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October 31, 2012

Carmacks Copper Project

NI 43-101 Technical Report
Feasibility Study
Volume I
Yukon Territory, Canada

REVISION: 0

Prepared For:

COPPER NORTH MINING CORP.

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DATE AND SIGNATURES PAGE

See Appendix A, Feasibility Study Contributors and Professional Qualifications, for Certificates of qualified persons. These Certificates are considered the date and signature of this report in accordance with Form 43-101F1.

The qualified persons contributing to this Technical Report are:

- Conrad E. Huss, P.E., Ph.D.; Senior Vice President and Chairman of the Board of M3 Engineering and Technology Corporation (M3); principal author of this technical report and responsible for sections 1 through 6 and sections 21 through 27.
- Thomas L. Drielick, P.E.; Senior Vice President of M3 Engineering and Technology Corporation (M3); responsible for section 13 and section 17 of this report.
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- Michael G. Hester, FAusIMM; Vice President of Independent Mining Consultants Inc.; responsible for section 15 and section 16.
- Gilles Arseneau, Ph.D., P.Geo.; Independent Geological Consultant; responsible for sections 7 through 12 and section 14

This report is current as of October 31, 2012.

(Signed) *"Conrad E. Huss"*	October 31, 2012
Conrad E. Huss, P.E., Ph.D	Date
(Signed) *"Thomas L. Drielick"*	October 31, 2012
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1 SUMMARY

This report has been prepared in accordance with the Canadian Standard NI 43-101. This technical report is based on the M3 Engineering & Technology Corporation (M3) study, "Carmacks Copper Project, Copper Mine and Process Plant, Feasibility Study," dated October 2012.

This feasibility study has been prepared by M3 to provide Copper North Mining Corp. and its wholly-owned subsidiary, Carmacks Mining Corp., with an up-to-date development plan, capital and operating cost estimate, and financial analysis for the Carmacks Copper Project. The study builds on the work done previously on this project by several other consultants on behalf of Copper North Mining Corp., its subsidiaries, and its predecessors (collectively referred to as CNMC) going back to the early 1990's. The study sets forth M3's conclusions and recommendations based on M3's experience and knowledge in the development of copper heap leach projects.

This study is based on end-of-September, 2012, technical and commercial parameters. All costs and financial analyses are in Canadian Dollars.

1.1 PROPERTY DESCRIPTION AND OWNERSHIP

On October 17, 2011, Western Copper Corp. (Western Copper) completed a plan of arrangement (the Arrangement) involving Western Copper and two of its subsidiaries formed on August 3, 2011 for the purposes of the Arrangement: Copper North Mining Corp. (together with its subsidiaries), and NorthIsle Copper and Gold Inc. (NorthIsle). Pursuant to the Arrangement, Western Copper transferred 100% interest in the Carmacks Copper Project, 100% interest in the Redstone Property, and $2 million in cash to CNMC and the Island Copper Property and $2.5 million in cash to NorthIsle in consideration for common shares of each respective company. Western Copper then changed its name to Western Copper and Gold Corp. (Western) and distributed the common shares of CNMC and NorthIsle to Western shareholders. Following the distribution, CNMC became a separate company, distinct from Western Copper.

Also pursuant to the Arrangement, the claims and leases that comprise the Carmacks Copper Project were transferred to, and are now held directly by Carmacks Mining Corp., a wholly-owned subsidiary of Copper North Mining Corp.

The Carmacks Copper Project is located in the Dawson Range at latitude 62°-21'N and longitude 136° - 41'W, 192 km north of Whitehorse, Yukon. 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 provincial scale. Figure 1-2 shows the location on a smaller scale.

The project site is located in the Whitehorse mining division of the Yukon and consists of 318 quartz claims, quartz claim fractions, 20 quartz leases, and quartz lease fractions as shown on Figure 1-3. This map was adapted from the Yukon government's mining recorders' website. The claims owned by Carmacks Mining Corp. are outlined in red.

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.

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. 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 Carmacks Mining Corp. on this project are overlain by crown land.

The property lies near, but does not encroach on, LSC R-9A, First Nations Surveyed Lands, Class A Land Reserve. Both Little Salmon Carmacks First Nation (LSFN) and Selkirk First Nation (SFN) consider the project area to be within their “traditional” territory.

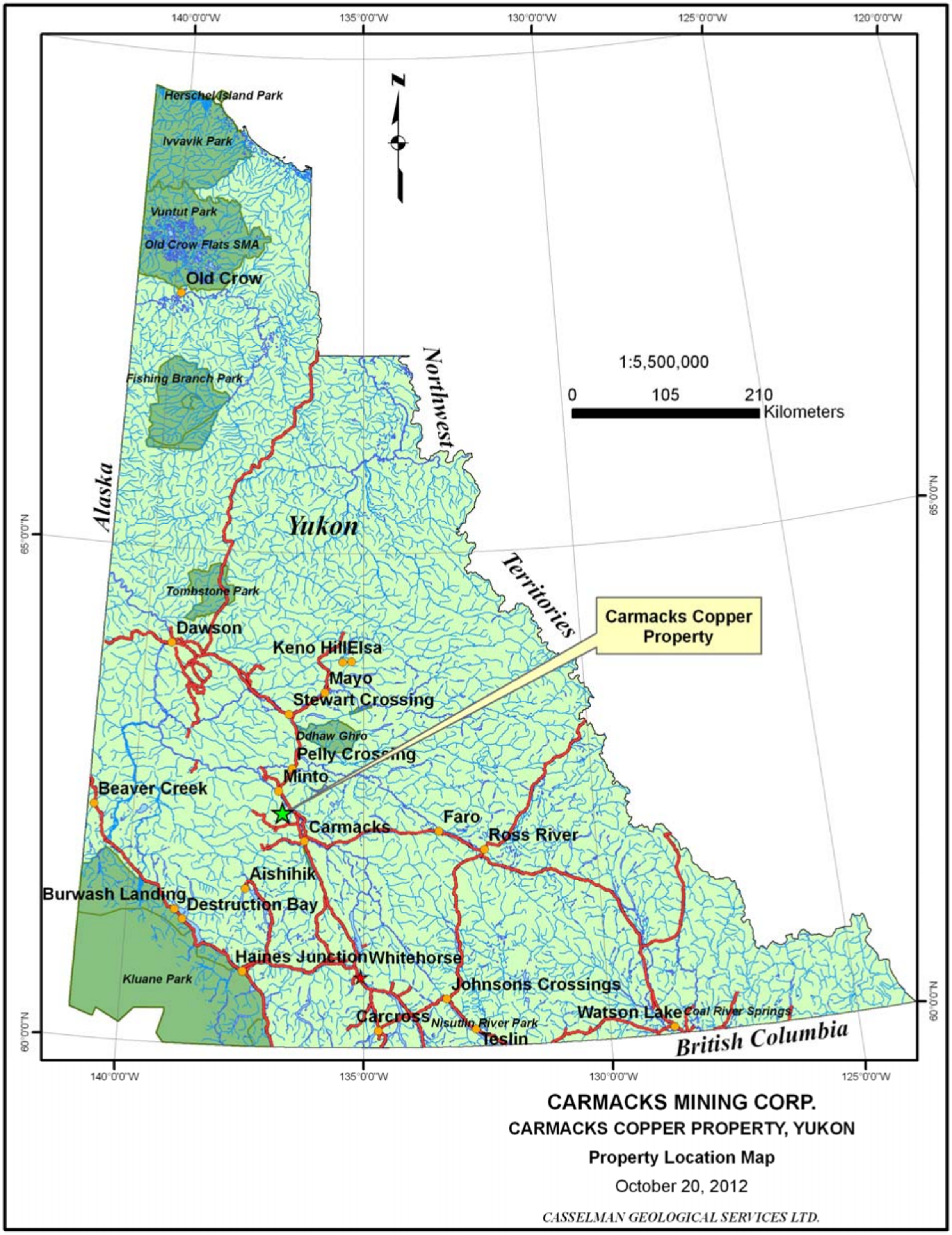


Figure 1-1: Project Location on a Provincial Scale

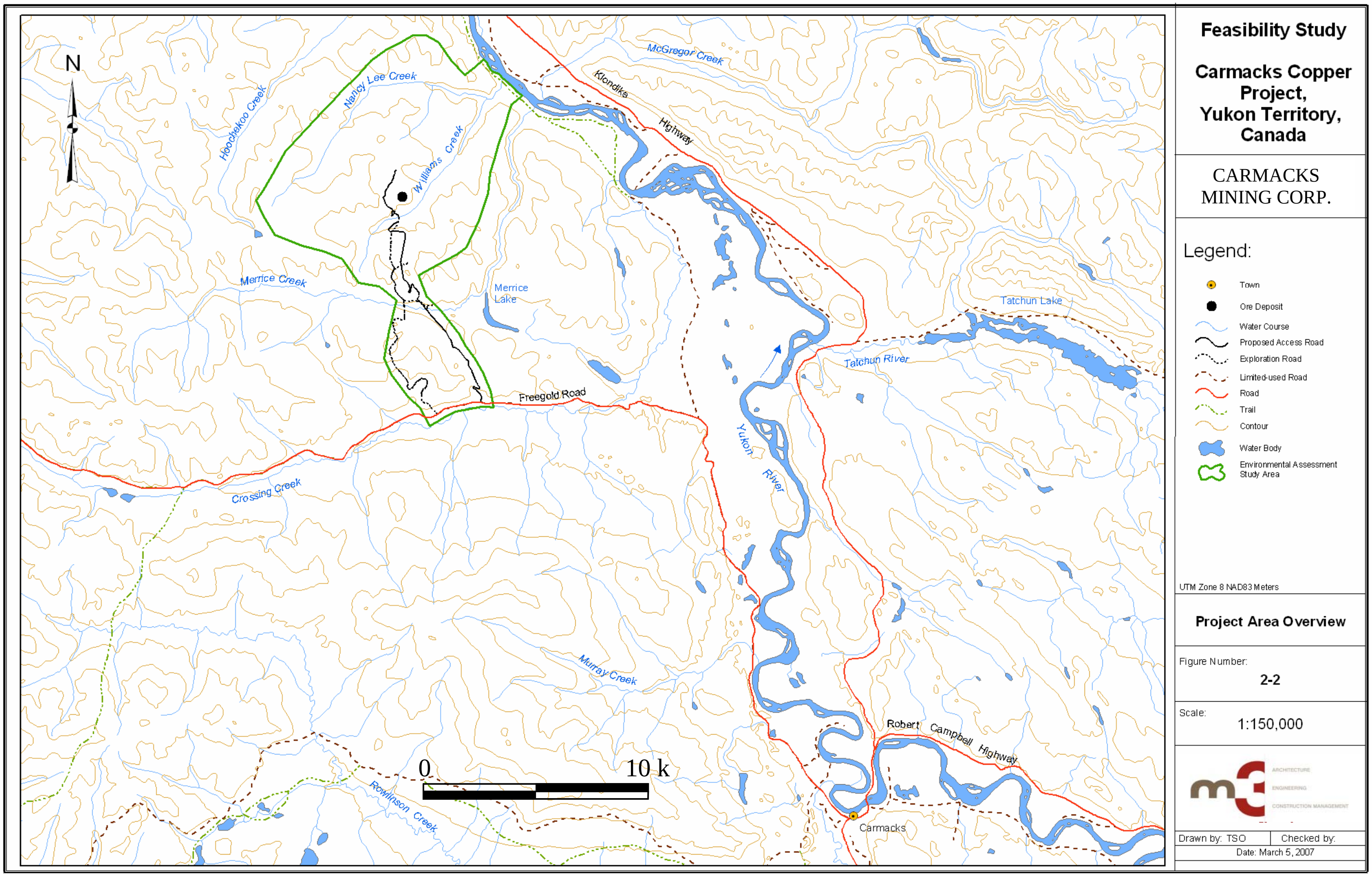


Figure 1-2: Project Location on a Local Scale

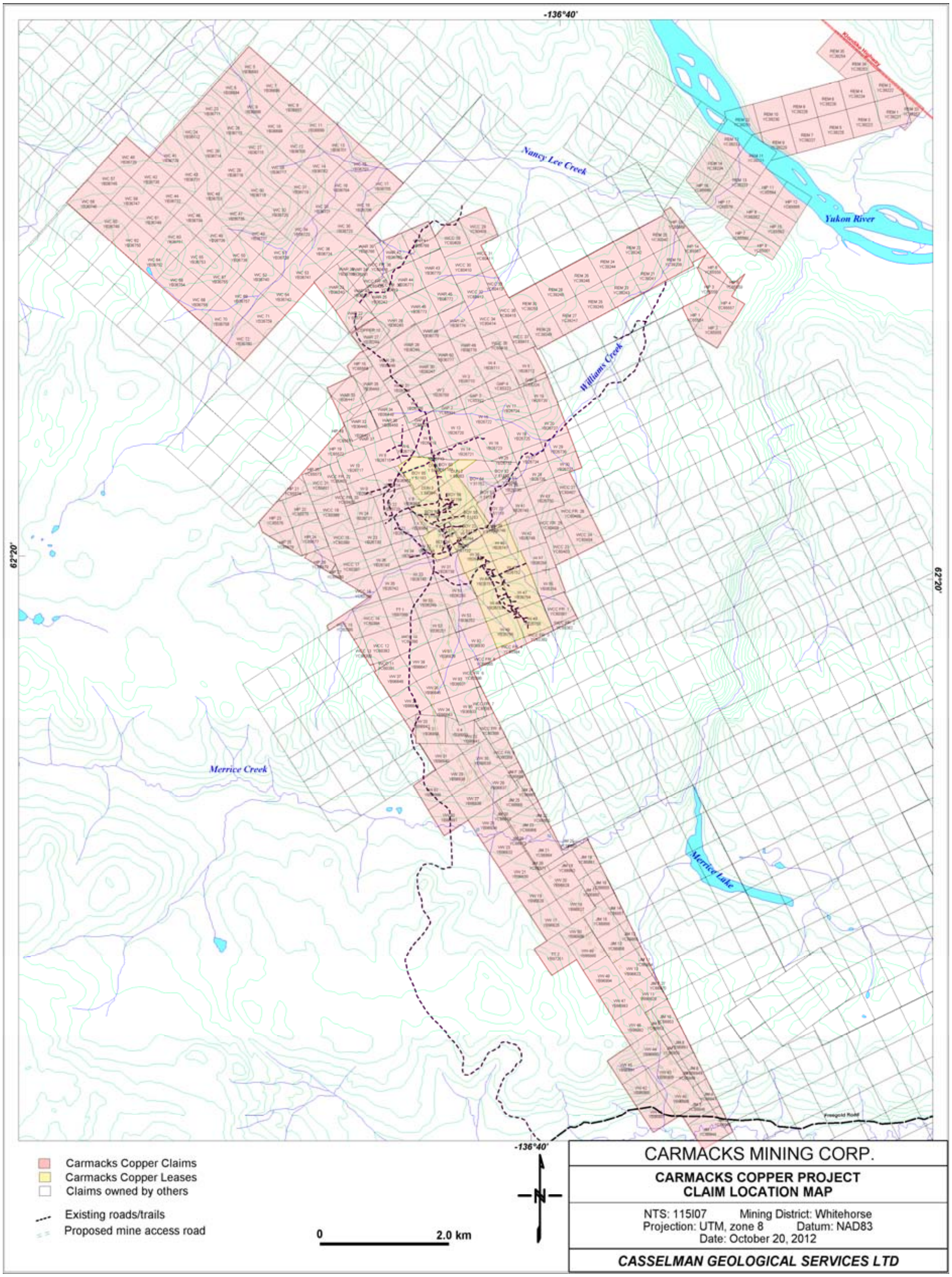


Figure 1-3: Claim Site Map

1.2 GEOLOGY AND MINERALIZATION

The Carmacks Copper copper-gold deposit lies within the Yukon Cataclastic Terrane. The deposit is hosted by feldspathic mafic gneisses (generally quartz deficient) that form a roof pendant within Upper Triassic hornblende-biotite granodiorite of the Granite Mountain Batholith. This study considers the development of only the No. 1, No. 4 and No. 7 Zones, 3 of 14 defined zones containing copper mineralization known on the property (Figure 7-2).

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 and fill fractures and rim sulphides. Primary sulphide minerals and magnetite are 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 subparallel 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 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.

1.3 METALLURGICAL TESTING

Metallurgical test work on various ore samples started in 1989 and has been ongoing since that time. These tests include:

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

Confirmatory test work continues at present to assist with detailed design.

Based on a careful review of the results of these tests the overall copper recovery has been estimated at 85% of the total copper content of the ore. For cash flow purposes, 80% recovery is assumed to occur in the first year the ore is placed on the pad, 1.25% recovery is assumed to occur in each of the following two years and a further 2.5% recovery is assumed to occur at the end of the mine life.

Tests most closely representing the planned operating condition indicate that acid consumption will be 20 kg per tonne of ore or less.

1.4 EXPLORATION STATUS

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, geophysical, and geochemical surveying. Prior to 2006, a total of 80 diamond drill holes and 11 reverse circulation holes, totalling 12,900 m of drilling, had been completed in the exploration of the property. In addition, over 8,000 m of surface trenching was 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 Copper property with a view to better defining the No.1 Zone and starting a more systematic exploration of the other known zones of mineralization. This consisted of diamond drilling and some rapid air blast drilling. The field program was suspended due to freezing weather on November 17, 2006 after a total of 7,100 m in 34 drill holes had been completed.

The exploration drilling program recommenced in March 2007, initially focusing on zones immediately south of the proposed open pit. The 2007 program consisted of 17,000 m of diamond drilling in 123 holes, 845 m of geotechnical drilling in 34 holes, 31.7 line km of induced polarization surveys and surveying of all drill hole locations including all the historic drill holes, geotechnical holes, and rapid air blast drill holes.

1.5 DEVELOPMENT AND OPERATIONS

The Carmacks Copper Project will be developed as an open-pit mine with an acid heap leach and a solvent extraction/electrowinning (SX/EW) process facility producing, on average, approximately 13,200 tonnes of LME Grade A cathode copper annually. Figure 1-4 is a simplified process flow sheet.

The mining operation is designed to produce an average 1.775 million tonnes of ore per year; at peak production approximately 37,500 tonnes (ore and waste) per day on a seven day per week, 24 hours per day operation. The mine will be operated year round but may temporarily suspend operations when winter temperatures are extreme. Ore production will likely be suspended in the coldest winter months but waste operations will continue.

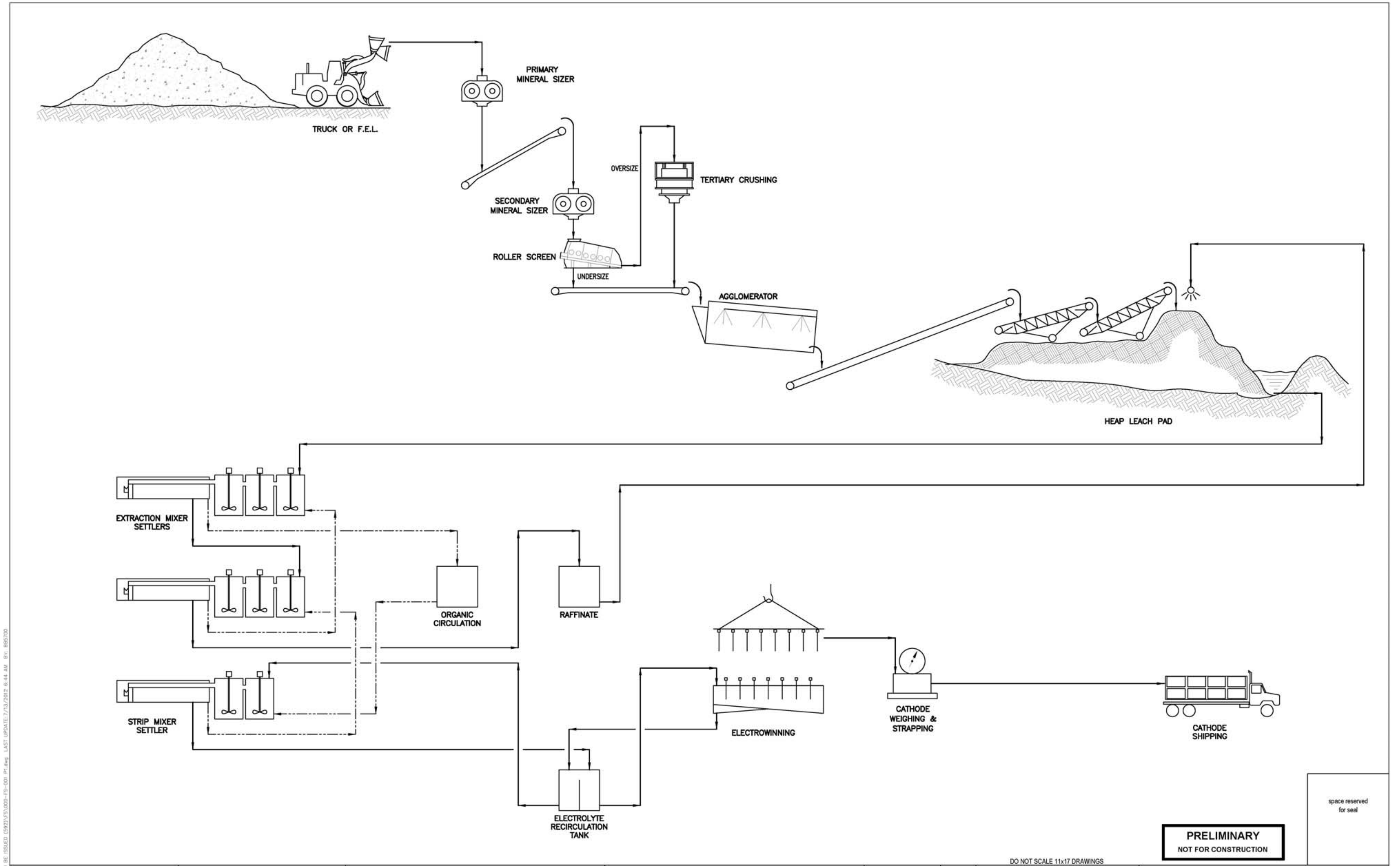


Figure 1-4: Simplified Process Flow Sheet

1.6 MINERAL RESOURCE

Wardrop Engineering constructed a block model of the No. 1, 4, and 7 Zones using historical data and data derived from the 2006 drilling campaign. Mineral resources were classified in accordance with CIM definitions as stipulated in NI 43-101. Carmacks block model contains 78,636 partial blocks coded as Zone No. 1, 4, and 7. There are 17,983 blocks classified as Measured, 43,955 as Indicated, and 16,698 as Inferred. There were no blocks within the mineralized units left unassigned. Based on this block model Wardrop estimated mineral resources as shown in Table 1-1 below.

Table 1-1: Mineral Resources at a 0.25% Total Copper Cut-Off

Zone	Class	Tonnage t (000)	TCu (%)	CuX (%)	CuS (%)	Au (g/t)	Ag (g/t)
Oxide	Measured (ME)	4,031	1.10	0.90	0.20	0.588	5.666
	Indicated (IN)	7,949	1.04	0.83	0.20	0.391	4.039
	ME+IN	11,980	1.07	0.86	0.21	0.456	4.578
	Inferred	90	0.73	0.53	0.20	0.128	1.809
Sulphide	Measured (ME)	695	0.80	0.02	0.77	0.261	2.542
	Indicated (IN)	3,645	0.74	0.03	0.71	0.205	2.296
	ME+IN	4,340	0.75	0.03	0.73	0.221	2.369
	Inferred	4,031	0.71	0.01	0.70	0.179	1.900

1.7 MINERAL RESERVE

The proven and probable reserves are contained within an engineered pit design based on a floating cone analysis of the resource block model using only measured and indicated resources. Inferred resources are not included in the reserve estimate.

Table 1-2: Mineral Reserve Estimate

Reserve Category	K tonnes	Tot Cu (%)	Sol Cu (%)	Nonsol Cu (%)	Gold (g/t)	Silver (g/t)
Proven Mineral Reserve Copper (M lbs)	4,127	1.039 94.5	0.851	0.188	0.559	5.39
Probable Mineral Reserve Copper (M lbs)	7,424	0.943 154.3	0.780	0.163	0.365	3.76
Proven /Probable Reserve Copper (M lbs)	11,551	0.977 248.9	0.805	0.172	0.435	4.34
Notes: Total material in Reserve Pit 69,957 Ktonnes. Waste to Ore: 5.1 Reserves are Fully Diluted and Based on a cutoff Grade of 0.18% Recoverable Copper						

1.8 CAPITAL COST

M3 specifically examined the capital to construct the mine site access road, required plant site roads, substations, water systems, and a crushing plant, heap leach facility, solvent extraction and electrowinning (SX/EW) processing facility and all other temporary and permanent facilities.

The estimate is based on the project as defined by the process and facility descriptions, design criteria, process flow diagrams and material balance, design drawings and sketches, equipment lists, and other documents developed or referenced in the feasibility study. Golder Associates provided a design report which forms the basis for the heap leach and waste rock facility quantities and estimated capital cost of these facilities.

The initial capital cost estimated for project is summarized as follows:

Table 1-3: Initial Estimated Capital Cost

Area	C$
Process & Infrastructure & Project Contingency	$162.1 million
Mine Development	$5.9 million
Mine Equip. Lease 2- Years	$3.8 million
Owner's Cost	$5.8 million
Total	$177.6 million

Life of mine sustaining capital amounts to C$4.7 million. An allowance equal to six months of operating costs is included in the cash flow for working capital. This amount is recovered at the completion of mining.

1.9 OPERATING COST

The operating and maintenance costs for the Carmacks operations have been estimated in detail and are summarized by areas of the plant. Cost centers include mine operations, process plant operations, and the General and Administration area. Operating costs were determined for a typical year of operations, based on an annual ore tonnage of 1.775 million tonnes and an average annual production of 13,200 tonnes of copper cathode. The life of mine unit cost per ore tonne is C$ 29.15 and the unit cost per copper pound is C$1.59. These figures are broken down as follows:

Table 1-4: Unit Cost per Ore Tonne

Area	C$ per tonne ore	C$ per lb copper
Mining	$15.88	$0.87
Processing	$9.71	$0.53
General and Administration	$3.29	$0.18
Shipping	$0.27	$0.01
Total	$29.15	$1.59

1.10 FINANCIAL ANALYSIS

Annual cash flow projections were estimated over the life of the mine based on the above estimates of capital expenditures, production cost, sales revenue, and salvage values. The cash flow model uses a copper price of C$3.20 which is a long term price provided by Copper North Mining Corp.

The after-tax financial indicators based on a 100% equity case are summarized as follows:

Table 1-5: Financial Indicators

IRR	10.0%
NPV @ 0%	C$98.9 million
NPV @ 5%	C$40.3 million
NPV @ 8%	C$14.5 million
NPV @ 10%	C$0.116 million
Payback Period	5.3 years

1.11 CONCLUSIONS AND RECOMMENDATIONS

M3 recommends that CNMC proceed with the development of this project, which is planned as an open-pit oxide mine with acid heap leach and solvent extraction/electrowinning process facilities producing cathode copper. The project will employ conventional, well tested technology throughout.

Initial capital investment in the project is estimated to be C$177.6 million including C$5.8 million for owner's costs. A further C$4.7 million of sustaining capital is required over the life of the mine. The life-of-mine operating costs are estimated to be C$1.59 per pound of copper produced. The base case cash flow model, assuming 100% equity returns an IRR of 10.0% and an NPV of C$40.3 million at 5% discount. This model uses a copper price of US$3.20 which is a long term price provided by CNMC. An exchange rate of C$1.00 = US$1.00 has been used throughout this study.

The project has a number of opportunities which are currently being investigated as it is moved towards development:

- Additional oxide ore reserves with present claim block,
- Reported additional oxide ore resources off-property but within trucking distance,
- Potential of processing oxide stockpile from nearby existing mine,
- Evaluate contract mining in lieu of self-performance, and
- Evaluate re-conditioned equipment for haulage and select process equipment, and
- Evaluate contract crushing.

2 INTRODUCTION

The Carmacks Copper project has been the subject of several prior studies. Two studies are of note; in 1995, Kilborn Engineering Pacific Ltd. (Kilborn) produced a study report titled "Carmacks Copper Project Feasibility Study" and in 1997, Kilborn produced a second report titled "Carmacks Copper Project, Yukon, Canada, Basic Design Report and Definitive Cost Estimate." Both studies examined development of the copper oxide mineral occurrence as an open pit mine with valley fill heap leaching followed by solvent extraction and electrowinning.

In 2007, an additional study report was produced by M3 Engineering & Technology Corp. titled "Carmacks Copper Project, Copper Mine and Process Plant, NI 43-101 Technical Report, Feasibility Study Volume I, Executive Summary". The study evaluated a similar development, but with some differences in mining and plant design.

This present-feasibility study has been prepared by M3 to provide Copper North Mining Corp. and its wholly-owned subsidiary, Carmacks Mining Corp., with an up-to-date development plan, capital and operating cost estimate, and financial analysis for the Carmacks Copper Project.

2.1 TERMS OF REFERENCE

This report was prepared by M3 Engineering & Technology Corp. (M3) at the request of Copper North Mining Corp. (CNMC). It was prepared in order to provide a Technical Report compliant with NI 43-101 which examines the technical and commercial feasibility of developing the copper mineralization at the Carmacks Copper Project in Yukon, Canada.

The study presents a definitive development plan sufficient in detail to confidently determine the economic prospects of the property. The report is based upon a significant body of metallurgical testing. This study will provide a sound basis for future development decisions for the property and will provide a basis for obtaining project financing. Finally, the study suggests areas of opportunities for improved economics and also examines areas of risk where further work would be valuable in mitigating the risks.

The estimate of mineral resources contained in this report conforms to the CIM Mineral Resource and Mineral Reserve definitions (December 2005) referred to in the National Instrument (NI) 43-101, Standards of Disclosure for Mineral Projects.

CNMC is a mineral exploration and development company engaged in the business of exploration and development of mineral properties. CNMC currently has interests in two properties in Canada. CNMC's current focus is on completing this feasibility study and moving the Carmacks Copper Project through to production.

Conrad Huss, P.E., Ph.D, is the principal author for this report and visited the site on June 12, 2012.

Daniel Roth of M3 Engineering & Technology is registered in the Yukon, and M3 is registered as a company in the Yukon.

2.2 SOURCES OF INFORMATION

This report is based in part on M3's corporate knowledge and experience of developing copper heap leach SX/EW projects. It also relies on maps, published government reports, CNMC letters and memoranda referring to historical work, previously conducted studies and reports and public information, all as listed in the "References" section (Section 27) at the conclusion of this report. Several sections from reports authored by other consultants have been directly quoted in this report, and are so indicated in the appropriate sections.

Site visits and areas of responsibility of qualified persons are summarized in Table 2-1.

Table 2-1: Qualified Persons Areas of Responsibility

Name of Qualified Person	Company	Area of Responsibility
Conrad E. Huss	M3	Sections 1, 2, 3, 4, 5, 6, 21, 22, 23, 24, 25, 26, 27
Thomas L. Drielick	M3	Sections 13,17
John Hull	Golder	Section 18, 20
Michael Hester	IMC	Sections 15, 16
Gille Arseneau	Formerly of Wardrop	Sections 7, 8, 9, 10, 11, 12, 14

2.3 ABBREVIATIONS

The following chart outlines common abbreviations used in this report.

This report generally uses the SI (metric) system of units. Exceptions are some common uses such as pounds of copper or use of inches for piping sizes. All engineering calculations are conducted using the SI system. The term "tonne" rather than "ton" is used to denote a metric ton, and is used throughout the report. All costs listed are reported in end of third quarter 2012 Canadian Dollars. Units used and their abbreviations are listed in the table below.

Table 2-2: Abbreviations

Units	Abbreviations
Amperes	A
Cubic meters	m^3
Cubic meters per hour	m^3/h
Current density	A/m^2
Density	t/m^3
Hectares	ha
grams/litre	g/L
Kilo (1000)	k
Kilogram	kg
Kilometer	km
Kilotonnes	ktonnes
Litres	L
Litres per second	L/s
Mega (1,000,000)	M
Meters	m
Millimeters	mm
Parts per Million	ppm
Specific gravity	S.G.
Square meters	m^2
Temperature Celsius	°C
Temperature Fahrenheit	°F
Tonnage factor or specific volume	m^3/tonne
Tonnes per day	t/d
Tonnes per year	t/y
Volts	V
Watts	W

3 RELIANCE ON OTHER EXPERTS

M3 has assumed that all the information and technical documents listed in the Reference section (Section 27) of this report are accurate and complete in all material aspects. While M3 carefully reviewed all the available information, it has not audited this work and cannot guarantee its accuracy and completeness. However, as a result of their review of the work, M3 believes the work has been performed diligently by qualified professionals and that the conclusions derived are reasonable. If any significant new information becomes known or available that would have a significant effect on the findings and conclusions contained in this report, M3 will revise this report.

M3 did not review any licenses, permits, or work contracts. Nor did M3 perform an independent verification of land title and tenure. M3 has 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, wildlife, and groundwater studies were conducted by Access Consulting Group under the direction of CNMC and its predecessors. The site hydrogeology model, site-wide water balance model, and water quality model were developed by Golder Associates under the direction of CNMC.

While M3 has relied largely on the documents listed in Section 27 for the information in this report, the conclusions and recommendations belong exclusively to M3. 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. M3 assumes no information has been withheld which would impact the conclusions or recommendations made herein. Should M3 become aware of facts or information that could materially alter the conclusions and recommendations of the report, M3 will make necessary revisions so the report is correct and accurate.

4 PROPERTY DESCRIPTION AND LOCATION

The Carmacks Copper Project is located in the Dawson Range at latitude 62°-21'N and longitude 136° - 41'W, some 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 1-1 shows the general project location on a provincial scale. Figure 1-2 shows the location on a smaller scale; proximate to the village of Carmacks and the Yukon River.

The Carmacks Copper Project site located in the Whitehorse mining division consists of 318 quartz claims, quartz claim fractions, 20 quartz leases and quartz lease fractions as shown on Figure 1-3. 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., 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 $2.5 million, 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 $900,000 in advance royalty has been paid. As a result, the maximum amount of royalties payable as of the date of this report is $1.6 million.

In the Yukon, claims 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.

The property lies near but does not encroach on LSC R-9A, First Nations Surveyed Lands, Class A Land Reserve, where both surface and mineral rights are reserved for First Nations, in this case the Little Salmon Carmacks First Nation. However, the project site is considered by both Little Salmon Carmacks First Nation and Selkirk First Nation to be in their traditional territory.

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

For exploration (and development) in the Yukon, the Quartz Mining Act and Quartz Mining Land Use Regulations require that:

(1) All areas disturbed must be left in a condition conducive to successful regeneration by native plant species.

(2) All areas disturbed must be re-sloped, contoured or otherwise stabilised to prevent long-term soil erosion.

(3) Structures must be removed and the site restored to a level of utility comparable to the previous level of utility.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND 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 Copper access road for 13 km to the property. The property access road is narrow and rough with steep sections and requires 4x4 capabilities in inclement weather conditions. The Freegold Road is maintained by the territorial government and is currently open seasonally, generally from April through September. Carmacks, on the Yukon River, is 175 km by paved road north of Whitehorse, which is 180 km north of the year-round port at Skagway, Alaska. A new 13 km access road is proposed to be constructed as part of the project development; brush clearing in preparation of this has occurred. Vegetation in wet areas, especially along the Williams Creek valley, consists of willows and alders. Drier areas are covered by spruce and pine trees. The property as a whole is below the tree line.

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 range from -30 °C for the month of January to 12 °C for the month of July. The location close to the Arctic Circle provides 22 hours of daylight in late June with similarly long nights in late December.

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 plummet to –50° C. The Cyprus Anvil open pit lead/zinc mine at Faro and the Brewery Creek open pit/heap leach gold mine not far from the project both operated year round successfully for many years in this climate.

Winter conditions may be considered to extend over the period where daytime maximum temperatures average below zero, which ranges 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 May to October.

5.3 LOCAL RESOURCES

Local commercial resources are limited. The Village of Carmacks, with a population of about 400, has some lodging capacity and a few stores and restaurants. Table 5-1 lists businesses currently based in Carmacks.

Table 5-1: Carmacks-Based Businesses

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
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

Human resources are likewise limited. A large part of the workforce will be drawn from other areas, probably from Whitehorse.

The Tantalus School serves the village of Carmacks by providing 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 new community recreation center with video games, table games, and other activities is a focal point for local youth. The center also has a gymnasium with fitness equipment and an outdoor covered skating rink.

Outdoor recreational opportunities abound. Fishing, hunting, and trapping are popular. Indeed, these activities are basic to the Yukon way of life and central to the sustenance of many people. In addition, summer canoeing down the Yukon River is a significant activity within the area with most canoeists coming from outside the area.

5.4 INFRASTRUCTURE

The project is approximately 220 km from Whitehorse, the capital of Yukon Territory. Whitehorse has a population of around 23,000, which is roughly three-quarters of the entire Yukon population. Whitehorse has an international airport which is 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 port at Skagway Alaska some 180 km to the south. Concentrate from the Faro mine was transported in this way after being trucked from the mine but when Faro closed down so did the railroad, except for tourist excursions. When the Faro mine reopened for a short period of time, the railway was not available and the concentrate was trucked all the way to Skagway for shipping over-seas. 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 town of Carmacks will provide a location for support and administrative services during construction and during plant 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 the 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 areas of the pit and leach pad locations. Areas sufficient for all leach pads, waste dumps, and other mine facilities have been located and designed in the feasibility study.

Carmacks has full communications services available including cell phone service.

5.5 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 lodgement boulder till with a sandy to silty matrix, generally less than 1 m thick.

Overburden is generally thin; a few centimeters 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 in 1996 to become Western Copper Holdings Ltd.

In 1993, Kilborn Pacific Engineering, Inc. (Kilborn) completed the first full feasibility study for the project. Kilborn updated that study in 1995. Based upon positive results reported by Kilborn, Western Copper Holding Ltd. made the decision to proceed with project development and filed for 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. Western Copper Holding Ltd. then began the process of obtaining proposals for the construction of the project. In 1998, after completing some early construction work, 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 permitting process and has been engaged since then in the environmental review process under the YEA process and more recently the newly enacted YESAA 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 Carmacks Copper Project.

In September 2006, Western Copper retained M3 Engineering & Technology Corporation (M3) to revise the earlier studies and to create a Bankable Level Feasibility Study fully compliant with NI 43-101.

In October 2011, Western Copper split spun off two separate companies, Copper North Mining Corp. and NorthIsle Copper & Gold Inc., and changed its name to Western Copper and Gold Corporation. Copper North Mining Corp. has continued to manage the Carmacks Copper Project.

7 GEOLOGICAL SETTING AND MINERALIZATION

The regional geology was described by Bostock in 1936 and more recently by Tempelman- Kluit in 1981 and 1985. Figure 7-1 is a geologic map of the region. The Carmacks region lies within the Intermontane Belt, which in the Carmacks map-area is divisible into the Yukon Cataclastic Terrane, Yukon Crystalline Terrane and Whitehorse Trough. Units of the Whitehorse Trough lie to the east of the Hoochekoo Fault, east of the Carmacks Copper Project. The Whitehorse Trough comprises Upper Triassic intermediate to basic volcanic (Povoas Formation) capped by carbonate reefs (Lewes River Group) and Lower Jurassic greywacke, shale and conglomerate (Laberge Group) derived from the underlying Upper Triassic granitic rocks. The Yukon Cataclastic Terrane includes hornblende-biotite-chlorite gneiss with interfoliated biotite granite gneiss, the Permian Selwyn Gneiss, intruded by the Upper Triassic Klotassin Suite- Minto Pluton and the Granite Mountain Batholith. Weakly foliated, mesocratic, biotite hornblende, Granite Mountain granodiorite contains screens or pendants of strongly foliated feldspar-biotite-hornblende quartz gneisses that host the Carmacks Copper deposit.

The Yukon Crystalline Terrane, extensively exposed southwest of the Carmacks Copper deposit, includes quartz-mica schist with quartzite, marble and amphibolite, Early Palaeozoic age and possibly equivalent to Pelly Gneiss, intruded by Cretaceous and Jurassic-aged granites and syenites. Tempelman-Kluit (1985) has included Upper Cretaceous Carmacks Group intermediate to basic volcanic and Cretaceous Mount Nansen intermediate to acid volcanic and sub-volcanic equivalents in the Yukon Crystalline Terrane.

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

Younger plutonic rocks intrude all three divisions of the Intermontane Belt and the contacts between them. Carmacks Group and Mount Nansen volcanic 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.

The predominant northwest structural trend is represented by the major Hoochekoo, Tatchun and Teslin faults to the east of the Carmacks Copper 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 Copper Project.

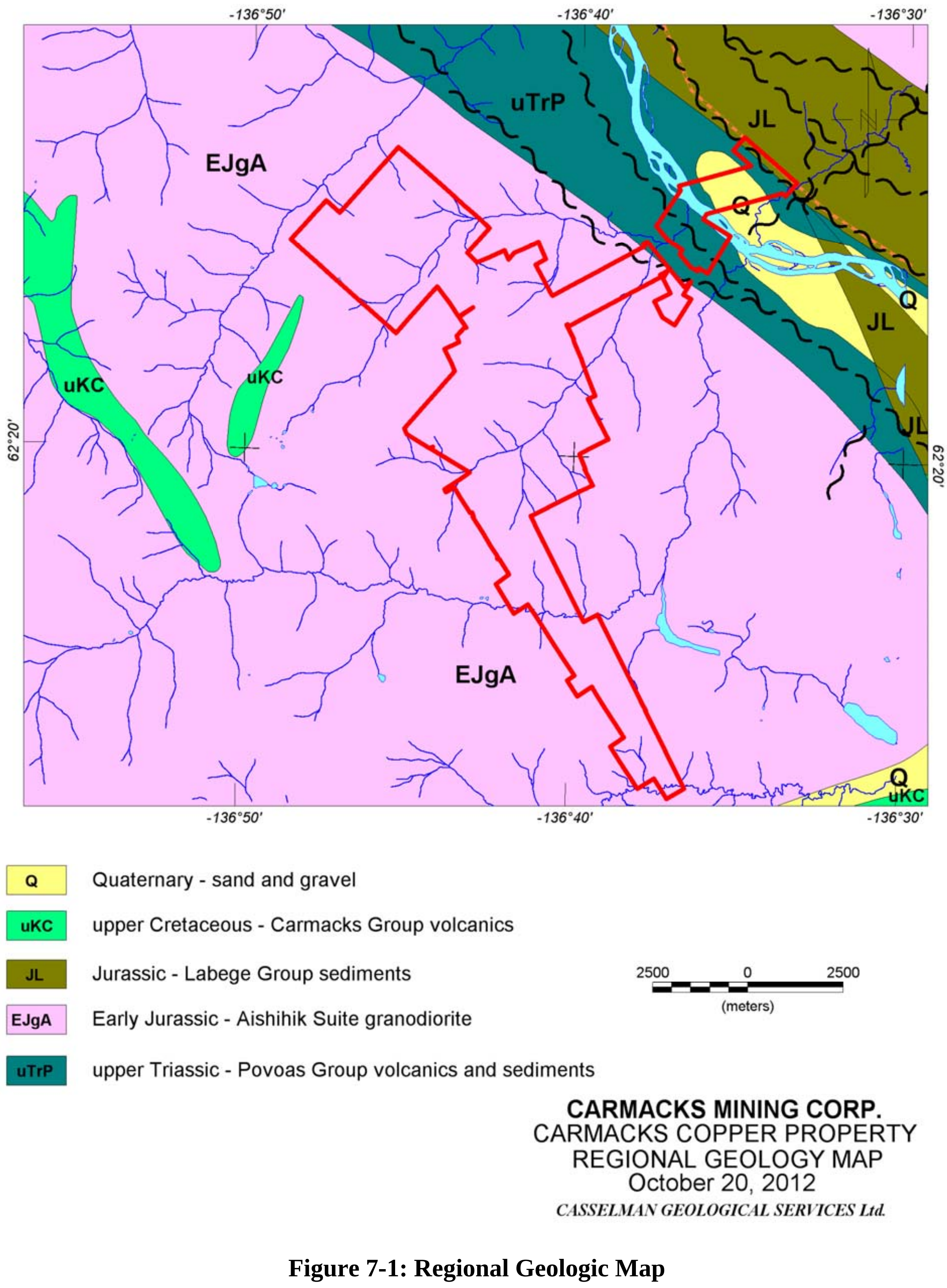


Figure 7-1: Regional Geologic Map

7.1 PROPERTY GEOLOGY

The Carmacks copper-gold deposit lies within the Yukon Cataclastic Terrane. The deposit area is underlain by intrusive and meta-intrusive rocks of the Granite Mountain Intrusion. Compositions range from granodiorite to diorite. These rocks are equigranular to porphyritic, and massive to moderately foliated. 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. Figure 7-2 shows the property geology.

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.

The host rocks for copper and gold mineralization at the No. 1, 4, and 7 Zones can be divided into three types:

1) biotite-rich gneiss and quartzofeldspathic gneiss;
2) ‘siliceous ore’; and
3) fine-grained ‘amphibolite’ and biotite schist. In addition, 13 identified zones containing Cu mineralization are known on or in the immediate vicinity of the property.

Most of the geological information comes from geophysics and drill core 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.

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

The general lack or very low quartz content and the high mafic content suggest a volcanic origin for the gneisses. Occasional drill intercepts of the “gneiss” in Zone No. 4 and in Zone No. 12 returned rock that resembled arkosic sediment, possibly derived from a mafic volcanic or indicating the gneiss is from a mixed volcano-sedimentary environment. An andesitic to basaltic pyroclastic volcanic, probably tuffaceous, agglomeratic or breccia precursor rock is considered the most likely.

Post mineralization aplite and pegmatites are common. They range in thickness from a few centimeters up to three meters. Quartz veins are uncommon and average two to five centimeters in thickness. Thin mafic dykes that were feeders for Carmacks Group volcanic are also uncommon. The only copper mineralization in these dykes and veins is non-sulphide secondary copper in aplite and pegmatite.

All of the historically estimated resources are contained in the No. 1 Zone which extends over a 700 m strike length and at least 450 m down dip. The deposit is open at depth and is oxidized to

250 m in depth. Copper-gold mineralization at Carmacks Copper is hosted by feldspathic-biotite-hornblende-quartz gneisses. These gneisses have been subdivided into nine categories based on coarseness and biotite-hornblende content. All of the gneisses are silica under saturated and mafic rich.

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 No.7 and 7A, terminating against sub-parallel faults down dip. Both the north and south ends of the deposit are offset by cross-cutting faults. The combined strike length of the No. 1, 7 and 7A is 800 m. The mineralized portions of zones No 7 and 7A extends down 120 m, below where the gneiss continues, but with very little copper mineralization. The copper and gold mineralization in No. 7 and 7A are similar to No. 1, with the exception that in some locations the depth of oxidation is shallower.

In the northern half of the zone, copper grades are higher in the footwall relative to the hanging wall. Oxide copper grades increase with depth in both the footwall and hanging wall. There is no association of copper values with rock type, mafic mineral content, or grain size. Gold values are higher in the north half of the deposit. They average 0.022 ounces gold per ton (0.75 g/t) compared with 0.008 ounces gold per ton (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, gold content does not correlate with rock type, mafic constituents or grain size. The majority of the gold occurs in a higher-grade zone between section 1700 N and section 1200 N.

Additional copper oxide resource potential has been identified in Zones 2, 12 and 13. Zones 12 and 13 are located 1.2 km south of Zone 1 and form a continuous body that measure 1.2 km in strike length and up to 110 m in width. Copper oxide mineralization, comparable to that observed in Zones 1, 4 and 7 has been trace to a depth of 60 m. Forty four drill holes have been completed in Zones 12 and 13, of which, 14 encountered significant copper oxide mineralization. Examples of some of the more significant intercepts are hole WC-132, which intercepted 18.0 m grading 0.888% Cu as oxide and hole WC-113, which intercepted 22.0 m grading 0.662% Cu oxide.

Zone 2 is located 3.1 km north of Zone 1. At Zone 2, significant copper oxide mineralization has been observed on surface. However, two historic drill holes failed to delineate copper oxides at depth. Further work is warranted to thoroughly evaluate this target.

The sulphide copper potential of the property has not been the focus of previous exploration efforts. Drill campaigns have historically focused on the oxide copper mineralization and, when sulphides were encountered at depth, it was generally determined to be below the oxide zone and drill holes were terminated. Below the oxide portions of zones 1, 4, 7, 12 and 13 are significant intervals of sulphide copper with significant gold concentrations. In fact, seventy seven (77) drill holes have encountered significant copper sulphide mineralization grading 0.2% Cu or greater. As well, drilling at Zone 2000S and at Zone 14 has encountered significant copper and gold intercepts. Examples of sulphide intercepts in Zone 1 included hole DDH-1-07 which encountered 0.54 g/t Au and 1.12% Cu over 44.81 m; and hole WC-002 which encountered 0.244 g/t Au and 0.93% Cu over 157.0 m.

Zone 2000S is located 600 m south of Zone 1. Drilling there in 2007 encountered 0.172 g/t Au and 0.88% Cu over 37.5 m in hole WC-034B, and 0.330 g/t Au and 1.61% Cu over 8.4 m in hole WC-037. At Zones 12 and 13 some of the more interesting sulphide intercepts included 0.224 g/t Au and 1.11% Cu over 17.75 m in hole WC-123 and 0.139 g/t Au and 0.62% Cu over 89.0 m in hole WC-022. Zone 14 is located 1.5 km southeast of Zone 1. Drilling there in 2007 intercepted 0.108 g/t Au and 1.39% Cu over 8.8 m in hole WC-130 and 0.93 g/t Au and 1.04% Cu over 16.37 m in hole WC-140. Additional drill testing of these targets is warranted to determine the size, grade and economics of the copper-gold sulphide mineralization at depth.

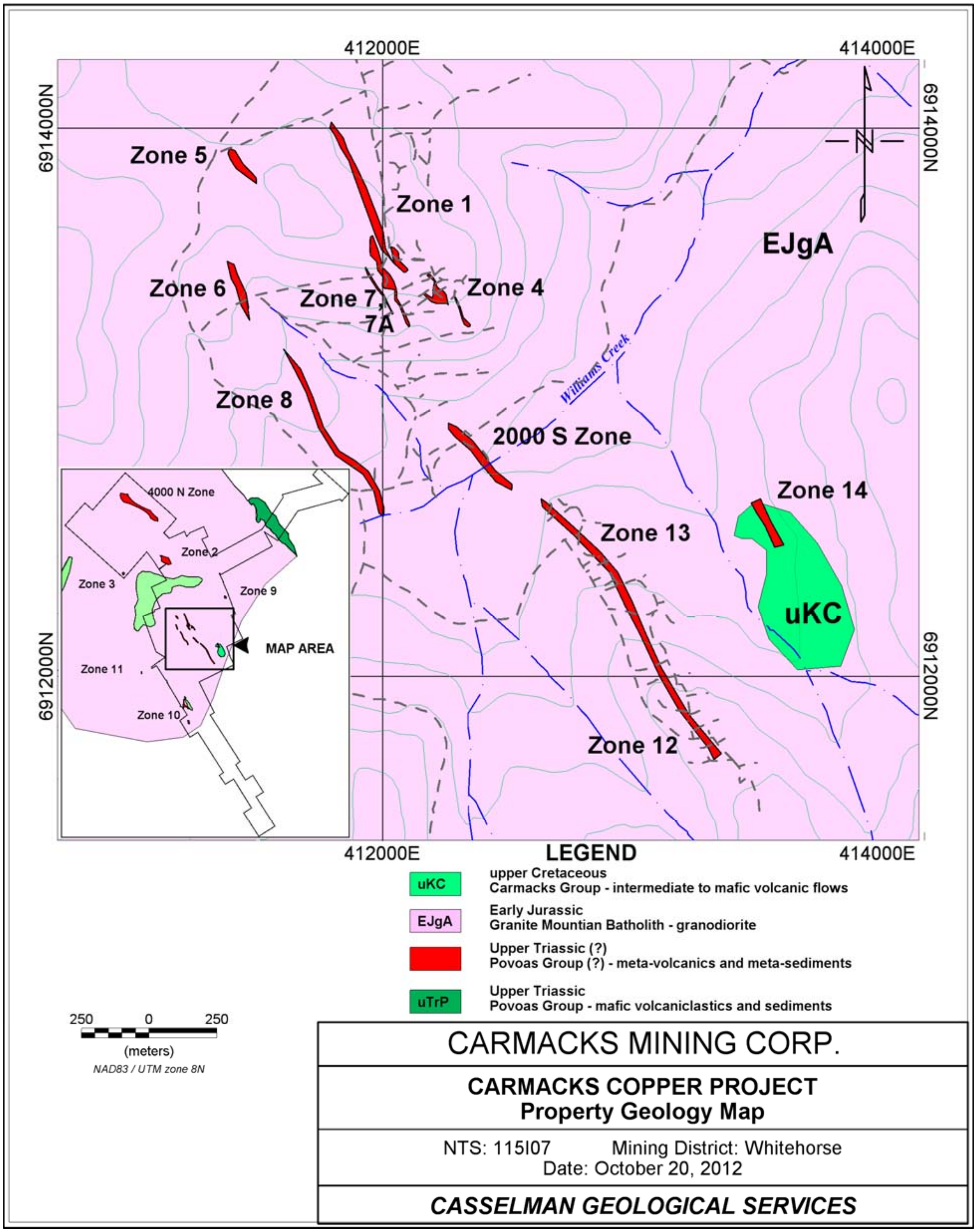


Figure 7-2: Property Geologic Map

7.2 MINERALIZATION

The majority of the copper found in oxide portion of the No. 1, No. 4, No. 7 and No. 7A Zones are 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.

Alteration minerals that could be considered strictly related to the mineralizing event rather than weathering or dyke intrusion are not recognizable. Epidotization and potassium feldspathization are obviously 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.

The upper 250 m of the No. 1 Zone is oxidized. 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 are disseminated and form narrow massive bands or heavy disseminations in bands. Non-copper sulphides are not common in the weathered zone and are usually intergrown or associated with each other when they do occur. They most commonly occur in hematite but also occur in copper sulphides and in the gangue minerals. 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 5 microns in size.

Secondary copper mineralization does not appear to be preferential to a particular rock type. In the north half of the No. 1 Zone, copper mineralization forms high and low grade zones that are reasonably consistent both along strike and down dip and these zones transcend lithologic 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 chalcopyrite, bornite, molybdenite, magnetite, pyrite and pyrrhotite. Primary copper mineralization appears to be zoned from bornite on the north to chalcopyrite and finally to pyrite and pyrrhotite on the south. Narrow veinlets of anhydride were found in the deepest drill hole.

8 DEPOSIT TYPES

The Carmacks Copper 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 Copper deposit have been postulate over the years and by different operators. Evidence from the 2006 and 2007 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 the No. 1, 7, 7A, 8, 12 and 13 zones. Evidence suggests the sulphide mineralization has been re-mobilized out of the rafts into the surrounding granodiorite and in some locations the sulphur has been driven off leaving native copper in the granodiorite matrix. At a later time, when the upper parts of the batholith where eroded and the gneiss was 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 of 2007. The Minto deposit has been interpreted as a either a metamorphosed stratiform sedimentary copper deposit or a metamorphosed porphyry copper deposit.

9 EXPLORATION

CNMC has not carried out any exploration on the Carmacks Copper property. The exploration programs described in this section were carried out by Western Copper Corp. (Western Copper) and its predecessors.

A considerable amount of exploration and drilling has been carried out on the property leading up to and during the discovery and definition of the Carmacks Copper deposit. In addition to drilling, the main mode of exploration has been trenching. The main No. 1, 4, 7, and 7A Zones have been trenched at 200 foot spacing and one or two trenches have been excavated on most of the other known anomalies (Figure 9-1). All trenches across the No.1 Zone were channel sampled with 5 or 10 foot (1.52 m or 3.05 m) sample lengths. Trenches parallel to the zone were not sampled.

Ground geophysics was carried out in 1991 over the No. 1 Zone area and continued north and south over a total 20,000-foot strike length. The survey was done at 200-foot line spacing for a total of 52.4 line miles. The VLF-EM and magnetometer survey identified numerous structures assumed to be faults as well as the main zone style mineralization.

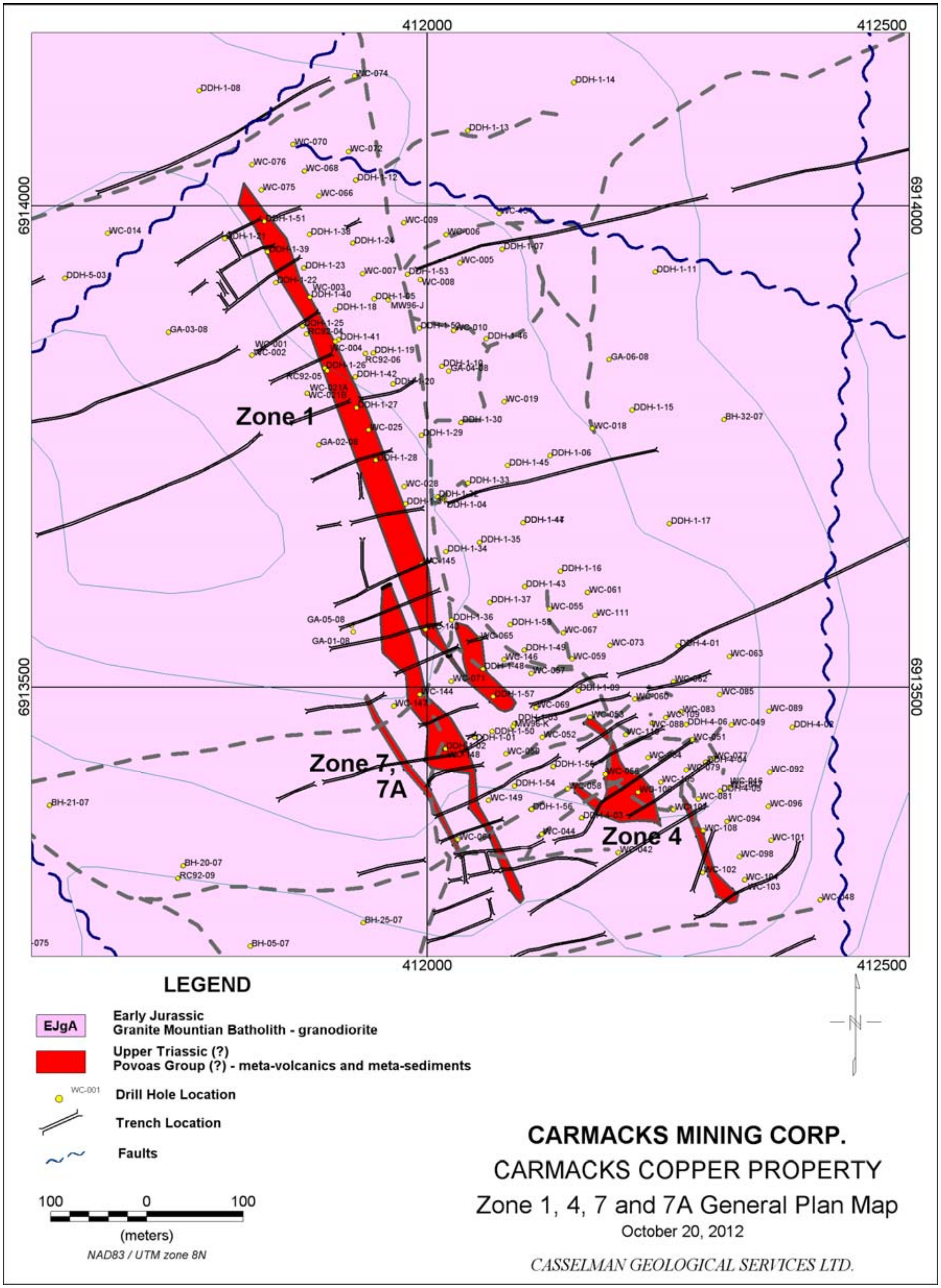


Figure 9-1: Zone 1, 4, 7, and 7A General Plan Map

10 DRILLING

Prior to 2006, a total of 80 diamond drill holes and 11 reverse circulation holes, amounting to approximately 12,900 m of drilling, were drilled in the exploration of the property. Five very short holes totalling 63 m were also drilled on the property. Drill holes are numbered by zone so hole 101 would be the first hole drilled on the No. 1 Zone and hole 1302 would be the second hole in No. 13 Zone.

Core drilling of the No. 1 Zone utilized BQ size in 1971, NQ size in 1990 and HQ size 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 down-hole hammer holes were drilled on the No. 1 Zone in 1992. They were drilled to twin diamond drill holes 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, 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 t-test comparison of reverse circulation holes versus diamond drill holes indicates there is 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 program was greater than 97%.

The object of the 2006 program was to examine the down-dip extension of the No. 1 Zone, with a goal to delineating 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 the No. 1 Zone and to expand the knowledge of some of the other mineralized zones.

In addition, a Rapid Air Blast (RAB) drilling program commenced in August 2006 which was designed to condemn areas of the property for future plant development. The field program was completed on November 17, 2006 and included a total of 34 drill hole for a total of 7,100 m of core.

In 2007, Western Copper continued the exploration and environmental sampling program and conducted geotechnical studies of the proposed heap leach pad, waste rock storage area, processing plant and camp location. The object of the 2007 program was to define the northern and southern limits of the No.1, 7 and 7A zones, to delineated the No. 4 Zone, to further test and define the N. 12 and 13 zones, exploration drilling at the newly discovered No. 14 Zone and condemnation drilling in the proposed waste rock storage, heap leach pad and the processing plant areas. The 2007 program consisted of 17,000 m of diamond drilling in 123 holes, 845 m of geotechnical drilling in 34 holes, 31.7 line km of induced polarization surveys and surveying of all drill hole locations including all the historic drill holes, geotechnical holes, and rapid air blast drill holes.

In 2008, Western Copper drilled 6 geotechnical holes (1,492 m) in the pit area, two (2) water wells in the camp area (253.5 m), and one (1) water monitoring well below the heap leach pad (151 m). Western Copper also conducted a small soil geochemical sampling program.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

Drill core in 1971 and 1990 was sampled in 3.05 m intervals. In 1991 and 1992, drill core was sampled by rock type for geological information but sampling was largely within 3.05 m intervals to facilitate later statistical analysis of assay data.

Reverse circulation holes were sampled over 1.52 m intervals within the No.1 Zone and at 3.05 m intervals for 7.62 m to 15.24 m on either side of the mineralization. Duplicate 12.5% splits were collected with one sample for assay and one sample kept at the core storage area.

All trenches across the No. 1 Zone were channel sampled with 1.52 m or 3.05 m sample lengths. Trenches parallel to the zone were not sampled. In 1971 rock assays were performed by Whitehorse Assay Office in Whitehorse. Two batches of sample rejects were sent to Chemex in Vancouver for check assays. In the first batch, the Chemex results were 5.9% higher than the originals but the second batch returned values 5.7% lower on average. In the programs from the 1990s, trench and drilling samples were sent for analysis to Chemex Labs Ltd. at 212 Brooksbank Avenue, North Vancouver, B.C. All samples were dried and crushed to better than 60% minus 10 mesh. An appropriate size split then underwent Cr-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 a 1/2 assay ton fire assay followed by AAS with a 0.002 oz/ton (0.0686 g/tonne) detection limit and an upper limit of 20 ounces per ton (685.71 g/tonne). Silver was assayed by aqua regia digestion followed by AAS with a 0.01 oz/tonne (0.34 g/tonne) detection limit and an upper limit of 20 oz/tonne (685.71 g/tonne).

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

For the 2006 program, all drill core sample intervals were marked at 1.0 m intervals by a qualified geologist. All samples were cut using a diamond core saw to obtain the best quality split core sample. Samples were packaged and shipped using industry standard secure packaging and were sent to ALS Chemex Laboratories in North Vancouver for processing.

Samples were processed by crushing to >70% -2 mm and pulverizing a 250 gram split to >85% -75 mm according to the ALS Chemex Prep 31 procedure. The samples were then analysed for 27 elements by "Near Total" digestion and Inductively Couple Plasma Emission Spectroscopy (ICP-ES) by ALS Chemex procedure ME-ICP61 or ME-ICP61a. As well, each sample was analysed for gold by fire assay and Atomic Absorption Spectroscopy (AAS) on a 30 g sample by procedure Au-AA23; for total copper content by four-acid (HF-HNO3-HClO4-HCl) digestion and Atomic Absorption according to procedure Cu-AA62; and for non-sulphide copper by sulphuric acid leach and AAS according to procedure Cu-AA05.

Duplicate samples were collected regularly, nominally every 20th sample, and were given unique sample numbers. For the first portion of the program, the duplicates were sent along with the original samples to ALS Chemex for processing and were processed as described above. For the latter portion of the program, the duplicates were sent to Acme Analytical Laboratories in Vancouver for analysis. The samples sent to Acme were processed by crushing to >70% -10 mesh and pulverizing a 250 gm split to >95% -150 mesh according to the Acme R1 50 procedure. The samples were then analysed for 43 elements by "Four Acid" digestion and Inductively Couple Plasma Mass Spectroscopy (ICP-MS) by Acme procedure 1T-MS. As well, all samples were analysed for gold by fire assay and (ICP-ES) on a 30 gm sample by procedure 3B ICPES; total copper content was determined by four-acid (HF-HNO3-HClO4-HCl) digestion and ICP-ES according to procedure 7TD; and for non-sulphide copper by sulphuric acid leach and AAS according to procedure 8.

11.1 SAMPLE SECURITY

Standard sample handling practices of the era were used on the property in 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.

The 2006 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 the lab in Vancouver. There has been no indication by the lab that any of the shipments have been tampered with.

12 DATA VERIFICATION

Wardrop carried out a test of digital assay data integrity by verifying 69% of the database records against the original electronic assay certificates. It should be noted that original assay sheets were missing for 24 of the drill holes; therefore, comparisons could not be made to the original assay certificates.

Of the 53 drill holes verified, a total of 8 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. Wardrop concluded that the assay and survey database is sufficiently free of error to be adequate for resource estimation of the Carmacks Copper deposit.

In August 2006, two historical drill holes were twinned to verify the validity of the historical assay results using current drilling, sample handling and assaying practices.

The twin holes, WC_003 and WC_004, were drilled to test historical holes 91-140 and 91-141, respectively, drilled in 1991. The locations and orientations of the holes are listed in Table 12-1 below:

Table 12-1: Coordinates of Twin Drill Holes

Hole	NAD83UTME	NAD83UTMN	Az_True	Dip
DDH 1-40-91	411878	6913907	248.5	-50
WC-003	411875	6913902	245	-50
DDH 1-41-91	411902	6913855	248.5	-50
WC-004	411905	6913857	245	-50

A comparison between the historical and current assay results can be found in Table 12-2 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 12-2: Comparison of Check Drilling and Historical Drilling

	1-40-91		WC 003		Difference (%) (new-old)	1-41-91		WC 004		Difference (%) (old-new)
	Total Cu	OX Cu	Total Cu	OX Cu		Total Cu	OX Cu	Total Cu	OX Cu	
Length	39.6m	39.6m	39m	39m	-1.54%	48.8	48.8m	48m	48m	-1.67%
Average	1.24	0.84	1.67	0.97	+15.77% (OX Cu)	1.23	0.98	1.13	0.99	+1% (OX Cu)
SD (%)	0.70	0.50	0.87	0.44		1.45	1.05	0.94	0.87	
Var (%)	0.59	0.41	0.70	0.34		0.91	0.66	0.65	0.59	

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 interval while the 2006 drilling was sampled on a three-meter interval. The small discrepancy between total copper values in hole 91-141 and WC 004 are caused by a short intersection of anomalously high grade copper (6.5%) over a length of 2.74 m in 91-141 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 selected by Aurora Geosciences Ltd. personnel and were collected by quartering remaining split core with a rock saw. The samples were collected at one-meter intervals, falling within 1991 sample intervals for comparison purposes. The sample handling, shipping, and preparation control procedures followed were the same as those employed for the 2006 diamond drill program.

It was not possible to sample exactly the same intervals of drill core as were sampled in 1991, but the results are nonetheless 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.

Table 12-3: Comparison of Check Drilling to Historical Drill Intersections

	1991 SAMPLE INTERVALS						2006 ONE METRE RE-ASSAYS					
Hole Number	From (m)	To (m)	Length (m)	Oxide Cu pct	Total Cu pct	Au ppm	From (m)	To (m)	Length (m)	Oxide Cu pct	Total Cu pct	Au ppm
1-22-91	38.40	42.06	3.66	0.77	1.60	1.100	39.92	40.84	0.92	0.51	1.32	0.748
1-27-91	34.75	37.80	3.05	2.95	3.11	0.340	36.88	37.79	0.91	2.43	2.80	0.289
1-28-91	26.52	26.82	3.05	1.61	1.72	0.410	24.68	25.60	0.92	3.00	3.34	1.925
1-32-91	50.90	53.95	3.05	1.81	2.02	0.070	51.81	52.70	0.89	2.93	3.25	0.250
1-35-91	77.42	80.47	3.05	1.82	1.96	0.270	77.41	78.33	0.92	3.14	3.54	0.296
1-38-91	117.81	119.18	1.37	1.12	1.20	0.550	118.56	119.48	0.92	0.93	1.04	0.399
1-50-91	64.53	67.00	2.47	0.90	1.00	0.070	64.31	65.22	0.91	0.90	1.14	0.454
1-56-91	54.86	57.91	3.05	1.86	1.90	0.450	54.86	55.77	0.91	1.28	1.39	0.944
1-57-91	78.64	81.69	2.44	1.20	1.33	3.630	78.94	79.85	0.91	0.81	1.03	0.181
1-58-91	88.39	91.44	3.05	0.18	0.19	0.000	88.39	89.30	0.91	0.37	0.42	0.013
WEIGHTED AVERAGE				**1.42**	**1.60**	**0.689**				**1.63**	**1.93**	**0.550**

In 2007, Dr. Arseneau 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 12-4.

Table 12-4: Assay Results of Representative Samples of No. 1 Zone

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

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.

The sample standards submitted with each batch of samples to each of the analytical labs for the 2007 program returned results that are considered consistent. 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 12-5 lists the statistical results from the standards analyses from both Chemex and Acme:

Table 12-5: 2007 Standard Samples Analytical Statistics

	AGL-1 (high grade Cu)			AGL-2 (moderate grade Cu)			AGL-3 (blank)		
	Au (ppm)	Non Sul. Cu (%)	Total Cu (%)	Au (ppm)	Non Sul. Cu (%)	Total Cu (%)	Au (ppm)	Non Sul. Cu (%)	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

The duplicate samples submitted in 2007 returned generally acceptable values. Figure 12-1 to Figure 12-6 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, non-sulphide copper and total copper analyses.

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.

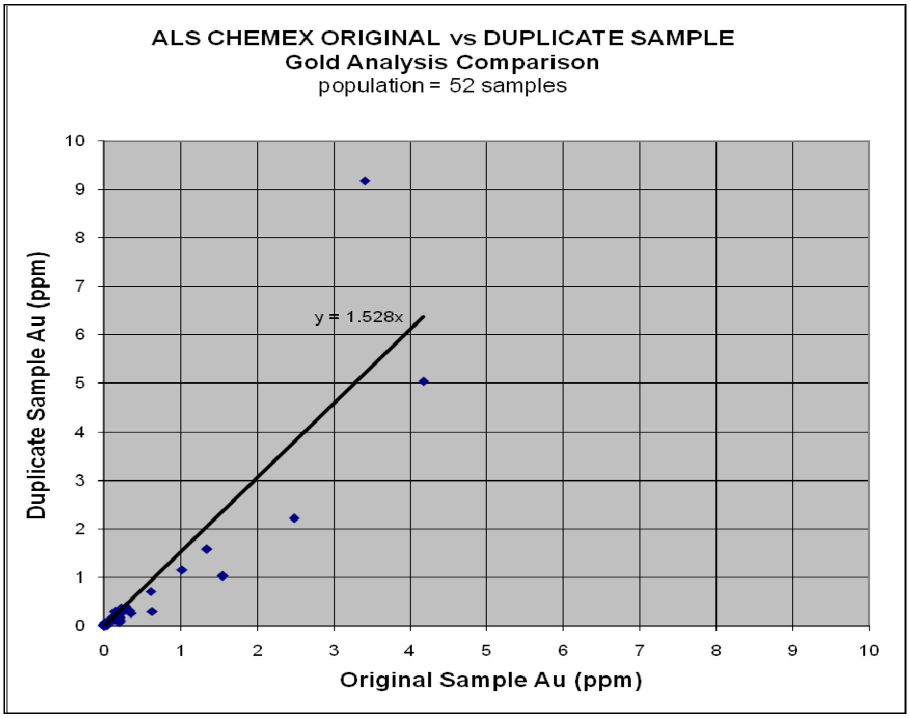


Figure 12-1: ALS Chemex Original vs Duplicate Sample Gold Analysis Comparison

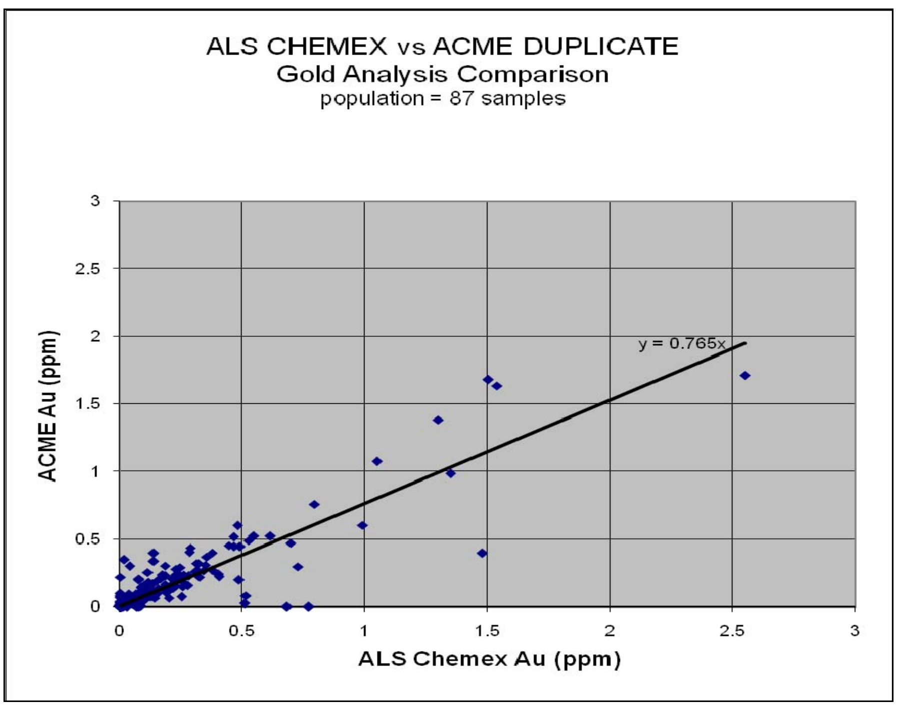


Figure 12-2: ALS Chemex Original vs Acme Duplicate Gold Analysis Comparison

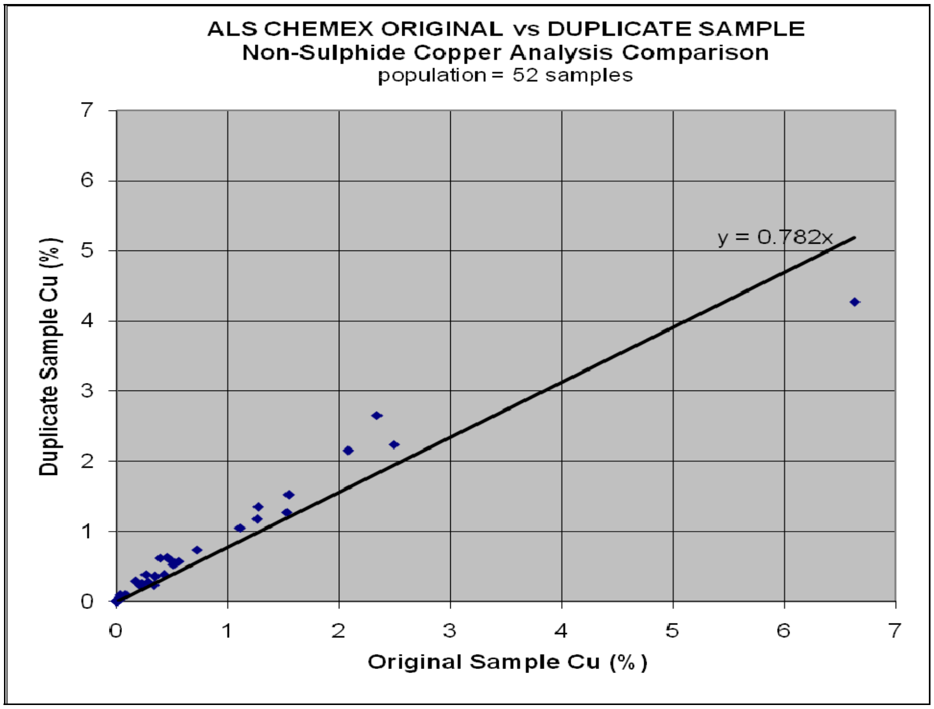


Figure 12-3: ALS Chemex Original vs Duplicate Non-Sulphide Copper Analysis Comparison

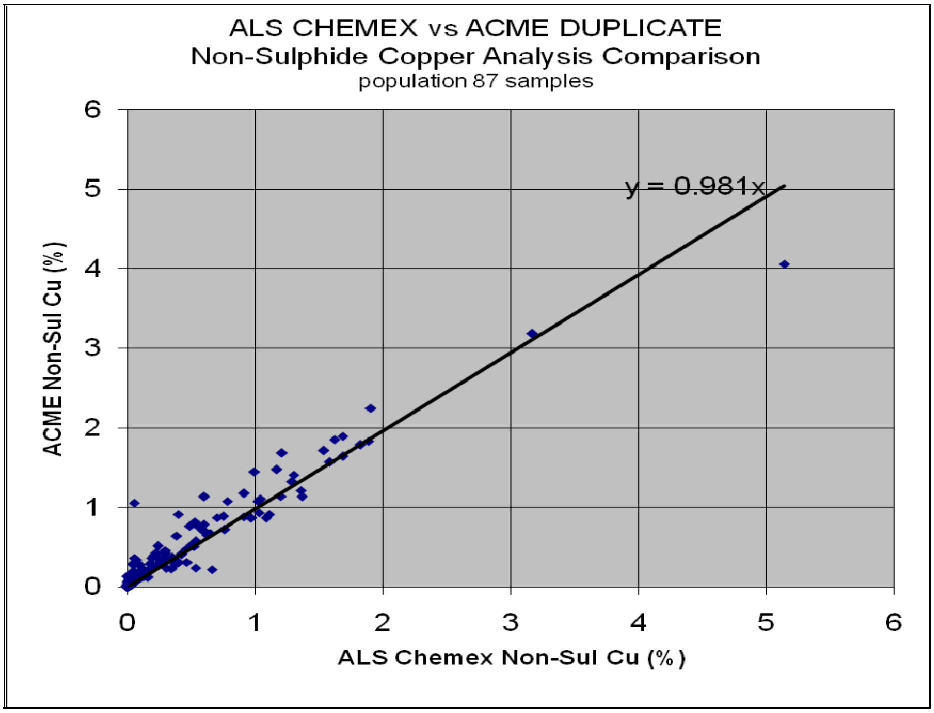


Figure 12-4: ALS Chemex Original vs Acme Duplicate Non-Sulphide Copper Analysis Comparison

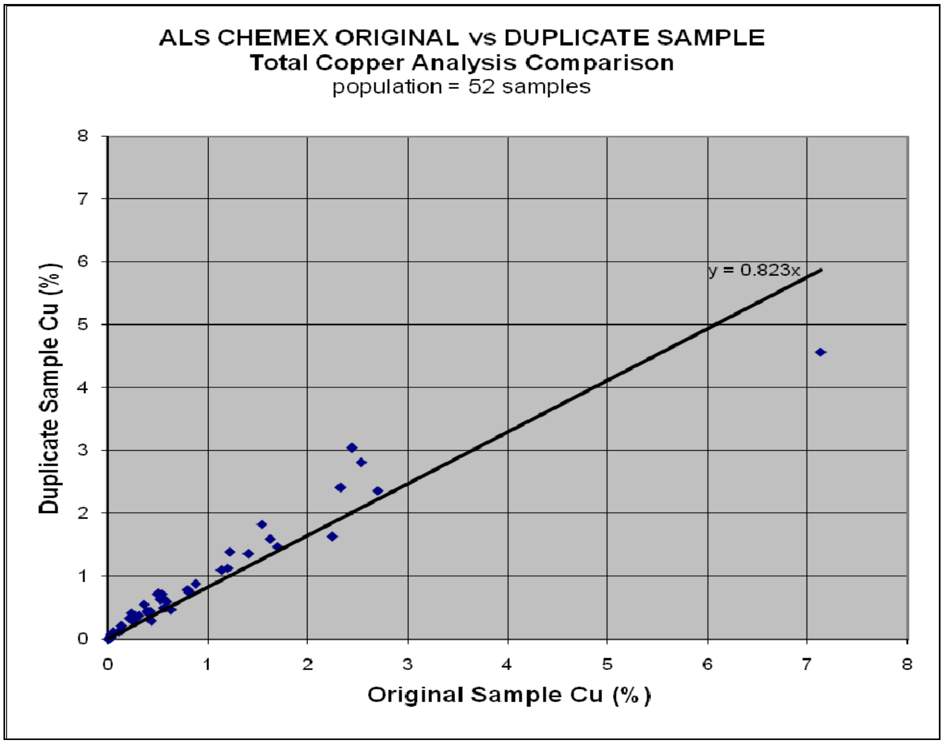


Figure 12-5: ALS Chemex Original vs Duplicate Total Copper Analysis Comparison

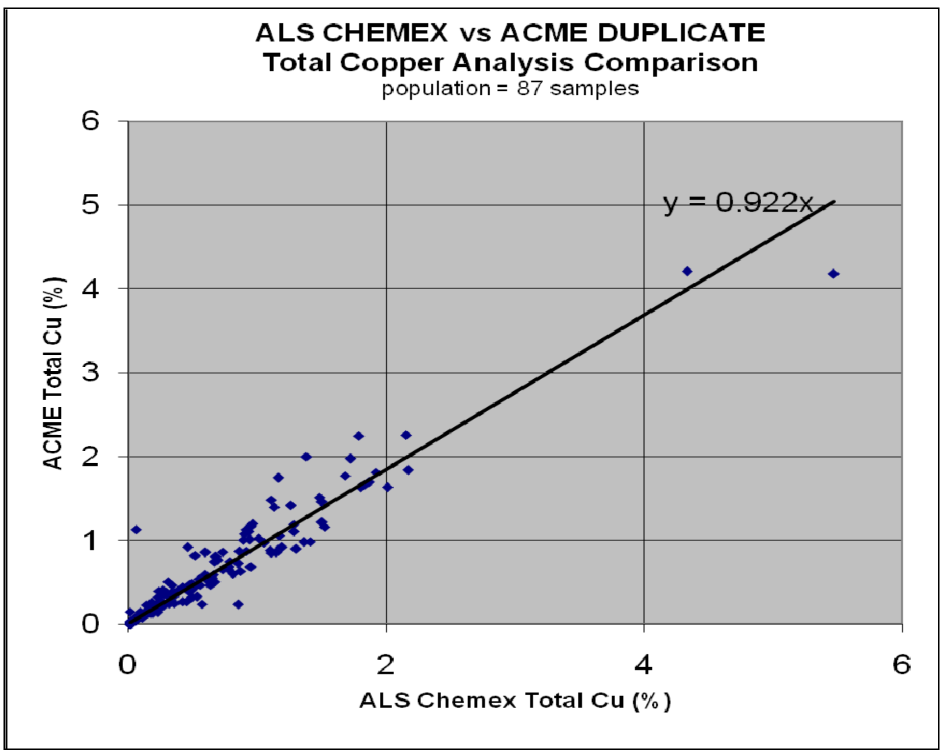


Figure 12-6: ALS Chemex Original vs Acme Duplicate Total Copper Analysis Comparison

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 METALLURGICAL TESTING

The metallurgical testing program on the Carmacks Copper Project focused on the recovery of acid soluble copper mineralization in the oxide cap of the Zone 1 deposit. The primary emphasis has been on development of design criteria and optimal operating parameters for heap leaching the crushed and agglomerated ore, 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) ore, examining leaching of the sulphide mineralization, and recovering gold following copper recovery.

Samples used in the Carmacks metallurgical testing program were taken from either surface trenching or drill core. Table 13-1 presents a list of the 13 metallurgical test programs undertaken since the first test in 1971.

Table 13-1: Historical Metallurgical Test Programs

Test Date	Company	Test By	Ore Classification, Sample or Composite Description	Test type
9/1971	Treadwell Corp.	Goodwin, J	Unknown	B. Roll
10/1989	Coastech Research	Lawrence, R	Unknown	Reactor and column
6/1990	BD&A	Unknown	Ore Composite	B. Roll
5/1992	BD&A	Beattie, M	Drill Core Composite	B. Roll
6/1992	Lakefield	Webster, S.	Drill Core Composite	B. Roll
4/1994	Brown & Root, Inc.	Schlitt, W.J.	Ore Composite	Crib
5/1994	Beattie Consulting, PRA	Beattie, M	Unknown	Column
2/1/1996	Beattie Consulting, PRA	Beattie, M	Drill Core Composite	Column
2/1/2001	Beattie Consulting, PRA	Beattie, M	Ore Composite	Column
4/20/2005	Westcoast Biotech	Bruynesteyn, A.	Ore Composite	Column
3/1/2006	Westcoast Biotech	Bruynesteyn, A.	Ore Composite	Column
4/15/2009	PRA Metallurgical Division	Tan, G.	Ore Composite	Column
2/28/2011	Inspectorate Exploration & Mining Services	Tan, G.	Ore Composite	Column

13.2 COPPER EXTRACTION AND RECOVERIES

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

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

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

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

The 85% extraction will be spread out over the life of the heap. It is assumed that 80% of the leached copper will be recovered in the first year, 1.25% will be recovered in each of the next two years, and an additional 2.5% will be recovered at the end of the mine life.

13.3 SULPHURIC ACID CONSUMPTION

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

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

13.4 OTHER REAGENT REQUIREMENTS

13.4.1 Organic Reagents

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

13.4.2 Other

Other reagents include:

- Guartec will be used as a plating aid in the electrowinning process at a rate of 3,534 kg/yr.

- Cobalt sulphate also assists the plating process. Consumption is estimated at 10,602 kg/yr.

13.5 ORE HYDRODYNAMIC CHARACTERIZATION

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

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

13.5.1 Stacking Test Results

Table 13-2 summarizes the conditions tested during the stacking tests. Agglomeration trials indicated that the optimal moisture addition to obtain full agglomeration of this Carmacks sample is about 6.5% for an acid addition equal to 80% the Net Acid Consumption. Six stacking tests were conducted on Fresh (head) samples while a single sample from the “leached” residue from one of the Hydrodynamic Column Tests (discussed below) was undertaken.

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

Table 13-2: Stacking – Test Matrix

Test	Top Size (mm)	H^+ (kg/Ton)	Ore Type	Heap/Lift Height (m)	Agglomerate Level
1	19	80% NAC*	Fresh	10	4
2	19	65% NAC	Fresh	10	4
3	19	50% NAC	Fresh	10	4
4	19	Optimal NAC	Fresh	10	4
5	19	80% NAC	Fresh	10	1
6	19	Optimal NAC	Fresh	32	4
7	19	Optimal NAC	"Leached"	32	5

* NAC = Net Acid Consumption throughout a complete leach cycle

The results from the Stacking Tests are summarized in terms of the density and conductivity profiles in Figure 13-1 and Figure 13-2.

Bulk density of the ore has a significant effect on the metallurgical performance of a sample and should be carefully considered during the planning and interpretation of metallurgical column tests. Ample empirical evidence shows the performance of a metallurgical test is strongly correlated to the ore density and thus metallurgical columns should be built to represent a realistic bulk density value. Without this information, an accurate scale-up of the results from the metallurgical column tests to the industrial scale is not possible. From the practical point of view the results presented in Figure 13-1 and Figure 13-2 indicate the following:

- The shape of the density profiles for the fresh ore indicates a relative competent porous structure and rock fragments which, as discussed below, lead to a good percolation capacity. This percolation capacity likely arises from the minimal content of fines (1.5% minus 105 um and 0.5% minus 74 um).
- Although the shape of the density profiles for all the fresh-ore samples agglomerated at a Level 4 is similar, agglomeration with an optimal level of moisture and 65% and 50% of the NAC produces a slightly higher as-placed density (1.48 ton/m^3) and a slightly steeper density profile than observed for the sample prepared with 80% if the NAC.
- A comparison of the density profiles indicates optimal agglomeration results from an acid addition of 80% of the NAC and a moisture content of 6.4%. As illustrated in Figure 13-1, these conditions achieve the maximum level of agglomeration possible for this sample (L4) and a reduced bulk density throughout the range of heap heights investigated by the STs (0.3 m to 30 m).
- The less-resilient nature of the "leached" sample results in density values which are larger than those of the fresh ore agglomerated at Level 4 once the heap height exceeds 3 m. For a heap height of 10 m, the density of the "leached" sample is 1.63 ton/m^3 and reaches 1.84 ton/m^3 once the heap height is 28 m which correspond to about 5% and 7% increase with respect to the density values obtained for the fresh ore.

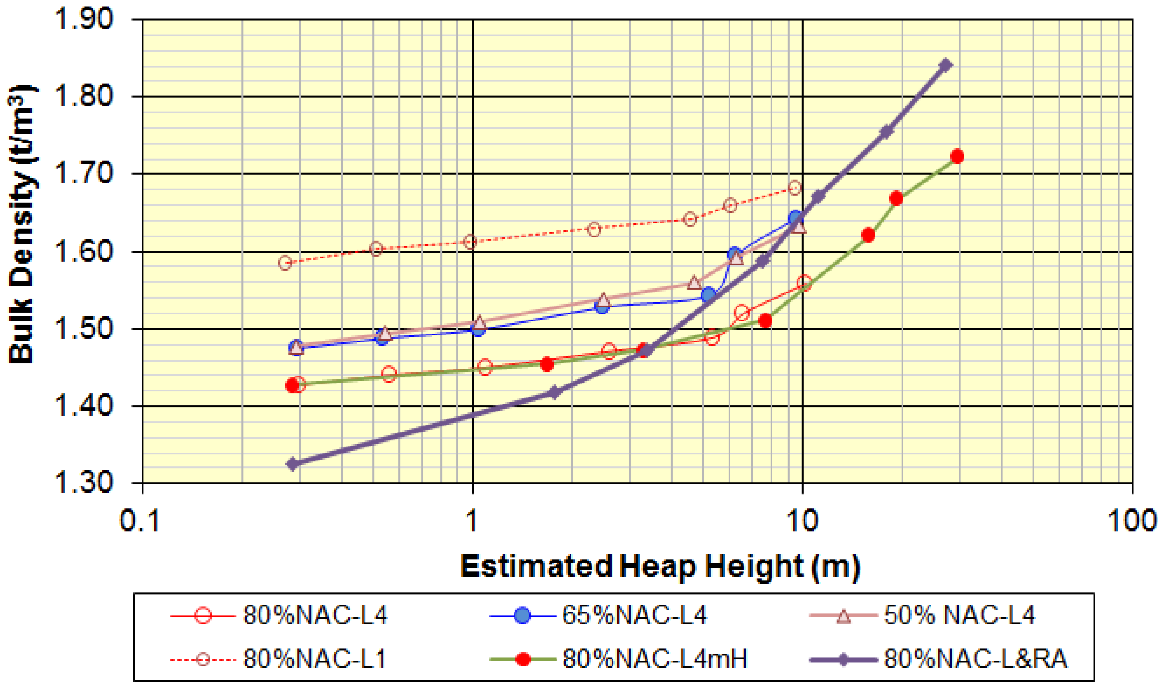


Figure 13-1: Carmacks Oxide Copper – Density Profiles

- Industrial experience shows that as long as the total porosity is larger than or equal to 30% an ore sample can still support heap leaching. Porosity values below this threshold result on too low saturated hydraulic conductivity and too high moisture retention capacity. Based on the specific gravity (SG) for the ore at 2.7 it is estimated that the maximum bulk density for this fresh Carmacks ore is 1.89 ton/m^3. Extrapolation from the density profile from the 80%NAC-L4mH sample suggest that this threshold density value will be reached for a heap height of about 70 m. For the "leached" sample, the heap height corresponding to this density threshold is about 35 m.
- The density profiles of the fresh ore indicate that this sample from the Carmacks ore would be a good candidate for a multi-lift heap design even after one leach cycle as long as the ore properly agglomerated.

Another key parameter for the design of a heap obtained from the Stacking Test is the reduction of the ore percolation capacity as the density (heap height) increases. The conductivity profiles obtained from the Stacking Tests are summarized graphically in Figure 13-2.

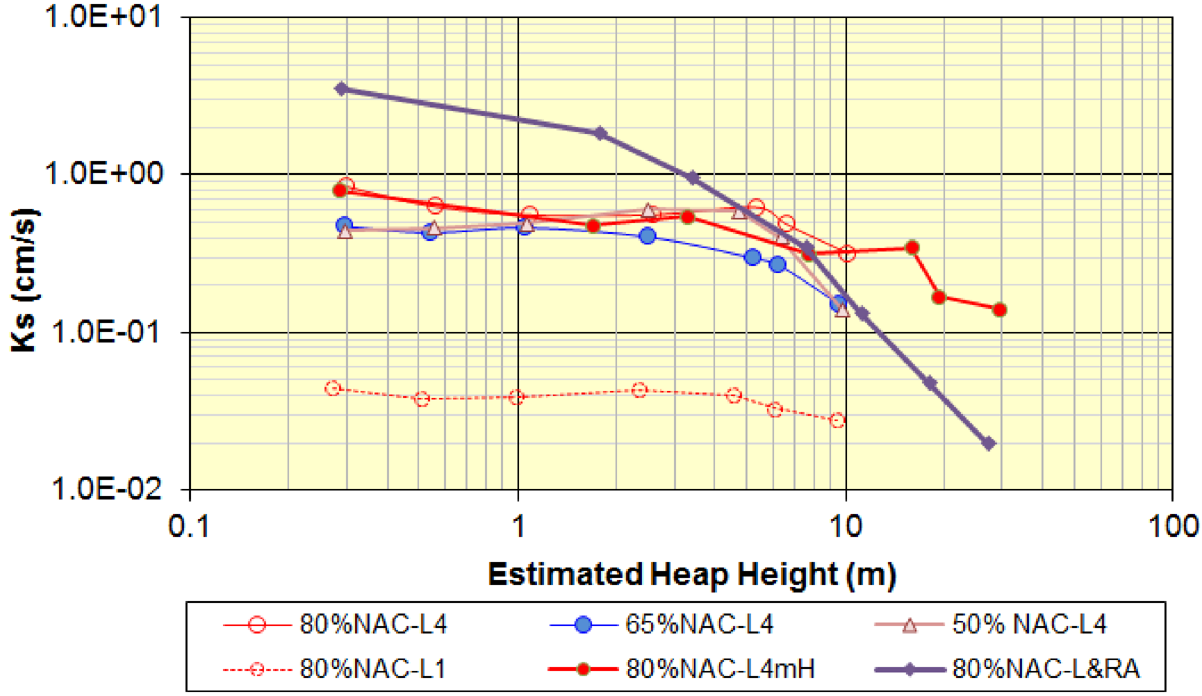


Figure 13-2: Carmacks Oxide Copper – Conductivity Profiles

Inspection of the conductivity profiles indicates that:

- All the samples subjected to the Stacking Tests procedure show reasonable as-placed saturated hydraulic conductivity (Ks > 1 x 10^{-2} cm/s). More importantly, the conductivity profiles show only a slight reduction in the conductivity of these samples as the heap height increases. The maximum reduction in conductivity seems to occur as the heap height increases beyond 8 m.
- Experimental evidence from a large number samples from a variety of ore type indicates that as long as the saturated hydraulic conductivity is larger than or equal to 10^{-2} cm/s; the ore should be a good candidate for heap leaching. All these samples (even the partly-agglomerated 80%NAC-L1) clearly satisfy this minimal requirement.
- The reduction in conductivity observed on these samples is relatively minor over a heap profile of 10 m. In fact, for the optimally agglomerated sample loaded to represent a 30-m heap the conductivity decreases only by a factor of 5.6. The minimum conductivity value (1.4 x 10^{-1} cm/s) obtained for a heap height of 30 m is still adequate to keep this sample as a candidate for heap leaching.
- Overall, the conductivity curves resulting from the Stacking Tests on this Carmacks sample is relatively flat suggesting that the fresh ore is relatively resilient. Typically, an increase in bulk density over the range of heap heights tested during this study (up to 30 m) results in a reduction of conductivity of one order of magnitude or larger.
- Comparison among the conductivity profiles for the various samples indicates that agglomeration to a minimum level of 4 has a positive impact on the percolation capacity

of the ore. For instance, agglomeration at a level 4 with moisture content of agglomeration of 6.5% results in an increase of the saturated hydraulic conductivity between a factor of 5 (for either the 50% or 65% NAC) and a factor of 12 (for the 80% NAC) with respect to a level 1 agglomerated under a 10-m lift .

- Optimal agglomeration will result in minimal bulk density, maximum total porosity and maximum saturated hydraulic conductivity. All these characteristics will provide an opportunity to reduce the risk of high liquid saturation along the heap profile and overall improvement in the metallurgical performance of the process.
- Significant reduction in hydraulic conductivity (two orders of magnitude) was observed on the re-agglomerated "leached" sample over a heap height of 28 m. Similar to the change observed on the density profile, the higher level of agglomeration produces an initially large hydraulic conductivity but the effect of decrepitation (likely due to increase level of fines) produces a sharper reduction in conductivity than that observed for the fresh ore.
- The slope of the hydraulic conductivity profile of the re-agglomerated "leached" ore is steeper than that of any of the fresh samples such that over the first 10 m the conductivity decreases from 3.5×10^{0} cm/s to 1.8×10^{-1} cm/s (a reduction of over an order of magnitude). As the heap height increases to 28 m, the ore conductivity further decreases to 2.0×10^{-2} cm/s (one more order of magnitude reduction).
- It is noted that at a heap height of about 30 m, the percolation capacity of the "leached" sample (2.0×10^{-2} cm/s) is about one order of magnitude (a factor of 10) smaller than that of the fresh ore (1.4×10^{-1} cm/s).
- Another important observation from the comparison between the conductivity profiles of the fresh ore with that of the "leached" sample is that the slope of the former is relatively flat while that for "leached" sample is steeper as the heap height increases beyond 8 m.
- From an operational point of view, the moisture content of the leached ore after drainage will be higher than that used for re-agglomerating the "leached" sample so over stacking of the first lift during the construction of the second lift may produce additional compaction and loss of conductivity than inferred from the results of the STs.
- Comparison among the conductivity profiles for the various samples indicates that agglomeration to a minimum Level of 4 will have a positive impact on the percolation capacity of the ore. Note that minimal agglomeration of the fresh ore at a Level 1 results in reduction of the saturated hydraulic conductivity of at least a factor of 10 with respect to the optimally agglomerated fresh ore (80%NAC- L1 versus 80%NAC- L4).

Although testing of the "leached" residue indicate an increase in bulk density and reduction of saturated hydraulic conductivity (likely related to an increase in the content of fines), the samples tested during this program retain a good level of porosity and percolation capacity so they can be considered good candidates to percolation leaching;

- The best performing ore from the point of view of the physical and hydraulic behavior is the fresh ore agglomerated at a Level 4.

- Clearly, the minimally-agglomerated ore is the worst performing sample (maximum bulk density and minimum saturated hydraulic conductivity).

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

13.5.2 Hydrodynamic Column Test

The test matrix for the Carmacks samples is summarized in Table 13-3 below. Two tests were completed on the fresh ore to represent lift-heights of 8 m and 32 m. A single test was conducted on the “leached” residue from tests 1 and 2. The “leached” sample was loaded to represent a 24 m heap to represent the conditions at the bottom of three 8-m lifts or two 12-m lifts. The HCT on the “leached” sample was initiated late in October and will be reported as soon as it is completed. A Hydrodynamic Column Test (HCT) is performed by placing the ore sample into six-inch diameter columns. The diameter of the columns is selected based on the top size of the ore (~19 mm) to minimize wall effects on the hydrodynamic parameters of the ore.

As indicated on the Test Matrix, the tests on the fresh ore were conducted on optimally agglomerated samples (Level 4) to simulate the response of 8-m lift and 32-m heap. The “leached” residue sample was air dried and rolled to minimize clumping and then re-agglomerated (Level 5) with raffinate to the optimal moisture addition determined for the fresh ore (8.8%).

Table 13-3: Hydrodynamic Column – Test Matrix

Test	Ore Type	Top Size (mm)	H+ (kg/Ton)	Lift Height (m)	Agglomerate Level
1	Fresh	Optimal	80%NAC	8	4
2	Fresh	Optimal	80%NAC	32	4
3	Leached	Optimal	As is	24	5

One of the key pieces of information derived from a HCT is the hydraulic conductivity curve; the relationship between solution application and degree of saturation. In general, the shape of the hydraulic conductivity curve is influenced by the particle size distribution, level of ore conditioning and moisture content, the blending ratio of the ore type, the type of solution used during the agglomeration process, and the bulk density. The hydraulic conductivity curve from the Carmacks fresh ore are summarized in graphical form on Figure 13-3.

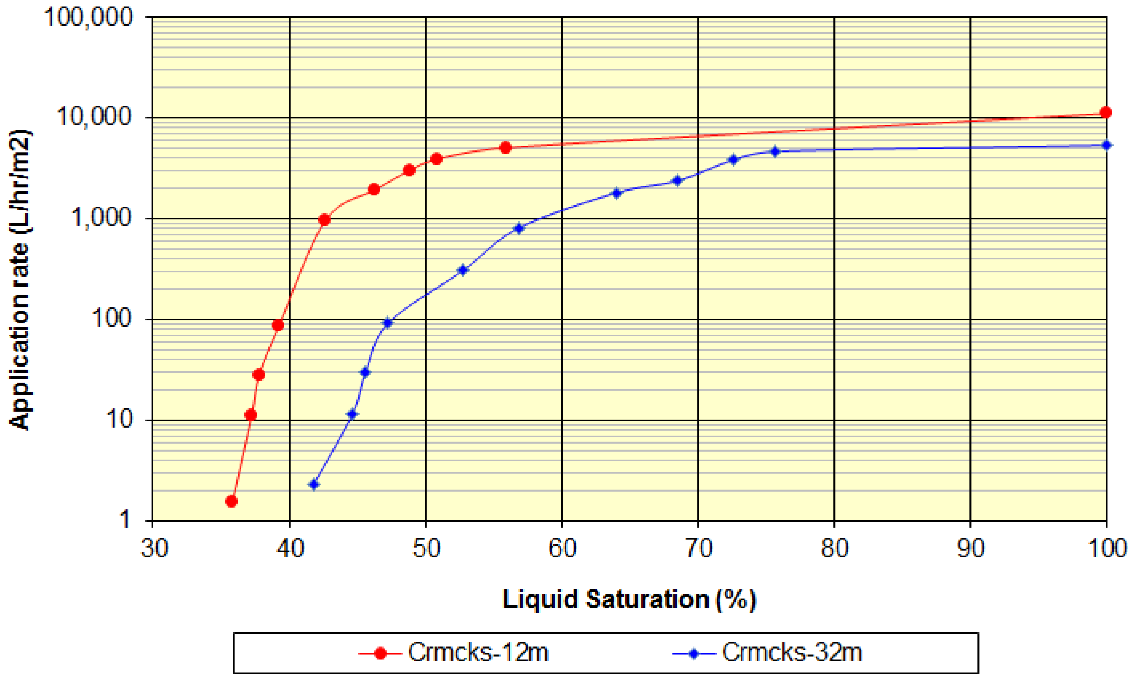


Figure 13-3: Carmacks Oxide Copper – Hydraulic Conductivity Curves

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

- For a typical range of solution application rates (5 L/h/m^2 to 15 L/h/m^2), the ore near the bottom of a 12-m lift would operate at a degree of saturation below 38%. As the heap height increases to 32 m , it is anticipated that the degree of saturation would increase to about 45%;
- Additional loading of the leached ore, as that resulting from a multi-lift heap, will result in higher liquid saturation near the bottom of the heap.
- Given that by design, the samples tested on the HCT represent the bottom of the lift/heap, the material higher on the profile of the lift/heap will in theory operate at a lower degree of saturation.

It is expected that the conductivity curve for the "leached" sample would be to the right and below the curve of the fresh ore loaded to represent a 32-m heap. The key question to be answered by the hydraulic conductivity curve of the "leached" sample is the degree of saturation resulting from a typical solution application rate. As long as the degree of saturation for the leach sample is below 75%, it could be concluded that the resulting might be mechanically stable. Given the multi-lift nature of the Carmacks heap, a degree of saturation higher than this value should be avoided.

Therefore, the results from the HCTs on the fresh sample confirm the preliminary determination obtained from the STs; the particular Carmacks sample tested during this characterization effort are sufficiently competent to support percolation leaching on either a dynamic heap with a minimum lift height of 12 m or a multi-lift heap with a total height of up to 32 m.

14 MINERAL RESOURCE ESTIMATES

Mineral resources were estimated for the Carmacks Project by Wardrop with the use of 3D modeling software, GEMS Version 6.04, provided by Gemcom Software International of Vancouver (Gemcom). Historical drill hole data (pre 2006) were converted from a local Imperial grid to the Metric coordinate system, Nad 83. Drill hole data from the completed 2006 drilling campaign were imported without conversion. Resources were estimated by Mr. Waldegger and verified and validated by Dr. Gilles Arseneau, P. Geo.; former Manager of Geology at Wardrop.

14.1 EXPLORATORY DATA ANALYSIS

Wardrop received a Gemcom project containing drill hole locations, survey, and assay data in imperial measurements for each hole drilled previous to the 2006 drilling campaign.

Collar locations were transformed into the Nad 83 coordinate system using AutoCAD software. The transformation was completed by moving, rotating, and scaling a drawing referenced in the local exploration grid coordinate system to match the same grid in a new drawing referenced in Nad 83. Collar locations were imported as points into Gemcom and pressed to the topographic surface to determine elevation. Drill hole location data were formatted in MS Excel prior to importing into a new, metric Gemcom project.

Survey and Assay data were converted to metric and imported into Gemcom. Limited lithological coding was also received, converted to metric, and imported into Gemcom.

Drill hole data from the 2006 campaign was received as spreadsheets. Location, survey, and assay data were formatted and imported into Gemcom.

14.2 ASSAYS

The two assay populations, pre-2006 and later drilling, were analyzed separately to determine any variances between the two drilling campaigns (Table 14-1 and Table 14-2) and determined that there are no appreciable differences between the historical data and the 2006 drill hole data.

Table 14-1: Descriptive Statistics of Assay Data in Zone No. 1 Oxide- Historical Drilling

Historical Assays	Length	Cu %	CUX	Au g/t	Ag g/t
Valid cases	673	673	673	673	673
Mean	2.266	1.16	0.98	0.557	5.009
Standard error of mean	0.035	0.04	0.03	0.042	0.247
Variance	0.840	0.92	0.70	1.203	41.208
Standard Deviation	0.916	0.96	0.83	1.097	6.419
Skew	-0.41	1.98	1.88	9.62	3.14
Kurtosis	-1.10	6.05	4.88	130.16	14.15
Minimum	0.18	0	0	0	0
25th percentile	1.52	0.47	0.40	0.137	1.029
Median	2.56	0.93	0.77	0.274	3.086
75th percentile	3.05	1.55	1.28	0.651	5.829
Maximum	4.27	7.07	5.70	17.143	59.200

Table 14-2: Descriptive Statistics of Assay Data in Zone No. 1 Oxide –Recent Drilling

Recent Assays	Length	Cu %	CUX	Au g/t	Ag g/t
Valid cases	1357	1357	1357	1357	1357
Mean	0.996	1.23	0.96	0.603	6.358
Standard error of mean	0.002	0.03	0.02	0.025	0.222
Variance	0.007	1.22	0.72	0.847	66.942
Standard Deviation	0.085	1.11	0.85	0.920	8.182
Skew	1.81	2.98	2.74	3.73	2.86
Kurtosis	76.25	14.37	12.16	20.11	10.32
Minimum	0.32	0.005	0.0005	0.0025	0.25
25th percentile	1.00	0.55	0.42	0.131	1.600
Median	1.00	0.93	0.73	0.283	3.500
75th percentile	1.00	1.59	1.25	0.662	7.400
Maximum	2.00	10.95	7.82	10.000	60.000

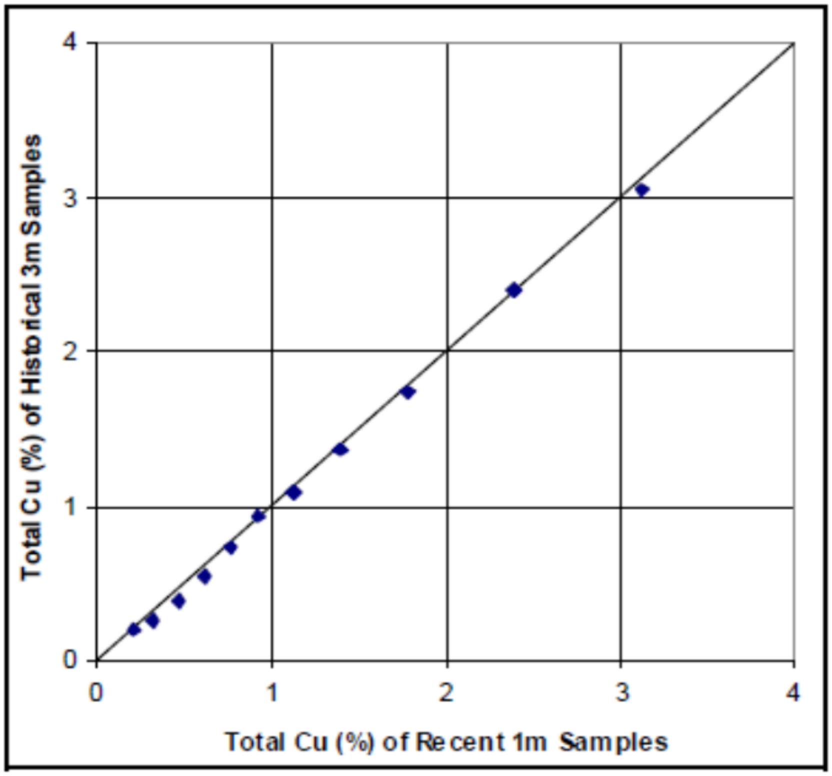


Figure 14-1: QQ Plot of Historical 3m Sample Grades vs. Recent 1m Sample Grades for Zone No.1 Oxide

14.3 CAPPING

Grade capping was considered and evaluated by examining the cumulative frequency distribution and histograms for copper oxide and total copper, while the assay data is log normal, the distribution did not appear to reflect multiple populations so Wardrop decided not to apply grade capping to the assay data.

14.4 COMPOSITES

Assays were composited to a fixed length of five meters, within the wireframes representing the mineralized zones. Composites were generated starting from the toe of the drill hole upwards and incorporated all assay data. Composite lengths were interrupted at geological contacts. A total of 920 composites for all zones modeled, were generated within the wireframes representing the mineralized zones (Table 14-3). A total of 134 composites were less than 5 m in length. Composites that were less than 2.5 m in length were incorporated in to their neighboring composites creating 62 composites between 5 m and 7.5 m in length. A total of 72 composites greater than 2.5 m and less than 5 m were used as is. One composite less than 2.5 m was used as it is in an area of very thin mineralization. A total of 3,094 composites were generated within the surrounding country rock in the same manner as described above. These composites were used to estimate a grade of mineralized material immediately outside of the wireframes.

Table 14-3: Descriptive Statistics of Composites Generated within the Mineralized Zones

	Length	Cu %	CUX	Au g/t	Ag g/t
Valid cases	920	920	920	920	920
Mean	4.984	1.00	0.71	0.441	4.506
Std. error of mean	0.019	0.02	0.02	0.019	0.164
Variance	0.320	0.51	0.43	0.338	24.643
Std. Deviation	0.566	0.72	0.66	0.582	4.964
Skew	-0.59	1.72	1.53	3.49	2.61
Kurtosis	10.14	4.96	3.98	18.02	8.46
Minimum	1.00	0	0	0	0
25th percentile	5.00	0.50	0.20	0.137	1.681
Median	5.00	0.83	0.57	0.249	3.008
75th percentile	5.00	1.34	1.04	0.474	5.338
Maximum	7.38	5.51	4.73	5.701	36.991

Table 14-4: Descriptive Statistics of Composites Generated within the Granodiorite

	Length	Cu %	CUX	Au g/t	Ag g/t
Valid cases	3094	3094	3094	3094	3094
Mean	4.984	0.01	0.01	0.003	0.085
Std. error of mean	0.008	0.00	0.00	0.000	0.007
Variance	0.176	0.00	0.00	0.000	0.169
Std. Deviation	0.419	0.06	0.03	0.018	0.411
Skew	-1.23	15.63	7.50	10.82	13.94
Kurtosis	24.00	433.66	69.60	165.78	286.80
Minimum	1.21	0	0	0	0
25th percentile	5.00	0.00	0.00	0.000	0.000
Median	5.00	0.00	0.00	0.000	0.000
75th percentile	5.00	0.00	0.00	0.000	0.000
Maximum	7.62	2.11	0.49	0.426	10.910

14.5 BULK DENSITY

In 1991, specific gravities were estimated by Chemex Labs, Ltd. on 21 drill core samples. Granodiorite comprised 5 samples, pegmatite 2 samples and gneiss 14 samples.

Granodiorite specific gravities from the hanging wall and footwall ranged from 2.69 to 2.71 for an average of 2.70. Gneiss specific gravities ranged from 2.59 to 2.97 although only one sample was greater than 2.73.

In 2006 and 2007, specific gravities were measured by Aurora Geoscience in the field on 1,358 drill core samples. An average specific gravity of 2.64 was determined for samples collected within the Zone No. 1 Oxide zone, and 2.75 within the Zone No. 1 Sulphide zone.

Table 14-5: Descriptive Statistics of Specific Gravity Data

	Zone No. 1 Oxide	Zone No. 4 Oxide	Zone No. 7 Oxide	Zone No. 1 Sulphide	Granodiorite
Valid cases	132	50	22	59	1095
Mean	2.643	2.646	2.663	2.749	2.661
Std. error of mean	0.009	0.010	0.016	0.014	0.003
Variance	0.010	0.005	0.005	0.012	0.008
Std. Deviation	0.100	0.068	0.074	0.110	0.088
Skew	-0.44	-0.05	0.58	-1.08	-1.38
Kurtosis	2.99	0.20	-0.37	2.28	16.27
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

14.6 GEOLOGICAL INTERPRETATION

Three mineralized zones (zone 1, 4, 7, and 7a) were interpreted on the basis of total copper grade. Surfaces were generated to represent the hanging wall and foot wall contacts with the mineralized zones. The surfaces honour the drill hole intersections in 3D. The solids were extended laterally approximately 15 m beyond the outermost drill hole intersections. The solids were generated by stitching the two non-intersecting surfaces together and then clipping the solids against the topographic surface.

The oxide sulphide boundary was modeled using a minimum 20% ratio of oxide copper to total copper. All assays that contained at least 20% of the total copper value as oxide copper were coded as oxide in the model. A polyline was generated on an inclined longitudinal section to represent the oxide-sulphide boundary. The polyline was snapped to the assays on the down dip drill holes, honouring the 3D points. A clipping solid was generated by extruding the polyline 100 m on either side of the section. The three mineralized zones were then clipped and intersected with the oxide clipping solid to create final oxide and sulphide solids for all three mineralized zones (Figure 14-2).

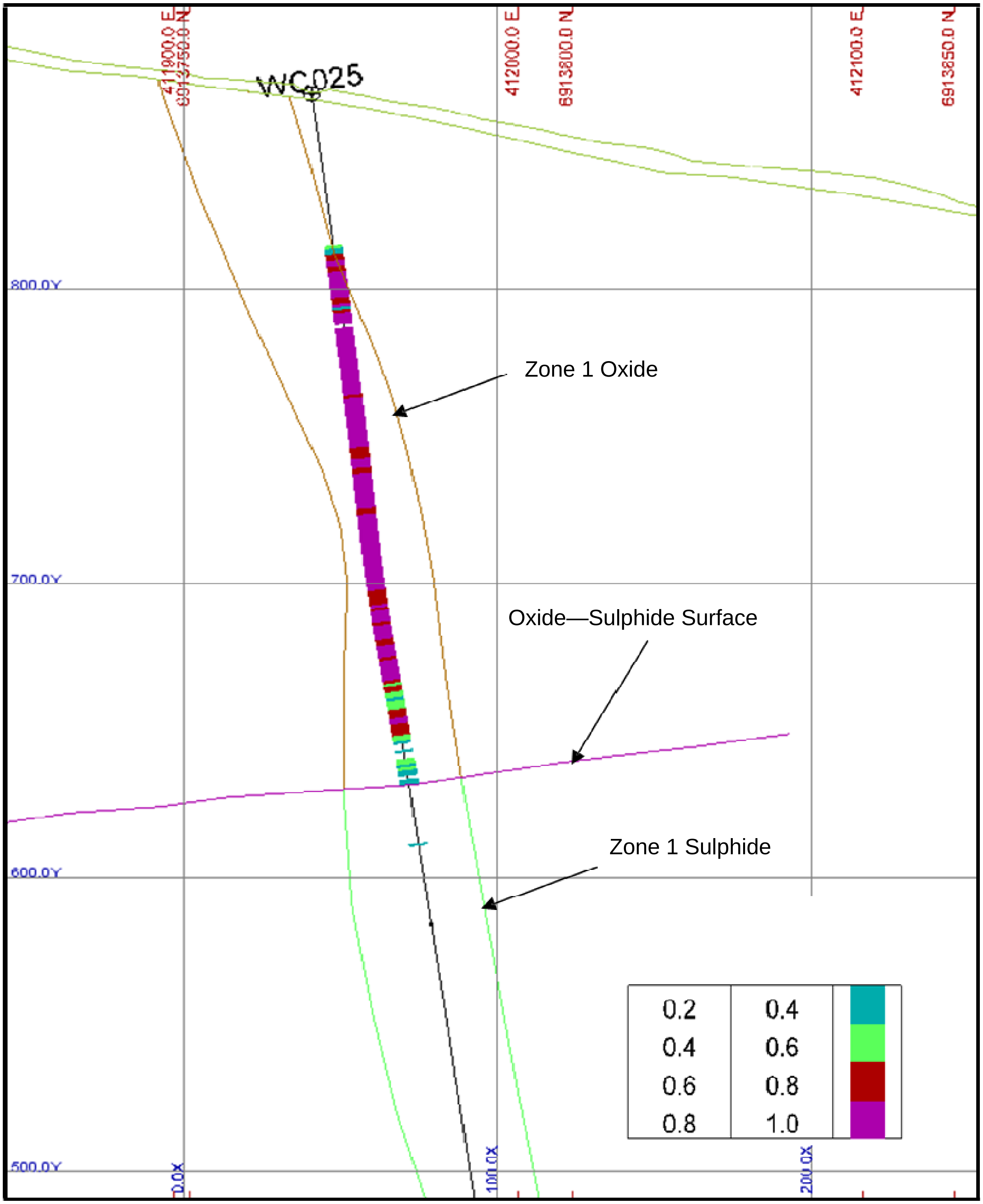


Figure 14-2: Cross Section Showing Oxide Copper to Total Copper Proportion in Drill Holes and Oxide & Sulphide Solids

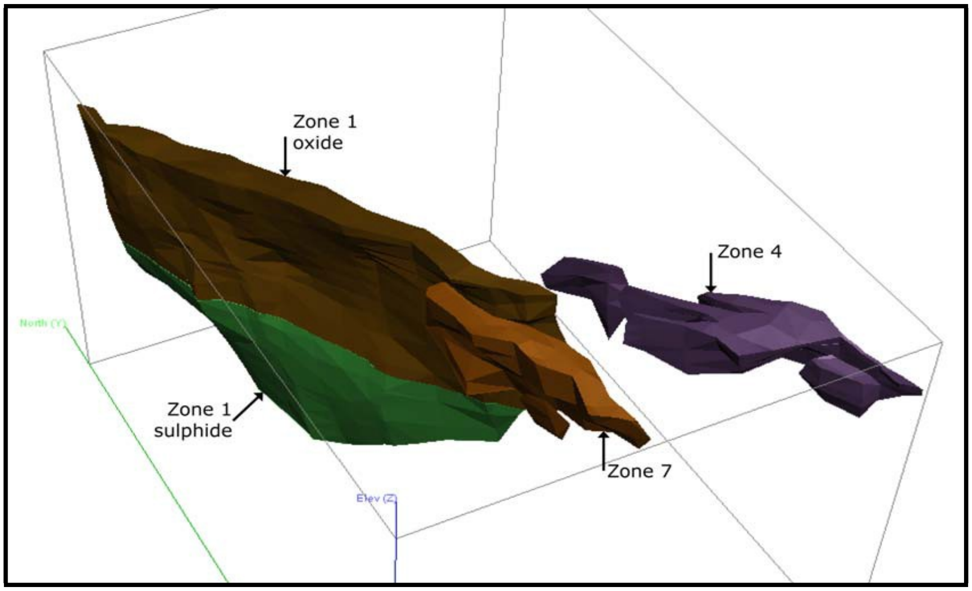


Figure 14-3: Mineralized Zones Clipped to Overburden, Viewed form Southwest

14.7 RESOURCE BLOCK MODEL

Mineral resources were estimated with 3-dimentional software provided by Gemcom. Grades were interpolated for total copper, oxide copper, gold and silver into 5 by 5 by 5 m blocks. The parameters defining the block model are presented in Table 14-6.

Table 14-6: Block Model Parameters

	Model Origin	No. of Blocks	Block Size
Easting	412050	70 columns	5 m
Northing	6913130	195 rows	5 m
Elevation	900	110 levels	5 m

The block model was rotated 24.2 degrees anti-clockwise around the origin, aligning it parallel to the strike of the deposit and the surface exploration grid.

14.8 ROCK TYPE MODEL

The rock type model was coded using the topographic surface and the modeled solids in the following sequence and as outlined in Table 14-7. The rock type model was first coded with waste, rock code (99), in the following sequence:

1. All blocks in the model were initialized to air, rock code (0)
2. All blocks below the topography surface were then initialized to overburden, rock code (9).
3. All blocks below the overburden-bedrock surface were then initialized to waste.

The rock type model in the Standard folder was then updated form the wireframes representing the mineralized zone. An accuracy level of nine needles per block oriented horizontally along rows was used to update the roc type model. Any block that was more than 0.001% by volume within the wireframe was re-code as being part of the mineralized zone according to the wireframe rock code.

Table 14-7: Block Model Rock Codes

Rock Type	Block Model Code
Air	0
Overburden	9
Waste	99
Zone 1 Oxide	101
Zone 4 Oxide	104
Zone 7 Oxide	107
Zone 7 Sulphide	207

14.9 PERCENT MODEL

The percent model was updated only from the mineralized zones clipped to topography, using horizontal needles by row. The percent model can be used to weight the volume of each block during resource reporting in order to estimate an accurate tonnage from the model.

14.10 DENSITY MODEL

Density was interpolated into blocks in two passes using isotropic inverse distance weighted to the second power. Interpolation occurred in two passes with sample support summarized in Table 14-8.

Table 14-8: Interpolation Parameters for Density Model

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

After the estimation process, any mineralized blocks that had a specific gravity value less than 2.5 were re-initialized to an average value based on their rock type as summarized in Table 14-9.

Table 14-9: Density by Rock Type

Rock Type	Density
Zone 1 Oxide	2.64
Zone 4 Oxide	2.64
Zone 7 Sulphide	2.64
Zone 1 Sulphide	2.75

14.11 GRADE MODEL

Geostatisticians use a variety of tools to describe the pattern of spatial continuity, or strength of the spatial similarity of a variable with separation distance and direction. The correlogram measures the correlation between data values as a function of their separation distance and direction. The distance at which the correlogram reaches the maximum variance 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; it is the distance over which sample values show some persistence or correlation. The shape of the correlogram describes the pattern of spatial continuity. A very rapid decrease near the origin is indicative of short scale variability. A more gradual decrease moving away from the origin suggests longer scale continuity.

Using Sage 2001 software, Variography was completed for the zones at Carmacks. Directional sample correlograms were calculated along horizontal azimuths of 0, 30, 60, 90, 120, 150, 180, 210, 240, 270, 300 and 330 degrees. For each azimuth, sample correlograms were also calculated at dips of 30 and 60 degrees in addition to horizontally. Lastly, a correlogram was calculated in the vertical direction. Using the twenty four correlograms an algorithm determined the best-fit model. This model is described by the nugget (C0), two nested structure variance contributions (C1, C2), ranges for the variance contributions and the model type (spherical or exponential). After fitting the variance parameters, the algorithm then fits an ellipsoid to the forty-eight ranges from the directional models for each structure. The final models of anisotropy are given by the lengths and orientations of the axes of the ellipsoids.

Results of the variography on the entire sample population were mixed because the sample density was too low for Zone 1 Sulphide and Zone 7, so variography was completed for only Zone 1 oxide. Correlograms were calculated for Cu Oxide, and modeled with a nugget and two nested spherical structures. The results are summarized in Table 14-10.

Rotation angles are set to correspond to Gemcom's rotational convention, which follows the right hand rule with rotation about Z axis being positive when X moves towards the Y axis, rotation about the Y axis is positive when Z moves towards the X axis. Grade models were interpolated for total copper, oxide copper, gold and silver grades. Sulphide copper was estimated as a function of the total copper and oxide copper contents as defined below.

14.11.1 Copper Grades

Copper grades (total copper percent and oxide copper percent) were interpolated into blocks using ordinary kriging with weighting parameters based on the correlogram data.

Table 14-10: Correlogram Data for Zone 1 (Oxide Only)

Element	Domain	Model	Z Rotation	Y Rotation	Z Rotation	Z Range	Y Range	X Range
Cu	Zone 1 Oxide							
		C_0=0.25						
		C_1=0.64	-24.8	39	-49	17.4	126.9	16.1
		C_2=0.335	-35.2	-23	-48	268.5	381.5	27.4

Grade interpolation search ellipses were designed from the correlogram information, trend of mineralization and sample data distribution. The grades were interpolated in three separate passes with differing sample support and search ellipses as summarized in Table 14-11 and Table 14-12.

Table 14-11: Sample Selection Criteria for Grade Interpolation

Rock Code	Codes used for Grade Interpolation	Domain
101	101	Zone 1 Oxide
104	104, 101	Zone 4 Sulphide
107	107, 101	Zone 7 Oxide
207	201	Zone 1 Sulphide
99	99	Waste

Table 14-12: Grade Interpolation Parameters for Copper

Pass	Axes Rotation	Ranges (m)	Occurrence per hole	Minimum Samples	Maximum Samples
1	Z=0 X=70 Z=0	X=100 Y=100 Z=15	1	3	10
2	Z=0 X=70 Z=0	X=150 Y=150 Z=50	1	2	12

Grades were only interpolated if at least three samples, no more than one sample per hole, were found within the search ellipse, and a maximum of twelve samples were used to interpolate any block for the first pass. The second pass only estimated grades in blocks that were un-interpreted in pass one. Blocks were assigned a grade in pass two if at least two samples, no more than one per hole, were found within the search ellipse. The third pass only estimated grades in blocks that were un-interpreted in pass one and two. Blocks were assigned a grade in pass three if at least two samples, no more than one per hole, were found within the larger search ellipse. Sample selections for grade interpolations were restricted by oxidation zones and by zones as indicated in Table 14-11.

14.11.2 Sulphide Copper Percent Calculation

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

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

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

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

14.11.3 Gold and Silver Grades

Gold and silver grades were interpolated into blocks using inverse distance weighted to the second power.

The same search ellipse from pass 3 for copper grades was used to interpolate gold and silver grades. The grades were interpolated in one pass with sample support summarized in Table 14-11 and search ellipse as summarized below in Table 14-13.

Table 14-13: Grade Interpolation Parameters for Gold and Silver

Axes Rotation	Ranges (m)	Occurrence per hole	Minimum Samples	Maximum Samples
Z=0 X=70 Z=0	X=150 Y=150 Z=50	1	3	8

14.12 MINERAL RESOURCE CLASSIFICATION

Mineral resources were classified in accordance with definitions provided by CIM as stipulated in NI 43-101. The Carmacks Copper mineral resources are classified by Wardrop as Measured, Indicated and Inferred.

The Carmacks Copper block model contains 78,636 partial blocks coded as Zone No. 1, 4, and 7. There are 17,983 blocks classified as Measured, 43,955 as Indicated, and 16,698 as Inferred. There were no blocks within the mineralized units left unassigned (Figure 14-4).

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

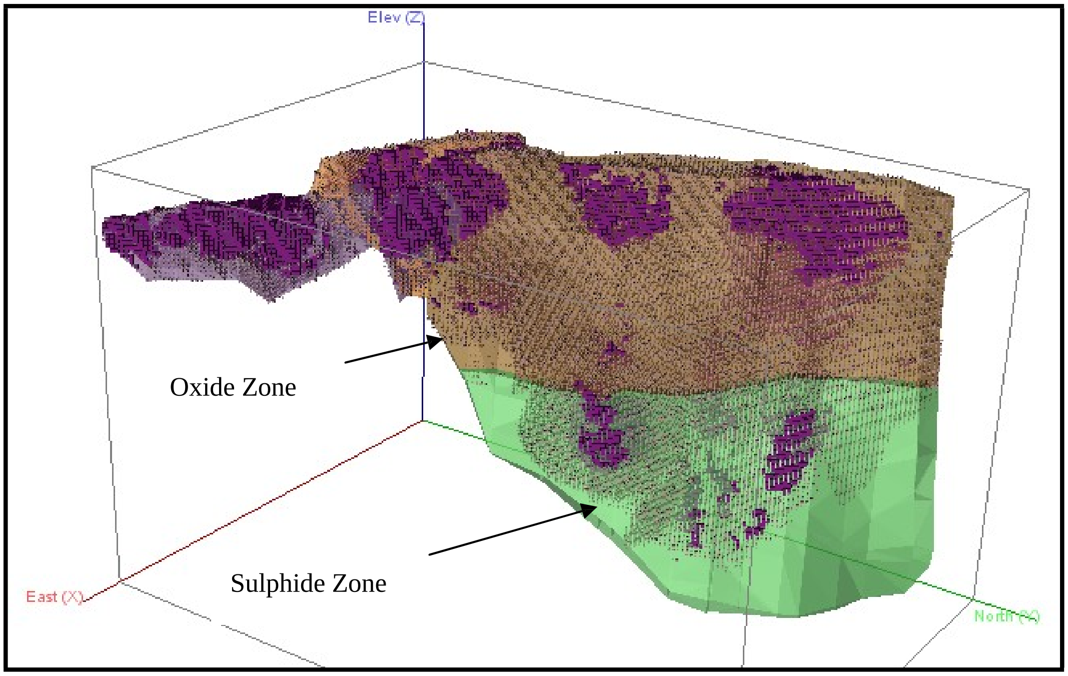


Figure 14-4: Three Dimensional Representation of Block Model Classification

Note: Magenta color blocks are measured mineral resource and the smaller grey blocks are indicated mineral resource.

14.13 MINERAL RESOURCE TABULATION

Wardrop estimated that the combined Zones 1, 4 and 7 contained approximately 12 million tonnes of oxide resource in the Measured plus Indicated categories grading 1.07 TCu, 0.86 CuX, 0.21% CuS, 0.46 g/t Au, and 4.58 g/t Ag at a 0.25% total copper (TCu) cut-off grade (Table 14-14).

Zone 1 also contained an additional 4.3 million tonnes of sulphide resource in the Measured plus Indicated categories grading 0.75% TCu, 0.03% CuX, 0.73% CuS, 0.22 g/t Au, and 2.37 g/t Ag.

In addition to the measured and indicated resource, the deposit contains 90,000 tonnes of oxide inferred resource grading 0.73% TCu, 0.53% CuX, 0.20 CuS, 0.12 g/t Au and 1.8 g/t Ag and 4 million tonnes of sulphide inferred resources grading 0.71 TCu, 0.01 CuX, 0.70 CuS, 0.18 g/t Au and 1.9 g/t Ag.

Table 14-14: Mineral Resources at 0.25% Total Copper Cutoff

Zone	Class	Tonnage t (000)	TCu (%)	CuX (%)	Cu S	Au (g/t)	Ag (g/t)
Z1 Oxide	Measured (ME)	2,985	1.25	1.02	0.23	0.696	6.514
	Indicated (IN)	7,058	1.07	0.86	0.21	0.405	4.094
	ME+IN	10,043	1.13	0.91	0.22	0.492	4.813
	Inferred	64	0.84	0.62	0.22	0.122	1.793
Z4 Oxide	Measured (ME)	614	0.48	0.37	0.11	0.211	2.414
	Indicated (IN)	257	0.51	0.35	0.16	0.184	2.230
	ME+IN	871	0.50	0.36	0.15	0.192	2.285
	Inferred	23	0.41	0.25	0.16	0.139	1.871
Z7 Oxide	Measured (ME)	432	0.97	0.82	0.15	0.376	4.430
	Indicated (IN)	634	0.90	0.74	0.16	0.317	4.155
	ME+IN	1,066	0.92	0.76	0.16	0.335	4.237
	Inferred	3	0.81	0.64	0.18	0.179	1.665
1+4+7	**Measured (ME)**	**4,031**	**1.10**	**0.90**	**0.20**	**0.588**	**5.666**
1+4+7	**Indicated (IN)**	**7,949**	**1.04**	**0.83**	**0.20**	**0.391**	**4.039**
1+4+7	**ME+IN**	**11,980**	**1.07**	**0.86**	**0.21**	**0.456**	**4.578**
1+4+7	**Inferred**	**90**	**0.73**	**0.53**	**0.20**	**0.128**	**1.809**
Z1 Sulphide	**Measured (ME)**	**695**	**0.80**	**0.02**	**0.77**	**0.261**	**2.542**
	Indicated (IN)	**3,645**	**0.74**	**0.03**	**0.71**	**0.205**	**2.296**
	ME+IN	**4,340**	**0.75**	**0.03**	**0.73**	**0.221**	**2.369**
	Inferred	**4,031**	**0.71**	**0.01**	**0.70**	**0.179**	**1.900**

Wardrop also estimated the mineral resources using a 0.50% TCu.

Table 14-15: Mineral Resources at 0.5% Total Copper Cutoff

Zone	Class	Tonnage t (000)	TCu (%)	CuX (%)	CuS (%)	Au (g/t)	Ag (g/t)
Z1 Oxide	Measured (ME)	2,827	1.30	1.06	0.24	0.726	6.787
	Indicated (IN)	6,767	1.10	0.89	0.21	0.416	4.202
	ME+IN	9,594	1.16	0.94	0.22	0.507	4.963
	Inferred	63	0.85	0.63	0.22	0.123	1.817
Z4 Oxide	Measured (ME)	252	0.63	0.48	0.15	0.274	3.001
	Indicated (IN)	116	0.69	0.47	0.22	0.238	2.838
	ME+IN	368	0.67	0.47	0.20	0.249	2.886
	Inferred	4	0.60	0.39	0.21	0.190	2.796
Z7 Oxide	Measured (ME)	405	1.00	0.85	0.16	0.390	4.607
	Indicated (IN)	623	0.91	0.74	0.17	0.321	4.203
	ME+IN	1,028	0.94	0.77	0.16	0.341	4.322
	Inferred	3	0.81	0.64	0.18	0.179	1.665
Z1 Sulphide	Measured (ME)	608	0.85	0.02	0.83	0.277	2.607
	Indicated (IN)	2,917	0.82	0.03	0.78	0.222	2.442
	ME+IN	3,525	0.83	0.03	0.80	0.238	2.491
	Inferred	3,082	0.81	0.02	0.80	0.216	2.291

14.14 BLOCK MODEL VALIDATION

Wardrop completed a detailed visual validation of the Carmacks block model. The model was checked for proper coding of drill hole intervals and block model cells, in both section and plan. 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).

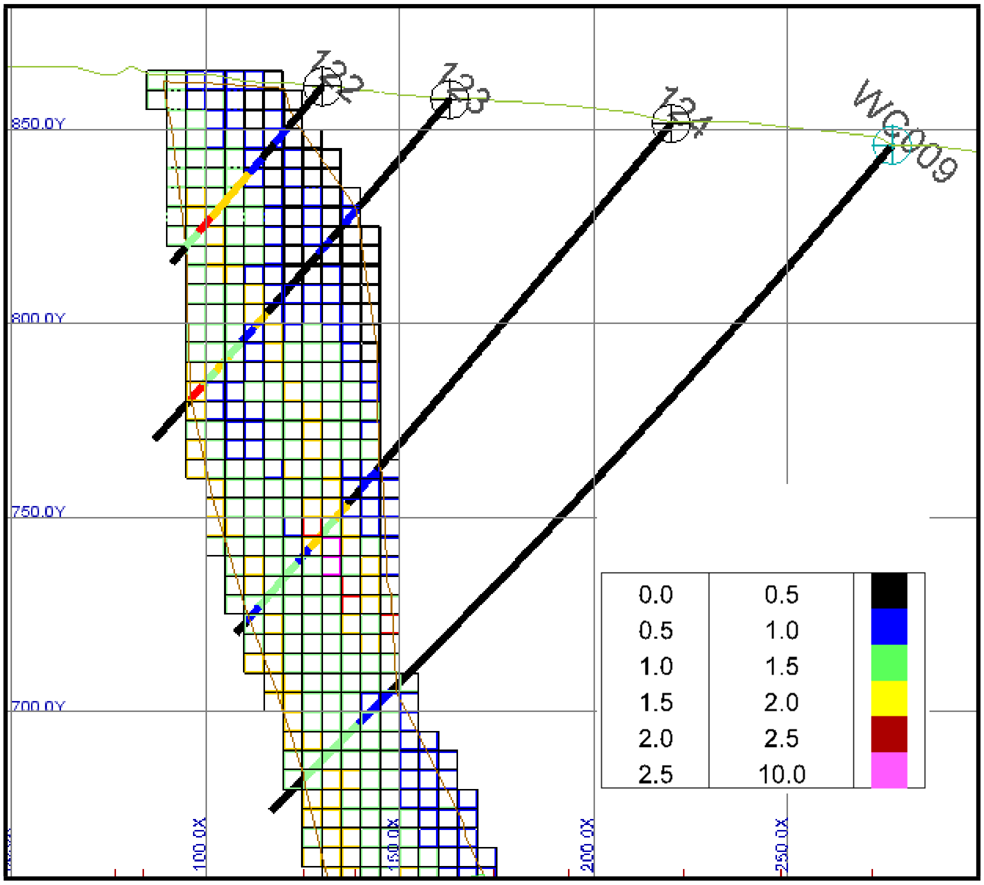


Figure 14-5: Cross Section with Drill Hole Composites Showing Total Cu Against Interpolated Block Model Grades

15 MINERAL RESERVE ESTIMATES

15.1 MINERAL RESERVE

It is the opinion of IMC that the mine/plant production schedules define the mineral reserve for a property. Table 15-1 shows the mineral reserve for the Carmacks property based on the current production schedule. This assumes that measured mineral resource inside the reserve pit is converted to proven mineral reserve and indicated mineral resource inside the reserve pit is converted to probable mineral reserve. The proven and probable mineral reserves amount to 11.6 million tonnes at 0.977% total copper, 0.805% soluble copper, 0.172% non-soluble copper, 0.435 g/t gold, and 4.34 g/t silver.

Table 15-1: Mineral Reserve Estimate

Reserve Category	K tonnes	Tot Cu (%)	Sol Cu (%)	Nonsol Cu (%)	Gold (g/t)	Silver (g/t)
Proven Mineral Reserve Copper (M lbs)	4,127	1.039 94.5	0.851	0.188	0.559	5.39
Probable Mineral Reserve Copper (M lbs)	7,424	0.943 154.3	0.780	0.163	0.365	3.76
Proven /Probable Reserve Copper (M lbs)	11,551	0.977 248.9	0.805	0.172	0.435	4.34
Notes:						
Total material in Reserve Pit		69,957 Ktonnes. Waste to Ore: 5.1				
Reserves are Fully Diluted and Based on a cutoff Grade of 0.18% Recoverable Copper						

IMC does not know of any mining, metallurgical, infrastructure, or other factors that might materially affect the mineral reserve. It is also the opinion of IMC that the resource block model was developed in such a way as to account for potential ore loss and mining dilution, so these mining factors have been accounted for. The mineral reserve is consistent with current CIM and NI 43-101 guidelines.

15.2 DESIGN ECONOMICS AND FLOATING CONE EVALUATION

A floating cone analysis was conducted to guide final pit design and mine phase designs for the mineral reserve estimate. Table 15-2 presents the preliminary economic parameters used in the design. The parameters are based on bulk open pit mining and processing the ore by crushing and heap leaching, followed by solvent extraction and electrowinning (SX/EW) to produce copper cathode at the site.

Table 15-2: Design Economics

	Units	Parameter
Copper Price Per Pound	(US$)	2.50
Mining Cost Per Total Tonne	(US$)	2.00
Process Cost Per Ore Tonne	(US$)	8.43
G&A Cost Per Ore Tonne	(US$)	1.22
Internal Recovered Cu Cutoff	(%)	0.18
Breakeven Recov Cu Cutoff	(%)	0.21
Internal Total Copper Cutoff	(%)	0.22
Breakeven Total Copper Cutoff	(%)	0.26
Note 1: Recovered based on total and soluble copper grades by block . For solcu / totcu $\geq$ 0.79, recovery = 85% of total copper For solcu / totcu > 0.79, recovery = 95% of solcu + 10% of totcu		

The mine design was based on a copper price of $2.50 per pound. For pit modeling, the mining cost was estimated by IMC to be about $2.00 per total tonne. This was based on applying updated equipment operating costs to the estimated operating shifts per year from the 2007 Feasibility Study. This also included current fuel and blasting agent costs. The process and G&A costs were provided by M3 Engineering and were also based on updating costs developed for the 2007 study.

The recovery equation was provided to IMC by Copper North and is as follows:

For Sol Cu / Tot Cu $\geq$ 0.79, recovery = 85% of total copper
For Sol Cu / Tot Cu < 0.79, recovery = 95% of Sol Cu + 10% of Tot Cu

IMC calculated recoverable copper grade as total copper x recovery on a block by block basis and incorporated it into the block model. Table 15-2 shows internal recovered copper cutoff grade as 0.18% copper and breakeven cutoff is 0.21% recoverable copper. Internal cutoff grade covers process and G&A costs, i.e. block routing is at the pit rim with mining as a sunk cost for blocks that have to be mined. Breakeven cutoff also pays for the mining cost for ore (but not for additional waste stripping). The mineral reserve is based on a cutoff grade of 0.18% recoverable copper.

Table 15-2 shows the average recovery to be 81.2%. This is based on the recoverable versus total copper grades and pounds in the mineral reserve. Based on this average recovery total copper cutoff grades are about 0.22% for internal cutoff and 0.26% for breakeven cutoff.

The pit designs also incorporate the slope angle recommendations from the report “Open Pit Slope Design – Carmacks Copper Project” by Golder Associates, dated October 22, 2008. Table 15-3 shows the recommended slope design parameters for each design sector. The inter-ramp slope angles are 52.6^{o} for all sectors. In addition, every 60 to 80 vertical meters a 12m catch bench, instead of an 8m catch bench, is specified. The design is also based on a double-bench configuration, i.e. two 10m benches faced up to a 20m height. Based on the Golder report and the 2007/2008 pit designs IMC estimated overall slope angles, with access roads, would be about 47 degrees on both sides of the pit; this was used for the floating cone runs.

The floating cones were run at copper prices from $3.00 per pound to $1.00 per pound in $0.25 increments. Table 15-4 shows the results based on the fully diluted model. The cutoff grade for the table is based on a 0.18% recovered copper, internal cutoff at the $2.50 copper price, prior to application of dilution. Also, only measured and indicated resources were used to generate the cone shells; inferred resource is considered waste.

The base case cone, at $2.50 per pound copper, contains 11.7 million ore tonnes at 0.799% recoverable copper, 0.977% total copper, and 0.810% soluble copper. Total material in the cone shell is 66.9 million tonnes. Cases 1 through 4, at prices from $3.00 to $2.25 copper are similar in size. The pit size decreases significantly at prices of $2.00 copper and less.

Table 15-3: Summary of Design Sectors and Pit Wall Design Recommendations

Design Sector	Pit Wall	Principal Wall Dip Direction (degrees)	Pit Wall Design Azimuth (degrees)	Bench Face Angle (BFA) (degrees)	Bench Height (metres)	Bench Width (metres)	Inter-ramp angle (IRA) (degrees)	Additional 12-metre wide Catchment Benches (bench elevation in metres)
1	West side of the pit. Footwall Zone.	055°	235°	70°	20	8	52.6°	At 840 metre bench elevation. At 760 metre bench elevation. At 700 metre bench elevation.
2	East Side of the pit. Hanging wall Zone.	235°	055°	70°	20	8	52.6°	None
3	North end of the pit.	105°	285°	70°	20	8	52.6°	At 760 metre bench elevation. At 700 metre bench elevation.
4	North end of the pit.	180°	360°	70°	20	8	52.6°	At 760 metre bench elevation
5	South end of the pit.	280°	100°	70°	20	8	52.6°	None
6	South end of the pit.	330°	150°	70°	20	8	52.6°	None

Table 15-4: Floating Cone Results – Diluted Model

Case	Cu Price ($/lb)	Ktonnes	Rec Cu (%)	Tot Cu (%)	Sol Cu (%)	Sulf Cu (%)	Gold (g/t)	Silver (g/t)	Waste Ktonnes	Total Ktonnes	Waste: Ore
1	3.00	12,027	0.791	0.980	0.801	0.179	0.432	4.32	58,543	70,570	4.9
2	2.75	11,918	0.794	0.983	0.804	0.179	0.434	4.34	57,753	69,670	4.8
3	**2.50**	**11,672**	**0.799**	**0.987**	**0.810**	**0.177**	**0.437**	**4.36**	**55,245**	**66,917**	4.7
4	2.25	11,380	0.802	0.989	0.814	0.175	0.438	4.37	52,146	63,526	4.6
5	2.00	9,409	0.798	0.982	0.811	0.171	0.459	4.50	32,965	42,375	3.5
6	1.75	8,467	0.805	0.989	0.821	0.169	0.459	4.50	25,821	34,288	3.0
7	1.50	7,289	0.819	1.006	0.833	0.174	0.478	4.68	19,096	26,385	2.6
8	1.25	5,675	0.836	1.026	0.848	0.178	0.505	4.97	11,478	17,153	2.0
9	1.00	4,215	0.864	1.058	0.876	0.181	0.530	5.16	6,647	10,862	1.6
Cutoff based on blocks above 0.18% recovered copper prior to application of dilution.											

15.3 RESOURCE BLOCK MODEL

The resource block model used for the project was developed by Dr. Gilles Arseneau, P. Geo. of Wardrop Engineering Inc. (Wardrop) during the 4th quarter of 2007. The model was provided to IMC at that time and has been in the possession of IMC since then. To IMC's knowledge, this is the most recent resource block model for the project.

The main Carmacks Copper ore body is hosted in an elongated structure with a sharp boundary between the ore and waste zones. The block model was based on 5m x 5m x 5m blocks and the percent of the block inside the ore zone was included for each block. For blocks on the perimeter of the deposit separate grades were estimated for the ore and waste portions of the block. For perimeter blocks IMC assumed about 1.5m of lateral dilution or 30% of a 5m block. For blocks with an ore fraction greater than or equal to 70% these were flushed out to full blocks at the weighted average grade of the ore and waste. For blocks less than 70% ore a 30% waste fraction was added at the waste grade.

The dilution calculation resulted in about 11.7% more ore tonnes at a 10.2% lower copper grade. The average grade of the dilution comes to about 0.03% total copper. The mineral reserve is based on blocks that were greater than 0.18% recoverable copper prior to dilution. IMC considers this a reasonable estimate of dilution and it is comparable to what was used for the 2007 Feasibility Study.

16 MINING METHODS

16.1 OPERATING PARAMETERS AND CRITERIA

Mine plans were developed for the Carmacks Copper deposit based on delivering ore to the crusher at the rate of 1,775 ktonnes per year or about 4,860 tonnes per day. The peak total material rate is 13.5 million tonnes per year.

Mining will be conducted on two 12 hour shifts per day for 335 days per year. It was specified that this was 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 is about 6 ½ years after a brief preproduction period.

16.2 PIT AND MINING PHASE DESIGN

Four mining phases were designed for the Carmacks Copper Project. Inter-ramp slope angles are 52.6 degrees, as specified by Golder. The design is also based on 10m mining benches in a double bench configuration for final walls. The main road is 25m wide at a maximum grade of 10%. This will accommodate trucks of approximately 90 metric tonnes 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.

Phase 4 (Figure 16-4) is the small southeast pit.

For the 2007/2008 pit designs IMC kept the roads off the highwall side of the pit. The bench set-backs recommended every 60 to 80 vertical meters in the Golder report significantly reduced the incentive to do this. They specified three set-backs on the highwall side of the pit. With the road on the highwall only one set-back, on about the 840 bench, should be required.

Table 16-1 shows the tonnages by mining phase. As with the cones, the tonnage tabulation is on a diluted basis. The cutoff grade for the table is based on blocks above 0.18% recovered copper, internal cutoff at the $2.50 copper price, prior to application of dilution. Also, only measured and indicated resources are tabulated; inferred resource is considered waste.

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: Ore
1	3,273	0.876	1.073	0.887	0.187	0.570	5.27	7,897	11,170	2.4
2	3,658	0.725	0.890	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

Cutoff based on blocks above 0.18% recovered copper prior to application of dilution.

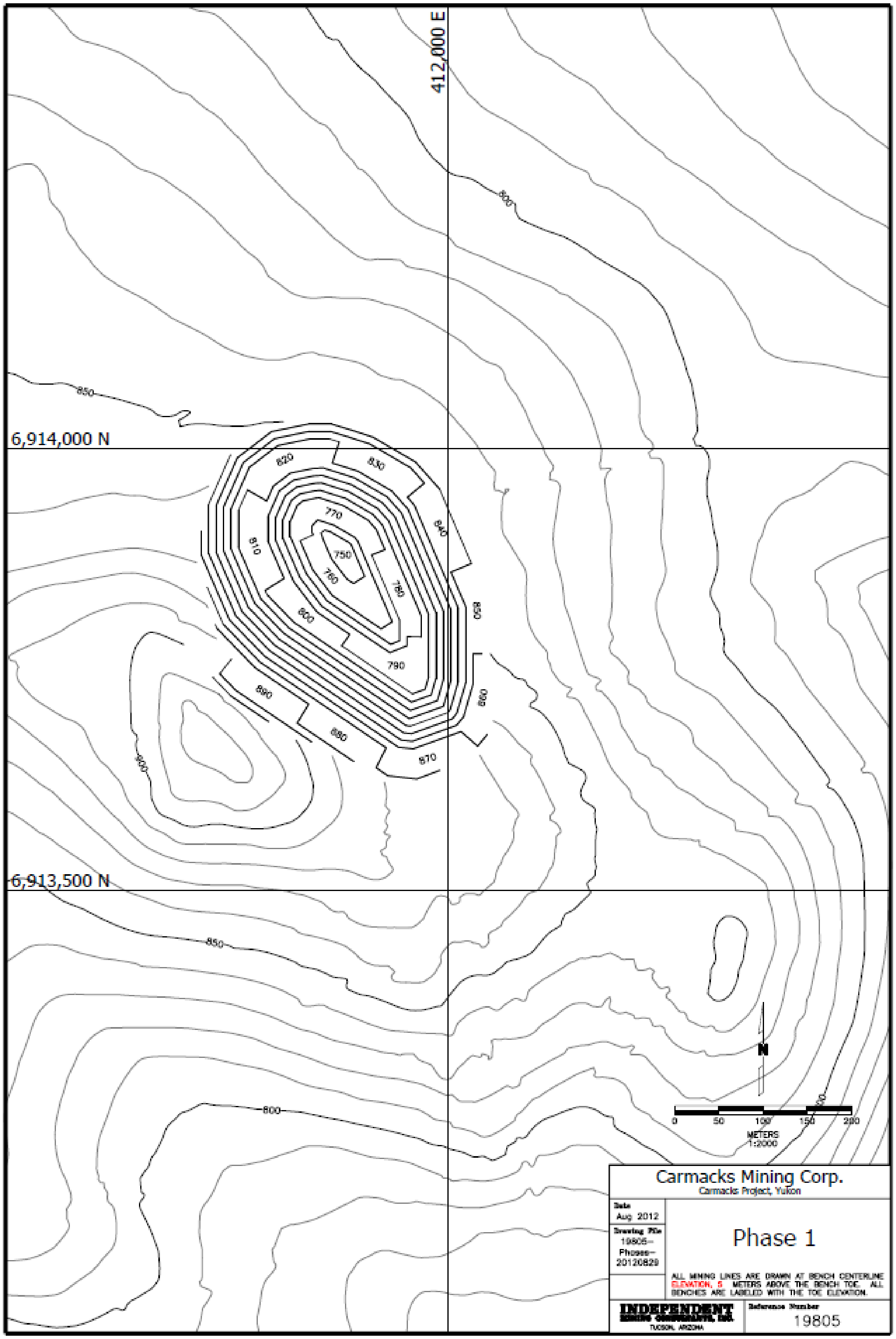


Figure 16-1: Mining Phase 1

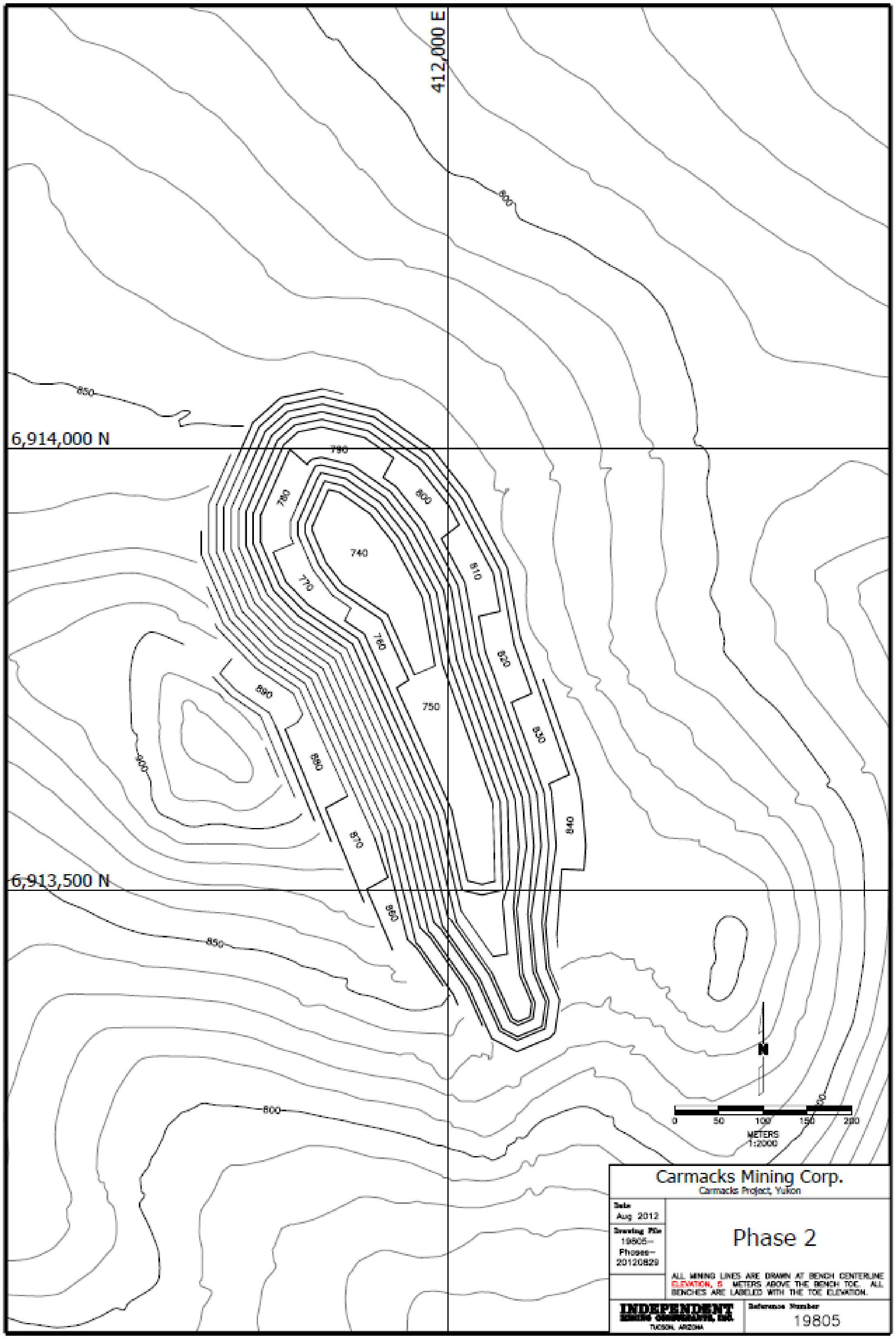


Figure 16-2: Mining Phase 2

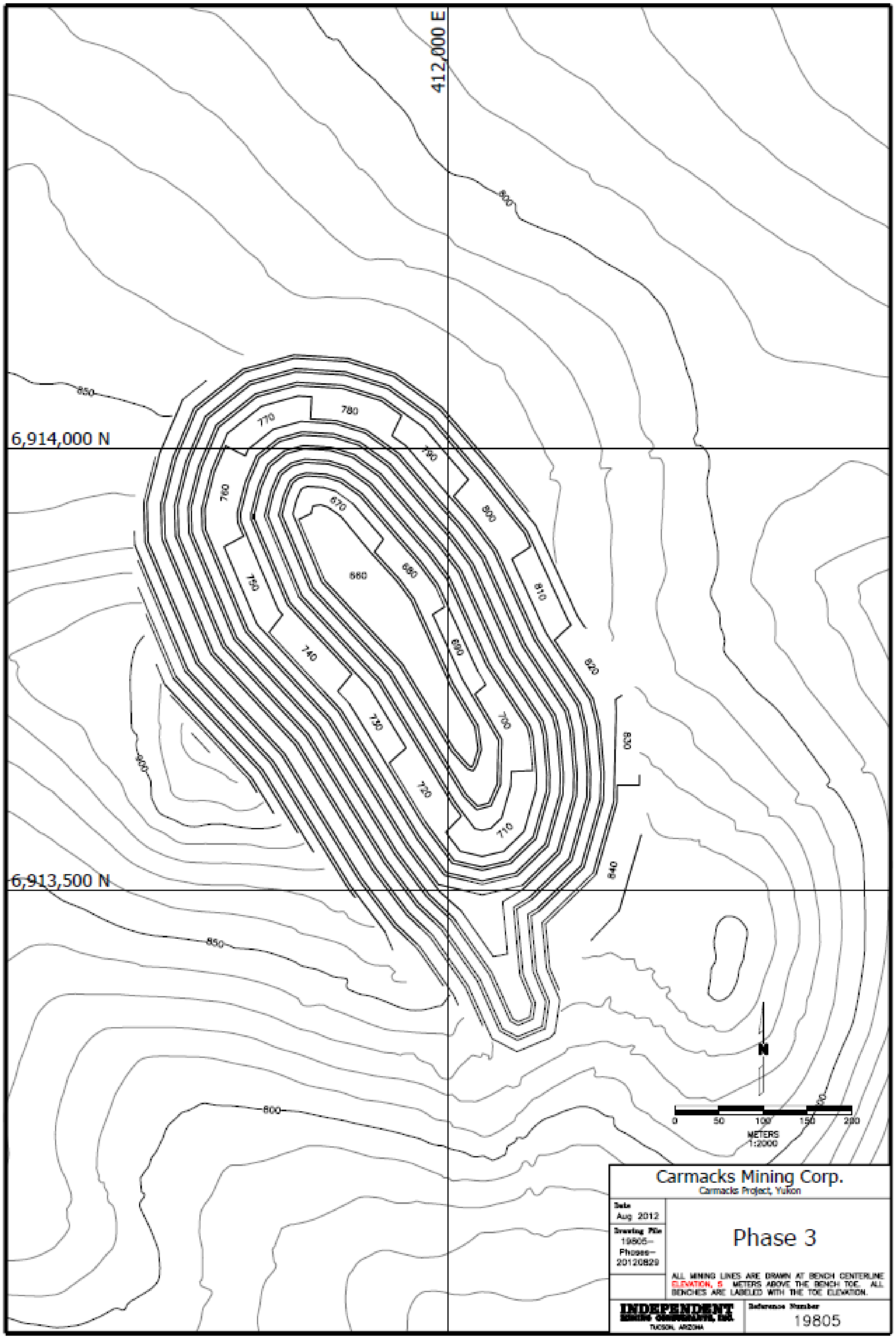


Figure 16-3: Mining Phase 3

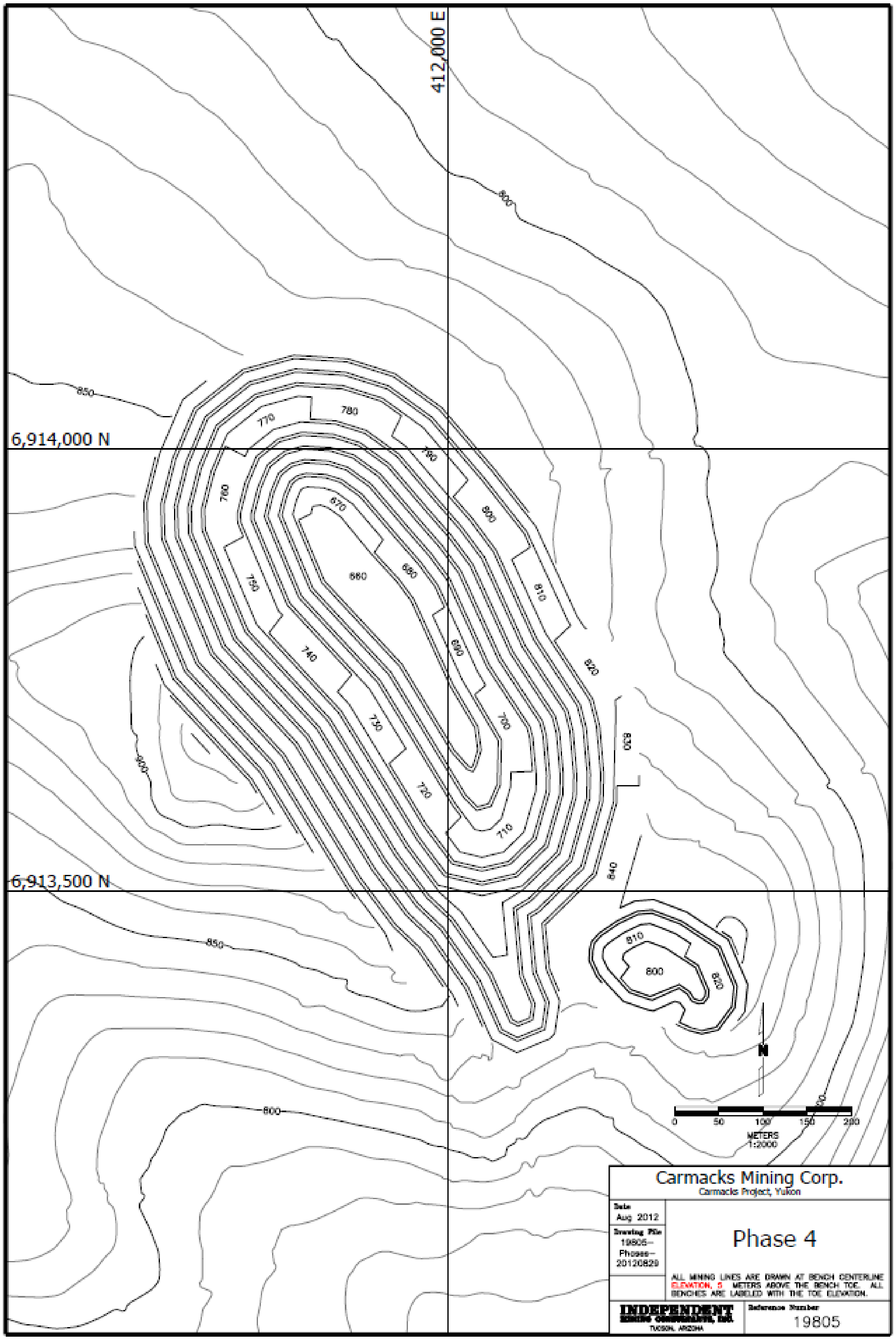


Figure 16-4: Mining Phase 4

16.3 MINE PRODUCTION SCHEDULE

A mine production schedule was developed to estimate annual ore and waste movements from the pit. The upper portion of Table 16-2 shows the mine production schedule. The schedule is based on mining 1,775 ktonnes per year of ore. Total leach ore is 11.6 million tonnes at 0.977% total copper and 0.805% soluble copper which comes to 0.793% recovered copper. This includes the estimated effect of dilution.

Total material is 70.0 million tonnes for a waste to ore ratio of 5.1 to 1. Preproduction is minimal at 953 ktonnes. The total material movement is 9.5 million tonnes during Year 1 and peaks at 13.5 million tonnes for Years 2 through 4. The waste to ore ratio is 6.6 to 1 during these peak years.

The lower portion of the table shows the proposed leach pad stacking schedule. Year 1 leach ore is the sum of ore mined during preproduction and Year 1. The average copper recovery is indicated to be 81.2% based on the recovery equation presented in Section 3.3 applied on a block by block basis.

Table 16-2: Mine Production Schedule and Proposed Leach Pad Stacking Schedule

		PP	1	2	3	4	5	6	7	TOTAL
Mine Production Schedule										
Ore Ktonnes	(k tonnes)	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	0.744	0.709	0.859	0.957	0.793
Total Copper	(%)	0.867	0.965	0.932	1.019	0.907	0.869	1.065	1.204	0.977
Soluble Copper	(%)	0.691	0.802	0.756	0.839	0.777	0.733	0.864	0.943	0.805
Sulfide Copper	(%)	0.176	0.164	0.176	0.180	0.130	0.136	0.201	0.261	0.172
Gold	(g/t)	0.306	0.472	0.411	0.490	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	5.40	4.34
Total Ktonnes	(k tonnes)	953	9,500	13,500	13,500	13,500	11,776	5,821	1,407	69,957
Waste Ktonnes	(k tonnes)	803	7,875	11,725	11,725	11,725	10,001	4,046	506	58,406
Waste to Ore Ratio	(none)	5.4	4.8	6.6	6.6	6.6	5.6	2.3	0.6	5.1
Proposed Leach Pad Stacking Schedule:										
Ore Ktonnes	(k tonnes)		1,775	1,775	1,775	1,775	1,775	1,775	901	11,551
Recovered Copper	(%)		0.784	0.752	0.828	0.744	0.709	0.859	0.957	0.793
Total Copper	(%)		0.957	0.932	1.019	0.907	0.869	1.065	1.204	0.977
Soluble Copper	(%)		0.793	0.756	0.839	0.777	0.733	0.864	0.943	0.805
Sulfide Copper	(%)		0.165	0.176	0.180	0.130	0.136	0.201	0.261	0.172
Average Recovery	(%)		82.0%	80.7%	81.3%	82.0%	81.6%	80.7%	79.5%	81.2%

16.4 WASTE ROCK STORAGE AREAS

Figure 16-6 shows the final mine site layout for the Carmacks Copper Project. The main waste storage area is north of the pit and was designed to contain 57.5 million tonnes of waste. It is constructed in 20m lifts, at angle of repose, with a 20m set-back between lifts to make the overall angle about 2.33H:1V.

In addition, 882 ktonnes of waste is placed in the small southeast pit. The crusher is located south of the pit and the leach pad (not shown) is west of the pit.

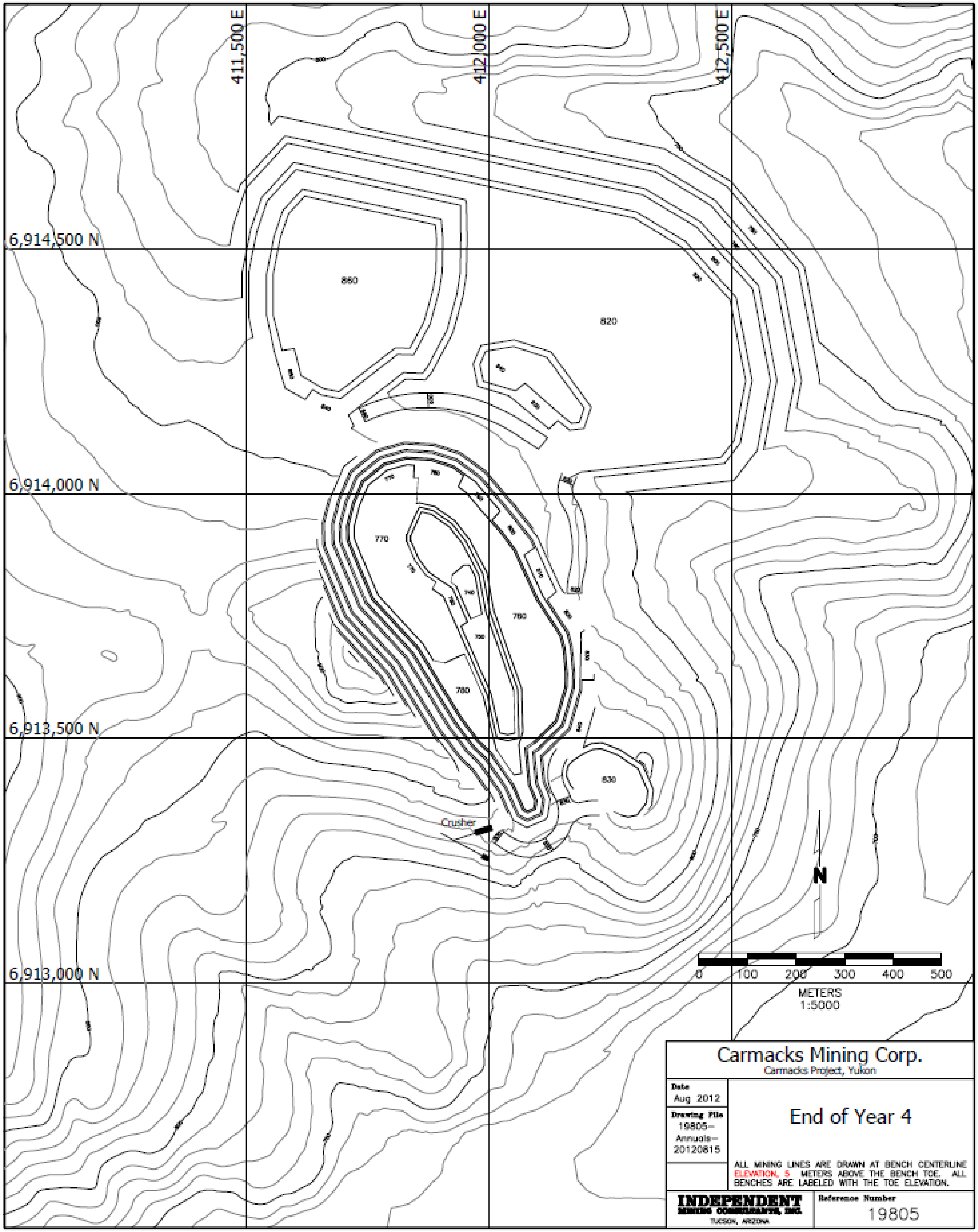


Figure 16-5: End of Year 4

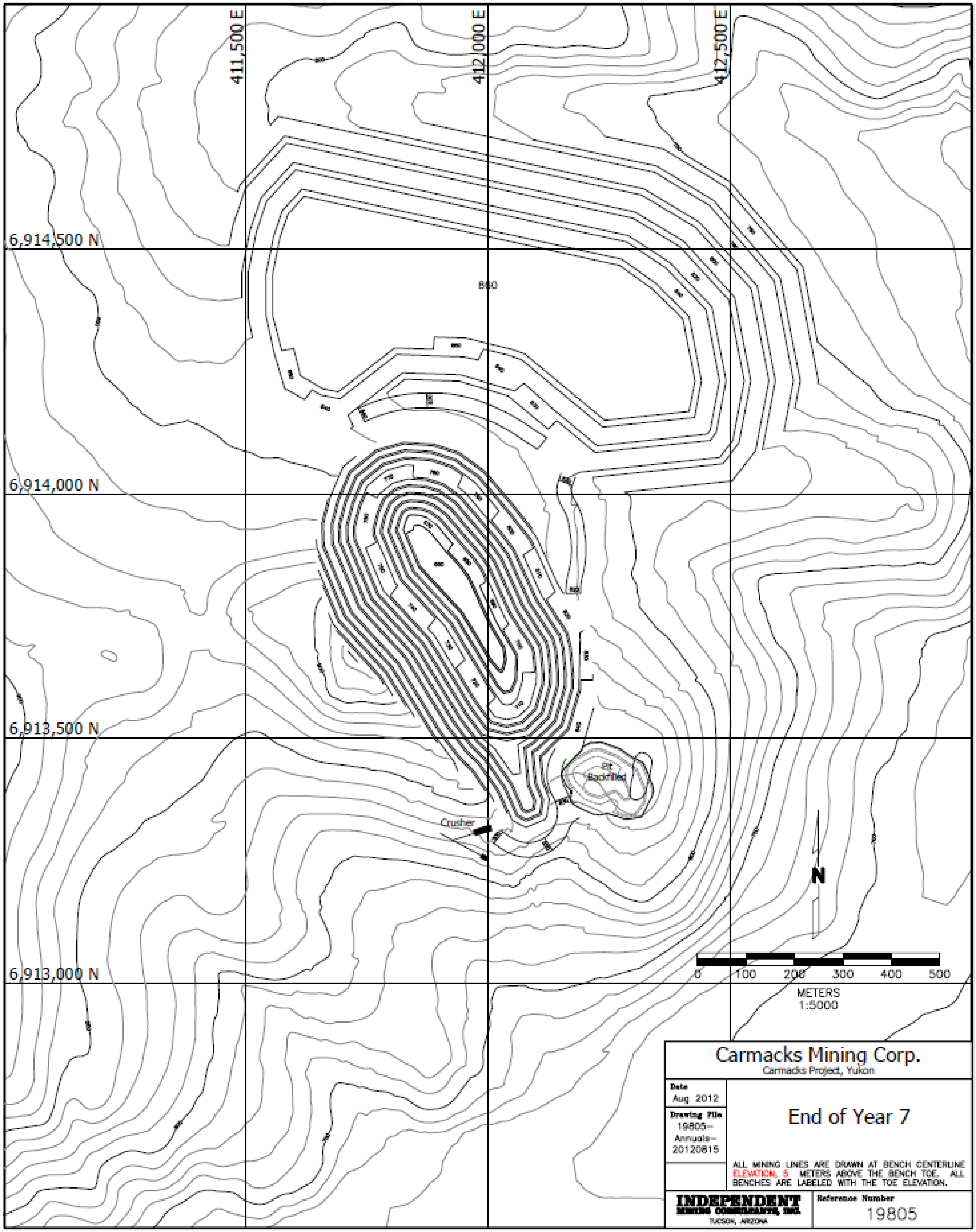


Figure 16-6: End of Year 7

16.5 MINING 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 owner operation and assumes a well-managed mining operation with a well-trained labor pool, and that all the equipment is new at the start of mining.

Table 16-3 shows major equipment requirements by time period for Carmacks.

Table 16-3: Mine Major Equipment Fleet Requirement

	Capacity/	Time Period								
Equipment Type	Power	PP	1	2	3	4	5	6	7	8
Caterpillar MD6240 Drill	(210 mm)	1	1	2	2	2	2	1	1	0
Komatsu PC2000 Hyd Shovel	(11 cu m)	0	1	1	1	1	1	1	1	0
Cat 992K Wheel Loader	(10.7 cu m)	1	1	1	1	1	1	1	1	0
Cat 777F Truck	(90 mt)	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 cu m)	1	1	1	1	1	1	1	1	0
TOTAL		11	15	17	18	18	18	15	12	0

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 ore 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 required for construction or operation of the plant and leach pad facilities.

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

	Mine Material Movements					Mine Operations Schedule						
Time Period	Ore (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	0.0%
TOTAL	11,551	2,445	55,961	150	70,107	2,346		4,524	4,524	54,288		

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

	(min)	(hr)
Scheduled Time Per Shift	720	12.00
Less Scheduled Nonproductive Times		
Travel Time/Shift Change/Blasting	15	0.25
Equipment Inspection	0	0.00
Lunch/Breaks	60	1.00
Fueling, Lube, & Service	0	0.00
Net Scheduled Productive Time (Metered Operating Time)	645	10.75
Job Efficiency Based on 50.0 Productive Minutes/Hour	83.3%	83.3%
Net Productive Operating Time Per Shift	538	8.96
Overall Mine Efficiency Factor	74.65%	74.65%

16.6.3 Material Characteristics

Table 16-6 summarizes the material characteristics used for equipment productivity calculations. In-situ bulk densities are 2.64 tonnes per cubic meter for ore, 2.66 tonnes per cubic meter for waste rock and about 2 tonnes per cubic meter 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 90mpa or about 13,000psi compressive strength. These were reported to be mostly in granodiorite wall rock which is probably stronger than the ore. IMC assigned a moderate strength index to waste rock and weak index to the ore.

Table 16-6: Material Characteristics

Parameter	Units	Leach Ore	Over Burden	Waste Rock	Ore Rehand
BULK DENSITY:					
Dry Bank Density	(mt/cu m)	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	(mt/cu m)	1.89	1.54	1.90	1.82
Wet Loose density	(mt/cu m)	1.94	1.62	1.96	1.87
MATERIAL STRENGTH:					
Strength Index (1-5)	(none)	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?	(none)	yes	yes	yes	no
NOTES:					
Strength Index: 1=very strong, 2=strong, 3=moderate, 4=weak, 5=very weak,6=not drilled/blasted					

Description of Strength Index

IMC Index	Brown Index	Description
1	R6	Specimen can only be chipped with a geologic hammer
2	R5	Specimen requires 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.

16.7 DRILLING

The drilling fleet consists of diesel powered drills with a pulldown of about 50,000 pounds 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 2m of subgrade drilling.

Shift productivities are estimated at 24,230 tonnes for ore and 17,184 tonnes for waste rock. Productivity in overburden is estimated at 40,602 tonnes per shift. Annual production is estimated at 12.4 million tonnes per drill for ore, 8.8 million tonnes per drill for waste rock, and 20.8 million tonnes for overburden.

The productivity calculations are based on a powder factor of 200 grams per tonne for ore, 250 g/t for waste rock, and 100 g/t for overburden. Drill penetration is estimated at 0.75 m/min for ore, 0.6 m/min for waste rock, and 1 m/min for overburden. The table also shows the spacing between holes is about 6m in ore, 5.5m in waste rock, and 8m 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 preproduction and Year 1 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 in the Roads and Dumps cost center.

Table 16-7: Penetration Rate and Peak Drilling Rate by Material Type

Caterpillar MD6240 Drill					
	Units		Leach Ore	Over Burden	Waste Rock
Hole Depth	(m)		12	12	12
Penetration Rate	(m/min)	*	0.75	1.00	0.59
Penetration Time Per Hole	(min)		16.1	11.9	20.2
Move Time	(min)	*	5.00	5.00	5.00
Pipe Length	(m)		12.80	12.80	12.80
Steel Changes	(none)	*	0	0	0
Time Per Steel Change	(min)	*	2.00	2.00	2.00
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

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 Ore	(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
Ore 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 Ore	(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
Ore Rehandle	(shifts)	0	0	0	0	0	0	0	0	0	0
Total Shifts	(shifts)	42	498	719	753	751	655	309	67	0	3,794
PRODUCTIVITY CALCULATIONS:											
Available Shifts Per Period	(shifts)	168	670	670	670	670	670	670	336	0	4,524
Mechanical Availability	(%)	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	0.0%	85.0%
Utilization of Availability	(%)	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	0.0%	90.0%
Maximum Utilization Per Drill	(%)	76.5%	76.5%	76.5%	76.5%	76.5%	76.5%	76.5%	76.5%	0.0%	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	1.40	1.47	1.47	1.28	0.60	0.26	0.00	
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%	0.0%	55.0%
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 Shift			Drl/Blst
Leach Ore	(t//shift)	24,230	yes
Overburden	(t/shift)	40,602	yes
Waste	(t/shift)	17,184	yes
Ore Rehandle	(t/shift)	52,852	no

16.8 LOADING

The primary loading fleet is based on hydraulic shovels with an 11 cubic meter bucket, such as the Komatsu PC2000 shovel, and wheel loaders with a 10.7 cubic meter bucket, 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 tonnes for rock and 13,209 tonnes for overburden. Annual production per shovel is estimated at 8.1 million tonnes for rock and 6.8 million tonnes for overburden. The loader shift productivity is estimated at 11,783 tonnes for rock and 10,051 tonnes for overburden. Annual production per loader is estimated at 6.0 million tonnes for rock and 5.2 million tonnes for overburden.

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 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 preproduction, 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 Grade

	Flat (kph)	Up (kph)	Down (kph)	Acl/Dcl (kph)
Loaded	45	10	21	10
Empty	45	24	39	10
Ramp Gradient				10.0%
Accel/Decel Distance			(m)	50

The first and last 50m of each profile was considered as acceleration/deceleration at an average speed of 10 kph. Table 16-12 shows that, life of mine, the productivity of the Cat 777 trucks is estimated at 3,659 tonnes per truck shift for a 12 hour shift.

Table 16-10: Shovel Requirements – Komatsu PC2000 Hyd Shovel (11 cu m)

		Units	PP	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	TOTAL
PERCENT LOADED BY SHOVEL:												
Leach Ore	60.0%	(%)	0.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	
Overburden	60.0%	(%)	0.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	
Waste	60.0%	(%)	0.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	
Ore Rehandle	0.0%	(%)	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	
LOADED BY SHOVEL:												
Leach Ore		(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
Ore 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 Ore		(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
Ore 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		(%)	0.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	0.0%	85.0%
Utilization of Availability		(%)	0.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	0.0%	90.0%
Maximum Utilization Per Shovel		(%)	0.0%	76.5%	76.5%	76.5%	76.5%	76.5%	76.5%	76.5%	0.0%	76.5%
Available Shifts Per Shovel		(shifts)	0	513	513	513	513	513	513	257	0	3,332
Fractional Number of Shovel		(none)	0.00	0.72	1.02	1.01	1.01	0.88	0.43	0.21	0.00	
Actual Number of Shovels		(none)	0	1	1	1	1	1	1	1	0	
Fleet Utilization		(%)	0.0%	55.0%	78.1%	77.0%	77.1%	67.1%	33.2%	16.0%	0.0%	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 Ore	(t/shift)	15,710
Overburden	(t/shift)	13,209
Waste	(t/shift)	15,710
Ore Rehandle	(t/shift)	15,710

Table 16-11: Loader Requirements – Cat 992K Wheel Loader (10.7 cu m)

		Units	PP	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	TOTAL
PERCENT LOADED BY LOADER:												
Leach Ore	40.0%	(%)	100.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	
Overburden	40.0%	(%)	100.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	
Waste	40.0%	(%)	100.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	
Ore Rehandle	100.0%	(%)	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	
LOADED BY LOADER:												
Leach Ore		(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
Ore 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 Ore		(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
Ore 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		(%)	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	0.0%	85.0%
Utilization of Availability		(%)	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	0.0%	90.0%
Maximum Utilization Per Loader		(%)	76.5%	76.5%	76.5%	76.5%	76.5%	76.5%	76.5%	76.5%	0.0%	76.5%
Available Shifts Per Loader		(shifts)	129	513	513	513	513	513	513	257	0	3,461
Fractional Number of Loadersl		(none)	0.67	0.66	0.91	0.89	0.90	0.78	0.39	0.19	0.00	
Actual Number of Loaders		(none)	1	1	1	1	1	1	1	1	0	
Fleet Utilization		(%)	51.0%	50.7%	69.4%	68.5%	68.5%	59.7%	29.5%	14.2%	0.0%	60.0%
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 Loader Production Per Shift		
Leach Ore	(t/shift)	11,783
Overburden	(t/shift)	10,051
Waste	(t/shift)	11,783
Ore Rehandle	(t/shift)	11,783

Table 16-12: Truck Requirements – Cat777F Truck (90mt)

	Units	PP	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	TOTAL
PRODUCTION REQUIREMENTS:											
Leach Ore	(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
Ore 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	4,524
Mechanical Availability	(%)	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	0.0%	85.0%
Utilization of Availability	(%)	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	0.0%	90.0%
Maximum Utilization Per Truck	(%)	76.5%	76.5%	76.5%	76.5%	76.5%	76.5%	76.5%	76.5%	0.0%	76.5%
Available Shifts Per Truck	(hours)	129	513	513	513	513	513	513	257	0	3,461
Fractional Number of Trucks	(none)	1.95	4.45	6.05	7.08	6.51	7.01	4.55	2.47	0.00	
Actual Number of Trucks	(none)	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%	0.0%	73.6%
NUMBER OF OPERATORS:											
Number of Mining Crews	(none)	2	3	3	3	3	3	3	3	0	
Number of Truck Operators	(none)	4	12	15	18	15	18	12	6	0	
Ton(ne)s Per Truck Shift	(ton(ne)s)	3,804	4,229	4,355	3,720	4,045	3,276	2,494	2,218	0	3,659

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 liter (1 unit).
- Excavator, 1.9 m^3 (1 unit).
- Drill, 140mm (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 RECOVERY METHODS

17.1 PROCESS PLANT DESCRIPTION

The Carmacks Copper Project will be developed as an open-pit mine with an acid heap leach and a solvent extraction/electrowinning (SX/EW) process facility producing, on average, approximately 13,200 tonnes of LME Grade A cathode copper annually. Figure 1-4 is a simplified process flow sheet.

The mining operation is designed to produce an average 1.775 million tonnes of ore per year or approximately 28,400 tonnes (ore and waste) per day on a seven day per week, 24 hours per day operation. The mine will be operated year round but may temporarily suspend ore stacking operations when winter temperatures are extreme.

The mine will use a conventional spread of mining equipment, the main units comprising 10.5 cubic meter hydraulic excavators, 11.5 m^3 loaders and 91-tonne haul trucks.

Ore will be hauled by truck and dumped directly into the primary crusher, from where it will be conveyed to secondary and tertiary crushers. The final product will have a maximum size of 19 mm and a P80 of 13 mm. The crushed product will first be agglomerated with sulphuric acid and water and then conveyed by a series of overland (grasshopper) conveyors to a lined valley fill leach pad where it will be placed by means of a radial stacker.

An Events Pond is located down gradient from the leach pad to provide capacity for an emergency drain down of the pad and to manage the plant water balance during various storm events.

The crushed ore on the leach pad will be irrigated with dilute sulphuric acid to leach copper from the ore. Pregnant leach solution will be collected and pumped to the solvent extraction plant where the dissolved copper in the solution will be concentrated. This concentrated solution passes to the electrowinning plant where the dissolved copper is plated onto cathodes. Copper is stripped from the cathode and is then transported to market.

Sulphuric acid is produced on site by means of a 131 tonne per day sulphuric acid plant. The plant will burn sulphur which will be transported to site in liquid form. Storage tanks will be provided for liquid sulphur to accommodate potential supply interruptions and for the concentrated acid to accommodate variations in demand for acid and allow for plant maintenance shutdowns.

Storage, mixing, and distribution are provided for other process reagents such as diluent, extractant, guartec, and cobalt sulphate.

The SX plant will consist of three mixer-settlers, two for extraction of the copper from the PLS (aqueous phase) into an organic phase containing an extractant reagent, and one for stripping the copper from the organic phase into a strong acid solution (the electrolyte).

The two extraction mixer settlers will operate in series with the aqueous and organic flowing counter to one another. In extraction, the transfer of copper from the PLS to the organic will be accompanied by the transfer of an equivalent amount of acid from the organic aqueous. The aqueous phase after extraction of most of the copper is called the raffinate. The raffinate will be pumped back to the heap to leach more copper.

In the single strip unit, the loaded organic will be contacted with a strong acid solution causing the extraction reaction to reverse and transferring the copper from the organic to the electrolyte while an equivalent amount of acid transfers from the electrolyte to the organic. The acid contained in the electrolyte is generated in the electrowinning cells. From the stripper settler the organic will return to the second of the two extraction mixer settler, while the electrolyte (the rich electrolyte) enriched in copper advances to the electrowinning cells in the tankhouse.

A direct electric current is passed through the cells causing the copper to plate out onto permanent stainless steel cathode blanks, and generating acid at the anode. From the tankhouse, the (lean) electrolyte returns to the strip mixer settler. The copper will be harvested on a weekly basis. Copper produced by this process, LME grade A, will be weighed and bundled into 2 to 3 tonne packages for sale on the world market.

Other facilities at site will include the following:

- A Truck Shop providing adequate space for the maintenance of two 91-tonne trucks and associated warehousing.
- An Administration building.
- A Laboratory facility.
- An Operations Camp to accommodate non-local workers.
- A Gatehouse/First Aid post.

The layout of the site process facilities are shown on Figure 17-1.

Utilities

CNMC anticipates Yukon Energy Crop. (YEC), the regional electrical utility company, will serve the mine from an existing Carmacks-Stewart 138 kV transmission line along the existing Klondike Highway. A new substation (tap-off) in the vicinity of McGregor Creek would feed an 11-kilometer 34.5 kV transmission spur line to the mine's main substation terminating on a dead-end structure. The schedule for completion of this spur line is the third quarter of 2015 which fits well with the present schedule for the development of this project. CNMC has a secure right-of-way for the power line from McGregor Creek to the site and is in discussions with YEC over terms of a future power supply agreement (PPA).

Total project electrical load is estimated to be about 10 megavolt-amperes (MVA). The mine is not a significant electrical power consumer, as all of the major mining equipment is proposed to be diesel powered.

Total fresh make-up water required varies depending on the precipitation but is expected to peak at a monthly average of about 27 m^3/hr. Approximately 45 m^3/day of potable water will be required and the remainder will go to the process. Potable water will be produced by means of a packaged treatment plant. Mine road watering will average about 190 tonnes per day, but that quantity is assumed to come from collected runoff and mine water infiltration.

Fresh water supply wells will be located in the bedrock-confined aquifer underlying the Williams Creek drainage. Each well will have the capacity to produce about 150 m^3/day of fresh water.

The fire water requirement is 280 m^3/hr. for two hours. This requirement will be satisfied by providing a dedicated fire reserve capacity of 560 m^3 in the lower portion of the fresh and firewater tank.

	ELEV.	CUT	FILL	DIFF.
PLANT	768.0	98,180	3,200	94,980
TRUCK SHOP	836.0	22,390	6,980	15,410
CRUSHER	832.0	19,130	17,640	1,490
CAMP	734.0	60,780	43,770	17,010
ROADS	VARIES	62,410	208,450	−146,040
TOTAL		262,890	280,040	−17,150

NOTE:
VOLUMES CALCULATED ARE IN CU. M. USING A 1.5H:1V CUT SLOPE AND 2H:1V FILL SLOPE.

PRELIMINARY
NOT FOR CONSTRUCTION

Figure 17-1: Site Grading Plan

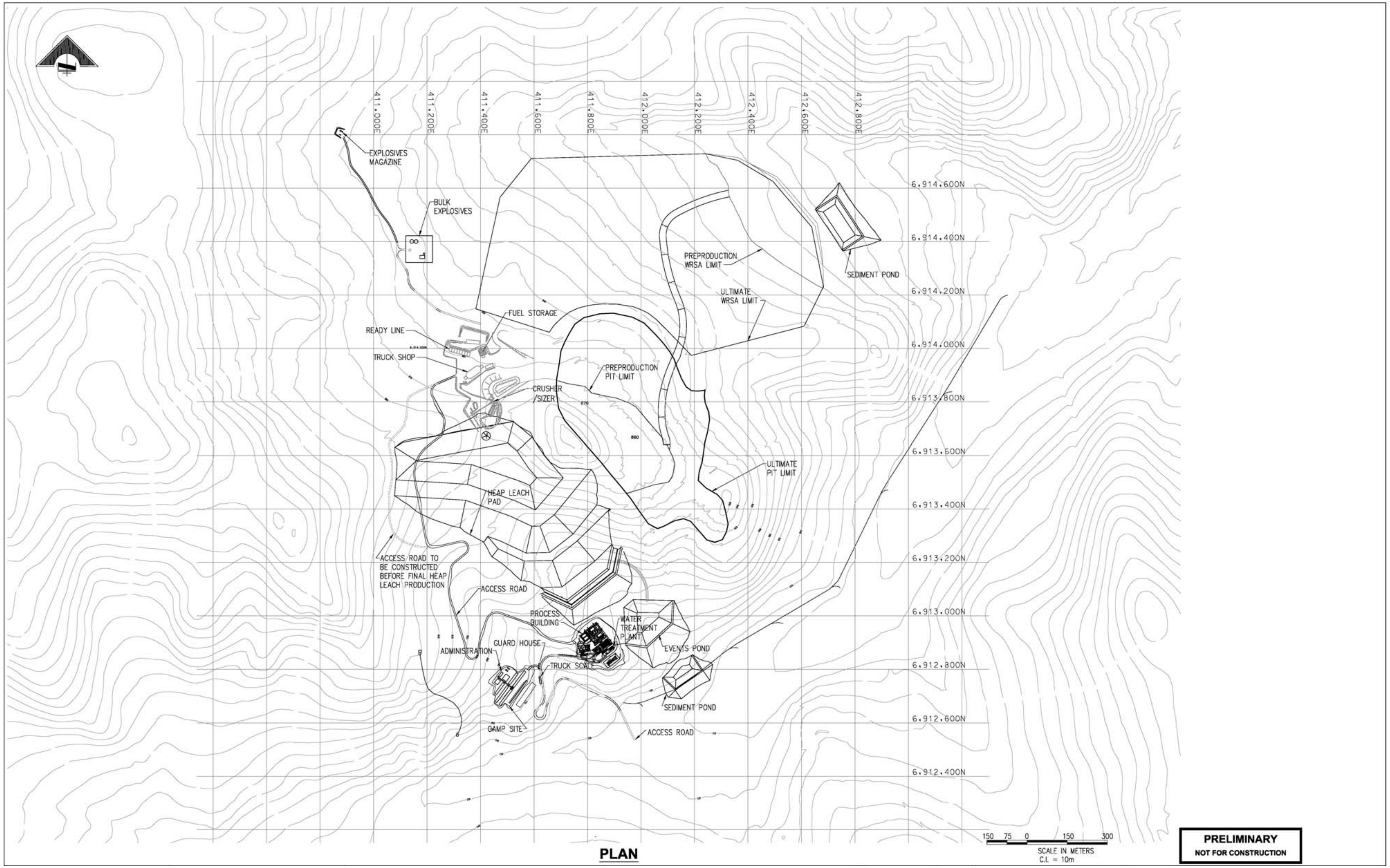


Figure 17-2: Overall Mine Site Plan

18 PROJECT INFRASTRUCTURE

The Project site is currently accessible by an existing 12 km exploration road that leads north from km 33 of the secondary, government maintained, unpaved roadway (Freegold Road) from Carmacks. A small airfield used by private aircraft exists near Carmacks.

The village of Carmacks lies on the Klondike Highway, a paved highway, 175 km north of Whitehorse which provides the main transportation link in the Yukon. Whitehorse has an international airport with daily flights to Vancouver.

Situated 180 km south of Whitehorse by paved road is the year-round port of Skagway Alaska. A narrow gauge railroad from Skagway to Whitehorse (Yukon & White Pass Route railway) has not operated commercially for several years. Skagway currently provides port facilities for cruise ships taking tourists to Yukon and Alaska. In the past it has accommodated facilities for the shipping of concentrate from Faro and other mines. Currently, Capstone Mining Corporation is shipping concentrate from this facility.

The nearest operational rail head is at Fort Nelson BC, approximately 1,200 km by paved road from Carmacks.

18.1 SITE LAYOUT AND ANCILLARY FACILITIES

Project facilities will lie on either side of a central hill with a 926 m elevation. The open pit mine will occupy more than 29 ha in an area lying to the northeast of the crest with the south-western pit rim nearly reaching the crest. The mine waste rock storage area (WRSA) will cover nearly 70 ha just north of the mine.

A small valley southwest of the central hill, of which the hill forms the northeast valley slope, will be filled with ore to form the leach heap. The heap will reach nearly to the summit of the hill on the southwest slope.

An earthen embankment at the southeast end of the heap will provide structural support for the heap. The leach heap and embankment combined will cover about 38.2 ha. A spill and runoff control collection pond termed the "Events Pond" will be built directly downhill from the heap leach. The Events Pond and its embankment will cover about 4.9 ha. A second pond, covering 1.9 ha will collect sediment from treated discharge before being released to Williams Creek.

The metallurgical recovery plant area will occupy somewhat more than 2.0 ha and will lie just south of the embankment.

An ore crushing plant and agglomerator will be constructed between the southern end of the pit and the heap leach embankment. The crusher will be fed either directly by mine haul trucks or by a front-end loader from a day pile adjacent to the crusher. A series of conveyor belts will carry ore from the agglomerator to the heap. The day pile, ore crushing, and agglomeration facility will cover some 2.7 ha.

18.2 HEAP LEACH FACILITY

18.2.1 Heap Leach Pad

The proposed 33 hectare heap leach pad will operate as a modified valley fill, with internal solution storage, and is designed to contain up to 13.3 million tonnes of ore at an assumed dry density of 1.7 tonnes/m³. The maximum elevation of the Heap Leach Pad is designed to be 898m. The ore will be crushed to 80% minus 13 mm, and placed on the heap leach pad in 8 m lifts using a system of overland and grasshopper conveyors and a stacking conveyor at a rate of approximately 1.78 Mtpa. A dilute sulphuric acid solution will be applied via a drip irrigation system with drip emitters plowed into the heap surface to a depth of approximately 1 m. Solution will be collected in the high permeability overliner unit at the base of the heap pad and from two collection systems located at the sides of the heap. Perforated collection pipes will be placed in the overliner unit to increase solution removal rates. The entire facility will be graded such that leach solution will drain to a collection sump at the toe of the confining embankment. The solution will then be pumped from the sump through a vertical riser to either the process plant or the Events Pond.

The heap leach development will make use of inter-lift liners at a maximum of every three lifts (24 m) to expedite the flow of PLS through the heap while allowing lower layers to commence drain-down once leaching in those layers is complete.

The heap leach pad will be constructed in stages throughout mine operation, with complete containment of leaching solutions at each stage of development.

18.2.1.1 Confining Embankment and Retaining Berms

An engineered earth confining embankment will be constructed at the toe of the heap leach pad and retaining berms will be constructed along the perimeter of the pad where the pad is founded on steep ground. These structures are required in order to provide stability to the heap leach pad, and to contain the leaching solution. The confining embankment will have a crest elevation of 808 m, maximum crest height of 32 m, crest width of 6 m, and a crest length of approximately 440 m, with an upstream slope of 3H:1V and a downstream slope of 2.5H:1V.

In the event of an emergency or other unforeseen circumstance a spillway connecting the Heap Leach Pad to the Events Pond will prevent overtopping of the confining embankment.

The retaining berms will have a variable crest elevations, but constant height of 8 m, crest widths of 3 m, and lengths of approximately 200 m and 50 m, with an upstream and downstream slopes of 3H:1V.

18.2.1.2 Surface Settlement and Stability

Operation of the heap leach pad is expected to generate heat throughout the life of mine due to the exothermic reaction of the sulphuric acid liberating the copper from the ore. Initial estimates predicted sustained elevated temperatures resulting in a depth of thaw beneath the heap leach pad of approximately 16 m within the first year, and complete thawing to 35 m (inferred maximum depth to bedrock) within approximately four to five years. Thawing of the foundation is expected to result in a decrease in strength of the foundation soils and thaw consolidation of the soils, resulting in surface settlements. However, recent thermistor readings indicate warming of the area below the proposed heap leach pad, attributed to clearing of vegetation from the area.

Ground surface settlements of up to 1.7 m can be expected under the heavily loaded areas of the heap leach pad where depths to bedrock of up to 35 m exist. In areas of more shallow bedrock, settlements of 0.3 to 0.6 m can be expected. The range in predicted settlements indicates that the differential settlements across the heap leach pad will be within acceptable limits for support of a composite liner and collection piping system. Foundation improvements under the heap leach pad are not considered necessary. The anticipated thaw related settlements are further reduced due to clearing of vegetation and associated ground heating and thaw of frozen ground.

The slope stability of the heap leach pad is governed by block-sliding type failures along the geomembrane of pad liner system. The general arrangement of the pad, including bench widths, has been arranged to provide adequate stability of the heap. Foundation drains will be installed beneath the footprint area of the pad to facilitate groundwater removal beneath the liner and to reduce generation of excess pore pressures beneath the heap.

18.2.1.3 Pad Liner System Design

The liner design for the base of the heap leach pad consists of a double composite liner with a continuous leak detection and recovery system (LDRS) across the lower portion of the facility where there is a potential for solution to accumulate and pond. The liner design for the upper portion of the pad, where solution is not expected to accumulate or pond to any significant height consists of a single composite liner with a LDRS beneath the collection piping.

The LDRS will be subdivided into zones and monitored for both fluid quality and quantity. The layout of the LDRS cells and the piping system design in the overliner will allow for contingent operation of the HLF if flow rates in select zones of the LDRS exceed acceptable limits. The LDRS will be designed to capture and convey any fluid that penetrates through the overlying composite liner system to monitoring and removal points (sumps). The collection trenches will be isolated from the underlying soil and natural groundwater system by the secondary geomembrane liner. PLS will be collected in the high permeability overliner layer at the base of the heap leach pad. The proposed overliner will consist of a 1-m thick layer of 25 mm minus crushed ore. Perforated collection pipes will be placed within the overliner to increase solution removal rates. Portions of the heap that may exceed 70 m in height will require HDPE SDR 17 class of pipe to avoid excessive pipe crushing. The remaining piping will be perforated, corrugated polyethylene piping. The PLS solution will be removed from the heap leach pad via submersible pumps in two stainless steel risers, each approximately 1 m in diameter, and placed within a sump at the upstream toe of the confining embankment.

18.2.2 Events Pond

The Events Pond is designed for short term storage of PLS during upsets within the plant or during large precipitation events for containment of a combination of PLS and contact storm water runoff from the heap. The structure will provide a storage capacity of 160,000 m^3 which has been demonstrated by the site specific water balance model to be adequate for all reasonably foreseeable events. The ultimate embankment configuration will have a crest elevation of 750 m, a crest width of 6 m, maximum crest height of 27 m, and an approximate crest length of 295 m, with upstream and downstream slopes of 3H:1V. In the unlikely event of an emergency or other unforeseen circumstance in which solution levels exceed the maximum design flood or storage capacity, discharge of excess water would be conveyed through the spillway in a controlled manner in order to avoid overtopping and damage to the embankment or liner.

The liner design for the Events Pond consists of a double composite liner system with continuous LDRS, with the primary geomembrane expected to remain exposed, and therefore subject to freeze-thaw conditions and thermal expansion and contraction. During operation in cold weather, to reduce the stress on the liner system resulting from sudden thermal expansion caused when relatively warm solution from the extraction plant or HLF is discharged into the pond, a minimum volume of solution will be maintained within the pond to act as a thermal buffer.

Foundation drains will be installed beneath the footprint area of the Events Pond to facilitate groundwater removal beneath the liner and to control generation of excess pore water pressures.

Construction and operation of the Events Pond are expected to have a lesser effect on thawing of the foundation soils compared to the Heap Leach Pad, due to the reduced ground surface temperature compared with the heap. The choice of sequence of construction of the Events Pond will affect the amount of strain that the liner may experience. Preliminary liner design has been carried out considering currently available geosynthetic products and based on performance and constructability criteria.

18.3 WASTE ROCK STORAGE AREA (WRSA)

The WRSA has been designed based on the guidelines set out in the B.C. Ministry of Energy, Mines and Petroleum Resources document for the “Investigation and Design of Mine Dumps, Interim Guidelines, May 1991”. The design is based on a projected capacity of 70 million tonnes of waste rock and testing to date suggests that the rock is not acid generating or metal leaching. The waste rock, would be 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. 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 area where vegetative covers are required at closure. The perimeter surface water ditches would 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 would 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 would 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 would 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 2 lifts. As the second lift nears completion, the western half of the footprint will be developed also. 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 800m. 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%.

18.4 POWER SUPPLY AND DISTRIBUTION

18.4.1 Power Supply

Electrical power for the project will be provided by Yukon Energy Corp. (YEC) via a new 11 km long, 34.5 kV overhead power line connecting a new 138/34.5 kV sub-station at the tap off point at McGregor Creek on the existing Carmacks-Stewart 138 kV grid to the mine-site 34.5/4.16 kV sub-station. YEC’s scope will include the tap-off sub-station and the 34.5 kV spur line. CNMC will provide the capital for the design, permitting, and construction of these facilities.

YEC has expressed interest in providing CNMC up to 10 MW of power under the current schedule of rates for industrial users in the Yukon. As the project progresses, YEC and CNMC will enter into a Power Purchase Agreement (PPA). Consistent with YEC’s policy, this PPA will also include provision to recover the capital invested in the existing Carmacks-Stewart 138 kV grid extension already in service as a monthly or annual charge to CNMC operations. If at the cessation of CNMC operations any capital recover balance exists, this balance will be paid out by CNMC.

18.4.2 Project Power Distribution

The total project electrical load is estimated to be less than 10 megavolt-amperes (MVA). The power is consumed by the crushing, agglomeration, and stacking system and the heap leach, SX/EW facilities. The mining operations contribute a comparatively minor portion of the total electrical load as all mining equipment is fuelled by diesel oil or Liquefied Natural Gas (LNG).

The main sub-station, provided by the project, will be located adjacent to the processing facilities. The sub-station will consist of a single main oil filled pad mounted (of mobile type), copper wound transformer, 10/13.3 MVA (OAFA); 34.5 kV delta incoming primary to 4,160/2400V grounded wye connected secondary voltage configuration along with associated gang operate air disconnect switches, SF6 insulated circuit breakers, neutral grounding resistor, grounding and lightning protection systems.

A mine in-plant electrical distribution system consists of multiple 4160 V, 3 phase, medium voltage feeders routed both underground in buried duct banks, and overhead pole lines using wooden poles and cross-arms to support un-insulated overhead line conductors to distribute electrical power to the different mine process areas. At each mine process area, smaller pad-mounted, oil-filled transformers will convert the 4160 V to the Canadian standard 600 V, 3 phase, 3 wire industrial utilization low voltage supply. The 600 V (low voltage) feeders and branch circuits will be derived from low voltage motor control centers (MCC's) and/or switchgear located in pre-fabricated electrical buildings. In addition, larger horsepower induction motor loads, depending on the type of motor drive controllers required by the process, will be served directly from 4160 V, 3 phase, 3 wire source using medium voltage MCC's and/or switchgear.

19 MARKET STUDIES AND CONTRACTS

19.1 MARKETS

No market study has been performed for this project and CNMC has not yet entered into any discussion with potential consumers regarding off-take agreements. However, LME Grade A cathode copper is a readily marketable commodity at prevailing copper prices. As such, no market study is deemed necessary.

19.2 CONTRACTS

As of the date of this study, CNMC has not entered into any contracts for the development of this project, for the purchase of supplies and services or for the sale of any product. However, to the maximum extent possible, all estimates of costs used in this study have been benchmarked against prevailing industry rates.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 ENVIRONMENTAL SETTING

The Carmacks Copper Project area lies within the Klondike Plateau and is part of the Pelly River Ecoregion (Oswald et. al. 1997), 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.

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. Approximately 3,400 ha were surveyed in 2006 and 97% was in forest cover. Black spruce is the dominant forest community type, covering 58% of the area surveyed, with Lodgepole pine (20%) and White spruce (17%) each approximately equally represented. Trembling aspen forest covers just 2% of the area surveyed. Willow fen (2%) and grassland (1%) also were 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. 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. In general, the project is located in a low density moose survey block in which moose occur year round in low numbers. The project area occurs outside the known range of Wood Bison with no known permanent occupancy in the area. 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.

The Little Salmon Carmacks First Nation has developed a fish and wildlife management plan for their traditional territory (LSCFN 2009). The plan identifies the need to protect the Yukon River between Tatchun Creek and Minto as this area is considered 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.

20.1.2 Aquatic Resources

20.1.2.1 Surface Water Hydrology

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

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

Flows were monitored on Williams and Nancy Lee creeks periodically between 1991 and 1994, and have been monitored annually from 2006 to the present. 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 freshet flow of 3.4 m^3/sec (293,000 m^3/day) (Golder 2012).

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, Waste Rock Storage Area, and Heap Leach Facility. 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 2012). 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. 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 upper Williams Creek (Golder 2012).

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 proposed HLF area ranges from 12 m to 65 m, and in the proposed Waste Rock Storage Area (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.

20.1.2.3 Surface Water Quality

Background surface water quality on and around the Carmacks project site has been extensively characterized. The current monitoring program has operated from 2005 to the present, 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.

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.

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. Exceptions include cadmium, which occasionally exceeds the CCME guideline of 0.000017 mg/L at several locations, and aluminum, copper, and iron, which typically exceed the applicable guidelines (Al, 0.1 mg/L; Cu, 0.003 mg/L; and Fe 0.3 mg/L) at all locations.

Given these background conditions, Site Specific Water Quality Objectives have been developed for aluminum (0.66 mg/L), copper (0.032 mg/L for hardness between 120 and 180 mg $CaCO_3$/L), and iron (1.1 mg/L) in the Williams Creek watershed. The SSWQOs for aluminum and iron were developed using the Background Concentration Procedure (BCP; Minnow 2008; CCME 2003). The SSWQO for copper based on the BCP is 0.004 mg/L (Minnow 2008), a marginal increase over the CCME criterion. Site-specific toxicity testing was conducted to support application of the Water Effect Ratio Procedure (WERP; CCME 2003) which resulted in a SSWQO for Cu of 0.032 mg/L for a water hardness range of 120 to 180 mg/L (Minnow 2009).

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. 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 with respect to 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; August 1992; Oct 2005; June 2006; July/August 2006; September 2006). Fish occur in the section of Williams Creek below the confluence with Nancy Lee Creek but have not been found at any other location in Williams Creek or Nancy Lee Creek. No fish have been found in the reach of Merrice Creek where the access road crossing is located. Species found in lower Williams Creek include juvenile Chinook salmon, arctic grayling, slimy sculpin, longnose sucker, burbot, and northern pike. Lower Williams Creek therefore is considered to provide rearing habitat for fish during the open water season.

20.1.2.6 Species at Risk

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

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

The project area does not provide critical habitat to any life stage of these species.

20.2 CURRENT LAND USES

20.2.1 Commercial and Industrial

The Carmacks Copper property is comprised of 338 claims, 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 HLF site. The CNMC exploration camp is currently in care and maintenance.

There is no commercial forest harvest activity in the project area due primarily to the low timber values and distance from markets. The property is located within Registered Outfitting Concession #13. The holder of this concession has indicated the project area is not generally hunted.

20.2.2 Traditional and Cultural Land and Resource Use

The property is located within both the Little Salmon Carmacks First Nation (LSCFN) and Selkirk First Nation (SFN) Traditional Territories.

The late summer/fall Chinook and Chum salmon spawning runs on the Yukon River support important 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 property is located within Registered Trapline #147. 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 land LSC R17-B 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

An archaeological impact assessment was conducted in the Williams Creek Valley for the proposed project by Antiquus Archaeological Consultants Ltd. (AAC) in August 1992. AAC also conducted “An Archaeological and Heritage Resource Overview Assessment of the Proposed Carmacks Copper 138 kV Transmission Line Project Route Options Near Carmacks, Yukon Territory” in September 1994. No archaeological sites were identified within the areas proposed for the open pit mine, leach pads and waste rock dumps. However, two historic archaeological sites were identified and recorded during the 1992 assessment. One at the confluence of Williams Creek and one of its tributaries about 1.25 km from the Yukon River, and a second on the Yukon River approximately 1.25 km from the mouth of Williams Creek. These sites are known and documented and will not be disturbed.

There are three locations near the proposed mine access road considered to have medium heritage site potential. One large medium heritage site potential area is located on both sides of Crossing Creek between the bridge over the creek on the existing Freegold Road and the turnoff to the mine access road. The remains of prehistoric or historic camps may be located in this area. The other two medium heritage site potential areas are located where the mine access road crosses Merrice and Williams Creeks.

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 130 ha cleared for the mine site and 12 ha cleared for the access road), 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.

The effects of the groundwater withdrawals, site development, open pit dewatering, and open pit filling on groundwater flows reporting to Williams Creek were examined in a FEFLOW model (Golder 2012 gw). Groundwater withdrawals for operations taken from the two planned groundwater wells are estimated to reduce baseflow in upper Williams Creek by about 15% through the 8 years of operations. Open pit development and operation will not start to have an effect on groundwater flows reporting to Williams Creek until year 5, when the pit bottom begins to intersect the bedrock aquifer. The effect of the pit on stream baseflow is not limited to the operations period, and continues through the period of pit filling, increasing incrementally to peak at an estimated 18% reduction of baseflow in year 10, and slowly declining to a 14% reduction in year 30, a 10% reduction by year 50, 6% by year 100, and 5% by year 200. There is a short period in which groundwater well pumping and pit dewatering/initial filling together have a cumulative effect, with aggregate reductions in baseflow of 23% to 31% in years 8 through 10.

These estimated reductions in baseflow translate to 1 to 2% reductions in steady state flow. These expected reductions in steady state flow would not be measureable and are not expected to significantly affect fish or the quality of fish habitat in lower Williams Creek.

Three sources of contact water will be managed during operations: groundwater seepage and precipitation pumped from the open pit sump; seepage and runoff from the waste rock storage facility; and seepage, runoff, and pregnant leachate solution (PLS) in the HLF. The HLF will be managed as a no-discharge facility. All surface runoff, PLS run through, and seepage from previously leached lifts will be collected in the Events Pond and directed back into the process stream. Operation of the HLF is therefore not expected to have an effect on water quality in Williams Creek.

Pit water will be produced from early in pit development to the end of mining at the start of year 7. Pit water will consist almost entirely of precipitation until year 5, when the pit bottom begins to intercept groundwater. Pit water will be pumped to the Heap Leach Facility Settling Pond (HLFSP), which also collects seepage and runoff from the Waste Rock Storage Area via the Waste Rock Storage Area Sediment Pond (WRSASP). Water quality in the HLFSP will be monitored and water can also be directed from the HLFSP to the water treatment plant if this monitoring indicates that treatment is required. The GoldSim water quality model indicates that discharges from the HLFSP during operations will not adversely affect water quality, with parameter concentrations meeting the proposed SSWQOs at monitoring station W12, upstream of the confluence with Nancy Lee Creek and upstream of any fish habitat in Williams Creek.

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 waste rock stockpile, groundwater seepage from the pit lake, and discharge from the HLF through the passive treatment system. The site-wide water quality model predicts that water quality at station W12 in upper Williams Creek, above the confluence with Nancy Lee Creek will meet the SSWQOs under average, 100 year wet, and 100 year dry annual precipitation conditions

Development, operations, closure, and post-closure conditions will not adversely affect water quality in Williams Creek.

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 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 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, with all project elements located on Registered Trapline #147. CNMC is working with the RTL holder to ensure access to lines is maintained, portions of lines 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 physical evidence of the project 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 Little Salmon Carmacks First Nation (LSCFN), the Selkirk First Nation (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 Yukon government 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.

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.

Once the assessment is complete and a positive decision (i.e. an approval) is issued by YESAB, the regulatory phase of permitting can be completed. Yukon Energy, Mines and Resources 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. The Yukon Water Board does not review a WUL license application until the YESAA process is complete and a decision document issued.

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.

20.6 SCHEDULE

The development of the project is highly dependent upon the issue of appropriate permits. Funding is unlikely to be made available before issue of the key permits and as noted above, a Quartz Mine Licence is required before construction can commence. A Type A Water Licence will be required during construction and prior to operation of the mine. Although no firm date is available for the issue of permits, CNMC is targeting late Q3 2013 for completion of screening under YEA and YESAA, leading to issue of the QML in Q4 2013 and early 2014 for the Water Licence.

Accordingly, CNMC plans to initiate basic engineering in Q3 of 2013, concentrate on project planning and bidding, evaluating and making conditional awards for key long lead equipment purchases and contracts. Only under special circumstances, such as to avoid overall schedule slippage, will full release of a purchase order or contract for fabrication or mobilisation be given prior to having permits in place and receipt of full project release.

Assuming permits are granted as targeted, purchase orders will be released for fabrication early 2014 and mobilisation for construction will begin as soon as weather conditions are appropriate in Q2 2014.

The 2014 construction season will focus on mine pre-stripping, the development of the leach pad confining embankment and the first stage of the leach pad, followed by other earthworks and concrete foundations. The target will be to have buildings closed in before winter.

Mechanical, electrical and instrumentation work will continue through winter inside the buildings. Once weather permits, in Q3 2015 the first stage of the leach pad liner and overliner will be completed along with the lining of the events pond. Construction will be substantially complete by the end of Q4 2015. Assuming pad loading and acid production commenced towards the end of Q4 2015, the first cathode copper is planned for Q1 2016.

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).

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

- Heap Leach Facility (HLF);
- Open Pit;
- Water Rock Storage Area (WRSA); and
- The Process Plant.

Site facilities are presented in Figure 20-1 and a process flow diagram for the proposed water management plan is provided in Figure 20-2. 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 can be discretized into three principle mine phases. These phases are discussed below.

20.7.1 Phase 1: Operations

Ore will be mined at the Project and copper will be extracted in the HLF for a period of seven years. This period is referred to as the operations phase. During operations, the pregnant leach solution (PLS) collected from the HLF will be pumped to the process plant. The Events Pond, located downstream of the HLF is designed to collect non-contact water within the HLF catchment, seepage through the HLF liner and overflow from the HLF in upset conditions. Water collected in the Events Pond is pumped back to the Process Plant to supplement freshwater demands. The Events Pond has an overflow spillway. The Events Pond was designed to store below the spillway invert water resulting from the combination of 1) the normal operating volume, 2) a 10 year snowmelt, 3) a 100 year 24hr storm event and 4) the HLF drain down. No discharge is expected from this facility during operations (Golder 2012b).

Water originating from the WRSA is collected in the waste rock storage area sedimentation pond (WRSASP). Water from this facility and the open pit will be pumped to the heap leach facility sedimentation pond (HLFSP) (Figure 20-1), if the quality of these drainages is acceptable for discharge. The water in the HLFSP will be reclaimed to the process plant to supplement freshwater demands or released to Williams Creek (Figure 20-2). Water draining from the open pit and the WRSASP will be monitored during operations and as noted above, if the quality of these drainages is acceptable for discharge, it will be pumped directly to the HLFSP. Conversely, if monitoring results indicate the water cannot be discharged, a contingency has been built into the water management plan to treat WRSASP and/or open pit water prior to pumping to the HLFSP (Figure 20-2). Water in the HLFSP can also be pumped to the treatment plant, in the event that routine monitoring of this facility indicates that its water quality is not suitable for discharge.

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 Events Pond and the WRSASP to the open pit to prevent discharges from these facilities; and
- Water in the Events Pond may be pumped to the treatment plant to maintain the designed operating levels.

Following the cessation of mining, pit dewatering will be discontinued and pit flooding will commence.

20.7.2 Phase 2: Closure

The initiation of the Project closure period corresponds to the shutdown of the process plant at the end of Year 7 of operations, when decommissioning of site facilities commences. Water management during this stage is similar to Phase 1; however pumping of the Events Pond and the HLFSP to the process plant will cease. Events Pond water will be pumped to the treatment plant and subsequently to the HLFSP. Discharge to Williams Creek from the HLFSP is expected during freshet and summer months. Runoff and seepage from the WRSASP will continue to be pumped to the HLFSP during closure phase and the open pit will continue to flood.

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 closure phase while the passive treatment system is implemented. This phase was considered to have duration of at least three years; however, in practice, it will continue until such time as it can be proven that water may be discharged directly from the WRSASP and the HLFSP without active treatment.

20.7.3 Phase 3: Post-Closure

Post closure will begin when the HLF water quality and the quality of water reporting to the two sediment ponds meet the site end of pipe (EOP) effluent quality standards (EQSs) that will be developed for the site. For the purpose of the water balance model this time is estimated to be at the start of mine year 11.

During the post-closure period, no active water management will occur. Pumping of water from the WRSASP will cease and drainage from this area will drain directly to north Williams Creek. Seepage and runoff from the closed HLF will report to the passive treatment system that will be developed following decommissioning of the Events Pond. The HLF flow will pass through Iron Terraces to a Surge Pond, then into a Biochemical Reactor (BCR) and through an aerobic treatment wetland to the HLFSP. Outflows from the HLFSP will drain to Williams Creek.

The open pit filled 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 2012a). There will be no surface water discharge form the open pit during the post-closure period. Seepage from the pit will drain towards Williams Creek. For the current assessment, it was assumed that all seepage would daylight in Williams Creek upstream of monitoring location W12 (Figure 20-3).

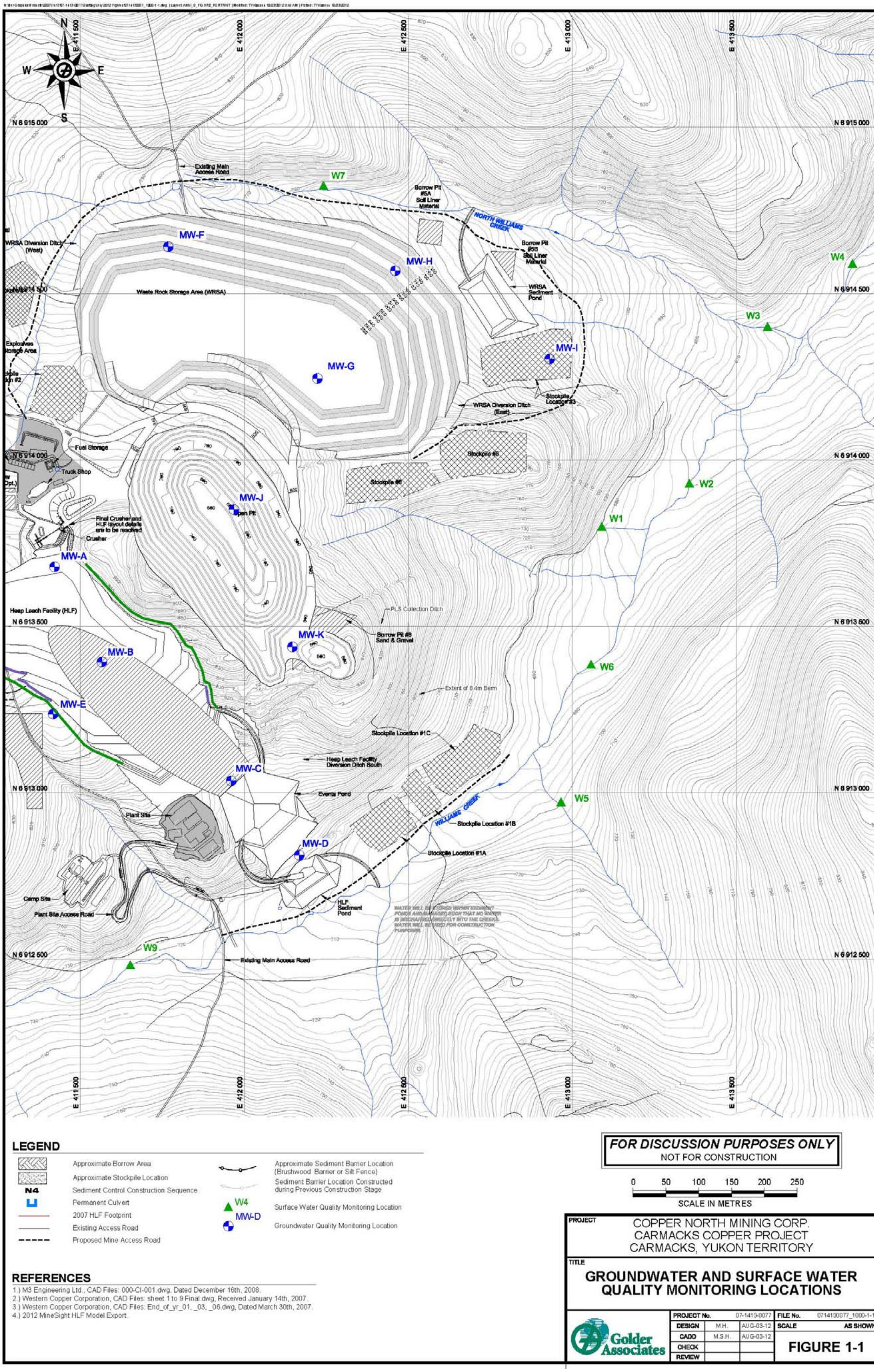


Figure 20-1: Groundwater and Surface Water Quality Monitoring Locations

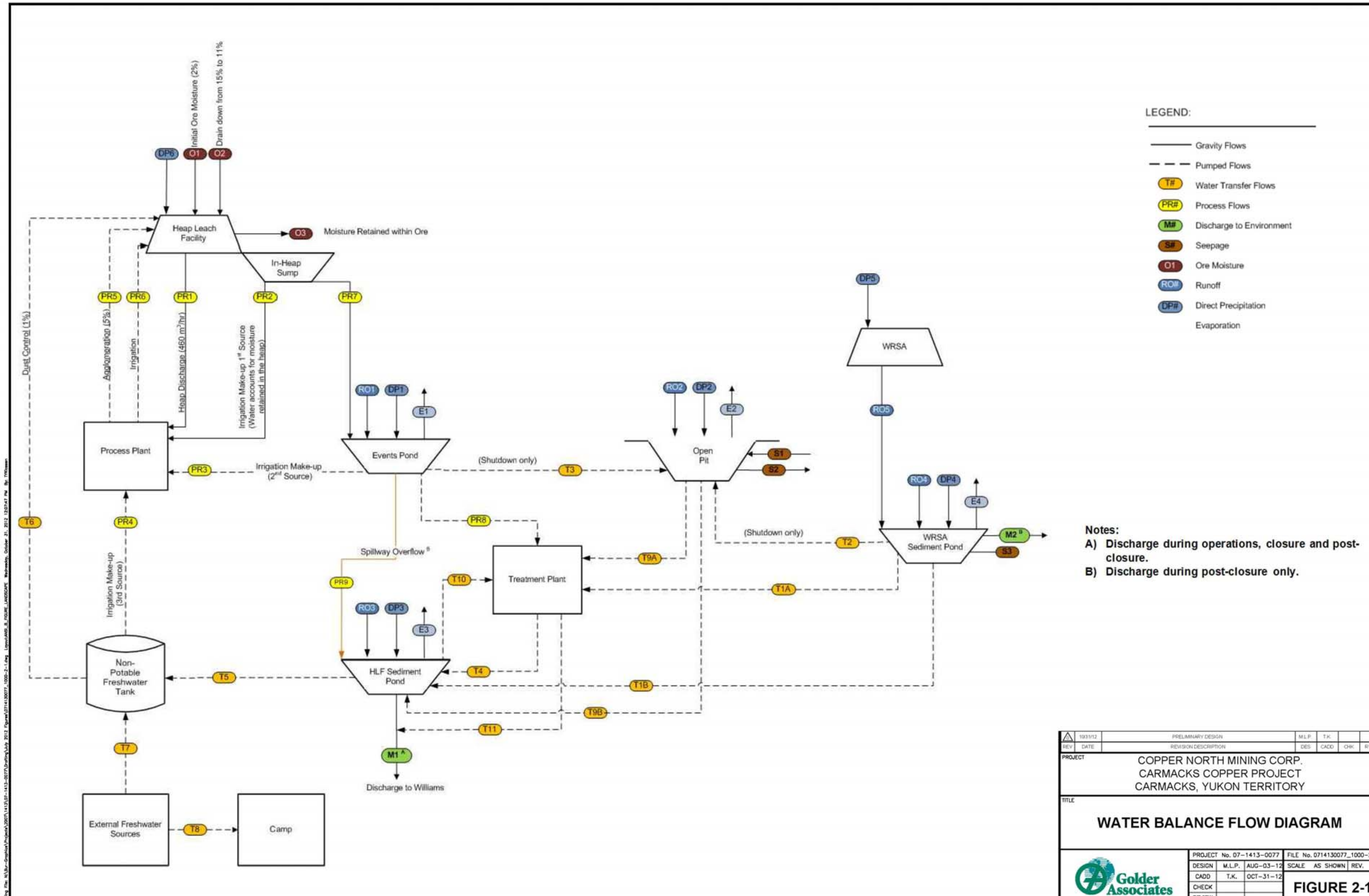


Figure 20-2: Water Balance Flow Diagram

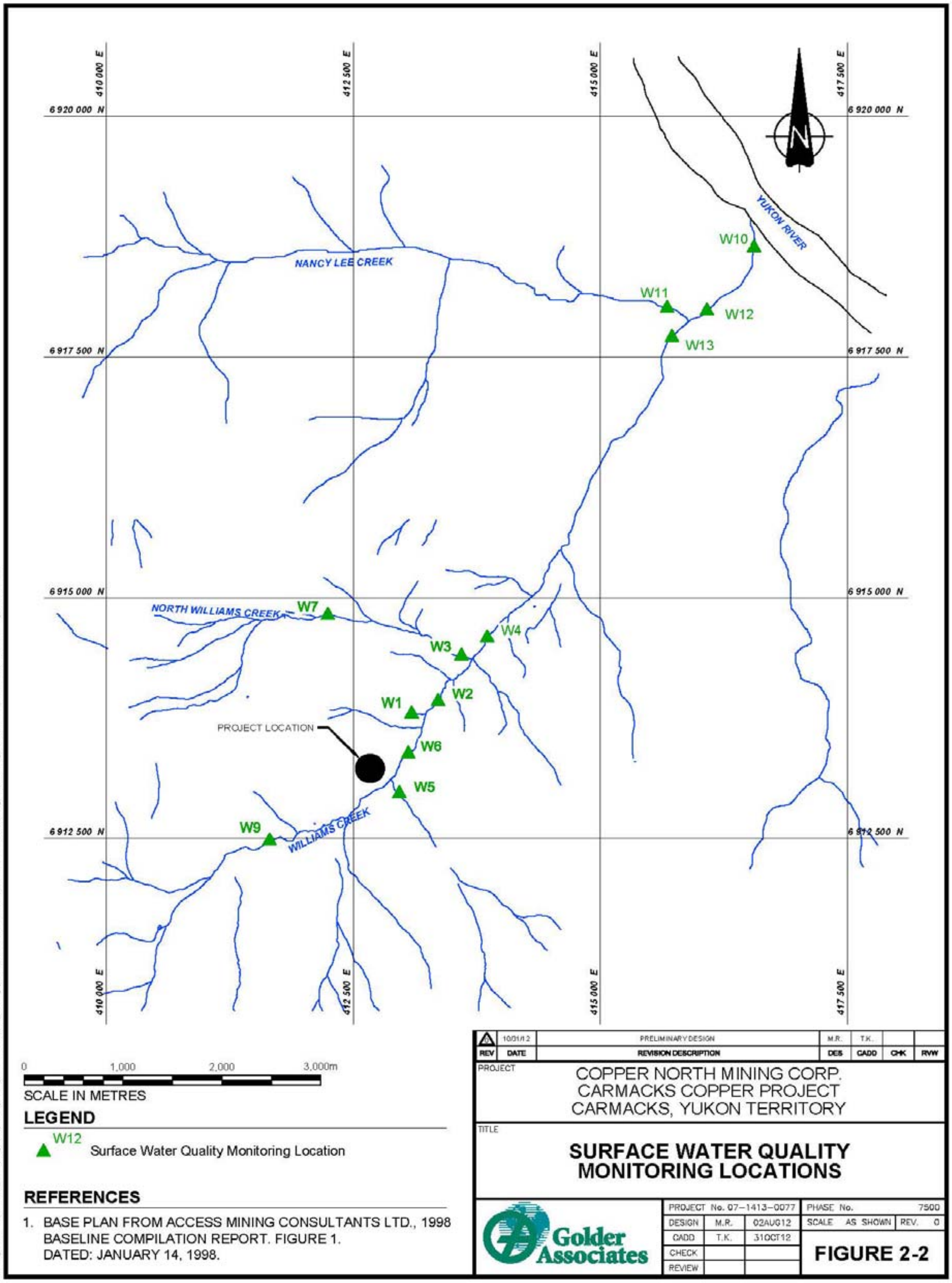


Figure 20-3: Surface Water Quality Monitoring Locations

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 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 2012 (hydrogeo report)). 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 2012 (water quality model)).

20.8.2 Heap Leach Facility

Final closure of the HLF will be undertaken immediately after copper recovery is complete. Closure involves drainage of the active heap layers on top of the final inter-lift liner, re-contouring of the heap surface to facilitate precipitation runoff, and placement of a store and release soil cover over the heap surface to minimize infiltration. A 1 m thick soil cover comprised of 30% fines will be placed and seeded to locally appropriate vegetation species. The soil will be taken from overburden stockpiles established during site development. The soil cover design has been selected to reduce infiltration of precipitation from an estimated 40% of total precipitation with no cover to approximately 12% under the established cover.

The heap layer under active leach will be allowed to drain to the Events Pond and from there the leachate will be treated in the water treatment plant and discharged. The underlying two interlift layers will be drained in advance of heap closure, with drainage of each layer beginning with installation of the inter-lift liner above. Consequently, approximately two-thirds of the heap mass will be drained by the time HLF closure is initiated.

Once drain down is complete and the cover is in place, HLF seepage and surface runoff will be collected in the Events Pond, treated, and discharged to Williams Creek.

20.8.3 Water Treatment

Water treatment during operations and closure will be carried out in the high density sludge water treatment plant. A passive treatment facility 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. An active treatment capability will be maintained on standby into the post-closure period until reliable passive water treatment has been demonstrated.

The passive treatment system will handle surface runoff and seepage from the HLF. The conceptual design for the passive treatment facility is a four-element system, consisting of:

- Iron Terraces for Fe, Mn, and As removal;
- Spring freshet equalization pond designed to hold 14 days of freshet flow;
- Biochemical reactor (BCR) loaded with organics (hay, wood, chips, etc.) and crushed limestone for removal of Al, Cu, Cd, Cr, Mo, Ni, Pb, Zn, and Se; and
- Aerobic polishing wetland for BOD adjustment.

The iron terraces, spring freshet pond, and BCR will be developed in the re-graded Events Pond, and the polishing wetland will be constructed in the re-graded HLF sedimentation pond.

20.8.4 Waste Rock Storage Area

The waste rock storage area (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 2012 (water quality model)).

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 offsite;
- 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 through closure and into post-closure as necessary to supplement the passive treatment system. The kerosene storage will be decommissioned and removed. Diesel generation capacity and fuel storage will be downsized to suit the reduced power demands of water treatment and closure.

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 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.

Closure and reclamation of the power line will be the responsibility of YEC and those 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 recontoured for stability and to reflect the natural local topography. Surfaces will be scarified and revegetated. Exploration trails typically require minimal contouring and stabilization prior. Any side-cast material will be recovered, trenches backfilled, and the trail left to natural revegetation. Reclamation of the main access road would involve removal of all culverts and the Merrice Creek bridge crossing, restoration of drainage, and scarification and revegetation.

Closure of the exploration road currently used to access the project site under Company authority nor is it a closure responsibility and costs for its closure have not been included in the closure estimate.

20.8.7 Vegetation Trials

Vegetation trials will be conducted to determine the most appropriate local vegetation species and planting protocols for the Project site. Detailed site revegetation plans will be developed for the project and incorporated into the closure and reclamation plan following completion of the trials.

20.8.8 Closure Costs

Estimated reclamation costs are shown in Table 20-1. It is expected that these costs would be reviewed during development of an updated Closure Plan.

Table 20-1: Reclamation Costs

Description	Total Estimated Cost
Closure Direct Cost	$5,157,000
Camp Cost	$473,000
Construction Indirects @ 15%	$845,000
Contingency @10	$648,000
Total	$7,123,000

21 CAPITAL AND OPERATING COSTS

21.1 BASIS OF CAPITAL COST

M3 specifically examined the capital to construct the mine site access road, required plant site roads, the power line and associated substations, water systems, and a crushing plant, heap leach facility, solvent extraction and electrowinning (SX/EW) processing facility.

The estimate is based on the project as defined by the process and facility descriptions, design criteria, process flow diagrams and material balance, design drawings and sketches, equipment lists, and other documents developed or referenced in the feasibility study. Golder Associates provided a design report which forms the basis for the heap leach facility quantities and estimated capital cost of this facility.

The total contracted capital cost including owner's cost and contingency is estimated to be C$177.6 million. Table 21-1 details capital costs.

Table 21-1: Production Cost Per Area

Area	C$
Process & Infrastructure & Project Contingency	$162.1 million
Mine Development	$5.9 million
Mine Equip. Lease 2- Years	$3.8 million
Owners Cost	$5.8 million
Total	$177.6 million

21.1.1 Currency

Estimate is expressed in Canadian dollars. Exchange rates used in the estimate are:

- US Dollars @ US$1.00 per C$
- Australian Dollars @ AU$1.00 per C$

Mineral sizers were quoted in Australian dollars. Some mobile equipment was quoted in Canadian dollars. All other equipment and raw materials were quoted in US dollars and converted to Canadian dollars.

21.1.2 Plant Equipment

Budgetary quotes were obtained from qualified Vendors for equipment exceeding $100,000 in value. M3 estimated costs for other equipment using inflated historical information and information from current comparable projects was used to support the estimate.

Total plant equipment cost exclusive of installation or shipping was estimated to be about C$34.4 million. Of that, about 93% was based on vendor quotes. The remainder came from M3's historical information or other sources such as catalogues or cost estimation services.

21.1.3 Bulk Material

Budgetary quotes were obtained for structural steel and concrete supply wire and cable, and tanks. A number of process tanks were originally specified to be stainless steel. Following discussions with tank fabricators and operating facilities, a number of tanks were changed to fibreglass reinforced plastic (FRP). Most FRP tank prices were based on budgetary quotes, but some prices were adjusted from the stainless steel prices using an adjustment factor supplied by the tank fabricator.

21.1.4 Contractor Installation Costs

Contractors with recent experience or currently working in the region provided information on contractor installation rates, productivity, and work schedules. The estimate assumes major earthworks as well as other construction on a rotation of 20 12-hour work days followed by 10 days off.

- Inflated M3 historical records from as-built construction records;
- Quotes from pre-engineered building suppliers;
- End of 3rd quarter 2012 construction bids and current equipment bids for like projects;
- Government publications that establish minimum costs to be charged for various work units; and
- Information from local contractors.

Civil (earthworks) will be suspended during the winter months. All other construction activity will continue year round and under cover.

21.1.5 Freight

Freight has been allowed at 10% of equipment and bulk material cost.

21.1.6 Spare Parts

An allowance for spare parts is included in the estimate as owner’s costs. An allowance of $100,000 plus 0.5% of plant equipment is included in the estimate for start-up and commissioning spare parts.

21.1.7 Common Distributable

M3 has estimated common distributable as a percentage of direct cost.

21.1.8 Construction Camp and Catering

The estimate assumes that a 200-man construction camp will be established early during the first construction season and will be available to all contractors at a fixed charge back rate. Similarly a catering contract will service all contractor personnel. Catering and camp operations costs are included at $100.00 per man-day.

21.1.9 Owner's Cost

Mine development, mining equipment were developed by IMC, and Owner cost were developed by CNMC.

21.1.10 Accuracy

The accuracy of this estimate for those items that are identified in the project scope is estimated to be in the range of +15% to -15%, meaning the cost could be as much as 15% higher than the estimate, or it could be up to 15% lower. Accuracy is an issue separate from contingency. Specific accounts are individually rated by percent depending on scope development and the detail level of the information. Data evaluation includes consideration of winter site conditions.

21.1.11 Time of Estimate

All costs are calculated in end of 3rd Quarter 2012 Canadian Dollars.

21.1.12 Escalation

No escalation is included in the cost estimate

21.1.13 Crews and Equipment Spreads

For major civil, mass excavation, and early concrete work, the estimate assumes working rotations of 20 days in and 10 days out. For process plant, steel erection, architectural, mechanical, electrical, and instrumentation work, the estimate assumes the same rotation.

21.1.14 Construction Equipment

The Owner will not supply any construction equipment (such as forklifts and cranes for unloading, water trucks for dust suppression, loaders, or dozers) for the project.

21.1.15 Site Availability

The construction site will be available to the Contractors 24 hours per day, seven days per week.

Construction work areas will be accessible during all scheduled working hours. Allowance is not included in this estimate for standby time for inefficiencies resulting from work stoppages for not utilizing seasonal weather opportunities.

On-site mobilization of heavy equipment will need to be done before spring breakup occurs and when load restrictions apply. The staging area for the equipment will need to be prepared and secured prior to moving on site.

21.1.16 Facilities

Early development of permanent facilities is for the Owner's and EPCM contractor's use and in general will not be available to construction personnel. A camp for construction personnel will be constructed and catering services will be provided in Q3 2014.

Early contractors are responsible for construction water, power, lighting, security services, and telephone until the owner brings the services onto the site. Thereafter, the contractors are required to connect to the power, water and communications facilities at a designated battery limit for these services. Telephone and wireless services will be chargeable to the contractors.

The Owner will provide offsite, permitted sources for:

- Sand;
- Gravel; and
- Aggregate.

21.1.17 Reference Unit Costs

M3 used the following sources for its unit costing:

a) Inflated M3 historical records from as-built construction records;
b) Third Quarter year 2012 construction bids and current equipment bids for like projects;
c) Government publications that establish minimum costs to be charged for various work units;
d) Inflated historical union rates; and
e) Information from local contractors.

21.1.18 Exclusions and Boundary Conditions

Exclusions

a) PST and GST Tax or associated financial costs on all goods and services;
b) Political or organized labour interruptions;
c) Financial holding costs of tax payments;
d) Future escalation;
e) Future currency variations; and
f) Unforeseen site conditions.

Boundary Conditions

a) Financial analysis is based on 100% equity-financing, excluding leased mining equipment;
b) Depreciation and depletion allowances are included in the financial model.
c) Permitting costs, after full project release, are part of Owner's costs.
d) Bonding costs are limited to payment and performance bonding which will be included in the contractors cost.
e) Cost of reclamation bond is included in Owner's cost.

21.2 DIRECT COSTS

21.2.1 Construction Labor Rates

Labour rates are based on M3 historical records and current construction bids. The base rate with fringes includes: taxes, benefits, burden, profit sharing, etc. All construction workers will be accommodated locally by the Owner.

The base rates with fringes are calculated from the base rate per day times a salary overhead factor. This factor includes paid vacations, paid holidays, termination days and taxes. The average rates for the salaries of various trades were taken from area local contractor rates with modifications for the estimate. Canadian salary for building trades is done on a per day basis. The work day is based on a ten-hour day. The daily base rates were divided by ten, and then averaged for each trade plus a salary overhead factor.

Construction Camp cost is excluded and not shown in the base hourly rate shown. The construction Camp cost is included in the owners cost.

21.2.2 Productivity Factor

As a point of reference, a productivity factor of 1.0 times the U.S. Labour is typical for the plant construction, (i.e. it is assumed that the work activities are scheduled and performed in proper sequence according to the seasonal climates). If the site is opened up in the fall and work resumes in the spring immediately after thaw with major excavation and concrete having continuity, good production rates will prevail. If normal production rates are adhered to during the spring and summer months, the same productivity rates should continue over the winter months inside the enclosed heated buildings.

21.2.3 Overtime

An overtime allowance will be included in the rates. Normal rates are calculated at 1.5 for hours over 40 hours. As previously mentioned, the rotation will be based on 20 10-hour work days followed by 10 days off. It will be on a rolling rotation basis so that work on-site will be continuous. This includes 13% overhead for vacation and holidays.

The estimate assumes all construction labour will maintain living quarters in the construction camp. This will decrease the fatigue factor and the resulting productivity decline on an overtime schedule. This rate will be paid to all employees living in accommodations provided by the Employer for vacating or check out of such accommodations when on leave.

21.2.4 Concrete Supply

It is assumed that local water, sand, and aggregate will be available for concrete and sand backfill. A site-enclosed batch plant is included in the estimate costs.

Construction equipment was included for each area. Equipment rates per hour were taken from historical/actual rates from Canadian government tables, local contractors (Pelly, Ketza), union rates, and road builders.

Not all sources had rates for all equipment. The operator cost is normally deducted from equipment rate to arrive at the well rounded equipment rate. Special condition mobilization costs will be included for moving equipment and staging prior to spring thaw. It is expected equipment rates will be higher than normal weather construction due to mobilization of equipment to site or near the construction site and not being able to perform work for days or weeks until the spring thaw.

21.3 INDIRECT COSTS

21.3.1 Indirect Field Costs

Indirect costs are accounted for in the estimate as follows:

- Mobilization in 0.5% of direct costs;
- Demobilization included with mobilization;
- Home office indirects supervision in contractors' directs (i.e., in labour rates);
- Temporary utilities and trailers in project directs;
- Site Safety officer – in project indirects (i.e., in EPCM cost);
- Insurance, included in Owner's cost;
- Construction permits in project directs;
- First aid cost in project indirects (i.e., in EPCM cost);
- Catering – in project indirects; and
- Camp – in project indirects.

21.4 MINE CAPITAL COST

21.4.1 Capital Cost Summary

The mine capital costs for the Carmacks mine, assuming owner operation, were estimated by IMC. The mine capital costs developed by IMC include the following items:

- Mine major equipment (leased);
- Mine support equipment and initial spare parts (leased);
- Mine preproduction development expense is $5.9 million; and
- Lease of mine equipment for the first two years is $3.8 million.

The estimated cost of the following facilities was developed by others and is included in the infrastructure capital budget:

- The mine shop and warehouse;
- Fuel and lubricant storage facilities;
- Explosive storage facilities; and

- Office facilities.

21.4.2 Mine Development

The first two years (preproduction) lease costs are capitalized in the financial model. Thereafter, lease costs are tabulated as operating costs in the financial model.

21.4.3 Major Equipment

Initial mine major equipment is estimated at C$ 19.4 million but as aforementioned will be under a lease agreement. Table 21-2 shows the equipment purchase schedule and cost by year. Most of the mine fleet is purchased for the preproduction period. During Year 1 the hydraulic shovel and two trucks are added to the fleet. Major equipment sustaining capital is $4.7 million and consists of an additional drill in Year 2 and an additional truck during Years 2 and 3 to bring the truck fleet to seven units.

The equipment quotes include freight, insurance, and assembly, but do not include any applicable sales taxes or duties.

The estimate is based on recent quotes from Equite Montevedeo Group (EMG), an international broker of mining equipment. Quotes were received from EMG for all equipment except the large drill, the hydraulic shovel, and the motor grader; these are based on quotes received by IMC for other recent projects. The EMG quotes were escalated 4%, as recommended by EMG, to account for anticipated mid-2012 price increases. None of the equipment will require replacement during the project life.

21.4.4 Support Equipment

An allowance for support equipment is based on 15% of the major equipment purchases for each year. Support equipment includes items such as fuel and lube trucks, tire handlers, mechanics trucks, welding trucks, cranes, shop forklifts, pickup trucks, etc. This also includes mine engineering and safety equipment such as a GPS system, surveying equipment, computers, etc. This allowance is also assumed to cover initial spare parts inventory.

21.4.5 Contingency

A contingency of 10% is added to the equipment cost estimate during the initial capital period.

Table 21-2: Mine Capital Cost

		Units	PP	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	TOTAL
MAJOR EQUIPMENT PURCHASE SCHEDULE:												
Caterpillar MD6240 Drill		(none)	1	0	1	0	0	0	0	0	0	2
Komatsu PC2000 Hyd Shovel		(none)	0	1	0	0	0	0	0	0	0	1
Cat 992K Wheel Loader		(none)	1	0	0	0	0	0	0	0	0	1
Cat 777F Truck		(none)	3	2	1	1	0	0	0	0	0	7
Cat D9T Track Dozer		(none)	2	0	0	0	0	0	0	0	0	2
Cat 824H Wheel Dozer		(none)	1	0	0	0	0	0	0	0	0	1
Cat 14M Motor Grader		(none)	1	0	0	0	0	0	0	0	0	1
Water Truck - 10,000 gal		(none)	1	0	0	0	0	0	0	0	0	1
Atlas Copco ECM 720 Drill		(none)	1	0	0	0	0	0	0	0	0	1
Cat 336D Excavator		(none)	1	0	0	0	0	0	0	0	0	1
Total Major Equipment Purchases		(none)	12	3	2	1	0	0	0	0	0	18
MAJOR EQUIPMENT CAPITAL COST:	Unit Price ($x1000)											
Caterpillar MD6240 Drill	1,742.0	($x1000)	1,742	0	1,742	0	0	0	0	0	0	3,484
Komatsu PC2000 Hyd Shovel	2,664.3	($x1000)	0	2,664	0	0	0	0	0	0	0	2,664
Cat 992K Wheel Loader	1,562.9	($x1000)	1,563	0	0	0	0	0	0	0	0	1,563
Cat 777F Truck	1,489.4	($x1000)	4,468	2,979	1,489	1,489	0	0	0	0	0	10,426
Cat D9T Track Dozer	891.6	($x1000)	1,783	0	0	0	0	0	0	0	0	1,783
Cat 824H Wheel Dozer	983.4	($x1000)	983	0	0	0	0	0	0	0	0	983
Cat 14M Motor Grader	546.0	($x1000)	546	0	0	0	0	0	0	0	0	546
Water Truck - 10,000 gal	1,581.4	($x1000)	1,581	0	0	0	0	0	0	0	0	1,581
Atlas Copco ECM 720 Drill	747.1	($x1000)	747	0	0	0	0	0	0	0	0	747
Cat 336D Excavator	350.0	($x1000)	350	0	0	0	0	0	0	0	0	350
MAJOR EQUIPMENT CAPITAL COST		($x1000)	13,764	5,643	3,231	1,489	0	0	0	0	0	24,128
SMALL EQUIPMENT AT	15.00%	($x1000)	2,065	846	485	223	0	0	0	0	0	3,619
TOTAL EQUIPMENT COST		($x1000)	15,829	6,489	3,716	1,713	0	0	0	0	0	27,747
CONTINGENCY AT	10.0%	($x1000)	1,583	649	0	0	0	0	0	0	0	2,232
MINE DEVELOPMENT		($x1000)	5,714	0	0	0	0	0	0	0	0	5,714
TOTAL MINE CAPITAL COST		($x1000)	23,126	7,138	3,716	1,713	0	0	0	0	0	35,693

21.5 SUMMARY OF OPERATING COST

The operating and maintenance costs for the Carmacks Copper operations are summarized by areas of the plant, and shown in Table 21-3. Cost centers include mine operations, process plant operations, and the General & Administration area. Operating costs were determined for a typical year of operations, based on an annual ore tonnage of 1.775 million tonnes and will produce an annual average of 13,200 tonnes of copper cathode. The proposed operation will process 11.6 million tonnes of ore at an average grade of 0.98% total copper and 58.4 million tonnes of waste over a project life of approximately 8 years. The life of mine unit cost per ore tonne is C$29.15 and the unit cost per copper pound is C$1.59, which includes mining, Process Plant, General & Administrative cost, and shipping.

Table 21-3: Production Cost Per Area

Area	Total Cost C$ (000)	Cost per tonne ore mined, C$
Mine	183,476	15.88
Process	112,209	9.71
G&A	38,004	3.29
Shipping	3,168	0.27
Total	**336,857**	**29.15**

21.6 PROCESS PLANT OPERATION COSTS

Operating Costs are detailed in Table 21-5. Typical operating process plant costs average $9.12/ tonne. The $9.71 includes cost for the last year residual leaching.

Table 21-4: Process Plant Operations Cost Average

Area	Cost per tonne, C$
Crushing	$1.137
Heap Leach	$3.718
SX/EW	$4.006
Ancillary Services	$0.259
Total	**$9.12**

Table 21-5: Operating Cost- Process Plant Cost Summary

Processing Units Base Rate (tonnes/year ore)	1,775,000		
	Total		
Process Plant Cost Area	Annual Cost - $		$/tonne ore
Crushing			
Labor and Fringes	$	938,000	0.528
Power		306,569	0.173
Wear Parts		68,604	0.039
Maintenance Parts		230,500	0.130
Maintenance Labor and Fringes		210,600	0.119
Maintenance Shop Power Allocation		300	0.000
Supplies & Services		263,100	0.148
Subtotal Crushing	**$**	**2,017,700**	**1.137**
Heap Leach			
Labor and Fringes	$	509,900	0.287
Power		141,667	0.080
Power - Allocation 90% Fresh / Fire Water System		152,556	0.086
Reagents (Sulfuric Acid)		5,003,864	2.819
Maintenance Parts		148,600	0.084
Maintenance Labor and Fringes		351,100	0.198
Maintenance Shop Power Allocation		100	0.000
Supplies and Services		291,200	0.164
Subtotal Heap Leach	**$**	**6,599,000**	**3.718**
SXEW			
Labor and Fringes	$	1,366,100	0.770
Power		3,554,535	2.003
Reagents		708,172	0.399
Maintenance Parts		915,900	0.516
Maintenance Labor and Fringes		280,900	0.158
Maintenance Shop Power Allocation		500	0.000
Supplies and Services		285,100	0.161
Subtotal SXEW	**$**	**7,111,200**	**4.006**
Acid Plant			
Labor and Fringes	$	821,900	0.463
Power		580,498	0.327
Sulphur		2,757,167	1.553
Wear Parts			
Maintenance Parts		495,500	0.279
Maintenance Labor and Fringes		280,900	0.158
Maintenance Shop Power Allocation		100	0.000
Supplies and Services		67,800	0.038
Annual Co-Generation/Heating Credit		0	-
Transfer Operating Cost to Heap Leach		(5,003,864)	(2.819)
Subtotal Acid Plant	**$**	-	-
Ancillary Services			
Power		0	-
Maintenance Parts		269,900	0.152
Maintenance Labor and Fringes		140,400	0.079
Supplies and Services		50,000	0.028
Subtotal Ancillary Services	**$**	**460,300**	**0.259**
Total Process Plant	**$**	**16,188,200**	**9.120**

21.7 MINE OPERATING COST

21.7.1 Operating Cost Summary

Table 21-6 summarizes the mine operating costs for the Carmacks mine, based on owner operation of the mine excluding leased mining equipment. Total cost, the cost per total tonne, and cost per ore tonne are shown by various time periods. The C$5.9 million preproduction development cost is the source of the mine development capital cost. During commercial production the unit costs for mining are $2.269 per total tonne and $13.58 per ore tonne.

Table 21-6: Summary of Total and Unit Mining Cost

Category	Total Material (kt)	Ore (kt)	Total Cost (US$)	Cost Per Total Ton(ne) (US$/t)	Cost Per Ore Ton(ne) (US$/t)
Mine Development (PP)	953	0	5,714	5.996	0.000
Commercial Production (Years 1 to 7)	69,154	11,551	156,913	2.269	13.584
All Time Periods (PP to 7)	70,107	11,551	162,627	2.320	14.079
Commercial Production Years 1 - 3	36,650	5,325	78,106	2.131	14.668
Commercial Production Years 4 - 6	31,097	5,325	73,128	2.352	13.733
Commercial Production Year 7	1,407	901	5,679	4.037	6.303

The costs are in 3rd quarter 2012 US dollars. The estimate is based on assumed prices for commodities such as fuel, explosives, parts, tires, etc. that are subject to wide variations depending on market conditions. The estimate is based on the following prices for key commodities:

- Diesel fuel delivered to the site for $1.041 per liter ($3.94/US gallon);
- Blasting agents at $0.60 per kg; and
- Tires at approximately 75% of US list prices.

Other than fuel, the cost estimate does not include any applicable sales taxes or duties on parts, tires, blasting agents and explosives, etc.

Table 21-7 presents the details of the cost estimate by cost center. The top section of the table shows total cost, the center section shows cost per total tonne, and the bottom section shows cost per ore tonne. The total and ore tonnages used as the divisor are also shown. Note that the total tonnes divisor includes re-handle quantities.

The cost estimate includes the following mining activities:

- Mining and transporting ore to the crusher;
- Mining and transporting waste to the waste storage facilities;
- Maintaining the haul roads and dumps;
- Operation and maintenance supervision; and
- Mine engineering and geology support, including ore grade control.

There is not an allowance in the mine operating cost estimate for pumping water from the pit.

Mine operating costs are broadly separated into parts and consumables costs and labor costs as will be discussed in the following sections.

21.7.2 Parts and Consumables Costs

21.7.2.1 General

Table 21-8 summarizes the mine parts and consumables operating costs by commodity (fuel, tires, parts, lubricants, explosives, etc.). Life of mine, parts and consumables account for $1.185 per total tonne and $7.190 per ore tonne, or about 52% of the mine operating costs.

Parts and consumables costs are divided into the flowing sections:

- Mine major equipment;
- Blasting supplies; and
- Allowances for small equipment and mine administration.

The line item “Other” on Table 21-8 occurs only during the mine development period and includes the mine development contingency discussed in Section 21.4, Mine Capital.

21.7.2.2 Mine Major Equipment

The supplies required to operate, maintain, and repair the mine major equipment contribute the most significant portion of the total mine parts and consumables costs. Table 21-9 summarizes the estimated cost per metered hour and per shift for the major equipment. Fuel cost is based on $1.041 per liter or $3.94 per US gallon delivered to the property. Tires, delivered to the property, are estimated at 75% of US list prices.

21.7.2.3 Blasting Supplies

Blasting consumables are based on powder factors of 200 g/t for ore, 250 g/t for waste rock, and 100 g/t for overburden. IMC has assumed the ore is relatively weak and the waste rock of moderate strength based on uniaxial compression testing performed by Golder. More details about hole diameter, spacing and burden, etc. are in Section 16.6 Mine Equipment.

Blasting agent costs are based on a mix of about 50% ANFO and 50% slurry delivered to the property for $0.60 per kilogram. There is also a 5% allowance for additional blasting agents for the small drill for road construction and wall control blasting. In addition to this, an allowance of 40% of the explosives costs is estimated for initiations supplies and loading the holes by a contractor. This is based on a March 2011 quotation from Orica for Western Copper’s Casino Project, also in the Yukon.

21.7.2.4 Allowances for Small Equipment

Allowances for parts and consumables for small equipment are factored from the major equipment as follows:

- Fuel, 15% of major equipment cost;
- Tires, 3% of major equipment;
- Repair parts, 3% of major equipment;
- Filters and lubricants, 3% of major equipment; and
- Ground engaging parts, 1% of major equipment.

Life of mine, these costs amount to about $0.079 per total tonne and are included in the Mine Services Cost Center.

21.7.2.5 Allowances for Administration Supplies

Administrative supplies are estimated at 20% of mine administrative salaries. This amounts to about $0.060 per total tonne and is in the Mine Administration Cost Center.

21.7.3 Mine Labor Costs

The number of salaried staff personnel (mine supervision), and costs are shown on Table 21-10 and Table 21-11 respectively. The annual rates include a burden of $20,783 per employee for worker's compensation, Canadian pension, employment insurance, medical, and travel costs, and are considered all-in rates. The costs on Table 21-11 report to the Mine Administration Cost Center on Table 21-7. Table 21-10 and Table 21-11 do not include mine equipment or maintenance training positions. These are considered a G&A Human Resources function.

Hourly labor requirements and costs are shown on Table 21-12 and Table 21-13 respectively. The rates include a large overtime premium due to the three crew rotation. 335 mine operating days x 24 hours/day divided into three crews comes to 2,680 hours per man per year. IMC has assumed this is 2,190 hours base time plus 490 hours of overtime. In addition, each person is paid for 160 hours of holiday/vacation pay (20 days x 8 hours). The payroll burden is assumed of $20,783 per person, as described above, is also included. This does not include camp costs for the personnel.

There is not any owner blasting personnel in the estimate; as discussed above this service is provided by the vendor.

Life of mine, labor costs amount to only about $1.13 per total tonne or about 48% of the operating costs.

Table 21-7: Operating Cost Summary by Cost Center

Cost Center	Units	PP	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	TOTAL	Percent
Drilling	($x1000)	252	1,857	2,639	2,741	2,735	2,323	1,297	261	0	14,105	8.7%
Blasting	($x1000)	160	1,915	2,753	2,889	2,881	2,518	1,205	271	0	14,593	9.0%
Loading	($x1000)	306	2,898	3,756	3,715	3,718	3,336	2,035	435	0	20,200	12.4%
Hauling	($x1000)	607	4,727	6,255	7,379	6,588	7,330	4,800	1,272	0	38,957	24.0%
Roads and Dumps	($x1000)	1,251	4,480	4,480	4,480	4,480	4,480	3,287	1,125	0	28,065	17.3%
Mine Services	($x1000)	1,628	3,073	3,283	3,352	3,317	3,298	2,814	943	0	21,709	13.3%
Mine Administration	($x1000)	1,511	3,811	3,811	3,811	3,811	3,811	3,065	1,372	0	25,000	15.4%
TOTAL OPERATING COST	($x1000)	5,714	22,761	26,978	28,367	27,529	27,097	18,502	5,679	0	162,627	100.0%
COST PER TOTAL TON(NE):												
Total Material	(kt)	953	9,650	13,500	13,500	13,500	11,776	5,821	1,407	0	70,107	
Drilling	(US$)	0.264	0.192	0.195	0.203	0.203	0.197	0.223	0.186	0.000	0.201	8.7%
Blasting	(US$)	0.168	0.198	0.204	0.214	0.213	0.214	0.207	0.192	0.000	0.208	9.0%
Loading	(US$)	0.321	0.300	0.278	0.275	0.275	0.283	0.350	0.309	0.000	0.288	12.4%
Hauling	(US$)	0.636	0.490	0.463	0.547	0.488	0.622	0.825	0.904	0.000	0.556	24.0%
Roads and Dumps	(US$)	1.313	0.464	0.332	0.332	0.332	0.380	0.565	0.800	0.000	0.400	17.3%
Mine Services	(US$)	1.708	0.318	0.243	0.248	0.246	0.280	0.483	0.670	0.000	0.310	13.3%
Mine Administration	(US$)	1.585	0.395	0.282	0.282	0.282	0.324	0.526	0.975	0.000	0.357	15.4%
OPERATING COST PER TOTAL TON(NE)	(US$)	5.996	2.359	1.998	2.101	2.039	2.301	3.178	4.037	0.000	2.320	100.0%
COST PER ORE TON(NE):												
Total Ore	(kt)	0	1,775	1,775	1,775	1,775	1,775	1,775	901	0	11,551	
Drilling	(US$)	0.000	1.046	1.487	1.544	1.541	1.309	0.730	0.290	0.000	1.221	8.7%
Blasting	(US$)	0.000	1.079	1.551	1.628	1.623	1.419	0.679	0.300	0.000	1.263	9.0%
Loading	(US$)	0.000	1.633	2.116	2.093	2.094	1.879	1.147	0.483	0.000	1.749	12.4%
Hauling	(US$)	0.000	2.663	3.524	4.157	3.712	4.129	2.704	1.412	0.000	3.373	24.0%
Roads and Dumps	(US$)	0.000	2.524	2.524	2.524	2.524	2.524	1.852	1.249	0.000	2.430	17.3%
Mine Services	(US$)	0.000	1.731	1.850	1.889	1.869	1.858	1.585	1.047	0.000	1.879	13.3%
Mine Administration	(US$)	0.000	2.147	2.147	2.147	2.147	2.147	1.726	1.522	0.000	2.164	15.4%
OPERATING COST PER ORE TON(NE)	(US$)	0.000	12.823	15.199	15.981	15.510	15.266	10.424	6.303	0.000	14.079	100.0%

*Excludes mine equipment lease cost.

Table 21-8: Parts and Consumables Summary

Commodity	Units	PP	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	TOTAL	Percent
Fuel	($x1000)	702	4,996	6,407	6,868	6,630	6,507	3,960	1,108	0	37,178	44.7%
Power	($x1000)	0	0	0	0	0	0	0	0	0	0	0.0%
Tires	($x1000)	120	746	974	1,082	1,021	1,047	663	182	0	5,835	7.0%
Replacement Parts	($x1000)	211	1,475	1,846	1,932	1,888	1,817	1,100	314	0	10,582	12.7%
Lubricants and Filters	($x1000)	135	966	1,241	1,341	1,291	1,276	782	217	0	7,249	8.7%
Ground Engaging Wear Parts	($x1000)	63	394	490	504	503	462	263	73	0	2,753	3.3%
Blasting	($x1000)	160	1,915	2,753	2,889	2,881	2,518	1,205	271	0	14,593	17.6%
Mine Administration	($x1000)	252	635	635	635	635	635	511	229	0	4,167	5.0%
Other	($x1000)	745	0	0	0	0	0	0	0	0	745	0.9%
TOTAL CONSUMABLES	($x1000)	2,387	11,127	14,345	15,251	14,849	14,263	8,484	2,394	0	83,101	100.0%
COST PER TOTAL TON(NE):												
Total Material	(kt)	953	9,650	13,500	13,500	13,500	11,776	5,821	1,407	0	70,107	
Fuel	(US$)	0.736	0.518	0.475	0.509	0.491	0.553	0.680	0.788	0.000	0.530	44.7%
Power	(US$)	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.0%
Tires	(US$)	0.126	0.077	0.072	0.080	0.076	0.089	0.114	0.130	0.000	0.083	7.0%
Replacement Parts	(US$)	0.221	0.153	0.137	0.143	0.140	0.154	0.189	0.223	0.000	0.151	12.7%
Lubricants and Filters	(US$)	0.142	0.100	0.092	0.099	0.096	0.108	0.134	0.154	0.000	0.103	8.7%
Ground Engaging Wear Parts	(US$)	0.066	0.041	0.036	0.037	0.037	0.039	0.045	0.052	0.000	0.039	3.3%
Blasting	(US$)	0.168	0.198	0.204	0.214	0.213	0.214	0.207	0.192	0.000	0.208	17.6%
Mine Administration	(US$)	0.264	0.066	0.047	0.047	0.047	0.054	0.088	0.162	0.000	0.059	5.0%
Other	(US$)	0.782	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.011	0.9%
CONSUMABLES PER TOTAL TON(NE)	(US$)	2.505	1.153	1.063	1.130	1.100	1.211	1.458	1.702	0.000	1.185	100.0%
COST PER ORE TON(NE):												
Total Ore	(kt)	0	1,775	1,775	1,775	1,775	1,775	1,775	901	0	11,551	
Fuel	(US$)	0.000	2.814	3.609	3.869	3.735	3.666	2.231	1.230	0.000	3.219	44.7%
Power	(US$)	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.0%
Tires	(US$)	0.000	0.420	0.549	0.609	0.575	0.590	0.373	0.202	0.000	0.505	7.0%
Replacement Parts	(US$)	0.000	0.831	1.040	1.088	1.063	1.024	0.620	0.348	0.000	0.916	12.7%
Lubricants and Filters	(US$)	0.000	0.544	0.699	0.756	0.727	0.719	0.441	0.241	0.000	0.628	8.7%
Ground Engaging Wear Parts	(US$)	0.000	0.222	0.276	0.284	0.283	0.260	0.148	0.082	0.000	0.238	3.3%
Blasting	(US$)	0.000	1.079	1.551	1.628	1.623	1.419	0.679	0.300	0.000	1.263	17.6%
Mine Administration	(US$)	0.000	0.358	0.358	0.358	0.358	0.358	0.288	0.254	0.000	0.361	5.0%
Other	(US$)	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.064	0.9%
CONSUMABLES PER ORE TON(NE)	(US$)	0.000	6.269	8.082	8.592	8.366	8.035	4.780	2.657	0.000	7.194	100.0%

Table 21-9: Capital and Operating Costs for Mining Equipment

US or Metric Units	Metric		US	
Diesel Fuel Cost ($/gal)	3.940	1.041	$/liter	Metric
Lubricants ($/gal)	9.970	2.634	$/liter	
Electrical Power ($/kwhr)	0.073			
Maintenance Cost Per Hour	47.52			
Parts Factor (x US Cost)	1.000			
Tire Factor (x US List Cost)	0.750			
Tire Life Factor (1-5)	3	Average	Tire Life Factor, 1=very low, 2=low, 3=average, 4=high, 5=very high	
Labor Factor (xUS Cost)	1.000			
Metered Minutes Per Shift	645	10.75	hours	
Abrasion Index (1-5) and Factor	3	1.00	Abrasion Index: 1=very high, 2=high, 3=moderate, 4=low, 5=very low, for wear parts, andbits/steel	

		Capital		Maint/Overhaul			Fuel		Tires				Lube/	Wear	Bits/	Cost Per Hour			Cost
Equipment Type	Capacity/ Power	Cost (US$)	Life (hours)	Maint Hrs/ Op Hrs	Mnt. Labor (US$)	Parts (US$)	Consum. (l/hr)	Cost (US$)	Per Tire (US$)	No. of Tires	Life (hours)	Cost (US$)	Filters (US$)	Parts (US$)	Steel (US$)	P&C (US$)	M. Labor (US$)	Total (US$)	Per Shift (US$)
Caterpillar MD6240 Drill	(210 mm)	1,742,000	60,000	0.90	42.76	37.35	120.5	125.38	0	0	0	0.00	31.44	0.00	38.67	232.83	42.76	275.60	2,962.66
Komatsu PC2000 Hyd Shovel	(11 cu m)	2,664,280	50,000	2.50	118.79	75.21	137.1	142.69	0	0	0	0.00	22.31	0.72	0.00	240.93	118.79	359.72	3,867.04
Cat 992K Wheel Loader	(10.7 cu m)	1,562,891	40,000	0.62	29.46	20.49	84.2	87.67	36,411	4	3,500	41.61	13.56	0.95	0.00	164.29	29.46	193.75	2,082.76
Cat 777F Truck	(90 mt)	1,489,374	80,000	0.36	17.11	13.51	62.8	65.33	10,990	6	3,500	18.84	15.59	0.00	0.00	113.27	17.11	130.38	1,401.59
Cat D9T Track Dozer	(306 kw)	891,649	50,000	0.46	21.86	11.68	47.0	48.96	0	0	0	0.00	7.65	13.76	0.00	82.04	21.86	103.90	1,116.90
Cat 824H Wheel Dozer	(264 kw)	983,424	50,000	0.42	19.96	17.85	33.8	35.22	6,199	4	3,000	8.27	8.63	1.48	0.00	71.44	19.96	91.40	982.54
Cat 14M Motor Grader	(193 kw)	546,000	50,000	0.32	15.21	11.02	29.7	30.93	1,624	6	2,500	3.90	4.95	1.19	0.00	51.99	15.21	67.19	722.30
Water Truck - 10,000 gal	(37,800 l)	1,581,386	50,000	0.67	31.84	36.67	34.1	35.52	6,205	6	3,500	10.64	14.98	0.00	0.00	97.81	31.84	129.65	1,393.69
Atlas Copco ECM 720 Drill	(140 mm)	747,136	40,000	1.08	51.32	30.12	44.2	45.97	0	0	0	0.00	10.88	0.00	10.46	97.43	51.32	148.75	1,599.04
Cat 336D Excavator	(1.93 cu m)	350,040	50,000	0.29	13.78	7.76	30.7	32.00	0	0	0	0.00	3.34	2.10	0.00	45.20	13.78	58.98	634.01

*Mine equipment is being leased.

Table 21-10: Supervisory Staff Labor Requirements

JOB DESCRIPTION	Cost ($x1000)	Year PP	1	2	3	4	5	6	7	8
MINE OPERATIONS:										
Mine Superintendent	116.5	1	1	1	1	1	1	1	1	0
General Forman	95.7	1	1	1	1	1	1	0	0	0
Drilling/Blasting Foreman	95.7	1	1	1	1	1	1	1	1	0
Shift Foreman	95.7	2	4	4	4	4	4	4	4	0
Administrative Assistant/Clerk	72.8	1	1	1	1	1	1	1	1	0
Mine Operations Total		6	8	8	8	8	8	7	7	0
MINE MAINTENANCE:										
Maint. Superintendent	116.5	1	1	1	1	1	1	1	1	0
Maint. General Foreman	95.7	1	1	1	1	1	1	0	0	0
Maint. Shift Foreman	95.7	2	4	4	4	4	4	4	4	0
Maint. Shop Foreman	95.7	2	3	3	3	3	3	3	2	0
Maint. Planner	85.3	1	1	1	1	1	1	1	1	0
Maint. Assistant/Clerk	72.8	1	1	1	1	1	1	0	0	0
Mine Maintenance Total		8	11	11	11	11	11	9	8	0
TECHNICAL SERVICES:										
Chief Mining Engineer	116.5	1	1	1	1	1	1	1	1	0
Mining Engineer	93.6	2	3	3	3	3	3	2	2	0
Chief Geologist	93.6	1	1	1	1	1	1	1	1	0
Geologist	93.6	2	3	3	3	3	3	2	1	0
Chief Surveyor	89.4	1	1	1	1	1	1	1	1	0
Technicians	85.3	6	6	6	6	6	6	4	3	0
Mine Technical Services Total		13	15	15	15	15	15	11	9	0
TOTAL PERSONNEL		27	34	34	34	34	34	27	24	0

Table 21-11: Supervisory Staff Labor Costs ($x1000)

JOB DESCRIPTION	Cost ($x1000)	Year PP	1	2	3	4	5	6	7	8	TOTAL
MINE OPERATIONS:											
Mine Superintendent	116.5	58.4	116.5	116.5	116.5	116.5	116.5	116.5	58.4	0.0	815.6
General Forman	95.7	48.0	95.7	95.7	95.7	95.7	95.7	0.0	0.0	0.0	526.3
Drilling/Blasting Foreman	95.7	48.0	95.7	95.7	95.7	95.7	95.7	95.7	48.0	0.0	669.9
Shift Foreman	95.7	95.9	382.7	382.7	382.7	382.7	382.7	382.7	191.9	0.0	2,583.8
Administrative Assistant/Clerk	72.8	36.5	72.8	72.8	72.8	72.8	72.8	72.8	36.5	0.0	509.7
Mine Operations Total		286.8	763.2	763.2	763.2	763.2	763.2	667.6	334.8	0.0	5,105.3
MINE MAINTENANCE:											
Maint. Superintendent	116.5	58.4	116.5	116.5	116.5	116.5	116.5	116.5	58.4	0.0	815.6
Maint. General Foreman	95.7	48.0	95.7	95.7	95.7	95.7	95.7	0.0	0.0	0.0	526.3
Maint. Shift Foreman	95.7	95.9	382.7	382.7	382.7	382.7	382.7	382.7	191.9	0.0	2,583.8
Maint. Shop Foreman	95.7	95.9	287.0	287.0	287.0	287.0	287.0	287.0	95.9	0.0	1,913.8
Maint. Planner	85.3	42.8	85.3	85.3	85.3	85.3	85.3	85.3	42.8	0.0	597.1
Maint. Assistant/Clerk	72.8	36.5	72.8	72.8	72.8	72.8	72.8	0.0	0.0	0.0	400.4
Mine Maintenance Total		377.5	1,039.8	1,039.8	1,039.8	1,039.8	1,039.8	871.4	389.0	0.0	6,837.0
TECHNICAL SERVICES:											
Chief Mining Engineer	116.5	58.4	116.5	116.5	116.5	116.5	116.5	116.5	58.4	0.0	815.6
Mining Engineer	93.6	93.9	280.7	280.7	280.7	280.7	280.7	187.2	93.9	0.0	1,778.6
Chief Geologist	93.6	46.9	93.6	93.6	93.6	93.6	93.6	93.6	46.9	0.0	655.4
Geologist	93.6	93.9	280.7	280.7	280.7	280.7	280.7	187.2	46.9	0.0	1,731.7
Chief Surveyor	89.4	44.8	89.4	89.4	89.4	89.4	89.4	89.4	44.8	0.0	626.2
Technicians	85.3	256.6	511.6	511.6	511.6	511.6	511.6	341.1	128.3	0.0	3,283.8
Mine Technical Services Total		594.5	1,372.5	1,372.5	1,372.5	1,372.5	1,372.5	1,014.9	419.3	0.0	8,891
TOTAL PERSONNEL		1,258.8	3,175.6	3,175.6	3,175.6	3,175.6	3,175.6	2,553.8	1,143.0	0.0	20,834
Partial Year Factor		50.1%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	50.1%	0.0%	

Table 21-12: Hourly Labor Requirements

JOB DESCRIPTION	Cost ($x1000)	Year PP	1	2	3	4	5	6	7	8
MINE OPERATIONS:										
Drill Operators	127.3	2	3	4	4	4	3	3	1	0
Shovel/Loader Operators	127.3	2	6	6	6	6	6	6	2	0
Truck Operators	127.3	4	12	15	18	15	18	12	6	0
Support Equipment Operators	127.3	8	12	12	12	12	12	9	6	0
Blasting Personnel	127.3	0	0	0	0	0	0	0	0	0
General Laborers	104.0	4	6	6	6	6	6	6	4	0
Mine Operations Total		20	39	43	46	43	45	36	19	0
MINE MAINTENANCE:										
Mechanics/Helpers/Welders	127.3	4.8	17.4	21.9	22.8	22.3	21.4	12.9	7.3	0.0
Fuel/Lube Technician	127.3	2	3	3	3	3	3	3	2	0
Tire Men	127.3	2	3	3	3	3	3	3	2	0
General Laborers	104.0	6	9	9	9	9	9	8	6	0
Mine Maintenance Total		14.8	32.4	36.9	37.8	37.3	36.4	26.9	17.3	0.0
TOTAL PERSONNEL		34.8	71.4	79.9	83.8	80.3	81.4	62.9	36.3	0.0
Maintenance/Operations Ratio		0.74	0.83	0.86	0.82	0.87	0.81	0.75	0.91	0.00

Table 21-13: Hourly Labor Costs ($x1000)

JOB DESCRIPTION	Cost ($x1000)	Year PP	1	2	3	4	5	6	7	8	TOTAL
MINE OPERATIONS:											
Drill Operators	127.3	128	382	509	509	509	382	382	64	0	2,866
Shovel/Loader Operators	127.3	128	764	764	764	764	764	764	128	0	4,840
Truck Operators	127.3	255	1,528	1,910	2,292	1,910	2,292	1,528	383	0	12,099
Support Equipment Operators	127.3	511	1,528	1,528	1,528	1,528	1,528	1,146	383	0	9,681
Blasting Personnel	127.3	0	0	0	0	0	0	0	0	0	0
General Laborers	104.0	209	624	624	624	624	624	624	209	0	4,163
Mine Operations Total		1,230	4,827	5,336	5,718	5,336	5,591	4,444	1,167	0	33,648
MINE MAINTENANCE:											
Mechanics/Helpers/Welders	127.3	309	2,221	2,784	2,901	2,839	2,722	1,636	468	0	15,880
Fuel/Lube Technician	127.3	128	382	382	382	382	382	382	128	0	2,548
Tire Men	127.3	128	382	382	382	382	382	382	128	0	2,548
General Laborers	104.0	313	936	936	936	936	936	832	313	0	6,140
Mine Maintenance Total		878	3,921	4,484	4,601	4,539	4,423	3,233	1,037	0	27,115
TOTAL PERSONNEL		2,108	8,748	9,820	10,319	9,875	10,013	7,677	2,203	0	60,764
Partial Year Factor		50.1%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	50.1%	0.0%	

22 ECONOMIC ANALYSIS

22.1 BASIS OF EVALUATION

Annual cash flows projections were estimated over the life of the mine based on estimates of capital expenditures, production cost, royalties, and sales revenue. The financial analysis is based on a number of project criteria. Following is a summary of criteria that have a definite influence on the estimate:

- The financial analysis is based on constant Canadian dollars (C$);
- The copper price used is a long term price which will be the base case. That price is C$3.20;
- No premiums for LME Grade A material were assumed in the price;
- Financial analysis is based on 100% equity financing, excluding leased mining equipment;
- Income tax rate is calculated at 30%;
- Labour costs were derived from a staffing plan and based on prevailing labour rates and included all applicable social security benefits as well as all applicable payroll taxes; and
- Preproduction operating costs are expensed in the year incurred. Tax losses associated with these expenses are carried forward and used to defer pre-tax earnings subsequent to the start of copper production.

22.2 ECONOMIC RESULTS

The base case using a copper price of C$3.20 per pound, the economic results based on a 100% equity calculation indicates that with an after-tax internal rate of return of 10.0% can be achieved. The corresponding after tax NPV is C$98.9 million at a zero discount rate, C$40.3 million at a 5% discount rate, C$14.5 million at a 8% discount rate, and C$116,000 at a 10% discount rate.

Table 22-1: Economic Indicators

Economic Indicators Before Taxes		Economic Indicators After Taxes	
NPV at 0% - ($000) before tax	$155,331	NPV at 0% - ($000) after tax	$98,920
NPV at 5% - ($000) before tax	$80,834	NPV at 5% - ($000) after tax	$40,307
NPV at 8% - ($000) after tax	$48,019	NPV at 8% - ($000) after tax	$14,451
NPV at 10% - ($000) before tax	$29,839	NPV at 10% - ($000) after tax	$116
IRR	14.1%	IRR	10.0%
		Payback - Years from Startup	5.3

22.3 PAYBACK

As shown in Table 22-1, the calculated payback period is 5.3 years.

22.4 MINE LIFE

The base case life-of-operation is a nominal eight years.

22.5 CAPITAL EXPENDITURE

22.5.1 Initial Capital

The base case financial indicators have been determined with 100% equity financing of the initial capital excluding leased mining equipment. Any acquisition cost or expenditures prior to start of the full project period have been treated as "sunk" cost and have not been included in the analysis.

22.5.2 Sustaining Capital

A schedule of capital cost expenditures during the production period was estimated and included in the financial analysis under the category of sustaining capital. The major component is the expansion of the leach pad. The total life of mine sustaining capital is estimated to be $4.7 million. This capital will be expended during a 4 year period, starting in Year 2 and ending in Year 5.

22.5.3 Working Capital

Operating working capital will vary by year depending on sales revenue. Operating working capital is allowed at six months of sales revenue to provide cash to meet operating expenses prior to receipt of sales revenue. In addition, working capital for plant consumable inventory is estimated in year 1. All the working capital is recaptured at the end of the mine life and the final value of the account is nil.

22.5.4 Salvage Value

A $6.3 million allowance for salvage value has been included in the cash flow analysis. This figure was arrived at by taking 20% of the plant equipment cost. This represents a conservative value based on information regarding the sales of assets from similar projects.

22.6 REVENUE

Annual revenue is determined by applying estimated metal prices to the annual payable metal estimated for each operating year. Sales prices have been applied to all life of mine production without escalation or hedging. The copper price used for the evaluation is C$3.20 per pound, which is long term price.

Revenue for the oxide ore metal sales is recognized at the time of production. The revenue is the gross value of payable metals sold before transportation charges.

22.7 TOTAL CASH OPERATING COST

Year 2 (typical) Total Cash Operating Cost is estimated to be C$29.64 per metric ton of oxide ore processed or C$1.77 per pound of copper, excluding the cost of the capitalized pre-stripping. The Total Cash Operating Cost includes mine operations, SX/EW operations, general administrative cost and shipping charges.

22.7.1 Mine

Mine operating cost was based on a detailed estimate previously discussed in Section 21.5.

22.7.2 Process

SX/EW operating cost was based on a detailed estimate previously discussed.

22.7.3 G&A

This operating category includes the site general and administrative cost. The general and administrative operating cost was also based on a detailed estimate as previously discussed. Also included in the financial analysis are the operations of the camp and the amortization of the cost charged by YEC relating to the capital recovery for the existing Carmacks-Steward 138 kV power grid.

22.7.4 Shipping, Smelting & Refining

The shipping charge for the copper cathode is estimated at $0.015 per pound of copper.

22.8 TOTAL CASH COST

Total Cash Cost is the Total Cash Operating Cost plus employee profit sharing, royalties, and property tax.

22.8.1 Employee Profit Sharing

Employee profit sharing was not applicable to this estimate.

22.8.2 Royalty

The Carmacks project is subject to an acquisition royalty payment of $2,500,000, of which $900,000 has been paid in advance. The payment is based on 3.0% of the net smelter return (NSR) from the first year of production, until the full amount is paid. This financial commitment was included in the cash flow.

22.8.3 Property Tax

No allowance was included in the cash flow.

22.9 TOTAL PRODUCTION COST

Total Production Cost is the Total Cash Cost plus the Reclamation & Closure Cost, and Depreciation.

22.9.1 Reclamation & Closure Bond Fee

This is included in the Owners' costs based on estimate of such insurers.

22.9.2 Reclamation & Closure

An allowance for the cost of final reclamation and closure of the property has been included in the cash flow projection. Continual early reclamation is done throughout the life of the mine and cost have included for such, e.g. borrow pits. Reclamation and closure cost are estimated to be $7.1 million.

22.9.3 Depreciation

Depreciation is calculated by 25% Declining Balance method starting with first year of production. The last year of production is the catch-up year if the assets are not fully depreciated by that time. An additional deduction for the initial capital is being taken in the early years until the initial capital is depreciated.

22.10 PROJECT FINANCING

It is assumed the project will be all equity financed, with the exception of mining equipment.

22.11 NET INCOME AFTER TAX

Net Income after Tax amounts to $93.8 million for the life of the mine.

22.12 NPV AND IRR

The base case economic analysis (Table 22-2) indicates that the project has an Internal Rate of Return (IRR) of 10.0% with a payback period of 5.3 years.

Table 22-2 compares the base case project financial indicators with the financial indicators for other cases when the sales price, the amount of capital expenditure, operating cost, and copper recovery are varied from the base case values. By comparing the results of this economic analysis, it can be seen that the project IRR's sensitivity to variation in sales price, variation of operating cost, variation of ore grade or copper recovery, and variation of capital cost are approximately equal.

Table 22-2: Results of Economic Analysis

	NPV @ 0% C$000	NPV @ 5% C$000	NPV @ 8% C$000	NPV @ 10% C$000	IRR %	Payback Years
Base Case	$98,920	$40,307	$14,451	$116	10.0%	5.3
SEC Price - $3.63	$155,993	$86,122	$55,013	$37,656	15.6%	4.1
Spot Price - $3.75	$171,740	$98,735	$66,166	$47,972	17.1%	3.8
Copper Price +20%	$183,523	$108,085	$74,390	$55,552	18.2%	3.7
Copper Price -20%	$11,405	($31,546)	($49,868)	($59,796)	1.1%	7.6
Capex +20%	$75,636	$15,284	($11,238)	($25,900)	6.6%	5.8
Capex -10%	$110,512	$52,629	$27,060	$12,868	12.1%	5.0
Opex +20%	$59,162	$6,205	($16,781)	($29,387)	5.7%	6.2
Opex -20%	$137,986	$73,321	$44,479	$28,371	14.4%	4.2
Recovery +5%	$119,723	$57,113	$29,381	$13,963	12.1%	4.9
Recovery -10%	$56,857	$5,898	($16,307)	($28,516)	5.7%	6.1

22.13 TAXATION

22.13.1 Corporate Income Tax

The Carmacks project is evaluated with a 30% combined federal and territorial corporate income tax rate of taxable income. The taxable income was reduced by loss carry forwards from the previous year of approximately $0.4 million and the first year's loss. In addition, a deduction of depreciation for CCA class 41A assets is being taken which results in no income tax being paid until the initial capital is fully depreciated. These deductions against income are applied each year, but cannot create a loss.

Corporate income taxes paid is estimated to be $41.6 million for the life of the mine.

22.13.2 Yukon Territorial Mining Royalty

The Yukon levies a net profits royalty based on the annual output of a mine up to a maximum rate of 12% on output of greater than $35 million. "Output" is determined by adjusting operating income for a number of factors and is different from net income for corporate tax purposes. Generally, output is mining revenues less operating expenses, capital cost depreciation (calculated using the declining balance method at a rate of 15%), and development cost (amortized over the life of the project). A number of items are excluded from this calculation including interest payments and third party royalty payments.

The Yukon mining royalty is calculated by applying a sliding scale rate of 3 – 12% based on the amount of output. It is estimated that $14.9 million will be paid in mining royalties over the life of the mine.

Table 22-3: Carmacks Copper Project Statistics

Total Reserves:	11.6 Mt @ 0.977% Copper
Production Quantities	
Mining Method:	Open Pit
Waste:Ore Ratio:	5.1:1
Total Material Moved:	70.0 Mt
Ore Processing Method	Crushed Heap Leach, 80% minus
Processing Rate:	nominal 5,000 t/d
Recovery (life-of-mine):	85%
Metal Production:	211,543,000 Pounds Cathode Copper
Production Life:	Pit 7 Years
	Pad 8 Years
Prices (In CDN Dollars)	
Copper Price:	C$3.20
Exchange Rate:	C$1 = US$1.00
Capital Cost (In CDN Dollars)	
Initial Capital Cost	$177,558,190
Sustaining Capital	$4,700,000
Project Economics with Equity Financing After Taxes (In CDN Dollars)- Base Case	
Net Present Value (NPV) at 0%	$98,920,000
Net Present Value (NPV) at 5%	$40,307,000
Net Present Value (NPV) at 8%	$14,451,000
Net Present Value (NPV) at 10%	$116,000
*Internal Rate of Return	10.0%
Payback	5.3 Years
**Cash Cost	$1.59/pound copper
Operating Cost – Year 2	$29.64/tonne of ore
* After Tax and Royalty	
** Before Reclamation, Interest, Tax, and Royalty	

23 ADJACENT PROPERTIES

There are no adjacent operational mining properties that would lead to a better understanding of this property. See Section 8.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT EXECUTION PLAN

24.1.1 Description

The Project Execution Plan describes, at a high level, how the project will be carried out. This plan contains an overall description of what the main work focuses are, project organization, the estimated schedule, and where important aspects of the project will be carried out.

The project execution proposed incorporates an integrated strategy for engineering, procurement and construction management (EPCM). The primary objective of the execution methodology is to deliver the project at the lowest capital cost, on schedule, and consistent with the project standards for quality, safety, and environmental compliance.

24.1.2 Objectives

The project execution plan has been established with the following objectives:

- To maintain the highest standard of safety so as to minimize incidents and accidents;
- To design and construct a process plant, together with the associated infrastructure, that is cost-effective, achieves performance specifications and is built to high quality standards;
- To design and operate the mine using proven methods, techniques and equipment;
- To optimize the project schedule to achieve an operating plant in the most efficient and timely manner within the various constraints placed upon the project; and
- To comply with the requirements of the conditions for the construction and operating license approvals.

24.1.3 Plan of Approach

24.1.3.1 Philosophy

This section describes the execution plan for advancing the Carmacks Copper Project from the current Feasibility Report stage to production. The project execution plan will ensure that key project processes and procedures are in place that will:

- Develop a Master Schedule;
- Consider significant project logistics;
- Develop a project procedures manual with a project communication and document control plan;
- Develop and implement site communications, construction infrastructure, and water supply for an early and efficient startup;
- Plan for early construction mobilization;
- Develop and execute project control procedures and processes;
- Perform constructability reviews;
- Implement project accounting and cost control best practices;

- Issue a cost control plan and a control budget; and
- Oversee project accounting.

CNMC intends to utilize an Engineering, Procurement and Construction Management (EPCM) approach utilizing multiple hard money and low unit cost prime contracts for CM, as the recommended method for executing the project. The capital cost estimate is based on this methodology. Mining and pre-production work activities as well as site road construction will be performed by contractors selected through a pre-qualification and pre-tending process. Based on the relatively modest size of the project, construction is highly to be performed exclusively by Yukon based companies.

Some items affecting the project are:

- Ability to start work that does not require engineering;
- Availability of construction and engineering resources;
- Experience of the qualified firms considered and their typical and proposed approach;
- An approach that utilizes the best resources available (matching contractors to the size of each contract)

As previously mentioned, M3 utilized an EPCM approach as the basis for the capital cost estimate. This approach provides for contracts that would include civil, concrete, structural steel, mechanical, piping, electrical and instrumentation.

The majority of mechanical and electrical equipment required for the project will be procured within North America. Concrete, building construction materials and timber products will be sourced primarily in the Yukon. Structural and miscellaneous steel, piping, tanks, electrical and miscellaneous process equipment will be sourced within Canada, and to the extent practical, within the region.

24.1.3.2 Engineering

The detailed engineering schedule is based on interim approval to be granted in early Basic Engineering starting in Q3 of 2013, and full EPCM release in Q1 of 2014. The design is scheduled to be 80% complete by December 2014.

Engineering will be done to match the plant protocol for drawing titles, equipment numbers and area numbers. Design will produce drawings in the International System of Units (Metric) format. Drawings and specifications will be done in English.

A site conditions specification will be done to ensure that vendors are aware of the site conditions. Individual equipment specifications will be done.

Engineering control will be maintained through drawing lists, specification lists, equipment lists, pipeline lists and instrument lists. Control of Engineering Requisitions for Quote (ERFQ) will be performed through an anticipated purchase orders list. Progress will be tracked through the use of the lists mentioned.

Concrete reinforcing steel drawings will be done using customary bar available in Canada. Reinforcing bar will be fully detailed to allow either site or shop fabrication.

Structural steel will be detailed by M3 using TEKLA software. Mechanical steel will be dictated by M3 utilizing either Inventor or TEKLA. This will allow fabrication of steel prior to the award of steel installation contracts.

Owner review of engineering progress and design philosophy will be an ongoing process.

24.1.3.3 Procurement

Procurement of long delivery equipment and materials is scheduled with their relevant engineering tasks. This will ensure that the applicable vendor information is incorporated into the design drawings and that the equipment will be delivered to site at the appropriate time and supports the overall project schedule. Particular emphasis will be placed on procuring the material and contract services required to establish the temporary construction infrastructure required for the construction program.

Procurement of major process equipment will be by the Engineering, Procurement, and Construction Management (EPCM) contractor (M3), acting as an agent for Copper North Mining Corp. (CNMC) through the use of owner approved purchase order forms. This will include all of the equipment in the equipment list as well as all of the instruments in the instrument list. Some instruments will be part of vendor equipment packages. In addition, structural steel, electrical panels, electrical lighting, major cable quantities, specialty valves and special pipe will be purchased. Contractors will be responsible for the purchase of common materials only.

Equipment and bulk material Suppliers will be selected via a competitive bidding process. Similarly, construction contractors will be selected through a pre-qualification process followed by a competitive bidding process. It is envisaged that the project will employ a combination of lump sum and unit price contracts as appropriate for the level of engineering and scope definition available at the time contract(s) are awarded.

It is intended that equipment will be sourced on a world-wide basis, assessed on the best delivered price and delivery schedule, fit-for-purpose basis. Preference will be made to procure goods from Canada.

Equipment will be purchased FOB at the point of manufacture or nearest shipping port for international shipments. A logistics contractor will be selected to coordinate all shipments of equipment and materials for the project and arrange for ocean and overland freight to the job site.

The EPCM contractor will be responsible for the receipt of the major equipment and materials at site. The equipment and materials will be turned over to the installation contractor for storage and safe keeping until installed. Bulk piping and electrical materials and some minor equipment will be made part of the construction contracts, and as such will be supplied by the various construction contractors. It is expected that each construction contractor provide for the receipt, storage, and distribution of materials and minor equipment they purchased.

The EPCM contractor will establish a list of recommended pre-qualified vendors for each major item of equipment for approval by Copper North. The EPCM contractor will prepare the tender documents, issue the equipment packages for the bid, prepare a technical and commercial evaluation, and issue a letter of recommendation for purchase for approval by Copper North Mining Corp. Copper North through the assistance of the EPCM contractor will conduct the commercial negotiations with the recommended vendor and advise the EPCM contractor of the negotiated terms for preparation of the purchase documents. When approved, the EPCM contractor will issue the purchase order, track the order, and expedite the engineering information and delivery of the equipment to the site.

24.1.3.4 Inspection

The EPCM contractor will be responsible to conduct QA/QC inspections for major equipment during the fabrication process to ensure the quality of manufacture and adherence to specifications. Levels of inspection for major equipment will be identified during the bidding stage, which may range from receipt and review of the manufacturer's quality control procedures to visits to the vendor's shops for inspection and witnessing of shop tests prior to shipment of the equipment. Where possible, inspectors close to the point of fabrication will be contracted to perform this service in order to minimize the travel cost for the project. Some assistance may also be provided by the EPCM engineering design team.

24.1.3.5 Expediting

The EPCM contractor will also be responsible to expedite the receipt of vendor drawings to support the engineering effort as well as the fabrication and delivery of major equipment to the site. An expediting report will be issued at regular intervals outlining the status of each purchase order in order to alert the project of any delays in the expected shipping date or issue of critical vendor drawings. Corrective action can then be taken to mitigate any delay.

The logistics contractor will be responsible to coordinate and expedite the equipment and material shipments from point of manufacture to site, including international shipments through customs.

24.1.3.6 Project Services

The EPCM contractor will be responsible for management and control of the various project activities and ensure that the team has appropriate resources to accomplish CNMC's objectives.

24.1.4 Construction

24.1.4.1 Construction Methodology

The 2014 construction program is scheduled to start in Q1 2014 before the frost is completely out of the ground. The work includes clearing and grubbing of the plant site, mass earthwork for site development, project access road and in-plant roads. Concrete foundations for the process building and other support structures will be constructed beginning in Q2 2014. The Sprung type structures for the process facilities and truck shop will be erected in the Q3 of 2014. The construction camp and associated services will be installed early in Q3 2014.

Construction work will continue under cover for the process facilities through the winter of 2014-2015 and will be complete by Q4 2015. Earthworks associated with the heap leach facility, confining embankment and related facilities will continue until late October or early November as the weather permits. Earthworks for the 2015 season will resume as soon as the weather allows. This work will include completion of the confining embankment, surface diversions, and ponds. Other work will include construction of the heap leach pad lining system, erection of the PLS pumping riser system, and crushing and application of the ore overliner. Ore stacking on the heap is scheduled to begin in Q4 2015.

24.1.4.2 Construction Management

Construction Management will be done as agents for the Owner using multiple prime contracts for each of the major work disciplines. The contracting plan is based on utilizing a series of local contractors to execute the construction work packages. The EPCM contractor will pre-qualify local contractors and prepare tender documents to bid and select the most qualified contractor for the various work packages. Some work packages will include the design, supply, and erection for specific facilities which are specialized in nature. The EPCM contractor will be comprised of individuals capable of coordinating the construction effort, supervising and inspecting the work, performing field engineering functions, administering contracts, supervising warehouse and material management functions, and performing cost control and schedule control functions. These activities will be under the direction of a resident construction manager and a team of engineers, and locally hired supervisors, and technicians. There would also be a commissioning team to do final checkout of the project.

Some site services will be contracted to third party specialists, working under the direction of the resident construction manager. Construction service contracts identified at this time include the following:

- Field survey services;
- QA/QC testing services; and
- Site security (If required).

24.1.5 Contracting Plan

Contracting is an integral function in the project’s overall execution. Contracting for the Carmacks Copper Project will be done in full accord with the provisions of the CNMC/EPCM contract.

A combination of vertical, horizontal, and design construct contracts may be employed as best suits the work to be performed, degree of engineering and scope definition available at the time of award. A site installed concrete batch plant will supply concrete to all construction contractors. The Owner furnished construction camp will be utilized by all construction contractors. Camp operations will be supplied by a contracted service provider.

The mass earthwork contract will cover all mine pre-stripping, clearing, grubbing, bulk excavation, and leach pad preparation. This approach will result in economy of scale and eliminate interfacing issues which would arise if multiple contractors were employed. The contractor will require only one major mobilization for all work.

As part of the contracting strategy, a list of proposed contract work packages has been developed to identify items of work anticipated to be assembled into a contract bid package. Depending upon how the project is ultimately executed and the timing, several work packages may be combined to form one contract bid package. The following table represents the Proposed Contract Work Package list:

Table 24-1: Proposed Contract Work Package List

No.	Bid Packages:	Comments
1	Materials Testing	Soils, Concrete & Structural Materials
2	Survey	Confirm Existing Terrain. Create Topo of Roadway, Heap Leach & Plant Site Areas
3	Access Road	Includes Roadway Drainage Culverts & Trenching
4	34 kV Power Line	Package with no. 5
5	34 kV Substation	Includes Emergency Generator Installation & Testing
6	Field Electrical Distribution - Sub Station to Process Areas, Camp & Water Pumping	Duct Banks from Switch Gear
7	Water Wells & Supply System - Yard Water Piping	Includes Fire Suppression
8	Septic System - Sewer Piping, Plant & Distribution Field	May include a Lift Station(s) if Site does not accommodate Gravity Sewer System
9	Clearing, Grubbing, Site Excavation & Site Preparation - All Areas	This could include Local Excavation for Building Footers
10	Heap Leach Excavation	Package with no. 10
11	Heap Leach Under Drains, Liner & Collection	Package with no. 9
12	Concrete Work - All Areas	This should Include Batch Plant Construction & Operation
13	Building Enclosures - Sprung Structures	from foundation bolts. Includes: SX, EW, Tank Farm, Truck Shop
14	Acid Plant Building Enclosure - Steel	From Foundation Bolts on Up
15	Tank Farm, SX & EW - Internal Structures, Mechanical, Electrical & Instrumentation	Includes all Equipment Installation & Testing
16	Acid Plant Complete - Mechanical, Electrical & Instrumentation	Includes all Equipment Installation & Testing
17	Crusher & Conveying - Structural, Mechanical, Electrical & Instrumentation	Includes all Equipment Installation & Testing
18	Field Erected Tanks	Should Include Field Weld Specifications
19	Yard Process Piping	Possibly Bundle with Tank Farm/Acid Plant
20	Camp Supply, Erect, Structural & Electrical	Includes Offices, Guardhouse, Generator, Plumbing and Furnishings
21	Camp Operation & Maintenance	Includes Catering, Site Services, Security, Transportation, etc.
22	Truck Shop & Wash - Internal Structures, Mechanical & Electrical	From Foundation Bolts. (Includes all items within the Building & Testing)
23	Fuel Station Installation Complete	Includes Propane
24	Fencing	Security & Substation
25	Exterior lighting	Plant Operational Area(s)

24.1.6 Labour

The labour market in northwestern Canada at this time continues to be minimal. According to the *Labour Market Bulletin for the Northwest Territories, Nunavut and Yukon, Spring 2012*,* while employment has been growing steadily over the past eight (8) years, there has been little to no growth over the past year to year and half due to the lack of new mines under construction. Currently mining construction is at a low level, although this should pick up within the next two (2) years as several key projects in all three territories begin development.

Many contractors in the Yukon are open shop. The local town of Carmacks (with a population of 400) and nearby communities will be able to supply a limited number of construction personnel. Most of the construction labour is expected to come from Whitehorse. Skilled trades may also have to come from Alberta and British Columbia.

*Labour Market Information (LMI) Division, Service Canada, Northwest Territories, Nunavut and Yukon. "Labour Market Bulletin for the Northwest Territories, Nunavut, and Yukon" 20 June 2012. statcan.gc.ca. LMI 20 June 2012 <http://www.statcan.gc.ca>

24.1.7 Project Schedule

At the present time, the study has developed a sequence of effort that should be followed as well as an estimated schedule through which the project will likely proceed. The schedule is comprised of three main components consisting of Milestones & Basic Engineering, Detail Engineering & Procurement and Construction & Start Up activities. The schedule (by component) is shown at the end of this section.

The schedule is largely self-explanatory and assumes that basic engineering will commence in the third quarter of 2013. The key milestone assumptions that drive the schedule are the completion of environmental assessment in Q3 of 2013 leading to a Quartz Mining Licence and full project release in Q2 of 2014. While it may be possible to recover schedule if these key dates slip, a significant slippage could lead to the loss of the entire 2014 construction season, due to weather constraints, and an associated project delay of one year.

24.1.7.1 Construction Completion and Turn-over Procedure

The Construction Completion Procedure is part of the Construction Quality Plan as well as the project specific Commissioning Plan. Contractors are to enter into contractual agreements with CNMC to perform certain portions of the work, which includes quality control of their work.

The Commissioning Plan (as further defined under 24.1.6) will be developed and implemented to insure a step-by-step, documented process and procedure for all mechanical, process, electrical/instrumentation completion, checkout and pre-operational testing. Pre-operational testing and commissioning will take place concurrent with mechanical completion. Pre-operational testing is currently scheduled to commence in Q3 of 2015 and wet commissioning and start-up is scheduled to commence in Q4 of 2015.

24.1.8 Quality Plan

A project specific, Quality Plan will be developed and implemented on the site. The Quality Plan is a management tool for the EPCM contractor, through the construction contractors, to maintain the quality of construction and installation on every aspect of a project. The plan, which consists of many different manuals and subcategories, will be developed during the engineering phase and available prior to the start of construction.

24.1.9 Commissioning Plan

The Commissioning Plan will also be project specific and is characterized as the transition of the constructed facilities from a status of “mechanically” or “substantially” complete to operational as defined by the subsystem list that will be developed for the project. The commissioning group will systemically verify the functionality of plant equipment, piping, electrical power and controls. This test and check phase will be conducted by discrete facility subsystems. The tested subsystems will be combined until the plant is fully functional. Start-up, also a commissioning group responsibility, will progressively move the functional facilities to operational status and performance.

In addition to these activities, the commissioning portion of the work will also include coordination of facilities operations training, maintenance training and turnover of all compiled commissioning documentation in an agreed form.

24.1.10 Health and Safety Plan

The Health and Safety Plan (HASP) will be established for the construction of the Carmacks Project and any other authorized work at the project site. The HASP covers all contractor personnel working on the project and any other authorized work for the project.

The HASP specifies regulatory compliance requirements, training, certifications and medical requirements necessary to complete the project for all personnel and contractors involved in the project. Along with the Operations Procedures, the HASP is to be followed by all Contractor personnel working at the site.

24.1.11 Traffic Management Plan

In order to minimize the disruption to the residents in the village of Carmacks and the public users of the Freegold Road during both construction and operations, CNMC will adopt a Traffic Management Plan to guide those travelling between the Klondike Highway and the mine site. The plan will be developed in collaboration with the EPCM contractor, construction contractors, suppliers and transportation companies.

24.1.12 Camp Transition

The intention is to build the camp initially as a construction camp. However, it is expected that a portion(s) of the camp will also become the permanent operations personnel camp. There currently exist two (2) potential camp site locations under consideration; (1) within the limits of the city of Carmacks, (2) the mine site location. The camp usage will transition from construction to operations during the latter stages of construction in 2015 and prior to start-up.

24.1.12.1 Project Organization

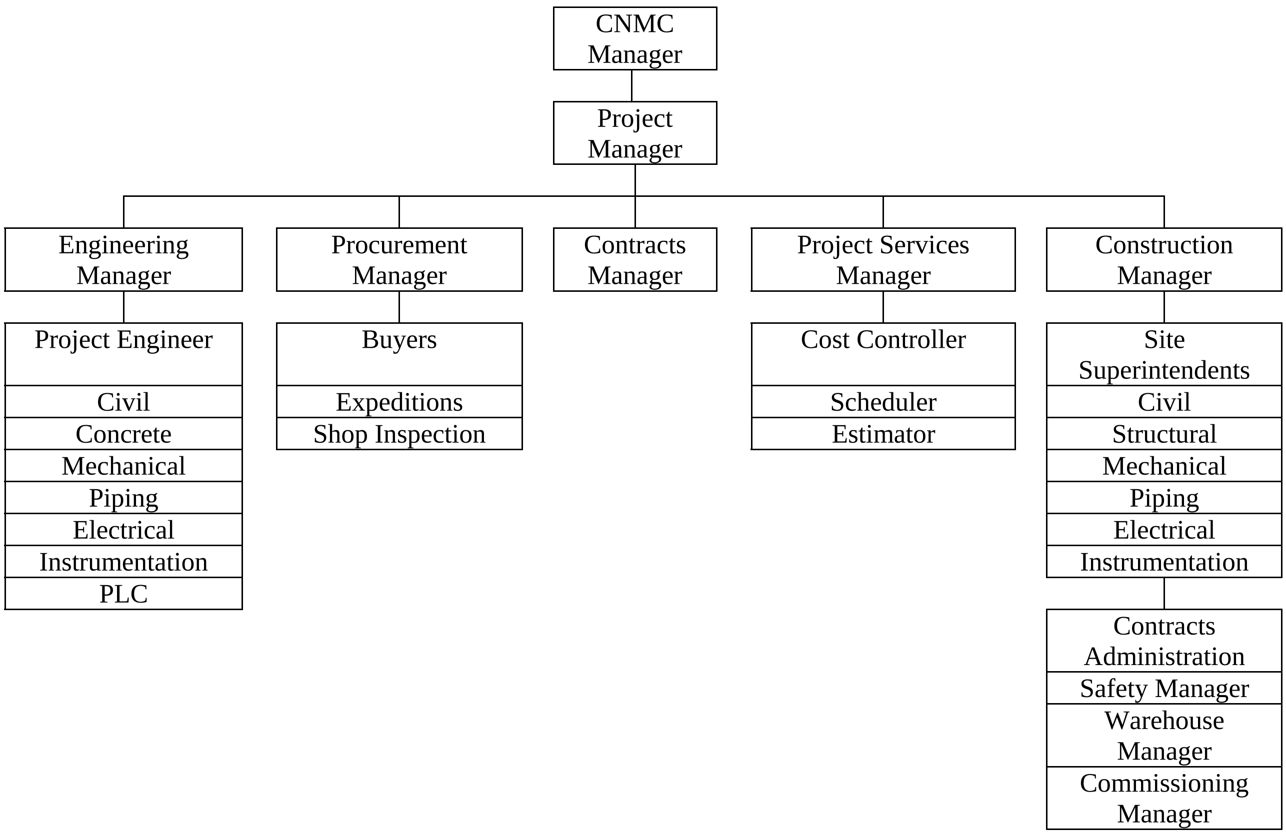
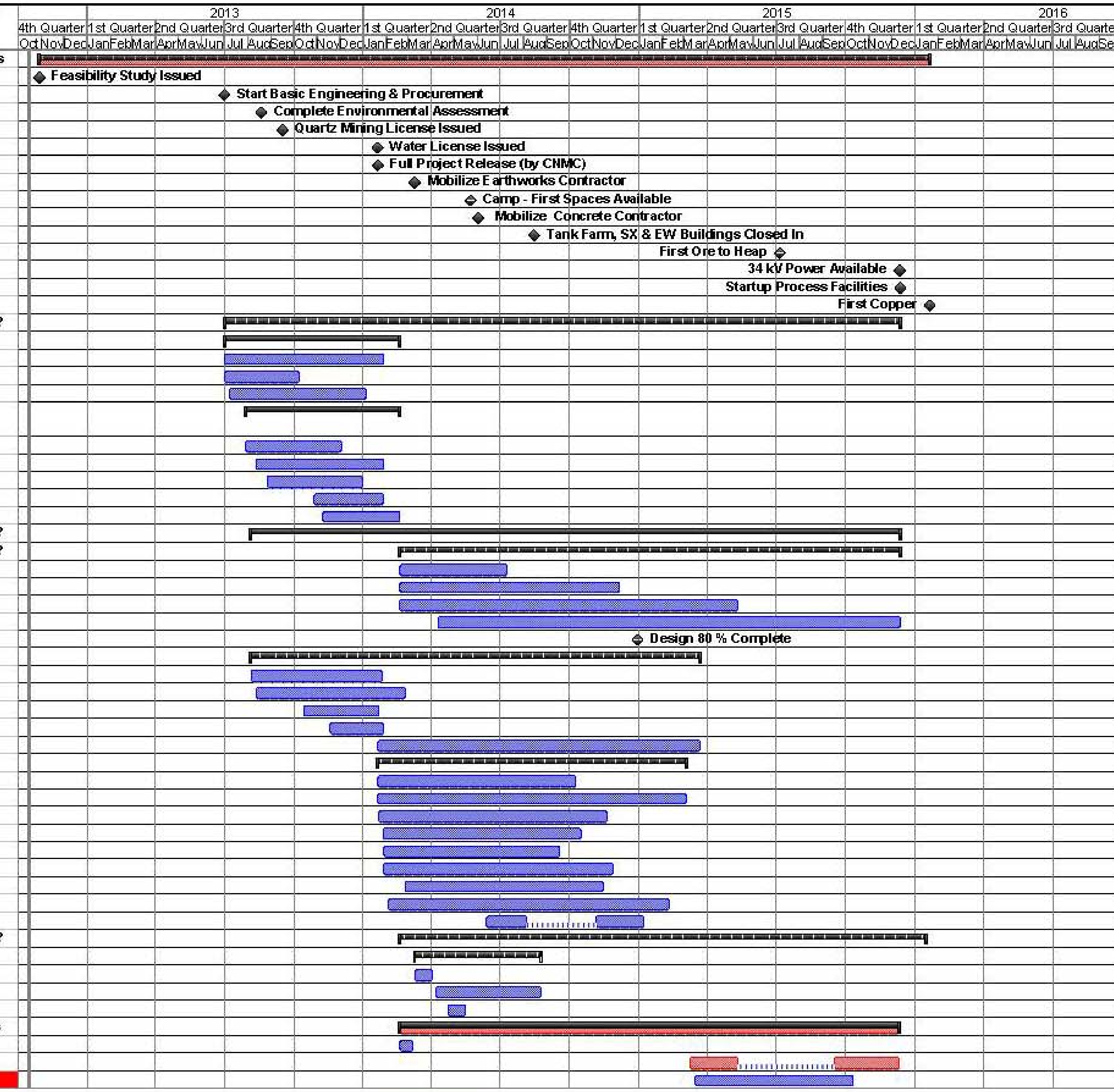


Figure 24-1: Project Organization Block Diagram

CARMACKS PROJECT
Summary Schedule
M3-PN120010

October 16, 2012

ID	Task Name	Est Start	Est Finish	Est Duration
1	**MILESTONES**	**Mon 10/29/12**	**Wed 1/20/16**	**1010.13 days**
2	Feasibility Study Issued	Mon 10/29/12	Mon 10/29/12	0 days
3	Start Basic Engineering & Procurement	Mon 7/1/13	Mon 7/1/13	0 wks
4	Complete Environmental Assessment	Mon 8/19/13	Mon 8/19/13	0 days
6	Quartz Mining License Issued	Mon 9/16/13	Mon 9/16/13	0 days
5	Water License Issued	Mon 1/20/14	Mon 1/20/14	0 days
7	Full Project Release (by CNMC)	Mon 1/20/14	Mon 1/20/14	0 days
8	Mobilize Earthworks Contractor	Mon 3/10/14	Mon 3/10/14	0 days
10	Camp - First Spaces Available	Thu 5/22/14	Thu 5/22/14	0 days
9	Mobilize Concrete Contractor	Mon 6/2/14	Mon 6/2/14	0 days
11	Tank Farm, SX & EW Buildings Closed In	Thu 8/14/14	Thu 8/14/14	0 days
12	First Ore to Heap	Mon 7/6/15	Mon 7/6/15	0 days
13	34 kV Power Available	Fri 12/11/15	Fri 12/11/15	0 days
14	Startup Process Facilities	Sat 12/12/15	Sat 12/12/15	0 days
15	First Copper	Wed 1/20/16	Wed 1/20/16	0 wks
16	**ENGINEERING and PROCUREMENT**	**Mon 7/1/13**	**Sat 12/12/15**	**767.63 days?**
17	**BASIC ENGINEERING**	**Mon 7/1/13**	**Mon 2/17/14**	**199 days?**
18	Design of Confining Embankment and Ponds	Mon 7/1/13	Mon 1/27/14	181 days?
19	Access Road, Ancillaries, Main Substation	Mon 7/1/13	Mon 10/7/13	85 days
20	34 kV Power Line (By Others)	Mon 7/8/13	Sat 1/4/14	156 days
21	**PREPARATION for BID,EVALUATION AND CONDITIONAL AWARD (SCHEDULE CRITICAL EQUIPMENT)**	**Mon 7/29/13**	**Mon 2/17/14**	**175 days**
22	Major Substation Components, conditional award	Mon 7/29/13	Mon 12/2/13	109 days
23	Acid Plant Supply / Erect Package, conditional award	Mon 8/12/13	Mon 1/27/14	145 days
24	Rectifiers, conditional award	Mon 8/26/13	Mon 12/30/13	109 days
26	Camp and Support Facilities, conditional award	Mon 10/28/13	Mon 1/27/14	79 days
25	Earthworks and Leach Pad, conditional award	Fri 11/8/13	Mon 2/17/14	87 days
27	**DETAIL ENGINEERING**	**Mon 8/5/13**	**Sat 12/12/15**	**737.63 days?**
28	**PLANT FACILITIES**	**Tue 2/18/14**	**Sat 12/12/15**	**568.63 days?**
29	Crushing and Conveying	Tue 2/18/14	Wed 7/9/14	122 days?
30	Tank Farm	Tue 2/18/14	Fri 12/5/14	250 days
31	SX/EW	Tue 2/18/14	Mon 5/11/15	384 days
32	Construction Support	Wed 4/9/14	Sat 12/12/15	525 days
33	Design 80 % Complete	Tue 12/30/14	Tue 12/30/14	0 wks
34	**BID PACKAGES**	**Mon 8/5/13**	**Sat 3/21/15**	**510 days**
35	Major Crushing / Conveying Components	Mon 8/5/13	Sat 1/25/14	150 days
36	Major SX/EW Components	Mon 8/12/13	Mon 2/24/14	169 days
39	Major Mining Equipment	Mon 10/14/13	Mon 1/20/14	85 days
37	Manufactured Buildings	Mon 11/18/13	Mon 1/27/14	61 days
38	Balance of Equipment and Construction Packages	Mon 1/20/14	Sat 3/21/15	366 days
40	**MAJOR EQUIPMENT SUPPLY & DELIVERY**	**Mon 1/20/14**	**Tue 3/3/15**	**350 days**
41	Rectifiers	Mon 1/20/14	Wed 10/8/14	225 days
42	Major Substation Components	Mon 1/20/14	Tue 3/3/15	350 days
47	Mining Equipment	Tue 1/21/14	Wed 11/19/14	260 days
45	Major Crushing / Conveying Components	Mon 1/27/14	Wed 10/15/14	225 days
43	Acid Plant Building Steel	Tue 1/28/14	Wed 9/17/14	200 days
44	Acid Plant Equipment	Tue 1/28/14	Wed 11/26/14	260 days
46	Major SX/EW Components	Tue 2/25/14	Thu 11/13/14	225 days
48	Structural/ Mechanical/Piping bulks	Mon 2/3/14	Mon 2/9/15	319 days
49	Electrical/Instrumentation bulks	Fri 6/13/14	Wed 1/7/15	100 days
50	**CONSTRUCTION**	**Tue 2/18/14**	**Fri 1/15/16**	**597.63 days?**
51	**TEMPORARY FACILITIES**	**Mon 3/10/14**	**Sat 8/23/14**	**144 days**
53	Setup Diesel Generators	Mon 3/10/14	Tue 4/1/14	20 days
54	Camp and Support Facilities	Mon 4/7/14	Sat 8/23/14	120 days
52	Major Earthworks Contractor Mobilization	Wed 4/23/14	Thu 5/15/14	20 days
55	**INFRASTRUCTURE**	**Tue 2/18/14**	**Fri 12/11/15**	**567.63 days**
56	Access Road Open for Construction	Tue 2/18/14	Thu 3/6/14	15 days
58	Main Substation	Mon 3/9/15	Fri 12/11/15	130 days
57	34 kV Power Line (by others)	Mon 3/16/15	Sat 10/10/15	180 days



Task		Summary		Inactive Milestone		Duration-only		Start-only
Split		Inactive Task		Inactive Summary		Manual Summary Rollup		Finish-only
Milestone		Inactive Task		Manual Task		Manual Summary		

Carmacks Summary Schedule 16-Oct-12

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Figure 24-2: Carmacks Project Summary Schedule

CARMACKS PROJECT
Summary Schedule
M3-PN120010

October 16, 2012

ID	Task Name	Est Start	Est Finish	Est Duration
59	**PRE PRODUCTION MINING (by Earthworks Contracor)**	**Sat 3/29/14**	**Mon 6/15/15**	**378.63 days**
60	Initial Clearing & Grubbing Completed by Earthworks Contractor	Sat 3/29/14	Sat 3/29/14	0 days
61	PreStrip and Stockpile	Mon 3/31/14	Mon 6/15/15	300 days
62	**EARTHWORKS**	**Fri 3/7/14**	**Fri 1/15/16**	**582.63 days?**
63	Plantsite Clearing / Grubbing	Fri 3/7/14	Sat 3/29/14	20 days
64	Rough Grade / Plant and Crusher Area Site Prep	Mon 3/10/14	Sat 5/17/14	60 days
65	**LEACH PAD DEVELOPMENT**	**Wed 4/2/14**	**Fri 1/15/16**	**560.63 days?**
66	Leach Pad Foundation and Drains	Wed 4/2/14	Sat 10/4/14	160 days
68	Embankment, Surface Diversions, Ponds - Phase I	Tue 4/8/14	Sat 10/4/14	155 days
67	Leach Pad soil and composite lining	Mon 4/13/15	Mon 8/24/15	115 days?
69	Embankment, Surface Diversions, Ponds - Phase II	Mon 4/13/15	Mon 9/14/15	133 days?
70	Crushing Overliner Material	Mon 6/8/15	Thu 7/23/15	40 days
71	Place Overliner Material (crushed ore)	Mon 7/6/15	Mon 8/17/15	37 days
72	Risers and Pump Station	Fri 12/11/15	Fri 1/15/16	30 days
73	Initial Irrigation	Fri 1/15/16	Fri 1/15/16	0 days
74	**CRUSHING and CONVEYING**	**Mon 8/4/14**	**Wed 7/29/15**	**309 days**
75	Excavation	Mon 8/4/14	Sat 9/6/14	30 days
76	Concrete	Mon 9/15/14	Sat 11/22/14	60 days
77	Structural Steel Erection	Mon 11/10/14	Wed 12/31/14	45 days
78	Building Closed In	Thu 1/1/15	Fri 1/23/15	20 days
79	Mechancial Steel and Equipment	Thu 1/1/15	Mon 7/6/15	160 days
80	Electrical and Instrumentation	Mon 6/8/15	Thu 7/23/15	40 days
81	Mechanical Completion/Pre Op Testing (with generators)	Tue 7/7/15	Wed 7/29/15	20 days
82	**PROCESS PLANT FACILITIES**	**Wed 4/2/14**	**Fri 12/11/15**	**530.63 days**
83	**Tank Farm, SX & EW**	**Wed 4/2/14**	**Tue 8/11/15**	**426 days**
84	Site Preparaion	Wed 4/2/14	Fri 5/23/14	45 days
85	Concrete	Mon 5/12/14	Sat 9/27/14	120 days
86	Erect Superstructure (fabric bldg.)	Mon 6/23/14	Wed 8/13/14	45 days
87	Building Closed In	Thu 8/14/14	Thu 8/14/14	1 day
88	Mechanical and Piping	Fri 8/15/14	Mon 5/4/15	225 days
89	Electrical and Instrumentation	Thu 10/30/14	Fri 6/19/15	200 days
90	Mechanical Completion/Pre Op Testing	Sat 6/20/15	Tue 8/11/15	45 days
91	**ACID PLANT**	**Wed 4/2/14**	**Fri 12/11/15**	**530.63 days**
92	Site Preparation	Wed 4/2/14	Fri 5/23/14	45 days
93	Concrete	Sat 5/24/14	Wed 8/13/14	70 days
94	Structural Steel Erection	Thu 9/18/14	Mon 12/8/14	70 days
95	Building Closed In	Tue 11/4/14	Mon 1/12/15	60 days
96	Mechanical and Piping	Sat 12/20/14	Mon 6/1/15	140 days
97	Electrical and Instrumentation	Tue 3/24/15	Fri 7/17/15	100 days
98	Mechanical Completion/Pre Op Testing	Wed 11/18/15	Fri 12/11/15	20 days
99	**STARTUP (introduction of feed to process facilities)**	Fri 12/11/15	Sat 12/12/15	1 day

Task | Split | Milestone | Summary | Inactive Task | Inactive Task | Inactive Milestone | Inactive Summary | Manual Task | Duration-only | Manual Summary Rollup | Manual Summary | Start-only | Finish-only

Carmacks Summary Schedule 16-Oct-12

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Carmacks Project Summary Schedule (Continued)

25 INTERPRETATION AND CONCLUSIONS

25.1 GENERAL

The Carmacks Copper oxide mineral occurrence can be successfully exploited by conventional open pit mining followed by heap leaching, and solvent extraction and electrowinning.

Under the study price of cathode copper, C$3.20 per pound, the internal rate of return for the project is calculated to be 10.0% with an undiscounted after-tax net present value of C$98.9 million.

25.2 OPPORTUNITIES

M3 recognizes substantial opportunities exist to enhance the project economics including:

- Additional oxide ore reserves with present claim;
- Reported additional oxide ore resources off-property but within trucking distance;
- Potential of processing oxide stockpile from nearby existing mine;
- Evaluate contract mining in lieu of self-performance; and
- Evaluate re-conditioned equipment for haulage and select process equipment.

26 RECOMMENDATIONS

This project has sound economics as presently constituted. Additional oxide reserves should be pursued as an extended mine life would enhance financial parameters.

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APPENDIX A: FEASIBILITY STUDY CONTRIBUTORS AND PROFESSIONAL QUALIFICATIONS

CERTIFICATE of QUALIFIED PERSON

Conrad E. Huss

I, Conrad E. Huss, P.E., Ph.D., do hereby certify that:

1. I am Senior Vice President and Chairman of the Board of:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Rd., Suite 101
   Tucson, Arizona 85704
   U.S.A.

2. I graduated with a Bachelor's of Science in Mathematics and a Bachelor's of Art in English from the University of Illinois in 1963. I graduated with a Master's of Science in Engineering Mechanics from the University of Arizona in 1968. In addition, I earned a Doctor of Philosophy in Engineering Mechanics from the University of Arizona in 1970.

3. I am a Professional Engineer in good standing in the State of Arizona in the areas of Civil (No. 9648) and Structural (No. 9733) engineering. I am also registered as a professional engineer in the States of California, Illinois, Maine, Minnesota, Missouri, Montana, New Mexico, Oklahoma, Texas, Utah, and Wyoming.

4. I have worked as an engineer for a total of forty three years since my graduation from the University of Illinois. I have taught at the University level part-time for five years and as an assistant professor for one year.

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.

6. I am the principal author for the preparation of the technical report titled "Carmacks Copper Project, NI 43-101 Technical Report Feasibility Study, Volume I, Yukon Territory, Canada" (the "Technical Report"), dated October 31, 2012, prepared for Copper North Mining Corp.; and am responsible for Sections 1 through 6 and 21 through 27. I have visited the project site on 12 June 2012.

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

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

9. 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.

10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 31st day of October, 2012.

Conrad E Huss

Signature of Qualified Person

Registered Professional Engineer (Civil)
CERTIFICATE NO.
9848
CONRAD
E. HUSS
Date Signed 31 OCT 2012
ARIZONA, U.S.A.

Conrad E. Huss, P.E., Ph.D.

Print Name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

Thomas L. Drielick

I, Thomas L. Drielick, P.E., do hereby certify that:

1. I am currently employed as Sr. Vice President by:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Road, Ste. 101
   Tucson, Arizona 85704
   U.S.A.

2. I am a graduate of Michigan Technological University and received a Bachelor of Science degree in Metallurgical Engineering in 1970. I am also a graduate of Southern Illinois University and received an M.B.A. degree in 1973.

3. I am a:

   - Registered Professional Engineer in the State of Arizona (No. 22958)
   - Registered Professional Engineer in the State of Michigan (No. 6201055633)
   - Member in good standing of the Society for Mining, Metallurgy and Exploration, Inc. (No. 850920)

4. I have practiced metallurgical and mineral processing engineering and project management for 41 years. I have worked for mining and exploration companies for 18 years and for M3 Engineering and Technology, Corporation for 23 years.

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.

6. I am responsible for the preparation of Section 13 "Mineral Processing and Metallurgical Testing" and Section 17 "Recovery Methods", of the technical report titled "Carmacks Copper Project, NI 43-101 Technical Report Feasibility Study, Volume I, Yukon Territory, Canada"," dated October 31, 2012 (the "Technical Report").

7. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

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

9. 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.

10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 31st day of October, 2012.

Signature of Qualified Person

Thomas L. Drielick
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

Daniel Roth

I, Daniel Roth, P.E., do hereby certify that:

1. I am currently employed as a Civil Engineer/Project Manager at M3 Engineering & Technology Corporation located at 2051 West Sunset Road, Suite 101, Tucson, AZ, 85704.

2. I graduated with a Bachelor's of Science degree in Civil Engineering from the University of Manitoba in 1990.

3. I am a registered professional engineer in good standing in the following jurisdictions:

   - Yukon, Canada (No. 1998)
   - Alberta, Canada (No. 62310)
   - Ontario, Canada (No. 100156213)
   - New Mexico, USA (No. 17342)
   - Arizona, USA (No. 37319)

   I am also a member in good standing with the Society of Mining, Metallurgy and Exploration.

4. I have practiced engineering in the civil and environmental fields for 20 years.

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.

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

7. The Technical Report contains information relating to mineral titles, permitting, environmental issues, regulatory matters and legal agreements. I am not a legal, environmental or regulatory professional, and do not offer a professional opinion regarding these issues.

8. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 31st day of October, 2012.

Signature of Qualified Person

Daniel Roth
Print Name of Qualified Person

PROFESSIONAL
YUKON
DANIEL K. ROTH
TERRITORY
31 October 2012
ENGINEER

CERTIFICATE OF QUALIFIED PERSON

John Hull, P. Eng.

I, John Hull do hereby certify that:

1. I am a Principal Geotechnical Engineer with Golder Associates Ltd., at 500 – 4260 Still Creek Drive, Burnaby, BC, V6C 6C6.
2. This certificate applies to the technical report titled "Carmacks Copper Project, NI 43-101 Technical Report Feasibility Study, Volume I, Yukon Territory, Canada", dated October 31, 2012 (the "Technical Report").
3. I fulfill the requirements of a qualified person for the purposes of NI 43-101 based on my academic qualifications, professional membership and relevant experience, as set out below:

   a. I hold the following academic qualifications:

BSc	Queen's University, Kingston	1972
MSc	Queen's University, Kingston	1973

   b. I am a member in good standing of the following professional and technical associations:

APEGBC	9835
APEGNWT	L1045
APEY	1562

   c. I have worked in the minerals industry as an engineer continuously since 1998, a period of 14 years.

4. I have most recently inspected the property on July 10, 2012 for 1 day.
5. I am responsible for section(s) 18 and 20 of the Technical Report.
6. I am independent of Copper North Mining Corp. as defined in section 1.5 of NI 43-101.
7. My prior involvement with the property includes: annual site inspection in 2010 and support of engineering effort for the heap leach pad and waste rock storage area since 2007.
8. I have read and am familiar with NI 43-101 and the sections of the Technical Report for which I am responsible. To the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 31st day of October, 2012

John Hull, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Michael G. Hester

I, Michael G. Hester, do hereby that:

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

2. This certificate applies to the Technical Report titled "Carmacks Copper Project, NI 43-101 Technical Report Feasibility Study, Volume 1, Yukon Territory, Canada" (the "Technical Report"), dated October 31, 2012.

3. I fulfill the requirements of a "Qualified Person" for the purposes of NI 43-101 based on my academic qualifications, professional membership, and relevant experience.

4. I hold the following academic qualifications:

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

5. 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)
   The Canadian Institute of Mining, Metallurgy and Petroleum (CIM Member # 100809)

6. I have worked in the minerals industry as an engineer continuously since 1979, a period of 33 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 also 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 was also employed as a staff engineer for Pincock, Allen & Holt, Inc. from 1979 to 1983.

7. I have most recently inspected the property on May 16-17, 2007 for a period of two days.

8. I am responsible for Section 15, Mineral Reserve Estimates, and Section 16, Mining Methods.

9. I am independent of Copper North Mining Corp. as defined in Section 1.5 of NI 43-101.

10. My prior involvement with the property includes work on the Feasibility Study conducted by Western Copper Corporation dated May 2007.

11. I have read and am familiar with NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with the instrument.

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

13. I consent to the filing of this report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

Dated this 31st day of October 2012, at Tucson, Arizona.

Michael G. Hester, FAusIMM

CERTIFICATE OF QUALIFIED PERSON

Gilles Arseneau, Ph.D., P. Geo.

I, Gilles Arseneau do hereby certify that:

1. I am an independent geological consultant with a business address at 2200-1066 West Hastings Street, Vancouver, BC, V6E 3X2.

2. This certificate applies to the technical report titled "Carmacks Copper Project, NI 43-101 Technical Report Feasibility Study, Volume I, Yukon Territory, Canada", dated October 31, 2012 (the "Technical Report").

3. I fulfill the requirements of a qualified person for the purposes of NI 43-101 based on my academic qualifications, professional membership and relevant experience, as set out below:

   a. I hold the following academic qualifications:

B.Sc.	University of New Brunswick	1979
M.Sc.	University of Western Ontario	1984
Ph.D.	Colorado School of Mines	1995

   b. I am a member in good standing of the following professional and technical associations:

Association of Professional Engineers and Geoscientists of British Columbia	#23474

   c. I have practiced my profession in mineral exploration continuously since graduation. I have over twenty years of experience in mineral exploration and I have over ten years of experience preparing mineral resource estimates using block-modeling software.

4. I have most recently inspected the property on May 15, 2007 for two days.

5. I am responsible for section(s) 7 to 12 and section 14 of the Technical Report.

6. I am independent of Copper North Mining Corp. as defined in section 1.5 of NI 43-101.

7. My prior involvement with the property includes: I am the co-author of a technical report titled "2011 Qualifying Report for the Carmacks Copper Deposit, Yukon Territory", dated 17 October, 2011 and the author of a technical report and mineral resource estimate prepared for the Carmacks property in 2007

8. I have read and am familiar with NI 43-101 and the sections of the Technical Report for which I am responsible. To the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

9. As of the effective date of this technical report, to the best of my knowledge, information and belief, Sections 7 to 12 and section 14 of Technical Report contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 31st day of October, 2012

G. Arseneau

Gilles Arseneau, Ph.D., P. Geo.