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METALLURGICAL INVESTIGATION
OF
WILLIAMS CREEK COPPER OXIDE ORE

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& ASSOCIATES LTD.

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METALLURGICAL INVESTIGATION OF WILLIAMS CREEK COPPER OXIDE ORE

Prepared for:

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Attention: Robert Quartermain

File Number: M90-001
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1.0 INTRODUCTION

At the request of Mr. Robert Quartermain of Consolidated Silver Standard Mines Ltd, metallurgical test work was carried out on bulk samples of oxide copper ore from the Williams Creek property. Two composites of run of mine sized ore were provided, with an average head grade of approximately 1.5% Cu. Total testwork in this investigation consisted of six bottle roll sulphuric acid leach tests, one bottle roll ammonia leach test, and two larger-scale sulphuric acid column leach tests. The purpose of the investigation was to provide detailed metallurgical information on the amenability of the Williams Creek ore to heap leaching with sulphuric acid.

This investigation was initiated by a phone call to Bacon, Donaldson & Associates (BDA) by Mr. Robert Quartermain on August 29, 1989. Further correspondence followed to define more fully the scope of the investigation. Actual testwork began on October 16, 1989. Throughout this investigation, testwork results were provided to Mr. Quartermain as soon as they became available.

2.0 SUMMARY

Bottle roll sulphuric acid leaching tests indicated that copper extractions of 80-85% Cu could be obtained for both ore composites at a fairly coarse crush (minus 3/4 inch). Net sulphuric acid consumption for these tests was 26-28 kg/tonne.

Two column leach tests conducted on 50:50 blends of composite Orange and composite Blue at a 3/4 inch crush yielded final copper extractions of 85.0% Cu (Column 1) and 86.7% Cu (Column 2). Both tests were identical, but Column 2 involved more intensive leaching in the beginning of the leach; solution strength was 20 g/l H_2SO_4 initially (compared to 15 g/l for Column 1), and solution flowrate was 100 ml/min initially (compared to 20 ml/min for Column 1). The more intensive leaching employed in Column 2 greatly increased the copper extraction rate, with Column 2 reaching 80% copper extraction within 15 days (compared to 60 days for Column 1).

With the shorter test duration, Column 2 had a marginally lower net sulphuric acid consumption; 48 kg/tonne compared to 51 kg/tonne for Column 1.

One bottle roll test utilizing ammonia leaching of the copper was performed. This test resulted in low copper extraction and this process route was not investigated further.

3.0 ORE CHARACTERIZATION

On September 6, 1989 approximately 2700 kg (6000 lbs) of damp, minus 15 cm (6 inch) copper oxide ore was received by Bacon, Donaldson and Associates at their facilities in Richmond, B.C. The shipment consisted of two separate ore composites: Composite Orange and Composite Blue. Approximately equal proportions of each composite were received.

The composites were jaw crushed separately to approximately minus 3 cm. It should be noted that some of the material crushed in a distinctly slabby manner, with pieces up to 10 or 12 cm along their long axis and less than 3 cm along their short axis. This was not a major occurrence however, with the total amount of slabby breakage estimated to be less than 1% by weight.

Sample cuts were taken from each composite for head analysis. This information is presented in Table 3.1 along with the average back calculated head grades from testwork.

Table 3.1
Head Assay Summary

Composite	Assay Type	Total Copper % Cu	Oxide Copper as % Cu	Total Iron % Fe
Orange	Assay	1.32	1.26	4.00
	Avg. Back-Calc.	1.54	1.43	3.84
Blue	Assay	1.68	1.60	4.60
	Avg. Back-Calc.	1.65	1.57	4.77
50:50 Mix	Assay	1.50	1.43	4.80
	Avg. Back-Calc.	1.56	-	4.11

4.0 BOTTLE ROLL LEACHING

In total, 7 bottle roll tests were conducted in this study. Six of these were sulphuric acid leaches conducted at various ore sizes to determine the optimum feed size for each composite, and to establish the plateau for copper extraction that could be expected in subsequent column leach testing. The final bottle roll test (L7) was an ammonia leach on a 50:50 mix of composites Orange and Blue to determine the general amenability of the ore to leaching with an ammonia solution. All bottle roll tests were conducted on 4 to 5 kg of ore at 50 to 55% solids. All leach solutions were changed and analyzed incrementally to provide extraction data.

4.1 Bottle Roll Sulphuric Acid Leaching

Six bottle roll sulphuric acid leach tests were conducted in this study. The tests investigated three different ore sizes on both Composite Orange and Composite Blue. All other test parameters were similar:

- 4000 - 5000 g dry ore
- 50 - 55% solids
- 15 g/l H_2SO_4 solution strength
- 166 hour duration (approximately 7 days)

The results of the six tests are summarized in Table 4.1 and the complete details are presented in Appendix I.

Table 4.1

Bottle Roll Sulphuric Acid Leaching

Test No.	Composite	Ore Size	Tailings %Cu^{Total}	Copper Extraction %	Net Acid Consumption Kg/tonne
L1	Blue	1-1.5 inch	0.38	74.7	29.3
L3	Blue	3/4 inch	0.35	80.6	27.9
L5	Blue	1/4-3/8 inch	0.34	80.3	23.8
L2	Orange	1-1.5 inch	0.52	66.3	28.0
L4	Orange	3/4 inch	0.28	82.3	26.4
L6	Orange	1/4-3/8 inch	0.20	86.6	26.5

The results in Table 4.1 indicate that Composite Orange shows greater sensitivity to crush size than Composite Blue. However, for both composites, the 3/4 inch size is adequate to achieve 80% copper extraction

In Table 4.1, the acid consumption figures shown are the net consumptions. These are calculated from the gross acid consumptions shown in the detailed balances (Appendix I) by correcting for the acid which occurs as copper.

Figures 4.1 and 4.2 illustrate the copper extraction as a function of time for the six tests. The curves indicate that the extraction rates have all levelled off by 166 hours, and it is likely that the final numbers have approached the ultimate extractions for these test parameters.

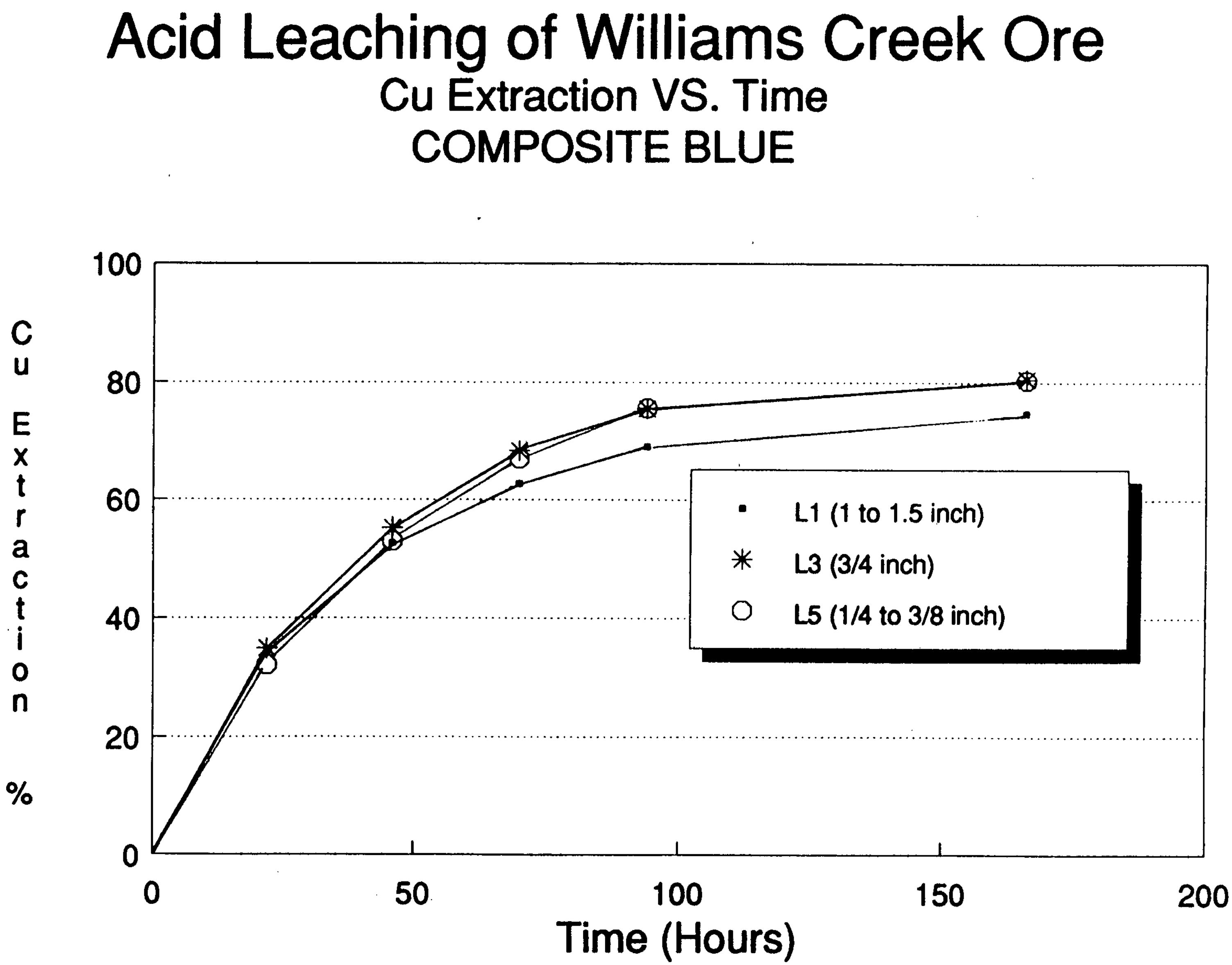

Acid Leaching of Williams Creek Ore
Cu Extraction VS. Time
COMPOSITE BLUE
Cu Extraction %
Time (Hours)
0
20
40
60
80
100
0
50
100
150
200
L1 (1 to 1.5 inch)
L3 (3/4 inch)
L5 (1/4 to 3/8 inch)

Figure 4.1

Acid Leaching of Williams Creek Ore

Cu Extraction VS. Time

COMPOSITE ORANGE

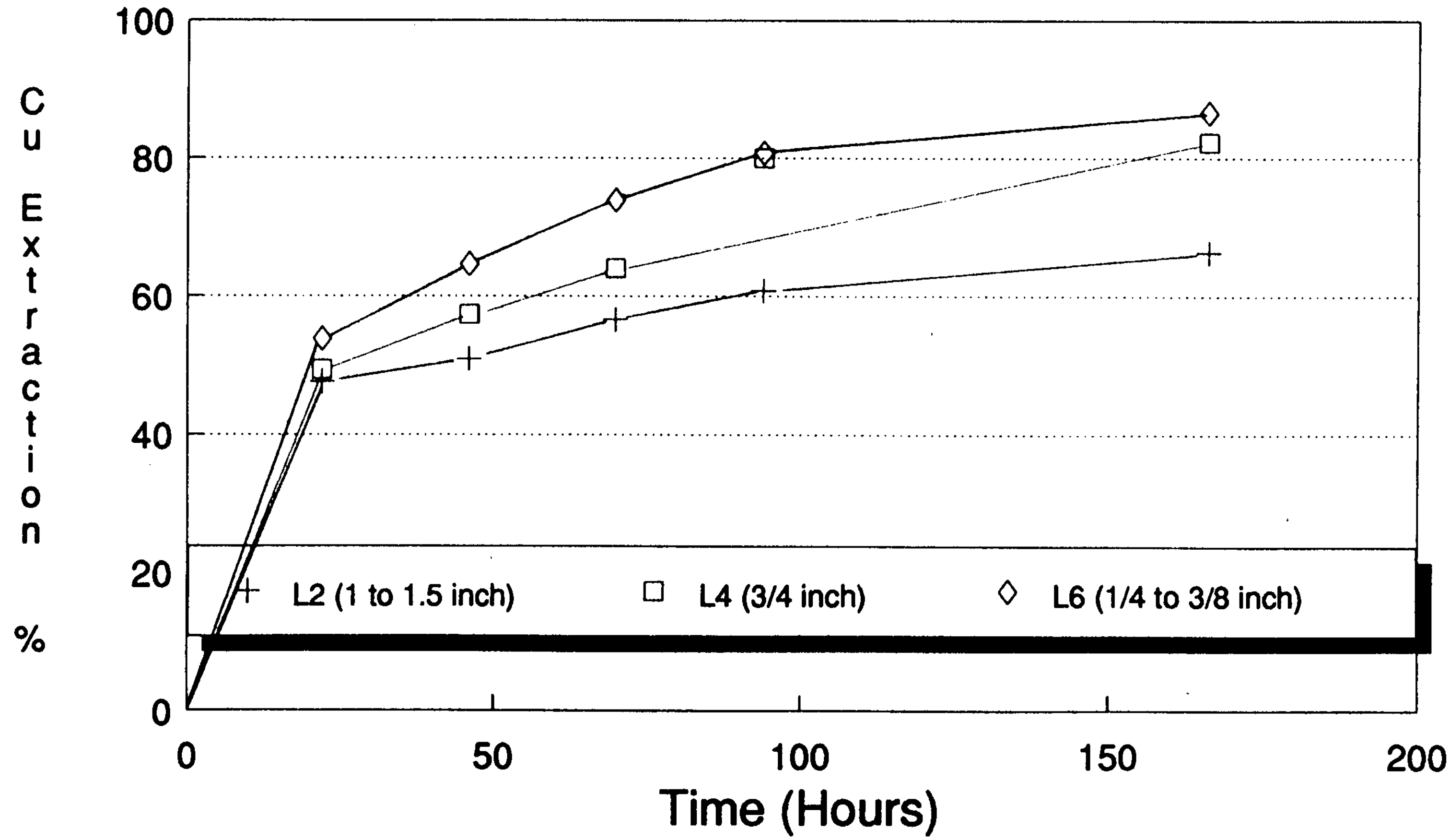


Figure 4.2

4.2 Bottle Roll Ammonia Leaching

A single bottle roll ammonia leach test was conducted on a 50:50 blend of Composite Orange and Composite Blue. The ore was crushed to 3/4 inch and leached for 10 days at 50% solids with a "0.25M NH_4OH + 0.25M $(NH_4)_2CO_3$" strength ammonia solution. The leach solution was changed daily for the first 96 hours and then after a total of 7 and 8 days.

Final extraction for this test was 57.7% copper and the final tailings assayed 0.70% Cu^T. These results are considerably worse than the comparable test (L3 and L4) with sulphuric acid. The ammonia leach extraction rate is compared to the rates of tests L3 and L4 in figure 4.3.

Since this test was preliminary in nature, no analyses were performed to establish reagent consumption. However, the motivation for the test was the fact that the contained iron and carbonate minerals would not be dissolved under the leach conditions.

Leaching of Williams Creek Ore

Cu Extraction VS. Time

Comparison of Acid & Ammonia Leaching

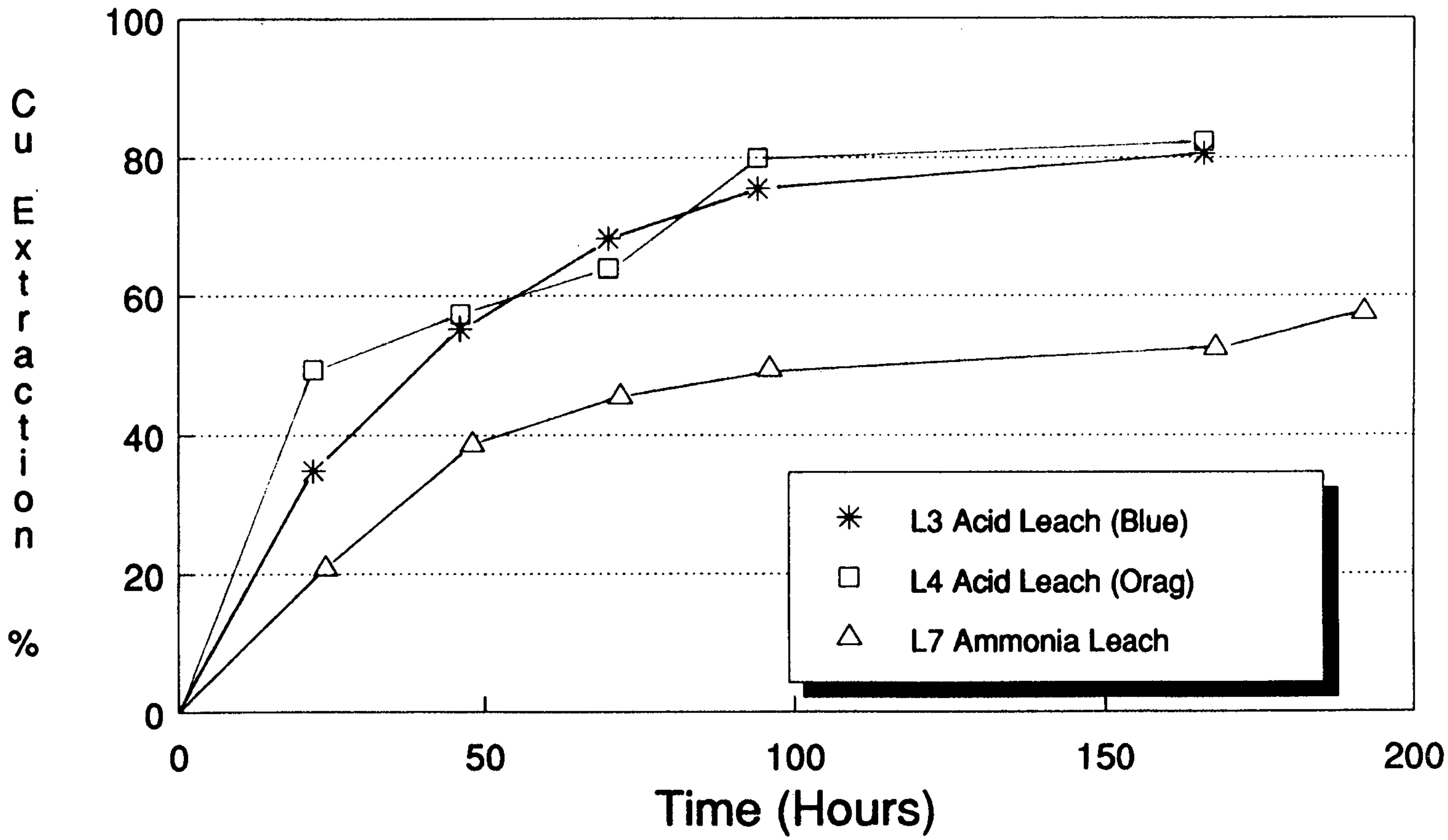


Figure 4.3

5.0 COLUMN LEACHING

Two separate column leach tests were conducted on 50:50 blends of minus 3/4 inch Orange and Blue composites. Both tests utilized a 12 foot by 1 foot diameter PVC plastic pipe for a column container. Sulphuric acid leach solutions were pumped continuously by metering pumps to the drip point at the top of the column. A special drip arrangement ensured that the contents of the column were uniformly wetted through the cross sectional area. Both columns were loaded by pre-wetting the blended ore with an 18 g/l H_2SO_4 solution at a solution - solids ratio of 3%. Both columns contained approximately 375 kg of ore.

Regular maintenance of the columns consisted of collecting and weighing the pregnant (out) solution in its entirety. This solution was then sampled for assay, with a further 2 litres saved in a glass bottle for future reference. The remaining pregnant solution was neutralized and discarded. Feed solution (in) was topped up at the appropriate solution strength and the metering pumps were checked periodically to ensure they were delivering the correct flowrate.

To end the columns, feed solution was stopped and the columns were allowed to drain for a period of days. Following this, the columns were flushed twice with water at a high flow rate. The out solutions from the washes were collected and assayed. Finally, the columns were dumped, the solids examined visually to determine if there were indications of solution channelling, then the solids were dried and sampled for assay and size analysis.

The two column tests in this investigation differed in that the first column was leached slowly, with low solution strengths and flowrates. This test had a duration of 78 days. The second column was leached more intensely, with high solution strengths and high flowrates for the initial part of the test. This type of intensive leaching at the beginning of a test is known as "pugging". Column 2 had a duration of 33 days.

5.1 Column Test 1

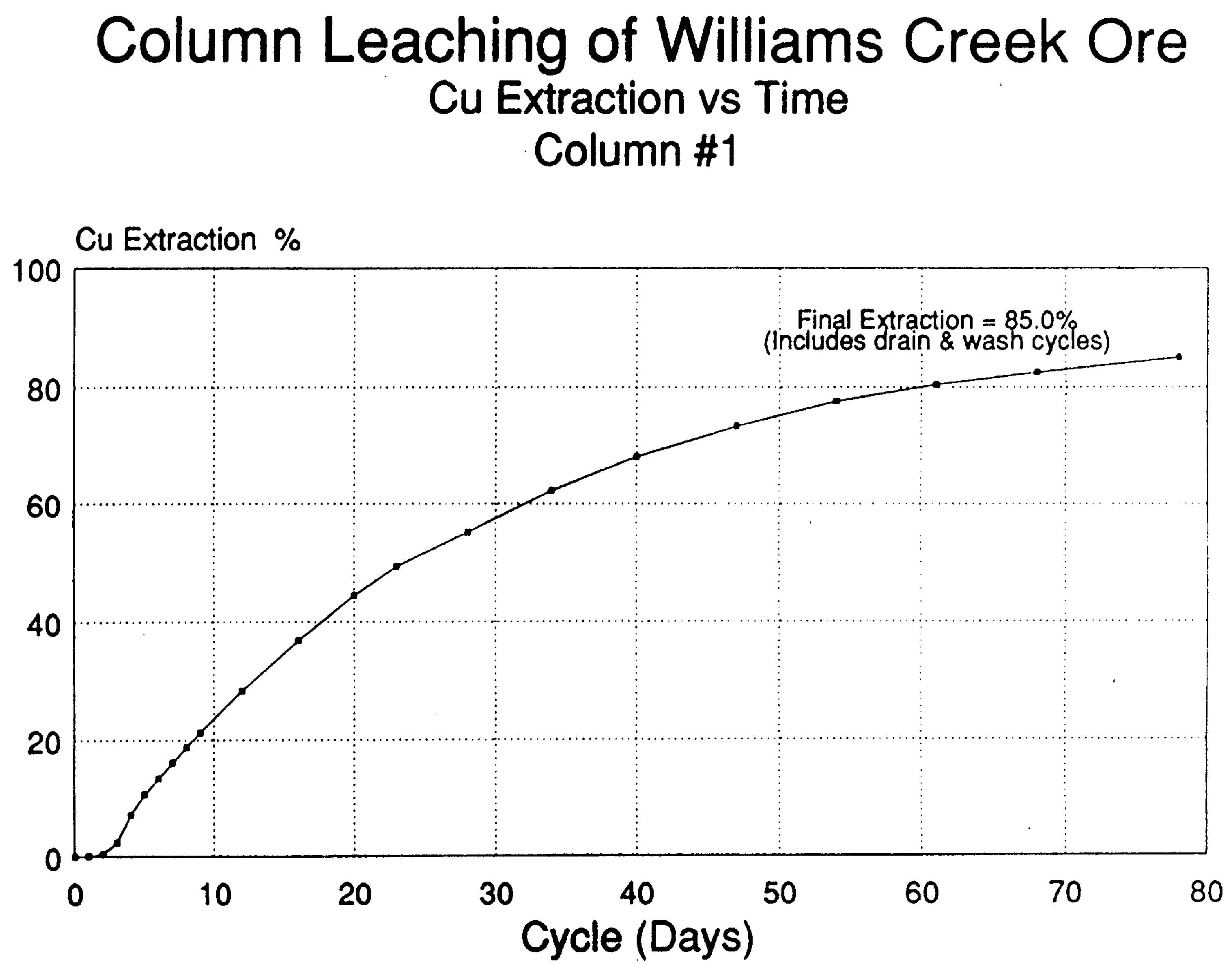
Column test 1 was leached for 78 days (including draining and washing) and reached a final copper extraction of 85.0%. Figure 5.1 illustrates the extraction rate for copper over the duration of the test. The curve depicts a long climb up to the final extraction of 85% Cu; 60% extraction was not reached until day 32, and 80% until day 60.

Column 1 began leaching with a solution strength of 15 g/l H_2SO_4 at a flowrate of 20ml/min and this was continued until day 54 when the iron content of the out solution reached 3.0 g/l Fe. After day 54, the feed strength was cut back to 10 g/l H_2SO_4 and this was maintained for the duration of the test. The lower feed strength had an immediate effect on the amount of iron being leached; the next out solution assayed only 1.84 g/l Fe. Figure 5.2 illustrates the iron and copper concentration of the pregnant solution over time. The test was ended when the copper concentration of the pregnant solution had decreased to approximately 0.5 g/l.

Column 1 had a total acid consumption of 70.6 kg H_2SO_4 per tonne of ore. Of this, 19.1 kg/tonne was consumed by the copper and would be recovered in a solution processing plant. Correcting for the dissolved copper, the net acid consumption for column 1 was 51.5 kg/tonne.

When column 1 was dismantled, visual inspection of the tailings indicated that all the material had leached evenly with no indications of channelling. The tailings were dried, then sampled for assay and size analysis.

At the request of the client, gold and silver assays were also conducted on the tailings for column 1. The assay indicated values of 0.51 g/tonne Au and 4.53 g/tonne Ag.



Column Leaching of Williams Creek Ore
Cu Extraction vs Time
Column #1
Cu Extraction %
100
80
60
40
20
0
0
10
20
30
40
50
60
70
80
Cycle (Days)
Final Extraction = 85.0%
(Includes drain & wash cycles)

Figure 5.1

Column Leaching of Williams Creek Ore

Cu and Fe Concentration vs Time

Column #1

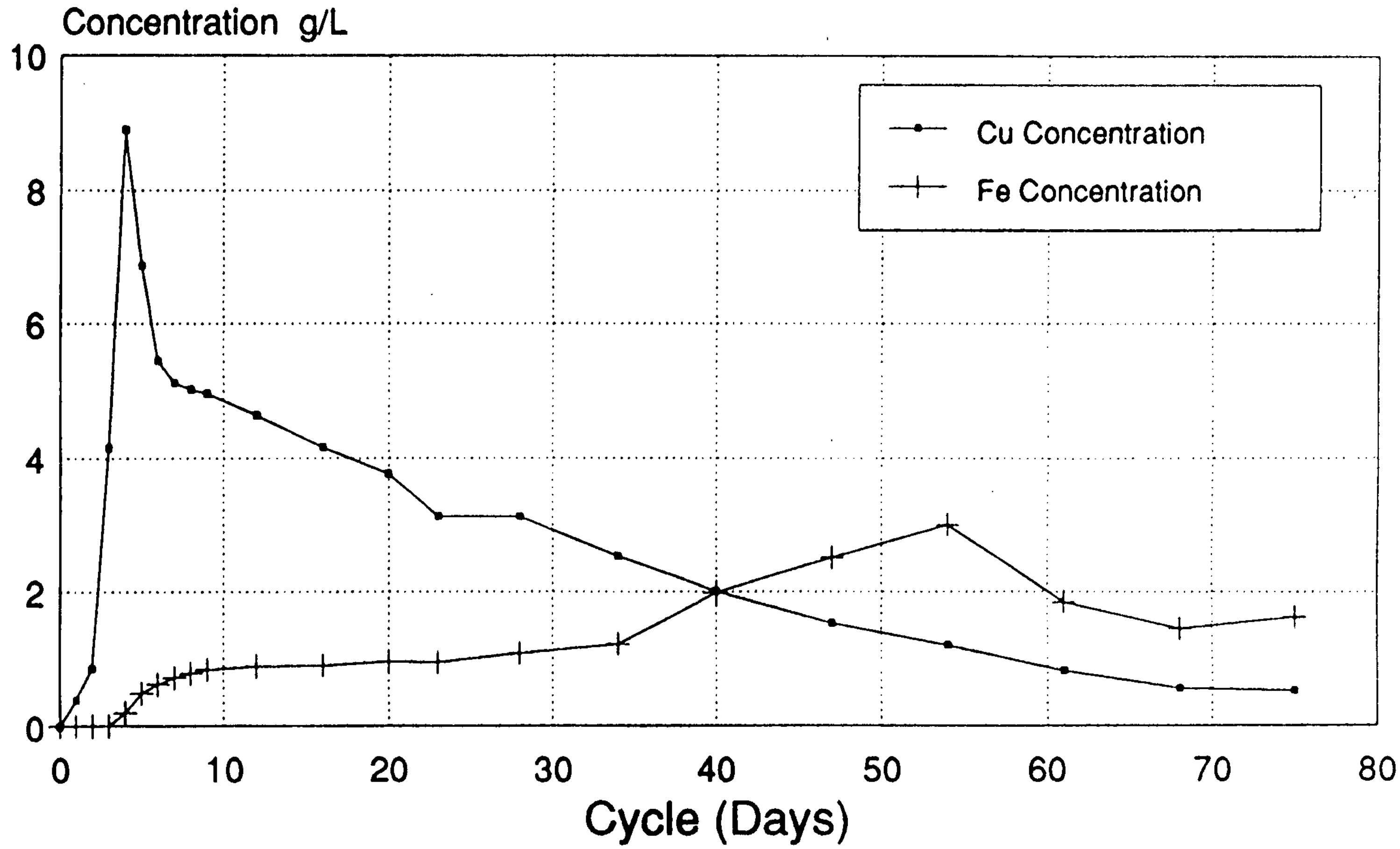


Figure 5.2

5.2 Column Test 2

Column test 2 was leached for 33 days (including draining and washing) and reached a final copper extraction of 86.7%. Figure 5.3 illustrates the rate of copper extraction over the course of the test. For this test, with its more intensive leaching, copper extraction progressed rapidly and reached 60% within 5 days, and 80% within 15 days.

Column 2 began leaching with a solution strength of 20 g/l H_2SO_4 and a flowrate of 100 ml/min (compared to 15 g/l H_2SO_4 and 20 ml/min for column 1) and maintained this until day 6 when the feed solution was cut back to 15 g/l H_2SO_4. Finally, on day 15 the feed strength was cut back to 10 g/l H_2SO_4. Throughout this test, iron content of the out solution did not exceed 2.0 g/l Fe. Flowrate of feed solution was cut back periodically through the course of the test and by the end of the test it was down to 20 ml/min. Figure 5.4 illustrates the copper and iron concentration of the pregnant solution over time. As for column 1, the test was ended when the copper concentration in the pregnant solution had decreased to approximately 0.5 g/L.

Column 2 had a total acid consumption of 69.1 kg H_2SO_4 per tonne of ore. Of this, 21.1 kg/tonne was consumed by this dissolving copper. After correcting for dissolved copper, net acid consumption for column 2 was 48 kg/tonne.

Upon dismantling of column 2, visual inspection of the tailings indicated even leaching with no apparent channelling. The dried tailings were sampled for assay and size analysis.

While column test 2 was in progress several pieces of the ore were placed in acid solution. This solution was changed periodically to ensure the availability of acid and to maximize the diffusion gradient for acid and copper. Upon the completion of column test 2 the pieces of rock were broken in half to observe the extent to which copper leaching had progressed. The copper oxide had been leached to a depth of 4 to 5mm. Inside this leached rim the rock contained visible copper "oxide" minerals.

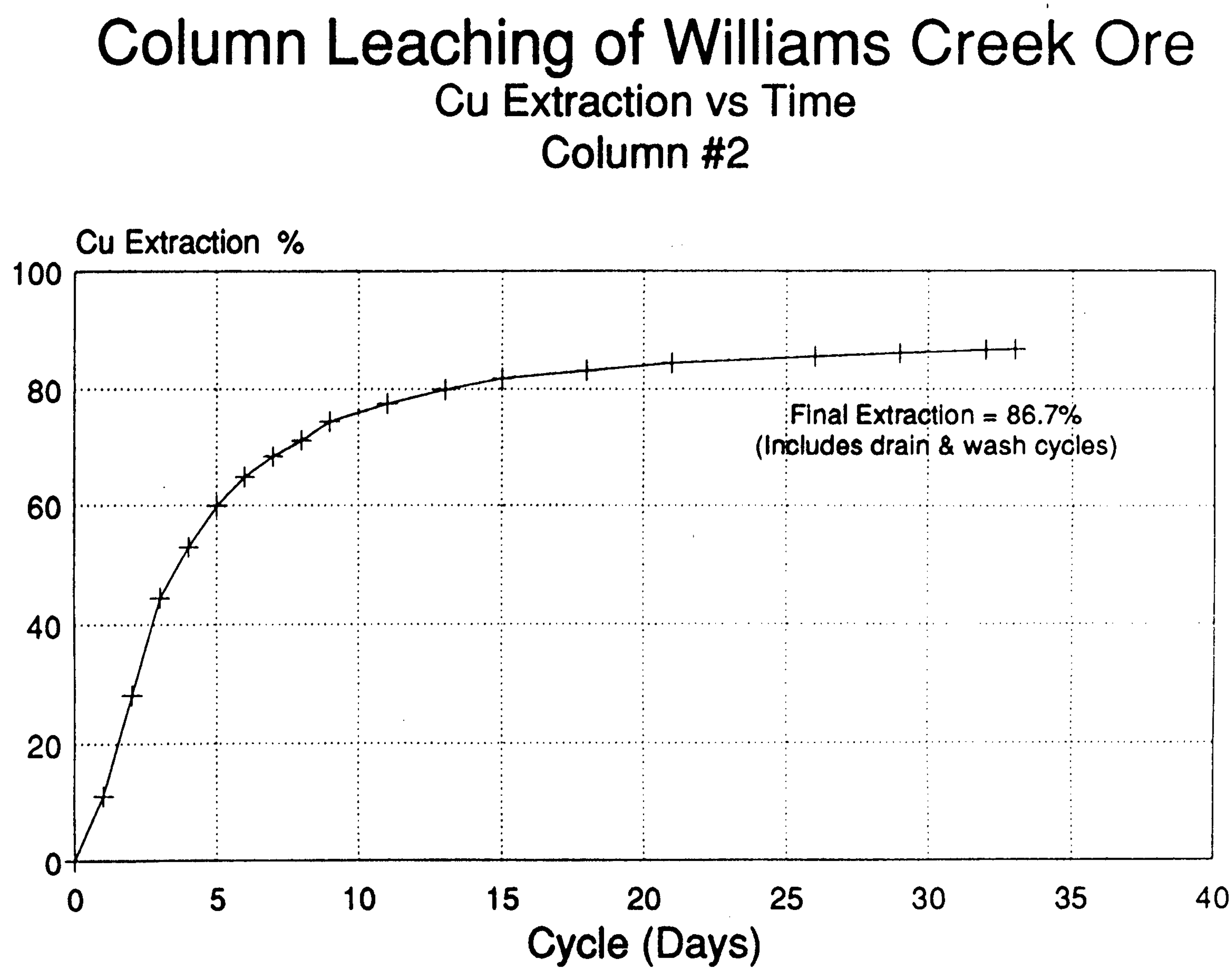
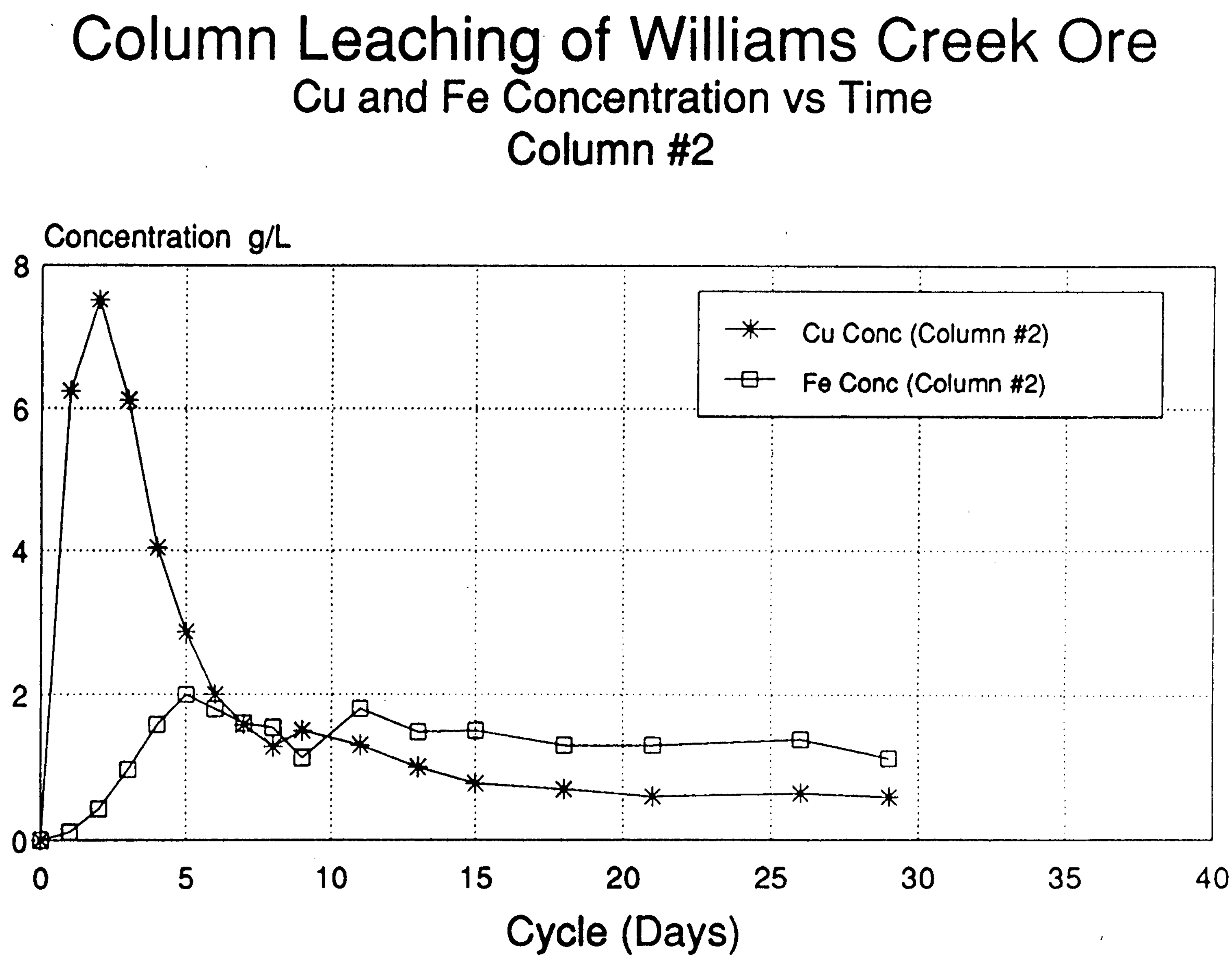

Column Leaching of Williams Creek Ore
Cu Extraction vs Time
Column #2
Cu Extraction %
100
80
60
40
20
0
0
5
10
15
20
25
30
35
40
Final Extraction = 86.7%
(Includes drain & wash cycles)
Cycle (Days)

Figure 5.3



Column Leaching of Williams Creek Ore
Cu and Fe Concentration vs Time
Column #2
Concentration g/L
8
6
4
2
0
Cu Conc (Column #2)
Fe Conc (Column #2)
0
5
10
15
20
25
30
35
40
Cycle (Days)

Figure 5.4

5.3 Comparison: Column 1 vs Column 2

Table 5.1 summarizes the important information from both column tests.

Table 5.1
Comparison: Column 1 and Column 2

Test No.	Composite	Ore Size	Duration (Days)	Tailings $\%Cu^{Total}$	Copper Extraction %	Net Acid Consumption Kg/tonne
Column 1	50:50 Mix	3/4"	78	0.22	85.0	51
Column 2	50:50 Mix	3/4"	33	0.21	86.7	48

Figure 5.5 illustrates the dramatic difference in copper extraction rates for the two tests.

It can be seen from the information in Table 5.1 and the graph in figure 5.5 that column 2 has given better results than column 1; copper extraction occurred much quicker and acid consumption was lower for column 2.

Column Leaching of Williams Creek Ore

Cu Extraction vs Time

Comparison of Column #1 & Column #2

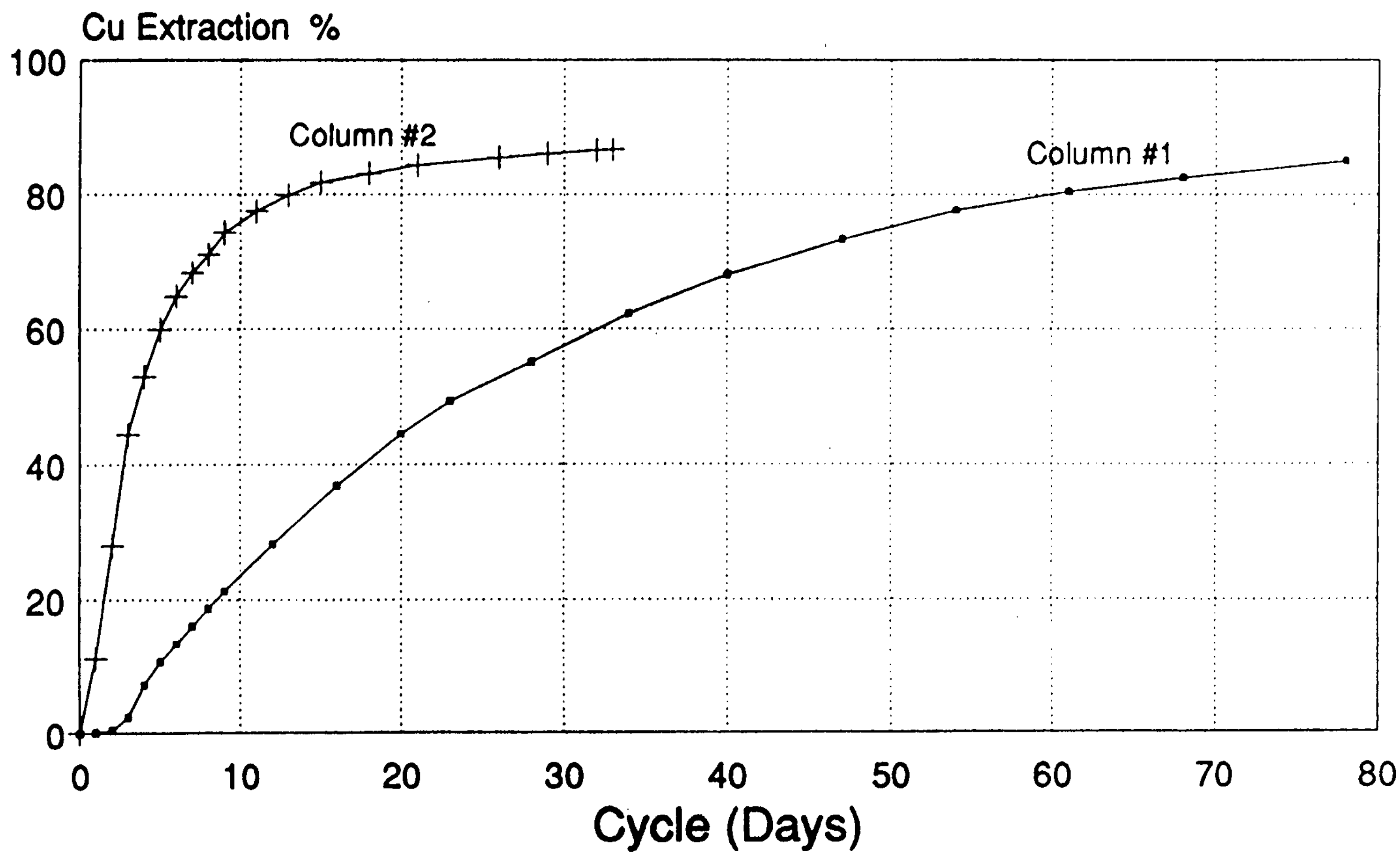


Figure 5.5

APPENDIX I

Bottle Roll Sulphuric Acid Leaching

BOTTLE ROLL SULPHURIC ACID LEACHING OF COPPER OXIDE ORES

File No: M90-001 Date: 16-Oct-89
Test No: L1
Sample Description: COMP BLUE (Size: 1 to 1.5 inch)

TEST CONDITIONS:

Solids:	4809.3 g
H2O:	4150.7 g
% Solids:	54%
Solution Strength:	15 g/l H2SO4
Test Duration:	166 hours

TEST DESCRIPTION:

- -solids and acid solution combined in large bottle
- -bottles placed on rollers
- -each day, acid solution decanted and replaced with fresh solution
- -decanted solution analyzed for Cu, Fe, free acid, and pH
- -test ended after 166 hours
- -solids washed and wash solutions analyzed
- -final solids assayed for TOTAL COPPER, OXIDE COPPER, IRON

TEST RESULTS:

Solution Analyses:

TIME	SAMPLE SIZE	COPPER		IRON		H2SO4		pH
	(g)	(g/l)	(g)	(g/l)	(g)	(g/l)	(g)	
22	2919	5.00	14.60	0.03	0.09	0.29	0.85	2.9
46	2780	5.60	15.57	0.24	0.67	0.45	1.25	2.8
70	2622	3.60	9.44	0.55	1.44	0.71	1.86	2.6
94	2548	2.44	6.22	0.09	0.23	0.91	2.32	2.7
166	2319	1.92	4.45	1.33	3.08	0.79	1.83	2.5
wash	4411	0.83	3.66	0.81	3.57	1.07	4.72	2.2
	TOTAL		53.93		9.08		12.83	

Solids Analyses:

TIME	SAMPLE SIZE	TOTAL COPPER		OXIDE COPPER		IRON	
	(g)	(%)	(g)	(%)	(g)	(%)	(g)
166	4809.3	0.38	18.28	0.28	13.47	4.60	221.23
	TOTAL		18.28		13.47		221.23

CALCULATIONS:

TIME	COPPER EXTRACTION		IRON EXTRACTION		ACID CONSUMPTION	
	INDV. %	CUM. %	INDV. %	CUM. %	INDV. kg/tonne	CUM. kg/tonne
22	34.55%	34.55%	0.06%	0.06%	12.70	12.70
46	18.07%	52.62%	0.41%	0.47%	8.79	21.49
70	10.08%	62.70%	0.86%	1.33%	8.19	29.67
94	6.31%	69.00%	-0.21%	1.12%	7.62	37.29
166	5.59%	74.59%	2.35%	3.47%	7.57	44.86
wash	0.10%	74.69%	0.47%	3.94%	1.73	46.59
	TOTAL	74.69%		3.94%		46.59

HEAD GRADE:

	TOTAL COPPER	OXIDE COPPER	IRON
ASSAY HEAD	1.68%	1.60%	4.60%
CALCULATED HEAD	1.50%	1.40%	4.79%

SIZE DISTRIBUTION

SAMPLE NO. M90-001 L1

Bottle Roll Tails

Size Fraction (mesh)		Individual Percentage Retained %	Cumulative Percentage Passing %
	+ 1"	16.6	83.4
- 1"	+ 3/4"	12.7	70.5
- 3/4"	+ 1/2"	5.1	65.4
- 1/2"	+ 3/8"	2.7	62.7
- 3/8"	+ 4 mesh	2.6	60.1
- 4	+ 6	0.6	59.6
- 6	+ 8	0.5	59.1
- 8	+ 10	0.6	58.5
- 10	+ 14	0.4	58.0
- 14	+ 20	0.5	57.6
- 20	+ 28	0.6	57.0
- 28	+ 35	2.0	55.0
- 35	+ 48	1.4	53.6
- 48	+ 65	1.3	52.3
- 65	+ 100	1.2	51.1
- 100	+ 150	1.4	49.7
- 150	+ 200	1.9	47.8
- 200		47.8	

BOTTLE ROLL SULPHURIC ACID LEACHING OF COPPER OXIDE ORES

File No: M90-001 Date: 16-Oct-89
Test No: L2
Sample Description: COMP ORANGE (Size: 1 to 1.5 inch)

TEST CONDITIONS:

Solids:	4897 g
H2O:	4458 g
% Solids:	52%
Solution Strength:	15 g/l H2SO4
Test Duration:	166 hours

TEST DESCRIPTION:

- solids and acid solution combined in large bottle
- bottles placed on rollers
- each day, acid solution decanted and replaced with fresh solution
- decanted solution analyzed for Cu, Fe, free acid, and pH
- test ended after 166 hours
- solids washed and wash solutions analyzed
- final solids assayed for TOTAL COPPER, OXIDE COPPER, IRON

TEST RESULTS:

Solution Analyses:

TIME	SAMPLE SIZE	COPPER		IRON		H2SO4		pH
	(g)	(g/l)	(g)	(g/l)	(g)	(g/l)	(g)	
22	3193	6.76	21.58	0.33	1.05	0.33	1.05	2.7
46	3019	4.12	12.44	0.66	1.99	1.13	3.41	2.3
70	2838	2.12	6.02	1.49	4.23	2.73	7.75	2.2
94	2701	1.43	3.86	1.16	3.13	3.99	10.78	2.2
166	2506	1.50	3.76	2.10	5.26	1.91	4.79	2.2
wash	5054	0.49	2.48	0.78	3.94	2.67	13.49	2.2
	TOTAL		50.14		19.61		41.27	

Solids Analyses:

TIME	SAMPLE SIZE	TOTAL COPPER		OXIDE COPPER		IRON	
	(g)	(%)	(g)	(%)	(g)	(%)	(g)
166	4897	0.52	25.46	0.31	15.18	3.20	156.70
	TOTAL		25.46		15.18		156.70

CALCULATIONS:

TIME	COPPER EXTRACTION		IRON EXTRACTION		ACID CONSUMPTION	
	INDV. %	CUM. %	INDV. %	CUM. %	INDV. kg/tonne	CUM. kg/tonne
22	47.62%	47.62%	1.00%	1.00%	13.35	13.35
46	3.28%	50.90%	1.14%	2.13%	8.84	22.19
70	5.71%	56.61%	3.09%	5.23%	7.09	29.29
94	4.15%	60.76%	1.61%	6.84%	5.96	35.25
166	5.48%	66.24%	3.97%	10.81%	7.97	43.22
wash	0.08%	66.32%	0.32%	11.12%	0.56	43.78
	TOTAL	66.32%		11.12%		43.78

HEAD GRADE:

	TOTAL COPPER	OXIDE COPPER	IRON
ASSAY HEAD	1.32%	1.26%	4.00%
CALCULATED HEAD	1.54%	1.33%	3.60%

SIZE DISTRIBUTION

SAMPLE NO. M90-001 L2

Bottle Roll Tails

Size Fraction (mesh)		Individual Percentage Retained %	Cumulative Percentage Passing %
	+ 1"	25.2	74.8
- 1"	+ 3/4"	18.8	56.1
- 3/4"	+ 1/2"	8.4	47.7
- 1/2"	+ 3/8"	4.6	43.1
- 3/8"	+ 4 mesh	3.4	39.7
- 4	+ 6	0.9	38.8
- 6	+ 8	0.6	38.2
- 8	+ 10	0.5	37.7
- 10	+ 14	0.3	37.4
- 14	+ 20	0.3	37.1
- 20	+ 28	0.1	37.0
- 28	+ 35	0.2	36.8
- 35	+ 48	0.1	36.7
- 48	+ 65	0.2	36.5
- 65	+ 100	0.2	36.3
- 100	+ 150	0.4	35.9
- 150	+ 200	0.7	35.2
- 200		35.2	

BOTTLE ROLL SULPHURIC ACID LEACHING OF COPPER OXIDE ORES

File No: M90-001 Date: 16-Oct-89
Test No: L3
Sample Description: COMP BLUE (Size: 3/4 inch)

TEST CONDITIONS:

Solids:	4538.2 g
H2O:	4450.8 g
% Solids:	50%
Solution Strength:	15 g/l H2SO4
Test Duration:	166 hours

TEST DESCRIPTION:

- -solids and acid solution combined in large bottle
- -bottles placed on rollers
- -each day, acid solution decanted and replaced with fresh solution
- -decanted solution analyzed for Cu, Fe, free acid, and pH
- -test ended after 166 hours
- -solids washed and wash solutions analyzed
- -final solids assayed for TOTAL COPPER, OXIDE COPPER, IRON

TEST RESULTS:

Solution Analyses:

TIME	SAMPLE SIZE	COPPER		IRON		H2SO4		pH
	(g)	(g/l)	(g)	(g/l)	(g)	(g/l)	(g)	
22	2812	5.32	14.96	0.02	0.06	0.29	0.82	2.7
46	2733	6.68	18.26	0.12	0.33	0.33	0.90	2.9
70	2524	5.00	12.62	0.40	1.01	0.50	1.26	2.8
94	2424	3.52	8.53	0.07	0.17	0.66	1.60	2.7
166	2362	2.54	6.00	1.08	2.55	0.56	1.32	2.6
wash	3595	1.50	5.39	0.75	2.70	0.63	2.26	2.5
	TOTAL		65.76		6.81		8.17	

Solids Analyses:

TIME	SAMPLE SIZE	TOTAL COPPER		OXIDE COPPER		IRON	
	(g)	(%)	(g)	(%)	(g)	(%)	(g)
166	4538.2	0.35	15.88	0.23	10.44	4.20	190.60
	TOTAL		15.88		10.44		190.60

CALCULATIONS:

TIME	COPPER EXTRACTION		IRON EXTRACTION		ACID CONSUMPTION	
	INDV. %	CUM. %	INDV. %	CUM. %	INDV. kg/tonne	CUM. kg/tonne
22	34.81%	34.81%	0.05%	0.05%	14.43	14.43
46	20.38%	55.20%	0.25%	0.30%	9.08	23.50
70	13.09%	68.29%	0.81%	1.11%	8.67	32.17
94	7.26%	75.55%	-0.24%	0.87%	7.91	40.08
166	5.36%	80.91%	2.44%	3.31%	7.76	47.84
wash	-0.36%	80.55%	0.14%	3.45%	2.36	50.20
	TOTAL	80.55%		3.45%		50.20

HEAD GRADE:

	TOTAL COPPER	OXIDE COPPER	IRON
ASSAY HEAD	1.68%	1.60%	4.60%
CALCULATED HEAD	1.80%	1.68%	4.35%

SIZE DISTRIBUTION

SAMPLE NO. M90-001 L3

Bottle Roll Tails

Size Fraction (mesh)		Individual Percentage Retained %	Cumulative Percentage Passing %
	+ 1"	0	100.0
- 1"	+ 3/4"	2.0	98.0
- 3/4"	+ 1/2"	12.5	85.5
- 1/2"	+ 3/8"	6.2	79.3
- 3/8"	+ 4 mesh	7.5	71.8
- 4	+ 6	2.1	69.7
- 6	+ 8	1.6	68.1
- 8	+ 10	1.8	66.3
- 10	+ 14	1.4	65.0
- 14	+ 20	1.5	63.5
- 20	+ 28	1.0	62.5
- 28	+ 35	2.7	59.9
- 35	+ 48	2.4	57.4
- 48	+ 65	2.6	54.8
- 65	+ 100	2.6	52.2
- 100	+ 150	2.5	49.7
- 150	+ 200	2.2	47.4
- 200		47.4	

BOTTLE ROLL SULPHURIC ACID LEACHING OF COPPER OXIDE ORES

File No: M90-001 Date: 16-Oct-89
Test No: L4
Sample Description: COMP ORANGE (Size: 3/4 inch)

TEST CONDITIONS:

Solids:	4805.3 g
H2O:	4167.7 g
% Solids:	54%
Solution Strength:	15 g/l H2SO4
Test Duration:	166 hours

TEST DESCRIPTION:

- solids and acid solution combined in large bottle
- bottles placed on rollers
- each day, acid solution decanted and replaced with fresh solution
- decanted solution analyzed for Cu, Fe, free acid, and pH
- test ended after 166 hours
- solids washed and wash solutions analyzed
- final solids assayed for TOTAL COPPER, OXIDE COPPER, IRON

TEST RESULTS:

Solution Analyses:

TIME	SAMPLE SIZE	COPPER		IRON		H2SO4		pH
	(g)	(g/l)	(g)	(g/l)	(g)	(g/l)	(g)	
22	2927	7.45	21.81	0.29	0.85	0.27	0.79	2.8
46	2914	5.16	15.04	0.58	1.69	0.62	1.81	2.6
70	2813	2.80	7.88	0.93	2.62	1.37	3.85	2.3
94	2637	3.80	10.02	0.15	0.40	2.36	6.22	2.2
166	2263	1.80	4.07	1.58	3.58	1.36	3.08	2.2
wash	3362	1.09	3.66	1.14	3.83	1.21	4.07	2.3
	TOTAL		62.48		12.96		19.82	

Solids Analyses:

TIME	SAMPLE SIZE	TOTAL COPPER		OXIDE COPPER		IRON	
	(g)	(%)	(g)	(%)	(g)	(%)	(g)
166	4805.3	0.28	13.45	0.22	10.57	3.60	172.99
	TOTAL		13.45		10.57		172.99

CALCULATIONS:

TIME	COPPER EXTRACTION		IRON EXTRACTION		ACID CONSUMPTION	
	INDV. %	CUM. %	INDV. %	CUM. %	INDV. kg/tonne	CUM. kg/tonne
22	49.38%	49.38%	0.78%	0.78%	12.78	12.78
46	7.93%	57.30%	0.98%	1.77%	8.67	21.44
70	6.71%	64.01%	1.70%	3.47%	8.07	29.51
94	15.94%	79.95%	-0.36%	3.11%	7.12	36.63
166	2.08%	82.03%	3.43%	6.55%	7.80	44.44
wash	0.25%	82.28%	0.42%	6.97%	2.05	46.49
	TOTAL	82.28%		6.97%		46.49

HEAD GRADE:

	TOTAL COPPER	OXIDE COPPER	IRON
ASSAY HEAD	1.32%	1.26%	4.00%
CALCULATED HEAD	1.58%	1.52%	3.87%

SIZE DISTRIBUTION

SAMPLE NO. M90-001 L4

Bottle Roll Tails

Size Fraction (mesh)		Individual Percentage Retained %	Cumulative Percentage Passing %
	+ 1"	0	100.0
- 1"	+ 3/4"	8.9	91.1
- 3/4"	+ 1/2"	25.5	65.6
- 1/2"	+ 3/8"	10.3	55.3
- 3/8"	+ 4 mesh	9.5	45.7
- 4	+ 6	2.0	43.8
- 6	+ 8	1.5	42.3
- 8	+ 10	1.5	40.8
- 10	+ 14	1.0	39.8
- 14	+ 20	1.0	38.9
- 20	+ 28	0.6	38.3
- 28	+ 35	1.3	37.0
- 35	+ 48	1.0	36.0
- 48	+ 65	0.9	35.1
- 65	+ 100	0.7	34.4
- 100	+ 150	0.6	33.8
- 150	+ 200	0.7	33.2
- 200		33.2	

BOTTLE ROLL SULPHURIC ACID LEACHING OF COPPER OXIDE ORES

File No: M90-001 Date: 16-Oct-89
Test No: L5
Sample Description: COMP BLUE (Size: 1/4 to 3/8 inch)

TEST CONDITIONS:

Solids:	4595.3 g
H2O:	4395.7 g
% Solids:	51%
Solution Strength:	15 g/l H2SO4
Test Duration:	166 hours

TEST DESCRIPTION:

- solids and acid solution combined in large bottle
- bottles placed on rollers
- each day, acid solution decanted and replaced with fresh solution
- decanted solution analyzed for Cu, Fe, free acid, and pH
- test ended after 166 hours
- solids washed and wash solutions analyzed
- final solids assayed for TOTAL COPPER, OXIDE COPPER, IRON

TEST RESULTS:

Solution Analyses:

TIME	SAMPLE SIZE	COPPER		IRON		H2SO4		pH
	(g)	(g/l)	(g)	(g/l)	(g)	(g/l)	(g)	
22	2750	4.80	13.20	0.03	0.08	0.17	0.47	3.4
46	2494	6.60	16.46	0.15	0.37	0.21	0.52	3.2
70	2430	5.36	13.02	0.39	0.95	0.38	0.92	2.9
94	2341	3.88	9.08	0.62	1.45	0.67	1.57	2.7
166	2166	2.88	6.24	1.03	2.23	0.44	0.95	2.7
wash	3387	1.70	5.76	0.69	2.34	0.36	1.22	2.7
	TOTAL		63.76		7.42		5.66	

Solids Analyses:

TIME	SAMPLE SIZE	TOTAL COPPER		OXIDE COPPER		IRON	
	(g)	(%)	(g)	(%)	(g)	(%)	(g)
166	4595.3	0.34	15.62	0.23	10.57	5.00	229.77
	TOTAL		15.62		10.57		229.77

CALCULATIONS:

TIME	COPPER EXTRACTION		IRON EXTRACTION		ACID CONSUMPTION	
	INDV. %	CUM. %	INDV. %	CUM. %	INDV. kg/tonne	CUM. kg/tonne
22	32.03%	32.03%	0.07%	0.07%	14.19	14.19
46	20.92%	52.95%	0.24%	0.31%	8.84	23.02
70	13.95%	66.90%	0.60%	0.91%	7.86	30.89
94	8.70%	75.60%	0.85%	1.76%	7.45	38.34
166	5.50%	81.10%	1.35%	3.11%	7.52	45.86
wash	-0.78%	80.32%	0.02%	3.13%	2.30	48.17
	TOTAL	80.32%		3.13%		48.17

HEAD GRADE:

	TOTAL COPPER	OXIDE COPPER	IRON
ASSAY HEAD	1.68%	1.60%	4.60%
CALCULATED HEAD	1.73%	1.62%	5.16%

SIZE DISTRIBUTION

SAMPLE NO. M90-001 L5

Bottle Roll Tails

Size Fraction (mesh)		Individual Percentage Retained %	Cumulative Percentage Passing %
	+ 1"	0	100.0
- 1"	+ 3/4"	0	100.0
- 3/4"	+ 1/2"	0.9	99.1
- 1/2"	+ 3/8"	4.1	95.1
- 3/8"	+ 4 mesh	15.6	79.5
- 4	+ 6	4.5	75.0
- 6	+ 8	3.1	71.8
- 8	+ 10	3.3	68.5
- 10	+ 14	2.6	65.9
- 14	+ 20	2.8	63.1
- 20	+ 28	1.8	61.3
- 28	+ 35	4.2	57.1
- 35	+ 48	3.7	53.3
- 48	+ 65	4.0	49.4
- 65	+ 100	3.7	45.7
- 100	+ 150	3.4	42.3
- 150	+ 200	2.8	39.6
- 200		39.6	

BOTTLE ROLL SULPHURIC ACID LEACHING OF COPPER OXIDE ORES

File No: M90-001 Date: 16-Oct-89
Test No: L6
Sample Description: COMP ORANGE (Size: 1/4 to 3/8 inch)

TEST CONDITIONS:

Solids: 4787 g
H2O: 4150 g
% Solids: 54%
Solution Strength: 15 g/l H2SO4
Test Duration: 166 hours

TEST DESCRIPTION:

- solids and acid solution combined in large bottle
- bottles placed on rollers
- each day, acid solution decanted and replaced with fresh solution
- decanted solution analyzed for Cu, Fe, free acid, and pH
- test ended after 166 hours
- solids washed and wash solutions analyzed
- final solids assayed for TOTAL COPPER, OXIDE COPPER, IRON

TEST RESULTS:

Solution Analyses:

TIME	SAMPLE SIZE	COPPER		IRON		H2SO4		pH
	(g)	(g/l)	(g)	(g/l)	(g)	(g/l)	(g)	
22	2750	7.40	20.35	0.28	0.77	0.24	0.66	2.7
46	2863	6.08	17.41	0.57	1.63	0.29	0.83	2.9
70	2689	3.52	9.47	0.85	2.29	0.90	2.42	2.5
94	2488	2.44	6.07	0.12	0.30	1.51	3.76	2.3
166	2281	1.98	4.52	1.45	3.31	1.03	2.35	2.3
wash	3777	1.03	3.89	0.90	3.40	0.84	3.17	2.4
	TOTAL		61.70		11.69		13.19	

Solids Analyses:

TIME	SAMPLE SIZE	TOTAL COPPER		OXIDE COPPER		IRON	
	(g)	(%)	(g)	(%)	(g)	(%)	(g)
166	4787	0.2	9.57	0.15	7.18	3.80	181.91
	TOTAL		9.57		7.18		181.91

CALCULATIONS:

TIME	COPPER EXTRACTION		IRON EXTRACTION		ACID CONSUMPTION	
	INDV. %	CUM. %	INDV. %	CUM. %	INDV. kg/tonne	CUM. kg/tonne
22	53.82%	53.82%	0.75%	0.75%	12.80	12.80
46	10.78%	64.60%	0.89%	1.64%	8.44	21.23
70	9.30%	73.90%	1.46%	3.10%	8.27	29.50
94	6.83%	80.73%	-0.42%	2.68%	7.39	36.89
166	6.01%	86.74%	3.12%	5.80%	7.43	44.32
wash	-0.17%	86.57%	0.24%	6.04%	2.12	46.44
TOTAL		86.57%		6.04%		46.44

HEAD GRADE:

	TOTAL COPPER	OXIDE COPPER	IRON
ASSAY HEAD	1.32%	1.26%	4.00%
CALCULATED HEAD	1.49%	1.44%	4.04%

SIZE DISTRIBUTION

SAMPLE NO. M90-001 L6

Bottle Roll Tails

Size Fraction (mesh)		Individual Percentage Retained %	Cumulative Percentage Passing %
	+ 1"	0	100.0
- 1"	+ 3/4"	0	100.0
- 3/4"	+ 1/2"	1.2	98.8
- 1/2"	+ 3/8"	4.4	94.4
- 3/8"	+ 4 mesh	30.1	64.3
- 4	+ 6	9.2	55.1
- 6	+ 8	5.0	50.1
- 8	+ 10	4.3	45.8
- 10	+ 14	2.6	43.3
- 14	+ 20	2.2	41.0
- 20	+ 28	1.4	39.7
- 28	+ 35	3.2	36.5
- 35	+ 48	2.8	33.7
- 48	+ 65	2.7	31.1
- 65	+ 100	2.3	28.8
- 100	+ 150	1.9	26.9
- 150	+ 200	1.4	25.5
- 200		25.5	

APPENDIX II

Bottle Roll Ammonia Leaching

BOTTLE ROLL AMMONIA LEACHING OF COPPER OXIDE ORES

File No: M90-001 Date: 12-Mar-90
Test No: L7
Sample Description: ORANGE ZONE:BLUE ZONE 50:50 COMPOSITE 3/4" CRUSH

TEST CONDITIONS:

Solids: 4000 g
H2O: 4000 g
% Solids: 50%
Solution Strength: .25 M NH40H + .25 M (NH4)2CO3
Test Duration: 10 days

TEST DESCRIPTION:

-solids and ammonia solution combined in large bottle
-bottles placed on rollers
-each day, ammonia sol'n decanted and replaced with fresh sol'n
-decanted solution analyzed for Cu, Fe, and pH
-test ended after ten (10) days
-solids washed and wash solutions analyzed
-final solids assayed for TOTAL COPPER, OXIDE COPPER, IRON

TEST RESULTS:

Solution Analyses:

TIME	SAMPLE SIZE	COPPER		IRON		pH
hours	(g)	(g/l)	(g)	(g/l)	(g)	
24	3359	3.36	11.29	0.00	0.00	9.5
48	2986	3.48	10.39	0.00	0.00	9.3
72	2961	2.04	6.04	0.00	0.00	9.3
96	2833	1.20	3.40	0.00	0.00	9.4
168	2810	0.86	2.42	0.00	0.00	9.4
192	2710	0.98	2.66	0.00	0.00	9.4
wash	4835	0.34	1.64	0.00	0.00	9.2
	TOTAL		37.84		0.00	

Solids Analyses:

TIME	SAMPLE SIZE	TOTAL COPPER		OXIDE COPPER		IRON	
	(g)	(%)	(g)	(%)	(g)	(%)	(g)
192	4000	0.7	28.00	0.66	26.40	4.80	192.00
	TOTAL		28.00		26.40		192.00

CALCULATIONS:

TIME	COPPER EXTRACTION		IRON EXTRACTION	
	INDV.	CUM.	INDV.	CUM.
hours	%	%	%	%
24	20.90%	20.90%	0.00%	0.00%
48	17.89%	38.79%	0.00%	0.00%
72	6.83%	45.61%	0.00%	0.00%
96	3.95%	49.56%	0.00%	0.00%
168	3.07%	52.63%	0.00%	0.00%
192	4.41%	57.04%	0.00%	0.00%
wash	0.67%	57.70%	0.00%	0.00%
TOTAL		57.70%		0.00%

HEAD GRADE:

	TOTAL COPPER	OXIDE COPPER	IRON
ASSAY HEAD	1.50%	1.43%	4.80%
CALCULATED HEAD	1.65%	1.61%	4.80%

APPENDIX III

Column Leaching

M90-001: WILLIAMS CREEK COPPER COLUMN #1

DATE		LEACH CYCLE	LITRES IN	LITRES OUT	FLOW (ml/min) IN (Aim)	FLOW (ml/min) OUT (Actual)	H2SO4 (g/l) IN	H2SO4 (g/l) OUT	CONSUMPTION H2SO4 GRAMS IND.	CONSUMPTION H2SO4 GRAMS CUM.	CONSUMPTION H2SO4 Kg/tonne (TOTAL)	CONSUMPTION H2SO4 Kg/tonne (As Cu)	PREGNANT SOLUTION Fe (g/l)	PREGNANT SOLUTION COPPER (g/l)	PREGNANT SOLUTION COPPER (g total)	CALCULATED % COPPER EXTRACTED IND.	CALCULATED % COPPER EXTRACTED CUM.
Nov	30	1	29.43	10.05	20	7.0	14.79	0.05	638.8	638.8	1.69	0.02	0.00	0.39	3.92	0.1%	0.1%
Dec	1	2	26.70	24.11	20	16.7	14.79	0.09	392.6	1031.5	2.73	0.10	0.00	0.86	20.74	0.4%	0.4%
	2	3	27.96	25.78	20	17.9	14.79	0.13	410.2	1441.6	3.81	0.54	0.00	4.16	107.25	2.0%	2.4%
	3	4	27.96	29.80	20	20.7	14.79	0.17	408.5	1850.2	4.90	1.62	0.20	8.90	265.19	4.8%	7.2%
	4	5	28.27	27.46	20	19.1	14.79	0.03	417.3	2267.5	6.00	2.39	0.49	6.88	188.92	3.4%	10.7%
	5	6	27.75	27.98	20	19.4	14.79	0.05	408.9	2676.4	7.08	3.01	0.62	5.45	152.51	2.8%	13.4%
	6	7	28.86	28.19	20	19.6	14.79	0.14	422.9	3099.3	8.20	3.60	0.72	5.12	144.35	2.6%	16.1%
	7	8	28.54	28.77	20	20.0	14.79	0.16	417.5	3516.8	9.31	4.19	0.78	5.02	144.44	2.6%	18.7%
	8	9	27.71	27.81	20	19.3	14.79	0.20	404.3	3921.1	10.38	4.75	0.84	4.96	137.93	2.5%	21.2%
	11	12	83.65	84.13	20	19.5	14.79	0.09	1229.6	5150.6	13.63	6.34	0.89	4.64	390.34	7.1%	28.3%
	15	16	114.55	112.27	20	19.5	14.79	0.28	1662.7	6813.3	18.03	8.24	0.90	4.16	467.06	8.5%	36.8%
	19	20	114.01	111.14	20	19.3	14.79	0.43	1638.4	8451.7	22.37	9.95	0.96	3.76	417.87	7.6%	44.4%
	22	23	85.18	85.00	20	19.7	14.79	0.86	1186.7	9638.5	25.51	11.03	0.95	3.12	265.20	4.8%	49.2%
	27	28	106.08	101.70	20	14.1	14.79	0.92	1475.3	11113.7	29.41	12.32	1.08	3.12	317.31	5.8%	55.0%
Jan	2	34	163.60	157.59	20	18.2	14.79	0.96	2268.4	13382.2	35.41	13.94	1.22	2.52	397.12	7.2%	62.2%
	8	40	167.24	160.49	20	18.6	14.79	0.40	2409.2	15791.4	41.79	15.25	1.98	2.00	320.98	5.8%	68.0%
	15	47	196.22	192.61	20	19.1	14.79	0.29	2846.2	18637.6	49.32	16.44	2.50	1.52	292.76	5.3%	73.4%
	22	54	199.00	193.27	20	19.2	14.79	1.00	2749.9	21387.5	56.60	17.38	3.00	1.20	231.93	4.2%	77.6%
	29	61	197.36	192.24	20	19.1	9.86	0.98	1757.6	23145.1	61.25	18.03	1.84	0.82	157.64	2.9%	80.4%
Feb	5	68	200.94	197.60	20	19.6	9.86	0.82	1819.3	24964.3	66.06	18.48	1.45	0.56	110.65	2.0%	82.5%
	12	75	187.00	181.18	20	18.0	9.86	0.56	1742.4	26706.7	70.67	18.86	1.62	0.52	94.21	1.7%	84.2%
	13	76	0.00	7.42	0	5.2	0.00	0.56	-4.2	26702.5	70.66	18.88	1.38	0.66	4.89	0.1%	84.3%
Wash 1		77	100.00	95.26	50	66.2	0.00	0.31	-29.5	26673.0	70.58	19.03	0.50	0.38	36.58	0.7%	84.9%
Wash 2		78	100.00	103.24	50	71.7	0.00	0.11	-11.4	26661.6	70.55	19.05	0.10	0.05	5.37	0.1%	85.0%
															4675.17		

more than half of the composite values have a zero copper oxide content, and there is clearly a distinct division between oxide copper and sulphide copper in the orebody.

After splitting the composite values into a sulphide zone, with zero copper oxide, and an oxide zone, the histograms of total copper for these two zones are shown in Figure 2(a) and (b). It is now seen that the oxide zone shows significantly higher assayed copper values, with an average value of 1.17% total copper as compared with 0.46% in the sulphide zone.

To determine the relationship between the total copper content and the degree of oxidation in the oxide zone, the scatter plot in Figure 3 shows that there is a tendency for higher total copper values to be associated with higher oxide ratios. Although this relationship is perhaps not statistically significant, it suggests that further emphasis should be given to establishing the possible metallurgical implications.

3.2.3 Spatial Data Analysis

To provide quick representations of the spatial distribution of total copper and oxide ratio, projected cross-sections of the composite values were produced. These are given in Figures 4(a) and (b), showing projected east/west and north/south cross sections respectively of the total copper composite values, and in Figures 5(a) and (b) for the oxide ratio.

Figures 4(a) and (b) show that higher copper values tend to be concentrated in the northern portion of the ore body, with a lower grade zone at depth towards the southeast. Initial potential for further ore definition would appear to be in a region bounded by 100530N and 101510N, as seen in Figure 4(b).

The projected sections of oxide ratio in Figures 5(a) and (b) show that the oxidized copper is confined to the shallower portions of the ore body, with little copper oxidation evident below an elevation of 640 meters.

WILLIAMS CREEK
Down-Hole Variogram - Total Cu (logs)

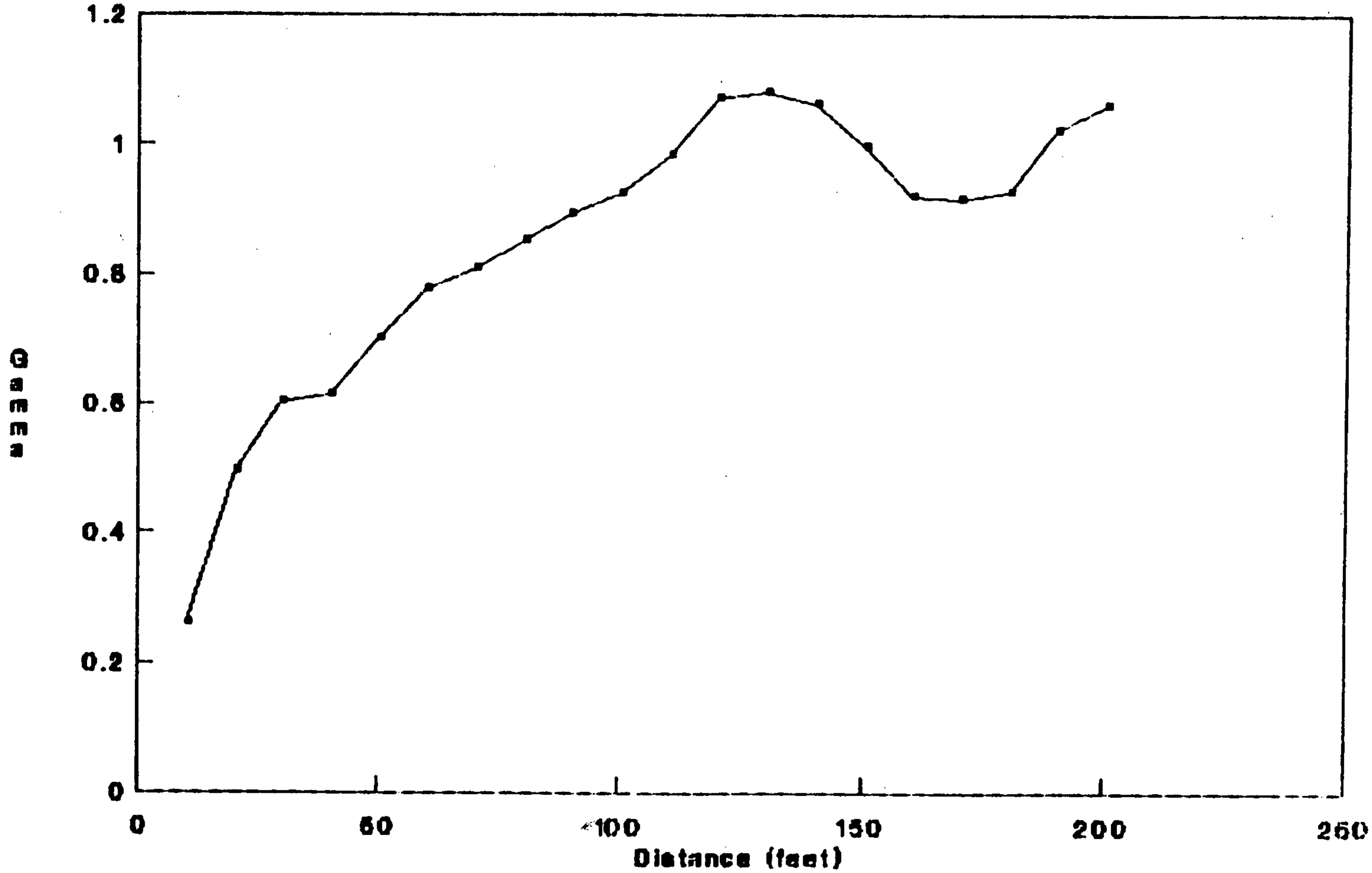
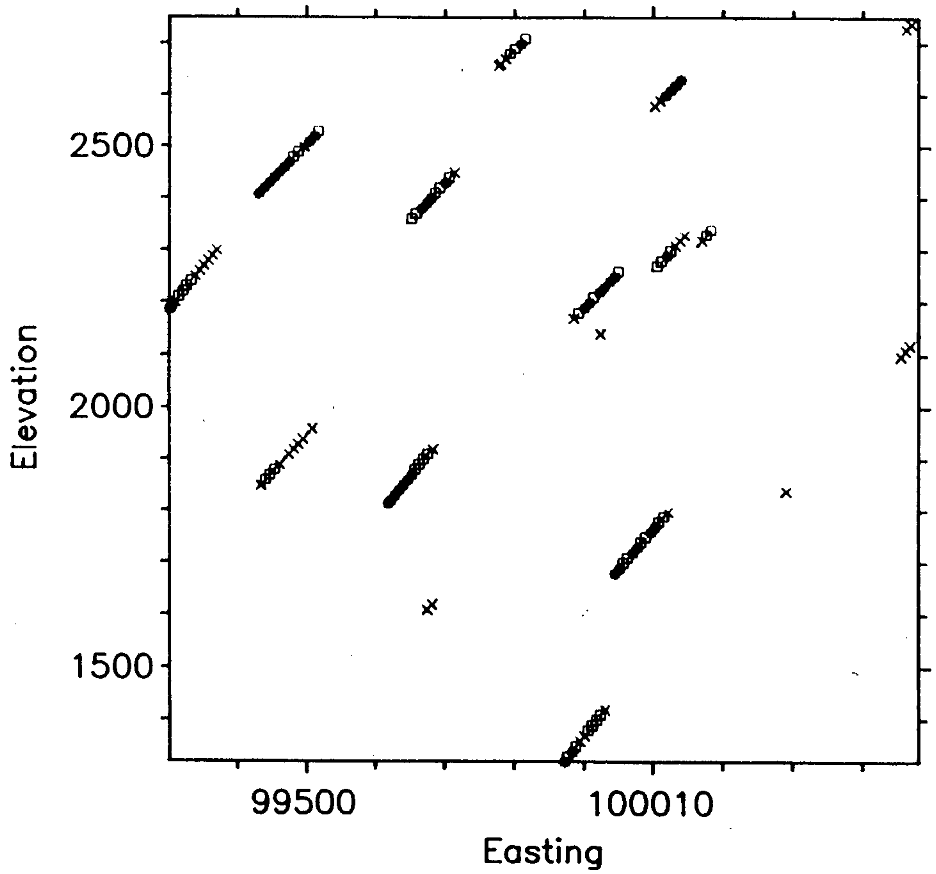


Figure 6

Williams Creek

Projected Cross–Section Looking North

Total Copper



2nd Quartile: .170 < × ≦ .510

3rd Quartile: .510 < □ ≦ 1.100

4th Quartile: 1.100 < ✱ ≦ 5.200

Figure 4(a)

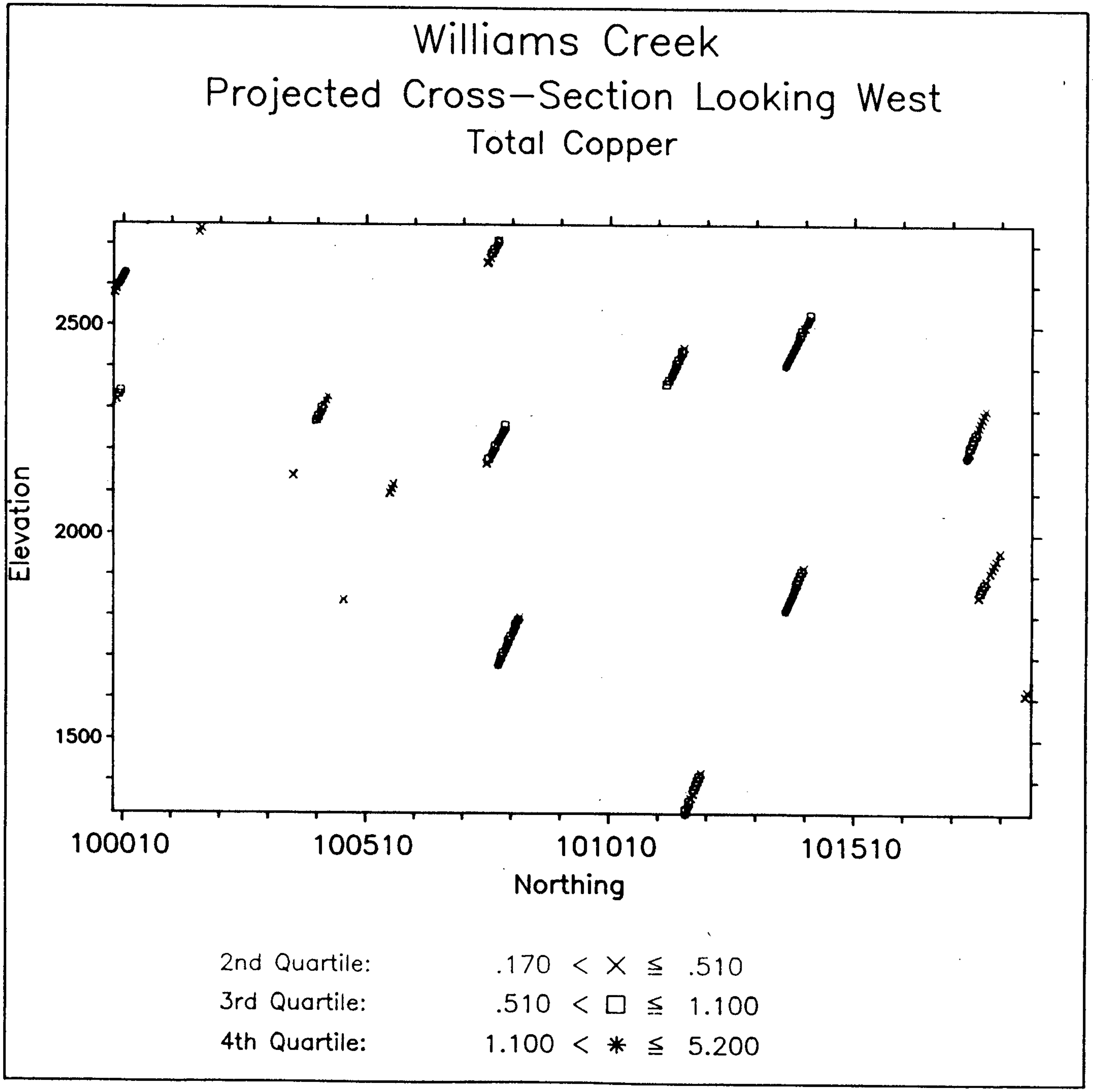
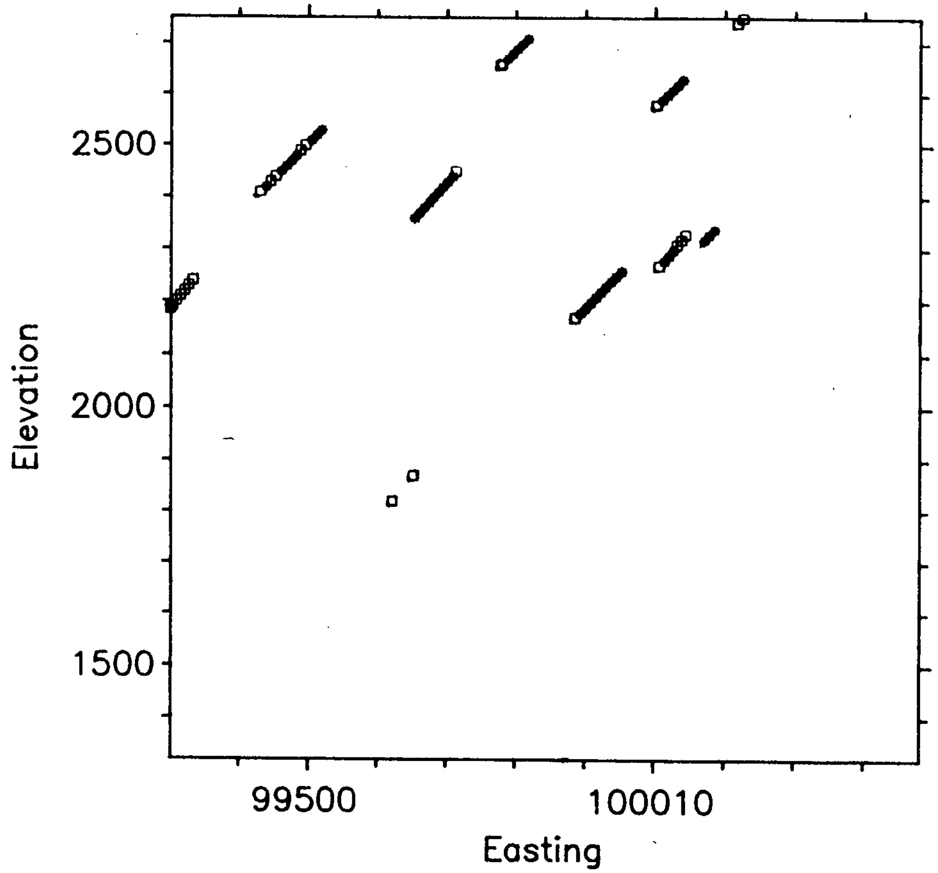

Williams Creek
Projected Cross–Section Looking West
Total Copper
Elevation
2500
2000
1500
100010
100510
101010
101510
Northing
2nd Quartile: .170 < × ≦ .510
3rd Quartile: .510 < □ ≦ 1.100
4th Quartile: 1.100 < ✱ ≦ 5.200

Figure 4(b)

Williams Creek

Projected Cross–Section Looking North

Oxide Ratio



2nd Quartile: .000 < × ≦ .000

3rd Quartile: .000 < □ ≦ 77.273

4th Quartile: 77.273 < ✱ ≦ 100.000

Figure 5(a)

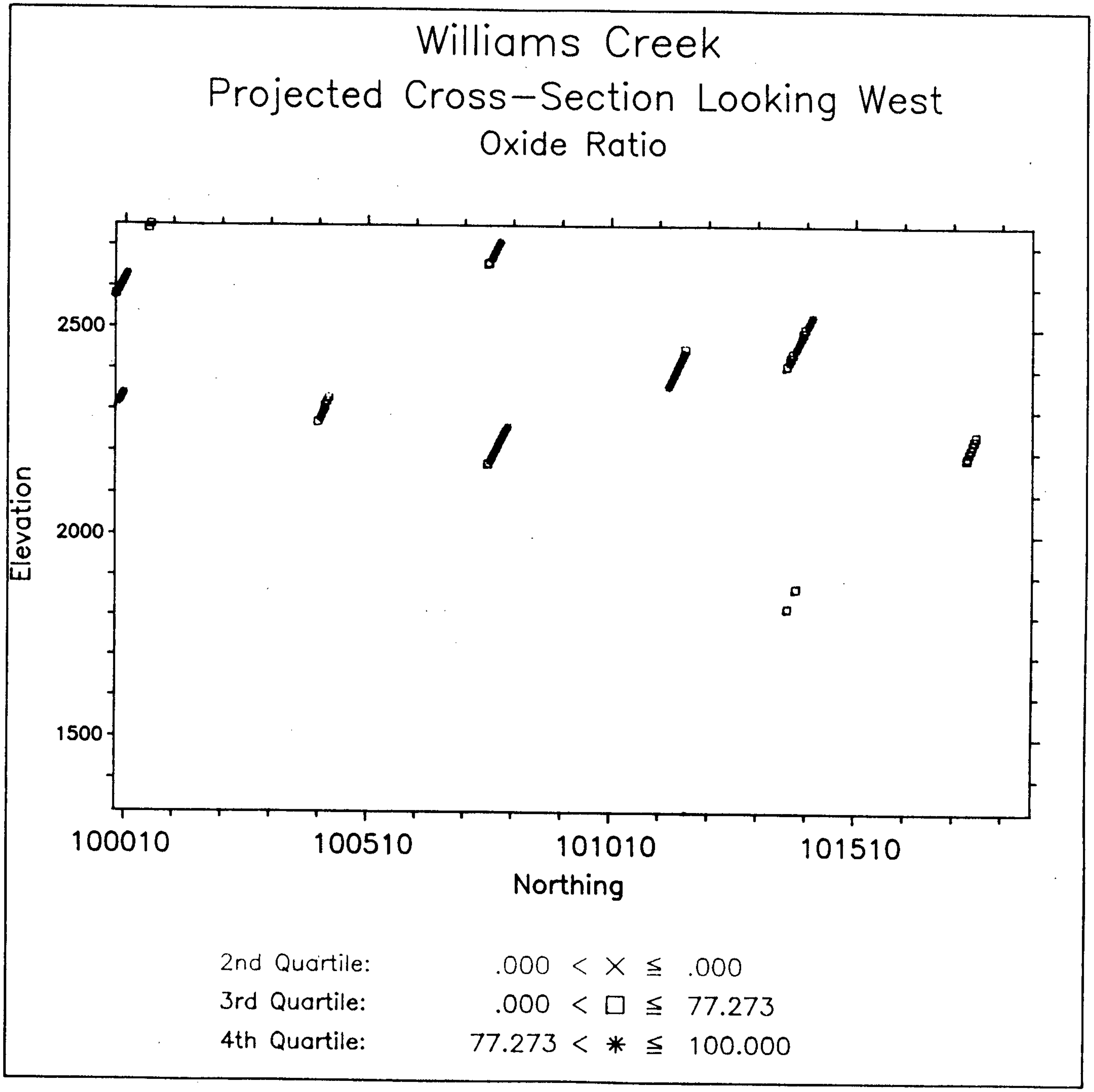

Williams Creek
Projected Cross-Section Looking West
Oxide Ratio
Elevation
2500
2000
1500
100010
100510
101010
101510
Northing
2nd Quartile: .000 < × ≦ .000
3rd Quartile: .000 < □ ≦ 77.273
4th Quartile: 77.273 < ✳ ≦ 100.000

Figure 5(b)

3.2.4 Variogram Analysis

In order to define the degree of spatial continuity in mineralization, an attempt was made to calculate directional variograms based on the composite data.

With the limited information available, it was not possible to obtain any clear definition of spatial correlational structure within the plane of the ore body. With a drillhole spacing of some 91 meters on strike, no distinct variogram was determined in the strike direction, and again sampling information was limited down dip.

Down hole variograms were computed on the 3 meter composites, and do show a distinct structure as shown in Figure 6. This variogram, calculated on logarithmically transformed total copper values, shows that there is a range of influence of some 36.6 meters, with a steady decrease in inter-sample correlation up to this distance.

It would be wrong to assume that the correlation structure seen in this direction, perpendicular to the plane of the orebody, is representative of the continuity in mineralization in other directions. Genetic control on mineralization determines the degree of variability in any particular direction, and lithologic and structural controls in the Williams Creek ore body will assist in defining any directions of preferential ore continuity.

However, the lack of any clear spatial structure within the plane of the ore body suggests that the sampling information is insufficient, and too widely spaced at this stage, to allow for any realistic grade estimation on a local basis. Further infill drilling is required, and based on the downhole variograms it is felt that a minimum hole spacing of 30 meters is required.

3.2.5 Recommendations

The sampling information available at this stage is insufficient for detailed ore body modelling or reserve estimation. Global estimates of in-situ geological reserves are possible, but no local definition would be realistic. No interpolation of grade values at this stage would be merited, and no detailed mine design should be attempted.

In order to define the ore body to an acceptable level of detail, further drilling on strike is required, and a minimum hole spacing of 30 meters is recommended at this stage. As further information becomes available, this can be re-evaluated.

It is suggested that the drilling be concentrated in the northern portion of the ore body, and that efforts be made to determine the boundary of oxidation.

3.3 REQUIREMENTS FOR DETAIL MINE DESIGN

To complete a final open pit design and associated production schedule a block model should be generated. The block model should incorporate known structural and lithological ore controls. It should be based upon a block size which can approximate the mineable selective unit. The block model should contain information regarding the following points:

- Copper grade - total and oxide
- Gold grade
- Lithologic units - rock type and alteration
- Rock quality determination - RQD, fracture frequency, etc.
- Specific gravity
- Pyrite - pyrhotite estimate for dump acid generation potential calculations

This information must come from the drill core and trench sample locations. The data should be available as computer files. The boreholes drilled for grade and geology information should be surveyed carefully and when appropriate logged for geotechnical information that may relate to the stability of the final wallslope.

Geostatistical techniques should be used to study the impact of loader selectivity at the digging face upon anticipated mine head grades.

3.4 PROPOSED MINING METHOD

The proposed mining is by conventional loader - truck methods. Alternate methods include combinations of hydraulic front shovels, in-pit crushers, conveyors and stackers. A contractor rate for mining and crushing was solicited.

CONTRACT MINING COST ESTIMATE

Mining	2,250,000 tonnes per year
Crushing	750,000 tonnes per year
Pit depth maximum	180 metres
Pit depth average	75 metres
Waste dump distance within	500 metres
Leach pad distance	500 metres
Contract mining	$1.75-$2.00/tonne
Contract crushing	$2.85/tonne

These preliminary estimates were considered high relative to the rates used by Silver Standard in their evaluation of the property. Wright have therefore produced a mining and crushing capital and operating cost estimate to evaluate the contractor rates.

The window of opportunity for mining has been defined as the period between May 15 - October 15. The production has been assumed at 750,000 tonnes at a 2:1 strip ratio. A total of 2,250,000 tonnes of material is mined annually. Waste stripping could in fact be carried on over an extended period but for this study it has been assumed that there are 130 days of operation available based upon 6 days per week operations utilizing two ten hour shifts. The employees will rotate out of the site as required on a monthly basis.

Production from the mine will be 17,300 tonnes per day or 8,650 tonnes per shift.

With the exception of the Caterpillar 992C, the equipment selected for mining at the Williams Creek property can all be moved by conventional tractor trailer or is self contained and transportable. Mobilization and set up of the entire plant can be accomplished in approximately three weeks. The 992C loaders must be erected on site.

3.4.1 Drilling and Blasting

The ore and waste will be drilled and blasted. For cost estimation purposes a bench height of 10 metres has been used. Further studies on dilution may dictate a lower bench height. For the production rates anticipated in the mine a single blasthole drill is required capable of single pass drilling 7-7/8" holes to 15 metres.

Drill holes will be sampled and assayed for ore grade control. Ore and waste contacts will be staked on the muck piles.

Pit conditions are anticipated to be relatively dry since it has been reported that diamond drill holes are losing water. The majority of the blasting operations will be carried out using AN/FO with liners used as required.

The oxidized ore zone is foliated and reported to be highly prefractured. The ore should be blasted to 100% passing 0.5 meters. All indications are that the ore will break well. A conservative powder factor has been included in the costs to ensure the plant feed is of a size suitable for size reduction in a moderate size jaw crusher.

Controlled blasting techniques will be required if maximum wall slope angles are to be attained. Presumably some form of trim blasting using the same drilling unit can be implemented. Geotechnical studies will be required to evaluate local and large scale stability factors and failure mechanisms.

A Bobcat skid steer loader has been included in the capital cost estimate to be used for blasthole stemming. This unit will also be useful around the crushing plant for cleanup around the conveyor transfer points and screening operations.

3.4.2 Loading

Loading will require two machines since ore and waste faces will be mined concurrently. Split faces will not always be available. The primary loading unit selected was a Caterpillar 992C. This machine has the capability to three pass load a 773B truck. A second loader, another Caterpillar 992C is required for loading ore. This second loader can also be used intermittently on the stockpile and around the plant area. Two large loaders represent a slight loader overcapacity but this ensures production will continue in the event of a major mechanical breakdown of the primary loading unit.

3.4.3 Hauling

Caterpillar 773B trucks were selected for mine haulage. These trucks have a high power to weight ratio, good manueverability and are transportable on provincial highways without complete dissassembly. With slight modifications to the hopper of the portable jaw crushing plant the 777B truck box size is suited to direct dumping.

With the limited information available with regards to locations of dumps and leach pads relative to the pit it is estimated that five trucks will be required initially to haul ore to the crusher, waste to the dump and product to the stockpile. The truck requirement will increase as the pit gets deeper unless the length of the operating season is extended.

3.4.4 General Support Equipment

Track mounted dozers will be required to maintain the leach pad as it is constructed in lifts and also in the mine pushing blast related backbreak, waste dump maintenance, and pit bench pioneering. Since these activities may be separated by a substantial distance a dozer has been proposed for the dump and another for the pit.

Road surfaces must be maintained properly to maximize truck speeds and to reduce operating costs. A used grader has been included in the capital cost estimate. This grader will also be used in the winter to maintain access to the plant site.

A single bay shop facility has been included in the capital cost. It is assumed that this would be a relatively simple structure on a concrete slab.

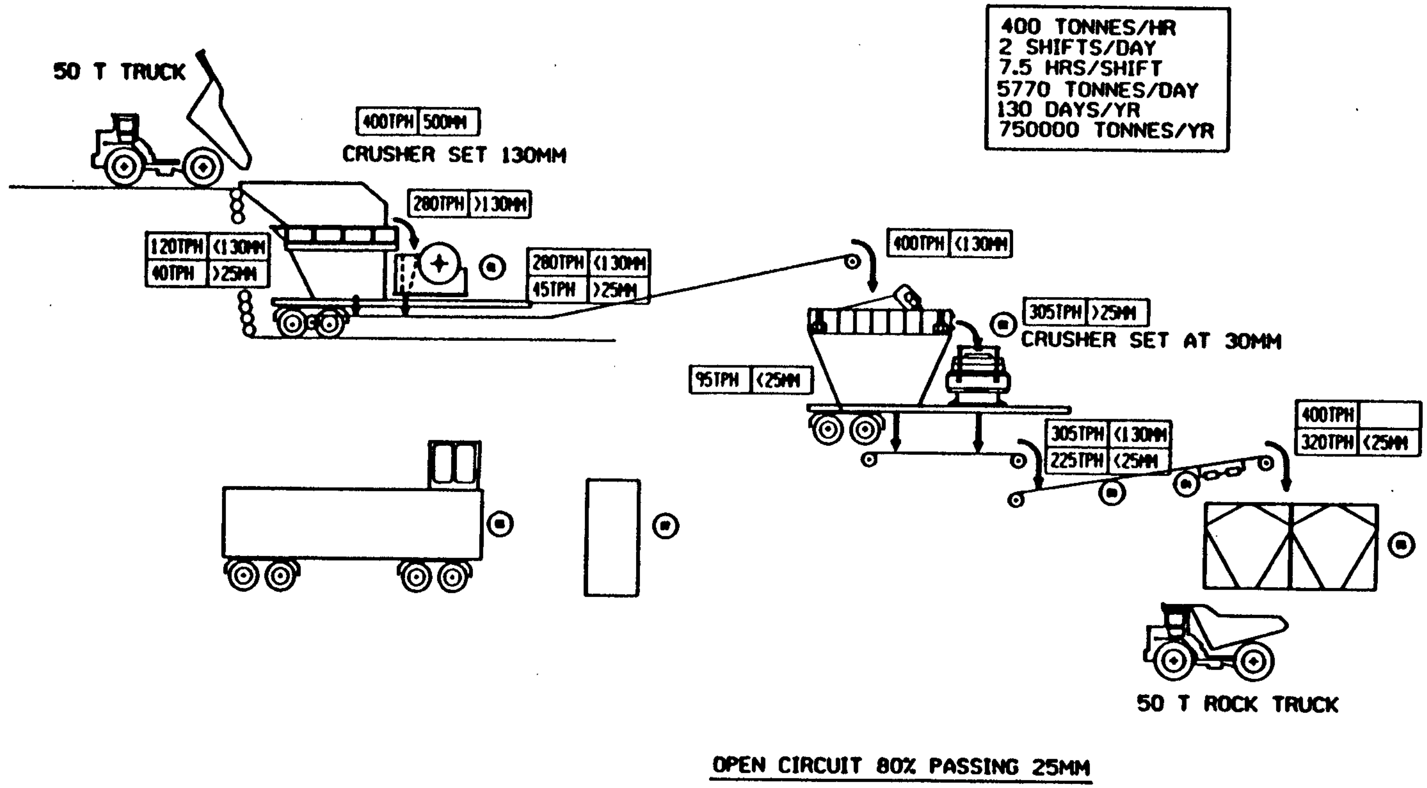
3.5 ORE CRUSHING

The metallurgical testing carried out to this point indicates copper recoveries in excess of 85% can be attained by crushing to 80% passing 25 mm.

A typical open circuit mobile crushing plant has been assembled for capital and operating cost estimation. A flowsheet of the process in shown on Figure 7. The first stage reduction is by jaw crusher. A LOKOMO C125 crusher has been selected. Maximum feed size is approximately 0.5 meters. A variable speed vibrating grizzly feeder with a spacing of 127 mm will be placed ahead of the jaw crusher. A retaining wall will be constructed at the primary crusher dump pocket. This wall will be a simple wing wall construction using vertical wide flange supports and locally available timber.

Product from the jaw crusher will pass over a 1.8 meter x 6.1 meter double deck screen. Material passing 25 mm will go to the product belt. Material retained on the screen will go directly to a 1560 OMNICONE crusher. This crusher will be equipped with coarse liners and set at approximately 29 mm. Product from the cone crusher will pass onto the final product belt.

A surge bin will be built to contain approximately 60 cubic metres of product. The bin will be elevated so as to allow a 773B truck to pass underneath. The trucks will be gravity



LEGEND

SILVER STANDARD RESOURCES INC.
WILLIAMS CREEK PROJECT

ORE CRUSHING CIRCUIT

D 1918 100

loaded by activating a hydraulically operated gate located in the bottom of the bin. An overflow spillway will be located on the side of the bin.

The crushing plant is supplied power from a 600 kW generator located in the control van. The entire plant is controlled by one operator located in the control booth of the generator - switch gear van. A second man will be required on the ground to clean up and to monitor areas out of sight of the operator. Trucks will control the loading operation. A belt weigh scale should be located on the product belt. A vertical 45,000 litre fuel storage tank will be located behind the van for the generator.

The generator set included with the crushing plant can be used for auxilliary backup power for the processing plant during the winter months.

SECTION 4

METALLURGY

SECTION 4

METALLURGY

The metallurgical test results from column and bottle roll leach tests conducted by Bacon, Donaldson and Associates Ltd. (BDA) and Coastech Research Inc. (Coastech) have been reviewed. Based on these preliminary tests the Williams Creek deposit appears to leach readily with a sulfuric acid lixiviant. If these tests are representative of the ore body in general, then a recoverable copper estimate of 85 percent (average) should be achievable in 30 days of leaching using ore crushed to 80% minus 25 mm.

4.1 TESTWORK

Two column leach tests conducted by BDA achieved 84.2 and 92.4 percent recovery in 75 and 26 days respectively. Both tests were continuing to recover copper at a rate of 0.24 percent per day when they were stopped. Minus 25 mm ore was leached in both columns with the only difference being leach solution application rate. A higher application rate significantly improves the initial leaching rate, leading to higher recoveries within a given period of time. This is very typical for oxidized copper deposits where a significant portion of the copper mineralization exists along fracture planes and faces in the host rock. Net acid consumption for the two tests averaged 51.8 and 46.9 kilograms per tonne of ore leached. A column test performed by Coastech using minus one inch ore recovered 73.7 percent of the copper contained in 22 days of leaching. Acid consumption for the test was 60 kilograms per tonne ore leached. The results of the Coastech tests indicates the potential variability in leaching performance.

Bottle roll testwork was also conducted by both BDA and Coastech. Various particle sizes were tested over 166 hours of leaching at BDA. Results are summarized in the following.

Facility	Size Fraction	% Cu Recovered	(Net) kg Acid/Tonne
Coastech	-200 mesh	92.4	[redacted]
BDA	1/4" -3/8"	80.3	23.8
BDA	1/4" -3/8"	86.6	26.5
BDA	3/4"	80.6	27.9
BDA	3/4"	82.3	26.4
BDA	1"-1.5"	74.7	29.3
BDA	1"-1.5"	66.3	28.0

In the BDA testwork, copper was still being leached at 166 hours. The Coastech recovery would probably be more indicative of ultimately leachable copper.

4.2 COPPER RECOVERY

The testwork conducted to date is very limited and utilized surface samples exclusively. What is not known is to what extent copper recoveries are affected with respect to location in the ore body. In general, recovery rates tend to decrease with depth in an ore body due to an increasing proportion of copper sulphide minerals. Overall recovery is very site specific and depends on many factors, such as:

- Host rock type(s)
- Mineralization (oxide, sulphide)
- Relative proportions of mineral types
- Geologic conditions
- Climatic conditions

Based on the physical characteristics of the drill work done to date, as described by R. Quartermain of Silver Standard, the ore body appears to be well oxidized with a very low occurence of sulphide minerals. All the material considered for leach at this time is contained in the oxidized zone with a sharply defined transitional zone between oxidized and undisturbed sulphide zones.

Allowing that some decrease in copper recovery will occur with depth and the probability of near ultimate copper recovery for surface material, an overall recovery of 85 percent should be achievable. This is based on at least 80 percent minus 25 mm ore using multiple lifts to an ultimate height of 32 meters in 4.5 meter increments. Recovery rates for sulphide copper minerals are dependent on the type and leaching conditions. A significant portion of the contained copper should be leached over time. A reasonable time frame for leaching to 85 percent should be 30 days over two cycles. An initial recovery of 80+ percent in the first leach cycle of 30 days should be achievable since the ore will be mined from the surface. Additional copper will be recovered as leaching solutions percolate through lifts of ore placed over previously leached material. Testwork on samples obtained at depth will be required to better define the potential recovery and rate of recovery for Williams Creek.

4.3 ACID CONSUMPTION

Acid consumption figures in this report are indicated as net. The net acid consumption is determined by subtracting the acid regenerated in the solvent extraction circuit from the total acid consumed as described by:

$$2\ RH + CuSO_4 \longrightarrow H_2SO_4 + R_2Cu$$

where R is the chelating organic extractant for copper. Acid required to reverse this reaction is generated in electrowinning as described by:

$$CuSO_4 + H_2O \xrightarrow{2.0\ V} Cu^{\circ} + H_2SO_4 + 1/2\ O_2$$

where Cu° is the copper metal plated.

Acid consumption described in this manner is representative of the acid that will be required to be purchased. It is therefore most relevant to examining costs.

Based on the testwork completed, acid consumptions range from 77.6 to 23.8 kilograms per tonne of ore leached. As with copper recoveries, acid consumption can also vary throughout the ore body and specifying an average consumption without benefit of testwork is highly speculative. For the purposes of this report an average consumption of 45 kilograms per tonne of ore was used. Dr. Morris Beattie of Bacon, Donaldson & Associates, who conducted the testwork, suggests [redacted]er tonne of ore appears to be a reasonable figure.

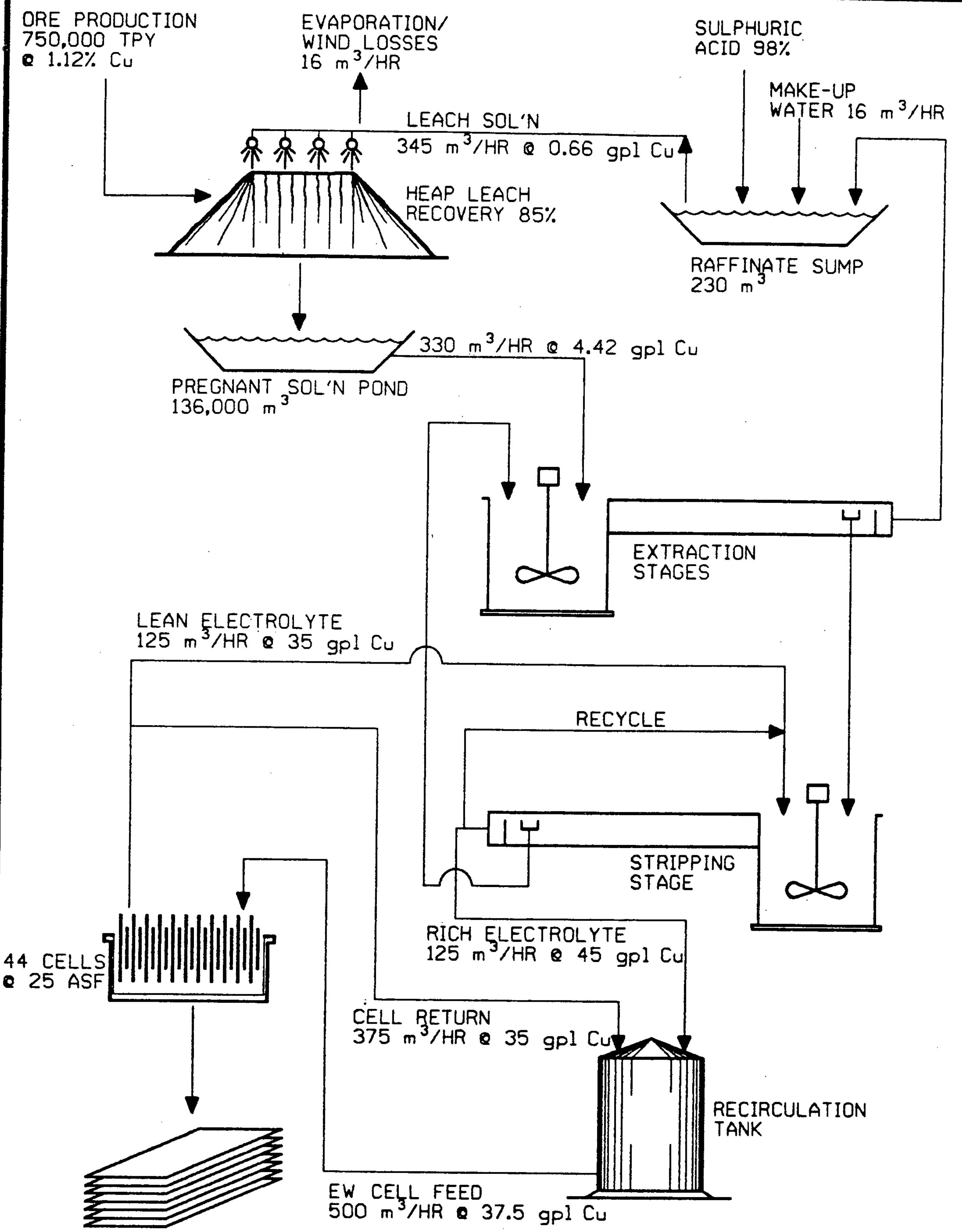
The cost of acid is the single highest operating cost in producing copper at Williams Creek (nearly 25%). It is therefore imperative that additional testwork be conducted to better discern what acid consumption rates are likely. An acid consumption of 45 kilograms per tonne ore is about average for oxidized copper ore bodies. Sulphide minerals may also play a role in acid consumption. As the mineral is leached sulphur is made availabile for conversion to sulphuric acid, usually by bacterial action.

4.4 DESIGN CONCEPTS

The basic process flowsheet that is used for this report is shown in Figure 8. Only principal flow streams and basic process equipment are shown. The plant is designed to process 330 m^3/hr of pregnant leach solution with an average copper tenor o[redacted] produce approximately [redacted]f copper cathode per day. The plant is expected to operate 8 months per year on a seasonal basis. [redacted] of cathode copper.

4.4.1 Solvent Extraction

A block diagram for solvent extraction is shown in Figure 9. The solvent extraction plant is to process 330 m^3/hr of pregnant leach solution. A series-parallel flow (extraction) arrangement is proposed to reduce capital costs and physical size of the operation. Krebs type mixer/settler units are also proposed to further reduce capital costs and physical area required. An extraction efficiency of 85 percent is expected. The entire plant will be located indoors to minimize cold weather problems and facilitate seasonal operation.



SILVER STANDARD RESOURCES INC.
WILLIAMS CREEK PROJECT
PROCESS FLOWSHEET

PLOTTED 15 MAY 1990

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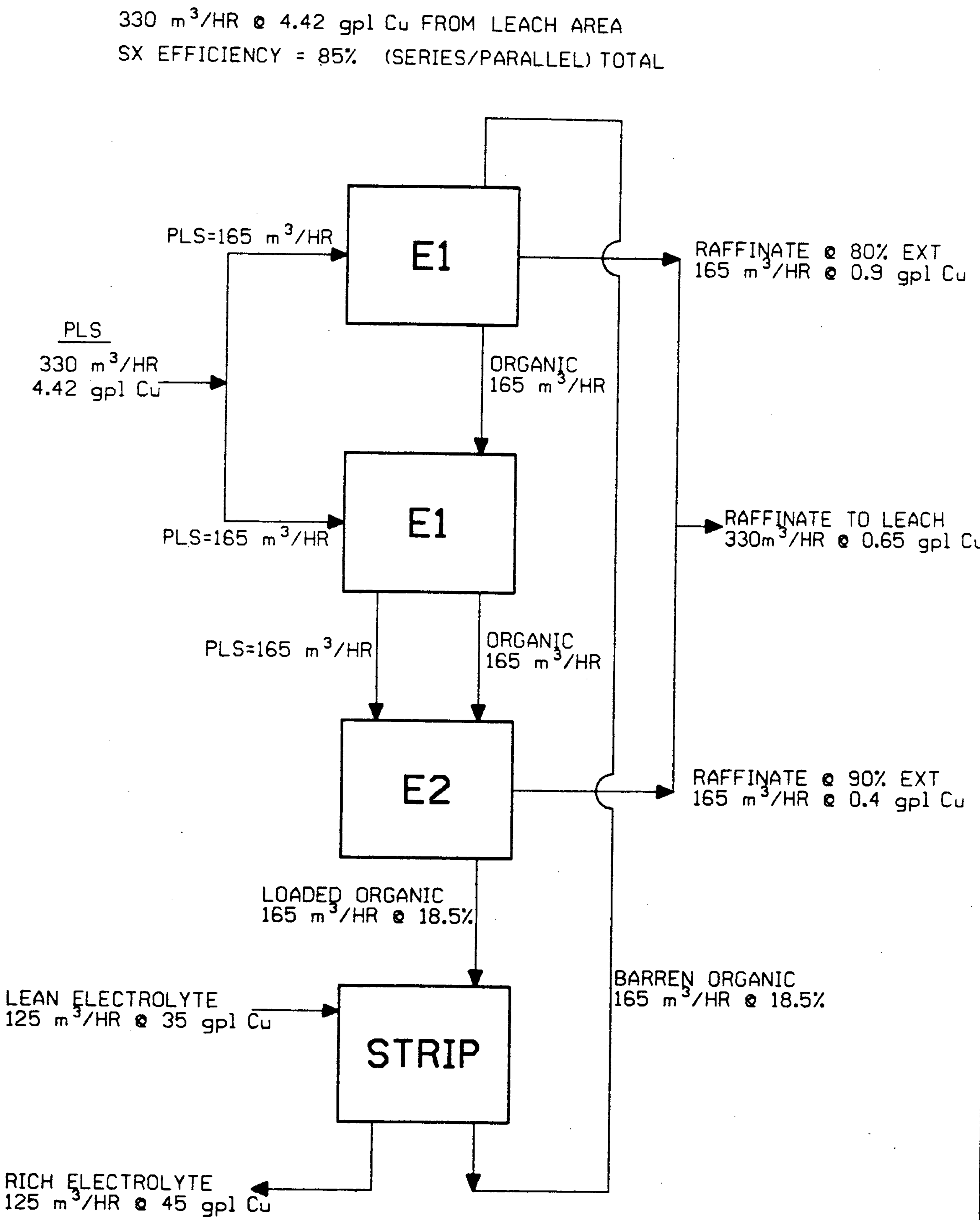
FIGURE 9



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WILLIAMS CREEK PROJECT
SOLVENT EXTRACTION

4.4.2 Electrowinning

The electrowinning plant is to utilize the ISA process of permanent stainless steel cathode mother blanks. Cells are to be Cominco style polymer concrete construction to minimize maintenance, installation and support structure costs. A very simple version of the Wemnec stripping machine is proposed. A nominal current density of 270 amps per square meter of plating area is to be used with 44 cells containing 48 cathodes and 49 insoluble lead anodes. The annual production capacity based on 8 months operating is 7,140 tonnes per year with a maximum of 8,640 tonnes per year at 325 amps per square meter. The current efficiency is expected to be 92+ percent. The solvent extraction building is to be extended to house the electrowinning plant with a fire wall separating the two plants. All process tanks will also be located in the building at an elevation below ground level to make use of gravity flow where possible.

4.4.3 Cold Weather

Extreme cold weather can significantly hamper any mining operation. Heap leach/solvent extraction/electrowinning operations are no exception. The Williams Creek property is located in central Yukon in a semi-arid region. A seasonal operating scenario is currently proposed to avoid the severe cold period from December through March. The practicality of this type of operation will have to be carefully studied. Freezing of process solutions, especially electrolyte, will make re-starting the plant very difficult. Thawing of heap piles and solution ponds will also make start-up difficult. Since most of the process equipment is located indoors minimal damage would be expected.

The use of temporary heating of process solutions (i.e. portable boiler units, etc.) may facilitate start-up after winter. Since the Williams Creek property is small and process flowstreams are not excessively large it may be more practical to consider year round operations. The following are some key points that will have to be considered for year round operations or cold weather operations:

- Heap leaching
  - Minimize surface area of pond and insulate surface
  - Heat solutions as necessary to prevent freezing
  - Use buried drip emitters for leach solution application or similar equipment
  - Bury all pipelines and/or insulate

- Solvent extraction
  - Adequate settling time for increased phase disengagement times must be included in design
  - Heat solutions as necessary to prevent freezing
  - Heat building and insulate as required
  - Determination of affects on copper transfer kenetics/proper reagent selection
  - Mixing design modifications for kinetics if required
  - Ventilate building for dangerous fumes
  - Use of explosion proof electrical components

- Electrowinning
  - Heat process solution (electrolyte) to prevent poor cathode deposits and prevent sulfate crystallization.
  - Reduce acid concentrations to prevent crystallization. This will affect solvent extraction performance.
  - Minimize heat losses with mist suppression system enhancements.

- Miscellaneous
  - Provide for road maintenance or adequate storage for consumables during road closures.
  - Camp facilities for workforce and supplies.
  - Plant equipment and enclosures designed for cold temperatures.

Some or all of these measures may be required even if a seasonal operation is maintained. Local climatic conditions will dictate the required measures and subsequent costs. The occurence of sulphide minerals may also play a role in cold weather operations. The sulphuric acid reaction created by the oxidation of free sulphur is exothermic and can provide an additional heat source.

SECTION 5

CAPITAL COSTS

SECTION 5

CAPITAL COSTS

5.1 MINING

An exploration program designed to drill the pit area on 30 meter centres will result in approximately 65 holes - 10,670 meters of drilling. A core size of "NQ" is recommended. Drilling should be coordinated to provide maximum geotechnical information while drilling for grade and structural targets.

The capital cost estimate for mining equipment is based upon new prices for the majority of the equipment. As indicated previously an important criteria in the selection process has been the mobility of the various items. As a result the fleet will closely represent what will be required by a contract mining company. If Silver Standard chooses to mine the deposit themselves this type of equipment will be readily mobilized to another similar deposit or for resale in the construction or aggregate markets. Even after mining 8 million tonnes of ore and associated waste the equipment should have excellent salvage value.

Some items not critical to operations were included as used. Sensitivity studies should be carried out to include the effect of decreased availabilities and costs associated with used equipment.

Mobile mine equipment capital cost estimates are based upon dealer budget quotes FOB Vancouver. Mining capital costs are shown in Table 1. The estimated number of highway truck loads have been indicated. These are then used to provide an estimate of the mobilization cost for the mining equipment.

Mine and crusher set-up time will be minimal due to the portable nature of the equipment. Concrete supports to elevate the crushers and screens are recommended if only to facilitate clean-up operations around conveyor transfer points.

TABLE 1 – WILLIAMS CREEK MINING CAPITAL COST

TYPE	MAKE	MODEL	STATU	UNIT	LOADS	UNIT COST	TOTAL
LOADING							
LOADER	CAT	992C	NEW	2	6	$1,050,000	$2,100,000
HAULAGE & ROADS							
HAUL TRUCKS	CAT	773	NEW	5	1	$612,000	$3,060,000
TRACTORS	CAT	D8N	NEW	2	3	$427,000	$854,000
GRADER	CAT	14G	USED	1	1	$50,000	$50,000
WATER TRUCK			USED	1	1	$50,000	$50,000
DRILLING							
ROTARY DRILL	DRILLTECH	D60K	NEW	1	2	$550,000	$550,000
SKID STEER LOADER	BOBCAT	600	NEW	1		$40,000	$40,000
BLASTING							
CAP MAGAZINE			NEW	1	0.5	$10,000	$10,000
POWDER MAGAZINE			NEW	1	0.5	$50,000	$50,000
POWDER/PUMP TRUCK			NEW	1	0.25	$40,000	$40,000
MAINTENANCE							
SERVICE TRUCK	GMC		NEW	1	0.5	$45,000	$45,000
WELDER	MILLER		NEW	1		$10,000	$10,000
MISCELLANEOUS							
SINGLE BAY SHOP			NEW	1	1	$50,000	$50,000
TOOLS			NEW				$40,000
FUEL TANKS	TIDY		NEW	1	0.5	$12,000	$12,000
PICK UP TRUCKS	GMC		NEW	3		$25,000	$75,000
AMBULANCE			USED	1	0.25	$40,000	$40,000
ENGINEERING							
SURVEY INSTRUMENTS			NEW				$10,000
COMPUTER HARDWARE			NEW				$15,000
COMPUTER SOFTWARE			NEW				$5,000
OFFICE EQUIPMENT			NEW				$25,000
MINING TOTAL							$7,131,000
CRUSHING							
JAW PLANT	NORDBERG	PC125	NEW	1	2	$325,000	$325,000
CONE/SCREEN PLANT	NORDBERG	1560	NEW	1	2	$620,000	$620,000
GENERATOR	NORDBERG	CONTROL	NEW	1	1	$300,000	$300,000
FIELD CONVEYOR	NORDBERG	36 "	NEW	1	1	$75,000	$75,000
WEIGH SCALE	RAMSEY REC		NEW	1		$10,000	$10,000
SURGE BIN	CUSTOM		NEW	1	1	$75,000	$75,000
FUEL TANKS	TIDY		NEW	1	0.5	$12,000	$12,000
TRAILER	FRUEHAUF		USED	2	2	$20,000	$40,000
RETAINING WALL	CUSTOM			1	1	$20,000	$20,000
CRUSHING TOTAL							$1,477,000
MOBILIZATION							
TRANSPORTATION				LOADS	28	$5,000	$140,000
CONCRETE & SERVICES							$30,000
SET UP COST							$50,000
MOBILIZATION TOTAL							$220,000
TOTAL MINING AND CRUSHING							$8,828,000

TABLE 1 WILLIAMS CREEK – CAPITAL REQUIREMENT (continued)

COST CENTRE		UNITS	UNIT COST	SUBTOTALS	SUBTOTALS
EXPLORATION					
DRILLING "NQ" CORE	(feet)	35000	$40.00	$1,400,000	
DRILLING "H" CORE	(feet)	2000	$75.00	$150,000	
METALLURGICAL TEST WORK				$150,000	$1,700,000
PREPRODUCTION SITE PREPARATION					
ROAD				$50,000	
STRIPPING/CLEARING				$200,000	
CAMP MOBILIZATION				$100,000	$350,000
HEAP CONSTRUCTION					
LINER DESIGN & CONSTRUCTION	(feet^2)	1500000	$2.00	$3,000,000	
POND	(gallons)	25000000		$550,000	
PIPING & IRRIGATION				$250,000	
PUMPS				$50,000	$3,850,000
ACCOMODATIONS					
TOWN HOUSING FIVE FAMILIES	(houses)	6	$50,000	$300,000	
SITE CAMP	(man)	15	$18,000	$270,000	
TOWN BUNK HOUSE	(man)	30	$10,000	$300,000	$870,000
VEHICLES					
PICK-UP TRUCKS	(units)	3	$25,000	$75,000	
FORKLIFT	(units)	1	$50,000	$50,000	
MOBILE CRANE (15 tonne)	(units)	1	$250,000	$250,000	$375,000
PROCESS PLANT					
SOLVENT EXTRACTION				$1,000,000	
ELECTROWINNING				$7,600,000	
POWER LINE				$1,600,000	
PRIMARY POWER DISTRIBUTION				$400,000	
WATER HEATER				$150,000	
BUILDINGS	(feet^2)	23000	$150	$3,450,000	$14,200,000
WORKING CAPITAL/FIRST FILL				$2,225,000	$2,225,000
PROCESS PLANT SUBTOTAL					$23,570,000
MINE OPERATIONS					
MINING EQUIPMENT				$7,131,000	
CRUSHING EQUIPMENT				$1,477,000	
MOBILIZATION				$220,000	$8,828,000
MINE OPERATIONS SUBTOTAL					$8,828,000
INVENTORY					$250,000
ENVIRONMENTAL STUDIES					$150,000
GEOTECHNICAL ENGINEERING					$150,000
SUBTOTAL CAPITAL COSTS					$32,948,000
EPCM ON PROCESS PLANT @ 12%					$2,828,000
CONTINGENCY ON PROCESS PLANT @ 20%					$4,714,000
TOTAL PROJECT COST					$39,940,000

A single bay shop is included in the estimate as well as an allowance for shop tools to include items such as hydraulic jacks, compressor, torches and the fundamental hand tools not normally provided by skilled trandemen.

The mine is allocated three 4-wheel drive pick-up trucks, a 1 ton truck for the blasting crew and an ambulance.

The Yukon Territory has no equivalent of provincial taxes. Since delivery of equipment is to the Yukon, no taxes have been included. All costs are in second quarter, 1990 Canadian dollars.

5.2 METALLURGY

The estimate of capital costs determined by Wright has been based on the following:

- Consideration of increased construction costs applicable to the Yukon.
- Information provided by Silver Standard Mines Ltd.
- Utilization of capital cost factors obtained from a feasibility study currently in progress for a southwestern U.S. leach/SX/EW facility.
- Use of modular or premanufactured designs where possible.
- Vendor budgetary quotes as required.
- A U.S. to Canadian currency conversion of 1:1.2.
- Accepted and published industry cost factors.
- Incorporation of best available proven technology.

It is recognized that construction costs in the Yukon will be significantly higher due to increased labour, materials and transportation costs. Based on the 1990 Marshall and Swift Valuation Quarterly, a differential cost multiplier of 1.75 should be applied to the above stated components when comparing and applying cost factors determined for a southwestern U.S. facility. Equipment costs, delivered to the Williams Creek site, are also expected to be higher due to increased transportation costs. An estimate of 32 percent of the capital cost of the equipment has been used to determine the additional costs incurred over similar equipment delivered to a site in the southwest U.S. The reasons for selecting a southwest U.S. facility for comparison are:

- Most cost factors for SX/EW facilities are based on southwest U.S. plants.
- Wright Engineers most recent cost data are based on a proposed SX/EW plant in Arizona. Data was collected in December 1989.
- Vendor data are more accurate and duplication of cost contingencies are minimized.

Since a significant portion of the Williams Creek facility is comprised of equipment or modular pre-manufactured components, an overall cost factor of 1.60 has been used. This factor includes the U.S. to Canadian currency conversion.

The estimated capital costs for the Williams Creek property are summarized in Table 1. A brief explanation of each cost area and major component items are presented below. All figures are in second quarter, 1990 Canadian dollars.

<u>Exploration</u>

A budget of $150,000 has been allotted for metallurgical testwork. This amount is based on a reasonable number of column and bottle roll leach tests to determine the expected leachability of the Williams Creek deposit and identify the methods and conditions for the economical production of copper. Large scale test heaps and extensive column testing are not considered necessary at this time and are not included. The level of testwork required ultimately will be determined by Silver Standard Mines Ltd. with costs adjusted accordingly.

<u>Preproduction</u>

Silver Standard has indicated that the government will upgrade the road from Carmacks to the Williams Creek turn-off at no cost and provide 75% of the funding to construct the road to the property.

Heap Leach

Ore is to be stacked in 4.5 meter lifts to a height of 32 meters. A lined area of 460,000 square meters will be required to contain the expected 8,000,000 tonnes of ore to be mined. Due to the potential clearing and grubbing requirements, the installed cost of a single 60 mil high density polyethylene liner with a fabric under liner and sized rock and sand overliner is estimated to be $21.53 per square meter. A 114,000 cubic meter pregnant leach solution collection pond is required for containment of the heap leach pile drain down for seasonal operation. A cost of $26.91 per square meter has been used for a 6 meter deep pond constructed with a double 30 mil high density polyethylene liner. Geotechnical engineering will be required for the pad site.

Accommodation

Housing costs in Carmacks are relatively low. An average three bedroom home is currently $50,000 to $60,000. Six homes have been included for management personnel. A bunkhouse for 30 persons has been included. All of these facilities are to be located in or around Carmacks.

A camp for 11 people is also included and is to be constructed at the mine site. Silver Standard believes that the balance of the employees required will be obtained from communities in the surrounding area.

Vehicles

Three 4 x 4 pick-up trucks have been included in the process plant capital costs for management and operations use. A 3.5 ton forklift for material handling including cathode production is provided. A 15 ton mobile crane is included for maintenance needs for the entire mine site.

Process Plant

Solvent extraction costs are based on Krebs mixer/settler technology using a series parallel extraction scenario. Krebs units are to be of fiberglass modular construction. A preliminary budget price from Krebs for the proposed plant excluding piping, other than interstage requirements, process tanks for solution circulation and instrumentation was $750,000 fully installed.

Electrowinning costs are based on ISA process technology with polymer concrete cells. Costs factored from a recent study are approximately $725 per annual tonne. The total allowance for the Williams Creek property for SX/EW construction is approximately $827 per annual tonne capacity. The electrowinning capacity is designed at 30 tonnes per day or an annual capacity of 10,400 tonnes.

The powerline extension to the property is approximately 40 kilometers. The expected rating is approximately 25-30 kV. A cost of $40,000 per kilometer has been used for an unsubsidized installation. Silver Standard believes that a subsidized cost could be $25,000 per kilometer.

A boiler rated at 10 MM BTU/hr has been selected to provide process solution heating requirements.

A 2,135 square meter building is required to house the process plant, control room, offices and laboratory. The estimated cost is $1,600 per square meter. Lab equipment and office furnishings are included at $125,000.

First fill and a 60 day working capital provision has also been included.

Excluded from this cost estimate is the water supply system for the process requirements. Approximately 200 gpm will be required. Water availability and quality data does not currently exist. A potential source might be Williams Creek itself. The Yukon River is the most likely source requiring a pumping/piping distribution system of 4 to 8 kilometers.

SECTION 6

OPERATING COSTS

SECTION 6

OPERATING COSTS

6.1 MINING

The mining costs have been estimated for the Williams Creek property by Tercon Contractors. Their estimate was kindly provided based upon a very limited amount of information which can be summarized as follows:

Site preparation:	
- Road upgrading	$25,000/km
- Clear and grub	$1,000/acre
	$0.25/square meter
Mining $1.75-$2.00/tonne:	
- Annual quantity mined	2,250,000 tonnes
- Maximum haul at pit crest	0.5 km
- Maximum pit depth	200 meters
- Pit depth - centre of gravity	80 meters
- Mining during a 4-5 month period	
Crushing and stockpiling $2.75/tonne:	
- Annual quantity crushed	750,000 tonnes
- Crushing jaw-cone-screen	80% <25 mm
- Loading haulage and stockpiling	

As more information and detail is developed regarding the nature of the mining operation a more accurate contract mining rate can certainly be developed. Clearly a long term contract will influence the unit cost. Mobilization expenses may not necessarily be incurred on an annual basis and the risk associated with equipment purchases will be spread over a greater volume of rock.

Wrights have prepared an estimate of mine and crusher operating costs which are summarized in Table 2.

TABLE 2

SUMMARY OF MINING AND SX-EW COSTS

Mining	$/Year	$/t Ore	$/lb Cu
Mining			
Equipment operation	2,089,000	2.79	0.13
Wages	1,101,000	1.47	0.07
Crushing			
Equipment operation	638,000	0.85	0.04
Wages	448,000	0.60	0.03
Total			
Equipment operation	2,727,000	3.64	0.17
Wages	1,549,000	2.07	0.10
Total Mining	4,276,000	5.70	0.27
SX-EW Leaching			
Labour			
Administration	556,250	0.74	0.04
Operations - Hourly	917,235	1.22	0.06
Maintenance - Hourly	251,790	0.34	0.02
Total	1,725,275	2.30	0.11
Power	848,232	1.13	0.05
Acid	3,769,920	5.03	0.24
Organics	659,736	0.88	0.04
Maintenance	549,780	0.73	0.03
Other costs	408,408	0.54	0.03
Shipping	801,108	1.07	0.05
Total	7,037,184	9.38	0.45
Total SX-EW	8,762,459	11.68	0.56
Housing and Camp	366,000	0.49	0.02
TOTAL SX-EW AND MINING	13,038,459	17.38	0.85

The annual wages have been calculated using a 25% loading factor. Mechanics have been scheduled for eight months. Operators were scheduled five months. Manpower requirements are detailed in Table 3.

Blasting consumable costs are calculated using budget quotes from Explosives Limited. Drill productivities do not include wall control measures. No allowance is made for serious water inflows to blastholes.

Equipment operating costs are shown in Table 5. Detail design of the mine is required. These operating costs provide a reasonable estimate assuming similar conditions to those described for contract mining. Equipment hours include allowance for mining crushing and stockpile construction.

6.2 PROCESS PLANT

The operating costs for the process plant have been estimated using expected loss or consumption rates typical for SX/EW facilities. The basis for copper production is 750,000 tonnes mined per year at 1.12 percent copper at 85 percent recovery. The average annual production will be 7,140 tonnes at 29.75 tonnes per day for 240 days of operation. A summary of these costs are shown in Table 2.

6.2.1 Labour

Process plant labour and administration costs are based on the manpower and wage rates shown in Table 3. These rates are typical of mining rates in Western Canada. More detailed evaluation should utilize definitive local rates.

6.2.2 Power

Yukon Power costs are approximately $0.06/kWh for commercial operations. Yukon Power has indicated to Silver Standard that they would have to start out at approximately

$0.09/kWh reducing to about $0.06/kWh with time. Yukon Power is currently charging $0.042/kWh to a zinc mining operation at Faro. For a first order estimate, $0.06/kWh is reasonable after negotiations, with an equal chance of being higher or lower at this time. Power consumption is based on 1900 kWh/tonne copper produced plus pumping and mixing costs for the leach/SX/EW plant and miscellaneous uses.

6.2.3 Acid

Acid is the single highest cost component to the Williams Creek operating costs. Acid consumption is estimated to be 45 kilograms per tonne of ore leached including an acid credit of 1.54 tonnes per tonne of cathode copper produced in electrowinning. The consumption rate has been estimated from very limited information and may or may not be representative of the ore body in general. Only additional testwork, as described in the recommendations in Section 7.2, will provide the necessary information to more accurately determine potential acid consumption rates.

The unit cost for sulphuric acid is $165 per tonne from either Fort St. John or Seattle, Washington. In the future it may be worth investigating the possibility of using acid plant waste or below specification acid. The additional trucking costs would have to be weighed against possible price concessions but may provide a means by which to reduce the overall cost of acid. Supply would also need to be investigated. For the purposes of this report $165/tonne has been used.

It is Wright's opinion that the incorporation of an acid production facility may provide a means to significantly improve the economics of the Williams Creek property. A sulphur burning acid plant capable of producing 165 tonnes per day of sulphuric acid will cost approximately $10 M - $12 M fully erected. The current price of liquid sulphur delivered to site is $138 per tonne. Approximately 3 tonnes of acid can be produced from 1 tonne of sulphur. The unit cost for acid would be reduced to approximately $46 per tonne which would reduce the unit cost to produce copper by 17¢ per pound. The net cost savings would be approximately $2.4 M per year at nominal production rates which would indicate a 4-5 year payback period. An

TABLE 3 – WILLIAMS CREEK SCHEDULE OF WAGE RATES – MINING & SX-EW

POSITION	HOURLY RATE ($/hr)	ANNUAL Salary	OWNERS COST ($/yr)/Man	MEN	TOTAL COST ($/YR)	$/T ORE	$/T MINED
ADMINISTRATION OF MINE OPERATIONS							
Mine Superintendent *		$75,000	$93,750	1	$93,750	$0.125	$0.042
Mine Foreman *		$55,000	$68,750	1	$68,750	$0.092	$0.031
Clerk		$35,000	$43,750	1	$43,750	$0.058	$0.019
Mine Geologist/Engineer *		$45,000	$56,250	1	$56,250	$0.075	$0.025
Surveyor - 6 Months		$20,000	$25,000	1	$25,000	$0.033	$0.011
Helper - 6 Months		$15,000	$18,750	1	$18,750	$0.025	$0.008
TOTAL MINING ADMINISTRATION				6	$306,250	$0.408	$0.136
MINE OPERATIONS - HOURLY							
Drillers	$18.00	$25,200	$31,500	2	$63,000	$0.084	$0.028
Truck Drivers	$17.00	$23,800	$29,750	8	$238,000	$0.317	$0.106
Wheeled Loaders	$18.00	$25,200	$31,500	4	$126,000	$0.168	$0.056
Tracked Dozers	$18.00	$25,200	$31,500	3	$94,500	$0.126	$0.042
Road Grader	$18.00	$25,200	$31,500	1	$31,500	$0.042	$0.014
Blasters	$18.00	$25,200	$31,500	1	$31,500	$0.042	$0.014
				19	$584,500	$0.779	$0.260
MINE MOBILE MAINTENANCE - HOURLY							
Shop Mechanics	$21.00	$29,400	$36,750	4	$147,000	$0.196	$0.065
Welder	$21.00	$29,400	$36,750	1	$36,750	$0.049	$0.016
Labourers	$15.00	$21,000	$26,250	1	$26,250	$0.035	$0.012
				6	$210,000	$0.280	$0.093
TOTAL MINING HOURLY				25	$794,500	$1.059	$0.353
CRUSHER OPERATIONS - HOURLY							
Crusherman I	$18.00	$25,200	$31,500	3	$94,500	$0.126	
Crusherman II	$16.00	$22,400	$28,000	1	$28,000	$0.037	
Labourers	$15.00	$21,000	$26,250	2	$52,500	$0.070	
				6	$175,000	$0.233	
CRUSHER MAINTENANCE - HOURLY							
Shop Mechanics	$21.00	$29,400	$36,750	4	$147,000	$0.196	
Welder	$21.00	$29,400	$36,750	2	$73,500	$0.098	
Labourers	$15.00	$21,000	$26,250	2	$52,500	$0.070	
				8	$273,000	$0.364	
TOTAL CRUSHING				14	$448,000	$0.597	
TOTAL MINING & CRUSHING HOURLY				39	$1,242,500	$1.657	
TOTAL MINING & CRUSHING				45	$1,548,750	$2.065	
UNIT COST $/T ORE 750000 TONNES					$2.065		
UNIT COST $/T MINED 2250000 TONNES					$0.688		
ADMINISTRATION OF SX-EW OPERATIONS							
Plant Manager *		$90,000	$112,500	1	$112,500		
Superintendent *		$70,000	$87,500	1	$87,500		
Shift Supervisors		$40,000	$50,000	4	$200,000		
Secretary/Clerk		$35,000	$43,750	1	$43,750		
Engineer *		$50,000	$62,500	1	$62,500		
Chemist		$40,000	$50,000	1	$50,000		
TOTAL WAGES ADMINISTRATION				9	$556,250	$0.742	
SX-EW OPERATIONS - HOURLY							
SX/EW	$18.00	$43,164	$53,955	8	$431,640		
Leaching	$18.00	$43,164	$53,955	7	$377,685		
Laboratory	$18.00	$43,164	$53,955	2	$107,910		
TOTAL WAGES OPERATIONS				17	$917,235	$1.223	
SX-EW MAINTENANCE - HOURLY							
Electricians/Instruments	$21.00	$50,358	$62,948	2	$125,895		
Mechanics	$21.00	$50,358	$62,948	2	$125,895		
TOTAL WAGES MAINTENANCE				4	$251,790	$0.336	
TOTAL WAGES HOURLY				21	$1,169,025	$1.559	
TOTAL WAGES HOURLY & SALARIED				30	$1,725,275	$2.300	
TOTAL MANPOWER COST MINING & SW-EW					$3,274,025	$4.365	
HOUSING COST					$60,000	$0.080	
CAMP COST					$368,800	$0.492	
TOTAL LABOUR COST					$3,702,825	$4.937	

TABLE 4 – WILLIAMS CREEK PRODUCTION BLASTHOLE DRILLING

ESTIMATE USING 7-7/8" PRODUCTION HOLES
EXPLOSIVES - AN/FO

		PRODUCTION HOLES 200 mm	COST NOT INCLUDING WAGES
DRILLING PARAMETERS			
COLLAR HOLE	minutes	2.0	
ADD STEEL	minutes	0.0	
REMOVE STEEL	minutes	0.5	
MOVE	minutes	2.0	
PENETRATION	metres/hr	22.0	
DRILLING	min./hole	37.2	
PRODUCTION	m/hr	19.3	
HOLE DEPTH	metres	12.0	
BURDEN	metres	5.2	
SPACING	metres	5.2	
BENCH HEIGHT	metres	10.0	
VOLUME	m^3	270.4	
S.G.	t/m^3	2.75	
TONNES/HOLE	tonnes	743.6	
EXPLOSIVES PARAMETERS			
CHARGE DIAMETER	mm	200	
COLLAR	metres	3.2	
LOADED LENGTH	metres	8.8	
S.G. POWDER	t/m^3	0.85	
VOLUME	m^3	0.28	
WEIGHT	kg	235.0	
POWDER FACTOR	kg/t	0.32	
POWDER FACTOR	kg/m^3	0.87	
POWDER AN/FO $100.00	$/100 kg	$234.99	
ACCESSORIES	$/hole	$20.00	
WAGES BLASTING	$/hr	$24.65	
TIME ALLOCATION	hr/hole	0.40	
WAGES BLASTING	$/hole	$9.86	
TOTAL BLASTING	$/hole	$264.85	$240.20
TOTAL BLAST COST	$/tonne	$0.356	$0.323
DRILLING COST CALCULATION			
DRILL BITS	each	$3,000	
EXPECTED LIFE	metres	1500	
COST	$/metre	$2.00	
FUEL COST	$/l	$0.280	
FUEL CONSUMPTION	l/hr	90	
FUEL COST	$/hr	$25.20	
R&M COSTS	$/hr	$40.00	
R&M/OPERATE RATIO		0.6	
WAGES R&M	$/hr	$28.00	
EFFECTIVE R&M LABOUR	$/hr	$16.80	
WAGES OPERATING	$/hr	$24.15	
TOTAL DRILL COST	$/hr	$144.83	$103.88
TOTAL DRILL COST	$/m	$7.49	
TOTAL DRILL COST	$/hole	$89.86	$64.45
TOTAL DRILL COST	$/tonne	$0.121	$0.087
COMBINED DRILLING AND BLASTING			
TOTAL DRILL & BLAST	$/tonne	$0.477	
DRILL FACTOR ESTIMATION			
DRILL FACTOR hrs/1000t		0.83439	

1.00 per kg ?

NOTE: Not enough is known about perimeter wall control blasting requirements to attach a cost to this aspect of the blasting operation.

TABLE 5 - MINE OPERATING COST ESTIMATE

EQUIPMENT	MAKE	TYPE	HRS/1000T	HOURS	$/HOUR WINC WAGES	ANNUAL COST	COST/TONNE MINED	COST/TONNE ORE
LOADING								
LOADER	CAT	992C	1.75	2625	$82.00	$215,250	0.10	0.29
HAULAGE & ROADS								
HAUL TRUCKS	CAT	773	2.50	7500	$60.00	$450,000	0.20	0.60
TRACTORS	CAT	D8N	2.00	4500	$41.00	$184,500	0.08	0.25
GRADER	CAT	14G	0.50	1125	$21.00	$23,625	0.01	0.03
DRILLING								
ROTARY DRILL	DRILLTECH	D60K	0.83	1877	$104.00	$195,156	0.09	0.26
SKID STEER LOADER	BOBCAT	600		400	$20.00	$8,000	0.00	0.01
BLASTING AGENTS						$787,500	0.35	1.05
CAP MAGAZINE								
POWDER MAGAZINE								
POWDER/PUMP TRUCK						$5,000	0.00	0.01
MAINTENANCE								
SERVICE TRUCK	GMC					$5,000	0.00	0.01
WELDER	MILLER					$2,000	0.00	0.00
MISCELLANEOUS								
SINGLE BAY SHOP						$50,000	0.02	0.07
TOOLS						$2,000	0.00	0.00
FUEL TANKS	TIDY							
PICK UP TRUCKS	GMC					$35,000	0.02	0.05
AMBULANCE						$1,000	0.00	0.00
ENGINEERING								
COMMUNICATIONS						$20,000	0.01	0.03
SUPPLIES						$30,000	0.01	0.04
CONSULTANTS						$25,000	0.01	0.03
MISCELLANEOUS						$50,000	0.02	0.07
TOTAL MINING						$2,089,031	0.93	2.79
CRUSHING								
JAW PLANT			2.50	1875	$100.00	$187,500		0.25
CONE/SCREEN PLANT			2.50	1875	$100.00	$187,500		0.25
GENERATOR			2.50	1875	$120.00	$225,000		0.30
FIELD CONVEYOR			2.50	1875	$5.00	$9,375		0.01
WEIGH SCALE			2.50	1875	$2.00	$3,750		0.01
SURGE BIN			2.50	1875	$10.00	$18,750		0.03
						$1,000		0.00
						$5,000		0.01
TOTAL CRUSHING						$637,875		0.85

additional benefit could be realized by effective use of the nearly 20 M BTU per hour waste heat produced. The impact on potential capital and/or operating costs would require a more in-depth study.

6.2.4 Organics

Organic costs are based on a loss rate of 60 ppm in both raffinate and electrolyte flowstreams. Budget prices of $0.61 per liter for Shelsol 160 and $15.5 per kilogram for Acorga M5640 delivered to site were used.

6.2.5 Maintenance

The modular plant equipment in new condition should not require excessive maintenance. Normal maintenance costs are 2 cents per pound of copper for southwest U.S. plants especially with an ISA tankhouse. A conversion factor of 1.75 for the Yukon was used for increased cost and currency exchange.

6.2.6 Shipping

Shipping costs were estimated at 17.5¢ per ton mile from the plant to Skagway. A cost of U.S. $44 per tonne for cathode shipment to Japanese ports in 4,550 to 6,360 tonne lots was used. Prices were obtained from Gearbulk.

6.2.7 Other

Remaining costs were estimated at 20 percent of the maintenance and operating supply cost (excluding acid). Due to the limited operating time only minimal electrode replacement will be required.

Owner's costs and applicable taxes have not been included in this estimate.

SECTION 7

RECOMMENDATIONS

SECTION 7

RECOMMENDATIONS

7.1 MINING

7.1.1 Drilling and Block Model Development

Diamond drilling directed towards ore reserve estimation at Williams Creek should be carried out on 30 meter centers and used to provide the maximum possible information for block model development. A block model is recommended since the results will then be available for pit limit economic analysis. The core size recommended is NQ wireline. The data logs should include information about the following:

- Total copper grade
- Oxide copper grade
- Gold grade
- Carbonate data
- Lithology
- Density
- Specific gravity
- Sulphide content
- Jointing data - fracture frequency - rock quality determination
- Major structural features

A three dimensional block model should be generated using a computer. This model should contain all the information that can be interpolated from the diamond drill hole data and surface trenches. Structural domains should be identified and the hanging wall and footwall boundaries used to define blocks within the model that should be modelled using common data. Gold and copper should be modelled separately.

An effort should be made to verify that the leaching characteristics of the material tested from surface samples can be projected to depth in the mineralized zone.

7.1.2 Mine Design

The open pit mine should be designed using a floating cone or Lerchs Grossman optimizer. A starter pit should be located to provide an initial pit which maximizes cash flow during the early years of the mine life. This is generally accomplished by locating an area with a lower than average strip ratio or higher than average grade or perhaps a combination of the two parameters. Practical push backs on the starter pit should be staged to increase the mine size to the final economic limits.

Areas of high gold values should be identified and mining should incorporate a plan to stockpile this material in a segregated area.

7.1.3 Mining Method

Substantial savings in mine operating costs can be made by eliminating the crushing plant requirement. Metallurgical testing directed towards defining the economics of recovery versus particle size will have an impact on this cost. Run of mine leaching should be explored.

Contractor estimates will be better defined as the orebody model and mining plan is completed. Long term contracts with contract mining companies will probably result in a lower unit cost. This option should be explored.

Leasing of mining equipment should be investigated as an alternative to purchasing.

The used equipment market should be investigated as a way to reduce mine capital cost.

Conveyors, stackers and self propelled crushers are alternate materials handling methods worth considering to reduce unit operating costs. The current plan incorporates loaders and trucks to provide flexibility in the mine operating environment. Conveyors if applicable will be cheaper to operate.

7.2 METALLURGY

The following recommendations apply to the metallurgical testwork program. It is necessary that these items be completed prior to further feasibility work to adequately describe process design criteria.

- [redacted] At least 4-6 "H" drill core holes must be drilled to provide material for leaching testwork. Material obtained should be as representative as possible of the ore body in general.

- Composite samples at regular intervals (3 meters) should be taken for bottle roll tests as conducted by Bacon, Donaldson & Associates. This should be conducted on all core samples.

- Material from at or near the expected "bottom" of the proposed pit should be leached under standard conditions determined for column test baseline.

- Analysis of head and tail samples for bottle roll tests should include determination of the mineralogy of the copper species.

- Various particle sizes should be tested in columns to determine the economic return of each crushing step. A good range would be 80 percent minus 100 mm material, 80 percent minus 50 mm material and 80 percent minus 25 mm material. Allow sufficient time to see if ultimate recoveries are impacted.

- Incorporation of an acid cure is typical for oxide copper leaching. This should be tested and optimized. An initial cure of 20 kg per tonne would be a starting point. The pH of the pregnant solution will determine limits.

- Optimization of flowrate per unit area. Typical ranges are 0.12 to 0.33 liters gpm per minute per square meter of area.

- Collection of pertinent climatic data for design considerations.

General recommendations for future design consideration are:

- Study economic benefits of incorporation of an acid plant complex. Sulphur supplies must be investigated, including below specification grade sources.

- Year round operations for SX/EW/leach facility. Acid plant waste heat recovery.

- Mining of sulphide material for acid and heat values. Additional copper recovery would also be realized.

- High grade mining in the first years would accelerate debt payback and reduce interest costs.

Gold recovery

- Gold mineralization appears to be of sufficient grade in a localized area to merit some testwork using neutralization/cyanide leach. This could be accomplished in practice by incorporating a segregated leach area for high gold content ore.

- Gold dissolution in acidic sulphate solutions with chloride may be possible. Gold recovery from copper leach solutions using an ion exchange resin is currently being piloted at an Arizona oxide copper leach facility.

- Ammonia leaching may still be worth investigating in a test column to determine if copper recovery can be achieved with additional time. The potential for gold recovery may merit consideration of finer crushing sizes and possibly vat leaching for copper and gold. Ammonia has also been used for leaching chalcopyrite and may allow economic recovery of sulphides.

- Thiourea leaching in an acidic environment may also be worth investigating.

It is generally expected that gold recovery is probably not economical with current technology. These recommendations are made with the idea that a limited high grade zone of 0.06 oz per ton may be treated separately.

SECTION 8

ADDENDUM

SECTION 8

ADDENDUM

At the request of Mr. Bob Quartermain of Silver Standard Resources Ltd. Wrights estimators reviewed the multipliers used to scale the capital costs for construction of an SX-EW plant in the Yukon Territory. They found no basis for changing these multipliers without more specific site related information.

Wrights have prepared an additional estimate of capital and operating costs based upon the following scenario:

1. Further testing may indicate that run of mine ore can be leached effectively. Copper recoveries are expected to be 75% as opposed to 85% used in the original study. Fragmentation of the ore will be adequate using the currently specified powder factors. The crushing plant capital cost is eliminated.

2. The electrowinning plant capital cost is reduced from $7.6 million to $6.7 million. The building size is reduced by 30 square metres.

3. The plant is operated 8 months of the year and placed in care and maintenance for 4 months. The workforce is reduced during winter months.

4. The mining equipment capital is reduced by purchasing more used mining equipment including rebuilt D8K tractors, used 773 trucks, used mobile crane, and one used 992 loader. Used mining equipment can be obtained at widely varying costs. The risk is that operating costs and availabilities become more difficult to predict and impact on the production capability of the fleet.

5. The power line costs were reduced to reflect verbal estimates from Yukon Power provided by Mr. Quartermain.

6. A 100 tonne capacity acid plant was included at a capital cost of $6 million with a resultant decrease in acid cost to $0.08/lb of copper produced.

The resultant capital cost summary is shown in Table 6. The associated operating cost summary is shown in Table 7. The wage schedule is shown in Table 8 with adjustments made to the operating periods.

In summary the operating costs are reduced to $0.63/lb of copper produced from $0.85/lb.

The total capital costs are increased to $41.7 million by the addition of an acid plant.

This exercise goes beyond the limits of the data provided upon which to base the property review but gives an indication of the impact various cost reducing methods can have upon the capital and operating costs. The basic exploration and metallurgical test work must be carried out to provide a foundation for this type of "what if?" analysis. Clearly management philosophies can have an impact on equipment, maintenance , and manpower costs. These factors are difficult to incorporate in a study of such a preliminary nature.

TABLE 6 – WILLIAMS CREEK MINING CAPITAL COST

TYPE	MAKE	MODEL	STATUS	UNIT	LOADS	UNIT COST	TOTAL
LOADING							
LOADER	CAT	992C	NEW	1	3	$1,050,000	$1,050,000
LOADER	CAT	992C	USED	1	3	$600,000	$600,000
HAULAGE & ROADS							
HAUL TRUCKS	CAT	773	USED	5	1	$350,000	$1,750,000
TRACTORS	CAT	D8K	USED	2	3	$150,000	$300,000
GRADER	CAT	14G	USED	1	1	$50,000	$50,000
WATER TRUCK			USED	1	1	$50,000	$50,000
DRILLING							
ROTARY DRILL	DRILLTECH	D60K	NEW	1	2	$550,000	$550,000
SKID STEER LOADER	BOBCAT	600	NEW	1		$30,000	$30,000
BLASTING							
CAP MAGAZINE			NEW	1	0.5	$10,000	$10,000
POWDER MAGAZINE			NEW	1	0.5	$50,000	$50,000
POWDER/PUMP TRUCK			NEW	1	0.25	$40,000	$40,000
MAINTENANCE							
SERVICE TRUCK	GMC		NEW	1	0.5	$45,000	$45,000
WELDER	MILLER		NEW	1		$10,000	$10,000
MISCELLANEOUS							
SINGLE BAY SHOP			NEW	1	1	$50,000	$50,000
TOOLS			NEW				$40,000
FUEL TANKS	TIDY		NEW	1	0.5	$12,000	$12,000
PICK UP TRUCKS	GMC		NEW	3		$25,000	$75,000
AMBULANCE			USED	1	0.25	$40,000	$40,000
ENGINEERING							
SURVEY INSTRUMENTS			NEW				$10,000
COMPUTER HARDWARE			NEW				$15,000
COMPUTER SOFTWARE			NEW				$5,000
OFFICE EQUIPMENT			NEW				$25,000
MINING TOTAL							$4,807,000
MOBILIZATION					18	$5,000	$90,000
TOTAL MINING							$4,897,000

TABLE 6 WILLIAMS CREEK – CAPITAL REQUIREMENT (continued)

COST CENTRE		UNITS	UNIT COST	SUBTOTALS	SUBTOTALS
EXPLORATION					
DRILLING "NQ" CORE	(feet)	35000	$40.00	$1,400,000	
DRILLING "H" CORE	(feet)	2000	$75.00	$150,000	
METALLURGICAL TEST WORK				$150,000	$1,700,000
PREPRODUCTION SITE PREPARATION					
ROAD				$50,000	
STRIPPING/CLEARING				$200,000	
CAMP MOBILIZATION				$100,000	$350,000
HEAP CONSTRUCTION					
LINER DESIGN & CONSTRUCTION	(feet^2)	1500000	$2.00	$3,000,000	
POND	(gallons)	25000000		$550,000	
PIPING & IRRIGATION				$250,000	
PUMPS				$50,000	$3,850,000
ACCOMODATIONS					
TOWN HOUSING FIVE FAMILIES	(houses)	6	$50,000	$300,000	
SITE CAMP	(man)	15	$18,000	$270,000	
TOWN BUNK HOUSE	(man)	30	$10,000	$300,000	$870,000
VEHICLES					
PICK-UP TRUCKS	(units)	3	$18,000	$54,000	
FORKLIFT (used)	(units)	1	$50,000	$50,000	
MOBILE CRANE (used) (15 tonne)	(units)	1	$125,000	$125,000	$229,000
PROCESS PLANT					
ACID PLANT 100 TPD CAPACITY				$6,000,000	
SOLVENT EXTRACTION				$1,000,000	
ELECTROWINNING				$6,700,000	
POWER LINE				$1,000,000	
PRIMARY POWER DISTRIBUTION				$400,000	
WATER HEATER				$150,000	
BUILDINGS	(feet^2)	22700	$150	$3,405,000	$18,655,000
WORKING CAPITAL/FIRST FILL				$2,225,000	$2,225,000
PROCESS PLANT SUBTOTAL					$27,879,000
MINE OPERATIONS					
MINING EQUIPMENT				$4,807,000	
MOBILIZATION				$90,000	$4,897,000
MINE OPERATIONS SUBTOTAL					$4,897,000
INVENTORY					$250,000
ENVIRONMENTAL STUDIES					$150,000
GEOTECHNICAL ENGINEERING					$150,000
SUBTOTAL CAPITAL COSTS					$33,326,000
EPCM @ 12%					$3,345,000
CONTINGENCY @ 20%					$5,574,000
TOTAL PROJECT COST					$41,695,000

TABLE 7 – SUMMARY OF MINING & SX-EW COSTS RUN OF MINE

RECOVERY AT 75%

	$/YR	$/T ORE	$/lb Cu
MINING			
EQUIPMENT OPERATION	$2,089,000	2.79	0.15
WAGES	$1,101,000	1.47	0.08
TOTAL MINING	$3,190,000	4.25	0.23
SX-EW LEACHING			
LABOUR			
ADMINISTRATION	$541,700	0.72	0.04
OPERATIONS - HOURLY	$611,500	0.82	0.04
MAINTENANCE - HOURLY	$167,900	0.22	0.01
TOTAL	$1,321,100	1.76	0.10
POWER	$694,260	0.93	0.05
ACID	$1,110,816	1.48	0.08
ORGANICS	$555,408	0.74	0.04
MAINTENANCE	$416,556	0.56	0.03
OTHER COSTS	$408,408	0.54	0.03
SHIPPING	$694,260	0.93	0.05
TOTAL	$3,879,708	5.17	0.28
TOTAL SX-EW	$5,200,808	6.93	0.37
CAMP & HOUSING	$322,000	0.43	0.02
TOTAL SX-EW & MINING	$8,390,808	11.19	0.63

TABLE 8 – WILLIAMS CREEK SCHEDULE OF WAGE RATES – MINING & SX-EW

POSITION	HOURLY RATE ($/hr)	ANNUAL Salary	OWNERS COST ($/yr)/Man	MEN	TOTAL COST ($/YR)	$/T ORE	$/T MINED
ADMINISTRATION OF MINE OPERATIONS							
Mine Superintendent *		$75,000	$93,750	1	$93,750	$0.125	$0.042
Mine Foreman *		$55,000	$68,750	1	$68,750	$0.092	$0.031
Clerk		$35,000	$43,750	1	$43,750	$0.058	$0.019
Mine Geologist/Engineer *		$45,000	$56,250	1	$56,250	$0.075	$0.025
Surveyor - 6 Months		$20,000	$25,000	1	$25,000	$0.033	$0.011
Helper - 6 Months		$15,000	$18,750	1	$18,750	$0.025	$0.008
TOTAL MINING ADMINISTRATION				6	$306,250	$0.408	$0.136
MINE OPERATIONS - HOURLY							
Drillers	$18.00	$25,200	$31,500	2	$63,000	$0.084	$0.028
Truck Drivers	$17.00	$23,800	$29,750	8	$238,000	$0.317	$0.106
Wheeled Loaders	$18.00	$25,200	$31,500	4	$126,000	$0.168	$0.056
Tracked Dozers	$18.00	$25,200	$31,500	3	$94,500	$0.126	$0.042
Road Grader	$18.00	$25,200	$31,500	1	$31,500	$0.042	$0.014
Blasters	$18.00	$25,200	$31,500	1	$31,500	$0.042	$0.014
				19	$584,500	$0.779	$0.260
MINE MOBILE MAINTENANCE - HOURLY							
Shop Mechanics	$21.00	$29,400	$36,750	4	$147,000	$0.196	$0.065
Welder	$21.00	$29,400	$36,750	1	$36,750	$0.049	$0.016
Labourers	$15.00	$21,000	$26,250	1	$26,250	$0.035	$0.012
				6	$210,000	$0.280	$0.093
TOTAL MINING HOURLY				25	$794,500	$1.059	$0.353
TOTAL MINING				31	$1,100,750	$1.468	
UNIT COST $/T ORE 750000 TONNES					$1.468		
UNIT COST $/T MINED 2250000 TONNES					$0.489		
ADMINISTRATION OF SX-EW OPERATIONS							
Plant Manager *		$90,000	$112,500	1	$112,500		
Superintendent *		$70,000	$87,500	1	$87,500		
Shift Supervisors		$40,000	$50,000	4	$200,000		
Secretary/Clerk		$35,000	$29,167	1	$29,167		
Engineer *		$50,000	$62,500	1	$62,500		
Chemist		$40,000	$50,000	1	$50,000		
TOTAL WAGES ADMINISTRATION				9	$541,667	$0.722	
SX-EW OPERATIONS - HOURLY							
SX/EW	$18.00	$43,164	$35,970	8	$287,760		
Leaching	$18.00	$43,164	$35,970	7	$251,790		
Laboratory	$18.00	$43,164	$35,970	2	$71,940		
TOTAL WAGES OPERATIONS				17	$611,490	$0.815	
SX-EW MAINTENANCE - HOURLY							
Electricians/Instruments	$21.00	$50,358	$41,965	2	$83,930		
Mechanics	$21.00	$50,358	$41,965	2	$83,930		
TOTAL WAGES MAINTENANCE				4	$167,860	$0.224	
TOTAL WAGES HOURLY				21	$779,350	$1.039	
TOTAL WAGES HOURLY & SALARIED				30	$1,321,017	$1.761	
TOTAL MANPOWER COST MINING & SW-EW					$2,421,767	$3.229	
HOUSING COST					$60,000	$0.080	
CAMP COST					$261,600	$0.349	
TOTAL LABOUR COST					$2,743,367	$3.658	

TABLE 9 – MINE OPERATING COST ESTIMATE

EQUIPMENT	MAKE	TYPE	HRS/1000T	HOURS	$/HOUR INC WAGES	ANNUAL COST	COST/TONNE MINED	COST/TONNE ORE
LOADING								
LOADER	CAT	992C	1.75	2625	$82.00	$215,250	0.10	0.29
HAULAGE & ROADS								
HAUL TRUCKS	CAT	773	2.50	7500	$60.00	$450,000	0.20	0.60
TRACTORS	CAT	D8K	2.00	4500	$41.00	$184,500	0.08	0.25
GRADER	CAT	14G	0.50	1125	$21.00	$23,625	0.01	0.03
DRILLING								
ROTARY DRILL	DRILLTECH	D60K	0.83	1877	$104.00	$195,156	0.09	0.26
SKID STEER LOADER	BOBCAT	600		400	$20.00	$8,000	0.00	0.01
BLASTING AGENTS						$787,500	0.35	1.05
CAP MAGAZINE								
POWDER MAGAZINE								
POWDER/PUMP TRUCK						$5,000	0.00	0.01
MAINTENANCE								
SERVICE TRUCK	GMC					$5,000	0.00	0.01
WELDER	MILLER					$2,000	0.00	0.00
MISCELLANEOUS								
SINGLE BAY SHOP						$50,000	0.02	0.07
TOOLS						$2,000	0.00	0.00
FUEL TANKS	TIDY							
PICK UP TRUCKS	GMC					$35,000	0.02	0.05
AMBULANCE						$1,000	0.00	0.00
ENGINEERING								
COMMUNICATIONS						$20,000	0.01	0.03
SUPPLIES						$30,000	0.01	0.04
CONSULTANTS						$25,000	0.01	0.03
MISCELLANEOUS						$50,000	0.02	0.07
TOTAL MINING						$2,089,031	0.93	2.79

Williams Creek - Drill Hole Assays
Other Zones
(Note: Number prefixing hole number designates Zone number)

Sample No	Interval (ft)	Total Cu%	Oxide Cu%	Sulfide Cu%(by subtr)	MoS_2 (conv)	Ag/oz/T	Au/oz/T
			Drill Hole 2-1				
			Unmineralized				
			Drill Hole 2-2				
1451	59-69	0.18					
1452	69-79	0.41	- - - - - - Not Assayed - - - - - - -				
1453	79-89	0.63					
			Drill Hole 2-3				
			Unmineralized				
			Drill Hole 5-1				
10302	227-230	0.02					
10303	230-240	0.04					
10304	240-250	0.06					
10305	250-260	0.08					
10306	260-270	0.06	- - - - - - Not Assayed - - - - - - -				
10307	270-280	0.04					
10308	280-290	0.13					
10309	290-300	0.08					
10310	300-310	1.38					
10311	310-319.5	0.05					

Williams Creek - Drill Hole Assays
No. 1 Zone and No. 4 Zone.

Drill Hole 3 - 0+00N

Sample No	Interval (ft)	Total Cu%	Oxide Cu%	Sulfide Cu% (by subtr)	MoS_2 (conv)	Ag/oz/T	Au/oz/T
4901	200-210	1.28	1.13	0.15	.002	0.30	0.01
4902	210-220	1.45	1.44	0.01	.002	0.28	0.02
4903	220-230	1.70	1.69	0.01	.002	0.38	0.02
4904	230-240	1.70	1.69	0.01	.003	0.30	0.02
4905	240-250	1.43	1.43	Nil	.003	0.20	0.01
4906	250-260	0.26	0.16	0.10	.002	Tr	0.01
4907	260-270	0.20	0.13	0.07	.002	Tr	Tr
4908	270-280	0.14	- - - - - - - Not Assayed - - - - - - -				
4918	33-43	0.08	0.06	0.02	Tr	0.06	Tr
4919	43-53	0.04	0.02	0.02	Tr	0.10	Tr
4920	53-63	0.02	0.01	0.01	Tr	0.12	Tr

Drill Hole 4 - 8+00N

Sample No	Interval (ft)	Total Cu%	Oxide Cu%	Sulfide Cu% (by subtr)	MoS_2 (conv)	Ag/oz/T	Au/oz/T
4909	130-134.5	0.04	- - - - - - - Not Assayed - - - - - - -				
4910	140-150	0.70	0.61	0.09	.005	Tr	0.01
4911	150-160	1.38	1.37	0.01	.008	0.30	0.03
4912	160-170	0.73	0.73	Nil	.003	0.08	0.02
4913	170-180	1.08	1.07	0.01	.010	0.28	0.02
4914	180-190	0.63	0.61	0.02	.002	0.20	0.02
4915	190-200	0.35	0.29	0.06	Tr	0.14	0.01
4916	200-210	0.27	0.19	0.08	Tr	0.06	0.01
4917	210-220	0.37	0.24	0.13	Tr	0.04	Tr

Drill Hole 5 - 16+00N

Sample No	Interval (ft)	Total Cu%	Oxide Cu%	Sulfide Cu% (by subtr)	MoS_2 (conv)	Ag/oz/T	Au/oz/T
4920A	310-315.2	0.02	0.01	0.01	Tr	0.02	Tr
4921	315.2-320	1.00	0.92	0.08	.002	0.18	0.02
4922	320-330	0.78	0.68	0.10	Tr	0.16	0.02
4923	330-340	1.58	1.44	0.14	.002	0.36	0.03
4924	340-350	1.35	1.09	0.26	.003	0.02	0.01
4925	350-360	0.38	0.23	0.15	.003	0.06	0.005
4176	360-370	0.26	0.15	0.11	Tr	0.16	0.01
4176A	370-380	0.73	0.53	0.20	Tr	0.16	0.02
4177	380-390	0.55	0.46	0.09	Tr	0.12	0.02
4177A	390-400	1.15	1.09	0.06	.008	0.18	0.02
4178	400-410	2.28	2.26	0.02	.003	0.28	0.01
4179	410-420	3.53	3.45	0.08	.005	0.30	0.02
4180	420-430	5.78	5.70	0.08	.082	0.98	0.07
4180A	430-440	2.70	1.61	1.09	.012	0.48	0.05
4181	440-450	4.28	2.44	1.84	.035	1.08	0.07
4181A	450-460	1.00	0.81	0.19	.010	0.28	0.07
4182	460-470	3.08	2.85	0.23	.070	0.68	0.07
4183	470-480	1.95	1.58	0.37	.030	0.50	0.06
4183A	480-490	0.13	- - - - - - - Not Assayed - - - - - - -				
4184	490-503	0.15	- - - - - - - Not Assayed - - - - - - -				

Drill Hole 6 - 8+00N
(under Hole 4)

Sample No	Interval (ft)	Total Cu%	Oxide Cu%	Sulfide Cu% (by subtr)	MoS_2 (conv)	Ag/oz/T	Au/oz/T
4185	580-585	0.60	.49	0.11	Tr	.08	.005
4186	585-590	0.65	.61	0.04	.005	.16	.01
4186A	590-600	0.60	.44	0.16	.002	.08	Tr
4187	600-610	0.98	.92	0.06	.002	.10	.01
4188	610-620	1.45	1.41	0.04	Tr	.10	.005
4189	620-630	2.00	1.99	0.01	.003	.12	.01
4190	630-640	2.70	2.70	Nil	.005	.20	.01
4191	640-650	2.05	2.03	0.02	.005	.26	.01
4192	650-660	0.80	.73	0.07	.003	.10	.01
4193	660-670	1.18	1.07	0.11	.005	.24	.01
4194	760-680	1.90	1.88	0.02	.022	.24	.01
4195	680-690	2.30	2.30	Nil	.010	.28	.01
4196	690-700	0.98	.84	0.14	.003	.14	.005
4197	700-712	0.11	.04	0.07	.002	.02	.005
4198	712-718	0.39	- - - - - - - Not Assayed - - - - - - -				

Drill Hole [illegible] - 16+00N
(under Hole 5)

Sample No	Interval (ft)	Total Cu%	Oxide Cu%	Sulfide Cu% (by subtr)	MoS_2 (conv)	Ag/oz/T	Au/oz/T
4199	934.3-940	0.17	NA		)		
4200	940-950	0.03	NA	N	)	- - - Not Assayed - - -	
4151	950-960	0.21	NA	O	)		
4152	960-970	0.33	NA	T	)		
4153	970-980	1.15	NA		.013	0.14	0.01
4154	980-990	0.58	NA		.012	0.14	0.02
4155	990-1000	0.70	NA	A	.007	0.14	0.02
4156	1000-1010	0.75	NA	P	.008	0.12	0.01
4157	1010-1020	0.68	NA	P	.027	0.18	0.01
4158	1020-1030	1.85	0.01	L	.008	0.18	0.02
4159	1030-1040	1.10	NA	I	.007	0.10	0.02
4160	1040-1050	1.53	NA	C	.003	0.10	0.02
4161	1050-1060	1.40	NA	A	.013	0.14	0.02
4162	1060-1070	1.38	NA	B	Tr	0.20	0.02
4163	1070-1080	1.70	NA	L	.010	0.28	0.02
4164	1080-1090	2.20	0.01	E	.002	0.28	0.03
4165	1090-1097	1.33	NA		.002	0.06	0.02

(NA - Not Assayed)

Drill Hole 8 - 24+00N
No mineralization in zone

Drill Hole 9 - 0+00N
(under Hole 3)

4173	20-25	0.02			Tr	
4174	25-30	1.13			Tr	
4175	30-40	0.09			Tr	
	(20'-40' is the west edge of the No. 4 zone)					
4166	530-535	0.08	NA	-	Tr	
4167	535-540	0.68	0.53	0.15	.002)	Composite Assay
4168	540-550	0.95	0.78	0.17	.002)	535' - 570'
4169	550-560	1.13	0.93	0.20	.003)	0.04 oz Ag/ton
4170	560-570	0.55	0.45	0.10	.002)	0.01 oz Au/ton
4171	570-580	0.06	NA	-	Tr	0.002% MoS_2
4172	580-590	0.03	NA	-	Tr	

Drill Hole 10 - 12+00N
(Hole A on X-Section)

1401	380-385	0.20	NA	-	
1402	385-390	0.53	0.44	0.09	Composite Assays
1403	390-400	0.49	0.36	0.13	385' - 450'
1404	400-410	0.83	0.74	0.09	0.13 oz Ag/ton
1405	410-420	1.55	1.54	0.01	0.03 oz Au/ton
1406	420-430	0.83	0.80	0.03	0.003% MoS_2
1407	430-440	0.65	0.54	0.11	
1408	440-450	0.70	0.64	0.06	450' - 505'
1409	450-460	1.40	1.35	0.05	0.13 oz Ag/ton
1410	460-470	3.90	3.68	0.22	0.03 oz Au/ton
1411	470-480	2.28	2.25	0.03	0.007% MoS_2
1412	480-490	1.20	1.10	0.10	
1413	490-500	1.00	0.98	0.02	
1414	500-505	1.05	0.93	0.12	
1415	505-510	0.12	NA	-	

Drill Hole 11 - 12+00N
(Hole C on section)

Sample No	Interval (ft)	Total Cu%	Oxide Cu%	Sulfide Cu% (by subtr)	MoS_2 (conv)	Ag/oz/T	Au/oz/T
1416	1435-1446.5	.34					
1417	1446.5-1465	.02	N				
1418	1465-1475	1.10	O				
1419	1475-1485	.43	T				
1420	1485-1495	.63					
1421	1495-1505	.98	A				
1422	1505-1515	.48	P				
1423	1515-1525	.49	P				
1424	1525-1535	.13	L				
1425	1535-1545	.49	I				
1426	1545-1555	.65	C				
1427	1555-1565	1.90	A				
1428	1565-1575	0.88	B				
1429	1575-1587.2	1.30	L				
			E				

Composite Assays
1465' - 1555'
0.02 oz Ag/ton
0.01 oz Au/ton

1555' - 1587.2'
0.04 oz Ag/ton
0.01 oz Au/ton

Drill Hole 12 - 19+00N

Sample No	Interval (ft)	Total Cu%	Oxide Cu%	Sulfide Cu% (by subtr)	MoS_2 (conv)	Ag/oz/T	Au/oz/T
1430	458-468	.24	N A		N		A
1431	527-537	.19	O S		O		S
1432	537-547	.23	T S		T		S
1433	547-557	.20	A				A
1434	557-567	.63	Y				Y
1435	567-577	.26	E				E
1436	577-587	.33	D				D
1437	587-597	.17					
1438	597-607	.32	....	....	....	....	....
1439	607-617	.73	0.54	0.19)			
1440	617-627	.60	0.32	0.28)			
1441	627-637	.50	0.31	0.19)	.002	0.02	0.01
1442	637-647	.63	0.46	0.17)			
1443	647-657	.30	0.13	0.17)			
1444	657-667	.70	0.38	0.32)	.003	0.02	0.01
1445	667-679	2.05	1.13	0.92		0.18	0.02

Drill Hole 13 - 19+00N

Sample No	Interval (ft)	Total Cu%	
1469	849-859	.01	
1470	859-869	.02	
1471	869-879	.01	
1472	879-889	.07	
1473	889-899	.22	
1474	899-909	.04	
1475	909-919	.02	
10301	919-925	.02	
1454	925-935	.30	
1455	935-945	.11	
1456	945-955	.35	- - - - - - Not Assayed - - - - - - -
1457	955-965	.50	
1458	965-975	.25	
1459	975-985	.15	
1460	985-995	.41	
1461	995-1005	.13	
1462	1005-1015	.13	
1463	1015-1025	.41	
1464	1025-1035	.65	
1465	1035-1045	.68	
1466	1045-1055	.65	
1467	1055-1065	.68	
1468	1065-1073	.34	

Sample No	Interval (ft)	Total Cu%	Oxide Cu%	Sulfide Cu% (by subtr)	MoS_2 (conv)	Ag/oz/T	Au/oz/T
10312	1243-1250	0.11					
10313	1250-1260	0.03					
10314	1260-1270	0.01					
10315	1270-1280	0.10					
10316	1280-1290	0.22					
10317	1290-1300	0.60					
10318	1300-1310	0.03					

- - - - - - - - Not Assayed - - - - - - - -

(Note: traces of mineralization found from 1310' to 1788')

Drill Hole 15 - 8+00N

Sample No	Interval (ft)	Total Cu%		
10338	1034.4-1040	0.63	N	)
10339	1040-1050	0.46	O	)
10340	1050-1060	0.55	T	)
10341	1060-1070	0.90		)
10342	1070-1080	0.85	O	)
10343	1080-1090	1.45	X	)
10344	1090-1100	1.40	I	)
10345	1100-1110	0.53	D	)
10346	1110-1120	1.35	I	)
10347	1120-1130	0.88	Z	)
10348	1130-1140	1.33	E	)
10349	1140-1150	1.93	D	)
10350	1150-1160	1.15		)
10251	1160-1170	1.05		)
10252	1170-1180	0.73	N	)
10253	1180-1190	1.70	O	)
10254	1190-1200	1.60	T	)
10255	1200-1203	0.75		)
10258	1248-1250	0.16	O	)
10259	1250-1260	0.17	X	)
10260	1260-1270	0.06	I	)
10261	1270-1280	0.07	D	)
10262	1280-1290	0.14	I	)
10263	1290-1300	0.06	Z	)
10264	1300-1310	0.05	E	)
10265	1310-1317	0.04	D	)

Composite Assay (10338–10342)
1034.4 to 1080
0.10 oz. Ag/ton, 0.01 oz. Au/ton and 0.008 % MoS_2

Composite Assay (10343–10251)
1080-1170
0.02 oz Ag/ton, 0.01 oz. Au/ton and 0.012% MoS_2

NOT ASSAYED (10252–10255)

(Note: 1203' to 1248' unmineralized

- - - NOT ASSAYED - - - (10258–10265)

Drill Hole 16 - 4+00N

No. 4 Zone

Sample No	Interval (ft)	Total Cu%	Oxide Cu%	Sulfide Cu% (by subtr)	MoS_2 (conv)	Ag/oz/T	Au/oz/T
10207	168-174	7.05	0.95	6.10	NA	0.88	0.01

No. 1 Zone

Sample No	Interval (ft)	Total Cu%	Oxide Cu%	Sulfide Cu% (by subtr)
10295	518-520	0.12	0.06	0.06
10296	520-530	0.19	0.10	0.09
10297	530-540	0.21	0.14	0.07
10298	540-550	0.30	0.23	0.07
10299	550-560	0.30	0.23	0.07
10300	560-570	0.53	0.39	0.14
10201	570-580	1.73	1.58	0.15
10202	580-590	1.55	1.46	0.09
10203	590-594	0.85	0.59	0.26
10204	594-600	0.65	0.41	0.24
10205	600-604	0.75	0.35	0.40
10206	641-646	0.55	0.49	0.06
10216	720-730	0.03	)	
10217	730-740	0.03	)	
10218	740-750	0.03	)	
10219	750-760	0.03	)	
10220	760-770	0.35	)	

- - - - Not Assayed- - - -

ORE RESERVES WILLIAMS CREEK PROPERTY No. 1 ZONE - 1971

FOOTWALL PLUS HANGING WALL

CATEGORY	BLOCK No	TONS (000)	TOTAL Cu %	SULFIDE Cu %	OXIDE Cu %	Ag oz/ton	Au oz/ton
LEACHED OXIDE (1)	1	968		NA	NA	NA	NA
	2	291	0.69	NA	NA	NA	NA
	3	1,011	0.86	NA	NA	NA	NA
	4	710	0.54	NA	NA	NA	NA
total no cutoff		2980	0.73				
total 0.6% Cu cutoff		2529	0.805				
total 0.8% Cu cutoff		1026	0.10				
ENRICHED OXIDE	5	183	1.56	NA	NA	NA	NA
	6	314	1.51	0.04	1.47	0.29	0.016
	7	308	0.85	0.15	0.70	0.04	0.0
	8	1339	0.69	0.13	0.56	0.10	0.02
	9	760	0.69	0.05	0.63	0.138	0.015
	10	1562	1.46	0.06	1.40	0.165	0.008
	11	1717	1.32	0.08	1.24	0.130	0.030
	12	2000	1.89	0.30	1.59	0.319	0.034
	13	1046	0.83	0.34	0.49	0.045	0.012
total no cutoff		9229	26	0.16	1.10	assumed	
total 0.6% Cu cutoff (2)		7065	1.52	0.17	1.35	not	
total 0.8% Cu cutoff (2)		4574	1.84	0.15	1.69	recoverable	
SULFIDE	14	3264	1.09	1.09	NIL	04	0.0
	15	2117	1.25	1.25	NIL	0.16	0.018
	16	1359	0.80	0.80	NIL	0.025	0.01
total no cutoff		6740	1.08	1.08	NIL	0.077	0.0125
total 0.6% Cu cutoff		6740	1.08	1.08	NIL	0.077	0.0125
total 0.8% Cu cutoff		4016	1.35	1.35	NIL	0.071	0.0135

FOOTWALL

CATEGORY	BLOCK No	TONS (000)	TOTAL Cu %	SULFIDE Cu %	OXIDE Cu %	Ag oz/ton	Au oz/ton
LEACHED OXIDE (1)	1	496	0.63	NA	NA	NA	NA
	2	153	0.47	NA	NA	NA	NA
	3	416	1.20	NA	NA	NA	NA
	4	412	0.78	NA	NA	NA	NA
total no cutoff		1477	0.785				
total 0.6% Cu cutoff		1324	0.79				
total 0.8% Cu cutoff		416	1.20				
ENRICHED OXIDE	5	NIL					
	6	NIL					
	7	NIL					
	8	532	1.27	0.19	1.08	0.10	0.02
	9	380	0.41	0.07	0.34	0.11	0.01
	10	1304	1.634	0.047	1.587	0.178	0.009
	11	816	1.87	0.09	1.78	0.13	0.03
	12	1045	2.86	0.44	2.42	0.474	0.05
	13	327	1.44	0.65	0.79	0.1	0.015
total no cutoff		4404	1.80	0.21	1.59	assumed	
total 0.6% Cu cutoff (2)		4024	1.93	0.225	1.705	.228 not	.027
total 0.8% Cu cutoff (2)		3697	1.98	0.19	1.79	recoverable	
SULFIDE	14	2380	1.24	1.24	NIL	0.02	0.01
	15	1282	1.57	1.57	NIL	0.172	0.02
	16	354	1.35	1.35	NIL	0.04	0.01
total no cutoff		4016	1.35	1.35	NIL	0.071	0.0135
total 0.6% Cu cutoff		4016	1.35	1.35	NIL	0.071	0.0135
total 0.8% Cu cutoff		4016	1.35	1.35	NIL	0.071	0.0135

HANGING WALL

CATEGORY	BLOCK No	TONS (000)	TOTAL Cu %	SULFIDE Cu %	OXIDE Cu %	Ag oz/ton	Au oz/ton
LEACHED OXIDE (1)	1	472	1.05	NA	NA	NA	NA
	2	138	0.93	NA	NA	NA	NA
	3	595	0.62	NA	NA	NA	NA
	4	298	0.22	NA	NA	NA	NA
total no cutoff		1503	0.702				
total 0.6% Cu cutoff		1205	0.823				
total 0.8% Cu cutoff		610	1.02				
ENRICHED OXIDE	5	183	1.56	NA	NA	NA	NA
	6	314	1.51	0.04	1.47	0.29	0.016
	7	308	0.85	0.15	0.70	0.04	0.01
	8	807	0.30	0.09	0.21	NA	NA
	9	380	0.97	0.03	0.94	0.165	0.02
	10	258	0.61	0.11	0.50	0.10	0.003
	11	901	0.82	0.07	0.75	0.13	0.03
	12	955	0.82	0.15	0.67	0.15	0.017
	13	719	0.55	0.20	0.35	0.02	0.01
total no cutoff		4825	768	0.108	0.66	assumed	
total 0.6% Cu cutoff (2)		3041	960	0.09	0.87	not	
total 0.8% Cu cutoff (2)		877	29	0.03	1.26	recoverable	
SULFIDE	14	884	0.68	0.68	NIL	0.10	0.01
	15	835	0.77	0.77	NIL	0.144	0.014
	16	1005	0.60	0.60	NIL	0.02	0.01
total no cutoff		2724	0.678	0.678	NIL	0.084	0.0113
total 0.6% Cu cutoff		2724	0.678	0.678	NIL	0.084	0.0113
total 0.8% Cu cutoff		NIL	NIL	NIL	NIL	NIL	NIL

NOTE (1) • Copper assay of leached oxide zone assumed to be 100% oxide copper
(2) •• Cutoff grade was applied to oxide assay rather than total copper assay

TOTAL RESERVES
(using only oxide copper assays for oxide zone)

CATEGORY	FOOTWALL PLUS HANGING WALL TONS (000)	FOOTWALL PLUS HANGING WALL COPPER %	FOOTWALL TONS (000)	FOOTWALL COPPER %	HANGING WALL TONS (000)	HANGING WALL COPPER %
no cutoff	18949	[illegible]	9887	1.37	9052	0.67
0.6% cutoff	16334	[illegible]	9364	1.43	6970	0.78
0.8% cutoff	9616	485	8129	.545	1487	[illegible]

DIAMOND DRILL HOLE ASSAYS

HOLE No.	WALL	TRUE WIDTH (FT)	TOTAL COPPER %	OXIDE COPPER %	SULFIDE COPPER %	SILVER oz/ton	GOLD oz/ton
1 & 2	1970 X-RAY HOLES, ASSAYS NOT RELIABLE						
3	HW	40	1.51	1.47	.04	0.29	0.016
	FW	24	0.20	0.13	.07	NA	NA
4	HW	33	0.97	0.94	.03	0.165	0.02
	FW	33	0.41	0.34	.07	0.1	0.01
5	HW	64	0.82	0.67	0.15	0.15	0.017
	FW	70	2.86	2.42	0.44	0.474	0.05
6	HW	17	0.61	0.50	0.11	0.10	.003
	FW	86	1.634	1.587	0.047	0.178	.009
7	HW	41	0.77	NIL	0.77	0.164	0.016
	FW	63	1.57	NIL	1.57	0.172	0.021
8	UNMINERALIZED						
9	HW	28	0.85	0.70	0.15	0.04	0.01
	FW	16	0.05	NA	NA	NA	NA
10	HW	57	0.82	0.75	0.07	0.13	0.03
	FW	48	1.87	1.78	0.09	0.13	0.03
11	HW	71	0.60	NIL	0.60	0.02	0.01
	FW	25	1.35	NIL	1.35	0.04	0.01
12	HW	44	0.55	0.35	0.20	0.02	0.01
	FW	20	1.44	0.79	0.65	0.10	0.015
13	HW	89	0.274	NOT ASSAYED			
	FW	36	0.67				
14	HW	TRACES OF MINERALIZATION					
	FW	16	0.41	NOT ASSAYED			
15	HW	39	0.60	NIL	0.60	0.10	0.01
	FW	105	1.24	NIL	1.24	0.02	0.01
16	HW	44	0.30	0.21	0.09	NOT ASSAYED	
	FW	29	1.27	1.08	0.19	0.02	0.01
17	HW	32	0.18	NIL	0.19	NOT ASSAYED	
	FW	TRACES OF MINERALIZATION					

LEGEND

Assays from surface trenching

○ Hole location on footwall of zone

NA - Not assayed

HW - Hanging wall

FW - Footwall

NOTE - Base elevation of 2800' obtained by aneroid barometer assuming Yukon River at Carmacks at 1550' above sea level

FIG W 25

ARCHER, CATHRO & ASSOCIATES LTD

FOOTWALL LONGITUDINAL SECTION
ORE RESERVES No. 1 ZONE

WILLIAMS CREEK PROPERTY

DAWSON RANGE JOINT VENTURE

SCALE IN FEET (VERTICAL AND HORIZONTAL)

200 0 200 400 800

To accompany report dated Jan 1977