



Western Copper  
Corporation

**Project Proposal**  
**Carmacks Copper Project**  
**Yukon Territory**

---

**Appendix E**

**Beattie Consulting Ltd. Report on Leaching and  
Decommissioning of Samples from  
Carmacks Oxide Copper Project (2001)**

---

**LEACHING AND  
DECOMMISSIONING OF  
SAMPLES FROM  
CARMACKS OXIDE  
COPPER PROJECT**

**WESTERN COPPER HOLDINGS  
FEBRUARY 2001**

**BEATTIE CONSULTING LTD  
VANCOUVER, BC CANADA**

# BEATTIE CONSULTING LTD

2955 W. 38<sup>th</sup> Avenue  
VANCOUVER, B.C.  
CANADA V6N 2X2

TEL: (604) 263 0695  
FAX: (604) 263 0695  
Email: mbeattie@attglobal.net

---

February 28, 2001

**Western Copper Holdings Limited**  
1650 – 1185 West Georgia Street  
Vancouver, BC  
V6E 4E6

**Attention: Mr. Dale Corman**

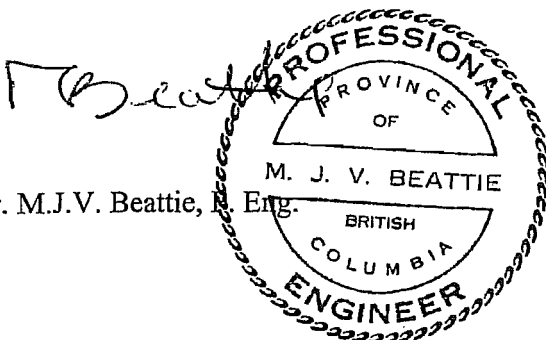
Dear Dale,

Enclosed please find our report entitled *Leaching and Decommissioning of Samples from Carmacks Oxide Copper Project*. The reports compile the testwork done for the project since early 1998 and draws conclusions based on this work within the context of all previously completed work.

The work completed to date provides a sound basis for predicting copper recovery requirements and neutralization requirements. We would be pleased to discuss any aspects of this project with you at any time.

Yours truly,  
**BEATTIE CONSULTING LTD**

Dr. M.J.V. Beattie, Eng.



**CARMACKS COPPER PROJECT  
LEACHING AND DECOMMISSIONING**

**TABLE OF CONTENTS**

	<u>Page</u>
<b>1.0 SUMMARY .....</b>	<b>1</b>
<b>2.0 INTRODUCTION .....</b>	<b>3</b>
<b>3.0 DISCUSSION .....</b>	<b>4</b>
<b>3.1 Sample Description.....</b>	<b>4</b>
<b>3.1.1 Bulk ROM Sample .....</b>	<b>4</b>
<b>3.1.2 Waste Sample.....</b>	<b>7</b>
<b>3.2 Leaching Testwork.....</b>	<b>9</b>
<b>3.2.1 Crushed Sample Leaching .....</b>	<b>9</b>
<b>3.2.2 Drain Down Testwork .....</b>	<b>22</b>
<b>3.2.3 Run of Mine Leaching.....</b>	<b>22</b>
<b>3.3 Decommissioning Testwork .....</b>	<b>31</b>
<b>3.3.1 Batch Neutralization of Leached Solids.....</b>	<b>31</b>
<b>3.3.2 Column Neutralization of Leached Solids .....</b>	<b>32</b>
<b>3.3.3 Decommissioning of Column C Leach Residue.....</b>	<b>39</b>
<b>3.3.4 Rinsing of Run of Mine Column.....</b>	<b>44</b>

**Appendix 1 Sample Collection Report**

**Appendix 2 Leach Column Test Details**

**Appendix 3 Waste Neutralization Batch Test Report**

**Appendix 4 Waste Column Neutralization Test Details**

**Appendix 5 Column C Neutralization Details**

## 1.0 SUMMARY

The most recently completed test program on samples from the Carmacks oxide copper deposit has confirmed the following:

- The copper is readily leachable by means of crushing to minus  $\frac{3}{4}$  inch followed by heap leaching. A copper extraction of just over 85% was achieved over a leach period of 187 days. Over the design leach time of 120 days the column achieved a copper extraction of 83%, corresponding to a commercial extraction of about 78%. For a multiple lift leach configuration it is recommended that the primary leach cycle be ended when about 70% copper extraction has been achieved in order to minimize acid consumption. For the current sample the primary leach cycle would be about 60 days.
- The predicted acid consumption for Carmacks is 22 kg/tonne. It is recommended that 20 kg/tonne be added to the cure stage with the balance being added to solution for control of feed pH.
- The recirculation of leach solutions through the column results in a gradual build-up of dissolved Al, Fe, Mg and Si. The presence of these constituents in solution indicates attack on the gangue minerals.
- Leaching of the Carmacks material at a Run of Mine size results in a significant loss in copper extraction of about 20% unless very long leach times, in excess of 1000 days are utilized. The test was conducted with a top size of 10 inches and some additional loss in copper extraction can be expected with the presence of even coarser material. The present test operated for 360 days and has not demonstrated the ultimate copper extraction achievable at a Run of Mine size.
- Even after extended leach times, the bulk sample was found to consume any free acid present in feed solutions. Following the completion of active leaching, a partial neutralization is achieved by continued circulation of the solutions through the heap. On this basis the leached material does not pose a threat for continued acid rock drainage.
- Previous column testwork indicated that the leached solids could be readily neutralized to result in an effluent of pH 4, which was buffered at this pH. Additional tests have indicated that when leached solids are freely mixed with solution containing an excess of base, the solution readily achieves a neutral pH and this pH is stable. Subsequent column tests have shown that sodium carbonate is effective at altering the leached solids so that effluents have a neutral pH. This condition appears to be stable over extended time periods.

- One effective rinsing procedure for the spent heaps appears to be to recirculate the solutions until the free acid is consumed and the copper concentration in solution becomes uneconomic to recover. At this point the leach solution should be neutralized with lime to precipitate sulphate and other deleterious constituents before being discharged. The heap should then be rinsed with groundwater (pH 7) in a series of pulses alternating with rest periods to allow dissolved sulphate, copper etc to diffuse from the rock particles. An effluent pH near pH 4 can be expected with this procedure. To achieve higher pH values, the addition of a base is required and sodium carbonate added as a pH 10.5 solution appears to be the most effective addition. The use of lime as a neutralizing agent will likely only be effective if the lime can be intimately mixed with the solids.

Alternatively, if an on-off leaching pad scenario is adopted, lime could be mixed with the spent ore as it is removed from the pad in order to achieve permanent neutralization.

- It has been observed that the leached solids tend to become less permeable due to both decrepitation and precipitation of secondary mineral phases. This loss of permeability will minimize the quantity of effluent to be treated with time. Consideration should be given to installing collection piping at least every few lifts to optimize the flow of solution from the heap during leaching and to minimize the inventory of dissolved copper in the heap.
- The alternative of using an on-off pad leach should be considered as an alternative to the multiple lift scenario contemplated to date. As well as facilitating the neutralization of the spent ore, this approach would provide for the possibility of gold recovery in a second process step.

## 2.0 INTRODUCTION

The leaching of oxide copper from Carmacks samples has been studied over a number of years and the results have been reported previously:

*Metallurgy of the Williams Creek Oxide Copper Deposit*  
Beattie Consulting Ltd  
May 1994

*Pilot Scale Column Testing of the Williams Creek Oxide Deposit*  
Beattie Consulting Ltd  
February 1996

The previous testwork demonstrated that copper can be leached successfully from Carmacks materials by crushing to minus ¾ inch followed by heap leaching with a weak sulphuric acid solution. Such a process is practiced widely for the recovery of copper from oxide copper ores and the parameters required for the Carmacks deposit are consistent with general practice in the industry.

The leaching testwork summarized in the present report was initiated to evaluate the potential for leaching of the Carmacks deposit at a Run-of Mine size. A parallel test on a portion of the same sample crushed to minus ¾ inch was conducted to enable the results for this sample to be compared to those obtained previously with drill core composites.

Previous decommissioning testwork had consisted of rinsing of test columns with water followed by solutions containing base additions to various pH levels. Those tests had demonstrated that the pH of the effluent from the columns could be raised quite readily to a value of about pH 4 but appeared to be buffered at this value and was resistant to further increases. The present program therefore included various tests to improve the understanding of what was controlling the effluent pH and to investigate alternatives for effective decommissioning of the leach pads.

All testwork included in this report was conducted at the laboratories of Process Research Associates Ltd of Vancouver.

### 3.0 DISCUSSION

#### 3.1 Sample Description

Two sources of material were used for the work included in this report. The primary sample consisted of a bulk sample taken from a series of surface trenches under the supervision of representatives of Kilborn Engineering Pacific Ltd. The Kilborn report describing the bulk sample collection is included as Appendix 1 to this report. The second sample consisted of trench material which had previously been crushed to minus  $\frac{3}{4}$  inch and used for a crib test at the site. This material had been partially leached of copper.

##### 3.1.1 Bulk ROM Sample

The bulk sample was obtained by taking samples from four separate trenches and coning and quartering these samples to provide a sub sample of each one. These sub-samples were combined into an overall composite which was then loaded into 18 bulk bags having an approximate moist weight of 1 tonne each for shipment to Vancouver.

At the laboratory, the entire sample was screened into a series of size fractions as summarized in Table 3.1.1. Sufficient material was available of all size fractions below 6 inches to take portions for assay. Fractions between 6 inches and 1 inch were sampled by coning and quartering while all finer fractions were sampled by means of riffing. The results of these assays are included in Table 3.1.1 and they indicate a general enrichment of copper into the finer fractions. The distribution of both total and soluble copper was obtained by assuming that the fractions coarser than 6 inches had the same copper content as the -6+4 inch fraction.

It was recognized during the sample collection that the removal of a bulk sample from a surface trench would result in a somewhat different size distribution than would be expected for an industrial-scale blast. Furthermore, even though the trenches were as deep as 11 meters in places, the near surface material could be somewhat weathered and therefore result in the generation of excessive fines during blasting and handling. The size distribution of the material placed in the ROM column was therefore adjusted to be closer to an expected size distribution so that the extraction would not be overstated. The size distribution of the bulk sample is summarized in Figure 3.1.1, together with the size distribution of the material placed in the column. Also included in this figure are the size distributions for several full-scale blast of oxide copper ores. It can be seen that the distribution of the material placed in the column has been adjusted to be similar to these other materials through the reduction of the fines content.



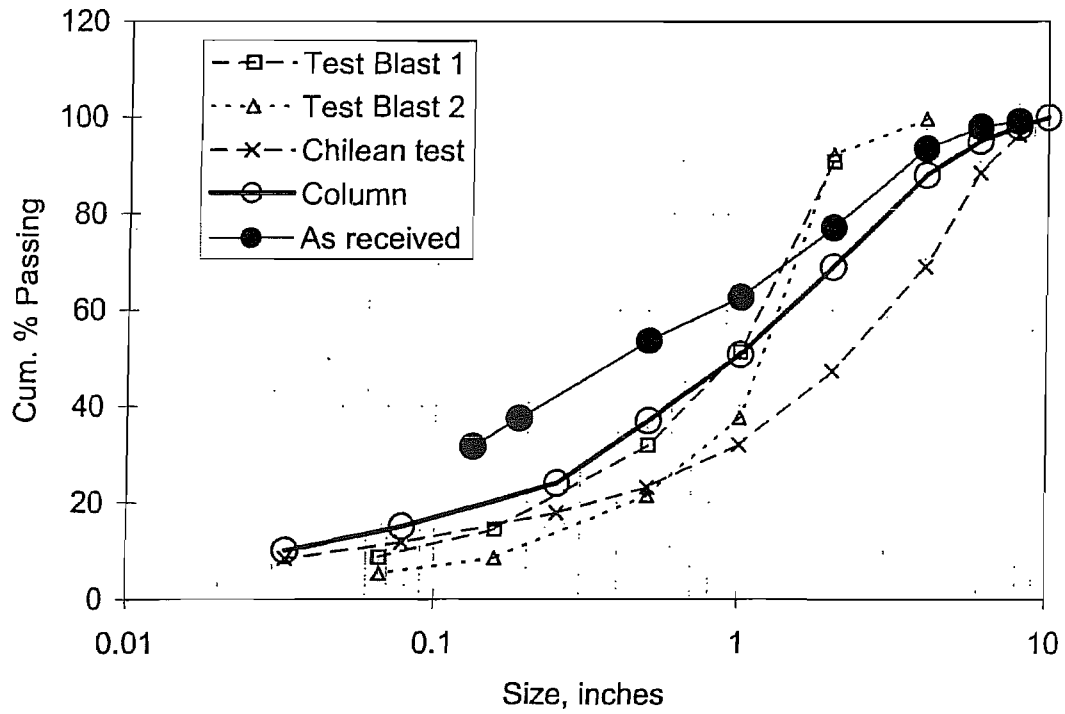


Figure 3.1.1 Size Distribution for Bulk Sample and Other ROM Samples.

**Table 3.1.1**  
**Assay by Size Fraction for ROM Sample**

Fraction	Weight %	Assay, %		Distribution, %	
		Cu <sub>T</sub>	Cu <sub>AS</sub>	Cu <sub>T</sub>	Cu <sub>AS</sub>
+12"	0.4	-	-	0.3	0.3
-12+8"	0.5	-	-	0.4	0.4
-8+6"	1.2	-	-	0.9	0.9
-6+4"	4.4	0.78	0.72	3.3	3.4
-4+2"	16.4	0.80	0.74	12.5	12.8
-2+1"	14.3	0.82	0.74	11.2	11.2
-1+1/2"	9.1	0.82	0.74	7.1	7.1
-1/2+ 4 mesh	16.2	0.94	0.84	14.6	14.4
-4+6	5.9	1.04	0.96	5.8	6.0
-6+10	3.5	0.82	0.76	2.7	2.8
-10+20	5.4	1.14	1.06	5.9	6.1
-20	22.7	1.62	1.50	35.2	36.2
Total		1.03	0.94		

A sub-sample of the bulk sample was taken and crushed to minus ¾ inch in order to conduct a column test on the sample at this crush size. The sample was taken by sampling each of the size fractions into which the bulk sample had been screened and then crushing the plus ¾ inch fractions to minus ¾ inch. The size analysis and assays for the crushed sample are summarized in Table 3.1.2.

**Table 3.1.2**  
**Assay by Size Fraction of Crushed Sample**

Fraction	Weight %	Assay, %		Distribution, %	
		Cu <sub>T</sub>	Cu <sub>AS</sub>	Cu <sub>T</sub>	Cu <sub>AS</sub>
-3/4 + ½	8.0	0.60	0.50	4.7	4.3
- ½ + 3/8	17.0	0.72	0.66	12.1	12.2
- 3/8 + ¼	18.0	0.76	0.72	13.5	14.1
- ¼ + 6m	15.9	0.86	0.76	13.5	13.1
-6m	41.1	1.38	1.26	56.1	56.3
Total		1.01	0.92		

Additional analyses for the bulk sample are summarized in Table 3.1.3. Process Research Associates conducted an acid-base accounting of the bulk sample by the Modified Sobek Method. The results of this analysis are summarized in Table 3.1.4 and show the material to be a net acid consumer.

**Table 3.1.3**  
**Analysis of Bulk Sample (Column C Head)**

	%
Cu <sub>T</sub>	1.02
Cu <sub>AS</sub>	0.90
Fe <sub>T</sub>	4.59
Fe <sub>AS</sub>	1.53
Al <sub>2</sub> O <sub>3</sub>	17.13
BaO	0.18
CaO	4.20
Fe <sub>2</sub> O <sub>3</sub>	8.03
K <sub>2</sub> O	2.55
MgO	2.78
MnO	0.12
Na <sub>2</sub> O	4.12
P <sub>2</sub> O <sub>5</sub>	0.43
SiO <sub>2</sub>	55.15
TiO <sub>2</sub>	0.86
LOI	4.00

**Table 3.1.4**  
**Acid Base Accounting of Bulk Sample**

Parameter	Value
Paste pH	8.1
24 Hour pH	1.82
S (T), %	0.12
S (SO <sub>4</sub> ), %	0.05
AP	2.2
NP	12.4
Net NP	10.2
NP/AP	5.7

### 3.1.2 Waste Sample

The sample of previously leached ore (designated by the laboratory as Sample L1 Waste) consisted of one bulk bag which was sampled at the laboratory by coning and quartering. This sample was analyzed to determine the extent of copper leaching and the overall make-up of the sample. The analysis summarized in Table 3.1.5 indicates a high total and soluble copper content. Assuming an initial copper content similar to that of the bulk sample, the copper appears to have been only about 50% leached in terms of copper extraction.

**Table 3.1.5**  
**Analysis of Sample L1 Waste**

	%
Cu <sub>T</sub>	0.58
Cu <sub>AS</sub>	0.42
Fe <sub>T</sub>	3.84
Fe <sub>AS</sub>	1.20
Al <sub>2</sub> O <sub>3</sub>	16.98
BaO	0.22
CaO	3.43
Fe <sub>2</sub> O <sub>3</sub>	6.21
K <sub>2</sub> O	2.43
MgO	2.24
MnO	0.07
Na <sub>2</sub> O	4.59
P <sub>2</sub> O <sub>5</sub>	0.30
SiO <sub>2</sub>	57.27
TiO <sub>2</sub>	0.68
LOI	5.64

## 3.2 Leaching Testwork

Additional column leach tests were conducted as part of the most recent program. One test (Column AB) was conducted on Run of Mine sized material to observe the leaching response of this material. A second test (Column C) was conducted on a portion of the same sample but crushed to the same size as the feed used for previous tests. The results of these column tests are discussed in this section. The details of the column tests are included in Appendix 2.

### 3.2.1 Crushed Sample Leaching

A column test (designated as Column C) was conducted on a portion of the composite sample crushed to minus  $\frac{3}{4}$  inch. This crush size had been selected previously as being appropriate for the Carmacks deposit. While determining the viability of conducting a ROM leach was a significant objective of the current program, testing of the sample crushed to minus  $\frac{3}{4}$  inch provided a basis of comparing the behaviour of the composite to previously tested materials and provided material for decommissioning studies. The following discussion compares the results for the current sample to the results obtained for previous samples.

Prior to being loaded in the column the material was mixed with an addition of 15 kg/tonne sulphuric acid and 19.9 kg/tonne water and tumbled to simulate the degree of agglomeration that the material would achieve through several stages of transfer on a grasshopper type stacking system. The material was placed in the column and allowed to cure for 5 days before leaching was started. Leaching was commenced with a 10 gpl sulphuric acid solution for the first 6 days of operation. After this time the feed was raffinate which had been blended to maintain a feed acid concentration of about 13 gpl acid. The balance of the test was a closed circuit test with the PLS (Pregnant Leach Solution) being processed through solvent extraction before being returned to the column feed. Further acid additions were made to the column feed from time to time to achieve specific objectives.

For the purpose of discussion, the results for column C are compared to those obtained previously for columns PC2 and FH. Column PC2 was conducted in 1992-1993 on sample material obtained from surface trenches while column FH was conducted on a drill core composite.

Figure 3.2.1 summarizes the copper extraction as a function of leach time for column C and Figure 3.2.2 shows the copper extraction as a function of time for samples C, FH and two tests on the PC sample. It can be seen in this figure that sample FH gave the same results as one of the PC tests while the other PC test gave a slower initial leach rate. Sample C on the other hand leached faster than any of the other test samples. The leaching rate can be influenced by several factors including particle size and rate of acid addition.

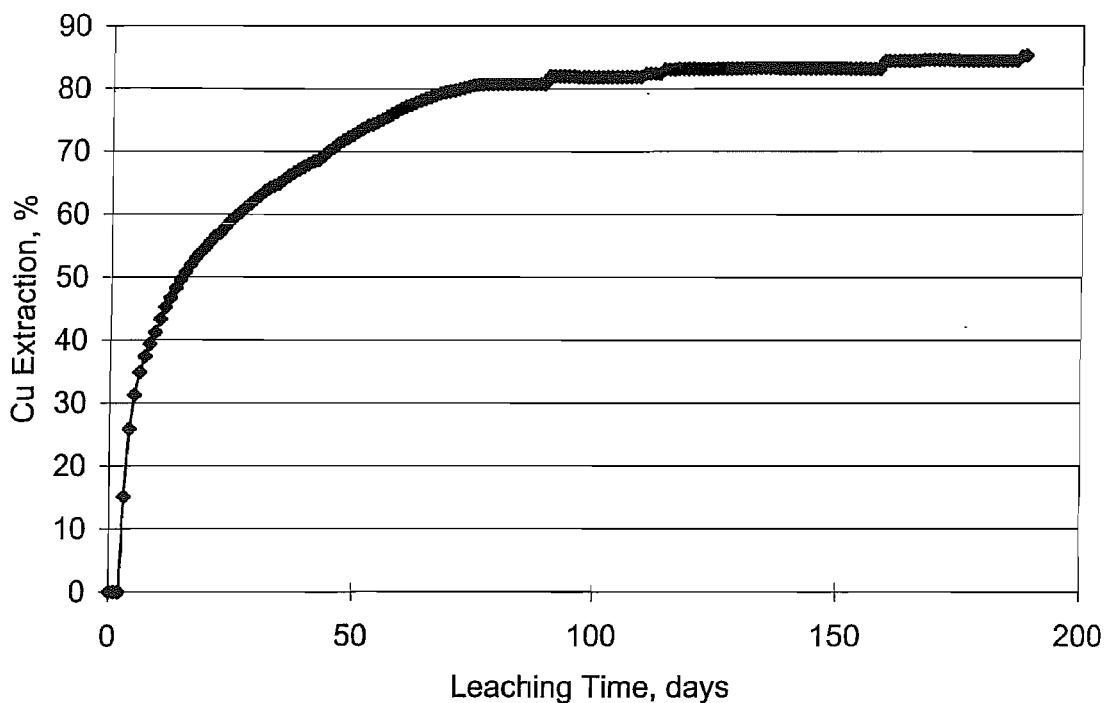


Figure 3.2.1 Extraction as a Function of Leach Time for Column C.

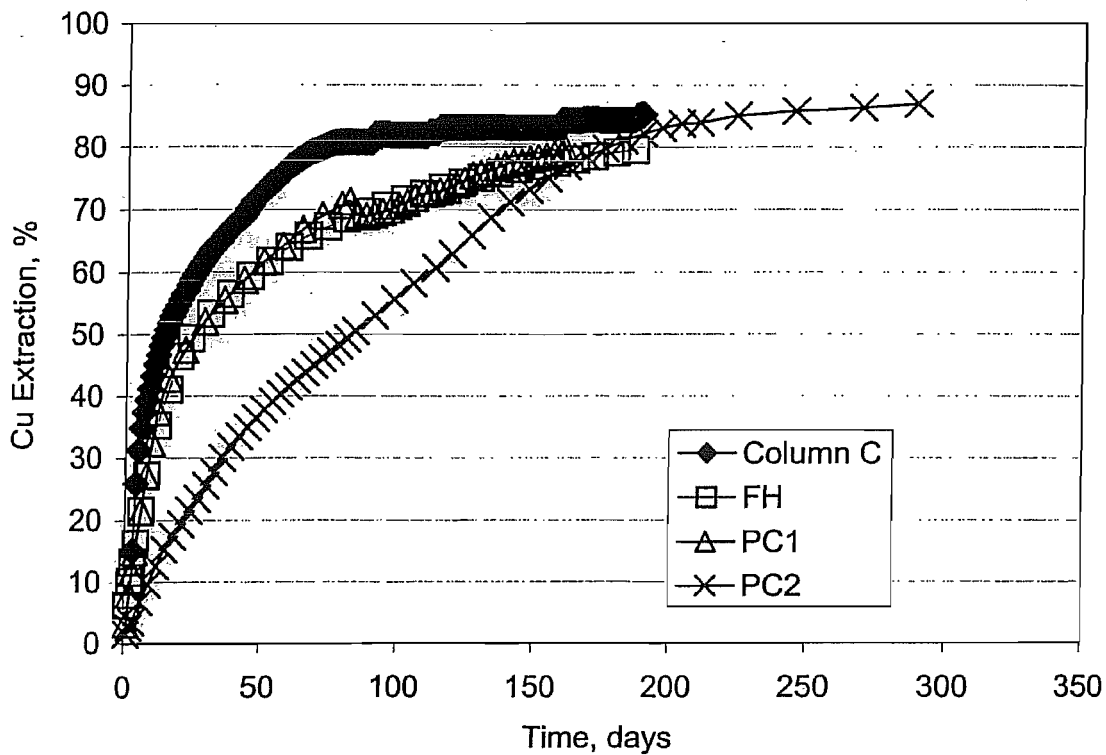


Figure 3.2.2 Copper Extraction Profiles for Various Carmacks Tests.

Figure 3.2.3 shows the size distribution of the three samples used for the various tests. The three samples have similar distributions at the coarse particle sizes but FH contains fewer fines than the other two samples and sample C is the finest of all. Sample FH likely has somewhat less fines than would be obtained when mining this material since much of the fines observed in mine products originate during blasting while the FH composite was produced by crushing drill core. However, it is expected that more fines will be experienced during the initial stages of mining, similar to the content of the PC and C samples and the presence of these fines would be expected to result in faster leach kinetics as well as increased acid consumption.

The slow initial rate experienced for column PC2 compared to PC1 can be explained in terms of the acid additions made to this column. Figure 3.2.4 shows the copper extraction as a function of acid addition for all the ¾ inch columns. It can be seen that column PC2 achieved comparable copper extraction (when adjusted for leach time) as the other columns but with much less acid. Column FH on the other hand maintained high acid concentrations, and therefore high acid consumption, throughout the test but obtained no benefit in terms of copper leaching for this extra acid. These results show that while it is very important to establish the correct conditions in terms of acid at the start of the leaching cycle, there is no benefit to maintaining aggressive conditions after the initial leaching period of 50 to 100 days. It does appear from Figure 3.2.2 combined with Figure 3.2.4 that sample C leaches somewhat more quickly than the other samples tested at similar acid additions. The slight increase in rate for sample C could be related to the fact that this is the finest sample tested, together with the fact that the sample has a lower head grade (1.0 vs 1.2% Cu).

The acid consumption expected for sample C is similar to that established for the other samples that were tested. While the current test (Column C) indicates a high acid consumption of 34.4 kg/t, consideration of Figure 3.2.4 shows this figure to be misleading. During the initial part of the test, the acid consumption for sample C was similar to that for PC2. However, while the consumption for column PC2 was held at about 22 kg/t, the consumption for column C continued to increase. The reason for the excessive acid consumption observed in column C is apparent from Figure 3.2.6. While most copper heap leach operations maintain a PLS pH in the range of 1.8 to 2.0, it has previously been established that for the Carmacks ore, the PLS pH should be at least 2.2. The initial PLS pH for Column C was well over pH 3 and acid was added over the period of 50 to 100 days to bring this value down to a more normal range. It appears that the target value of 2.2 is still too low for this material and based on a comparison of the pH profiles for the various tests as summarized in Figure 3.2.7, a more appropriate target appears to be pH 2.5 to 2.7. In order to achieve this target efficiently a cure acid addition of 20 kg/t, rather than the 15 kg per tonne used in Column C, is indicated.

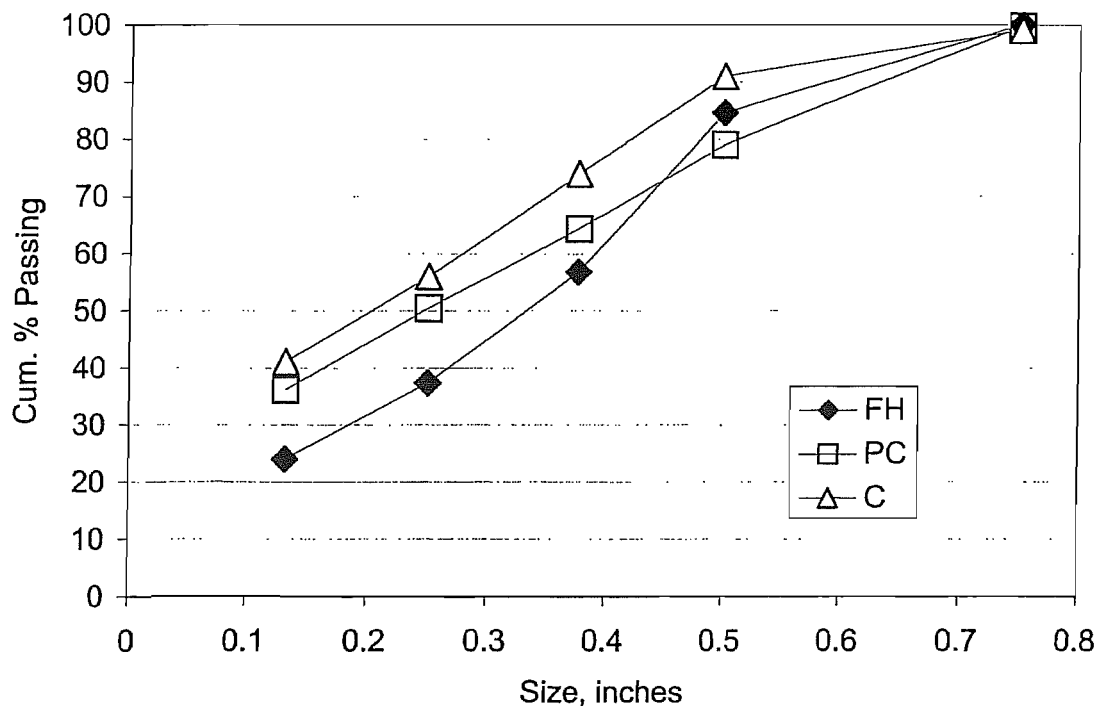


Figure 3.2.3 Size Distribution for Carmacks Test Samples.

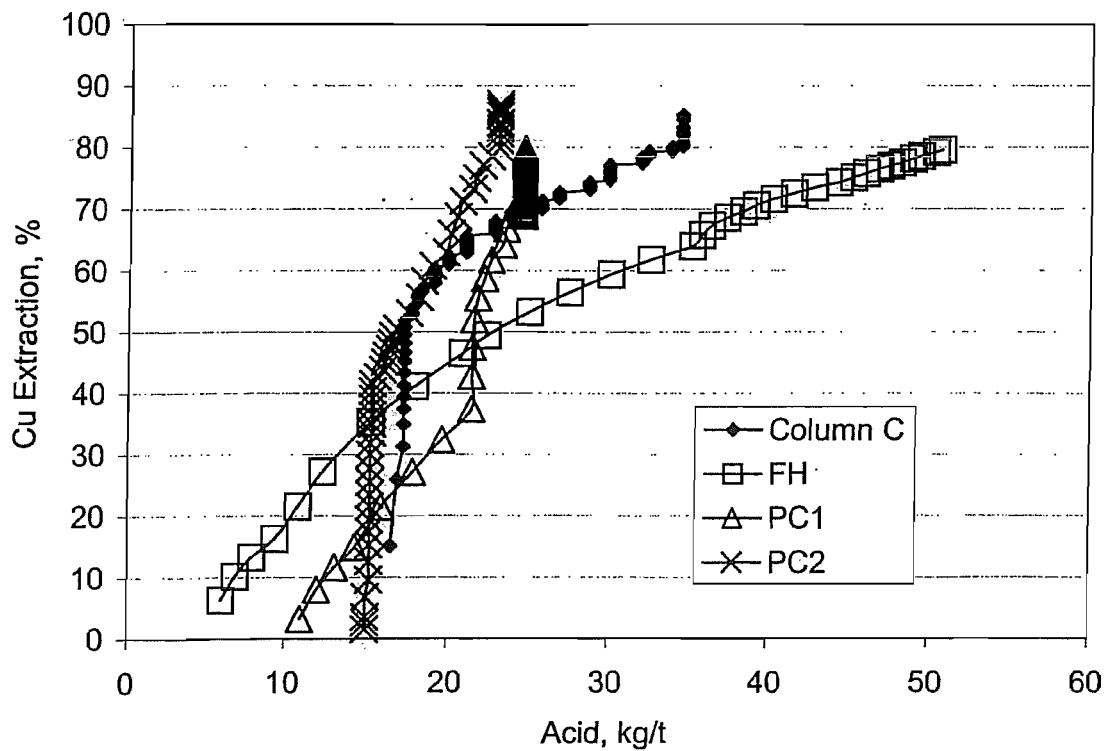


Figure 3.2.4 Copper Extraction as a Function of Acid Addition.



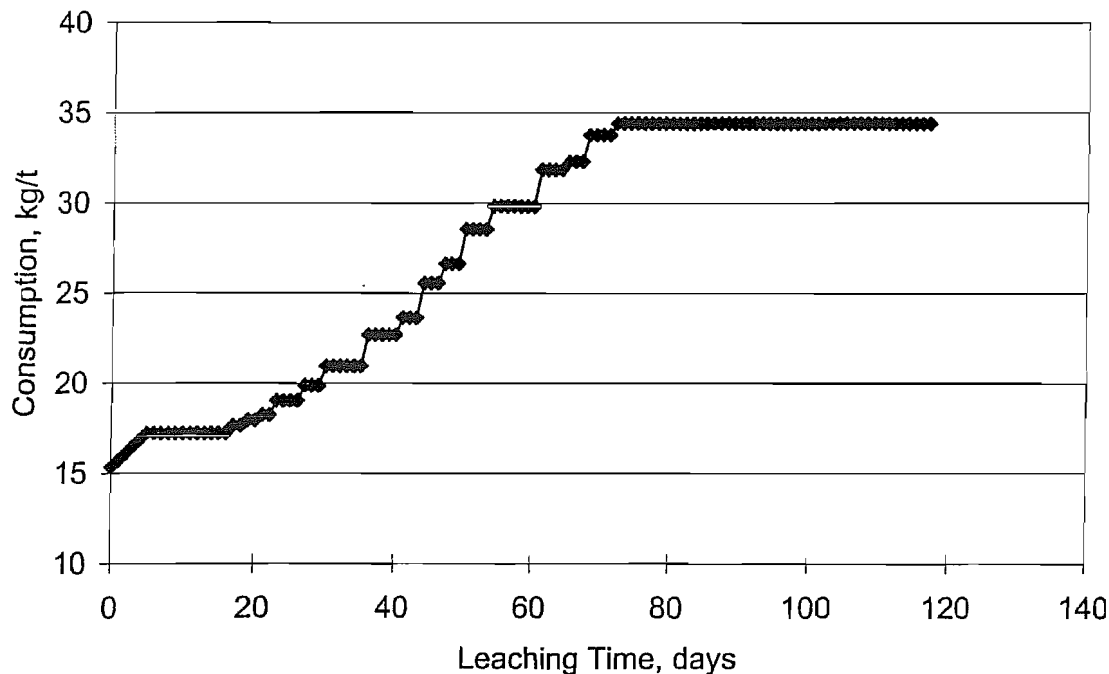


Figure 3.2.5 Acid Consumption for Column C.

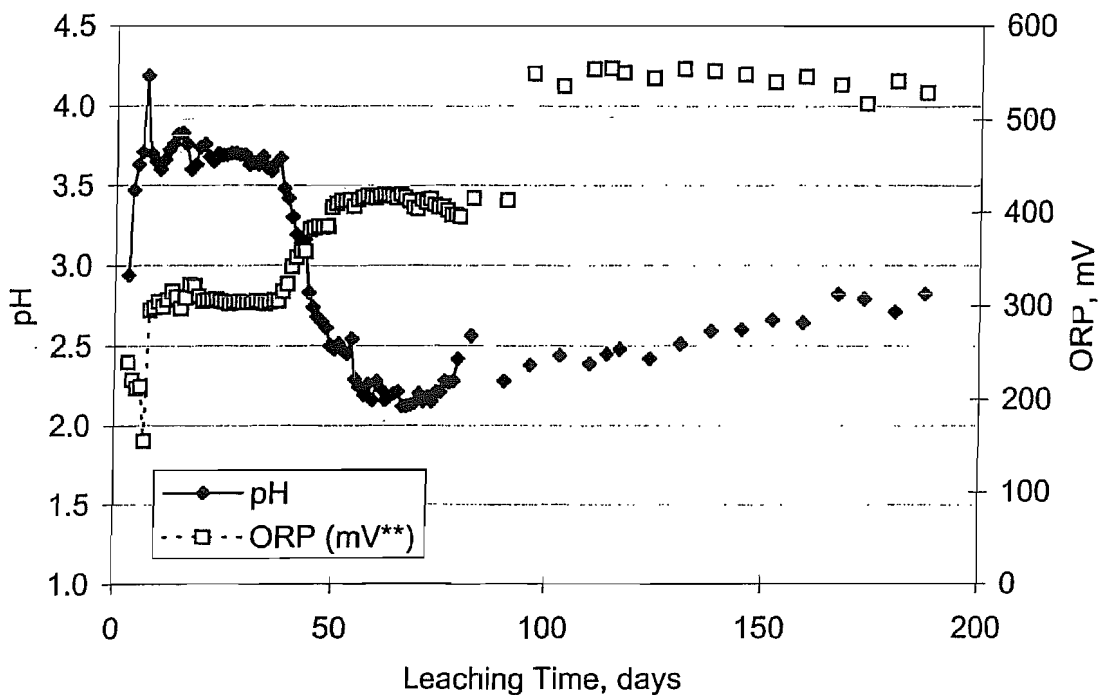


Figure 3.2.6 pH and ORP Profile for Column C.

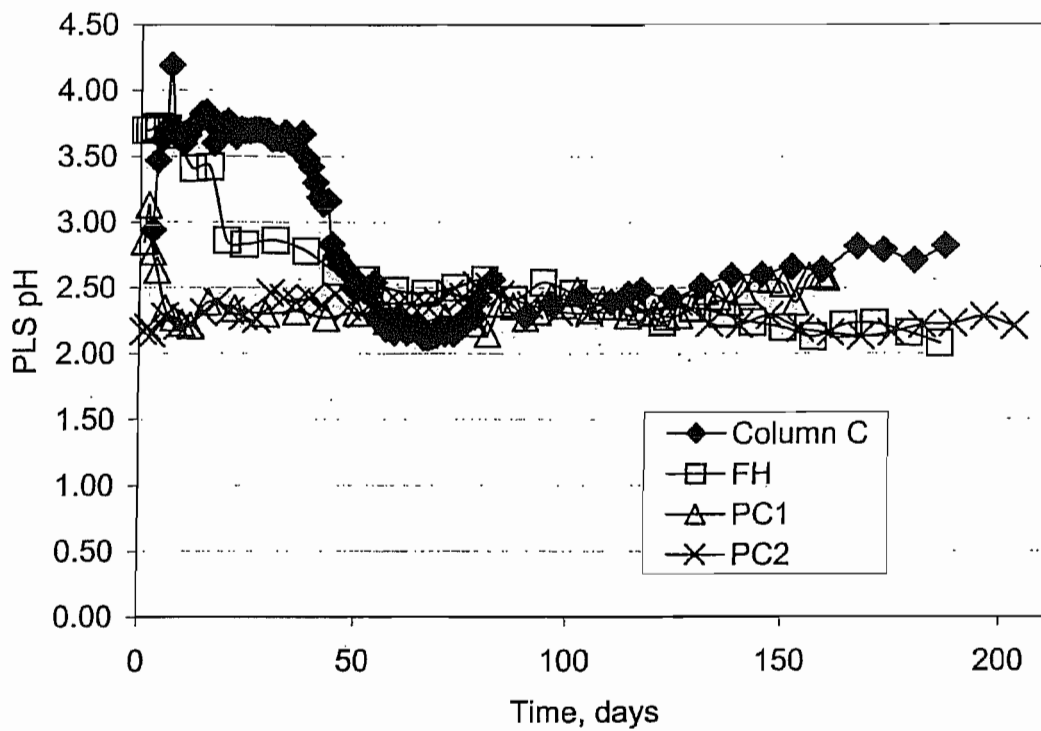


Figure 3.2.7 PLS pH Profiles for Carmacks Column Tests.

After the interval from day 72 to day 75, no further acid additions were made to this test. The solution was allowed to drain down from the column for eight days to provide an indication of the drain-down characteristics as discussed in Section 3.2.2. After the drain down test was completed the PLS solution was continuously recirculated through the material until the copper grade reached about 0.5 gpl Cu. At this point solvent extraction was performed to remove the copper and the solution was then recycled again. As is summarized in Figure 3.2.8, the pH of both the feed and PLS increased over this period as the free acid was consumed. The downward spikes in the feed pH in this figure show the points where solvent extraction was carried out, resulting in a temporary increase in the feed acid concentration. The acid added through solvent extraction was consumed quickly by the ore each time.

The net conclusion is that Sample C leaches somewhat more readily than the other materials tested and requires a high PLS pH in order to minimize acid consumption. The copper extraction of about 83% achieved for the column test over the design leaching period of about 120 days would translate into a commercial copper extraction of about 76 to 78%, depending on operating efficiencies. When processing materials with a leaching profile such as that observed for column C, the primary leach cycle should be reduced to about 60 days. About 70% copper extraction (column) would be achieved during this period with the remaining extraction achieved during subsequent leaching of overlying lifts. The acid consumption will be minimized in this manner.

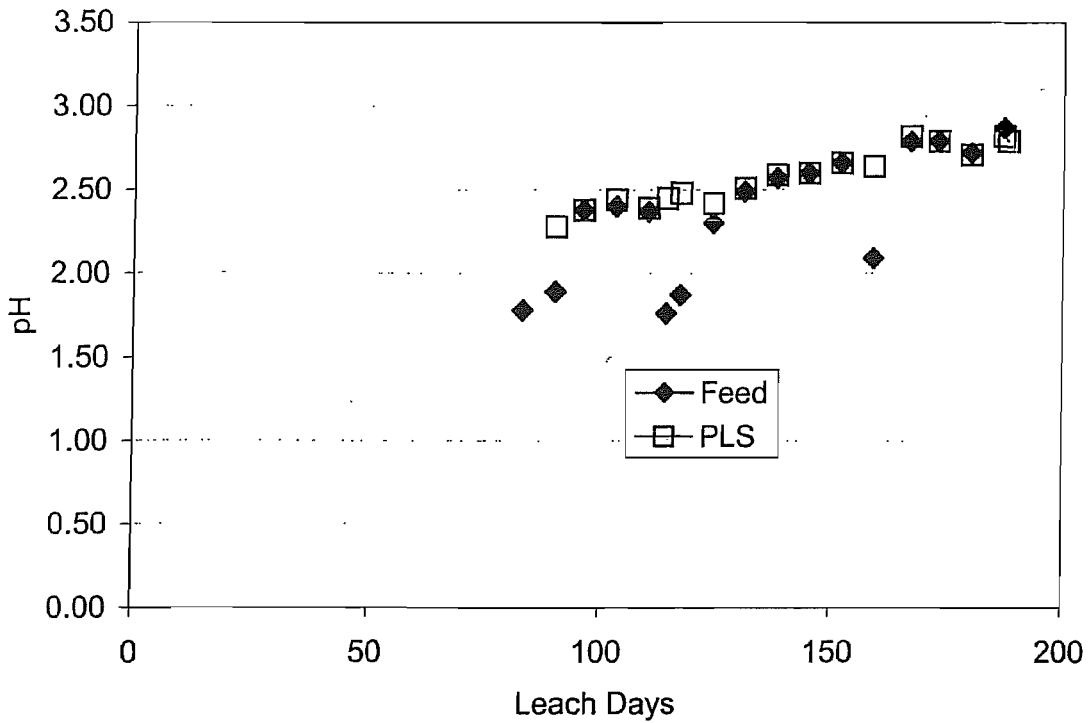


Figure 3.2.8 pH of Feed and PLS During Solution Recirculation.

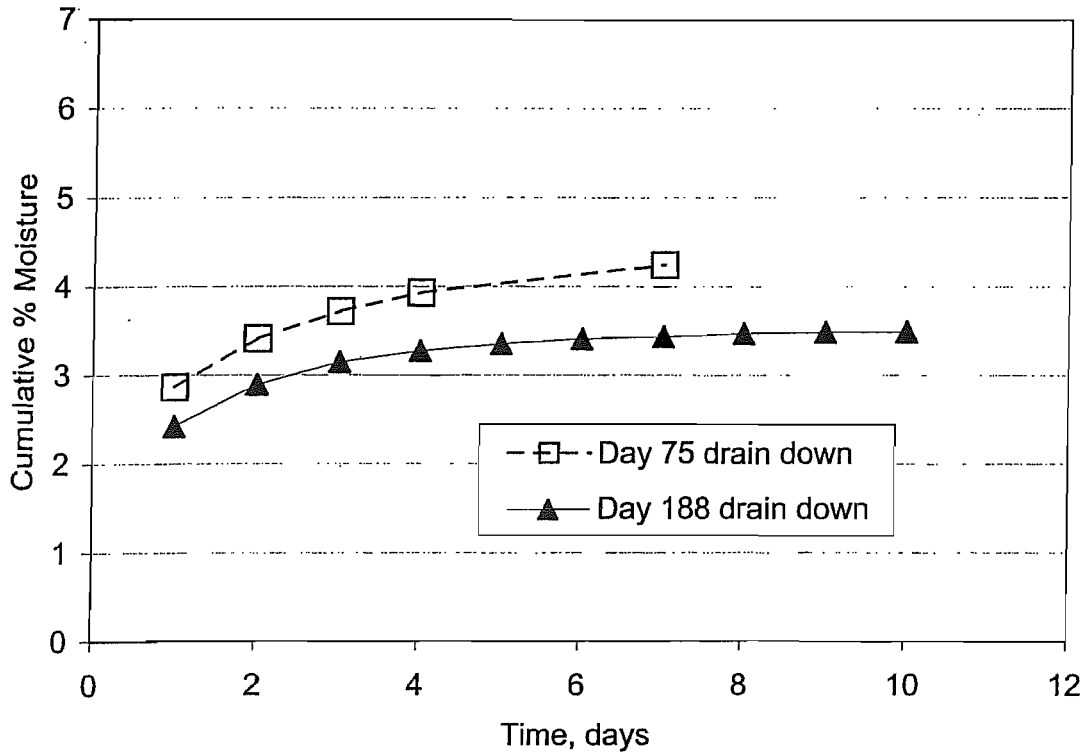


Figure 3.2.9 Drain Down Curves for Column C.

The recovery by size fraction for column C is summarized in Table 3.2.1. The recovery is seen to be quite uniform for all fractions below about ½ inch. Above this size there is a slight loss in copper extraction. It would be expected that the coarsest particles would continue to leach but at a very slow rate. Similarly, even larger particles than included in this test would be expected to achieve an even lower copper extraction in a given leach time.

**Table 3.2.1**  
**Recovery of Copper by Size Fraction at ¾ Inch Crush Size**

Fraction	Final Residue			Recovery
	Wt. %	Cu <sub>T</sub> , %	Cu <sub>AS</sub> , %	Cu <sub>T</sub> %
- ¾ + ½	5.5	0.18	0.17	79.5
- ½ + 3/8	13.5	0.12	0.12	86.8
- 3/8 + ¼	14.7	0.11	0.09	88.2
- ¼ + 6m	14.1	0.10	0.09	89.7
- 6m	52.3	0.12	0.11	88.9
Total		0.12	0.11	87.8

The trace element concentration for the Column C PLS for a number of days is summarized in Table 3.2.2 and for the Raffinate is summarized in Table 3.2.3. It is apparent that the gangue mineral constituents are being leached by the solution as evidenced by the increase in dissolved Al, Mg, Fe, and Si. It is likely that the biotite in the sample is undergoing significant leaching and accounts for much of the acid consumption.

**Table 3.2.2**  
**ICP Analysis of Pregnant Leach Solution**

Element	Unit	Column C PLS					
		Day 5	Day 12	Day 19	Day 26	Day 33	Day 40
SO <sub>4</sub> <sup>=</sup>	g/L	39.97	16.46	16.22	19.88	22.12	26.98
Al	mg/L	1502.2	814.0	1175.7	1485.2	1950.2	2417.4
Sb	mg/L	0.1	< 0.1	0.1	< 0.1	< 0.1	0.1
As	mg/L	2.7	1.0	1.5	1.7	1.6	2.4
Ba	mg/L	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01
Bi	mg/L	< 0.1	< 0.1	< 0.1	0.9	0.2	< 0.1
Cd	mg/L	< 0.01	0.16	0.20	0.21	0.24	0.27
Ca	mg/L	497.9	466.4	463.9	467.0	470.0	496.1
Cr	mg/L	0.21	< 0.01	0.01	0.23	0.50	0.83
Co	mg/L	9.80	4.51	5.02	5.42	5.98	6.68
Cu	mg/L	15.78 g	5083.31	2720.73	2194.03	1722.71	1270.35
Fe	mg/L	503.13	667.70	740.58	1176.52	1443.16	1787.69
La	mg/L	2.17	1.50	1.30	1.26	1.31	1.38
Pb	mg/L	0.48	0.14	< 0.05	< 0.05	< 0.05	< 0.05
Mg	mg/L	1325.5	765.1	911.2	1171.8	1453.1	1588.1
Mn	mg/L	359.61	153.19	173.98	175.80	199.37	206.85
Hg	mg/L	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05
Mo	mg/L	0.33	0.17	0.16	0.13	0.24	0.17
Ni	mg/L	2.03	1.04	1.22	1.58	1.72	2.15
P	mg/L	187.9	59.4	31.0	26.5	24.6	21.4
K	mg/L	17	11	18	22	13	5
Sc	mg/L	0.73	0.27	0.28	0.84	1.31	1.60
Si	mg/L	49.9	58.7	62.4		107.61	
Ag	mg/L	< 0.02	< 0.02	< 0.02	< 0.02	< 0.02	< 0.02
Na	mg/L	39	< 1	22	21	17	15
Sr	mg/L	2.28	1.51	1.15	0.82	0.59	0.61
Th	mg/L	0.2	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1
Ti	mg/L	0.6	0.2	0.1	< 0.1	< 0.1	< 0.1
W	mg/L	3.7	0.7	0.4	0.7	0.5	0.3
V	mg/L	0.10	0.03	0.02	0.02	0.03	0.53
Zn	mg/L	45.23	19.68	18.91	20.34	22.21	25.70
Zr	mg/L	0.02	0.02	< 0.01	< 0.01	< 0.01	0.06

Element	Unit	Column C PLS					
		Day 47	Day 54	Day 61	Day 68	Day 103	Day 246
SO <sub>4</sub> <sup>=</sup>	g/L	33.06	42.53	36.47	44.42		
Al	mg/L	2935.3	3705.5	2935.4	3778.7	3424.8	3596
Sb	mg/L	< 0.1	<0.1	<0.1	0.1	0.2	0.3
As	mg/L	1.7	2.2	1.0	1.7	6.6	12.9
Ba	mg/L	< 0.01	<0.01	<0.01	< 0.01	<0.01	<0.01
Bi	mg/L	< 0.1	<0.1	<0.1	< 0.1	<0.1	<0.1
Cd	mg/L	0.26	0.34	0.22	0.28	0.25	0.41
Ca	mg/L	508.9	474.9	481.9	491.2	497.5	493.2
Cr	mg/L	1.48	2.14	1.84	2.50	2.19	0.86
Co	mg/L	7.26	9.15	5.80	7.01	6.74	7.65
Cu	mg/L	797.00	932.20	484.80	323.90	427.44	271.54
Fe	mg/L	3537.02	4616.31	4291.79	5015.75	1608	28.6
La	mg/L	1.54	1.69	1.07	1.32	0.60	<0.05
Pb	mg/L	< 0.05	<0.05	<0.05	< 0.05	<0.05	0.35
Mg	mg/L	1966.0	2449.8	1952.5	2574.1	2458	3629
Mn	mg/L	224.68	265.10	167.52	217.77	189.7	341.5
Hg	mg/L	< 0.05	<0.05	<0.05	< 0.05	<0.05	0.16
Mo	mg/L	0.94	1.55	3.43	2.99	0.54	0.26
Ni	mg/L	2.36	2.84	1.83	2.90	2.75	3.06
P	mg/L	160.7	209.2	219.9	210.6	88.5	6.9
K	mg/L	106	92.	154.	236	41	12
Sc	mg/L	1.90	2.17	1.68	2.12	1.76	0.45
Si	mg/L	279.42	261.05	264.96	284.13	211.9	
Ag	mg/L	< 0.02	<0.02	,0.02	< 0.02	<0.02	0.04
Na	mg/L	6	8.	<1.	< 1	16	40
Sr	mg/L	0.99	1.04	1.05	1.12	0.29	0.06
Th	mg/L	< 0.1	<0.1	<0.1	< 0.1	<0.1	0.2
Ti	mg/L	0.3	0.5	1.1	3.0	0.6	<0.1
W	mg/L	0.4	<0.1	0.6	< 0.1	<0.1	<0.1
V	mg/L	8.16	12.81	12.37	16.05	1.52	<0.01
Zn	mg/L	27.97	31.46	23.27	29.62	31.43	37.42
Zr	mg/L	0.05	0.09	0.06	0.08	<0.01	0.01

Table 3.2.3  
ICP Analysis of Column C Raffinate

Element	Unit	Column C SX Raffinate					
		Day 5	Day 12	Day 19	Day 26	Day 33	Day 40
SO <sub>4</sub> <sup>=</sup>	g/L	46.02	19.42	17.60	20.77	24.62	27.68
Al	mg/L	1655.0	822.2	1151.1	1532.4	1937.1	2402.0
Sb	mg/L	0.3	< 0.1	< 0.1	< 0.1	< 0.1	0.2
As	mg/L	2.3	0.9	1.1	1.6	1.7	2.3
Ba	mg/L	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01
Bi	mg/L	< 0.1	< 0.1	0.1	< 0.1	< 0.1	< 0.1
Cd	mg/L	0.42	0.18	0.19	0.22	0.23	0.27
Ca	mg/L	505.1	466.8	463.8	472.5	463.2	495.1
Cr	mg/L	0.28	< 0.01	< 0.01	0.24	0.52	0.91
Co	mg/L	10.71	4.47	5.02	5.39	5.93	6.67
Cu	mg/L	235.70	6.55	12.70	9.11	9.58	7.42
Fe	mg/L	586.58	678.84	740.20	1213.93	1426.50	1831.20
La	mg/L	2.32	1.49	1.32	1.20	1.31	1.38
Pb	mg/L	0.11	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05
Mg	mg/L	1413.6	756.9	890.5	1127.4	1399.7	1563.7
Mn	mg/L	385.20	152.11	170.16	179.91	197.54	205.64
Hg	mg/L	< 0.05	< 0.05	< 0.05	0.17	< 0.05	< 0.05
Mo	mg/L	0.09	0.08	0.09	0.07	0.04	0.15
Ni	mg/L	2.33	1.06	1.17	1.55	1.73	2.08
P	mg/L	15.4	0.3	1.1	1.4	2.8	8.0
K	mg/L	19	12	20	15	18	5
Sc	mg/L	0.69	0.22	0.25	0.83	1.28	1.66
Si	mg/L	56.6	57.9	62.5		108.75	
Ag	mg/L	< 0.02	< 0.02	0.04	< 0.02	< 0.02	0.05
Na	mg/L	41	< 1	21	21	16	16
Sr	mg/L	2.36	1.52	1.16	0.83	0.58	0.61
Th	mg/L	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1
Ti	mg/L	0.1	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1
W	mg/L	< 0.1	< 0.1	< 0.1	< 0.1	< 0.01	< 0.1
V	mg/L	0.10	< 0.01	< 0.01	< 0.01	0.03	1.01
Zn	mg/L	27.70	13.38	15.34	17.66	19.83	23.95
Zr	mg/L	0.03	< 0.01	0.02	< 0.01	0.03	0.03



Element	Unit	Column C SX Raffinate					
		Day 47	Day 54	Day 61	Day 68		
SO <sub>4</sub> <sup>=</sup>	g/L	32.07	40.53	41.11	46.36		
Al	mg/L	2765.7	3426.5	3304.5	3868.0		
Sb	mg/L	< 0.1	<0.1	<0.1	< 0.1		
As	mg/L	2.0	2.4	1.6	1.7		
Ba	mg/L	< 0.01	<0.01	<0.01	< 0.01		
Bi	mg/L	< 0.1	<0.1	<0.1	< 0.1		
Cd	mg/L	0.25	0.33	0.28	0.28		
Ca	mg/L	503.4	479.4	494.1	488.8		
Cr	mg/L	1.32	1.96	2.10	2.52		
Co	mg/L	7.05	7.80	7.07	7.18		
Cu	mg/L	3.81	4.98	6.09	2.67		
Fe	mg/L	3061.59	4203.01	4344.62	4974.65		
La	mg/L	1.46	1.60	-1.36	1.38		
Pb	mg/L	< 0.05	<0.05	<0.05	< 0.05		
Mg	mg/L	1881.6	2252.7	2197.9	2615.2		
Mn	mg/L	219.39	254.99	209.25	224.90		
Hg	mg/L	< 0.05	<0.05	<0.05	< 0.05		
Mo	mg/L	0.25	0.70	1.50	0.64		
Ni	mg/L	2.27	2.92	2.37	2.91		
P	mg/L	117.4	207.8	232.1	241.7		
K	mg/L	80	87.	147.	241		
Sc	mg/L	1.83	2.14	2.07	2.15		
Si	mg/L	259.23	243.53	267.58	305.63		
Ag	mg/L	< 0.02	<0.02	<0.02	< 0.02		
Na	mg/L	9	10.	4.0	2		
Sr	mg/L	0.97	1.06	1.06	1.12		
Th	mg/L	< 0.1	<0.1	<0.1	< 0.1		
Ti	mg/L	0.2	0.5	1.1	3.2		
W	mg/L	< 0.1	<0.1	0.5	< 0.1		
V	mg/L	7.05	11.34	13.17	15.74		
Zn	mg/L	25.97	28.53	27.16	29.75		
Zr	mg/L	0.01	0.07	0.05	0.09		

### 3.2.2 Drain Down Testwork

Based on the solution balance during the first 6 days of column operation, the total solution hold-up in the column was calculated to be equivalent to 14.8% by weight. Two drain down tests were conducted on this column, one after 75 days leaching and the other at the end of the test. The results are summarized in Figure 3.2.9. The drain down at the end of the test showed a moisture release of about 3.5%. The moisture content of the column residue was determined to be 10.8%, which is in reasonable agreement with the hold-up moisture less the drain down. The decrease in drain down from day 75 to day 188 could be the result of decrepitation with time, resulting in increased moisture content of the leached solids.

A drain down test was also conducted on the ROM leach column. The solution hold-up in this column was determined to be equivalent to 10.2% moisture. Over 7 days the drain down moisture was 1.8% by weight and extending these results to 30 days indicates a drain down of 2.4%.

### 3.2.3 Run of Mine Leaching

A run of mine leaching test (Test AB) was conducted with the bulk sample, using two column sections each 36 inches (~0.9m) in diameter and 4 meters tall. The sample was loaded in each section to a height of about 3.5 meters. All solutions were pumped to the top of the first section of column (Column A) and the PLS from this section became the feed to the second section (Column B) resulting in an effective overall height for the test of 7 meters. Solvent extraction was conducted on the PLS from column B before the solution was recirculated to column A. All results for this test will be discussed on the basis that the two sections together comprised one test. The details for this test are included in Appendix 2.

Once the sample had been placed in the columns, a solution containing 235 gpl sulphuric acid was pumped onto Column A and advanced through B just until solution appeared at the bottom of B. This acid addition, amounting to 12.58 kg/t H<sub>2</sub>SO<sub>4</sub>, was the cure acid addition. Curing was allowed to proceed for 7 days before the column was irrigated with leach solution.

Leaching was initiated by pumping an approximately 8 gpl sulphuric acid solution onto column A at a rate of 7 l/h/m<sup>2</sup> for the first four days of operation. After this time the raffinate was recycled to the feed and was initially diluted with water to maintain the target acid concentration in the feed. Over the course of the test the feed acid concentration was varied as reflected by the feed pH as summarized in Figure 3.2.12, in order to achieve specific objectives. Since the determination of free acid in the leach solutions is quite unreliable the feed conditions are best judged on the basis of the feed pH.

The progress of leaching of the ROM test is summarized on a linear time scale in Figure 3.2.10 and on a log time scale in Figure 3.2.11. In both Figures the results for column C are included for the purpose of comparison. It is apparent from the outset of the tests that the crushed sample leaches much faster than the ROM sample as would be expected. It also appears that the ROM test will have an ultimate copper extraction that is significantly lower than that of the crushed sample. Specific aspects of the ROM test need to be considered to determine whether a ROM leach could achieve higher extractions under different conditions or with an extended leach time.

The extraction curve is flat across the period from day 84 to 91 since no feed solution was being added at this time. The column was allowed to drain down during this period to provide drain down data as discussed in Section 3.2.2. Following the drain down, the feed pH was allowed to increase so that no acid additions were made over the period from day 83 to day 164. The copper extraction curve is therefore somewhat flatter than prior to this period. Following day 164 the feed acid concentration was increased as is apparent from the drop in both the feed pH as shown in Figure 3.2.12 and PLS pH, summarized in Figure 3.2.13. No significant benefit other than a slight increase in the leach rate resulted from this increase in acid concentration and it was therefore gradually allowed to decrease as is apparent from the acid consumption as a function of leach time as summarized in Figure 3.2.14. After day 304, no further acid additions were made to the column and the pH was allowed to increase naturally.

The copper extractions as a function of acid addition for both the ROM and  $\frac{3}{4}$  inch tests are summarized in Figure 3.2.15. It can be seen that in the range of 40 to 65% Cu extraction, the two samples gave very similar results. The shapes of the two curves are consistent with the variations in operating conditions used for the column tests. It is apparent from these results that there is no benefit in terms of copper extraction derived from the use of high acid additions once the initial leach period which achieves somewhere in the range of 50% copper extraction is complete. It also appears that since increased acid consumption for the ROM column was resulting in low incremental copper extraction, a very long leach time of more than 1000 days (with the attendant increase in acid consumption) would be required to achieve a copper extraction similar to that achieved at a  $\frac{3}{4}$  inch crush size.

The residue from the ROM test was not screened into size fractions for assay but it is expected that there would be a significant decrease in copper extraction for the coarser particle sizes.

Tables 3.2.4 and 3.2.5 summarize the dissolved element concentrations for the PLS and raffinate from column AB respectively. As for the column containing crushed sample, the build up of dissolved trace elements is consistent with the leaching process.

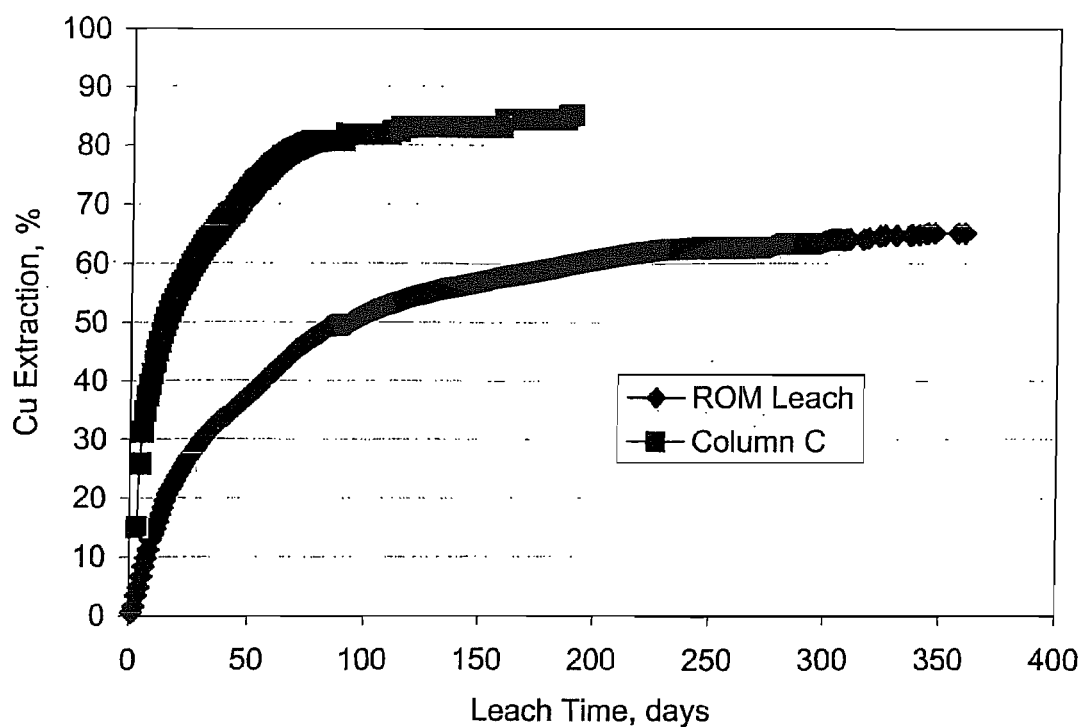


Figure 3.2.10 Cu Extraction Versus Time for Columns C and AB.

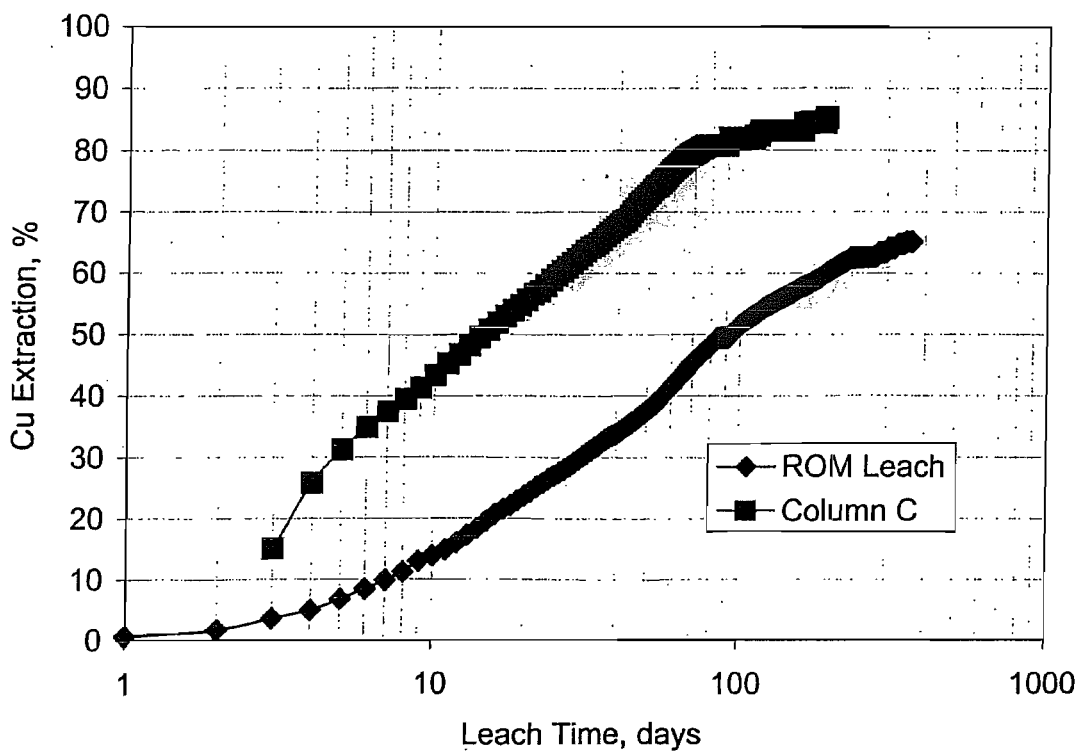


Figure 3.2.11 Cu Leaching vs Log Time for Columns C and AB.

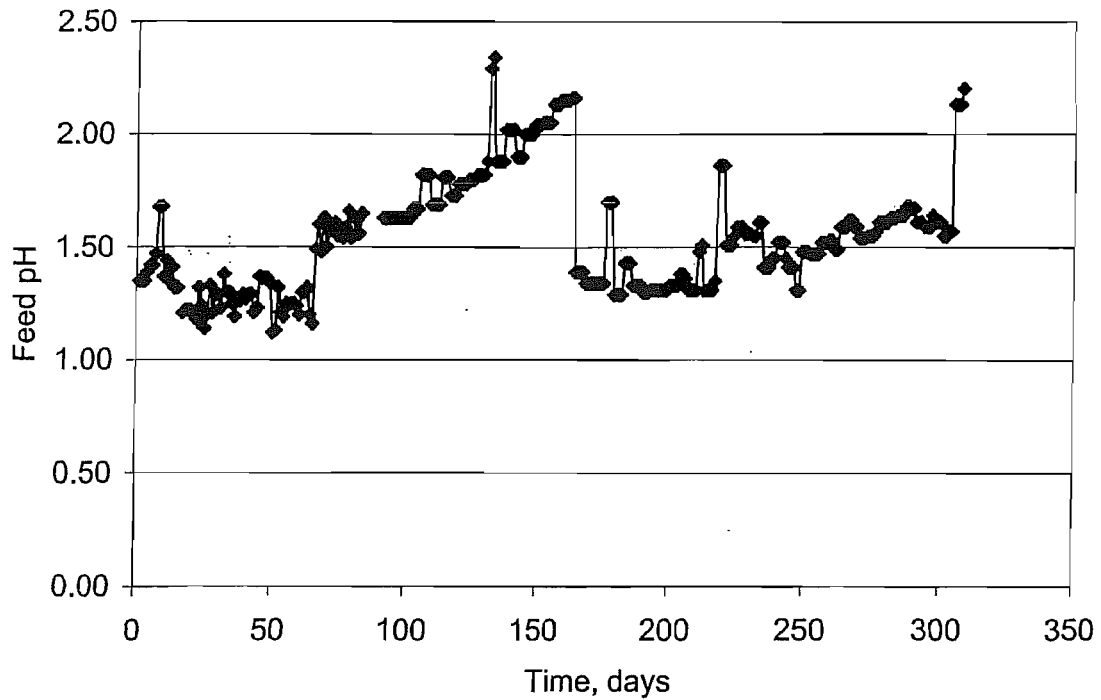


Figure 3.2.12 Feed pH for Column AB.

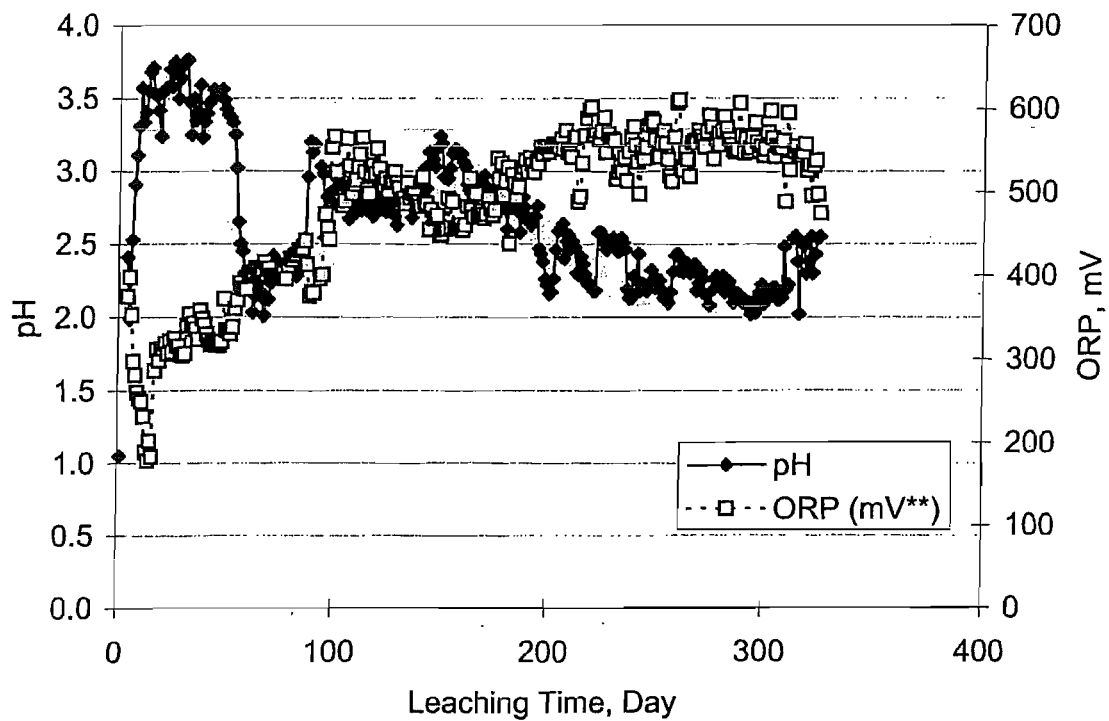


Figure 3.2.13 pH and ORP for Column AB PLS.

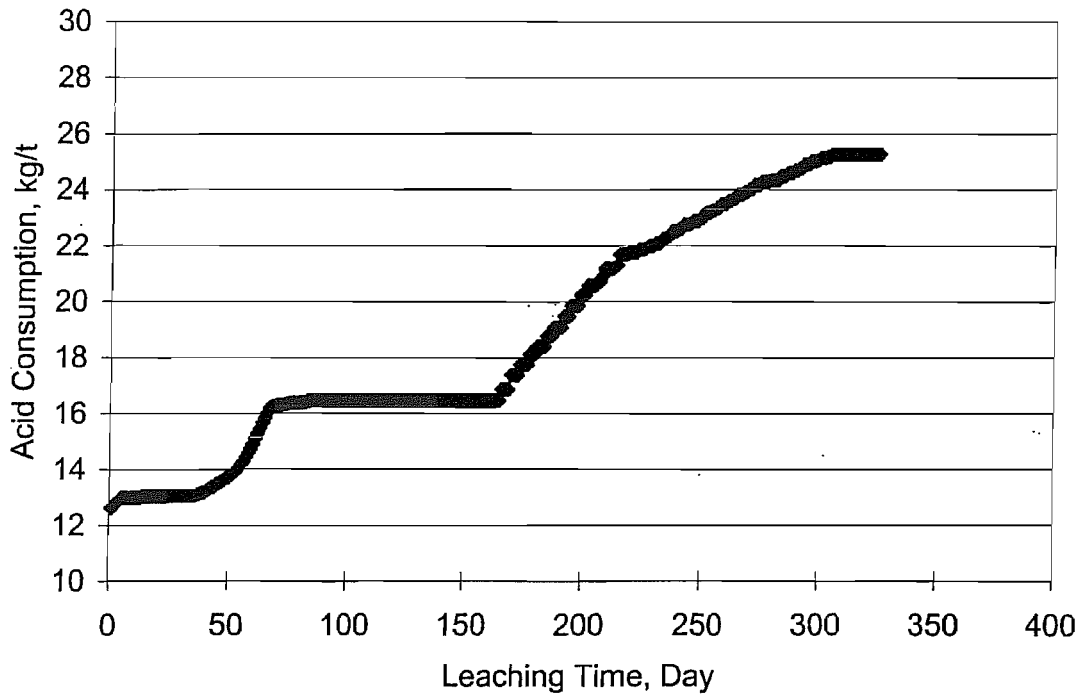


Figure 3.2.14 Acid Consumption vs Time for Column AB.

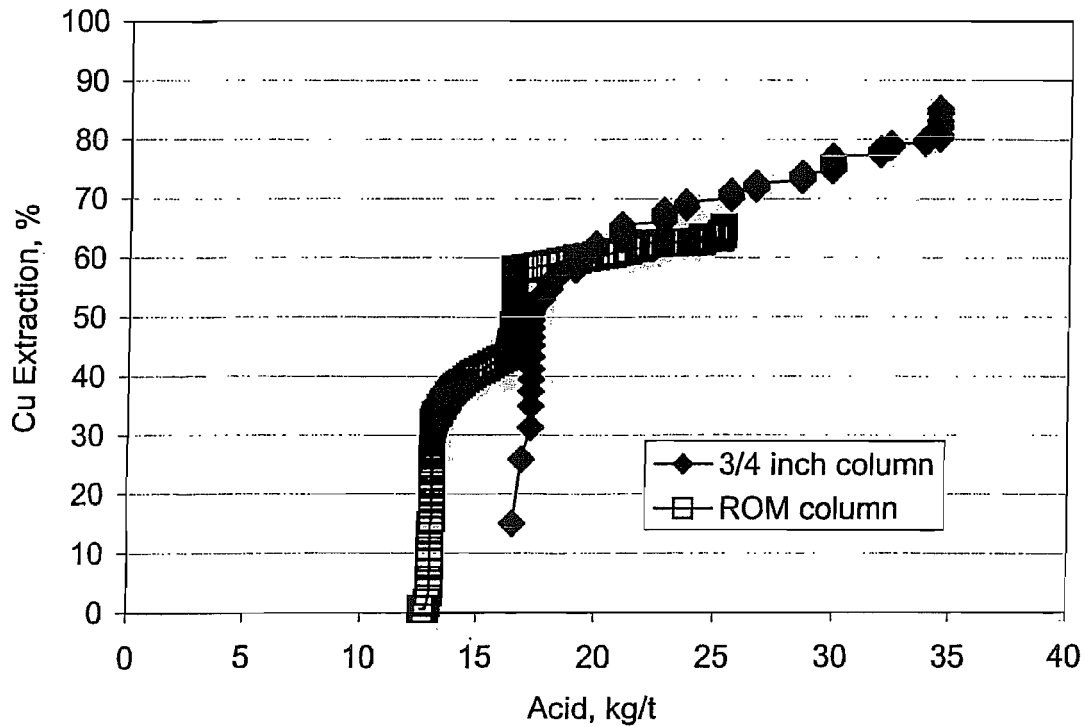


Figure 3.2.15 Cu Extraction vs Acid Consumption for Columns C and AB.

Table 3.2.4  
ICP Analysis of Column B PLS

Element	Unit	Column B PLS					
		Day 7	Day 14	Day 21	Day 28	Day 35	Day 42
SO <sub>4</sub> <sup>=</sup>	g/L	65.76	39.86	28.38	23.38	25.55	26.38
Al	mg/L	3912.9	2037.3	1389.6	1299.3	1413.9	1750.0
Sb	mg/L	< 0.1	0.2	< 0.1	< 0.1	< 0.1	0.1
As	mg/L	4.0	2.6	2.1	1.6	2.1	1.7
Ba	mg/L	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01
Bi	mg/L	< 0.1	0.6	< 0.1	0.6	2.6	1.7
Cd	mg/L	< 0.01	0.32	0.27	0.24	0.26	0.31
Ca	mg/L	482.2	470.2	459.3	455.9	467.7	463.9
Cr	mg/L	0.92	0.20	0.04	0.03	0.05	0.11
Co	mg/L	13.34	9.86	7.37	6.33	7.33	7.96
Cu	g/L	15.69	10.04	6734.65	5005.51	4703.37	3924.04
Fe	mg/L	3075.71	1493.27	1013.21	1025.27	1093.43	1290.37
La	mg/L	2.01	1.32	1.08	0.89	0.87	0.91
Pb	mg/L	< 0.05	< 0.05	< 0.05	0.24	< 0.05	0.10
Mg	mg/L	3496.8	2301.7	1748.2	1616.6	1775.9	2149.5
Mn	mg/L	505.18	368.08	266.33	233.77	260.13	290.49
Hg	mg/L	< 0.05	< 0.05	< 0.05	< 0.05	0.11	< 0.05
Mo	mg/L	1.69	0.35	0.12	0.15	0.21	0.25
Ni	mg/L	3.28	2.26	1.79	1.53	2.04	2.15
P	mg/L	217.8	96.2	87.5	55.6	49.8	49.3
K	mg/L	213	125	70	66	54	61
Sc	mg/L	1.49	0.77	0.46	0.42	0.48	0.65
Si	mg/L	125.6	79.6	72.9	70.2	C0251	85.66
Ag	mg/L	< 0.02	< 0.02	< 0.02	< 0.02	< 0.02	< 0.02
Na	mg/L	29	26	14	23	30	26
Sr	mg/L	1.36	0.96	0.77	0.69	0.64	0.57
Th	mg/L	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1
Ti	mg/L	1.7	0.5	0.3	0.2	0.2	0.2
W	mg/L	3.4	2.2	1.0	0.7	1.6	1.6
V	mg/L	3.85	0.69	0.09	0.13	0.39	0.47
Zn	mg/L	57.44	39.66	29.39	25.19	27.10	28.80
Zr	mg/L	0.10	0.02	0.03	0.03	< 0.01	< 0.01

Element	Unit	Column B PLS					
		Day 49	Day 56	Day 63	Day 70	Day 77	Day 112
SO <sub>4</sub> <sup>=</sup>	g/L	27.68	30.94	36.04	38.58	39.75	
Al	mg/L	1842.8	2274.4	2737.9	3120.1	3319.9	3783
Sb	mg/L	< 0.1	< 0.1	0.3	<0.1	0.3	0.2
As	mg/L	2.6	2.3	2.4	3.0	2.9	9.0
Ba	mg/L	< 0.01	< 0.01	<0.01	<0.01	< 0.01	<0.01
Bi	mg/L	1.0	0.2	<0.1	1.2	< 0.1	<0.1
Cd	mg/L	0.34	0.34	0.38	0.41	0.43	0.54
Ca	mg/L	500.9	517.2	482.0	491.1	479.8	515.2
Cr	mg/L	0.15	0.46	0.72	0.79	0.82	0.52
Co	mg/L	8.59	9.02	9.01	9.78	10.00	12.32
Cu	g/L	3207.61	1915.31	1661.69	1442.53	1222.32	1080
Fe	mg/L	1391.77	2044.03	2576.85	2677.90	2273.35	267.3
La	mg/L	0.88	1.13	1.12	1.13	1.13	0.52
Pb	mg/L	< 0.05	< 0.05	<0.05	<0.05	< 0.05	0.80
Mg	mg/L	2025.7	2354.4	2589.1	2772.0	2917.8	3320
Mn	mg/L	281.31	299.20	318.25	321.35	350.77	394.6
Hg	mg/L	< 0.05	< 0.05	<0.05	<0.05	< 0.05	<0.05
Mo	mg/L	0.26	0.91	1.57	1.86	1.05	0.25
Ni	mg/L	2.46	2.45	2.77	2.82	3.15	4.15
P	mg/L	44.3	80.0	92.8	71.4	70.7	26.1
K	mg/L	69	152	117.	104.	71	6
Sc	mg/L	0.68	1.04	1.35	1.55	1.66	1.60
Si	mg/L	C0302	204.68	177.9	171.99	147.63	99.36
Ag	mg/L	< 0.02	< 0.02	<0.02	<0.02	< 0.02	<0.02
Na	mg/L	26	26	25.	26.	28	37
Sr	mg/L	0.52	0.83	0.80	0.73	0.54	0.13
Th	mg/L	< 0.1	< 0.1	<0.1	<0.1	< 0.1	<0.1
Ti	mg/L	0.2	2.4	4.4	2.7	2.5	0.4
W	mg/L	0.9	1.1	<0.1	0.4	< 0.1	0.2
V	mg/L	0.68	3.67	4.67	4.71	3.46	<0.01
Zn	mg/L	31.56	31.22	31.09	33.32	35.25	45.48
Zr	mg/L	< 0.01	< 0.01	0.04	0.02	0.07	<0.01



**Table 3.2.5**  
**ICP Analysis of Column B Raffinate**

Element	Unit	Column B SX Raffinate					
		Day 7	Day 14	Day 21	Day 28	Day 35	Day 42
SO <sub>4</sub> <sup>=</sup>	g/L	67.50	39.93	28.37	23.73	25.48	26.99
Al	mg/L	3879.9	2054.3	1420.7	1291.7	1391.1	1775.8
Sb	mg/L	< 0.1	< 0.1	0.3	0.1	< 0.1	< 0.1
As	mg/L	2.9	2.3	1.5	1.5	1.6	1.6
Ba	mg/L	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01
Bi	mg/L	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1
Cd	mg/L	0.50	0.36	0.29	0.26	0.29	0.31
Ca	mg/L	493.4	472.9	463.0	462.6	469.7	465.6
Cr	mg/L	0.94	0.22	0.07	0.03	0.07	0.09
Co	mg/L	13.54	9.93	7.35	6.47	7.32	8.02
Cu	mg/L	1561.49	2299.47	49.27	184.54	71.37	52.15
Fe	mg/L	3056.49	1487.32	1067.34	998.51	1078.72	1284.69
La	mg/L	2.05	1.40	1.00	0.87	0.94	0.93
Pb	mg/L	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05
Mg	mg/L	3531.4	2321.5	1751.3	1566.5	1750.9	2052.7
Mn	mg/L	499.30	368.77	268.84	233.30	256.84	289.32
Hg	mg/L	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05
Mo	mg/L	1.44	1.22	0.20	0.21	0.15	0.13
Ni	mg/L	3.18	2.38	1.71	1.71	1.75	2.06
P	mg/L	41.4	29.1	1.5	3.0	2.2	3.2
K	mg/L	227	99	59	59	132	61
Sc	mg/L	1.38	0.69	0.40	0.38	0.44	0.63
Si	mg/L	130.6	78.6	73.8	71.2	C0251	87.16
Ag	mg/L	< 0.02	< 0.02	< 0.02	< 0.02	< 0.02	< 0.02
Na	mg/L	27	26	15	24	31	28
Sr	mg/L	1.36	0.96	0.78	0.70	0.64	0.58
Th	mg/L	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1	< 0.1
Ti	mg/L	1.2	0.2	< 0.1	< 0.1	< 0.1	< 0.1
W	mg/L	< 0.1	0.2	< 0.1	< 0.1	< 0.01	< 0.1
V	mg/L	3.85	0.53	0.31	0.16	0.39	0.44
Zn	mg/L	40.11	29.65	20.71	19.12	21.24	24.21
Zr	mg/L	0.09	0.03	0.01	0.03	< 0.01	< 0.01

Element	Unit	Column B SX Raffinate					
		Day 49	Day 56	Day 63	Day 70	Day 77	Day 112
SO <sub>4</sub> <sup>=</sup>	g/L	27.89	30.92	36.46	38.72	39.33	
Al	mg/L	1850.7	2282.0	2797.4	3051.9	3317.1	3804
Sb	mg/L	< 0.1	< 0.1	0.2	<0.1	< 0.1	0.1
As	mg/L	2.2	2.3	3.0	2.8	3.2	8.1
Ba	mg/L	< 0.01	< 0.01	<0.01	<0.01	< 0.01	<0.01
Bi	mg/L	< 0.1	< 0.1	<0.1	0.1	< 0.1	<0.1
Cd	mg/L	0.36	0.34	0.39	0.41	0.43	0.56
Ca	mg/L	497.0	517.4	489.1	487.8	485.5	509.5
Cr	mg/L	0.16	0.49	0.71	0.78	0.84	0.56
Co	mg/L	8.51	9.05	9.15	9.71	10.12	12.21
Cu	mg/L	36.02	14.07	12.03	11.89	6.00	3.39
Fe	mg/L	1388.87	2019.78	2549.83	2580.55	2131.68	170.2
La	mg/L	0.92	1.11	1.08	1.11	1.08	0.59
Pb	mg/L	< 0.05	0.21	<0.05	<0.05	< 0.05	0.78
Mg	mg/L	2049.9	2375.6	2605.4	2790.5	2957.9	3347
Mn	mg/L	281.62	301.58	322.29	317.63	351.93	396.4
Hg	mg/L	< 0.05	< 0.05	0.16	<0.05	0.11	<0.05
Mo	mg/L	0.13	0.21	0.45	1.32	0.69	0.21
Ni	mg/L	2.41	2.54	2.83	2.82	3.14	3.97
P	mg/L	2.9	54.9	73.3	51.2	57.6	14.5
K	mg/L	67	222	107	102	63	15
Sc	mg/L	0.75	1.03	1.39	1.51	1.67	1.60
Si	mg/L	C0302	203.66	183.59	169.90	148.07	99.12
Ag	mg/L	< 0.02	< 0.02	<0.02	<0.02	< 0.02	<0.02
Na	mg/L	27	27	27	26	29	38
Sr	mg/L	0.52	0.83	0.84	0.71	0.54	0.14
Th	mg/L	< 0.1	< 0.1	<0.1	<0.1	< 0.1	<0.1
Ti	mg/L	< 0.1	2.4	4.3	2.4	2.3	0.3
W	mg/L	< 0.1	< 0.1	<0.1	0.2	< 0.1	<0.1
V	mg/L	0.94	3.68	4.70	4.48	3.35	<0.01
Zn	mg/L	26.86	29.10	29.72	31.46	34.26	43.69
Zr	mg/L	< 0.01	0.03	0.02	0.03	0.05	0.01

### 3.3 Decommissioning Testwork

Column neutralization testwork conducted on the pilot columns reported in February 1996 indicated that while an effluent pH of just over pH 4 was readily achieved, the pH appeared to be buffered at this point. Changing the feed pH across the range of pH 7 to 11 did not impact the effluent pH nor did the use of lime in place of sodium hydroxide. Since elevated concentrations of rock forming elements were detected in solution it appears that the acid leaching of the Carmacks material results in an alteration process, likely to a clay-like mineral phase, and this altered product leaches when a base is added.

Various test series were conducted as part of the present program to establish the requirements for decommissioning of the heap leach. These tests included batch and column tests to study the neutralization properties of the leached solids. The objective of this testwork was to establish a procedure whereby the leach residue could be stabilized so that it would not release acid or deleterious elements into the environment once leaching was complete.

#### 3.3.1 Batch Neutralization of Leached Solids

Some scoping testwork was conducted on the L1 waste sample prior to the completion of the column copper leach tests in order to establish the conditions most likely to achieve the decommissioning objectives. A report on this work was completed in May 1998 and is included as Appendix 3.

An initial test on this sample was conducted by leaching the sample with deionized water for 24 hours. The pH of the solution stabilized at just under pH 4. An analysis of the solution indicated that Al, Ca, Fe, Mg, Mn and K were leaching, together with copper and sulphate. It was apparent that this material had not been completely leached and had also not been washed effectively.

Since the waste sample had been only partially leached during the field pilot test, it was re-acidified with a 15 gpl H<sub>2</sub>SO<sub>4</sub> solution prior to neutralization tests. After 48 hours contact time, the acid solution was drained from the sample and it was rinsed with a 2 gpl acid solution for a further 24 hours. Analyses for this sample (AL1) obtained during the subsequent testwork indicated that this procedure had not removed all the soluble copper.

Four tests were conducted on the sample with different neutralizing agents. The test procedure involved agitating the sample with deionized water and then adding the neutralizing agent to target pH levels. The test are summarized in Table 3.3.1.

**Table 3.3.1**  
**Neutralization Tests on Carmacks Waste Sample AL1.**

Test No.	Neutralizing Agent
AL1 – T1	NaOH
AL1 – T2	Lime
AL1 – T3	Na <sub>2</sub> CO <sub>3</sub>
AL1 – T4	Water

The test results indicate that while the solution with no base added remained at about pH 4, with the addition of lime, sodium carbonate or sodium hydroxide, the pH could be adjusted quite readily to about pH 7 and was stable once this value was achieved. These test results suggest that when a basic solution is in free contact with leached solids, a neutralizing effect is achieved and there is not unlimited release of acid. These results are somewhat at odds with the results of the pilot column neutralization where even the top section of column material remained at about pH 4 after considerable basis solution had passed through. It is likely that the large volume of solution available in the present titration tests is overwhelming the surface reactions that are detected under percolation conditions.

### 3.3.2 Column Neutralization of Leached Solids

Several column tests were conducted to study the neutralization of the waste sample (L1) under different conditions. Two columns, N1 and N2, were conducted initially using 15 cm x 5.5 meter (6" x 216") columns filled with the waste sample. Test N1 was conducted by feeding a sodium carbonate solution to the column while test N2 was conducted with sodium hydroxide. The test details are attached in Appendix 4. In each case, the feed rate of the solution was comparable to that used during leaching of the copper. The feed pH to column N1 was varied over the course of the test, in response to the effluent pH while the feed pH to test N2 was fairly constant.

The feed pH and resulting effluent pH for these two tests are summarized in Figures 3.3.1 and 3.3.2. The response of the waste to the two neutralizing agents was drastically different. For test N2, with sodium hydroxide the effluent pH remained near 4.0 over the period of the test. Although the feed pH was about 9 throughout the test, this feed pH translates to a very low cumulative addition of sodium hydroxide of 0.013 kg/tonne. For test N1 using a 5% solution of sodium carbonate, the effluent pH increased very rapidly to values over pH10. After 20 days irrigation the column was rested for one week and when irrigation resumed the feed was changed to a pH near 7, representing a very low rate of sodium carbonate addition. A feed pH of 7 was selected as being near the groundwater pH in the area of the deposit. The effluent pH remained high following the rest and the decrease in feed pH. The volume of pH 7 water pumped through the column was far greater than the amount of moisture which would have been held in the column following the rest

period so that the continued high effluent pH is not simply the result of flushing out the high pH solution which had been pumped into the column initially.

Based on the observed results, together with information in the published literature it appears that the presence of sodium carbonate in solution converts the surface of the alteration products formed during leaching so that they become stable and cease releasing their constituents such as aluminum into solution. This observation is supported by the analysis of the day 20 effluent for column N1 compared to the day 4 effluent as summarized in Table 3.3.2.

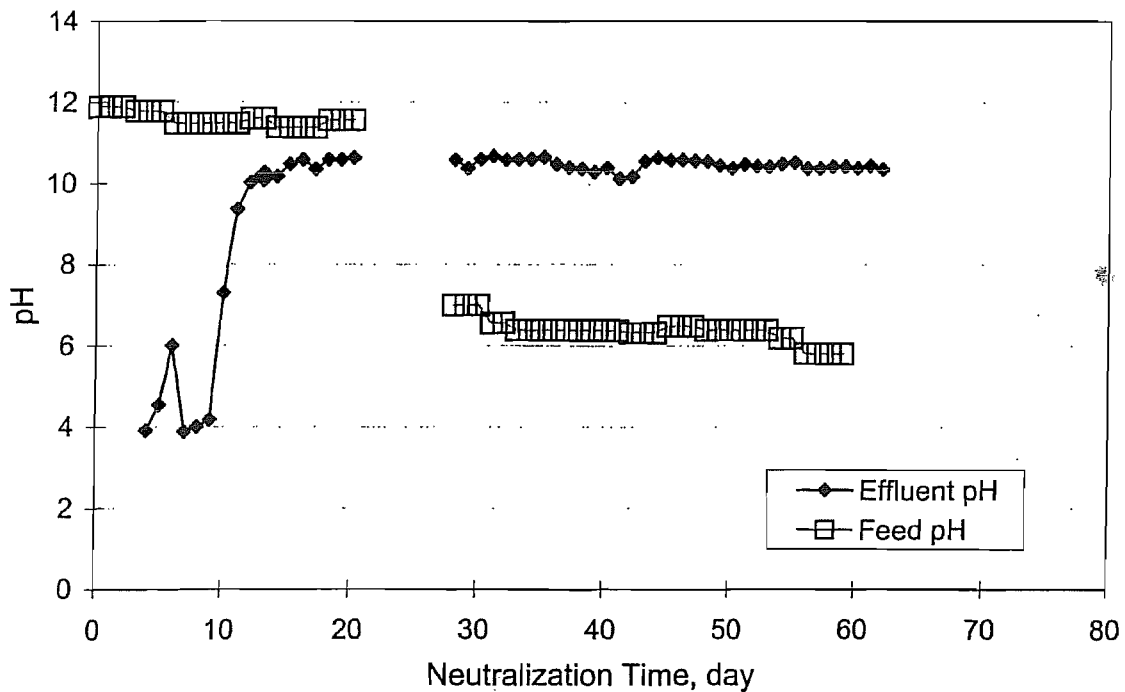


Figure 3.3.1 pH of Feed and Effluent for Neutralization Column N1.

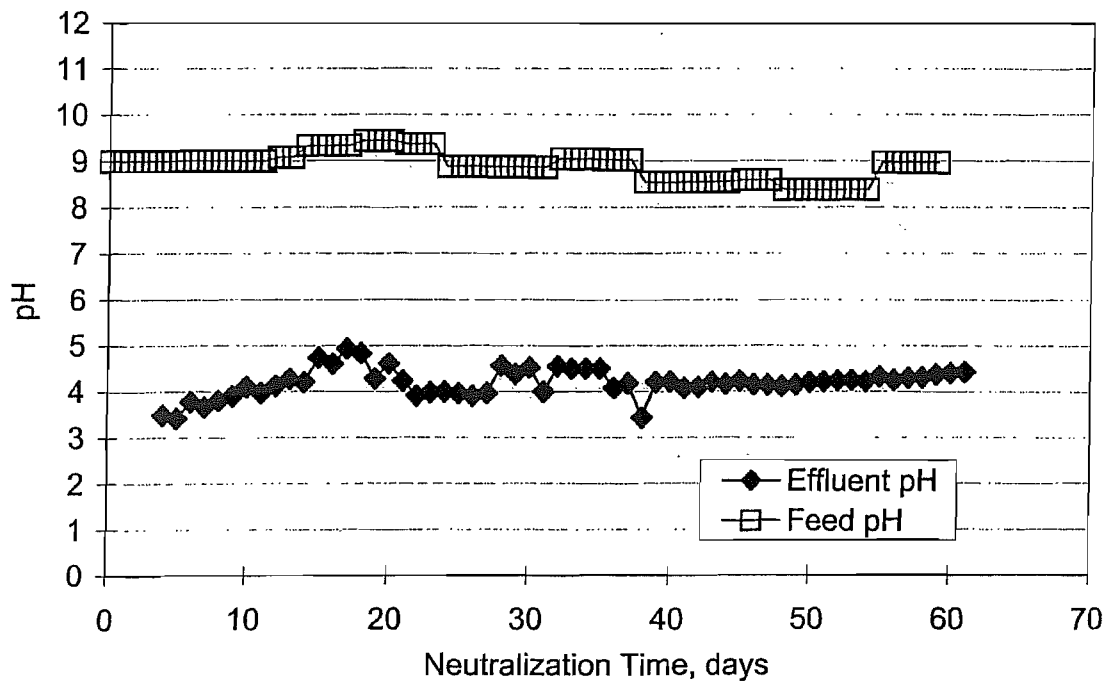


Figure 3.3.2 Feed and Effluent pH for Neutralization Test N2.

**Table 3.3.2**  
**Multi-element ICP Analysis of Column N1 Effluent, mg/L**

Element	Day 4	Day 20
Al	2213.8	<0.2
Sb	0.5	0.1
As	6.9	0.3
Ba	<0.01	<0.01
Bi	<0.1	<0.1
Cd	0.08	<0.01
Ca	481.2	2.1
Cr	<0.01	<0.01
Co	13.59	<0.01
Cu	25180	5.86
Fe	28.48	0.23
La	0.09	<0.05
Pb	0.76	<0.05
Mg	5155.2	<0.1
Mn	755.04	0.02
Hg	<0.05	0.10
Mo	0.05	4.29
Ni	7.18	0.94
P	334.0	88.3
K	33	49
Sc	0.47	<0.01
Ag	<0.02	<0.02
Na	1077	18000
Sr	0.18	<0.01
Th	0.5	<0.1
Ti	0.9	<0.1
W	6.1	0.2
V	0.08	1.11
Zn	82.99	<0.01
Zr	0.01	0.04
SO <sub>4</sub>	70800	600
CO <sub>3</sub>	<1	9000
HCO <sub>3</sub>	<1	37000

The consumption of sodium carbonate is not apparent from this test. While the day 4 effluent analysis indicates that the feed carbonate is being entirely consumed at this point, the day 20 effluent contains 45 gpl combined carbonate and bicarbonate, representing most of the feed carbonate. Since analyses are not available for all the effluents a balance is not possible.

A third column test was conducted on the waste material under more moderate additions of sodium carbonate to establish the sodium carbonate consumption required to achieve a neutral effluent pH and to observe the effects of aging of the neutralized solids. This test, N3, was conducted in a smaller 10 cm x 1.5 meter (4" x 60") column since limited sample material remained at this point. The test was started by feeding a solution of water adjusted to pH 7 with sodium carbonate to the top of the column at a rate comparable to that used during leaching of copper, of 14.8 l/h/m<sup>2</sup>.

The feed and effluent pHs for test N3 are summarized in Figure 3.3.3. Since the effluent pH remained near pH 4 initially, the feed pH was increased in stages until a significant increase in effluent pH was observed. The effluent pH increased above about pH 4 once the feed pH had been increased to pH 10 and increased above pH 6 once the feed pH was increased to pH 10.5. The reason for the lack of response at the lower feed pH is likely that it required an insignificant amount of sodium carbonate to increase the pH of the feed water to these levels. The sodium carbonate addition as a function of time is summarized in Figure 3.3.4. It can be seen that the consumption started to increase after day 32 when the feed pH was increased to pH 10.

Over the period where the feed pH was at 10.5, a somewhat erratic behaviour is apparent for the effluent pH. This variation is believed to result from channeling of the solution through the column. Both columns N1 and N2 had become essentially impermeable over the course of the test, likely (at least in part) as a result of the high fines content and weathered nature of the waste sample used for this testwork.

Towards the end of the initial test period the feed pH was dropped at first to pH 7 and then to pH 5.3 to observe whether the effluent pH would respond in a downward manner. The variation in the effluent pH is within the order of variation, which had been observed previously under steady feed conditions. Irrigation was stopped to this column and it was allowed to rest for 20 days before being irrigated again with pH 7 solution. The effluent pH following this rest period is comparable to that observed prior to the rest.

Following the initial test period above, column N3 was allowed to rest for 509 days before being irrigated again with pH 7 solution for one week. As shown in Figure 3.3.5, the effluent pH over this week varied from an initial value of 5.9 to an average value of 6.3. The sulphate concentration in the effluent decreased from a value of 160 mg/L on day 1 to a value of 8 mg/L on day 7 and the copper concentration decreased from 2.85 ppm on day 1 to 0.69 ppm on day 7. It appears that during the extended rest period, a small amount of sulphuric acid and copper migrated out of the rock fragments. Once the surface concentration of weak acid and copper was removed from the column, a stable discharge situation was achieved.



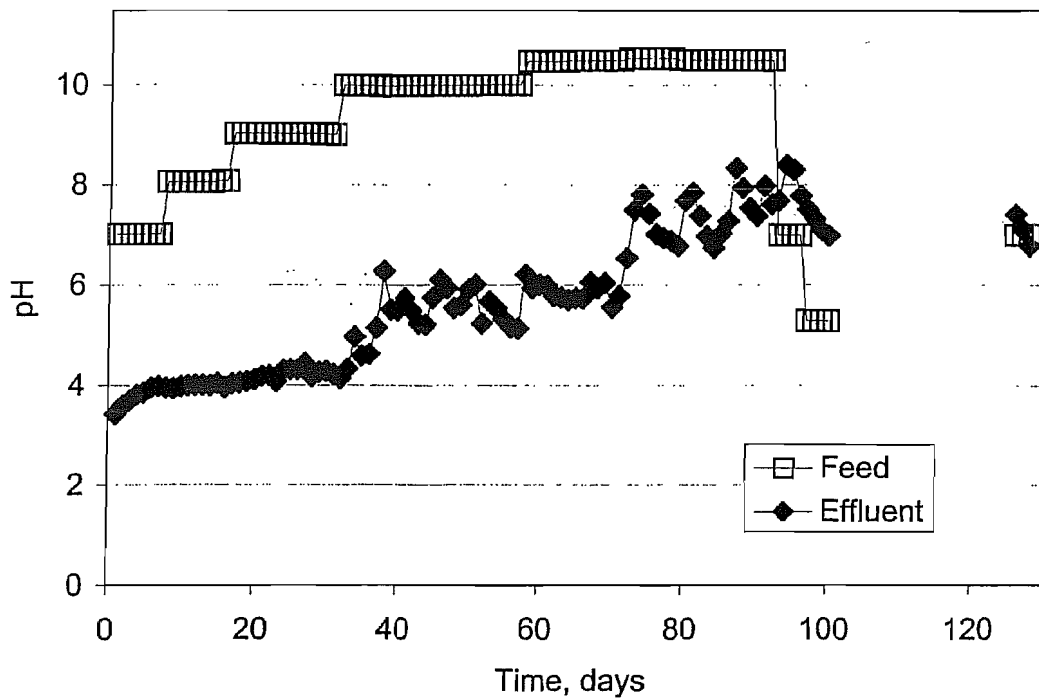


Figure 3.3.3 Feed and Effluent pH for Neutralization Test N3.

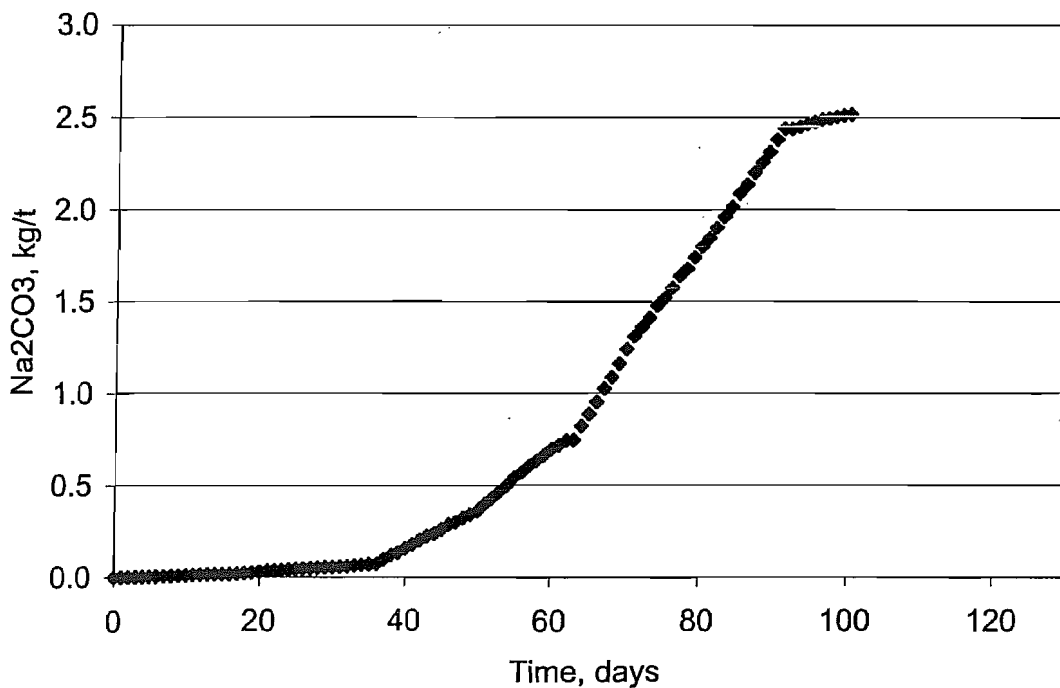


Figure 3.3.4 Sodium Carbonate Addition vs Time for Test N3.

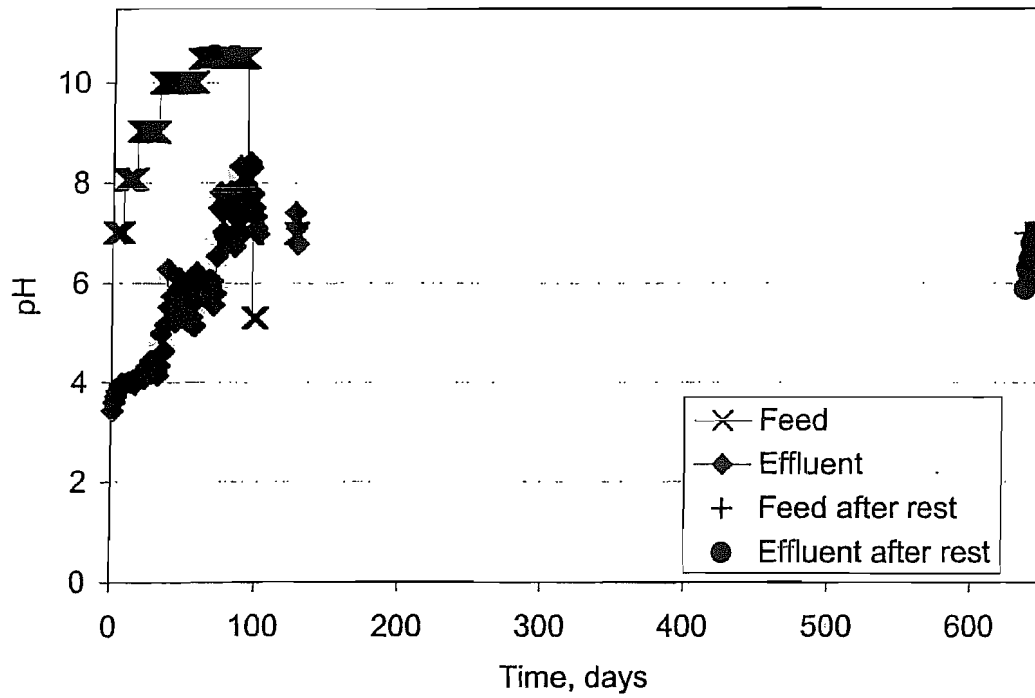


Figure 3.3.5 Test N3 pH Profiles Over Extended Time.

### 3.3.3 Decommissioning of Column C Leach Residue

As discussed in Section 3.2, once copper leaching of the ¾ inch crushed material in Column C had achieved target levels, acid addition to this test was stopped and the pH was allowed to increase naturally. The solution was recirculated continuously with periodic solvent extraction to remove the dissolved copper. This procedure resulted in an increase in PLS pH from a value of 2.28 to 2.79 over a period of about 100 days. Although the feed pH spiked downwards several times over this period, the effluent pH maintained an upward trend showing that even after prolonged leaching with acid, the Carmacks material has a tendency to consume free acid.

After the leach solution had been allowed to drain from the column for one week, a rinsing test was undertaken. The details of this test are included as Appendix 5. Initially, the column was rinsed with a total of 9 liters of water adjusted to pH 7 with minor additions of sodium carbonate. This volume is estimated at about 1/3 pore volume. The effluent from this initial rinse had a pH of 3.43 – 3.12. After this initial rinse the column was irrigated with a solution of sodium carbonate having a pH of 10.5. The pH of the feed and effluent with time is summarized in Figure 3.3.6. Over the test period of 141 days the effluent pH just achieved a value of pH 4. Consideration of the results summarized in Figure 3.3.7 together with those in Figures 3.3.8 and 3.3.9 reveals the reason for a lack of increase in the effluent pH during the initial rinse procedure. The initial rinse with pH 7 solution involved insufficient solution to rinse out the sulphate from the leached ore. The rinsing with pH 10.5 solution involved greater volumes of solution per day and the rate of rinsing of sulphate, aluminum and magnesium from the spent ore increased. However, the sulphate in the effluent over the first 22 days of the test period initially remained near 5 gpl. By day 111 the sulphate concentration in the effluent had decreased to about 1 gpl. The aluminum in the effluent decreased from over 4000 ppm at the start of the rinse to about 200 ppm after 22 days rinsing. It would appear that rinsing with much greater volumes of pH 7 solution than used in the present test is warranted in order to rinse out sulphate and other impurities before any attempt is made to add sodium carbonate for neutralization purposes. Groundwater would be suitable for the initial rinse. The most effective rinse procedure will likely involve a series of pulses so that sulphate is able to diffuse out of the rock between flushes.

Following the initial rinse test, the column was allowed to rest for 425 days prior to additional rinsing being undertaken. The objective of the later rinse was to determine if any rebound to lower pH would be observed. The results summarized in Figure 3.3.10 indicate that after the rest, the effluent pH, even when irrigating with only pH 7 solution, remained at about pH 4. As noted previously, in the absence of intentional neutralization the Carmacks material appears to buffer at about pH 4. The initial effluent to be obtained from the column after the rest contained 13.7 ppm Cu and the concentration dropped to 5.81 ppm after 5 days rinsing. Following the extended rest period the effluent contained 4.2 ppm Al with 22.5 ppm Mg on day 3 and 1.2 ppm with 9.7 ppm Mg on day 7 of the second rinse period.

The apparent reason that column C did not achieve comparable discharge pH to that achieved for column N3, is that sulphate rinsing was not complete before sodium carbonate addition was started. Comparing Figure 3.3.3 with 3.3.6 and Figure 3.3.4 with 3.3.7 supports this conclusion. The carbonate addition made to column C of about 1.2 kg/t would be expected to achieve an effluent pH of just below pH 6 if flushing of impurities including sulphate removal was more complete prior to carbonate addition.

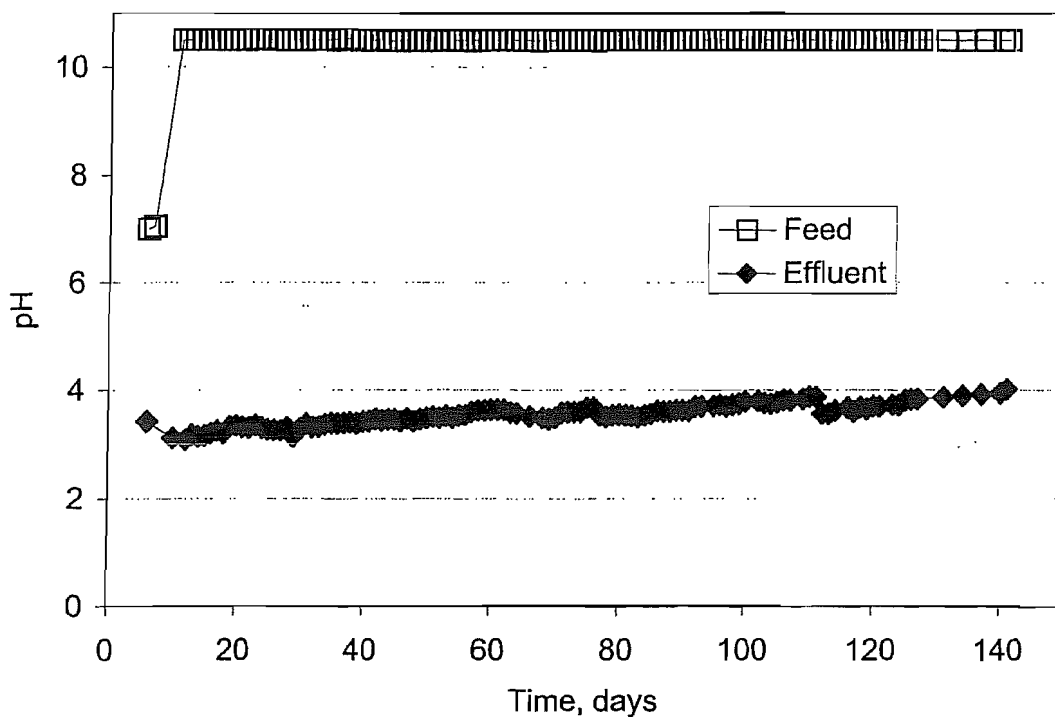


Figure 3.3.6 pH as a Function of Rinse Time for Column C.

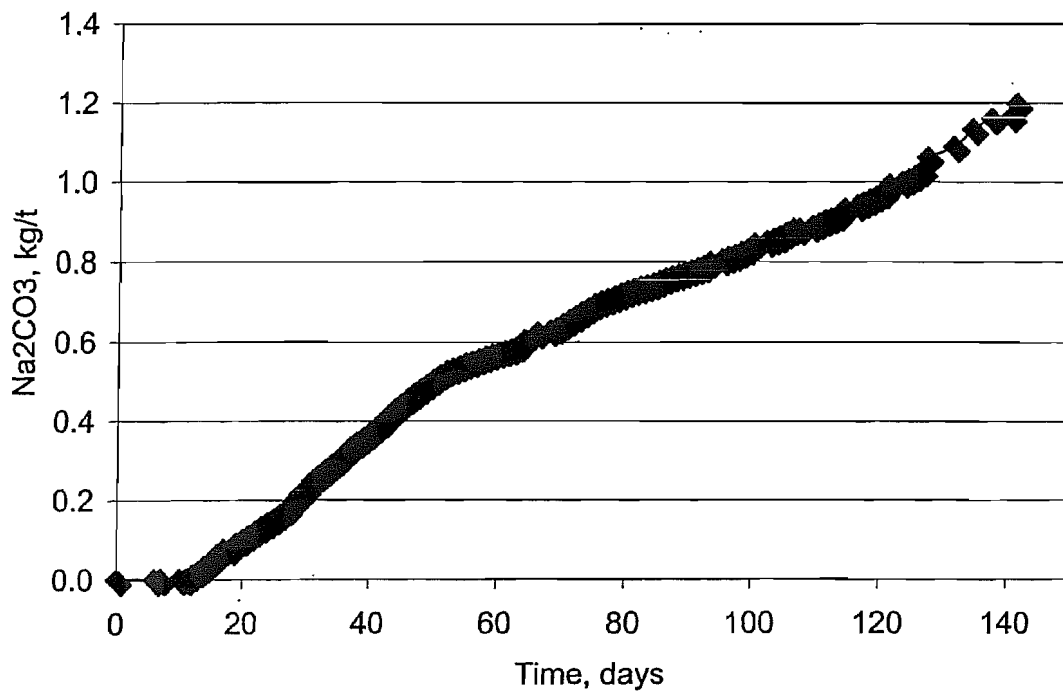


Figure 3.3.7 Sodium Carbonate Consumption for Rinsing of Column C.

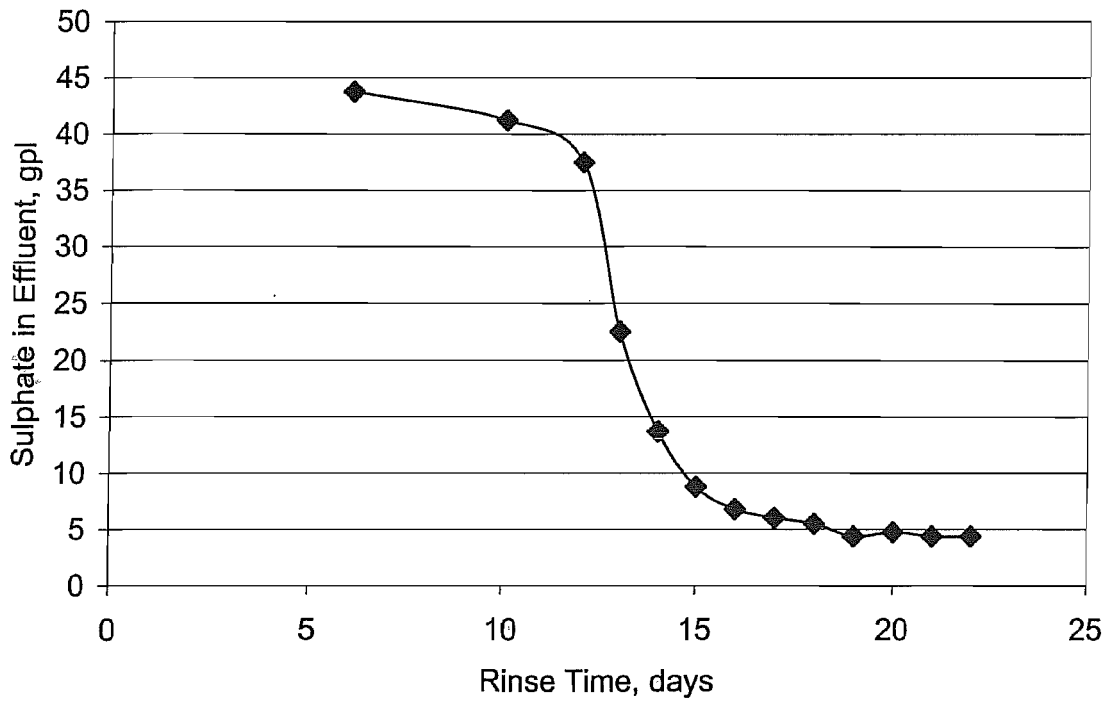


Figure 3.3.8 Sulphate in Effluent as a Function of Rinse Time.

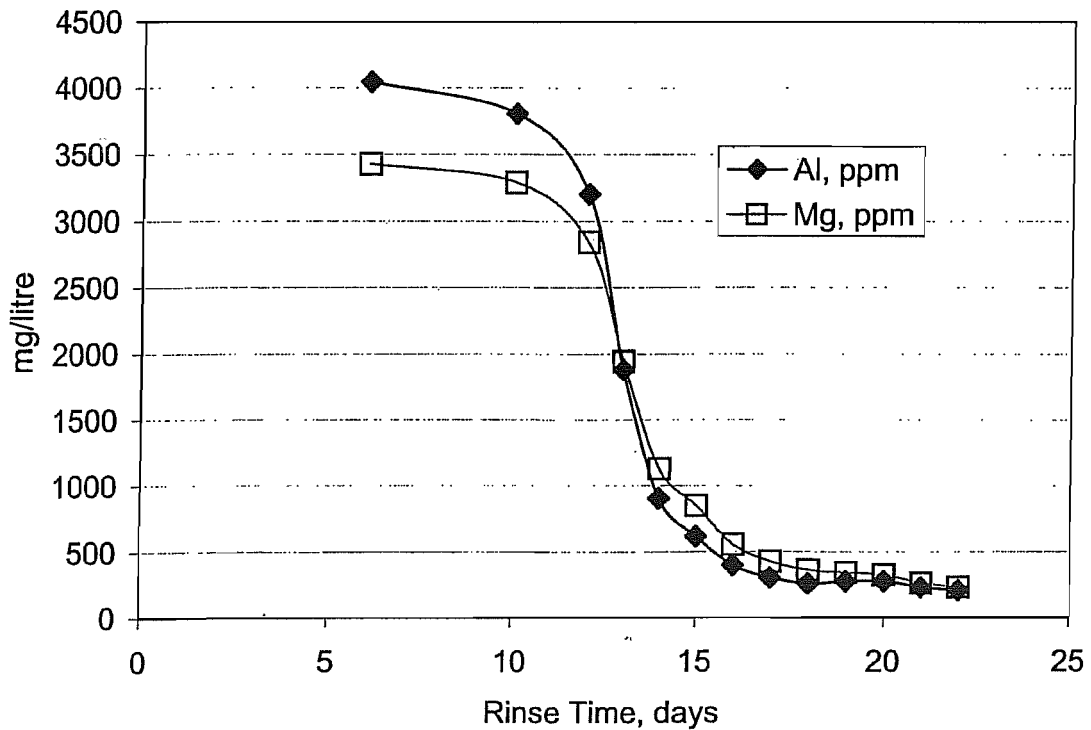


Figure 3.3.9 Al and Mg in Effluent as a Function of Rinse Time.

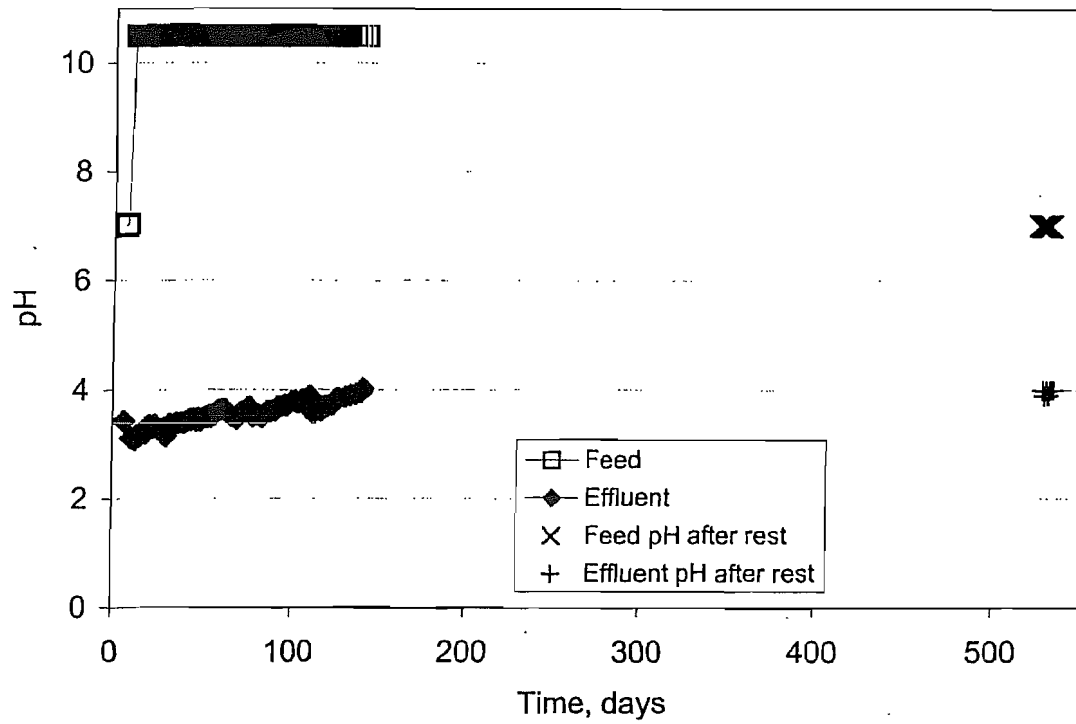


Figure 3.3.10 Feed and Effluent pH After Extended Rest Period.

### 3.3.4 Rinsing of Run of Mine Column

No active neutralization of the column AB contents was undertaken. At the end of the leach period the solution was recirculated through this column and the pH was allowed to increase naturally. The PLS pH at the end of the test was about pH 2.5. After sitting from February 1999 until May 2000, the column was rinsed for 6 days with water having a pH of 7.0. The details of this rinse are included in Appendix 5. The effluent pH from this rinse was 3.0 to 3.2, indicating that some neutralization of the retained solution had occurred. Since leaching of this material had not been taken to its ultimate level the retained acid resulted in additional copper leaching with the day 1 effluent containing 2.4 grams per litter copper. By day 3 of the rinse this concentration had decreased to 1.5 grams per litter. Subsequent to the rinse period the column remained in place for more than six months. At the end of this period the contents were emptied and it was found that the bottom section had become cemented by iron precipitates.



## **Appendix 1**

### **Sample Collection Report**

**WESTERN COPPER HOLDINGS LIMITED**

**CARMACKS, YUKON TERRITORY**

**CARMACKS COPPER PROJECT**

Project No. 8555-25

**RUN-OF-MINE BULK SAMPLE COLLECTION**

**KILBORN ENGINEERING PACIFIC LTD.  
400 - 1380 Burrard Street  
Vancouver, B.C.  
V6Z 2B7**

**JANUARY 1998**

**WESTERN COPPER HOLDINGS LIMITED  
CARMACKS COPPER PROJECT  
RUN-OF-MINE BULK SAMPLE COLLECTION**

**TABLE OF CONTENTS**

<u><b>SECTION</b></u>	<u><b>PAGE</b></u>
1.0 INTRODUCTION.....	1-1
1.1 Scope of Work .....	1-1
1.2 Geology .....	1-1
1.3 Disclaimer .....	1-2
2.0 PROCEDURES.....	2-1
2.1 Identification of Trenches and Mineralized Zones .....	2-1
2.2 Equipment .....	2-2
2.3 Trench Development .....	2-3
2.4 Blasting.....	2-6
2.5 Bulk Sampling.....	2-8
2.5.1 Sampling .....	2-8
2.5.2 Mixing and Blending .....	2-9
2.6 Channel-Chip Samples.....	2-10
3.0 PROGRAM SCHEDULE .....	3-1
4.0 COMMENTS .....	4-1
5.0 REFERENCES .....	5-1

**APPENDICES**

APPENDIX A	FIELD REPORTS
APPENDIX B	SAMPLE DESCRIPTION MEMO
APPENDIX C	DRAWINGS

## **1.0 INTRODUCTION**

## 1.0 INTRODUCTION

### 1.1 SCOPE OF WORK

Western Copper (Western) requested Kilborn to obtain a bulk composite sample from the Carmacks Copper Project located at Williams Creek near Carmacks, Yukon (Kilborn letter March 10, 1997). The 18,000 kg bulk sample is to be used for copper heap leach metallurgical testing for run-of-mine ore.

The bulk sample was to be taken from four trenches and blended at the proportions as listed in Table 1.1-1. The material taken from the each trench was designed to obtain a representative sample from the near surface mineralization at the Carmacks Copper Mine Zone 1 deposit. Kilborn has relied on Max Holtby's detailed knowledge of the deposit to select the trenches to be sampled and determined the proportion of sample to be taken from each trench. Trench locations are shown on Drawing No. 1.1-1.

**Table 1.1-1  
Trench Sample Proportions**

<b>Trench</b>	<b>Proportion</b>
T91-22	30
T91-20	30
T92-68	20
T91-18	20

### 1.2 GEOLOGY

The Carmacks Copper deposit occurs as a gneissic roof pendant within the Upper Triassic Granite Mountain Batholith. This Batholith is principally of granodioritic composition including rocks of diorite, tonalite and quartz monzonite composition. The mineralized roof pendant consist of a feldspathic-biotite-hornblende-quartz gniesses. The gniesses are generally quartz deficient. Gneiss compositions vary from basaltic to

granodioritic. The gneisses are well foliated with coarse to fine grained textures. Aplite and pegmatite dykes intrude both the batholithic rocks and the gneisses. Rocks of this deposited are oxidized to a depth of 260 m (Holtby and McNoughton, 1993).

Copper and gold mineralization occurs in the Carmacks Copper deposit. Holtby and McNoughton (1993) indicate that 86% of the deposit consists of secondary copper minerals mainly malachite, cuprite, azurite and tenorite. Unoxidized mineralization consist of bornite and chalcopyrite. Gold mineralization is commonly associated with limonite but is also associated with malachite, plagioclase, chlorite and quartz. Gold occurs as native grains rarely greater than 5  $\mu\text{m}$  in size.

The geological resources for Zone 1 of the deposit has been estimated at 20.7 mt at a grade of 0.98% copper, using a cut off grade of 0.20% copper (Kilborn, 1994).

### 1.3 DISCLAIMER

Kilborn has followed standard professional procedures in preparing the contents of this report, but it is based in part on details, information and assumptions provided by others. Kilborn, therefore cannot guarantee the correctness of all the information, but to the extent of its investigation and within the scope of the work delegated to it, Kilborn believes that the report is substantially correct.

## **2.0 PROCEDURES**

## 2.0 PROCEDURES

### 2.1 IDENTIFICATION OF TRENCHES AND MINERALIZED ZONES

Trenches 91-18, 92-68, 91-20 and 91-22 were previously marked by Lamerton & Associates. Kilborn confirmed trench locations by locating nearby drill holes and other marked trenches.

Discussions with Silver Standard (Holtby, per. com) indicated that the footwall contact in each trench could be readily located. It was suggested that the hanging wall contact could be located approximately by measuring from the foot wall. The distance to be measured could be obtained from trench plans or sections.

Kilborn was readily able to identify footwall contacts in each of the trenches. Hangingwall contacts were determined by measuring distances from trench footwall contact and by identifying the geological contact based on examination of other trenches where hangingwall contacts were marked with pickets. A comparison of mineralized trench intersections from Holtby and McNaughton (1993) sections and Kilborn's determinations is shown in Table 2.1-1. The distance between footwall and hangingwall determined by Kilborn was from 0.3 to 9.1 m shorter than distances determined from sections. These differences are believed to be within the accuracy of the 1" to 100' map scale of the sections used to determine the distances.



**Table 2.1-1  
Comparison of Trench Mineralization Distances**

Trench	Length of Mineralized Intersection		Difference
	Sections*	Sampled	
91-18	39.6	39.3	0.3
92-68	73.1	66.1	7
91-20	36.6	33.9	2.7
91-22	33	23.9	9.1

\* Mineralization distances from cross-sections in appendix 1 of the Holtby and McNaughton (1993) report.

## 2.2 EQUIPMENT

The following equipment was used:

- 1 Cat 225LC excavator;
- 1 Cat D7G bulldozer;
- 2 Dump trucks, tandem axle, 29 t gvwt;
- 1 Air track percussion drill;
- 1 450 compressor for drill; and
- 1 Flat deck truck with Hiab crane.

As a contractor based in Carmacks was not available, a contractor from Whitehorse was chosen. The contractor chosen to supply the equipment and personnel was H. Coyne & Sons Ltd. The contractor set up a camp for equipment operators beside the Freegold Road on the access road to Williams Creek. The camp was established to reduce travel time for equipment operators to and from the site.

The excavator and bulldozer were used to deepen and widen trenches. Dump trucks were necessary to move excavated ground from trench 91-20 as the trench was too deep to pile ground on trench sides. Ground removed from trench 91-20 was stock piled in a nearby pit. Care was taken to pile waste (footwall and hangingwall rock) material separately. The dump trucks were also used to move the bulk sample out of trenches to mixing sites.

Additionally a Hiab-crane truck was used for weighing bulk sample bags, loading bulk sample bags and loading equipment.

### 2.3 TRENCH DEVELOPMENT

Trenches selected by Western were widened to approximately 6 m to allow rotation of the excavator for loading of bulk sample on to dump trucks. Initially existing trenches were about 2 to 4 m deep. Generally these trenches were deepened an additional 2 to 3 m to ensure that the ground was below previous blast fragmentation (Figures 2.3-1, 2.3-2, 2.3-3). Trench 91-20 was deepened from the original 3-4 m from surface to 9 to 11 m from the surface (Figure 2.3-4). This trench was deepened to confirm an unexpected footwall contact that paralleled the trench direction for 15 to 20 m. This contact was excavated to ensure that sampling began at the proper contact. The trench was deepened to a depth where rocks were more competent and therefore more representative of run-of-mine material.

The side walls of trenches, in particular trench 91-20, were benched to prevent rock falls into the trench and to allow clearance for the excavator bucket. Ground removed from trench 91-20 was stockpiled in a nearby pit. Care was taken to pile waste (hangingwall and footwall rock) material separately.



Figure 2.3-1 Trench 91-18 after excavation,  
before blasting.



Figure 2.3-2 Trench 92-68 after bulk sampling.



Figure 2.3-3 Trench 91-22. After bulk sampling.

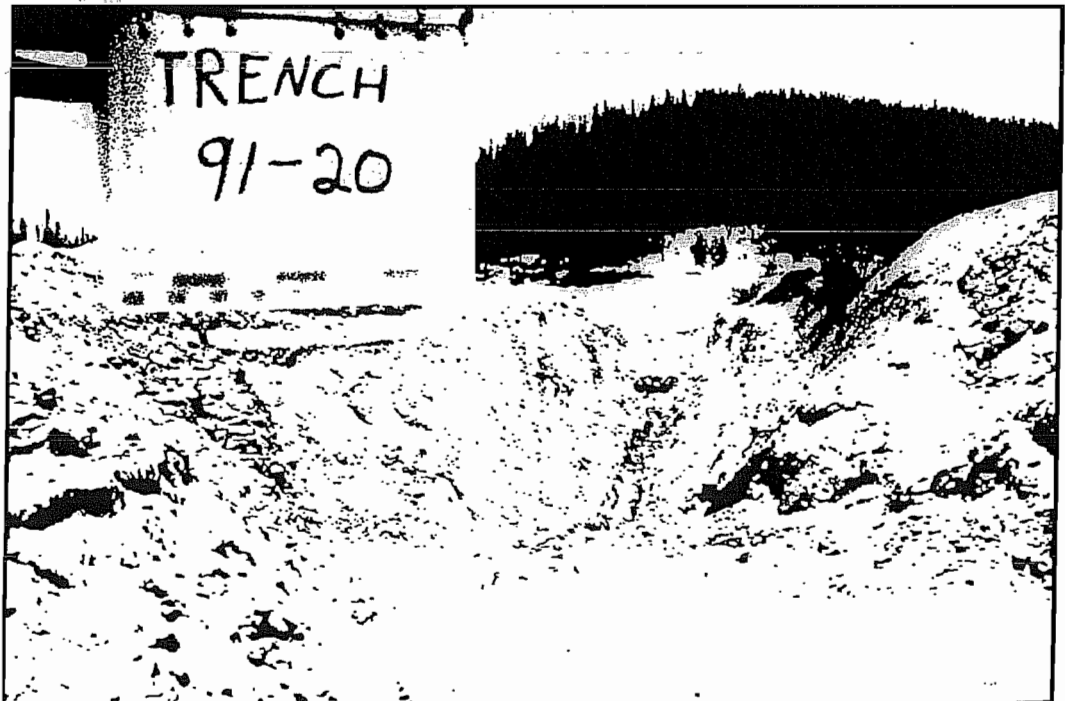


Figure 2.3-4 Trench 92-20. After bulk sampling.

## 2.4 BLASTING

The bulk sample was obtained by drop cut blasting of the trench floor. The blast was designed to ensure that blasted material was slightly wider than the excavator bucket (.95 m wide, 1.2 m deep). Blast holes were initially drilled by a Mini Air Track drill which was replaced by an Air Track drill because of break downs. Blast holes were drilled vertically at a depth of 1.5 m and a diameter of 0.05 m. A three blast hole pattern was drilled centered across the width of the trench at a hole spacing of 0.75 m. This three hole pattern was spaced along the length of the trench at 1.5 m.

Inclined blasted hole fans were drilled over the eastern end of trench 91-18 over a hill of competent rock.

In areas where the rock was highly friable no blast holes were drilled. In trench 91-68 an intervening 4.5 m section of granodiorite waste rock was not blasted.

92-68

Blast holes were charged with a half stick of Unimax Yellow TS (32 mm x 40 mm) powder and loaded 0.6 m from the collar with anfo. Holes were then stemmed to the collar with drill cuttings and fine grained ground. Non-electric detonators with 25 ms delays were used. The blasting pattern schematic is shown in Figure 2.4-1.

## 2.5 BULK SAMPLING

### 2.5.1 Sampling

Each trench was sampled using the excavator. A continuous sample about 1.0 m wide and 1.0 m deep was taken from each trench. The sample was taken from the footwall contact to the hangingwall contact and in the case of trench 91-68 the intervening granodiorite was excluded. Each bucket near these contacts was inspected to ensure that no waste rock was included.

The following steps were taken to ensure that contamination from trench side walls was excluded from the bulk sample:

- The top 0.15 m to 0.2 m from blasted material was scraped off before sampling to ensure that no debris from the trench walls was included;
- Material that fell into the trench from sidewalls was moved aside before the next bucket of sample was taken. This ensured no contamination from sidewall material. Unblasted very friable sections of the trenches were sampled in the same manner.

Each excavator bucket of sample was loaded directly into a dump truck. The bulk sample was then transported to and dumped at a mixing site nearby each trench.

\* In addition to the bulk sample weathered previously leached ore from trench 91-20 was collected. Approximately 800 lbs of this material was placed separately in a bulk sample bag.

### 2.5.2 Mixing and Blending

The bulk sample from each trench was rolled, moved and repiled for one-half hour. A new pile was made from this material by moving the material bucket load by bucket load on to a new pile. Care was taken to keep the new pile in a regularly shaped cone, this process is called coning. A skin of 3 to 5 cm of bulk sample was left on the mixing pad surface to reduce dilution from introduction of ground material.

The mass of the bulk sample was estimated from the volume of sample moved out of the trench multiplied by a density of 2.68 (ore density (Holtby and McNaughton, 1993)). The masses of bulk samples from each trench were reduced to masses which when combined would maintain the mixing ratio suggested by Western/Silver Standard:

<u>Trench</u>	<u>Ratio</u>
91-18	20%
92-68	20%
91-20	30%
91-22	30%

The estimated masses were obtained by physically splitting the pile in quarters or multiples of quarters. Each successive split was re-coned for further splitting. The final mass from each trench was loaded into one dump truck and dumped on to a new mixing pad. This sample containing a proportion from other trenches was then mixed by rolling, moving and repiling for about half an hour. The bulk sample was then coned twice to ensure good mixing. This bulk sample was then loaded into bulk sample bags with two 3 mil thick polyethylene liners. The plastic liners ensure that fines were not lost during transport and to ensure that the sample did not get wet during transport. Each bag was then weighed to ensure that approximately one tonne of material was loaded in each bag.

Despite a calculated bulk sample excess of 5 tonnes a shortfall of three tonnes of bulk sample was discovered upon weighing bulk sample material. This shortfall was made up by blending and mixing excess sample material from each trench. The same relative proportions from each trench was maintained.

Bulk sample bags were sealed, loaded onto the dump trucks and transported to Whitehorse. The bulk samples were then trans-shipped to Vancouver by truck.

## 2.6 CHANNEL-CHIP SAMPLES

Two meter channel-chip samples were taken along each trench. A channel-chip sample is a continuous sample of rock similar to a channel sample. The channel-chip sample is different from a channel sample because the width and depth of the channel-chip sample is roughly the same. In a channel sample the width and depth of the sample is exactly uniform as equipment such as a diamond saw would be used to cut the sample from rock. Kilborn channel-chip sampled the side walls of the trenches. All samples consisted in-situ blasted rock and in place unblasted rock. Only the portions of the trench used in the bulk sample were channel-chip sampled. Geological notes on the channel-chip samples are attached.



## TRENCH 92-68 CHANNEL-CHIP SAMPLING

Samples taken from footwall contact to hanging wall.  
Sample start marked with picket.

SAMPLE NUMBER	FROM [m]	TO [m]	DESCRIPTION
588820	0.0	2.0	Biotite+feldspar gneiss, fine to medium grained, rusty brown, mod. weathered, mod foliated, tr-0.5% malachite, 2-5% ?tenorite+lim., minor granodiorite dyke
588821	2.0	4.0	Bt+fspar gneiss, f-m. gr. rusty brown, mod. weathered, 5% ?tenorite+lim., 1-2% malachite, minor pegmatite dykes
588822	4.0	6.0	Bt+fspar gneiss, f-m. gr. rusty brown, mod. weathered, 5% ?tenorite+lim., 1-2% malachite, minor pegmatite dykes
588823	6.0	8.0	Bt+fspar gneiss, f-m. gr. rusty brown, mod. weathered, 5% ?tenorite+lim., 1-2% malachite, minor pegmatite dykes
588824	8.0	10.0	Bt+fspar gneiss, f-m. gr. rusty brown, mod. weathered, 5% ?tenorite+lim., 1-3% malachite, some pegmatite dykes
588825	10.0	12.0	Bt+fspar gneiss, m-c. gr. rusty brown, mod. weathered, very friable, 3-6% ?tenorite+lim., 1-2% malachite, tr. azurite
588826	12.0	14.0	Bt+fspar gneiss, f-c. gr. rusty brown, mod. weathered, very friable, 3-6% ?tenorite+lim., 1-2% malachite, tr. azurite
588827	14.0	16.0	Bt+fspar gneiss, f-c. gr. rusty brown, mod. weathered, very friable, 3-6% ?tenorite+lim., 1-2% malachite, tr. azurite
588828	16.0	18.0	Bt+fspar gneiss, f-c. gr. rusty brown, mod. weathered, friable, 3-6% ?tenorite+lim., 1-3% malachite, mod. competent
588829	18.0	20.0	Bt+fspar gneiss, f-m. gr. rusty brown-yellow, mod. weathered, friable, 3-6% ?tenorite+lim., 1-2% malachite, mod. competent
588830	20.0	22.0	Bt+fspar gneiss, f-m. gr. rusty brown-yellow, mod. weathered, friable, 3-6% ?tenorite+lim., 1-2% malachite, mod. competent
588831	22.0	24.0	Bt+fspar gneiss, f-m. gr. rusty brown, mod. weathered, friable, 3-6% ?tenorite+lim., 2-3% malachite, mod. competent
588832	24.0	26.9	Bt+fspar gneiss, f-m. gr. rusty brown, mod-v. weathered, mod-very friable, 3-6% ?tenorite+lim., 2-3% malachite, minor granodiorite dykes, contact with granodiorite unit
588833	31.4	34.0	Bt+fspar gneiss, f-m. gr. rusty brown-yellow, mod-v. weathered, mod-very friable, 5% ?tenorite+lim., 1-2% malachite, contact with granodiorite unit
588834	34.0	36.0	Bt+fspar gneiss, f-m. gr. rusty brown-yellow, mod-v. weathered, mod-very friable, 5% ?tenorite+lim., 2-3% malachite, contact with granodiorite unit
588835	36.0	38.0	Bt+fspar gneiss, f-m. gr. rusty brown-yellow, mod-v. weathered, mod-very friable, 2-5% ?tenorite+lim., 2-4% malachite
588836	38.0	40.0	Bt+fspar gneiss, f-m. gr. rusty brown-yellow, mod-v. weathered, mod-very friable, 2-5% ?tenorite+lim., 2-3% malachite
588837	40.0	42.0	Bt+fspar gneiss, m-c. gr. rusty brown, mod-v. weathered, mod friable to mod competent, 5% ?tenorite+lim., 1-2% malachite
588838	42.0	44.0	Bt+fspar gneiss, m-c. gr. rusty brown, mod. weathered, mod competent, 5% ?tenorite+lim., 1-2% malachite
588839	44.0	46.0	Bt+fspar gneiss, f-c. gr. rusty brown-yellow, mod. weathered, 7% ?tenorite+lim., 2-3% malachite
588840	46.0	48.0	Bt+fspar gneiss, f-c. gr. rusty brown-yellow, mod. weathered, 4-50% ?tenorite+lim., 2-3% malachite
588841	48.0	50.0	Bt+fspar gneiss, f-c. gr. rusty brown-yellow, mod. weathered, 3-7% ?tenorite+lim., 2-3% malachite
588842	50.0	52.0	Bt+fspar gneiss, f-c. gr. rusty brown-yellow, mod-v. weathered, 3-7% ?tenorite+lim., 2-3% malachite
588843	52.0	54.0	Bt+fspar gneiss, c. gr. rusty brown-yellow, v. weathered, mod-v. friable, 5-6% ?tenorite+lim., 3-4% malachite
588844	54.0	56.0	Bt+fspar gneiss, c. gr. rusty brown-yellow, v. weathered, mod-v. friable, 5-6% ?tenorite+lim., 3-4% malachite
588845	56.0	58.0	Bt+fspar gneiss, c. gr. rusty brown-yellow, v. weathered, mod-v. friable, 5-6% ?tenorite+lim., 3-4% malachite
588846	58.0	60.0	Bt+fspar gneiss, c. gr. rusty brown-yellow, v. weathered, mod-v. friable, 4-6% ?tenorite+lim., 2-3% malachite, 1-2% azurite
588847	60.0	62.0	Bt+fspar gneiss, c. gr. rusty brown-yellow, v. weathered, mod-v. friable, 4-6% ?tenorite+lim., 2-4% malachite, 2-3% azurite, transitional change from gneissic to massive igneous texture
588848	62.0	64.0	Bt+fspar gneiss, c. gr. rusty brown-yellow, v. weathered, mod-v. friable, 4-6% ?tenorite+lim., 2-4% malachite, 2-3% azurite, transitional change from gneissic to massive igneous texture

End of Sampling

## TRENCH 91-22 CHANNEL-CHIP SAMPLING

Samples taken from footwall contact to hangingwall.  
Sample start marked with picket.

SAMPLE NUMBER	FROM [m]	TO [m]	DESCRIPTION
586568	0.0	2.0	Biotite-feldspar gneiss, fine to medium grained, rusty brown-yellow, v. oxidized, mod. friable, 2-3% ?tenorite+lim, 3-4% malachite 1-2% azurite,
586269	2.0	4.0	Bt+fspar gneiss, f-c. grained, rusty brown-yellow, v. oxidized, blocky, 3-6% malachite, 2-3 ?tenorite+lim,
586270	4.0	6.0	Bt+fspar gneiss, f-c. grained, rusty brown-yellow, v. oxidized, blocky, 2-3% malachite, 2-3 ?tenorite+lim, minor pegmatite dyke
586271	6.0	8.0	Bt+fspar gneiss, m-c. grained, rusty brown-yellow, v. oxidized, blocky, 3-4% malachite, 2-3 ?tenorite+lim, 2-3% azurite
586272	8.0	10.0	Bt+fspar gneiss, m-c. grained, rusty brown-yellow, v. oxidized, mod. friable, 2-3% malachite, 2-4% ?tenorite+lim, tr-13% azurite
586273	10.0	12.0	Bt+fspar gneiss, m-c. grained, rusty brown, v. oxidized, v. friable, 2-3% malachite, 2-4% ?tenorite+lim, tr-13% azurite, clayey-texture, intense hematite staining
586274	12.0	14.0	Bt+fspar gneiss, m-c. grained, rusty brown, v. oxidized, v. friable, 2-3% malachite, 2-4% ?tenorite+lim, tr-13% azurite,
586275	14.0	16.0	clayey-texture, intense hematite staining Bt+fspar gneiss, m-c. grained, rusty brown, v. oxidized, v. friable, 2-3% malachite, 4-6% ?tenorite+lim, tr % azurite, mod. hematite staining
586276	16.0	18.0	Bt+fspar gneiss, m-c. grained, rusty brown, v. oxidized, v. friable, 2-3% malachite, 4-6% ?tenorite+lim, tr % azurite, mod. hematite staining
586277	18.0	20.0	Bt+fspar gneiss, m-c. grained, rusty brown, v. oxidized, v. friable, 2-3% malachite, 3-5% ?tenorite+lim, 1-2% azurite, mod. hematite staining
586278	20.0	22.0	Bt+fspar gneiss, f-c. grained, rusty brown, v. oxidized, blocky wk. friable, 1-3% malachite, 2-3% ?tenorite+lim, 1% azurite
586279	22.0	23.5	Bt+fspar gneiss, f-c. grained, rusty brown, v. oxidized, blocky mod.. friable, mod. hematization, increasing abundance of gnodiorite dykes, hanging wall contact.
			End of Sampling

## TRENCH 91-20 CHANNEL-CHIP SAMPLING

Samples taken from footwall contact to hangingwall.

Sample start marked with picket.

SAMPLE NUMBER	FROM [m]	TO [m]	DESCRIPTION
586251	0.0	2.0	Biotite+felspar gneiss, fine grained, rusty brown-yellow, v. oxidized, very friable, 2-5% ?tenorite+lim, well foliated
586252	2.0	4.0	Bt+fspar gneiss, fine grained, rusty brown-yellow, v. oxidized, very friable, 2-5% ?tenorite+lim, locally 2-3%chalcocite
582563	4.0	6.0	Bt+fspar gneiss, fine grained, rusty brown-yellow, v. oxidized, mod. friable, 2-5% ?tenorite+lim, tr-2%malachite
582654	6.0	8.0	Bt+fspar gneiss, fine grained, rusty brown-yellow, v. oxidized, weakly friableto blocky, 2-53% ?tenorite+lim, tr-2%malachite
586255	8.0	10.0	Bt+fspar gneiss, fine grained, rusty brown-yellow, v. oxidized, blocky, mod fractured, 10-13% ?tenorite+lim, 1-3%malachite,
586256	10.0	12.0	Bt+fspar gneiss, fine grained, rusty brown-yellow, v. oxidized, blocky, mod fractured, 3-5% ?tenorite+lim, 3-54%malachite, minor granodiorite dykes
586257	12.0	14.0	Bt+fspar gneiss, fine grained, rusty brown-yellow, v. oxidized, blocky, mod fractured, 3-5% ?tenorite+lim, 2-4%malachite, pegmatite dykes 20cm wide
586258	14.0	16.0	Bt+fspar gneiss, f-c. grained, rusty brown-yellow, v. oxidized, mod-v. friable, 2-4%malachite, 3-5 ?tenorite+lim,
586259	16.0	18.0	Bt+fspar gneiss, f. grained, rusty brown-yellow, v. oxidized, mod-v. friable, 1-3%malachite, 1-3 ?tenorite+lim,
586260	18.0	20.0	Bt+fspar gneiss, f. grained, rusty brown-yellow, v. oxidized, mod-v. friable, 1-3%malachite, 1-3 ?tenorite+lim,
586261	20.0	22.0	Bt+fspar gneiss, f-c. grained, rusty brown-yellow, v. oxidized, v. friable, 1-3%malachite, 1-3 ?tenorite+lim,
586262	22.0	24.0	Bt+fspar gneiss, f-c. grained, rusty brown-yellow, v. oxidized, mod-v. friable, clayey texture, 2-4%malachite, 6-7% ?tenorite+lim,
586263	24.0	26.0	Bt+fspar gneiss, f. grained, rusty brown-yellow, v. oxidized, v. friable, clayey texture, 1-3%malachite, 3-5% ?tenorite+lim
586264	26.0	28.0	Bt+fspar gneiss, f. grained, rusty brown-yellow, v. oxidized, v. friable, clayey texture, 1-3%malachite, 3-5% ?tenorite+lim
586265	28.0	30.0	Bt+fspar gneiss, f-c. grained, rusty brown-yellow, v. oxidized, v. friable, clayey texture, 2-3%malachite, 3-5% ?tenorite+lim
586266	30.0	32.0	Bt+fspar gneiss, c. grained, rusty brown-yellow, v. oxidized, mod. friable, 2-3%malachite, 3-5% ?tenorite+lim.
586267	32.0	33.9	Bt+fspar gneiss, c. grained, rusty brown-yellow, v. oxidized, mod. friable, 3-4%malachite, 2-3% ?tenorite+lim, contact with granodiorite hanging wall
			End of Sampling

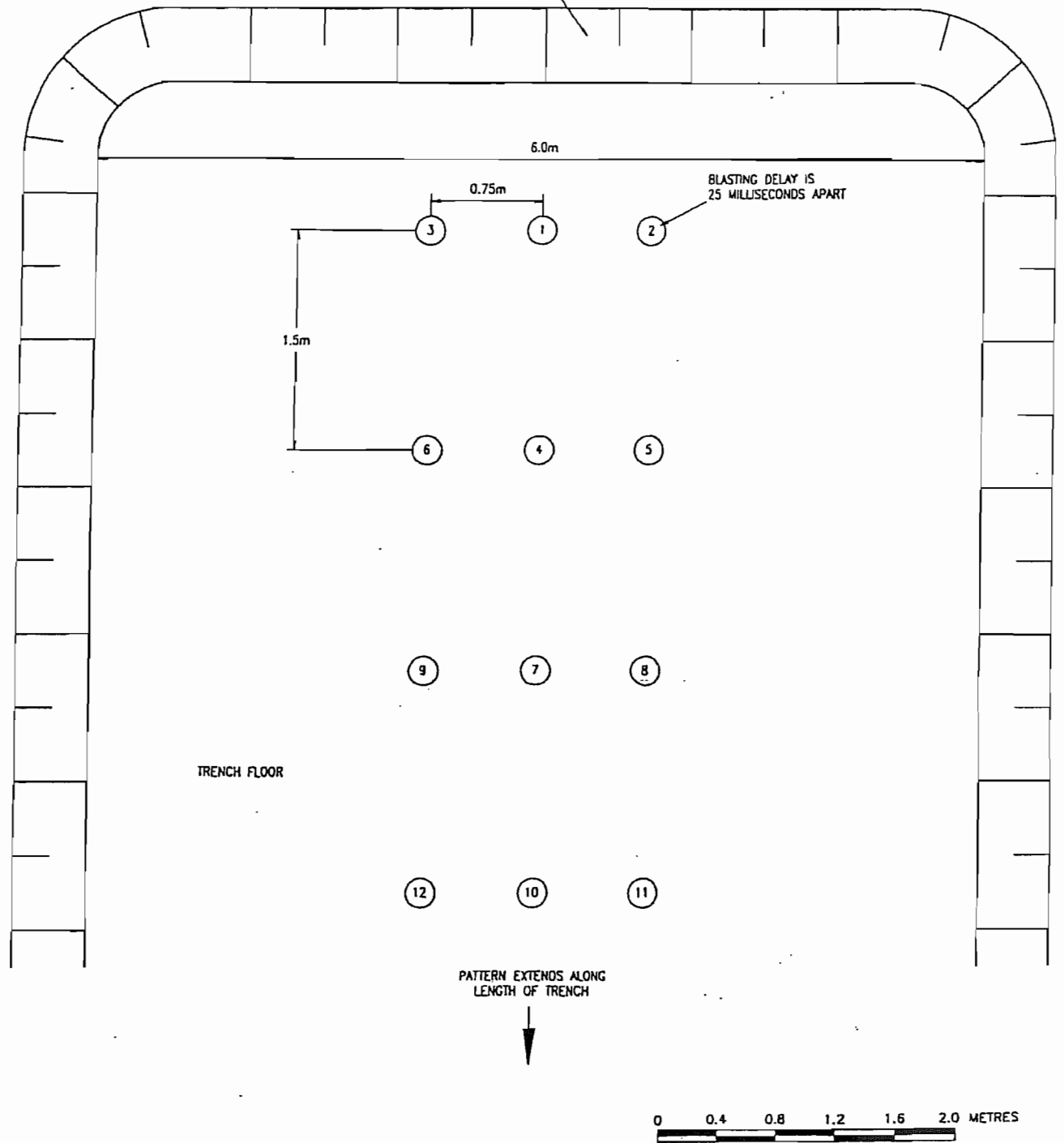
## TRENCH 91-18 CHANNEL-CHIP SAMPLING

Samples taken from footwall contact to hanging wall.

Sample start marked with picket.


SAMPLE NUMBER	FROM [m]	TO [m]	DESCRIPTION
588801	0.0	2.0	Biotite+felspar gneiss, fine grained, rusty brown, mod. weathered, foliated at 128o 84oNE, tr-1% malachite, 2-5% ?tenorite+lim.
588802	2.0	4.0	Bt+fspar gneiss, f. gr. rusty brown, mod. weathered, 2-5% ?tenorite+lim., minor granodiorite dykes
588803	4.0	6.0	Bt+fspar gneiss, f. gr., rusty brown-yellow, mod. to intense clay weathering, 2-5% ?tenorite+lim.
588804	6.0	8.0	Bt+fspar gneiss, f. gr., rusty brown, very friable, tr-1% malachite (locally 2-3%), 5% ?tenorite+lim.
588805	8.0	10.0	Bt+fspar gneiss, m-f gr., rusty brown, mod-very friable, 1-2% malachite, 5% ?tenorite+lim., minor pegmatite veins
588806	10.0	12.0	Bt+fspar gneiss, m gr., rusty brown, mod. friable, tr-1% malachite, 2-3% ?tenorite+lim., biotite bands
588807	12.0	14.0	Bt+fspar gneiss, m gr., rusty brown-yellow, friable, highly fractured, tr-1% malachite, 2-3% ?tenorite+lim., biotite 20-30%
588808	14.0	16.0	Bt+fspar gneiss, m gr., intense to mod. weathering, rusty brown-yellow, v. friable, clayey to bt sand texture, 5-10% ?tenorite+lim., tr-1% malachite,
588809	16.0	18.0	Bt+fspar gneiss, f-m gr., mod. weathering, rusty brown-yellow, v. friable, clayey to bt sand texture, 2-5% ?tenorite+lim., 1-2% malachite
588810	18.0	20.0	Bt+fspar gneiss, f-m gr., mod. weathering, rusty brown-yellow, v. friable, clayey to bt sand texture, 2-5% ?tenorite+lim., 1-2% malachite
588811	20.0	22.0	Bt+fspar gneiss, f-m gr., mod. weathering, rusty brown-yellow, v. friable, clayey to bt sand texture, 2-5% ?tenorite+lim., 1-2% malachite
588812	22.0	24.0	Bt+fspar gneiss, f-m gr., mod. weathering, rusty brown, mod. competent and v. friable sections, 2-5% ?tenorite+lim., 2-3% malachite, minor pegmatite (?qz veins) dykes
588813	24.0	26.0	Bt+fspar gneiss, c-f gr., mod. weathering, rusty brown, mod. fractured, 5% ?tenorite+lim., 2-4% malachite
588814	26.0	28.0	Bt+fspar gneiss, f gr., mod. weathering, rusty brown, mod. foliated, mod. competent fractured, 5% ?tenorite+lim., 2-3% malachite (locally 4%)
588815	28.0	30.0	Bt+fspar gneiss, f gr., mod. weathering, grey-brown, mod. foliated, mod. competent, fractured, 2% ?tenorite+lim., 3-4% malachite, minor pegmatite
588816	30.0	32.0	Bt+fspar gneiss, c gr., wk weathered, grey-black, mod. foliated, competent, mod. fractured, 3-4% malachite, forms resistant outcrop, mineralization occurs along fracture planes, weakly foliated
588817	32.0	34.0	Bt+fspar gneiss, c gr., wk weathered, grey-black, mod. foliated, competent, mod. fractured, 3-5% malachite, resistant, wk foliated
588818	34.0	36.0	Bt+fspar gneiss, c gr., wk weathered, grey-black, mod. foliated, competent, mod. fractured, 3-5% malachite, resistant, wk. foliated
588819	36.0	39.3	Bt+fspar gneiss, c gr., wk weathered, grey-black, mod. foliated, competent, mod. fractured, 2-3% malachite, resistant, wk. foliated
			End of Sampling

TRENCH WALL (NOT TO SCALE)



SHEET NUMBER

SECTION: MINING	CLIENT: WESTERN COPPER HOLDINGS LTD.
SCALE AS SHOWN	DATE
DESIGNED: D.K.	23OCT97
DRAWN: T.M.	23OCT97
CHECKED: D.K.	23OCT97
APPROVED: D.K.	23OCT97
LOCATION: CARMARKS, YUKON TERRITORY	
THE ENGINEERING DATA ON THIS DRAWING IS SOLELY FOR THE PURPOSE AND PROJECT FOR WHICH THIS DRAWING IS ISSUED	



**KILBORN  
SNC-LAVALIN**

**CARMARKS COPPER PROJECT**

FIG 2.4-1  
TRENCH BLASTING SCHEMATIC

FIG 2.4-1.DWG

EQUIP/UNIT/LOOP No.

---

PROJECT No. DIVISION No.

8555 25

### **3.0 PROGRAM SCHEDULE**

### 3.0 PROGRAM SCHEDULE

Removal of the bulk sample commenced on September 23, 1997 and was completed October 9, 1997. The sample was received at Process Research's Vancouver Laboratory on October 27, 1997. Activities during collection of samples are outlined in Table 3.0-1.

**Table 3.0-1  
Activity Schedule for Bulk Sampling Program**

DATE	EVENT
September 22	Mobilization to Carmacks.
September 23	Equipment and operators on site. Start excavation of trench 91-18.
September 25	Excavation of trench 91-18 complete. Start excavation of trench 92-68.
September 28	Excavation of trench 92-68 complete.
September 29	Final work on trench 91-22 completed by excavator. Start excavation of trench 91-20. Dump trucks and Mini-Air Trac drill arrive on site.
September 30	Blast hole drilling started on trench 91-18.
October 02	Blast hole drilling on trench 91-18 complete. Trench 91-18 loaded and blasted.
October 04	Blast hole drilling for trench 92-68 started. Drill break down. Channel-chip sampling completed for trench 91-18.
October 06	Blast hole drilling for trench 92-68 completed. Blast hole drilling for trench 91-20 completed. Blast hole drilling for trench 91-22 completed. Trenches 92-68, 91-20, and 91-22 loaded and blasted.
October 07	Bulk sample taken from trenches 92-68, 91-20 and 91-22. Bulk sample from trench 92-68 mixed.
October 08	Bulk sample from trenches 91-20 and 91-22 mixed. Composite bulk sample blended and mixed. Composite bulk sample loaded into ore bags. Channel-chip sampling of trench 92-68 complete.
October 09	Additional composite bulk sample blended and mixed. Additional composite bulk loaded into ore bags.
October 10	Channel-chip sampling of trenches 91-20 and 91-22. Demobilization from Carmacks to Whitehorse.
October 11	Demobilization from Whitehorse.



## **4.0 COMMENTS**

## 4.0 COMMENTS

### Mineralization

Kilborn observed that malachite azurite and chalcocite mineralization occurred principally along rock fractures and as small aggregates in fine grained highly friable zones. Some malachite mineralization was observed along mineral grains in coarse grained gneisses that were moderately to weakly friable. In resistant competent coarse grained gneisses malachite appeared to occur only along fracture planes.

### Run-of-Mine Simulation

Fine grained material from coarse sand size to fine silt size and smaller particles appeared to dominate the bulk sample material. The main source of this fine material is probably in-situ. This is evidenced by some sections in trenches 91-18 and 91-20 which were not blasted because the material was fine grained and could be excavated without blasting. It was also observed that much of the trench material was moderately to very friable and broke into small particles with a minimum of force. This fine grained material was observed to a depth of 11 m in trench 91-20. It was not possible for Kilborn to determine to what depths this fine grained friable rock extends.

Trench blasting compared to production blasting can generate finer material. Blasting in a trench confines the blast and typically results in finer fragmentation. A production blast that is less confined typically results in coarser fragmentation. Since all material for the bulk sample was necessarily extracted from trenches, the size of blast fragments may be expected to be finer than from a production blast.

The bulk sample extracted from trenches may be biased towards finer grained material. Blast fragmentation from trenches and possibly finer and more friable ore material in the upper portion of the deposit may contribute to higher amounts of fine materials than would be expected from a run-of-mill production blast.

## **5.0 REFERENCES**

## 5.0 REFERENCES

1. Holtby, M. and K. McNaughton. Williams Creek Project: Yukon Territory, Silver Standard Resources Inc. February 1993.
2. Kilborn Engineering Pacific Ltd. Western Copper Holdings Limited, Carmacks Copper Project: Feasibility Study. October 1994.
3. Holtby, M. Personal Communication. September 9, 1997.