COPPER NORTH MINING CORP.

2011 ANNUAL QUARTZ MINING LICENCE REPORT

Submitted to Yukon Government, Energy Mines and Resources

Yukon Quartz Mining Licence QML-0007

March 2012

Prepared by:

ACCESS

2011 ANNUAL QUARTZ MINING LICENCE REPORT

Submitted to Yukon Government Energy Mines and Resources

Yukon Quartz Mining Licence QML-0007

Carmacks Copper Project, Yukon Territory

Submitted by:

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EXECUTIVE SUMMARY

Activities at the mine site during the period 1 January 2011 to 31 December 2011 have been limited. Site activities consisted of five site visits to monitor water quality and flows, the Annual Engineer's Inspection and maintenance work undertaken in response to that inspection, as well as an exploration program to map and sample geological material along the proposed mine access road. Copies of the associated documents have been attached to this report. No development activities were undertaken in 2011.

Geochemical testing on spent ore from column tests was been carried out in 2008, with a second set beginning in 2010. Results of the static testing and 56 week humidity cell tests from the 2010 program, as well as a comparison with 2008 results, is appended.

Security in the amount of $80,300 has been posted with Yukon against the liability incurred to date as a result of exploration activities.

This report has been formatted to respond to the specific requirements in the QML even though there may be no corresponding project undertakings.

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1.0 INTRODUCTION

This Annual Report has been prepared for Copper North Mining Corp. and covers the period from January 1, 2011 to December 31, 2011 as required by Clauses 16.5 and 16.6 of Quartz Mining Licence (QML) QML-0007. As of January 19, 2012 the assignment of QML-007 was authorized from Carmacks Copper Limited to Carmacks Mining Corp, a wholly-owned subsidiary of Copper North Mining Corp. A "2011 Qualifying Report for the Carmacks Copper Deposit" (NI 43-101 Technical Report) was prepared to support the spin out of Copper North Mining Corp. from Western Copper Corporation and is provided in Appendix A.

This report provides a summary of activities at the Carmacks Copper Property for the reporting year, including: but not limited to physical stability inspection and minor maintenance activities, and water quality monitoring.

Few site activities occurred that would normally form a part of this report in future years, once major project permitting is completed. Additional sections and information will be added as necessary to accommodate expanded reporting requirements from future mine development and related plans.

The preliminary mine layout (not yet constructed) is illustrated in Figure 1.

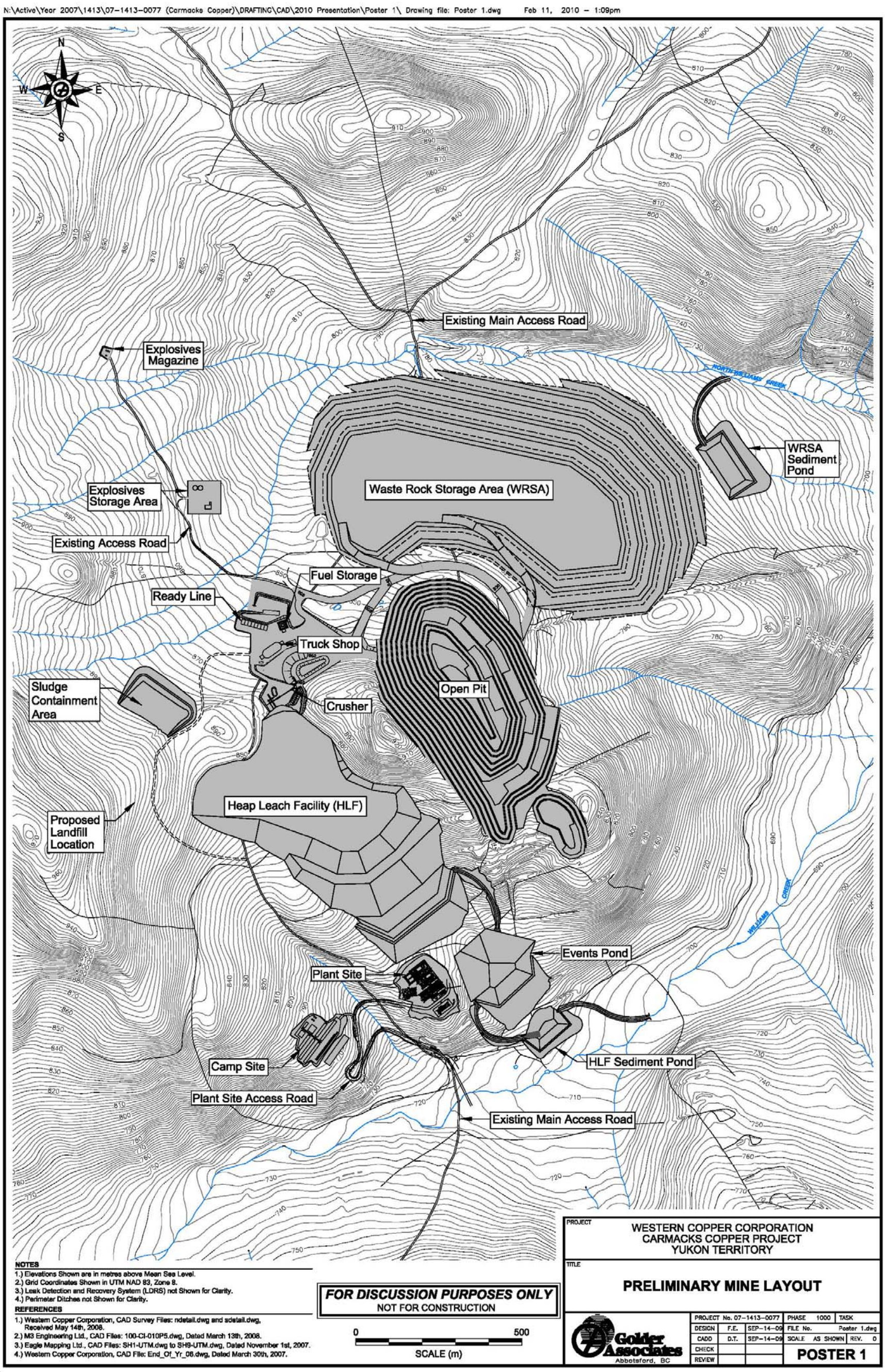


Figure 1. Preliminary Mine Layout

2.0 SITE ACTIVITIES

2.1 EXPLORATION

Exploration activities occurred during September 2011 to map and sample surficial geological material along the proposed mine access road. Test pitting and soil sampling were conducted on the southern portion of the property in the future mine access road area. Documentation of this program for assessment purposes on the Quartz Claims is provided in Appendix B.

2.2 CONSTRUCTION AND DEVELOPMENT

2.2.1 Overview of Activities by Quarter

No construction or development activities occurred on the property in 2011.

2.2.2 As built drawings

No “as-built” drawings were produced in 2011.

2.3 MINING ACTIVITIES

2.3.1 Overview of Activities by Quarter

No mining activities took place in 2011.

2.3.2 Production Schedule – Ore and Waste Removal

Not applicable for this reporting period; no mining activities took place in 2011.

2.3.3 Average Head Grades

Not applicable for this reporting period; no mining activities took place in 2011.

2.3.4 Open Pit Stability

Not applicable for this reporting period; no mining activities took place in 2011.

2.3.5 Heap Leach Cells – Status of Leaching (including layout drawing)

Not applicable for this reporting period; no mining activities took place in 2011.

2.3.6 Copper Production

Not applicable for this reporting period; no mining activities took place in 2011.

2.3.7 Spills

No spills occurred during the reporting period.

2.3.8 On-going Reclamation

Reclamation activities that occurred during the reporting period were conducted in response to the Annual Engineer's Inspection (see 2.3.9 below). One trip to the property was made on October 20 to clean up the site and remove 3 drums of fuel, as recommended by the Mines Inspector.

2.3.9 Actions Undertaken in Response to Annual Engineer's Inspection

The Annual Engineer's Inspection of the entire project area was undertaken on June 29, 2011 (see Appendix C) and recommended some minor maintenance activities. In response to these recommendations, follow up work was conducted on September 14 as summarized in the letter report provided in Appendix D. Maintenance work on the Williams Creek culvert, along the access road on the north side of North Williams Creek, and down slope of the project area within the floodplain of Williams Creek was conducted.

2.3.10 Access Road

The access road to the site has not been constructed; however, as mentioned in Section 2.1, exploration activities occurred during September 2011 to map and sample surficial geological material along the proposed mine access road.

2.4 RESOURCES AND RESERVES

The resource and reserve estimates for the property are shown in Tables 1 and 2 respectively. The reserves remain as stated in the "Feasibility Study" prepared in 2007, while the resource was updated later that year.

Table 1. Updated Resource Estimate for Zones 1, 4 and 7 at 0.25% Total Copper Cut-Off

	Category	Tonnes	Copper (%)		Gold	Silver	Contained Metal		
		(000)'s	Total	Oxide	(g/t)	(g/t)	Copper (000)'s lb.	Gold oz.	Silver oz.
Oxide	Measured	4,031	1.10	0.90	0.59	5.7	98,130	76,000	734,000
	Indicated	7,949	1.04	0.84	0.39	4.0	182,448	100,000	1,032,000
	M+I	**11,980**	**1.06**	**0.86**	**0.46**	**4.6**	**280,577**	**176,000**	**1,766,000**
	Inferred	90	0.73	0.53	0.13	1.8	1,452	370	5,000
Sulphide	Measured	695	0.80	0.02	0.26	2.5	12,192	6,000	57,000
	Indicated	3,645	0.74	0.03	0.20	2.3	59,195	24,000	269,000
	M+I	**4,340**	**0.75**	**0.03**	**0.21**	**2.3**	**71,387**	**30,000**	**326,000**
	Inferred	4,031	0.71	0.01	0.18	1.9	63,383	23,000	246,000

Table 2. Mineral Reserve

Reserve Class	Ore t (000)	Total Cu (%)	Cu Oxide (%)	Cu Non-Oxide (%)	Au (g/t)	Ag (g/t)
Proven Mineral Reserve	3,190	1.227	1.028	0.199	0.659	6.20
Probable Mineral Reserve						
Open Pit Ore	6,462	1.099	0.938	0.162	0.466	4.49
Estimated Dilution	960	0.065	0.043	0.021	0.018	0.20
Total Probable Mineral Reserve	7,422	0.965	0.822	0.144	0.408	3.93
Proven and Probable Mineral Reserve	10,611	1.044	0.884	0.160	0.483	4.62

2.5 CARE AND MAINTENANCE

No activities to report.

2.6 PROPOSED DEVELOPMENT AND PRODUCTION FOR UPCOMING YEAR

There are presently no development or production plans for the 2012 year.

3.0 MONITORING PROGRAMS AND STUDIES

The QML contains a number of requirements for studies and monitoring programs. The following sections outline work done with respect to these studies and programs. Copies of the actual reports relating to these are appended.

3.1 ON-GOING METALLURGICAL STUDIES

3.1.1 Field Tests

No metallurgical field tests were in progress as of 2011.

3.1.2 Laboratory Tests

Geochemical testing on spent ore from column tests was been carried out in 2008, with a second set beginning in 2010. Results of the static testing and 56 week humidity cell tests from the 2010 program, as well as a comparison with 2008 results, is provided in Appendix E.

3.2 HEAP LEACH PAD LINER PERFORMANCE MONITORING

No liner has been placed and no performance monitoring is in progress.

3.3 WATER QUALITY SURVEILLANCE PROGRAM

During 2011, five separate site visits were made for the purpose of collecting water quality data. These visits took place on the following dates:

May 19th

May 31st

June 28th – 29th

September 7th

October 27th – 28th

Table 3 lists the locations (shown on Figure 2) that have been established for the routine monitoring of surface water quality to date. Further locations will be added as the mine is brought into production.

Table 3. Water Quality Surveillance Program Site Descriptions and Locations

Station	Description	Northing	Easting
W2	Williams Creek Upstream of North Williams Creek Confluence	6914145	413499
W3	Lower North Williams Creek Upstream of Confluence with Williams Creek	6914379	413640
W4	Williams Creek Downstream of Confluence with North Williams Creek	6914653	413888
W5	South East Tributary to Williams Creek	6912947	412978
W6	Williams Creek Downstream of South East Tributary	6913373	413042
W7	Upper North Williams Creek Tributary Upstream of Road Crossing	6914810	411778
W9	Williams Creek Upstream of Access Road Crossing	6912511	411907
W10	Williams Creek Upstream of Yukon River	6919033	416606
W11	Nancy Lee Creek (Tributary of Williams Creek)	6918096	415803
W12	Williams Creek Downstream of Confluence with Nancy Lee Creek	6918000	416102
W13	Williams Creek Upstream of Confluence with Nancy Lee Creek	6917984	415912
Y1*	Yukon River Upstream of Williams Creek	6918974	416752
Y2*	Yukon River Downstream of Williams Creek	6919308	416249

Notes: Coordinates are UTM Zone 8 NAD83

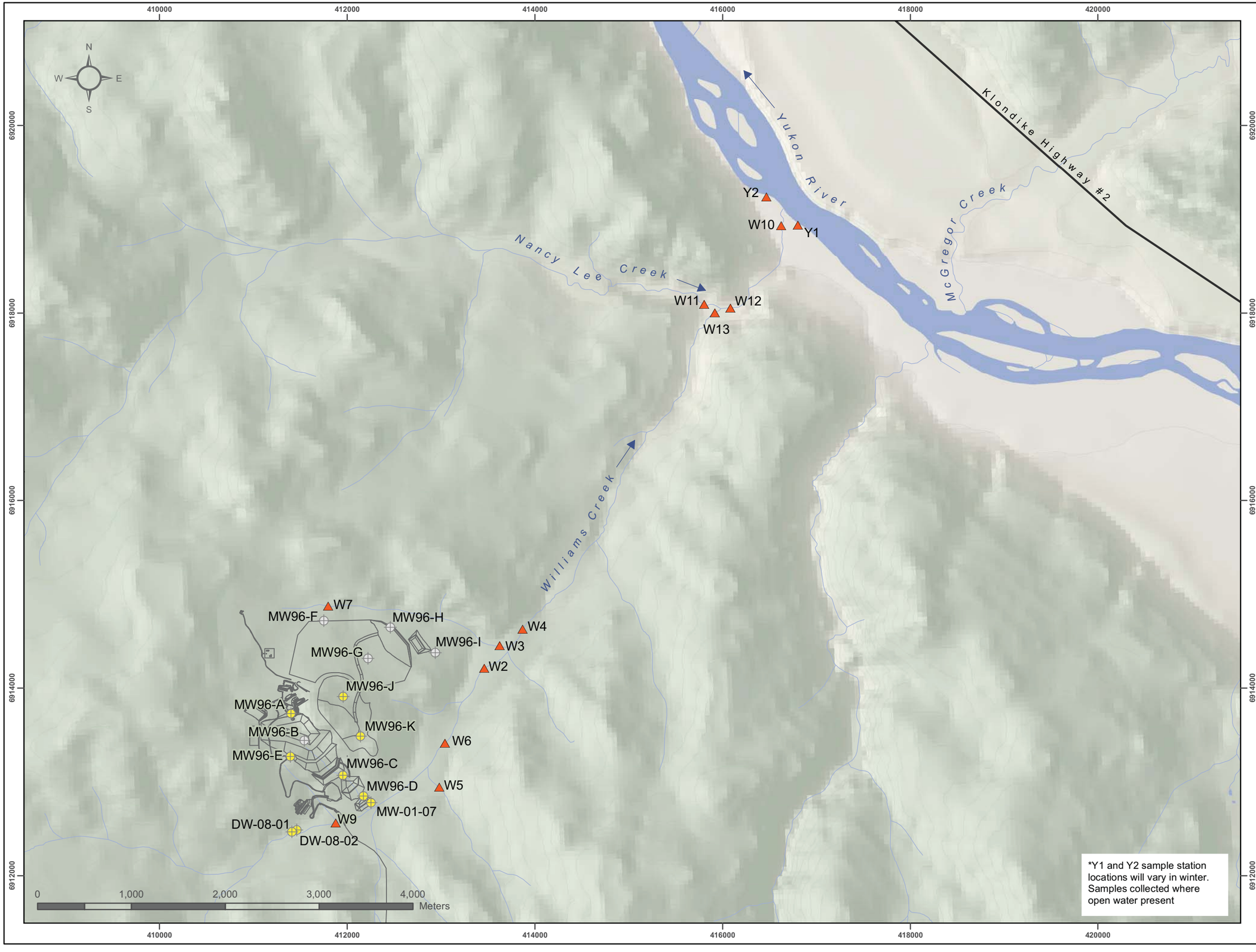
* Yukon River open water sample station locations; during freezing conditions samples collected where open water available.

Surface water quality monitoring activities took place at ten of the thirteen stations in 2010 and 2011. Williams Creek near the confluence with Yukon River (station W10) as well as the Yukon River (stations Y1 and Y2) were not monitored in 2010 and 2011 due to the logistical challenge of accessing these locations. The data collected from these locations in previous years is considered relatively robust at this stage prior to project advancement.

CARMACKS COPPER PROJECT

2010 ANNUAL QUARTZ MINING LICENSE REPORT

Western Copper Corporation



- Actively Monitored Well
- Inactive Well
- Surface Water Monitoring Station
- Contour
- Watercourse
- Waterbody

Datum: NAD83; Projection: UTM Zone 8N

National Topographic Data Base (NTDB) compiled by Natural Resources Canada at a scale of 1:50,000. Cadastral data compiled by Natural Resources Canada. Reproduced under license from © Her Majesty the Queen in Right of Canada, Department of Natural Resources Canada. All rights reserved.

FIGURE 2

MONITORING STATION LOCATIONS

ACCESS CONSULTING GROUP

Drawn by MD | MARCH 2010 | Checked by NS

D:\Project\AllProjects\WCH-01\gis\mxd\Envtl_Mon_Update_Rep\2009_RevSFig_X_Monitoring_Stns.mxd

3.3.1 Surface Water Quality

Surface water quality sampling was conducted during the four site visits in 2011 between May 31 and October 28. The reader is directed to the 2011 Environmental Monitoring Update and Data Summary, included in Appendix F for details of the 2011 sampling and results.

3.3.2 Groundwater Quality

Groundwater monitoring was also conducted where possible during the June and September water quality sampling events. The reader is referred to Appendix F for details of the sampling and results.

3.4 HYDROGEOLOGY STUDIES

No hydrogeologic studies were required or conducted in 2011.

3.5 WATER TREATMENT AND MANAGEMENT

No water treatment studies or water management studies were required or conducted in 2011.

3.6 CLIMATE DATA AND SNOW SURVEY MONITORING PROGRAM

Climate data from the meteorological station maintained by Yukon Government close to the property was logged throughout 2011. Copper North Mining Corp. plans to install and maintain its own climate monitoring station at the site once a decision to proceed with construction has been made. Refer to Appendix F for a summary of 2011 temperature and precipitation data as well as the latest Yukon Government survey results at the Williams Creek snow course station.

3.7 GEOCHEMICAL STUDIES AND ACID-BASE ACCOUNTING

As mentioned in Section 3.1.2, geochemical testing on spent ore from column tests was been carried out in 2008, with a second set beginning in 2010. Results of the static testing and 56 week humidity cell tests from the 2010 program, as well as a comparison with 2008 results, is provided in Appendix E.

3.8 PHYSICAL MONITORING PROGRAM

Physical monitoring of structures and facilities in 2011 is limited to the Annual Engineer's Inspection (Appendix C).

3.9 ENGINEER'S ANNUAL PHYSICAL INSPECTION REPORTS

As required under Section 16.1 and 16.2 of the QML, Copper North Mining Corp. engaged Golder Associates Ltd. to perform an Annual Physical Inspection of the site. The inspection was carried out on June 29, 2011 (QML Section 16.1). The complete report is contained in Appendix C. This report has been previously submitted to Government of Yukon, Department of Energy, Mines and Resources, Mineral Resources Branch.

Since no development has taken place on site, the report focused on existing site conditions and limited infrastructure. Areas observed on June 29 to require maintenance were adequately repaired on September 14.

3.10 RECLAMATION AND REVEGETATION STUDIES

In 2007 a test patch of seeding was completed on an approximately 500 m x 12 m area adjacent (west) of the access road south of the Williams Creek crossing and the helicopter pad area. The seeding and resulting vegetation was intended to help stabilize sediments in this area and has been observed in the past four years to be performing well.

3.11 SUBMISSION AND APPROVAL OF PLANS

No plans were submitted during 2011.

4.0 OUTSTANDING FINANCIAL LIABILITY

4.1 HEAP LEACH

At present there has been no update to the assessment of the liability associated with the Heap Leach Facility which was presented in the May 2009 revision of the Preliminary Detailed Closure and Reclamation Plan.

4.2 WASTE ROCK STORAGE

There has also been no update to the assessment of the liability associated with the Waste Rock Storage Facility which was presented in the May 2009 revision of the Preliminary Detailed Closure and Reclamation Plan.

4.3 OVERALL LIABILITY

The estimated maximum overall liability associated with the development and operation of the mine remains as set out in the May 2009 revision of the Preliminary Detailed Closure and Reclamation Plan

Facility or Area Description	**Cost**	
OPEN PIT	$	23,000
HEAP LEACH FACILITY	$	17,295,000
HLF EVENTS AND SEDIMENT PONDS	$	296,000
WASTE ROCK STORAGE AREA	$	740,000
PLANT AND ANCILLARY FACILITIES	$	467,000
CAMP	$	103,000
TRUCK SHOP SERVICE COMPLEX	$	70,000
MISCELLANEOUS FACILITIES	$	95,000
ACCESS AND HAUL ROADS	$	248,000
SITE MANAGEMENT	$	1,103,000
TOTAL	$	20,440,000

An additional $2.675 million is estimated to cover costs associated with rinsing and neutralization of the heap leach facility, should it extend to a 9 year period as opposed to the initially estimated 4.5 year period.

To date security in the amount of $80,300 has been posted with Yukon Government. This represents the liability incurred to date due to exploration activities on the site.

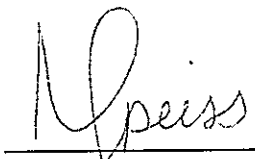
4.4 ENGINEERING CONTINGENCIES

In accordance with Section 11.0 of the QML, Copper North Mining Corp. prepared a Contingency Plan on the basis of a workshop held in October 2009. The plan was submitted to the Chief of Mining Land Use in January 2010. The main purpose of the Contingency Plan was to identify possible alternative approaches to decommissioning the Heap Leach Facility, however, other facilities were also examined. The plan identified a number of possible failure modes and contingency measures for each of the facilities and recommended further work that should be undertaken. The report was issued in draft format pending comments from government. No comment from government has been received to date. No further work has been undertaken at this time to develop any of the contingency plans identified.

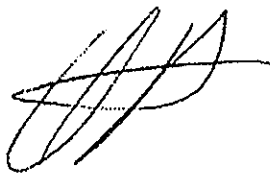
5.0 CLOSING STATEMENTS

Access Consulting Group of Whitehorse, Yukon, has prepared this Annual Report on behalf of Copper North Mining Corp. We trust this document fulfills your present requirements. If you have any questions or require further details, please contact the undersigned.

ACCESS CONSULTING GROUP

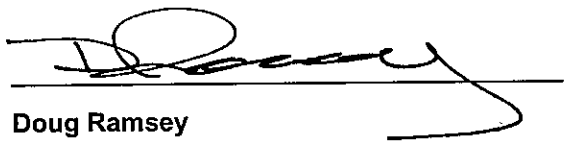


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2011 ANNUAL QUARTZ MINING LICENCE REPORT

SUBMITTED TO YUKON GOVERNMENT, ENERGY MINES AND RESOURCES

YUKON QUARTZ MINING LICENCE QML-0007

Appendix A

2011 Qualifying Report for the Carmacks Copper Deposit

2011 Qualifying Report
for the
Carmacks Copper Deposit,
Yukon Territory

NTS Map Sheet: 115I07
Latitude 62°21'N
Longitude 136°41'W

Report By:

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Report For:

Western Copper Corporation,
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Copper North Mining Corporation and
Carmacks Mining Corporation
2050 – 1111 West Georgia Street
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August 15, 2011

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1.0 SUMMARY

The Carmacks Copper Project is located approximately 192 km north of Whitehorse, Yukon, within the Whitehorse Mining District. The property consists of 338 full-size and fractional claims in two claim blocks; one containing 312 claims, the other 26 claims. All of the claims owned 100% by Carmacks Copper Corp., a 100% owned subsidiary of Western Copper Corporation (**Western Copper**).

Western Copper (formerly Western Copper Holdings Ltd.) and Thermal Exploration Company (**Thermal Exploration**) acquired the Carmacks Copper Project in 1989 from Archer, Cathro & Associates subject to a 3% Net Smelter Royalty. Western Copper completed a feasibility study on the property which was subsequently updated in 1995. Following this, Western Copper made the decision to proceed with project development and filed for environmental review together with Quartz Mining and Water Licence Applications. In 1998, after completing some early construction work on the project Western Copper suspended work on the project indefinitely due to low copper prices. In 2006, an exploration program was re-initiated on the Carmacks property consisting of diamond drilling, Rapid Air Blast (**RAB**) drilling and environmental baseline studies. Western Copper continued exploration drilling, environmental studies and geotechnical investigations, seasonally, in 2007 and 2008.

To date, a total of 241 diamond drill holes, 11 reverse circulation holes, and 61 Rapid Air Blast holes, amounting to approximately 38,207 m of drilling, were drilled in the exploration of the property. In addition, 71 geotechnical holes have been drilled for an additional 3,847 m.

The Carmacks 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 Triassic hornblende-biotite granodiorite of the Granite Mountain Batholith. The deposit constitutes the No. 1, No. 4 and No. 7 Zones, which are 3 of 14 defined zones containing copper mineralization known on or in the immediate vicinity of the property. 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 250m 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.

An updated mineral resource was estimated by Wardrop Engineering Inc, in 2007. The updated resource estimate was calculated using historical and recent drill hole data. The oxide sulphide boundary was modelled using a minimum 20% ratio of oxide copper to total copper. Wardrop used a specific gravity of 2.65 for the granodiorite, 2.68 for the oxide mineralization and 2.71 for the sulphide mineralization. Based on these parameters, Wardrop estimated that the Carmacks deposit contained approximately 10 million tonnes of oxide resources in the combined measured plus indicated categories, grading 0.96% Cu Oxide (**CuX**), 0.17% Cu Sulphide (**CuS**), 0.519 g/t Au, and 4.938 g/t Ag. The deposit contains an additional 5.7 million tonnes of sulphide resources in the combined measured plus indicated categories grading 0.05% CuX, 0.71% CuS, 0.227 g/t Au, and 2.244 g/t Ag. The mineral resources were reported at 0.25% total copper cut-off grade.

In June 2005, the project entered the environmental assessment process and in April 15, 2009 the company received a Quartz Mining Licence. The company then entered the Yukon Water Board review for a water licence. On May 10, 2010 the company was notified that the Water Licence had been denied with a list of deficiencies in the application and design process.

Western Copper is currently studying alternative heap leach designs to resolve the Water Licence issues, including modifications to the plant design. These may require additional geotechnical investigations to support the design change and an updated capital cost estimate for the project.

Exploration potential for oxide copper and sulphide resources exists in the Zone 2, 12, 13 and 14 areas. Additional drilling will be required to define these zones. Drilling to test and define these areas is estimated at 5,000 to 15,000 m.

The Phase 1 budget estimate for the alternative heap leach and plant design, geotechnical investigation and updating the feasibility study is $600,000. The Phase 2 budget estimate for evaluation of the under-explored zones is $1,000,000 to $3,000,000. Although the project does not require the additional zones to be economic.

2.0 INTRODUCTION AND TERMS OF REFERENCE

The Carmacks Copper Project site is located 192 kilometres (**km**) north of Whitehorse in the Whitehorse Mining Division of the Yukon Territory, Canada. The property consists of 312 full-size and fractional claims in one contiguous blocks, and 26 claims in a second contiguous block.

In 1995, Western Copper commissioned Kilborn Engineering Pacific Ltd. (**Kilborn**) to prepare a Feasibility Study for the Carmacks Copper Project. This study contains resource estimates that were prepared prior to the inception of National Instrument 43-101 (**NI 43-101**) but were verified by the authors in a 2007 Aurora Geosciences Ltd. (**Aurora**) report. Aurora concluded that the resource estimates presented in the Kilborn feasibility study were a reasonable interpretation of the quantity and grade of the mineralization present, and satisfied the definitions and standards of the Canadian Institute of Mining, Metallurgy, and Petroleum (**CIM**) and as defined under NI 43-101. In the spring of 2007, Wardrop re-estimated the mineral resources for the Carmacks deposit using a 3 dimensional block modelling approach based on both historical (pre-2006) and the new data from the 2006 campaign.

The 2007 drilling program commenced in March and was completed in mid-October. The program further defined zones of mineralization at the No. 1, No. 7 and No. 7A zones, delineated mineralization at the No. 4, No. 12 and No. 13 zones and identified copper-molybdenum mineralization in a new zone, the No. 14 Zone.

This report relies on information and data gathered from existing reports prepared by other consultants and companies. This report relies primarily on information contained in the Basic Engineering Report, dated December 1997 and the Feasibility Study, dated September 1995, both of which were prepared by Kilborn. This report also relies on a technical report prepared by Aurora dated January 8, 2007 and the Resource Estimate Report prepared by Wardrop and Casselman.

In this report, all dollar amounts are expressed in Canadian currency unless otherwise specified.

2.1 TERMS of REFERENCE

Western Copper has commissioned Casselman Geological Services Ltd. to prepare an updated report utilizing the 43-101 standard in preparation for a spin-out of assets from Western Copper to a newly formed company. This report relies on the resource estimate of 2007 and is based on historical data plus data from the 2006, 2007 and 2008 exploration campaigns. The mineral resource estimate of 2007 was in conformance with the CIM Mineral Resource and Mineral Reserve definitions referred to in NI 43-101 Standards and Disclosure for Mineral Projects.

Scott Casselman, B.Sc., P.Geo., of Casselman Geological Services Ltd is the qualified person as defined by National Instrument 43-101, responsible for the preparation of this report. Mr. Casselman is independent of Western Copper and managed the exploration programs on site in 2006, 2007 and 2008.

3.0 RELIANCE ON OTHER EXPERTS

The author relied on data provide by Western Copper for drill data, assay data, resource estimation, surveying, claim status, environmental studies and permitting.

4.0 PROPERY DESCRIPTION AND LOCATION

The Carmacks Copper Project is located in the Dawson Range Mountains on NTS map sheet 115I07 at latitude 62°21'N and longitude 136°41'W, some 192 km north of Whitehorse, Yukon as shown on Figure 4.1. 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.

The Carmacks Copper Project site, located in the Whitehorse Mining District, consists of 318 quartz claims and quartz claim fractions and 20 quartz leases and quartz lease fractions as shown on Figure 4.2. The term ‘quartz’ for a claim in the Yukon is the nomenclature used to distinguish between a bedrock or lode mineral rights and ‘placer’ claims for placer mineral rights. The registered owner of the claims is Carmacks Copper Corp., a 100% owned subsidiary of Western Copper Corporation.

A list of claims derived from the Yukon Mining Records and Claim Status Report, which describe the status of the Mineral claims and leases comprising the Carmacks property is presented in 4.1 and 4.2, respectively.



WESTERN COPPER CORPORATION

CARMACKS COPPER PROPERTY

Figure 4.1 Property Location Map

Table 4.1 Quartz Claim Status

Claim Name and No.	Grant No.	Expiry Date	Registered Owner	% Owned
BOY 20	Y51118	2012/03/09	Carmacks Copper Corp.	100
BOY 51, 53	Y51149, Y51151	2012/03/09	Carmacks Copper Corp.	100
BOY 52, 54	Y51150, Y51152	2014/03/09	Carmacks Copper Corp.	100
BOY 83	Y51181	2012/03/09	Carmacks Copper Corp.	100
DUN 1	Y59382	2012/03/09	Carmacks Copper Corp.	100
TT 1	YB97068	2015/03/09	Carmacks Copper Corp.	100
TT 2	YB97251	2012/03/09	Carmacks Copper Corp.	100
VW 11, 13	YB96620, YB 96622	2012/03/09	Carmacks Copper Corp.	100
VW 17-18	YB96626-YB6627	2015/03/09	Carmacks Copper Corp.	100
VW 19–21	YB96628–YB96630	2016/03/09	Carmacks Copper Corp.	100
VW 23, 25	YB96632, YB6634	2015/03/09	Carmacks Copper Corp.	100
VW 27–38	YB96636-YB96647	2015/03/09	Carmacks Copper Corp.	100
VW 40-50	YB96986-YB96996	2012/03/09	Carmacks Copper Corp.	100
VW 60-61	YB96997-YB96998	2015/03/09	Carmacks Copper Corp.	100
REM 1-14	YC39221-YC39234	2012/04/11	Carmacks Copper Corp.	100
REM 19-30	YC39239-YC39250	2012/04/11	Carmacks Copper Corp.	100
REM 32-35	YC39251-YC39254	2012/04/11	Carmacks Copper Corp.	100
W 1-6	YB26708-YB26713	2012/03/09	Carmacks Copper Corp.	100
W 7-10	YB26714-YB26717	2015/03/09	Carmacks Copper Corp.	100
W 11,13,15,17,19,20	YB26718,20,22,24,26,27	2012/03/09	Carmacks Copper Corp.	100
W 12,14,16,18	YB26719,21,23,25	2013/03/09	Carmacks Copper Corp.	100
W 21-24	YB26728-YB26731	2015/03/09	Carmacks Copper Corp.	100
W 25-28	YB26732-YB26735	2013/03/09	Carmacks Copper Corp.	100
W 29-30	YB26736-YB26737	2012/03/09	Carmacks Copper Corp.	100
W 31-37	YB26738-YB26744	2015/03/09	Carmacks Copper Corp.	100
W 41-43	YB26748-YB26750	2013/03/09	Carmacks Copper Corp.	100
W 50-53	YB36249-YB36252	2015/03/09	Carmacks Copper Corp.	100
W 55 ,W 57	YB36254, YB36256	2013/03/09	Carmacks Copper Corp.	100
W 91-93	YB36929-YB36931	2015/03/09	Carmacks Copper Corp.	100
W 95	YB36933	2015/03/09	Carmacks Copper Corp.	100
X 3-4	YB36898, 899	2015/03/09	Carmacks Copper Corp.	100
X 5-7	YB36962-YB36964	2015/03/09	Carmacks Copper Corp.	100
WC 5-18	YB36693-YB36706	2012/03/09	Carmacks Copper Corp.	100
WC 23-36	YB36711-YB36724	2012/03/09	Carmacks Copper Corp.	100
WC 40-54	YB36728-YB36742	2012/03/09	Carmacks Copper Corp.	100
WC 57-72	YB36745-YB36760	2012/03/09	Carmacks Copper Corp.	100
WAR 22	Y59373	2016/03/09	Carmacks Copper Corp.	100
WAR 23-29	YB36240-YB36246	2015/03/09	Carmacks Copper Corp.	100
WAR 30-31	YB36247-YB36248	2012/03/09	Carmacks Copper Corp.	100
WAR 32-37	YB36446-YB36451	2015/03/09	Carmacks Copper Corp.	100
WAR 38-44	YB36765-YB36771	2015/03/09	Carmacks Copper Corp.	100
WAR 45	YB36772	2012/03/09	Carmacks Copper Corp.	100
WAR 46	YB36773	2015/03/09	Carmacks Copper Corp.	100
WAR 47-50	YB36774-YB36777	2012/03/09	Carmacks Copper Corp.	100
WCC 1-40	YC60381-YC60420	2013/05/10	Carmacks Copper Corp.	100
GAP 1-5	YC65320-YC65324	2013/03/09	Carmacks Copper Corp.	100
HIP 1-12	YC65554-YC65565	2013/03/09	Carmacks Copper Corp.	100
HIP 13, 14	YC65566-YC65567	2012/14/08	Carmacks Copper Corp.	100
HIP 15-27	YC65568-YC65580	2013/03/09	Carmacks Copper Corp.	100
JIM 1-30	YC66844-YC66873	2012/03/09	Carmacks Copper Corp.	100

Table 4.2 Quartz Lease Status

Claim Name and No.	Grant No.	Expiry Date	Registered Owner	% Owned	Quartz Lease
AC # 2-3	Y91722–Y91723	2019/10/28	Carmacks Copper Corp.	100	OW00070-71
BOY 22	Y51120	2019/10/28	Carmacks Copper Corp.	100	OW00061
BOY 24	Y51122	2019/10/28	Carmacks Copper Corp.	100	OW00062
BOY 55–58	Y51153–Y51156	2019/10/28	Carmacks Copper Corp.	100	OW00063-66
BOY 85	Y51183	2019/10/28	Carmacks Copper Corp.	100	OW00067
DUN 2-3	Y59383–Y59384	2019/10/28	Carmacks Copper Corp.	100	OW00068-69
W 38-40	YB26745-YB26747	2019/10/28	Carmacks Copper Corp.	100	OW00072-74
W 44-48	YB26751-YB26755	2019/10/28	Carmacks Copper Corp.	100	OW00075-79
W 49	YB26756	2025/03/09	Carmacks Copper Corp.	100	OW00080

The above claim information has been verified with the Yukon Mining Recorder NMRS database, current to August 12, 2011, which states that the expiry dates listed are valid and that all the claims and leases are in good standing (www.yukonminingrecorder.ca). Archer, Cathro & Associates (1981) Limited, at the election of Western Copper retain a 3.0% NSR royalty to a maximum of $2.5 million. Annual advanced royalty payments of $100,000 are made whenever the average annual price of copper is above US$1.10/pound. To date $300,000 of the royalty has been paid.

In the Yukon, claims are good 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 per claim per year of work recorded is also applicable to record the assessment work. 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.

The property lies near 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.

In 1972, the leases on the property were surveyed as per one of the requirements of obtaining a lease. 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. The far northern claims (WC claims), southern claims (VW, TT and X) and eastern claims (REM) have not been legally surveyed.

Figure 4.2 shows the location of the claims and leases.

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

- all areas disturbed must be left in a condition conducive to successful regeneration by native plant species.
- all areas disturbed must be re-sloped, contoured, or otherwise stabilized to prevent long-term soil erosion.
- structures must be removed and the site restored to a level of utility comparable to the previous level of utility.

Western Copper has had environmental baseline monitoring programs in place since 1992. An Initial Environmental Evaluation (**IEE**) was compiled in 1994 and since then four addendums to the IEE have been filed by Hallam, Knight Piésold Ltd. (**HKP**), Knight Piésold Ltd. and Kilborn to detail additional information on the environmental parameters of the project. Programs have consisted of, but are not limited to, the following:

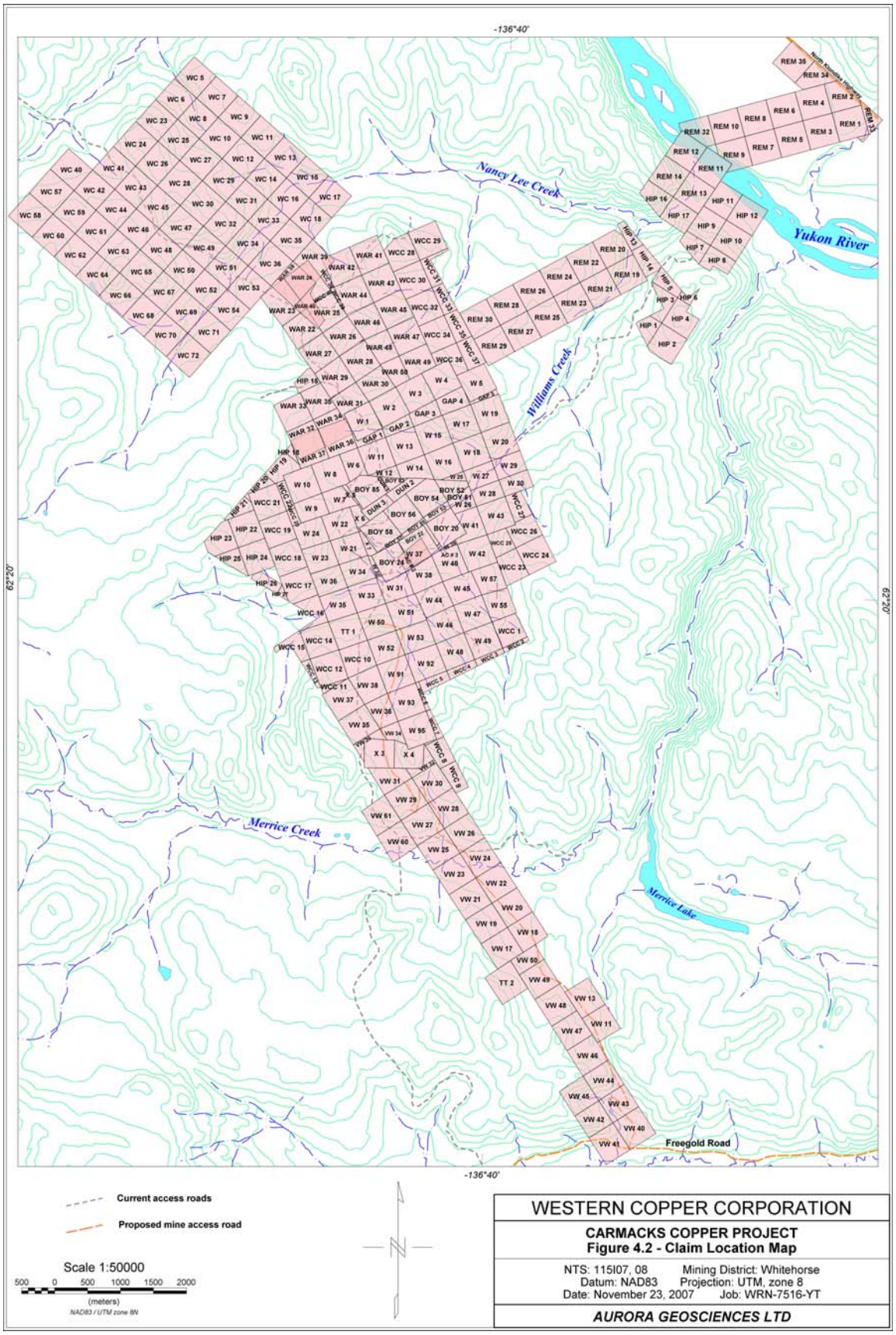
- overburden and waste characterization studies.
- in 23 waste rock samples, only one was over detection limit in elemental sulphur, overall Neutralization potential was NP/AP of 90:1, recent work using rainwater through waste sections of core, all three samples returned net NP greater than five and, although metals were present in the rock, the leachate metal content was low to very low for all metals.
- leach pad neutralization, initial difficulty in increasing pH above 4 has been rectified by using sodium carbonate instead of lime for treatment of the leached material.

In June 2005, Western Silver filed a Project Description with the Yukon Territorial Government (**YTG**) to initiate an environmental assessment under the Yukon Environmental Assessment (**YEA**) process and at the same time made application for both a Water License and a Quartz Mining License. Federal legislation passed in November of 2005 required Western Copper to enter into a second, parallel, assessment process pursuant to the Yukon Environmental and Socioeconomic Assessment Act (**YESAA**). A Project Proposal prepared in accordance with YESAA requirements was formally submitted in February 2006 and deemed adequate for public review in February 2007. On April 15, 2009 the company received a Quartz Mining Licence.

Prior to conducting the review of the Water Licence application, the Yukon Water Board required the recommendations of the Quartz Mining Licence. This review commenced shortly after April 15, 2009. On May 10, 2010 the company was notified that the Water Licence had been denied. The Water Board indicated a

number of deficiencies and concerns in the application, which the company is currently working to address.

Western Copper will need to place environmental reclamation bonding to cover future and current potential environmental liabilities that at this time would consist of re-contouring and re-vegetating the camp areas and any old exploration trenches that remain intact.


-136°40'
62°20'
Nancy Lee Creek
Yukon River
Williams Creek
Merrice Creek
Merrice Lake
Freegold Road
Current access roads
Proposed mine access road
Scale 1:50000
500 0 500 1000 1500 2000
(meters)
NAD83 / UTM zone 8N
WESTERN COPPER CORPORATION
CARMACKS COPPER PROJECT
Figure 4.2 - Claim Location Map
NTS: 115I07, 08 Mining District: Whitehorse
Datum: NAD83 Projection: UTM, zone 8
Date: November 23, 2007 Job: WRN-7516-YT
AURORA GEOSCIENCES LTD

5.0 ACCESIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE and PHYSIOGRAPHY

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 a 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.

The climate in the Carmacks area is marked by warm summers and cold winters. Mean daily temperatures range from -30°C in January to 12°C in July. The location close to the Arctic Circle provides 22 hours of daylight in late June but 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 375 mm (water equivalent) with one third falling as snow. July is the wettest month. Mean annual evaporation is approximately 404 mm to yield a net loss of 29 mm. Maximum evaporation occurs in July. 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 successfully for many years in this climate.

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

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. 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 centimetres **(cm)** 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, 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 till and colluvium to 55 m deep has been measured in the centre of the valley. Northeast of the deposit, in the North Williams Creek valley, overburden has been measured up to 100 m deep. Permafrost is present at varying depths on most north facing slopes and at depth in other areas. Facilities should be located to avoid frost-susceptible, poorly drained soils.

The project is 220 km from Whitehorse, the capital of Yukon Territory. Whitehorse has a population of around 26,000, which is roughly 3/4 of the entire Yukon. Whitehorse is serviced by daily commercial flights from British Columbia and Alberta to the south and other northern communities and all-weather paved highways to the south and west to Alaska. Historically, mining has been the Yukon's most important industry. In the past, the Yukon & White Pass Route (**Y&WPR**) railroad provided rail service from Whitehorse to port at Skagway, Alaska. In fact, 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. Concentrate from the Minto Mine is currently trucked to Skagway, for shipping over-seas.

Carmacks has little infrastructure but may provide a location for camp or town site for the project. The nearest access point to the power grid is 9.5 km east-northeast of the deposit, to the power line along the Klondike Highway. Running a transmission line from here to the property would require the line to be strung across the Yukon River.

There are no permanent facilities 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. Although it is planned to use a local well water source, make up water could also be brought from the Yukon River, which runs year round.

6.0 History

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

In the late 1960's, exploration for porphyry copper deposits in the Dawson Range led to the discovery of the Casino porphyry copper deposit, 104 km to the northwest of the Carmacks Copper deposit. This discovery precipitated a staking rush that led to the staking of the Williams Creek property in 1970 by G. Wing and A. Arsenault of Whitehorse. The Dawson Range Joint Venture (Straus Exploration Inc., Great Plains Development of Canada Ltd., Trojan Consolidated Minerals Ltd., and Molybdenum Corporation of America) optioned the property and contracted Archer, Cathro and Associates to conduct reconnaissance prospecting and geochemical sampling. During this program, the present No. 1 and No. 2 Zones were discovered.

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

The property was purchased by Western Copper Holdings and Thermal Exploration Ltd in 1991. Later that year, they performed 3,464 m of diamond drilling in 36 holes and initiated a baseline environmental study. In 1992, they drilled 1,164 m in 11 diamond drill holes and 856 m in 11 reverse circulation holes. The companies also conducted additional metallurgical test work, baseline environmental testing, a biophysical assessment of the area and contracted Knight Piesold Ltd to conduct geotechnical studies on the deposit consisting of test pit excavation, overburden sampling, oriented diamond drill core logging and geologic mapping.

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

In 1995, Western Copper Holdings and Thermal Exploration Ltd. merged to become Western Copper Holdings Ltd. The company contracted Knight and Piesold Ltd. to initiate a preliminary mine design and also initiated clearing and

grubbing of a site access road and leach pad area. The company submitted a mine permit application later that year.

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

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

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

In early 2006, Glamis Gold Ltd. purchased Western Silver Corporation and spun off a separate firm named Western Copper Corporation (**WCC**). WCC retained the rights to the Carmacks Copper Project.

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

Table 6.1 Historical Tonnage & Grade Estimates of the Carmacks Copper Deposit

Year	Source	Tons	Cu ox. %	Cu %	Au oz/t	Comments
1991	MPH Consulting Ltd.(No. 1 Zone)	14,564,600	0.90	1.05	-	Conventional by section 76% proven, 13% probable
1991	MPH Consulting Ltd.(No. 1 Zone)	14,564,600	0.88	1.00	-	IDS block model 78% proven, 10% probable
1993	Western Copper Audited by Kilborn	12,984,240	0.911	1.195	0.016	Measured and indicated at cutoff of 0.8% total copper
1993	Western Copper Audited by Kilborn	15,867,140	0.829	1.096	0.014	Measured and indicated at cutoff of 0.5% total copper
1993	Western Copper Audited by Kilborn	19,062,390	0.725	0.972	0.013	Measured and indicated at cutoff of 0.01% total copper
1997	Western est. Audit by Kilborn/SNC	13,300,000	-	0.97	-	Cutoff grade 0.29%T Cu Mine use 4.6:1 strip ratio
2007	Wardrop (No. 1, 4 and 7 Zones)	10,000,000	0.96	1.13	0.017	Oxide Resource, Measured and indicated at cutoff of 0.25% total copper

With the exception of the 2007 resource estimate, the mineral resource calculations presented in Table 6.1 have not been classified in accordance with the CIM approved standards as required in NI 43-101. These estimates have been obtained from sources believed reliable and conform to disclosure standards in use at the time of their publication, but have not all been independently verified. The historical resource estimates are no longer relevant and should not be relied upon, as they are replaced by the 2007 estimate which is re-presented in Section 16.0 of this report.

7.0 GEOLOGICAL SETTING

The regional geology was described by Bostock in 1936 and more recently by Tempelman- Kluit in 1981 and 1985 and is shown in Figure 7.1. The Carmacks region lies within the Intermontane Belt, which in the Carmacks map-area is divisible into the Yukon Cataclastic Terrane, Yukon Crystalline Terrane and Whitehorse Trough.

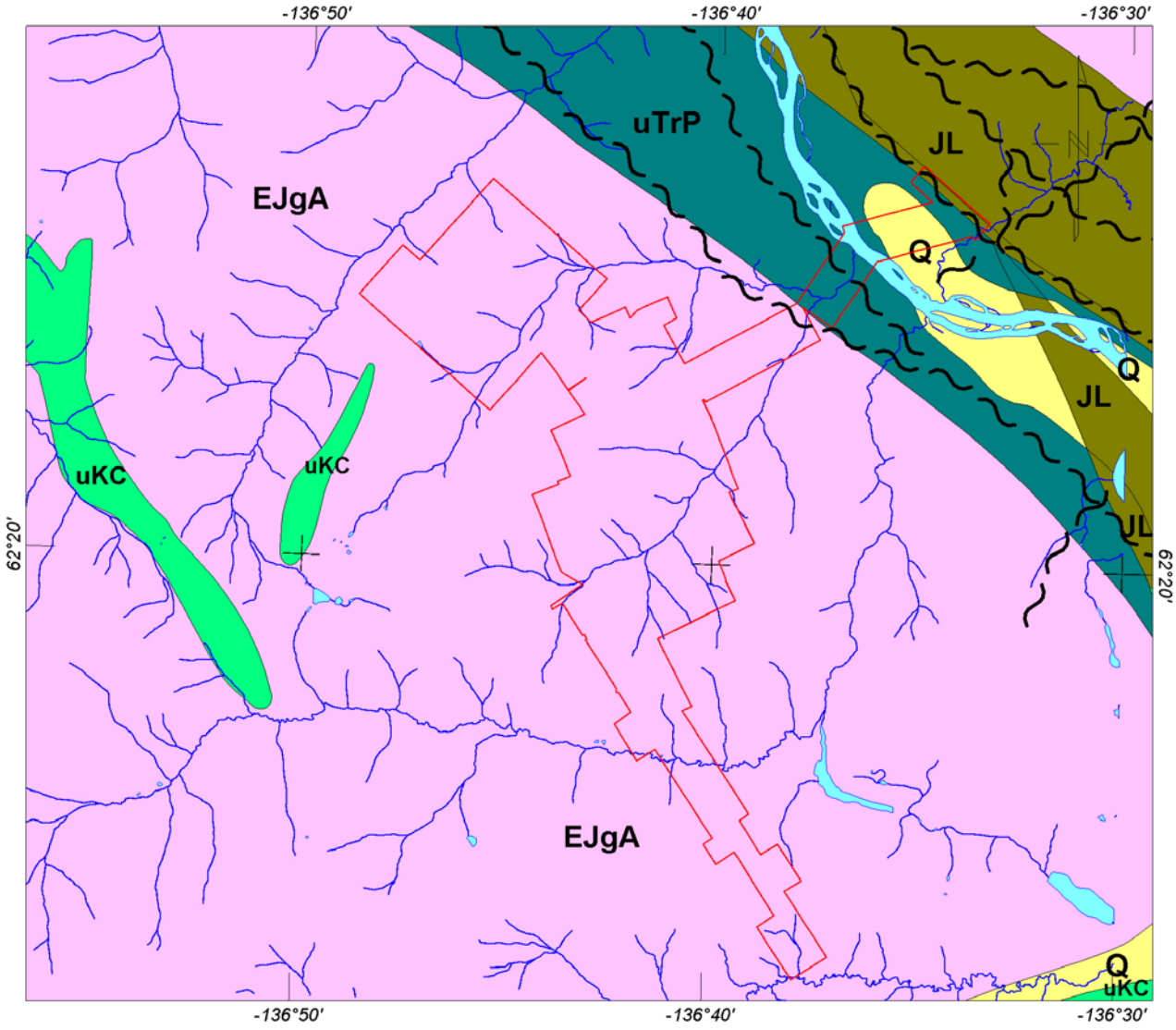
The Whitehorse Trough lies to the east of the Hoochekoo Fault, east of the Carmacks Copper Project. The Whitehorse Trough comprises Upper Triassic intermediate to basic volcanic capped by carbonate reefs (Povoas Formation) and Lower Jurassic greywacke, shale and conglomerate, derived from the underlying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

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

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

Mesozoic strata of the Whitehorse Trough are only exposed in fault contact with the Yukon Crystalline Terrane and Yukon 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.

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.



Q Quaternary - sand and gravel

uKC upper Cretaceous - Carmacks Group volcanics

JL Jurassic - Labege Group sediments

EJgA Early Jurassic - Aishihik Suite granodiorite

uTrP upper Triassic - Povoas Group volcanics and sediments

2500 0 2500
(meters)

WESTERN COPPER CORPORATION
CARMACKS COPPER PROPERTY
Figure 7.1 - REGIONAL GEOLOGY 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.

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:

- biotite-rich gneiss and quartzo-feldspathic gneiss
- 'siliceous ore'
- 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 shown in Figure 7.2 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 an 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 with some sedimentary re-working is considered the most likely.

Post mineralization aplite and pegmatites are common. They range in thickness from a few centimetres up to three metres. Quartz veins are uncommon and average two to five centimetres 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.

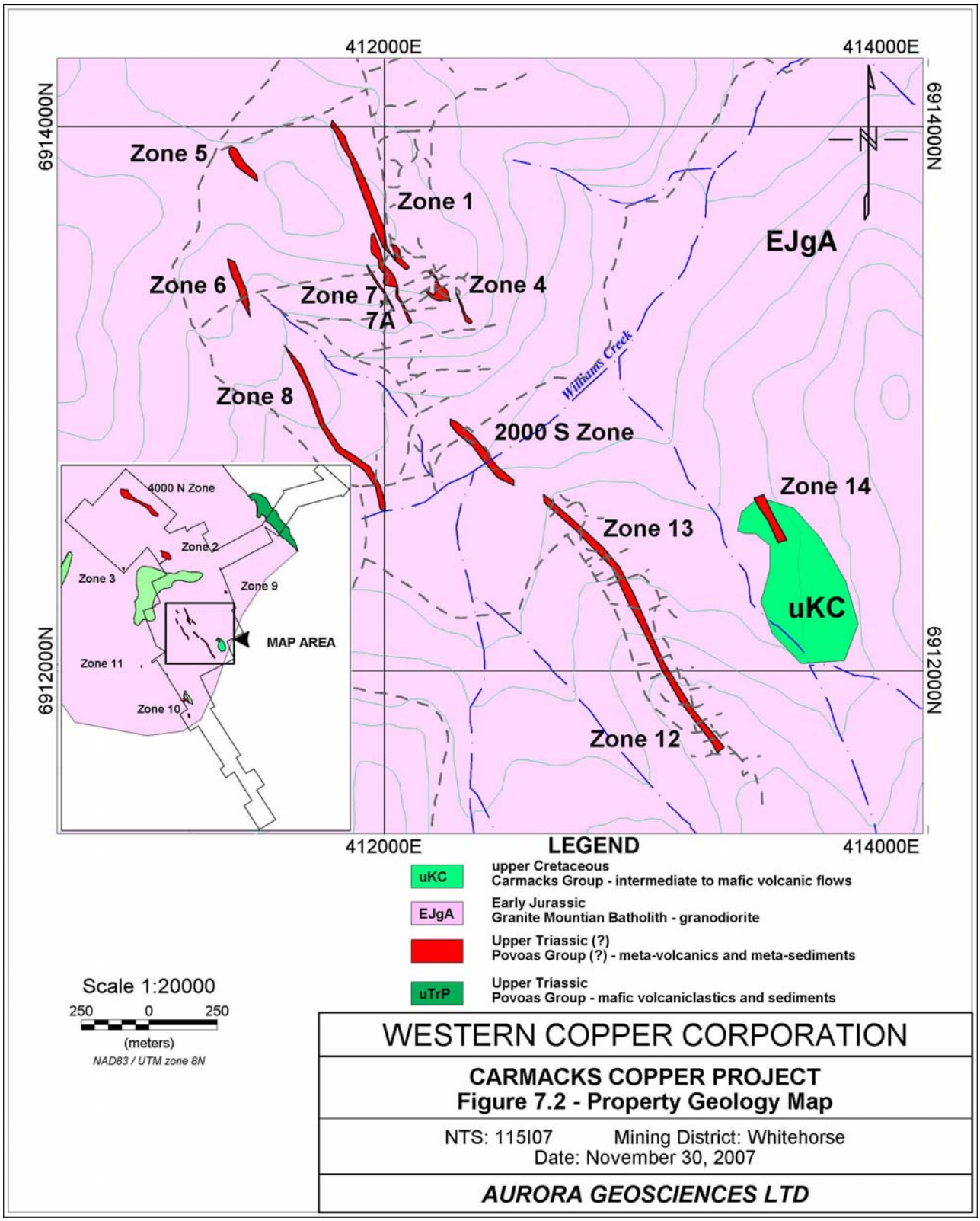
All of the historically estimated resources are contained in the No. 1 Zone which extends over a 700 m strike length and at least 450 m down dip. The deposit is open at depth and is oxidized to approximately 230 metres in depth. Copper-gold mineralization at Carmacks Copper is hosted by feldspathic-biotite-hornblende-quartz gneisses. These gneisses have been subdivided into nine categories based on coarseness and biotite-hornblende content. All of the gneisses are silica undersaturated and mafic rich.

The character of the No. 1 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 and appears to terminate against sub-parallel faults down dip. At the north end the deposit pinches out. At the south end the No. 1 deposit appears to splay into the No. 7, 7A zones and was pulled apart, or fault offset into No. 4 Zone. 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.

The No. 4 Zone varies from biotite-hornblende gneiss, as seen in the No. 1 zone, in the northern part to a more granular, meta-volcano-sedimentary host rock in the south. The copper content is lower in the No. 4 zone and there is a shallower depth of oxidation. The shape of the No. 4 zone is an irregular, wavy, shallowly dipping body, which forms a large “bowl” shape in the centre of the zone.

In the northern half of the No. 1 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 (**oz**) gold per ton (0.75 grams per tonne (**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.


412000E
414000E
6914000N
6912000N
Zone 5
Zone 1
Zone 6
Zone 7
7A
Zone 4
Zone 8
2000 S Zone
Zone 14
Zone 13
Zone 12
EJgA
uKC
Williams Creek
4000 N Zone
Zone 2
Zone 3
Zone 9
MAP AREA
Zone 11
Zone 10
LEGEND
uKC
upper Cretaceous
Carmacks Group - intermediate to mafic volcanic flows
EJgA
Early Jurassic
Granite Mountian Batholith - granodiorite
Upper Triassic (?)
Povoas Group (?) - meta-volcanics and meta-sediments
uTrP
Upper Triassic
Povoas Group - mafic volcaniclastics and sediments
Scale 1:20000
250 0 250
(meters)
NAD83 / UTM zone 8N
WESTERN COPPER CORPORATION
CARMACKS COPPER PROJECT
Figure 7.2 - Property Geology Map
NTS: 115I07 Mining District: Whitehorse
Date: November 30, 2007
AURORA GEOSCIENCES LTD

8.0 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 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 Sherwood 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.

9.0 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). Native copper occurs as dendritic secondary precipitates on fractures, occasionally up to 50 m from the deposits. 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 a 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 230 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, and 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 five 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.

10.0 EXPLORATION

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 has been trenched at 200-foot spacing and one or two trenches have been excavated on most of the other known anomalies (Figure).

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.

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

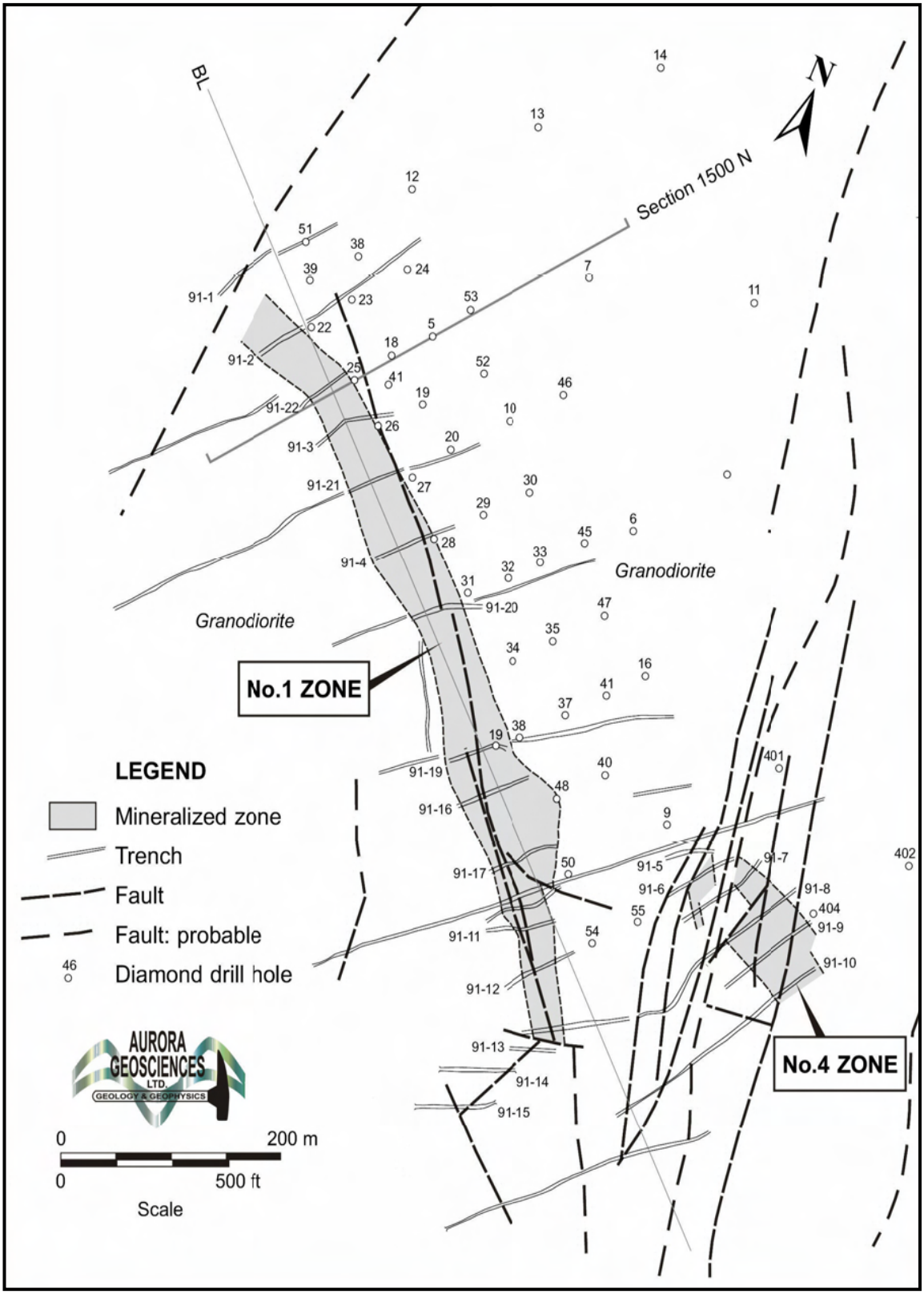
The Carmacks area in general was also covered by a regional fixed wing airborne geophysical survey conducted by Fugro Airborne Surveys for the Yukon government in 2001 (Shives et al, 2002).

In 2006, Western Copper conducted 7,100 m of diamond drilling in 34 holes, 1,200 m of rotary air blast drilling in 61 holes, and re-initiated environmental baseline studies. In 2007, the company 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 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.

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

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

Figure 10.1 General Plan Map



11.0 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. The drill holes prior to 2006 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 and 2007 programs 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 delineate the oxidation-reduction front at depth on the deposit; confirm historic drill results by twinning two of the previously drilled holes and explore along strike to search for lateral extensions of the No. 1 Zone, and to expand the knowledge of some of the other mineralized zones.

In addition, a RAB drilling program commenced in August 2006, which was designed to condemn areas of the property for future plant development.

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 2008 program was designed to complete the geotechnical studies initiated in 2007.

12.0 SAMPLING METHOD and APPROACH

Drill core in 1971 and 1990 was sampled in 10-foot (3.05 m) intervals. In 1991 and 1992, drill core was sampled by rock type for geological information but sampling was largely within 10-foot intervals to facilitate later statistical analysis of assay data.

Reverse circulation holes were sampled over five foot (1.52 m) intervals within the No. 1 Zone and at 10-foot intervals for 25 to 50 feet (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 5 or 10 foot (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 ALS Chemex Labs Ltd. (**Chemex**) in North Vancouver, B.C. for check assays. The first batch results from Chemex were 5.9% higher than the originals but the second batch returned values 5.7% lower on average. In the 1990s programs, trench and drilling samples were sent to Chemex for analysis. All samples were dried and crushed to better than 60% minus 10 mesh. An appropriate size split then underwent Chrome-steel ring pulverization until >90% was minus 150 mesh size.

Total copper was assayed by $HClO_4$ – HNO_3 digestion followed by Atomic Absorption Spectrometry (**AAS**) with a 0.01% detection limit. Non-sulphide copper was assayed by dilute H_2SO_4 digestion followed by AAS with a 0.01% detection limit. Gold was assayed by 1/2 assay ton fire assay followed by AAS with a 0.002 oz/ton (0.0686 g/t) detection limit and an upper limit of 20 oz/ton (685.71 g/t). Silver was assayed by aqua regia digestion followed by AAS with a 0.01 oz/ton (0.34 g/t) detection limit and an upper limit of 20 oz/ton (685.71 g/t).

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

Samples were processed by crushing to >70% <2 mm and pulverizing a 250-g split to >85% -75 mm according to Chemex's Prep 31 procedure. The samples were then analyzed for 27 elements by "Near Total" digestion and Inductively Couple Plasma Emission Spectroscopy (**ICP-ES**) by Chemex's ME-ICP61 or ME-ICP61a procedures.

As well, each sample was analyzed for:

- gold by fire assay and AAS on a 30-g sample by procedure Au-AA23
- total copper content by four-acid (HF-NNO_3-$HClO_4$-HCl) digestion and Atomic Absorption according to procedure Cu-AA62
- non-sulphide copper by sulphuric acid leach and AAS according to procedure Cu-AA05.

Duplicate samples were collected regularly, nominally every 20th sample, and were given unique sample numbers. For the first portion of the program, the duplicates were sent along with the original samples to Chemex for processing and were processed as described above. For the latter portion of the program the duplicates were sent to Acme Analytical Laboratories (**ACME**) in Vancouver for analysis. The samples sent to Acme were processed by crushing to >70% <-10 mesh and pulverizing a 250 g split to >95% -150 mesh according to the Acme R1 50 procedure. The samples were then analyzed for 43 elements by "Four Acid" digestion and Inductively Couple Plasma Mass Spectroscopy (**ICP-MS**) by Acme's 1T-MS procedure. As well, all samples were analyzed for gold by fire assay and Inductively Coupled Plasma Emission Spectroscopy (**ICP-ES**) on a 30 gm sample by procedure 3B ICP-ES. Total copper content was determined by four-acid (HF-NNO_3-$HClO_4$-HCl) digestion and ICP-ES according to procedure 7TD and, for non-sulphide copper, by sulphuric acid leach and AAS according to procedure 8.

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

12.1 SECURITY OF SAMPLES

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

13.0 DATA VERIFICATION

In 2007, 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 these 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 deposit.

In August 2006, two historical diamond drill holes were twinned to verify the validity of the historical assay results using current diamond 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 below:

Table 13.1 Coordinates of Twin Drill Holes

Hole	NAD83UTME	NAD83UTMN	Az_True	Dip
1-40-91	411878	6913907	248.5	-50
WC-003	411875	6913902	245	-50
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 below. The hanging wall and footwall contacts were well defined in all four drillholes. 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 13.2 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 (%)	0.7	0.5	0.87	0.44		1.45	1.05	0.94	0.87	
Var (%)	0.59	0.41	0.7	0.34		0.91	0.66	0.65	0.59	

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-metre 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 9 feet (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-metre intervals falling within 1991sample 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.

Table 13.3 Comparison of Check Drilling to Historical Drill Intersections

Hole No.	1991 Sample Intervals						2006 One Metre Re-Assays					
	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	1.10	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	0.34	36.88	37.79	0.91	2.43	2.80	0.289
1-28-91	23.77	26.82	3.05	1.61	1.72	0.41	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	0.07	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	0.27	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	0.55	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	0.07	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	0.45	54.86	55.77	0.91	1.28	1.39	0.944
1-57-91	79.25	81.69	2.44	1.20	1.33	3.63	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	0.00	88.39	89.30	0.91	0.37	0.42	0.013
Average				**1.42**	**1.60**	**0.689**				**1.63**	**1.93**	**0.550**

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

In addition, Wardrop carried out a site visit between May 15 and 16, 2007. The purpose of the visit was to examine the property access, surface geology, take representative samples of the mineralization, and locate drill hole collars with GPS. A total of sixteen drill hole collars were located and all the locations agreed well with the data entered in the geology database used for resource estimation.

Wardrop 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 .

Table 13.4 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

Wardrop check 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. The following table lists the statistical results from the standards analyses from both Chemex and Acme:

Table 13.5 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. Below are six tables that show the comparison between original samples and duplicate samples, submitted to Chemex and between original samples submitted to Chemex and duplicate samples submitted to ACME for gold, non-sulphide copper and total copper analyses:

Figure 13.1

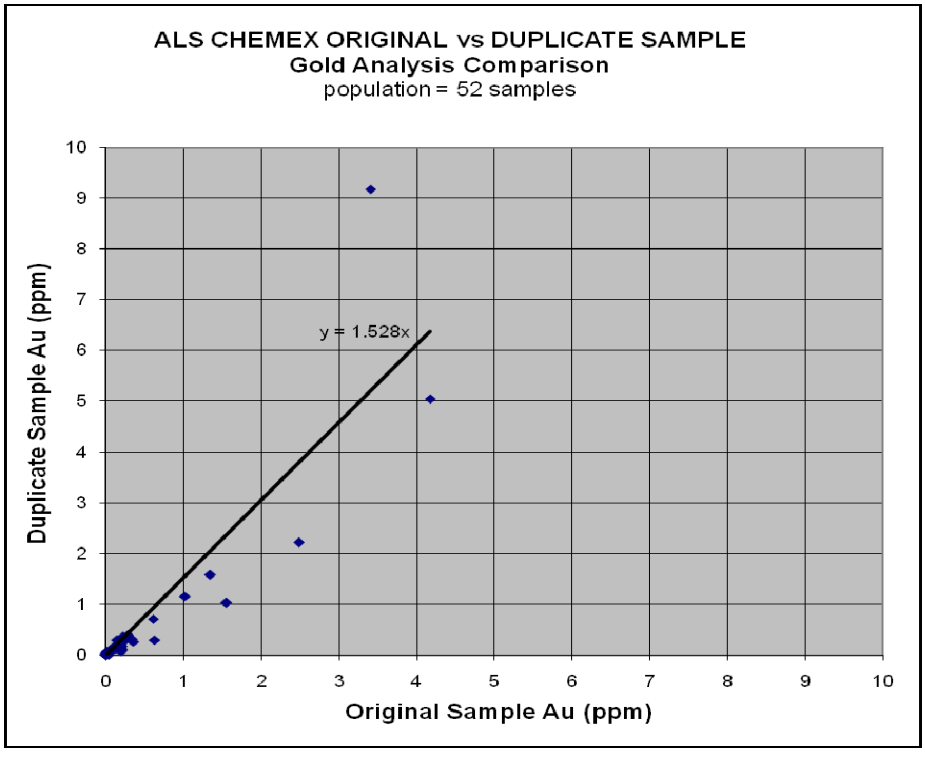


Figure 13.2

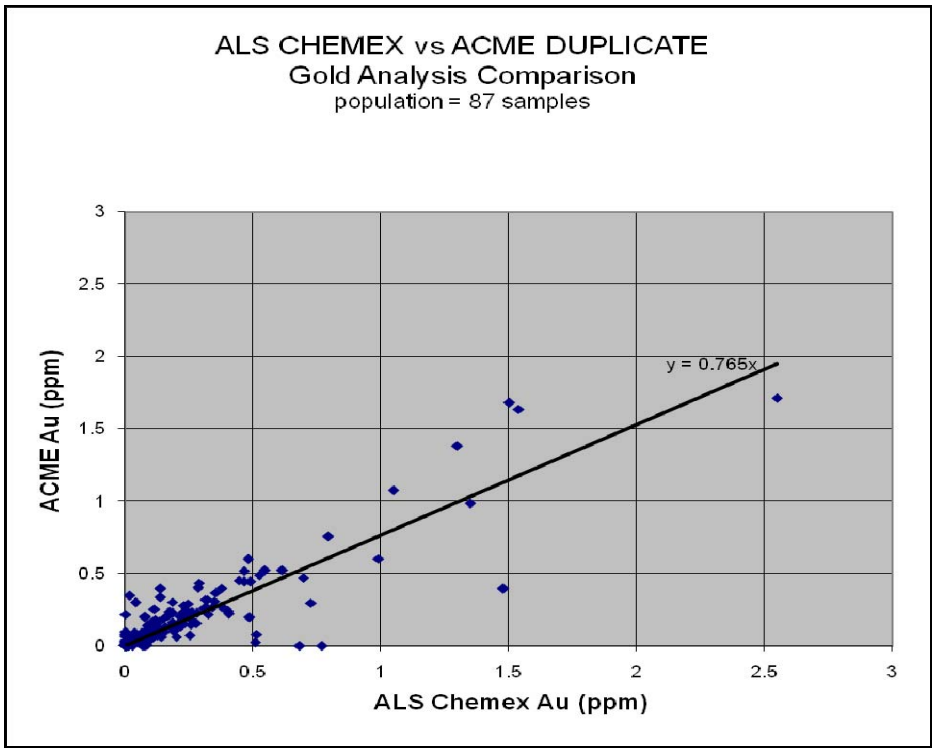


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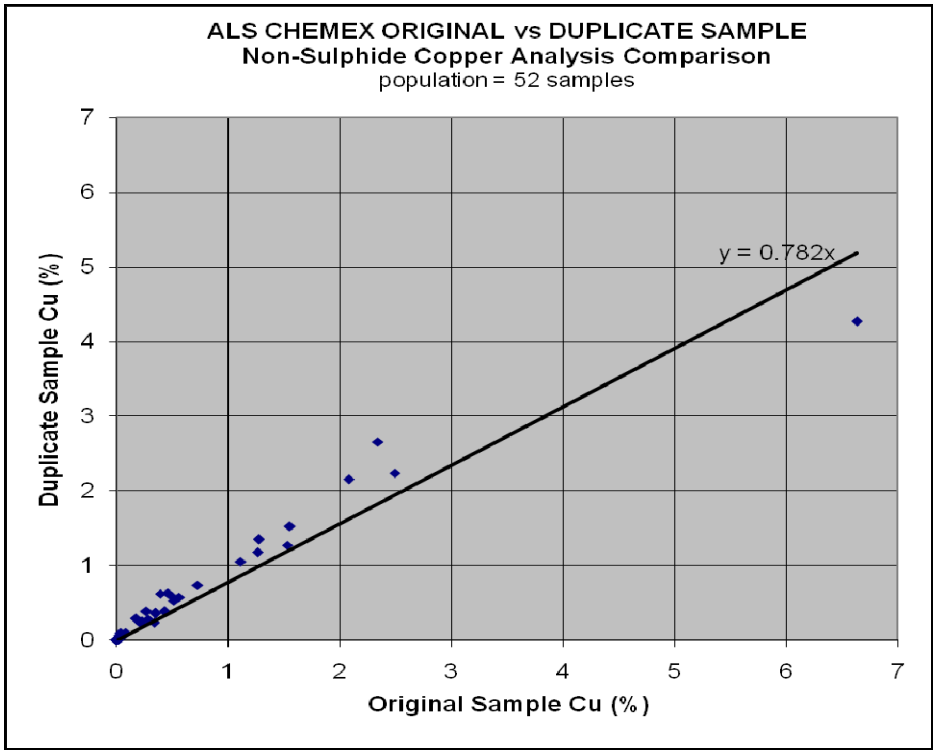


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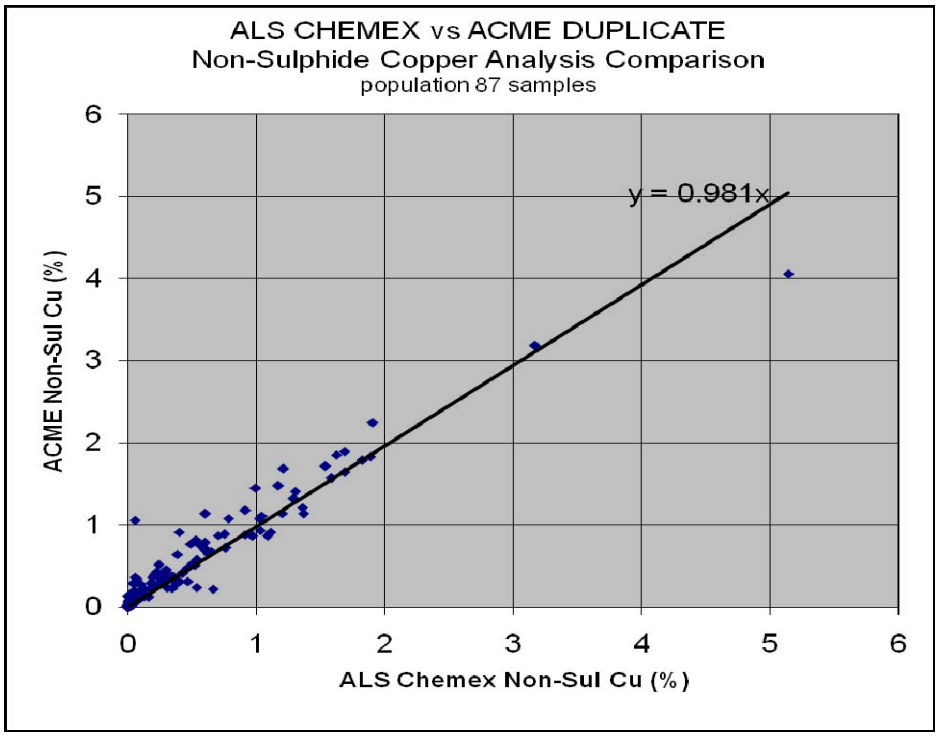


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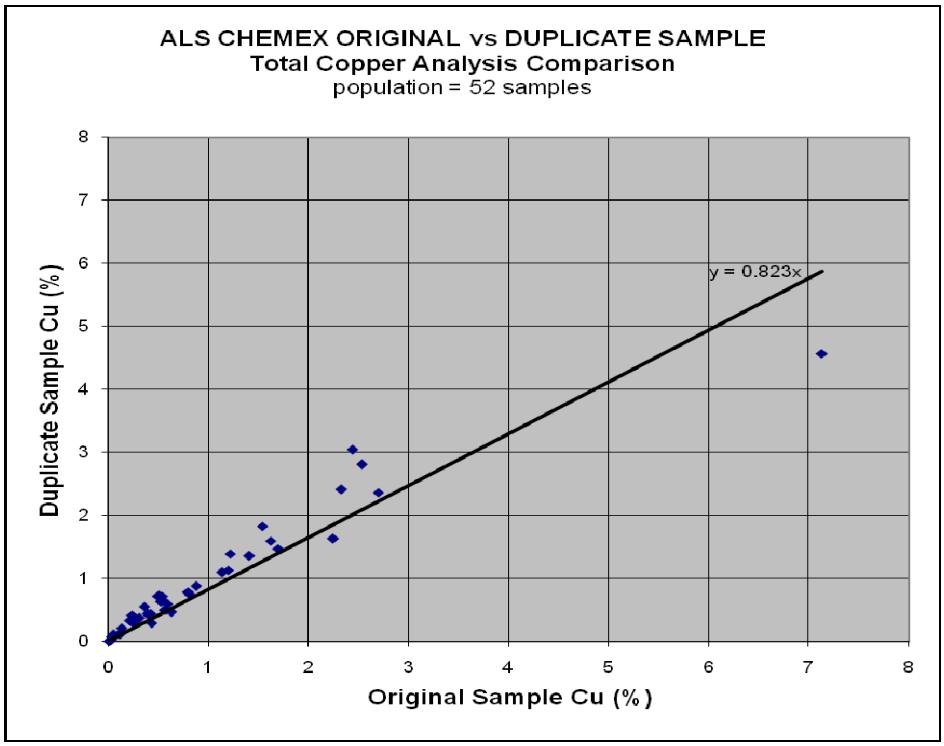
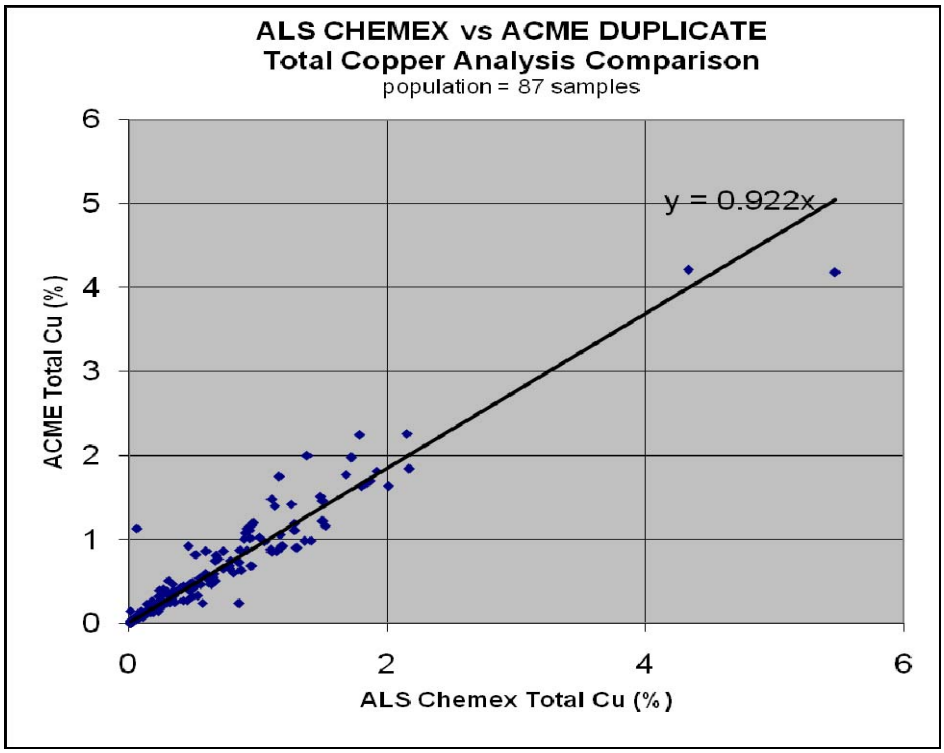


Figure 13.6



The greatest variability is seen in the gold analyses, which can be expected due to the coasre 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.

14.0 ADJACENT PROPERTIES

BC Gold has claims immediately north, south and east of the Carmacks Copper Property. In 2007 and 2008, they conducted exploration programs consisting of airborne geophysical surveying, mobile metal ion soil geochemical sampling (MMI sampling), prospecting, geological mapping and 1,360 m of diamond drilling in 7 holes and in 2008 drilled a further 5 holes for 1,235 m. The most significant results included 4.69 m grading 0.46% copper in hole ICE 07-02, 5.47 m grading 0.39% copper in hole ICD07-04 and 63.1 m grading 0.17% copper in hole WS08-08.

15.0 MINERAL PROCESSING and METALLURGICAL TESTING

A series of metallurgical tests have been carried out on the Carmacks property beginning in 1971 and continuing, intermittently, to the present. The testing programs focused mainly on the recovery of acid soluble copper mineralization in the oxide cap of the No. 1 Zone as summarized in Table 15.1.

Table 15.1 Summary of Metallurgical Testing on the No. 1 Zone

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 & column
6/1990	BD&A	Unkown	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

Tests carried out by Beatties Consulting PRA (**Beattie**) between 1996 and 2001 closely mimic the conditions being considered commercially. These use samples crushed to -20 mm and agglomerated; columns were greater than 5 m in height, and the columns were irrigated with solution at a pH of 1-1.5 for 160-190 days.

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

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

If Cu(oxide)/Cu(total) > 0.79, Leachable Copper = 85%

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

The acid consumption rates calculated during the sequential leaching tests indicated average acid consumption of 25 kilograms per tonne (**kg/t**). These tests were performed on pulverized samples; the actual acid consumption from this material at the planned coarser crush size would be expected to be lower.

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.

16.0 MINERAL RESOURCE and MINERAL RESERVE ESTIMATES

The current resource estimate for the Carmacks Deposit was calculate in 2007 by Mr. Michael Waldegger and Dr. Gilles Arseneau, of Wardrop. The estimate can found in the report, *"Resource Estimate of the No. 1 and No. 7 Zones, Carmacks Deposit, Yukon Territory"*. Mineral resources were classified in accordance with definitions provided by CIM as stipulated in NI 43-101. The resources were estimated using 3D modeling software and are classified as Measured, Indicated, and Inferred.

Historical drill hole data (pre-2006) was converted from a local Imperial grid to the Metric coordinate system, NAD83. Drill hole data from the completed 2006 and 2007 drilling campaigns was imported without conversion.

The two assay populations, pre-2006 and 2006 to 2007 drilling, were analyzed separately to determine any variances between the two drilling campaigns. Wardrop determined that there are no appreciable differences between the historical data and the 2006/2007 drill hole data.

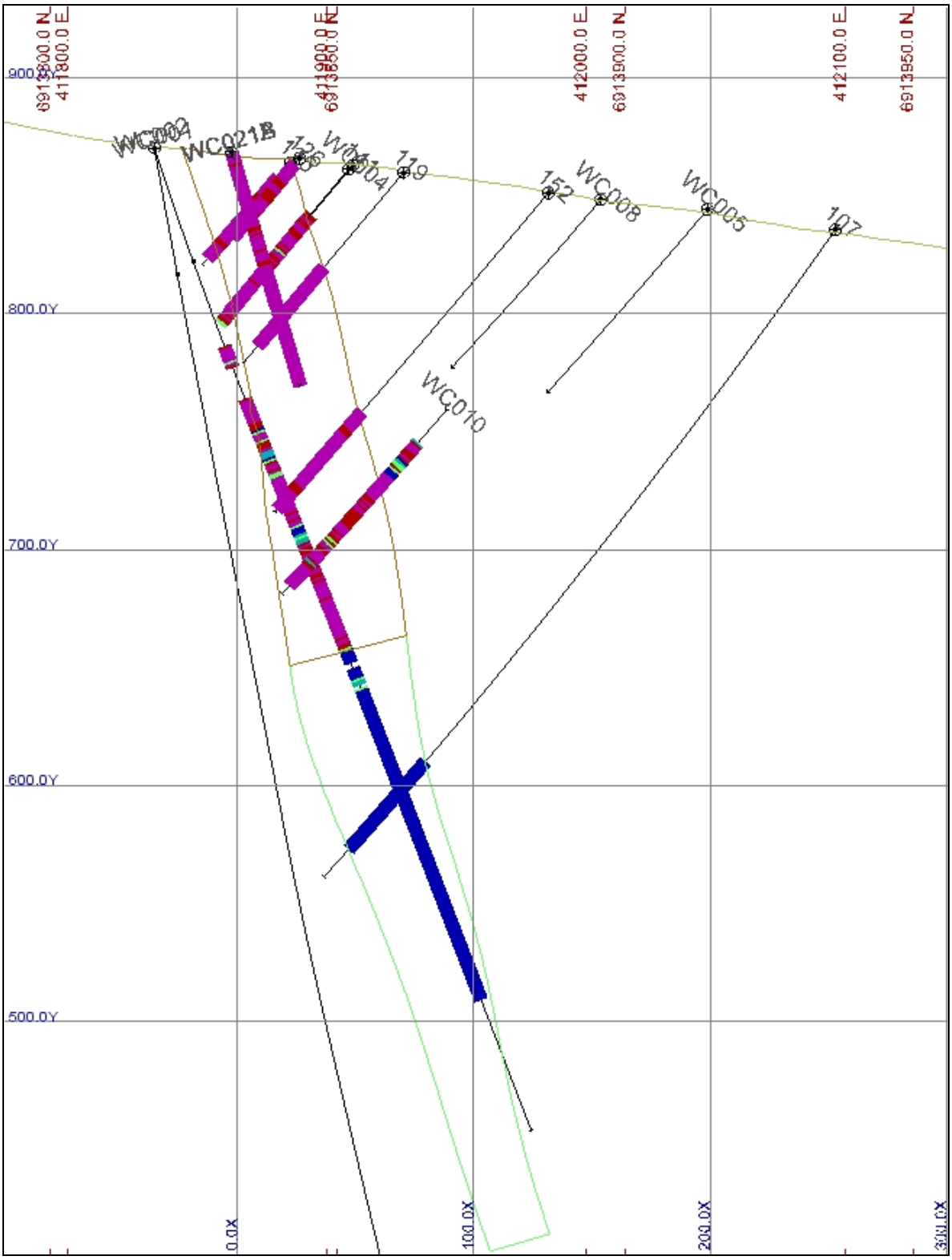
Grade capping was considered and evaluated by examining the cumulative frequency distribution and histograms for copper oxide and total copper. Wardrop decided not to apply grade capping to the assay data because the assay data were observed to be log normal distributed and did not appear to reflect multiple populations or contain significant high-grade outliers.

Assays were composited to a fixed length of five metres, within the geological solids. Composites were generated from the base of the hole upwards to ensure that the higher gold values found associated with the footwall of the zones did not get eliminated by the compositing routine.

The mineralized zones were interpreted on the basis of total copper grade. Surfaces were generated based on grade data to represent the hangingwall and footwall contacts of the mineralized zones.

The oxide sulphide boundary was modelled 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. The oxide and sulphide boundary is illustrated in Figure 16.1.

Figure 16.1 Cross Section Showing Oxide Copper to Total Copper Proportion in Drill Holes in Oxide & Sulphide Solids



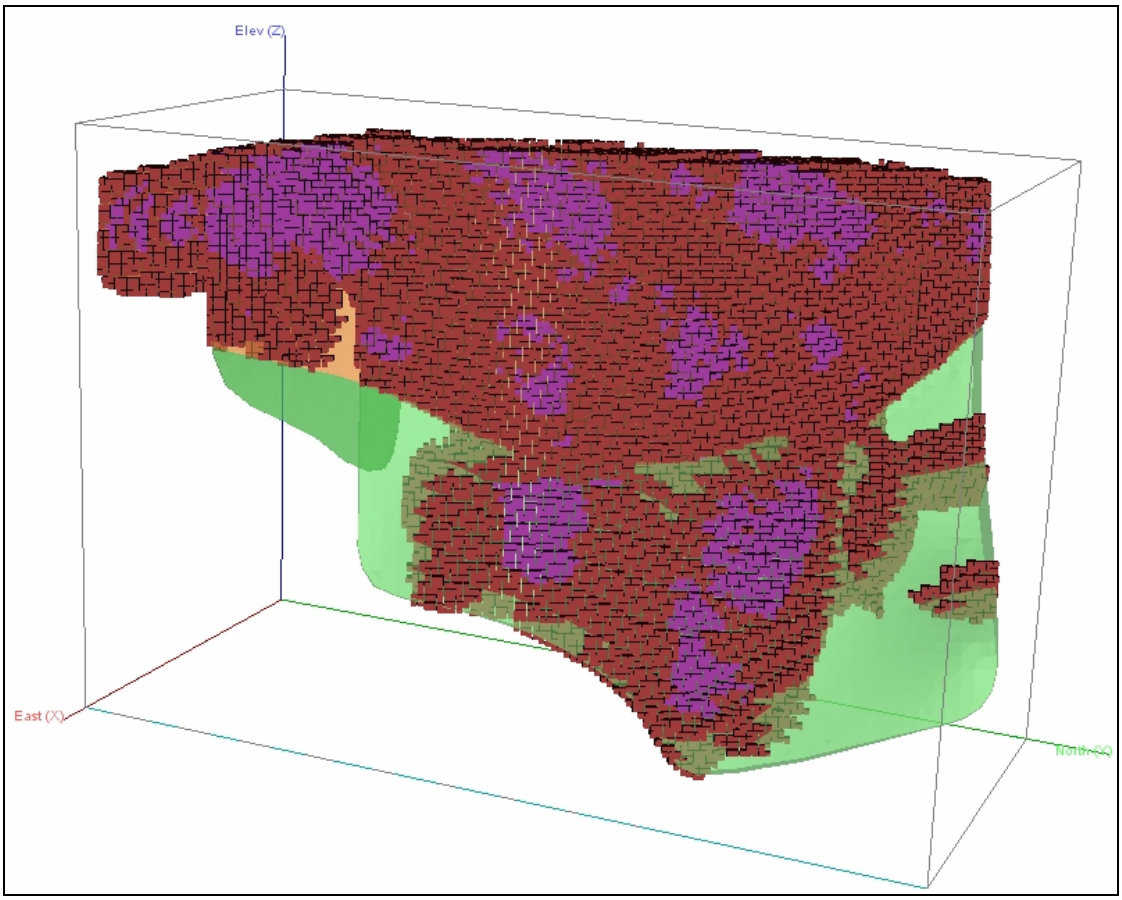
The density model was created by rock type to values as outlined in Table 16.1.

Table 16.1 Density by Rock Type

Rock Type	Density (g/cc)
Air	0
Waste	2.65
Zone 1 Oxide	2.68
Zone 7 Oxide	2.68
Zone 1 Sulphide	2.71
Zone 7 Sulphide	2.71

Metal grades were interpolated into blocks using ordinary kriging with weighting parameters based on correlogram data. The block model contained 16,037 blocks classified as Measured, 41,643 as Indicated, and 14,492 as Inferred (Figure 16.2).

Figure 16.2 Three Dimensional Representation of Block Model



Note: Magenta colour blocks are Measured mineral resource and Red blocks are Indicated mineral resource.

Table 16.2 Mineral Resources at 0.25% Total Copper Cutoff

Rock Group	Grade Group	Density (t/m^3)	Tonnage t(000)	Tcu (%)	CuX (%)	CuS (%)	Au (g/t)	Ag (g/t)	Class
Oxide	Meas	2.680	3288	1.23	1.03	0.20	0.652	6.135	1
	Ind	2.680	7084	1.09	0.93	0.16	0.458	4.382	2
	Meas+Ind	2.680	10372	1.13	0.96	0.17	0.519	4.938	
	Inf	2.684	82	0.82	0.62	0.21	0.207	1.502	3
Sulphide	Meas	2.710	1256	0.72	0.04	0.67	0.227	2.398	1
	Ind	2.710	4399	0.78	0.06	0.72	0.227	2.200	2
	Meas+Ind	2.710	5655	0.77	0.05	0.71	0.227	2.244	
	Inf	2.710	3161	0.69	0.07	0.62	0.180	1.644	3

Wardrop also estimated the mineral resources using a 0.50% TCu and 0.80% TCu cutoff, as indicated in Table 16.13 and Table 16.14.

Table 16.3 Mineral Resources at 0.50% Total Copper Cutoff

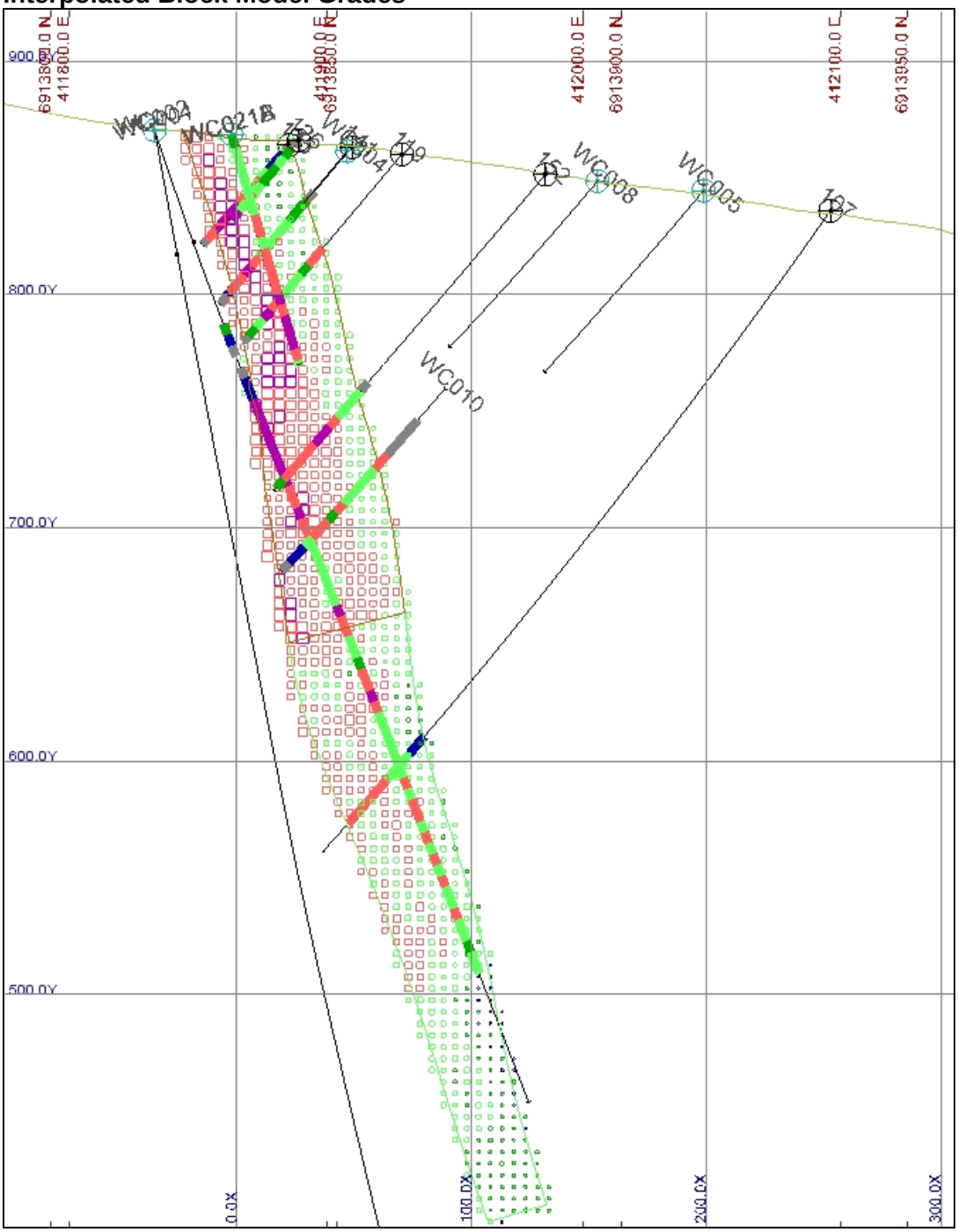
Rock Group	Grade Group	Density (t/m^3)	Tonnage t(000)	Tcu (%)	CuX (%)	CuS (%)	Au (g/t)	Ag (g/t)	Class
Oxide	Meas	2.680	3164	1.26	1.05	0.20	0.672	6.318	1
	Ind	2.680	6820	1.12	0.95	0.16	0.468	4.481	2
	Meas+Ind	2.680	9983	1.16	0.98	0.18	0.533	5.063	
	Inf	2.684	81	0.83	0.62	0.21	0.209	1.516	3
Sulphide	Meas	2.710	1055	0.77	0.05	0.73	0.239	2.478	1
	Ind	2.710	3474	0.89	0.06	0.82	0.247	2.437	2
	Meas+Ind	2.710	4528	0.86	0.06	0.80	0.245	2.446	
	Inf	2.710	1952	0.89	0.10	0.78	0.255	2.380	3

Table 16.4 Mineral Resources at 0.80% Total Copper Cutoff

Rock Group	Grade Group	Density (t/m^3)	Tonnage t(000)	Tcu (%)	CuX (%)	CuS (%)	Au (g/t)	Ag (g/t)	Class
Oxide	Meas	2.680	2657	1.37	1.15	0.22	0.753	7.054	1
	Ind	2.680	5721	1.20	1.02	0.18	0.503	4.846	2
	Meas+Ind	2.680	8378	1.25	1.06	0.19	0.582	5.546	
	Inf	2.687	18	1.13	0.67	0.46	0.333	3.292	3
Sulphide	Meas	2.710	372	1.00	0.04	0.96	0.294	2.632	1
	Ind	2.710	1862	1.10	0.08	1.03	0.276	2.662	2
	Meas+Ind	2.710	2235	1.09	0.07	1.02	0.279	2.657	
	Inf	2.710	1106	1.09	0.12	0.97	0.341	3.287	3

Wardrop completed a detailed visual validation of the Carmacks block model by checking for proper coding of drill hole intervals and block model cells, in both section and plan. Coding was found to be properly done and the checks showed good agreement between drill hole composite values and model cell values (Figure 16.3).

Figure 16.3 Cross Section with Drill Hole Composites Showing Total Cu Against Interpolated Block Model Grades



17.0 INTERPRETATION and CONCLUSIONS

In the 2007 resource estimate, Wardrop concluded that both the historical and current drilling databases were of sufficient quality for the purpose of resource estimation. The Carmacks deposit is sufficiently defined for the estimation of Measured, Indicated, and Inferred mineral resources.

Wardrop concluded that, at a 0.25% total copper cutoff, the Carmacks deposit contains approximately 10 million tonnes of oxide resources in the combined Measured plus Indicated categories grading 0.96% CuX, 0.17% CuS, 0.519 g/t Au, and 4.938 g/t Ag.

Wardrop estimated the deposit contains an additional 5.7 million tonnes of sulphide resources in the combined Measured plus Indicated categories grading 0.05% CuX, 0.71% CuS, 0.227 g/t Au, and 2.244 g/t Ag.

Western Copper is currently studying alternative heap leach designs to resolve the Water Licence issues, including modifications to the plant design. These may require additional geotechnical investigations to support the design change and an updated capital cost estimate for the project.

Exploration potential for oxide copper and sulphide resources exists in the Zone 2, 12, 13 and 14 areas. Additional drilling will be required to define these zones. Drilling to test and define these areas is estimated at 5,000 to 15,000 m.

The Phase 1 budget estimate for the alternative heap leach and plant design, geotechnical investigation and updating the feasibility study is $600,000. The Phase 2 budget estimate for evaluation of the under-explored zones is $1,000,000 to $3,000,000. Although the project does not require the additional zones to be economic.

Respectfully Submitted

Scott Casselman, P.Geo

18.0 CERTIFICATE of QUALIFIED PERSON

I, Scott Casselman of Whitehorse Yukon, do hereby certify that as author of this "2011 Qualifying Report for the Carmacks Copper Deposit, Yukon Territory", dated 15 August, 2011, make the following statements:

- I am employed of Casselman Geological Services Ltd, 33 Firth Road, Whitehorse, Yukon, Y1A 4R5.
- I have a B.Sc. in Geology from the Carleton University of Ottawa, obtained in 1985.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #20032.
- I have practiced my profession in mineral exploration continuously since graduation. I have over twenty five years of experience in mineral exploration.
- 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 purpose of NI 43-101.
- I am responsible for this technical report. I was field supervisor on this project from July 10 to October 21, 2006, from February 3 to October 16, 2007 and from March 31 to May 30, 2008.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- As of the date of this Certificate, to my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
- I am independent of the Issuer as described in Section 1.4 of National Instrument 43-101.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

Signed and dated this 15th day of August, 2011 at Whitehorse, Yukon.

"signed and sealed'

Scott Casselman, P.Geo.
Geologist
Casselman Geological Services Ltd

19.0 REFERENCES

Arsenault, G. and Casselman, S, November, 2007, Resource Estimate of the No. 1, No. 4 and No. 7 zones, Carmacks Deposit, Yukon Territory.

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COPPER NORTH
MINING CORP.

2011 ANNUAL QUARTZ MINING LICENCE REPORT

SUBMITTED TO YUKON GOVERNMENT, ENERGY MINES AND RESOURCES

YUKON QUARTZ MINING LICENCE QML-0007

Appendix B

2011 Assessment Report for the Carmacks Copper Project

2011 Assessment Report
for the
Carmacks Copper Project

Whitehorse Mining District
Yukon Territory
NTS 115I/07
Latitude: 62° 21'
Longitude: 136° 41'

Quartz Claims:

Claim	Grant #
BOY 20	Y 51118
BOY 51	Y 51149
BOY 52	Y 51150
BOY 53	Y 51151
BOY 54	Y 51152
BOY 83	Y 51181
WAR 22	Y 59373
DUN 1	Y 59382
W 1 - 37	YB26708 - 44
W 41 - 43	YB26748 - 50
WAR 23 - 31	YB36240 - 48
W 50 - 53	YB36249 - 52
W 55	YB36254
W 57	YB36256
WAR 32 - 37	YB36446 - 51
WC 5 - 18	YB36693 - 706
WC 23 - 36	YB36711 - 724
WC 40	YB36728
WC 41 - 54	YB36729 - 42
WC 57 - 72	YB36745 - 60
WAR 38 - 50	YB36765 - 77

Claim	Grant #
X 3 - 4	YB36898 - 99
W 91 - 93	YB36929 - 31
W 95	YB36933
X 5 - 7	YB36962 - 64
VW 11	YB96620
VW 13	YB96622
VW 17 - 21	YB96626 - 30
VW 23	YB96632
VW 25	YB96634
VW 27 - 38	YB96636 - 47
VW 40 - 50, 60, 61	YB96986 - 98
TT 1	YB97068
TT 2	YB97251
REM 1 - 14	YC39221 - 34
REM 19 - 30	YC39239 - 50
REM 32 - 35	YC39251 - 54
WCC 1 - 40	YC60381 - 420
GAP 1 - 5	YC65320 - 24
HIP 1 - 27	YC65554 - 80
JIM 1 - 30	YC66844 - 73

By:
S. G. Casselman, B.Sc., P. Geo.
Casselman Geological Services Ltd.
33 Firth Road, Whitehorse, Yukon, Y1A 4R5

For:
Copper North Mining Corp.
2050 - 1111 West Georgia Street
Vancouver, BC, V6E 4M3

SUMMARY

In September, 2011, Copper North Mining Corp. contracted Casselman Geological Services Ltd to conduct an exploration program consisting of surficial geological mapping and soil sampling on the Carmacks Copper Property. The program was designed to evaluate the area proposed for the future mine access road, in the southern part of the claim group.

Copper North Mining Corp. was formed in November, 2011 by spin-off from Western Copper Corporation. Copper Norths' mining properties included the Carmacks Copper Property and Redstone Copper Property in Northwest Territories.

The Carmacks Copper Property is located on Williams Creek, 192 km north of Whitehorse, or 38 km northwest of Carmacks, Yukon. It is accessible by 4x4 vehicles via the Freegold Road and the Carmacks Copper access road. The property consists of 318 Quartz Claims and 20 Mining Leases owned 100% by Carmacks Copper Ltd., a 100% owned subsidiary of Copper North Mining Corp.

The exploration history of the region dates back to the Klondike Gold Rush of 1898, when placer miners traveling the Yukon River started prospecting along the route. Between 1969 and 1995 a number of companies and partnerships were involved in large exploration programs, and an extensive array of metallurgical test work, engineering and environmental studies were completed on the property. Sixteen zones containing copper mineralization known on or in the immediate vicinity of the property had been identified during that time, of which the No. 1 Zone was the most advanced.

In 1995, the company submitted a Quartz Mining License application to the federal government. However, the permit was not forthcoming and, due to changing market conditions, the company withdrew the application a few years later. In 2005, the company re-submitted the project for environmental screening with the Yukon Environmental and Socio-Economic Assessment Board and in 2009 received a Quartz Mining License.

From 2006 to 2008, Western Copper Corporation conducted extensive exploration on the property, including 24,930 m of diamond drilling, 1,201 m of rapid air-blast drilling, 997 m of geotechnical drilling, induced polarization surveying, road and camp upgrades, prospecting, baseline environmental studies, engineering studies, prepared a pre-feasibility study and initiated a feasibility study.

The property hosts copper oxide mineralization in large meta-volcanic xenoliths that have been consumed by granodiorite of the Granite Mountain Batholith. In 2007, the company calculated a NI43-101 compliant resource estimate in the combined measured plus indicated categories for the No. 1, 4, 7 and 7A zones of 10.0 million tonnes grading 0.96% oxide copper, 0.519 g/t gold and 4.938 g/t silver. The deposit contains an additional 5.7 million tonnes of sulphide resources in the combined measured plus indicated categories grading 0.71% sulphide copper 0.227 g/t gold, and 2.244 g/t silver. The mineral resources were reported at 0.25% total copper cut-off grade.

The 2011 exploration program mapped and sampled surficial geological material along the proposed mine access road and identified a 1.4 km long esker immediately adjacent to the proposed road. The esker is comprised of washed sand and cobbley-gravelly sand that would be suitable for road construction. It is recommended that the road alignment be adjusted slightly eastward and be built on the esker. The current planned road is located in low-lying, swampy ground. Moving the road approximately 50 to 75 m eastward would locate it on the esker, on raised, well drained ground and would reduce the requirement for transport of fill.

Ten soil samples were collected from test pits along the esker. The test pits were mapped and photographed. The samples did not return any anomalous copper or deleterious elements, thus corroborating the suitability of the proposed re-alignment location.

Recommendations for future exploration work on the property are to complete the detailed surficial mapping program over the remainder of the proposed access road. This program is estimated to cost $40,000.

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1.0 INTRODUCTION

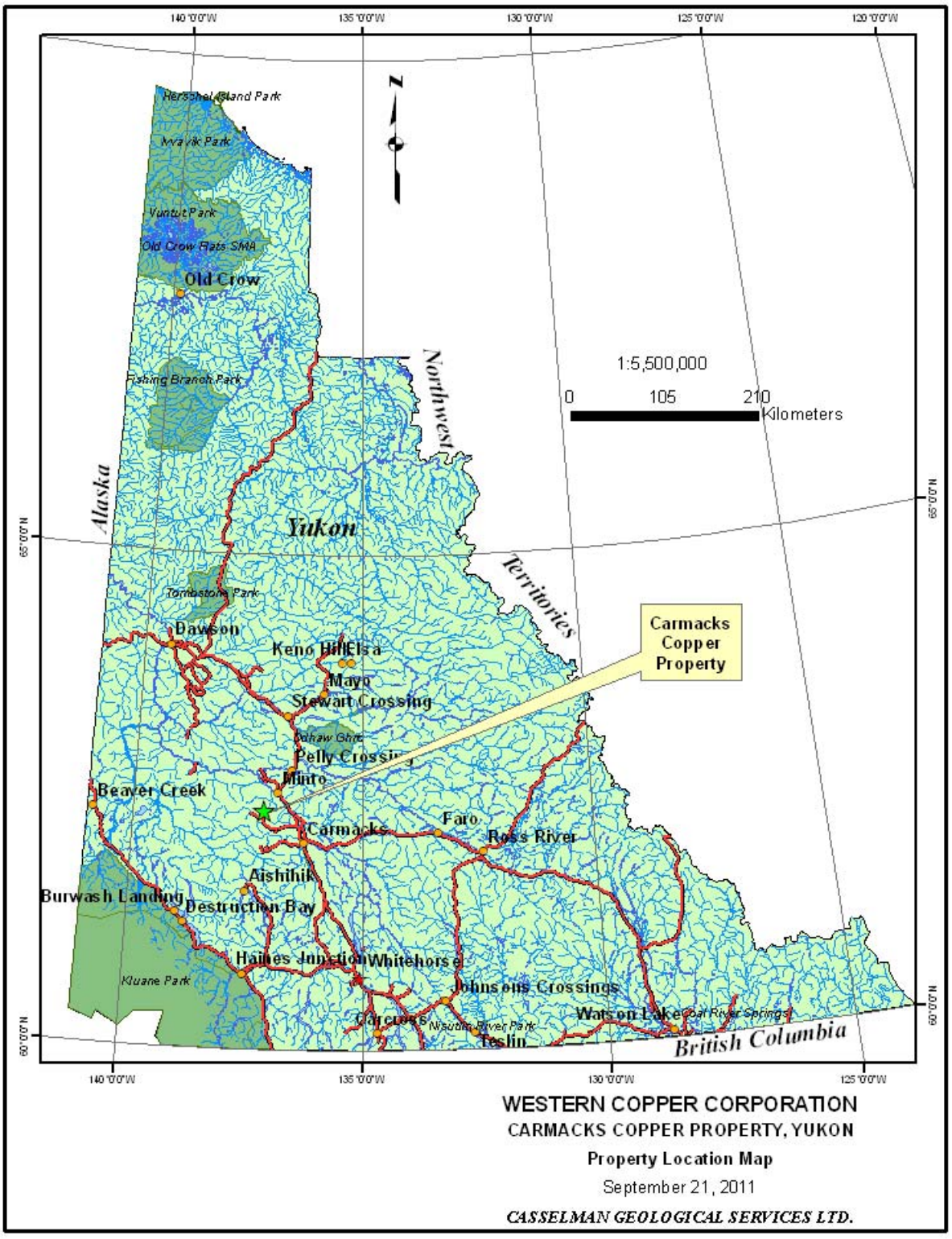
In the fall of 2011, Western Copper Corporation (later re-organized to form Copper North Mining Corp.) contracted Casselman Geological Services Ltd to conduct an exploration program on the Carmacks Copper Property. The program consisted of surficial geological mapping, test pitting and sampling on the southern portion of the property, in an area proposed for the future mine access road. The program also involved one day of repair work on the old access road to provide access to the camp. This report documents the program for assessment purposes on the Quartz Claims.

The exploration program was managed by Scott Casselman, B.SC, P.Geo, of Casselman Geological Services Ltd of Whitehorse, Yukon. Heavy equipment services were provided by H. Coyne and Sons Ltd of Whitehorse. The field program commenced on September 12 and was completed on September 16, 2011. An additional trip to the site was conducted on October 20, to complete the cleanup of the site and remove 3 drums of fuel, as recommended by the Mines Inspector.

2.0 LOCATION AND ACCESS

The Carmacks Copper Project is located in the Dawson Range Mountains on NTS map sheet 115I07 at latitude 62°21'N and longitude 136°41'W, some 192 km north of Whitehorse, Yukon as shown on Figure 1. 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.

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 centre of 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 a part of the project development; brush clearing in preparation of this has occurred.


140°0'0"W
135°0'0"W
130°0'0"W
125°0'0"W
120°0'0"W
65°0'0"N
60°0'0"N
Herschel Island Park
Ivvavik Park
Vuntut Park
Old Crow Flats SMA
Old Crow
Fishing Branch Park
Alaska
Yukon
Northwest
Territories
British Columbia
Tombstone Park
Dawson
Keno Hill
Elsa
Mayo
Stewart Crossing
Tatchun Creek
Pelly Crossing
Minto
Beaver Creek
Carmacks
Faro
Ross River
Aishihik
Burwash Landing
Destruction Bay
Haines Junction
Whitehorse
Kluane Park
Johnsons Crossings
Watson Lake
Coal River Springs
Nisutlin River Park
Teslin
Carmacks Copper Property
1:5,500,000
0
105
210
Kilometers
WESTERN COPPER CORPORATION
CARMACKS COPPER PROPERTY, YUKON
Property Location Map
September 21, 2011
CASSELMAN GEOLOGICAL SERVICES LTD.

3.0 CLAIM INFORMATION

The Carmacks Copper Property is in the Whitehorse Mining District and consists of 318 Quartz Claims and 20 Mining Leases acquired in accordance with the Yukon Quartz Mining Act. The claims are register in the name of, and owned 100% by Carmacks Copper Ltd., a 100% owned subsidiary of Copper North Mining Corp. The claim location map is included in Figure 2. A list of lease information is included in Table 1 and claim information in Table 2.

The claim information has been verified with the Yukon Mining Recorder NMRS database, current to November 1, 2011, which states that the expiry dates listed are valid and that all the claims and leases are in good standing (www.yukonminingrecorder.ca).

Archer, Cathro & Associates (1981) Limited, at the election of Copper North retain a 3.0% NSR royalty to a maximum of $2.5 million. Annual advanced royalty payments of $100,000 are made whenever the average annual price of copper is above US$1.10/pound. To date $300,000 of the royalty has been paid.

The property is located on Crown Land and the surface rights are unencumbered. Immediately west of the property is First Nations Class A Land Reserve LSC R-9A belonging to the Little Salmon Carmacks First Nation. Both surface and mineral rights are reserved for the First Nation on Class A land.

In 1972, the leases on the property were surveyed as per one of the requirements of obtaining a lease. 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. The far northern claims (WC claims), southern claims (VW, TT, X and JIM) and eastern claims (REM and HIP) have not been legally surveyed.

Table 1. Quartz Lease Status

Claim Name and No.	Grant No.	Expiry Date	Registered Owner	% Owned	Quartz Lease
AC # 2-3	Y91722–Y91723	2019/10/28	Carmacks Copper Ltd.	100	OW00070-71
BOY 22	Y51120	2019/10/28	Carmacks Copper Ltd.	100	OW00061
BOY 24	Y51122	2019/10/28	Carmacks Copper Ltd.	100	OW00062
BOY 55–58	Y51153–Y51156	2019/10/28	Carmacks Copper Ltd.	100	OW00063-66
BOY 85	Y51183	2019/10/28	Carmacks Copper Ltd.	100	OW00067
DUN 2-3	Y59383–Y59384	2019/10/28	Carmacks Copper Ltd.	100	OW00068-69
W 38-40	YB26745-YB26747	2019/10/28	Carmacks Copper Ltd.	100	OW00072-74
W 44-48	YB26751-YB26755	2019/10/28	Carmacks Copper Ltd.	100	OW00075-79
W 49	YB26756	2025/03/09	Carmacks Copper Ltd.	100	OW00080

Table 2. Quartz Claim Status

Claim Name and No.	Grant No.	Expiry Date	Registered Owner	% Owned
BOY 20	Y51118	2012/03/09	Carmacks Copper Ltd.	100
BOY 51, 53	Y51149, Y51151	2012/03/09	Carmacks Copper Ltd.	100
BOY 52, 54	Y51150, Y51152	2014/03/09	Carmacks Copper Ltd.	100
BOY 83	Y51181	2012/03/09	Carmacks Copper Ltd.	100
DUN 1	Y59382	2012/03/09	Carmacks Copper Ltd.	100
TT 1	YB97068	2015/03/09	Carmacks Copper Ltd.	100
TT 2	YB97251	2012/03/09	Carmacks Copper Ltd.	100
VW 11, 13	YB96620, YB 96622	2012/03/09	Carmacks Copper Ltd.	100
VW 17-18	YB96626-YB6627	2015/03/09	Carmacks Copper Ltd.	100
VW 19–21	YB96628–YB96630	2016/03/09	Carmacks Copper Ltd.	100
VW 23, 25	YB96632, YB6634	2015/03/09	Carmacks Copper Ltd.	100
VW 27–38	YB96636-YB96647	2015/03/09	Carmacks Copper Ltd.	100
VW 40-50	YB96986-YB96996	2012/03/09	Carmacks Copper Ltd.	100
VW 60-61	YB96997-YB96998	2015/03/09	Carmacks Copper Ltd.	100
REM 1-14	YC39221-YC39234	2012/04/11	Carmacks Copper Ltd.	100
REM 19-30	YC39239-YC39250	2012/04/11	Carmacks Copper Ltd.	100
REM 32-35	YC39251-YC39254	2012/04/11	Carmacks Copper Ltd.	100
W 1-6	YB26708-YB26713	2012/03/09	Carmacks Copper Ltd.	100
W 7-10	YB26714-YB26717	2015/03/09	Carmacks Copper Ltd.	100
W 11,13,15,17,19,20	YB26718,20,22,24,26,27	2012/03/09	Carmacks Copper Ltd.	100
W 12,14,16,18	YB26719,21,23,25	2013/03/09	Carmacks Copper Ltd.	100
W 21-24	YB26728-YB26731	2015/03/09	Carmacks Copper Ltd.	100
W 25-28	YB26732-YB26735	2013/03/09	Carmacks Copper Ltd.	100
W 29-30	YB26736-YB26737	2012/03/09	Carmacks Copper Ltd.	100
W 31-37	YB26738-YB26744	2015/03/09	Carmacks Copper Ltd.	100
W 41-43	YB26748-YB26750	2013/03/09	Carmacks Copper Ltd.	100
W 50-53	YB36249-YB36252	2015/03/09	Carmacks Copper Ltd.	100
W 55 ,W 57	YB36254, YB36256	2013/03/09	Carmacks Copper Ltd.	100
W 91-93	YB36929-YB36931	2015/03/09	Carmacks Copper Ltd.	100
W 95	YB36933	2015/03/09	Carmacks Copper Ltd.	100
X 3-4	YB36898, 899	2015/03/09	Carmacks Copper Ltd.	100
X 5-7	YB36962-YB36964	2015/03/09	Carmacks Copper Ltd.	100
WC 5-18	YB36693-YB36706	2012/03/09	Carmacks Copper Ltd.	100
WC 23-36	YB36711-YB36724	2012/03/09	Carmacks Copper Ltd.	100
WC 40-54	YB36728-YB36742	2012/03/09	Carmacks Copper Ltd.	100
WC 57-72	YB36745-YB36760	2012/03/09	Carmacks Copper Ltd.	100
WAR 22	Y59373	2016/03/09	Carmacks Copper Ltd.	100
WAR 23-29	YB36240-YB36246	2015/03/09	Carmacks Copper Ltd.	100
WAR 30-31	YB36247-YB36248	2012/03/09	Carmacks Copper Ltd.	100
WAR 32-37	YB36446-YB36451	2015/03/09	Carmacks Copper Ltd.	100
WAR 38-44	YB36765-YB36771	2015/03/09	Carmacks Copper Ltd.	100
WAR 45	YB36772	2012/03/09	Carmacks Copper Ltd.	100
WAR 46	YB36773	2015/03/09	Carmacks Copper Ltd.	100
WAR 47-50	YB36774-YB36777	2012/03/09	Carmacks Copper Ltd.	100
WCC 1-40	YC60381-YC60420	2013/05/10	Carmacks Copper Ltd.	100
GAP 1-5	YC65320-YC65324	2013/03/09	Carmacks Copper Ltd.	100
HIP 1-12	YC65554-YC65565	2013/03/09	Carmacks Copper Ltd.	100
HIP 13, 14	YC65566-YC65567	2012/14/08	Carmacks Copper Ltd.	100
HIP 15-27	YC65568-YC65580	2013/03/09	Carmacks Copper Ltd.	100
JIM 1-30	YC66844-YC66873	2012/03/09	Carmacks Copper Ltd.	100

4.0 PHYSIOGRAPHY AND CLIMATE

The property is located in the Dawson Range Mountains. Topography at the property area is subdued. Topographic relief for the entire property is 515 metres (**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.

North facing slopes are generally underlain by permafrost and are generally swampy or boggy with much less tree growth and thick sphagnum moss cover. South facing slopes are generally drier and, in some locations, are free of permafrost. The hills are covered by black spruce, white spruce, pine, poplar, birch and alder trees at lower elevations and alpine grasses and scrub willows at higher elevations and in the alpine terrain.

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. Overburden is generally thin; a few centimetres **(cm)** 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.

The climate in the Carmacks area is marked by warm summers and cold winters. Mean daily temperatures range from -30°C in January to 12°C in July. The location close to the Arctic Circle provides 22 hours of daylight in late June but 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 375 millimetres (**mm**) (water equivalent) with one third falling as snow. July is the wettest month. Mean annual evaporation is approximately 404 mm to yield a net loss of 29 mm. Maximum evaporation occurs in July. 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.

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

5.0 PROPERTY HISTORY

The exploration history of the region dates back to the Klondike Gold Rush of 1898, when placer miners traveling the Yukon River started prospecting along the route. The earliest exploration work in the area was directed to the few outcrops in the Williams Creek, Merrice Creek, Nancy Lee Creek and Hoochekoo Creek canyons. The first claims staked in the region were staked west of the Yukon River on Nancy Lee Creek. There are a number of small adits and workings on the claims targeting vein copper mineralization. A few tons of copper ore were shipped to the Granby Smelter in 1917. These claims are now Crown Grants.

In 1969, the Casino Porphyry Deposit was discovered, which prompted a staking rush in the region. In March of 1970, the Boy Claims were staked by Whitehorse businessmen, G. Wing and A. Arsenault. The original claims consisted of 134 units. The property was optioned to the Dawson Range Joint Venture (DRJV), later that year. The DRJV consisted of Straus Exploration Inc., Great Plains Development of Canada Ltd., Trojan Consolidated Minerals Ltd. and Molybdenum Corporation of America.

The DRJV conducted a program of prospecting and geochemical sampling in the summer of 1970 and discovered two outcrops with copper oxide mineralization; the No. 1 and No. 2 zones. The discovery prompted the staking of a further 185 claims, some trenching and drilling of two x-ray diamond drill holes in the No. 1 Zone for a total of 103 feet (31.4 m).

In 1971, the DRJV conducted a program consisting of 24.5 km of road building, bulldozer trenching, 108 line-km of grid geochemistry, 27 line-km of VLF-EM geophysical surveying, 48 km of line-cutting, geological mapping, an airphoto survey and 5,583 m of diamond drilling in 25 holes in five separate zones (Zones 1, 2, 3, 5 and 6). Highlights of this program included drill indicated reserves in the No. 1 Zone of 16,334,000 tons grading 1.15% copper at a 0.6% copper cut-off (this reserve figure is not 43-101 compliant). The program also identified copper oxide mineralization in the No. 3 and 4 zones.

In 1972, the DRJV conducted a program consisting of an additional 2.1 km of road construction, bulldozer trenching, 31 km of line cutting, 150 line-km of soil sampling, and 1,531 m of diamond drilling in 8 holes in the No.1, No 4, and No. 8 zones. A recommendation for additional drilling in the No. 3, No. 12 and No. 13 zones was made following the exploration program. However, the mining industry went into a slump and no further work was performed for 17 years.

In 1982, the DRJV returned its' interest in the property to Archer, Cathro & Associates, which later sold the property to Archer, Cathro & Associates (1981) Ltd. In 1989, Archer, Cathro optioned the property to Western Copper Holdings Ltd. and Thermal Exploration Company. Western Copper and Thermal conducted metallurgical test work later that year.

In 1990, Western Copper and Thermal upgraded the access road to the property and drilled 322 m in three holes in the Zone 1. Each of the 3 holes intersected copper oxide mineralization. The following year the two companies conducted a program consisting of 3,464 m of diamond drilling in 36 holes; 35 in Zone 1 and 1 in Zone 4. They also dug 22

trenches in zones 1 and 4 for a total of 1,856 m of trenching, surveyed 83.2 line-km of magnetic and VLF-EM geophysics, and initiated baseline environmental studies. The program was successful in delineating and expanding the area of mineralization in Zone 1 and identifying additional copper oxide mineralization in Zone 4. The geophysical survey also identified a number of magnetic anomalies on the property similar to the magnetic lows occurring in Zone 1 and identified a number of VLF-EM conductors worthy of follow-up.

In 1992, Western Copper and Thermal drilled 1,164 m in 11 holes in zones 1, 4, 12 and 13 and drilled 856 m in 11 Reverse Circulation holes at various locations on the property. The diamond drilling in Zone 4 encountered some copper oxide mineralization, but, unlike Zone 1, it was very disrupted by faulting and difficult to trace. The program also identified copper mineralization in gneissic rocks in zones 12 and 13 that was similar to Zone 1, but contained more sulphide minerals and less oxide minerals. The Reverse Circulation drilling identified a new zone, the 2000S zone, where hole RC-92-01 encountered 0.62% oxide Cu over 12.2 m and hole RC-92-10 encountered 0.47% oxide Cu over 12.2 m. The companies also conducted additional metallurgical test work, baseline environmental testing, a biophysical assessment of the area and contracted Knight Piesold Ltd to conduct geotechnical studies on the deposit consisting of test pit excavation, overburden sampling, oriented diamond drill core logging and geologic mapping.

In 1994, the companies expanded the grid on the property and conducted further magnetic and VLF-EM surveying, soil sampling and prospecting. A new area of copper mineralization was identified at the far northern part of the property, the 4000 Zone.

Also in 1994, Kilborn Engineering Pacific Ltd. was contracted to conduct a Feasibility Study. The study indicated that, based on the copper price at the time, the project was viable using open pit mining methods and solvent extraction-electowinning.

In 1995, the company contracted Knight and Piesold Ltd. to initiate a preliminary mine design and also initiated clearing and grubbing of a site access road and leach pad area. The company submitted a mine permit application later that year.

While the company was awaiting a mine permit, they contracted Kilborn Engineering to produce a basic engineering report, in 1997. The permit was not forthcoming and, due to changing market conditions the company withdrew the permit application. The property sat dormant until the re-initiation of permitting in 2005 and exploration in 2006. During this time the property was consolidated into a single company and it changed its' name to Western Silver Corporation.

In February, 2006 Western Silver Corporation was taken over by Glamis Gold and a new company, Western Copper Corporation, was formed. The Carmacks Copper Property was spun off to Western Copper as part of the arrangement.

In 2006, Western Copper resumed mineral exploration activities on the Carmacks Copper Project after it had lain dormant for 11 years. The company conducted and exploration program that consisted of 7,100 m of diamond drilling in 34 holes, 1,201 m of Rotary Air Blast drilling (RAB) in 61 holes, access road upgrade work, 9.2 km of line-cutting, re-

initiation of environmental baseline studies, surveying of drill hole collars and claim posts and re-initiation of the mine permitting process.

In 2007 the company conducted an exploration program that consisted of 17,830 m of diamond drilling in 122 holes, 942 m of overburden geotechnical drilling, in 33 holes, and one 55 m hydrogeological monitoring hole. As well, the company conducted access road upgrade work, 22.1 km of line-cutting, 49.3 km of Induced Polarization surveys, prospecting in the Zone 2 area and continued baseline environmental studies.

In 2008, the company conducted additional geotechnical investigations in preparation for development. In April 2009, Western Copper received a Quartz Mining License for the project. On May 10, 2010 the company was notified that the Water License had been denied with a list of deficiencies in the application and design process. Copper North is now working in collaboration with the regulators and agencies to address the issues which were identified in the Water Use Application QZ08-084.

The Carmacks deposit has been subject to some historical tonnage and grade estimations over the years as summarized in Table 3. The historic resources are presented here to show the progression of development of the resources over the years on the property.

Table 3. Historical Tonnage & Grade Estimates of the Carmacks Copper Deposit

Year	Source	Tonnes	Cu ox. %	Cu total %	Au g/t	Comments
1991	MPH Consulting Ltd.(No. 1 Zone)	13,212,743	0.90	1.05	-	Conventional by section 76% proven, 13% probable
1991	MPH Consulting Ltd.(No. 1 Zone)	13,212,743	0.88	1.00	-	IDS block model 78% proven, 10% probable
1993	Western Copper Audited by Kilborn	11,779,104	0.911	1.195	0.500	Measured and indicated at cutoff of 0.8% total copper
1993	Western Copper Audited by Kilborn	14,394,427	0.829	1.096	0.438	Measured and indicated at cutoff of 0.5% total copper
1993	Western Copper Audited by Kilborn	17,293,109	0.725	0.972	0.406	Measured and indicated at cutoff of 0.01% total copper
1997	Western est. Audit by Kilborn/SNC	12,065,557	-	0.97	-	Cutoff grade 0.29% total Cu Mine use 4.6:1 strip ratio
2007	Wardrop (No. 1, 4 and 7 Zones)	10,000,000	0.96	1.13	0.519	Oxide Resource, Measured and indicated at cutoff of 0.25% total copper

The mineral resource calculations prior to the 2007 calculation presented in Table 3 have not been classified in accordance with the CIM approved standards as required in NI 43-101. These estimates have been obtained from sources believed reliable and conform to disclosure standards in use at the time of their publication, but have not all been independently verified. The 2007 calculation was completed according to CIM approved standards and is 43-101 compliant.

6.0 GEOLOGY

6.1 REGIONAL GEOLOGY

The regional geology was described by Bostock in 1936 and more recently by Tempelman-Kluit in 1975, 1980 and 1985 and is shown in Figure 3. Much of the regional geology is taken from Cavey, et. al. (2006).

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.

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

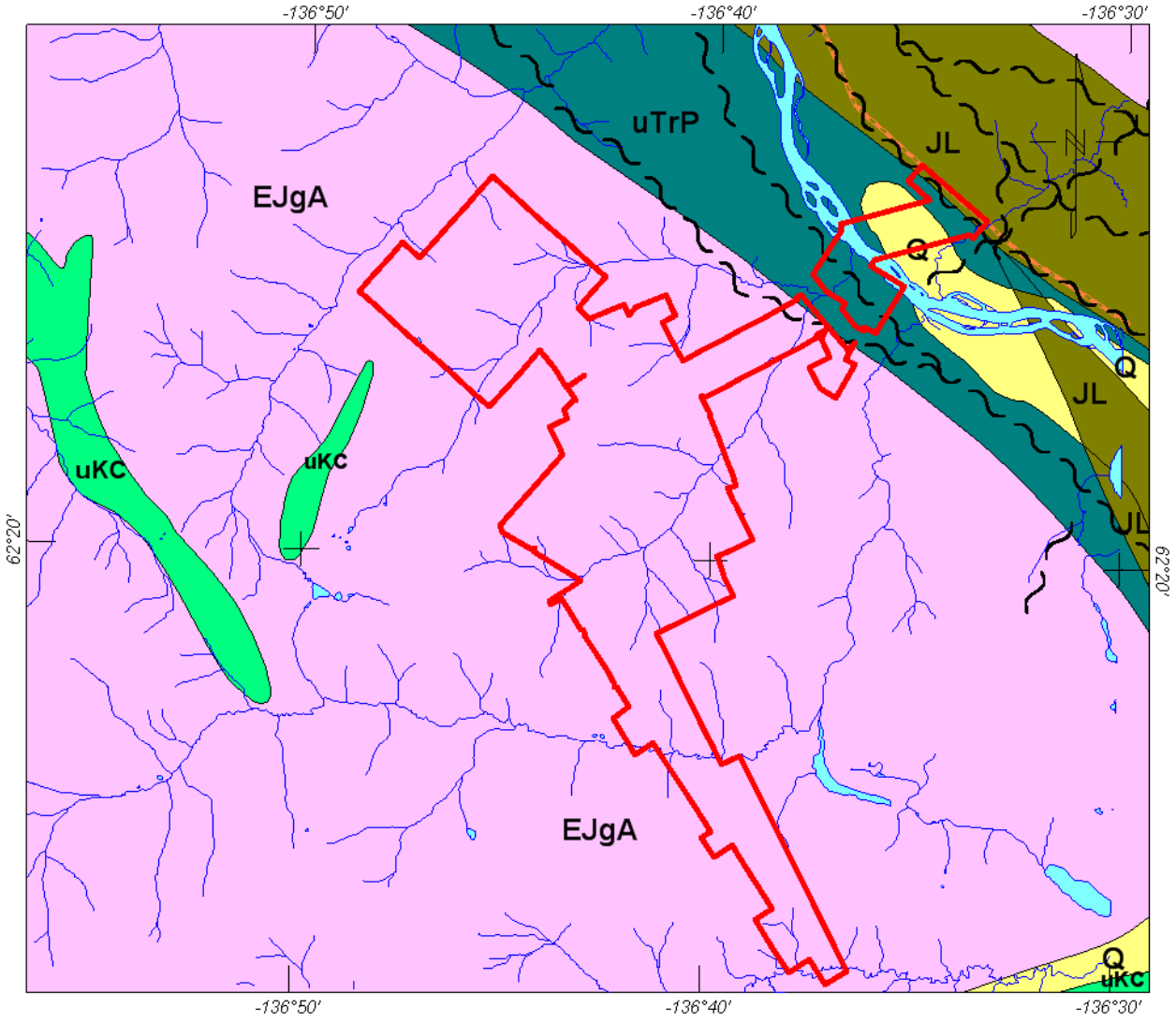
The Yukon Cataclastic Terrane includes hornblende-biotite-chlorite gneiss with interfoliated biotite granite gneiss, Permian Selwyn Gneiss, intruded by Early Jurassic Aishihik Suite Granite Mountain Batholith. Weakly foliated, mesocratic, biotite-hornblende, Granite Mountain granodiorite contains screens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

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

Mesozoic strata of the Whitehorse Trough are only exposed in fault contact with the Yukon Crystalline Terrane and Yukon 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.

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.



Q	Quaternary - sand and gravel
uKC	upper Cretaceous - Carmacks Group volcanics
JL	Jurassic - Labege Group sediments
EJgA	Early Jurassic - Aishihik Suite granodiorite
uTrP	upper Triassic - Povoas Group volcanics and sediments

2500 0 2500
(meters)

WESTERN COPPER CORPORATION
CARMACKS COPPER PROPERTY
REGIONAL GEOLOGY MAP
September 22, 2011
CASSELMAN GEOLOGICAL SERVICES Ltd.

6.2 PROPERTY BEDROCK GEOLOGY

The Carmacks Copper Property lies within the Yukon Cataclastic Terrane. The deposit is hosted by variably metamorphosed volcanic to volcano-sedimentary rocks that form roof pendants and large xenoliths within Jurassic hornblende-biotite granodiorite of the Granite Mountain (Aishihik Suite) Batholith (Figure 4).

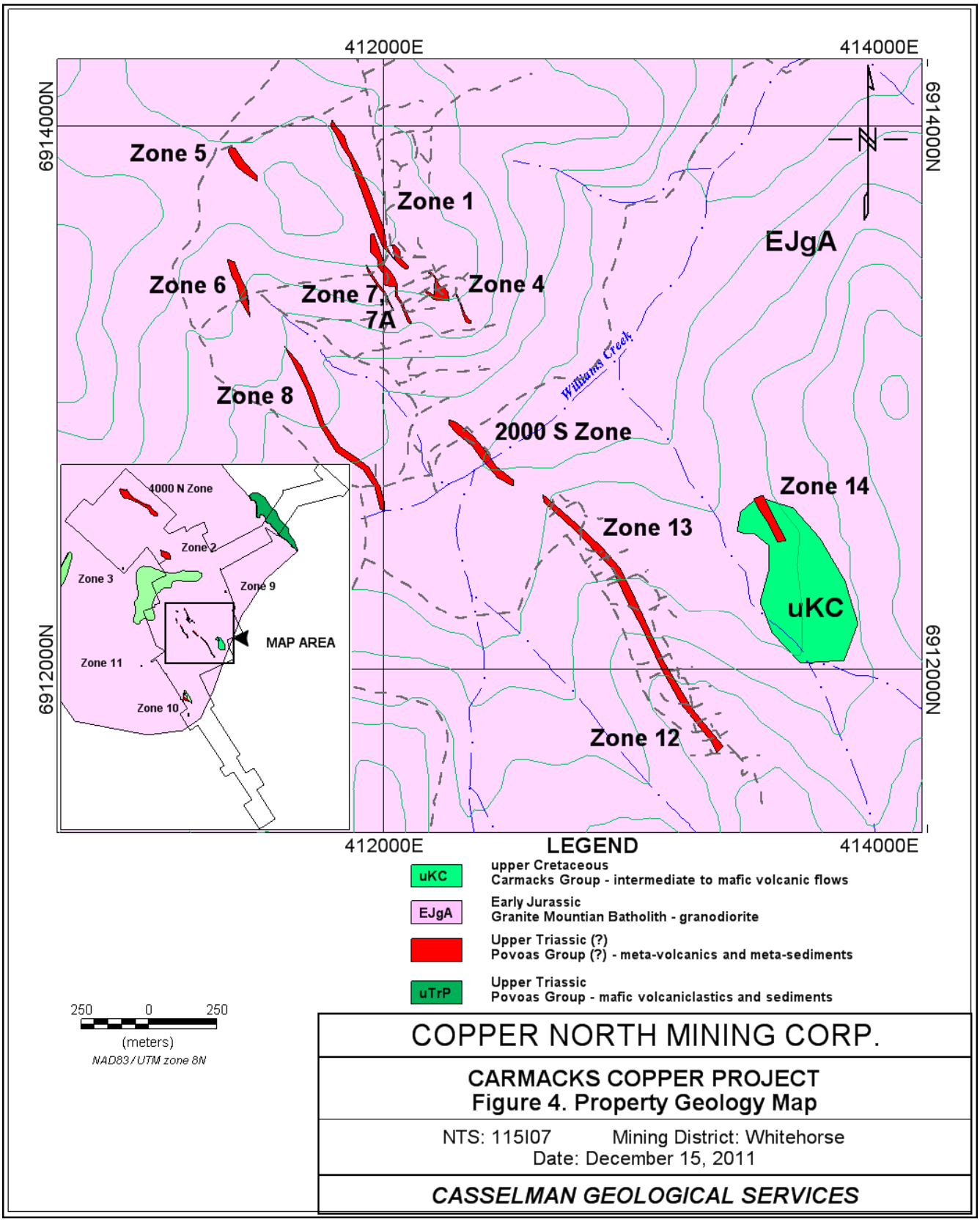
The host rocks generally lack or have very low quartz content and high mafic content. An andesitic to basaltic pyroclastic volcanic, probably tuffaceous, agglomerate or breccia precursor rock is considered the most likely source. In some locations, where the degree of metamorphism is low, arkosic sediments, believed to be derived from a basic volcanic or plutonic regime, are observed intermixed with the volcaniclastic rocks. The host rocks are believed to have originated from the upper Triassic Povoas Formation and have been consumed by later intrusions of the Granite Mountain Batholith. The host rocks vary from relatively unmetamorphosed; to slightly metamorphosed; to amphibolites gneiss; to completely re-crystallized hornblendite.

The Granite Mountain granodiorite is generally massive in appearance, medium to coarse grained to mega-crystic porphyritic. It occasionally develops a weak foliation near contacts with the meta-volcanic / gneiss units, where the flow has been restricted by the xenoliths. The granodiorite has been separated into five divisions; four based on quartz, biotite, hornblende and K-feldspar contents and a fifth based on assimilated gneiss.

Post mineralization aplite and pegmatites are common. They range in thickness from a few centimetres up to three metres. Quartz veins are uncommon and average two to ten centimetres 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.

There are 16 identified zones containing copper mineralization known on or in the immediate vicinity of the property (Figure 4). All of the estimated resources are contained in the No. 1, No. 4 and No. 7 zones which extend over a 1000 m strike length and at least 450 m down dip. The deposit is open at depth and is oxidized to 250 meters in depth. Copper-gold mineralization at Carmacks Copper is hosted primarily in the meta-volcanic rocks, although some copper oxide minerals have “bled out” into fractured zones in the adjacent granodiorite. In a few instances this “bleeding out” has occurred up to 50 m away from the meta-volcanic source.

The copper mineralization in the upper, oxidized, portion of the No.1 Zone consists of malachite>tenorite>azurite and lesser chalcopyrite, bornite, covelite and cuprite. The boundary between oxidized and un-oxidized mineralization is generally sharp and undulating at approximately 250 m vertically below surface. In the transition zone, chalcopyrite and bornite content increase. In the sulphide zone chalcopyrite is the dominant copper mineral. The deposit and encompassing granodiorite contains very little non-copper bearing sulphide minerals, including pyrite, which is estimate at less than 0.5%. Native copper is occasionally observed as dendritic fracture filling in the meta-volcanic and in the granodiorite.


412000E
414000E
6914000N
6912000N
Zone 5
Zone 1
Zone 6
Zone 7,
7A
Zone 4
EJgA
Williams Creek
Zone 8
2000 S Zone
Zone 14
Zone 13
uKC
Zone 12
4000 N Zone
Zone 2
Zone 3
Zone 9
MAP AREA
Zone 11
Zone 10
LEGEND
uKC
upper Cretaceous
Carmacks Group - intermediate to mafic volcanic flows
EJgA
Early Jurassic
Granite Mountian Batholith - granodiorite
Upper Triassic (?)
Povoas Group (?) - meta-volcanics and meta-sediments
uTrP
Upper Triassic
Povoas Group - mafic volcaniclastics and sediments
250 0 250
(meters)
NAD83 / UTM zone 8N
COPPER NORTH MINING CORP.
CARMACKS COPPER PROJECT
Figure 4. Property Geology Map
NTS: 115I07 Mining District: Whitehorse
Date: December 15, 2011
CASSELMAN GEOLOGICAL SERVICES

The character of the deposit changes along strike leading to a division into northern and southern halves. The northern half is more regular in thickness, dip angle, width and down dip characteristics. The southern half splays into irregular intercalations, terminating against sub-parallel faults down dip. Both the north and south ends of the deposit are offset by cross-cutting faults. The No. 7, 7A and 4 zones are interpreted as separate rafts or pull-aparts of a larger raft of meta-volcanic. Although, it is possible they are fault offsets of the No. 1 Zone.

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

6.3 SURFICIAL GEOLOGY

The surficial geology presented in Figure 5 was obtained in digital format from the Yukon Geological Survey website. The data was digitized from original work by Jackson (1997). This data is plotted on Figures 5 and 6.

The property is predominantly underlain by thin, till veneer (<1 m) of diamicton that is comprised of stony, silty sandy matrix (Unit **MvR**) of the Middle Pliestocene, Reid Glaciation. This unit covers the majority of north and south facing slopes and ridge tops throughout the region with the exception of the slopes between Merrice Creek and Williams Creek in the centre of the property. Here the unit **Cv** of the Pleistocene and Halocene dominates. Unit **Cv** is similar in composition to unit **MvR** and is comprised of diamicton, stony with a sandy matrix. It is <1 m thick and discontinuous over bedrock.

The smaller valley bottoms and creek beds are generally underlain by sands and gravels from glaciofluvial ice stagnation sediments of the Late Pleistocene, McConnell Glaciation (**Gx**) or Middle Pleistocene, Reid Glaciation (**GxR**), or alluvial complex sediments of the Mid-Pleistocene, Pre-McConnell period (**AxPM**). These are occasionally overlain by thick bog, fen and swamp deposits (**O**), particularly in the Merrice Creek Valley.

The Yukon River valley is underlain by sand, cobble, and silt from flood plain deposits in the Holocene, Post McConnell Glaciation (**Au, Ap, Af** and **At**). A small patch of well sorted eolian sands and dunes (**Eb**) is observed west of the property in the Merrice Creek Valley.

Table 4. Table of Surficial Geological Formations

Holocene – Post McConnell Glaciation

O **Bog, fen, and swamp deposits:** undivided; thickness <1 m to 10 m

Au **Alluvial sediments, undivided:** sediments forming floodplains, fans, and terraces as above that cannot be subdivided at this map scale

Ap **Floodplain sediments:** gravel, cobble to pebble; massive to thick bedded capped by sand and silt; flat lying; includes lacustrine and organic deposits in abandoned channels and back-swamp areas subject to periodic inundation and reworking by floods; thickness 1 to 5 m

Pleistocene and Holocene (Undivided)

Eb **Eolian sands:** sand, well sorted; massive; forms crescent-shape and linear dunes and features or gently undulating inter-dune eolian plains; thickness 1-5 m

Cv **Colluvial veneer sediments**: diamicton, stony with a sandy matrix; massive; thickness <1 m to discontinuous over bedrock

Late Pleistocene (Wisconsinan) - McConnell Glaciation

Gx **Glaciofluvial ice stagnation complex sediments:** gravel, sand, diamicton, poorly to moderately sorted, and minor silt and clay; bedding thick to massive and commonly folded and faulted from syndepositional ice melt out; surface consists of hummocks, kettles, esker and crevasse-fill ridges with minor elements of units **Gp**, **Gd**, and **Gt**

Mb **Till blanket:** diamicton, stony with a silty, sandy matrix; massive to crudely stratified; surface conforms to the underlying topography; thickness 1 to 5 m

Mid-Pleistocene - Pre-McConnell

AxPM **Alluvial complex sediments:** gravel and sand, poorly to moderately sorted; thin to thick bedded, interstratified with colluvial diamicton, reworked loess, peat, and woody detritus; sediments underlie the floors and margins of narrow upland valleys and grade laterally (upslope) into colluvial blankets. They contain segregated ice lenses and ice wedges and are normally capped by blanket bog; sediments may represent several depositional cycles; thickness may exceed 10 m in mid-valley locations.

Middle Pliestocene – Reid Glaciation

GpR **Glaciofluvial plain sediments:** gravel and sand, moderately to well sorted; thick bedded to massive; planar surface; thick 1 to 10 m or more.

GxR **Glaciofluvial ice stagnation complex sediments:** gravel, sand, diamicton, poorly to moderately sorted, and minor silt and clay; bedding thick to massive and commonly folded and faulted from syndepositional ice melt out; surface consists of hummocks, kettles, esker and crevasse-fill ridges.

MvR **Till veneer;** diamicton, stony, silty sandy matrix; massive; discontinuous and may contain extensive areas of thin (<1 m) and patchy colluvium and bedrock.

7.0 2011 EXPLORATION PROGRAM

The 2011 exploration program on the property consisted of surficial geological mapping, test pitting, collection of 10 sediment samples, one from each of the test pits, and repairs to the current access road.

The surficial mapping was conducted along the southern part of the property, covering approximately 4 sq. km. The test pitting was conducted on an esker discovered proximal to the proposed mine access road to characterize the material for suitability for road construction. Ten test pits were excavated and the profile was mapped and a representative sample of material from the pit was collected. The samples were nominally 2 kg and 0.5 kg subset of material was sent to the lab for analysis. Photos of each of the test pits is included below.

The repairs to the current access road involved repairs to a culvert at Williams Creek and were required to provide access to the camp area. A loader was used to haul approximately 7 cubic metres of coarse fill to cover the culvert and make the road passable.

8.0 GEOCHEMICAL ANALYTICAL PROCEDURE

The 10 sediment samples were delivered to the ALS Chemex Labs Ltd. in Whitehorse for processing and analysis. The samples were screened and a -180um pulp was analyzed at ALS Chemex in North Vancouver, BC. The analytical procedure involved analysis for 51 elements by Aqua Regia digestion and Inductively Coupled Plasma – Mass Spectrometry finish according to the ME-MS41 analytical package and for gold by Fire Assay and ICP-Atomic Emission Spectrometry (Au-ICP21). Geochemical Analytical Certificates are included in Appendix II, and sample locations are plotted on Figure 6.

9.0 RESULTS

The sediment samples did not return any significant copper values. The highest value is 45.2 ppm Cu from the sample from Test Pit #2. Other minerals of economic interest such as gold, silver, lead, zinc, and molybdenum are all at low levels. These values are considered background values and would indicate that there is no significant mineralization proximal the proposed road in this area.

The program was successful in delineating a large esker located in the southern part of the property and proximal to the proposed mine access road. The esker measures approximately 1,400 m north-south, by 250 m east-west. The small excavator dug 10 pits to test the esker. Five of the test pits reached bedrock at between 0.9 to 1.9 m, one test pit was abandoned in permafrost at 1.24 m (Test Pit #10) and four test pits bottomed in sand and gravel, having reached the depth limit of the excavator (between 1.9 and 2.5 m). The thickness of the sands and gravels is greater to the west and thinner to the east varying from over 2.5 m in Test Pit #7 to 0.9 m in Test Pit #5. The average thickness of the esker is estimated at 1.8 to 2.0 m.

The test pit results are:

Test Pit #1



0.0 m	
	organics
0.10 m	
	White River ash
0.15 m	
	gravelly sand
1.50 m	
	regolith–decomposed granite
1.90 m	bedrock

Test Pit #1 reached decomposed bedrock at 1.50 m and ended in bedrock at 1.90 m.

Test Pit #2



0.0 m	organics
0.05 m	White River ash
0.15 m	
	well sorted cobbles and sand with cobbles to 20 cm and well rounded.
1.90 m	limit of excavator

Test Pit #3



0.0 m	
0.05 m	organics
0.15 m	White River ash
	Clean, well sorted sand
2.30 m	limit of excavator

Test Pit #4



0.0 m	
0.10 m	organics
	White River ash
0.38 m	
	decomposed granite
1.10 m	bedrock

Test Pit #5



0.0 m	
0.10 m	organics
	White River ash
0.23 m	
	decomposed granite
0.90 m	bedrock

Test Pit #6



0.0 m	
0.08 m	organics
	White River ash
0.18 m	
	washed gravel and sand
1.18 m	
	decomposed granite
1.40 m	bedrock

Test Pit #7



0.0 m	
0.10 m	organics
0.28 m	White River ash
	sand and 10% cobbles
2.30 m	
	sand and 30% cobbles
2.50 m	limit of excavator

Test Pit #8



0.0 m	
0.05 m	organics
	White River ash
0.20 m	
	granitic sand
0.90 m	
	decomposed granite
1.40 m	bedrock, water in bottom of hole

Test Pit #9



0.0 m	
0.05 m	organics
	White River ash
0.28 m	
	Sand with 15 to 20% gravel and cobble-sized material (cobbles to 10 cm and rounded)
2.20 m	limit of excavator

Test Pit #10



0.0 m	
0.06 m	organics
0.16 m	White River ash
	washed sand
1.04 m	permafrost

An excellent source of gravel and cobble bearing aggregate was identified at the site labelled “photo” on Figure 6 (see photo below). This photo was taken from the road cut as originally proposed. The area was not test pitted as there was not sufficient time.



Photo 1. Gravel- and cobble-bearing esker on the proposed Carmacks Mine access road

10.0 CONCLUSIONS AND RECOMENDATIONS

The discovery and delineation of the esker in the southern part of the property and proximal to the proposed mine access road is significant and will prove valuable for road construction aggregate. The bulk of the material in the esker is washed sand, but patches of gravel and cobble-rich sand in significant quantities are present.

It is recommended that the road alignment be adjusted slightly to the east to take advantage of the slightly raised and drier ground conditions along the western margin of the esker, as indicated on Figure 6. This would also have the advantage of locating the roadway closer to the source of aggregate reducing trucking distances and costs for the sections of the road that will require fill.

Recommendations for future exploration work on the property are to complete the detailed surficial mapping program over the remainder of the proposed access road. In particular, the section from Merrice Creek to Williams Creek should be mapped to determine the most readily available sources of aggregate. It is estimate that a 10 day exploration program at a cost of $40,000 should be sufficient to complete this proposal.

Respectfully submitted

Scott Casselman, BSc., P.Geo.

STATEMENT OF EXPENDITURES

Contracting Costs

Casselman Geological Services Ltd

Scott Casselman	71 hours @ $ 75 / hour	$ 5,325.00
Truck rental	6 days @ $150 /day	900.00
Tools and Phone rental	5 days @ $100 /day	500.00
Satellite phone rental	6 days @ $15 /day	90.00

Kluane Drilling Ltd

Operator hours	12 hours @ $ 50 /hour	600.00
Kubota Excavator	20 hours @ $120 /m	2,400.00
920 loader	14 hours @ $130 /day	1,820.00
ATV Rental	5 days @ $100/day	500.00
Transport equipment to and from site		2,960.00

Analytical costs 10 samples @ $/ sample	486.54
Groceries	316.35
Diesel, Gas, Propane	298.58
Report Writing and reproduction costs	3,000.00
Total	**$19,196.47**

12.0 REFERENCES

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APPENDIX I

STATEMENT OF QUALIFICATIONS

Statement of Qualifications

I, Scott Casselman, P. Geo., certify that:

1. I am a geologist employed by Casselman Geological Services Ltd. and reside at 33 Firth Road, Whitehorse, Yukon Territory, Y1A 4R5.

2. I graduated from Carleton University in Ottawa, Ontario with a Bachelor of Science Degree in Geology in 1985 and have worked as a geologist since that time.

3. I am a member of the Association of Professional Engineers and Geoscientists of British Columbia, Registration No. 20032.

4. I supervised the 2011 exploration program on the Carmacks Copper Project in Central Yukon, for Copper North Mining Corp. described in this report.

Dated this ___day of _____, 20121, at Whitehorse, Yukon Territory.

Scott G. Casselman, BSc., P.Geo.

Appendix II

GEOCHEMICAL ANALYTICAL CERTIFICATES

ALS Canada Ltd.
2103 Dollarton Hwy
North Vancouver BC V7H 0A7
Phone: 604 984 0221 Fax: 604 984 0218 www.alsglobal.com

To: WESTERN COPPER CORPORATION LTD.
2050 - 1111 WEST GEORGIA STREET
VANCOUVER BC V6E 4M3

Finalized Date: 28- OCT- 2011
Account: WESCOP

CERTIFICATE WH11188314

Project: Carmacks

P.O. No.: 11- 01

This report is for 10 Soil samples submitted to our lab in Whitehorse, YT, Canada on 16- SEP- 2011.

The following have access to data associated with this certificate:

SCOTT CASSELMAN	WESTERN COPPER CORPORATION	

SAMPLE PREPARATION

ALS CODE	DESCRIPTION
WEI- 21	Received Sample Weight
LOG- 22	Sample login - Rcd w/o BarCode
SCR- 41	Screen to - 180um and save both

ANALYTICAL PROCEDURES

ALS CODE	DESCRIPTION	INSTRUMENT
Au- ICP21	Au 30g FA ICP- AES Finish	ICP- AES
ME- MS41	51 anal. aqua regia ICPMS	

To: WESTERN COPPER CORPORATION LTD.
ATTN: SCOTT CASSELMAN
33 FIRTH ROAD
WHITEHORSE YT Y1A 4R5

This is the Final Report and supersedes any preliminary report with this certificate number. Results apply to samples as submitted. All pages of this report have been checked and approved for release.

Signature:
Colin Ramshaw, Vancouver Laboratory Manager

ALS Canada Ltd.
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North Vancouver BC V7H 0A7
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Finalized Date: 28- OCT- 2011
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Project: Carmacks

CERTIFICATE OF ANALYSIS WH11188314

Sample Description	Method Analyte Units LOR	WEI- 21 Recvd Wt. kg 0.02	Au- ICP21 Au ppm 0.001	ME- MS41 Ag ppm 0.01	ME- MS41 Al % 0.01	ME- MS41 As ppm 0.1	ME- MS41 Au ppm 0.2	ME- MS41 B ppm 10	ME- MS41 Ba ppm 10	ME- MS41 Be ppm 0.05	ME- MS41 Bi ppm 0.01	ME- MS41 Ca % 0.01	ME- MS41 Cd ppm 0.01	ME- MS41 Ce ppm 0.02	ME- MS41 Co ppm 0.1	ME- MS41 Cr ppm 1
Test Pit #1		0.42	0.002	0.03	1.62	2.4	<0.2	<10	950	0.91	0.02	1.21	0.09	78.3	7.6	6
Test Pit #2		0.44	0.008	0.10	1.62	9.8	<0.2	<10	270	0.52	0.08	0.81	0.19	36.1	14.7	33
Test Pit #3		0.34	0.012	0.07	0.92	8.8	<0.2	<10	130	0.39	0.07	0.59	0.15	21.3	8.9	25
Test Pit #4		0.48	0.001	0.03	2.89	3.8	<0.2	<10	240	0.62	0.02	1.49	0.04	58.1	10.0	14
Test Pit #5		0.36	0.006	0.04	2.72	6.3	<0.2	<10	280	0.54	0.08	0.97	0.05	30.3	9.9	22
Test Pit #6		0.48	0.002	0.03	1.53	3.9	<0.2	<10	630	0.83	0.02	5.56	0.08	49.7	10.9	12
Test Pit #7		0.46	0.004	0.07	0.95	9.5	<0.2	<10	270	0.35	0.08	1.92	0.22	23.0	11.0	26
Test Pit #8		0.40	0.002	0.04	0.82	7.7	<0.2	<10	110	0.33	0.06	0.47	0.04	17.95	7.4	23
Test Pit #9		0.46	0.008	0.04	0.95	7.3	<0.2	<10	150	0.37	0.06	0.37	0.08	23.3	8.7	31
Test Pit #10		0.34	0.005	0.05	2.18	3.1	<0.2	<10	420	0.65	0.04	1.11	0.03	32.1	10.9	16

***** See Appendix Page for comments regarding this certificate *****

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VANCOUVER BC V6E 4M3

Finalized Date: 28- OCT- 2011
Account: WESCOP

Project: Carmacks

CERTIFICATE OF ANALYSIS WH11188314

Sample Description	Method	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41
	Analyte	Cs	Cu	Fe	Ga	Ge	Hf	Hg	In	K	La	Li	Mg	Mn	Mo	Na
	Units	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	%	ppm	ppm	%
	LOR	0.05	0.2	0.01	0.05	0.05	0.02	0.01	0.005	0.01	0.2	0.1	0.01	5	0.05	0.01
Test Pit #1		2.95	9.0	2.68	5.13	0.14	0.07	0.02	0.035	0.26	36.4	6.2	0.44	728	0.32	0.01
Test Pit #2		1.95	45.2	3.66	6.26	0.15	0.19	0.04	0.029	0.17	17.7	10.9	0.78	1090	1.13	0.04
Test Pit #3		0.65	31.4	2.57	3.35	0.13	0.26	0.03	0.018	0.07	10.7	5.3	0.50	539	0.68	0.03
Test Pit #4		0.36	5.9	3.51	11.80	0.20	0.07	0.01	0.028	0.27	20.5	9.4	1.05	746	0.32	0.03
Test Pit #5		0.48	16.3	3.29	10.25	0.11	0.03	<0.01	0.027	0.13	13.5	9.6	0.79	560	0.61	0.02
Test Pit #6		2.01	8.0	3.87	7.07	0.13	0.05	0.01	0.057	0.28	28.1	4.6	0.43	1060	0.39	0.01
Test Pit #7		0.91	31.4	2.71	3.75	0.10	0.20	0.06	0.021	0.10	11.8	6.4	0.49	571	0.88	0.03
Test Pit #8		0.35	20.5	2.10	3.01	0.08	0.10	0.06	0.017	0.04	10.4	4.3	0.36	351	0.47	0.02
Test Pit #9		0.57	22.5	2.66	3.20	0.09	0.18	0.01	0.016	0.07	12.6	4.9	0.36	374	0.62	0.02
Test Pit #10		0.34	17.1	3.78	10.10	0.10	0.06	0.03	0.042	0.21	17.1	6.5	0.77	737	0.35	0.02

***** See Appendix Page for comments regarding this certificate *****

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VANCOUVER BC V6E 4M3

Finalized Date: 28- OCT- 2011
Account: WESCOP

Project: Carmacks

CERTIFICATE OF ANALYSIS WH11188314

Sample Description	Method	ME- MS41	ME- MS41	ME- MS41	ME- MS41	ME- MS41	ME- MS41	ME- MS41	ME- MS41	ME- MS41	ME- MS41	ME- MS41	ME- MS41	ME- MS41	ME- MS41	ME- MS41
	Analyte	Nb	Ni	P	Pb	Rb	Re	S	Sb	Sc	Se	Sn	Sr	Ta	Te	Th
	Units	ppm	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
	LOR	0.05	0.2	10	0.2	0.1	0.001	0.01	0.05	0.1	0.2	0.2	0.2	0.01	0.01	0.2
Test Pit #1		0.12	4.2	2070	11.5	13.7	<0.001	<0.01	0.58	6.4	0.5	0.4	33.4	0.01	<0.01	4.1
Test Pit #2		0.35	26.8	1260	10.0	20.2	<0.001	<0.01	0.40	8.0	0.7	0.6	77.0	<0.01	0.02	4.7
Test Pit #3		0.25	23.4	760	6.6	5.5	<0.001	<0.01	0.34	5.5	0.5	0.3	57.5	<0.01	0.02	2.8
Test Pit #4		0.51	8.5	2480	4.4	12.9	<0.001	<0.01	0.12	5.4	0.4	0.7	75.2	<0.01	0.01	7.8
Test Pit #5		1.07	13.0	1380	6.4	10.7	<0.001	<0.01	0.19	4.5	0.4	0.7	56.1	<0.01	0.02	3.6
Test Pit #6		0.22	8.9	2410	8.2	17.7	<0.001	0.02	0.69	10.1	0.7	0.8	61.7	0.01	0.02	3.9
Test Pit #7		0.34	24.6	930	7.3	7.6	<0.001	0.02	0.45	6.0	0.4	0.4	75.9	<0.01	0.02	3.3
Test Pit #8		0.40	17.0	580	4.9	3.2	<0.001	0.01	0.30	5.2	0.3	0.3	47.1	<0.01	0.02	2.5
Test Pit #9		0.37	21.8	600	5.2	6.8	<0.001	0.01	0.28	5.2	0.4	0.3	33.6	<0.01	0.02	4.7
Test Pit #10		0.25	10.2	1960	7.0	13.6	<0.001	0.01	0.14	8.6	0.4	0.8	38.3	0.01	0.01	4.2

***** See Appendix Page for comments regarding this certificate *****

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CERTIFICATE OF ANALYSIS WH11188314

Sample Description	Method	ME- MS41	ME- MS41	ME- MS41	ME- MS41	ME- MS41	ME- MS41	ME- MS41	ME- MS41
	Analyte	Ti	Tl	U	V	W	Y	Zn	Zr
	Units	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm
	LOR	0.005	0.02	0.05	1	0.05	0.05	2	0.5
Test Pit #1		0.012	0.10	0.20	40	0.07	16.85	68	1.6
Test Pit #2		0.115	0.29	0.86	82	0.15	14.90	76	8.8
Test Pit #3		0.070	0.10	0.63	60	0.10	12.20	49	10.9
Test Pit #4		0.197	0.05	0.30	80	0.14	6.80	90	2.1
Test Pit #5		0.153	0.06	0.34	79	0.21	5.02	70	1.4
Test Pit #6		0.053	0.10	0.25	86	0.07	21.7	86	0.5
Test Pit #7		0.066	0.18	0.59	64	0.12	9.27	56	8.8
Test Pit #8		0.057	0.05	0.52	53	0.11	8.85	36	5.0
Test Pit #9		0.064	0.09	0.70	68	0.12	7.86	38	8.1
Test Pit #10		0.035	0.07	0.38	89	0.26	13.50	86	1.9

***** See Appendix Page for comments regarding this certificate *****

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VANCOUVER BC V6E 4M3

Finalized Date: 28- OCT- 2011
Account: WESCOP

Project: Carmacks

CERTIFICATE OF ANALYSIS WH11188314

Method	CERTIFICATE COMMENTS
ME- MS41	Gold determinations by this method are semi- quantitative due to the small sample weight used (0.5g).

-136°40'

62°20'

Nancy Lee Creek

Yukon River

Klondike Highway

Williams Creek

Merrice Creek

Merrice Lake

Freegold Road

Carmacks Copper Claims
Carmacks Copper Leases
Claims owned by others

Existing roads/trails
Proposed mine access road

0 2.0 km

N

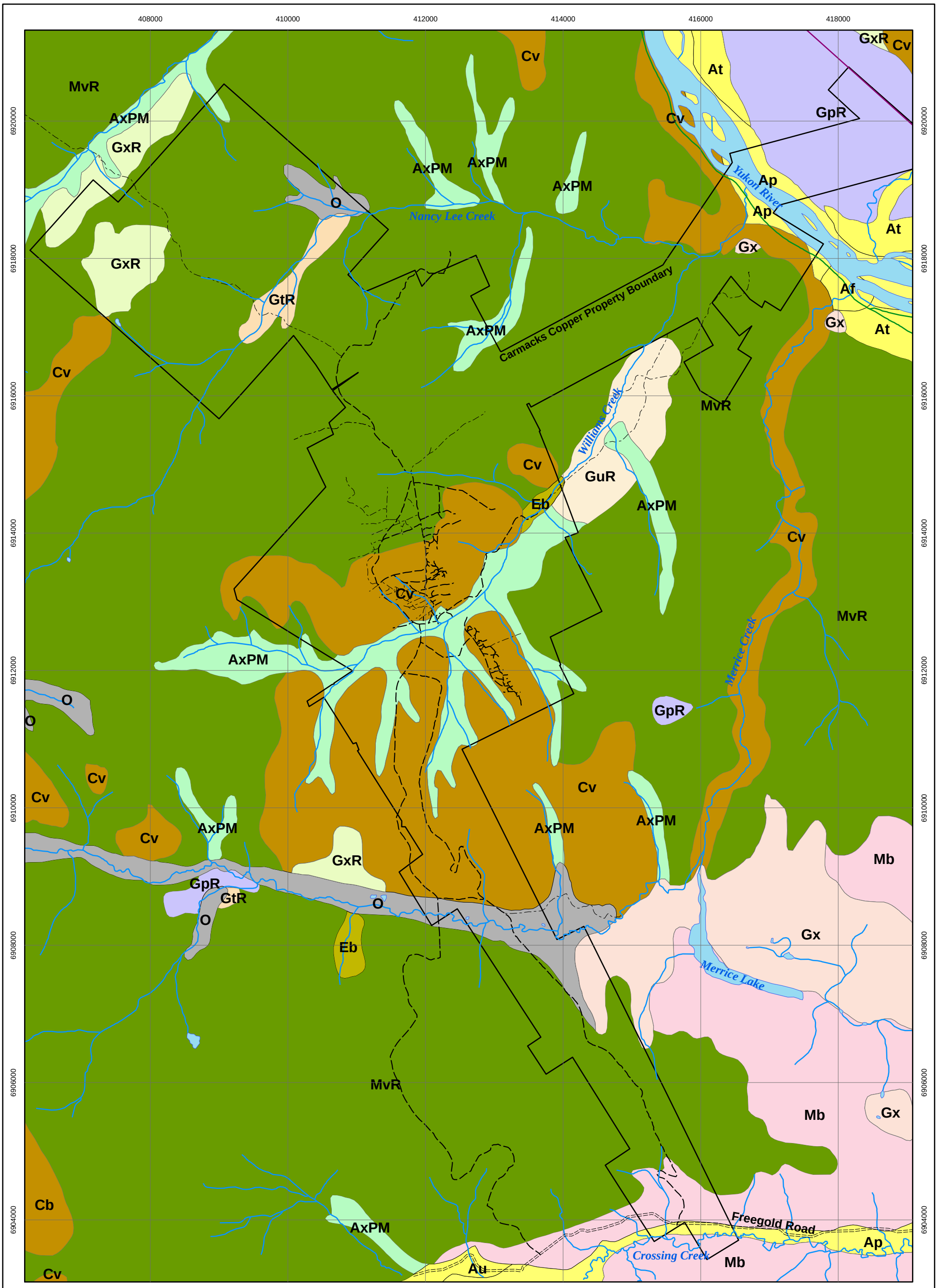
-136°40'

COPPER NORTH MINING CORP.

CARMACKS COPPER PROJECT
Figure 2. CLAIM LOCATION MAP

NTS: 115I07 Mining District: Whitehorse
Projection: UTM, zone 8 Datum: NAD83
Date: January 5, 2012

CASSELMAN GEOLOGICAL SERVICES LTD



Holocene - Post McConnell Glaciation

- O — Bog, fen, and swamp deposits
- Au — Alluvial sediments, undivided
- Ap — Floodplain sediments

Pleistocene and Holocene (Undivided)

- Eb — Eolian sands
- Cv — Colluvial veneer sediments

Late Pleistocene (Wisconsian) - McConnell Glaciation

- Gx — Glaciofluvial ice stagnation complex sediments
- Mb — Till blanket

Mid-Pleistocene - Pre-McConnell

- AxPM — Alluvial complex sediments

Middle Pleistocene - Reid Glaciation

- GpR — Glaciofluvial plain sediments
- GxR — Glaciofluvial ice stagnation complex sediments
- MvR — Till veneer

COPPER NORTH MINING CORP.
CARMACKS COPPER PROPERTY

Figure 5. Property Surficial Geology

NTS 115I07 Mining District: Whitehorse
Datum: NAD83 Projection: UTM, Zone 8
December 13, 2011

Casselman Geological Services Ltd.

414000 416000

6908000 6908000

6906000 6906000

6904000 6904000

Merrice Creek

O

Gx

Mb

MvR

Ap

Test Pit 10
Test Pit 9
Test Pit 8
Test Pit 7
Test Pit 6
Test Pit 5
Test Pit 4
Test Pit 3
Photo
Test Pit 2
Test Pit 1

Freegold Road

Crossing Creek

YB96630 YB96628 YB96626 YB96996 YB96995 YB97251 YB96994 YB96622 YB96620 YB96993 YB96992 YB96991 YB96990 YB96989 YB96988 YB96987 YB96986

YC66871 YC66861 YC66860 YC66859 YC66858 YC66857 YC66856 YC66855 YC66854 YC66853 YC66852 YC66851 YC66850 YC66849 YC66848 YC66847 YC66846 YC66845 YC66844 YC66870

YC60304 YC60305 YC60306 YC60307 YC60308 YC60309 YC60310 YC60311 YC60312 YC60313 YC60314 YC60315 YC60316 YC60317 YC60318 YC60319 YC60321 YC60323 YC60325 YC60327

YC60280 YC60281 YC60282 YC60283 YC60284 YC60285 YC60286 YC60287 YC60288 YC60289 YC60291 YC60293 YC60294 YC60320 YC60322 YC54032 YC54031

Test pit

Measured Esker outline

Mine Access Road

Proposed Access Road Adjustment

O	Bog, fen, and swamp deposits
Ap	Floodplain sediments
Gx	Glaciofluvial ice stagnation complex sediments
Mb	Till blanket
MvR	Till Veneer

COPPER NORTH MINING CORP.
CARMACKS COPPER PROPERTY

Figure 6. 2001 Work Program

NTS 115I07 Mining District: Whitehorse

Datum: NAD83 Projection: UTM, Zone 8

December 13, 2011

Casselman Geological Services Ltd.



2011 ANNUAL QUARTZ MINING LICENCE REPORT

SUBMITTED TO YUKON GOVERNMENT, ENERGY MINES AND RESOURCES

YUKON QUARTZ MINING LICENCE QML-0007

Appendix C

Annual Engineer's Inspection Report, June 2011

August 23, 2011

Project No. 07-1413-0077
Doc. No. 162 Rev. 0

Mr. Paul West-Sells
President & Chief Operating Officer
Western Copper Corporation
2060 - 1111 West Georgia Street
Vancouver, BC
V6E 4M3

ANNUAL INSPECTION, JUNE 29, 2011
CARMACKS COPPER PROJECT, CARMACKS, YUKON

Dear Mr. West-Sells,

Golder Associates Ltd. (Golder) completed an inspection of the Carmacks Copper project site for Western Copper Corporation on June 29, 2011. The inspection of the proposed future site of the Carmacks Copper mine was completed as part of the requirement of the Quartz Mining License (QML – 0007) for an annual inspection. The inspection was to evaluate the condition and stability of the existing facilities in the area of the proposed new mine and the proposed facilities which will include a heap leach pad, an open pit mine area, a waste rock storage area, a processing plant and related facilities, ore preparation facilities, ore stockpiles, any water diversion structures and/or other related operations or facilities. The inspection was limited in that there are no structures on site at present with the exception of the mine exploration camp and there are no current activities currently at the site. An inspection was however, completed of the entire project area.

1.0 INSPECTION

The inspection by Golder was completed by Ms. Fiona Esford along with Mr. Scott Casselman – a representative of Western Copper Corporation. The inspection focused on the existing site conditions and the limited site infrastructure. Photographs of the site at the time of the inspection are presented in Appendix A.

The project is in the advanced exploration stage and final permitting for the proposed future mine. As noted the only infrastructure on site, at present, is the exploration camp and a series of access roads to the proposed open pit mine area and other areas of the property. The access roads were developed to provide access for exploration activities.

The inspection of the camp area indicated the camp is well maintained and there is no erosion of sediments from the pad area into the surrounding natural area. The slope behind or west of the camp is stable and is not impacting or causing any safety issues with the camp structures or operations. There is minor slumping of small

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sections of the slope, but these are not impacting camp safety nor would they represent an issue to workers on the project.

In addition to the camp, the inspection included the area proposed for the heap leach facility, events pond, and the heap leach sediment pond. This portion of the site also includes the area planned for the process plant west of the heap leach embankment. None of these structures have been developed and site preparation in these areas has been limited to the clearing of trees and organic soils (complete in 1997-1998), development of a series of access roads, and drilling platforms established as part of the exploration programs. Since the initial clearing, vegetation re-growth has been occurring. Erosion and sediment control measures in this area were initially put into place in September 2008 and further maintained in September 2009 and August 2010. These include a series of ditches and berms to divert water into vegetated areas and to break up flow to reduce the potential for erosion, sediment catch basins, and silt fences. The inspection indicated that there has been some erosion of sediments along the access roads and that the sediment basins are trapping and containing the sediment adequately. Additional capacity remains within these catch basins.

Further down slope of the proposed HLF sediment pond, and within the floodplain of Williams Creek, a series of silt fences have been installed and the area seeded, where sediment from previous site erosion has accumulated. At the time of the inspection, it was observed that the silt fencing surrounding these areas is working effectively and there is no evidence of recent sediment movement. Grass seed used in 2009 to promote the re-establishment of vegetation and further stabilize the sediment has generally been effective. However, re-growth in one particular area has not progressed well as it seems water is periodically pooling thereby impacting growth. It is recommended that either small ditches be made to promote drainage from the area into other vegetated areas, or alternatively, the area is re-vegetated with woody plants (*e.g.* willow cuttings) that are better able to tolerate saturated soil conditions. No movement of sediment beyond the existing silt fences was observed, nor does it appear that sediment has entered Williams Creek. These erosion control measures are of a temporary nature that should be inspected annually and ongoing minor maintenance activities should be anticipated.

The area where the open pit is to be developed was inspected. The excavation slopes of the trenches developed as part of the effort to obtain bulk samples during exploration activities were observed to be in reasonable condition and there was no observed slumping of the excavation slopes or failures of these slopes. There were no visual signs of erosion observed in the area of the proposed open pit. Several of the closed drill pads were inspected and there did not appear to be any erosion noted from these areas that require attention.

The area of the proposed waste rock storage area and the present access road crossing at North Williams Creek was inspected. The waste rock storage area is still tree covered and the drill pads in the area of the proposed waste rock storage area did not have any signs of sediment movement into the surrounding area. The small sediment catch basins at the drill pads still have capacity to manage more sediment, if required. Small erosion of the access road which crosses the waste rock storage area was observed. Sediment from the access road is being naturally diverted to adjacent vegetated areas as a result of road grades and was not observed to be entering any waterways.

Minor erosion along the ditches of the access road on the north side of North Williams Creek was observed. A series of diversion ditches were created along the access road to divert water and sediment from the road into adjacent vegetated areas in 2010. These diversions appear to be working effectively; however, it is recommended that one additional ditch be added on the west side of the road further down slope, closer to

North Williams Creek. In addition the silt fence on the bank of North Williams Creek has deteriorated and therefore should be re-installed. Annual inspection and the requirement for minor maintenance of these sediment control measures should be anticipated.

The process plant and the crusher / truck shop areas were also inspected. The process plant site is partially cleared and the crusher / truck shop site is still tree covered. There was no apparent erosion along any of the access roads in these two areas or at the weather station located just to the west of the crusher / truck shop area.

The new camp site was inspected and it is still tree covered. The site adjacent to the camp where the new water wells were installed was also inspected and there is no apparent movement of sediment beyond the work pads around the wells.

The site access road crossing of Williams Creek has a single culvert. The sand bags that were previously placed surrounding the culvert entrance and exit have now deteriorated. We would recommend re-establishing the support of the fill material adjacent to the culvert. In addition, it appears that flow has passed over two segments of the road near the crossing. It is anticipated that this was a temporary condition, which likely occurred at the beginning of the freshet, potentially when the culvert was still frozen/blocked by snow. Some erosion of the roadbed material has occurred. At some point in the future if additional activities are to occur at the site, then upgrading the crossing would be recommended. In the interim, given the lack of activity at the site, it is recommended to re-establish the bank protection on the upstream and downstream side of the culvert and extend the protection to the location where flow has occurred across the access road. Further, a coarse gravel material should be placed across the road at the low points (where erosion occurred) so that future flows across the road do not result in new erosion of the road embankment.

The general site development has not started yet. Therefore, there are no stability concerns associated with the undeveloped facilities and no maintenance required. No permanent water diversion structures are in place. There are however, temporary water management diversion structures (*i.e.*, ditches and sediment catch basins) in place that are appropriate for the exploration stage of the property. It is recommended that they continue to be inspected annually and that ongoing maintenance be conducted, as deemed necessary.

2.0 ADDITIONAL HEAP LEACH GEOCHEMISTRY TEST RESULTS

As part of the annual review, we summarize the ongoing geochemical testing of the leached ore obtained from columns using typical ore samples from the site.

Two samples of material were obtained from a large 8 m high column of leached, rinsed, and neutralized ore that was dismantled in the summer of 2010. The testing program includes: ABA, SFE, whole rock analysis, humidity cell testing, and residue testing following completion of the humidity cell testing. This work began in August 2010 and is still ongoing. The humidity cell testing is approaching week 56 and sample results appear consistent. It is anticipated that at the completion of week 56 this testing program will be terminated and the completion testing conducted. A report presenting the results of the testing will be completed at that time.

3.0 RECOMMENDED ACTIONS

The inspection of the proposed mine site for the Carmacks Copper was completed on June 29, 2011 and indicated that as the site development has not been started yet, there is limited infrastructure and limited requirements for maintenance or further investigations. It is understood that the recommended maintenance work will be carried out by Western Copper Corporation prior to the onset of winter conditions at the site.

We trust that this letter satisfies your requirements. If you require additional information, please do not hesitate to contact us.

Yours very truly,

GOLDER ASSOCIATES LTD.

ORIGINAL SIGNED AND SEALED

Fiona Esford, P.Eng.
Senior Geotechnical Engineer

ORIGINAL SIGNED

John Hull, P.Eng. (BC, NWT, NU, YK)
Principal

FE/JAH/fe/aw/cf

Attachments: Appendix A – Photographs

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APPENDIX A
Photographs











Photographs 1 - 5: *Carmacks Exploration Camp Infrastructure, which includes an office, core shack, equipment shed, temporary trailers for accommodations and kitchen facilities, and drill core storage area.*

Date Taken: *June 29, 2011*





Photographs 6 - 7: *Drill pad access road located north of Williams Creek and downstream of the area of the proposed heap leach facility sediment pond. Photographs show vegetation re-growth which has occurred and is assisting in stabilizing previously transported sediments.*

Date Taken: *June 29, 2011*







Photographs 8 - 10: *Silt fences located between the area of the proposed heap leach facility sediment pond and Williams Creek and area re-vegetated area. Re-establishment of grass vegetation in this area appears to be hampered by the periodic pooling of water. Therefore planting of woody vegetation (e.g. willow cuttings) which are more tolerant to saturated soil conditions is recommended.*

Date Taken: *June 29, 2011*



Photograph 11: View of proposed heap leach facility and plant site area, taken from area of proposed open pit, looking southwest.

Date Taken: June 29, 2011



Photograph 12: View of proposed heap leach facility area taken from near the proposed plant area.

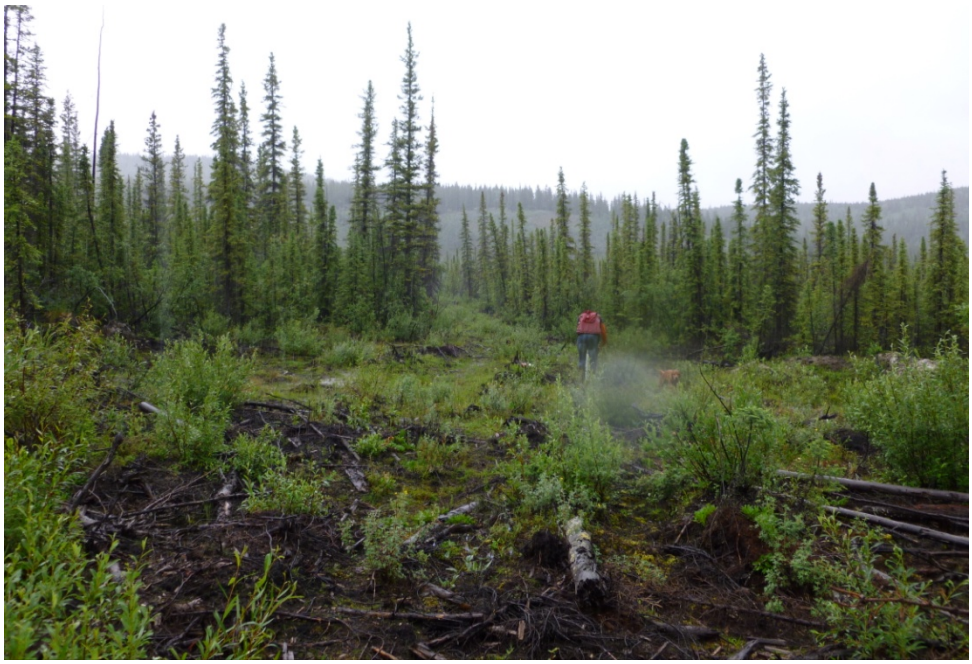
Date Taken: June 29, 2011

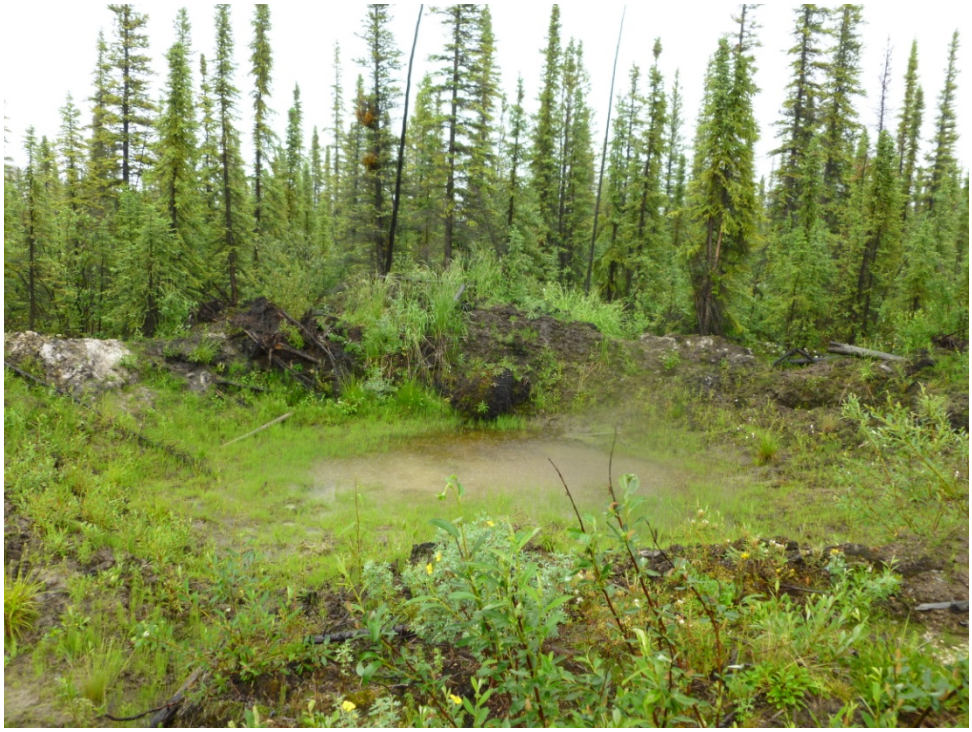




Photographs 13 - 14: *Access roads within the proposed waste rock storage area.*

Date Taken: *June 29, 2011*





Photograph 15 (top): *Access road within the proposed waste rock storage area.*

Photograph 16 (bottom): *Sediment basin trap near drill pad in proposed waste rock storage area.*

Date Taken: *June 29, 2011*







Photographs 17 - 19: *View of culvert and access road crossing Williams Creek. Note sand bags used for embankment support on upstream and downstream side of the culvert which have deteriorated.*

Date Taken: *June 29, 2011.*



Photographs 20 - 21: *Access road near Williams Creek crossing where water appears to have flowed across and where some road bed material has eroded.*

Date Taken: *June 29, 2011*

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COPPER NORTH
MINING CORP.

2011 ANNUAL QUARTZ MINING LICENCE REPORT

SUBMITTED TO YUKON GOVERNMENT, ENERGY MINES AND RESOURCES

YUKON QUARTZ MINING LICENCE QML-0007

Appendix D

Reclamation Work at Carmacks Copper Project, September 2011

CASSELMAN GEOLOGICAL SERVICES Ltd

33 Firth Road
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September 19, 2011

Paul West-Sells
President
Western Copper Corporation
2050-1111 West Georgia Street
Vancouver, BC
V6E 4M3

Dear Paul,

Re. Reclammation work at Carmacks Copper Project

From September 12 to 16, 2011, Al Serafinchon of Kluane Drilling Ltd and I conducted exploration and reclamation work at the Carmacks Copper Project. The exploration work was conducted on Sept 13 and 15, the reclamation work on Sept 14. The reclamation work was conducted to address 3 issues identified by Fiona Esford of Golder Associates, during her annual inspection of the site. The 3 issues she identified were:

1 – The culvert at Williams Creek on the exploration access road required rip-rap to prevent erosion of the road bed during spring thaw
2 – Diversion of silt from erosion on the road at North Williams Creek away from the pond on the upstream side of the road
3 – Plant trees in an area in the Williams Creek valley where grass seeding has not taken due to damp ground conditions

Kluane Drilling provided a loader and operator (Al Serafinchon) to repair the road bed at Williams Creek with rip-rap and create a diversion ditch at North Williams Creek. The tree planting involved planting cuttings of locally acquired willows at a density of 1 per square metre in an area measuring approximately 25 by 25 m where previously planted grass was not able to establish due to wet ground conditions.

The photos below show the results of the reclamation work.



Rip-rap placed around culvert on Williams Creek, upstream side

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Rip-rap placed around culvert on Williams Creek, downstream side



Diversion ditch on North Williams Creek, looking uphill



Diversion ditch on North Williams Creek, looking downhill

CASSELMAN GEOLOGICAL SERVICES Ltd

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Whitehorse, Yukon, Y1A 4R5
Tel: (867) 456-2706
Cell: (867) 334-1108
Email: casselmangeo@northwestel.net



Willow cuttings planted in wet ground in Williams Creek valley

As well, we removed 9 old fuel drums, which were partially filled with fuel, from the site. These drums had been in lined fuel berms, but having being 4 years old, were becoming stale dated.

Sincerely

"Scott Casselman"

Scott Casselman, B.Sc., P.Geo.

COPPER NORTH
MINING CORP.

2011 ANNUAL QUARTZ MINING LICENCE REPORT

SUBMITTED TO YUKON GOVERNMENT, ENERGY MINES AND RESOURCES

YUKON QUARTZ MINING LICENCE QML-0007

Appendix E

Results of 2010 Geochemical Test Work on Neutralized Spent Ore

---

February 15, 2012

REPORT ON

Results of the 2010 Geochemical Test Work on Neutralized Spent Ore Carmacks Copper Project, Yukon Territory

Submitted to:
Western Copper Corporation
2050 – 1111 West Georgia
Vancouver, BC
V6E 4M3

Attention: Mr. Jonathan Clegg

REPORT

A world of capabilities delivered locally

Project Number: 07-1413-0077/9100
Document Number 173 Rev. 0
Distribution:

2 Copies - Western Copper Corporation

2 Copies - Golder Associates Ltd.

Golder Associates

Executive Summary

Western Copper Corporation (WCC) is currently in the process of permitting the Carmacks Copper Project (Carmacks), Yukon Territory. As part of the characterization program, WCC has conducted metallurgical column tests on ore samples from the Site. A follow-on step to the metallurgical recovery test program has been to conduct post-acid leach neutralization and rinsing to simulate the current proposed closure plan for the heap leach pad following cessation of mining, Golder Associates Ltd. (Golder) has conducted two sets of geochemical testing to characterize the acid rock drainage (ARD) and metal leaching (ML) properties of the neutralized spent ore; the first in 2008 (Golder, 2010), and the second beginning in 2010. This report summarizes the results of the static testing and 56 week humidity cell tests of the neutralized spent ore of the 2010 program and compares the results of the 2010 tests with the results of the 2008 work.

Two samples of neutralized spent ore (Col 7 A+B Left and Col 7 A+B right) were collected as part of the 2010 study from a large scale column leach test conducted at the PRA Laboratory, Richmond BC. The samples were composites of the entire column and consisted of minus 19 mm crushed ore (P_{80} of ½ inch). The samples underwent a suite of standard geochemical tests including acid base accounting (ABA), whole rock major and trace element analyses, short term shake flask extraction (SFE) leach tests, Net Acid Generation (NAG) tests and NAG leachate analyses, mineralogical analyses, and humidity cell tests.

The two 2010 composite samples (Col A+B Left and Right) were classified as non-potentially acid generating based upon current ABA criteria and the circum-neutral NAGpH values (6.84 to 6.96). The samples were sulphate dominant with low sulphide sulphur concentrations (at or below the detection limit of 0.01%) and had alkaline paste pH and measurable bulk acid neutralizing potential (NP) and carbonate acid neutralizing potentials ($CaCO_3$ NP). Whole rock elemental concentrations were generally low in the 2010 samples except for silver, copper, and molybdenum which were elevated compared to similar unmineralized rock. The leachate from the SFE and NAG leach tests met all Metal Mining Effluent Regulation (MMER) guidelines for pH and metal concentrations and were within the Canadian Council of Ministers of the Environment (CCME) criteria except for select metals, amongst them silver, copper and molybdenum. The MMER guidelines and CCME reference standards however do not directly apply to the SFE, NAG leachates, or humidity cell leachates and are provided for reference purposes only.

The humidity cell leachate chemistry from the 2010 samples was very similar to the leachate from samples T1 and Col 6 of the 2008 neutralized spent ore tests. The initial pH values were neutral to alkaline and declined to circum neutral (pH of 7.8) over time. Alkalinity and sulphate concentrations were initially high but declined to values of 40 mg/l and approximately 25 mg/l, respectively, when the tests were terminated. Metal concentrations were low and declined with time. Concentrations were below the MMER guidelines and below the CCME criteria except for selected metals. The similarity between the results of the 2010 humidity cell tests and the 2008 tests, which ran for up to 102 weeks, suggested that the 2010 neutralized spent ore samples would likely maintain the circum neutral pH and low metals concentrations.

Study Limitations

Golder Associates Ltd. (Golder) has prepared this document in a manner consistent with that level of care and skill ordinarily exercised by members of the engineering and science professions currently practising under similar conditions in the jurisdiction in which the services are provided, subject to the time limits and physical constraints applicable to this document. No warranty, express or implied, is made.

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APPENDIX II
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XRD Mineralogy

APPENDIX IV
2008 and 2010 Static and Kinetic Testing Results

1.0 INTRODUCTION

Western Copper Corporation (WCC) is currently in the process of permitting the Carmacks Copper Project (Carmacks), near Carmacks, Yukon Territory. Golder conducted geochemical testing in 2008 using two sets of neutralized spent ore samples from metallurgical testing as part of the project feasibility study. The objectives of the Golder investigation were to characterize the acid rock drainage (ARD) and metal leaching (ML) properties of the neutralized spent ore which is likely to compose the heap leach pad at the site following closure of the mine (Golder, 2010). The spent ore samples consisted of two splits, one sample from the upper portion of the metallurgical test column and a second sample from the lower portions of the column (samples labelled T1 and B1 for the top and bottom samples, respectively) and a third sample that was a composite of the entire column (sample Col 6). The results of the 2008 tests indicated that the ARD/ML properties of the neutralized spent ore varied within the column but when combined in a single composite sample had low ARD potential owing to low sulphide sulphur concentrations as well as measurable residual acid neutralizing capacity and low ML properties. Thus, both static leach tests and long term kinetic testing (102 weeks for T1 and B1 and 58 weeks for Col 6) of these samples met Metal Mining Effluent Regulation (MMER) guidelines for pH and dissolved metal concentrations.

Subsequent to completion of the 2008 study, a large-scale metallurgical column test was completed and the neutralized spent ore made available to Golder for additional geochemical testing. As with the 2008 neutralized spent ore samples, the ore had been leached with acid to recover the copper; rinsed to flush acid out of the spent ore; and then neutralized with alkali to return the spent ore to a neutral pH. This memorandum summarizes the results of the static geochemical testing, the 56 cycles of the kinetic testing, and the close down testing following termination of the kinetic tests for the 2010 neutralized spent ore samples. The 2010 testing was designed to confirm the findings of the 2008 test work concerning the ARD/ML behaviour of the entire pad by using composite samples from the entire test column.

2.0 SAMPLING PROGRAM

The geology of the Carmacks deposit is simple and consists of two major rock types: granodiorite and gneiss (ACG, 2006) with volumetrically minor igneous textural subfacies. The ore is, however, confined to the gneiss. Therefore, when major and trace element chemical analyses are compared to unmineralized rock, the comparison is made to either granodiorite or similar igneous rocks.

Ore samples for metallurgical testing were collected by ACG (ACG, 2007) and details of the sampling procedure are provided in Golder, 2010. The ore material was sent to PRA for metallurgical testing where the bulk samples were mixed to make one composite sample. The composite sample of ore was crushed to minus 19 mm (3/4 inch) with a P_{80} of ½ inch (Redfearn 2010, per. comm.). The ore for the 2010 testing was loaded into two vertical columns, 8 meters tall by 200 mm in diameter that were plumbed in sequence. Raffinate was applied to the top of the first column (Column A) and discharge from Column A was pumped to the top of the second column (Column B). Pregnant solution was recovered from the bottom of Column B. To simulate closure, the two columns were rinsed with water, rinsed with gypsum saturated water, neutralized with sodium carbonate, and then rinsed a final time (Redfearn 2010, per. comm..). The two columns were unloaded on August 13, 2010 and a Golder geochemist recovered representative samples of the neutralized spent ore for geochemical and geotechnical testing. An internal Golder memorandum summarizing the sampling procedure is included in Appendix I.

The geochemical samples were analyzed at SGS, Burnaby, BC. The samples for static testing were crushed to the required grain size for analyses. The samples used for the kinetic test were as delivered from PRA.

3.0 GEOCHEMICAL TESTING

The 2010 test program for the neutralized spent ore included a broad spectrum of static and kinetic testing and was intended to be compatible with historic geochemical investigations of both waste rock and neutralized spent ore. The test program included:

- Whole rock major and trace elemental analyses (pre- and post-kinetic tests);
- Acid base Accounting (ABA) (pre- and post-kinetic tests);
- Shake Flask Extraction (SFE) 24-hour leach tests;
- Net Acid Generation (NAG) tests (pre- and post-kinetic test) and NAG leachate analyses;
- Mineralogical analyses; and
- Kinetic humidity cell tests.

A discussion of past test programs, a description of each test protocol, and the comparative ARD/ML criteria to assess the ARD impacts from the test results are included in Appendix II. The mineralogical report is attached in Appendix III.

Results of the 2008 kinetic testing (up to 102 weeks duration) suggested that the leachate derived from the neutralized spent ore was likely to remain neutral into the distant future. However, pre- and post-humidity cell testing sulphide sulphur analyses suggested that even after protracted cycles of oxidation and flushing, the sulphide sulphur, though much diminished in concentration (between 0.03 and 0.07 wt%, reduced from 0.05 to 0.13 wt%) remained in the test residue (Golder, 2010b). Depletion calculations using the post-test total sulphur, sulphide sulphur and neutralizing potential indicated that the pH of the leachate may decline over time. NAG testing was included in the 2010 testing program in order to provide more conclusive evidence of the likelihood of neutral drainage from the neutralized spent ore.

4.0 RESULTS

The results of the static and kinetic geochemical testing of the 2010 neutralized spent ore samples are presented below. The complete results of the 2010 geochemical testing are available in Appendix IV, Tables IV.1 to IV.8. Results of the analyses for the 2008 neutralized spent ore samples can be found in Attachment IV and are used in the text and figures of this report to provide a broader context for the geochemistry of the neutralized spent ore from the Carmacks Project.

4.1 Major and Trace Element Chemistry

Major and trace elemental concentrations of the samples were compared to the composition of an "average" rock of similar composition to identify either potential elemental enrichment or depletion. The spent ore was composed of primarily alumino-silicate minerals. Figure 4-1 presents plots of the average major element chemistries for the 2008 and 2010 neutralized spent ore samples as compared with an "average" high calcium granite (*i.e.*, a granodiorite) (Price, 1997). The figure shows that the major element compositions of the five neutralized spent ore samples are similar and equivalent to mineralized granodiorite with the exception of iron which is consistently elevated in all of the samples compared to the "average" granodiorite.

The trace metal concentrations of the neutralized spent ore samples are similar to each other and to the equivalent unmineralized rock with the exception of silver, copper, and molybdenum. Silver, copper, and molybdenum concentrations are elevated by greater than an order of magnitude in all samples as compared to the "average" high calcium granite (Price, 1997). The actual availability of an element to leach depends upon factors not measured in the whole rock chemical analyses such as mineralogy or water/rock interaction. Figure 4-2 presents plots of the average metal concentrations for the neutralized spent ore samples as compared with "average" high calcium granitic rock (Price, 1997) with respect to MMER and CCME regulated parameters.

4.2 Shake Flask Extraction (SFE) and NAG Leachate

SFE tests measure the concentrations of readily leachable elements released from minerals in the samples while the NAG leachate analyses is a measure of the elements that may be released as a result of sulphide mineral oxidation, or elements released under acidic conditions resulting from sulphide mineral oxidation. The NAG test and the NAG leachate ideally are representative of "terminal" conditions. The SFE and NAG generated leachates and leachate from the humidity cell tests are compared to both the Metal Mining Effluent Regulations (MMER) and the Canadian Council of Ministers of the Environment (CCME) fresh water aquatic life water quality standards for reference purposes only. The use of MMER and CCME guidelines is strictly qualitative and intended only to provide a mechanism to estimate which elements are more or less mobile compared to a set of reference concentrations. The measured concentrations in the SFE, NAG and humidity cell leachate cannot be directly used as estimates of the actual site discharge concentrations because:

- The water to rock ratio may or may not simulate the actual water to rock ratio at the site (*e.g.,* precipitation infiltrating a waste storage facility);
- Both tests are aggressive, which may over estimate the rates of dissolution of soluble minerals and mineral crusts; and
- The grain size and surface area of the sample material prior to leaching is also not representative of actual conditions at the site.

The concentrations in the SFE and NAG leachate for the neutralized spent ore samples are shown graphically in Figure 4-3 and Figure 4-4 with respect to the MMER and the CCME fresh water aquatic life water quality standards, respectively, and summarized in Table 4.1. Figures 4-3 and 4-4 use one half the detection limit for concentrations reported below the detection limit. SFE leachate from the neutralized spent ore samples was generally alkaline to near neutral. SFE leachate for the 2010 samples was alkaline (7.95 to 8.52), similar to the pH of T1 and Col-6 (8.09 and 7.67, respectively) and within the MMER and CCME acceptable range. The leachate from B1 was slightly acidic (5.96) and below both the MMER and CCME minimum acceptable value for pH of 6.0.

The SFE leachate concentrations for the neutralized spent ore were all below the MMER Maximum Monthly Mean Concentration (the most rigorous of the MMER Guidelines) (Figure 4-3). However, leachate from sample B1 had elevated copper and zinc concentrations as compared to leachate from the 2010, T1 and Col-6 samples. Figure 4-4 shows that all of the neutralized spent ore samples exceed the CCME- fresh water aquatic life guidelines for selected parameters (Table 4.1).

The NAG leachate was near neutral (6.65 to 6.92) for both of the 2010 samples and within the MMER and CCME guidelines. All of the other parameters met the MMER guidelines. Figure 4-4 also shows the measured concentrations for the 2010 NAG leachate samples which met the CCME guidelines except for selected elements (Table 4.1).

Table 4.1: Summary of SFE and NAG Leachate Analyses Exceedences

	SFE		NAG Leachate	
	MMER	CCME	MMER	CCME
Col 7 A+B Left Sample	none	Al, As, Hg, Mo, Se, Ag, Cu, Fe,	none	Se, Ag, Cr, Cu
Col 7 A+B Right Sample	none	Hg, Mo, Se, Ag, Cu	none	As, Se, Ag, Cr, Cu
B 1 (-1/2 inch)	pH	pH, Al, Se, Cd, Cu, Zn	na	na
T 1 (-1/2 inch)	none	Mo, Se,	na	na
Col-6 (0700601)	none	Mo, Se, Cd, Cu	na	na

4.3 ABA

ABA testing is used to measure the sulphur species concentrations as an indicator of the acid generating potential (AP) and acid neutralizing potential (NP) of the samples. Two methods of determining the NP were used based upon standard industry practice: the bulk NP of the sample and the carbonate or CaCO3 NP (Appendix III). The ABA results for the 2010 and 2008 Golder samples are summarized in Figures 4-5 to 4-9. The ABA results for the 2008 Golder samples are presented in Figures 4-5 to 4-9 for comparative purposes. Table 4.2 summarizes the results of the ABA analyses.

4.3.1 Paste pH and Sulphide Sulphur Concentration

The 2010 samples had higher paste pH and lower sulphide sulphur concentrations than the 2008 samples. Figure 4-5 is a plot of paste pH and sulphide sulphur for the 2010 and 2008 neutralized spent ore samples. Paste pH ranged from 8 to 8.93 for the 2010 and sulphide sulphur concentrations were 0.01 wt%. Samples T1 and Col-6 reported paste pH values of 7.85 and 7.89 and similar, very low sulphide sulphur concentrations. Sample B1 reported slightly lower paste pH and a slightly higher sulphide sulphur concentration.

Table 4.2: ABA Summaries for the 2008 Waste Rock and Spent Ore Samples

Rock Type	Paste pH	S(T)	S(SO4)	S(S-2)	NP	$CaCO_3$ NP
	su	%	%	%	Kg $CaCO_3$/tonne	Kg $CaCO_3$/tonne
Col 7 A+B Left Sample	8.93	0.29	0.28	0.01	22.5	9.2
Col 7 A+B Right Sample	8.00	0.52	0.51	0.01	26.8	13.3
B 1 (-1/2 inch)	5.91	0.55	0.42	0.13	6.3	0.8
T 1 (-1/2 inch)	7.85	0.12	0.07	0.05	10.8	6.7
Col-6 (0700601)	7.89	0.38	0.03	0.08	8.8	3.3

S(T) = total sulphur concentration, S(SO4) = sulphate sulphur concentration, and the S(S-2) = sulphide sulphur concentration.

4.3.2 Sulphur Speciation

Sulphate sulphur was the dominant sulphur species for all of the neutralized spent ore samples. Figure 4-6 is a plot of sulphide content as a function of total sulphur for the neutralized spent ore samples. A reference line of 1:1 is presented in the figure to assess the dominant form of sulphur.

Unlike the 2008 samples with 0.05 to 0.13% sulphide sulphur shown in Figure 4-5, the 2010 samples had lower sulphide sulphur concentrations near the detection limits (< 0.01%). The higher measured sulphate sulphur concentration is consistent with spent ore having undergone oxidation as part of the leaching process and is likely derived from jarosite [$K_2Fe_6(SO_4)_4(OH)_{12}$], a common secondary hydrous sulphate mineral associated with oxidized or weathered ore (Section 5.5).

4.3.3 Neutralization Potential

Two static test procedures were used to measure the neutralization potential of the waste rock samples: 1) Bulk NP (NP) and 2) Carbonate NP ($CaCO_3$ NP). The bulk NP is the total neutralizing potential of a rock from carbonate and other forms of acid-neutralizing chemical species such as the hydroxide ion (OH^-) and aluminosilicate minerals. The $CaCO_3$ NP is an estimate of the amount of carbonate ion (HCO_3^- and CO_3^{-2}) neutralizing material present in a rock. The use of $CaCO_3$ NP can be considered a more conservative estimate of the acid neutralizing potential of a rock.

Bulk NP was the dominant neutralizing species for all of the samples suggesting a non-carbonate source for the majority of the neutralizing potential of the spent ore. Figure 4-7 is a plot of the NP versus $CaCO_3$ NP for the neutralized spent ore samples. The bulk NP for the 2010 samples was twice the bulk NP of the 2008 samples (22.5 to 26.8 kg $CaCO_3$ /tonne and 6.3 to 10.8 kg $CaCO_3$ /tonne, respectively) which in part reflects the higher $CaCO_3$ NP of the 2010 samples compared with the 2008 samples (9.2 to 13.3 kg $CaCO_3$ /tonne and 0.8 to 6.7 kg $CaCO_3$ /tonne, respectively). The likely source of the $CaCO_3$ NP is calcite ($CaCO_3$) derived from the sodium carbonate used in the neutralization step (Section 5.5).

4.3.4 ARD Criteria for the Supplemental Waste Rock Samples

ARD criteria are based upon terms derived from either the ABA or NAG tests: the net potential ratio (NPR), which is the ratio of the NP to the AP of the rock (NPR = NP/AP), the net neutralization potential which is the difference between the acid generating and acid neutralizing potentials of the sample (NNP = NP-AP), and the NAGpH which is the pH of the leachate derived from the NAG test. Results of the three ARD criteria are summarized in Table 4.3

4.3.4.1 NPR Criteria

The 2010 samples had NPR values well within the field of non-potentially acid generating material under the MEND (2009) criteria. Figure 4-8 is a plot of Bulk NP versus AP for the neutralized spent ore samples. Two of the three 2008 samples likewise were classified as non-potentially acid generating. Sample B1 is classified as having uncertain acid generating potential.

The 2010 samples were both non-potentially acid generating owing to the higher CaCO3 NP and very low AP (*i.e.*, sulphide sulphur concentration), while the 2008 samples ranged from non-potentially acid generating to potentially acid generating. Figure 4-9 is a plot of the $CaCO_3$ NP versus AP for spent ore samples. Figure 4-10 is a plot of sulphide sulphur versus NPR which shows the effects of the higher carbonate and lower sulphide sulphate concentrations in the 2010 samples compared with the 2008 samples and the subsequent impact on the NPR.

Table 4.3: ARD Criteria Summary for the Neutralized Spent Ore Samples

	Net NP	NPR	$CaCO_3$ Net NP	CaNPR	NAG pH	NAG Acidity (to pH 4.5) kg H_2SO_4/t
Units	Kg $CaCO_3$/tonne		Kg $CaCO_3$/tonne	Kg $CaCO_3$/tonne	su	
Col 7 A+B Left Sample	22.2	72.0	8.9	29.33	6.97	n/a
Col 7 A+B Right Sample	26.5	85.8	13.0	42.67	6.84	n/a
B 1 (-1/2 inch)	2.2	1.6	-3.2	0.21	-	-
T 1 (-1/2 inch)	9.2	6.9	5.1	4.27	-	-
Col-6 (0700601)	6.3	3.5	0.8	1.33	-	-

4.3.4.2 NNP Criteria

The NNP is an alternative criterion often used with the NPR, although most commonly used in situations with either high sulphide sulphur concentration or high neutralizing potential. Figure 4-11 is a plot of sulphide sulphur versus NNP which shows that the 2010 samples both fell into the range of non-acid generating material owing to the high bulk NP and very low sulphide sulphur concentrations. The 2008 samples were of uncertain acid-generating potential based upon the NNP criteria. Figure 4-12 is a plot of the NPR versus NNP; the 2010 samples both fell in the field of non-acid generating material.

4.3.4.3 NAG Criteria

The NAG pH of both of the 2010 samples was circum-neutral (6.84 to 6.97) which was consistent with the very low sulphide sulphur concentration and the high bulk NP and CaCO3 NP (Table 4.3). The NAG Acidity to pH 4.5 was non-detectible owing to the absence of sources of acidity and therefore both samples are classified as non-acid generating based upon the AMIR (2002) criteria.

4.4 Mineralogy

The two 2010 samples and two of the three 2008 neutralized spent ore samples (B1 and T1) were analysed for their mineralogy using X-Ray Diffraction. The test method is discussed in Appendix II and III and the full results of the mineralogical analytical test are provided in Appendix III and IV.

All of the neutralized spent ore samples were composed of primarily plagioclase, K-feldspar, quartz, biotite, and actinolite. The 2008 samples contained from 12 to 19.7% clay (montmorillonite) while the 2010 samples contained significantly lower proportions of clay 1.9 to 2.3% (kaolinite) and higher proportions of plagioclase (62 to 62.5% compared with 52.1 to 56.8%). The differences in feldspar and clay abundances may reflect different raw ore samples with different levels of primary hydrothermal alteration and/or different levels of weathering. All of the samples contained jarosite and the two 2010 samples and the 2008 T1 sample contain between 0.6 to 1% calcite. None of the samples contained measurable sulphide mineralization.

4.5 Kinetic Testing

The humidity cell testing for the 2010 samples began on August 31, 2010 and terminated on September 20, 2011, after 56 cycles of testing. The results of the 2010 humidity cell tests and the 2008 tests for samples B1, T1,and Col-6 are presented in Appendix IV, Table IV.8 and IV.9 for Col 7 A+B Left and Col 7 A+B Right and Tables IV.9 to IV.11 for , B1, T1 and Col-6 samples, respectively.

4.5.1 Weekly Analyses

Figures 4-13 to 4-15 are time series plots for the results of the weekly pH, alkalinity, and sulphate measurements for all of the neutralized spent ore sample humidity cell tests. pH is a MMER regulated parameter and therefore Figure 4-13 includes the MMER guideline pH range. Table 4.4 summarizes bulk and $CaCO_3$ NPR, the maximum and minimum pH, alkalinity and sulphate for cycles 1 to 56 for Col 7 A+B Left and Right, and cycles 1-102 for T1 and B1 and the average pH, alkalinity and sulphate for the last five cycles. The table also includes the same set of data for cycles 1-58 of humidity cell tests for Col-6.

The bulk NPR and CaCO3 NPR are included in Table 4.4 because these are measures of the long-term ARD potential of the rock. Both 2010 samples were classified as non-acid generating based upon the bulk NPR and CaCO3 NPR and results of the humidity cell tests confirm the long term prediction; average pH values are 7.75 and 7.82 respectively. The bulk NPR and CaCO3 NPR of the two 2010 samples are an order of magnitude greater than the equivalent values for T1 and Col 6, confirming the likelihood of the 2010 samples maintaining long-lived neutral to alkaline pH values.

Results of the weekly analyses of the 2010 humidity cell tests suggested that the leachate from Col 7 A+B Left and Right were alkaline and within MMER guidelines. The minimum and maximum pH values ranged from 7.66 to 9.24 for Col 7 A+B Left, roughly an order of magnitude higher than the pH for Col 7 A+B Right, T1, and Col 6. As Figure 4-13 shows however, the pH of Col 7 A+B Left declined with time and by cycle 38, the pH of leachate from Col 7 A+B Right, T1 and Col 6 were essentially identical and remained so for duration of the tests; 56, 58, and 102 cycles, respectively. The pH for sample B1 was consistently lower than the pH for the other samples.

The alkalinity of Col 7 A+B Left ranged from 36 and 254 mg/L for the 56 cycles of the test and like the pH, was significantly higher than the measured alkalinity in the leachate from Col 7 A+B Right, Ti and Col 6. However, as with pH, Figure 4-14 shows that the alkalinity concentration in leachate from Col 7 A+B Left quickly declined and by cycle 38 or 39, matched the concentrations in Col 7 A+B Right, T1, and Col 6 and remained so for the rest of the test cycles suggesting that the two 2010 samples will likewise maintain elevated alkalinity concentrations into the future.

The sulphate concentrations in leachate from the two 2010 samples were similar to the starting sulphate concentrations in all of the 2008 leachate samples. The sulphate concentrations in the 2010 samples declined with time but generally less rapidly than the 2008 samples however since neither Col 7 A+B Left or Right contained measurable sulphide sulphur, the long term sulphate concentration is likely to decline in a similar fashion to the measured sulphate in T1 and Col 6.

Table 4.4: ABA Results and Summary of Leachate Chemistry for Weekly Humidity Cell Results

Sample	ABA Results		Humidity Cell Results								
	Bulk NPR	$CaCO_3$ NPR	Minimum/Maximum Results						Average for Last Five Analyses		
			pH		Alkalinity		SO_4		pH	Alkalinity	SO_4
			Max	Min	Max	Min	Max	Min			
Col 7 A+B Left	72.0	29.33	9.24	7.66	254.5	36.5	557	20	7.75	41.4	28
Col 7 A+B Right	85.5	42.67	8.22	7.58	90.2	34.9	797	19	7.82	41.2	23.8
T1	6.91	4.27	7.84	7.73	60.7	49.9	3	1	7.79	56.6	2
	1.55	0.21	5.66	5.48	1.3	0.8	7	2	5.58	1.2	5
Col-6	3.52	1.33	7.74	7.52	44.4	32.2	3	1	7.62	37.3	2

4.5.2 Full Chemical Analyses

Results of the 56 cycles of the two 2010 humidity cell testing showed that the metal concentrations in humidity cell leachate were low, below the MMER guidelines, and similar to the concentrations observed the 2008 samples particularly the 102 cycles of T1 and 58 cycles for Col-6. Figures 4-16 to 4-20 are time series plots of the results of MMER parameters arsenic, copper, nickel, lead and zinc. The figures include the MMER maximum monthly mean and maximum grab sample concentrations.

Several elements were elevated in concentration in leachate from the first four cycles of the 2010 humidity cell tests compared with the CCME criteria. Table 4.5 lists the elements in high concentration for the first four analyses of the 2010 and 2008 humidity cell tests, and the average of the last five analyses for tests. Figures 4-21 to 4-25 are time series plots of antimony, cadmium, molybdenum, mercury and selenium - elements that are mobile under neutral pH conditions. Figures 4-16 to 4-25 and Table 4.5 demonstrate that the concentrations of dissolved chemical species rapidly declined over the first few weeks of the 2010 and 2008 tests by up to several orders of magnitude. The concentrations of the 2010 leachate generally declined similar to the pattern for the concentrations observed in the 2008 leachate samples; and based upon Figures 4-16 to 4-25, are likely to decline over time as observed in leachate from cell T1. As noted in Section 4.2, the MMER guidelines and CCME reference standards do not directly apply to the humidity cell leachates but were provided for reference purposes and used as a metric for relative mobility.

Table 4.5: Average Metal Concentrations for Elements Exceeding Guidelines for the Last Five Analyses of the Humidity Cell Tests

	First four analyses				Last 5 Analyses	
	Col 7 A+B Left and Right		2008 Samples		Col 7 A+B Left and Right	2008 Samples
	Minimum	Maximum	Minimum	Maximum	Average	Average
MMER	none	none	none	none	none	none
CCME	Al, Cd, Cu, Fe, Pb, Hg, Mo, Se, Ag	Al, As, Cd, Cr, Cu, Fe, Pb, Hg, Mo, Se, Ag,	Al, Cd, Cu, Mo, Se, Zn	Al, Cd, Cu, Hg, Mo, Se, Zn	Al, Cd, Mo, Se, Ag	Al, Cd, Cu, Mo, Se

4.6 Closedown Test Results

Following completion of the 56 cycles of the 2010 humidity cell tests, samples of test residue from the Col 7 A+B Right and Left cells underwent several of the same set of static tests as conducted at the onset of the geochemical testing program. These test results (post-kinetic test samples) are compared below with the initial static test results (pre-kinetic test samples) to document changes in ARD and whole rock chemistry that may have occurred as a result of prolonged leaching.

Comparison of the pre- and post-kinetic test static testing results for the Col 7 A+B Left and Right samples must be done with caution because 1) there is an inherent variability in the chemical composition between any two splits of heterogeneous material (sampling variability) and 2) there is a variability in the analytical results from analyses done on different batches (analytical variability). Also, the sample splits for the two sets of tests were from run-of-mine material. In both instances, a 200 g split was collected from the coarse grain spent ore samples (minus 19 mm (3/4 inch) with a P_{80} of ½ inch) and pulverized for the static test work. The presence or absence of even a few larger fragments can therefore skew the geochemistry of the test samples.

4.6.1 Acid Base Accounting

The closedown tests consisted of full ABA analysis including paste pH, NP, sulphur species and TIC concentrations. Table 4.6 summarizes the results of the pre-kinetic and post-kinetic ABA testing and the derived NPR and $CaCO_3$ NPR for the 2010 samples.

The paste pH values of the pre-kinetic test samples were consistently higher than the paste pH of the post-kinetic test samples, which was consistent with higher NP and to a lesser extent, the higher $CaCO_3$ NP of the pre-kinetic samples. Sulphur species concentrations between the two samples were similar and neither sample had measurable sulphide sulphur concentrations.

Both sets of pre- and post-kinetic test samples were non-potentially acid generating using the MEND NPR guidelines. However, as a result of the higher NP and $CaCO_3$ NP concentrations, the pre-kinetic test samples have higher NPR and CaCO3 NPR values.

Table 4.6: ABA Summaries for the 2008 Waste Rock and Spent Ore Samples

Rock Type	Paste	S(T)	S(SO4)	S(S-2)	NP	$CaCO_3$ NP	NPR	CaCO3 NPR	NAG pH
	pH	%	%	%	Kg $CaCO_3$/tonne	Kg $CaCO_3$/tonne			su
Pre-Kinetic Test Samples									
Col 7 A+B Left Sample	8.93	0.29	0.28	0.01	22.5	9.2	72.0	29.33	6.97
Col 7 A+B Right Sample	8.00	0.52	0.51	0.01	26.8	13.3	85.8	42.67	6.84
Post Kinetic Test Samples									
Col 7 A+B Left Sample	7.68	0.4	0.49	<0.01	11.6	7.5	37.12	24.00	6.76
Col 7 A+B Right Sample	7.74	0.43	0.52	<0.01	9.9	3.3	31.68	10.67	6.82

4.6.2 Net Acid Generation NAG Test

The NAG test is an alternative to the ABA analyses to estimate the ARD potential of the samples (Table 4.6). Both sets of pre-and post-kinetic test NAG pH values were identical and circum-neutral indicating that the Col 7 A+B Left and Right samples were non-potentially acid generating and, as suggested by the ABA tests, lacked any indication of pyrite, which in the presence of the hydrogen peroxide would have released acidity and depressed the NAG pH.

4.6.3 Major and Trace Element Chemistry

The differences in both the major element and trace element concentrations of the pre- and post-kinetic test samples were inconclusive. Tables 4.7 and 4.8 summarize the results of the pre- and post-kinetic tests major elements and trace metal analyses, respectively.

The major element concentrations generally showed an increase between the pre- and post-kinetic tests. The source of the differences may have been simply the differences that arose between analyses of two different samples of otherwise heterogeneous material. The similarity of silica concentrations between the pre- and post-kinetic test samples precludes a systematic analytical variability.

Table 4.7: Major Element Concentrations

Sample ID	SiO2	Al2O3	Fe2O3	CaO	MgO	Na2O	K2O
	%	%	%	%	%	%	%
Pre-Kinetic Test Samples[1]							
Col 7 A+B Left Sample	60.11	16.87	5.33	4.3	1.91	4.62	1.96
Col 7 A+B Right Sample	58.17	15.97	6.27	4.55	2.03	4.46	1.88
Post-Kinetic Test							
Col 7 A+B Left Sample	59.12	20.13	8.51	5.38	2.69	5.03	2.39
Col 7 A+B Right Sample	59.26	19.31	7.62	4.27	2.32	5.16	2.40

The trace element chemistry showed a similar overall increase in concentrations between the pre- and post-kinetic test samples. The observed differences may have been due to differences in bulk chemistry as well as potential analytical variability. The two sets of analyses include different analytes suggesting that the analytical methods, standards, or calibration may have changed over time.

Table 4.8: Trace Element Concentrations

Sample ID	Al	Ag	As	Cd	Cr	Cu	Fe	Hg	Mo	Ni	Pb	Se	Tl	Zn
	ppm	Ppm	ppm	ppm	Ppm	Ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
	82000	0.051	1.9	0.13	22	30	29600	0.08	1	15	15	0.05	0.72	60
Pre-Kinetic Test Samples														
Col 7 A+B Left Sample	10700	2.3	1	0.2	40	588.4	29100	0.211	77.8	5.9	3.2	0.8	0.2	40
Col 7 A+B Right Sample	10200	2.1	1.5	0.3	33	634.9	32400	0.156	103.9	4.7	2.9	0.9	0.3	42
Post-Kinetic Test Samples														
Col 7 A+B Left Sample	17200	3.49	4	0.1	45	886	43300	0.03	110	3.8	6.4	2	0.3	51
Col 7 A+B Right Sample	16700	3.16	5	0.12	61	1000	45400	0.07	103	4.8	10.2	2	0.31	59

[1] The pre-kinetic test data have been normalized to 100% including LOI. The post-kinetic data reported in Appendix IV are the raw, un-normalized data. For comparison purposes, the post-kinetic test data in Table 4.6 has been normalized to include LOI.

5.0 DEPLETION CALCULATIONS

Time estimates for complete consumption of sulphide sulphur and NP (depletion calculations) for both the Col 7 A+B Left and Right samples have been calculated using:

- the results of the pre- and post kinetic tests ABA analyses; and
- release rates of sulphate and alkalinity from the last five cycles of the humidity cell tests.

Depletion calculations were constructed using the total sulphur and sulphide sulphur concentrations. The total sulphur concentration may have over estimated the time for depletion of sulphide sulphur because it assumed all sulphur present was sulphide sulphur and the sulphate measured in the humidity cell leachate was derived entirely from pyrite oxidation. Using the sulphide sulphur concentration may underestimate the depletion time because the sulphide sulphur concentrations are low (at or below the detection limits) and the primary source of soluble sulphate was dissolution of sulphate minerals not pyrite oxidation. Table 5.1 summarizes the estimates for time of depletion for pyrite oxidation for total and sulphide sulphur models and depletion of NP. The results of the pre- and post-kinetic test sample depletion time estimates were very similar; using the total sulphur concentration the sulphur will outlast the NP and using the sulphide sulphur, the NP will outlast the sulphur.

Table 5.1: Results of Depletion Calculations

Sample ID	S	S	NP
	(Based on Total Sulphur)	(Based on Sulphide Sulphur)	(Alkalinity)
Pre-Kinetic Test Samples			
Col 7 A+B Left Sample	14	0.5	3.1
Col 7 A+B Right Sample	27	0.5	5.5
Post-Kinetic Test Samples			
Col 7 A+B Left Sample	20	0.5	4.3
Col 7 A+B Right Sample	23	0.5	4.5

6.0 CONCLUSION

Results of the 2010 neutralized spent ore sampling and analyses program for the Carmacks Copper Project lead to the following conclusions.

The two 2010 composite samples (Col A+B Left and Right) were non-potentially acid generating based upon current ARD criteria for the NPR and CaNPR ratios owing to (1) low sulphide sulphur concentrations (0.01%) and (2) measurable bulk and carbonate neutralizing potential. The dominant sulphur species was sulphate sulphur which was consistent with the presence of jarosite, a potassium iron sulphate mineral, and the absence of detectable sulphide minerals. The paste pH values were alkaline (8.0 to 8.93 and 7.68 to 7.74 for pre- and post-kinetic tests, respectively) and the NAGpH values were circum-neutral (6.84 to 6.96 and 6.78 to 6.82 for pre-and post-kinetic tests, respectively) confirming the lack of ARD potential over both the short-term and long-term, respectively.

Whole rock elemental concentrations were generally low in the 2010 samples except for silver, copper, and molybdenum which are elevated compared to similar unmineralized rock. The low elemental concentrations were consistent with the low dissolved metal concentrations in leachate from the short term leach tests. The leachate from the SFE and NAG leach tests met all MMER guidelines for pH and metal concentrations and were within the CCME criteria except for select metals, amongst them silver, copper and molybdenum. The MMER guidelines and CCME reference standards, however, do not directly apply to the SFE or NAG leachates and are provided for reference purposes only.

The two 2010 samples were geochemically similar to the top sample (T1) and the composite sample (Col 6) from the 2008 test program. Results of the Col 7 A+B Right cell matched the weekly behaviour of T1 and Col 6, leachate with slightly alkaline pH and slightly elevated and declining alkalinity and sulphate concentrations over the duration of the tests. Leachate from Col 7 A+B Left had elevated starting pH and strongly alkalinity and sulphate concentrations which declined rapidly within the first 40 cycles of the test to track thereafter the pH, alkalinity and generally sulphate concentrations of the Col 7 A+B Right and the longer-lived T1 and Col 6 tests. By the end of the tests, the average pH was near neutral (approximately 7.8) alkalinity and sulphate values were low, 41 and 25 mg/L, respectively.

Metal concentrations in the leachate from Col 7 A+B Right and Left were low and within the MMER guidelines for both the early cycles of the humidity cell tests as well as the later cycles of the tests as were the 2008 samples. The leachate from the 2010 samples also met the CCME reference standards except for select metals.

Results of the closedown tests showed that the NP declined over the course of the test, however the paste pH remained neutral to slightly alkaline and the sulphur species concentrations remained unchanged. The sulphide sulphur began the tests at the detection limit and declined to below the detection limit over the 56 cycles of the test. Comparison of the metals concentrations between the pre- and post-kinetic tests was inconclusive.

The depletion calculations were prepared using both the total sulphur concentration as a proxy for the sulphide sulphur concentration, a conservative estimate, and using the sulphide sulphur concentration directly. The more conservative estimate of the total sulphur indicated that the sulphur may out last the NP; using the actual sulphur concentration, NP outlasted the sulphur concentration. The very low sulphide sulphur concentrations suggests that sulphide oxidation is unlikely in the Carmacks neutralized spent ore

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SAA/FEC/JAH/np

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Figure 4-1
Average Major Element Chemistry for 2007 and 2008 Samples
Carmacks Copper Project, YT

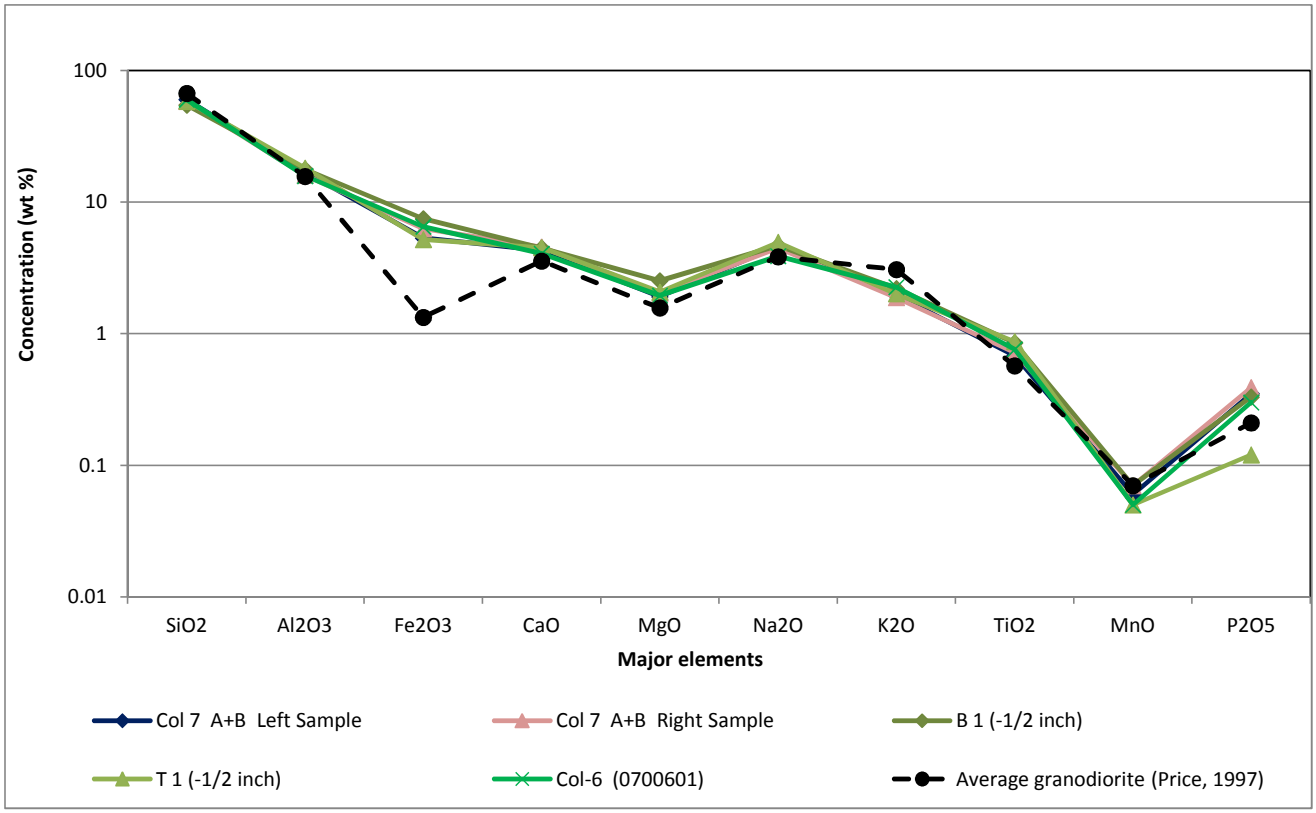


Figure 4-2
Average Trace Element Concntrations for 2007 and 2008 Samples
Carmacks Copper Project, YT

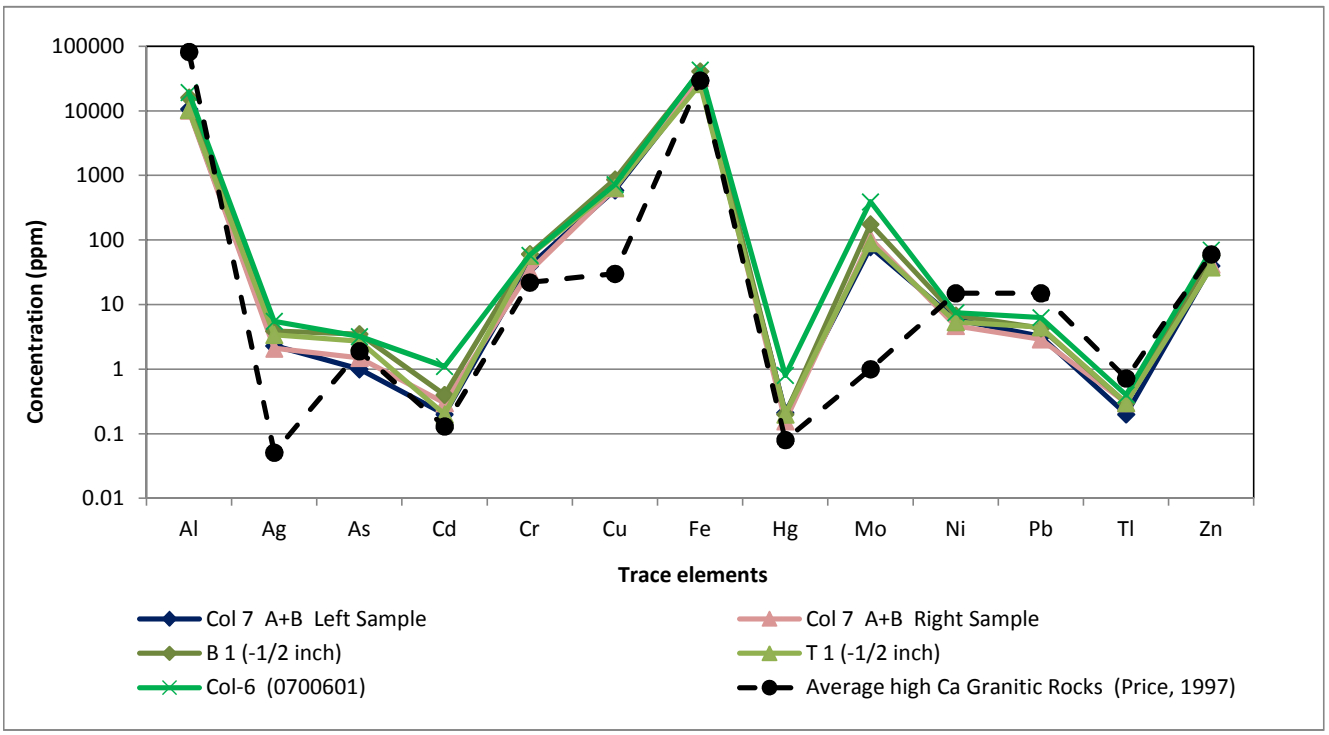
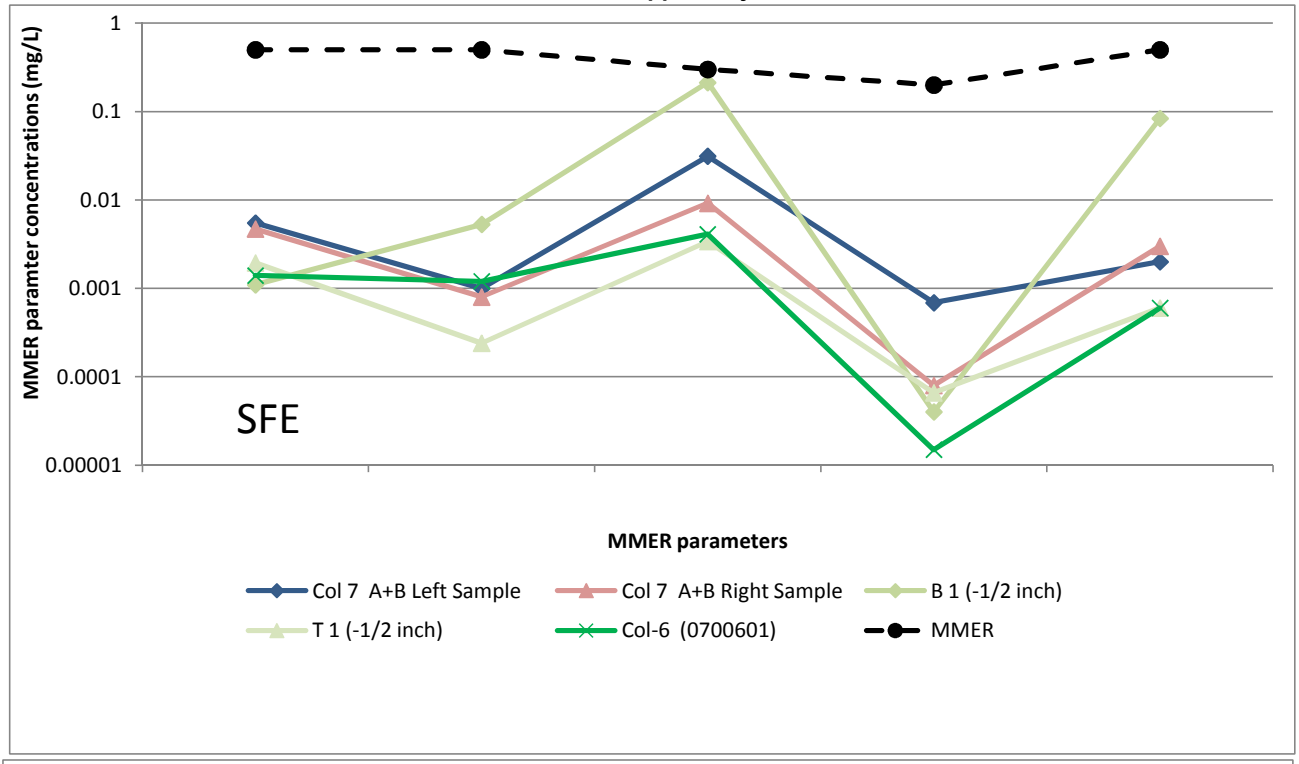


Figure 4-3
Average SFE Leachate Concentrations of MMER Parameters for 2007 and 2008 Samples
Carmacks Copper Project, YT



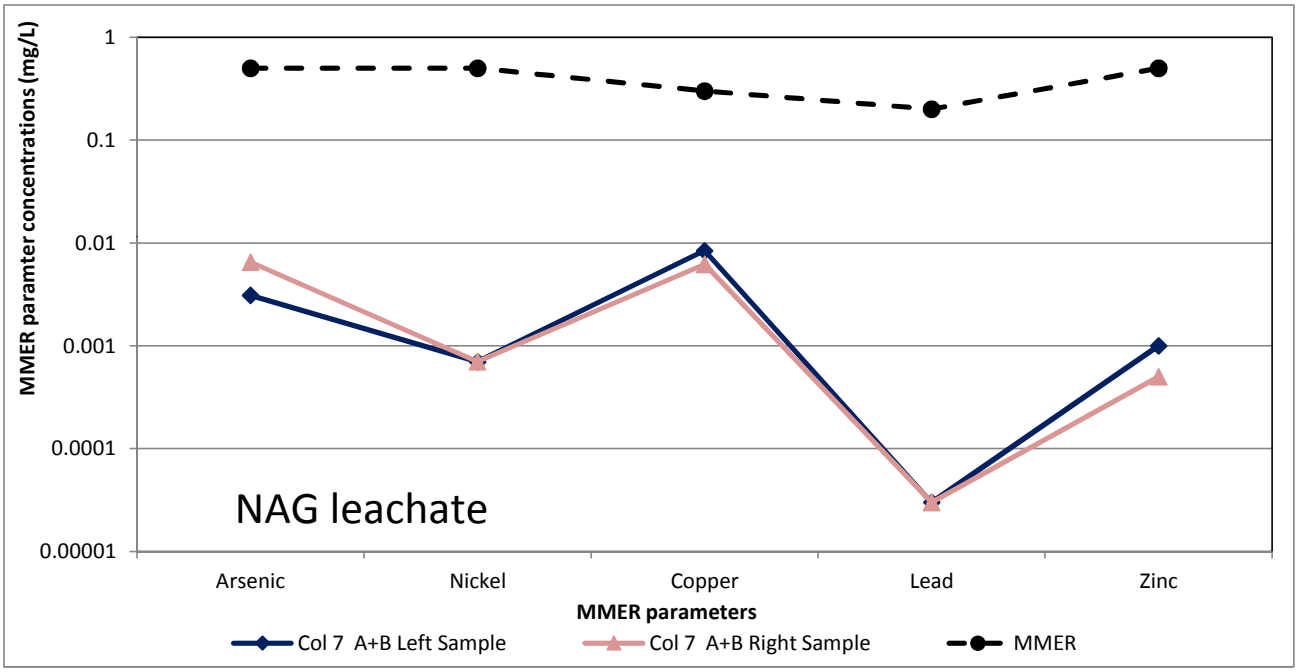


Figure 4-4
Average SFE Leachate Concentrations of CCME Parameters for 2007 and 2008 Samples
Carmacks Copper Project, YT

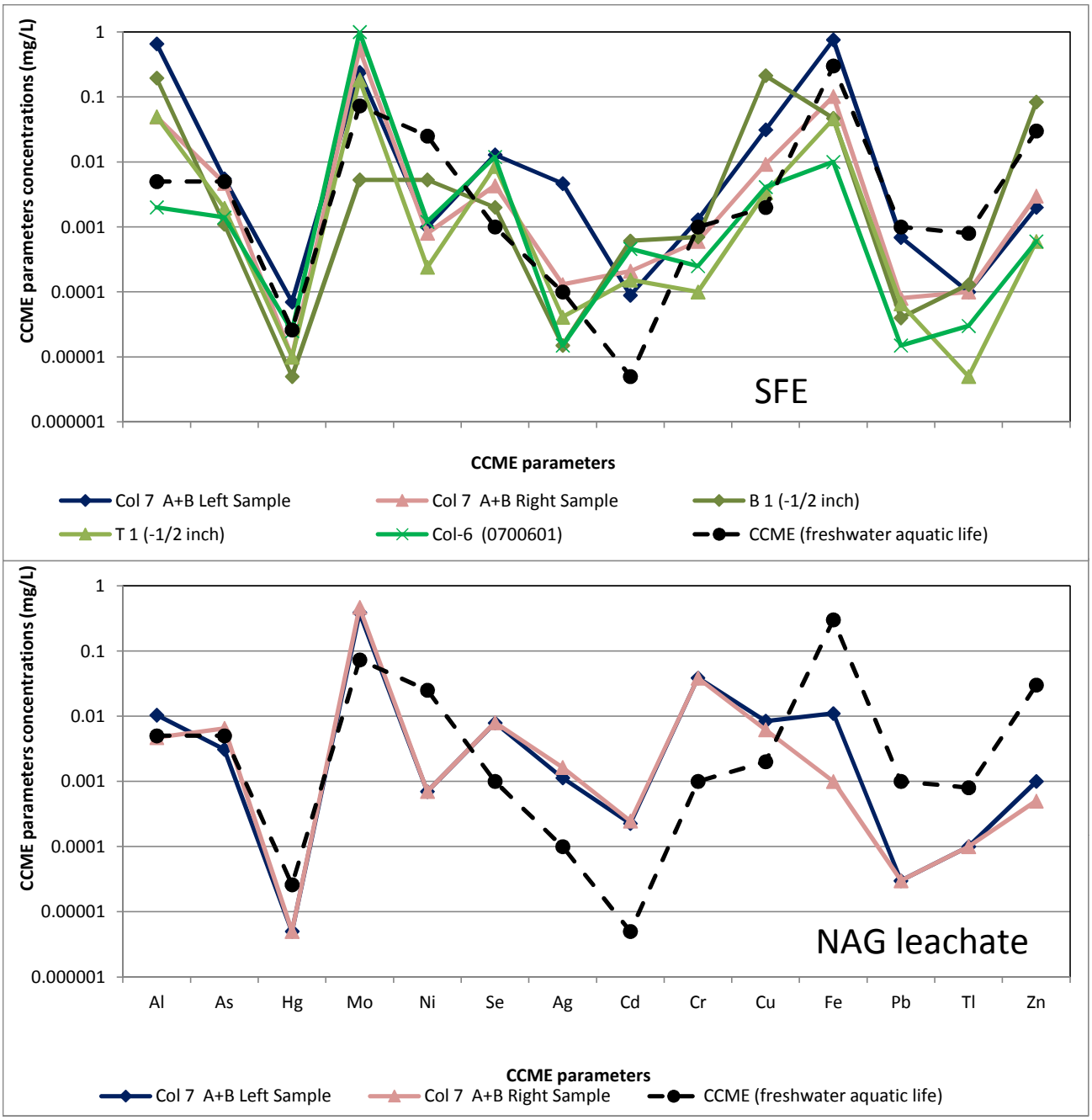


Figure 4-5
Sulphide Sulphur vs. Paste pH for 2007 and 2008 Samples
Carmacks Copper Project, YT

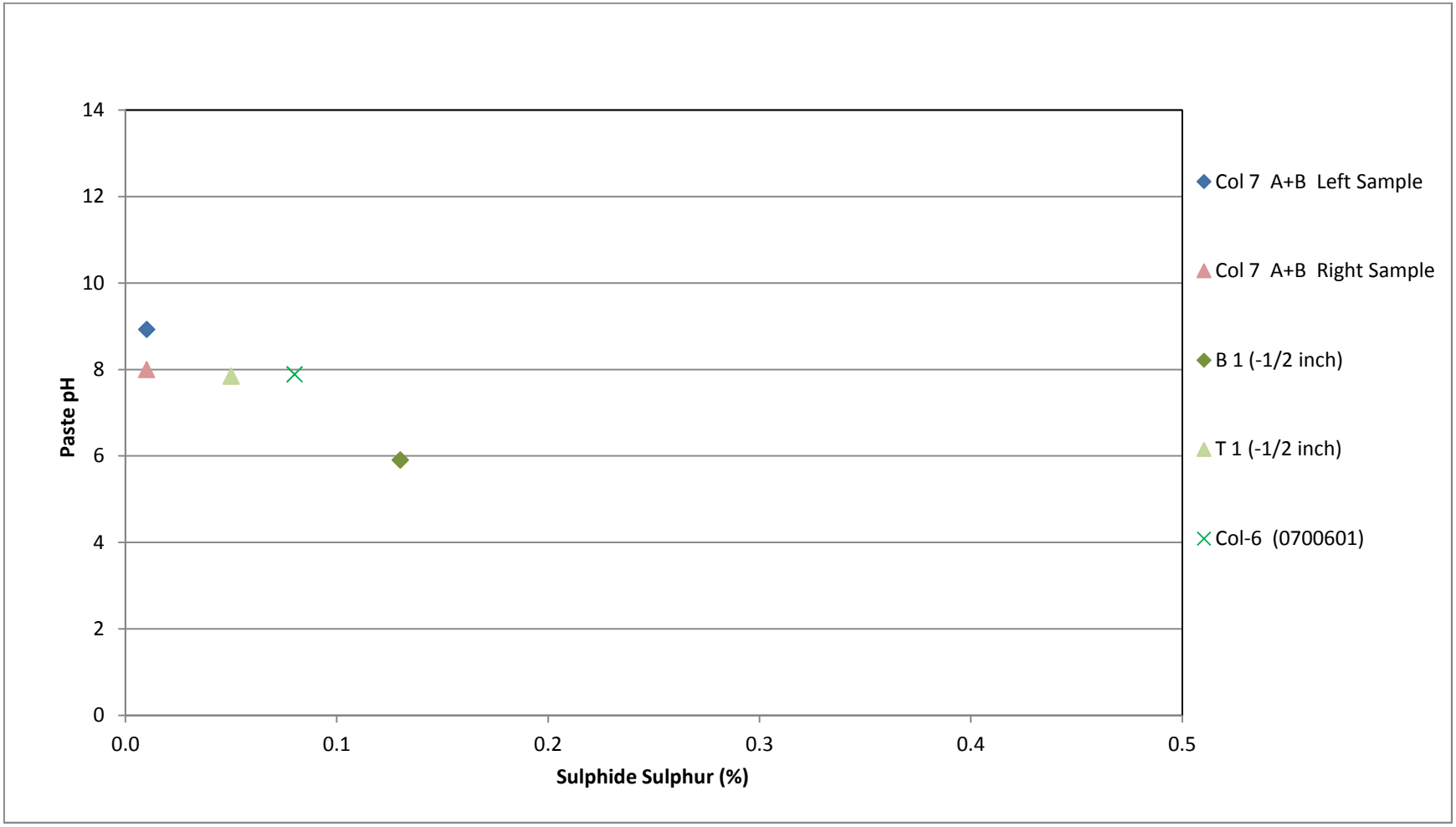


Figure 4-6
Sulphide Sulphur vs. Total Sulphur for 2007 and 2008 Samples
Carmacks Copper Project, YT

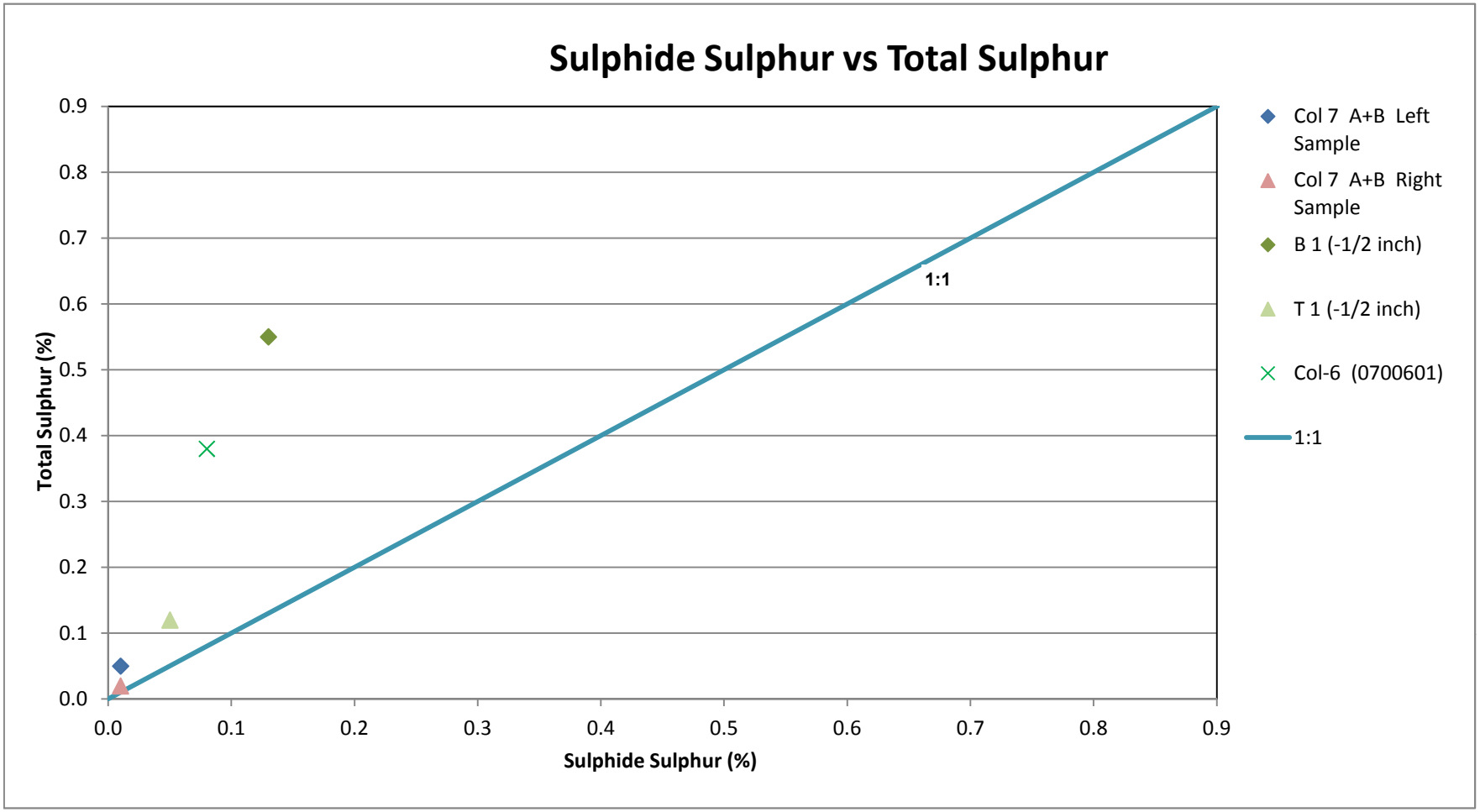


Figure 4-7
CaCO3 NP vs. Bulk NP for 2007 and 2008 Samples
Carmacks Copper Project, YT

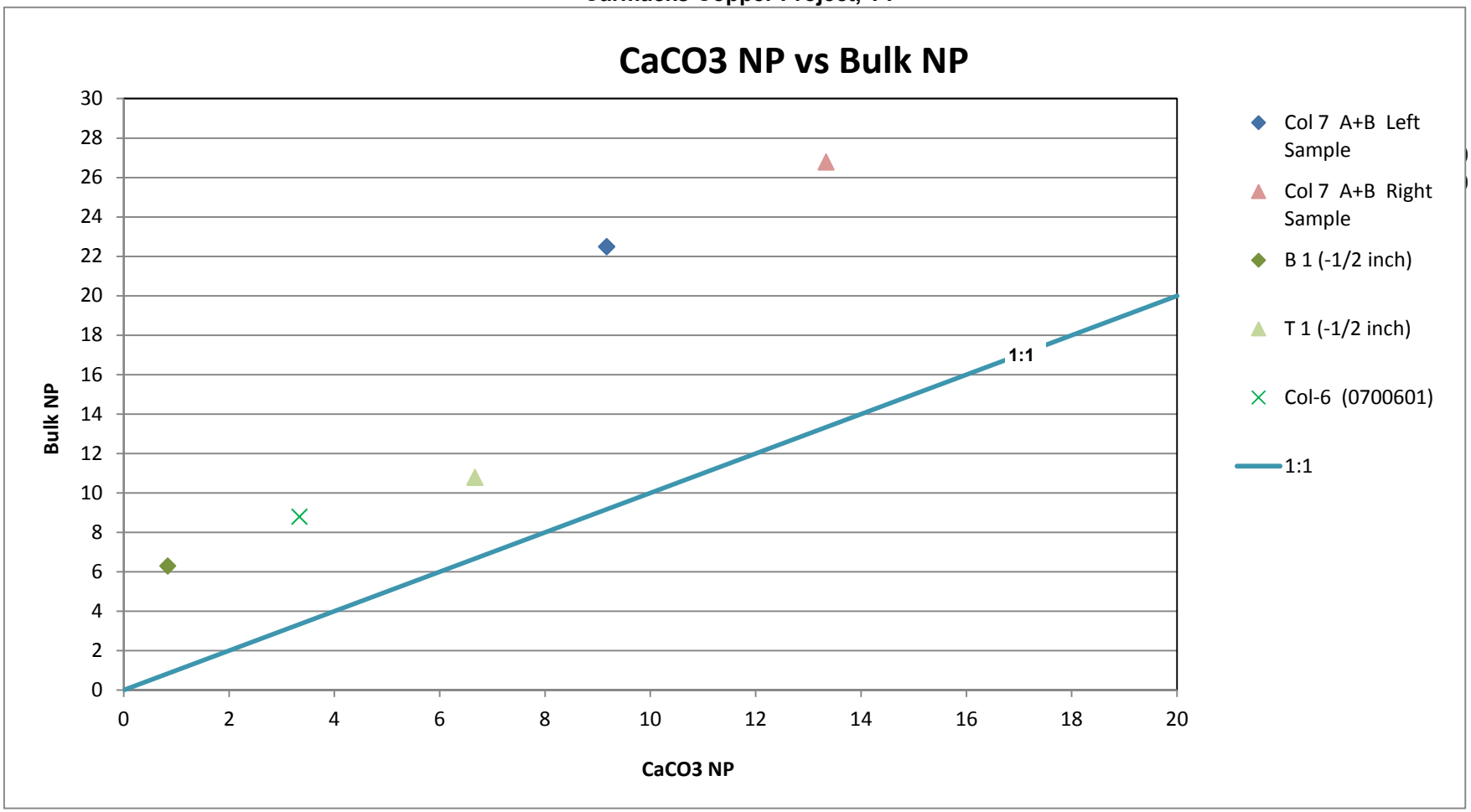


Figure 4-8
Bulk NP vs. AP for 2007 and 2008 Samples
Carmacks Copper Project, YT

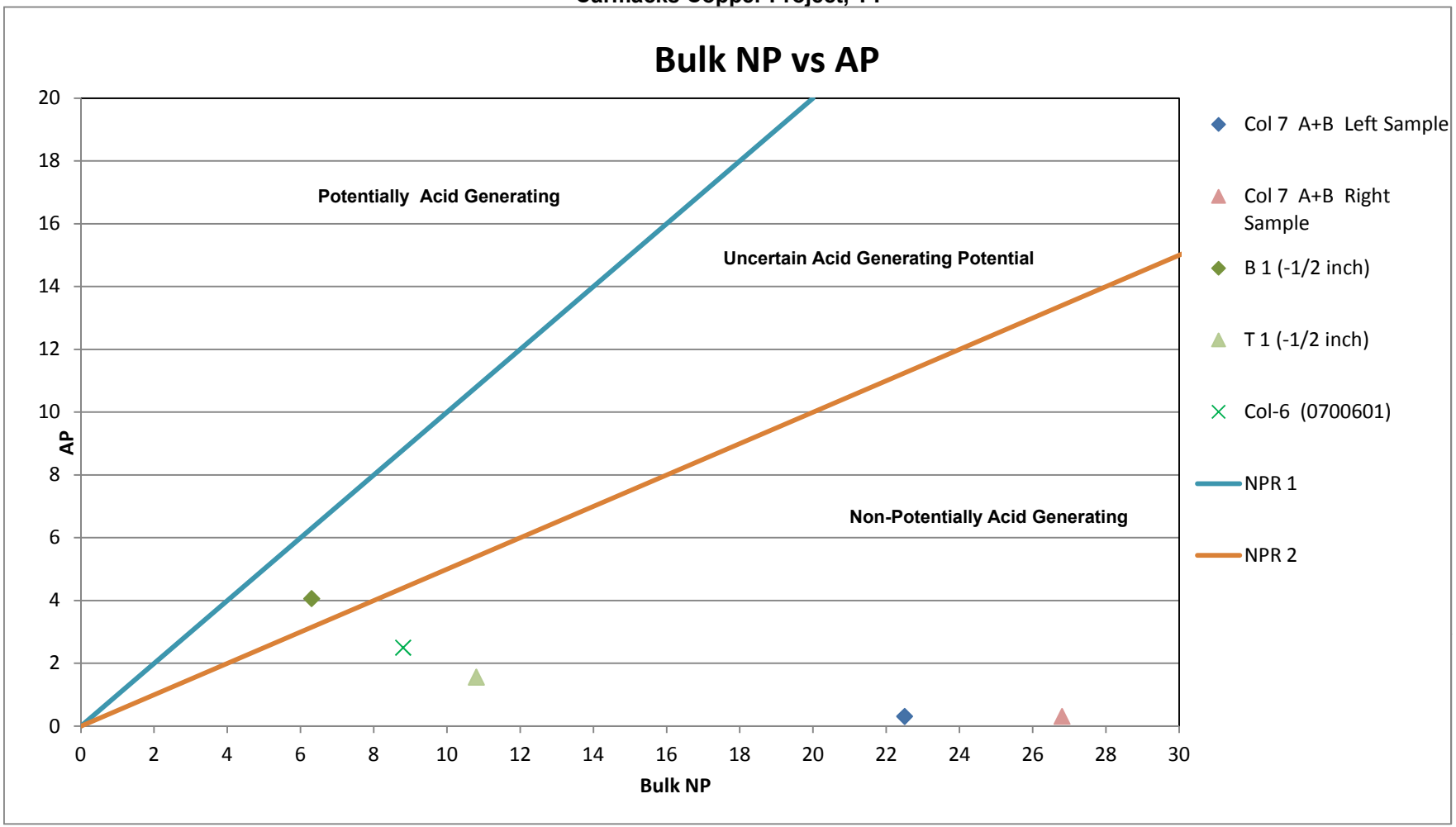


Figure 4-9
CaCO3 NP vs. AP for 2007 and 2008 Samples
Carmacks Copper Project, YT

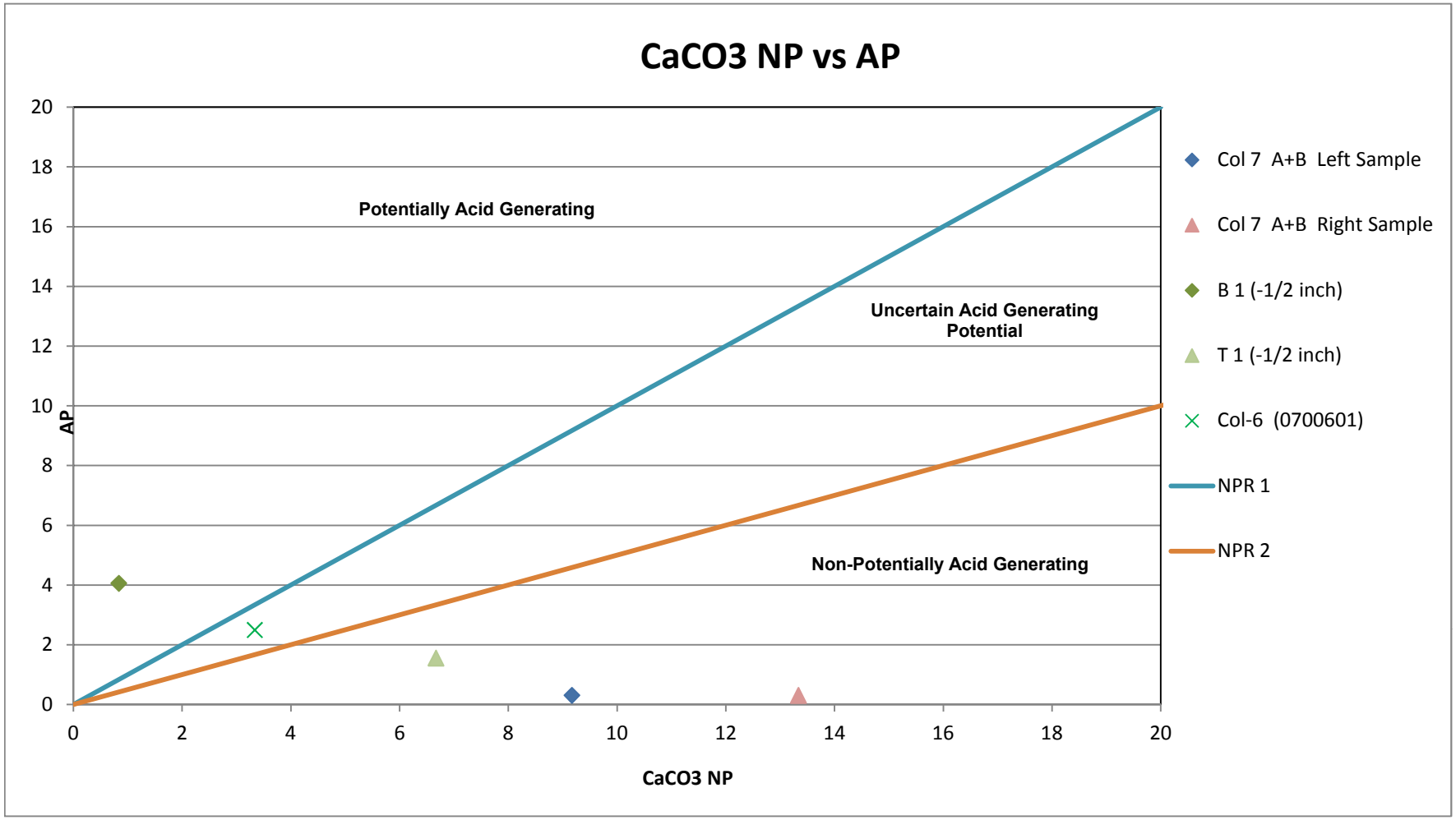


Figure 4-10
Sulphide Sulphur vs. NPR for 2007 and 2008 Samples
Carmacks Copper Project, YT

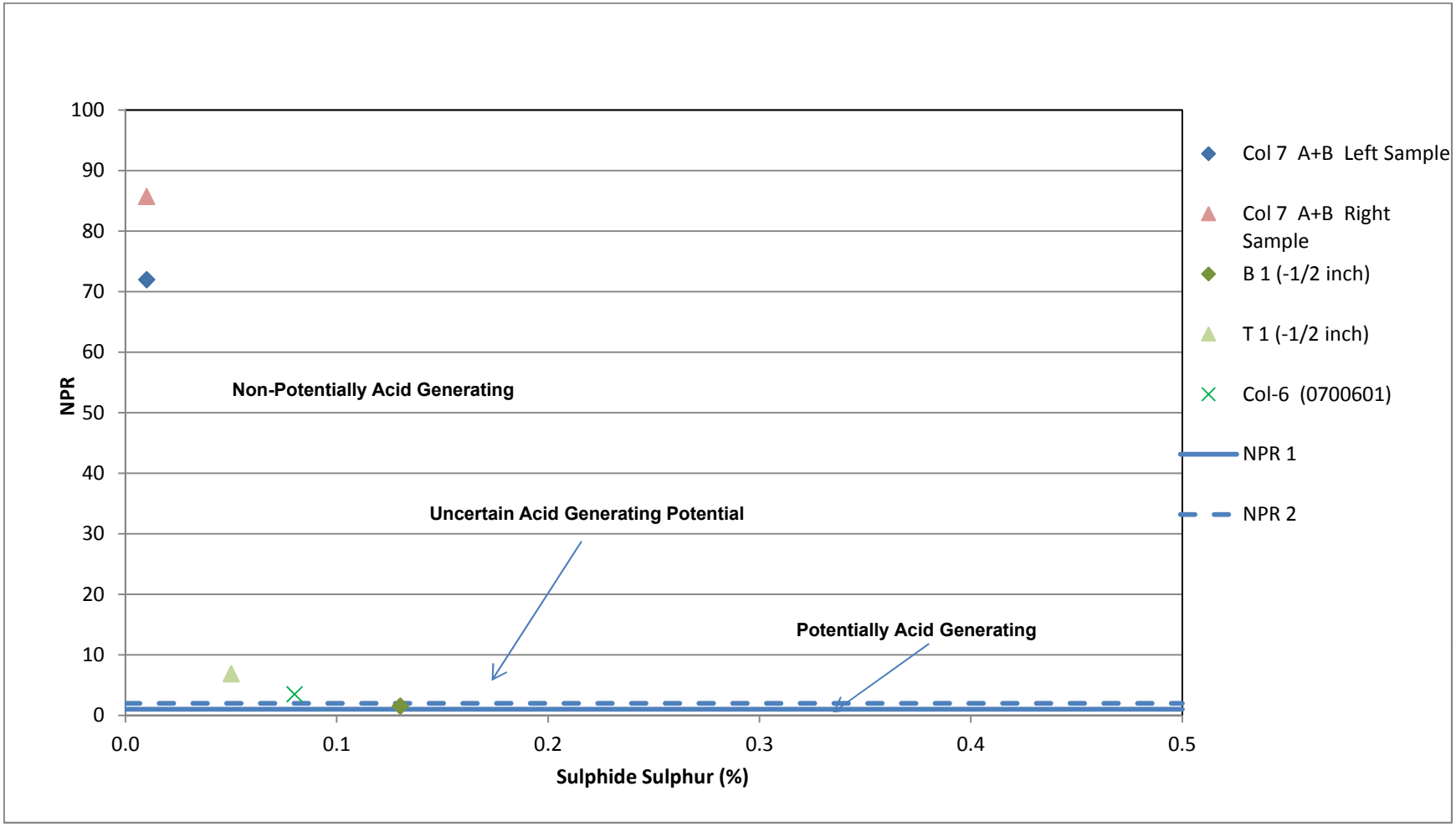


Figure 4-11
Sulphide Sulphur vs. NNP for 2007 and 2008 Samples
Carmacks Copper Project, YT

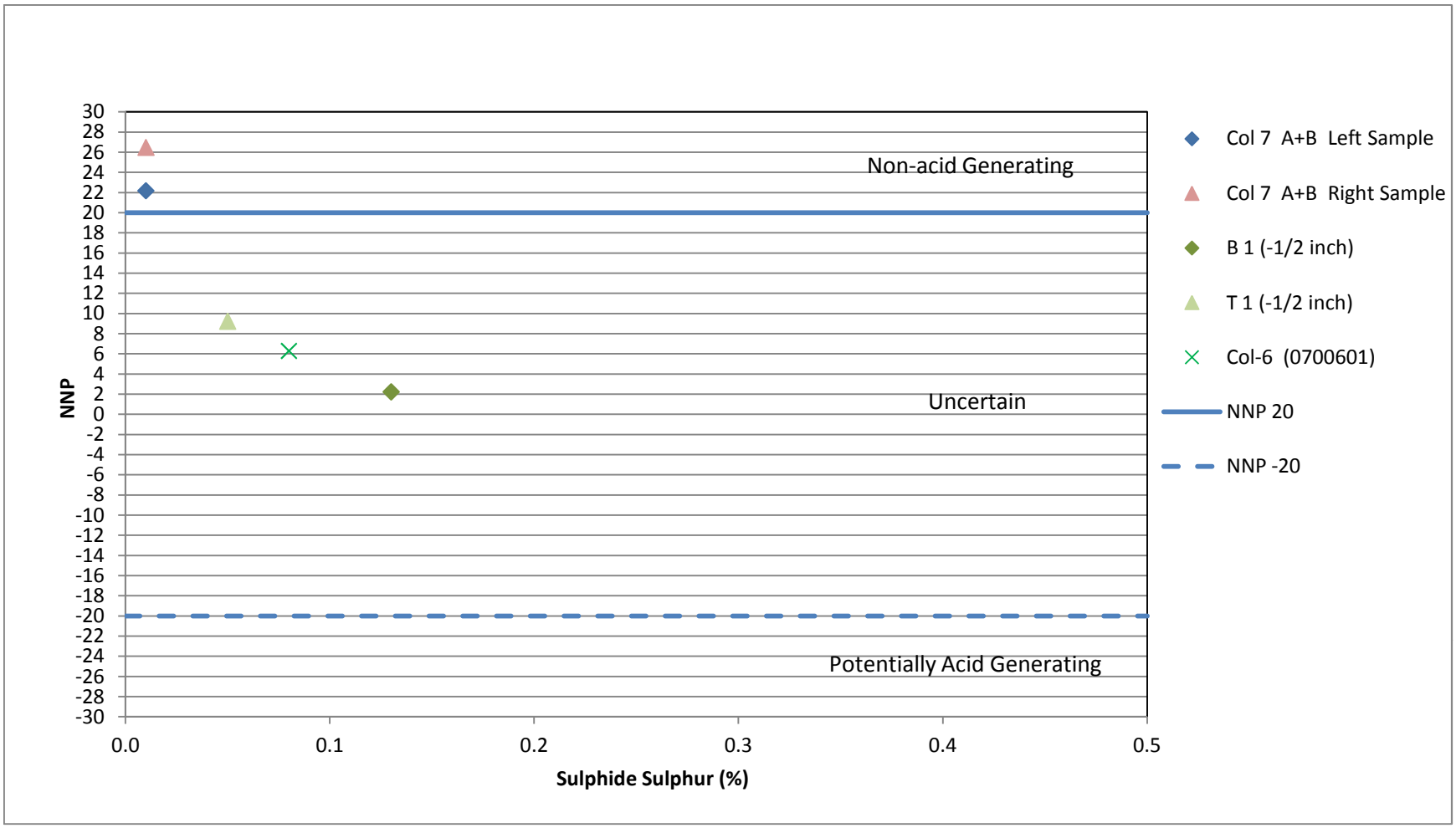


Figure 4-12
NNP vs. NPR for 2007 and 2008 Samples
Carmacks Copper Project, YT

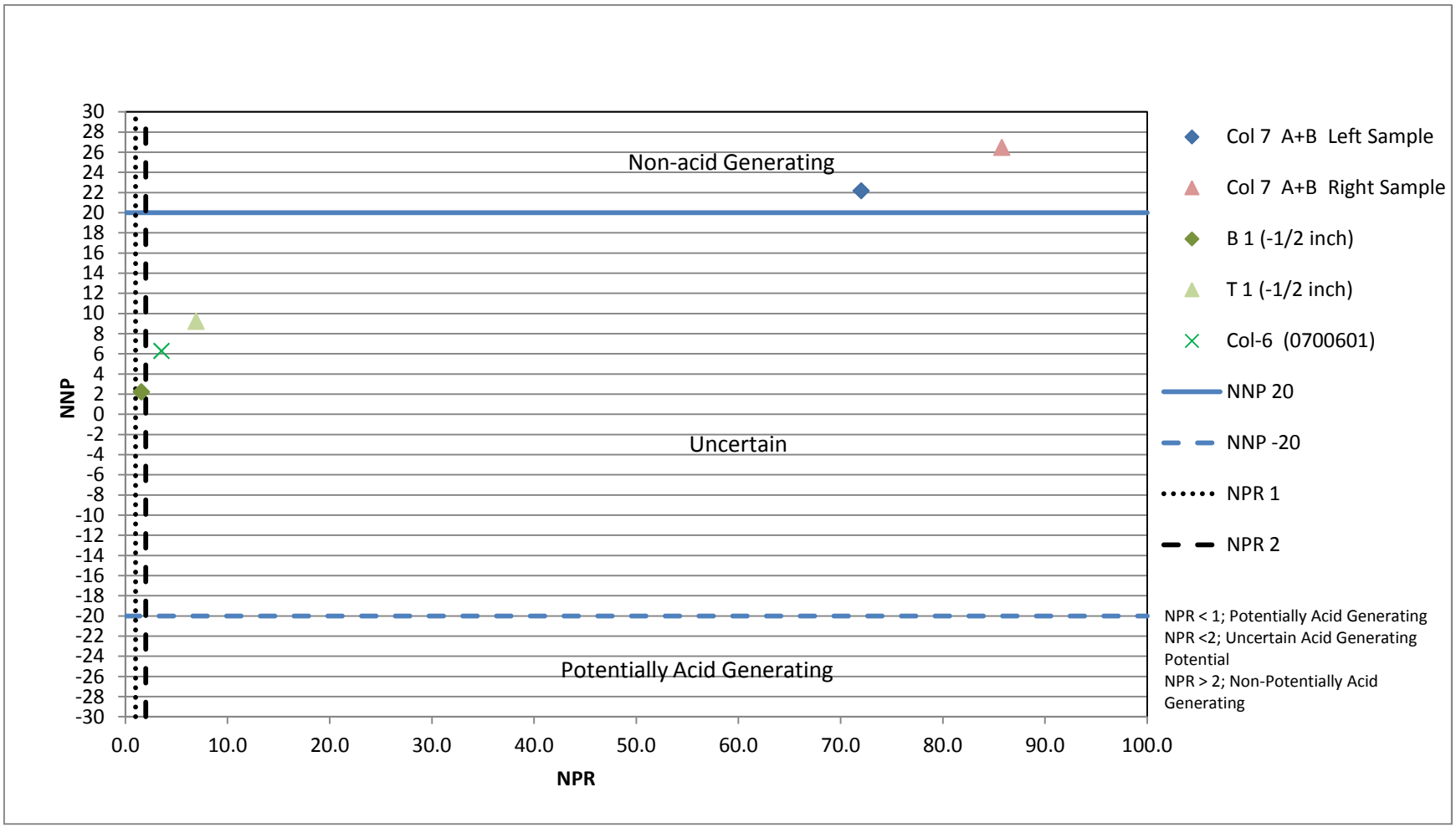


Figure 4-13
pH results for Spent Ore Kinetic Testing
Carmacks Copper Project, YT

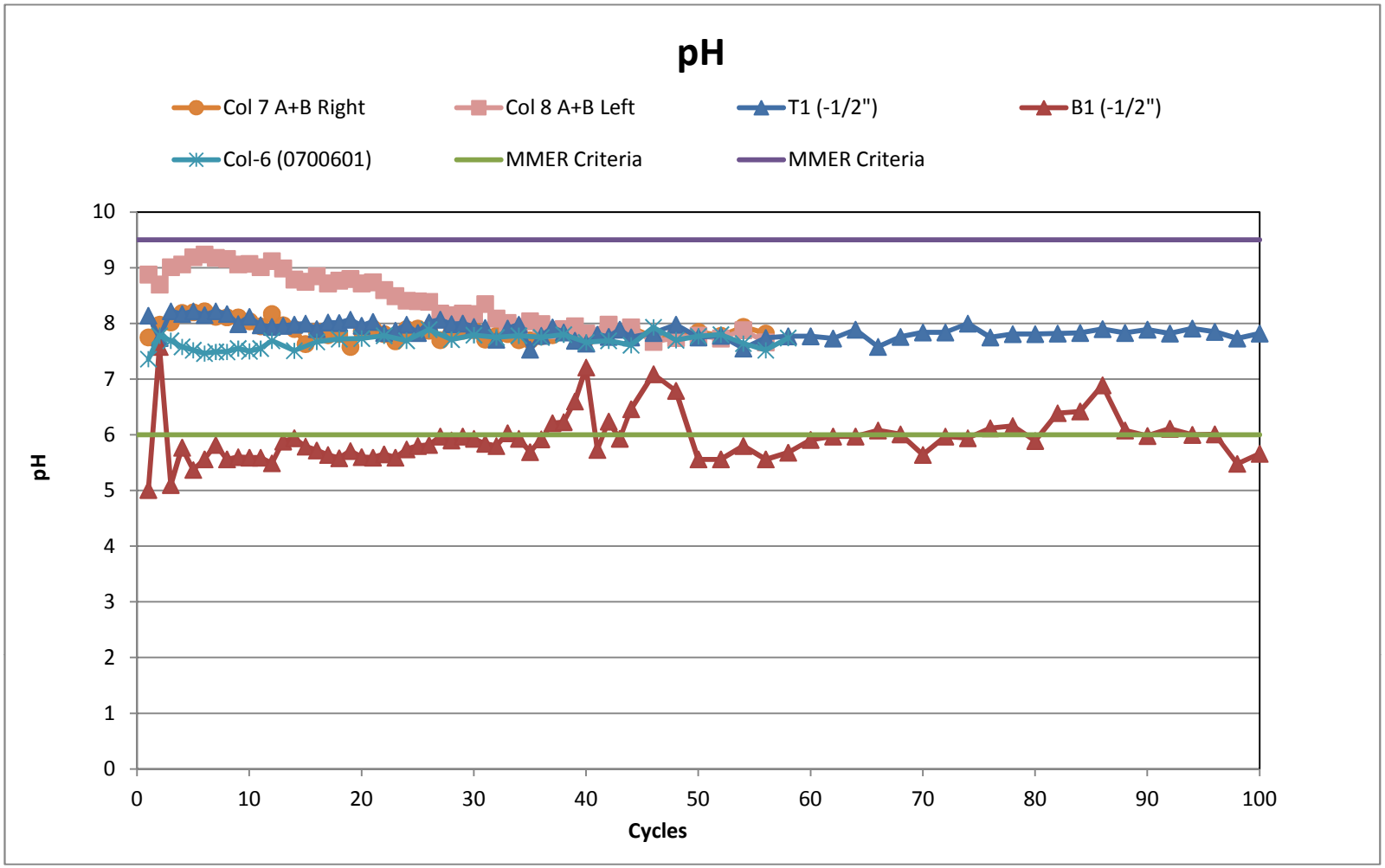


Figure 4-14
Alkalinity results for Spent Ore Kinetic Testing
Carmacks Copper Project, YT

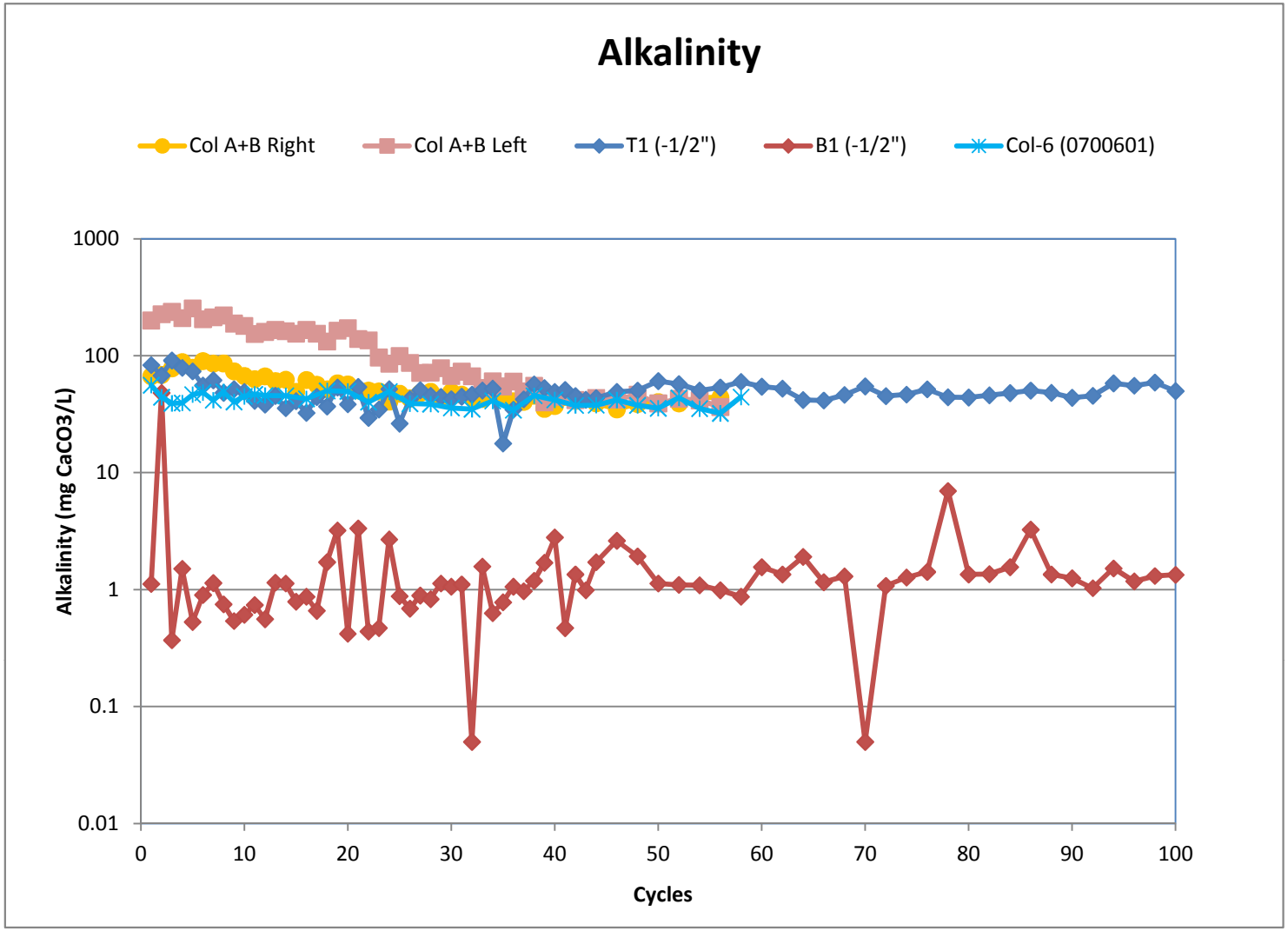


Figure 4-15
Alkalinity results for Spent Ore Kinetic Testing
Carmacks Copper Project, YT

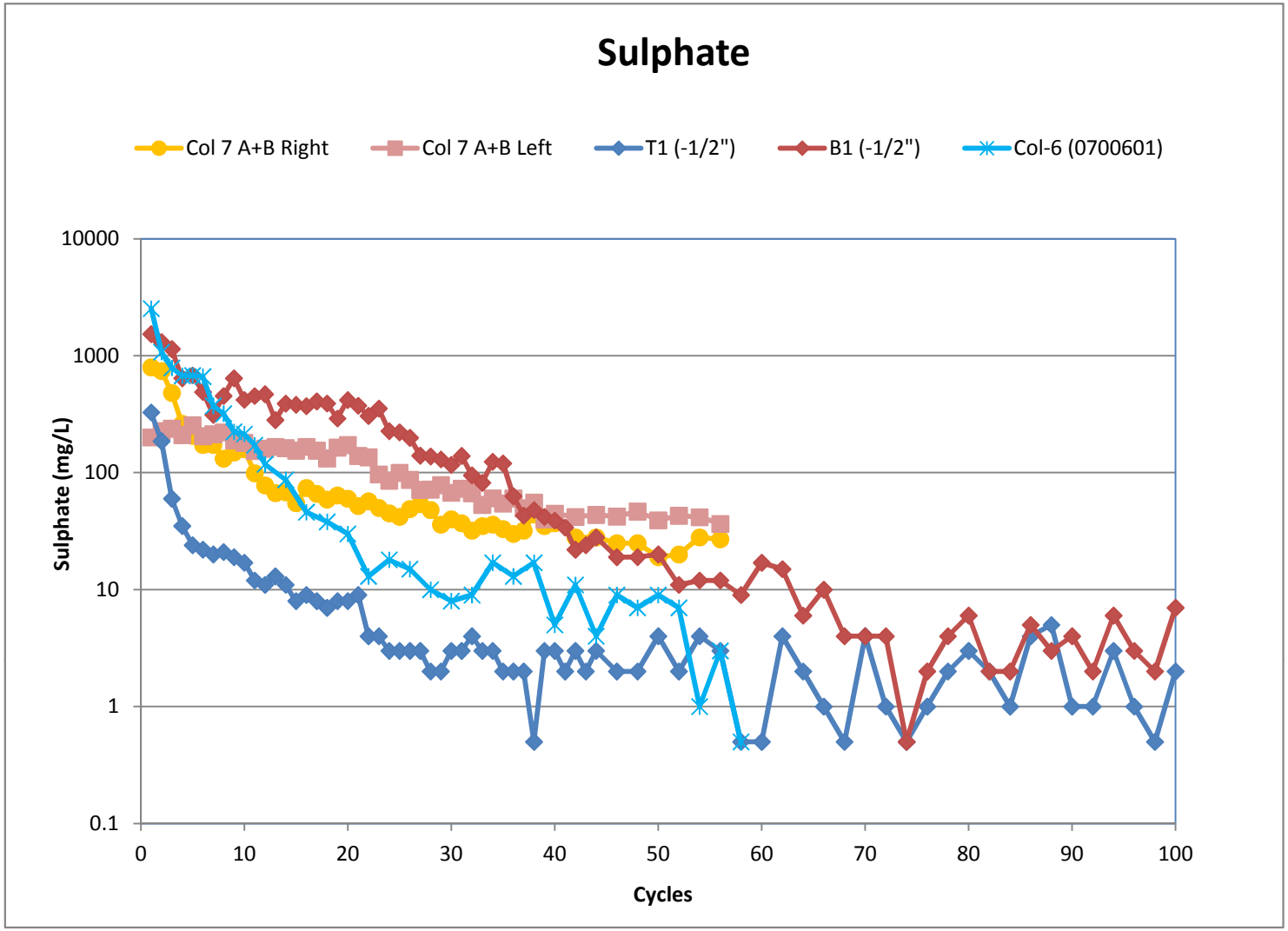


Figure 4-16
Arsenic Results for Spent Ore Kinetic Test
Carmacks Copper Project, YT

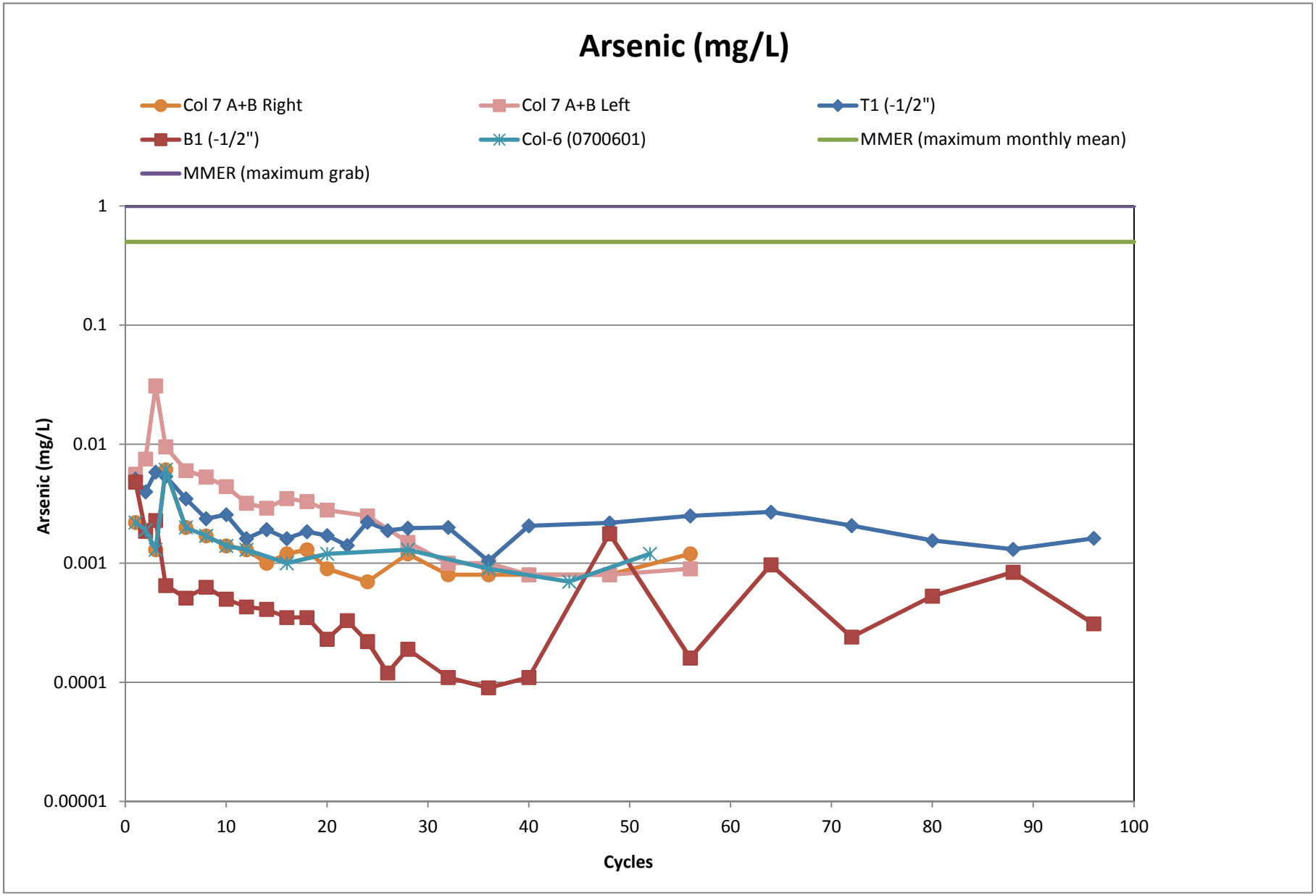


Figure 4-17
Copper Results for Spent Ore Kinetic Test
Carmacks Copper Project, YT

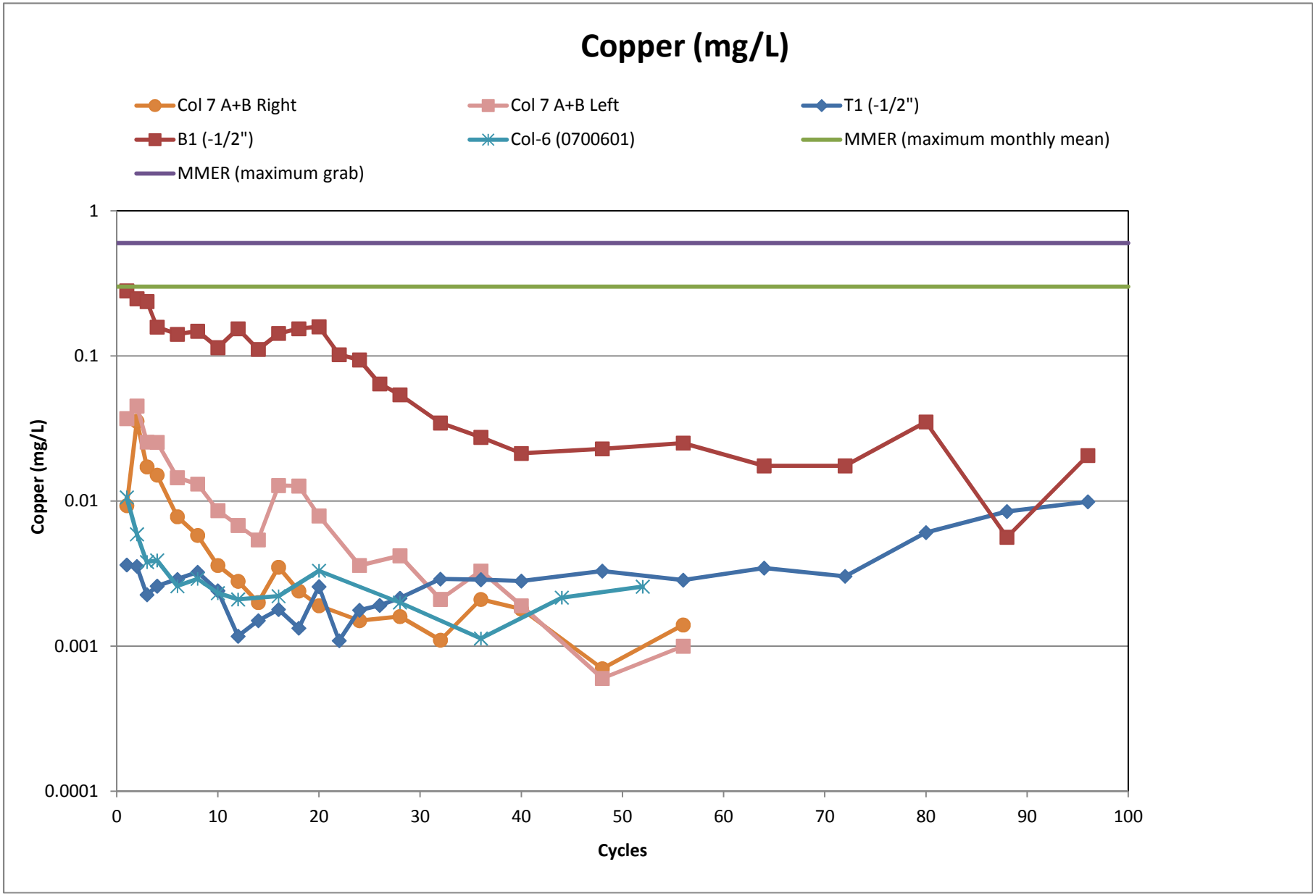


Figure 4-18
Nickel Results for Spent Ore Kinetic Test
Carmacks Copper Project, YT

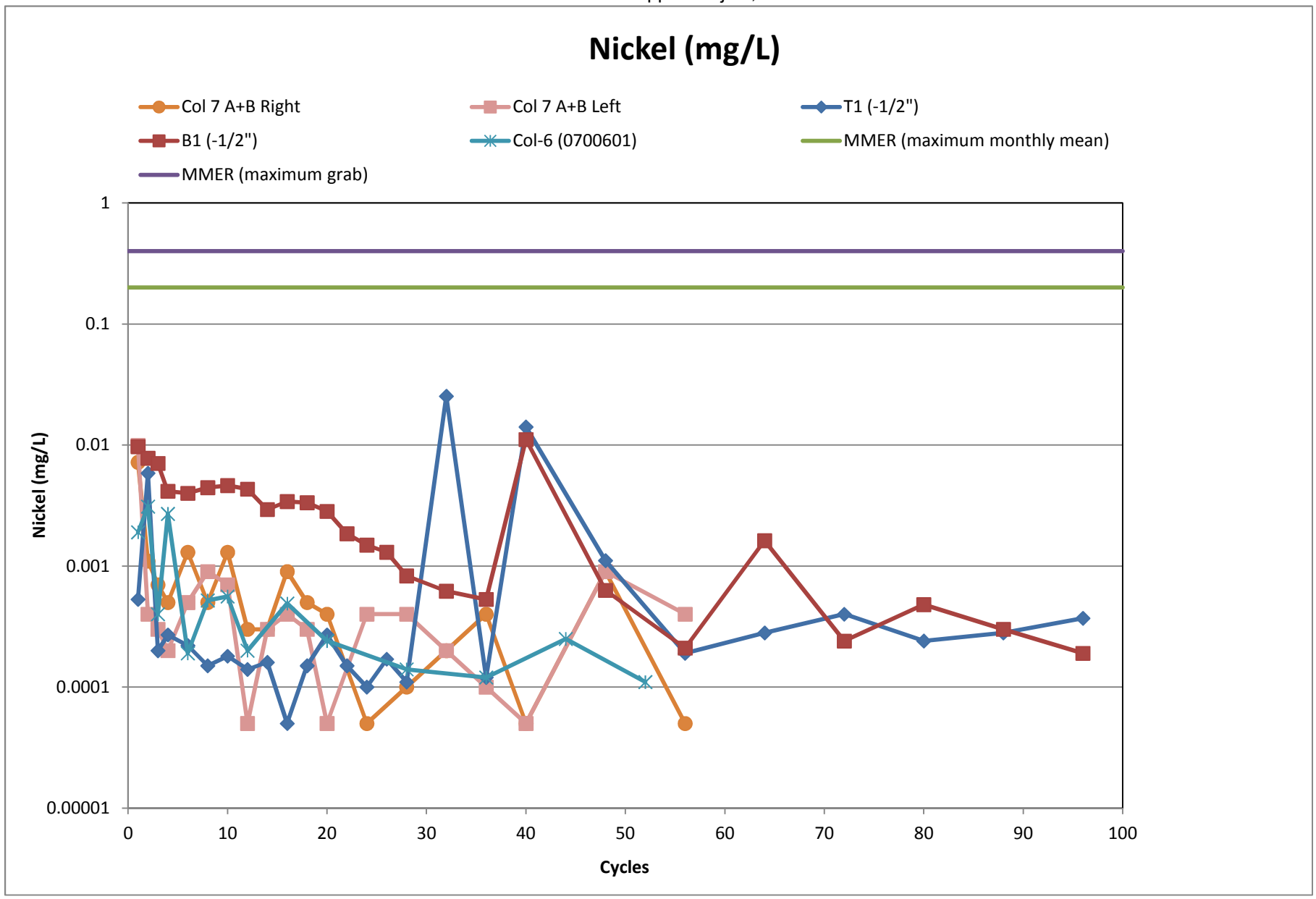


Figure 4-19
Lead Results for Spent Ore Kinetic Test
Carmacks Copper Project, YT

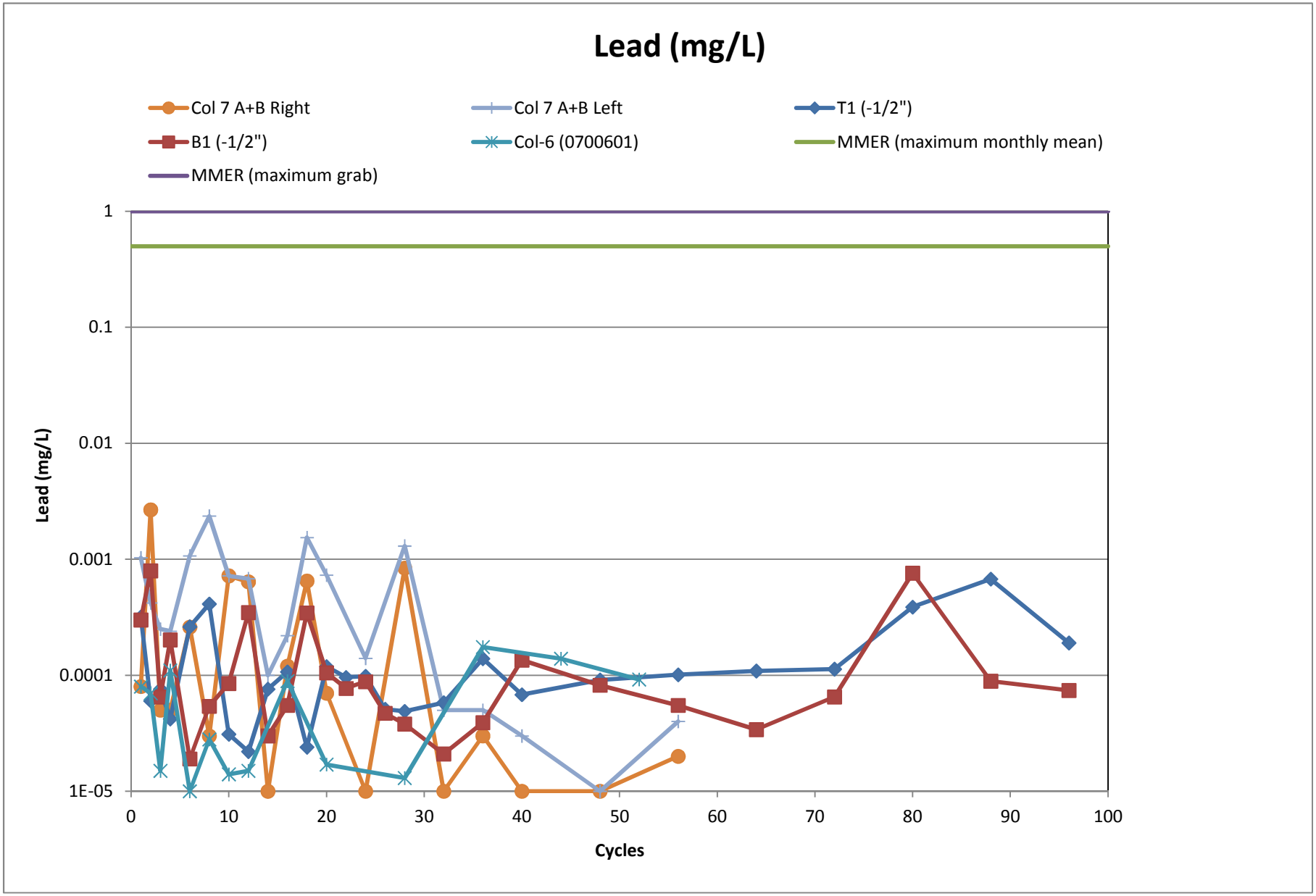


Figure 4-20
Zinc Results for Spent Ore Kinetic Test
Carmacks Copper Project, YT

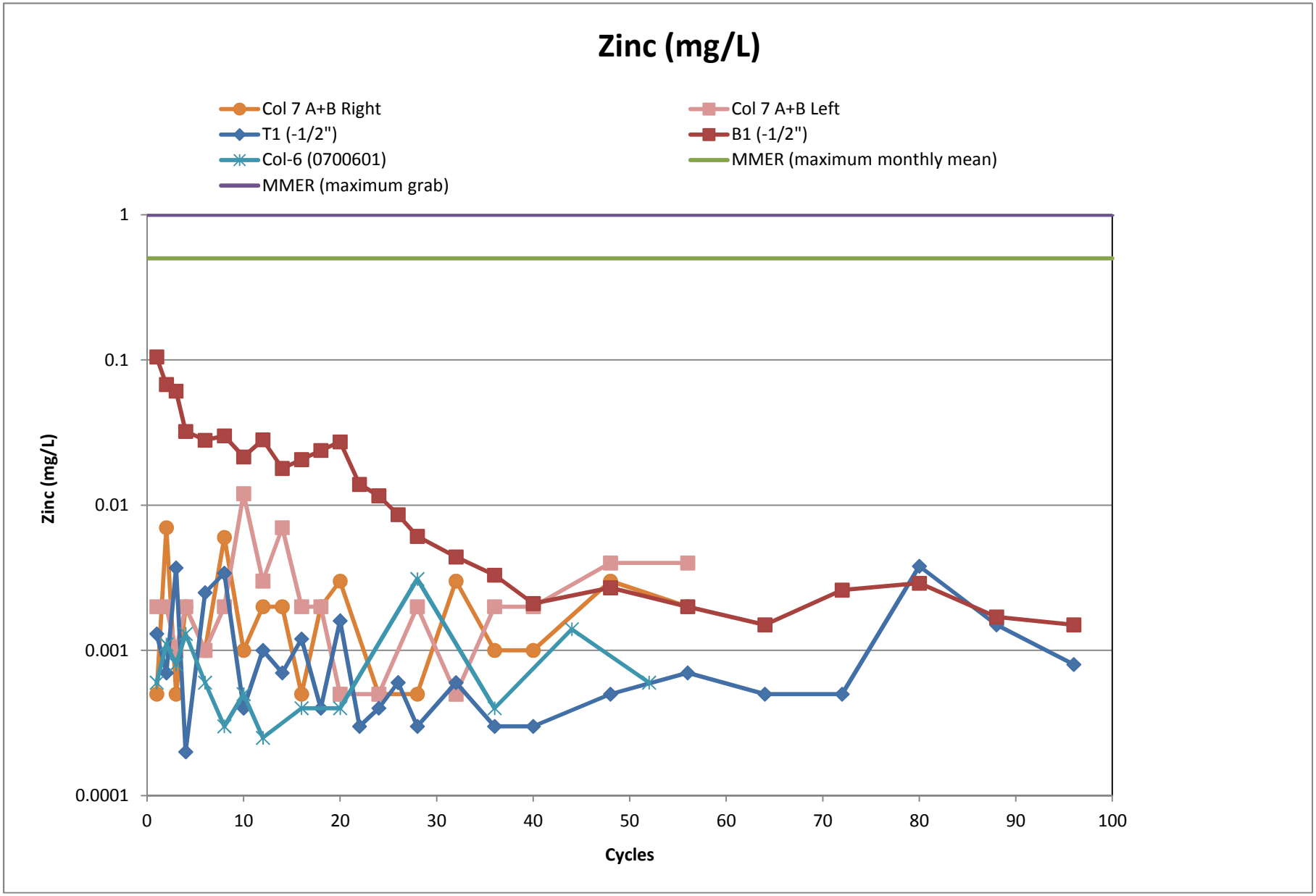


Figure 4-21
Antimony Results for Spent Ore Kinetic Test
Carmacks Copper Project, YT

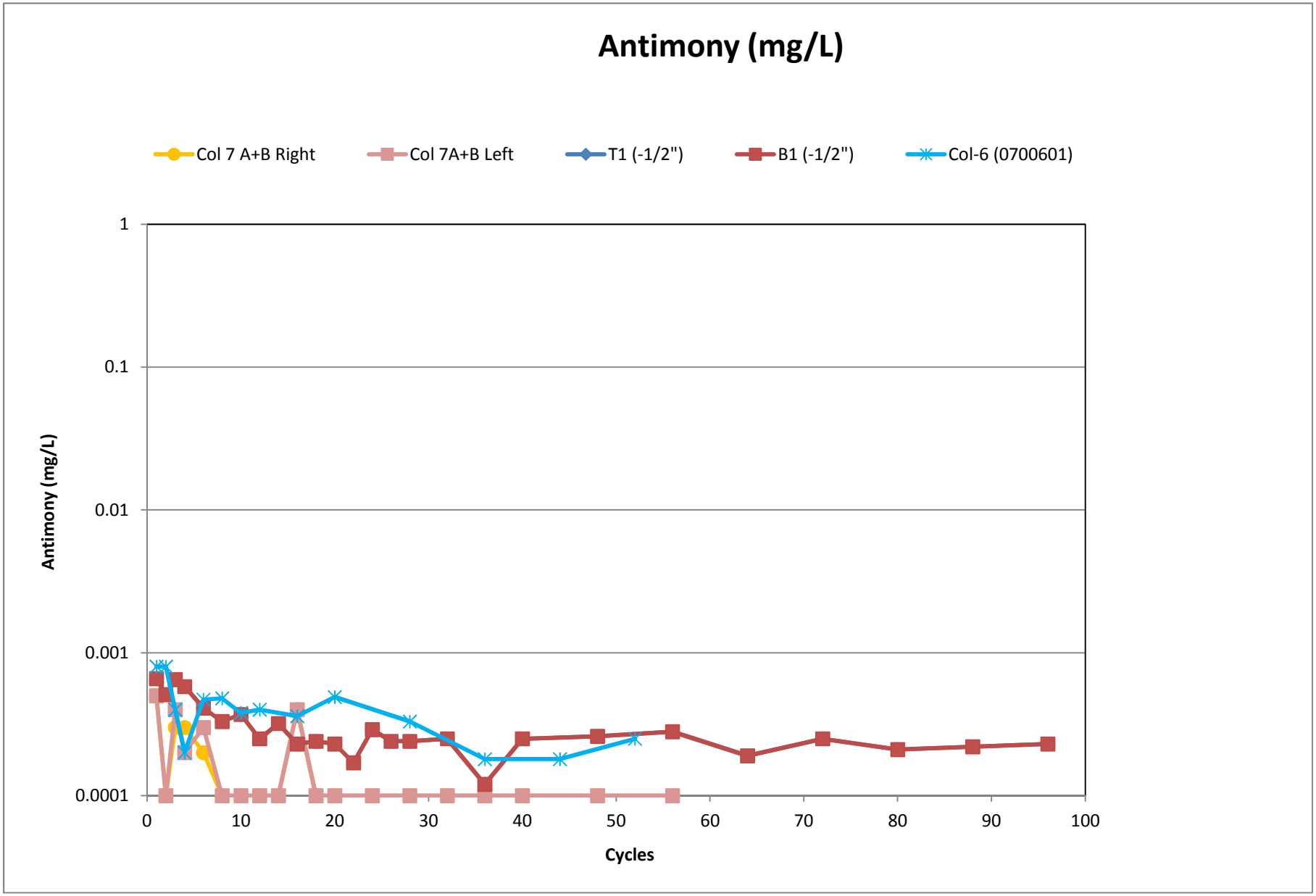


Figure 4-22
Cadmium Results for Spent Ore Kinetic Test
Carmacks Copper Project, YT

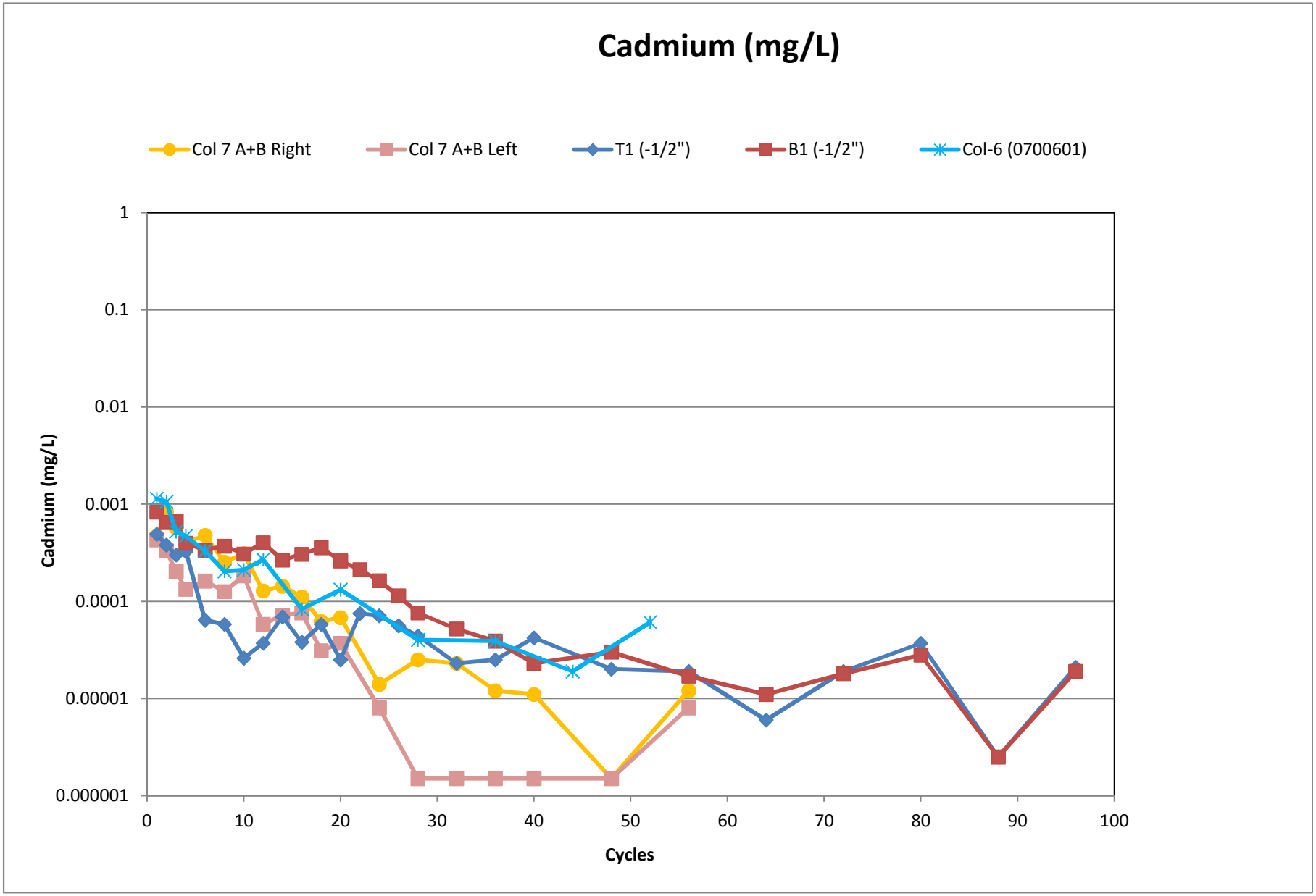


Figure 4-23
Molybdenum Results for Spent Ore Kinetic Test
Carmacks Copper Project, YT

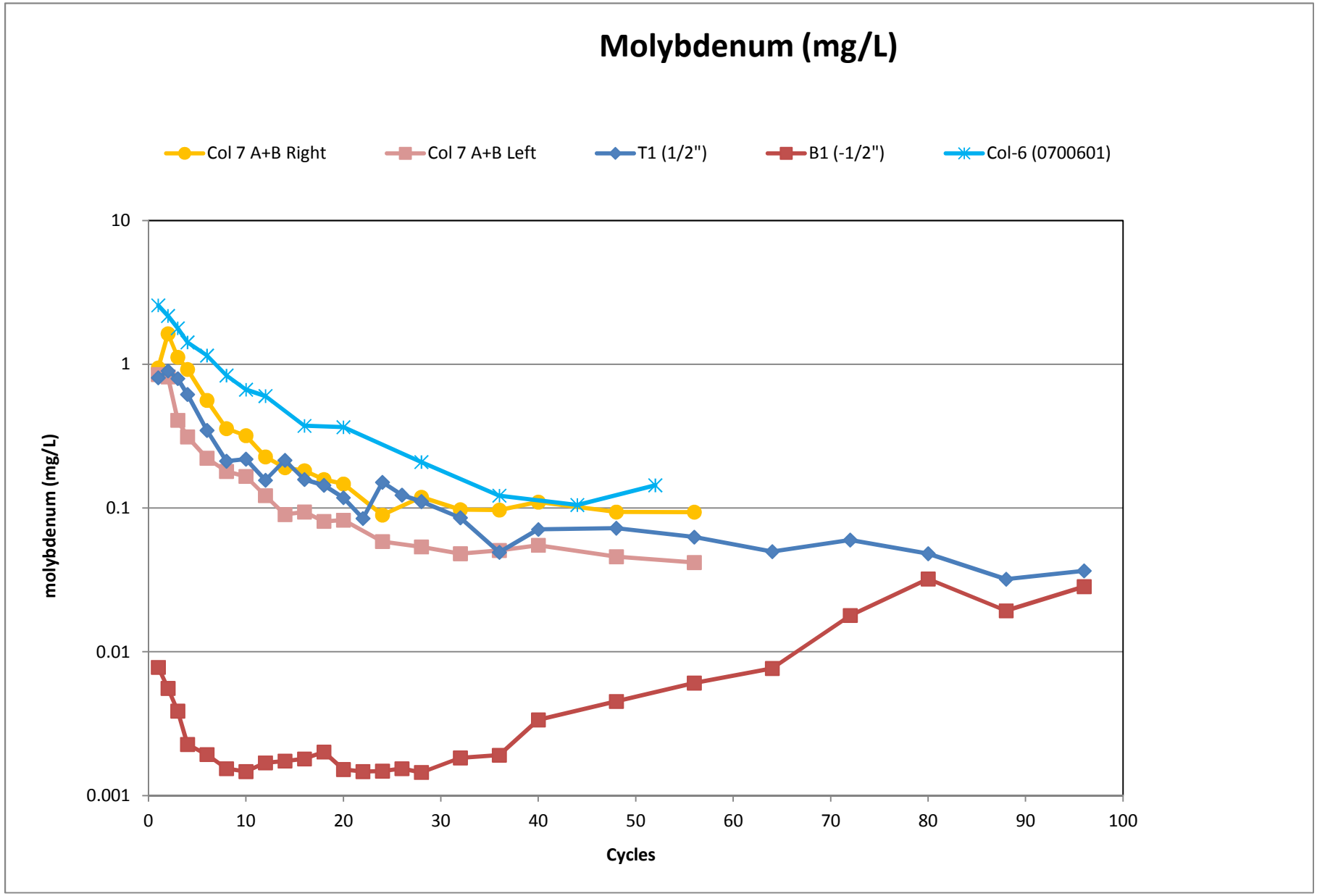


Figure 4-24
Mercury Results for Spent Ore Kinetic Test
Carmacks Copper Project, YT

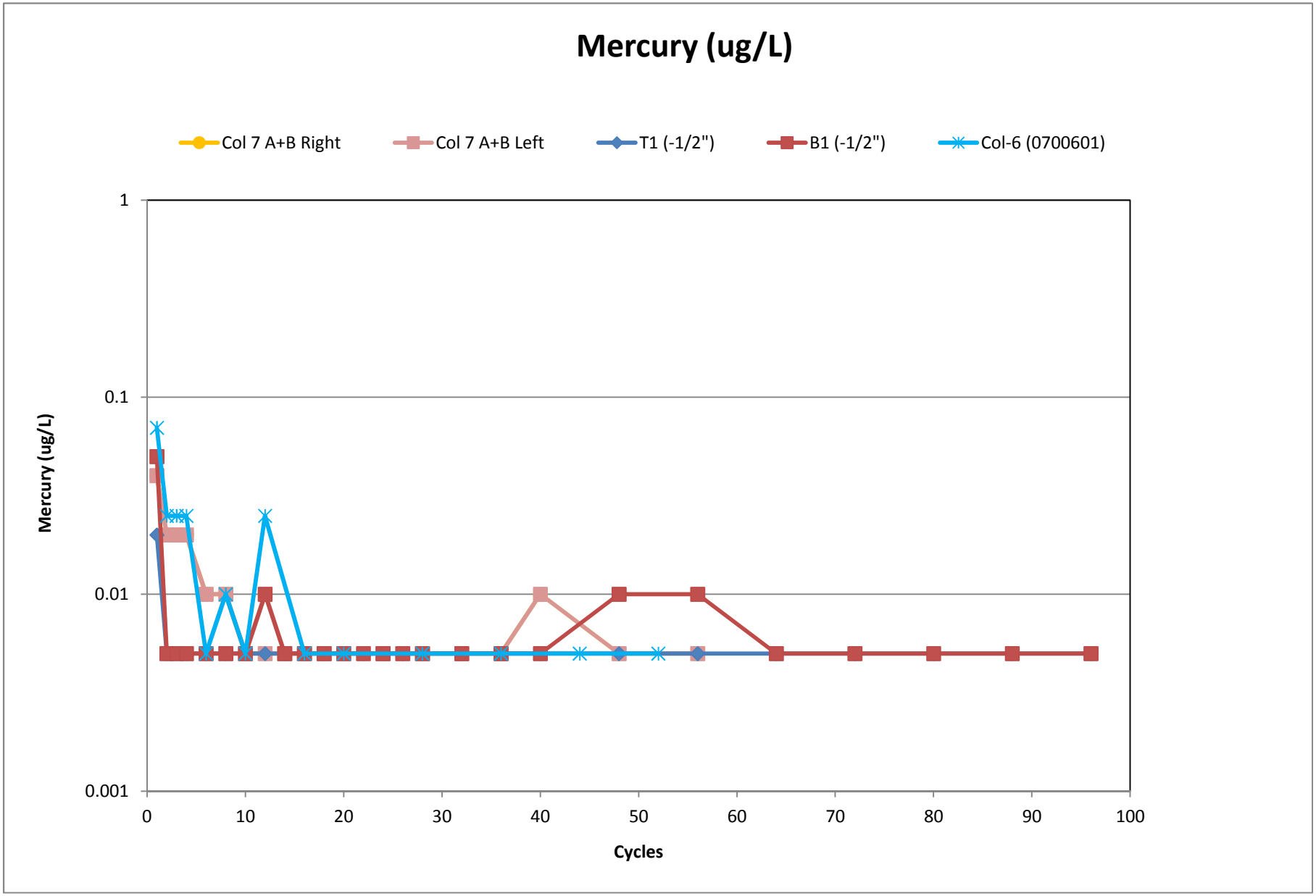
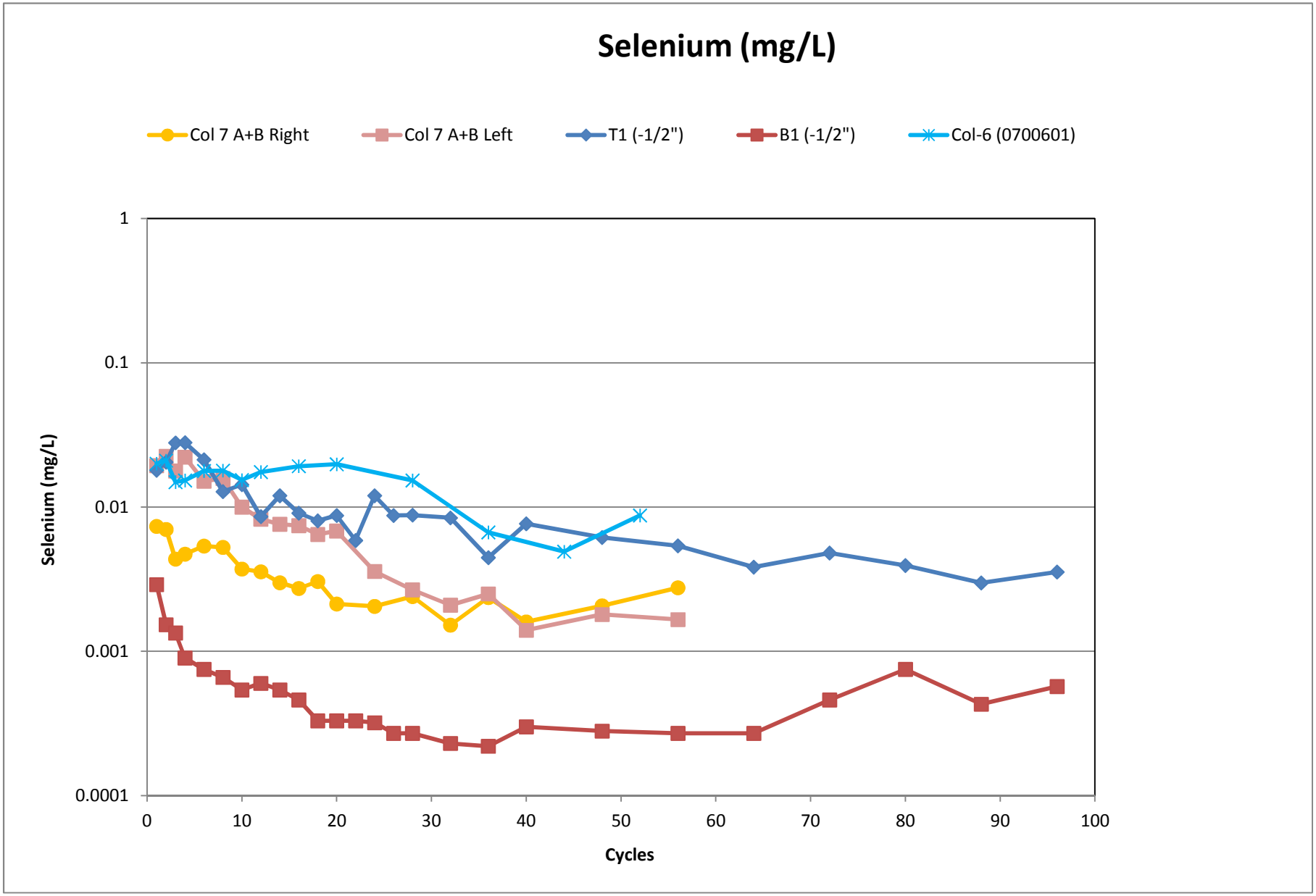


Figure 4-25
Selenium Results for Spent Ore Kinetic Test
Carmacks Copper Project, YT



APPENDIX I

Sampling Procedure

TECHNICAL MEMORANDUM

DATE August 17, 2010 **DOCUMENT No.** 07-1413-0077/9100

TO Ms. Fiona Esford
Golder Associates

FROM Steven Atkin, Ph.D., P.Geo. **EMAIL** satkin@golder.com

SAMPLE COLLECTION COLUMN 7, 2010 NEUTRALIZED SPENT ORE SAMPLES

1.0 INTRODUCTION

Dr. Steven Atkin met with Mr. Paul West-Sells of Western Copper Corp. (WCC) and Dr. Gie Tan of Inspectorate America Corporation, PRA Division, at the PRA laboratory, Richmond, BC, on August 13, 2010, to observe the unloading of neutralized spent ore from column test cells and to collect samples of neutralized spent ore for geochemical and geotechnical testing. This memorandum summarizes the procedures used for sample collection.

2.0 SPENT ORE RECOVERY

The metallurgical tests were conducted using two columns measuring 8 meters in length and 8 inches in diameter. The two columns, labelled column 7A and 7B, where connected in series such that when standing vertically, solution was applied to the top of column 7A and the discharge of column 7A was pumped to the top of column 7B. The final solution was recovered at the base of column 7B. The test material consisted of ore from the Carmacks Copper Project crushed to a maximum grain size of ¾ inch with a P80 of ½ inch (personal communication, 2010).

The columns were removed from their mounts in the lab and moved to a laydown area outside of the lab building. The columns containing the neutralized spent ore were then lifted using a forklift and unloaded on to a tarpaulin. Figure 1 shows the method used to lift and unload the column. Figure 2 shows the two sets of neutralized spent ore column loads displayed in rows as discharged from the columns.

3.0 SAMPLE COLLECTION

Two separate sets of samples were collected from the neutralized spent ore column loads: two (2) geochemical samples that were composites of columns 7A and 7B and four (4) geotechnical samples that were composites of the upper half of column 7A, the bottom half of column 7A, the top half of column 7B and the bottom half of column 7B.

A channel sample was cut along the length of each row for each sample. Figures 3, 4, 5, and 6 show the channel samples for the geochemical samples for the right sample (Figures 3 and 4) and the left sample (Figures 5 and 6). A larger sample was collected for the geotechnical testing. Figures 7 to 14 show the channel samples collected for the geotechnical work and are paired including the first pass channel sample (the "before" photo) and the final configuration of load after the actual sample was collected (the "after" photo). The differences between the "before" and "after" photos arise because of the large volume of sample required for the geotechnical testing and the need to make several sampling passes to collect enough neutralized spent ore to meet the required 20+ kg sample weight.

Once the samples were collected, each bag was labelled and weighed and packed in 20 L plastic buckets for shipment. Table 1 summarizes the mass of each of the geochemical and geotechnical samples.

Table 1: Table of sample sizes

	Geochemical samples		**Geotechnical samples**			
Sample	7 A+B Left	7 A+B Right	7A Top	7 A Bottom	7 B Top	7 B Bottom
Mass (kg)	11.9	12.5	23.02	20.18	20.47	21.47

4.0 SAMPLE DESCRIPTION

The neutralized spent ore was granular, consisting of fine and coarse grain-size fractions (Figure 15) and showed little evidence of grain size segregation when discharged from the column. Care was taken to insure that the samples were representative of all size fractions. The material appeared to be well drained and although moist, showed no evidence of saturation. The bottom of column 7B was slightly more competent than the neutralized spent ore from column 7A suggesting a higher moisture content.

The material was everywhere brown and strongly iron oxide stained. The upper portions of column 7A contained a white crust (Figure 3). Figure 16 is a close-up view of crystals that makeup the crust, which may be gypsum. Although the finer size fraction may be completely leached, the coarser fragments show only a thin veneer of oxidation. Figures 17 and 18 are close-up views of gneiss and granodiorite fragments, respectively, showing the depth of oxidation. Although the interior of the granodiorite remained generally unaltered, oxidation, either supergene or a result of leaching, extended along fractures or along foliation in the gneiss.

5.0 CLOSING

We trust the above summary of the sampling conducted at the PRA laboratory provides adequate documentation of the procedures used in securing the materials for future geochemical and geotechnical test work. If you have any questions about the sampling program, please do not hesitate to contact me.

Steven A. Atkin, Ph.D., P.Geo.
Senior Geochemist

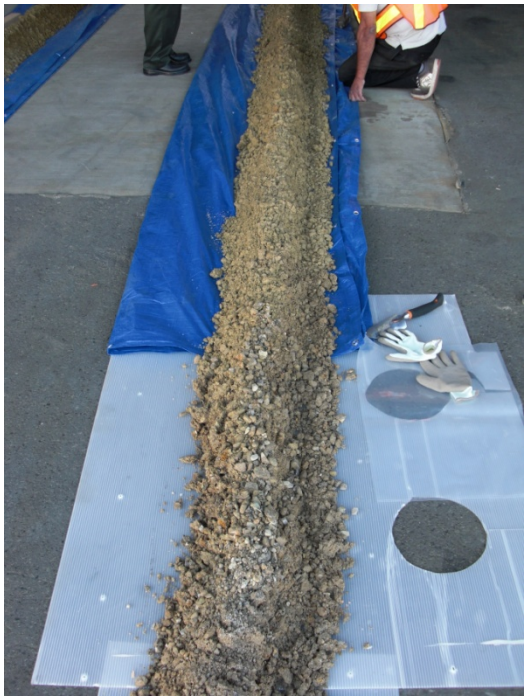
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Photograph 1: Unloading column



Photograph 2: Column 7A (left) and 7B (right) (view from bottom of column)



Photograph 3: Right geochemical sample Column 7A (view from top of column)



Photograph 4: Right geochemical sample Column 7B (view from top of column)



Photograph 5: Left geochemical sample Column 7A (view from bottom of column)



Photograph 6: Left geochemical sample Column 7B (view from bottom of column)



Photograph 7: 7A top geotechnical sample (before sampling)(view from bottom of column)



Photograph 8: 7A top geotechnical sample (after sampling)



Photograph 9: 7A bottom geotechnical sample (before sampling)



Photograph 10: 7A bottom geotechnical sample (after sampling)



Photograph 11: 7B top geotechnical sample (before sampling)



Photograph 12: 7B top geotechnical sample (after sampling)



Photgraph 13: 7B bottom geotechnical sample (before sampling)



Photograph 14: 7B bottom geotechnical sample (after sampling)



Photograph 15: Grain size of column discharge



Photograph 16: White crust upper column 7A



Photograph17: Gneiss fragments with thin leached rind



Photograph 18: Granodiorite fragments with thin leached rind

APPENDIX II

Historic Geochemical Testing and Test Protocols

February 15, 2012
Project No. 07-1413-0077/9100
Doc. No. 173 Rev. 0

Golder Associates

1.0 HISTORIC GEOCHEMICAL TESTING

Geochemical testing of waste rock and ore samples began in the mid 1990's and has been supplemented with additional testing as the project has evolved. Table II-1 and II-2 list the various test program that have taken place over the years, the tests conducted and the number and types of test per test program.

Table 1.1: Summary of Historic and Current Geological and Geochemical Investigations of Waste Rock

Geochemical Test Protocol	Hallam Knight Piesold 1995	ACG Consulting Group 2006	Golder Associates Ltd. 2007	Golder Associates Ltd. 2008
Petrography	n/a	27	n/a	n/a
Acid Base-Accounting (ABA)	23	27	7	8
Whole Rock	23	27	7	8
24 hour Leach Tests	6	27	7	8
Tessier Extraction	1	n/a	n/a	n/a
Kinetic Test	n/a	n/a	6	n/a

Table 1.2: Summary of Historic Geological and Geochemical Investigations of Ore and Spent Ore

Studies	Hallam Knight Piesold Ltd. 1995	Beattie Consulting Ltd. 1994	Beattie Consulting Ltd. 1996	Beattie Consulting Ltd. 1998	Beattie Consulting Ltd. 2001	Alexco Resource Corp. 2006	Golder Associates Ltd. 2008	Golder Associates Ltd. 2010
	Raw Ore	Metallurgical Leach/Neutralized Leached Ore						
Test Material	Core and Trench Samples	Core and Bulk Surface Samples	Core and Pilot Heap Leach Plant Trench Samples	Trench Samples (Kilborn)	Trench Samples (Kilborn)	Unspecified Material (Source Process Research Ass.)	Trench Samples (ACG); Rinsed and Neutralization (PRA)	Trench Samples (ACG); Rinsed and Neutralization (PRA)
Geochemical Test Protocol	10 ABA, 3 BC LEP	None	Mineralogy Prior to Test	ICP Prior to Test	None	2 ABA, 4 Petrography	Petrography, ABA, Whole Rock, 24 hour Leach Test, Kinetic Test (Humidity Cell)	Petrography, ABA, Whole Rock, 24 hour Leach Test, NAG/NAG leachate, Kinetic Test (Humidity Cell)

2.0 TESTS PROTOCOLS

2.1 Whole Rock Analysis

The whole rock major and trace element chemical composition of mineralized rock is used to identify samples with anomalous (e.g., elevated or depleted) concentrations of metals when compared to equivalent, unmineralized rock. The application of whole rock analyses is based on the assumption that water in contact with unmineralized rock will have a "normal" chemical composition, meeting standard water quality guidelines. Elevated whole rock metal concentrations suggest the increased potential for metal leaching and elevated metal concentrations in the adjacent water.

The chemical composition of each of the samples was determined by whole rock analysis using 1) lithium metaborate fusion and nitric acid digestion for major element concentrations and 2) multi-acid digestion for trace element analyses. Whole rock analysis of major rock-forming elements provides total elemental weight percent compositions as oxides (*e.g.,* Na_2O, CaO, SiO_2, etc.) and is used to determine the general characteristics of different lithologies. Multi-acid digestion provides elemental concentrations of trace metals as well as major elements. Multi-acid digestion of samples prior to ICP analysis is aggressive, however, the process may leave trace amounts of solids undissolved and therefore not analyzed. Thus, lithium metaborate fusion analytical results are considered to be more representative of the actual concentration of major elements as compared to major element results from multi-acid digestion.

2.2 Shake Flask Extraction (SFE)

The shake flask extraction test (SFE) is a short term leaching test (24 hours or less) that measures the solubility of acids and metals produced from readily solubilized mineral acidity and soluble salts (*e.g.,* oxidation products), respectively, within the sample.

The SFE test uses 250 g of minus 9.52 mm (3/8 inch) crushed material is mixed with 750 ml of deionized water (3:1 water to rock ratio). The water and rock mixture is then agitated on a shaker table for 24 hours. The resulting leachate is filtered and analyzed for; pH, conductivity, acidity, alkalinity, sulphate, and low-level metals (by ICP-MS).

2.3 Acid Base-Accounting (ABA)

ABA tests were performed to evaluate the ARD potential of the waste rock and neutralized spent ore material. ABA testing is a screening level tool that is used to estimate the amount of acid generating and acid neutralizing material in a sample. It is designed to provide the following:

- The maximum acid generation potential of a material assuming all of the sulphide sulphur can be attributed to pyrite and is allowed to oxidize; and
- The maximum acid neutralization potential of the material assuming all acid buffering capacity of a sample, as fast acting carbonate neutralizing potential, is allowed to react.

ABA tests were performed to evaluate the ARD potential of the neutralized spent ore material. The samples underwent a standard suite of ABA analyses, including the measurement of the paste pH, sulphur species (total sulphur - S_T, sulphate sulphur – S_{SO4}, and sulphide sulphur - S_{S-2}), total inorganic carbon (TIC), and carbonate and bulk neutralizing potentials ($CaCO_3$ NP and NP, respectively). From these parameters, derived functions were calculated to estimate the ARD potential of the samples: acid generating potential (AP), net neutralizing potential (NNP = NP – AP or $CaCO_3$ NNP = $CaCO_3$NP – AP) and acid neutralizing potential ratio (NPR = NP/AP or $CaCO_3$NPR = $CaCO_3$NP/AP).

The ABA analyses are conducted using the modified Sobek method. Paste pH is the pH measured for a paste consisting of water and crushed rock. Total sulphur (S_T) concentration is analyzed by pyrolysis and infrared spectroscopy (using a LECO furnace), and acid soluble sulphate sulphur (S_{SO4}) is measured by ICP analysis after reaction with HCl. Sulphide sulphur (S_{S-2}) is calculated by difference using S_T and S_{SO4} ($S_{S-2} = S_T - S_{SO4}$). S_{S-2} content is then used to calculate the acid generating potential (AP). Bulk NP is measured by titrating the sample with a strong acid and measuring the amount of strong base required to return the solution to a specified pH.

2.4 Net Acid Generation (NAG) Test and NAG Leachate Analysis

NAG tests use hydrogen peroxide to promote rapid sulphide mineral oxidation and ideally, generate leachates that are representative of “terminal” conditions. The pH of the leachate at the end of the test is termed the NAGpH. A portion of the leachate is titrated with NaOH to pH 4.5 and 7, NAG Acidity to pH 4.5 and NAG Acidity to pH 7, respectively. The acidity measured to pH 4.5 is generally believed to represent acidity due to iron, aluminum, and hydrogen ions; the acidity measured between 4.5 and 7 is attributed to soluble metals such as copper and zinc (Miller, 1998). The balance of the leachate is then analyzed for the same elements as the SFE test. The NAG leachate analysis is an indicator of the metals that can be released from sulphide mineral oxidation. However, it is a conservative estimate of the metals concentrations for naturally weathered sulphide mineral-bearing rock because all of the metals are released in a single “slug” rather than released slowly over time.

2.5 Mineralogy

The neutralized spent ore samples were analysed for their mineralogy using the Rietveld method of quantitative X-Ray Diffraction (XRD) analysis (see Attachment II). The samples were reduced <10 µm by grinding under ethanol in a vibratory McCrone Micronising Mill for 7 minutes. Step-scan X-ray powder-diffraction data were collected over a range 3-80°2θ with CoKα radiation on a standard Siemens (Bruker) D5000 Bragg-Brentano diffractometer equipped with an Fe monochromator foil, 0.6 mm (0.3°) divergence slit, incident- and diffracted-beam Soller slits and a Vantec-1 strip detector. The long fine-focus Co X-ray tube was operated at 35 kV and 40 mA, using a take-off angle of 6°.

The X-ray diffractograms were analyzed using the International Centre for Diffraction Database PDF-4 using Search-Match software by Siemens (Bruker). X-ray powder-diffraction data were refined with Rietveld program Topas 3 (Bruker AXS).

2.6 Kinetic Test

The humidity cell test, often called the accelerated weathering test, is a long term laboratory protocol designed to simulate the natural cycles of precipitation that occur in mine facilities. The long duration of the test coupled with the short cycle times of flushing followed by rest periods (typically one week) promotes the aggressive oxidation of metal-bearing sulphide minerals, acid production, and metals leaching.

The results of humidity cell testing can be compared with water quality guidelines but are most often applied to predictive geochemical modeling to estimate the long term discharge water chemistry from waste rock piles or other mine facilities and pit wall runoff for pit lake water chemistry estimates. Although ABA analyses are commonly used to estimate the acid generating and neutralizing potential of mined material, humidity cell tests are considered to be a more reliable measure of the on-set of sulphide oxidation and acid generation, particularly for low sulphur and low NP systems where silicate neutralizing potential may be significant (Maest et al., 2005).

Humidity cell tests use a standard test format. A 1 kg sample of 80% minus 6.35 mm (¼ inch) material is placed in a humidity cell vessel and flushed with 740 mL of demineralised water (cycle 1). The leachate is recovered, filtered, and preserved for analysis. Weekly cycles follow in which dry air is passed over and through the sample for three days, followed by moist air passed over and through the sample for three days, followed by flushing of the sample with 500 mL of demineralised water on the seventh day. The weekly cycle is repeated for a minimum of 20 weeks or until the leachate achieves a steady state chemical composition.

Weekly samples for this study are analyzed for pH, conductivity, acidity, alkalinity, and sulphate up to cycle 40 and every two cycles thereafter. A full range of anions and cations that include the weekly parameters plus 35 major and trace element concentrations were collected for cycles, 1, 2, 3, 4, 6, 8, 10, 12, 14, 16, 18, 20, 24, 28, 32, 36, and 40 and every 8 cycles (*i.e.*, cycle 48, 56 etc.) thereafter. A grain size analysis of the samples is also conducted to estimate the reactive surface area of the test material in the cell.

3.0 COMPARATIVE GUIDELINES

3.1 Metal Leaching Potential

Results of the SFE tests, the NAG leachate analyses, and the leachate from the humidity cells were compared with the Metal Mining Effluent Regulations (MMER) and the Canadian Council of Ministers of the Environment (CCME) fresh water aquatic life water quality standards for reference purposes only. The MMER guidelines are the federal environmental regulations applied to all metal mines (except placer mines) in Canada. The CCME fresh water aquatic life standards are receiving water quality guidelines.

3.2 ARD Potential

3.2.1 Net Potential Ratio (NPR) Criteria

ABA results were compared to the Mine Environment Neutral Drainage (MEND) Program outlined in and provided in Table 3.1: *Prediction Manual for Drainage Chemistry from Sulphidic Geologic Materials* (MEND, 2009). The criteria shown in Table 5.1 is a revision from the criteria used in earlier reports and represents the current understanding of ARD generation in sulphidic rock.

Table 3.1: NPR Acid Rock Drainage Screening Criteria (MEND, 2009)

Potential for ARD	Initial Screening Criteria
Potentially Acid Generating	NPR <1
Uncertain Acid Generating Potential	1<NPR<2
Non-Potentially Acid Generating	NPR >24

3.2.2 Net Neutralizing Potential (NNP) Criteria

The criteria for the ARD potential based on the NNP are from the BC *Draft Acid Rock Drainage Technical Guide* (SRK, 1992) and provided in Table 3.2:

Table 3.2: NNP Acid Rock Drainage Screening Criteria (SRK, 1992)

Potential for Acid Generation	NNP Criteria
	t $CaCO_3$ /1000 t
Non-acid generating	NNP > 20
Uncertain acid generating potential	20 > NNP > -20
Potentially acid generating	-20 > NNP

3.2.3 Net Acid Generation Criteria (NAGpH)

Results of NAG tests are compared to criteria defined by AMIRA from the *ARD Test Handbook* (AMIRA, 2002) to identify the ARD potential of the test material, as summarized in Table 3.3

Table 3.3: Net Acid Generation Criteria (AMIRA, 2009)

NAG pH	NAG Acidity (to pH 4.5) kg H_2SO_4/t	Potential for Acid Generation
≥ 4.5	0	Non-acid generating.
< 4.5	≤ 5 [1]	Low potential to be acid generating.
< 4.5	> 5 [1]	Potentially acid generating.

1- The NAG criteria used may vary from site to site with a cut-off up to 10 kg H_2SO_4/t.

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APPENDIX III

XRD Mineralogy

QUANTITATIVE PHASE ANALYSIS OF TWO POWDER SAMPLES USING THE RIETVELD METHOD AND X-RAY POWDER DIFFRACTION DATA.

(Project: 0715 Carmack – PO# 42208)

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September 13, 2010

EXPERIMENTAL METHOD

The two samples of **Project 0715 Carmack** were reduced to the optimum grain-size range for quantitative X-ray analysis (<10 μm) by grinding under ethanol in a vibratory McCrone Micronising Mill for 7 minutes. Step-scan X-ray powder-diffraction data were collected over a range 3-80°2θ with CoKa radiation on a Bruker D8 Focus Bragg-Brentano diffractometer equipped with an Fe monochromator foil, 0.6 mm (0.3°) divergence slit, incident- and diffracted-beam Soller slits and a LynxEye detector. The long fine-focus Co X-ray tube was operated at 35 kV and 40 mA, using a take-off angle of 6°.

RESULTS

The X-ray diffractograms were analyzed using the International Centre for Diffraction Database PDF-4 and Search-Match software by Siemens (Bruker). X-ray powder-diffraction data of the samples were refined with Rietveld program Topas 4.2 (Bruker AXS). The results of quantitative phase analysis by Rietveld refinements are given in Table 1. These amounts represent the relative amounts of crystalline phases normalized to 100%. The Rietveld refinement plots are shown in Figures 1 – 2.

Table 1. Results of quantitative phase analysis (wt.%)

Mineral	**Ideal Formula**	**Col 7 A+B Left**	**Col 7 A+B Right**
Quartz	SiO_2	11.6	9.9
Biotite	$K(Mg,Fe)_3(AlSi_3O_{10})(OH)_2$	6.7	6.9
Kaolinite	$Al_2Si_2O_5(OH)_4$	2.3	1.9
Actinolite	$Ca_2(Mg,Fe)_5Si_8O_{22}(OH)_2$	7.0	8.5
K-feldspar	$KAlSi_3O_8$	7.3	5.2
Plagioclase	$NaAlSi_3O_8 – CaAl_2Si_2O_8$	62.0	62.5
Calcite	$CaCO_3$	1.0	0.9
Dolomite	$CaMg(CO_3)_2$		0.7
Jarosite	$K_2Fe_6^{3+}(SO_4)_4(OH)_{12}$	1.6	3.0
Anatase ?	TiO_2	0.5	0.6
Total		100.0	100.0

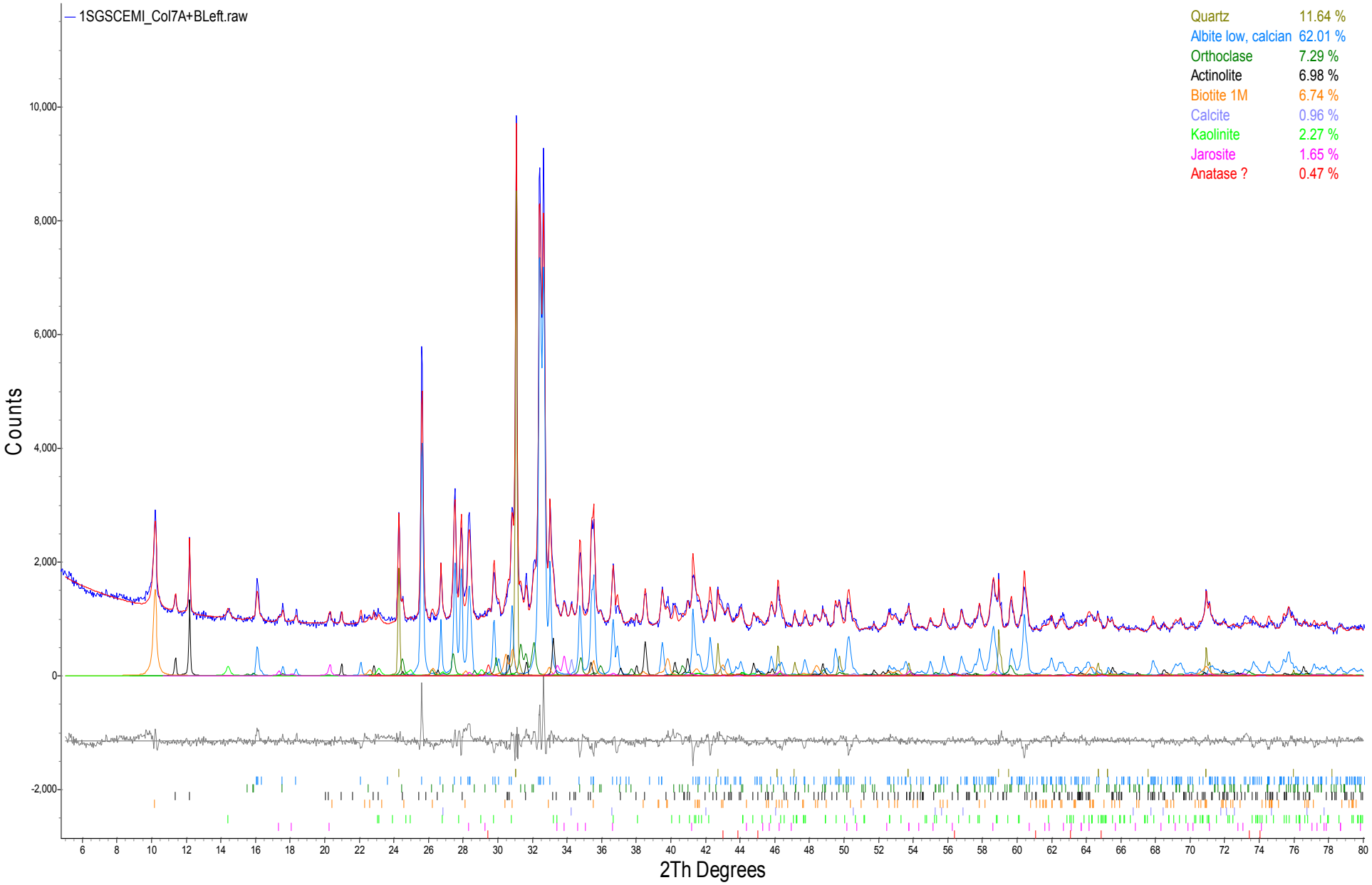


Figure 1. Rietveld refinement plot of sample **SGS CEMI "Col 7 A+B Left"** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

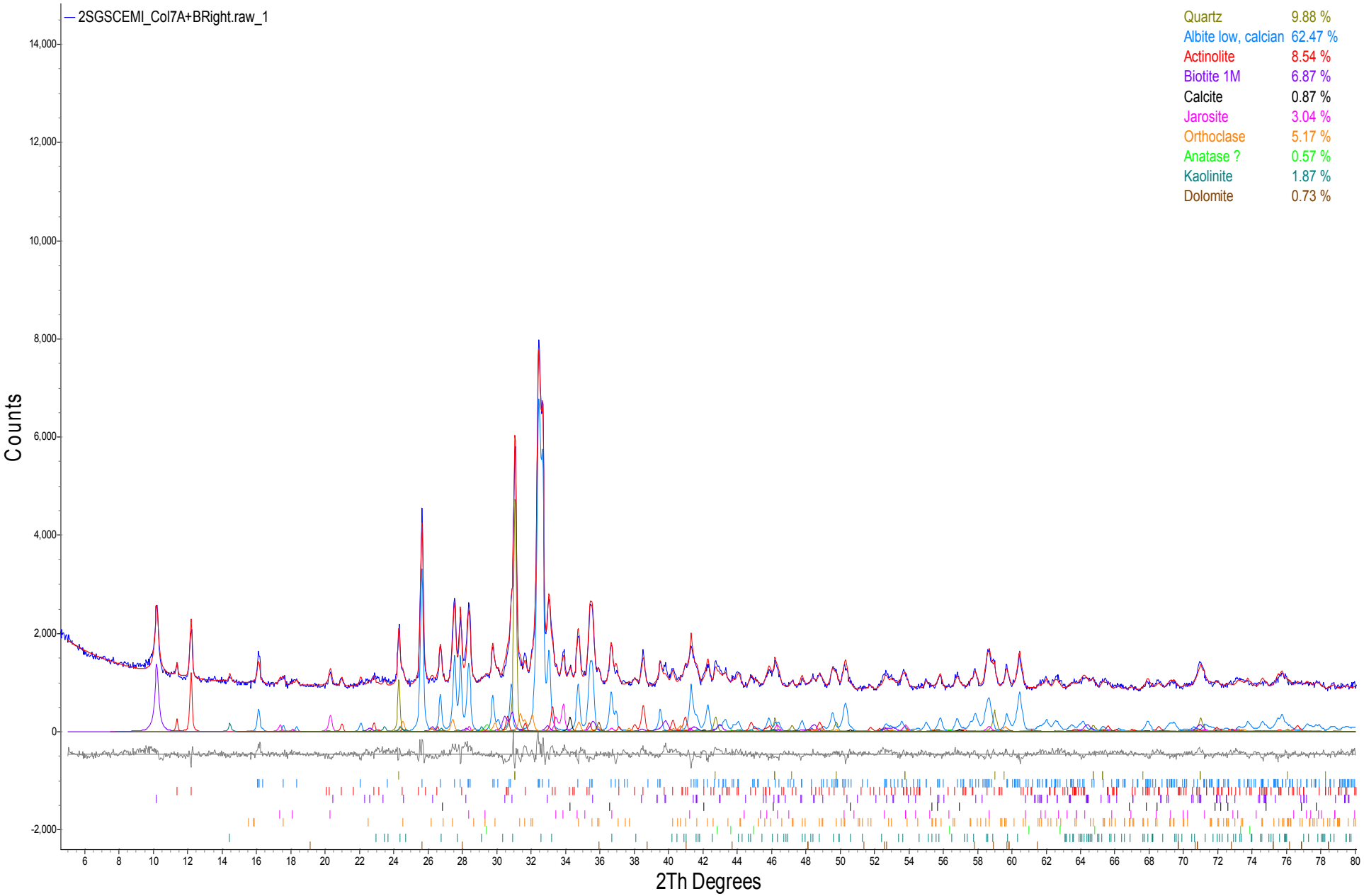


Figure 2. Rietveld refinement plot of sample **SGS CEMI "Col 7 A+B Right"** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

APPENDIX IV

2008 and 2010 Static and Kinetic Testing Results

February 15, 2012
Project No. 07-1413-0077/9100
Doc. No. 173 Rev. 0

Golder Associates

TABLE IV.1
X-Ray Diffraction Mineralogy
2010 Neutralized Spent Ore Samples
Carmacks Copper Project, YT

Mineral	Ideal formula	Col 7 A+B Left	Col 7 A+B Right	T1	B1
Quartz	SiO_2	11.6	9.9	9.9	7.5
Biotite	$K(Mg,Fe^{2+})_3AlSi_3O_{10}(OH)_2$	6.7	6.9	5.4	7
Kaolinite	$Al_2Si_2O_5(OH)_4$	2.3	1.9		
Muscovite	$KAl_2AlSi_3O_{10}(OH)_2$			1.4	
Actinolite	$Ca_2(Mg,Fe^{2+})_5Si_8O_{22}(OH)_2$	7	8.5	6.1	5.7
K-feldspar	$KAlSi_3O_8$	7.3	5.2	6.2	5.1
Plagioclase	$NaAlSi_3O_8 – CaAl_2Si_2O_8$	62	62.5	56.8	52.1
Calcite	$CaCO_3$	1	0.9	0.6	
Gypsum	$CaSO_4 \cdot 2H_2O$				0.9
Dolomite	$CaMg(CO_3)_2$		0.7		
Jarosite	$K_2Fe_6^{3+}(SO_4)_4(OH)_{12}$	1.6	3	1.7	2.1
Montmorillonite model	$(Na,Ca)_{0.3}(Al,Mg)_2Si_4O_{10}(OH)_2 \cdot nH_2O$			12	19.7
Anatase ?	TiO_2	0.5	0.6		
Total		100	100	100	100

TABLE IV.2
Whole Rock Analysis Results
2010 Neutralized Spent Ore Samples
Carmacks Copper Project, YT

Sample ID	Rock type	Alteration	Mineralization	SiO2 %	Al2O3 %	Fe2O3 %	CaO %	MgO %	Na2O %	K2O %	TiO2 %	P2O5 %	MnO %	BaO %	Cr2O3 %	LOI %	Total %	C %	S %
Current Spent Ore																			
Pre-Kinetic Test Samples																			
Col 7 A+B Left Sample	Neutralized Spent Ore	-	-	60.11	16.87	5.33	4.3	1.91	4.62	1.96	0.68	0.35	0.06	0.2	0.01	3.33	99.72	0.11	0.29
Col 7 A+B Right Sample	Neutralized Spent Ore	-	-	58.17	15.97	6.27	4.55	2.03	4.46	1.88	0.72	0.39	0.07	0.17	0.01	4.5	99.19	0.17	0.52
Post-Kinetic Test [1]																			
Col 7 A+B Left Sample	Neutralized Spent Ore			56.4	19.2	8.12	5.13	2.57	4.8	2.28	0.97	0.31	0.06	0.135	0.01	4.83	104.82		
Col 7 A+B Right Sample	Neutralized Spent Ore			57.4	18.7	7.38	4.14	2.25	5	2.32	0.83	0.35	0.06	0.157	0.02	4.63	103.24		
Historical Spent Ore																			
B 1 (-1/2 inch)	Neutralized Spent Ore	-	-	54.56	17.73	7.47	4.46	2.52	4.68	2.2	0.85	0.33	0.07	0.15	0.02	4.79	99.81	0.01	0.55
T 1 (-1/2 inch)	Neutralized Spent Ore	-	-	58.36	18.08	5.19	4.5	2.07	4.95	2.01	0.87	0.12	0.05	0.17	0.02	3.07	99.46	0.12	0.12
Col-6 (0700601)	Neutralized Spent Ore	-	-	59.93	15.78	6.5	4.05	1.95	3.89	2.26	0.76	0.3	0.05	0.15	0.01	4.16	99.8	0.07	0.37

Notes:
< = less than detection limit
(1) Metal concentrations for the Post-Kinetic Test Samples hae been normalized to 100% without the LOI

TABLE IV.3
Trace Element Analysis Results
2010 Neutralized Spent Ore Samples
Carmacks Copper Project, YT

Sample ID	Rock type	Alteration	Mineralization	Ag ppm	Al %	As ppm	B ppm	Ba ppm	Be ppm	Bi ppm	Ca %	Cd ppm	Ce ppm	Co ppm	Cr ppm	Cs ppm	Cu ppm	Fe %	Ga ppm	Ge ppm	Hf ppm	Hg ppm	In ppm	K %	La ppm	Li ppm	Lu ppm	Mg %	Mn ppm	Mo ppm
Current Spent Ore																													
Pre-Kinetic Test																													
Col 7 A+B Left Sample	Neutralized Spent Ore	-	-	2.3	1.07	1	n/a	524	<1	2	0.56	0.2	11	7.1	40	1.2	588.4	2.91	6	<0.1	0.1	0.211	0.08	0.65	6	5.9	n/a	0.72	235	77.8
Col 7 A+B Right Sample	Neutralized Spent Ore	-	-	2.1	1.02	1.5	n/a	416	<1	1.9	0.72	0.3	11	6.9	33	1.2	634.9	3.24	7	<0.1	0.1	0.156	0.09	0.72	7	4.3	n/a	0.73	240	103.9
Post-Kinetic Test																													
Col 7 A+B Left Sample	Neutralized Spent Ore			3.49	1.72	4	60	590	0.1	3.91	0.89	0.1	13.4	9.2	45	1.63	886	4.33	8.7	0.2	0.13	0.03	0.13	0.96	7.6	7	0.07	1.02	330	110
Col 7 A+B Right Sample	Neutralized Spent Ore			3.16	1.67	5	60	569	0.2	2.05	0.63	0.12	15.9	9.7	61	1.53	1000	4.54	9.1	0.2	0.12	0.07	0.16	1.02	9	6	0.07	1.02	329	103
Historic Spent Ore																													
B 1 (-1/2 inch)	Neutralized Spent Ore	-	-	3.9	1.61	3.5	n/a	470	<1	3.4	0.71	0.4	13	10	61	1.6	866.9	4.1	11	0.1	0.1	0.2	0.14	0.9	8	5.1	n/a	1.03	306	175.5
T 1 (-1/2 inch)	Neutralized Spent Ore	-	-	3.4	1.04	2.7	n/a	571	<1	1.9	0.59	0.2	11	7.1	57	1.4	647.2	2.62	6	0.1	0.1	0.2	0.06	0.59	6	3.5	n/a	0.71	234	90.1
Col-6 (0700601)	Neutralized Spent Ore	-	-	5.5	1.93	3.2	n/a	660	<1	5.3	0.72	1.1	16	11.9	58	2.1	721	4.28	12	0.2	0.1	0.8	0.11	1.05	10	5.8	n/a	1.04	337	390.4

Sample ID	Rock type	Alteration	Mineralization	Na %	Nb ppm	Ni ppm	P %	Pb ppm	Rb ppm	Re ppb	S %	Sb ppm	Sc ppm	Se ppm	Nb ppm	Sn ppm	Sr ppm	Ta ppm	Tb ppm	Te ppm	Th ppm	Ti %	Tl ppm	U ppm	V ppm	W ppm	Y ppm	Yb ppm	Zn ppm	Zr ppm
Current Spent Ore																													
Pre-Kinetic Test																													
Col 7 A+B Left Sample	Neutralized Spent Ore	-	-	0.16	0.2	5.9	0.077	3.2	40	35	0.24	<0.1	5.2	0.8	n/a	0.9	30	<0.1	n/a	2.3	1.3	0.145	0.2	0.4	99	3.9	3.3	n/a	40	1.7
Col 7 A+B Right Sample	Neutralized Spent Ore	-	-	0.14	0.2	4.7	0.088	2.9	41	25	0.43	<0.1	5.5	0.9	n/a	0.9	34	<0.1	n/a	0.8	1.5	0.154	0.3	0.4	104	3.6	3.8	n/a	42	1.7
Post-Kinetic Test																													
Col 7 A+B Left Sample	Neutralized Spent Ore			0.09	n/a	3.8	1130	6.4	53.5	n/a	0.44	<0.05	7.5	2	0.29	1.4	37.4	<0.05	0.17	0.97	1.5	0.2	0.3	0.45	133	4.9	4.65	0.5	51	3.3
Col 7 A+B Right Sample	Neutralized Spent Ore			0.08	n/a	4.8	1290	10.2	55	n/a	0.47	<0.05	8.5	2	0.34	1.4	40.7	<0.05	0.19	0.75	2	0.2	0.31	0.53	134	10.9	5.1	0.5	59	3.6
Historic Spent Ore																													
B 1 (-1/2 inch)	Neutralized Spent Ore	-	-	0.07	0.2	6.8	0.136	4.3	60.2	39	0.57	0.1	6.9	1.1	n/a	2.1	47	<0.1	n/a	0.7	2	0.191	0.3	0.5	122	6.8	4.3	n/a	58	2.2
T 1 (-1/2 inch)	Neutralized Spent Ore	-	-	0.07	0.2	5.4	0.037	4.5	46.6	26	0.1	<0.1	5.3	<0.5	n/a	1.8	42	<0.1	n/a	0.6	1	0.175	0.3	0.5	90	5.4	4.2	n/a	38	2
Col-6 (0700601)	Neutralized Spent Ore	-	-	0.13	0.3	7.5	0.115	6.3	68	246	0.42	0.1	7.5	2.7	n/a	2	43	<0.1	n/a	1.1	2.4	0.229	0.4	0.7	133	17.7	5.2	n/a	70	2.6

Notes:
< = less than detection limit
Non-detect values assumed to be equal to one half of the detection limit in calculations of average, median and standard deviation, where applicable.
n/a = not analyzed

TABLE IV.4
Shake Flask Analysis Results
2010 Neutralized Spent Ore Samples
Carmacks Copper Project, YT

CCME (CEQG) (freshwater aquatic life)* (mg/L)				**6.5 - 9.0**							**0.005 - 0.1**		**0.005**				**0.000026**	**0.073**	**0.025 - 0.15**			**0.001**		**0.0001**
Metal Mining Effluent Regulations (MMER) (mg/L)**				6.0 - 9.5									0.5						0.5					
Sample ID	**Rock type**	**Alteration**	**Mineralization**	**pH s.u.**	**REDOX mV**	**Conductivity uS/cm**	**Acidity (pH 8.3) mg/L as CaCO3**	**Alkalinity mg/L as CaCO3**	**Hardness mg/L as CaCO3**	**Sulphate (mg/L)**	**Aluminum[2] (mg/L)**	**Antimony (mg/L)**	**Arsenic (mg/L)**	**Barium (mg/L)**	**Magnesium (mg/L)**	**Manganese (mg/L)**	**Mercury (mg/L)**	**Molybdenum (mg/L)**	**Nickel[4] (mg/L)**	**Phosphorus (mg/L)**	**Potassium (mg/L)**	**Selenium (mg/L)**	**Silicon (mg/L)**	**Silver (mg/L)**
Current Spent Ore																								
Col 7 A+B Left Sample	Neutralized Spent Ore	-	-	8.52	208.98	731.89	#N/A	194.84	11.20	194.00	**0.66**	0.0004	**0.0055**	0.018	0.17	0.0065	**0.07**	**0.24**	0.0010	0.91	2.55	**0.013**	46.60	**0.0047**
Col 7 A+B Right Sample	Neutralized Spent Ore	-	-	7.98	279.29	786.47	4.16	77.72	116.00	321.00	0.05	< 0.0002	0.0047	0.02	0.042	0.0052	**0.01**	**0.54**	0.0008	0.11	3.19	**0.0043**	29.10	**0.00013**
Historical Spent Ore																								
B 1 (-1/2 inch)	Neutralized Spent Ore	-	-	**5.96**	487	2359	8.0	2.2	1590	1463	**0.195**	<0.0001	0.0011	0.0103	3.09	0.151	<0.00001	0.0053	0.0053	0.021	4.41	**0.002**	19.9	<0.00003
T 1 (-1/2 inch)	Neutralized Spent Ore	-	-	8.09	417	266	2.6	87.2	54.7	38	0.0498	0.0004	0.00194	0.107	<0.05	0.00039	0.00001	**0.185**	0.00024	0.575	1.84	**0.00855**	34.6	0.000041
Col-6 (0700601)	Neutralized Spent Ore	-	-	7.67	398	2197	5.8	52.7	693	1339	0.002	0.0004	0.0014	0.0277	0.4	0.0005	<0.00005	**0.996**	0.0012	0.247	4.48	**0.0119**	35	<0.00003

CCME (CEQG) (freshwater aquatic life)* (mg/L)							**0.000005 - 0.000338**		**0.001/0.0089**		**0.002 - 0.004**	**0.3**	**0.001 - 0.007**					**0.0008**					**0.03**	
Metal Mining Effluent Regulations (MMER) (mg/L)**											0.3		0.2										0.5	
Sample ID	**Rock type**	**Alteration**	**Mineralization**	**Beryllium (mg/L)**	**Bismuth (mg/L)**	**Boron (mg/L)**	**Cadmium[4,7] (mg/L)**	**Calcium (mg/L)**	**Chromium[3] (mg/L)**	**Cobalt (mg/L)**	**Copper[4] (mg/L)**	**Iron (mg/L)**	**Lead[4] (mg/L)**	**Lithium (mg/L)**	**Sodium (mg/L)**	**Strontium (mg/L)**	**Sulphur (mg/L)**	**Thallium (mg/L)**	**Tin (mg/L)**	**Titanium (mg/L)**	**Uranium (mg/L)**	**Vanadium (mg/L)**	**Zinc (mg/L)**	**Zirconium (mg/L)**
Current Spent Ore																								
Col 7 A+B Left Sample	Neutralized Spent Ore	-	-	< 0.00002	0.0001	0.01	0.000089	4.2	0.0013	0.000171	**0.0312**	**0.756**	0.00069	< 0.001	158	0.0056	65.7	< 0.0002	0.00097	0.0498	0.000076	0.0712	0.002	0.00029
Col 7 A+B Right Sample	Neutralized Spent Ore	-	-	< 0.00002	0.00001	0.006	0.000209	46.3	0.0006	0.000098	**0.0092**	0.102	0.00008	< 0.001	131	0.0197	115	< 0.0002	0.00021	0.006	0.000033	0.0118	0.003	0.00004
Historical Spent Ore																								
B 1 (-1/2 inch)	Neutralized Spent Ore	-	-	<0.00005	<0.00003	0.031	**0.00061**	631	0.0007	0.00288	**0.213**	0.047	0.00004	<0.003	15.7	0.455	552	0.00013	<0.00005	0.01	<0.00001	0.002	**0.0836**	<0.0005
T 1 (-1/2 inch)	Neutralized Spent Ore	-	-	<0.00001	0.000008	0.006	0.000153	21.9	0.0001	0.000014	**0.00339**	0.046	0.000066	<0.0005	35.8	0.101	17	0.000005	0.00002	0.0054	0.000006	0.0016	0.0006	<0.0001
Col-6 (0700601)	Neutralized Spent Ore	-	-	<0.00005	<0.00003	<0.3	**0.00046**	277	<0.0005	<0.00003	**0.0041**	0.01	<0.00003	<0.003	256	0.349	441	0.00003	<0.00005	<0.003	<0.00001	<0.001	0.0006	<0.0005

Notes:
1. CEQG freshwater guidelines and criteria are based on**total** metal concentrations, except for aluminum (dissolved aluminum criterion).
2. Freshwater aquatic life criterion for aluminum depends on pH, [Ca2+] and DOC. In this table, only the pH criterion has been applied to highlight exceedances.
3. Freshwater aquatic life criteria for chromium depends on the valence of chromium ion. In the above table, the Cr(VI) criterion of 0.001 mg/L is shown.
4. Freshwater aquatic life criterion or guideline is hardness dependant.
5. Maximum authorized monthly mean concentration (based o**total** concentration).
6. < = less than the analytical detection limit.
7. Cadmium guideline = 10^(0.86*log(hardness)-3.2)

* December 2003
** June 6, 2002
Non-detect values assumed to be equal to one half the detection limit in calculations of average, median and standard deviation, where applicable.

TABLE IV.5
Acid Base Accounting Results
2010 Neutralized Spent Ore Samples
Carmacks Copper Project, YT

Sample ID	Sample ID	Rock type	Alteration	Mineralization	Paste pH	S(T) %	S(SO4) %	S(S-2) %	AP	NP	TIC %	CaCO3 NP	Net NP	NPR	CaCO3 Net NP	CaNPR
Current Spent Ore	**Current Spent Ore**															
							Pre-Kinetic Test Samples									
Col 7 A+B Left Sample	Col 7 A+B Left Sample	Neutralized Spent Ore	-	-	8.93	0.29	0.28	0.01	0.3	22.5	0.11	9.2	22.2	72.0	8.9	29.33
Col 7 A+B Right Sample	Col 7 A+B Right Sample	Neutralized Spent Ore	-	-	8.00	0.52	0.51	0.01	0.3	26.8	0.16	13.3	26.5	85.8	13.0	42.67
							Post Kinetic Test Samples									
Col 7 A+B Left Sample	Col 7 A+B Left Sample	Neutralized Spent Ore			7.68	0.4	0.49	<0.01	0.3	11.6	0.09	7.5	11.3	37.12	7.2	24.00
Col 7 A+B Right Sample	Col 7 A+B Right Sample	Neutralized Spent Ore			7.74	0.43	0.52	<0.01	0.3	9.9	0.04	3.3	9.6	31.68	3.0	10.67
Historical Spent Ore	**Historical Spent Ore**															
B 1 (-1/2 inch)	B 1 (-1/2 inch)	Neutralized Spent Ore	-	-	5.91	0.55	0.42	0.13	4.1	6.3	0.01	0.8	2.2	1.6	-3.2	0.21
T 1 (-1/2 inch)	T 1 (-1/2 inch)	Neutralized Spent Ore	-	-	7.85	0.12	0.07	0.05	1.6	10.8	0.08	6.7	9.2	6.9	5.1	4.27
Col-6 (0700601)	Col-6 (0700601)	Neutralized Spent Ore	-	-	7.89	0.38	0.3	0.08	2.5	8.8	0.04	3.3	6.3	3.5	0.8	1.33

Notes:
< = less than detection limit
* S(S2-) sulphide sulphur calculated as S(T)total suphur - S(SO4)sulphate sulphur
Where S(SO4) is reported as <0.01%, it is assumed to be zero for the AP calculation.
TIC = Total inorganic carbon, used in calculation of CaNP
AP = Acid potential in kg CaCO3 equivalent per tonne of rock. AP is determined from calculated sulphide sulphur content: S(T) - S(SO4), assuming total conversion of sulphide to sulphate.
Bulk NP = Neutralization potential in kg CaCO3 equivalent per tonne of rock.
CaNP = Carbonate mineral neutralization potential in kg CaCO3 equivalent per tonne of rock
NET NP = Net neutralization potential in kg CaCO3 equivalent per tonne of rock. Calculated as the difference between NP and AP.
NPR = Neutralization potential ratio: NP/AP
CaNPR = Neutralization potential ratio calculated using CaNP: CaNP/AP
Non-detect values assumed to be equal to one half of the detection limit in calculations of average, median and standard deviation, where applicable.

TABLE IV.6
NAG Test Results
2010 Neutralized Spent Ore Samples
Carmacks Copper Project, YT

Sample ID	NAG pH after reaction	NAG (kg H2SO4/tonne) to pH 4.5	NAG (kg H2SO4/tonne) to pH 7.0	NAG (kg H2SO4/tonne) to pH 4.5	NAG (kg H2SO4/tonne) to pH 7.0
Pre-Kinetic Test Samples					
Col 7 A+B Left Sample	6.97	#N/A	0.25	#N/A	0.5
Col 7 A+B Right Sample	6.84	#N/A	0.5	#N/A	1
Post-Kinetic Test Samples					
Col 7 A+B Left Sample	6.76	#N/A	0.35	#N/A	0.7
Col 7 A+B Right Sample	6.82	#N/A	0.25	#N/A	0.5

TABLE IV.7
NAG Leachate Analysis Results
2010 Neutralized Spent Ore Samples
Carmacks Copper Project, YT

CCME (CEQG) (freshwater aquatic life)* (mg/L)				**6.5 - 9.0**							**0.005 - 0.1**		**0.005**				**0.000026**	**0.073**	**0.025 - 0.15**			**0.001**		**0.0001**
Metal Mining Effluent Regulations (MMER) ** (mg/L)**				6.0 - 9.5									0.5						0.5					
Sample ID	**Rock type**	**Alteration**	**Mineralization**	**pH s.u.**	**REDOX mV**	**Conductivity uS/cm**	**Acidity (pH 8.3) mg/L as CaCO3**	**Alkalinity mg/L as CaCO3**	**Hardness mg/L as CaCO3**	**Sulphate (mg/L)**	**Aluminum[2] (mg/L)**	**Antimony (mg/L)**	**Arsenic (mg/L)**	**Barium (mg/L)**	**Magnesium (mg/L)**	**Manganese (mg/L)**	**Mercury (mg/L)**	**Molybdenum (mg/L)**	**Nickel[4] (mg/L)**	**Phosphorus (mg/L)**	**Potassium (mg/L)**	**Selenium (mg/L)**	**Silicon (mg/L)**	**Silver (mg/L)**
Current Spent Ore																								
Col 7 A+B Left Sample	Neutralized Spent Ore	-	-	6.92	297.85	292.03	63.24	63.77	21.9	20	0.01	0.0028	0.0031	0.0011	0.99	0.0082	< 0.01	0.39	0.0007	24.00	4.84	**0.0079**	24.50	**0.0011**
Col 7 A+B Right Sample	Neutralized Spent Ore	-	-	6.65	317.87	313.09	63.55	47.74	34.5	48	0.0047	0.0024	**0.0065**	0.0005	1.23	0.0053	< 0.01	0.47	0.0007	22.00	4.85	**0.0080**	23.30	**0.0016**
Historical Spent Ore																								
B 1 (-1/2 inch)	Neutralized Spent Ore	-	-																					
T 1 (-1/2 inch)	Neutralized Spent Ore	-	-																					
Col-6 (0700601)	Neutralized Spent Ore	-	-																					

CCME (CEQG) (freshwater aquatic life)* (mg/L)							**0.000005 - 0.000338**		**0.001/0.0089**		**0.002 - 0.004**	**0.3**	**0.001 - 0.007**				**0.0008**						**0.03**	
Metal Mining Effluent Regulations (MMER) ** (mg/L)**											0.3		0.2										0.5	
Sample ID	**Rock type**	**Alteration**	**Mineralization**	**Beryllium (mg/L)**	**Bismuth (mg/L)**	**Boron (mg/L)**	**Cadmium[4,7] (mg/L)**	**Calcium (mg/L)**	**Chromium[3] (mg/L)**	**Cobalt (mg/L)**	**Copper[4] (mg/L)**	**Iron (mg/L)**	**Lead[4] (mg/L)**	**Lithium (mg/L)**	**Sodium (mg/L)**	**Strontium (mg/L)**	**Sulphur (mg/L)**	**Thallium (mg/L)**	**Tin (mg/L)**	**Titanium (mg/L)**	**Uranium (mg/L)**	**Vanadium (mg/L)**	**Zinc (mg/L)**	**Zirconium (mg/L)**
Current Spent Ore																								
Col 7 A+B Left Sample	Neutralized Spent Ore	-	-	< 0.00002	0.00001	0.025	0.00023	7.14	**0.039**	0.00	**0.0084**	0.011	0.00003	0.002	50.10	0.0044	6.32	< 0.0002	0.0012	0.13	0.00014	0.12	0.001	0.00009
Col 7 A+B Right Sample	Neutralized Spent Ore	-	-	< 0.00002	< 0.00001	0.027	0.00025	11.80	**0.038**	0.00	**0.0062**	< 0.002	0.00003	0.002	47.60	0.0042	13.80	< 0.0002	0.0006	0.12	0.0001	0.13	< 0.001	0.00007
Historical Spent Ore																								
B 1 (-1/2 inch)	Neutralized Spent Ore	-	-																					
T 1 (-1/2 inch)	Neutralized Spent Ore	-	-																					
Col-6 (0700601)	Neutralized Spent Ore	-	-																					

Notes:
1. CEQG freshwater guidelines and criteria are based on **total** metal concentrations, except for aluminum (dissolved aluminum criterion).
2. Freshwater aquatic life criterion for aluminum depends on pH, [Ca2+] and DOC. In this table, only the pH criterion has been applied to highlight exceedances.
3. Freshwater aquatic life criteria for chromium depends on the valence of chromium ion. In the above table, the Cr(VI) criterion of 0.001 mg/L is shown.
4. Freshwater aquatic life criterion or guideline is hardness dependant.
5. Maximum authorized monthly mean concentration (based on **total** concentration).
6. < = less than the analytical detection limit.
7. Cadmium guideline = 10^(0.86*log(hardness)-3.2)

* December 2003
** June 6, 2002
Non-detect values assumed to be equal to one half the detection limit in calculations of average, median and standard deviation, where applicable.

Col 7 A+B Left Sample

Sieve Designation	Aperture	Weight Retained		
				Cumulative
	(mm)	(g)	(%)	(%)
+ 3/4	19.000	14.30	7.2%	7.2%
-3/4 + 1/2	12.500	39.70	19.9%	27.0%
-1/2 + 3/8	9.500	40.40	20.2%	47.3%
-3/8 + 1/4	6.300	19.70	9.9%	57.1%
-1/4 + 10	2.000	27.80	13.9%	71.1%
-10 + 35	0.425	24.70	12.4%	83.4%
-35 + 100	0.150	19.00	9.5%	92.9%
-100 + 270	0.053	9.80	4.9%	97.8%
-270	-0.053	4.30	2.2%	100.0%
TOTAL		199.70	100.0%	

Col 7 A+B Right Sample

Sieve Designation	Aperture	Weight Retained		
				Cumulative
	(mm)	(g)	(%)	(%)
+ 3/4	19.000	34.20	17.1%	17.1%
-3/4 + 1/2	12.500	38.30	19.2%	36.3%
-1/2 + 3/8	9.500	23.80	11.9%	48.2%
-3/8 + 1/4	6.300	22.90	11.5%	59.7%
-1/4 + 10	2.000	31.20	15.6%	75.3%
-10 + 35	0.425	23.30	11.7%	87.0%
-35 + 100	0.150	15.10	7.6%	94.5%
-100 + 270	0.053	7.50	3.8%	98.3%
-270	-0.053	3.40	1.7%	100.0%
TOTAL		199.70	100.0%	

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COPPER NORTH
MINING CORP.

2011 ANNUAL QUARTZ MINING LICENCE REPORT

SUBMITTED TO YUKON GOVERNMENT, ENERGY MINES AND RESOURCES

YUKON QUARTZ MINING LICENCE QML-0007

Appendix F

2011 Environmental Monitoring Program Update and Data Summary

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COPPER NORTH MINING CORP.

2011 Environmental Monitoring Program Update and Data Summary

Carmacks Copper Project
Yukon Territory

Prepared by:

ACCESS

March 2012

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1.0 BACKGROUND & INTRODUCTION

The Carmacks Copper project is a proposed open pit copper mine and solvent extraction and electrowinning processing facility to be developed by Copper North Mining Corp., (spun out from Western Copper Corporation of Vancouver, British Columbia in October 2011). The ore body is located in Yukon (Figure 1), about 38 km northwest of the Village of Carmacks, or 192 km northwest of Whitehorse (Figure 2). The project site is located in the upper reaches of Williams Creek, approximately 9 km upstream of the confluence with the Yukon River. Williams Creek is a small erosional creek originating in the Dawson Range and draining northeast into the Yukon River downstream of Carmacks.

An initial assessment of baseline environmental conditions at the project site was undertaken between 1992 and 1994, with periodic data collection occurring in 1997 and 1999. In August 2005 the environmental monitoring program (EMP) was re-activated to update baseline data for the project area and has continued on a regular basis during ice free months, roughly between April and October. The EMP focuses primarily on collecting groundwater and surface water quality data, conducting fisheries, wildlife, vegetation, benthic and sediment studies (as required or permissible), and noting general site conditions throughout the years.

During 2011 no construction or development activities occurred at Carmacks Copper. Access Consulting Group conducted five site visits, which focused on the collection of baseline surface water and groundwater quality data, and noting site conditions. This report provides a summary of EMP activities completed and data collected to incorporate 2011 results and analysis. The five site visits in 2011 took place in May (2), June, September and October.

Table 1 summarizes environmental monitoring activities that have occurred at the Carmacks Copper site between 2005 and 2011.

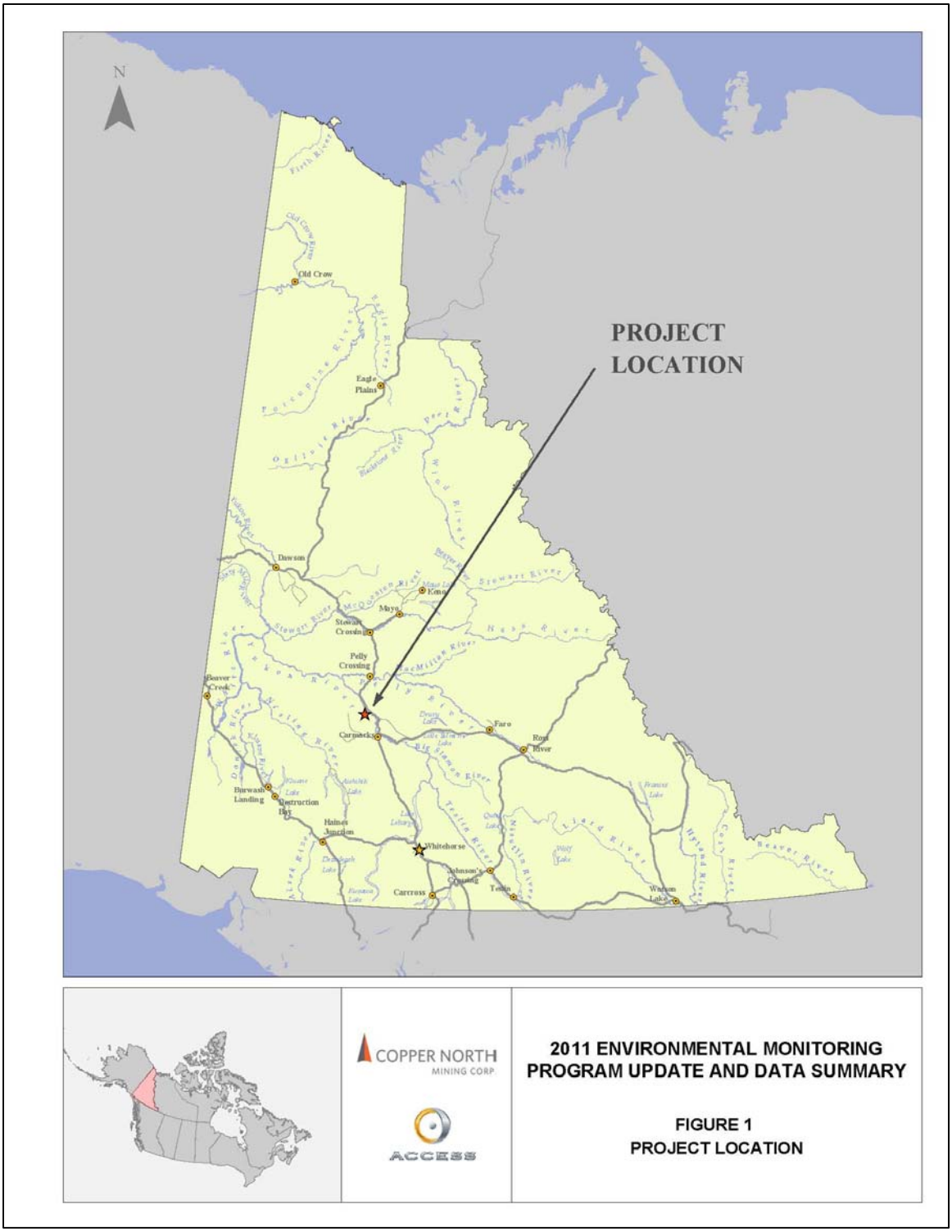
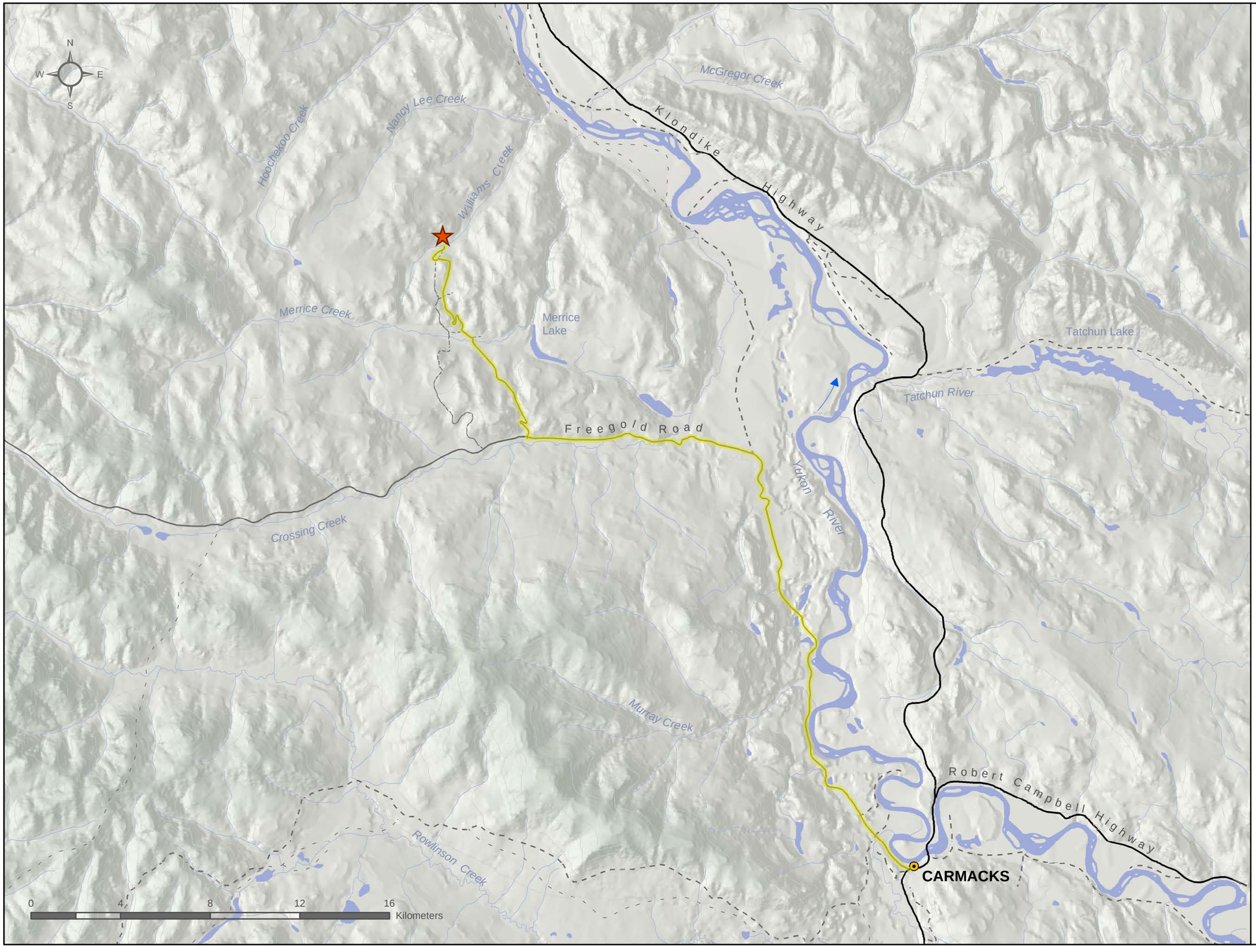


Figure 1 General Location Map of the Yukon

CARMACKS COPPER PROJECT

2011 ENVIRONMENTAL MONITORING PROGRAM UPDATE AND DATA SUMMARY

COPPER NORTH MINING CORP.



- Project Location
- Access Road
- Road
- Exploration Road
- Contour
- Watercourse
- Waterbody

Datum: NAD83; Projection: UTM Zone 8N

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FIGURE 2

PROJECT OVERVIEW

ACCESS

Drawn by JP | FEBRUARY 2012 | Checked by NS

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(*Last edited by: jpan; 2/23/2012/12:22 PM*)

Table 1 Summary of Environmental Monitoring Conducted Between 2005 and 2011

Date	Surface Water	Groundwater	Sediment	Benthos	Fisheries	Wildlife	Vegetation
Aug-05	√		√				
Oct-05	√		√		√		
Dec-05						√	
Mar-06	√						
Jun-06	√	√	√		√		
Jul-06	√	√	√		√	√	√
Aug-06	√	√	√				
Sep-06	√	√	√	√	√		
Oct-06	√	√	√				
Apr-07	√	√					
May-07	√	√					
Jun-07	√	√					
Jul-07	√	√					
Aug-07	√	√					
Sep-07	√	√	√	√			
Oct-07	√	√					
Mar-08	√						
Apr-08	√						
May-08	√	√					
June-08	√	√					
July-08	√	√					
Aug-08	√	√					
Sept-08	√	√	√				√
Oct-08	√	√	√				
Nov-08	√						
May-09	√	√					
July-09	√	√			√		
Sept-09	√				√		
Oct-09	√	√					
Apr-10	√*						
May-10	√	√					
June-10	√	√					
Aug-10	√	√					
Oct-10	√						
May-11	√						
Jun-11	√	√					
Sept-11	√	√					
Oct-11	√						

*Hydrology at W4 and W12 only

2.0 SURFACE WATER QUALITY MONITORING

Surface water quality samples have been routinely collected from Williams Creek, Nancy Lee Creek, and the Yukon River for analysis of general chemical parameters, nutrients, and total and dissolved metals. Field measurements were collected for pH, conductivity, temperature and dissolved oxygen.

2.1 SURFACE WATER QUALITY SAMPLE STATION LOCATIONS & SAMPLING FREQUENCY

The thirteen surface water quality monitoring stations located on mainstem Williams Creek (seven), tributaries to Williams Creek (four), and on the Yukon River (two) are described in Table 2 and illustrated on Figure 3.

Table 2 Surface Water Quality Monitoring Station Locations in Williams Creek Watershed

Station	Description / Location
W2	Williams Creek Upstream of North Williams Creek Confluence
W3	Lower North Williams Creek Upstream of Confluence with Williams Creek
W4	Williams Creek Downstream of Confluence with North Williams Creek
W5	South East Tributary to Williams Creek
W6	Williams Creek Downstream of South East Tributary
W7	Upper North Williams Creek Tributary Upstream of Road Crossing
W9	Williams Creek Upstream of Access Road Crossing
W10	Williams Creek Upstream of Yukon River
W11	Nancy Lee Creek (Tributary of Williams Creek)
W12	Williams Creek Downstream of Confluence with Nancy Lee Creek
W13	Williams Creek Upstream of Confluence with Nancy Lee Creek
Y1	Yukon River Upstream of Williams Creek
Y2	Yukon River Downstream of Williams Creek

Surface water quality monitoring activities took place at ten of the thirteen stations in 2010 and 2011 (see Table 3). Williams Creek near the confluence with Yukon River (station W10) as well as the Yukon River (stations Y1 and Y2) were not monitored in 2010 and 2011 due to the logistical challenge of accessing these locations. The data collected from these locations in previous years is considered relatively robust at this stage prior to project advancement.

CARMACKS COPPER PROJECT

2011 ENVIRONMENTAL MONITORING PROGRAM UPDATE AND DATA SUMMARY

COPPER NORTH MINING CORP.

N
W E
S

Yukon River
Klondike Highway #2
McGregor Creek
Nancy Lee Creek
Williams Creek

Y2
W10
Y1
W11
W12
W13
W7
MW96-F
MW96-H
W4
MW96-I
W3
MW96-G
W2
MW96-J
MW96-A
MW96-K
MW96-B
W6
MW96-E
MW96-C
MW96-D
W5
MW-01-07
W9
DW-08-01
DW-08-02

0 1,000 2,000 3,000 4,000 Meters

*Y1 and Y2 sample station locations will vary in winter. Samples collected where open water present

- Actively Monitored Well
- Inactive Well
- Surface Water Monitoring Station
- Contour
- Watercourse
- Waterbody

Datum: NAD83; Projection: UTM Zone 8N

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FIGURE 3

MONITORING STATION LOCATIONS

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(Last edited by: jpan: 2/23/2012/12:00 PM)

Table 3 Water Quality Monitoring between 2005 and 2011

Sample Event	Sample Stations												
	W2	W3	W4	W5	W6	W7	W9	W10	W11	W12	W13	Y1*	Y2*
Aug-05	Y	Y	Y	N	N	Y	Y	N	N	N	N	-	-
Oct-05	Y	Y	Y	N	Y	Y	Y	Y	Y	Y	Y	-	-
Mar-06	N	Y	Y	N	N	N	Y	Y	N	Y	Y	Y	Y
Jun-06	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y
Jul-06	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y
Aug-06	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y
Sep-06	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y
Oct-06	Y	Y	Y	Y	Y	N	Y	Y	Y	Y	Y	Y	Y
Apr-07	Y	Y	Y	Y	Y	N	Y	Y	Y	Y	Y	Y	Y
May-07	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	N	Y
Jun-07	Y	Y	Y	Y	Y	Y	Y	N	N	N	N	N	N
Jul-07	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y
Aug-07	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y
Sep-07	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y
Oct-07	Y	Y	Y	Y	Y	Y	Y	N	N	N	N	N	N
Mar-08	Y	N	Y	N	N	N	Y	N	Y	Y	Y	Y	Y
Apr-08	Y	Y	Y	N	Y	N	Y	N	Y	Y	Y	Y	Y
May-08	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y
Jun-08	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y
Jul-08	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y
Aug-08	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y
Sep-08	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y
Oct-08	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y
Nov-08	Y	Y	Y	N	N	N	N	Y	Y	Y	Y	Y	Y
May-09	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y
July-09	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y
Sep-09	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y
Oct-09	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y	Y
Apr-10	N	N	Y**	N	N	N	N	N	N	Y**	N	N	N
May-10	Y	Y	Y	Y	Y	Y	Y	N	Y	Y	Y	N	N
Jun-10	Y	Y	Y	Y	Y	Y	Y	N	Y	Y	Y	N	N
Aug-10	Y	Y	Y	Y	Y	Y	Y	N	Y	Y	Y	N	N
Oct-10	Y	Y	Y	Y	Y	Y	Y	N	Y	Y	Y	N	N
May-11	Y	Y	Y	Y	Y	Y	Y	N	Y	Y	Y	N	N
Jun-11	Y	Y	Y	Y	Y	Y	Y	N	Y	Y	Y	N	N
Sep-11	Y	Y	Y	Y	Y	Y	Y	N	Y	Y	Y	N	N
Oct-11	Y	Y	Y	Y	Y	Y	Y	N	Y	Y	Y	N	N

* Yukon River sample stations established in 2006

**Hydrology only

2.2 PROTOCOL

Surface water quality sampling was undertaken using Industry recognized Best Management Practices and following Access Consulting Field Protocols for the collection of surface water samples. Field duplicate samples were collected during each sampling event to quantify natural variability between samples. Additional field quality control samples include field blanks and trip blanks, where de-ionized water is handled, processed and analyzed in the same manner as water samples. Blanks can provide indication of sample contamination occurring in the field (field blanks) or lab (laboratory method blanks) and at any point in between (trip blanks).

2.3 RESULTS AND DISCUSSION

Water quality data collected since 2005 are stored in an EQWin database that is customized to manage environmental sampling data. Historic data has not been entered into the database as original laboratory files are not available. Further, evaluation of the older data revealed that the laboratory's method detection limit (MDL) for many parameters was greater than applicable Canadian Water Quality Guidelines (CWQG; CCME 1999).

A summary of water quality data for each station sampled between 2005 and 2011 is provided for a number of parameters of interest, most of which have an associated CWQG for freshwater aquatic life including: pH, total suspended solids, total aluminum, arsenic, cadmium, copper, iron, lead, mercury, molybdenum, selenium, nickel, zinc, ammonia, nitrate and nitrite. Summary statistics are provided at each station for:

- average;
- sample size (count);
- minimum;
- maximum;
- geometric mean;
- count of samples less than laboratory Reportable Detection Limit (< DL);
- standard deviation;
- 1st quartile;
- median;
- 3rd quartile; and

- Count and percent of samples above the CWQG for protection of freshwater aquatic life, if applicable.

Where concentrations were below the laboratory reportable detection limit (RDL), one half the RDL value was used in for statistical calculation.

pH

For the most part pH falls within the CWQG range of 6.5 – 9.0. However, an occasional measurement of pH below 6.5 has been noted, with values as low as 4.64 (W5 on May 9, 2007). The few occasions where pH was measured below 6.5 tend to occur during spring freshet when high water levels were flowing over moss and organics present along creek banks. As water seeps through layers of decaying organic matter it can dissolve carbonic dioxide produced by decay which can result in lower pH levels. Measurements have also been observed to be near pH 9.0, with one measurement above at 9.97 at W2 on September 9, 2009. On average (as shown in Table 4) pH ranges between 7.37 and 7.97 for all sites.

Table 4 Field pH – Surface Water Quality Summary Statistics for 2005 – 2011

Statistic	W2	W3	W4	W5	W6	W7	W9	W10	W11	W12	W13	Y1	Y2
Average	7.9	7.37	7.79	7.39	7.75	7.38	7.58	7.78	7.88	7.94	7.94	7.96	7.97
Count	33	33	33	27	31	28	32	23	30	32	31	20	19
Minimum	7.01	6.36	6.98	4.64	7.03	6.23	6.53	6.6	5.13	6.57	5.63	6.92	6.93
Maximum	9.97	8.5	9.03	8.1	8.31	8.4	8.84	8.65	8.8	8.9	8.71	8.7	8.63
Geometric Mean	7.89	7.36	7.77	7.36	7.75	7.37	7.56	7.75	7.85	7.93	7.91	7.94	7.96
Count <DL	0	0	0	0	0	0	0	0	0	0	0	0	0
Standard Deviation	0.55	0.4	0.49	0.64	0.34	0.47	0.41	0.6	0.63	0.52	0.6	0.46	0.5
1st Quartile	7.56	7.12	7.29	7.21	7.5	7.09	7.4	7.33	7.79	7.77	7.83	7.8	7.82
Median	7.9	7.34	7.83	7.53	7.8	7.38	7.6	7.96	7.96	7.94	8.06	8.05	8.1
3rd Quartile	8.09	7.62	8.1	7.75	7.96	7.61	7.8	8.27	8.21	8.23	8.25	8.21	8.25
Count Over CWQG	1	1	1	1	0	1	0	0	1	0	1	0	0
% Over CWQG	3	3	3	3.7	0	3.6	0	0	3.3	0	3.2	0	0

Total Suspended Solids

Total suspended solids (TSS) values were generally low during sampling events, with average values ranging between 3 mg/L and 14 mg/L for all sites (Table 5). Elevations in TSS or spikes tend to occur during freshet as well as following rain events.

Table 5 Total Suspended Solids – Surface Water Quality Summary Statistics for 2005 – 2011

Statistic	W2	W3	W4	W5	W6	W7	W9	W10	W11	W12	W13	Y1	Y2
Average	6	5	6	6	13	3	3	6	5	6	13	14	13
Count	33	33	34	27	31	28	35	22	32	32	32	23	24
Minimum	0	0	0	0	0	0	0	0	0	0	0	1	1
Maximum	100	26	76	35	123	29	24	42	34	40	120	98	90
Geometric Mean	2	2	2	3	4	1	2	2	2	3	4	5	5
Count <DL	17	16	17	11	9	21	19	14	16	12	13	7	5
Standard Deviation	18	6	14	9	25	7	6	12	8	9	26	23	21
1st Quartile	1	1	1	1	1	1	1	1	1	1	1	1	2
Median	1	2	1	2	3	1	1	1	2	2	2	6	5
3rd Quartile	3	5	4	7	12	1	3	4	3	7	12	14	14

Total Aluminum

As shown in Table 6, total aluminum concentrations regularly exceed the CWQG of 0.1 mg/L (at pH $\geq$ 6.5, or 0.005 mg/L at pH < 6.5). A site specific water quality objective (SSWQO) of 0.66 mg/L has been derived using CCME protocols by Minnow Environmental Inc. (assessing water quality data collected between 2005 and 2008). Average concentrations for all sites between 2005 and 2011 range between 0.0724 mg/L (W7) and 0.3937 mg/L (Y2). Downstream stations where fish have been observed (W10, W11, and W13) have exceeded the site specific objective as have most other stations including the Yukon River. However, as shown in Figure 4, since 2009 no exceedances of the SSWQO have been observed aside from once at W12 and W13 in May 2011 during spring freshet.

Table 6 Total Aluminum – Surface Water Quality Summary Statistics for 2005 – 2011

Statistic	W2	W3	W4	W5	W6	W7	W9	W10	W11	W12	W13	Y1	Y2
Average	0.1053	0.1303	0.1154	0.1487	0.1828	0.0724	0.0754	0.1778	0.1064	0.1413	0.1887	0.3135	0.3937
Count	35	37	36	27	33	32	37	23	33	33	34	23	24
Minimum	0.01	0.006	0.01	0.01	0.01	0.025	0.0055	0.005	0.0088	0.01	0.0025	0.019	0.01
Maximum	1.44	1.76	1.48	1.68	2.09	0.405	0.623	1.08	0.746	1.25	1.92	2.1	2.04
Geometric Mean	0.048	0.0356	0.0478	0.0653	0.0764	0.0539	0.0368	0.0619	0.0485	0.0647	0.0547	0.1147	0.1764
Count <DL	1	1	1	0	1	0	2	1	1	1	3	0	1
Standard Deviation	0.2404	0.3299	0.2559	0.3216	0.3768	0.0856	0.1185	0.2767	0.1706	0.2376	0.379	0.4964	0.5281
1st Quartile	0.0205	0.0122	0.0238	0.0295	0.0308	0.0358	0.017	0.0191	0.0226	0.0274	0.0157	0.0279	0.07
Median	0.0431	0.026	0.0367	0.043	0.0727	0.0426	0.029	0.078	0.04	0.05	0.041	0.0894	0.229
3rd Quartile	0.092	0.08	0.0675	0.151	0.14	0.0607	0.058	0.1585	0.096	0.156	0.1555	0.4155	0.4632
Count Over CWQG	7	10	7	9	14	7	7	9	8	11	13	12	15
% Over CWQG	20	27	19.4	33.3	42.4	21.9	18.9	39.1	24.2	33.3	38.2	52.2	62.5

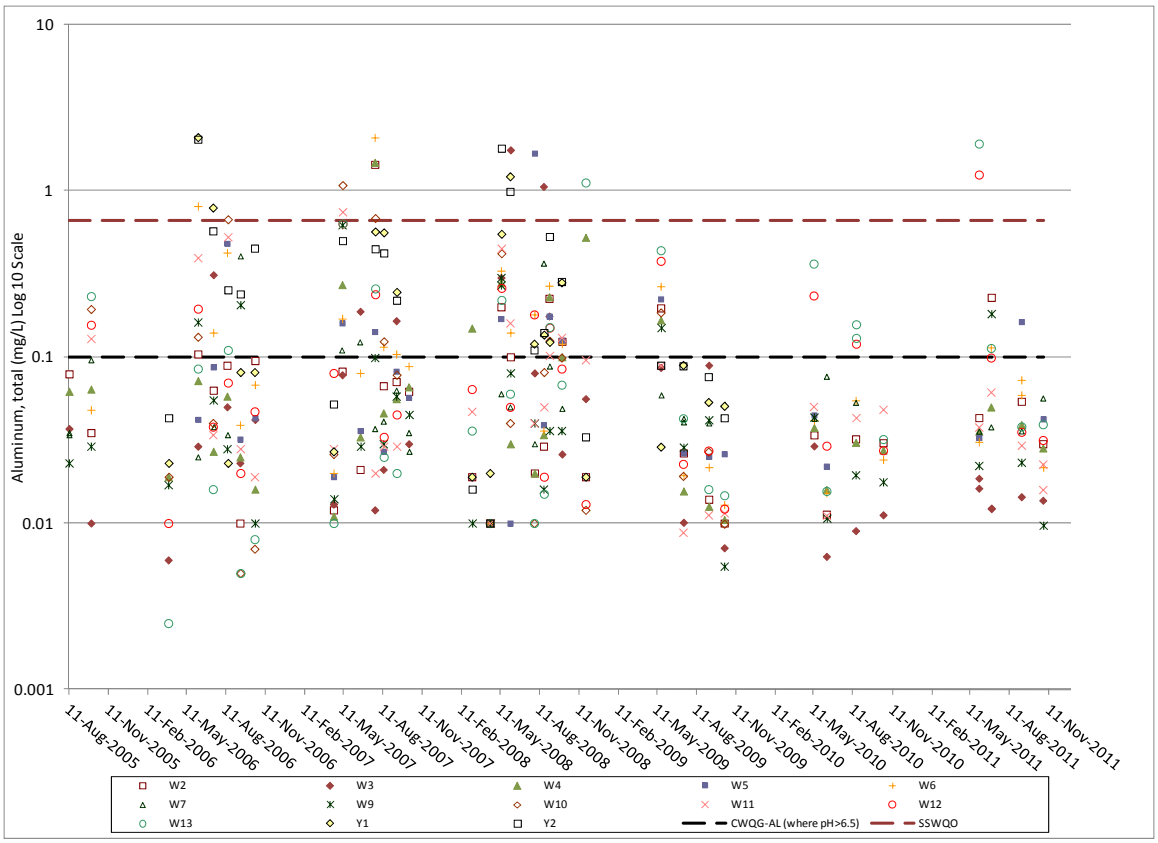


Figure 4 Williams Creek Watershed, Total Aluminum

Total Arsenic

Table 7 shows that site arsenic concentrations are well below the CWQG of 0.005 mg/L. Average arsenic values for the sites are estimated to range between 0.00044 and 0.00059 mg/L.

Table 7 Total Arsenic – Surface Water Quality Summary Statistics for 2005 – 2011

Statistic	W2	W3	W4	W5	W6	W7	W9	W10	W11	W12	W13	Y1	Y2
Average	0.00056	0.0005	0.00054	0.00052	0.00055	0.00048	0.00051	0.00048	0.0005	0.00044	0.00047	0.00059	0.00057
Count	35	37	36	27	33	32	37	23	33	33	34	23	24
Minimum	0.0001	0.0001	0.0001	0.0002	0.0001	0.0001	0.0001	0.0002	0.0002	0.0001	0.0001	0.0001	0.0002
Maximum	0.0016	0.0022	0.0014	0.0023	0.0013	0.0026	0.0014	0.0018	0.0016	0.00115	0.00155	0.0026	0.0016
Geometric Mean	0.0005	0.00044	0.00049	0.00045	0.00051	0.0004	0.00044	0.00044	0.00046	0.0004	0.00042	0.00048	0.00053
Count <DL	2	2	3	1	2	4	4	1	0	4	3	3	1
Standard Deviation	0.00028	0.00034	0.00025	0.00039	0.00023	0.00042	0.00027	0.00031	0.00024	0.0002	0.00025	0.0005	0.00027
1st Quartile	0.0004	0.0004	0.0004	0.00034	0.0004	0.00037	0.00038	0.00038	0.0004	0.0003	0.00035	0.0004	0.00044
Median	0.0005	0.00042	0.0005	0.0005	0.0005	0.0004	0.0005	0.0004	0.00044	0.0004	0.0004	0.0005	0.0005
3rd Quartile	0.00065	0.0005	0.0006	0.00055	0.00067	0.0005	0.0006	0.0005	0.0005	0.00051	0.00053	0.00055	0.0006

Total Cadmium

The CWQG for total cadmium is dependent on hardness, and Minnow Environmental Inc. calculated the guideline to range between 0.00003 and 0.00006 mg/L (depending on the station) in their *Evaluation of Water Quality and Development of Background-Based Site-Specific Water Quality Objectives for Williams Creek* (May 2008), which evaluated data collected between 2005 and 2007. Prior to 2009 most results were below a somewhat elevated RDL; improved laboratory RDLs were implemented in 2009 and since then data shows periodic exceedances of the hardness dependent guideline. Figure 5 shows cadmium results from 2009 to 2011 with a CWQG for cadmium calculated at 0.00004 mg/L based on an average hardness (130 mg/L) at all stations for the same period.

Table 8 Total Cadmium – Surface Water Quality Summary Statistics for 2005 – 2011

Statistic	W2	W3	W4	W5	W6	W7	W9	W10	W11	W12	W13	Y1	Y2
Average	0.000018	0.000015	0.000021	0.000013	0.000017	0.000028	0.000018	0.00002	0.000029	0.000411	0.000019	0.000018	0.000019
Count	35	37	36	27	33	32	37	23	33	33	34	23	24
Minimum	0.000003	0.000003	0.000003	0.000005	0.000003	0.000003	0.000003	0.000005	0.000003	0.000003	0.000003	0.000005	0.000005
Maximum	0.000077	0.00006	0.00012	0.000041	0.000089	0.00024	0.00008	0.00006	0.00034	0.0127	0.000076	0.000054	0.00005
Geometric Mean	0.000012	0.00001	0.000013	0.00001	0.000011	0.000014	0.000012	0.000014	0.000015	0.000019	0.000013	0.000012	0.000015
Count <DL	19	21	19	16	18	15	20	12	12	13	16	14	12
Standard Deviation	0.000017	0.000014	0.000024	0.000012	0.000017	0.000046	0.000017	0.000016	0.000059	0.002206	0.000016	0.000016	0.000013
1st Quartile	0.000005	0.000005	0.000005	0.000005	0.000005	0.000005	0.000005	0.000005	0.000009	0.000009	0.000005	0.000005	0.00001
Median	0.00001	0.000009	0.00001	0.00001	0.000011	0.00001	0.00001	0.000013	0.000012	0.000018	0.000012	0.00001	0.000016
3rd Quartile	0.000028	0.00002	0.000031	0.000015	0.00002	0.000028	0.000032	0.000031	0.000023	0.000035	0.000031	0.000027	0.000024
Count Over CWQG	24	24	25	19	21	23	26	19	21	25	24	20	21
% Over CWQG	68.6	64.9	69.4	70.4	63.6	71.9	70.3	82.6	63.6	75.8	70.6	87	87.5

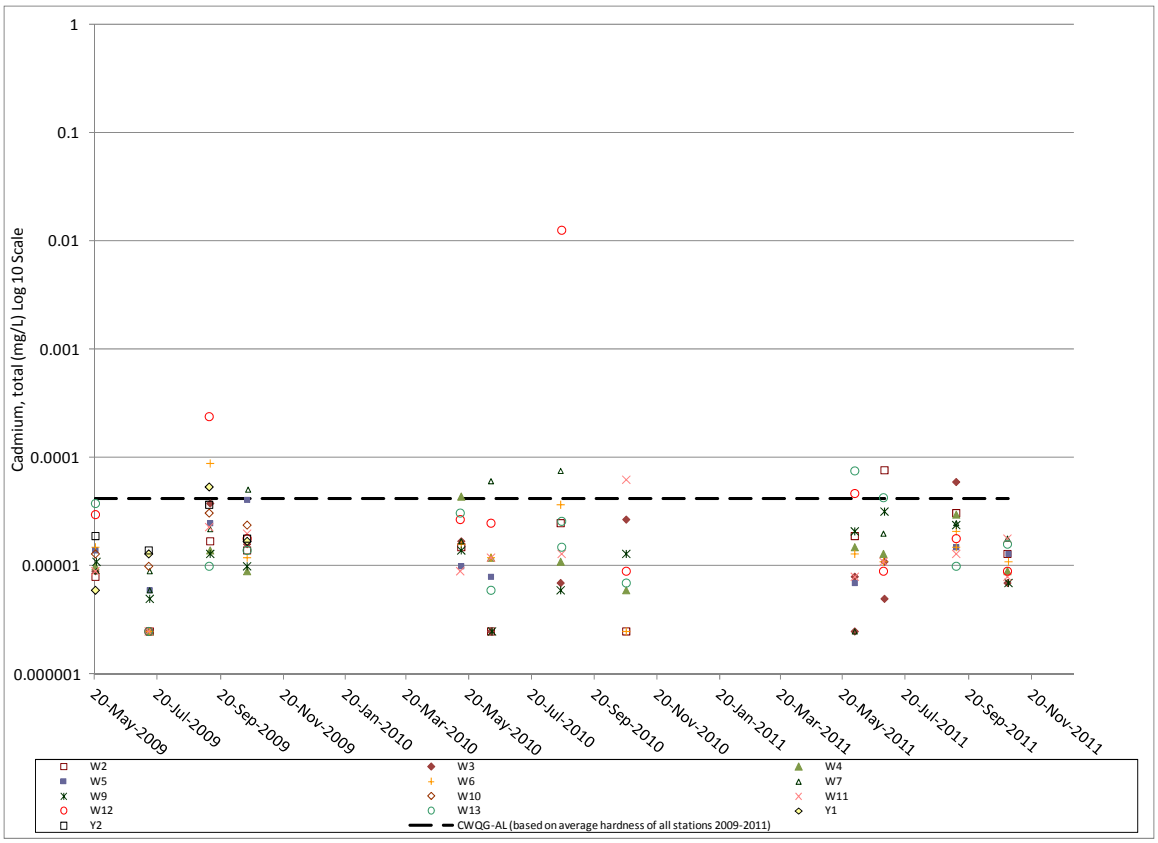


Figure 5 Williams Creek Watershed, Total Cadmium (2009-2011)

Total Copper

The CWQG for total copper is hardness dependent and regularly exceeded as shown in Table 10:

- 0.002 mg/L at hardness less than 120 mg/L;
- 0.003 mg/L at hardness from 120 – 180 mg/L; and
- 0.004 mg/L at hardness greater than 180 mg/L.

Minnow Environmental calculated the following hardness dependent, CCME derived site specific water quality objective for total copper:

- 0.021 mg/L at hardness less than 120 mg/L;
- 0.032 mg/L at hardness from 120 – 180 mg/L; and
- 0.042 mg/L at hardness greater than 180 mg/L.

Table 9 shows average concentrations for all sites ranging between 0.00137 mg/L (Y1) and 0.00464 mg/L (W7). The maximum total copper concentration at W7 of 0.066 mg/L is thought to be erroneous as North Williams Creek is narrow and shallow and mobilization of sediment or soil may have incidentally contaminated the sample. Figure 6 shows copper results with a CWQG calculated at 0.003 mg/L and the SSWQO of 0.032 mg/L based on an average hardness (130 mg/L) at all stations from 2009-2011. Aside from the erroneous measurement at W7, no values exceed the SSWQO.

Table 9 Total Copper – Surface Water Quality Summary Statistics for 2005 – 2011

Statistic	W2	W3	W4	W5	W6	W7	W9	W10	W11	W12	W13	Y1	Y2
Average	0.00167	0.00189	0.00211	0.002	0.00174	0.00464	0.00146	0.00248	0.00251	0.00247	0.00224	0.00137	0.00182
Count	35	37	36	27	33	32	37	23	33	33	34	23	24
Minimum	0.00048	0.0005	0.0005	0.0005	0.0005	0.001	0.00043	0.001	0.0007	0.0005	0.0005	0.0005	0.0005
Maximum	0.008	0.005	0.014	0.007	0.00555	0.066	0.004	0.004	0.005	0.0059	0.007	0.004	0.008
Geometric Mean	0.00134	0.00167	0.00154	0.00171	0.00144	0.00278	0.00122	0.00232	0.00234	0.00226	0.00189	0.0011	0.0013
Count <DL	6	3	6	2	5	1	7	1	1	2	4	7	8
Standard Deviation	0.00139	0.00101	0.00234	0.00129	0.00118	0.01124	0.00085	0.0009	0.0009	0.00105	0.00148	0.00101	0.00177
1st Quartile	0.001	0.001	0.001	0.00103	0.001	0.002	0.00088	0.002	0.002	0.002	0.00146	0.00059	0.00064
Median	0.00136	0.0018	0.00152	0.002	0.0015	0.00255	0.00118	0.002	0.00239	0.002	0.002	0.001	0.001
3rd Quartile	0.002	0.002	0.00212	0.00288	0.002	0.003	0.002	0.003	0.003	0.003	0.00226	0.002	0.002
Count Over CWQG	5	8	9	9	6	20	5	9	16	12	7	2	5
% Over CWQG	14.3	21.6	25	33.3	18.2	62.5	13.5	39.1	48.5	36.4	20.6	8.7	20.8

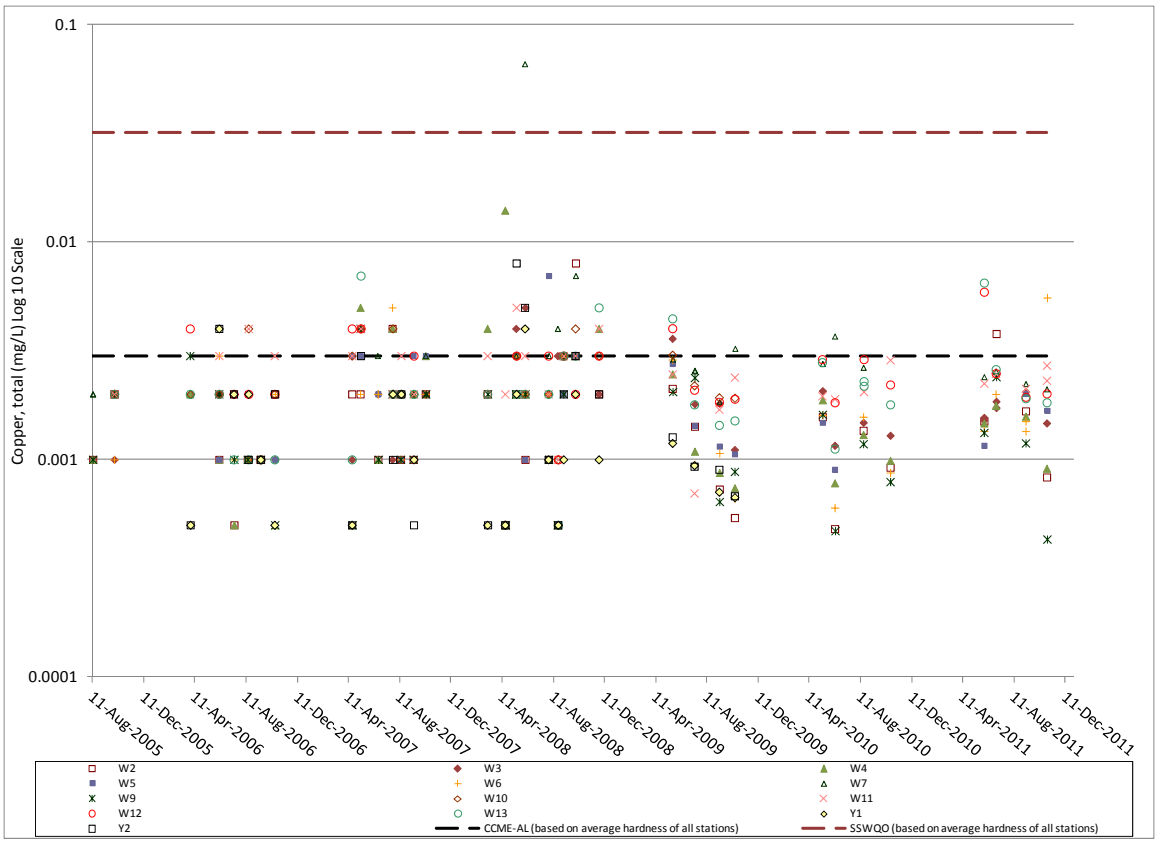


Figure 6 Williams Creek Watershed, Total Copper

Total Iron

In addition to total aluminum and copper, total iron concentrations also regularly exceed the CWQG of 0.3 mg/L (Table 10). Therefore, a CCME derived site specific water quality objective of 1.1 mg/L has been calculated by Minnow Environmental (assessing water quality data collected between 2005 and 2008). Average concentrations shown in Table 10 for all sites range between 0.235 mg/L (W9) and 0.652 mg/L (W5). Downstream stations where fish have been observed (W10, W11, and W13) periodically exceed the site specific objective as do most other stations including the Yukon River. However, as shown in Figure 7, since 2009 no exceedances of the SSWQO have been observed aside from once at W12 and W13 in May 2011 during spring freshet.

Table 10 Total Iron – Surface Water Quality Summary Statistics for 2005 – 2011

Statistic	W2	W3	W4	W5	W6	W7	W9	W10	W11	W12	W13	Y1	Y2
Average	0.399	0.407	0.376	0.652	0.498	0.273	0.235	0.285	0.239	0.252	0.304	0.421	0.578
Count	35	37	36	27	33	32	37	23	33	33	34	23	24
Minimum	0.01	0.01	0.01	0.05	0.01	0.05	0.04	0.029	0.02	0.03	0.02	0.04	0.04
Maximum	2.2	4.66	2.2	6.09	2.8	2.2	0.84	1.2	1.2	2.02	3.04	2.4	3.44
Geometric Mean	0.298	0.186	0.281	0.421	0.354	0.17	0.172	0.158	0.145	0.14	0.121	0.187	0.27
Count <DL	3	4	2	1	2	3	2	3	8	7	9	5	3
Standard Deviation	0.362	0.828	0.367	1.113	0.501	0.41	0.214	0.337	0.279	0.368	0.571	0.63	0.829
1st Quartile	0.29	0.1	0.257	0.3	0.3	0.1	0.1	0.067	0.055	0.053	0.05	0.079	0.129
Median	0.31	0.181	0.3	0.373	0.343	0.12	0.147	0.11	0.154	0.1	0.1	0.16	0.26
3rd Quartile	0.4	0.3	0.355	0.567	0.47	0.201	0.3	0.381	0.23	0.3	0.296	0.405	0.625
Count Over CWQG	18	9	12	19	21	7	6	7	7	8	7	7	10
% Over CWQG	51.4	24.3	33.3	70.4	63.6	21.9	16.2	30.4	21.2	24.2	20.6	30.4	41.7

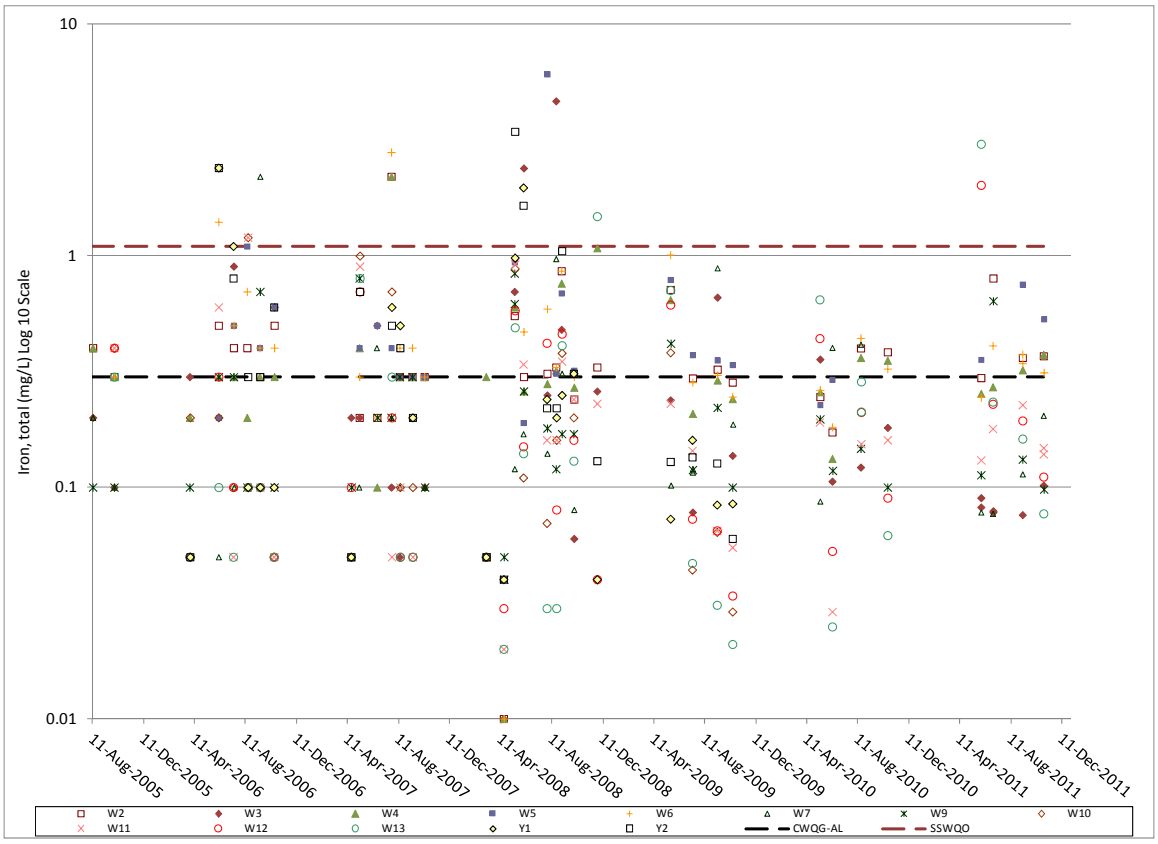


Figure 7 Williams Creek Watershed, Total Iron

Total Lead

The CWQG for total lead is dependent on hardness (typically ranging between 0.002 mg/L and 0.007 mg/L). Table 11 shows occasional exceedances of the guideline, while average concentrations range between 0.00014 mg/L and 0.00055 mg/L.

Table 11 Total Lead – Water Quality Summary Statistics for 2005 – 2011

Statistic	W2	W3	W4	W5	W6	W7	W9	W10	W11	W12	W13	Y1	Y2
Average	0.000259	0.000286	0.000185	0.000139	0.00015	0.000551	0.000344	0.000186	0.000378	0.000246	0.000388	0.000213	0.000374
Count	35	37	36	27	33	32	37	23	33	33	34	23	24
Minimum	0.000011	0.000008	0.000011	0.000014	0.000008	0.000009	0.000013	0.000036	0.000003	0.00002	0.000022	0.000024	0.00005
Maximum	0.0033	0.0038	0.0014	0.001	0.0006	0.0111	0.0037	0.0005	0.0071	0.0016	0.0037	0.0008	0.0019
Geometric Mean	0.000105	0.00011	0.000101	0.000093	0.000105	0.000122	0.000122	0.00014	0.000101	0.000146	0.000156	0.000137	0.000229
Count <DL	16	7	12	8	9	10	11	7	12	5	9	8	4
Standard Deviation	0.000581	0.000651	0.000274	0.000183	0.000133	0.001952	0.00072	0.000133	0.001234	0.000315	0.000697	0.000205	0.00043
1st Quartile	0.00005	0.00005	0.00005	0.00005	0.00005	0.00005	0.00005	0.000075	0.00005	0.000085	0.00005	0.00005	0.0001
Median	0.0001	0.0001	0.0001	0.0001	0.0001	0.0001	0.0001	0.000146	0.000083	0.000113	0.0001	0.00013	0.000218
3rd Quartile	0.000163	0.0002	0.000198	0.000187	0.0002	0.00022	0.0003	0.0003	0.0002	0.0003	0.000293	0.0003	0.000325
Count Over CWQG	1	2	1	0	0	2	3	0	2	1	3	0	1
% Over CWQG	2.9	5.4	2.8	0	0	6.2	8.1	0	6.1	3	8.8	0	4.2

Total Mercury

Total mercury concentrations infrequently register above laboratory detection levels (RDL = 0.00001 mg/L since October 2007). The CWQG for inorganic mercury is 0.000026 mg/L and average total mercury has been estimated to range between 0.000006 mg/L and 0.000012 mg/L using ½ the RDL to calculate (Table 12). The one exceedance noted at each station save Y1 is due to a poor detection level in May 2007 (0.0001 mg/L) which is above the guideline.

Table 12 Total Mercury – Water Quality Summary Statistics for 2005 – 2011

Statistic	W2	W3	W4	W5	W6	W7	W9	W10	W11	W12	W13	Y1	Y2
Average	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001
Count	26	27	26	20	25	24	27	15	25	24	25	16	17
Minimum	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001
Maximum	0.00005	0.00005	0.00005	0.00005	0.00005	0.00005	0.00006	0.00005	0.00005	0.00005	0.00005	0.00001	0.00005
Geometric Mean	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001
Count <DL	25	23	23	17	21	18	25	14	23	22	23	16	16
Standard Deviation	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0	0.00001
1st Quartile	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001
Median	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001
3rd Quartile	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001	0.00001
Count Over CWQG	1	1	1	1	1	1	2	1	1	1	1	0	1
% Over CWQG	3.8	3.7	3.8	5	4	4.2	7.4	6.7	4	4.2	4	0	5.9

Total Molybdenum

The CWQG for total molybdenum is 0.073 mg/L and has not been exceeded at any of the stations. Average total molybdenum ranges between 0.00037 mg/L and 0.00397 mg/L.

Table 13 Total Molybdenum – Water Quality Summary Statistics for 2005 – 2011

Statistic	W2	W3	W4	W5	W6	W7	W9	W10	W11	W12	W13	Y1	Y2
Average	0.00394	0.00077	0.00397	0.00059	0.00307	0.00037	0.00085	0.00133	0.00064	0.00141	0.00239	0.00097	0.00095
Count	35	37	36	27	33	32	37	23	33	33	34	23	24
Minimum	0.0005	0.00028	0.00066	0.00026	0.00062	0.00001	0.00015	0.00034	0.00019	0.00037	0.0007	0.0005	0.00033
Maximum	0.009	0.00183	0.026	0.001	0.007	0.001	0.004	0.003	0.002	0.003	0.005	0.00131	0.00138
Geometric Mean	0.00326	0.00071	0.00301	0.00056	0.00252	0.00025	0.00064	0.00122	0.00058	0.00126	0.00221	0.00096	0.00092
Count <DL	2	16	2	13	1	16	11	4	11	5	1	3	5
Standard Deviation	0.00212	0.00031	0.00416	0.0002	0.0018	0.00028	0.00084	0.00057	0.00034	0.00065	0.00087	0.00017	0.00022
1st Quartile	0.00208	0.0005	0.00195	0.0005	0.0014	0.00015	0.00049	0.001	0.00049	0.00095	0.002	0.00098	0.00091
Median	0.004	0.00073	0.00334	0.0005	0.00256	0.0004	0.00053	0.00118	0.00051	0.00138	0.00251	0.001	0.001
3rd Quartile	0.00509	0.001	0.00481	0.0007	0.00414	0.0005	0.001	0.00166	0.00077	0.00191	0.00299	0.00101	0.001
Count Over CWQG	0	0	0	0	0	0	0	0	0	0	0	0	0
% Over CWQG	0	0	0	0	0	0	0	0	0	0	0	0	0

Total Nickel

The CWQG for total nickel is dependent on hardness, and typically ranges between 0.065 mg/L and 0.15 mg/L (depending on the station) (Minnow, 2008). Average concentrations (Table 14) range between 0.00105 mg/L and 0.00244 mg/L.

Table 14 Total Nickel – Water Quality Summary Statistics for 2005 – 2011

Statistic	W2	W3	W4	W5	W6	W7	W9	W10	W11	W12	W13	Y1	Y2
Average	0.00106	0.0013	0.00105	0.00193	0.00128	0.00151	0.00107	0.0011	0.00125	0.00112	0.00114	0.00141	0.00244
Count	35	37	36	27	33	32	37	23	33	33	34	23	24
Minimum	0.0005	0.00025	0.00025	0.00025	0.00025	0.00025	0.00025	0.00025	0.00025	0.00025	0.00025	0.00025	0.00025
Maximum	0.005	0.01	0.006	0.017	0.009	0.004	0.007	0.008	0.01	0.006	0.005	0.009	0.03
Geometric Mean	0.00092	0.00088	0.00085	0.00117	0.00102	0.00137	0.00073	0.00077	0.00096	0.00084	0.00083	0.00086	0.00107
Count <DL	5	7	7	3	4	2	9	7	5	10	9	7	6
Standard Deviation	0.00081	0.00186	0.00097	0.00333	0.00146	0.00065	0.00139	0.00159	0.00162	0.00111	0.0011	0.00197	0.00597
1st Quartile	0.00069	0.00061	0.0006	0.00084	0.00076	0.00119	0.0005	0.0005	0.0008	0.0005	0.0005	0.0005	0.00059
Median	0.00086	0.00072	0.00079	0.00101	0.001	0.00138	0.00078	0.00065	0.001	0.0007	0.00087	0.00063	0.00095
3rd Quartile	0.00108	0.001	0.00106	0.00135	0.0013	0.002	0.001	0.0009	0.0011	0.0012	0.00118	0.0011	0.00145
Count Over CWQG	0	0	0	0	0	0	0	0	0	0	0	0	1
% Over CWQG	0	0	0	0	0	0	0	0	0	0	0	0	4.2

Total Selenium

Total selenium concentrations (shown in Table 15) are below the CWQG of 0.001 mg/L.

Table 15 Total Selenium – Water Quality Summary Statistics for 2005 – 2011

Statistic	W2	W3	W4	W5	W6	W7	W9	W10	W11	W12	W13	Y1	Y2
Average	0.00019	0.00014	0.00019	0.00013	0.00015	0.00015	0.00018	0.00017	0.00017	0.00019	0.0002	0.00023	0.0002
Count	35	37	36	27	33	32	37	23	33	33	34	23	24
Minimum	0.00007	0.00002	0.00007	0.00002	0.00006	0.00005	0.00006	0.00008	0.00005	0.00005	0.00008	0.0001	0.0001
Maximum	0.0005	0.0008	0.0009	0.0003	0.0003	0.0003	0.0009	0.0003	0.001	0.0006	0.0007	0.0009	0.0003
Geometric Mean	0.00015	0.0001	0.00015	0.0001	0.00013	0.00012	0.00014	0.00015	0.00013	0.00015	0.00016	0.00019	0.00019
Count <DL	18	22	19	21	19	17	22	19	19	18	17	16	17
Standard Deviation	0.00013	0.00014	0.00018	0.0001	0.00009	0.0001	0.00017	0.00009	0.00017	0.00013	0.00014	0.00018	0.00008
1st Quartile	0.00009	0.00006	0.00009	0.00006	0.0001	0.00007	0.0001	0.0001	0.00009	0.0001	0.0001	0.0001	0.00012
Median	0.0001	0.0001	0.0001	0.0001	0.0001	0.0001	0.0001	0.00013	0.0001	0.00011	0.00013	0.00019	0.0002
3rd Quartile	0.0003	0.0002	0.00022	0.0002	0.0002	0.00022	0.0003	0.00025	0.0002	0.0003	0.00027	0.0003	0.0003
Count Over CWQG	0	0	0	0	0	0	0	0	0	0	0	0	0
% Over CWQG	0	0	0	0	0	0	0	0	0	0	0	0	0

Total Zinc

Total zinc exceeded the CWQG (0.03 mg/L) once during 2011 sampling at W13 in June. Average total zinc concentrations (Table 16) range between 0.0036 mg/L and 0.0057 mg/L.

Table 16 Total Zinc – Water Quality Summary Statistics for 2005 – 2011

Statistic	W2	W3	W4	W5	W6	W7	W9	W10	W11	W12	W13	Y1	Y2
Average	0.0054	0.0036	0.0057	0.0042	0.0042	0.0054	0.0044	0.0048	0.0049	0.005	0.0057	0.0047	0.0054
Count	35	37	36	27	33	32	37	23	33	33	34	23	24
Minimum	0.0005	0.0003	0.0002	0.0005	0.0003	0.0005	0.0004	0.0007	0.0005	0.0004	0.0003	0.0005	0.001
Maximum	0.022	0.012	0.027	0.016	0.023	0.027	0.02	0.02	0.022	0.0168	0.0518	0.014	0.019
Geometric Mean	0.0037	0.0025	0.0031	0.0029	0.0028	0.0034	0.0031	0.0036	0.0035	0.0035	0.0032	0.0034	0.004
Count <DL	1	3	2	1	0	3	2	0	0	1	1	2	0
Standard Deviation	0.0048	0.0031	0.0066	0.0038	0.0045	0.0058	0.0039	0.0043	0.0042	0.0039	0.0089	0.0034	0.0044
1st Quartile	0.0019	0.0012	0.0017	0.002	0.0016	0.0019	0.0019	0.002	0.002	0.002	0.0017	0.002	0.0027
Median	0.004	0.0026	0.004	0.0031	0.003	0.0035	0.003	0.004	0.004	0.0038	0.0033	0.005	0.0046
3rd Quartile	0.0071	0.0056	0.0053	0.0055	0.004	0.0058	0.006	0.0055	0.007	0.008	0.0067	0.007	0.0063
Count Over CWQG	0	0	0	0	0	0	0	0	0	0	1	0	0
% Over CWQG	0	0	0	0	0	0	0	0	0	0	2.9	0	0

Total Ammonia

The CWQG for total ammonia is pH and water temperature dependent. The lowest applicable guideline determined by Minnow Environmental (May 2008) was 0.12 mg/L. Average total ammonia concentrations range between 0.021 mg/L and 0.035 mg/L (Table 17).

Table 17 Ammonia Nitrogen – Water Quality Summary Statistics for 2005 – 2011

Statistic	W2	W3	W4	W5	W6	W7	W9	W10	W11	W12	W13	Y1	Y2
Average	0.028	0.026	0.026	0.029	0.034	0.035	0.025	0.021	0.025	0.028	0.027	0.021	0.022
Count	35	35	36	27	33	30	37	23	33	32	32	23	24
Minimum	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003
Maximum	0.15	0.11	0.18	0.097	0.17	0.19	0.17	0.025	0.16	0.18	0.19	0.025	0.025
Geometric Mean	0.022	0.02	0.019	0.022	0.024	0.026	0.018	0.017	0.018	0.021	0.018	0.017	0.019
Count <DL	25	25	30	20	22	17	30	23	25	24	26	23	22
Standard Deviation	0.024	0.02	0.028	0.022	0.031	0.035	0.026	0.009	0.026	0.03	0.032	0.009	0.008
1st Quartile	0.025	0.022	0.025	0.025	0.025	0.025	0.025	0.025	0.017	0.024	0.022	0.025	0.025
Median	0.025	0.025	0.025	0.025	0.025	0.025	0.025	0.025	0.025	0.025	0.025	0.025	0.025
3rd Quartile	0.025	0.025	0.025	0.025	0.025	0.025	0.025	0.025	0.025	0.025	0.025	0.025	0.025

Nitrate Nitrogen

The CWQG for nitrate nitrogen is 2.9 mg/L, which has not been exceeded at any station. Average nitrate nitrogen concentrations range between 0.02 mg/L and 0.13 mg/L (Table 18).

Table 18 Nitrate Nitrogen – Water Quality Summary Statistics for 2005 – 2011

Statistic	W2	W3	W4	W5	W6	W7	W9	W10	W11	W12	W13	Y1	Y2
Average	0.09	0.09	0.05	0.03	0.04	0.02	0.03	0.06	0.13	0.04	0.05	0.03	0.05
Count	30	29	30	23	28	26	32	17	28	26	26	17	18
Minimum	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Maximum	1.2	2	0.2	0.1	0.2	0.15	0.2	0.3	2.55	0.2	0.22	0.2	0.2
Geometric Mean	0.03	0.02	0.03	0.02	0.02	0.01	0.02	0.03	0.02	0.02	0.03	0.02	0.03
Count <DL	15	20	16	18	16	22	22	6	18	13	13	8	7
Standard Deviation	0.22	0.37	0.07	0.03	0.05	0.03	0.04	0.09	0.48	0.04	0.06	0.05	0.06
1st Quartile	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Median	0.03	0.01	0.02	0.01	0.02	0.01	0.01	0.04	0.01	0.01	0.03	0.02	0.03
3rd Quartile	0.07	0.03	0.06	0.05	0.05	0.03	0.04	0.06	0.05	0.05	0.07	0.04	0.05
Count Over Standard	0	0	0	0	0	0	0	0	0	0	0	0	0

Nitrite Nitrogen

The CWQG for nitrite nitrogen is 0.06 mg/L and has not been exceeded (Table 19). With most results falling below the RDL, average nitrite nitrogen concentrations range between 0.01 mg/L and 0.016 mg/L.

Table 19 Nitrite Nitrogen – Water Quality Summary Statistics for 2005 – 2011

Statistic	W2	W3	W4	W5	W6	W7	W9	W10	W11	W12	W13	Y1	Y2
Average	0.01	0.011	0.01	0.012	0.01	0.015	0.012	0.014	0.01	0.011	0.013	0.014	0.016
Count	30	29	30	23	28	26	32	17	28	26	26	17	18
Minimum	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003
Maximum	0.04	0.05	0.04	0.04	0.04	0.07	0.06	0.04	0.04	0.04	0.04	0.04	0.05
Geometric Mean	0.006	0.006	0.006	0.007	0.006	0.008	0.007	0.009	0.006	0.006	0.007	0.008	0.01
Count <DL	28	26	27	20	26	19	27	13	24	22	21	13	11
Standard Deviation	0.011	0.013	0.011	0.012	0.011	0.018	0.014	0.012	0.011	0.011	0.013	0.013	0.013
1st Quartile	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.003	0.004
Median	0.004	0.003	0.004	0.005	0.003	0.005	0.005	0.01	0.003	0.005	0.005	0.01	0.01
3rd Quartile	0.01	0.02	0.01	0.025	0.014	0.025	0.014	0.025	0.013	0.018	0.025	0.025	0.025
Count Over Standard	0	0	0	0	0	1	0	0	0	0	0	0	0

3.0 SURFACE WATER HYDROLOGY

Surface water hydrology, or discharge, relates the dynamics of flow in surface water systems with correlations drawn between precipitation and surface runoff. Discharge quantities are generally determined by topography, season, drainage, land-use, and soil conditions. Instantaneous flow measurements are collected when conditions permit (ice free/sufficient flow; typically May through October) during water sampling events. Data logging hydrometric instrumentation at stations W4, W10 and W12 has also been used to record stage and discharge records on a more continuous basis since 2006.

During 2009 spring freshet, the hydrology station at W10 became dislodged from the creek while the one at W4 was moderately affected. In July 2009 new hydrology stations were constructed at W4 and W12. The W10 hydrology station was not re-established as W12 was determined to be the proposed point of compliance for mine receiving water quality standards. In May 2011, hydrology stations W4 and W12 were visited to install data logging instruments. Barometric and water level data was logged until October 2011 when the loggers were last downloaded.

3.1 PROTOCOL

Manual water flow measurements may not always be possible due to ice and flow conditions. Bucket flows or estimates may be used as a substitute. Flow measurements are obtained by following Industry recognized Best Management Practices and Standard Operating Procedures.

3.2 RESULTS

Manual flow measurements collected from Williams Creek during 2006 to 2011 and subsequent calculations of discharge are shown on Figure 8. Figure 9 shows level logger data processed for discharge at station W4 as well as manually metered measurements collected between 2006 and 2011. Figure 10 shows level logger data processed for discharge at station W10 between 2006 and 2008, and at W12 for 2009 and 2011 as well as manual measurements.

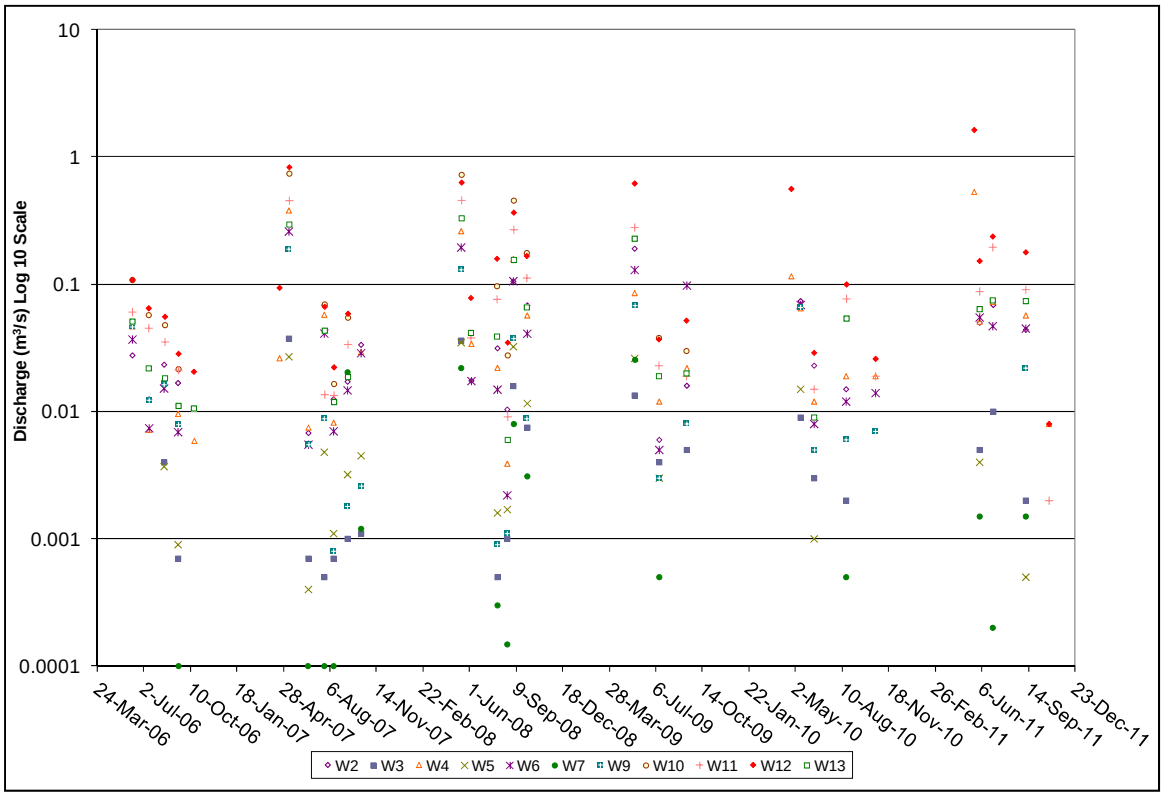


Figure 8 Williams Creek Watershed Manual Measurements of Discharge 2006 to 2011

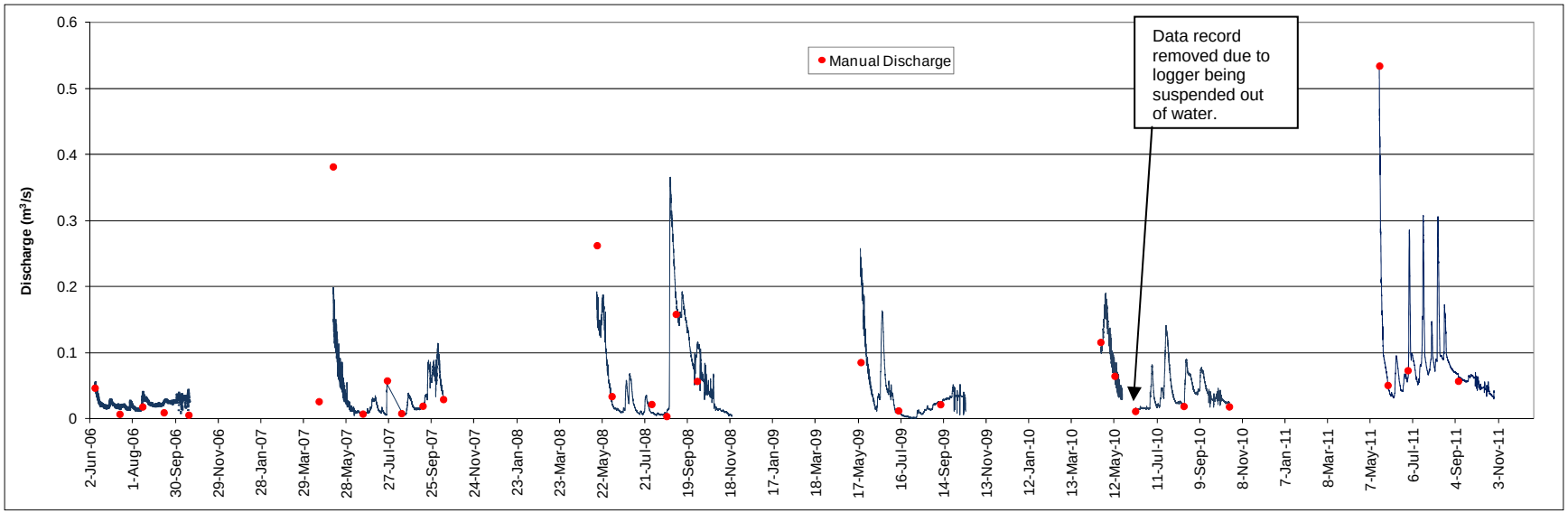


Figure 9 Williams Creek Station W4 Discharge for 2006 to 2011 Using Logger Instrumentation & Manual Methods

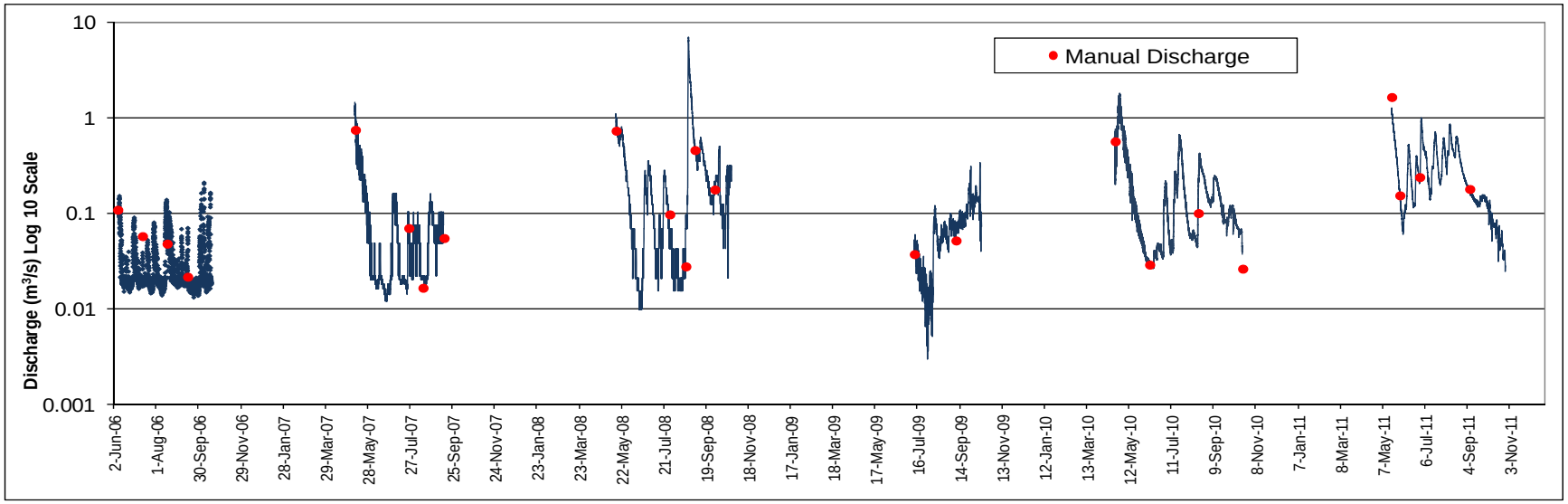


Figure 10 Williams Creek Station W10 Discharge for 2006 to 2008, and Station W12 for 2009 and 2011, Using Logger Instrumentation & Manual Methods

3.3 DISCUSSION

Figures 8 to 10 show peak flows occur during spring freshet with less dramatic increases in flow during the summer following rainfall events. A substantial increase in flow on August 25, 2008 was observed after a heavy rainfall event which resulted in 62.3 mm falling in little more than 24 hours. By October ice usually starts to form on the stream banks and beds and discharge can become elevated when ice that has formed melts during the day and break up occurs.

In 2006 and 2009 the data loggers were installed after freshet in lower Williams Creek, however, in 2007 and 2008 freshet was partially captured at W10. In 2007, the loggers were launched on May 10 at station W10; the following statistics are provided for the period of May 10 to June 10 2007:

- Minimum discharge = 0.016 m^3/s;
- Maximum discharge = 1.456 m^3/s.

In 2008, the loggers were launched on May 14 at station W10; the following summary statistics are provided for the period of May 14 to June 14, 2008:

- Minimum discharge = 0.015 m^3/s;
- Maximum discharge = 1.107 m^3/s.

In 2010 loggers were launched on April 22 at W4 and W12 to capture spring freshet. Because the loggers were installed over creek-bed ice, confidence in the data before mid-May is lowered and could be +/- 25%. The W4 level logger became suspended out of water in May and resulted in part of the data record being removed until the logger was dropped to the creek bed in June.

In 2011 loggers were launched on May 19 at W4 and W12 and last downloaded October 27. Maximum discharge recorded at W12 during freshet as well as the rest of the year was on the same day the logger was installed, at 1.266 m^3/s

4.0 CLIMATE

In 1994, Water Resources Division of DIAND established an automatic meteorological station at the site. The station monitors air and soil temperature, solar radiation, wind speed, and precipitation. The station is operated by Yukon Government, Water Resources Branch and continuous records are available from September 1994 to September 2008, except where gaps occur due to equipment malfunctions. No data is available between September 2008 and May 2010 due to logger malfunction. Available temperature and rainfall data between 2007 and 2011 are summarized in Table 20, while Figures 11 and 12 show 2011 results. While it is assumed that rainfall recorded during winter is a result of snowmelt no data has been omitted.

Table 20 Williams Creek Temperature and Rainfall 2007 – 2011

Month	Average Monthly Temperature ˚C				2007-2011 Average	Total Rainfall mm				2007-2011 Average
	2007	2008*	2010	2011**		2007	2008*	2010	2011**	
January	-13.7	-20.3		-16.0	**-16.7**	0	0		0.5	**0.2**
February	-19.6	-16.7		-16.6	**-17.6**	0	0		11	**3.7**
March	-15.2	-6.4		-10.9	**-10.8**	0.4	3.4		1.7	**1.8**
April	1.2	0.4		0.5	**0.7**	4.3	7.6		1.4	**4.4**
May	7.8	7.4		8.8	**8.0**	13.4	15.1		9.5	**12.7**
June	14.1	11.2	12.7	12.2	**12.6**	36.7	44.2	51	72.9	**51.2**
July	15.0	12.1	14.5	13.6	**13.8**	63	49	83.3	74.4	**67.4**
August	13.9	10.1	14.0	10.1	**12.0**	45.6	111	40.6	51.8	**62.3**
September	5.7	8.2	6.4	6.7	**6.8**	36.9	15.9	21	7.7	**20.4**
October	-3.2		-0.9	-1.4	**-1.8**	20.1		12.8	5.8	**12.9**
November	-7.4		-8.1	-17.1	**-10.9**	0.4		0.9	1.5	**0.9**
December	-18.3		-21.3	-8.1	**-15.9**	0		0	0.2	**0.1**

* 2008 data available to September 12, 2008.

** 2011 data available to December 20, 2011.

Figure 11 Williams Creek 2011 Air Temperature

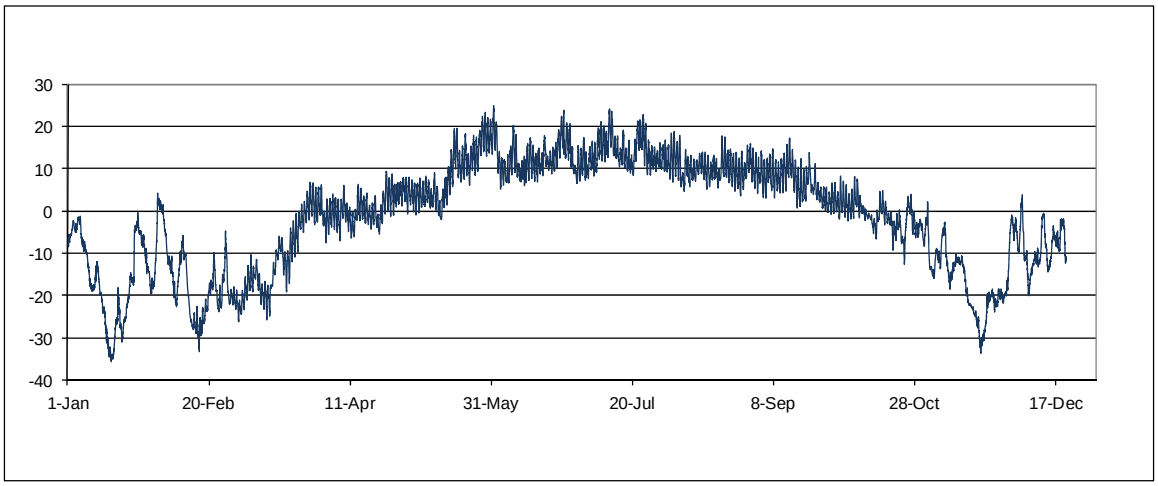
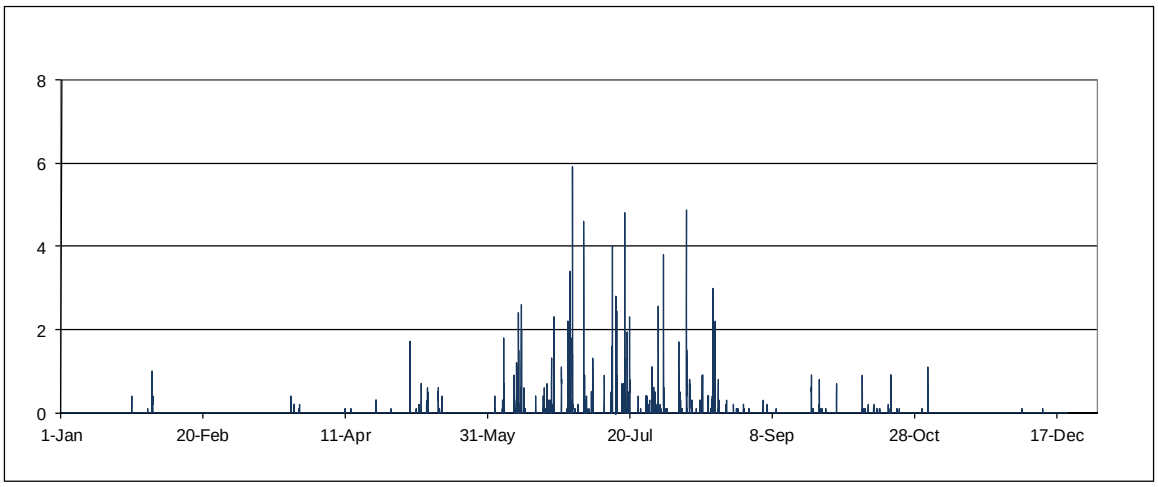


Figure 12 Williams Creek 2011 Rainfall



A snow course station also exists at the site and is monitored by Yukon Government in February, March and April. The Williams Creek snow course station data for February between 2008 and 2011 follows. The station was not surveyed in February 2012.

- February 23, 2011
  - snow depth = 72 cm
  - water content = 153 mm
- February 23, 2010
  - snow depth = 53 cm
  - water content = 96 mm
- February 24, 2009

  - snow depth = 66 cm
  - water content = 134 mm
- February 27, 2008
  - snow depth = 62 cm
  - water content = 103 mm.

Average February water content over 16 years of monitoring at Williams Creek is 83 mm. As described in Yukon Government's Yukon Snow Survey Bulletin and Water Supply Forecast (March, 2012), snowpack conditions in the Carmacks area are normal for March 1st and an area wide average has been estimated to be 118 percent of normal.

5.0 GROUNDWATER MONITORING

Groundwater monitoring wells are typically inspected during site visits and groundwater quality samples are collected when possible. Samples collected are sent to an accredited laboratory for analysis of the same parameters as surface water samples including physical parameters (pH, conductivity, total suspended solids, total dissolved solids, turbidity, hardness); inorganic nonmetallic parameters (ammonium, nitrogen, phosphorous, orthophosphate, dissolved organic carbon, total inorganic carbon, total organic carbon); weak acid dissociable cyanide; and metals. In-situ measurements (temperature, pH, conductivity and ORP) are also collected during sample collection.

5.1 MONITORING WELL SAMPLE STATION LOCATIONS & SAMPLING FREQUENCY

A total of 11 monitoring wells were established on site in 1996 in the heap leach pad area, waste rock storage area, and open pit area (see Figure 3):

- Leach pad MW96-A to E
- Waste rock storage area MW96-F, G, H & I
- Open pit MW96-J and K

During initial inspections and monitoring of these wells, a few were removed from the monitoring program; one from the leach pad area and all in the waste rock storage area:

- MW96-B: bailer lodged in well;
- MW96-F: ice plugged;

- MW96-G: no PVC pipe (only casing);
- MW96-H: ice plugged;
- MW96-I: ice plugged.

New wells have been installed and are also being checked.

- HYD-WELL-1 or MW-01-07: installed late 2007 below heap leach pad sediment pond area (appears ice plugged at each visit, no samples able to be collected);
- DW08-01: shallow well installed summer of 2008 in proposed camp location;
- DW08-02: shallow well installed summer of 2008 in proposed camp location.

Most functioning monitoring wells were visited in June 2011 and a few were also checked in September 2011. No water sample was collected from the camp potable water well in 2011 as the camp was not used by Copper North.

Most of the existing monitoring wells will be decommissioned once project components are constructed. For the purposes of monitoring groundwater quality during operations of the Carmacks Copper mine, additional wells (or other complementary approaches) will be established on site, particularly downgradient of specific project components (i.e. heap leach area and waste rock storage area).

5.2 PROTOCOL

Sampling of groundwater was undertaken using Industry recognized Best Management Practices and Standard Operating Procedures for the collection of groundwater samples.

5.3 RESULTS

As with surface water quality data, groundwater quality data collected since 2005 are stored in an EQWin database that is customized to manage environmental sampling data. A summary of results for actively monitored and sampled wells is provided in Table 21.

Table 21 Groundwater Quality Summary Statistics 2006-2011

Monitoring Well	pH-F pH Units	pH-L pH Units	Ag-D mg/L	Al-D mg/L	As-D mg/L	Cd-D mg/L	Cr-D mg/L	Cu-D mg/L	Fe-D mg/L	Hg-D mg/L	Mo-D mg/L	Ni-D mg/L	Pb-D mg/L	Se-D mg/L	Tl-D mg/L	Zn-D mg/L	CN-WAD mg/L	Ammonia mg/L	Nitrite mg/L	Nitrate mg/L
MW96-A																				
Average	7.8	8.1	0.000037	0.0207	0.00106	0.000026	0.0004	0.00197	0.014	0.00001	0.00844	0.00055	0.000266	0.00042	0.000019	0.0068	0.0004	0.054	0.011	0.97
Count	12	10	15	15	15	15	15	15	15	7	15	15	15	15	15	15	4	15	10	10
Minimum	7.24	7.8	0.000003	0.0025	0.0001	0.000005	0.0001	0.0005	0.005	0.00001	0.0005	0.00025	0.00005	0.0001	0.000001	0.0005	0.0003	0.003	0.003	0.81
Maximum	8.47	8.3	0.00005	0.0559	0.00241	0.000079	0.0007	0.006	0.086	0.00001	0.022	0.0016	0.000884	0.00126	0.000025	0.0178	0.001	0.5	0.025	1.38
Geometric Mean	7.79	8.1	0.000022	0.0145	0.00064	0.000017	0.0003	0.00148	0.009	0.00001	0.00357	0.00045	0.000168	0.00025	0.000012	0.0046	0.0004	0.023	0.007	0.96
Count <DL	0	0	15	1	2	5	6	3	7	7	6	6	5	7	13	1	3	11	10	0
Median	7.79	8.11	0.00005	0.018	0.0008	0.00002	0.0003	0.0017	0.006	0.00001	0.011	0.00043	0.0002	0.0002	0.000025	0.0042	0.0003	0.025	0.006	0.96
MW96-B																				
Average	7.89	8.04	0.0001	0.008	0.0001	0.00006	0.0007	0.001	0.01	0.00002	0.001	0.0003	0.0001	0.0008	0.00003	0.005		0.02	0.021	1.03
Count	7	5	8	8	8	8	8	8	8	4	8	8	8	8	8	8		8	5	5
Minimum	7.2	7.81	0.0001	0.006	0.0001	0.00001	0.0003	0.001	0.01	0.00001	0.001	0.0003	0.0001	0.0005	0.00003	0.002		0.03	0.003	1
Maximum	8.3	8.15	0.0001	0.01	0.0002	0.00041	0.0013	0.001	0.03	0.00005	0.001	0.0003	0.0002	0.0009	0.00003	0.009		0.03	0.025	1.1
Geometric Mean	7.88	8.04	0	0.008	0.0001	0.00001	0.0006	0.001	0.01	0.00001	0	0.0002	0.0001	0.0008	0.00003	0.005		0.02	0.016	1.03
Count <DL	0	0	8	0	7	5	2	4	5	4	8	8	5	0	8	0		8	5	0
Median	8.03	8.1	0.0001	0.007	0.0001	0.00001	0.0005	0.001	0.01	0.00001	0.001	0.0003	0.0001	0.0008	0.00003	0.005		0.03	0.025	1
MW96-C																				
Average	7.57	8	0.000028	0.0334	0.00028	0.000081	0.0018	0.00171	0.036	0.00001	0.02443	0.00086	0.000274	0.00249	0.000016	0.0082	0.001	0.085	0.02	1.23
Count	14	12	14	14	14	14	14	14	14	11	14	14	14	14	14	14	3	15	13	13
Minimum	6.98	7.8	0.000003	0.0025	0.0001	0.000005	0.0003	0.0005	0.005	0.00001	0.00681	0.00025	0.000027	0.00053	0.000001	0.002	0.0003	0.003	0.003	0.75
Maximum	8.1	8.22	0.00005	0.328	0.00078	0.000873	0.0034	0.006	0.407	0.00005	0.0375	0.00419	0.00238	0.0039	0.000025	0.038	0.0025	0.61	0.087	3.6
Geometric Mean	7.57	8	0.000016	0.0108	0.00023	0.000019	0.0015	0.00111	0.01	0.00001	0.02315	0.00051	0.000106	0.00224	0.00001	0.0058	0.0005	0.033	0.011	1.06
Count <DL	0	0	12	3	4	5	0	7	6	11	0	9	6	0	13	0	3	12	9	0
Median	7.61	7.94	0.000033	0.011	0.0003	0.000019	0.0016	0.00057	0.007	0.00001	0.0249	0.00043	0.000075	0.00245	0.000025	0.0052	0.0003	0.025	0.007	0.9
MW96-E																				
Average	7.54	7.96	0.000039	0.0239	0.00032	0.000074	0.0013	0.00331	0.031	0.00001	0.0197	0.00061	0.001017	0.0007	0.000015	0.0151	0.0003	0.037	0.01	0.21
Count	15	14	16	16	16	16	16	16	16	11	16	16	16	16	16	16	5	16	14	14
Minimum	7.09	7.63	0.000003	0.0025	0.00009	0.00001	0.0001	0.00119	0.004	0.00001	0.0105	0.00025	0.00005	0.0001	0.000001	0.0073	0.0003	0.003	0.003	0.07
Maximum	8.2	8.36	0.000156	0.247	0.0007	0.000272	0.0035	0.00646	0.394	0.00005	0.032	0.00168	0.00892	0.0027	0.000025	0.024	0.0003	0.2	0.025	0.48
Geometric Mean	7.54	7.96	0.000018	0.0079	0.00024	0.000056	0.0008	0.00302	0.008	0.00001	0.0184	0.00049	0.000376	0.00052	0.000007	0.0143	0.0003	0.024	0.006	0.18
Count <DL	0	0	15	4	4	0	1	0	8	11	0	7	1	3	15	0	5	10	14	0
Median	7.5	7.97	0.00005	0.0072	0.0003	0.000054	0.0011	0.003	0.005	0.00001	0.018	0.00049	0.00025	0.00057	0.000025	0.015	0.0003	0.025	0.004	0.2
DW08-01																				
Average	7.64	7.89	0.000004	0.0048	0.00015	0.00001	0.0009	0.00052	9.68	0.00001	0.00072	0.00124	0.000088	0.00018	0.000009	0.0021	0.003	0.077	0.005	0.01
Count	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	1	4	4	4
Minimum	7.27	7.15	0.000003	0.0011	0.00001	0.00001	0.0001	0.00007	1.23	0.00001	0.0001	0.00021	0.00005	0.00002	0.000001	0.0014	0.003	0.009	0.005	0.01
Maximum	8.23	8.25	0.000005	0.007	0.0004	0.00002	0.002	0.001	31.2	0.00001	0.00117	0.004	0.0002	0.0003	0.000025	0.003	0.003	0.12	0.007	0.01
Geometric Mean	7.63	7.88	0.000004	0.0039	0.00008	0.00001	0.0005	0.00036	4.25	0.00001	0.0005	0.00057	0.000071	0.00012	0.000005	0.002	0.003	0.054	0.005	0.01
Count <DL	0	0	4	1	3	2	1	2	0	4	1	2	2	4	4	0	1	0	3	3
Median	7.53	8.09	0.000005	0.0055	0.0001	0.00001	0.0008	0.0005	3.15	0.00001	0.00081	0.00038	0.000051	0.0002	0.000005	0.002	0.003	0.09	0.005	0.01
DW08-02																				
Average	7.64	7.21	0.000004	0.0071	0.00009	0.000005	0.0004	0.0005	5.16	0.00001	0.00152	0.00033	0.000073	0.00011	0.000006	0.0026	0.0005	0.097	0.005	0.01
Count	7	7	7	7	7	7	7	7	7	5	7	7	7	7	7	7	3	7	7	7
Minimum	6.38	6.2	0.000003	0.0013	0.00001	0.000003	0.0001	0.00006	0.05	0.00001	0.0003	0.00012	0.000017	0.00002	0.000001	0.0005	0.0003	0.023	0.003	0.01
Maximum	8.3	7.85	0.000005	0.025	0.0002	0.00001	0.0013	0.002	20.7	0.00001	0.00322	0.00062	0.0002	0.0003	0.000025	0.006	0.0009	0.22	0.015	0.03
Geometric Mean	7.61	7.18	0.000003	0.0046	0.00005	0.000004	0.0002	0.00026	1.91	0.00001	0.00088	0.00028	0.000053	0.00005	0.000003	0.0021	0.0004	0.08	0.004	0.01
Count <DL	0	0	7	1	2	5	4	2	0	5	0	3	2	7	7	0	2	0	6	5
Median	7.93	7.31	0.000003	0.005	0.00004	0.000003	0.0003	0.0002	3.6	0.00001	0.00045	0.00025	0.00005	0.00002	0.000001	0.0018	0.0003	0.08	0.005	0.01

Table 22 provides monitoring well locations (shown in Figure 3), well depths, depth to water measured between 2006 and 2011, and includes whether or not the site was visited or if water was present.

Groundwater quality summary graphs for dissolved aluminum, cadmium, copper, iron and selenium are presented in Figures 13 through 17.

Table 22 Groundwater Monitoring Well Sample Locations, Events and Depth (m) to Water Measurements 2006 - 2011

Monitoring Well #	MW96-A	MW96-B	MW96-C	MW96-D	MW96-E	MW96-F	MW96-H	MW96-I	MW96-J	MW96-K**	HYD-WELL-1 or MW-01-07	DW08-01	DW08-02
Easting	411395	411535	411946	412165	411385	411745	412453	412931	411950	412086	412219	411455	411404
Northing	6913718	6913433	6913059	6912838	6913271	6914708	6914635	6914367	6913899	6913459	6912688	6912473	6912458
Drilled Depth (m)	91.4	91.4	50.0	41.1	91.4	62.5	55.2	54.9	90.5	93.0			
Measured Depth (m)	89.67	91.97	50.0	10.38	93.25	6.59*	7.72*	2.81*	91.52	93.9			
Date													
Jun-06	62.09	42.22	40.38	No water	Not located	4.96	N/A	N/A	N/A	N/A	N/A	N/A	N/A
Jul-06	62.14	42.53	No water	No water	Not located	No water	N/A	N/A	No water	N/A	N/A	N/A	N/A
Aug-06	62.43	42.34	42.06	No water	54.5	4.96	2.9	2.1	No water	N/A	N/A	N/A	N/A
Sep-06	64.53	42.14	40.45	No water	44.5	6.04	6.23	No water	91.58	N/A	N/A	N/A	N/A
Oct-06	62.65	43.01	42.64	No water	42.38	3.95*	No water	1.84	91.08	N/A	N/A	N/A	N/A
Apr-07	frozen	frozen	frozen	frozen	frozen	Little water	Not visited	Not visited	frozen	No water	N/A	N/A	N/A
May-07	no water	42.93	41.08		50.62	6.35	frozen	No water	91.57	No water	N/A	N/A	N/A
Jun-07	55.6	43.55	41.3	No water	52.8	6.44	1.84	No water	No water	No water	N/A	N/A	N/A
Jul-07	50.5	42.7	no water	No water	53.12	6.29	No water	2.11	No water	No water	N/A	N/A	N/A
Aug-07	48.2	43.52	41.4	No water	53.34	Little water	No water	No water	No water	No water	N/A	N/A	N/A
Sep-07	48.3	43.5	41.4	No water	53.55	Ice plug	No water	Ice plug	No water	No water	N/A	N/A	N/A
Oct-07	59.2	N/A***	19.5	No water	61.6	Ice plug	Ice plug	Ice plug	No water	No water	new well installed	N/A	N/A
May-08	7.18*	-	41.17	Not visited	54.56	-	-	-	Not visited	Not visited	Not visited	N/A	N/A
Jun-08	50.65	-	40.96	No water	55.28*	-	-	-	91.48	91.45	Not visited	N/A	N/A
Jul-08	47.87	-	40.98	Not visited	54.8	-	Ice plug	-	Not visited	Not visited	plugged	N/A	N/A
Aug-08	61.35	-	Not recorded	No water	54.8	-	-	-	No water	No water	plugged	5.18	6.68
Sep-08	64.35	-	41.14	Not visited	55.43	-	-	-	91.2	88.55	plugged	5.23	6.73
Oct-08	58	-	51.08	No water	82	-	-	-	75	83	frozen	6.5	6.8
Nov-08	No water	-	waterra frozen	Not visited	Not visited	-	-	-	Not visited	Not visited	Not visited	Not visited	Not visited
May-09	44.47*	-	39.48	No water	51.3	-	-	-	ice build-up	ice build-up	frozen	Not visited	Not visited
Jul-09	ice build-up	-	38.57	No water	No water	-	-	-	ice build-up	ice build-up	frozen	Not visited	Not visited
Oct-09	55.87	-	47.53	No water	46	-	-	-	90.61	91.6	Not visited	Not visited	Not visited
May-10	56.9	-	38.27	No water	43.65	-	-	-	91.62	91.84	Not visited	No water	4.97
Jun-10	57.8	-	10.83 to ice	No water	43.835	-	-	-	91.605	91.88	frozen	No water	5.55
Jun-11	Not Measured	-	Not Measured	No water	Not Measured	-	-	-	Not visited	Not visited	frozen	Ice blockage	Not Measured
Sep-11	Not visted	-	10.6	Not visited	Not visited	-	-	-	Not visited	Not visited	Not visited	2.93	Dry

* Depth to ice

**Assumed buried by trenching until located in April 2007

*** Bailer jammed at ~29 m Sept07

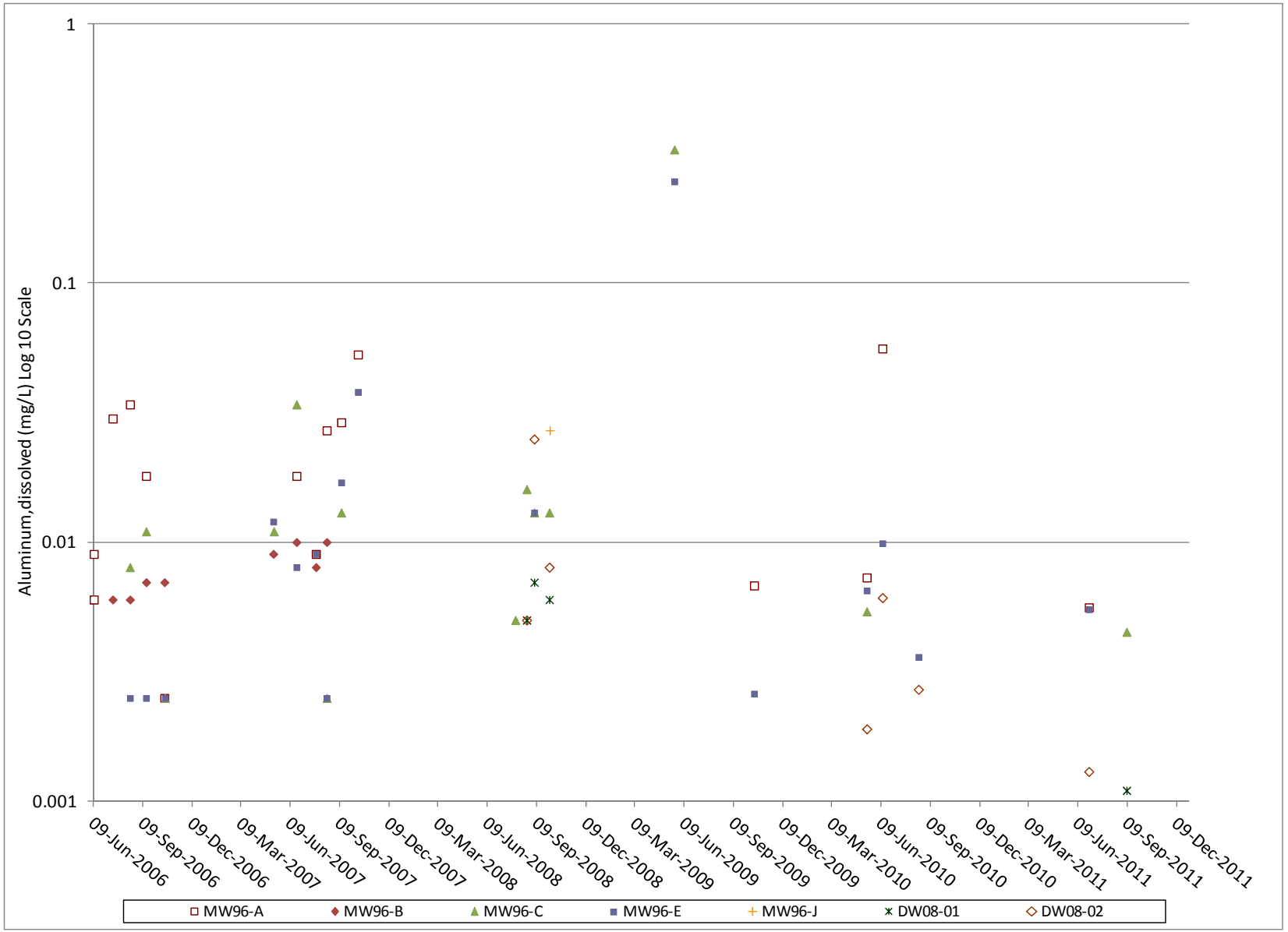


Figure 13 Carmacks Copper Monitoring Wells, Dissolved Aluminum

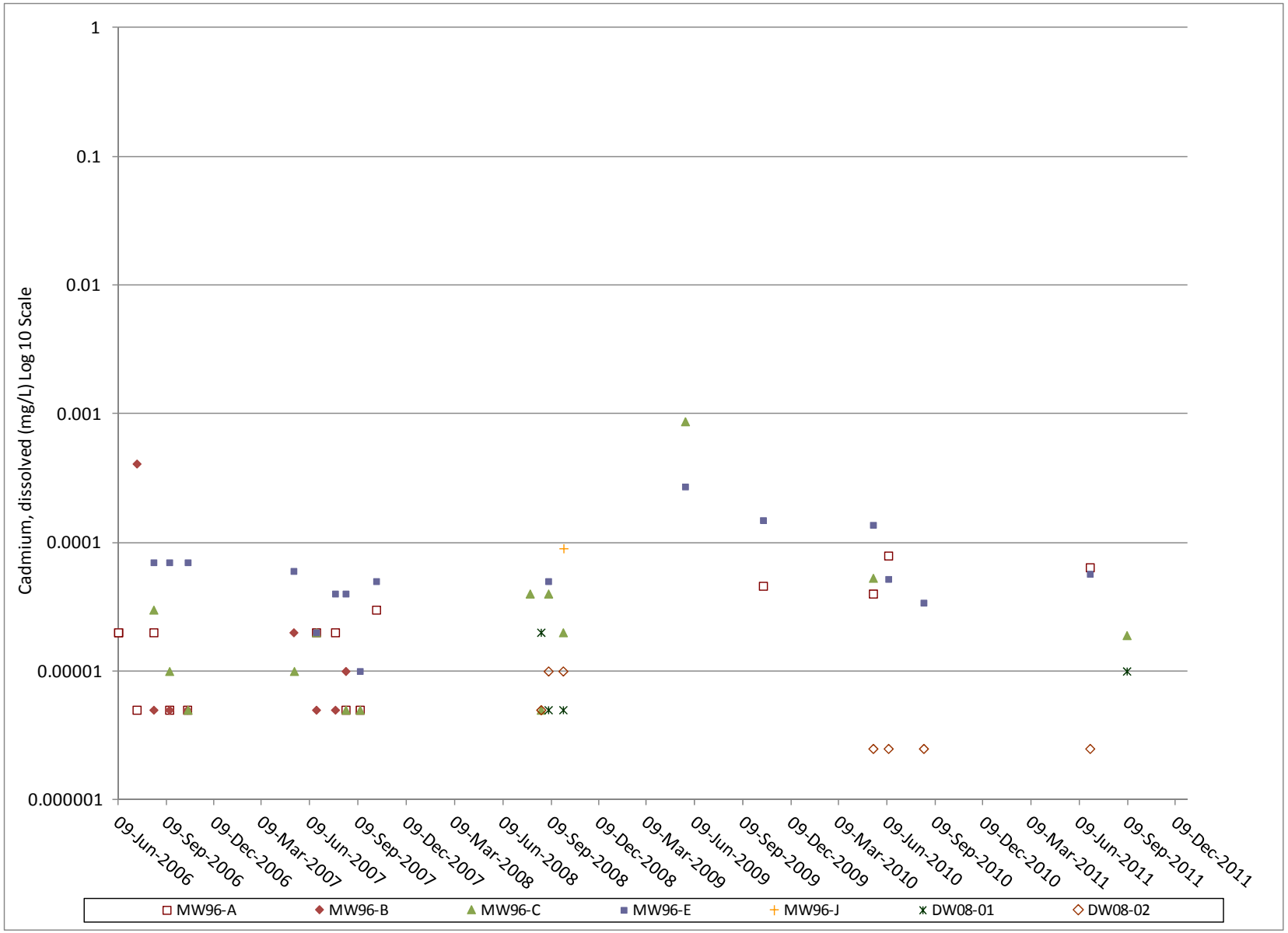


Figure 14 Carmacks Copper Monitoring Wells, Dissolved Cadmium

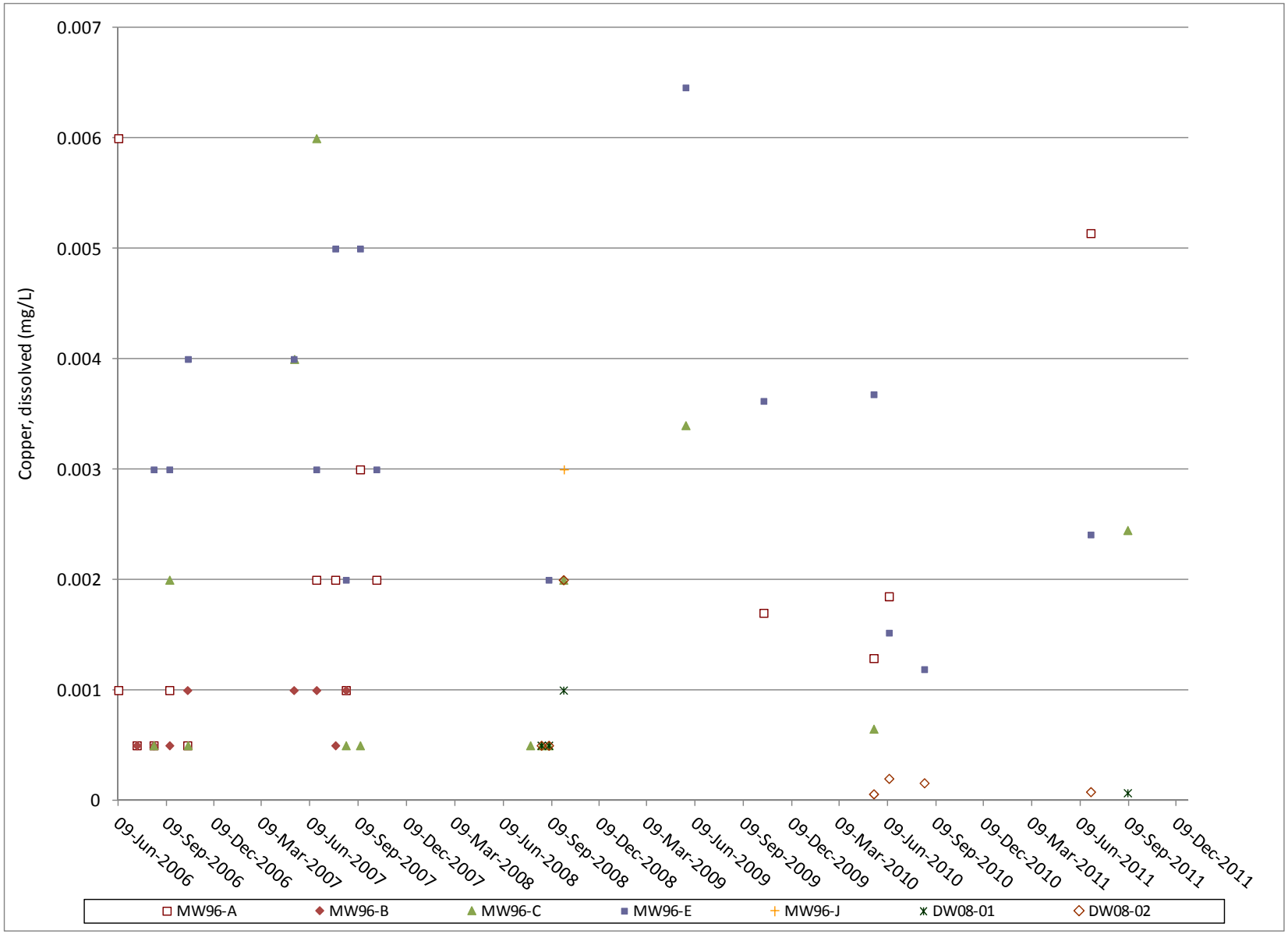


Figure 15 Carmacks Copper Monitoring Wells, Dissolved Copper

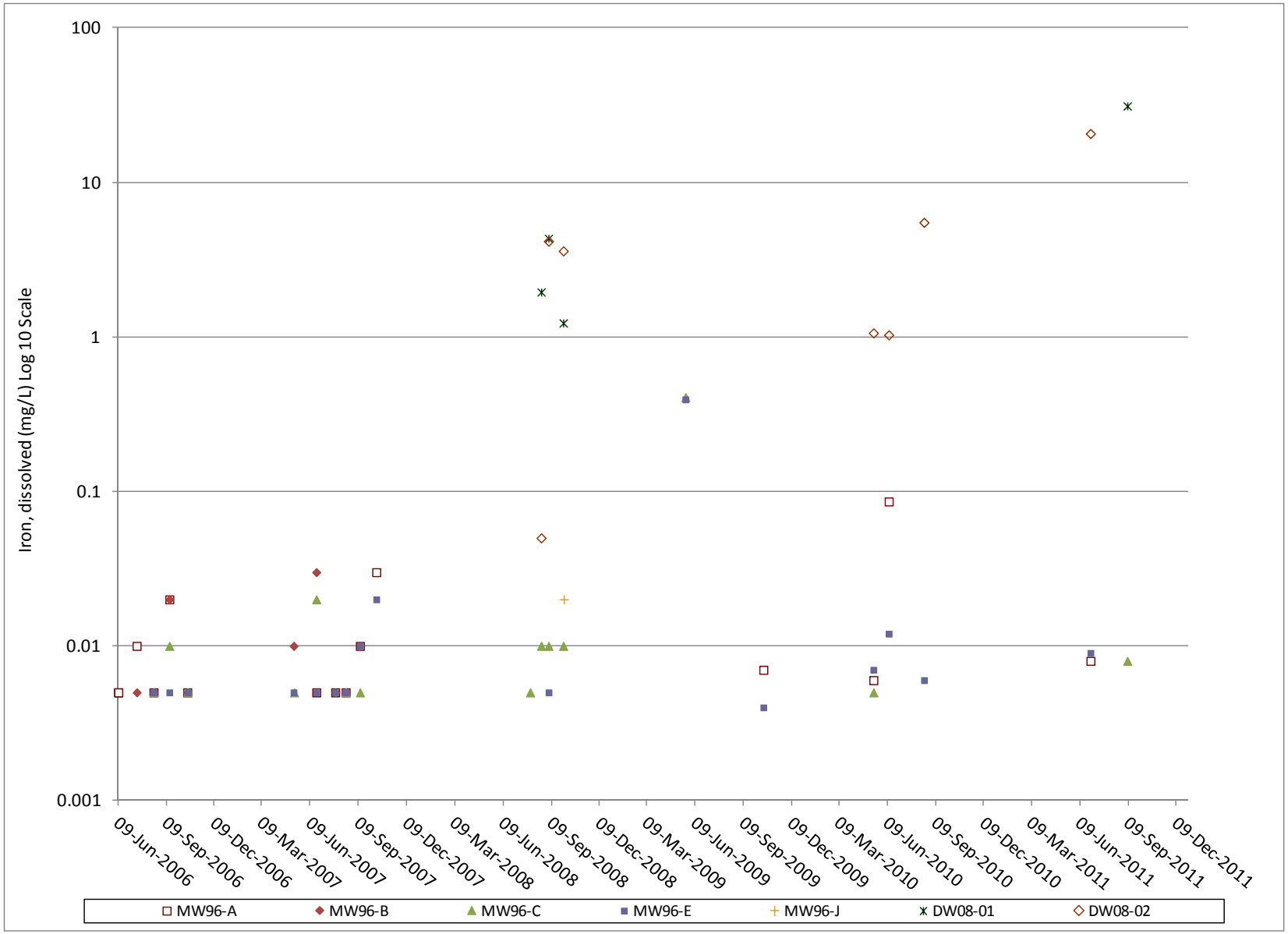


Figure 16 Carmacks Copper Monitoring Wells, Dissolved Iron

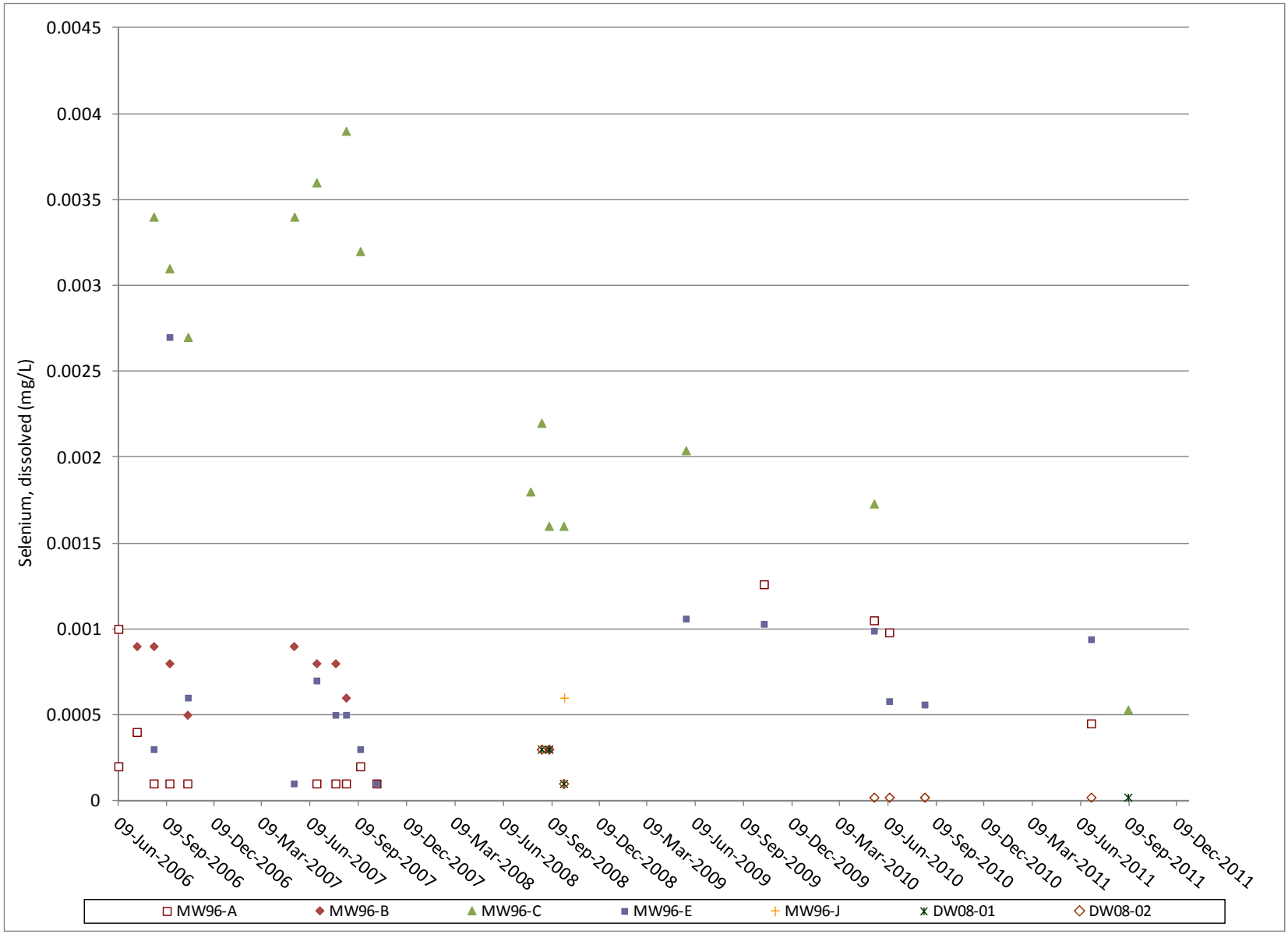


Figure 17 Carmacks Copper Monitoring Wells, Dissolved Selenium

6.0 CLOSING STATEMENT

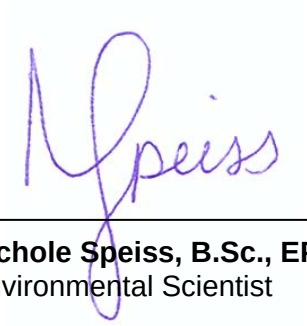
Access Consulting Group of Whitehorse, Yukon, has prepared this *2011 Environmental Monitoring Program Update and Data Summary* for the Carmacks Copper Project for the exclusive use of Copper North Mining Corp. and is based on data and information collected during site investigations. Access Consulting Group has followed standard professional procedures in conducting the investigations and in preparing the contents of this report. The material in this report reflects Access Consulting Group's best judgment in light of the information available at the time of the preparation of this report. Any use that a third party makes of this report, or any reliance on decisions to be made based on it, is the responsibility of the third parties. Access Consulting Group accepts no responsibility for damages, if any, suffered by any third party as a result of decisions made or actions based on this report. Access Consulting Group believes that the contents of this report are substantively correct.

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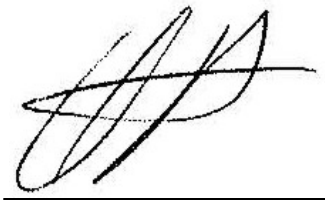
If you have any questions or require further details, please contact the undersigned.

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