Zones 1, 4, and 7 are generally higher grade than the Mineral Resources for Zones 12, 13 and 2000S. The reader is cautioned that the figures presented in these figures should not be misconstrued as a Mineral Resource statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade.

Figure 14.7: Grade tonnage Curve for Measured and Indicated Oxide Mineral Resource for Zones 1, 4 and 7



Source: ACS (2016)

Figure 14.8: Grade tonnage curve For Measured and Indicated Oxide Mineral Resource for Zones 12, 13 and 2000S



Source: ACS (2016)

15 Mineral Reserve Estimates

This PEA report does not state a mineral reserve.

16 Mining Methods

16.1 Mining Methods

16.1.1 Operating Parameters and Criteria

The mine plan for the Carmacks Project remains unchanged from that developed by IMC (2012) and detailed in the 2012 FS. Mineralized material production is envisioned to be 1.775 Mt/a, at an average rate of 4,860 t/d. The peak total material rate is 13 Mt/a. Mining will be conducted on two 12- hour shifts per day for 335 days per year. It was specified that this was to be envisioned to be conducted with three mining crews using a 20-day on/10-day off rotation. This will result in a high amount of overtime pay compared to most mining operations.

With the current mine production schedule, the commercial project life will be about 6.5 years after a brief pre-production period.

16.2 Pit and Mining Phase Design

Four mining phases were designed for the Carmacks Project. Inter-ramp slope angles are 52.6°. The design is also based on 10 m mining benches in a double bench configuration for final walls. The main road is 25 m wide at a maximum grade of 10%. This will accommodate trucks of approximately 90 t such as Caterpillar 777 class trucks.

- Phase 1 (Figure 16.1) is based on the northwest end of the $1.25 copper floating cone. This was not designed as a double bench configuration; there are no final walls in this phase;
- Phase 2 (Figure 16.2) is a push to the southeast along about the $1.75 copper cone economic boundary. The southeast end of the pit is at the final wall and is shown in the double bench configuration;
- Phase 3 (Figure 16.3) is the final pit configuration for the main pit. It is based on the $2.50 copper floating cone; and
- Phase 4 (Figure 16.4) is the small southeast pit.

Figure 16.1: Mining Phase 1

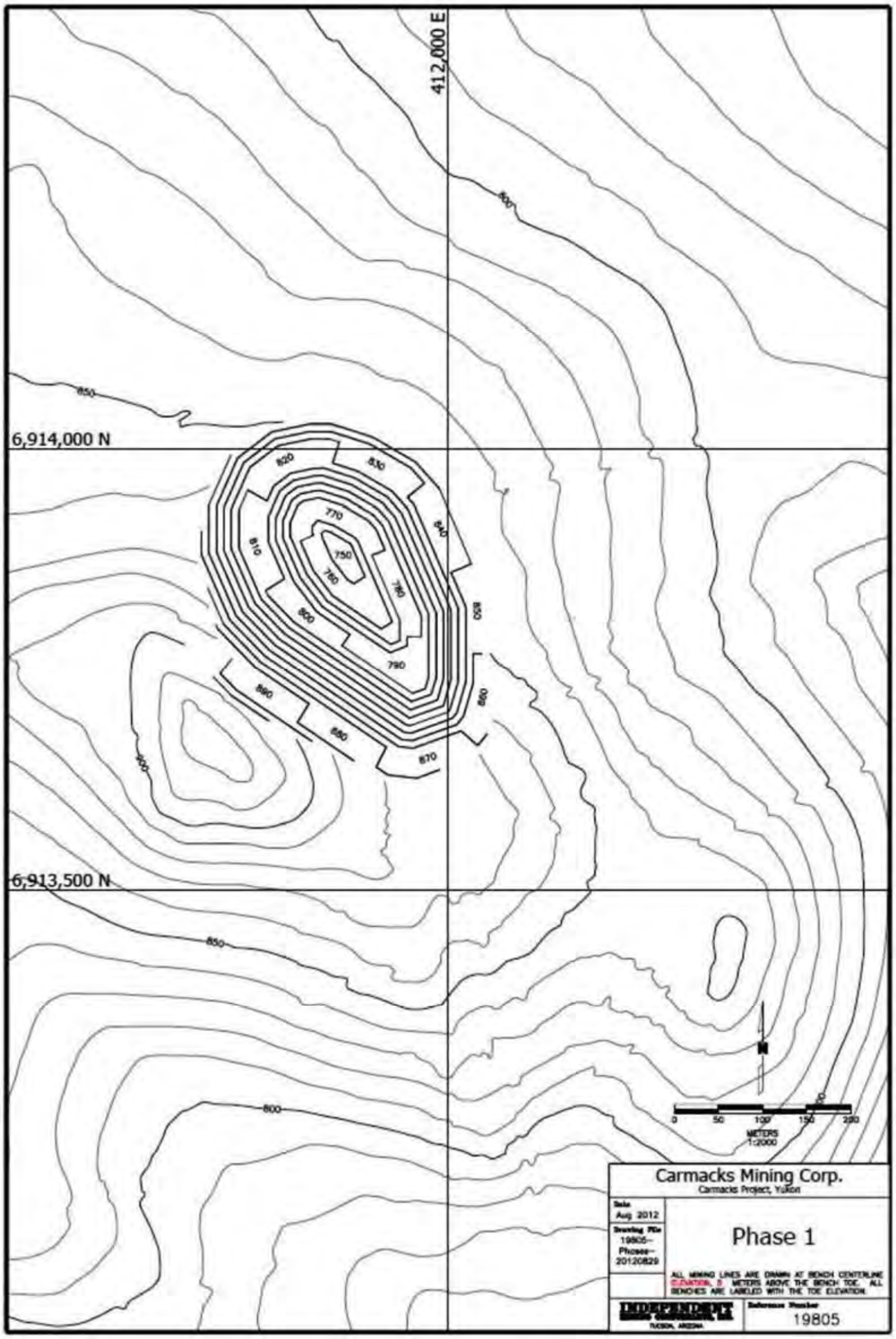


Figure 16.2: Mining Phase 2

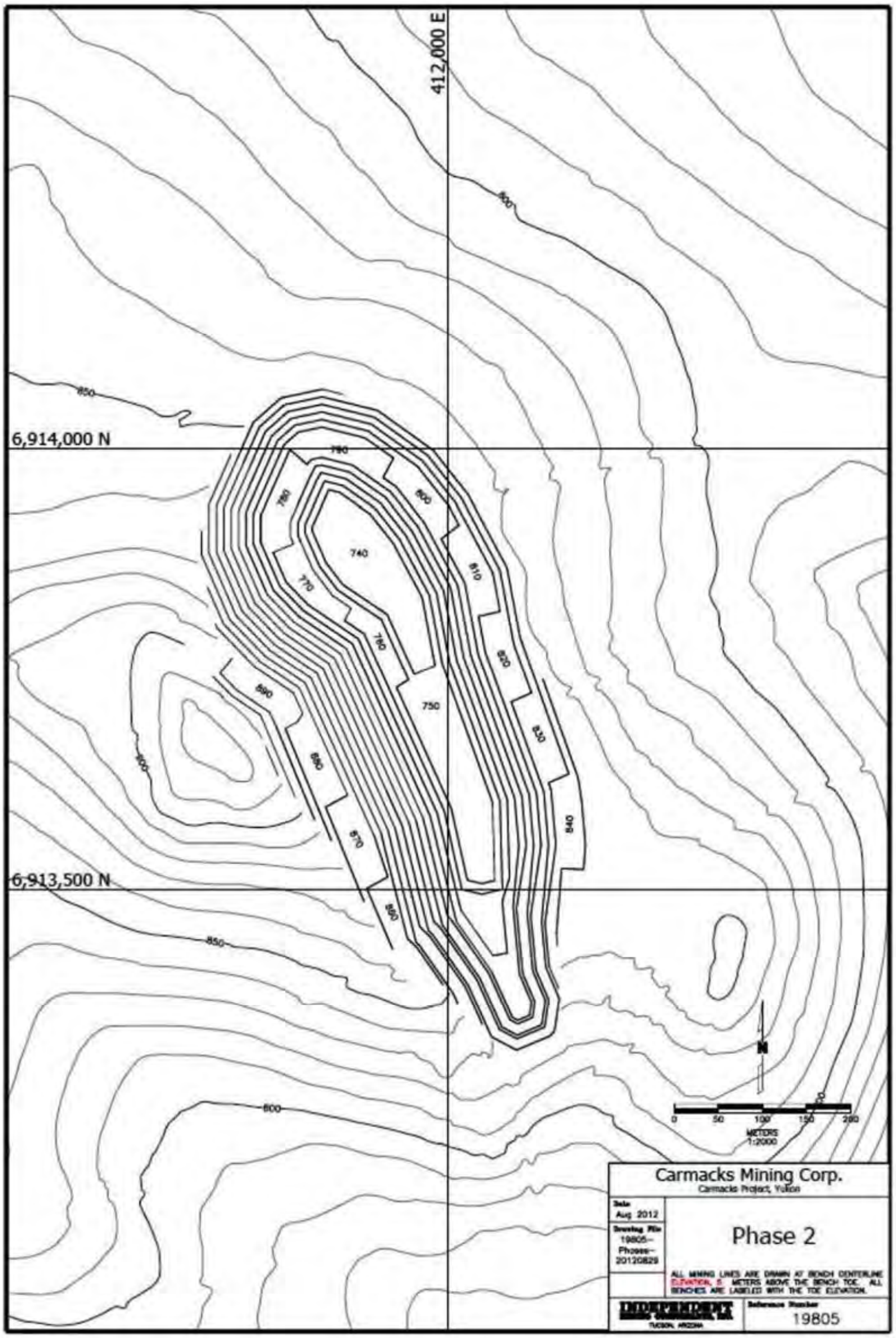


Figure 16.3: Mining Phase 3

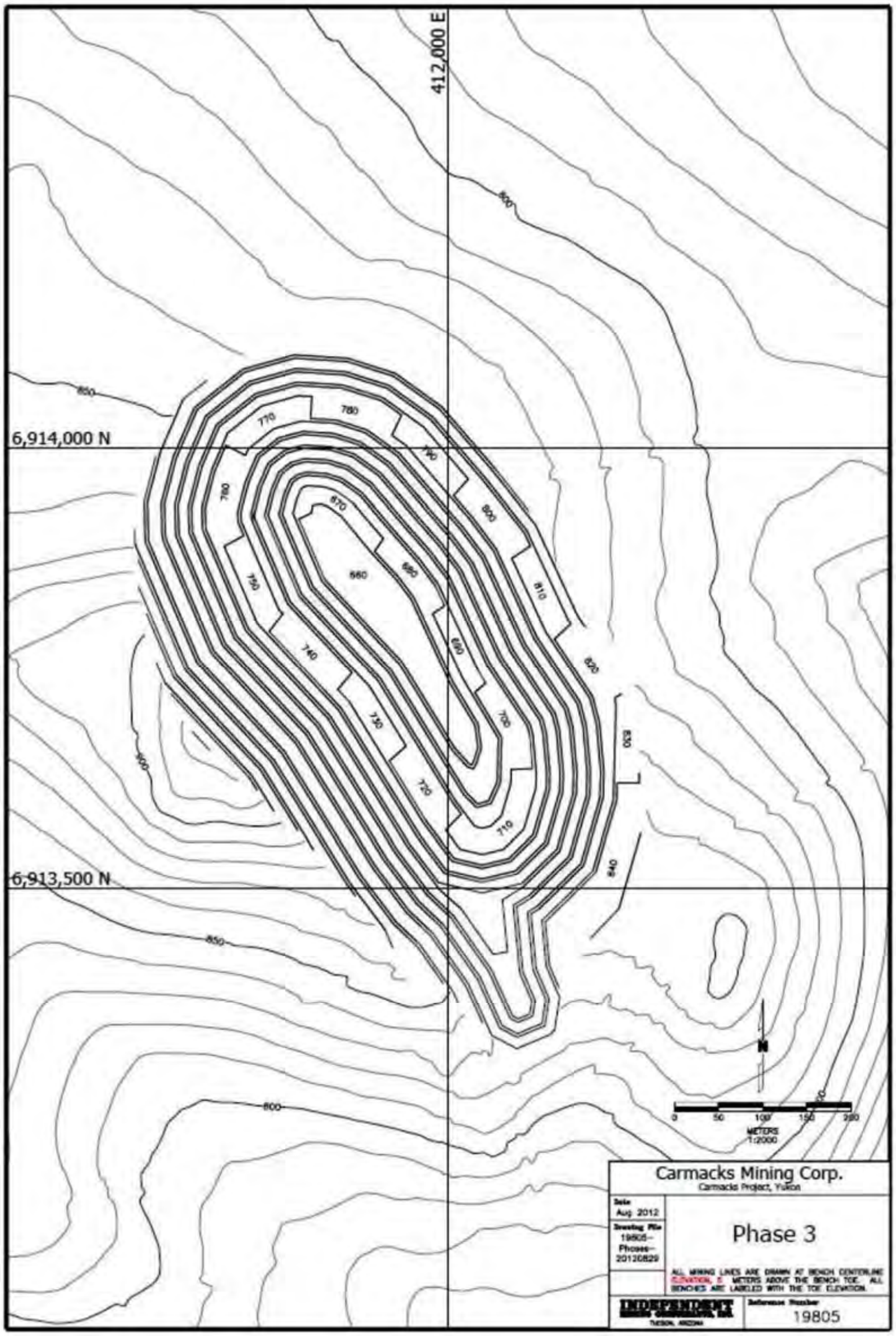
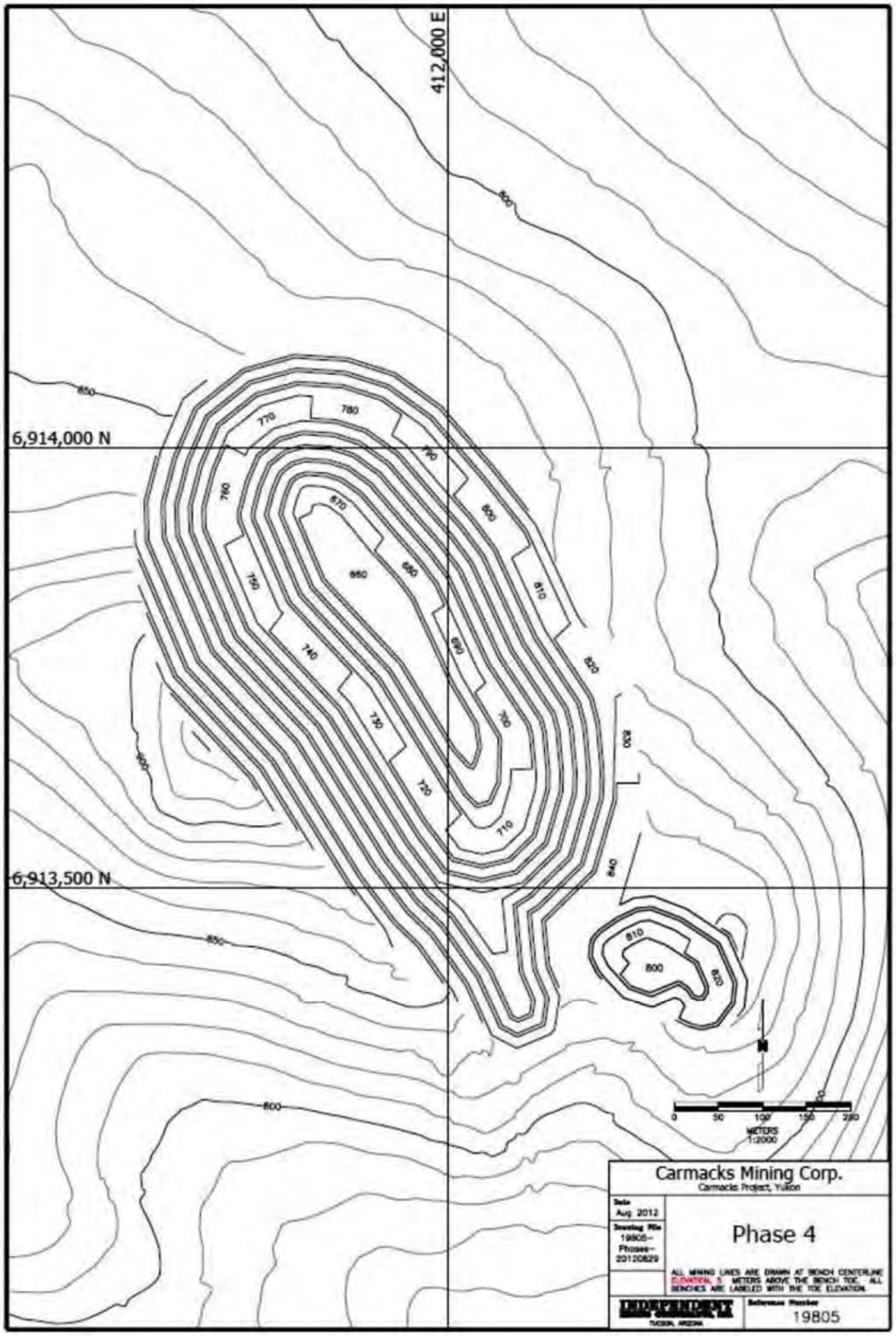


Figure 16.4: Mining Phase 4



Earlier (2007/2008) pit designs for the Carmacks project kept the roads off the highwall side of the pit. However, by moving the road to the highwall, the number of bench set-backs will be reduced from three, as recommended by Golder (2008a), to one set-back, on about the 840 bench. Table 16.1 shows the tonnages by mining phase. As with the cones, the tonnage tabulation is on a diluted basis. The cut-off grade for the table is based on blocks above 0.18% recovered copper, internal cut-off at the $2.50 copper price, prior to application of dilution.

Table 16.1: Carmacks Mining Phases Diluted Model

Phase	Ktonnes	Rec Cu (%)	Tot Cu (%)	Sol Cu (%)	Sulf Cu (%)	Gold (g/t)	Silver (g/t)	Waste Ktonnes	Total Ktonnes	Waste: mineralized material
1	3,273	0.876	1.073	0.887	0.187	0.57	5.27	7,897	11,170	2.4
2	3,658	0.725	0.89	0.744	0.146	0.318	3.53	15,726	19,384	4.3
3	4,083	0.848	1.051	0.857	0.194	0.463	4.59	34,144	38,227	8.4
4	537	0.341	0.431	0.335	0.096	0.188	2.22	639	1,176	1.2
Total	11,551	0.793	0.977	0.805	0.172	0.435	4.34	58,406	69,957	5.1

Note: Cut-off based on blocks above 0.18% recovered copper prior to application of dilution.
Source: IMC (2012)

16.3 Mine Production Schedule

A mine production schedule was developed to estimate annual mineralized material and waste movements from the pit. Table 16.2 shows the mine production schedule. The schedule is based on mining 1.775 Mt/a of mineralized material. Total material is 70.0 Mt for a waste to mineralized material ratio of 5.1 to 1. Pre-production will be minimal at 953 kt. The total material movement will be 9.5 Mt during Year 1 and will peak at 13.5 Mt for Years 2 through 4. The waste to mineralized material ratio will be 6.6 to 1 during these peak years.

Table 16.2: Mine Production Schedule

Mine Production Schedule										
		PP	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**Total**
Mill Feed	Kt	150	1,625	1,775	1,775	1.775	1,775	1,775	901	11,551
Recovered Copper	%	0.701	0.792	0.752	0.828	0.774	0.709	0.859	0.957	0.793
Total Copper	%	0.867	0.965	0.932	1.019	0.907	0.896	1.065	1.204	0.977
Soluble Copper	%	0.691	0.802	0.756	0.839	0.777	0.773	0.864	0.943	0.805
Sulphide Copper	%	0.176	0.164	0.176	0.18	0.13	0.136	0.201	0.261	0.172
Gold	g/t	0.306	0.472	0.411	0.49	0.343	0.412	0.462	0.497	0.435
Silver	g/t	2.99	4.43	4.26	4.84	3.59	3.91	4.59	5.4	4.34
Total Tonnes Moved	Kt	953	9,500	13,500	13,500	13,500	11,776	5,821	1,407	69,957
Waste Kt	Kt	803	7,875	11,725	11,725	11,725	10,001	4,046	506	58,406
Strip Ratio	Waste t: Mill Feed	5.4	4.8	6.6	6.6	6.6	5.6	2.3	0.6	5.1

16.4 Waste Rock Storage Areas

The WRSA is located north of the pit (Figure 16.6) and was designed to contain the required 60 Mt of waste that is to be mined, as detailed in Section 18.5.2.

Figure 16.5: End of Year 4

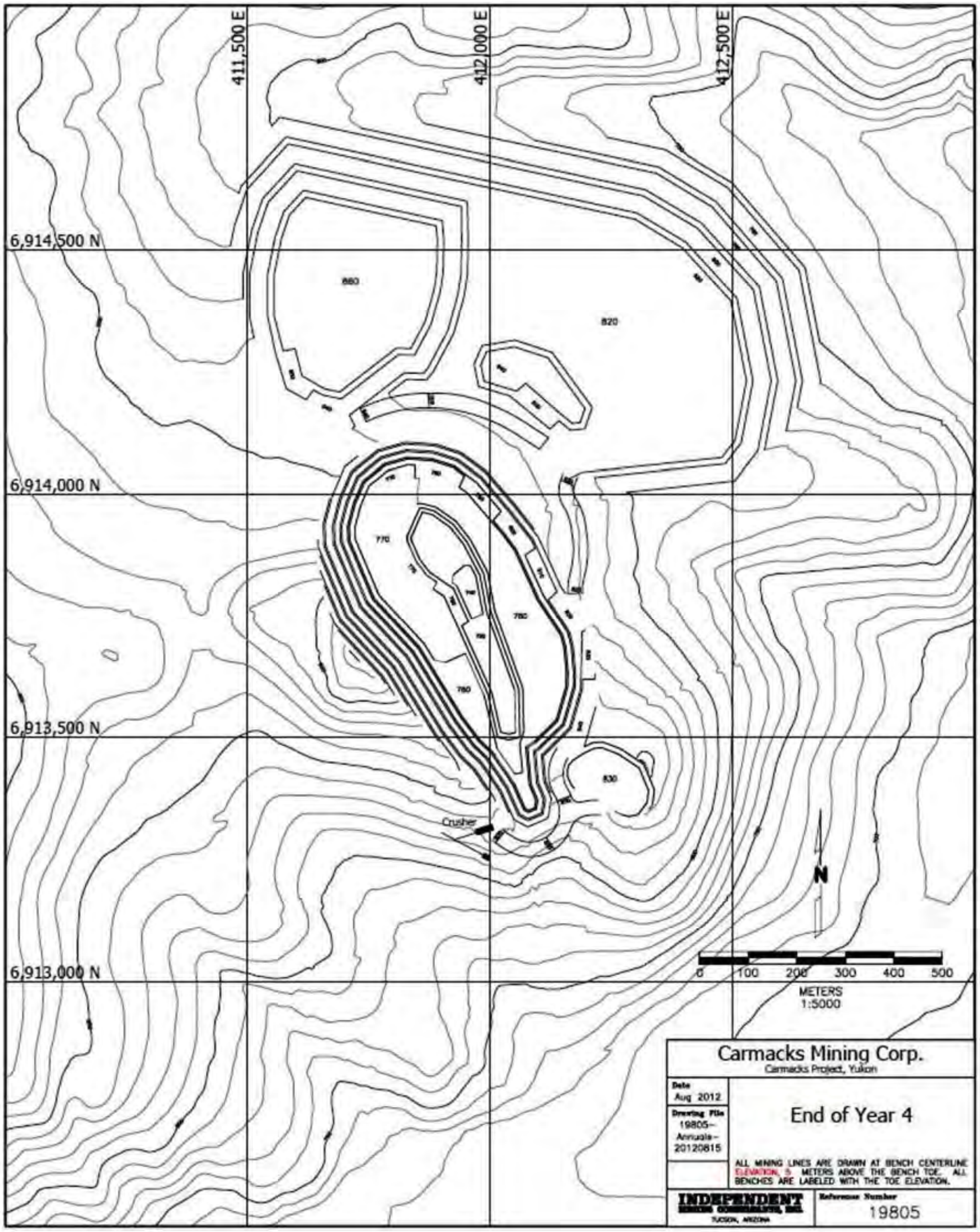
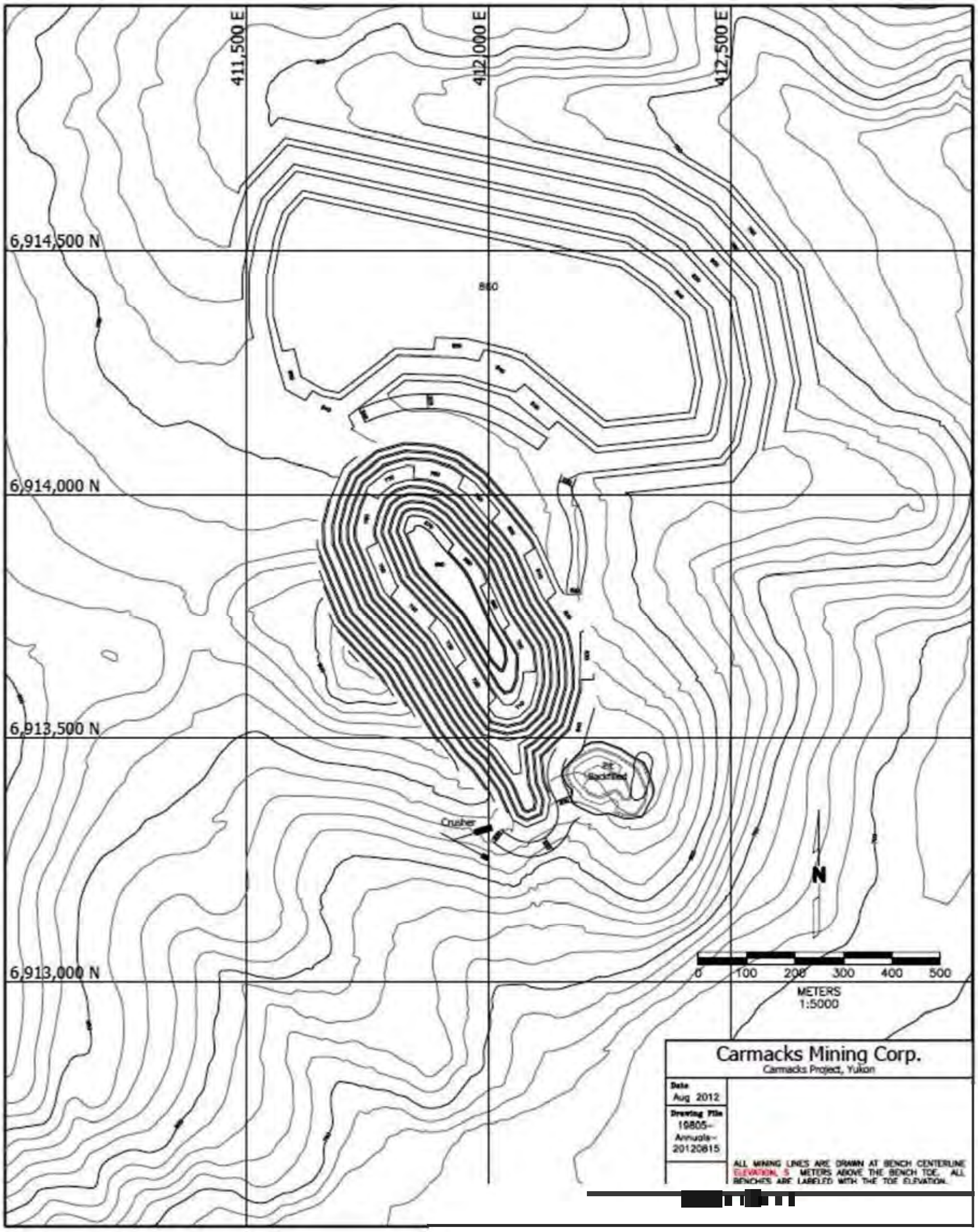


Figure 16.6: End of Year 7



16.5 Mine Equipment

16.5.1 Summary of Equipment Requirements

Mine major equipment requirements for the Carmacks copper mine were sized and estimated on a first principles basis based on the mine production schedule, the mine work schedule, and estimated equipment productivity rates. The work schedule is based on two 12-hour shifts per day for 335 days per year. The mine equipment estimate is based on an owner operation and assumes a well-managed mining operation with a well-trained labour pool, and new equipment at the start of mining development. Table 16.3 shows major equipment requirements by mining period.

Table 16.3: Mine Major Equipment Fleet Requirement

Equipment Type	Capacity /Power	Time Period								
		PP	1	2	3	4	5	6	7	8
Caterpillar MD6240 Hyd Drill	210 mm	1	1	2	2	2	2	1	1	0
Komatsu PC2000 Hyd Shovel	11 m^3	0	1	1	1	1	1	1	1	0
Cat 992K Wheel Loader	10.7 m^3	1	1	1	1	1	1	1	1	0
Cat 777F Truck	90 m	2	5	6	7	7	7	5	3	0
Cat D9T Track Dozer	306 kw	2	2	2	2	2	2	2	1	0
Cat 824H Wheel Dozer	264 kw	1	1	1	1	1	1	1	1	0
Cat 14M Motor Grader	193 kw	1	1	1	1	1	1	1	1	0
Water Truck – 10,000 gal	37,800 L	1	1	1	1	1	1	1	1	0
Atlas Copco ECM 720 Drill	140 mm	1	1	1	1	1	1	1	1	0
Cat 336D Excavator	1.93 m^3	1	1	1	1	1	1	1	1	0
Total		**11**	**15**	**17**	**18**	**18**	**18**	**15**	**12**	**0**

Source: IMC (2012)

This represents the equipment required to perform the following duties:

- Developing access roads from the mine to the crusher and waste dumps;
- Mining and transporting mineralized material to the crusher;
- Mining and transporting waste to the various waste storage facilities; and
- Maintaining the haul roads and dumps.

The equipment list does not include equipment for construction or operation of the plant and the TMA.

16.6 Production and Operating Parameters

16.6.1 Mine Operating Schedule

Table 16.4 shows the mine operating schedule used as the basis of the equipment calculations. The left half of the table shows the mine material movements by material type by time period. The right half of Table 16.4 shows the mine operating schedule. It can be seen that the mine is scheduled to operate two shifts per day (12 hours per shift) for 335 days per year for 670 available shifts per year. CNMC specified that three mining crews would be used on a 20-day on/10-day off rotation as shown in the table. This will result in a relatively high overtime pay allowance compared to most mining operations.

Table 16.4: Summary of Mine Material Movements and Mine Operations Schedule

Time Period	Mine Material Movements					Miner Operations Schedule						
	Mineralized Material (kt)	OB (kt)	Waste (kt)	Rehandle (kt)	Total (kt)	Sched Days	Shifts/ Day	Sched Shifts	Avail Shifts	Avail Hours	Mining Crews	Partial Year (%)
PP	150	329	474	0	953	168	1	168	168	2,016	2	50.1
Year 1	1,625	816	7,059	150	9,650	335	2	670	670	8,040	3	100.0
Year 2	1,775	1,097	10,628	0	13,500	335	2	670	670	8,040	3	100.0
Year 3	1,775	72	11,653	0	13,500	335	2	670	670	8,040	3	100.0
Year 4	1,775	130	11,595	0	13,500	335	2	670	670	8,040	3	100.0
Year 5	1,775	1	10,000	0	11,776	335	2	670	670	8,040	3	100.0
Year 6	1,775	0	4,046	0	5,821	335	2	670	670	8,040	3	100.0
Year 7	901	0	506	0	1,407	168	2	336	336	4,032	3	50.1
Year 8	0	0	0	0	0	0	0	0	0	0	0	0
Total	**11,551**	**2,445**	**55,961**	**150**	**70,107**	**2,346**		**4,524**	**4,524**	**54,288**		

Source IMC (2012)

16.6.2 Operating Time per Shift

Operating time per shift represents the actual time during the shift that the equipment is “productive”. This is equal to the total shift time less all scheduled and unscheduled delays.

Table 16.5: Summary of Operating Time per Shift

Scheduled Time Per Shift	(Minutes)	(Hours)
Less Schedule Nonproductive Times	720	12
Travel Time/Shift Change/Blasting	15	0.25
Equipment Inspection	0	0
Lunch/Breaks	60	1
Fueling, Lube & Services	0	0
Net Scheduled Productive Time (Metred Operating Time)	645	10.75
Job Efficiency Based on 50.0 Productive Minutes/Hour	83.30%	83.30%
Net Productive Operating Time Per Shift	538	8.96
Overall Mine Efficiency Factor	74.65%	75.65%

Source: IMC (2012)

16.6.3 Material Characteristics

Table 16.6 summarizes the material characteristics used for equipment productivity calculations. In-situ bulk densities are 2.64 t/m^3 for mineralized material, 2.66 t/m^3 for waste rock and about 2 t/m^3 for overburden. IMC assumed a material handling swell factor of 40% for rock and 30% for overburden. Moisture content of the material is considered negligible for material handling purposes. An estimated strength index is also shown that is used in the drilling and blasting requirement calculations. Based on uniaxial compression tests, performed under the supervision of Golder, the materials appear to be of moderate strength. Golder presented results from 35 uniaxial compressive strength tests that averaged 90 mpa or about 13,000 psi compressive strength. These were reported to be mostly in granodiorite wall rock which is probably stronger than the mineralized material. IMC assigned a moderate strength index to waste rock and weak index to the mineralized material.

Table 16.6 Material Characteristics

Parameter	Units	Leach Mineralized material	Over Burden	Waste Rock	Mineralized material Rehand
Bulk Density					
Dry Bank Density	m/m^3	2.64	2.00	2.66	2.00
Material handling Swell	%	40.0	30.0	40.0	10.0
Moisture Content	%	3.0	5.0	3.0	3.0
Dry Loose Density	m/m^3	1.89	1.54	1.90	1.82
Wet Loose Density	m/m^3	1.94	1.62	1.96	1.87
Material Strength					
Strength Index (1-5)	no units	4	5	3	6
Nominal Compressive Strength	psi	10,000	5,000	15,000	1,000
Nominal Compressive Strength	mpa	69	34	103	7
Drill/Blast This Material?	No units	yes	yes	yes	no
Notes:					
Strenth Index: 1= Very Strong, 2=Strong, 3= Moderate, 4= Weak, 5=very Weak, 6=non-Drilled/Blasted					
Description of Strength					
Index					
IMC	Brown				
Index	Index	Description			
1	R6	Specimen can only be chipped with a geologic hammer			
2	R5	Specimen required many blows with hammer to fracture			
3	R4	More than one blow to fracture			
4	R3	Can be Fractured with single blow			
5	R2	Can be peeled with knife with difficulty, can indent with firm hammer blow			
6	R0-R1	Crumbles under firm blow with hammer, can be peeled with pocket knife			

Source: IMC (2012)

16.7 Drilling

The drilling fleet consists of diesel powered drills with a pulldown of about 50,000 lbs or 22,680 kg, such as the Caterpillar MD6240 drill (formerly a Bucyrus/Terex SKFX drill). Material will be drilled with 210 mm diameter holes on 10 m mining benches with 2 m of subgrade drilling.

Shift productivities are estimated at 24,230 t for mill feed material and 17,184 t for waste rock. Productivity in overburden is estimated at 40,602 t per shift. Annual production is estimated at 12.4 Mt per drill for mill feed material, 8.8Mt per drill for waste rock, and 20.8Mt for overburden.

The productivity calculations are based on a powder factor of 200 g/t for mill feed material, 250 g/t for waste rock, and 100 g/t for overburden. Drill penetration is estimated at 0.75 m/min for mill feed material, 0.6 m/min for waste rock, and 1 m/min for overburden. The table also shows the spacing between holes is about 6 m in mill feed material, 5.5 m in waste rock, and 8 m in overburden. Table 16.7 shows the relationship between drill penetration rate and the average drilling rate, which allows for moving the drill, etc.

Table 16.8 summarizes drilling requirements by year. This includes the required drilling shifts per year, the fractional drill fleet, the actual drill fleet, and fleet utilization. One drill is required for pre-production and Year 1 production and two drills are required for Years 2 through 5.

The equipment list also includes a small drill capable of drilling about 140 mm or 5.5 inch holes. This will be used as backup to the primary production drills, construction activities, such as roads, and will also be used for wall control blasting for the final pit wall. Shifts for this drill are included under the support equipment section at the end of this chapter. The costs are included in the roads and dumps cost centre.

Table 16.7: Penetration Rate and Peak Drilling Rate by Material Type

Caterpillar MD6240 Drill					
	Units		Leach Mineralized material	Over Burden	Waste Rock
Hole Depth	m		12	12	12
Penetration Rate	m/min	*	0.75	1	0.59
Penetration Time Per Hole	min		16.1	11.9	20.2
Move Time	min	*	5	5	5
Pipe Length	m		12.8	12.8	12.8
Steel Changes	m		0	0	0
Time Per Steel Change	none	*	2	2	2
Total Time Per Hole	min	*	21.1	16.9	25.2
Holes Per Hour	holes		2.85	3.54	2.38
Average Drilling Rate	m/hr		34.2	42.5	28.6

Source: IMC (2012)

Table 16.8: Drill Requirements Caterpillar MD6240 Drill (210 mm)

	Units	PP	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Drilled Material											
Leach Mineralized material	kt	150	1,625	1,775	1,775	1,775	1,775	1,775	901	0	11,551
Overburden	kt	329	816	1,097	72	130	1	0	0	0	2,445
Waste	kt	474	7,059	10,628	11,653	11,595	10,000	4,046	506	0	55,961
Mineralized material Rehandle	kt	0	0	0	0	0	0	0	0	0	0
Total Material	kt	953	9,500	13,500	13,500	13,500	11,776	5,821	1,407	0	69,957
Required Drill Shifts											
Leach Mineralized material	Shifts	6	67	73	73	73	73	73	37	0	477
Overburden	Shifts	8	20	27	2	3	0	0	0	0	60
Waste	Shifts	28	411	618	678	675	582	235	29	0	3,257
Mineralized material Rehandle	Shifts	0	0	0	0	0	0	0	0	0	0
Total Shifts	Shifts	42	498	719	753	751	655	309	67	0	3,796
Productivity Calculations											
Available Shifts Per Period	shifts	168	670	670	670	670	670	670	336	0	4,524
Mechanical Availability	%	85	85	85	85	85	85	85	85	0	85
Utilization of Availability	%	90	90	90	90	90	90	90	90	0	90
Maximum Utilization Per Drill	%	76.5	76.5	76.5	76.5	76.5	76.5	76.5	76.5	0	76.5
Available Shifts Per Drill	Shifts	129	513	513	513	513	513	513	257	0	
Fractional Number of Drills	none	0.33	0.97	1.4	1.47	1.47	1.28	0.6	0.26	0	
Actual Number of Drills	none	1	1	2	2	2	2	1	1	0	
Fleet Utilization	%	24.9	74.3	53.6	56.2	56.1	48.9	46.1	19.8	0	55
Number of Operators:											
Number of Mining Crews	None	2	3	3	3	3	3	3	3	0	
Number of Drill Operators	None	2	3	4	4	4	3	3	1	0	

Average Drill Production Per Shifts		Drill/Blast	
Leach Mineralized material	t/shift	24,230	Yes
Overburden	t/shift	40,602	Yes
Waste	t/shift	17,184	Yes
Mineralized material Rehandle	t/shift	52,852	No

16.8 Loading

The primary loading fleet is based on hydraulic shovels with an 11 m^3 bucket, such as the Komatsu PC2000 shovel, and wheel loaders with a 10.7 m^3bucket, such as the Caterpillar 992K loader. Both are matched with trucks with a nominal capacity of about 90 metric tonnes such as the Caterpillar 777F truck. The shovel shift productivity (12-hour shift) is estimated at 15,710 t for rock and 13,209 t for overburden. Annual production per shovel is estimated at 8.1 Mt for rock and 6.8 Mt for overburden.

The loader shift productivity is estimated at 11,783 t for rock and 10,051 t for overburden. Annual production per loader is estimated at 6.0 Mt for rock and 5.2 Mt for overburden.

The first and last 50 m of each profile was considered as acceleration/deceleration at an average speed of 10 km/h. Table 16.12 shows that, over the life of mine, the productivity of the Cat 777 trucks is estimated at 3,659 t per truck shift for a 12 h-shift.

Table 16.10 summarizes the shovel requirements by year, including required shifts, fractional fleet, actual fleet, and fleet utilization. One shovel is required for Years 1 through 7.

Table 16.11 summarizes the loader requirements by year. One loader is required for all time periods. Note also that the loading requirements assume 60% of the material is loaded by the shovel and 40% by the loader.

16.9 Hauling

Table 16.12 summarizes haul truck requirements by year. It includes truck shifts, the fractional fleet, actual fleet, and fleet utilization. Two trucks are required for pre-production, five trucks for Year 1, six trucks for Year 2, and seven trucks for Years 3 through 7. To develop the truck haulage requirements, the truck haulage profiles were measured for each material type, for each mining bench, for each mining phase per year. Data collected for each profile was the total distance, total elevation rise and total elevation drop along the profile. Ramps were assumed at a grade of 10%. Average truck speeds were as follows:

Table 16.9: Average Travel Speeds and Ramp Guide

	Flat (kph)	Up (kph)	Down (kph)	Acl/Dcl (kph)
Loaded	45	10	21	10
Empty	45	24	39	10
Ramp Gradient				10.00%
Accelerate/Decelerate Distance			m	50

The first and last 50 m of each profile was considered as acceleration/deceleration at an average speed of 10 k/h. Table 16.12 shows that, over the life of mine, the productivity of the Cat 777 trucks is estimated at 3,659 t per truck shift for a 12-hour shift.

Table 16.10: Shovel Requirements Komatsu PC2000 Hydraulic Shovel (11 m^3)

	Units	PP	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Percent Loaded By Shovel											
Leach Mineralized material	%	0	60	60	60	60	60	60	60	60	
Overburden	%	0	60	60	60	60	60	60	60	60	
Waste	%	0	60	60	60	60	60	60	60	60	
Mineralized material Rehandle	%	0	0	0	0	0	0	0	0	0	
Loaded By shovel											
Leach Mineralized material	kt	0	975	1,065	1,065	1,065	1,065	1,065	541	0	6,841
Overburden	kt	0	490	658	43	78	1	0	0	0	1,270
Waste	kt	0	4,235	6,377	6,992	6,957	6,000	2,428	304	0	33,292
Mineralized material Rehandle	kt	0	0	0	0	0	0	0	0	0	0
Total Material	kt	0	5,700	8,100	8,100	8,100	7,066	3,493	844	0	41,402
Required Shovel Shifts											
Leach Mineralized material	shifts	0	62	68	68	68	68	68	34	0	435
Overburden	shifts	0	37	50	3	6	0	0	0	0	96
Waste	shifts	0	270	406	445	443	382	155	19	0	2,119
Mineralized material Rehandle	shifts	0	0	0	0	0	0	0	0	0	0
Total Shifts	shifts	0	369	524	516	517	450	222	54	0	2,651
Productivity Calculations											
Available Shifts Per Period	shifts	168	670	670	670	670	670	670	336	0	4.524
Mechanical Availability	%	0	85	85	85	85	85	85	85	0	85
Utilization of Availability	%	0	90	90	90	90	90	90	90	0	90
Maximum Utilization Per Shovel	%	0	76.5	76.5	76.5	76.5	76.5	76.5	76.5	0	76.5
Available Shifts Per Shovel	Shifts	0	513	513	513	513	513	513	257	0	3.332
Fractional Number of Shovel	none	0	0.72	1.02	1.01	1.01	0.88	0.43	0.21	0	
Actual Number of Shovels	none	0	1	1	1	1	1	1	1	0	
Fleet Utilization	%	0	55	78.1	77	77.1	67.1	33.2	16	0	67.6
Number of Operators:											
Number of Mining Crews	None	0	3	3	3	3	3	3	3	0	
Number of Shovel Operators	None	0	3	3	3	3	3	3	1	0	

Average Shovel Production Per Shift		
Leach Mineralized material	t/shift	15,710
Overburden	t/shift	13,209
Waste	t/shift	15,710
Mineralized material Rehandle	t/shift	15,710

Source: IMC (2012)

Table 16.11: Loader Requirements Cat 992K Wheel Loader (10.7 m^3)

	Units	PP	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Percent Loaded By Loader											
Leach Mineralized material	%	100	40	40	40	40	40	40	40	40	
Overburden	%	100	40	40	40	40	40	40	40	40	
Waste	%	100	40	40	40	40	40	40	40	40	
Mineralized material Rehandle	%	100	100	100	100	100	100	100	100	100	
Loaded By Loader											
Leach Mineralized material	kt	150	650	710	710	710	710	710	360	0	4,710
Overburden	kt	329	326	439	29	52	0	0	0	0	1,175
Waste	kt	474	2,824	4,251	4,661	4,638	4,000	1,618	202	0	22,669
Mineralized material Rehandle	kt	0	150	0	0	0	0	0	0	0	150
Total Material	kt	953	3,950	5,400	5,400	5,400	4,710	2,328	563	0	28,705
Required Loader Shifts											
Leach Mineralized material	shifts	13	55	60	60	60	60	60	31	0	400
Overburden	shifts	33	32	44	3	5	0	0	0	0	117
Waste	shifts	40	240	361	396	394	339	137	17	0	1,924
Mineralized material Rehandle	shifts	0	13	0	0	0	0	0	0	0	13
Total Shifts	shifts	86	340	465	459	459	400	198	48	0	2,453
Productivity Calculations											
Available Shifts Per Period	shifts	168	670	670	670	670	670	670	336	0	4,524
Mechanical Availability	%	85	85	85	85	85	85	85	85	0	85
Utilization of Availability	%	90	90	90	90	90	90	90	90	0	90
Maximum Utilization Per Loader	%	76.5	76.5	76.5	76.5	76.5	76.5	76.5	76.5	0	76.5
Available Shifts Per Loader	Shifts	129	513	513	513	513	513	513	257	0	3,461
Fractional Number of Loaders	none	0.67	0.66	0.91	0.89	0.9	0.78	0.39	0.19	0	
Actual Number of Loaders	none	1	1	1	1	1	1	1	1	0	
Fleet Utilization	%	51	50.7	69.4	68.5	68.5	59.7	29.5	14.2	0	60
Number of Operators:											
Number of Mining Crews	None	2	3	3	3	3	3	3	3	0	
Number of Loader Operators	None	2	3	3	3	3	3	3	1	0	

Average Shovel Production Per Shift		
Leach Mineralized material	t/shift	11,783
Overburden	t/shift	10,051
Waste	t/shift	11,783
Mineralized material Rehandle	t/shift	11,783

Source: IMC (2012)

Table 16.12: Truck Requirements Cat 777F Truck (90 m)

	Units	PP	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Production Requirements											
Leach Mineralized material	kt	150	1,625	1,775	1,775	1,775	1,775	1,775	901	0	11,551
Overburden	kt	329	816	1,097	72	130	1	0	0	0	2,445
Waste	kt	474	7,059	10,628	11,653	11,595	10,000	4,046	506	0	55,961
Mineralized material Rehandle	kt	0	150	0	0	0	0	0	0	0	150
Total Material	kt	953	9,650	13,500	13,500	13,500	11,776	5,821	1,407	0	70,107
Productivity Calculations											
Required Truck Shifts	shifts	251	2,282	3,100	3,629	3,337	3,594	2,334	634	0	19,162
Required Truck Hours	hours	3,006	27,385	37,198	43,552	40,050	43,131	28,009	7,614	0	229,946
Available Shifts Per Period	shifts	168	670	670	670	670	670	670	336	0	4,525
Mechanical Availability	%	85	85	85	85	85	85	85	85	0	85
Utilization of Availability	%	90	90	90	90	90	90	90	90	0	90
Maximum Utilization Per Truck	%	76.5	76.5	76.5	76.5	76.5	76.5	76.5	76.5	0	76.5
Available Shifts Per Truck	Shifts	129	513	513	513	513	513	513	257	0	3,461
Fractional Number of Trucks	trucks	1.95	4.45	6.05	7.08	6.51	7.01	4.55	2.47	0	
Actual Number of Trucks	trucks	2	5	6	7	7	7	5	3	0	
Fleet Utilization	%	74.6	68.1	77.1	77.4	71.2	76.6	69.7	62.9	0	73.6
Number of Operators											
Number of Mining Crews	crews	2	3	3	3	3	3	3	3	0	
Number of Truck Operators	operator	4	12	15	18	15	18	12	6	0	
Tonnes Per Truck Shift	tonnes	3,804	4,229	4,355	3,720	4,045	3,276	2,494	2,218	0	3,659

Source: IMC (2012)

16.10 Support Equipment

The mine support equipment includes the following equipment types. This equipment is used to maintain roads and dumps and to support the primary drilling, loading, and hauling fleet.

- Track Dozer, 306 kw (2 units);
- Wheel Dozer, 264 kw (1 unit);
- Motor Grader, 193 kw (1 unit);
- Water Truck, 37,800 litre (1 unit);
- Excavator, 1.9 m^3 (1 unit); and
- Drill, 140 mm (1 unit).

In addition to road construction activities, the small drill will also be used for wall control blasting on the final pit wall and backup to the primary production drills.

17 Process Description/Recovery Methods

17.1 Introduction

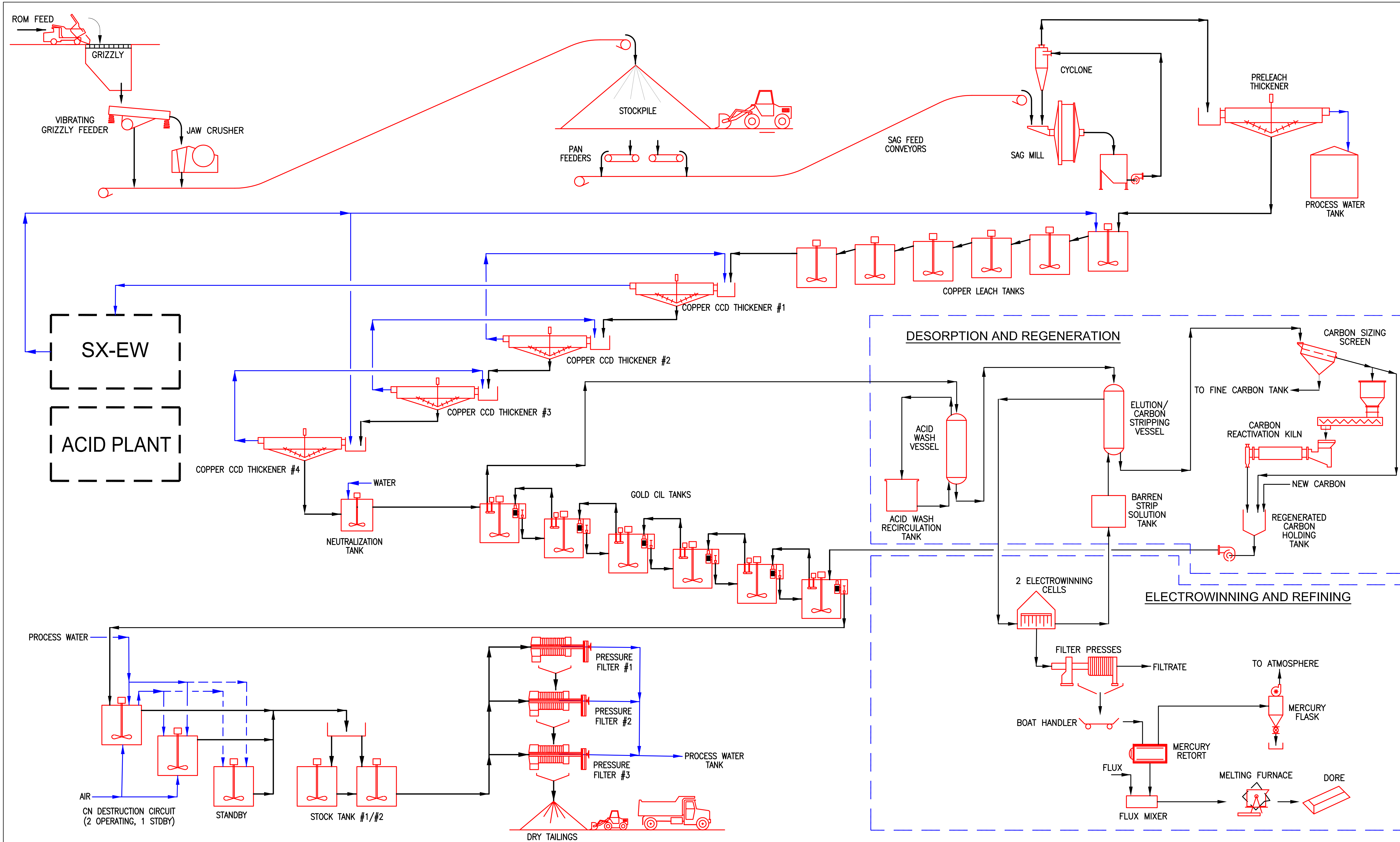
This section describes the recovery methods proposed for the project. Flowsheet development, operating parameters and design criteria were based on results from metallurgical test work presented in Section 13. The copper and gold recovery process was designed on the basis of 4,860 t/d with average head grades of 0.98% Cu and 0.435 g/t Au.

A jaw crushing plant will operate at a nominal crushing rate of 312 t/h, 16 hours a day for 365 days per year. The process plant will operate 24 hours per day for 365 days per year with a plant availability of 92% and a processing rate of 220 t/h. The copper will be leached with sulphuric acid, recovered in a solvent extraction / electrowinning circuit (SX-EW) and shipped as cathode copper. The copper circuit tailings is leached in CIL tanks and the carbon is processed in a 2 t/d carbon ADR plant for gold extraction and the production of gold doré. This process will achieve an estimated recovery of 85.2% Cu and 84.4% Au.

The mineralized material processing facilities will include the following unit operations:

- Crushing and mineralized material Handling:
  - Primary Crushing: A vibrating grizzly screen and jaw crusher in open circuit, producing a final product P_{80} of approximately 114 mm; and
  - Fine Mineralized material Stockpile: 5,000 t fine mineralized material stockpile and reclaim feeders;
- Process plant:
  - Primary Grinding: a SAG mill operating in closed circuit with a cyclone cluster, producing a final product P_{80} of approximately 664 µm;
  - Copper Leaching and Recovery: a pre-leach thickener, six copper leach tanks, four counter current decantation (CCD) thickeners and an SX-EW circuit; and
  - Gold Leaching: 6 – CIL tanks, an ADR plant and cyanide destruction.
- Tailings management: tailings filtration, load-out and dry stack at the tailings management facility.

A process flowsheet and process plant layout are presented in Figure 17.1 and Figure 17.2 respectively.



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SCALE: None	DATE:
DESIGNED: J.D.S.	Sept 22 16
DRAWN: B. Wong	Sept 22 16
CHECKED:	
APPROVED:	

JDS Energy & Mining Inc.

TITLE
COPPER NORTH MINING CORP.
THE CARMACKS COPPER PROJECT
SIMPLIFIED PROCESS FLOWSHEET

PROJECT No.	DRAWING No.	REV No.
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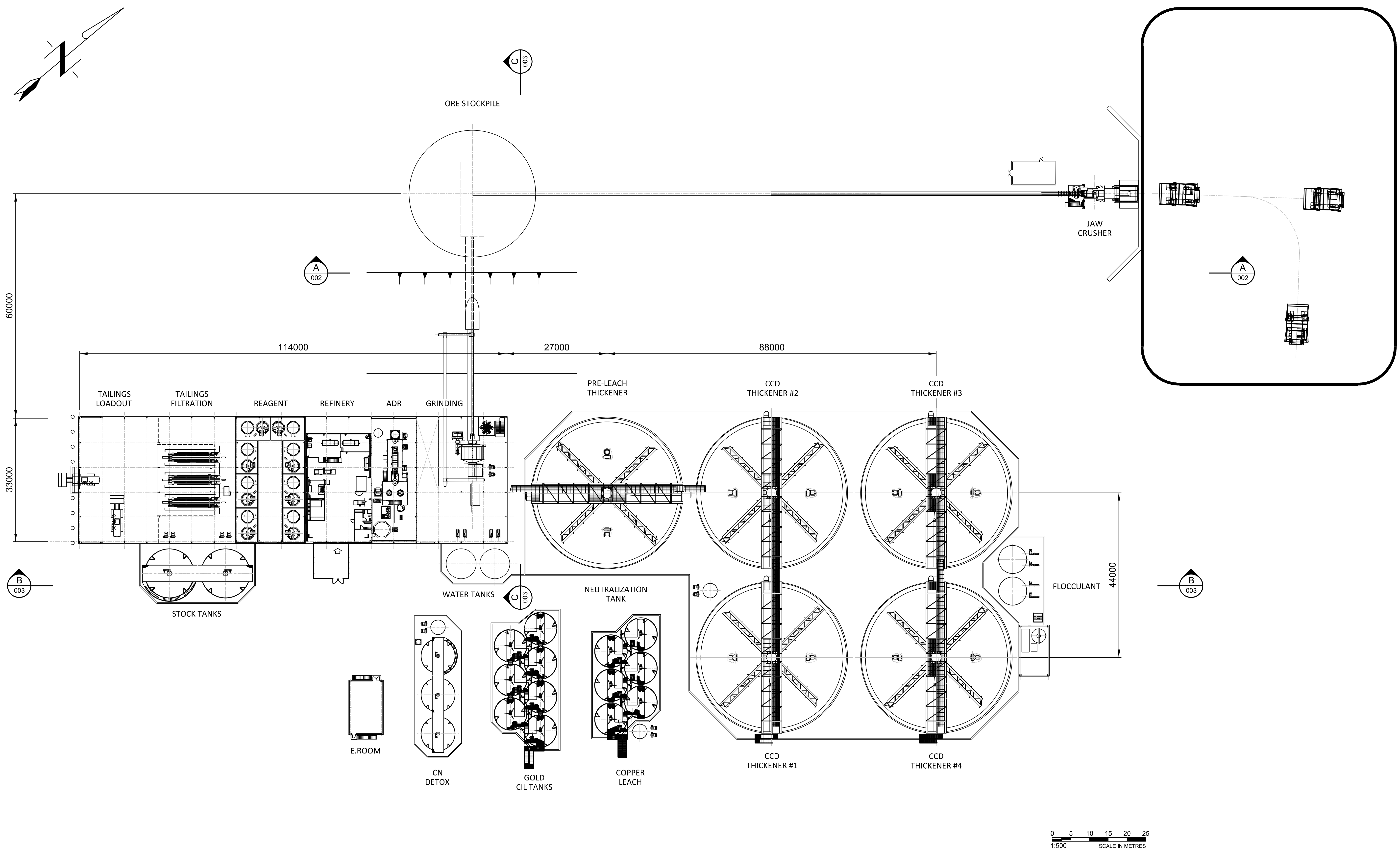

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DESIGNED: J.D.S.	19 MAY 2016
DRAWN: S PERKINS	19 MAY 2016
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APPROVED:	

JDS Energy & Mining Inc.

TITLE
COPPER NORTH PROJECT
PLANTSITE
PLAN

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17.2 Process Plant Design Criteria

The process design criteria and mass balance detail the annual mineralized material production, major flows, and plant availability. The key process design criteria are summarized in Table 17.1.

Table 17.1: Process Design Criteria

Criteria	Description	Units	Design	Source
Plant Throughput		t/d	4,860	Mine Plan
		Mt/a	1,825	Mine Plan
Crusher Availability		%	65	Client
Crusher Throughput		t/h	312	Engineering Calculation
Crusher Selection		Size, in	36x48	Vendor Recommended
		Number	1	
Mill Availability		%	92	Client
Mill Throughput		t/h	220	Engineering Calculation
Physical Characteristics	Cwi	kWh/t	-	Not Available
	Rwi	kWh/t	9.1	BV Minerals – Phase 2 Report
	Bwi	kWh/t	15.2	BV Minerals – Phase 2 Report
Primary Grind Size	P_{80}	µm	664	Client
Average Head Grade		% Cu	0.98	Client
		g/t Au	0.435	Client
Circuit Recovery	Copper	%	85.2	Dreisinger Consulting – BV Minerals Feb. 23, 2016
	Gold	%	84.4	Dreisinger Consulting – BV Minerals Feb. 23, 2016
	Silver	%	9.4	Dreisinger Consulting – BV Minerals Feb. 23, 2016
Pre-leach Thickener Settling Rate		$t/h/m^2$	0.25	BV Minerals – Phase 2 Report
Thickener Underflow Density		% solids	70	Client
Copper Leach Tank Retention Time		h	6	BV Minerals – Phase 2 Report
Copper Leach Density		% solids	50	BV Minerals – Phase 2 Report
Copper Leach Temperature		Deg. C	80	Dreisinger Consulting – BV Minerals Feb. 23, 2016
SX –EW Plant Feed Rate		m^3/h	530	Mass Balance Calculation
Gold/Silver CIL Leach Tank Retention Time		h	12	BV Minerals – Phase 2 Report
Gold/Silver CIL Leach Density		% solids	40	BV Minerals – Phase 2 Report
ADR Capacity		t Carbon	2	Engineering Calculation
Carbon Loading		g Au / t Carbon	3,000	No test work available, Vendor Recommended
Cyanide Destruction		h	3	BV Minerals – Phase 2 Report

Source: Multiple Sources

17.3 Process Plant Description

17.3.1 Crushing

The crushing circuit consists of a stationary grizzly, rock breaker, truck dump pocket, vibrating feeder, jaw crusher, and belt feeder. A vibrating grizzly feeder will draw material out of the dump pocket and constantly feed the jaw crusher. Crushed product, at a P_{80} of 114 mm, will discharge onto a belt conveyor and be transferred to a 5,000 t stockpile.

Two belt feeders will reclaim feed from the crushed material stockpile and discharge it onto the SAG mill feed conveyor. Each feeder will be capable of delivering full tonnage to the mill. A weightometer on the SAG mill feed conveyor will control the speed of the feeders to provide a constant feed rate of 226 t/h to the SAG mill.

17.3.2 Grinding

Reclaimed material will feed a 6.1 m dia x 3.4 m long SAG mill driven by a 1,380 Kw variable speed induction motor. This will allow the SAG mill to vary the power draw and optimize circuit parameters to accommodate changing feed conditions. Grinding media will be added to the SAG mill via the SAG mill feed conveyor. The SAG will operate in closed circuit with flat bottom cyclones. The cyclone underflow will feed the SAG mill feed chute and the overflow, at a target P_{80} grind size of 664 µm, will flow by gravity to the pre-leach copper dewatering thickener.

17.3.3 Copper Leaving and Recovery

17.3.3.1 Copper Leach and Counter Current Decantation Circuits

Cyclone overflow will feed the pre-leach thickener. The thickener overflow will be collected in the process water tanks and circulated to the grinding circuit as make-up water. The thickener underflow, at a higher solids density, will feed the first of six 8.5 m dia. X 9.0 m copper leach tanks, providing 6 hours of total leach time. Raffinate from the SX-EW circuits, at 80°C, will be used to dilute the copper leach slurry to 50% solids.

Sulphuric acid will leach copper into solution before being recovered in four CCD thickeners operated in series. The slurry, thickener underflow, will flow from one thickener to the next, while thickener overflow will flow counter current to the slurry. The process allows the solids to be adequately washed, recovering any entrained pregnant leach solution (PLS). The copper bearing PLS, or the first thickener overflow, will be sent to the SX-EW circuit to plate the copper in solution onto cathodes.

17.3.3.2 Solvent Extraction – Electrowinning Circuit

The SX-EW circuits were sized by the vendor based on the solution flow rate from the CCD circuit and the anticipated copper recovery. The PLS will be pumped through two solvent extraction stages where copper is transferred from the aqueous phase to the organic phase. The barren aqueous solution, or raffinate, will be recycled to the leach circuit.

The organic solution will be washed in the loaded organic tank and then flow to the stripping stage where the copper is transferred from the organic phase to the electrolyte. The rich electrolyte is pumped into the electrowinning cells where a current is used to deposit Cu^{2+} onto stainless steel cathodes. The loaded cathodes are stripped periodically to remove the copper, and then washed and replaced in the electrowinning cells. The cathodes will be sold directly into the market.

17.3.4 Gold Leaching and Recovery

17.3.4.1 Gold Leaching

The last copper CCD thickener underflow will be pumped to a neutralization tank where lime will be used to raise the Ph of the leached slurry from acidic to basic conditions. The CIL and an ADR plant will be used to recover gold and silver from the neutralized slurry.

The leach circuit will consist of six 11 m dia. X 12 m high tanks operating in series and will provide a 12 hour retention time. Sodium cyanide will be used to leach the gold and silver from the rock into solution, while activated carbon will simultaneously adsorb the precious metals into its pores. Carbon is pumped countercurrent to slurry flow, with fresh carbon being added in the sixth tank. Once a day, the loaded carbon is pumped from the first tank to the ADR plant and the gold silver is recovered as doré bars. Lime will be used for Ph control to prevent HCN gas from forming.

In projects with high silver to gold grade ratios, such as Carmacks, Merrill Crowe is the recommended process option to handle the extra silver produced; however, this requires multiple CCD wash thickeners to ensure a clean PLS solution for zinc cementation. Compared with CIL and ADR, a CCD wash circuit is more expensive and requires a larger plant footprint. Since silver recoveries are estimated to be much lower than gold recoveries, the PLS solution will have a silver to gold ratio amenable to CIL and ADR. Test work is recommend in the next stage of engineering to establish carbon loading and CIL circuit performance.

17.3.4.2 Carbon Stripping (Elution)

The carbon stripping (elution) process uses a cyanide / caustic soda mixture to strip precious metals from the carbon and create a pregnant solution suitable for electrowinning.

The strip column will be a carbon steel tank with the capacity to handle 2 t of carbon. During the strip cycle, barren solution containing approximately 1% sodium hydroxide and 0.2% sodium cyanide, at a temperature of 140°C (284°F) and 450 kPa (65 psi), will be circulated through the strip vessel. Solution exiting the top of the elution vessel will be cooled below its boiling point by the heat recovery heat exchanger. Heat from the outgoing solution will be transferred to the incoming cold solution for temperature increase, prior to the cold solution passing through the solution heater. A diesel powered boiler will be used as the heat source. Pregnant strip solution from the elution circuit will be transported to the electrowinning circuit to recover the gold and silver on stainless steel cathodes.

17.3.4.3 Carbon Regeneration

A recessed impeller pump will transfer the stripped carbon from the elution vessel to the kiln feed dewatering screen. The kiln feed screen doubles as a dewatering screen and a carbon sizing screen, where fine carbon particles will be removed. Oversize carbon from the screen will discharge by gravity to the carbon regeneration kiln feed hopper.

Screen undersize, containing carbon fines and water, will drain by gravity into the carbon fines tank. Subsequently, the carbon fines will be filtered and collected into bags for disposal. A diesel fired horizontal kiln with a residual heat dryer will be utilized to treat 2 t of carbon per day, equivalent to 100% carbon regeneration. The regeneration kiln discharge will be transferred to the carbon quench tank by gravity, and cooled by fresh water and/or carbon fines water prior to being pumped back to the CIL circuit.

To compensate for carbon losses by attrition, fresh carbon will be added to the circuit via the carbon attrition tank and mixed with water to activate the carbon pores. The fresh carbon will then drain into the quench tank.

17.3.4.4 Gold Electrowinning and Refining

Pregnant solution from the strip vessel will be pumped to the two-stage, single pass electrowinning circuit. Precious metals will be recovered onto the stainless steel cathodes, while the barren strip solution is pumped back to the barren solution tank and recycled in the elution circuit. To prevent impurities building up in solution, a bleed is periodically sent to the CIL circuit.

Gold rich sludge will be washed off the steel cathodes using high pressure water and collected in a holding tank. Once or twice per week, the sludge will be drained, filtered, dried and combined with fluxes. The resulting mixture will be smelted in a diesel powered, direct-fire furnace to create gold doré. This process will take place within a secure and supervised area and the gold doré will be stored in a vault before shipping off-site.

17.3.5 Cyanide Destruction

The cyanide destruction circuit will consist of three mechanically agitated tanks, two operating and one standby, with a total capacity of 600 m^3, equivalent to 3-hours retention time. Cyanide will be detoxified using the SO_2/Air process. Treated slurry from the cyanide destruction circuit will feed the final tailings stock tanks.

Process air will be sparged into the bottom of the tanks, while sodium metabisulphite will be dosed into the system as a solution to generate SO_2. Lime slurry will also be added to maintain the optimum Ph of 8.0 – 8.5. Copper sulphate will be added as a catalyst if required. This system has been designed, based on test work results reported in BV Minerals – Phase 2 Report, to reduce the total cyanide concentration (TCN) from 311 mg/L CN_{WAD} to less than 1 mg/L, prior to transfer to the final tailings stock tank. Additional test work is recommended in the next phase of engineering to confirm the design criteria for this flowsheet.

17.3.6 Tailings

The slurry from the cyanide destruction circuit will be pumped to one of two 8-hour capacity final tailings stock tanks. The stock tanks will feed one of three pressure filters to reduce the moisture content to approximately 17%. The dry solids will be loaded into trucks and hauled to the dry stack tailings facility. Filtrate from the filters will be recovered and used as make-up water in the process facility.

17.4 Reagents

Reagents consumed within the circuits will be prepared and distributed by the reagent handling systems. All reagent areas will be bermed with sump pumps to collect spillages. The reagents will be mixed, stored and then delivered through a supply loop, with dosage controlled by flow metres and manual control valves. The storage tanks have been sized for a minimum storage capacity of one day. The reagents will generally be delivered in powder form or as liquids in 1-tonne totes.

Lime will be delivered by bulk road tanker and stored in a 200 t capacity silo.

17.5 Sulphuric Acid Plant

An acid plant is planned for on-site production of sulphuric acid for use in the copper leaching circuit. The acid plant cost was carried over from the last study. In the next stage of engineering, plant design and equipment sizing should be confirmed. When the acid plant is offline for maintenance sulphuric acid totes will be stored on-site as a backup supply.

17.6 Air Supply

An instrument and plant air system, with compressors, dryers, filters, and receivers, will be included in the plant facility design. The compressors will be located in a compressor room inside the plant building and receivers will be strategically located throughout the plant as required.

17.7 Water Management

The following types of water will be used in the process plant:

- **Process Water**: Process water will consist of overflow water from the pre-leach thickener and tailings filtrate. This water will predominantly be used in the grinding circuit to dilute slurry to the target solids density; and
- **Fresh Water**: Process plant fresh water will be pumped from wells and will be used as reagent make-up water and gland water.

18 Project Infrastructure and Services

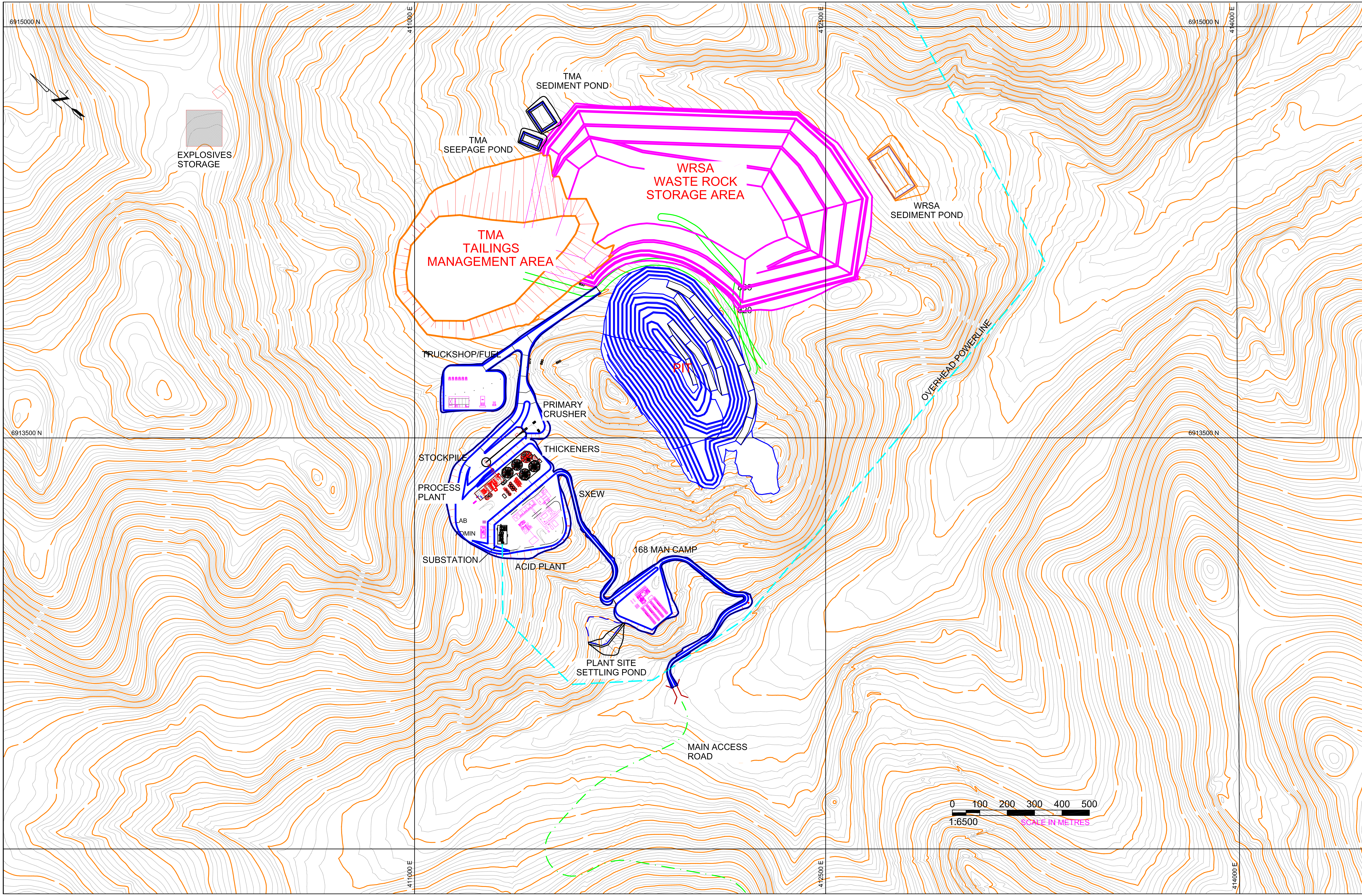
The project envisions the upgrading or construction of the following key infrastructure items:

- Approximately 12 km of all-seasonal access road from Freegold Road to the project site;
- Dry stack tailings management area (TMA);
- Waste rock storage area (WRSA);
- Crushing and grinding circuits;
- Gold extraction plant;
- Solvent Extraction / Electrowining plant;
- Electrical connection to Yukon Energy, transmission line and on-site substation and distribution network;
- Process and fire water storage and distribution;
- Sewage collection system;
- Truck shop and warehouse building;
- Administration building; and
- Mine dry and camp facility.

18.1 General Site Arrangement

An overall site plan for the plant site area is shown in Figure 18.1 with a detailed plant site layout shown in Figure 18.2.

The plant facilities have been located to the west of the open pit where the terrain is less steep and minimal earthworks will be required to provide suitable footprint for the surface structures and equipment. The dry stack tailings management area would be located to the north of the plant site to allow for short haulage distances. The mine WRSA will cover nearly 70 ha northeast of the open pit.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TAILINGS
MANAGEMENT AREA
TRUCKSHOP/FUEL
PIT
PRIMARY
CRUSHER
OVERHEAD POWERLINE
STOCKPILE
THICKENERS
PROCESS
PLANT
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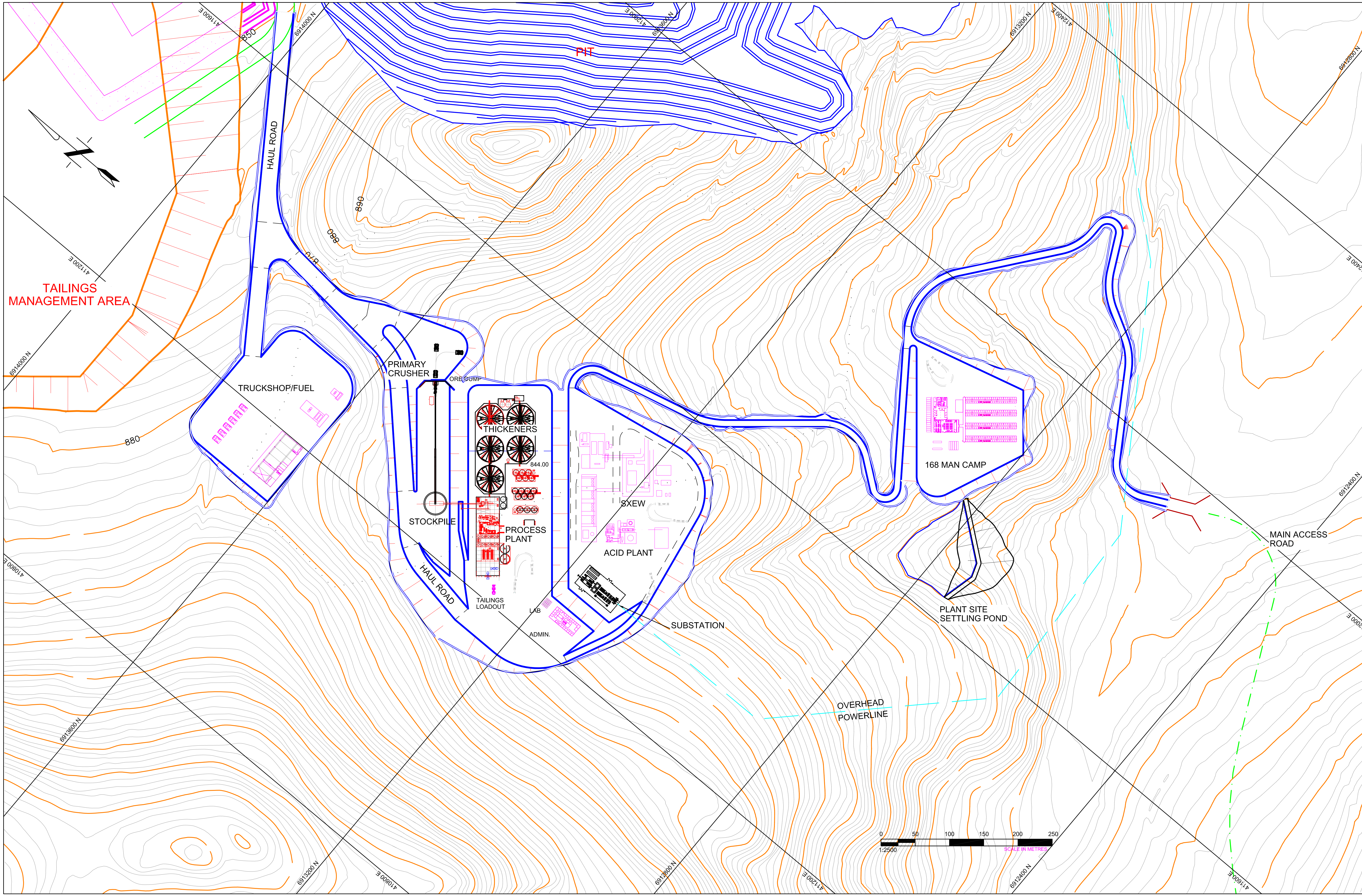
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TRUCKSHOP/FUEL
PRIMARY
CRUSHER
ORE DUMP
THICKENERS
844.00
STOCKPILE
PROCESS
PLANT
SXEW
ACID PLANT
HAUL ROAD
TAILINGS
LOADOUT
LAB
ADMIN.
SUBSTATION
168 MAN CAMP
PLANT SITE
SETTLING POND
MAIN ACCESS
ROAD
OVERHEAD
POWERLINE
850
890
880
870
880
6914000 N
411600 E
6913600 N
6913200 N
6912800 N
6912400 N
411200 E
410800 E
412000 E
412400 E
0
50
100
150
200
250
1:2500
SCALE IN METRES

18.2 Site Access

Approximately 12 km of all-season access road is required to connect the Freegold Road to the plant site area. The road is planned to be a radio assist, single-lane, gravel road with inter-visible turnouts. The road would be approximately 5 m wide and include a ditch at one side with a maximum designed grade of the road of 10%.

18.3 Buildings and Structures

18.3.1 Process Buildings

The process plant, SX/EW and acid plant are planned to be located in an insulated fabric buildings with the areas as shown in Table 18.1.

Table 18.1: Process Building Areas

Process Descriptions	Area (m^2)
SX/EW/TF Plant Building	6,250
Acid Plant Building	2,400
Process Plant Building	4,000

Source: JDS (2016)

18.3.2 Truck Shop & Warehouse Building

The surface maintenance shop will be a single-bay, 9 m x 30 m insulated fabric covered structure suitable for preventative maintenance services, basic repairs and component replacement. More extensive repair work would be conducted off-site.

The warehouse will be located within close proximity to the surface maintenance shop. The warehouse will be a single-bay, 12 m x 24 m uninsulated fabric covered structure.

Overhead cranes will be provided for equipment maintenance. The building will be heated to 5°C by electric unit heaters.

18.3.3 Office Complex

The 385 m^2 office complex will be constructed from modular units manufactured off-site and in compliance with highway transportation size restrictions. Modules will be supported on wood cribbing. The complex will comply with all building and fire code requirements and be provided with sprinklers throughout. The site office facility will contain the following items:

- Private offices;
- Main boardroom; and
- Mine operations line-up area.

18.3.4 Mine Dry Complex

The 385 m^2 mine dry will be constructed from modular units manufactured off-site and in compliance with highway transportation size restrictions. Modules will be supported on wood cribbing. The complex will comply with all building and fire code requirements and be provided with sprinklers throughout. Arctic corridors will connect the mine dry with the camp core facilities and rooms.

The mine dry facility will service construction and operations staff during the life of the project. It will be capable of servicing 150 workers during shift change and contain the following:

- Male and female clean and dirty lockers; and
- Showers and washroom facilities with separate male and female sections.

18.3.5 Camp

The camp will comprise single-occupancy rooms with central washrooms. It will be used during the construction stage and throughout the operations stage. There will be four dormitory wings, each capable of housing 42 people for a total of 168 beds.

The kitchen / dining / recreation complex will include the following:

- Kitchen complete with cooking, preparation and baking areas, dry food storage and walk-in freezer/cooler. The kitchen will be provided with appropriate specialized fire detection and suppression systems;
- Dining room with serving and lunch preparation areas;
- First aid room;
- Mudroom complete with coat and boot racks, benches and male-female washrooms;
- Housekeeping facilities;
- Reception desk and lobby; and
- Recreation area.

The camp will be constructed from modular units manufactured off-site in compliance with highway transportation size restrictions. Camp modules will rest on wood cribbing. The camp will comply with all building and fire code requirements and be provided with sprinklers throughout. Arctic corridors will connect the main camp complex and dormitory wings.

18.3.6 Fuel Storage

Diesel will be trucked to the project site on an as needed basis and stored in a 150,000 L Enviro-Tank including an integrated dispensing system. Surface mobile equipment will fuel-up at the storage tank and fixed equipment will be supplied by the fuel & lube truck.

18.4 Power

Electrical power for the project will be provided by Yukon Energy Corp. (YEC). A new 11 km long, 34.5 Kv overhead power line will connect to a new 138/34.5 Kv substation at the tap-off point at McGregor Creek on the existing Carmacks Stewart 138 Kv grid to the mine site substation, supplying 4.16Kv to the mine facilities.

The total project electrical load is estimated to be approximately 10 MW with the total average annual power consumption estimated to be approximately 81,200,000 kWh/a.

18.5 Tailings and Waste Rock Management

18.5.1 Dry Stack Tailings Storage Area

CNMC is planning to employ the best available technology (BAT) principles for tailings management at the Carmacks Project, as recommended by the Independent Expert Engineering Investigation and Review Panel that prepared the report on the Mount Polley Tailings Storage Facility Breach (Province of British Columbia, 2015).

The primary objective of the BAT principles is to assure the physical stability of the tailings deposits both during operation and closure. To achieve this, BAT comprises three components:

- Eliminate surface water from the impoundment;
- Promote unsaturated conditions in the tailings with drainage provisions; and
- Achieve dilatant conditions throughout the tailings deposit by compaction.

BAT also incorporates the adoption of technology to provide chemical stability of the tailings in closure.

The Mount Polley Tailings Storage Facility Breach Report (Province of British Columbia, 2015) specifically identifies filtered tailings as a technology that embodies BAT. It is in this context that regulatory authorities within Canada are increasingly encouraging mining companies to adopt filtered tailings and ‘dry stack’ technology.

18.5.1.1 Design Concept

The dry stack TMA has been designed by Golder (2016) to:

- Satisfy the regulatory requirements specified by the YG; and,
- Comply with the following guidelines:
    - Canadian Dam Association: *Dam Safety Guidelines* (2007, and Revised in 2013);
    - Canadian Dam Association: Technical Bulletin – Application of Dam Safety Guidelines to Mining Dams (2014);
    - ‘British Columbia Mined Rock and Overburden Investigation and Design Manual, Interim Guidelines, May 1991’;

- ‘Draft Acid Rock Drainage Technical Guide’ (British Columbia Acid Mine Drainage Task Force 1989);
- ‘Draft Guidelines and Recommended Methods for Predictions of Metal Leaching and Acid Rock Drainage at Minesites in British Columbia’ (Price, 1997);
- Global Acid Rock Drainage Guide (INAP 2009); and
- Prediction Manual for Drainage Chemistry from Sulphidic Geologic Materials (Mend 2009).

The TMA will be located on a side-hill immediately adjacent to and west of the proposed WRSA (Figure 18.1). The site slopes from south-west to north-east between an elevation of 818 m and 877 m.

Development of the TMA will be staged over the life of mine. In general, the TMA construction will proceed upslope (toward the south-west), with initial placement in the topographical low, near the north-east corner. Sequencing of the filtered tailings and waste rock placement has not been modelled, so the general plan of TMA development may change during more detailed design phases.

The TMA starter facilities will be constructed prior to the placement of filtered tailings in Year 1. The starter facilities will include the permanent and temporary surface water diversion drains, a sediment pond for non-contact water, a seepage collection pond for contact water, and a lined and bounded area for placement of the tailings. The liner is expected to comprise a geosynthetic clay liner (GCL) that will be keyed into a starter berm, constructed around the perimeter of the TMA for containment. This area will be extended as required over the life of the facility. Underliner (foundation) drains will be installed beneath the liner and a toe-drain will collect seepage from on top of the liner, as described in Section 18.5.1.2.

With the exception of the toe-drain, the current design does not include allowance for blanket or finger drains on top of the GCL liner. This is based on the relatively high permeability of the tailings, sloping grade of the site, and the assumption that the tailings will not leach acid or metals at levels of concern. It also assumes that subsequent lifts placed during winter months will remain frozen, limiting seepage in future years. Modelling should be carried out as part of future studies to confirm whether finger or blanket drains are required on top of the liner. Performance should be considered in terms of seepage management and excess porewater pressure generation.

The TMA will occupy an initial lined area of approximately 13.4 ha. This area allows for an average rate of rise in the first year of about 7 m/year. As the area of the TMA is extended, the landform will grow to occupy an ultimate footprint area of approximately 36.2 ha. Allowing for 4H:1V side slopes, this results in an average rate of rise for the tailings of approximately 4 to 5 m/year. Waste rock and filtered tailings will be transported to the site by truck and zoned within the landform. The waste rock shell will be constructed ahead of the filtered tailings to provide stability, reduce dust and the potential for erosion of the tailings. Waste rock will be placed in layers, spread using a dozer and compacted by haul traffic. The intent is to reduce segregation and provide a dense matrix for stability. The filtered tailings will be placed behind the waste rock shell, spread by dozer, and compacted using a vibratory smooth drum roller.

The facility will be constructed with 4H:1V side slopes. The slope angle provides for long-term stability of the landform and will allow progressive reclamation of the slopes to commence during operations.

18.5.1.2 Water Management

18.5.1.2.1 Non-Contact Water

Two ephemeral streams flow through the TMA site. Water from these streams, together with runoff from the upstream catchment, will be diverted to the north, around the western perimeter of the TMA. The West Diversion Channel will drain to a sediment pond, prior to discharging to North Williams Creek. Consistent with the diversion channel geometry for the current WRSA design (Golder 2008b), it is assumed that the West Diversion Channel will be designed to accommodate the 1 in 10 year 24-hour design storm and 1 in 10 year freshet with a minimum freeboard of 300 mm. It is also assumed that the channel will be lined with geotextile filter fabric and rip-rap.

Temporary diversion channels will be constructed as the TMA footprint is extended and will be used to divert minor surface water runoff into the West Diversion Channel. We have assumed that the temporary diversion channels will be unlined and designed to accommodate the 1 in 5-year 24-hour design event.

An underliner drainage system will be installed in the foundation of the TMA to drain groundwater and relieve excess porewater pressure from thawing permafrost. The drain design will be similar to that proposed for the current WRSA (Golder 2008b). The drains excavated into the natural drainage lines of the foundation and backfilled with a free-draining granular material and a 150 mm diameter perforated pipe surrounded by non-woven geotextile.

It is assumed that water from the underliner drainage system will be classified as non-contact water. On this basis, the drains will be designed to drain under gravity into the West Diversion Channel for discharge into North Williams Creek.

The design includes allowance for construction of an unlined sediment pond, prior to discharge of water from the West Diversion Channel. The sediment pond has been included in consideration of the potential for sediment loads in runoff from the cleared areas, unlined temporary diversion drains, waste rock slopes and progressive reclamation activities. It is possible that the need for this pond could be negated by implementing other sediment control measures in the design and the possible duplication of diversion drains. This would allow the direct discharge of water from the West Diversion Drain into North Williams Creek and should be considered as part of future design iterations. For the purposes of the conceptual design, the TMA sediment pond has been sized to hold the 1 in 10-year 24-hour design storm and 1 in 10-year freshet with a 50% dead storage volume, assuming a runoff coefficient of 80%. Flows from the underliner drainage and runoff from the TMA landform have not been accounted for in the preliminary sizing the TMA sediment pond. The pond will incorporate a spillway and rip-rap lined plunge pool, designed to pass larger storm events.

18.5.1.2.2 Contact Water

Contact water includes runoff and seepage that has come into contact with the filtered tailings. Contact water will be isolated from non-contact water. The seepage from the filtered tailings will be collected and directed to the TMA seepage pond via the seepage collection system. Water from the TMA seepage pond will be recycled to the plant for use as process make-up water.

Liner System

The entire TMA will be underlain by a single layer GCL liner that will be keyed into the starter berm. GCL is a geosynthetic product composed of two layers of geotextile with an internal bentonite core. The hydraulic conductivity is typically on the order of 10^{-10} to 10^{-12} m/s. GCL has some capacity to self-heal damage to the GCL as the hydrated bentonite will naturally deform with stress application.

Bentonite has a low internal angle of friction (shear strength). It is therefore anticipated that a reinforced GCL will be required over the entire area of the TMA. Further design studies and testing will be required to specify the GCL required for the proposed application.

The foundation for placement of the GCL will be cleared, grubbed and stripped of topsoil prior to compaction with a smooth drum roller. We have assumed that the natural soils will provide a suitable subgrade for placement of the liner, accepting that it may be necessary to remove soft or wet soils and ice from some areas in preparation for liner placement.

Seepage Collection System

The phreatic surface in the TMA will be managed by the internal seepage collection system. The seepage collection system will initially comprise a toe-drain, constructed along the north and east perimeter of the TMA. The toe-drain will drain by gravity to the TMA seepage pond. The toe-drain will include a perforated polyethylene pipe, wrapped in non-woven geotextile and installed in a granular filter zone.

The phreatic surface within the tailings will be monitored as the height of the landform increases. If necessary, finger or blanket drains may be constructed within the filtered tailings stack to manage the phreatic surface within the tailings. Finger or blanket drains may be constructed from select waste rock and would be connected into the seepage collection starter facilities. The installation of finger or blanket drains is considered a contingency measure and is not included in the conceptual cost estimate for the TMA.

Based on publicly available information for the Keno Hill mine, drainage from the filtered tailings in the subarctic climate may be minimal (Alexco Keno Hill Mining Corp 2015). Proper compaction and maintenance of the TMA including grading to prevent ponded water on the TMA surface will reduce water infiltration into the filtered tailings. Further, the development of frozen layers within the tailings stack is likely to reduce vertical seepage from the facility.

TMA Seepage Pond

The TMA seepage pond is sized to contain runoff from a 1 in 100 year 24-hour storm event on the starter facility area assuming a runoff coefficient of 100%. This is considered a ‘worst-case’ scenario for runoff from a storm event during the first year of filtered tailings placement. In later stages of the TMA’s development, incident rainfall and seepage will be attenuated by infiltration into the tailings.

The pond will be lined with a single layer HDPE liner to reduce seepage of contact water into groundwater. It will also incorporate a lined spillway that will allow it to safely pass runoff exceeding the design storage event.

The TMA seepage pond has been sized assuming a maximum operating water depth of 4 m and 2.5H:1V slopes. Further design studies and geotechnical investigations will be required to confirm the pond geometry.

18.5.2 Waste Rock Storage Area

The WRSA has been designed by Golder (2008b) based on the guidelines set out in the BC Ministry of Energy, Mines and Petroleum Resources document for the “Investigation and Design of Mine Dumps, Interim Guidelines, May 1991”. Refer to Golder (2008b) for the complete preliminary design of the WRSA, which is summarized below.

The WRSA design is based on a projected capacity of 70 Mt of non-acid-generating waste rock. Testing to date indicates the rock is not acid generating or metal leaching. The waste rock is a durable granodiorite or biotite gneiss and would be placed from the east limit of the WRSA progressing west in lifts up to 25 m thick.

The WRSA has been sited to the north of the open pit in an area that has a thick overburden layer and is understood to be beyond the area to be mined with the open pit operation (Figure 18.1). The north limit of the storage area was determined by the local drainage and the storage area is to stay south of the first major creek north of the mine area.

The WRSA will be cleared before the mine starts operation to remove the upper organic layer and the ash. The material will be stockpiled to be reused later for areas where vegetative covers are required at closure. The perimeter surface water ditches will be developed at this time along with the WRSA sediment pond with a capacity of 53,000 m^3. The eastern half of the footprint will be cleared to allow the permafrost to thaw. The thawing of the permafrost is important, as the interim stability of the slopes of the WRSA control the slope stability. If the permafrost remains in the ground, the interim slopes will have to be flattened or a wide “runout” zone developed around the perimeter of the site to “catch” small slope slumps or failures that will occur. As the WRSA expands and the upper lifts of the facility are developed, the permafrost will disappear under the WRSA and the stability of the interim slopes will be defined by the strength of the waste rock.

The WRSA will be built to elevation 800 m over the eastern half of the WRSA in two lifts. As the second lift nears completion, the western half of the footprint will also be developed. The first lift above elevation 800 m will be to 820 m and then in equal lifts to the anticipated maximum elevation of 880 m. The ramp starts from the southeast corner and will continue up the south slope to approximately elevation 800 m. The ramp will then move to a point near the northeast corner of the open pit or some 400 m west (ramp to start at ground elevation 795 m near pit slope). The ramp will then “climb” on the south slope of the WRSA to the top elevation of the WRSA at 880 m. This will result in a main haul ramp with a grade of ~10%.

18.6 Freight

Freight will be delivered to site on the access road and offloaded at the warehouse or other designated area.

18.7 Explosive Storage

Explosives will be stored at a secured and monitored site located approximately 1.6 km from process facilities. Bulk explosives agents will be stored in silos while boosters and detonators would be stored in locked and barricaded sea-containers and separated according to Natural Resource Canada guidelines.

18.8 Information Technology & Communications

The administration offices, mill complex, maintenance facility and the camp will include a wireless computer network and satellite phone system. Hand-held radios will be used to provide voice-communication between personnel on surface.

19 Market Studies and Contracts

19.1 Market Studies

Detailed market studies on the potential sale of copper cathode and doré from the Carmacks Project were not completed. The terms applied to the economic analysis were reviewed and considered to be acceptable for the purposes of this PEA by QP Gord Doerksen, P.Eng.

No contractual arrangements for shipping, port usage, or refining exist at this time. Table 19.1 outlines the terms used in the economic analysis.

Table 19.1: NSR Assumptions Used in the Economic Analysis

Assumptions	Unit	Value
Payable Copper Cathode	%	100
Payable Gold	%	100
Payable Silver	%	100
Copper Cathode Shipping Charge	US$/lb	0.015
Au Refining Charge	US$/payable oz	4.00
Ag Refining Charge	US$/payable oz	0.40

Source: JDS (2016)

19.2 Contracts and Royalties

The Carmacks property is subject to an acquisition royalty payment of C$2.5M to Archer, Cathro & Associates (1981) Limited. The terms of this royalty are outlined below:

- The project is subject to advance royalty payments of C$100,000 per year prior up until commercial production is reached, in any year in which the average daily copper price reported by the London Metal Exchange is US$1.10 or more per pound. To date, $1.3M in advance royalty has been paid. As a result, the maximum amount of royalty that remains payable as of the date of this report is $1.2M.
- Once commercial production is achieved, the property is either subject to a 3% NSR Royalty or 15% NPI (Net Profits Interest), at the election of CNMC. If CNMC elects to pay the net smelter royalty, it has the right to purchase the royalty for $2.5 M, less any advance royalty payments made to that date.

This financial commitment is included in the cash flow. Total third party royalties for the project amount to $1.2M over the LOM.

19.3 Metal Prices

The precious metal markets are highly liquid and benefit from financial markets around the world (London, New York, Tokyo, and Hong Kong). Historical copper, gold and silver prices are shown in Figure 19.1, Figure 19.2 and Figure 19.3. Historical average US$:C$ exchange rates are shown in Figure 19.4.

Figure 19.1: Historical Copper Price

Source: London Metal Exchange (2016)

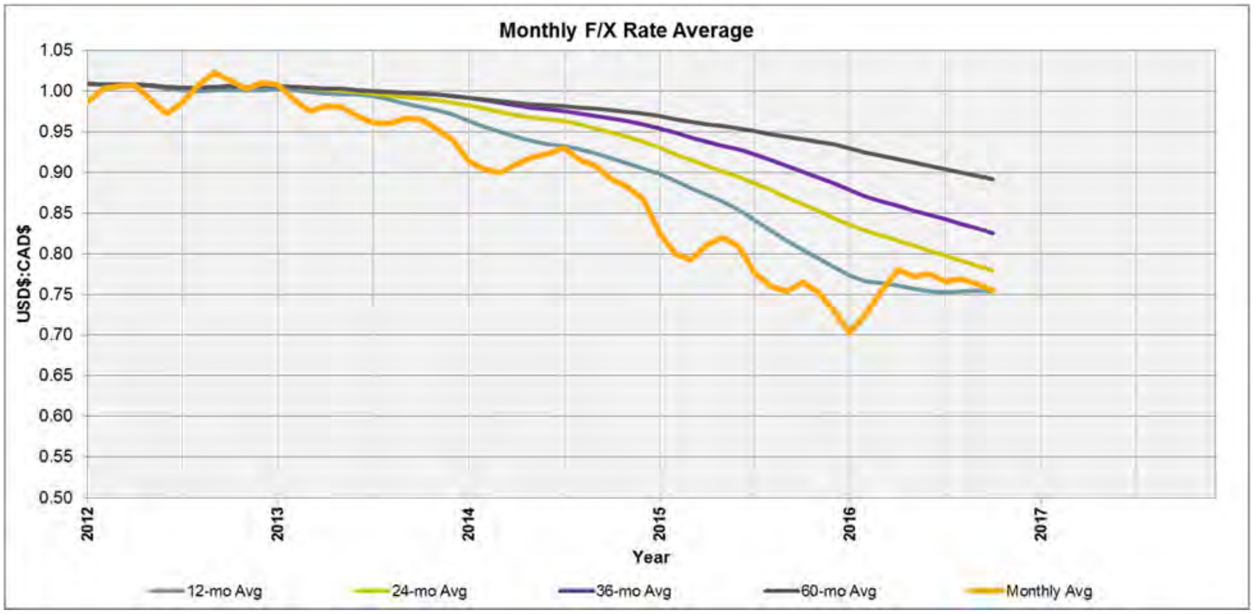
Figure 19.2: Historical Gold Price

Source: Kitco (2016)

Figure 19.3: Historical Silver Price

Source: Kitco (2016)

Figure 19.4: Monthly Average US$:C$ Foreign Exchange



Source: Bank of Canada (2016)

The gold and silver prices used in the economic analysis are based on the 6-month average spot rate as at September 2016. For copper, the price selected was based on recently released comparable Technical Reports. The US$:C$ exchange rate used in the economic analysis is between the 18-month and 24-month trailing average as at September 2016. A sensitivity analysis was completed as part of the overall economic analysis. The results of this are discussed in Section 23. Table 19.2 outlines the metal price and exchange rate used in the economic analysis.

It must be noted that metal prices and exchange rates are highly variable and are driven by complex market forces and are extremely difficult to predict.

Table 19.2: Metal Price and Exchange Rate

Assumptions	Unit	Value
Cu Price	US$/lb	2.50
Au Price	US$/oz	1,300
Ag Price	US$/oz	17.50
Exchange Rate	US$:C$	0.78

Source: JDS (2016)

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Environmental Setting

The Carmacks Project area lies within the Klondike Plateau and is part of the Pelly River Ecoregion (Oswald and Senyk 1977), which is comprised of portions of the Stewart, Macmillan, Lewes, and Klondike Plateaus and Tintina Valley physiographic subdivisions (Bostock, 1970). Surface drainage flows both north and east from the study area. A number of valley streams, of which Williams Creek is the largest, drain northeastward to the Yukon River.

Environmental baseline conditions on and around the Carmacks Project site have been documented in numerous surveys conducted since 1989. The terrestrial and aquatic resources, current land uses, and heritage resources potentially affected by the project are summarized below, followed by an overview of project effects on the natural and human environments.

20.1.1 Terrestrial Resources

20.1.1.1 Vegetation

The project area is predominantly forested with approximately 97% of the project area in forest cover. Black spruce is the dominant forest community type (58%), with Lodgepole pine (20%) and White spruce (17%) each approximately equally represented (Access 2007a). Trembling aspen forest (2%), willow fen (2%), and grassland (1%) are minor vegetation community types.

20.1.1.2 Wildlife

Environment Yukon has identified key wildlife areas for important wildlife species occurring in the territory. These areas may be important in one or more stages of a species' life history, such as winter range, calving/lambing, salt licks, or summer nesting habitat and are considered important to the long-term management of the species. The mine site is located well outside any key wildlife area. Key wildlife areas occurring in the general vicinity of the project site include: summer breeding habitat for golden eagles located in the northern portion of the project study area, overlapping lower Williams Creek and the adjacent Yukon River; and, winter range for moose south of the main project development area that includes the area of the mine access road corridor but does not include the mine site or immediate surrounding area (Access 2007b). The powerline corridor crosses the Yukon River and passes through the golden eagle habitat. The corridor route was surveyed for golden eagle and other raptor nesting sites during the 2013 nesting season so the final powerline alignment will avoid any raptor habitat.

In general, the project is located in a low density moose survey block in which moose occur year-round in low numbers (PAH 1993, HKP 1995, Access 2007b, Markel and Larsen 1988, O'Donoghue et al. 2008a and 2008b).

The project area occurs outside the known range of wood bison with no known permanent occupancy in the area (PAH 1993, Access 2007b). The project area also is well west and north of key habitat areas for wood bison. Black bears are common in the project area.

Grizzly bears are much less abundant but grizzlies have been observed along the Freegold Road and the Yukon Quest trail (PAH 1993, Access 2007b).

The Little Salmon/Carmacks First Nation (LSCFN 2011) has identified the Yukon River between Tatchun Creek and Minto as important habitat for moose, salmon, and other wildlife. This reach of the river includes several sloughs and islands and provides important calving, summer range, and winter range habitat for moose. Moose were commonly observed through this reach in the 1960s, but are less commonly observed currently, perhaps due to increased river travel traffic during summer. Hunting does not appear to be responsible for the reduced frequency of moose observance. Few people are hunting along the river and licensed harvests are low. Dog Salmon Slough is another important habitat area located approximately 2.5 km downstream of the confluence of Williams Creek with the Yukon River. Bears use this area for fishing. None of the project elements interacts with these areas.

20.1.2 Aquatic Resources

20.1.2.1 Surface Water Hydrology

The project site is located in the Williams Creek watershed, a local tributary of the Yukon River. The watershed is comprised of two sub-basins, Nancy Lee Creek and Williams Creek. The entire project site is located in the Williams Creek sub-basin (Figure 20-1).

Nancy Lee Creek drains approximately 44 km^2, flowing into Williams Creek approximately 1 km from the Yukon River. Williams Creek drains approximately 42 km^2 upstream of the confluence with Nancy Lee Creek, with approximately 2 km^2 contributing to flows below the confluence. Williams Creek discharges to the Yukon River approximately 40 km northwest of the village of Carmacks. The Yukon River above the mouth of Williams Creek drains approximately 90,600 km^2. Based on the ratio of drainage areas, the Williams Creek watershed accounts for approximately 0.1% of the total Yukon River flow below the confluence.

Flows were monitored on Williams and Nancy Lee creeks periodically between 1991 and 1994 (HKP 1995), and from 2006 through the 2012 open water season (Access 2013). Flows are highly seasonal, typically peaking in May during freshet and then dropping to steady state flow maintained by a combination of baseflow and precipitation runoff in June through September, and finally dropping to baseflow only in October through to the next freshet. Baseflow in upper Williams Creek above the confluence with Nancy Lee Creek is estimated at <0.02 m^3/sec (1,500 m^3/day) compared to average steady state flow of 0.3 to 0.6 m^3/sec (25,000 to 50,000 m^3/day) and average freshet flow of 3.4 m^3/sec (293,000 m^3/day) (Golder 2012b).

20.1.2.2 Hydrogeology

The understanding of the groundwater system on and around the project site has been developed through the monitoring of numerous wells on the project site, and particularly in the vicinities of the planned open pit, WRSA, and formerly considered the heap leach facility (HLF). Pump tests, determinations of hydraulic conductivity, and monitoring of piezometric levels informed the development of an updated FEFLOW groundwater model for the project site (Golder 2012c). Additional investigation of hydrogeology conditions in the area of the proposed TMA is planned for spring 2017 and this information will be incorporated into an updated groundwater model for the site.

The general project area is characterized by a regional groundwater flow system within bedrock. Groundwater is recharged by precipitation at higher elevations in the upland areas and flows toward the valleys of Nancy Lee Creek, Williams Creek, Merrice Creek, and the Yukon River. Overall, the water table mimics ground surface topography and the depth to groundwater generally increases with increasing ground surface elevation. Based on groundwater levels in the monitoring wells, the depth to groundwater in the previously proposed WRSA ranges from 2 m (near Williams Creek) to 50 m. In the vicinity of the proposed open pit, the depth to groundwater exceeds 91 m. The presence of permafrost may have resulted in the development of perched water tables in some areas; however, these are assumed to be isolated and discontinuous. The permafrost likely acts as a barrier to infiltration in some areas, thereby reducing recharge and potentially resulting in the overall depression of the regional water table.

In the vicinity of the mine site, groundwater flow direction is toward Williams Creek and maintains baseflow in the creek. Mine site development, operation, and closure therefore only have the potential to affect groundwater reporting to, and affecting baseflow in upper Williams Creek (Golder 2012c). Peak flows are not expected to be measurably affected.

20.1.2.3 Surface Water Quality

Background surface water quality on and around the Carmacks project site has been extensively characterized, with monitoring conducted periodically beginning in 1989. The most recent monitoring program operated from 2005 through the 2012 open water season, with 11 monitoring stations located on Williams Creek and its tributaries and two stations on the Yukon River, 100 m upstream and 300 m downstream of the mouth of Williams Creek (Access 2013; Figure 20-1).

Yukon typically manages water quality through application of the CCME water quality guidelines appropriate to the water use being protected (aquatic life, drinking, recreation, or agriculture) with protection of aquatic life being the focus for the local streams and the adjacent Yukon River. The applicable British Columbia guideline may be used in place of the CCME guideline in cases where the BC criterion is considered more appropriate to the local conditions. Site Specific Water Quality Objectives (SSWQO) are developed for locations where background concentrations of one or more parameters typically exceed the established guideline.

The local surface waters are slightly alkaline, with mean Ph at all monitoring stations between 7.6 and 8.1. Consistent with the alkaline Ph, the waters of the Williams Creek watershed are well-buffered. Mean total alkalinity is highest on the Williams Creek mainstem and on Nancy Lee Creek. Similar patterns of spatial variation are evident in hardness, sulphate concentration, specific conductance, and total dissolved solids. The spatial patterns of alkalinity and hardness are likely a reflection of the relative contribution of groundwater to baseflow, with the higher concentrations indicating a greater groundwater influence.

In the Williams Creek watershed and in the Yukon River near the mouth of Williams Creek, most parameters consistently occur at concentrations below the applicable CCME or BC Guideline for the Protection of Freshwater Aquatic Life. Several parameters occasionally exceeded the applicable guidelines in the 393 samples collected between 2005 and 2012, including: cadmium (in three samples), lead (in two samples), silver (in six samples), and zinc (in one sample).

Aluminum, copper, and iron concentrations frequently (Al in 28% of samples, Cu in 11%, and Fe in 36%) exceeded the applicable guidelines and SSWQOs have been proposed for these parameters using the Background Concentration Procedure (CNMC 2016).

20.1.2.4 Sediment Quality

Stream sediment quality was initially sampled in July 1992 and then in each of the 2005, 2006, and 2007 open water seasons (Access 2008). Parameters analyzed included: Ph, Al, As, Cd, Cu, Fe, Pb, Ni, Se, and Zn. Sediment Ph was circumneutral at all locations. Arsenic, cadmium, lead, and zinc concentrations were below the respective CCME Interim Sediment Quality Guidelines (ISQG) at all locations in the watershed. Mean Al concentrations ranged between 6,525 and 9,191 µg/g, with no evident trend regarding the location in the watershed. Mean Fe concentrations ranged between 14,355 and 23,725 µg/g, also with no evident spatial trend in the watershed. Mean Ni concentrations ranged between 2.6 and 18.5 µg/g, with the highest concentrations occurring near and below the confluence with Nancy Lee Creek. Selenium concentrations were typically below the reportable detection limit throughout the watershed. No CCME guidelines have been set for Al, Fe, Ni, or Se in sediment.

20.1.2.5 Fish and Fish Habitat

Several fish surveys have been conducted in Williams Creek, Nancy Lee Creek, Merrice Creek near the access road crossing, and at the mouth of Williams Creek at the Yukon River (August 1991 and August 1992 (PAH 1993); Oct 2005; June 2006; July/August 2006; and, September 2006 (Access 2007c); July and September 2009 (Access 2010b)). Fish have consistently been found in the section of Williams Creek below the confluence with Nancy Lee Creek (i.e., at stations W10 and W12), but have not been found at any other location in Williams Creek or Nancy Lee Creek. Fish captures or observations upstream of station W12 have been limited to the capture of a single slimy sculpin at W13 in October 2005 and the observation of a single adult grayling in the pool at W11 in July 2009. No fish have ever been captured upstream of station W13 and no fish have been found in the reach of Merrice Creek where the access road crossing is located, consistent with the absence of fish from the upper reaches of Williams Creek.

Fish are thought to avoid moving up Williams Creek because of the consistently colder water than in the lower creek and in the Yukon River. Benthic invertebrate standing stocks also are much lower in upper Williams Creek.

Lower Williams Creek, below the confluence with Nancy Lee Creek, provides rearing habitat for fish during the open water season. Species found in lower Williams Creek include juvenile Chinook Salmon, Arctic Grayling, Slimy Sculpin, Longnose Sucker, Burbot, and Northern Pike.

20.1.2.6 Species at Risk

Species at risk with the potential to occur in the project area include:

- Threatened – wood bison, peregrine falcon (Anatum subspecies);
- Special Concern – grizzly bear, wolverine, short-eared owl; and,
- At Risk in Yukon but not elsewhere – mule deer, elk, cougar.

The project area does not provide critical habitat to any life stage of these species and is not expected to adversely affect any of these species (Access 2006).

20.2 Current Land Uses

20.2.1 Commercial and Industrial

The Carmacks copper property is comprised of 373 mineral claims and 20 leases, all of which are 100% owned by CNMC. Site activities to date have included access road and exploration camp construction, exploration drilling and trenching, environmental baseline studies, and limited site preparation in the form of forest clearing from portions of the area previously designated for a HLF. The CNMC exploration camp is operated seasonally under a Class 3 Quartz Mining Exploration Permit.

There is no commercial forest harvest activity in the project area due primarily to the low timber values and distance from markets. The mine site, access road, and powerline corridor west of the Yukon River are located within Registered Outfitting Concession #13, and the powerline corridor on the east side of the Yukon River is in Registered Outfitting Concession #14. The holder of Concession #13 has indicated the project area is not generally hunted.

20.2.2 Traditional and Cultural Land and Resource Use

The property is located within the Traditional Territories of the LSCFN and the Selkirk First Nation (SFN).

The late summer/fall Chinook and Chum salmon spawning runs on the Yukon River support aboriginal food and commercial fisheries. Members of the LSCFN fish at many sites along the Yukon River between Carmacks and Fort Selkirk as well as at sites upstream of Carmacks. Fishing locations vary annually depending upon flow conditions on the river. Most fishing is along the mainstem of the Yukon River, although traditional fishing may at times occur at the mouth of Williams Creek. Some sport fishing may also occur at the mouth of Williams Creek as recreational canoeists make their way down the river to Dawson City.

The mine site, access road, and powerline corridor west of the Yukon River are located within Registered Trapline #147. The powerline corridor east of the Yukon River is located in Registered Trapline #143. Trapline production statistics are not publicly available. Expected harvest includes mink, beaver, fox, marten, squirrel, lynx, coyote, and wolverine.

The property is part of the LSCFN traditional hunting grounds. The LSCFN collects native plants for medicinal and traditional purposes throughout the region. The property does not provide a unique source of any plants used by LSCFN.

20.2.3 Settlement Land and Land Claims

None of the project components or activities is located on settlement lands and any nearby settlement lands are held by LSCFN. The closest settlement lands downstream of the project, LSC S-30B1, are located approximately 4.5 km downstream of the mouth of Williams Creek. Settlement lands LSC R17-B are situated on the east bank of the Yukon River approximately 4.8 km downstream of the mouth of Williams Creek. Six LSCFN settlement land parcels occur adjacent or near to the Freegold Road.

There is one land claim selection located near the project. LSCFN has selected parcel R-9A west of the project site. This parcel extends into the project environmental assessment study area but does not include any of the mineral claims or leases or any of the areas in which project activities are proposed. The land selection is upstream of any project components or activities and is not expected to be affected by the project.

20.2.4 Heritage Resources

Archaeological impact assessments were conducted in the Williams Creek valley for the proposed project in August 1992 (Antiquus 1993) and along potential powerline routings in 1994 (Antiquus 1995) and in 2013 (EcoFor 2015). In 1998, Hare (1999) conducted archaeological assessments at several locations in southern Yukon in connection with various development proposals and community requests. Thomas (2006) also examined heritage resources in the area of McGregor Creek as part of the Heritage Resources Impact Assessment conducted in preparation for the Carmacks Stewart/Minto Spur Transmission line. The collective findings of these studies are that no archaeological sites occur on the proposed mine site; five sites occur along the powerline corridor (three sites near the confluence of McGregor Creek and the Yukon River and two sites adjacent to Williams Creek between Nancy Lee Creek and the Yukon River); and, there are two locations with medium heritage resource potential on the planned access road corridor, at the crossings of Merrice and Williams creeks. All of the sites along the powerline can be avoided through minor adjustment to the line alignment within the corridor. Site specific surveys of the stream crossing locations will be conducted during detailed design and the final crossing locations adjusted as necessary on the basis of these survey findings to avoid interaction.

20.3 Environmental and Social Effects

20.3.1 Terrestrial Resources

The project is not expected to have significant adverse effects on terrestrial resources. This represents the combined result of the small footprint of disturbance (approximately 155 ha cleared for the mine site, 12 ha cleared for the access road, and 38 ha cleared for the powerline right-of-way; total 205 ha), absence of critical wildlife habitat in and near the areas of disturbance, and absence of vegetation species at risk.

20.3.2 Aquatic Resources

The project site and all infrastructure in the Williams Creek watershed are located well upstream of any waterbodies that directly provide habitat for any fish species. The closest fish habitat on Williams Creek is located more than 3.5 km downstream of any site development, below the confluence with Nancy Lee Creek. Similarly, the existing and proposed bridge crossings of Merrice Creek are located upstream of any fish bearing waters. No direct interaction with fish habitat is part of any phase of the project plan and no Fisheries Act authorizations are required.

Potential effects of the project on aquatic resources therefore are related to how the project will affect the quantity and quality of water leaving the site. Effects on quantity are limited to the Williams Creek sub-watershed, and arise from: surface runoff management on the mine site, groundwater withdrawals from the water supply wells, open pit dewatering for mining, and then open pit filling after mining has been completed.

A FEFLOW groundwater model has been developed for the site to examine the effects of project plans on groundwater conditions (Golder 2012c) but the model has not yet been updated to incorporate the project changes since the 2012 FS. The model indicated local groundwater flows maintain baseflow in the adjacent upper Williams Creek and that project-related effects on groundwater flows would reduce baseflows in upper Williams Creek. The model will be updated during subsequent project planning phases prior to finalizing the YESAB Project Proposal.

Four sources of contact water will be managed during operations: groundwater seepage and precipitation pumped from the open pit sump; seepage and surface runoff from the WRSA; surface runoff from the TMA; and, excess process water from the process plant.

Water inflow to the open pit will originate from precipitation, runoff from the local contributing watershed, and groundwater seepage. Precipitation and runoff are the primary inflows to the pit through the first two years of mining, with groundwater seepage increasing through mining Years 3 to 6 and remaining high in year 7. Excess pit water will be pumped to the Waste Rock Storage Area Sediment Pond (WRSASP). Water quality in the WRSASP will be monitored and excess water will either be discharged directly to North Williams Creek if quality is suitable, or will be directed to the high density sludge (HDS) water treatment plant for treatment before discharge.

A GoldSim water quality model has been developed for the site to examine the effects of project plans on receiving water quality (Golder 2012d). The model remains to be updated to reflect the current project plan and this will be done during subsequent project planning phases prior to finalizing the YESAB Project Proposal. Project specific effluent quality objectives have already been proposed to protect downstream water quality. Pre-discharge quality monitoring to identify treatment as needed and the availability of active water treatment during operations and closure provide protection for receiving water quality through these project phases.

Once mining ends, pumping will be discontinued and the pit will be allowed to fill with water. The final pond elevation of approximately 712 masl is nominally 80 m below the rim of the pit. There will be no surface discharge from the pit lake. The final water level in the pit will be balanced by precipitation, evaporation, and seepage to groundwater. Pit lake seepage will daylight to upper Williams Creek above the confluence with Nancy Lee Creek.

Project site discharges to surface waters during the post-closure period will include runoff from the WRSA and TMA and groundwater seepage from the pit lake.

20.4 Socio-Economic Effects

An assessment of the socio-economic effects of the project was completed in 2007 for the previous project proposal to YESAB (Vector Research and Research Northwest 2007). This assessment indicated the project as then proposed would not have any adverse socio-economic effects on local communities or Yukon as a whole, and there were several identified significant positive effects associated with the project. The changes to the project detailed in the present study are not expected to alter these findings. Specific issues are examined below.

20.4.1 Commercial Land Use

There is limited commercial land use activity in the project area, currently amounting to occasional commercial hunting. The only concern related to the project is the potential of increased bear control actions related to site management. CNMC plans to manage the mine site and overall project operation to minimize the attraction of bears, and therefore the requirement for bear control actions. This approach is both for protection of site personnel and the bears. No adverse effects on commercial land use are expected to occur during project development, operations, closure, or post-closure.

20.4.2 Traditional Resource Use

The primary traditional resource use in the project area is trapping. The mine site, access road, and powerline west of the Yukon River are located on Registered Trapline #147. The powerline and substation east of the Yukon River are located on Registered Trapline #143. CNMC will work with the RTL holders to ensure access to lines is maintained, portions of trapline trails that are disturbed by project elements are relocated, overall effects are minimized, and non-mitigable effects appropriately compensated.

No effects on fishing success are expected to occur during project development, operations, closure, or post-closure. Similarly, no effects on hunting success are expected to occur during project development, operations, closure, or post-closure.

20.4.3 Recreational Land Use

The project is not expected to affect recreational land use. No recreational uses will be displaced by the project and the mine site will not be visible from the Yukon River, an important recreational waterway.

20.4.4 Community Engagement

Engagement with the local communities related to the Carmacks Copper Project has been undertaken in several periods since interest in developing the deposit was first expressed in 1991. The local stakeholder communities include the LSCFN, the SFN, and the village of Carmacks. The project is located on the traditional resource areas of both First Nations and primary project access passes through the Village.

The first period of community engagement extended from 1991 to at least 1997, and included public meetings, as well as exchanges of technical documents and correspondence. The project did not fully complete environmental permitting at that time, instead being put on hold due to market conditions. Interest in project development returned in 2004, when then owner Western Silver initiated enquiries into the permitting process in consideration of legislative changes since the initial project submission. The communities were again engaged using a combination of meetings, information sessions, and exchanges of technical documentation and correspondence in 2005, 2006, and 2007, and the public process of both the Yukon Environmental and Socio-Economic Assessment review in 2007/2008 and the Yukon Water Board review in 2009/2010. Concerns expressed by the communities were primarily related to the potential environmental effects of the project and, in particular, the post-closure effects.

Since formation in October 2011, CNMC has been working on project design changes to address the environmental concerns of the local communities and to communicate these changes to the communities. All technical documentation submitted to the YG agencies is also provided to the communities; a public open house information session was held in the village of Carmacks in August 2012 and an information sharing meeting was held with LSCFN administration in August 2012.

CNMC signed a letter of intent with LSCFN in December 2012, which initiated consultation on the project and its potential environmental and socio-economic effects. Funding was provided to LSCFN by CNMC to enable the First Nation to conduct an independent technical review of the project plans, which will continue through the project re-engineering and subsequent permitting.

20.5 Permits

Major hard rock mining projects in Yukon are required to satisfy a two-step regulatory review and approval process before mining activity may commence. The first step is an environmental and socio- economic assessment conducted in accordance with the Yukon Environmental and Socio-Economic Assessment Act (YESAA) which is administered by the Yukon Environmental and Socio-Economic Assessment Board (YESAB). The YESAA review typically takes from 9 to 18 months to complete, depending on the project, the issues, and the need for supplementary information beyond that initially submitted by the proponent.

The second step is the regulatory phase involving two enabling licenses, the Quartz Mining License (QML) and the Water Use License (WUL). The QML process is administered by Yukon Energy, Mines, and Resources (EMR) and the QML regulates the following mining related activities:

- The area and mineral deposits to be mined;
- Allowable mining and milling rates;
- Pre-construction plans and drawings;
- Post-construction as-built drawings;
- Monitoring programs;
- Design of mine workings, including underground and open pit development and production, and waste dumps;
- Site infrastructure, including buildings, roads, fuel storage, etc.;
- Solid waste disposal;
- Reclamation, including slope stability, erosion control, and re-vegetation;

- Financial security; and,
- Annual reporting requirements.

The WUL process is administered by the Yukon Water Board and regulates the use of water, the deposit of waste into water, receiving water quality, and all water conveyance and retention structures associated with a development. Any WUL issued for the project will set limits on the quality and quantity of discharges to water and on the quantities of any surface or groundwater takings. The WUL also will set monitoring and reporting requirements for surface and ground waters, for water discharges, and for water management structures such as dams, dykes, and ponds. A Type B WUL would be necessary for the project construction phase, in order to provide the necessary water to supply the construction camp and other construction activities as well as for the sanitary septic system. A Type A WUL will be required for project operation, involving the taking of process water, pit dewatering, and the discharge of any treated water.

The environmental assessment phase must be completed and a positive decision (i.e., an approval) issued by YESAB before the regulatory phase of permitting can be completed. Yukon EMR will review a QML submission in advance of a YESAB decision but cannot issue a QML until the decision document for the YESAA review has been issued. With a QML application developed and submitted in advance of a YESAB decision, the QML decision can proceed quickly following a positive decision by YESAB. The Yukon Water Board does not review a WUL license application until the YESAA process is complete and a decision document issued, and the WUL review process can take several months, particularly for a Type A licence review which also requires a public hearing.

The project, as it was previously proposed in 2007, received a positive environmental and socio-economic assessment determination from YESAB in 2008 and a Quartz Mining License in 2009. The nature of the project changes proposed in the present plan are such that that the project proposal must again pass an Executive Committee Level Environmental Screening Assessment before the regulatory phase, in which the QML is either amended or a new QML is issued and a WUL issued. Much of the project information and potential environmental effects have already been reviewed by YESAB, as part of the previous copper-only, project submission which should assist in expediting the next YESAB review.

20.6 Schedule

The project schedule has been developed on the basis of the expected schedules for the project environmental assessment and permitting and the time required for project engineering and ordering of long lead capital equipment. The schedule assumes that some construction can proceed on the basis of the existing and valid QML, along with a Type B WUL that would allow operation of the construction camp. Other factors that may potentially affect the project schedule are project financing and metal prices.

Accordingly, CNMC plans to initiate pre-feasibility engineering in Q1 of 2017, with an emphasis on metallurgical testing and related engineering costing studies for refinement of the copper and gold/silver recovery processes. Basic and detailed engineering would follow a positive pre-feasibility study, with basic engineering complete around the end of Q1 2018 and detailed engineering continuing through Q4 of 2018.

Development of the updated YESAB Project Proposal would occur in parallel with the feasibility study work, leading to a submission to YESAB in late Q4 of 2017. Based on discussion to date with YG and YESAB, the YESAB review is expected to take approximately 9 to 12 months to complete given that much of the information was assessed in the previous YESAB review. Yukon EMR will determine if an amended or new QML will be issued upon review of the QML application, with the new or amended QML expected in Q1 of 2019, prior to the start of construction.

The construction schedule is based on starting construction of project elements covered by the existing QML, with the key elements being the access road and construction camp, which need to be ready for the start of the 2019 construction season. The start of construction also is dependent on availability of project financing.

20.7 Water Management Plan

The climate observed at the project is defined by distinct seasons. In winter (October to April), precipitation is accumulated as snow. Peak flows occur during the freshet month corresponding to the snowmelt in May. Steady state flows are then established during the remaining months (June to September). Net water production from the site can either be used as process make-up water or must be managed for off-site discharge to local receiving waters.

The conceptual water management plan has been developed to manage water from the following site facilities:

- Open pit;
- TMA;
- WRSA; and,
- The process plant site.

The current water management plan builds on the plan developed by Golder (2012b) for the copper leach project as it was previously detailed in the 2012 FS. The site layout is shown on Figure 17-2 and the conceptual water management flow sheet is on Figure 20-3. Water will be managed to minimize discharges to Williams Creek by supplementing fresh water requirements in the process plant with site water. The water management strategy is summarized below by project phase.

20.7.1 Phase 1: Operations

The operations period extends from the start of mining until the completion of metal recovery. Mineralized material will be mined for approximately 6.5 years, and copper, gold, and silver recovery will continue for a short period thereafter, for a toal project life of approximately seven years.

The process plant contains the copper and gold leaching, recovery, and refining facilities. The leaching tanks and CCD thickeners will be located in a lined/bermed area that will provide secondary containment of any potential leakage. The metallurgical process will initially be supplied by fresh water drawn from groundwater, but ongoing operation will largely be supplied by water reclaimed from the process streams or from the tailings management area sediment pond (TMASP) and the WRSASP. Fresh make-up water will only be necessary for reagent make-up and gland water.

Seepage from TMA underdrains and surface runoff diverted around the TMA will be managed as non-contact water and will be directed through an unlined settling pond for discharge to North Williams Creek. Runoff from the TMA will be managed as contact water and will be collected in the TMASP. The collected water can be reclaimed to the process or directed to the WRSASP for management and discharge.

Runoff and seepage originating from the WRSA is collected in the WRSASP. Precipitation, local runoff, and groundwater seepage will collect in the pit sump. The pit sump refers to the combined water contained in the pit bottom sump as well as in the satellite pit once that becomes available for water storage. Water from the pit sump will be pumped to the WRSASP. Water may be reclaimed to the process from the WRSASP or will be discharged to North Williams Creek. Water quality in the WRSASP will be monitored during operations and will be released if the quality is acceptable for discharge, If water quality is not acceptable for discharge the water will be directed to the HDS water treatment plant for treatment before release. Excess water is expected to be produced beginning in about year 3 of operations, when groundwater seepage to the open pit increases.

The following additional contingencies have also been built into the operations water management strategy:

- If an operational process plant shutdown occurs, water can be pumped from the WRSASP to the open pit to prevent discharges from these facilities; and
- The SSP provides a third level of containment for the copper and gold leach circuits. In the event of a major leak in a leach circuit the leakage would be contained by the underlying liner. The collected leakage could then be pumped to the SSP to allow access to the leach circuit for repairs. The leakage transferred to the SSP could then be returned to the process stream

Following the cessation of mining, pit dewatering will be discontinued and pit flooding will commence. The site-wide water balance developed for the previous project plan will be updated and optimized during subsequent project design phases.

20.7.2 Phase 2: Closure

The initiation of the project closure period corresponds to the shutdown of the process plant in Q2 of Year 7 of operations. Some decommissioning activities will be initiated earlier as part of the progressive reclamation plan (e.g., WRSA and TMA), while the remainder of site facilities decommissioning will commence at this time.

Water management during closure is similar to Phase 1, with the exception that pumping from the open pit will cease and no water will be reclaimed to the process plant. Surface runoff and seepage from tailings in the TMA will be directed through the TMASP to the WRSASP. Underdrain seepage and water diverted around the TMA will continue to be discharged directly to North Williams Creek. Runoff and seepage from the WRSA will report to the WRSASP. Water collected in the WRSASP will be monitored for quality and will either be discharged directly to North Williams Creek or will be treated prior to discharge. Previous water quality modelling determined that runoff and seepage from the WRSA would not require treatment following closure (Golder 2008b). Further geochemical testing and modelling of the TMA runoff and seepage is in progress and, until otherwise determined, active treatment has remained part of the post-closure water management plan.

A key component of the closure plan for the project is the transition of the active treatment plant into a passive treatment system. For the current assessment, it was assumed only active water treatment would occur during the closure phase while the passive treatment system is implemented. This phase was considered to have a duration of two to three years.

20.7.3 Phase 3: Post-Closure

Post-closure will begin when the passive treatment system becomes fully operational, receiving all water from the WRSASP. During the post-closure period, no active water management will occur.

Outflows from the passive treatment system will drain to North Williams Creek and from there to the Williams Creek main stream.

The open pit filling or flooding would be ongoing for part of the post-closure phase. Starting from the pit bottom at 640 masl, the quasi-static surface water level is predicted to be at an elevation 675 masl within approximately 10 years, 695 masl within approximately 30 years and 712 masl within approximately 200 years following mine closure (Golder 2012b). There will be no surface water discharge from the open pit during the post-closure period. Groundwater seepage from the pit will drain toward Williams Creek.

20.8 Closure and Reclamation

All quartz mines in Yukon are required to have an approved Closure and Reclamation Plan and agreed upon financial security in place prior to starting operations. There is a current approved plan in place for the project based on an earlier project design, but this plan will need to be modified to reflect the project design changes described in this report. Closure plans are then reviewed and updated at two year intervals through construction and operation to ensure the plan reflects the project as it is developed and to account for progressive reclamation measures that would reduce the final overall closure cost.

The updated conceptual closure plan for the project as described in this PEA is summarized below by major project component mine reclamation.

20.8.1 Open Pit

Closure of the open pit will involve the removal of all equipment and installations, blocking of access to the pit ramp with boulders, placement of a boulder fence along accessible sections of the pit rim, and erection of signage to warn of the open pit hazard. Once mining is complete dewatering will be terminated and the pit will be allowed to gradually fill, creating a pit lake over a period of 200 years. The final surface elevation of the pit lake will be approximately 712 masl, which will be approximately 90 m below the pit rim. The pit lake will not have a surface outflow, but seepage to groundwater will daylight in upper Williams Creek (Golder 2012b). Pit lake water quality will be similar to local groundwater and the GoldSim water quality model indicates the pit lake seepage will not adversely affect water quality in Williams Creek (Golder 2012d).

20.8.2 Tailings Management Area

Closure planning for the TMA is at the conceptual stage, consistent with the design stage of the facility. The closure concept involves creating a stable re-vegetated landform from the TMA and the abutting WRSA. The slopes of the TMA are built to 4H:1V during operation to allow progressive reclamation of the landform. Therefore, the closure landform is expected to have the same approximate footprint and size as the ultimate TMA landform. Post-decommissioning earthworks are expected to be limited to the construction of swales on the upper TMA surface and integration of the drainage with the WRSA. Drainage paths will be designed to limit down-slope flow distances and reduce surface erosion.

Progressive reclamation during operations reduces future reclamation costs and enhances environmental protection. Progressive reclamation is also valuable in establishing which closure measures will be effective during permanent closure. This can reduce the length of the active care closure phase.

Testing to date indicates that acid generation and metal leaching are not a concern for the waste rock. It has been assumed that this will also be true of the filtered tailings. Therefore, an evapo-transpirative type cover may not be required. Instead the objective of the cover would be to minimize erosion, and facilitate establishment of a vegetative cover that is consistent with the final land use and harmonious with the surrounding environment.

Organic material stripped from the area before mining and stockpiled will be re-spread on the TMA surface. The organic material will be initially seeded with native seed mixtures to minimize erosion. In closure, re-vegetation of the general site, TMA and WRSA would follow the general guidelines for reclamation in the Yukon.

20.8.3 Water Treatment

Water treatment during operations and closure will be carried out in the water treatment plant. The plant will incorporate a HDS treatment circuit for management of metal concentrations,. The HDS circuit may be comprised of more than one treatment train to allow the effective treatment of the variable water flows expected over the course of project operations. This will be examined in detail in later design stages.

A passive treatment facility (PTS) will be constructed and commissioned during closure and will be progressively brought on line during the closure period, with all water treatment carried out in the passive facility by the end of the closure period..

The PTS will handle surface runoff and seepage from the closed TMA and WRSA. The design for the PTS will be developed during subsequent design phases of the project.

20.8.4 Waste Rock Storage Area

The WRSA will be developed so that a minimum of slope re- contouring is necessary for closure; slope grading on bench surfaces will be maintained and operational slopes will be established and maintained at a stable 2.25H:1V slope. Closure will involve placement of 0.3 to 0.5 m of organic soils on the flat bench areas. Soil will be sourced from the overburden stockpiles. Lodgepole pine will be seeded on areas facing south and west and white spruce will be seeded on areas facing north and all soil placement areas will have an initial seeding of native grasses to control erosion while the seeded and native trees become established. Slopes will not be seeded. Surface runoff collection ditches and the sediment control pond (WRSASP) will be maintained as long as necessary to control sediment in WRSA runoff – typically until vegetation is well-established on the WRSA.

Geochemical testing to date has indicated the rock to be placed in the WRSA is not acid-generating and is not a metal leaching source concern, so a cover to control infiltration is not necessary. In consideration of the expected runoff quality determined in humidity cell tests, the WRSASP overflow will be directed to Williams Creek at closure. The expected WRSASP overflow quality will be verified by monitoring prior to directing the discharge to surface waters. The GoldSim water quality model results indicate that treatment of this discharge source is not expected to be necessary in order to protect receiving water quality (Golder 2012d). The mine plan, waste rock quantities and properties have not changed since the GoldSim model was developed. However, the model will need to be updated to incorporate the TMA.

20.8.5 Other Mine Site Facilities

The general approach to closure and reclamation of the other site facilities and infrastructure is to:

- Remove equipment from the site that is no longer required, typically for sale or salvage;
- Remove supplies form the site that are no longer needed – either returned to the supplier for credit or sold;
- Remove, dismantle, or demolish (as appropriate) buildings and structures – for sale, salvage, recycling of key components, or disposal, either on-site or off-site;
- Survey and remediation of all areas of soil contamination;
- Demolition of foundations to grade;
- Grading to stabilize slopes, maintain natural drainage patterns, and fit with the natural local topography;
- Cover of pads, and other disturbed areas as needed, with overburden to support vegetation; and
- Scarification of other areas and seeding of all disturbed areas to locally appropriate vegetation.

All facilities not required for reclamation and water treatment purposes will be dismantled and removed during the closure period. The water treatment plant will be maintained into closure as necessary to supplement the PTS system.

The kerosene storage will be decommissioned and removed. Backup diesel generation capacity and fuel storage will be maintained until active water treatment is discontinued and power is no longer needed on the site.

Explosives storage facilities will be owned by the explosives supplier. Closure and reclamation will also be the responsibility of the supplier.

Final closure of the solid waste facility will require the filing of a final closure plan to the Yukon Government (YG) documenting the contained materials and the conditions of the facility. Prior to final closure, any hazardous materials will be removed to a licensed handling facility and salvageable materials (metal, tires) may be recovered for salvage/recycling. Final closure will involve coverage with two compacted lifts (each 200 mm thickness) of soil, grading for drainage, and seeding.

Closure of the land treatment facility also is subject to the submission of a formal closure plan to the YG, including sampling results to document the final concentrations of contaminants in the soils being treated. Once contaminant levels have been reduced to regulated concentrations, the treated soil can be removed from the facility and used for site reclamation. The land treatment facility will be one of the last facilities to be closed on the site to ensure there is the capacity to properly manage any soil contamination identified in the course of site closure.

The power line will be removed and the right-of-way reclaimed once line power to the site is no longer required. Reclamation will be limited to contouring and vegetation of disturbed areas. These costs are included in the capital cost estimate.

20.8.6 Roads

Roads used for exploration, for access to the site (access road), and access around the site (site roads) will be decommissioned and reclaimed once they are no longer required. CNMC expects that the final disposition of the access road will be determined in consultation with the local communities and the Yukon Government. For the purpose of this study the closure cost estimate includes costs for reclamation of the 13 km site access road.

The general closure approach for roads is to ensure physical stabilization of the surface, natural drainage is not impeded (i.e., culverts removed and adjacent banks are stable), and locally appropriate vegetation is established along the cleared right-of-way. Site roads will be reclaimed during closure. Culverts will be removed, and slope surfaces re-contoured for stability and to reflect the natural local topography.

Surfaces will be scarified and re-vegetated. Exploration trails typically require minimal contouring and stabilization. Any side-cast material will be recovered, trenches backfilled, and the trail left to natural re-vegetation. Reclamation of the main access road would involve removal of all culverts and the Merrice Creek bridge crossing, restoration of drainage, and scarification and re-vegetation.

The exploration trail currently used to access the project site is not under company authority and consequently it is not a CNMC closure responsibility. Costs for access trail closure have not been included in the closure cost estimate.

20.8.7 Closure Costs

Reclamation costs are estimated to be $5.6M, based on self-performance of the works using company staff and equipment to the extent possible. Any closure plan filed with and accepted by regulatory agencies would include costing based on third party contracting of the works, as required by Yukon closure regulations.

Closure costs have been estimated based on the typical closure, reclamation, and monitoring activities for an open pit mine. Typical activities include:

- Removal of all surface infrastructure and buildings;
- Closure and capping of the TMA (performed progressively through the mine life until final closure post-operations);
- Access road closure;
- Power transmission line and substation removal;
- Re-vegetation and seeding; and
- Ongoing site monitoring.

Figure 20.1: Monitoring Station Locations

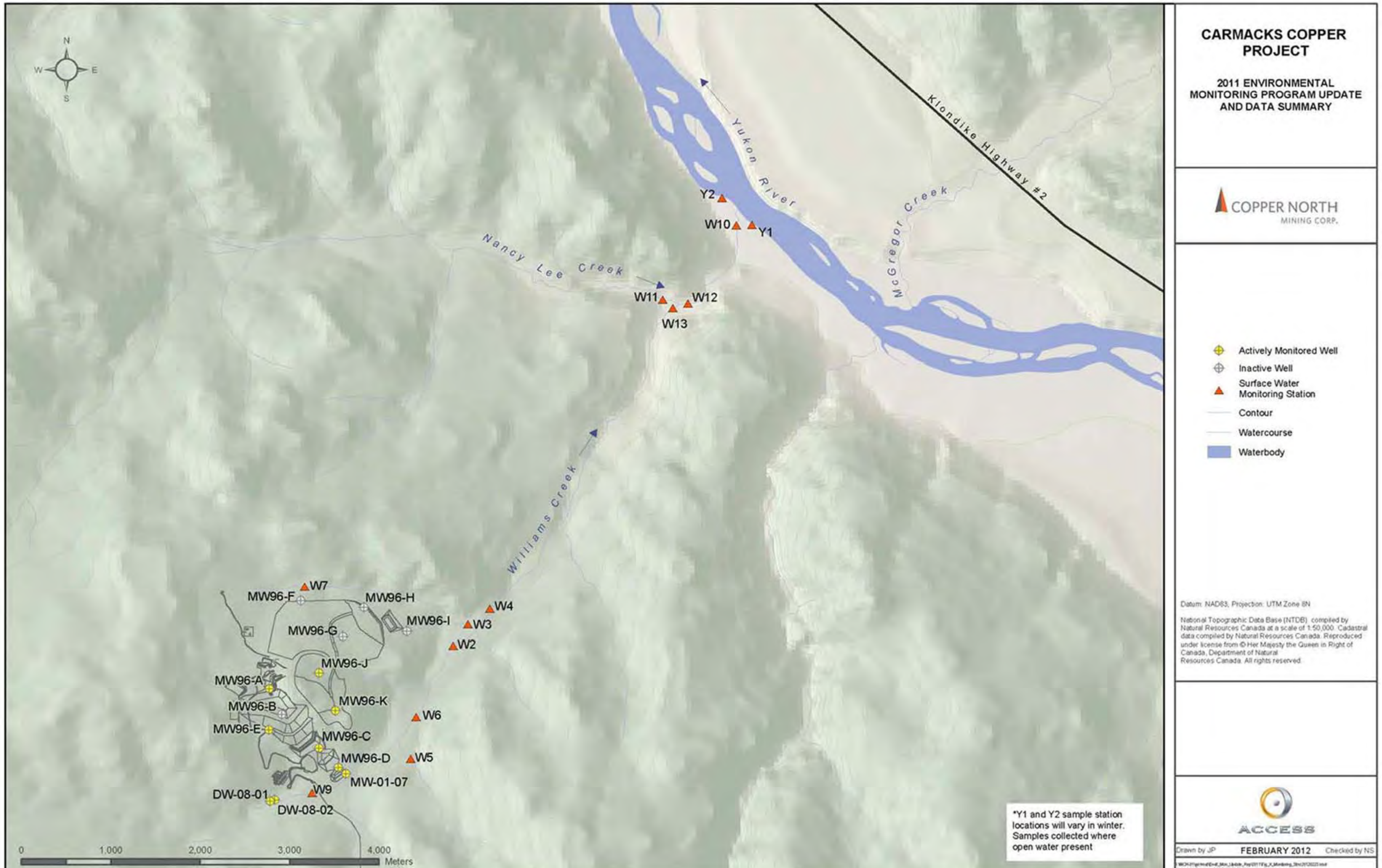
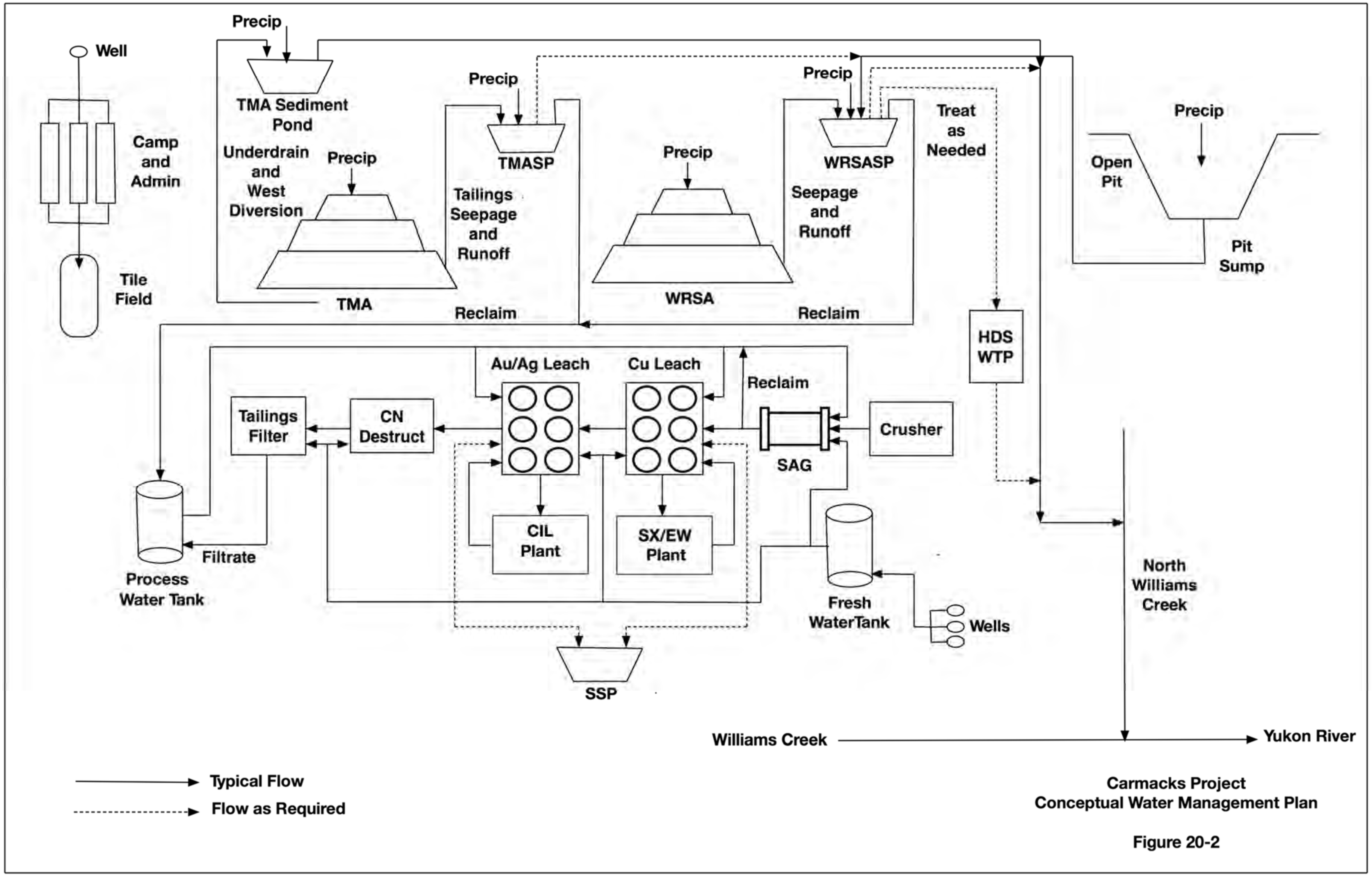


Figure 20.2: Conceptual Water Management Plan



21 Capital Cost Estimate

21.1 Summary & Estimate Results

LOM project capital costs total C$264M, consisting of the following distinct phases:

- Pre-production Capital Costs – includes all costs to develop the property to a 4,850 t/d production. Initial capital costs total $241M (including $26M contingency) and are expended over a 23-month pre-production construction and commissioning period; and
- Sustaining & Closure Capital Costs – includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations. Sustaining capital costs total $23M (including $3M in contingency) and are expended in operating years 1 through 10.

The capital cost estimate was compiled using a combination of quotations, database costs, and database factors.

Table 21.1 presents the capital estimate summary for initial, sustaining, and closure capital costs in Q3 2016 dollars with no escalation. The estimate assumes that the mining equipment will be leased.

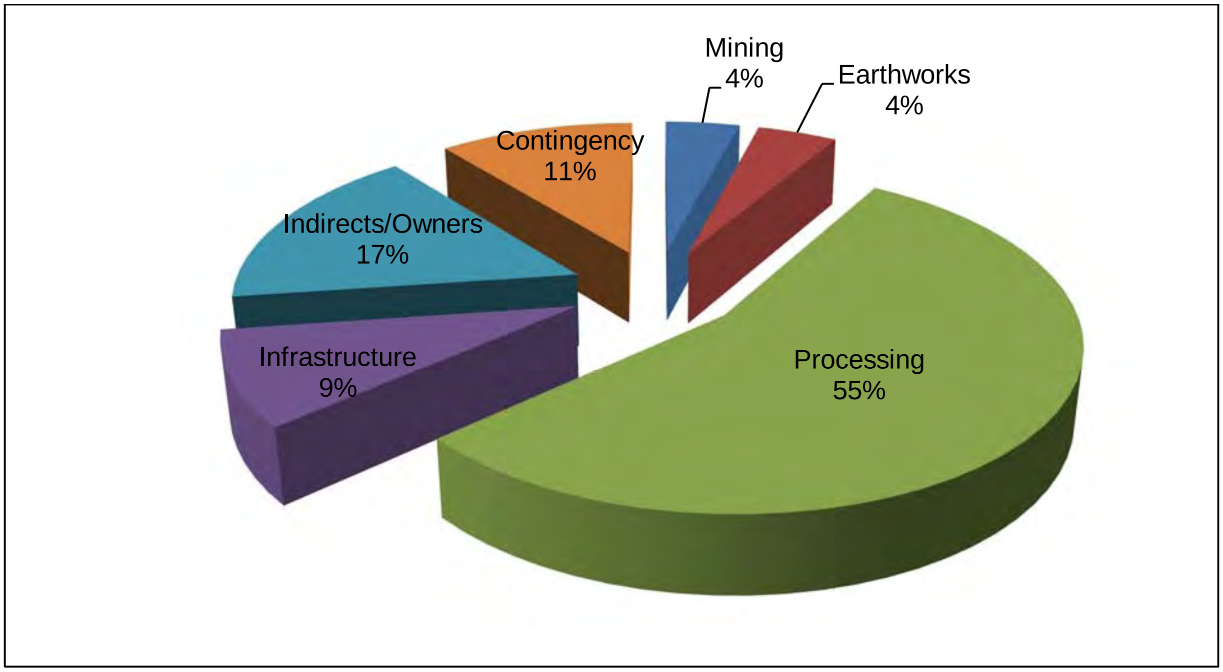
Table 21.1: Capital Cost Summary

WBS	Area	Pre-Production (M$)	Sustaining/ Closure (M$)	Total (M$)
1000	Mining	9.9	3.0	12.9
2000	Site Development	10.6	6.9	17.5
3000	Ore Crushing & Handling	3.0	-	3.0
4000	Process Plant	129.2	1.9	131.1
5000	On-Site Infrastructure	15.0	1.8	16.8
6000	Off-Site Infrastructure	7.3	-	7.3
7000	Indirect Costs	8.3	1.3	9.6
8000	EPCM	16.8	-	16.8
9000	Owners Costs	14.4	-	14.4
C100	Closure Costs	-	5.6	5.6
	Subtotal Pre-Contingency	**214.7**	**20.5**	**235.2**
9900	Contingency	25.9	2.5	28.4
	Total Capital Costs	**240.6**	**23.0**	**263.6**

Source: JDS 2016

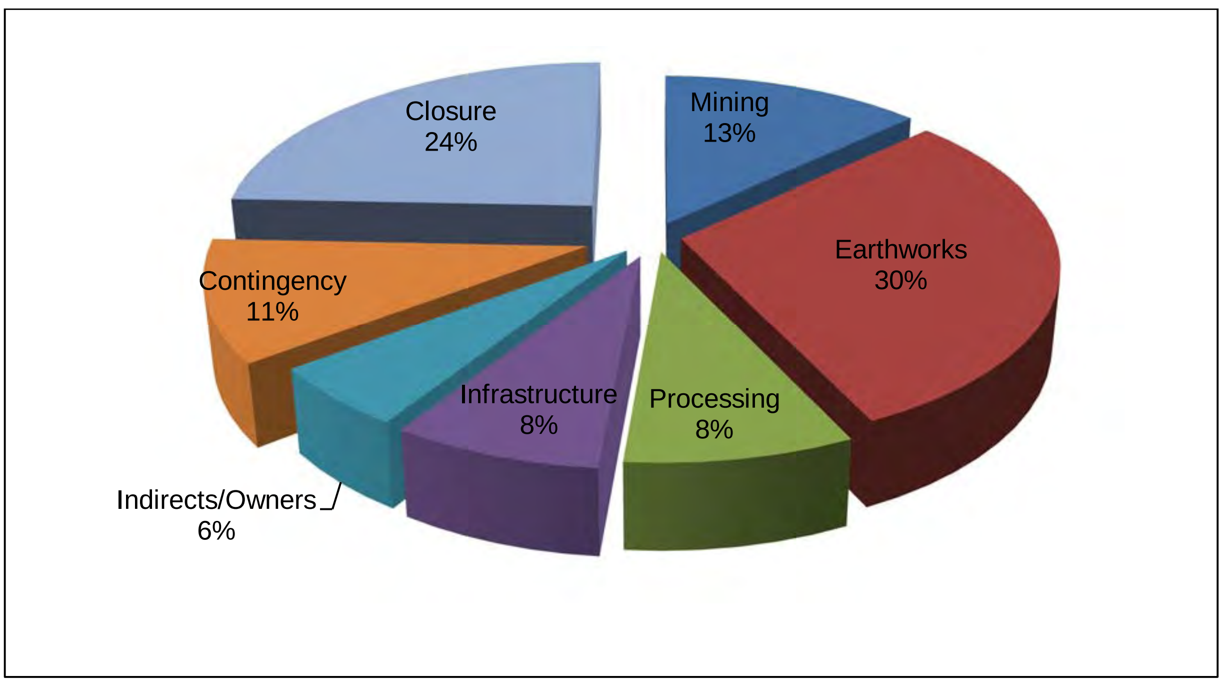
Figure 21.1 and Figure 21.2 present the capital cost distribution for the pre-production and sustaining phases. The majority of the sustaining capital estimate relates to expansion of the TMA and Waste Rock Storage Facility foundations.

Figure 21.1: Initial Capital Cost Distribution



Source: JDS (2016)

Figure 21.2: Sustaining/Closure Capital Cost Distribution

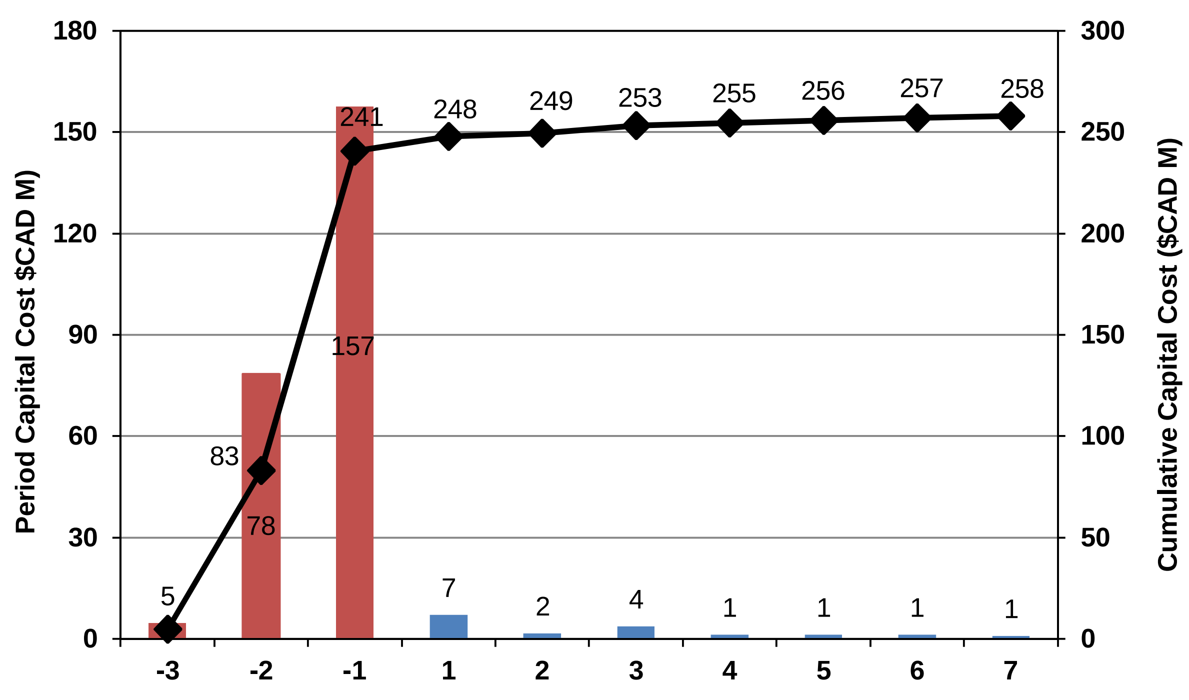


Source: JDS (2016)

21.2 Capital Cost Profile

All capital costs for the project have been distributed against the development schedule in order to support the economic cash flow model. Figure 21.3 presents an annual LOM capital cost profile (excluding closure years).

Figure 21.3: Capital Cost Profile (Excluding Closure Years)



Source: JDS (2016)

21.3 Key Estimate Assumptions

The following key assumptions were made during development of the capital estimate:

- Open pit mine development activities will be performed by an Owner-operated team; and
- All surface construction (including earthworks) will be performed by contractors under the management of an EPCM contractor.

21.4 Key Estimate Parameters

The following key parameters apply to the capital estimates:

- Estimate Class: The capital cost estimates are considered Class 4 estimates (-15%/+25%). The overall project definition is estimated to be 10%;

- Estimate Base Date: The base date of the estimate is October 1st, 2016. No escalation has been applied to the capital cost estimate for costs occurring in the future;
- Units of Measure: The International System of Units (SI) is used throughout the capital estimate; and
- Currency: All capital costs are expressed in Canadian Dollars (C$ or $). Portions of the estimate were estimated in US Dollars and converted to C$ using an exchange rate of 0.78 US$:C$.

21.5 Basis of Estimate

21.5.1 Labour Rates

The majority of installation costs within the estimate have been factored based on mechanical equipment costs. Where applicable within the estimate, an average all-in contractor crew labour-rate of $95/hr has been applied, based on buildups from other recent and similar studies.

Operational labour rates were built up from first principles. Base rates are based prevailing wages in the area, and legal premiums and benefits were built up to create all-in rates. Operational labour rates and staffing levels are described further within Section 22.

21.5.2 Fuel & Energy Supply

Where applicable, a delivered fuel price of $0.762/L and a grid power energy supply price of $0.105/kWh has been used throughout the estimate.

21.5.3 Mine Capital Costs

Mine capital cost estimates have been assembled from first principals, based on the mine production schedule.

21.5.3.1 Pre-Stripping

Pre-stripping costs include the labour, fuel, equipment usage, and consumables costs for the removal of barren waste material at the open pit prior to mineralized material processing. Costs have been assembled from first principles, based on the mine schedule and database unit costs for labour, equipment operations, and consumables.

21.5.3.2 Mine Mobile Equipment

Open pit mining equipment quantities and costs were determined through buildup of mine plan quantities and associated equipment utilization requirements. Database unit costs were applied to the required quantities.

21.5.3.3 Fixed Mine Equipment

The fixed mine equipment sector includes costs for dewatering piping and accessories, shop tools, survey equipment, computers, and engineering software.

21.5.4 Site Development & Road Works

Site development costs are generally based on high level material take-offs and database unit pricing.

21.5.4.1 Site Development

Material take-offs were developed from preliminary 3D models for earthen pads. Database unit costs were applied for excavations, fills, and surfacing materials. Allowances were made for settling ponds, surface water control, and temporary roads.

21.5.4.2 On-Site Roads

On-site road costs are based on road lengths from the general arrangement drawings and database $/km unit rates for gravel roads in similar ground conditions.

21.5.5 Process Plant

The process plant capital costs include all of the direct costs to construct the 1.8 tonne per annum processing plant. A $320,000 annual allowance is applied during operations for miscellaneous sustaining projects, rebuilds, and modifications required to maintain the nameplate throughput.

The process plant capital cost estimate was assembled form a combination of engineered take-offs, supplier quotations, contractor quotations, and database allowances. Table 21.2 presents a summary basis of estimate for the various commodity types within the process plant estimate.

Table 21.2: Process Plant Basis of Estimate

Commodity	Estimate Basis
Equipment	
Major Equipment	Budget quotations were solicited from qualified suppliers for the major equipment identified in the flow sheets and equipment register.
Minor Equipment	In-house data (firm and budgetary quotations from recent projects) was used for minor or low value equipment.
Installation (Labour & Materials)	
Concrete	High level take-off quantities were developed from general arrangement drawings and database concrete quantity ratios per facility area (m3/m^2). Database unit rates were applied to the take-off quantities.
Internal Structural Steel	Factored based on mechanical equipment costs.
Process Plant Building	Database unit costs ($/m^2) applied to areas determined from the general arrangement drawings. Fabric walled buildings assumed for the process area buildings. Lump sum allowances included for modular control and lunch rooms.
Mechanical Equipment Installation	Factored based on mechanical equipment costs.
Piping	Factored based on mechanical equipment costs.
Electrical & Instrumentation	Factored based on mechanical equipment costs.

Source: JDS (2016)

21.5.6 On-Site Infrastructure

On-site infrastructure at the Carmacks project includes a camp, office complex, mine dry, on-site power distribution, water, and waste handling infrastructure, maintenance facilities (shops and warehouses), the surface mobile support fleet, bulk fuel storage and information technology (IT) and communications systems. Table 21.3 presents a summary basis of estimate for the various commodity types within the process plant estimate.

Table 21.3: Infrastructure Basis of Estimate

Component	Estimate Basis
Accommodations, Office Complex, and Mine Dry	Factored database costs, based on the accommodations, dry, and office requirements determined for the operations and construction phases.
Maintenance Facilities	Database unit costs ($/m^2) applied to areas determined from the general arrangement drawings. Fabric walled buildings assumed for the maintenance and warehouse buildings.
Explosives Management Facilities	Lump sum allowances based on experience at similar operations.
Site Utilities	Site utilities include on-site power distribution, emergency power generation, a chlorinator water treatment plant, incinerator, and septic field. Lump sum allowances have been applied to these facilities based on experience at similar operations.
Contact Water Treatment Plant	Lump sum allowance based on other recently quoted facilities.
Surface Mobile Equipment	Surface equipment fleet requirements are determined based on material movement requirements and experience at similar operations, and considering site conditions specific to the project. Waste rock/tailing handling equipment requirements are based on equipment utilization requirements for the haulage operations. No equipment replacements are anticipated for the surface equipment fleet due to the short mine life and relatively low utilization of equipment. Database unit pricing has been applied to the surface equipment fleet quantities.
Bulk Fuel Storage & Distribution	Lump sum allowance based on other recently constructed facilities.
IT & Communications	Lump sum allowances based on experience at similar remote operations.

Source: JDS (2016)

21.5.7 Off-Site Infrastructure

The project requires a reliable road connection for access and concentrate shipments and a power connection for energy supply.

21.5.7.1 Main Access Road

Database unit rates ($/km roads, $/m bridges) were applied to the road lengths for upgrade of the main access road.

21.5.7.2 Power Transmission Line

A factored database costs for the overhead power line (including right-of-way clearing and grubbing) and mobile substations have been included in the estimate.

21.6 Indirect Cost Estimate

Indirect costs are those that are not directly accountable to a specific cost object. Table 21.4 presents the basis of estimate for each of the indirect cost categories. The majority of indirect costs in the estimate are factors or allowances based on recently completed definitive estimates for similar projects.

Table 21.4: Indirect Cost Basis of Estimate

Commodity	Basis
Construction Support Services	Included in Owners costs, refer to Section 21.7
Construction Support Services	Time based cost allowance for general construction site services (temporary power, heating & hoarding, contractor support, etc.) applied against the surface construction schedule
Temporary Facilities & Utilities	Allowance for construction offices and ablution facilities Allowance for diesel construction power
Contractor Mobilization	Factored allowance (0.5% of direct costs) for contractor mobilization and miscellaneous expenses; Note that contractor profit on labour and materials are included in the direct cost unit rates
Logistics & Freight	Factored allowance (0.5% of direct equipment and material costs) for all freight and logistics
Start-up and Commissioning	Factored allowance (2.0%) for spare parts Factored allowance (1.0%) for the provision of vendor services for commissioning support
Detailed Engineering & Procurement	Factored allowances (5%) of total direct construction costs (excluding mining) for detailed engineering and procurement
Project & Construction Management	Staffing plan built up against the development schedule for project management, health and safety, construction management, field engineering, project controls, and contract administration Database unit (hourly) rates

Source: JDS (2016)

21.7 Owners Cost Estimate

Owner's costs are items that are included within the operating costs during production. These items are included in the initial capital costs during the construction phase and capitalized. The cost elements described below are described in more detail within Section 22.

- Pre-production processing: Costs of the Owner's processing labour, power, and consumables incurred before declaration of commercial production;
- Pre-production general & administration: Costs of the Owner's labour and expenses (camp and catering, safety, finance, security, purchasing, support labour, maintenance, equipment usage, management, etc.) incurred prior to commercial production.

21.8 Closure Cost & Salvage Value

Closure costs have been estimated based on the typical closure, reclamation, and monitoring activities for an open pit mine. Typical activities include:

- Removal of all surface infrastructure and buildings;
- Closure and capping of the TMA (performed progressively through the mine life until final closure post-operations);
- Access road closure;
- Power transmission line and substation removal;
- Re-vegetation and seeding; and
- Ongoing site monitoring.

A total lump-sum closure cost of $5.6M has been used for the estimate, based on factored costs from similar projects. Progressive closure costs begin in Year 2 with the capping of the TMA. Final closure costs are incurred in Years 8 through 10.

21.9 Cost Contingency

Contingency was evaluated by major WBS area, based on the level of design and pricing confidence. The result was an overall blended contingency of 12% or $28.4M LOM.

21.10 Capital Estimate Exclusions

The following items have been excluded from the capital cost estimate:

- Working capital (included in the financial model);
- Financing costs;
- Currency fluctuations;
- Lost time due to severe weather conditions beyond those expected in the region;
- Lost time due to force majeure;
- Additional costs for accelerated or decelerated deliveries of equipment, materials or services resultant from a change in project schedule;
- Warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares;
- Any project sunk costs (studies, exploration programs, etc.);
- Closure bonding; and
- Escalation cost.

22 Operating Cost Estimate

The OPEX estimate is based on a combination of experience, reference project, budgetary quotes and factors as appropriate with a preliminary study.

The operating cost estimate in this study includes the costs to mine, process the mineralized material to produce copper cathode and doré and general and administrative expenses (G&A). These items total the mine operating costs and are summarized in Table 22.1. The estimate is based on leasing equipment and owner operating the mining and services fleet. The target accuracy of the operating cost is -25/+30%.

The operating cost estimate is broken into three major sections:

- Open Pit Mining;
- Processing; and
- General & Administrative.

The total operating unit cost is estimated to be $45.45/t processed. Average annual, total LOM and unit operating cost estimates are summarized in Table 22.1. Figure 22.1 illustrates the operating cost distribution. Equipment lease costs are included in the mine operating cost estimate.

Operating costs are expressed in Canadian dollars with a fixed exchange rate of US$:C$ = 0.78. No allowance for inflation has been applied.

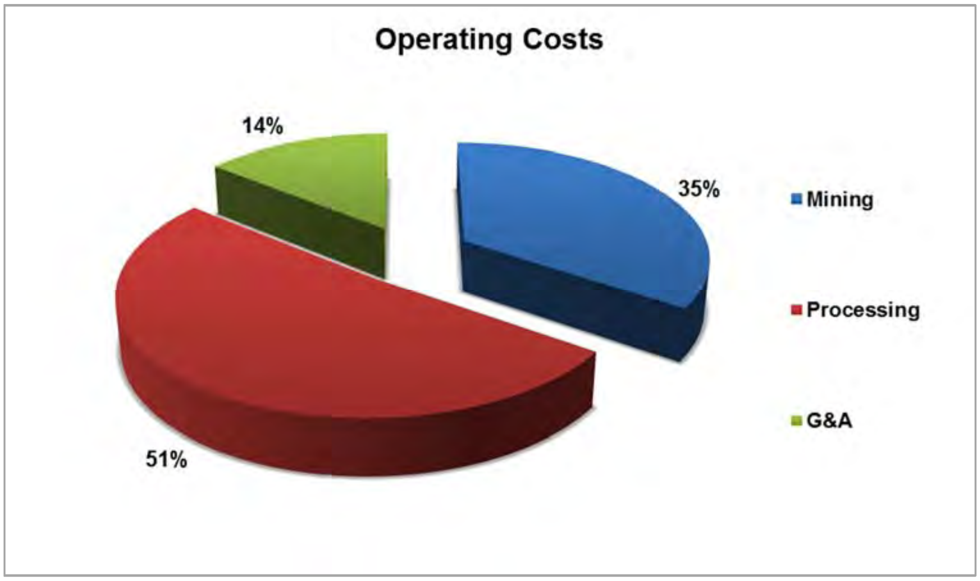
The main OPEX component assumptions are outlined in Table 22.1 and shown graphically in Figure 22.1.

Table 22.1: Breakdown of Estimated Operating Costs

Operating Costs	Avg Annual (M$)	$/t processed	LOM (M$)
Mining*	26	15.73	182
Processing	38	23.27	269
G&A	11	6.45	75
Total	**75**	**45.45**	**525**

*Average LOM Mining cost amounts to $2.63/t mined at a 5.1:1 strip ratio (excluding pre-production tonnes mined).
Totals may not add due to rounding
Source: JDS (2016)

Figure 22.1: Operating Cost Distribution



Source: JDS (2016)

The main operating cost component assumptions are shown in Table 22.2.

Table 22.2: Main OPEX Component Assumptions

Item	Unit	Value
Electrical Power Cost	$/kWh	0.105
Overall Power Consumption (all facilities)	kWh/t processed	45.75
Diesel Cost (delivered)	$/litre	0.762
LOM Average Manpower	employees	260

Source: JDS (2016)

22.1 Operations Labour

This section provides an overview of total workforce and the methods used to compile the labour rates.

Table 22.3 summarizes the total planned workforce during project operations.

Table 22.3: Summary of Personnel

Department	Total Persons Employed (Peak)
Mining	115
Processing	76
General & Administration*	70
Total Personnel – All Areas	**261**

*Includes Owner and Contractor personnel
Source: JDS (2016)

Labour base rates were determined by reference to other active northern Canadian operations and benchmarked against Costmine (Canadian Mine Salaries, Wages, Benefits 2015 Survey Results). Labour burdens were assembled using first principles. The following items are included in the burdened labour rates:

- Scheduled overtime costs based on individual employee rotation;
- Unscheduled overtime allowance of 10% for hourly employees;
- CPP, EI, WCB as legislated in Yukon;
- Statutory holiday allowance of 6% of scheduled hours;
- Vacation pay allowance of 6% of scheduled hours;
- Pension allowance of 5% of scheduled hours; and
- Flexible benefits package of $2,500 annually per employee.

22.2 Basis of Estimate

22.2.1 Mine Operating Cost Estimate

The mine operating costs include all open pit mining activities including pit and dump operations, road maintenance, mine supervision, technical services and equipment leasing. The average LOM mine operating costs (excludes pre-production) are estimated to be $15.73/t processed or $2.63/t mined and are presented in Table 22:4 by category.

Table 22.4: Summary of Mine Operating Costs

Cost Category	$/t processed	$/t mined	LOM (M$)
Labour	5.16	0.86	60
Fuel	2.61	0.44	30
Operating & Maintenance Consumables	4.72	0.79	54
Tools, Supplies & Contract Services	0.89	0.15	10
Lease Payments	2.35	0.39	27
Total Mine Costs	**15.73**	**2.63**	**182**

22.2.2 Processing Operating Cost Estimate

Process operating costs were developed using labour rates based on operating mines in the area and sufficient personnel to operate the process plant, factored maintenance cost, budget quotes for consumables and a factored power requirement. Process operating costs are summarized below in Table 22.5. Costs are subdivided into operating categories.

Table 22.5: Processing Operating Cost by Category

Category	$/tonne processed
Labour	4.04
Equipment Maintenance & Consumables (Reagents, Media, Liners and other Wear Parts)	14.80
Power & Fuel	4.42
Grand Total by Activity	**23.27**

Source: JDS (2016)

Process labour includes burden for salaried and hourly employees to account for in-country benefits, training, production bonus and potential ex-patriot benefits & costs.

Equipment maintenance was calculated by applying a factor of 4% to major process equipment cost. Costs for media were determined using engineering calculations based on mill power draw, abrasion index and vendor quotes for media as a cost per tonne. Reagent requirements from recent test work and budget quotes from vendors were used to calculate the cost of reagents. Mill liners and wear parts for major equipment were based on vendor recommended requirements and quotes.

Power costs were calculated from the total installed power assuming $0.105/kWh.

22.3 General and Administration Operating Cost Estimate

The general and administration costs include all off-site and on-site activities including personnel transportation, camp catering, surface support equipment, water treatment, and all associated labour. The G&A operating cost is estimated to be $6.45 per tonne processed and can be attributed to two categories:

- Labour; and
- On-site items.

Table 22.6: Summary of G&A Costs

Cost Category	$/t Processed	LOM (M$)
Labour	2.4	28
On-site Items	4.05	47
Total G&A Costs	6.45	75

Source: JDS (2016)

23 Economic Analysis

23.1 Summary of Analysis

An engineering economic model was developed to estimate annual cash flows and sensitivities of the project. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed and are likely to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Univariate sensitivity analyses were performed for variations in metal prices, US$:C$ exchange rates, operating costs, capital costs, and discount rates to determine their relative importance as project value drivers.

This technical report contains forward-looking information regarding projected mine production rates, construction schedules and forecasts of resulting cash flows as part of this study. The mill head grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits to construct and operate a mine, or to obtain major equipment or skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis.

The estimates of capital and operating costs have been developed specifically for this project and are summarized in Section 21 and Section 22 of this report (presented in 2016 dollars). The economic analysis has been run with no inflation (constant dollar basis).

23.2 Assumptions

The summary of the mine plan and payable metals produced is outlined in Table 23.1.

Table 23.1: LOM Plan Summary

Parameter	Unit	Value
Mine Life	Years	7.0
Resource Mined	kt	11,551
Throughput Rate	kt/d	32
Average Cu Head Grade	%	0.977
Average Au Head Grade	g/t	0.43
Average Ag Head Grade	g/t	4.34
Cu Payable	Mlbs	213
	Mlbs/a	30
Au Payable	koz	136
	koz/a	19
Ag Payable	koz	151
	koz/a	22

Source: JDS (2016)

Other economic factors include the following:

- Discount rate of 8% (sensitivities using other discount rates have been calculated);
- Closure cost of $6M;
- Nominal 2016 dollars;
- Revenues, costs, taxes are calculated for each period in which they occur rather than actual outgoing/incoming payment;
- Working capital calculated as one and a half months of operating costs (mining, processing, and G&A) in Year 1;
- Results are presented on 100% ownership; and
- No management fees or financing costs (equity fund-raising was assumed).

The model excludes all pre-development and sunk costs up to the start of detailed engineering (i.e. exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, financing costs, etc.).

Table 23.2 outlines the metal prices and US$:C$ exchange rate assumptions used in the economic analysis. The gold and silver prices used in the economic analysis are based on the 6 month average spot rate as at September 2016. For copper, the price selected was based on recently released comparable Technical Reports.

The reader is cautioned that the metal prices and exchange rates used in this study are only estimates based on recent historical performance and there is absolutely no guarantee that they will be realized if the project is taken into production. The metal prices are based on many complex factors and there are no reliable long-term predictive tools.

Table 23.2: Metal Price & Foreign Exchange Rates used in Economic Analysis

Parameter	Unit	Value
Copper Price	US$/lb	2.50
Gold Price	US$/oz	1,300
Silver Price	US$/oz	17.50
Exchange Rate	US$:C$	0.78

Source: JDS (2016)

23.3 Revenues & NSR Parameters

Mine revenue is derived from the sale of copper cathode and doré bars into the international marketplace. No contractual arrangements for refining exist at this time. Details regarding the terms used for the economic analysis can be found in the market studies (Section 19) of this report.

Table 23.3 indicates the NSR parameters that were used in the economic analysis.

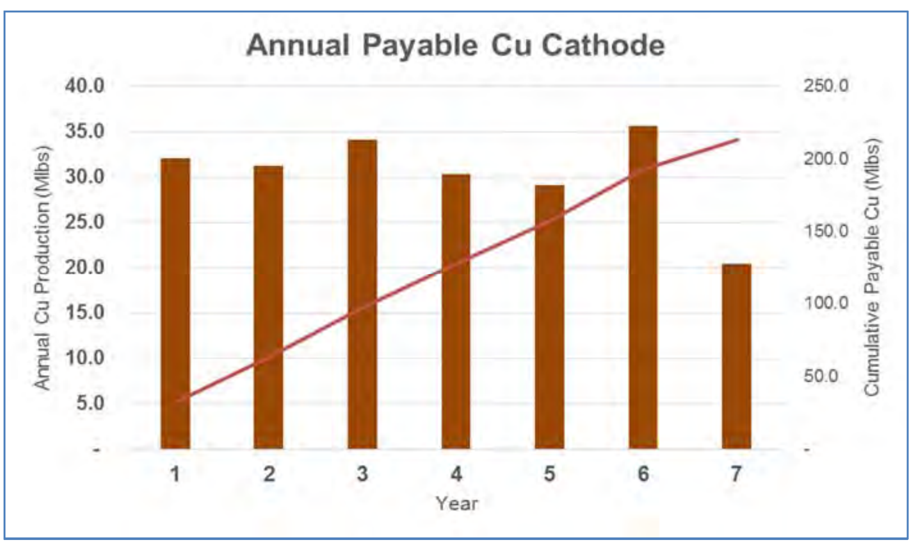
Table 23.3: NSR Parameters Used in Economic Analysis

Assumptions & Inputs	Unit	Value
Mine Operating Days	days/a	365
Archer Cathro Royalty*	% NSR	3
	% NPI**	15
Recoveries		
Cu	%	85.5
Au	%	84.4
Ag	%	9.4
NSR Parameters		
Cu Cathode Payable	%	100
Cu Cathode Shipping Charge	US$/lb	0.015
Au Payable	%	100
Au Refining Charge	US$/pay oz	4.00
Ag Payable	%	100
Ag Refining Charge	US$/pay oz	0.40

*At the election of Copper North – further terms on the royalty are outlined in Section 19 (currently capped at $1.2M).
** Net profit interest
Source : JDS (2016)

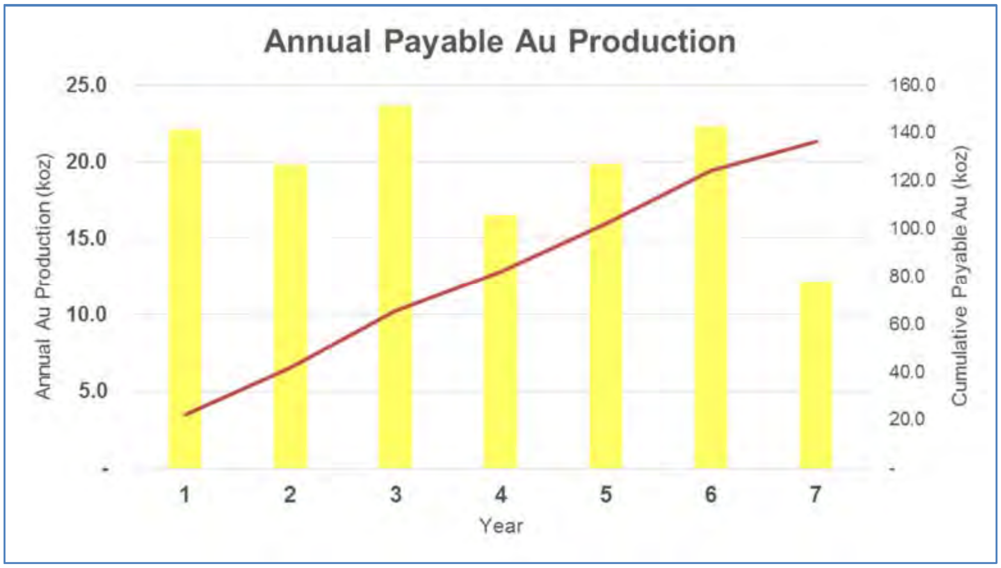
Figure 23.1, Figure 23.2, and Figure 23.3 show breakdowns of the amount of copper, gold and silver recovered during the mine life and the amount of payable metal for the project. A total of 213 Mlbs of copper, 136 koz of gold, and 151 koz of silver are projected to be produced during the mine life. Copper accounts for about 75% of gross project revenues, gold for about 25% and silver accounting for less than 1%.

Figure 23.1: Payable Cu Cathode Production by Year



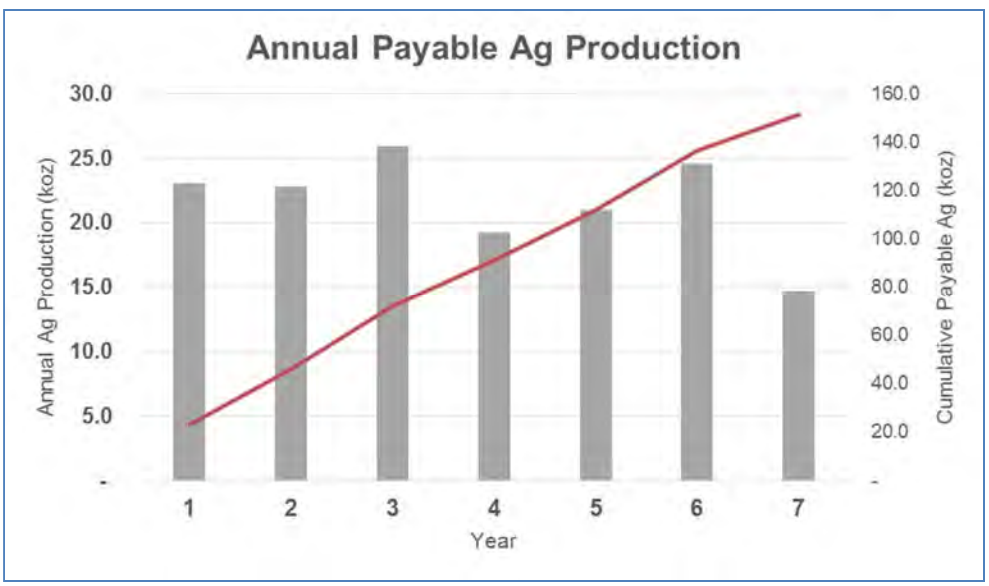
Source: JDS (2016)

Figure 23.2: Payable Au Production by Year



Source: JDS (2016)

Figure 23.3: Payable Ag Production by Year



Source: JDS (2016)

23.4 Taxes

The project has been evaluated on an after-tax basis in order to provide a more indicative, but still approximate, value of the potential project economics. A tax model was prepared by Wentworth Taylor, a specialized mining tax accountant with applicable Yukon mineral tax regime experience. Tax pools were used in the analysis. The tax model contains the following assumptions:

- 15% federal income tax rate;
- 15% Yukon territorial tax rate;
- The Yukon Mining Quartz Tax has also been evaluated as part of the after-tax analysis. The Crown royalty applies to all mineralized material, minerals, or mineral bearing substances mined in the Yukon on a calendar year basis;
- The royalty is calculated based on the value of the output mine which is the value of minerals produced exceeded by the various deductions allowable; and
- The royalty rate ranges from 0 to 12% based on the taxable revenue from saleable metals minus deductions.

Total taxes for the project amount to $43 M.

23.5 Royalties

Due to the royalties already paid by Copper North and subtracted from the maximum $2.5 M, total third party royalties for the project amount to $1.2M over the LOM. More details on the structure of this royalty can be found in Section 19.

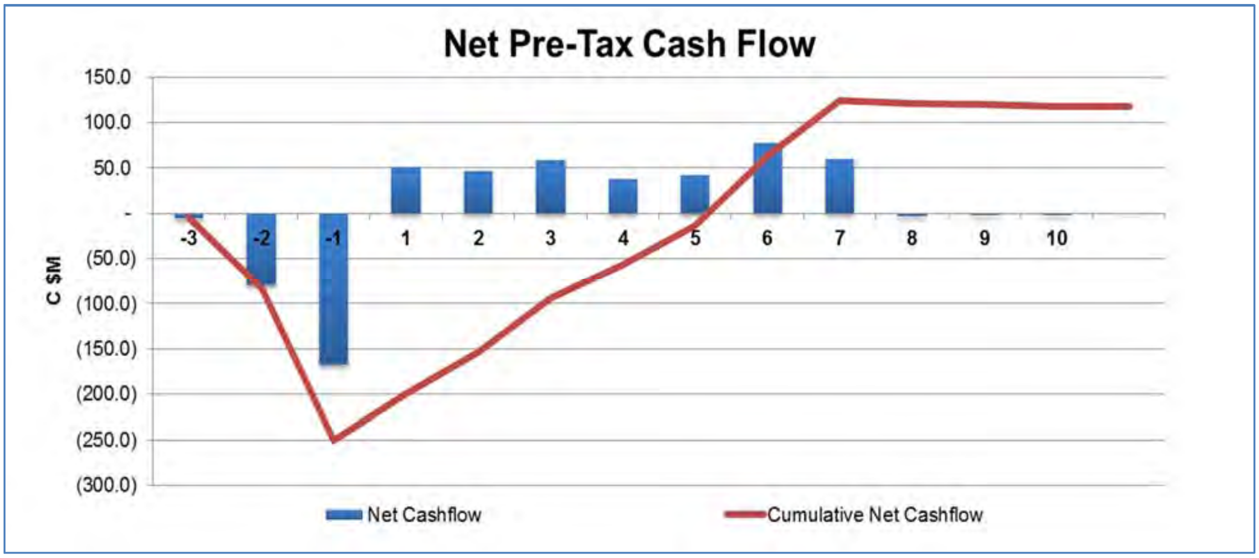
23.6 Results

At this preliminary stage, the project has a pre-tax IRR of 9.4% and a net present value using an 8% discount rate ($NPV_{8\%}$) of $12 M using the metal prices described in Section 19.

Figure 23.4 shows the projected cash flows, and Table 23.4 summarizes the economic results of the Carmacks project.

The pre-tax break-even copper price for the project is approximately US$2.43/lb, based on the LOM plan presented herein, a gold price of US$1,300/oz and a silver price of US$17.50/oz.

Figure 23.4: Pre-Tax Annual Cash Flows



Source: JDS (2016)

Table 23.4: Summary of Results

Summary of Results	Unit	Value
Cash Cost (Net of Byproduct)	US$/lb Cu	1.08
Cash Cost (incl. Sustaining and Closure CAPEX)	US$/lb Cu	1.16
Capital Costs		
Pre-Production Capital	M$	215
Pre-Production Contingency	M$	26
Total Pre-Production Capital	**M$**	**241**
Sustaining & Closure Capital	M$	21
Sustaining & Closure Contingency	M$	3
Total Sustaining & Closure Capital	M$	23
Total Capital Costs Incl. Contingency	**M$**	**264**
Working Capital	M$	10
Pre-Tax Cash Flow	LOM M$	118
	M$/a	17
Taxes	LOM M$	43
After-Tax Cash Flow	LOM M$	75
	M$/a	11
Economic Results		
Pre-Tax NPV$_{8\%}$	**M$**	**12**
Pre-Tax IRR	**%**	**9.4**
Pre-Tax Payback	**Years**	**5.2**
After-Tax NPV$_{8\%}$	**M$**	**-11**
After-Tax IRR	**%**	**6.6**
After-Tax Payback	**Years**	**5.3**

Source: JDS (2016)

23.7 Sensitivities

A univariate sensitivity analysis was performed to examine which factors most affect the project economics when acting independently of all other cost and revenue factors. Each variable evaluated was tested using the same percentage range of variation, from -15% to +15%, although some variables may actually experience significantly larger or smaller percentage fluctuations over the LOM.

The sensitivity analysis was performed on the following variables independently, assuming all others remained constant: copper price, FX rate, head grade, CAPEX, and OPEX. For instance, the copper price was evaluated at a +/- 15% range to the base case, while the FX rate and all other variables remained constant. This may not be truly representative of market scenarios, as metal prices and FX rates may be correlated and may fluctuate in a similar trend. The variables examined in this analysis are those commonly considered in similar studies – their selection for examination does not reflect any particular uncertainty.

Notwithstanding the above noted limitations to the sensitivity analysis, which are common to studies of this sort, the analysis revealed that the project is most sensitive to FX rate, followed by head grades and copper price. The project showed the least sensitivity to capital and operating costs. Table 23.5 and Figure 23.4 show the results of the sensitivity tests.

A sensitivity analysis was also performed on gold and copper prices and tested under various discount rates. The results of these tests are demonstrated in Table 23.5 and Table 23.6. Project economics improve considerably at a copper price of US$2.75/lb and higher.

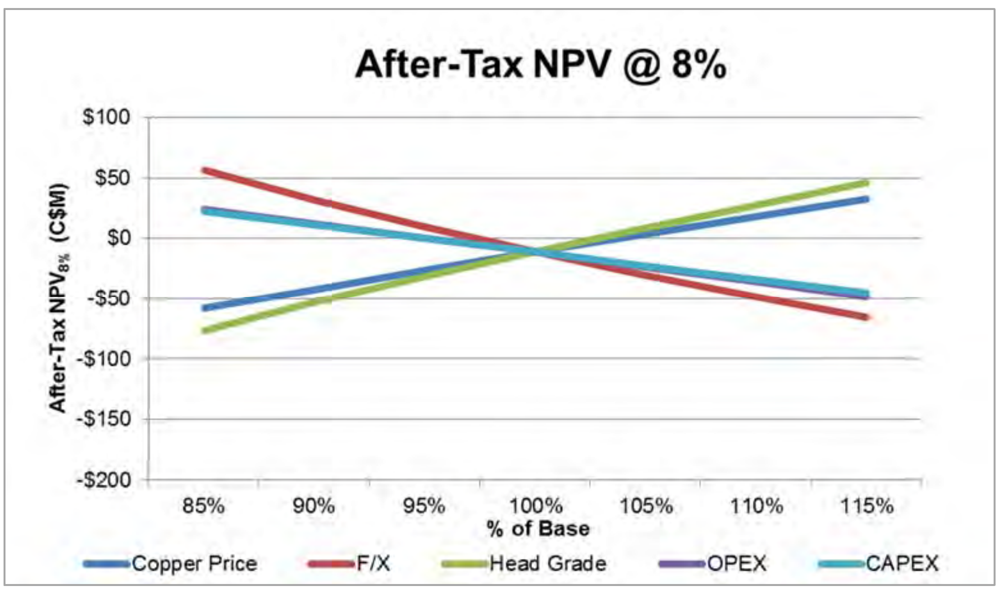
The economic cash flow model for the project is illustrated in Figure 23.6.

Table 23.5: Pre-Tax and After-Tax Sensitivity Results on $NPV_{8\%}$

Variable	Pre-Tax NPV8% (M$)			After-Tax NPV8% (M$)		
	-15% Variance	0% Variance	15% Variance	-15% Variance	0% Variance	15% Variance
Copper Price	-54	12	78	-58	-11	32
FX Rate	115	12	-65	57	-11	-66
Head Grade	-76	12	100	-77	-11	47
CAPEX	46	12	-22	23	-11	-46
OPEX	64	12	-40	24	-11	-48

Source: JDS (2016)

Figure 23.5: Sensitivity Results After-Tax $NPV_{8\%}$



Source: JDS (2016)

Table 23.6: Metal Price Sensitivity, Pre-Tax $NPV_{8\%}$

	Copper Price US$/lb						
Au US$/oz	**$1.75**	**$2.00**	**$2.25**	**$2.50**	**$2.75/lb**	**$3.00**	**$3.25**
1,000	-154	-110	-66	-22	22	66	110
1,100	-143	-99	-55	-11	33	77	121
1,200	-131	-87	-43	1	45	89	133
1,250	-126	-82	-38	6	50	94	138
1,300	-120	-76	-32	12	56	100	144
1,400	-109	-65	-21	23	67	111	155
1,500	-97	-53	-9	35	79	123	166

Source: JDS (2016)

Table 23.7: Base Case Scenario Discount Rate Sensitivity

Discount Rate (%)	Pre-Tax NPV (M$)	After-Tax NPV (M$)
0	118	75
5	44	14
7	22	-4
8	12	-11
10	-5	-25
12	-19	-37

Source: JDS (2016)

Item	Source	Unit	PP	Production	LOM	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
METAL PRICES AND EXCHANGE RATE																		
Cu	link	US$/lb	**2.50**	**2.50**	**2.50**	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50
Au	link	US$/oz	**1,300**	**1,300**	**1,300**	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Ag	link	US$/oz	**17.50**	**17.50**	**17.50**	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50
Exchange Rate	link	US$:C$	**0.78**	**0.78**	**0.78**	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78
MINE SCHEDULE																		
Mine Production Schedule																		
Resource	link	ktonnes	**150**	**11,401**	**11,551**	-	-	150	1,625	1,775	1,775	1,775	1,775	1,775	901	-	-	-
Waste	link	ktonnes	**803**	**57,603**	**58,406**	-	-	803	7,875	11,725	11,725	11,725	10,001	4,046	506	-	-	-
Total Mined	**calc**	**ktonnes**	**953**	**69,004**	**69,957**	**-**	**-**	**953**	**9,500**	**13,500**	**13,500**	**13,500**	**11,776**	**5,821**	**1,407**	**-**	**-**	**-**
Rehandle	link	ktonnes	**-**	**150**	**150**	-	-	-	150	-	-	-	-	-	-	-	-	-
Strip Ratio	calc	w:o	**5.4**	**5.1**	**5.1**	-	-	5.4	4.8	6.6	6.6	6.6	5.6	2.3	0.6	-	-	-
Mining Rate	calc	ktpd	**2.6**	**32.2**	**31.8**	-	-	2.6	26.0	37.0	37.0	37.0	32.3	15.9	3.9	-	-	-
Head Grades																		
Copper	input	%	**0.867%**	**0.979%**	**0.977%**	-	-	0.867%	0.965%	0.932%	1.019%	0.907%	0.869%	1.065%	1.204%	-	-	-
Gold	input	g/t	**0.31**	**0.44**	**0.43**	-	-	0.31	0.47	0.41	0.49	0.34	0.41	0.46	0.50	-	-	-
Silver	input	g/t	**2.99**	**4.36**	**4.34**	-	-	2.99	4.43	4.26	4.84	3.59	3.91	4.59	5.40	-	-	-
MILL SCHEDULE																		
Mill Production Schedule																		
Resource Processed	link	ktonnes	**-**	**11,551**	**11,551**	-	-	-	1,775	1,775	1,775	1,775	1,775	1,775	901	-	-	-
Total Resource Processed	**calc**	**ktonnes**	**-**	**11,551**	**11,551**	**-**	**-**	**-**	**1,775**	**1,775**	**1,775**	**1,775**	**1,775**	**1,775**	**901**	**-**	**-**	**-**
CONTAINED & RECOVERED METALS																		
Copper																		
Contained	calc	Mlbs	**-**	**248.9**	**248.9**				37.4	36.5	39.9	35.5	34.0	41.7	23.9	-	-	-
Recovery	link	%	**0.0%**	**85.5%**	**85.5%**	-	-	-	85.5%	85.5%	85.5%	85.5%	85.5%	85.5%	85.5%	-	-	-
Recovered	calc	Mlbs	**-**	**212.9**	**212.9**	-	-	-	32.0	31.2	34.1	30.4	29.1	35.6	20.5	-	-	-
Gold																		
Contained	calc	koz	**-**	**161.4**	**161.4**				26.1	23.5	28.0	19.6	23.5	26.4	14.4	-	-	-
Recovery	link	%	**0.0%**	**84.4%**	**84.4%**	-	-	-	84.4%	84.4%	84.4%	84.4%	84.4%	84.4%	84.4%	-	-	-
Recovered	calc	koz	**-**	**136.3**	**136.3**	-	-	-	22.1	19.8	23.6	16.5	19.9	22.3	12.2	-	-	-
Silver																		
Contained	calc	koz	**-**	**1,611.6**	**1,611.6**				245.9	243.1	276.2	204.9	223.1	261.9	156.4	-	-	-
Recovery	link	%	**0.0%**	**9.4%**	**9.4%**	-	-	-	9.4%	9.4%	9.4%	9.4%	9.4%	9.4%	9.4%	-	-	-
Recovered	calc	koz	**-**	**151.2**	**151.2**	-	-	-	23.1	22.8	25.9	19.2	20.9	24.6	14.7	-	-	-
PAYABLE METALS																		
Copper																		
Payable Copper Cathode	link	%	**100%**	**100%**	**100%**	-	-	-	100%	100%	100%	100%	100%	100%	100%	-	-	-
	calc	Mlbs	**-**	**212.9**	**212.9**	-	-	-	32.0	31.2	34.1	30.4	29.1	35.6	20.5	-	-	-
	calc	US$M	**-**	**532.2**	**532.2**	-	-	-	80.1	78.0	85.3	75.9	72.7	89.1	51.1	-	-	-
	calc	C$M	**-**	**682.3**	**682.3**	-	-	-	102.6	100.0	109.3	97.3	93.2	114.2	65.6	-	-	-
Gold																		
Payable Au	link	%	**100%**	**100%**	**100%**	-	-	-	100%	100%	100%	100%	100%	100%	100%	-	-	-
	calc	koz	**-**	**136.3**	**136.3**	-	-	-	22.1	19.8	23.6	16.5	19.9	22.3	12.2	-	-	-
	calc	US$M	**-**	**177.2**	**177.2**	-	-	-	28.7	25.7	30.7	21.5	25.8	28.9	15.8	-	-	-
	calc	C$M	**-**	**227.2**	**227.2**	-	-	-	36.8	33.0	39.4	27.5	33.1	37.1	20.3	-	-	-
Silver																		
Payable Ag	link	%	**100%**	**100%**	**100%**	-	-	-	100%	100%	100%	100%	100%	100%	100%	-	-	-
	calc	koz	**-**	**151.2**	**151.2**	-	-	-	23.1	22.8	25.9	19.2	20.9	24.6	14.7	-	-	-
	calc	US$M	**-**	**2.6**	**2.6**	-	-	-	0.4	0.4	0.5	0.3	0.4	0.4	0.3	-	-	-
	calc	C$M	**-**	**3.4**	**3.4**	-	-	-	0.5	0.5	0.6	0.4	0.5	0.6	0.3	-	-	-
TOTAL REVENUE																		
Total Gross Revenue	calc	C$M	**-**	**912.8**	**912.8**	-	-	-	139.9	133.5	149.2	125.3	126.8	151.9	86.2	-	-	-
TC/RC/ROYALTIES																		
Copper																		
Copper Cathode Premium	input	US$/lb cont	**-**	**-**	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-
	calc	C$M	**-**	**-**	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-
Copper Cathode Shipping Charge	link	US$/lb	**0.015**	**0.015**	**0.015**	0.015	0.015	0.015	0.015	0.015	0.015	0.015	0.015	0.015	0.015	0.015	0.015	0.015
	calc	C$M	**-**	**4.1**	**4.1**	-	-	-	0.6	0.6	0.7	0.6	0.6	0.7	0.4	-	-	-
Gold																		
Au Refining Costs	link	US$/payable oz	**4.00**	**4.00**	**4.00**	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00
	calc	C$M	**-**	**0.7**	**0.7**	-	-	-	0.1	0.1	0.1	0.1	0.1	0.1	0.1	-	-	-
Silver																		
Ag Refining Costs	link	US$/payable oz	**0.40**	**0.40**	**0.40**	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40
	calc	C$M	**-**	**0.1**	**0.1**	-	-	-	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-	-	-
Royalties																		
Archer Cathro Royalty	calc	C$M	**0.3**	**0.9**	**1.2**	0.1	0.1	0.1	0.9	-	-	-	-	-	-	-	-	-
TOTAL																		
Total Charges + Royalties	**calc**	**C$M**	**0.3**	**5.8**	**6.1**	**0.1**	**0.1**	**0.1**	**1.6**	**0.7**	**0.8**	**0.7**	**0.7**	**0.8**	**0.5**	**-**	**-**	**-**

Item	Source	Unit	PP	Production	LOM	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
NSR																		
NSR																		
Net Smelter Return	**calc**	**C$M**	**(0.3)**	**907.0**	**906.7**	**(0.1)**	**(0.1)**	**(0.1)**	**138.3**	**132.8**	**148.5**	**124.6**	**126.1**	**151.1**	**85.7**	**-**	**-**	**-**
OPEX																		
Mining	calc	C$/t processed	-	15.73	15.73	-	-	-	15.43	18.16	18.78	18.60	16.71	11.13	6.97	-	-	-
	link	C$M	-	181.7	181.7	-	-	-	27.4	32.2	33.3	33.0	29.7	19.8	6.3	-	-	-
Processing	link	C$/t processed	-	23.27	23.27	-	-	-	23.27	23.27	23.27	23.27	23.27	23.27	23.27	-	-	-
	calc	C$M	-	268.8	268.8	-	-	-	41.3	41.3	41.3	41.3	41.3	41.3	21.0	-	-	-
G&A	calc	C$/t processed	-	6.45	6.45	-	-	-	7.06	6.23	6.23	6.26	6.26	6.16	8.60	-	-	-
	link	C$M	-	74.5	74.5				11.5	11.1	11.1	11.1	11.1	10.9	7.7	-	-	-
Unit Operating Cost	**calc**	**C$/t processed**	**-**	**45.45**	**45.45**	**-**	**-**	**-**	**49.32**	**47.66**	**48.28**	**48.13**	**46.24**	**40.56**	**38.84**	**-**	**-**	**-**
Total OPEX	**calc**	**C$M**	**-**	**524.9**	**524.9**	**-**	**-**	**-**	**80.1**	**84.6**	**85.7**	**85.4**	**82.1**	**72.0**	**35.0**	**-**	**-**	**-**
Copper Cash Costs (Net of By-products)	**calc**	**C$/payable lb Cu**	**-**	**1.38**	**1.38**	**-**	**-**	**-**	**1.34**	**1.64**	**1.34**	**1.89**	**1.67**	**0.96**	**0.70**	**-**	**-**	**-**
	calc	**US$/payable lb Cu**	**-**	**1.08**	**1.08**	**-**	**-**	**-**	**1.04**	**1.28**	**1.05**	**1.48**	**1.30**	**0.75**	**0.55**	**-**	**-**	**-**
Net Operating Cashflow	**calc**	**C$M**	**(0.3)**	**382.1**	**381.8**	**(0.1)**	**(0.1)**	**(0.1)**	**58.1**	**48.2**	**62.8**	**39.2**	**44.0**	**79.1**	**50.7**	**-**	**-**	**-**
CAPEX																		
Mining	link	C$M	**9.9**	**3.0**	**12.9**	-	-	9.9	2.4	0.3	0.3	-	-	-	-	-	-	-
Site Development and Earthworks	link	C$M	**10.6**	**6.9**	**17.5**	-	5.7	5.0	2.9	0.7	0.7	0.7	0.7	0.7	0.7	-	-	-
Ore Crushing and Handling	link	C$M	**3.0**	**-**	**3.0**	-	0.6	2.4	-	-	-	-	-	-	-	-	-	-
Process Plant	link	C$M	**129.2**	**1.9**	**131.1**	-	25.8	103.4	0.3	0.3	0.3	0.3	0.3	0.3	-	-	-	-
On-Site Infrastructure	link	C$M	**15.0**	**1.8**	**16.8**	1.3	13.8	-	-	-	1.8	-	-	-	-	-	-	-
Off-Site Infrastructure	link	C$M	**7.3**	**-**	**7.3**	1.3	6.1	-	-	-	-	-	-	-	-	-	-	-
Project Indirects	link	C$M	**8.3**	**1.3**	**9.6**	-	3.2	5.2	1.0	0.1	0.1	-	-	-	-	-	-	-
EPCM	link	C$M	**16.8**	**-**	**16.8**	1.6	11.7	3.5	-	-	-	-	-	-	-	-	-	-
Owner Costs	link	C$M	**14.4**	**-**	**14.4**	-	2.8	11.6	-	-	-	-	-	-	-	-	-	-
Closure	link	C$M	**-**	**5.6**	**5.6**	-	-	-	-	0.1	0.1	0.1	0.1	0.1	0.1	2.5	1.3	1.3
Subtotal	**calc**	**C$M**	**214.7**	**20.5**	**235.2**	**4.1**	**69.6**	**141.0**	**6.6**	**1.5**	**3.3**	**1.1**	**1.1**	**1.1**	**0.8**	**2.5**	**1.3**	**1.3**
Contingency	link	C$M	**25.9**	**2.5**	**28.4**	0.6	8.8	16.5	0.6	0.2	0.4	0.2	0.2	0.2	0.1	0.4	0.2	0.2
CAPEX incl. Contingency	**calc**	**C$M**	**240.6**	**23.0**	**263.6**	**4.7**	**78.4**	**157.5**	**7.2**	**1.7**	**3.8**	**1.2**	**1.2**	**1.2**	**0.9**	**2.9**	**1.4**	**1.4**
CAPEX Breakdown																		
Pre-Production	*calc*	*C$M*	**240.6**	**-**	**240.6**	*4.7*	*78.4*	*157.5*										
Sustaining & Closure	*calc*	*C$M*	**-**	**23.0**	**23.0**				*7.2*	*1.7*	*3.8*	*1.2*	*1.2*	*1.2*	*0.9*	*2.9*	*1.4*	*1.4*
WORKING CAPITAL																		
Working Capital																		
Working Capital	*calc*	C$M	**10.0**	**(10.0)**	**-**			*10.0*							*(10.0)*			
TAXES																		
Taxes																		
Income Taxes	link	C$M	**-**	**33.8**	**33.8**	-	-	-	-	-	-	-	-	20.1	13.7	-	-	-
Yukon Mining Royalty	link	C$M	**-**	**9.1**	**9.1**	-	-	-	1.4	0.6	1.7	0.0	0.3	3.4	1.7	-	-	-
Total Taxes	**link**	**C$M**	**-**	**43.0**	**43.0**	**-**	**-**	**-**	**1.4**	**0.6**	**1.7**	**0.0**	**0.3**	**23.5**	**15.4**	**-**	**-**	**-**
CASH FLOWS																		
Pre-Tax																		
Net Cashflow	**calc**	**C$M**	**(250.9)**	**369.1**	**118.2**	**(4.8)**	**(78.5)**	**(167.6)**	**51.0**	**46.5**	**59.0**	**37.9**	**42.8**	**77.9**	**59.8**	**(2.9)**	**(1.4)**	**(1.4)**
Cumulative Net Cashflow	calc	C$M				(4.8)	(83.3)	(250.9)	(200.0)	(153.5)	(94.5)	(56.6)	(13.8)	64.1	123.9	121.1	119.6	118.2
Net Cashflow	**calc**	**US$M**	**(195.7)**	**287.9**	**92.2**	**(3.7)**	**(61.3)**	**(130.7)**	**39.8**	**36.2**	**46.0**	**29.6**	**33.4**	**60.7**	**46.7**	**(2.2)**	**(1.1)**	**(1.1)**
Cumulative Net Cashflow	calc	US$M				(3.7)	(65.0)	(195.7)	(156.0)	(119.7)	(73.7)	(44.1)	(10.7)	50.0	96.7	94.4	93.3	92.2
After-Tax																		
Net Cashflow	**calc**	**C$M**	**(250.9)**	**326.1**	**75.2**	**(4.8)**	**(78.5)**	**(167.6)**	**49.5**	**45.9**	**57.3**	**37.9**	**42.5**	**54.3**	**44.4**	**(2.9)**	**(1.4)**	**(1.4)**
Cumulative Net Cashflow	calc	C$M				(4.8)	(83.3)	(250.9)	(201.4)	(155.5)	(98.2)	(60.3)	(17.8)	36.5	81.0	78.1	76.6	75.2
Net Cashflow	**calc**	**US$M**	**(195.7)**	**254.4**	**58.7**	**(3.7)**	**(61.3)**	**(130.7)**	**38.6**	**35.8**	**44.7**	**29.6**	**33.2**	**42.4**	**34.6**	**(2.2)**	**(1.1)**	**(1.1)**
Cumulative Net Cashflow	calc	US$M				(3.7)	(65.0)	(195.7)	(157.1)	(121.3)	(76.6)	(47.0)	(13.9)	28.5	63.1	60.9	59.8	58.7
ECONOMIC INDICATORS																		
Pre-Tax																		
Pre-Tax IRR	**calc**	**%**			**9.4%**													
Pre-Tax Payback	**calc**	**Years**			**5.2**													
Pre-Tax NPV @ 8%	**calc**	**C$M**			**11.9**													
	calc	**US$M**			**9.3**													
Pre-Tax NPV @ 0%	**calc**	**C$M**			**118.2**													
	calc	**US$M**			**92.2**													
After-Tax																		
After-Tax IRR	**calc**	**%**			**6.6%**													
After-Tax Payback Period	**calc**	**Years**			**5.3**													
After-Tax NPV @ 8%	**calc**	**C$M**			**11.4**													
	calc	**US$M**			**8.9**													
After-Tax NPV @ 0%	**calc**	**C$M**			**75.2**													
	calc	**US$M**			**58.7**													

24 Adjacent Properties

There are no adjacent operational mining properties that would lead to a better understanding of this property.

25 Other Relevant Data and Information

There is no other relevant data or information relative to the scope of this report.

26 Interpretations and Conclusions

It is the conclusion of the QPs that the PEA summarized in this technical report contains adequate detail and information to support the economic result herein contained. The PEA proposes the use of industry standard equipment and operating practices. To date, the QPs are not aware of any fatal flaws for the project.

Using the assumptions highlighted in this report, the Carmacks project would require improved input parameters such as an extension of the mine life through resource expansion, improvements in silver metal recoveries, capital and/or operating cost reduction, US$:C$ exchange rate improvement, or an increase in gold and copper prices to be advanced to the next stage of study (Preliminary Feasibility Study).

26.1 Risks

As with any proposed mining project, there are risks. The most significant potential risks associated with the Carmacks project are the level of Mineral Resource estimate, level of metallurgical testing and process design, operating and capital cost escalation, permitting and environmental compliance, unforeseen schedule delays, changes in regulatory requirements, retention of mining personnel due to the remote location, the ability to raise financing, and commodity price variability.

These risks are common to most mining projects, many of which may be mitigated, at least to some degree, with adequate engineering, planning and pro-active project management.

Table 26.1 identifies what are currently deemed to be the most significant internal project risks, potential impacts, and possible mitigation approaches.

Table 26.1 Main Project Risks

Risk	Explanation/Potential Impact	Possible Risk Mitigation
Mineral Resource Estimate	All Mineral Resource estimates carry some risk and are one of the most common issues with project success. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.	There are sufficient Measured and Indicated tonnes in Zones 1, 4 and 7 to potentially be considered a Mineral Reserve once the project progresses to the Pre-Feasibility stage.
Metallurgical Recoveries	Changes to metallurgical assumptions could lead to reduced metal recovery, increased processing OPEX costs, and/or changes to the processing circuit design. If LOM metal recovery is lower than assumed, the project economics would be negatively impacted.	Additional sampling and test work optimization to be conducted as part of ongoing studies.
CAPEX and OPEX	The ability to achieve the estimated CAPEX and OPEX costs are important elements of project success. If OPEX increases then the mining cut-off grade would increase and, all else being equal, the open pit recovered tonnage would reduce yielding fewer mineable tonnes.	Further cost estimation accuracy with the next level of study, as well as the active investigation of potential cost reduction measures would assist in the support of reasonable cost estimates.
Permit Acquisition	The ability to secure all of the permits to build and operate the project is of paramount importance. Failure to secure the necessary permits could stop or delay the project.	Continued fostering and further development of close relationship with the local communities. Development of relationships with government along with a thorough Environmental and Social Impact Assessment and a project design that gives appropriate consideration to the environment and local people is required.
Development Schedule	The project development could be delayed for a number of reasons and could impact project economics. A change in schedule would alter the project economics.	If an aggressive schedule is to be followed, early engagement with all stakeholders is important.
Ability to Attract and Retain Experienced Professionals	The ability to attract and retain competent, experienced professionals is a key success factor for the project, particularly due to the remove nature of the project. High turnover or the lack of appropriate technical and management staff at the project could result in difficulties meeting project goals.	The early search for professionals as well as competitive salaries, flexible work schedules and benefits all help to identify, attract and retain critical people,

Source: JDS (2016)

26.2 Opportunities

There are significant opportunities that could improve the economics of the project. The major opportunities that have been identified at this time are summarized in the following sections. Further information and assessments are required before these opportunities could be included in the project economics.

26.2.1 Process

The metallurgical process detailed in this PEA has effectively maximized the recovery of Cu and Au from the oxide material. However, there remains an opportunity to address improvement in the silver recovery, which is only 9.4% based on the metallurgical test work completed to date. This low Ag recovery appears related to changes in the metallurgical process, given that earlier process configurations achieved much higher Ag recovery (61 to 72%, mean 67.7%; Beattie 2015). Increasing silver recovery to 67% has the potential to add another $18.8 M in gross LOM revenue.

There are several opportunities related to further refinement of the metallurgical process to be examined that have the potential to reduce CAPEX and/or OPEX, including:

- Examination of alternative solid/liquid separation technology in the copper circuit and for tailings filtration, with the potential for both CAPEX (equipment cost) and OPEX (energy and reagent consumption reductions);
- Optimization of leach temperature and reagent additions for copper leaching (potential for energy and reagent consumption reductions); and,
- Reagents account for 54% of processing OPEX, and the cyanide destruction reagents account for approximately 28% of that reagent cost. Consideration of alternative methods of cyanide destruction in the final tailings slurry, with a focus on reduction of reagent costs, represents a potentially significant OPEX reduction.

26.2.2 Extend Mine Life

Extension of the mine life beyond 7 years has the potential to provide the single largest increase in NPV of all the opportunities to be examined. Recent exploration drilling in 2014 and 2015 identified additional near-surface oxide Mineral Resources, in Zones 2000S, 12, and 13 (ACS 2016), that remain to be brought into the project plan. These additional resources remain open along strike indicating a realistic potential to add further oxide resources to the project. Additional drilling, metallurgical testing, and mine planning need to be completed to bring these resources into the project plan.

If these hypothetical tonnes were to be brought into the resource model with an extension of the mine life by four years at the LOM average throughput, grade and recoveries there is potential to increase the NPV by approximately $90 M and the IRR by approximately 6%.

26.2.3 Other Potential Opportunities

Other potential opportunities include:

- Mine and plant construction efficiency and timelines;
- Global sourcing of used equipment for operations; and
- Evaluation of processing sulphide Mineral Resource at Carmacks, for mine extension.

27 Recommendations

JDS recommends a staged approached to future work on the Carmacks project due to the risks of the project and the current economic results. Due to the marginal nature of the project at the base case metal prices, it is recommended that CNMC initially investigate improving the project economics through the improvement and opportunities listed in Section 26.2.

The first stage of the recommended work should focus on process and metallurgical improvements. **Error! Reference source not found.** Provides a breakdown of the recommended task and their associated costs.

Table 27.1: Process and Metallurgical Improvement Costs

Item	Cost
Improve Ag recovery, test work to confirm CIL flowsheet, leach testing and carbon loading	$75,000
Optimization of leach temperature and reagent additions(variability and confirmation samples)	$50,000
Examination of alternative solid/liquid separation technology	$50,000
Reagent Optimization	Included Above
Total Estimate	**$175,000**

Source: JDS (2016)

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29 Units of Measure, Abbreviations and Acronyms

Symbol/Abbreviation	Description
‘	Minute (Plane Angle)
“	Second (Plane Angle) or Inches
°	Degree
°C	Degrees Celsius
3D	Three-Dimensions
A	Ampere
a	Annum (Year)
AA	Atomic Absorption
AAS	Atomic Absorption Spectrometry
ac	Acre
ADR	Adsorption-Desorption-Recovery
AES	Atomic Emission Spectroscopy
AI	Abrasion index
ALS	ALS Chemex Ltd
amsl	Above Mean Sea Level
ANFO	Ammonium nitrate fuel oil
ARD	Acid Rock Drainage
Au	Gold
BAT	Best available technology
BD	Bulk Density
BFA	Bench Face Angles
BTU	British Thermal Unit
BV/h	Bed Volumes Per Hour
BWI	Ball mill work index
C$	Dollar (Canadian)
Ca	Calcium
CCD	Counter current decantation
CDP	Cyanide Detoxification Plant
CF	Cumulative Frequency
cfm	Cubic Feet Per Minute
CHP	Combined Heat And Power Plant
CIC	Carbon-In-Column
CIL	Carbon in Leach
CIM	Canadian Institute Of Mining And Metallurgy
cm	Centimetre
CM	Construction Management
cm^2	Square Centimetre
cm^3	Cubic Centimetre
CNMC	Copper North Mining Company

Symbol/Abbreviation	Description
COG	Cut-Off Grades
Cr	Chromium
CSA	Canadian Securities Administrators
CSRM	Certified standard reference materials
Cu	Copper
CV	Coefficient of Variation
d	Day
d/a	Days per Year (Annum)
d/wk	Days per Week
Db	Decibel
dBa	Decibel Adjusted
DCIP	Direct current induced polarization
DCS	Distributed Control System
DDH	Diamond drill holes
DGPS	Differential Global Positioning System
dmt	Dry Metric Ton
DSTSF	Dry Stack Tailings Storage Facility
DTM	Digital terrain model
EA	Environmental Assessment
EDA	Exploratory Data Analysis
ELOS	Equivalent linear over-break/slough
EMR	Energy, Mines and Resources
EP	Engineering and Procurement
EPCM	Engineering, Procurement and Construction Management
FEL	Front-End Loader
FS	Feasibility Study
ft	Foot
ft^2	Square Foot
ft^3	Cubic Foot
ft^3/s	Cubic Feet Per Second
g	Gram
G&A	General and administrative
g/cm^3	Grams Per Cubic Metre
g/L	Grams Per Litre
g/t	Grams Per Tonne
gal	Gallon (Us)
GCL	Geosynthetic clay liner
GJ	Gigajoule
Gpa	Gigapascal
gpm	Gallons Per Minute (US)
GRG	Gravity recoverable gold
GSC	Geological Survey of Canada
GW	Gigawatt

Symbol/Abbreviation	Description
h	Hour
h/a	Hours Per Year
h/d	Hours Per Day
h/wk	Hours Per Week
ha	Hectare (10,000 M2)
HDS	High density sludge
HG	High Grade
HLF	Heap leach facility
HLP	Heap Leaching Pads
HMI	Human Machine Interface
hp	Horsepower
HPGR	High-Pressure Grinding Rolls
HQ	Drill Core Diameter Of 63.5 Mm
HSE	Health, Safety and Environmental
HVAC	Heating, Ventilation, and Air Conditioning
HW	Hanging Wall
Hz	Hertz
IFC	International Finance Corporation
in	Inch
in^2	Square Inch
in^3	Cubic Inch
IP	Internet Protocol
IRR	Internal Rate Of Return
ISQG	Interim Sediment Quality Guidelines
IT	Information technology
IT	Information technology
JDS	JDS Energy and Mining Inc.
K	Hydraulic Conductivity
k	Kilo (Thousand)
KE	Kriging Efficiency
kg	Kilogram
kg/h	Kilograms Per Hour
kg/m^2	Kilograms Per Square Metre
kg/m^3	Kilograms Per Cubic Metre
km	Kilometre
km/h	Kilometres Per Hour
km^2	Square Kilometre
KNA	Kriging Neighbourhood Analysis
kPa	Kilopascal
kt	Kilotonne
Kv	Kilovolt
KV	Kriging Variance
Kva	Kilovolt-Ampere

Symbol/Abbreviation	Description
Kw	Kilowatt
kWh	Kilowatt Hour
kWh/a	Kilowatt Hours Per Year
kWh/t	Kilowatt Hours Per Tonne
L	Litre
L/min	Litres Per Minute
L/s	Litres Per Second
LAN	Local Area Network
LDD	Large-Diameter Drill
LG	Low Grade
LG	Lerchs- Grossman
LH	Long hole
LHD	Load-haul-dump
LOI	Letter of Intent
LOM	Life Of Mine
LSCFN	Little Salmon/Carmacks First Nation
m	Metre
M	Million
m/min	Metres Per Minute
m/s	Metres Per Second
m^2	Square Metre
m^3	Cubic Metre
m^3/h	Cubic Metres Per Hour
m^3/s	Cubic Metres Per Second
Ma	Million Years
mamsl	Metres Above Mean Sea Level
MAP	Mean Annual Precipitation
masl	Metres Above Mean Sea Level
Mb/s	Megabytes Per Second
mbgs	Metres Below Ground Surface
mbs	Metres Below Surface
mbsl	Metres Below Sea Level
MCC	Motor Control Centres
MCF	Mechanized cut and fill
mg	Milligram
mg/L	Milligrams Per Litre
min	Minute (Time)
Ml	Milliliter
Mm^3	Million Cubic Metres
MMER	Metal Mining Effluent Regulations
mo	Month
Mpa	Megapascal
MRE	Mineral Resource Estimate

Symbol/Abbreviation	Description
Mt	Million Metric Tonnes
MVA	Megavolt-Ampere
MW	Megawatt
MWMT	Meteoric Water Mobility Tests
MWTP	Mine Water Treatment Plant
NAD	North American Datum
NG	Normal Grade
Ni	Nickel
NI 43-101	National Instrument 43-101
Nm^3/h	Normal Cubic Metres Per Hour
NPI	Net profits interest
NPV	Net present value
NPVS	NPV Scheduler
NQ	Drill Core Diameter of 47.6 Mm
NRC	Natural Resources Canada
NSR	Net smelter return
OEM	Original Equipment Manufacturers
OIS	Operator Interface Stations
OP	Open Pit
OSA	Overall Slope Angles
oz	Troy Ounce
P.Geo.	Professional Geoscientist
Pa	Pascal
PAG	Potential acid generating
PAX	Potassium Amyl Xanthate
PEA	Preliminary Economic Assessment
PEP	Project Execution Plan
PFS	Preliminary Feasibility Study
PLC	Programmable Logic Controller
PLS	Pregnant Leach Solution
PMF	Probable Maximum Flood
ppb	Parts Per Billion
PPE	Protective personal equipment
ppm	Parts Per Million
psi	Pounds Per Square Inch
PTS	Passive treatment
QA	Quality Assurance
QA/QC	Quality Assurance/Quality Control
QC	Quality Control
QKNA	Qualitative Kriging Neighbourhood Analysis
QMA	Quartz Mining Act
QML	Quartz Mining License
QMS	Quality Management System

Symbol/Abbreviation	Description
QP	Qualified Person
QQ	Quartile-Quartile
RAB	Rotary air blast
RC	Reverse Circulation
RCD	Reverse circulation drilling
RDI	Resource Development Inc
RMR	Rock Mass Rating
ROM	Run-Of-Mine
rpm	Revolutions Per Minute
RQD	Rock quality designation
RWI	Rod mill work index
s	Second (Time)
S.G.	Specific Gravity
SARA	Species At Risk Act
SART	Sulphidization, acidification, recycling and thickening
Scfm	Standard Cubic Feet Per Minute
SD	Standard deviations
SEDEX	Sedimentary Exhalative
SFN	Selkirk First Nation
SG	Specific Gravity
SIA	Socio-economic Impact Assessment
SMR	South Mcquesten Road
SPMDD	Standard Proctor Maximum Dry Density
SRM	Standards reference material
SSWQO	Site Specific Water Quality Objectives
SVOL	Search Volume
t	Tonne (1,000 Kg) (Metric Ton)
t/a	Tonnes Per Year
t/d	Tonnes Per Day
t/h	Tonnes Per Hour
TCR	Total Core Recovery
TMA	Tailings management area
TMASP	Tailings management area sediment pond
tph	Tonnes Per Hour
ts/hm^3	Tonnes Seconds Per Hour Metre Cubed
TSF	Tailings storage facility
TSS	Total Suspended Solids
UCS	Uniaxial compression
US	United States
US$	Dollar (American)
UTM	Universal Transverse Mercator
V	Volt
VEC	Valued Ecosystem Components
VoIP	Voice Over Internet Protocol

Symbol/Abbreviation	Description
VSAT	Very Small Aperture Terminal
VSEC	Valued Socio-Economic Components
w/w	Weight/Weight
WAD	Weak-Acid-Dissociable
WBS	Work Breakdown Structure
wk	Week
wmt	Wet Metric Ton
WRS	Waste Rock Stockpile
WRSA	Waste Rock Storage Area
WRSASP	Waste Rock Storage Area Sediment Pond
WUL	Water Use License
XRF	X-ray fluorescence
Y&WPR	Yukon & White Pass Route
YESAA	Yukon Environmental and Socio-Economic Assessment Act
YESAB	Yukon Environmental and Socio-Economic Assessment Board
YG	Yukon Government
µm	Microns

APPENDIX A – QP CERTIFICATES

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CERTIFICATE OF AUTHOR

I, Gordon Doerksen, P.Eng., do hereby certify that:

1. I am currently employed as V.P. Technical Services with JDS Energy & Mining Inc. with an office at Suite 900-999 West Hastings Street, Vancouver, BC, V6C 2W2;

2. This certificate applies to the technical report titled "NI 43-101 Preliminary Economic Assessment Technical Report on the Carmacks Project, Whitehorse, Yukon, Canada", with an effective date of October 12, 2016, (the "Technical Report") prepared for Copper North Mining Corp. ("the Issuer");

3. I am a Professional Mining Engineer (P.Eng. #32273) registered with the Association of Professional Engineers, Geologists of British Columbia. I am also a registered Professional Mining Engineer in Yukon Territory. I am a Member of the Canadian Institute of Mining and Metallurgy and a Registered Member of the Society of Mining Engineers of the AIME.

   I am a graduate of Montana Tech with a B.Sc. in Mining Engineering (1990). I have been involved in mining since 1985 and have practiced my profession continuously since 1990. I have held senior mine production and mine technical positions in mining operations in Canada, the US and in Africa. I have worked as a consultant for over eight years and have performed mine planning, project management, cost estimation, scheduling and economic analysis work, as a Qualified Person, for a significant number of engineering studies and technical reports many of which were located in Latin America.
   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 with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4. I have visited the Carmacks Project site on July 26, 2016;

5. I am responsible for Section numbers 1, 2, 3, 15, 18, 19, 20, 21, 22, 23, 24, 25, 26, 27, 28 and 29 of the Technical Report;

6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have had no prior involvement with the property that is the subject of the Technical Report;

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;

9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective Date: October 12, 2016
Signing Date: November 25, 2016

(original signed and sealed) "Gordon Doerksen, P.Eng."

Gordon Doerksen, P.Eng.

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CERTIFICATE OF AUTHOR

I, Kelly S. McLeod, P. Eng., do hereby certify that:

1. I am currently employed as a Senior Engineer, Metallurgy, with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;
2. This certificate applies to the technical report titled "NI 43-101 Preliminary Economic Assessment Technical Report on the Carmacks Project, Whitehorse, Yukon, Canada", with an effective date of October 12, 2016, (the "Technical Report") prepared for Copper North Mining Corp. ("the Issuer");
3. I am a graduate of McMaster University with a Bachelor's of Engineering, Metallurgy, 1984. I have practiced my profession intermittently since 1984;
4. I am a Professional Metallurgical Engineer (P.Eng. #15868) registered with the Association of Professional Engineers, Geologists of British Columbia;
5. 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 with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;
6. I did not visited the Carmacks Project site;
7. I am responsible for Section 17 of this Technical Report;
8. I have had no prior involvement with the property that is the subject of this Technical Report;
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;
10. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: October 12, 2016
Signing Date: November 25, 2016

(original signed and sealed) "Kelly S. McLeod, P.Eng."

Kelly S. McLeod, P. Eng.

ARSENEAU Consulting Services

Certificate of Qualified Person

I, Dr. Gilles Arseneau, P. Geo., do hereby certify that:

1. I am President of ARSENEAU Consulting Services Inc. ("**ACS**"), a corporation with a business address of Suite 900, 999 West Hastings Street, Vancouver, British Columbia, Canada.

2. I am co-author of a technical report entitled "NI 43-101 Preliminary Economic Assessment Technical Report on the Carmacks Project, Whitehorse, Yukon, Canada", with an effective date of October 12, 2016, (the "Technical Report") prepared for Copper North Mining Corp. ("the Issuer").

3. I am a graduate of the University of New Brunswick with a B.Sc. (Geology) degree obtained in 1979, the University of Western Ontario with an M.Sc. (Geology) degree obtained in 1984 and the Colorado School of Mines with a Ph.D. (Geology) obtained in 1995.

4. I have practiced my profession continuously since 1995. I have worked in exploration in North and South America and have extensive experience modelling copper mineralization similar to the Carmacks deposits.

5. I am Professional Geoscientist registered as a member, in good standing, with the Association of Professional Engineers & Geoscientists of British Columbia (no. 23474).

6. I have read the definition of "qualified person" set out in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* ("**NI 43-101**") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" within the meaning of NI 43-101**.**

7. My most recent personal inspection of the Carmacks copper project occurred on October 14, 2015.

8. I am responsible for sections 4 to 12 and Section 14 of the Technical Report and accept professional responsibility for the Technical Report.

9. I am independent of the Issuer as defined in Section 1.5 of NI 43-101.

10. I have had prior involvement with the Carmacks copper project. I am the author of technical reports for the property as stated in the reference section of this Technical Report.

11. I have read NI 43-101, Form 43-101F1 and the Technical Report, which have been prepared in compliance with that instrument and form.

12. As of the effective date of the Technical Report, 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.

Dated this 25 day of November, 2016 in Vancouver, British Columbia.

["Original signed and sealed"]

Dr. Gilles Arseneau, P. Geo.

CERTIFICATE OF QUALIFIED PERSON

I, David Dreisinger, P.Eng., do hereby certify that:

1. I am the President of Dreisinger Consulting Inc. with a business office at 5233 Bentley Crescent, Delta, British Columbia. I am a graduate of Queen's University in Kingston, Canada with a B.Sc. (Metallurgical Engineering, 1980) and a Ph.D. (Metallurgical Engineering, 1984). I am a Fellow of the Canadian Institute of Mining, Metallurgy and Petroleum. I am a Fellow of the Canadian Academy of Engineering;
2. This certificate applies to the technical report titled "NI 43-101 Preliminary Economic Assessment Technical Report on the Carmacks Project, Whitehorse, Yukon, Canada", with an effective date of October 12, 2016, (the "Technical Report") prepared for Copper North Mining Corp. ("the Issuer");
3. I have practiced my profession continuously since graduation. I have been employed in research and teaching at the University of British Columbia since 1984 and currently hold the title of Professor and Chairholder, Industrial Research Chair in Hydrometallurgy in the Department of Materials Engineering. I have provided consulting services to the global metallurgical industry since 1987. I have been the President of Dreisinger Consulting Inc. since 1998;
4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (Registration Number 15803, May 6, 1987);
5. As a result of my experience and qualifications, I am a Qualified Person as defined under National Instrument 43-101;
6. I have made visits to the Bureau Veritas Laboratory in Richmond, B.C. to oversee metallurgical testing related to the Carmacks Project in 2016;
7. I am responsible for section 13 of the technical report;
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;
9. I have had no prior involvement with the property that is the subject of the Technical Report;
10. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
11. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective Date: October 12, 2016
Signing Date: November 25, 2016
(original signed and sealed) "David Dreisinger, P.Eng."

David Dreisinger, P.Eng.

INDEPENDENT
MINING CONSULTANTS, INC.

3560 E. Gas Road
Tucson, Arizona 85714 USA
Tel: (520) 294-9861 Fax: (520) 294-9865

CERTIFICATE OF AUTHOR

I, Michael G. Hester, do hereby certify that:

1. I am currently employed as Vice President and Principal Mining Engineer by Independent Mining Consultants, Inc. ("IMC") of 3560 East Gas Road, Tucson, Arizona, 85714, USA, phone number (520) 294-9861.

2. This certificate applies to the technical report titled "NI 43-101 Preliminary Economic Assessment Technical Report on the Carmacks Project, Whitehorse, Yukon, Canada", with an effective date of October 12, 2016, (the "Technical Report") prepared for Copper North Mining Corp. ("the issuer").

3. I hold the following academic qualifications:

B.S. (Mining Engineering)	University of Arizona	1979
M.S. (Mining Engineering)	University of Arizona	1982

4. I am a Fellow of the Australian Institute of Mining and Metallurgy (FAusIMM #221108), a professional association as defined by National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101"). As well, I am a member in good standing of the following technical associations and societies:

   Society for Mining, Metallurgy, and Exploration, Inc. (SME Member # 1423200)

   Member of Resources and Reserves Committee of the Society of Mining, Metallurgy, and Exploration.

   The Canadian Institute of Mining, Metallurgy and Petroleum (CIM Member #100809)

5. I have worked in the minerals industry as an engineer continuously since 1979, a period of 37 years. I am a founding partner, Vice President, and Principal Mining Engineer for Independent Mining Consultants, Inc. ("IMC"), a position I have held since 1983. I have been employed as an Adjunct Lecturer at the University of Arizona (1997-1998) where I taught classes in open pit mine planning and mine economic analysis. I am also a member of the Resources and Reserves Committee of the Society of Mining, Metallurgy, and Exploration since March 2012. I was employed as a staff engineer for Pincock, Allen & Holt, Inc. from 1979 to 1983. During my career I have had extensive experience reviewing and auditing deposit sampling methods, analytical procedures, and QA/QC analysis. I also have many years of experience developing mineral resource models, developing open pit mine plans and production schedules, calculating equipment requirements for open pit mining operations, developing mine capital and operating cost estimates, performing economic analysis of mining operations and managing various PEA, Pre-Feasibility, and Feasibility Studies.

6. I have read the definition of "Qualified Person" ("QP") set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

7. I visited the Carmacks site on May 16-17, 2007.

8. I am the responsible for Section 16 of the Technical Report.

9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

10. My prior involvement with the property includes work on the May 2007 and October 2012 Technical Reports for Western Copper and Copper North respectively.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. As of the effective date of the Technical Report and the date of the this certificate, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: October 12, 2016
Signing Date: November 25, 2016

(original signed) "Michael G. Hester"

<u>Michael G. Hester, FAusIMM_</u>

Golder Associates

November 24, 2016 Project No. 1655978

CERTIFICATE OF AUTHOR

I, David Anstey, P.Eng., do hereby certify that:

1) I am currently employed as Associate, Senior Tailings Engineer with Golder Associates Ltd. with an office at 102, 2535 3rd Avenue SE, Calgary, Alberta, T2A 7W5.

2) This certificate applies to the technical report titled "NI 43-101 Preliminary Economic Assessment Technical Report on the Carmacks Project, Whitehorse, Yukon, Canada", with an effective date of October 12, 2016, (the "Technical Report") prepared for Copper North Mining Corp. ("the Issuer").

3) I am a registered Professional Engineer in Yukon Territory. I am also a Professional Engineer (P.Eng. # 237989) registered with the Association of Professional Engineers and Geoscientists of Alberta.

   I am a graduate of University of Western Australia with a BEng Civil (2000) and BCom (2000). I have worked continuously as a consultant to the mining industry since 2002. In this role, I have performed investigation, design, analysis, permitting, construction and operational review of tailings storage facilities and mine water management facilities. I have worked on projects for conventional, paste thickened, co-mingled, filtered and in-line polymer treated tailings; across eleven different commodities; and more than nine countries.

   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 with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4) I have visited the Carmacks Project site on July 12, 2016.

5) I am responsible for Section number 18.5.1 of the Technical Report.

6) I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7) My prior involvement with the property was to oversee tailings management studies commencing in 2015 and to carry out the 2016 Engineering Inspection.

8) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9) As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: October 12, 2016
Signing Date: November 24, 2016

GOLDER ASSOCIATES LTD.

PROFESSIONAL
11/24/16
YUKON
DAVID ANSTEY
TERRITORY
ENGINEER

David Anstey, P.Eng.
Associate, Senior Tailings Engineer

BD/DRA

https://capws.golder.com/sites/p1655978tailingsprefeasibilityandfeasiblity/technical/pea review/november 23 review/1655978 certificate of author_dra.docx

November 24, 2016 Project No. 1655978

CERTIFICATE OF AUTHOR

I, Fiona Esford, P.Eng., do hereby certify that:

1) I am currently employed as a Geotechnical Engineer with Golder Associates Ltd, in the office located at 2920 Virtual Way, Suite 200, Vancouver, British Columbia, V5M 0C4.

2) This certificate applies to the technical report titled "NI 43-101 Preliminary Economic Assessment Technical Report on the Carmacks Project, Whitehorse, Yukon, Canada", with an effective date of October 12, 2016, (the "Technical Report") prepared for Copper North Mining Corp. ("the Issuer").

3) I am a Professional Engineer registered with the Association of Professional Engineers of Yukon. I am a Member of the Canadian Institute of Mining and Metallurgy and Canadian Dam Association.

   I am a graduate of University of Waterloo with a B.A.Sc. degree in Geological Engineering (1995), and a graduate of the University of British Columbia with an M.A.Sc. degree in Geotechnical Engineering (2002). I have 19 years of engineering experience in the field of geotechnical and environmental engineering. My work has been carried out in Canada and internationally, including Australia, South America, Mexico, and the United States. I currently fulfill the role of engineer of record for several dams and tailings facilities. My experience has included design of tailings dams and dike design entailing: site selection evaluation, conducting geotechnical investigations, design and analysis, material selection, geomembrane selection, preparation of construction drawings, specifications and bid packages, quality assurance monitoring, as built reporting, preparation of OMS manuals and emergency response plans, and performing annual inspections. Experience has also involved heap leach pad design entailing: site characterization and selection, heap layout and design for pad and valley fills, overseeing laboratory testing programs, liner system design, piping layout, and preliminary stacking plan development. Additional work has included, conducting back-analyses for slope failures, conducting investigations and assessment to determine mechanisms of landslide failures, and participating in a multi-disciplined team assessing a case of mine subsidence.

   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 with a professional association (as defined in NI 43-101) and relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4) I have visited the Carmacks Project site on July 8, 2010.

5) I am responsible for Section number 18.5.2 of the Technical Report.

6) I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7) I began to provide consulting engineering services for the Carmacks Copper Project in 2006 through 2010 including aspects of the heap leach facility design, waste rock storage area design, site investigations, water balance and water management work, geochemical testing and laboratory testing, conducting annual inspections and provide support to Western Copper Corporation for permitting. I have resumed working in a senior review capacity for work carried out by other Golder Associates Ltd. representatives for the Carmacks Copper Project in 2015 and 2016 for the re-design of the tailings management area.

8) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9) As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: October 12, 2016
Signing Date: November 24, 2016

GOLDER ASSOCIATES LTD.

Fiona Esford

ORIGINAL COPY STAMPED

Fiona Esford, P.Eng.
Associate, Geotechnical Engineer

BD/FE

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