Landore Resources Limited

~

BAM Project

Metallurgical Report
Phase 1

26 January 2019

Job 1801

Allard Engineering Services LLC
6080 W Peregrine Way
Tucson, AZ USA

TABLE OF CONTENTS

APPENDICES

LIST OF FIGURES

1.0 SUMMARY:

Whole core from the Landore's BAM Project at the Junior Lake Concession was delivered to Base Metals Laboratory in Kamloops, BC for Phase 1 metallurgical testing. Phase 1 testing consisted of basic testing of various potential processing unit operations for the BAM composite material. This testing included gravity separation, flotation, and cyanide leaching for both agitated leaching and heap leaching.

1.1 Conclusions:

Phase 1 metallurgical testwork has provided the following insights:

1. Significant free gold is present in the composite tested.
2. High extractions of gold are achievable with grinding, gravity separation (+65%), and cyanide leaching (+95%) with overall extractions around 98%.
3. Cyanide and lime consumptions were low in the leaching tests.
4. Liberation of gold particles is reduced in size-fractions above 300 microns.
5. Flotation of the BAM composite achieved reasonable extractions of gold, albeit at low concentrate grades.
6. Heap leaching with fine crushing and agglomeration can achieve acceptable extractions of gold (±84% at test conditions).
7. In fine-ground material, gold occurs predominantly as coarse liberated particles and as attachments and inclusions in chlorite and cobaltite. Minor quantities of gold are associated with tellurides.
8. Cyanide leach extractions of gold at sizes below 300 microns do not appear to be dependent on particle size.
9. Sparging agitation leach tests with oxygen improves the extraction of gold over sparging with air.
10. Reasonable variations of cyanide concentration and percent solids do not appear to influence gold extractions from agitated leach tests at typical grind sizes.
11. Agitation leach pulps are amenable to Carbon in Leach/Carbon in Pulp operations.
12. Silver and copper species are present in the ore but only partially dissolved by cyanide.

1.2 Processing Options:

The current testing indicates that various processing options are possible with the attendant capital and operating costs and gold extractions. Capital and operating costs are not part of this exercise however, the following estimates of extraction and fundamental reagent consumptions should allow reasonable recommendations to be made.

1.2.1 Grind-Gravity-Agitation Leach:

A conventional mill with grinding to 100% passing 212 microns with a robust gravity circuit on the classifying cyclone underflow followed by a CIP or CIL agitation leach would likely achieve the following:

Gold Extraction	±98% (not accounting for soluble losses)
Cyanide Consumption:	±0.40 kg/t
Lime Consumption	±0.60 kg/t as CaO

It should be understood that the concentrate developed in this phase of the testing was insufficient to determine the reagent consumptions of concentrate leaching, which will be in addition to these values.

1.2.2 Grind-Gravity-Flotation:

A conventional mill with grinding to 100% passing 212 microns with a robust gravity circuit on the classifying cyclone underflow and a flotation circuit would likely achieve the following:

Gold Extraction ±88.5%

It should be noted that the concentrates in this option did not achieve a gold grade sufficient to be sold to a smelter. Additional downstream processing and the attendant reduction in extraction would have to be considered if this option were to be pursued.

1.2.3 Coarse Crushed Heap Leach:

Crushing to 100% minus 31 mm and leaching in a heap would likely achieve the following:

Gold Extraction:	±52% (based on a 4% discount for lab vs. field extractions)
Cyanide Consumption:	0.06 kg/t
Lime Consumption:	0.06 kg/t as CaO

1.2.4 Fine Crushed Heap Leach:

Crushing to 100% minus 6 mm, agglomeration with cement, and leaching in a heap would likely achieve the following:

Gold Extraction	±84% (based on a 4% discount for lab vs. field extractions)
Cyanide Consumption:	0.12 kg/t
Lime Consumption:	0.07 kg/t as CaO
Cement Consumption:	3-4 kg/t (assumed requirement for agglomeration)

This does not account for the difficulty in operating a heap leach with the site conditions: lower extractions and weather curtailments of production are likely.

1.3 Recommendations:

Based on the current level of testing, the following recommendations are made:

1. Eliminate flotation as a viable unit operation.
2. Develop further understanding of a milling/gravity/leach circuit. This would require investigation of the following:
    a. Variability of the deposit for physical properties and amenability to the flowsheet.
    b. HPGR (High Pressure Grinding Roll) crushing as a way to increase the gold liberation.
    c. Rheology/filtration/thickening tests on ground and cyanided pulps.
    d. Gravity-recoverable gold tests to establish a baseline.
    e. Cyanide destruction in slurried tails.
    f. Gold loading tests on activated carbon from pulps.
    g. Cyanidation of gravity concentrate.
3. Develop additional understanding on the viability of heap leaching. This would require investigation of the following:
    a. Cold temperature leach extraction rate.
    b. Heap stability testing to determine agglomeration requirements and allowable heap height.
    c. HPGR crushing as a way to increase gold extraction rate.
    d. Effect of cyanide cure on extraction rate.
    e. Effect of application rate on leach extraction.
    f. Gold loading tests on activated carbon in leach solutions.

2.0 SAMPLES:

Samples were obtained in the field by Landore staff. These samples represented whole core intervals from the BAM deposit. Intervals were selected to represent a single metallurgical type based on geological interpretation. Detailed descriptions of the intervals are included in Appendix A, attached to this report, and a summary is included in Figure 2.1. The two samples were combined to create a bulk composite for testing.

Met	Drill Hole	Depth, meters From	To	Meters Total	Weight,kg
1	0418-653	138	196	58	465.38
2	0418-654	124	172	48	388.12

Figure 2.1 – Field Sampling Data.

The drill-hole locations for the samples are identified in Figure 2.2.

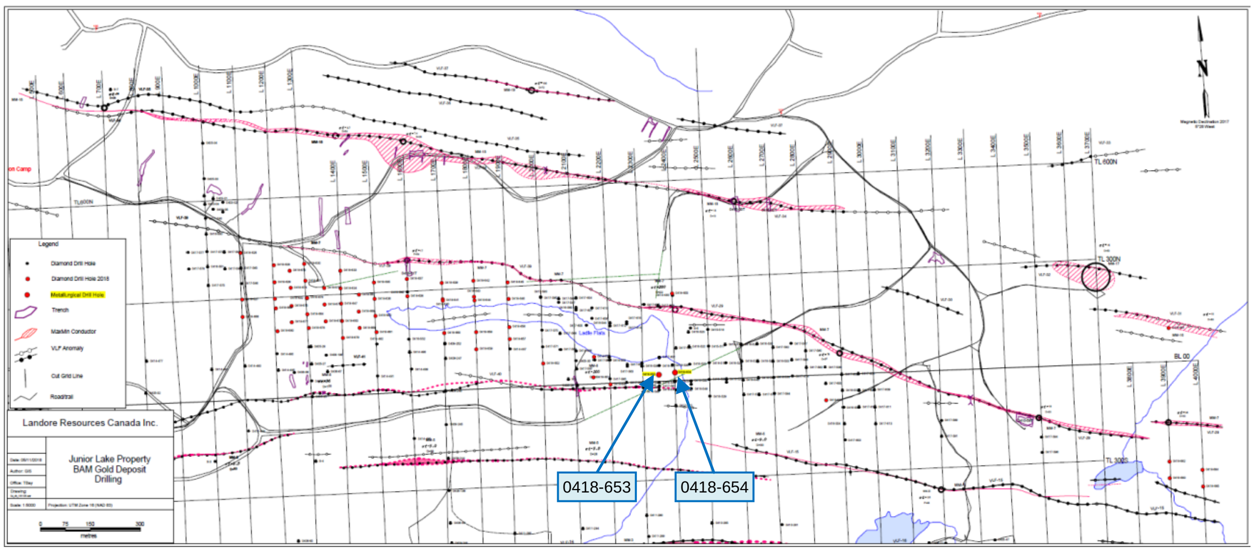


Figure 2.2 – Field Sampling Location Plan.

2.1 Head Assays:

Eight splits were taken from the master composite and assayed for gold as shown in Figure 2.3. The remaining elements were assayed in duplicate.

	Au, ppm	Ag, ppm	Cu, %	Fe, %	S, %
Hd 1	0.88	1	0.013	2.32	0.13
Hd 2	0.53	1	0.013	2.30	0.11
Hd 3	0.97	--	--	--	--
Hd 4	0.60	--	--	--	--
Hd 5	1.24	--	--	--	--
Hd 6	2.10	--	--	--	--
Hd 7	0.58	--	--	--	--
Hd 8	0.66	--	--	--	--
Average	0.95	1	0.013	2.31	0.12

Figure 2.3 – Head Assays.

As can be seen, the variations in the gold value are significant indicating the presence of free gold. Other significant observations are that there is minimal silver and the copper content is two orders of magnitude higher than the gold.

An additional head assay using a screened metallic assay technique was used to help account for the coarse gold. The coarsest fraction is separated and assayed in its entirety to avoid the nugget effect. The results of the assay are included in Figure 2.4.

	Mass – g	Au – g/tonne
+106	22.1	7.34
-106 Cut 1	--	0.63
-106 Cut 2	--	0.73
-106 Average	908	0.68
Total	930	0.84

Figure 2.4 – Screened Metallics Head Assay.

Whole rock analysis was conducted on the sample. The analysis is included in Appendix B. Nothing unusual was noted in the analysis.

2.2 Head Screen Analysis:

Bulk sample was crushed to make two master composites, one crushed to 100% passing 31.5 mm particle size and the second to 100% passing 6.3 mm using conventional crushing equipment. Screening and assaying by size of these composites resulted in the particle size distribution in Figure 2.5 and the distributions in Figure 2.6. Detailed test sheets are included in Appendix B. It should be noted that the value corresponding to the 10 micron size is the "pan fraction" which is indeterminate in mean particle size. This approximation allows the fines data to be represented on the graph. The coarse material was observed to be very platy.

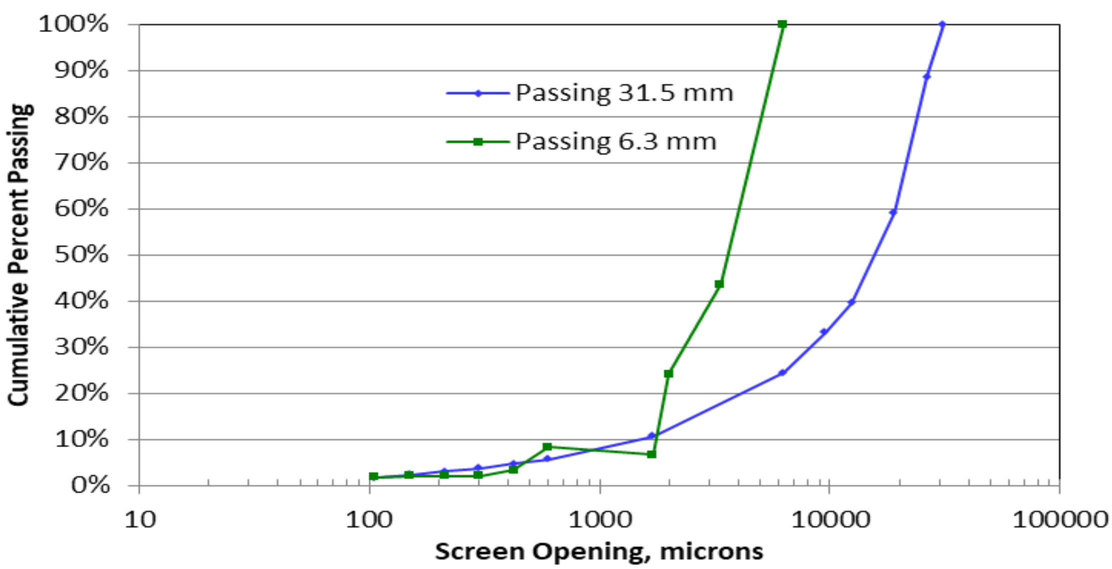


Figure 2.5 – Master Composite Particle Size Distributions.

Figure 2.6 shows the distribution of weight, gold and copper by size for the coarse (100% passing 31.5 mm) master composite.

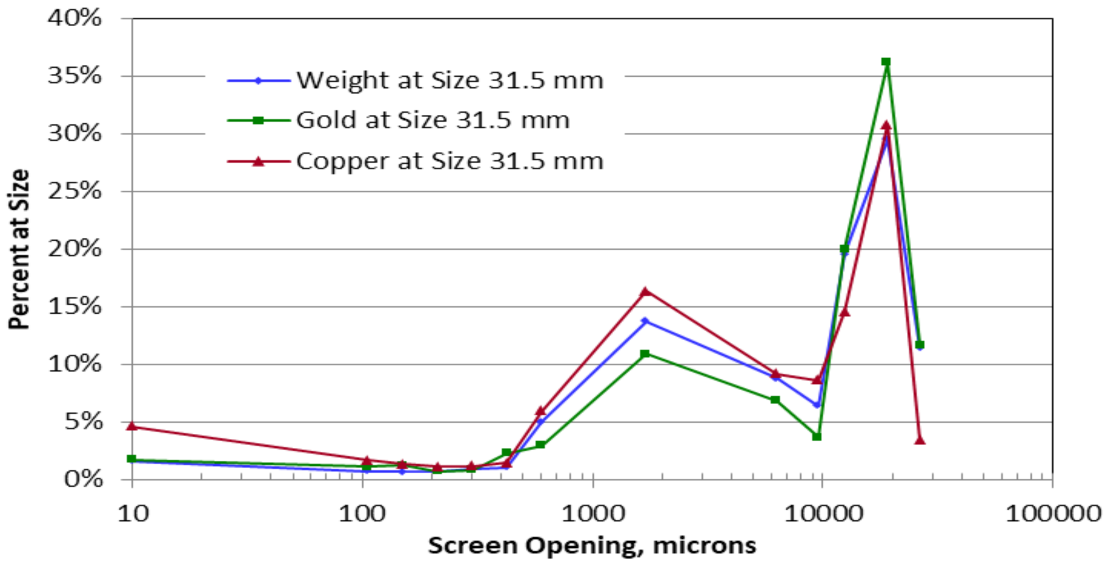


Figure 2.6 – Coarse Master Composite Percent at Size.

What is interesting with Figure 2.6 is that the distribution of weight, gold and copper are very similar with the bulk of the metals weight in the coarser fractions. Copper does show a slight upgrading in the finest size. Silver was assayed as well, but the value was low and uniform across all the sizes and, as such, would have presented a line identical to the weight line.

Figure 2.7 shows the distribution of metals by size for the finer composite (100% passing 6.3 mm).

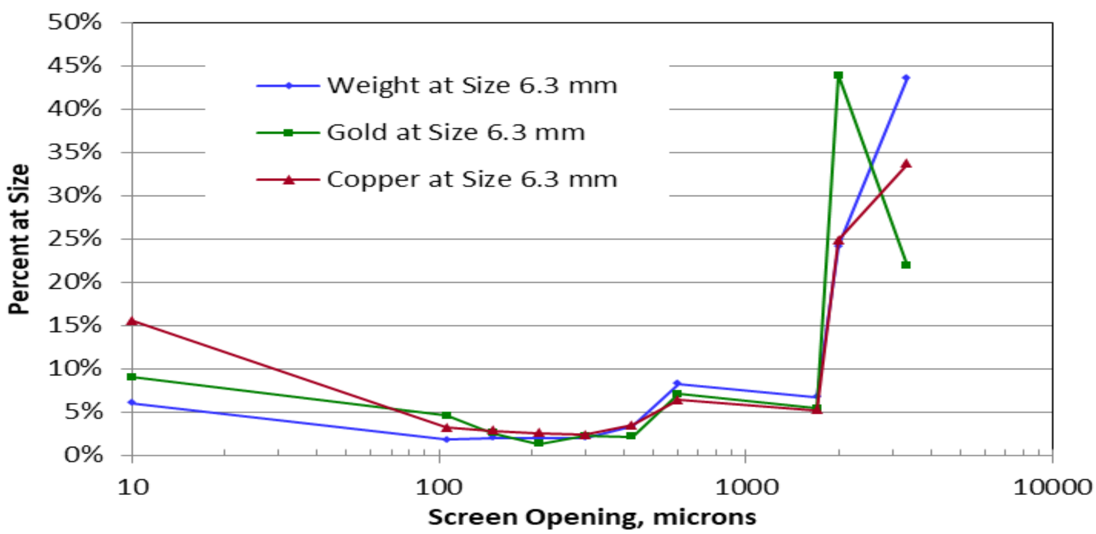


Figure 2.7 – Fine Master Composite Percent at Size.

The distribution of metals in the finer master composite as shown in Figure 2.7 follows a similar pattern to the coarse master composite with the exception of a greater upgrading in the fines.

2.3 <u>Physical Properties</u>:

Tests for preliminary Sag Mill (SMC), Abrasion Index, Rod and Ball Mill Work Indices were completed. Figure 2.8 summarizes these results and detailed test reports are included in Appendix B.

Item	Units	Value
Rod Work Index, Wi	kWhr/t	16.2
Ball Work Index, Wi	kWhr/t	15.9
Sag Mill Comminution, Axb	--	52.2
Abrasion Index, Ai	--	0.14

Figure 2.8 – Physical Properties.

The abrasion index is low, which would corroborate the waxy feel of the samples. The Rod and Ball work indices are on the higher side. It is atypical that the Rod Mill Work Index is greater than the Ball Mill Work Index and some additional testing should be conducted as the project progresses.

2.4 Quality Control:

Certified assay standards were run with each set of analyses. Figure 2.9 shows the results of these analyses over the course of the testing.

Analysis	Certified Value	<0.01	0.47	6.87	37.08
Head Assays	July 18/18	<0.01	0.46	6.56	--
Head Assays	July 18/18	<0.01	0.48	6.60	--
Head Assays	July 23/18	<0.01	0.46	6.58	--
Head Assays	July 23/18	<0.01	0.46	6.62	--
Bottle Rolls 1, 2	July 24/18	<0.01	0.47	6.46	--
Bottle Rolls 1, 3	July 24/18	<0.01	0.47	6.54	--
Flotation 3, 4	Aug 01/18	<0.01	0.50	6.81	36.55
Flotation 3, 5	Aug 01/18	<0.01	0.48	6.77	36.50
Size Analysis	Aug 08/18	<0.01	0.50	6.87	37.59
Size Analysis	Aug 08/18	<0.01	0.50	6.83	37.59
GRG	Aug 17/18	<0.01	0.52	6.80	37.12
GRG	Aug 17/18	<0.01	0.52	6.84	37.17
GRG	Aug 21/18	<0.01	0.50	6.85	37.66
CL-01, 02 Carbons	Aug 21/18	<0.01	0.50	6.81	37.71
Gravity Conc.	Sep 18/18	<0.01	0.46	6.87	36.95
Gravity Conc.	Sep 18/18	<0.01	0.46	6.83	37.30
BR 13 & Ro 14	Sept 25/18	<0.01	0.46	6.76	--
BR 13 & Ro 15	Sept 25/18	<0.01	0.48	6.76	--
mean		--	0.48	6.73	37.22
Std. Dev.		--	0.02	0.13	0.44

Figure 2.9 – Assay Standards.

2.5 Conclusions:

Additional Bond work indices and abrasion index should be determined using samples from a variety of locations that represent the depth and breadth of the orebody.

3.0 Baseline Cyanide Leach Testing:

In order to determine the cyanide leach amenability of the BAM master composite, several bottle roll leach tests were conducted using a variety of conditions. In each test, a 1.0 kg split of the master composite was ground or pulverized to a fine size and placed in a plastic jar. The material in these tests was slurried with a cyanide solution at 33% solids and sparged with oxygen. pH control was by hydrated lime addition. The tests ran for either 96 or 72 hours. Samples were taken at 1 hr, 2 hr, 4 hr, 8 hr, 24 hr, 48 hr, 72 hr and 96 hours. Intermediate samples were extracted from the pulp and centrifuged to obtain a clear supernatant. A known volume of the clear liquid was taken for analysis. A quantity of cyanide solution equivalent to that taken for analysis was replaced in the centrifuge tube and agitated. The pulp was then returned to the test. At the end of the test the entire contents of the bottle were filtered and the filtrate sampled. The filtercake was then rinsed and dried for fire assay. The bottle roll test apparatus is shown in the background of Figure 5.2. Detailed test sheets are included in Appendix B.

3.1 Bottle Rolls – Whole Ore:

Figure 3.1 is a summary of the whole ore bottle roll test conditions completed in Phase 1.

Test	P_{80}	NaCN	Leach	Extraction			Calculated Head			NaCN	Lime
No.	Size, µ	gpl	hrs	Au, %	Ag, %	Cu, %	Au, gm/t	Ag, gm/t	Cu, gm/t	Kg/t	Kg/t
01	Pulv.	1	96	99.2%	22.5%	14.0%	1.23	0.64	55.7	0.71	0.63
02	Pulv.	5	96	94.6%	51.0%	21.8%	1.48	0.49	95.9	1.86	0.29
05	Pulv.	1	96	98.8%	29.1%	21.8%	1.29	0.71	63.4	0.46	0.5
06	Pulv.	1	96	99.3%	33.2%	24.6%	1.14	0.75	65.0	0.45	0.5

Figure 3.1 – Summary of Whole Ore Bottle Roll Tests.

Two bottle roll tests (Tests 01 and 02) were conducted, using the conditions in Figure 3.1, and the results are presented in Figure 3.2 and 3.3 based on calculated head. Cyanide and lime consumptions were low. As can be seen, the calculated head for gold is higher than the head analysis and varies somewhat between samples which is indicative of free gold.

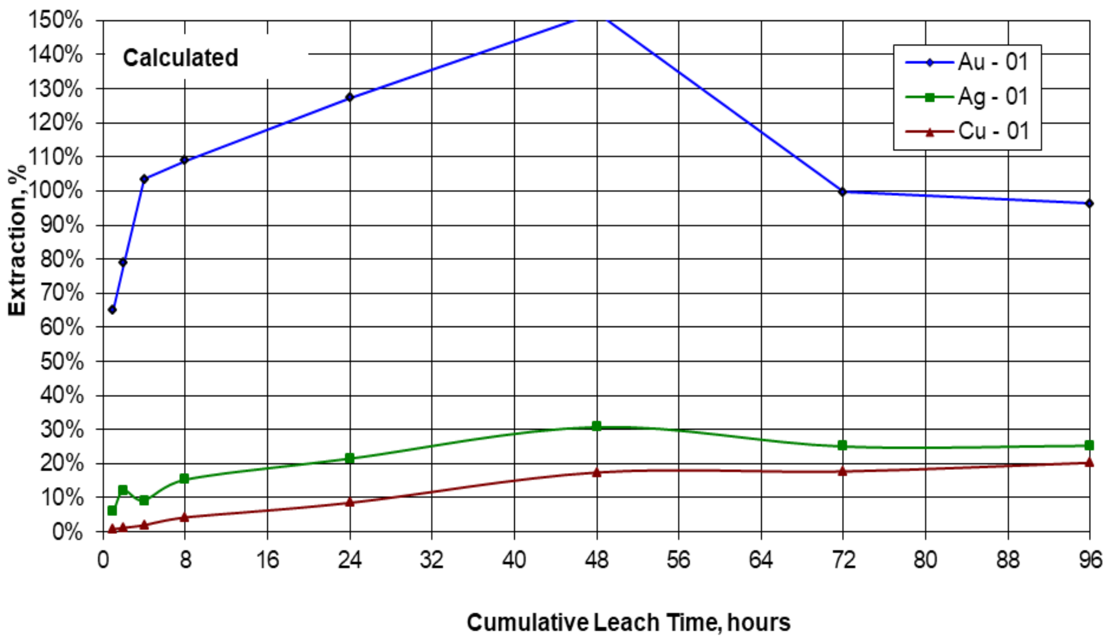


Figure 3.2 – Cumulative Percent Extraction from Bottle Roll Test 01.

Figure 3.2 shows that copper and silver are extracted gradually over the course of the test. An odd feature is that the gold extraction increases in solution to the middle of the test and then declines. The 150% extraction is a mathematical artefact of using the 96 hour solution concentration of gold to develop the calculated head.

Figure 3.3 shows that the higher cyanide concentration in this test increases the dissolution of copper but does not influence the extraction of silver. Of note in Figure 3.3 is that the gold extraction curve has a similar shape as Figure 3.2 but the reduction in solution concentration later in the test is less pronounced.

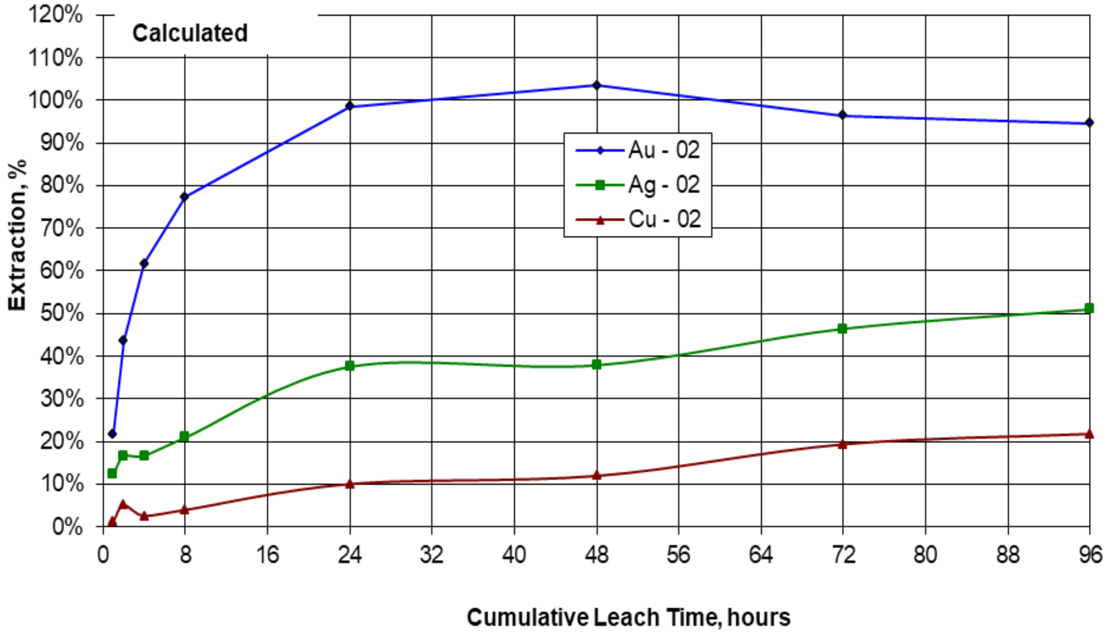


Figure 3.3 – Cumulative Percent Extraction from Bottle Roll Test 02.

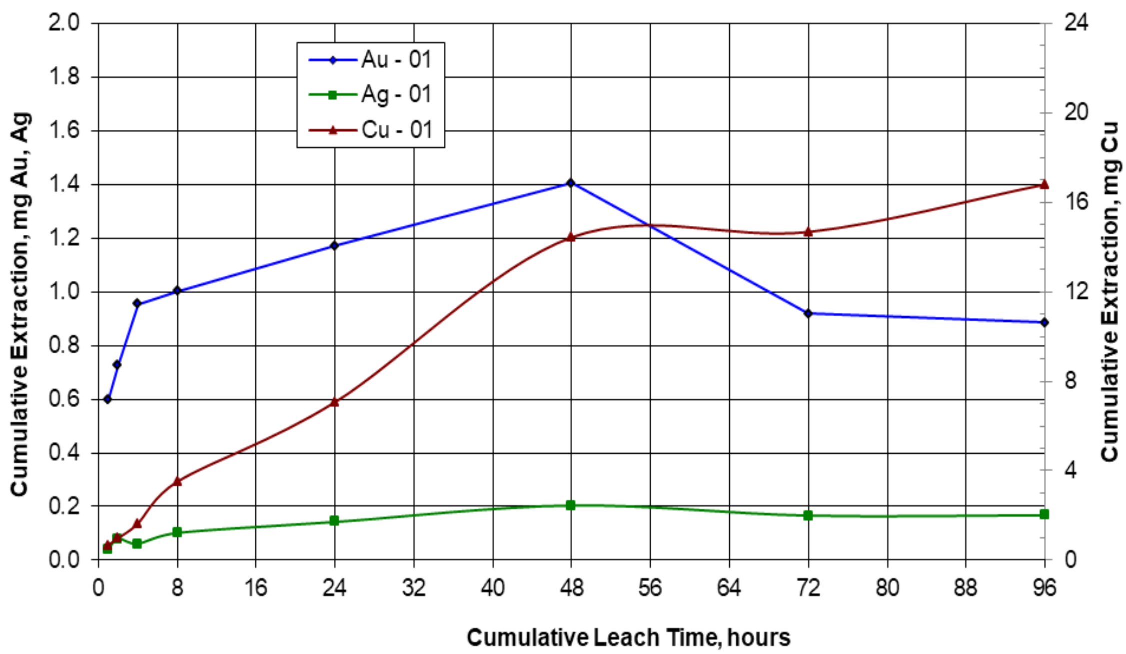


Figure 3.4 – Cumulative Extraction vs. Time for Bottle Roll Test 01.

Figure 3.4 shows the data from bottle roll Test 01 showing the total milligrams of metal extracted over the duration of the test. What this shows is that the extraction of silver in this test is minimal but the extraction of copper is well over 10 times higher than the gold in solution and still increasing at the end of the test. Copper complicates cyanide leaching and recovery of gold.

Figure 3.5 shows a similar graph for bottle roll Test 02. As can be seen the silver is unaffected, but the copper is enhanced by the higher cyanide concentration and is still extracting at 96 hours.

In order to investigate the behavior shown in Test 01 two additional duplicate tests (bottle roll 05 & 06) were conducted using the conditions of Test 01 with the exception that the feed was ground to 80% passing 75 microns. The results of these tests are shown in Figure 3.6.

The results in Figure 3.6 show that, with some variation, the extraction for silver and copper is similar in both Tests 05 and 06, and with Test 01. The drop in gold extraction was not reproduced in these tests and gold extraction was very high.

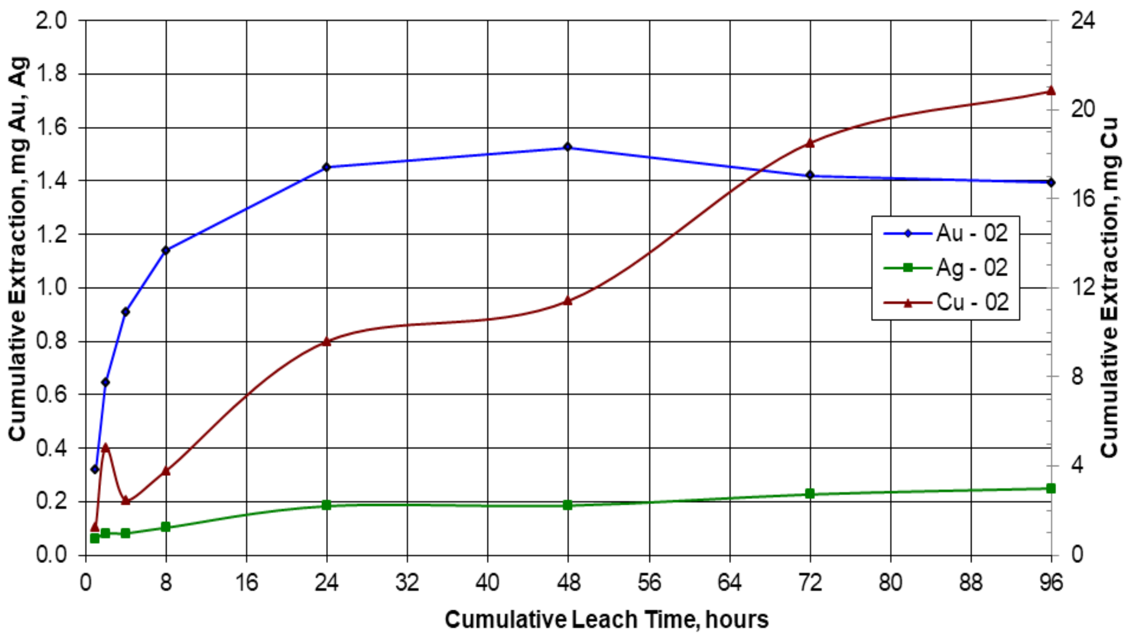


Figure 3.5 – Cumulative Extraction vs. Time for Bottle Roll Test 02.

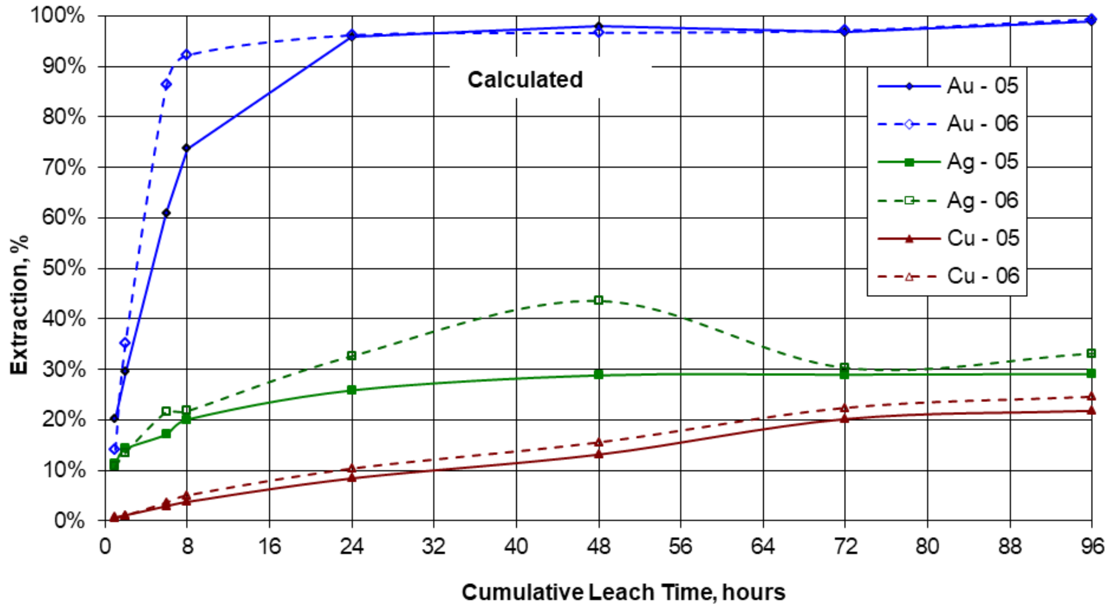


Figure 3.6 – Cumulative Extraction vs. Time for Bottle Roll Tests 05 & 06.

The final solution from Tests 01 and 02 was submitted for ICP analysis. The results are included in Appendix B. No unusual species were noted, however, the higher cyanide concentration of Test 02 increased the concentration of some of the species.

Figure 3.7 shows the same information as Figure 3.4 for the replicated tests. The shape of these curves is similar to those in Figure 3.4, except that the reduction in gold at the end of the test was not reproduced. The offset between the curves is an indication of the variation in calculated gold head between samples.

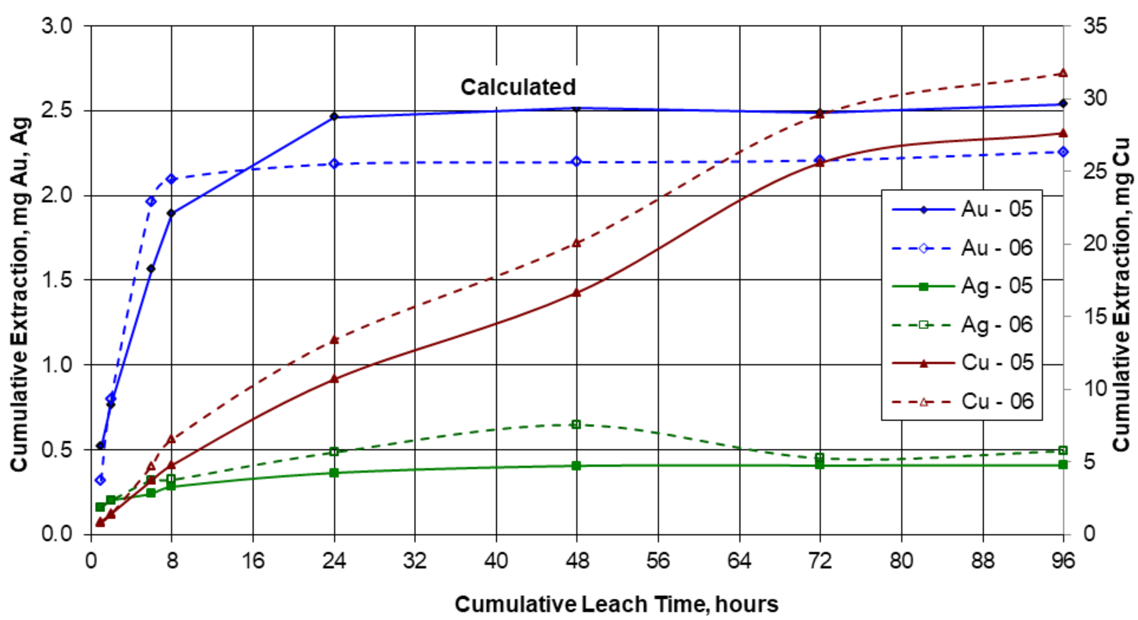


Figure 3.7 – Cumulative Extraction vs. Time for Bottle Roll Test 05 & 06.

3.2 Bottle Rolls – Gravity and Grind Size:

Tests 07 thorough 10 were conducted to investigate the effect of grind size on gravity recovery and the subsequent leachability of the gravity tails. The feed was ground to size and then passed through the Knelson Concentrator. The gravity concentrate was then hand panned to generate a pan concentrate with the pan tails returned to the leach feed. Figure 3.8 summarizes the results of these tests.

Test No.	P_{80} Size, µ	NaCN gpl	Leach hrs	Gravity Extraction Au, %	Ag, %	Cu, %	Grav. Tail Leach Extr. Au, %	Ag, %	Cu, %	NaCN Kg/t	Lime Kg/t
07	150	1	72	70.3%	--	--	97.2%	22.5%	14.0%	0.45	0.3
08	106	1	72	71.3%	--	--	94.9%	22.6%	16.1%	0.39	0.3
09	75	1	72	55.3%	--	--	94.4%	27.1%	16.6%	0.45	0.3
10	53	1	72	61.5%	--	--	96.1%	25.1%	19.0%	0.45	0.3
Test No.	P_{80} Size, µ	NaCN gpl	Leach hrs	Overall Extraction Au, %	Ag, %	Cu, %	Calculated Head Au, gm/t	Ag, gm/t	Cu, gm/t		
07	150	1	72	99.2%	22.5%	14.0%	1.23	0.64	55.7		
08	106	1	72	98.5%	22.6%	16.1%	1.37	0.64	60.1		
09	75	1	72	98.4%	27.1%	16.6%	0.91	0.69	65.5		
10	53	1	72	98.5%	25.1%	19.0%	1.01	0.67	66.2		

Figure 3.8 – Summary of Bottle Roll Tests 07 to 10.

The removal of coarse gold from the feed was somewhat independent of grind size with a slight trend towards reduced recovery at finer grinds. This trend may be expected because the finer gold has a greater chance of escaping the Knelson Concentrator.

The extraction figures shown in Figure 3.8 for the leach extraction of the gravity tails is the percentage extraction of the gold remaining in the tails. As can be seen, the gold extraction of the tails is high. The silver and copper extraction in the gravity concentrate was not recorded due to the very small quantity of concentrate after hand panning.

The overall extraction reported in Figure 3.8 is the combination of the gravity concentrate and the leach extraction of the tails based on calculated head. The calculated head shows some variability, as would be expected in a sample with coarse gold.

Comparing this data with that for the whole ore bottle roll tests in Figure 3.1 indicates that leach extractions are similarly high.

Figure 3.9 shows the leach kinetics of Tests 07 to 10 (gravity tails leach) and the effects of varying the grind size on the cumulative extraction of gold from the tails. It is apparent that the variation of grind size between 150 micron and 53 micron does not have a significant impact on the gold leach extraction, which is rapid and practically complete after 24 hours.

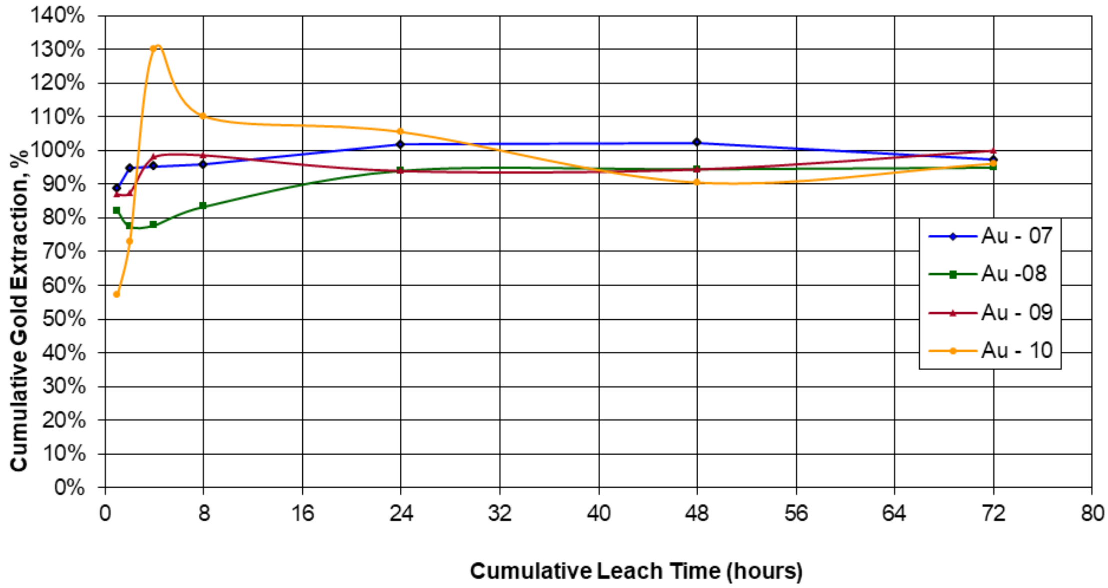


Figure 3.9 – Cumulative Gold Extraction vs. Time for Bottle Roll Tests 07 to 10.

Figure 3.10 shows that the extraction of copper versus time and the particle size distribution (PSD) in the range of study does not appear to have an impact.

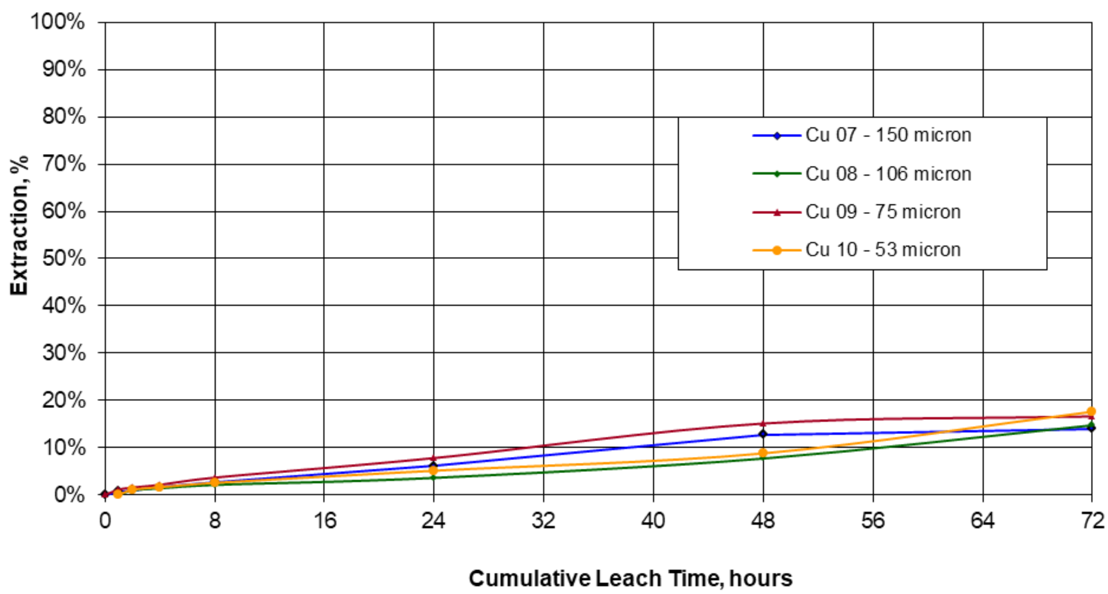


Figure 3.10 – Cumulative Copper Extraction vs. Time for Bottle Roll Tests 07 to 10.

3.3 <u>Reagent Consumption:</u>

Cyanide and lime consumptions are included in Figure 3.1 and 3.8. With the exception of the initial Tests 01 and 02, the cyanide consumption remained reasonably consistent at 0.45 kg/t, which is low. Lime consumption was also low (excluding Tests 01 & 02) at 0.3 kg/t to 0.5 kg/t (reported as hydrated lime or 0.22 kg/t to 0.36 kg/t as CaO).

3.4 Conclusions:

The following conclusions can be made from the results of the leach tests:

1. Cyanide leach extractions are very high with fine ground material.
2. Removing the coarser gold by gravity separation before leaching reduces the required residence time for leaching.
3. Reagent consumptions are low at 0.45 kg/t for NaCN and from 0.3 kg/t to 0.5 kg/t for $Ca(OH)_2$.
4. Particle size distribution (in the range tested) has limited impact on the gold extractions.
5. Copper and silver dissolve to a limited extent.
6. Significant variations exist between assay heads and calculated heads, likely due to free gold.

4.0 Flotation Testing:

Flotation tests were conducted using a 2 kg split of the master composite ground in a rod mill at 57% solids to a nominal P80 of 75 microns. The ground charge was placed in a 4.4 liter flotation cell and made up with water. Flotation was carried out at a natural pH of 8.4 to 8.5. 20 gm/t Potassium Amyl Xanthate (PAX) was added to the test and MIBC as a frother. The pulp was agitated at 800 rpm and air introduced; concentrate was collected for two minutes. Additional PAX was added and concentrate collected for an additional 2 minutes. This was repeated until four concentrates were obtained. The concentrates were assayed separately. Detailed test sheets are included in Appendix B. Gravity concentration of both the feed and tails was conducted. The laboratory flotation apparatus is shown in Figure 4.1.

Figure 4.1 – Denver D12 Laboratory Flotation Machine.

4.1 Rougher Flotation:

Rougher flotation Test 03 is summarized in Figure 4.1 where 4 concentrates were obtained and the flotation tails passed through a Knelson concentrator. The tails concentrate was then hand-panned to create a pan concentrate and a pan tail. Detailed test sheets are included in Appendix B.

Product	Weight %	Weight grams	Assay, g/t Au	Assay, g/t Ag	Distribution, % Au	Distribution, % Ag
(1) Ro Con 1	2.9	58.2	19.7	3.0	69.2	14.9
(2) Ro Con 2	2.7	54.1	4.16	0.5	13.6	2.3
(3) Ro Con 3	2.3	44.9	0.35	0.5	1.0	1.9
(4) Ro Con 4	1.7	33.5	0.17	0.5	0.3	1.4
(5) Ro Tail Pan Con	0.3	5.3	14.5	2.0	4.6	0.9
(6) Ro Tail Pan Tail	2.4	48.2	1.28	1.0	3.7	4.1
(7) Rougher Tail	87.8	1750.5	0.07	0.5	7.5	74.5
Recalc. Feed	100.0	1994.7	0.83	0.6	100	100
Cum. Product	**Weight %**	**Weight grams**	**Assay, g/t Au**	**Assay, g/t Ag**	**Distribution, % Au**	**Distribution, % Ag**
Product 1	2.9	58.2	19.7	3.0	69.2	14.9
Products 1 to 2	5.6	112.3	12.2	1.8	82.8	17.2
Products 1 to 3	7.9	157.2	8.82	1.4	83.8	19.1
Products 1 to 4	9.6	190.7	7.30	1.3	84.1	20.5
Products 1 to 5	9.8	196.0	7.49	1.3	88.8	21.4
Products 1 to 6	12.2	244.2	6.26	1.2	92.5	25.5
Product 7	87.8	1750.5	0.07	0.5	7.5	74.5
Recalc. Feed	100.0	1994.7	0.83	0.6	100	100

Figure 4.2 – Flotation Summary for Test 03.

Figure 4.2 shows that up to 84.1% of the gold and 20.5% of the silver (Products 1-4) are recoverable into 9.6% of the weight by froth flotation. The numbers in parenthesis in the upper panel of Figure 4.2 represent the product number referred to in the lower panel. Adding in the Knelson pan concentrate (Product 5) brings the cumulative recovery to 88.8% of the gold and 21.5% of the silver into 9.8% of the weight. Silver upgrading was minimal. Copper was not assayed.

Figure 4.3 summarizes the results from Flotation Test 04 where the feed was passed through the Knelson concentrator before flotation. The gravity concentrate was hand panned and the pan tails included with the flotation feed. Figure 4.3 shows that 89.5% of the gold can be recovered into 8.7% of the weight. Silver upgrading is minimal in this test as well. It should be noted that the tails grade is similar between both flotation tests.

Product	Weight %	Weight grams	Assay, g/t Au	Assay, g/t Ag	Distribution,% Au	Distribution,% Ag
(1) Pan Con	0.2	4.7	206	14.0	79.6	5.3
(2) Ro Con 1	2.9	58.9	1.11	3.0	5.4	14.3
(3) Ro Con 2	2.2	43.5	0.85	1.0	3.0	3.5
(4) Ro Con 3	1.9	37.1	0.32	0.5	1.0	1.5
(5) Ro Con 4	1.4	28.8	0.22	0.5	0.5	1.2
(6) Rougher Tail	91.3	1825.3	0.07	0.5	10.5	74.1
Total	100.0	1998.3	0.61	0.6	100	100
Cum. Product	**Weight %**	**Weight grams**	**Assay, g/t Au**	**Assay, g/t Ag**	**Distribution, % Au**	**Distribution, % Ag**
Product 1	0.2	4.7	206	14.0	79.6	5.3
Products 1 to 2	3.2	63.6	16.24	3.8	84.9	19.7
Products 1 to 3	5.4	107.1	9.99	2.7	88.0	23.2
Products 1 to 4	7.2	144.2	7.50	2.1	88.9	24.7
Products 1 to 5	8.7	173.0	6.29	1.8	89.5	25.9
Product 6	91.3	1825.3	0.07	0.5	10.5	74.1
Total	100.0	1998.3	0.61	0.6	100.0	100.0

Figure 4.3 – Flotation Summary for Test 04.

Figure 4.4 shows the grade-recovery cure for Flotation Tests 03 and 04. The data shows that the bulk of the recovery occurs early in the flotation (Test 03). Gravity separation after flotation increases the recovery approximately 5% (last point on lower right of curve) with a very slight improvement in grade.

The curve for Test 04 is a bit misleading, because the gravity concentrate prior to flotation was included in the curve. The data point for the gravity concentrate was not included in the graph, because the grade (206 gm/t) would have expanded the graph and eliminated the detail. To get a sense of the contribution of flotation after gravity separation the dashed line was added to represent flotation on the residual gold in the gravity tail. Flotation of the gravity tail recovers a maximum of around 50% of the residual values at a low concentrate grade.

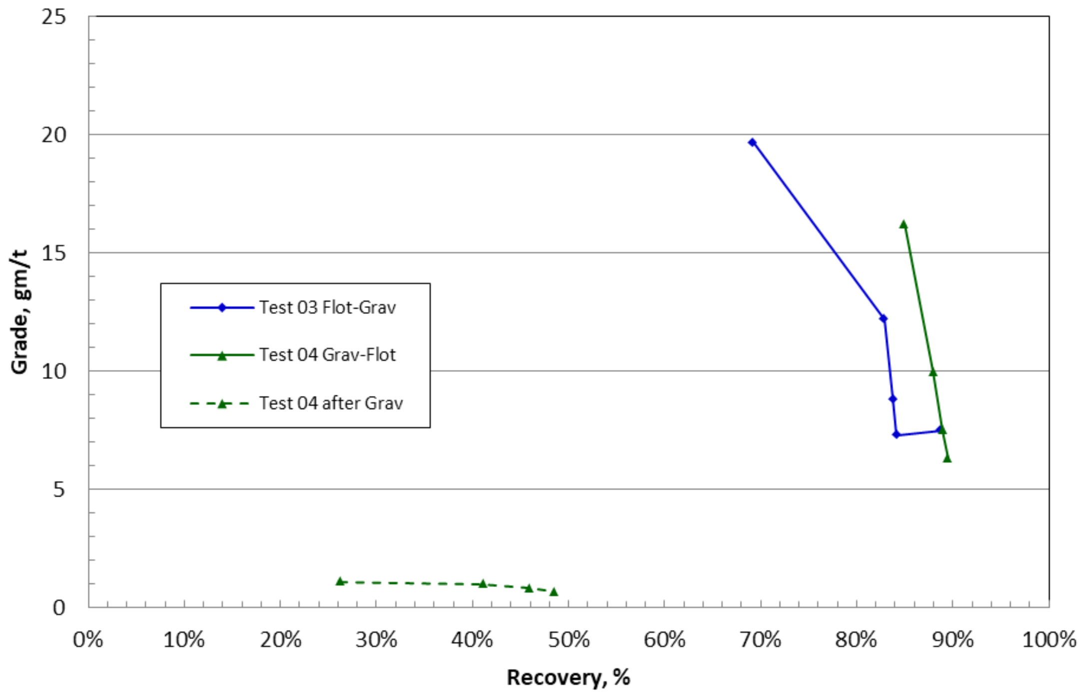


Figure 4.4 – Grade/Recovery Curve for Test 04.

4.2 <u>Conclusions:</u>

The following conclusions can be made from the data obtained in the rougher flotation tests.

1. Rougher flotation achieves recoveries in the high 80 percent range.
2. Results of the combination of flotation and gravity is similar irrespective of the order that the unit operations are employed.
3. The grade of the concentrate drops quickly as higher recovery is obtained.
4. The grade of the rougher concentrate is too low to sell to a smelter.

5.0 <u>Gravity Recoverable Gold (GRG):</u>

The gravity recoverable gold (GRG) test is an empirical test designed to mimic using gravity concentrators on cyclone underflows of a grinding circuit to recover free gold.

Feed to the laboratory Knelson Concentrator is initially ground to nominally 1700 microns and a split taken for head screen assays. The remaining feed is passed through the unit. The tails are collected and filtered. A split of the tails is taken for particle size analysis. The concentrate is flushed from the machine, dried, weighed and assayed by screen fraction.

For the second stage the tails from the first stage are ground to nominally 212 microns and passed through the concentrator. The procedure above is repeated at a grind size of 75 micron. The final tails are collected and sampled for tails screen assay. Detailed test data sheets are included in Appendix B.

Operating conditions used for the Knelson Concentrator are shown in Figure 5.1.

g-force, g's:	60
Water pressure to bowl, kPa:	13.8
Fluidization Water Flowrate, l/min:	4.5

Figure 5.1 – Laboratory Scale Knelson Concentrator Conditions.

The Knelson laboratory gravity concentrator is shown in Figure 5.2

Figure 5.2 – Laboratory Scale Knelson Concentrator.

5.1 Gravity Tests:

Gravity recoverable gold tests were conducted on the master composite. Figure 5.3 summarizes the first concentrate and tails particle size distributions (PSD). In the following figures an assay with an asterisk represents size fractions that were too small in mass to assay. Adjacent fractions were combined for assay and the result was applied to each fraction. Figure 5.4 shows the data for the second concentrate and tails. Figure 5.5 shows the data for the third concentrate and tails. Figure 5.6 shows the feed and final tails screened assay results.

Opening	Concentrate 1					Tails 1		
micron	Wt, gm	Wt, %	Cum %	Au, gm/t	Au, %	Wt, gm	Wt, %	Cum %
1700	9.9	10.3%	89.7%	0.34	0.1%	29.6	3.3%	96.7%
1180	28.5	29.6%	60.1%	2.08	1.0%	163.9	18.2%	78.6%
850	17.8	18.5%	41.6%	8.40	2.6%	146.3	16.2%	62.3%
600	10.8	11.2%	30.4%	3.35	0.6%	116.4	12.9%	49.4%
425	7.1	7.4%	23.1%	0.17	0.0%	87.1	9.7%	39.8%
300	5.1	5.3%	17.8%	175	15.4%	67.9	7.5%	32.3%
212	4.0	4.2%	13.6%	138	9.5%	56.0	6.2%	26.1%
150	3.4	3.5%	10.1%	298	17.5%	45.6	5.1%	21.0%
106	2.8	2.9%	7.2%	249	12.0%	41.1	4.6%	16.4%
75	2.2	2.3%	4.9%	390	14.8%	33.4	3.7%	12.7%
53	1.6	1.7%	3.2%	352	9.7%	28.8	3.2%	9.6%
38	1.3	1.3%	1.9%	323	7.3%	25.1	2.8%	6.8%
25	1.0	1.0%	0.8%	323	5.6%	18.8	2.1%	4.7%
-25	0.8	0.8%	0.0%	282	3.9%	42.3	4.7%	0.0%
Calc Head	96.3	100.0%		60.2	100.0%	902.3	100.0%	0.0%
						P80 = 1214 micron		

Figure 5.3 – Summary of GRG Products – Pass 1.

Opening micron	Concentrate 2 Wt, gm	Wt, %	Cum %	Au, gm/t	Au, %	Tails 2 Wt, gm	Wt, %	Cum %
1700	0	0.0%	100.0%		0.0%	0.0	0.0%	100.0%
1180	0.0	0.0%	100.0%		0.0%	0.0	0.0%	100.0%
850	0.0	0.0%	100.0%		0.0%	0.0	0.0%	100.0%
600	0.6	0.9%	99.1%	27.2*	0.2%	0.1	0.1%	99.9%
425	0.5	0.7%	98.4%	27.2*	0.2%	0.9	0.9%	99.0%
300	2.6	3.9%	94.5%	27.2*	0.9%	3.3	3.3%	95.7%
212	6.8	10.1%	84.4%	65	5.5%	6.5	6.5%	89.2%
150	10.5	15.6%	68.7%	130	16.9%	8.9	8.9%	80.3%
106	10.9	16.2%	52.5%	150	20.4%	10.7	10.7%	69.6%
75	10.6	15.8%	36.7%	127	16.8%	10.8	10.8%	58.8%
53	8.3	12.4%	24.3%	117	12.1%	9.9	9.9%	48.9%
38	7.2	10.7%	13.6%	127	11.4%	9.5	9.5%	39.4%
25	4.4	6.6%	7.0%	121	6.6%	9.5	9.5%	29.9%
-25	4.7	7.0%	0.0%	154	9.0%	29.9	29.9%	0.0%
Calc Head	67.1	100.0%		119.8	100.0%	100.0	100%	
						P80 = 148 micron		

Figure 5.4 – Summary of GRG Products – Pass 2.

Opening micron	Concentrate 3 Wt, gm	Wt, %	Cum %	Au, gm/t	Au, %	Tails 3 Wt, gm	Wt, %	Cum %
1700	0	0.0%	100.0%		0.0%	0.0	0.0%	100.0%
1180	0.0	0.0%	100.0%		0.0%	0.0	0.0%	100.0%
850	0.0	0.0%	100.0%		0.0%	0.0	0.0%	100.0%
600	0.1	0.2%	99.8%	12.4*	0.0%	0.0	0.0%	100.0%
425	0.1	0.2%	99.7%	12.4*	0.0%	0.0	0.0%	100.0%
300	0.3	0.5%	99.2%	12.4*	0.1%	0.0	0.0%	100.0%
212	1.2	2.0%	97.2%	12	0.5%	0.1	0.0%	100.0%
150	4.8	7.9%	89.3%	27	4.6%	3.5	0.7%	99.3%
106	9.3	15.3%	74.0%	13	4.1%	21.3	4.3%	95.0%
75	13.4	22.0%	52.0%	19	9.0%	52.4	10.5%	84.5%
53	11.4	18.8%	33.2%	42	17.1%	72.0	14.4%	70.1%
38	9.5	15.6%	17.6%	59	20.1%	95.3	19.1%	51.1%
25	5.4	8.9%	8.7%	92	17.6%	157.2	31.4%	19.6%
-25	5.3	8.7%	24.5%	142	26.7%	98.2	19.6%	50.5%
Calc Head	60.8	100.0%		46.2	100.0%	500.0	100.0%	
						P80 = 67 micron		

Figure 5.5 – Summary of GRG Products – Pass 3.

Opening micron	Head Wt, gm	Wt, %	Cum %	Au, gm/t	Au, %	Tails Wt, gm	Wt, %	Cum %	Au, gm/t	Au, %
1700	31.5	3.2%	96.8%	0.10	0.5%	0		100.0%		0.0%
1180	180.2	18.1%	78.7%	0.59	15.8%	0.0		100.0%		0.0%
850	158.0	15.9%	62.8%	0.50	11.9%	0.0		100.0%		0.0%
600	124.3	12.5%	50.3%	0.67	12.5%	0.0		100.0%		0.0%
425	95.4	9.6%	40.7%	0.57	8.1%	0.0		100.0%		0.0%
300	75.2	7.6%	33.1%	0.61	6.9%	0.0		100.0%		0.0%
212	60.2	6.1%	27.0%	0.63	5.7%	0.1		100.0%	0.17	0.0%
150	46.0	4.6%	22.4%	0.47	3.2%	3.5	0.7%	99.3%	0.17	0.6%
106	40.2	4.0%	18.4%	1.42	8.5%	21.3	4.3%	95.0%	0.13	2.6%
75	31.6	3.2%	15.2%	1.16	5.5%	52.4	10.5%	84.5%	0.12	6.1%
53	26.9	2.7%	12.5%	1.79	7.2%	72.0	14.4%	70.1%	0.14	9.3%
38	27.4	2.8%	9.7%	1.06	4.3%	95.3	19.1%	51.1%	0.13	11.7%
25	20.1	2.0%	7.7%	1.27	3.8%	157.2	31.4%	19.6%	0.12	17.9%
-25	76.3	7.7%	0.0%	0.55	6.3%	98.2	19.6%	50.5%	0.56	51.9%
Calc Head	993.3	100.0%		0.7	100.0%	500.0	100.0%		0.20	100.0%
	P80 = 1214 micron						P80 = 67 micron			

Figure 5.6 – Summary of GRG Feed – Final Tails.

Figure 5.7 shows the assay by size data in Figure 5.6 in graphical form. It is apparent the gold un-extractable by gravity is reasonably uniform in grade across several particle sizes. Also, the gold concentration in the finest fraction is relatively unchanged.

Figure 5.8 shows the assay at size for the three GRG concentrates. This graph also shows that very little liberation occurs above a particle size of 300 microns and additional grinding improves the recovery of gold from the finest sizes.

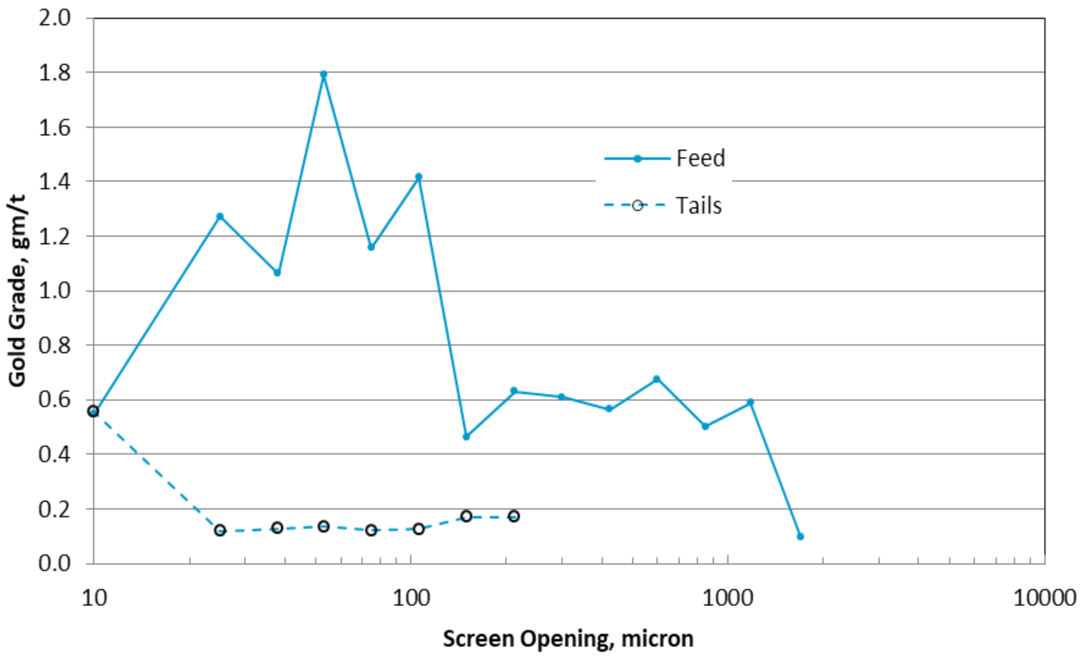


Figure 5.7 – Gold Assay at Size for GRG Feed and Tails.

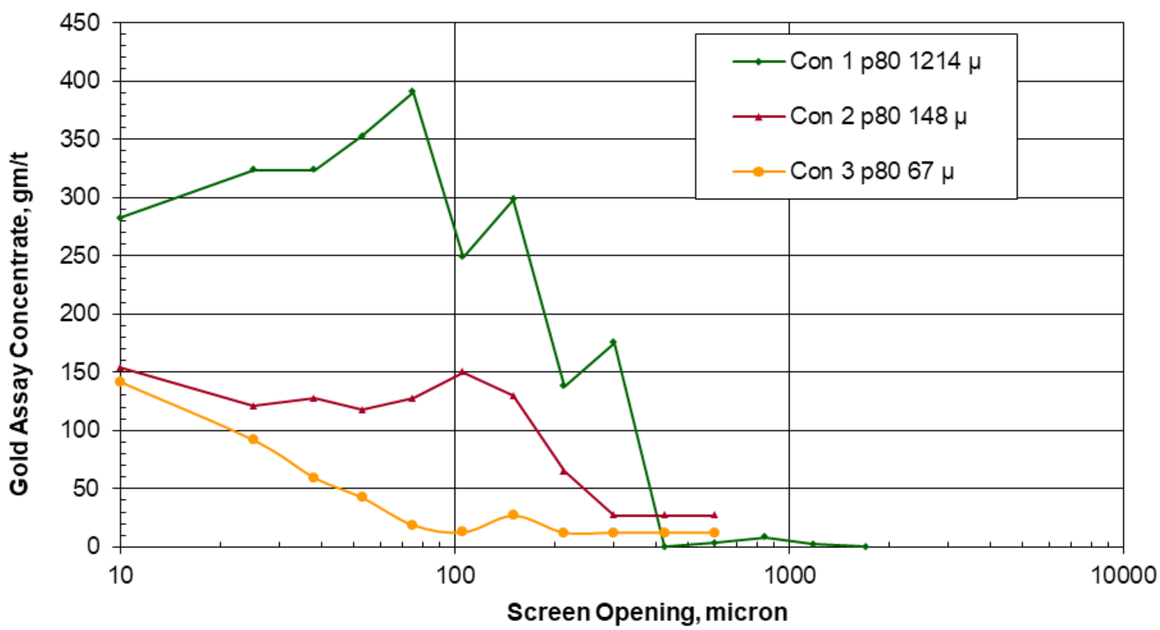


Figure 5.8 – Gold Assay at Size for GRG Concentrates.

Figure 5.9 shows the cumulative percent retained gold by size for the GRG concentrates. The intersection with the left hand axis is equivalent to the cumulative gravity recoverable gold at each successive grind, including the previous grind size recovery. This graph also shows that little gold is liberated above approximately 300 microns. The cumulative extractions per stage are 28.7%, 68.6% and 82.5% for passes 1 through 3 respectively.

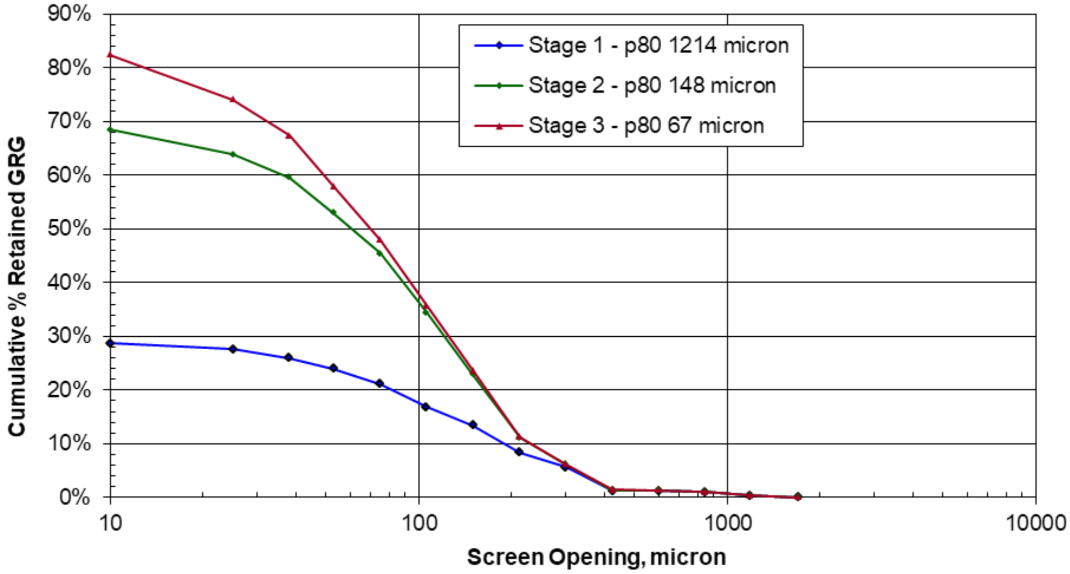


Figure 5.9 – Cumulative Percent Retained Gold by Size for GRG Concentrates.

Figure 5.10 shows the distribution of mass and gold of the gravity tails. What is apparent from Figure 5.10 is that the gold that is not recovered in the Knelson concentrate is predominantly in the finest fraction.

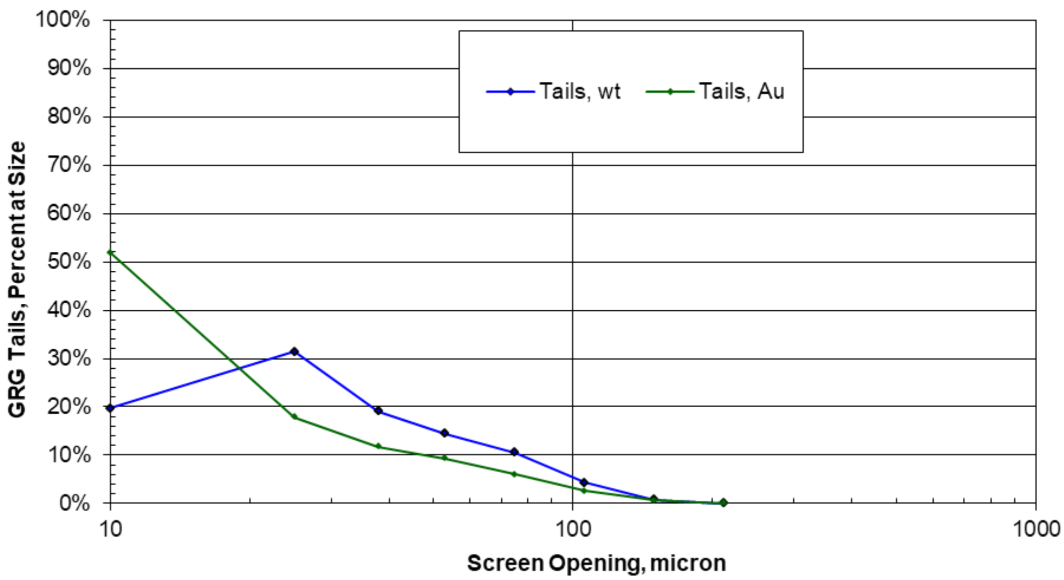


Figure 5.10 – Weight and Gold Distribution by Size for GRG Tails.

5.2 Bulk Gravity Separation:

Approximately 30 kg of the master composite was ground to a nominal passing 212 microns and passed through the Knelson Concentrator in Test 12. The concentrate obtained from the gravity separation represented 59 grams with a gold content of 278 gm/tonne representing 65.5% of the contained gold in the feed. Summary data for the Feed and tails from this test is shown in Figure 5.11.

Opening micron	Head Wt, gm	Wt, %	Cum %	Au, gm/t	Au, %	Tails Wt, gm	Wt, %	Cum %	Au, gm/t	Au, %
212	8.2	1.6%	98.4%	0.47	0.8%	15.0	3.1%	96.9%	0.36	3.7%
150	86.4	17.3%	81.1%	1.94	32.9%	67.1	13.9%	83.0%	0.33	15.5%
106	61.6	12.3%	68.8%	1.72	20.8%	60.7	12.5%	70.5%	0.44	18.7%
75	57.6	11.5%	57.3%	1.15	13.1%	55.8	11.5%	59.0%	0.28	10.9%
53	48.5	9.7%	47.6%	0.89	8.5%	48.3	10.0%	49.0%	0.28	9.5%
38	56.6	11.3%	36.2%	0.71	7.9%	56.8	11.7%	37.2%	0.13	5.2%
-38	181.3	36.2%	0.0%	0.45	16.1%	180.2	37.2%	0.0%	0.29	36.5%
Calc	500.2	100.0%		1.0	100.0%	483.9	100.0%		0.30	100 %
	P80 = 145 micron					P80 = 139 micron				

Figure 5.11 – Bulk Gravity Test Summary.

Figure 5.12 shows the weight and gold distribution of the feed and tails for Test 12. These graphs show that the fines are not recovered in the concentrator.

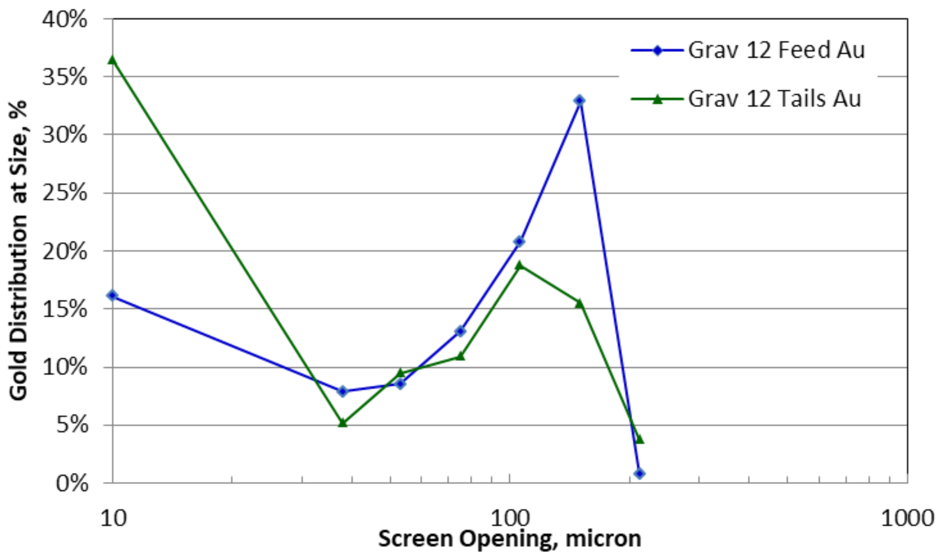


Figure 5.12 – Weight and Gold Distribution by Size for Bulk Gravity Feed & Tails.

5.2.1 Leaching of Bulk Gravity Tails:

A variety of bottle roll leach tests were conducted on the bulk gravity tails using different conditions. These tests were run for 72 hours with the conditions listed in Figure 5.13. Detailed test data sheets are included in Appendix B.

Test No.	NaCN gpl	Solids %	Air/O_2	Carbon	Extraction Au, %	Au, %	Cu, %	Calculated Head, gm/t Au	Ag	Cu	NaCN Kg/t	Lime Kg/t
13	1.00	33%	Air	No	87.23%	11.02%	9.98%	0.31	0.56	63	0.27	0.80
16	1.00	33%	O_2	No	94.69%	11.18%	13.10%	0.38	0.56	50	0.34	0.57
17	0.75	33%	O_2	No	95.00%	11.27%	10.77%	0.40	0.56	58	0.21	0.55
18	0.50	33%	O_2	No	94.68%	11.25%	8.95%	0.38	0.56	58	0.02	0.63
19	0.25	33%	O_2	No	93.60%	11.02%	8.59%	0.31	0.56	54	0.04	0.60
20	1.00	25%	O_2	No	92.51%	15.65%	12.88%	0.40	0.59	59	0.43	0.85
21	1.00	40%	O_2	No	95.27%	13.66%	11.11%	0.32	0.58	60	0.43	0.42
22	1.00	33%	O_2	Yes	93.73%	--	7.22%	0.40	0.52	55	1.72	0.70

Figure 5.13 – Bottle Roll Tests on Test 12 Gravity Tails.

The extraction values for gold in Figure 5.13 in the leach tests do not include the gold extracted in the gravity concentrate. Adding the 65.5% extraction in the bulk gravity test to the cumulative extraction from the leach tests provides the overall gold extraction. This results in the overall extractions in Figure 5.14. It is apparent that the BAM master composite is amenable to gravity concentration followed by cyanide leaching.

Test No.	NaCN gpl	Solids %	Air/O2	Carbon	Extraction Au, %
13	1.00	33%	Air	No	95.6%
16	1.00	33%	O2	No	98.2%
17	0.75	33%	O2	No	98.3%
18	0.50	33%	O2	No	98.2%
19	0.25	33%	O2	No	97.8%
20	1.00	25%	O2	No	97.4%
21	1.00	40%	O2	No	98.4%
22	1.00	33%	O2	Yes	97.8%

Figure 5.14 – Overall Extraction from Bottle Roll Tests with Bulk Gravity Concentration.

5.2.2 <u>Leaching Oxidation Variable</u>:

Tests 13 and 16 were run to determine the necessity of using oxygen to sparge the leach reactor. Other conditions are identified in Figure 5.13.

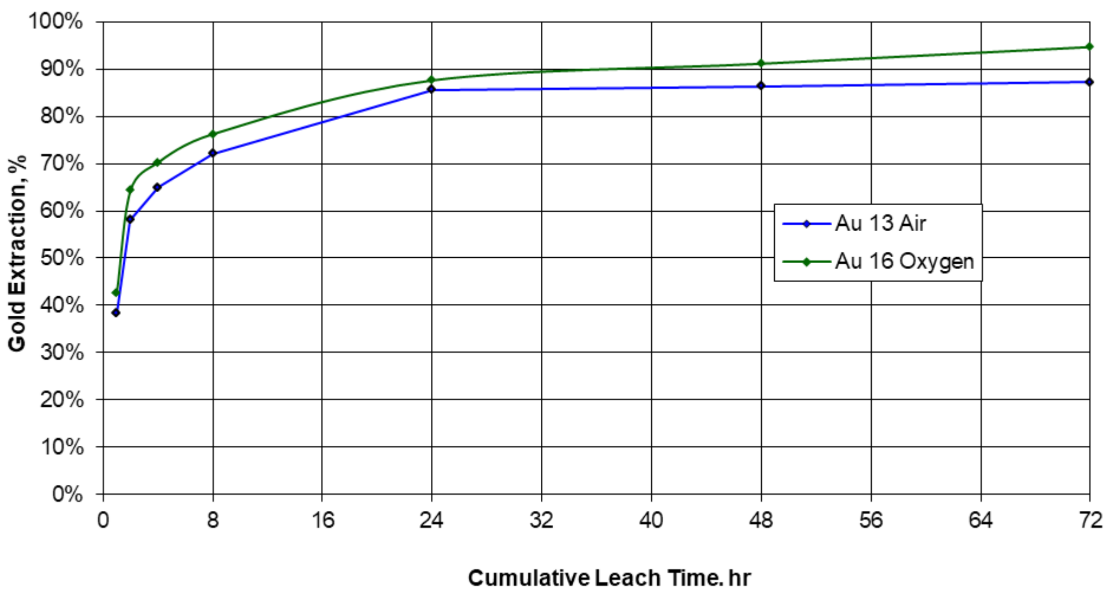


Figure 5.15 – Bottle Roll Tests on Test 12 Gravity Tails – Air vs. O_2.

As can be seen in Figure 5.15 the addition of oxygen showed a small but beneficial improvement in extraction. Based on this test the remainder of the tests identified in Figure 5.13 employed oxygen sparging.

5.2.3 Leaching Cyanide Variable:

Tests 16, 17; 18 and 19 were run with varying concentrations of NaCN. The tests used 1.0 gpl, 0.75 gpl; 0.5 gpl and 0.25 gpl, respectively. The cumulative gold extraction rate curves are presented in Figure 5.16. As can be seen in this figure, increased cyanide concentration had a mild impact on initial leach rate but overall recovery was virtually identical for all concentrations.

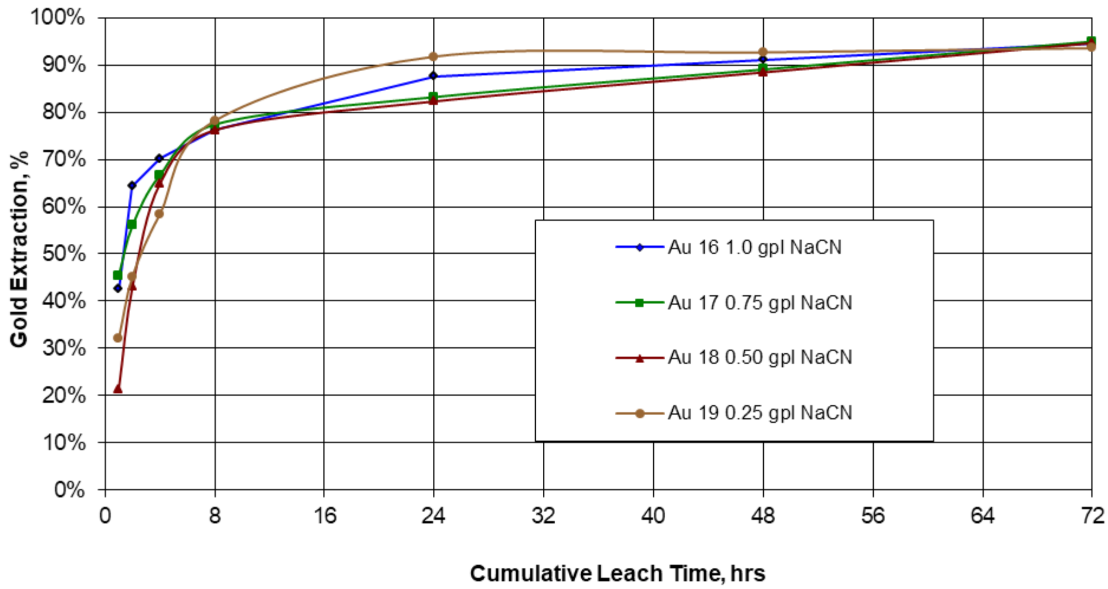


Figure 5.16 – Bottle Roll Tests on Test 12 Gravity Tails – NaCN Concentration.

5.2.4 Leaching Percent Solids Variable:

The tests shown in Figure 5.17 varied the percent solids in the leach while maintaining 1 gpl NaCN concentration and sparging with oxygen. The variability shown in the graph is counterintuitive, in that the lower percent solids demonstrate lower ultimate extraction than the higher density slurries, which is usually not the case. This trend is corroborated by the tails assay which is lower for the higher percent solids tests.

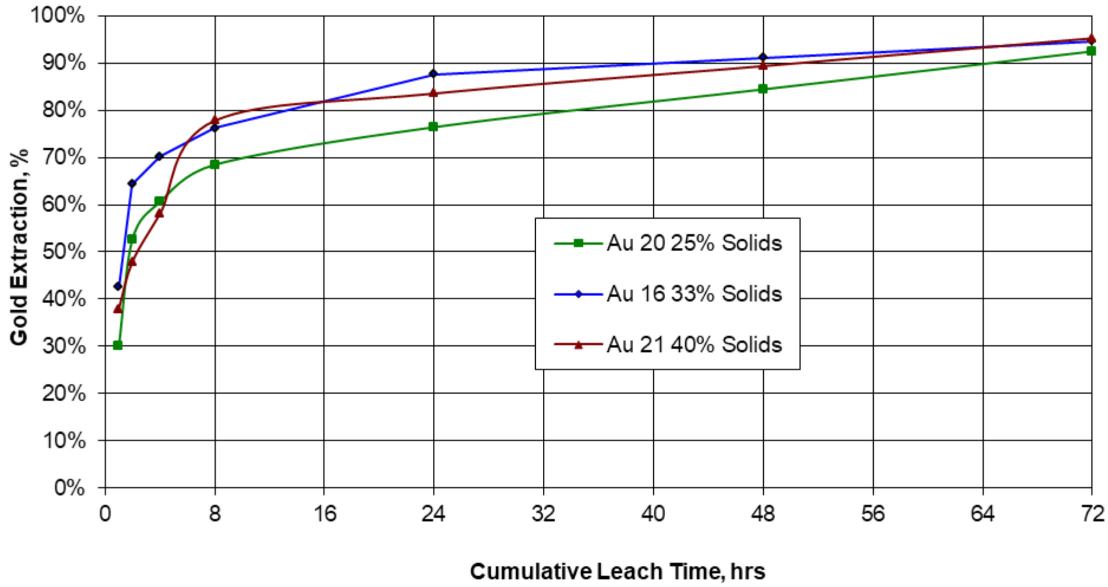


Figure 5.17 – Bottle Roll Tests on Test 12 Gravity Tails – Percent Solids.

5.2.5 Leaching Activated Carbon Variable:

Test 22 was run to simulate a carbon in leach (CIL) process. Activated carbon equivalent to 50 gm/liter was added to a bottle roll test. The conditions were otherwise identical to Test 16. This data is shown in Figure 5.18. Only the ultimate extraction for Test 22 is included, based on the assay of gold on carbon as identified by the point at the 72 hours leach time. As can be seen there is very little difference between the two tests. A silver extraction is not included in Figure 5.13, because the lower limit of detection for the assay method using carbon would give an erroneous high number; the silver on the carbon was non-detectable.

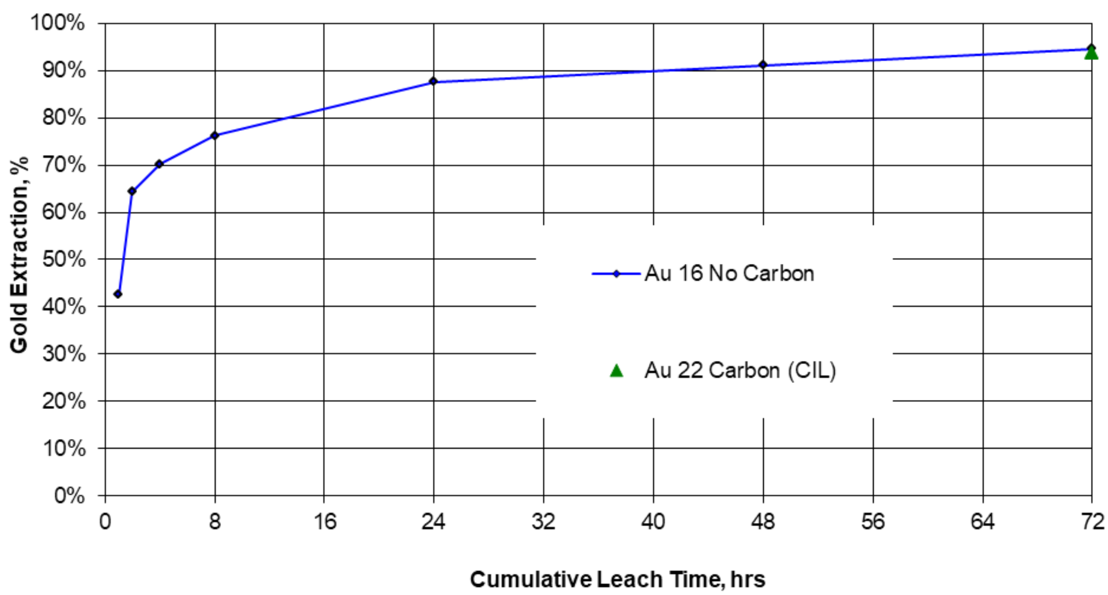


Figure 5.18 – Bottle Roll Tests on Test 12 Gravity Tails – Activated Carbon.

5.2.6 Leaching Reagent Consumption:

Consumption of cyanide and lime are included in Figure 5.13. With the exception of the CIL test sparged with oxygen (Test 22) the NaCN consumptions are low, with very low consumptions for the reduced cyanide addition tests.

Lime consumption for all the leach tests in Figure 5.13 was of a similar magnitude and average 0.64 kg/tonne as $Ca(OH)_2$ (0.46 kg/t as CaO).

5.2.7 Flotation of Bulk Gravity Tails:

A split of the bulk gravity tails was floated (Test 14) using the procedures in Section 4.0, with the exception that the grind size was a P80 of 139 micron (per Figure 5.11). A summary of the flotation results is included in Figure 5.19. Flotation recovered 66.7% of the residual gold values in the bulk gravity tails. Silver did not concentrate appreciably with flotation.

Product	Weight		Assay, g/t		Distribution, %	
	%	grams	Au	Ag	Au	Ag
(1) Ro Con 1	1.9	36.6	9.59	3.0	51.8	10.2
(2) Ro Con 2	1.0	20.2	2.92	0.5	8.7	0.9
(3) Ro Con 3	0.8	15.6	1.81	0.5	4.2	0.7
(4) Ro Con 4	0.8	15.2	0.93	0.5	2.1	0.7
(5) Rougher Tail	95.5	1872.9	0.12	0.5	33.3	87.4
Recalc. Feed	100.0	1960.5	0.35	0.5	100	100
Cum. Product	Weight		Assay, g/t		Distribution, %	
	%	grams	Au	Ag	Au	Ag
Product 1	1.9	36.6	9.59	3.0	51.8	10.2
Products 1 to 2	2.9	56.8	7.22	2.1	60.5	11.2
Products 1 to 3	3.7	72.4	6.05	1.8	64.6	11.9
Products 1 to 4	4.5	87.6	5.17	1.5	66.7	12.6
Products 5	95.5	1872.9	0.12	0.5	33.3	87.4
Recalc. Feed	100.0	1960.5	0.35	0.5	100	100

Figure 5.19 – Rougher Flotation Summary for Test 12 Tails.

Figure 5.20 shows the grade-recovery curve for test 14, including the bulk gravity concentrate. Overall extraction of gold is 88.5%. The extraction and shape of the curve is nearly identical to that shown in Figure 4.4.

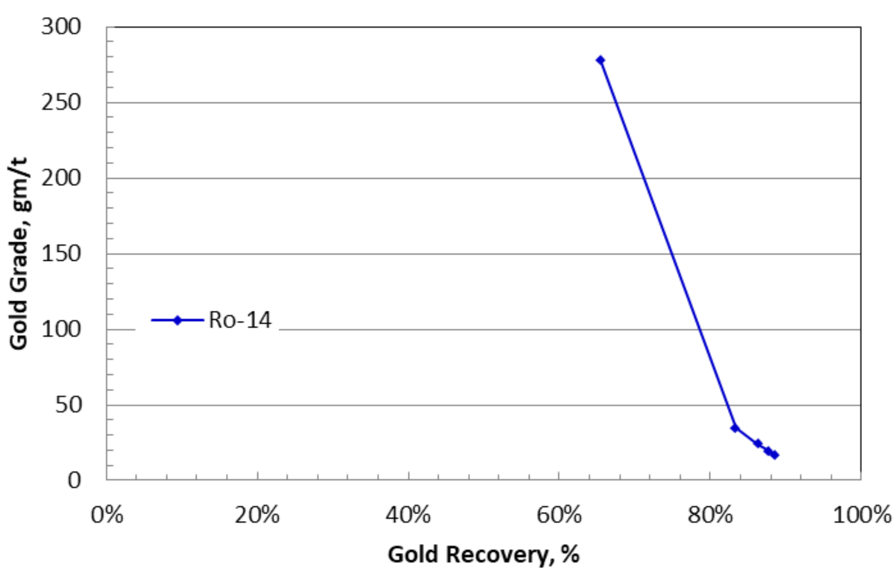


Figure 5.20 – Rougher Flotation Tests on Test 12 Gravity Tails.

Test 12 gravity tailings were floated using a Hydrofloat column which is summarized in Figure 5.21. It is apparent that the elutriating action of the column dilutes the concentrate considerably. Flotation using the hydrofloat column recovered roughly 41% of the gold in the feed which when combined with the gravity concentrate represents an overall recovery of 79.6% of the gold. This is substantially lower than conventional rougher flotation.

Product Test 23	Weight		Assay		Distribution, %	
	%	grams	Au, g/t	S, %	Au	S
(1) Elutriation O/F	65.7	10909.4	0.24	0.05	44.6	49.9
(2) Float Con 1	5.2	859.7	2.23	0.441	32.7	36.7
(3) Float Con 2	3.5	584.3	0.82	0.158	8.2	8.9
(4) Hydrofloat Tail	25.6	4260.6	0.20	0.011	14.5	4.5
Recalc. Feed	100.0	16614	0.35	0.06	100.0	100.0
Cum. Product	Weight		Assay		Distribution, %	
	%	grams	Au, g/t	S, %	Au	S
Product 1	65.7	10909.4	0.24	0.05	44.6	49.9
Products 1 to 2	70.8	11769.1	0.39	0.08	77.3	86.5
Products 1 to 3	74.4	12353.4	0.41	0.08	85.5	95.5
Products 2 to 3	8.7	1444.0	1.66	0.33	40.8	45.6
Product 4	25.6	4260.6	0.20	0.01	14.5	4.5
Recalc. Feed	100.0	16614.0	0.35	0.06	100.0	100.0
Flotation Only	Weight		Assay		Distribution, %	
	%	grams	Au, g/t	S, %	Au	S
Product 2	15.1	859.7	2.23	0.44	59.0	73.1
Products 2 to 3	25.3	1444.0	1.66	0.33	73.8	91.0
Product 4	74.7	4260.6	0.20	0.01	26.2	9.0
Recalc. Feed	100.0	5704.6	0.57	0.09	100.0	100.0

Figure 5.21 – Hydrofloat Tests on Test 12 Gravity Tails.

5.3 Conclusions:

Various conclusions can be made from the gravity testing:

1. The BAM master composite appears to be very amenable to gravity concentration due to free gold.
2. The liberation size for the gold appears to be finer than 300 micron.

Several conclusions can be made from the testing of the bulk gravity tails:

1. Sparging the leach tests with oxygen increased the extraction several percent over sparging with air resulting in an overall extraction decrease of 2% to 3%.
2. Variation of the concentration of NaCN in the range of 0.25 gm/liter to 1.0 gram/liter did not influence the ultimate extraction of gold in the leach tests.
3. Varying the slurry density between 25% solids and 40% solids did not have an impact on the gold extraction.
4. Adding activated carbon to the leach tests resulted in similar extraction of gold.
5. Flotation of the bulk gravity tails increased the gold extraction, however, this extraction was in a low grade concentrate and to a lesser extent than cyanide leaching.
6. Column flotation does not appear to provide a benefit.

6.0 Heap Leach Testing:

The fine and coarse master composites were placed in 8 inch diameter PVC pipes for column leach testing. The column was separated into two 3 meter sections by a flange. A 75mm tube was lowered into the center of the column and filled with ore. As more material was added to the top the tube was lifted to allow the material in the bottom to discharge gently and uniformly into the column. This was continued until each section was filled and a new section added. The total column height was approximately 6 meters.

Barren solution was made from tap water, adjusted to pH 10 – 10.5 with lime and made up to 0.60 gpl NaCN in a 5-gallon bucket. A separate bucket was made for each column. The bucket was weighed every morning to determine how much solution had been applied to each column in the preceding 24 hours.

Pregnant solution was allowed to free-drain from the bottom of the column and was collected in a bucket. This bucket was weighed each day, sampled and replaced with an empty bucket. Activated carbon was added to the pregnant solution and stirred, until the next morning when the carbon was removed, then added to the next day's pregnant solution. The solution after contact with the activated carbon was added to the barren solution bucket, mixed, assayed and made up for feed to the column.

Operating conditions of each of the column tests are summarized in Figure 6.1.

Solution application rate was set at 12 ℓ/hr.m^2. Each bucket was sampled for gold, silver, copper, pH, ORP and WAD cyanide. Pregnant and barren leach solutions generated during laboratory column and bottle-roll tests were analyzed for weak and dissociable (WAD) cyanide. WAD cyanide was the preferred method for determining cyanide, since competing metal ions in solution render the conventional free cyanide analysis inaccurate (free cyanide by silver nitrate titration). The WAD cyanide analytical method selected for the work was the MP-WAD technique as described by Botz et al. (2013). WAD cyanide QA/QC standards containing sodium cyanide and copper cyanide were routinely analyzed during the work. This data was recorded and is included as part of the lab report in Appendix B.

Test	Type	Wt., kg	NaCN gm/l	Leach ℓ/hr.m^2	Leach days	pH Target
CL-01	Coarse	149.2	0.60	12.0	89	10-10.5
CL-02	Fine	149.8	0.60	12.0	127	10-10.5

Figure 6.1 – BAM Column Leach Conditions.

Each column was leached to extinction and then rinsed with fresh water and drained. The "Leach Days" noted in Figure 6.1 are the active leach days and do not account for the rinse time.

6.1 Fine Scoping Column:

The fine texture of the minus 6.3 mm master composite raised concerns about the ability of the material to percolate in a heap leach. A small scoping column was set up in a clear 3" diameter column to a depth of 1.74 meters as shown in Figure 6.2. Solution was applied at a rate of 12 ℓ/hr.m^2. Observation of the material in the column indicates the movement of fines, as indicated by the layers of finer material accumulating in layers around the coarse particles as shown in Figure 6.3.

Figure 6.2 – Scoping Column on -6.3 mm Master Composite.

Once the column was fully wetted and no ponding observed, the column was filled with solution and then allowed to free drain. Figure 6.3 shows the level of the solution interface versus time. Also shown in Figure 6.3 is the drain down rate identified as an order of magnitude greater than the average application rate of 12 ℓ/hr.m^2. A typical rule-of-thumb for predicting the long term porosity of a heap is that the drain down rate of an unconsolidated column of ore must be 2 orders of magnitude greater than the average application rate to ensure adequate porosity is available in the heap to prevent formation of a phreatic head. Figure 6.4 shows that the drain down of the scoping column did not achieve 2 orders of magnitude greater rate with a typical application rate of 12 ℓ/hr.m^2.

Figure 6.3 – Scoping Column Detail.

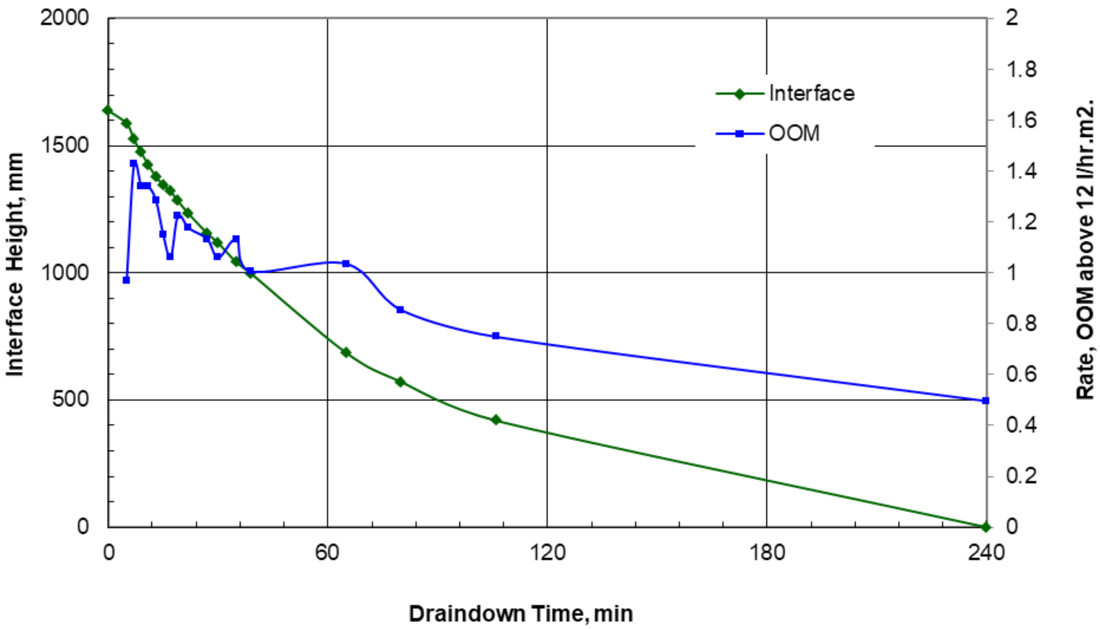


Figure 6.4 – Scoping Percolation Data.

After draining, the level of the ore column was measured and a slump of 9.9% was calculated. This level of slump is typically associated with a marginally acceptable heap leach material. It is unlikely fine BAM ore could be leached commercially without agglomeration.

Figure 6.5 shows the scoping column material after discharge. The observation that the material retains the shape of the column is a good indicator that fine crushed BAM ore may be unsuitable for heap leaching.

Figure 6.5 – Scoping Column Material after Discharge.

6.2 <u>Leach Columns:</u>

Six meter high, 6 inch diameter PVC columns were filled with each of the master composites. Both of these columns were placed under leach at 12 ℓ/hr.m^2 with leach solution containing 600 ppm NaCN and a pH of 10.5. The columns are being operated in locked cycle with carbon.

Figure 6.6 summarizes the results of the column leach tests. The percent extractions were calculated using solution assays and "calculated head" using the tails assays. This method takes into account any feed grade variation.

	Leach Time	Extraction			Consumption, kg/t	
	days	Gold, %	Silver, %	Copper,%	NaCN	CaO
Coarse - CL-01	89	56.4%	14.2%	11.2%	0.06	0.06
Fine CL-02	127	88.8%	22.7%	10.2%	0.12	0.07

Figure 6.6 –Column Test Results.

Figure 6.7 shows the leach rate curves for gold, silver and copper for both the coarse crushed column (CL-01) and the fine crushed column (CL-02). The extraction rate for gold shows a typical response to column leaching. The initial rapid rate from the easily leached component is followed by a slower rate for unliberated gold. The extraction of gold is higher in the fine crush column, which is also typical.

Silver and copper extraction is low in both columns although the finer crushing increased the extraction of silver.

Obviously the coarse composite extraction rate may be solely dependent on the coarse size. However, platy material can be a poor candidate for heap leaching due to a "shingle" effect where the flat plates stack and align in the heap like shingles on a roof to create percolation "shadows" that are not effectively leached.

The abrupt change in the extraction rate for the fine column at Day 72 was due to reduced free cyanide in the pregnant solution. The CN to copper ratio requires values greater than 4:1 to maintain leaching rates. The CN:Cu ratio dropped to below this value prior to being increased by the addition of cyanide. After the addition of cyanide the extraction rate increased.

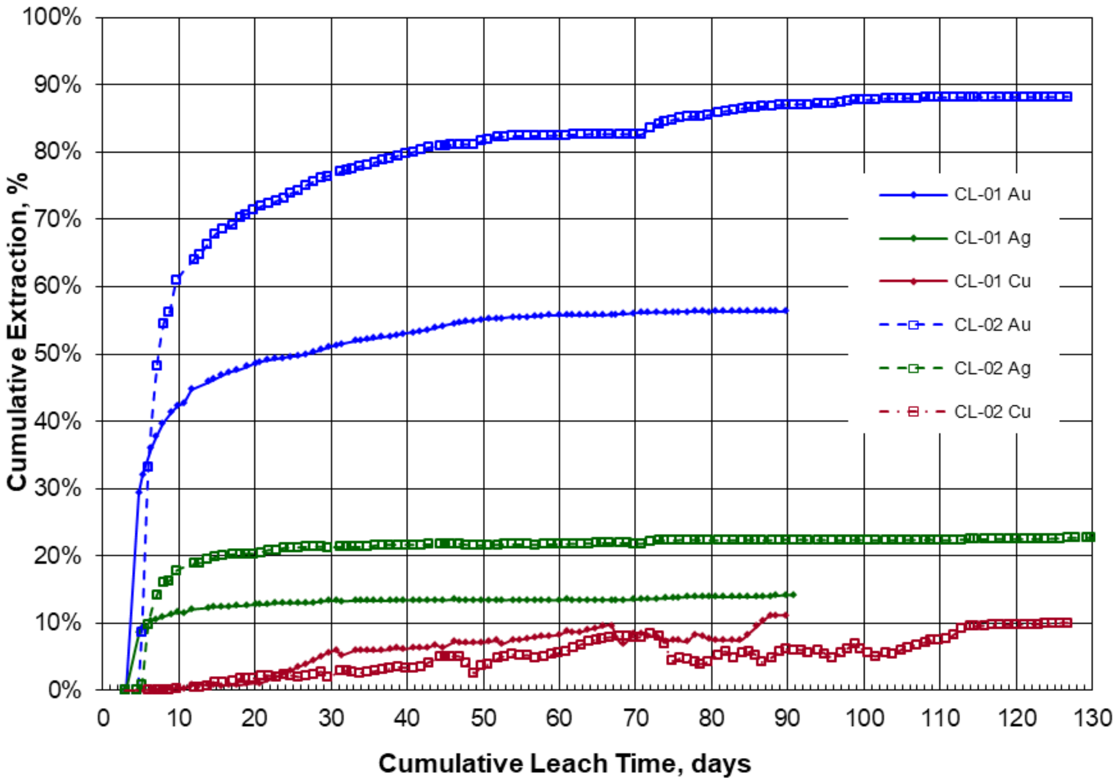


Figure 6.7 – Column Test Extraction vs. Time.

Figure 6.8 shows the cumulative weight percent passing of the feed and tails from the column tests. The only notable feature in Figure 6.8 is the difference between the feed and tails particle size distributions (PSD) for the fine crushed column. The tails material was rescreened to confirm this difference. Due to the irregularity of the feed PSD, it is difficult to place much significance in the difference.

Figure 6.9 shows the gold assay at each size interval for the column test feed and tails at each size interval. What is evident from this figure is that crushing below about 425 microns is required for high extraction of gold. The bulk of the un-extracted gold is in the fractions larger than 425 microns.

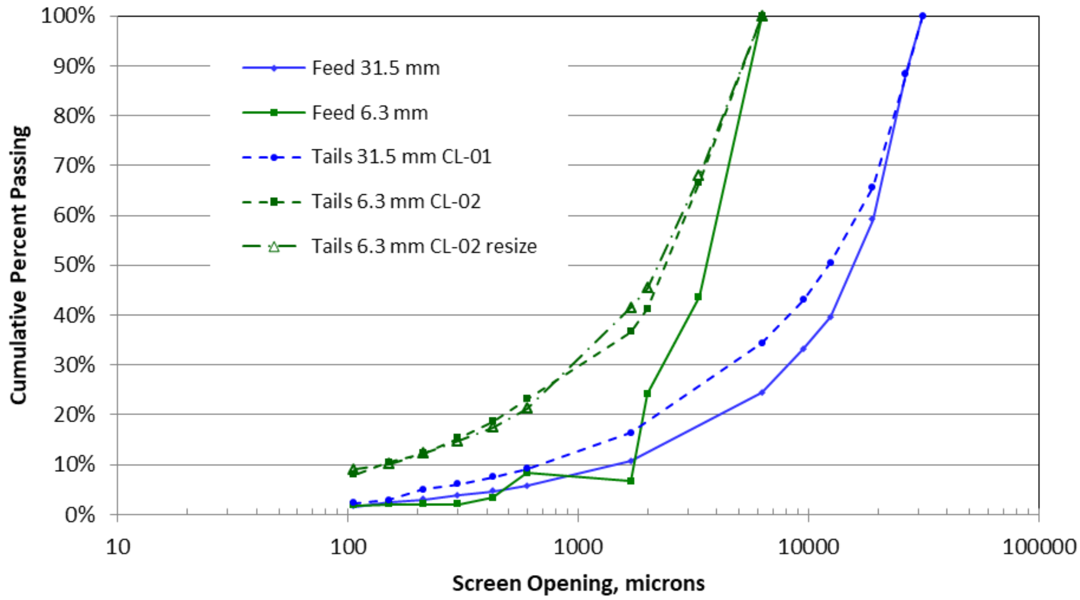


Figure 6.8 – Column Test PSD of Feed and Tails.

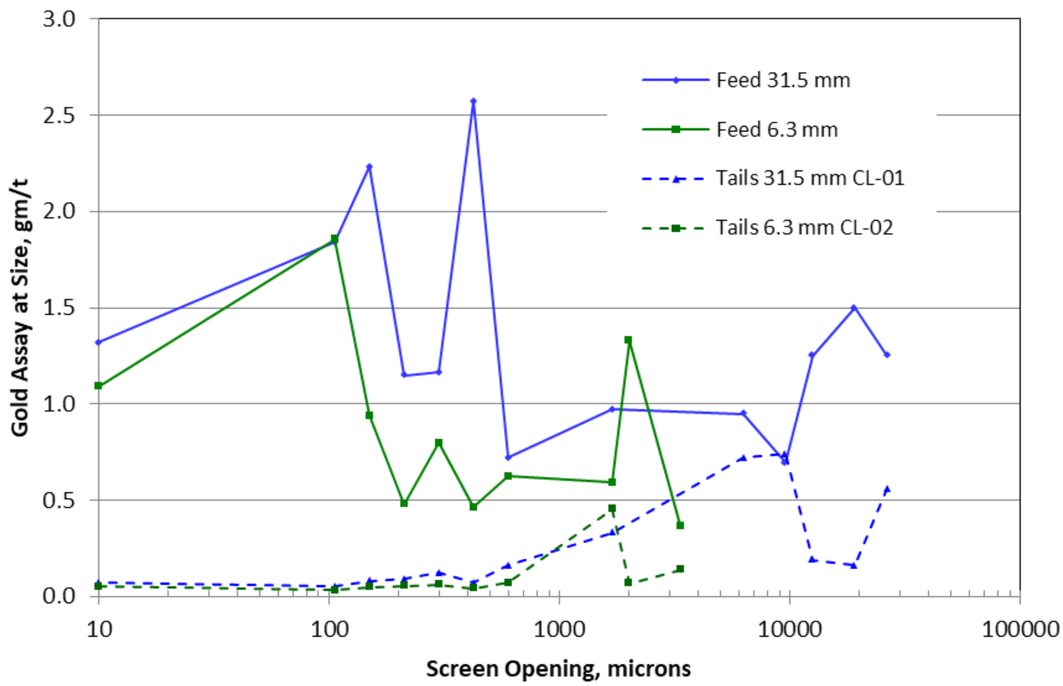


Figure 6.9 – Column Test Gold Assay by Size.

Figure 6.10 shows the distribution of weight, gold and copper as percentage at each screen size for the coarse crushed column (CL-01). Comparing this with Figure 2.6 for the minus 31.5 mm feed distributions shows a similar trend for the distributions. The distribution of weight in the tails shows a similar distribution, in comparison to the feed, with peaks at 1,700 micron and 12,500 microns. However, the curve for the tails appears to show that some of the coarser material reports to the peak at 1,700 microns. Since the amount of material in the finest fraction did not increase, the change in the PSD is unlikely to be due to degradation during leaching, but may be an artefact of sampling and screening. Materials that have a high aspect ratio, as demonstrated by the coarse crushed composite are sensitive to residence time during screening.

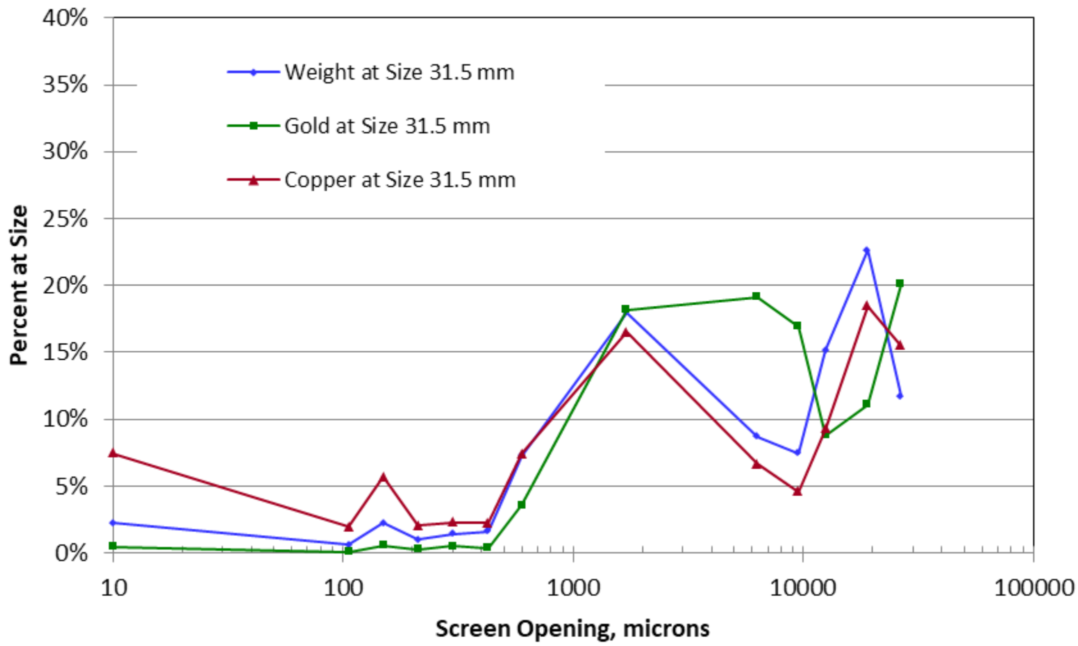


Figure 6.10 – Column Tails Weight, Gold and Copper Distribution at Size, CL-01.

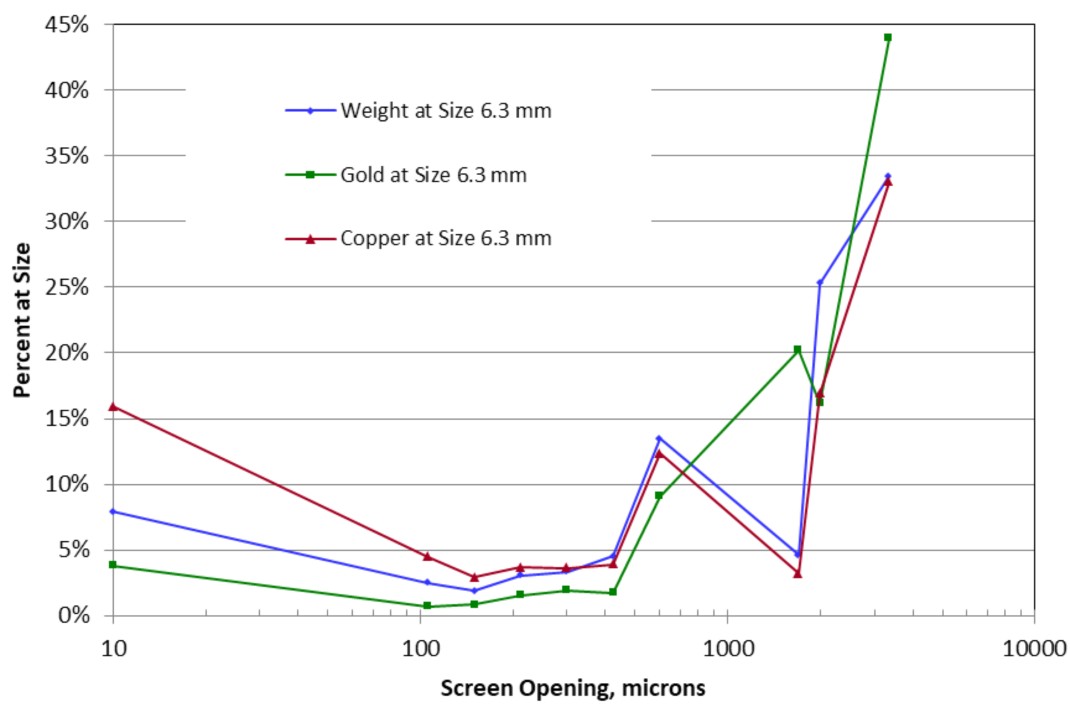


Figure 6.11 – Column Tails Weight, Gold and Copper Distribution at Size, CL-02.

Comparing the gold curves for the feed in Figure 2.6 with those for the tails in Figure 6.10 shows that the bulk of the un-leached gold resides in the coarsest fractions (79% retained on 6,300 microns), which is as would be expected. The presence of coarse gold in the composites results in the variation in the gold distribution in the coarsest fractions.

Silver was assayed in the tails at a low and constant level in all the fractions. This would result in a silver distribution curve identical to the weight distribution curve in Figure 6.10.

The copper distribution is relatively unchanged between the feed and tails. This response would indicate that the copper species present is only slightly soluble in cyanide and not closely dependent on particle size.

Figure 6.11 shows the distribution of weight, gold and copper as a percentage at each screen size for the fine crushed column (CL-02). Comparing this with Figure 2.7 for the minus 6.3 mm feed indicates small changes in PSD that are likely due to sampling/screening variations. Degradation from leaching is not readily apparent from these figures.

Figure 6.11 shows that the majority of the un-leached gold in the tails is in the coarser sizes, which is consistent with the coarse crushed column.

Silver tails assays are low and uniform across all the size intervals so that the curve would be identical to the weight distribution curve in Figure 6.11. The copper distribution is reasonably unchanged between the feed and tails. This is consistent with the coarse column.

6.3 <u>Summary</u>:

Column leach testing examines the ultimate extractions under closely controlled conditions. It is impractical to expect field operations to maintain similar control of the heap leach. In order to predict a practical field extraction, the laboratory extractions are reduced by a factor of 3% to 5 % to account for field inefficiencies. For the purposes of this exercise a 4% deduct was applied to the interim laboratory results to obtain a field extraction. This methodology results in the following field gold extractions:

Coarse Master Composite Gold Extraction:	±52%
Fine Master Composite Gold Extraction:	±85%

The 4% deduct is typical for a well run heap in a benign operating environment. The climate at the site does not qualify as a benign environment for heap leaching. Additional deduct should be expected, although the magnitude of this deduct cannot be determined without significant additional effort.

Cyanide and lime consumptions, as identified in Figure 6.6, are low. This is surprising due to the presence of pyrrhotite which is a known cyanide consumer.

Cement consumption for a fine crushed heap leach is estimated at 3 to 4 kg/t which is an assumption based on experience. The actual cement requirement was not tested during the Phase 1 program.

6.4 <u>Conclusions/Discussion</u>:

Leach column testing of the BAM master composite resulted in several conclusions:

1. Fine crushing is required for acceptable extractions of gold.
2. Fine crushed BAM material will most likely require agglomeration to maintain percolation in a multi-lift heap.
3. Extraction of gold from coarse crushed material is low.
4. Cyanide and lime consumptions are low.

It should be noted that cold temperature operation of heap leaches can be difficult. Lower temperatures reduce the extraction rate which will extend the leach time. Agglomeration in freezing conditions reduces the quality of the agglomerates due to poor mixing and reduced strength of the cured cement.

Freezing conditions may reduce operating cycles to the summer months. Freezing on the heap surfaces will break up agglomerates leaving a low permeability zone between lifts. Conventional carbon columns can freeze or plug with ice, curtailing production.

7.0 Mineral Association Analysis:

Samples of various concentrates of the master composite were sent to the Center for Advanced Mineral Processing (CAMP) at Montana Tech. The test report is included in Appendix C. The following samples were sent:

Test 03 – Gravity Tails
Test 03 – Gravity Concentrate
Test 12 – Bulk Gravity Tails
Test 12 – Bulk Gravity Concentrate
Test 14 – Bulk Gravity Tails Rougher Concentrate
Test 14 – Bulk Gravity Tails Rougher Tails

Each sample was inspected by scanning electron microscopy/energy dispersive x-ray (SEM-EDS) to determine the mineral species and occurrences and their associations with gold.

7.1 Summary:

Gold occurrences in the concentrates were predominantly grains with a high aspect ratio with the length to width ratio around 3:1. The gold particles were observed as free grains and associated with chlorite. Gold also occurred attached to and included in cobaltite (CoAsS) mineral grains. A minor association was observed with silver and bismuth tellurides. Only one grain of gold was observed in the tails of indeterminate genesis. Observed gold grains had a gold content greater than 90%.

Sulfides and arsenides were concentrated in the gravity concentrates and flotation concentrates. The primary sulfides were pyrite and pyrrhotite. Concentrations of these minerals were higher in the gravity concentrate than the flotation concentrate. Tramp iron was increased in the gravity concentrate.

Gangue minerals in the sample were predominantly silicates.

7.2 Discussion:

Very little in the report identifies the specific mechanism that allows gold to report to the tailings. The gravity concentrate is coarser than the gravity tails, which would be expected. It is likely gold losses to the gravity tails consist of very fine gold particles either attached or included in gangue minerals. High cyanide extractions of the tails would support this.

The presence of gold associated with tellurides may account for the incomplete cyanide leaching of the tails and for the improved leach extraction with oxygen over air spaging.

Cyanide dissolution rate of cobaltite is unknown, however cobalt cyanides are soluble.

Tramp metal and pyrrhotite identified in the gravity concentrate will consume cyanide during leaching and should be removed with magnetic separation prior to leaching.

<< End >>

APPENDIX A

Sample Intervals

Legend
Diamond Drill Hole
Diamond Drill Hole 2018
Metallurgical Drill Hole
Trench
MaxMin Conductor
VLF Anomaly
Cut Grid Line
Road/trail
Landore Resources Canada Inc.
Junior Lake Property
BAM Gold Deposit
Drilling
Date: 05/11/2018
Author: GIS
Office: TBay
Scale: 1:5000
Projection: UTM Zone 16 (NAD 83)
0 75 150 300
metres
Magnetic Declination 2017
5°28 West
N
Ladle Flats
TL 600N
TL 300N
BL 00
TL 300S

Hole ID	MET	From	To	Sample ID	Bag Number	Received Wt
0418-653	MET 1	138	139	W868716	Bag 1	
0418-653	MET 1	139	140	W868717	Bag 2	15.52
0418-653	MET 1	140	141	W868718	Bag 3	
0418-653	MET 1	141	142	W868719	Bag 4	16.10
0418-653	MET 1	142	143	W868720	Bag 5	
0418-653	MET 1	143	144	W868721	Bag 6	16.24
0418-653	MET 1	144	145	W868722	Bag 7	
0418-653	MET 1	145	146	W868723	Bag 8	16.20
0418-653	MET 1	146	147	W868724	Bag 9	
0418-653	MET 1	147	148	W868725	Bag 10	15.78
0418-653	MET 1	148	149	W868726	Bag 11	
0418-653	MET 1	149	150	W868727	Bag 12	16.14
0418-653	MET 1	150	151	W868728	Bag 13	
0418-653	MET 1	151	152	W868729	Bag 14	15.78
0418-653	MET 1	152	153	W868730	Bag 15	
0418-653	MET 1	153	154	W868731	Bag 16	15.68
0418-653	MET 1	154	155	W868732	Bag 17	
0418-653	MET 1	155	156	W868733	Bag 18	16.46
0418-653	MET 1	156	157	W868734	Bag 19	
0418-653	MET 1	157	158	W868735	Bag 20	16.14
0418-653	MET 1	158	159	W868736	Bag 21	
0418-653	MET 1	159	160	W868737	Bag 22	15.94
0418-653	MET 1	160	161	W868738	Bag 23	
0418-653	MET 1	161	162	W868739	Bag 24	15.90
0418-653	MET 1	162	163	W868740	Bag 25	
0418-653	MET 1	163	164	W868741	Bag 26	15.36
0418-653	MET 1	164	165	W868742	Bag 27	
0418-653	MET 1	165	166	W868743	Bag 28	15.50
0418-653	MET 1	166	167	W868744	Bag 29	
0418-653	MET 1	167	168	W868745	Bag 30	16.38
0418-653	MET 1	168	169	W868746	Bag 31	
0418-653	MET 1	169	170	W868747	Bag 32	16.02
0418-653	MET 1	170	171	W868748	Bag 33	
0418-653	MET 1	171	172	W868749	Bag 34	16.32
0418-653	MET 1	172	173	W868750	Bag 35	
0418-653	MET 1	173	174	W868751	Bag 36	15.74
0418-653	MET 1	174	175	W868752	Bag 37	
0418-653	MET 1	175	176	W868753	Bag 38	16.60
0418-653	MET 1	176	177	W868754	Bag 39	
0418-653	MET 1	177	178	W868755	Bag 40	16.30
0418-653	MET 1	178	179	W868756	Bag 41	
0418-653	MET 1	179	180	W868757	Bag 42	16.78
0418-653	MET 1	180	181	W868758	Bag 43	
0418-653	MET 1	181	182	W868759	Bag 44	16.46
0418-653	MET 1	182	183	W868760	Bag 45	
0418-653	MET 1	183	184	W868761	Bag 46	16.10
0418-653	MET 1	184	185	W868762	Bag 47	
0418-653	MET 1	185	186	W868763	Bag 48	16.30
0418-653	MET 1	186	187	W868764	Bag 49	
0418-653	MET 1	187	188	W868765	Bag 50	15.88
0418-653	MET 1	188	189	W868766	Bag 51	
0418-653	MET 1	189	190	W868767	Bag 52	15.54
0418-653	MET 1	190	191	W868768	Bag 53	
0418-653	MET 1	191	192	W868769	Bag 54	16.16
0418-653	MET 1	192	193	W868770	Bag 55	
0418-653	MET 1	193	194	W868771	Bag 56	16.02
0418-653	MET 1	194	195	W868772	Bag 57	
0418-653	MET 1	195	196	W868773	Bag 58	16.04
Total						**465.38**

Hole ID	MET	From	To	Sample ID	Bag Number	Received Wt
0418-654	MET 2	124	125	W868863	Bag 1	
0418-654	MET 2	125	126	W868864	Bag 2	16.36
0418-654	MET 2	126	127	W868865	Bag 3	
0418-654	MET 2	127	128	W868866	Bag 4	16.08
0418-654	MET 2	128	129	W868867	Bag 5	
0418-654	MET 2	129	130	W868868	Bag 6	15.88
0418-654	MET 2	130	131	W868869	Bag 7	
0418-654	MET 2	131	132	W868870	Bag 8	15.82
0418-654	MET 2	132	133	W868871	Bag 9	
0418-654	MET 2	133	134	W868872	Bag 10	16.22
0418-654	MET 2	134	135	W868873	Bag 11	
0418-654	MET 2	135	136	W868874	Bag 12	16.14
0418-654	MET 2	136	137	W868875	Bag 13	
0418-654	MET 2	137	138	W868876	Bag 14	16.16
0418-654	MET 2	138	139	W868877	Bag 15	
0418-654	MET 2	139	140	W868878	Bag 16	15.96
0418-654	MET 2	140	141	W868879	Bag 17	
0418-654	MET 2	141	142	W868880	Bag 18	16.32
0418-654	MET 2	142	143	W868881	Bag 19	
0418-654	MET 2	143	144	W868882	Bag 20	16.24
0418-654	MET 2	144	145	W868883	Bag 21	
0418-654	MET 2	145	146	W868884	Bag 22	16.46
0418-654	MET 2	146	147	W868885	Bag 23	
0418-654	MET 2	147	148	W868886	Bag 24	16
0418-654	MET 2	148	149	W868887	Bag 25	
0418-654	MET 2	149	150	W868888	Bag 26	15.94
0418-654	MET 2	150	151	W868889	Bag 27	
0418-654	MET 2	151	152	W868890	Bag 28	16.02
0418-654	MET 2	152	153	W868891	Bag 29	
0418-654	MET 2	153	154	W868892	Bag 30	16.24
0418-654	MET 2	154	155	W868893	Bag 31	
0418-654	MET 2	155	156	W868894	Bag 32	16.28
0418-654	MET 2	156	157	W868895	Bag 33	
0418-654	MET 2	157	158	W868896	Bag 34	16.16
0418-654	MET 2	158	159	W868897	Bag 35	
0418-654	MET 2	159	160	W868898	Bag 36	16.54
0418-654	MET 2	160	161	W868899	Bag 37	
0418-654	MET 2	161	162	W868900	Bag 38	16.46
0418-654	MET 2	162	163	W868901	Bag 39	
0418-654	MET 2	163	164	W868902	Bag 40	16.5
0418-654	MET 2	164	165	W868903	Bag 41	
0418-654	MET 2	165	166	W868904	Bag 42	16.74
0418-654	MET 2	166	167	W868905	Bag 43	
0418-654	MET 2	167	168	W868906	Bag 44	15.74
0418-654	MET 2	168	169	W868907	Bag 45	
0418-654	MET 2	169	170	W868908	Bag 46	15.62
0418-654	MET 2	170	171	W868909	Bag 47	
0418-654	MET 2	171	172	W868910	Bag 48	16.24
Total						388.12

APPENDIX B

Base Metals Laboratory

Metallurgical Flowsheet Evaluation on a Composite Sample from the BAM East Project

BL0295

January 10, 2019

METALLURGICAL FLOWSHEET EVALUATION ON A COMPOSITE SAMPLE FROM THE BAM EAST PROJECT BL0295 – FINAL REPORT

Contents

Appendices:

BRADLEY ANGOVE
P. ENG., MAUSIMM(CP)

PROFESSIONAL PROVINCE OF B. S. ANGOVE # 40973 BRITISH COLUMBIA ENGINEER
Jan 10 2019

CAMERON BRUIN
EIT

January 10, 2019
Final Report

1.0 Introduction

This project investigated one composite from the BAM East Project, located in Ontario. Geoff Allard, of Allard Engineering Services LLC, is the client's representative for this project, and provided direction and guidance for the test work program.

The objective of this program was to evaluate flowsheet options for the processing of the BAM East Gold ore; these being a heap leaching process, or a milling process with cyanidation or flotation flowsheet. Previous testing had indicated a high proportion of the gold reporting to a gravity concentrate, these details were provided by the client as the testing had been conducted at another laboratory. In addition, this test program obtained data related to ore physical properties.

To complete the above objectives, a single metallurgical composite was prepared and designated Master Composite (MC). The composite was stage crushed to 100% passing 31.5mm (-31.5mm), thoroughly homogenized and split in half. Ore hardness testing sub-samples were taken from the coarse fraction with the remainder being stored until required for testing. One split was stage crushed to 100% passing 6.3mm (-6.3mm) and thoroughly homogenized. Representative head cuts were removed and assayed for elements of interest and were also subjected to head assays by size. Sub-samples were also removed and subjected to bottle roll leach testing, scoping column leach testing and column leach testing with all products being assayed for gold and silver. Selected test products were also assayed for copper and elements of interest.

Samples for this test program were received at Base Metallurgical Laboratories Ltd. on July 4, 2018. The shipment contained about 854 kilograms of sample, received as whole drill core. Details can be found in Appendix A.

This report summarizes key results from the test program, using data summaries and graphical displays. Detailed results, such as condition sheets and full sizing distributions, can be found in the Appendices as follows:

- Appendix A: Chain of Custody
- Appendix B: Metallurgical Testing
- Appendix C: Assays
- Appendix D: Comminution Testing
- Appendix E: Sizings

2.0 Comminution

Bond ball and rod mill work index, SAG Mill Comminution (SMC) tests, and Abrasion index tests were conducted on the MC sample. Full comminution test results and the JKTech report are provided in Appendix D. A summary of the results are shown in Table 1.

TABLE 1: COMMINUTION TEST RESULT SUMMARY

Sample	Bond BWI	Bond RWI	Abrasion Index	Axb	**SCSE** KWh/tonne
	kWh/tonne	kWh/tonne	Ai		
MC	15.9	16.2	0.14	52.2	8.74

The Bond ball mill work index test was conducted using a closing screen sizing of 106µm, resulting in a product sizing of 75µm K_{80}. At this closing screen size, a Bond ball work index of 15.9 kWh/tonne was determined. The Bond rod mill work index of the MC was determined to be 16.2 kWh/tonne. These values indicate the mineralization to be moderately hard with respect to ball and rod mill grinding. The Abrasion index for this sample was determined to be 0.14, classifying the sample as mildly abrasive. The SMC derived A x b value was determined to be 52.2 for this sample, while the SCSE measured 8.74 kWh/tonne, indicating the sample is moderately hard with respect to breakage in a SAG Mill.

3.0 Chemical Content

Previous testing and information provided by the client indicated a high proportion of the gold to be free native gold, which can cause variation between head assays. Eight representative head cuts were removed from the MC sample and assayed for gold along with a screen metallic assay. Duplicate assays were also performed for iron, sulphur, silver and copper. A single ICP and WRA assay was also conducted. A summary of average results for the tested elements are shown in Table 2.

TABLE 2: HEAD ASSAYS

Sample	Assay - percent or g/tonne				
	Au	Fe	S	Cu	Ag
MC	0.95	2.3	0.12	0.007	1
Screen Metallics	0.84				

Note: Gold and silver are reported in g/tonne, others in percent.
Detailed data can be found in Appendix C.

Gold content in the sample was assayed at 0.95 g/tonne, silver at 1 g/tonne and copper at 0.007 percent.

4.0 Metallurgical Testing

A series of metallurgical tests were conducted on the MC sample, evaluating several flowsheet options. Results are discussed in detail in the following subsections.

4.1 GRG Testing

A three stage Gravity recoverable gold test was conducted on the sample. About 19 kg of sample was stage crushed to 100% passing 1.7mm and subjected to gravity concentration. The tailings were ground to a K_{80} of 150µm, and again processed through the Knelson concentrator. A final grind and pass was conducted at a K_{80} of about 70µm. A summary of results is shown in Table 3.

TABLE 3: GRG SUMMARY

Sample ID	Product	Weight		Assay - g/t	Dist'n
		%	grams	Au	(%)
Master Composite	Knelson Con 1	0.6	96.3	60.2	28.7
	Knelson Con 2	0.4	67.1	120	39.8
	Knelson Con 3	0.4	60.8	46.2	13.9
	Knelson Tail 3	98.7	16812	0.21	17.5
Recalculated Feed				17036	1.18
GRG (%)					82.5

Note: Detailed test data can be found in Appendix B.

The total gold recovered to the gravity concentrates from all three stages measured about 82.5 percent, at a combined gold grade of 74 g/tonne. This indicates the sample is very amenable to gravity concentration. The majority of the gold was recovered during the second stage, indicating that grinding of the sample is required to achieve significant (>50% recovery) gold recovery via gravity separation.

4.2 Heap Leach Amenability Tests

Two column leach tests were conducted, one on each of the coarse crushed (-31.5mm) and fine crushed (-6.3mm) MC sample. Test conditions were chosen by the client and results are discussed below.

The columns were operated continuously with frequent solution and carbon assays to determine kinetic leach extraction rates of the samples. A summary of the results are presented in Table 4 and Figures 1 and 2.

The tests were conducted in locked cycle with activated carbon. The ore was placed in 6-inch diameter columns to a depth of 6 meters. Test CL-01 (coarse crush) ran for a duration of 91 days, while Test CL-02 (fine crush) ran for a duration of 131 days. The feed and pregnant solution from the tests were weighed and assayed for gold, silver and copper every day. Pregnant leach solution was contacted with carbon for adsorption of the metals, before recycling the barren solution to the column feed.

For CL-01, 57 percent of the gold was extracted, while silver and copper extractions were low at 14 and 7 percent, respectively. Gold extraction for the finer crush test was significantly higher; 89 percent of the gold from CL-02 was extracted, while silver and copper extractions were low at 22 and 9 percent, respectively.

For CL-01, leach kinetics show that the gold leach extraction levelled off after about 50 days, while for CL-02, the gold extraction levelled off after about 90 days.

Detailed data sheets can be found in Appendix B.

TABLE 4: COLUMN LEACH TESTS SUMMARY

Test	Duration	Flowrate	pH		Extraction - percent			NaCN Consumption
	days	ℓ/hr.m2	range	average	Au	Ag	Cu	kg/tonne
CL-01, coarse crush	91	12	9.2-12.5	10.7	56.6	14.0	7.1	0.05
CL-02, fine crush	131	12	9.4-12.4	10.5	89.1	22.4	9.0	0.07

FIGURE 1: KINETIC LEACH RATES CL-01

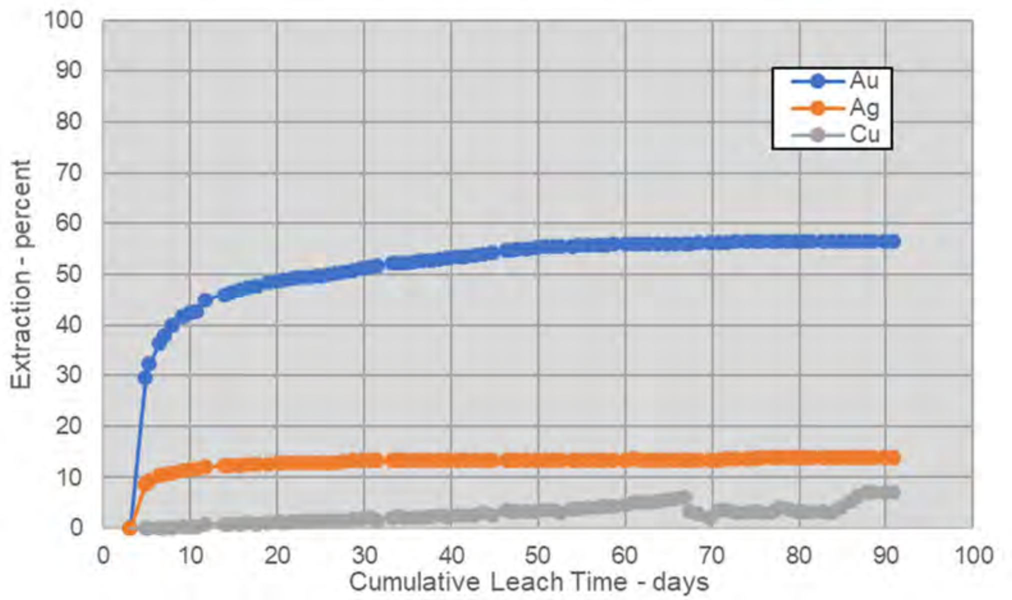
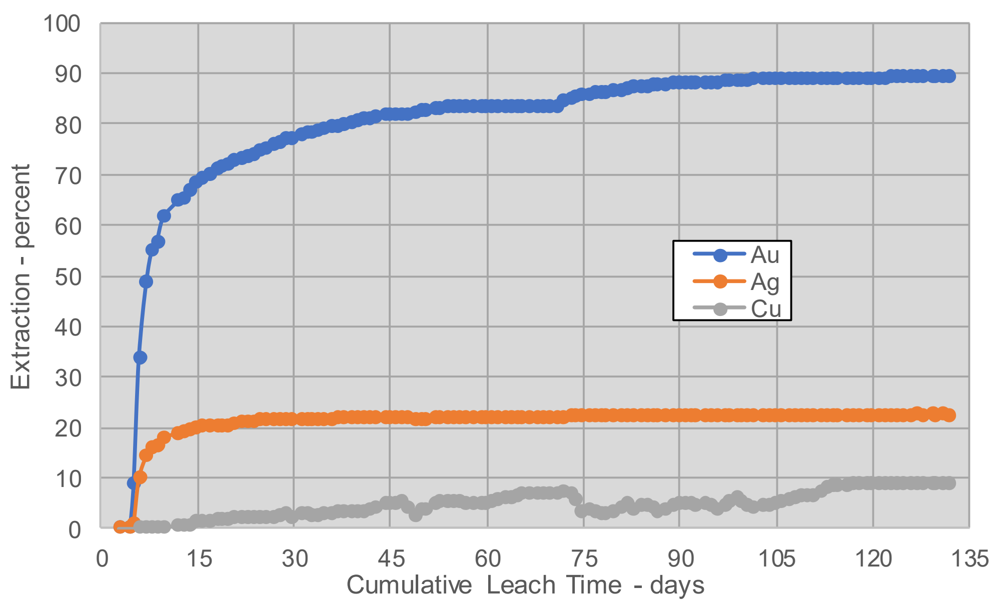


FIGURE 2: KINETIC LEACH RATES CL-02



4.3 Assay by Size and Recovery

A 30-kilogram sub-sample from both the fine crushed (-6.3mm) and coarse crushed (-31.5mm) material of the Master Composite was split for a head screen analysis. The splits were screened into 10 to 13 size fractions and assayed for gold, silver and copper. A portion of the tailings from each column test was also split into these fractions and assayed, to determine extraction by size fraction. Detailed assays and particle size distribution data can be found in Appendix C. A summary of the gold assays by size, along with the calculated extraction by size for each column is also shown in Table 5.

In the coarse crushed material, just over 90 percent of the gold was contained in the +600µm fraction and a similar distribution pattern was noted for the silver and copper. In the fine crushed material, over 75 percent of the gold and silver and 70 percent of the copper were contained in the +600µm fraction, while 9 percent of the total gold was contained in the -106µm fraction.

The extraction by size data indicates that higher extractions are achieved at the finer particle size fractions. It should be noted that there was significant variation for some fraction sizes between the head cuts and triplicate gold assays, indicating the presence of coarse gold or gold nugget effect.

TABLE 5: ASSAY BY SIZE – COLUMN TEST RESIDUE

Sample ID	Size Fraction µm	Feed		Tails		Extraction By Size Fraction percent
		Mass	Assay - g/t	Mass	Assay - g/t	
		%	Gold	%	Gold	
MC Coarse Crush	26500	11.4	1.25	11.7	0.94	25
	19000	29.5	1.50	22.6	0.16	89
	12500	19.6	1.25	15.2	0.19	85
	9500	6.4	1.27	7.5	1.08	15
	6300	8.8	1.17	8.7	0.99	16
	1700	13.7	0.97	18.0	0.33	66
	600	5.0	0.72	7.3	0.16	78
	425	1.1	2.57	1.6	0.07	97
	300	0.9	1.16	1.4	0.12	90
	212	0.7	1.15	1.0	0.09	92
	150	0.7	2.23	2.2	0.08	96
	106	0.7	1.84	0.6	0.05	97
	-106	1.6	1.32	2.2	0.07	95
Calc Head		100	1.28	100	0.42	
MC - Fine Crush	3350	43.6	0.37	33.4	0.11	69
	2000	24.2	1.33	25.3	0.09	93
	1700	6.7	0.59	4.6	0.31	47
	600	8.3	0.62	13.5	0.07	89
	425	3.4	0.46	4.5	0.04	91
	300	2.0	0.80	3.3	0.06	92
	212	2.0	0.48	3.0	0.05	89
	150	2.0	0.94	1.9	0.05	95
	106	1.8	1.86	2.5	0.03	98
	-106	6.0	1.09	7.9	0.05	95
Calc Head		100	0.73	100	0.10	

4.4 Free Milling Amenability Tests

A series of cyanide bottle roll tests, rougher tests, and gravity concentration tests were conducted to assess gold extraction from the MC sample. A summary of test data and results are shown below in Table 6.

TABLE 6: TEST SUMMARY

Test	Method	PG µm K80	Sparge Gas	% Solids w/w	Duration hrs	NaCN ppm	Au Extraction - percent Leach	Gravity	Rougher	Overall	Tls Grade Au- g/tonne	Rgnt Cons. - kg/t NaCN	Lime
1	Leach	Pulv	O2	33	96	1000	99.6	-	-	**99.6**	**0.04**	0.71	0.63
2	Leach	Pulv	O2	33	96	5000	96.4	-	-	**96.4**	**0.08**	2.86	0.29
3	Rougher+Grav on RoTls	75	-	-	-	-	-	8.4	84.1	**92.5**	**0.07**	-	-
4	Gravity + Rougher	75	-	-	-	-	-	79.6	9.9	**89.5**	**0.07**	-	-
5	Leach	75	O2	33	96	1000	96.8	-	-	**96.8**	**0.02**	0.46	0.50
6	Leach	75	O2	33	96	1000	97.1	-	-	**97.1**	**0.01**	0.45	0.50
7	Grav - Leach	150	O2	33	72	1000	28.8	70.3	-	**99.2**	**0.01**	0.45	0.30
8	Grav - Leach	106	O2	33	72	1000	27.3	71.3	-	**98.5**	**0.02**	0.39	0.30
9	Grav - Leach	75	O2	33	72	1000	43.0	55.3	-	**98.4**	**0.02**	0.45	0.30
10	Grav - Leach	53	O2	33	72	1000	37.0	61.5	-	**98.5**	**0.02**	0.45	0.30
11	GRG	-	-	-	-	-	-	82.5	-	**82.5**	**0.21**	-	-
12	Bulk Gravity	150	-	-	-	-	-	65.5	-	**65.5**	**0.30**	-	-
12,13	Bulk Grav + Leach	150	Air	33	72	1000	30.1	65.5	-	**95.6**	**0.04**	0.27	0.80
12,14	Bulk Grav + Ro	150	-	-	-	-	-	65.5	23.0	**88.5**	**0.12**	-	-
12,15	Bulk Grav + Seq. Grav	150	-	-	-	-	-	65.5+11.8	-	**77.3**	**0.24**	-	-
12,16	Bulk Grav + Leach	150	O2	33	72	1000	32.7	65.5	-	**98.2**	**0.02**	0.34	0.57
12,17	Bulk Grav + Leach	150	O2	33	72	750	32.8	65.5	-	**98.3**	**0.02**	0.21	0.55
12,18	Bulk Grav + Leach	150	O2	33	72	500	32.7	65.5	-	**98.2**	**0.02**	0.02	0.63
12,19	Bulk Grav + Leach	150	O2	33	72	250	32.3	65.5	-	**97.8**	**0.02**	0.04	0.60
12,20	Bulk Grav + Leach	150	O2	20	72	1000	31.9	65.5	-	**97.4**	**0.03**	0.43	0.85
12,21	Bulk Grav + Leach	150	O2	40	72	1000	32.9	65.5	-	**98.4**	**0.02**	0.43	0.42
12,22	Bulk Grav + CIL Leach	150	O2	33	72	1000	33.2	65.5	-	**98.7**	**0.02**	1.72	0.70
12,23	Bulk Grav + Hydrofloat*	150	-	-	-	-	-	65.5	29.5	**95.0**	**0.20**	-	-

Note: Hydrofloat recovery includes elutriation O/F which would need further processing.
PG indicates Primary Grind Sizing
Detailed Results can be found in Appendix B.

The sample was subjected to 96-hour whole ore cyanidation bottle roll tests. Tests 1 and 2 investigated extraction on pulverized sample at two cyanide concentrations, while Tests 5 to 6 investigated a 75µm K_{80} primary grind, conducted in duplicate at 1000ppm cyanide. High gold extractions of up to 99.6 percent were obtained in these tests. Increasing the cyanide concentration did not increase gold extraction.

A gravity-leach flowsheet was tested at primary grind sizing K_{80s} ranging from 53 to 150µm, Tests 7 to 10. Changing the feed size distribution did not affect gold extraction over the range tested; gold extraction ranged from 98.4 to 99.2 percent. The majority of the gold was recovered in gravity concentration, extracting between 55 and 71 percent of the gold.

Tests at a coarser primary grind sizing are recommended to further establish the effect of primary grind.

A whole ore rougher flotation test followed by gravity concentration of the tailings (Test 3), as well as a gravity concentration test followed by rougher flotation (Test 4) was conducted on the MC sample, ground to 75μm K_{80}. For these tests, total gold recovered to the gravity and rougher flotation concentrates measured 92.5 and 89.5 for Tests 3 and 4, respectively.

A single stage bulk gravity test, Test 12, was conducted on the MC sample ground to 150μm K_{80} in order to generate product for downstream evaluation. The gravity stage recovered about 66 percent of the gold in the feed. The gravity tail from this test was subjected to additional testing including gravity, flotation, and cyanidation testing.

Cyanidation tests on the bulk gravity tailings evaluated pulp density, air versus oxygen addition, cyanide concentration, and Carbon in Leach (CIL) conditions. Results indicate overall gold extractions of 95.6 to 98.7 percent were obtainable. Air sparging as opposed to oxygen measured the lowest extraction, at 95.6 total gold extraction. Cyanide concentration was reduced from 1000 to 250ppm and maintained gold extractions of around 98 percent. Sodium cyanide and lime consumption at the low cyanide concentration test (Test 19 - 250ppm), measured 0.04 and 0.60 kg/tonne, respectively, which are considered to be low. Carbon in Leach and various pulp densities also had limited effect on gold extraction, with extractions measured between 97.4 and 98.8 percent.

A single rougher test was also conducted on the bulk gravity tailings, resulting in overall gold recovery of 88.5 percent.

A 2-stage sequential gravity concentration test on the bulk gravity tailings was also conducted, resulting in overall gold recovery of 77.3 percent.

Lastly, hydrofloat processing of the bulk gravity tailings was evaluated, resulting in overall gold recovery of 95 percent.

5.0 Summary and Recommendations

The objective of this program was to evaluate flowsheet options for the processing of the BAM East Gold ore. Samples for this test program were received at Base Metallurgical Laboratories Ltd. on July 4, 2018. The shipment contained about 854 kilograms of sample, received as whole drill cores. A single composite was generated for the testing program.

Comminution testing resulted in a Bond ball work index of 15.9 kWh/tonne, and a Bond rod mill work index of 16.2 kWh/tonne. These values indicate the mineralization to be moderately hard with respect to ball and rod mill grinding. The Abrasion index for this sample was determined to be 0.14, classifying the sample as mildly abrasive. The SMC derived A x b value was determined to be 52.2 for this sample, indicating the sample is moderately hard with respect to breakage in a SAG Mill.

Due to the assumed coarse nature of the gold in the composite, eight separate sub-samples were taken and assayed for gold. A separate sub-sample was utilized for screen metallics. The 8 sub-samples average 0.95 g/tonne gold, whilst the screen metallics returned a value of 0.84 g/tonne gold. There is very little silver and copper in the composite, about 1 and 7 g/tonne, respectively.

A coarse crush (100% passing 31.5mm) leach column conducted in locked cycle extracted about 57 percent of the gold in the feed after 91 days. A fine grind (100% passing 6.3mm) leach column was also conducted in locked cycle and resulted in much higher gold extractions at 89 percent gold extraction, with the test lasting 132 days.

Free milling flowsheet evaluations indicated that a gravity-leach flowsheet could extract up to about 98 percent of the gold in the feed, with sodium cyanide and lime consumptions of 0.04 and 0.60 kg/tonne being measured for the low cyanide concentration test (Test 19 – 250ppm), which are considered to be low. An extended gravity recoverable gold test indicated about 82 percent of the gold in the feed is recoverable to gravity concentrates, whilst a gravity-flotation flowsheet could recover about 89 percent of the gold in the feed. A single hydrofloat test was evaluated on gravity tailings, this test recovered about 95 percent of the gold including gravity and hydro float processing. Additional processing would be required on the fines portion of hydrofloat testing.

Due to the nature of the gold, further testing should include variability testing across the deposit. If a gravity-leach flowsheet is selected additional testing should also include; carbon adsorption, oxygen uptake testing, rheology/viscosity, settling/filtration and cyanide detoxification.

APPENDIX A – CHAIN OF CUSTODY

APPENDIX A
CHAIN OF CUSTODY

Samples were received at Base Metallurgical Laboratories in a single shipment on July 4, 2018. Table A-1 provides the sample identification as provided by Landore Resources Ltd and mass information for the samples received. Pictures of the samples as received are provided as Photos A-1. Samples were received well labelled, with rice sacks of each drill core intersection contained in buckets.

All sample was combined into one Master Composite. The composite was coarsely crushed to -31.5mm. A portion of the sample was then split out for comminution testing, which was further crushed to the required size for each comminution test. Another 205 kg split of <31.5mm Master Composite material was used for the coarse column test charge, CL01. A 30 kg cut was also used for a feed – size by assay. A portion of about 250kg was also set aside for HPGR testing. This remains in storage.

The remaining sample of Master Composite, 350 kg, was stage crushed to <6.3mm. A second column charge of 205kg was used for the fine column, CL02. The remaining material was further crushed and split for other metallurgical testing, including gravity, bottle roll cyanidation, and flotation testing.

<u>TABLE A-1A</u>
<u>SAMPLE RECEIVED - MET 1</u>

Hole ID	MET	From	To	Sample ID	Bag	Received Weight (kg)
0418-653	MET 1	138	139	W868716	Bag 1	
0418-653	MET 1	139	140	W868717	Bag 2	15.52
0418-653	MET 1	140	141	W868718	Bag 3	
0418-653	MET 1	141	142	W868719	Bag 4	16.10
0418-653	MET 1	142	143	W868720	Bag 5	
0418-653	MET 1	143	144	W868721	Bag 6	16.24
0418-653	MET 1	144	145	W868722	Bag 7	
0418-653	MET 1	145	146	W868723	Bag 8	16.20
0418-653	MET 1	146	147	W868724	Bag 9	
0418-653	MET 1	147	148	W868725	Bag 10	15.78
0418-653	MET 1	148	149	W868726	Bag 11	
0418-653	MET 1	149	150	W868727	Bag 12	16.14
0418-653	MET 1	150	151	W868728	Bag 13	
0418-653	MET 1	151	152	W868729	Bag 14	15.78
0418-653	MET 1	152	153	W868730	Bag 15	
0418-653	MET 1	153	154	W868731	Bag 16	15.68
0418-653	MET 1	154	155	W868732	Bag 17	
0418-653	MET 1	155	156	W868733	Bag 18	16.46
0418-653	MET 1	156	157	W868734	Bag 19	
0418-653	MET 1	157	158	W868735	Bag 20	16.14
0418-653	MET 1	158	159	W868736	Bag 21	
0418-653	MET 1	159	160	W868737	Bag 22	15.94
0418-653	MET 1	160	161	W868738	Bag 23	
0418-653	MET 1	161	162	W868739	Bag 24	15.90
0418-653	MET 1	162	163	W868740	Bag 25	
0418-653	MET 1	163	164	W868741	Bag 26	15.36
0418-653	MET 1	164	165	W868742	Bag 27	
0418-653	MET 1	165	166	W868743	Bag 28	15.50
0418-653	MET 1	166	167	W868744	Bag 29	
0418-653	MET 1	167	168	W868745	Bag 30	16.38
0418-653	MET 1	168	169	W868746	Bag 31	
0418-653	MET 1	169	170	W868747	Bag 32	16.02
0418-653	MET 1	170	171	W868748	Bag 33	
0418-653	MET 1	171	172	W868749	Bag 34	16.32
0418-653	MET 1	172	173	W868750	Bag 35	
0418-653	MET 1	173	174	W868751	Bag 36	15.74
0418-653	MET 1	174	175	W868752	Bag 37	
0418-653	MET 1	175	176	W868753	Bag 38	16.60
0418-653	MET 1	176	177	W868754	Bag 39	
0418-653	MET 1	177	178	W868755	Bag 40	16.30
0418-653	MET 1	178	179	W868756	Bag 41	
0418-653	MET 1	179	180	W868757	Bag 42	16.78

TABLE A-1A Continued
SAMPLE RECEIVED - MET 1

Hole ID	MET	From	To	Sample ID	Bag	Received Weight (kg)
0418-653	MET 1	180	181	W868758	Bag 43	
0418-653	MET 1	181	182	W868759	Bag 44	16.46
0418-653	MET 1	182	183	W868760	Bag 45	
0418-653	MET 1	183	184	W868761	Bag 46	16.10
0418-653	MET 1	184	185	W868762	Bag 47	
0418-653	MET 1	185	186	W868763	Bag 48	16.30
0418-653	MET 1	186	187	W868764	Bag 49	
0418-653	MET 1	187	188	W868765	Bag 50	15.88
0418-653	MET 1	188	189	W868766	Bag 51	
0418-653	MET 1	189	190	W868767	Bag 52	15.54
0418-653	MET 1	190	191	W868768	Bag 53	
0418-653	MET 1	191	192	W868769	Bag 54	16.16
0418-653	MET 1	192	193	W868770	Bag 55	
0418-653	MET 1	193	194	W868771	Bag 56	16.02
0418-653	MET 1	194	195	W868772	Bag 57	
0418-653	MET 1	195	196	W868773	Bag 58	16.04
Total						465.38

TABLE A-1B

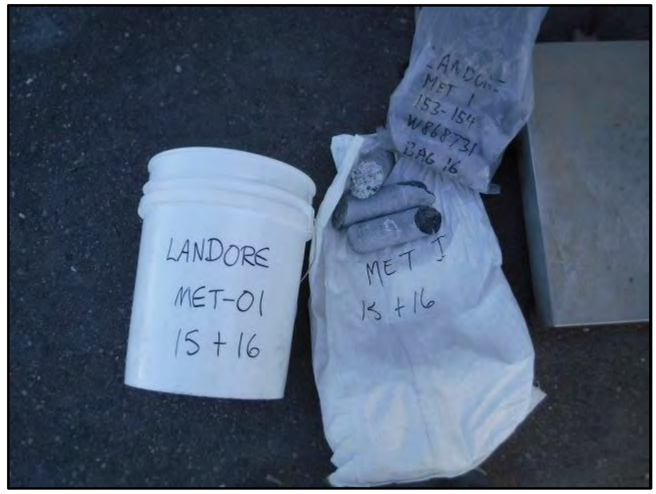
SAMPLE RECEIVED - MET 2

Hole ID	MET	From	To	Sample ID	Bag	Received Weight (kg)
0418-654	MET 2	124	125	W868863	Bag 1	
0418-654	MET 2	125	126	W868864	Bag 2	16.36
0418-654	MET 2	126	127	W868865	Bag 3	
0418-654	MET 2	127	128	W868866	Bag 4	16.08
0418-654	MET 2	128	129	W868867	Bag 5	
0418-654	MET 2	129	130	W868868	Bag 6	15.88
0418-654	MET 2	130	131	W868869	Bag 7	
0418-654	MET 2	131	132	W868870	Bag 8	15.82
0418-654	MET 2	132	133	W868871	Bag 9	
0418-654	MET 2	133	134	W868872	Bag 10	16.22
0418-654	MET 2	134	135	W868873	Bag 11	
0418-654	MET 2	135	136	W868874	Bag 12	16.14
0418-654	MET 2	136	137	W868875	Bag 13	
0418-654	MET 2	137	138	W868876	Bag 14	16.16
0418-654	MET 2	138	139	W868877	Bag 15	
0418-654	MET 2	139	140	W868878	Bag 16	15.96
0418-654	MET 2	140	141	W868879	Bag 17	
0418-654	MET 2	141	142	W868880	Bag 18	16.32
0418-654	MET 2	142	143	W868881	Bag 19	
0418-654	MET 2	143	144	W868882	Bag 20	16.24
0418-654	MET 2	144	145	W868883	Bag 21	
0418-654	MET 2	145	146	W868884	Bag 22	16.46
0418-654	MET 2	146	147	W868885	Bag 23	
0418-654	MET 2	147	148	W868886	Bag 24	16
0418-654	MET 2	148	149	W868887	Bag 25	
0418-654	MET 2	149	150	W868888	Bag 26	15.94
0418-654	MET 2	150	151	W868889	Bag 27	
0418-654	MET 2	151	152	W868890	Bag 28	16.02
0418-654	MET 2	152	153	W868891	Bag 29	
0418-654	MET 2	153	154	W868892	Bag 30	16.24
0418-654	MET 2	154	155	W868893	Bag 31	
0418-654	MET 2	155	156	W868894	Bag 32	16.28
0418-654	MET 2	156	157	W868895	Bag 33	
0418-654	MET 2	157	158	W868896	Bag 34	16.16
0418-654	MET 2	158	159	W868897	Bag 35	
0418-654	MET 2	159	160	W868898	Bag 36	16.54
0418-654	MET 2	160	161	W868899	Bag 37	
0418-654	MET 2	161	162	W868900	Bag 38	16.46
0418-654	MET 2	162	163	W868901	Bag 39	
0418-654	MET 2	163	164	W868902	Bag 40	16.5
0418-654	MET 2	164	165	W868903	Bag 41	
0418-654	MET 2	165	166	W868904	Bag 42	16.74

TABLE A-1B Continued
SAMPLE RECEIVED - MET 2

Hole ID	MET	From	To	Sample ID	Bag	Received Weight (kg)
0418-654	MET 2	166	167	W868905	Bag 43	
0418-654	MET 2	167	168	W868906	Bag 44	15.74
0418-654	MET 2	168	169	W868907	Bag 45	
0418-654	MET 2	169	170	W868908	Bag 46	15.62
0418-654	MET 2	170	171	W868909	Bag 47	
0418-654	MET 2	171	172	W868910	Bag 48	16.24
Total						388.12

PHOTOS A-1
SAMPLES RECEIVED



APPENDIX B – METALLURGICAL TESTING

APPENDIX B
METALLURGICAL TESTING

Test No:	BL295-01		Au, gm/t	Ag, gm/t	Cu, %
Date:	16-Jul-18	Head Assay, Ave:	0.950	1.00	0.01%
Test Type:	Standard bottle roll procedure.	Head Assay 1:	0.95	1	0.007%
Test Objective:	Evaluate gold extraction.	Head Assay 2:	0.95	1	0.007%
Sample:	Master Comp	Head Assay 3:	0.95	1	0.007%
Grind:	Pulverized				
Target Maintenance NaCN:	1000 ppm	Total Mass, Initial:	3238.4	grams including bottle	

Cyanidation Leaching @ pH 10.5 (lime), 1000ppm NaCN, O2 sparged,

	Au, gm/t	Ag, gm/t	Cu, %
Tails Assay, Ave:	0.035	0.5	0.007%
Tail Assay 1:	0.040	0.5	0.007%
Tail Assay 2:	0.030	0.5	0.007%

Sample Size 20 ml

Calculated Head

	Au	Ag	Cu
Head, mg:	0.950	1.000	70
Tails, mg:	0.035	0.500	67
Calculated Head, mg:	0.921	0.664	83

Parameter	Time	Added (g)		WAD CN	pH		Dissolved	Redox	Temp
	Cum	NaCN	Lime	mg/l CN	Measured	Adjusted	O_2 (mg/L)	mV	Deg. C
Natural	-	-	-		8.6	-	7.1		
Leach 0	0	2.00	0.48		10.5	10.5	7.1	-590	19.2
Leach 1	1	0.10	-	548	10.7	-	>20	-587	24.8
Leach 2	2	0.06	-	540	10.7	-	>20	-596	25.0
Leach 3	4	0.00	-	462	10.6	-	>20	-608	25.5
Leach 4	8	0.18	-	493	10.6	-	>20	-593	25.7
Leach 5	24	0.00	0.15	502	10.3	-	>20	-644	24.1
Leach 6	48	0.08		531	10.9	-	>20	-588	24.4
Leach 7	72	0.18	-	496	10.8	-	>20	-580	19.2
Leach 8	96	-	-	515	10.5	-	>20	-578	19.5
Total	96	2.60	0.63		-	-	-	-	
CaO Titration			0.12						

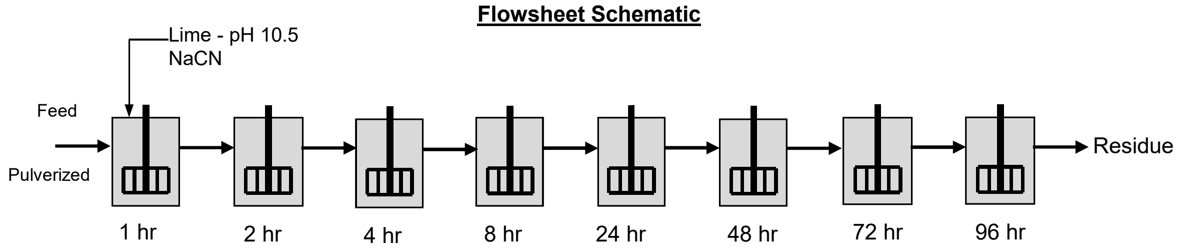
Mass of Sample	1000
Volume of Water	2000
Pulp Density	33

NaCN Consumption	0.71	kg/tonne
Lime Consumption	0.63	kg/tonne

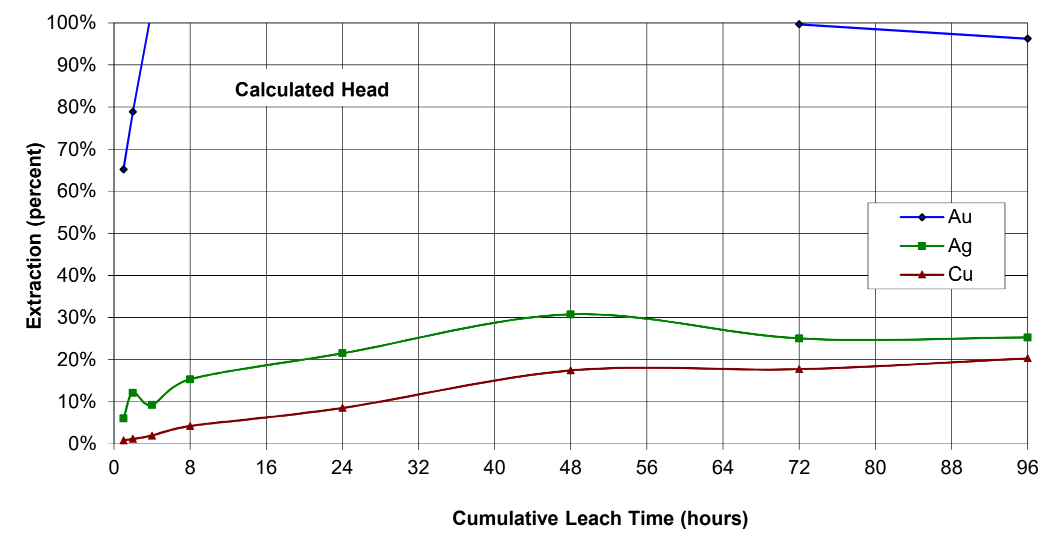
Cumulative Metallurgical Balance

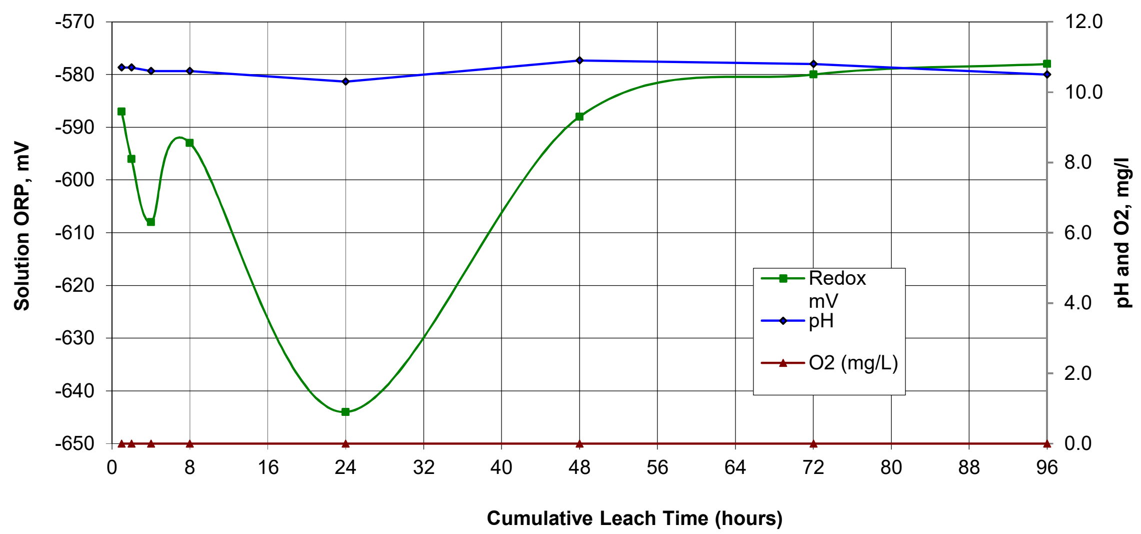
Product	Cumulative	Vol or	Units	Free CN	Free NaCN	mol WAD/	Assay - g/tonne (ppm)			Distribution - percent		
	Time - Hrs	Mass		mg/l, calc'd	mg/l	mol Cu	Au	Ag	Cu	Au	Ag	Cu
Cyanide Liquor (1 hr)	1	2000	mL	547.3	1,031	3935.88	0.30	0.02	0.34	65.17%	6.03%	0.82%
Cyanide Liquor (2 hr)	2	2000	mL	539.6	1,016	2694.00	0.36	0.04	0.49	78.85%	12.11%	1.19%
Cyanide Liquor (4 hr)	4	2000	mL	460.3	867	1392.26	0.47	0.03	0.81	103.53%	9.22%	1.98%
Cyanide Liquor (8 hr)	8	2000	mL	490.2	923	692.19	0.49	0.05	1.74	108.89%	15.34%	4.24%
Cyanide Liquor (24hr)	24	2000	mL	496.6	935	350.53	0.57	0.07	3.50	127.34%	21.51%	8.54%
Cyanide Liquor (48hr)	48	2000	mL	519.4	978	181.37	0.68	0.10	7.15	152.47%	30.76%	17.44%
Cyanide Liquor (72hr)	72	2000	mL	484.6	913	168.28	0.43	0.08	7.20	99.64%	25.04%	17.74%
Cyanide Liquor (96hr)	96	2000	mL	501.6	945	153.47	0.41	0.08	8.19	96.23%	25.28%	20.30%
Cyanidation Tails	-	992	g				0.035	0.50	67	3.77%	74.72%	79.70%
Calculated Feed, gm/t		992	g				0.928	0.67	83	100.00%	100.00%	100.00%
Head Assay, gm/t		1000					0.950	1.00	70			

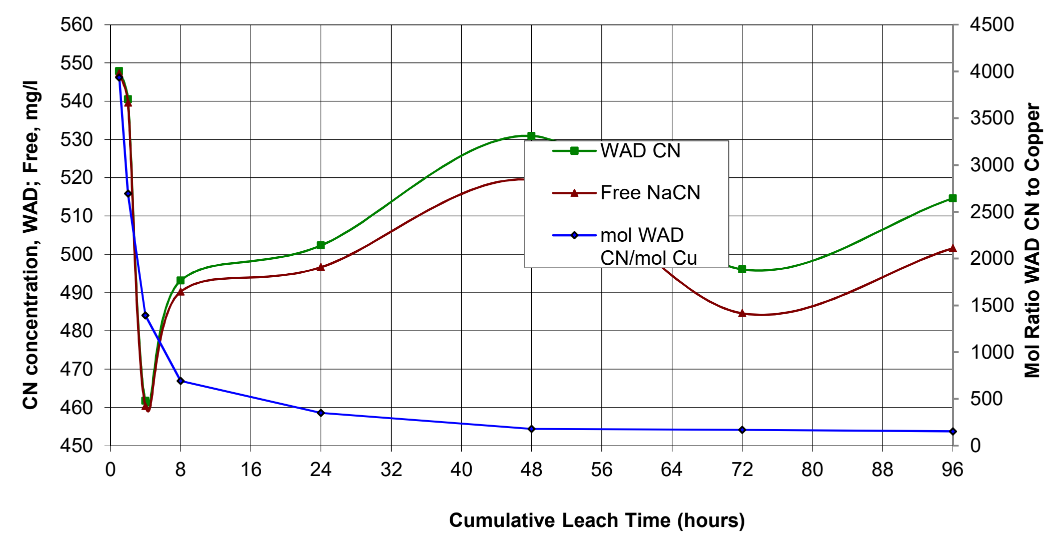
Flowsheet Schematic



Cyanide Leach Kinetic Curves







Test No:	BL295-02		Au, gm/t	Ag, gm/t	Cu, %	
Date:	16-Jul-18	Head Assay, Ave:	0.950	1.00	0.01%	
Test Type:	Standard bottle roll procedure.	Head Assay 1:	0.95	1	0.01%	
Test Objective:	Repeat Test 01 at higher NaCN.	Head Assay 2:	0.95	1	0.01%	
Sample:	Master Comp	Head Assay 3:	0.95	1	0.01%	
Grind:	Pulverized					
Target Maintenance NaCN:	5000 ppm	Total Mass, Initial:	3242.7	grams including bottle		

	Au, gm/t	Ag, gm/t	Cu, %	
Tails Assay, Ave:	0.080	0.5	0.01%	
Tail Assay 1:	0.080	0.5	0.01%	
Tail Assay 2:	0.080	0.5	0.01%	
Sample Size	20	ml		
Calculated Head				

	Au	Ag	Cu
Head, mg:	0.950	1.000	70
Tails, mg:	0.080	0.500	75
Calculated Head, mg:	1.472	0.748	96

Cyanidation Leaching @ pH 10.5 (lime), 5000ppm NaCN, O2 sparged,

Parameter	Time	Added (g)		WAD CN	pH		Dissolved	Redox	Temp
	Cum	NaCN	Lime	mg/l CN	Measured	Adjusted	O_2 (mg/L)	mV	Deg. C
Natural	-	-	-		8.5	-	7.2		
Leach 0	0	10.00	0.29		10.5	10.5	7.2	-701	19.2
Leach 1	1	0.84	-	2,581	11.1	-	>20	-705	25.0
Leach 2	2	1.00	-	2,416	11.2	-	>20	-712	25.2
Leach 3	4	0.00	-	2,720	11.0	-	>20	-708	25.6
Leach 4	8	0.00	-	2,902	11.0	-	>20	-706	26.0
Leach 5	24	0.16	-	3,115	10.9	-	>20	-722	24.2
Leach 6	48	0.56	-	2,891	11.1	-	>20	-702	23.8
Leach 7	72	0.44	-	3,174	11.3	-	>20	-704	19.3
Leach 8	96	0.00	-	2,709	10.9	-	>20	-703	19.5
Total	96	13.00	0.29		-	-	-	-	
CaO Titration			0.28						

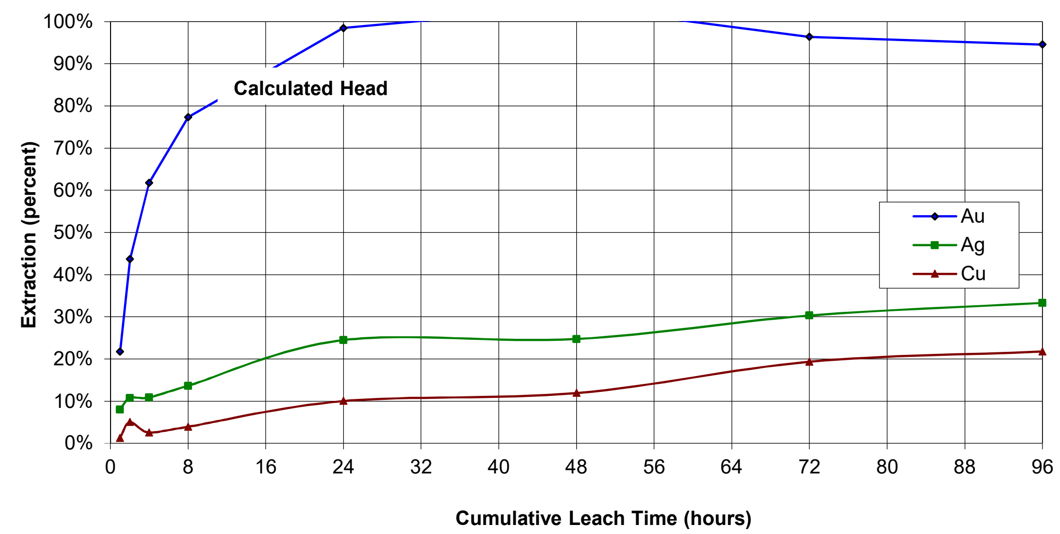
Mass of Sample	1000
Volume of Water	2000
Pulp Density	33

NaCN Consumption	2.86	kg/tonne
Lime Consumption	0.29	kg/tonne

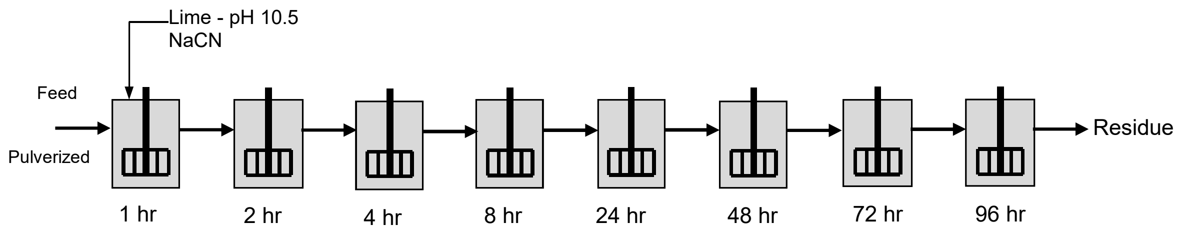
Cumulative Metallurgical Balance

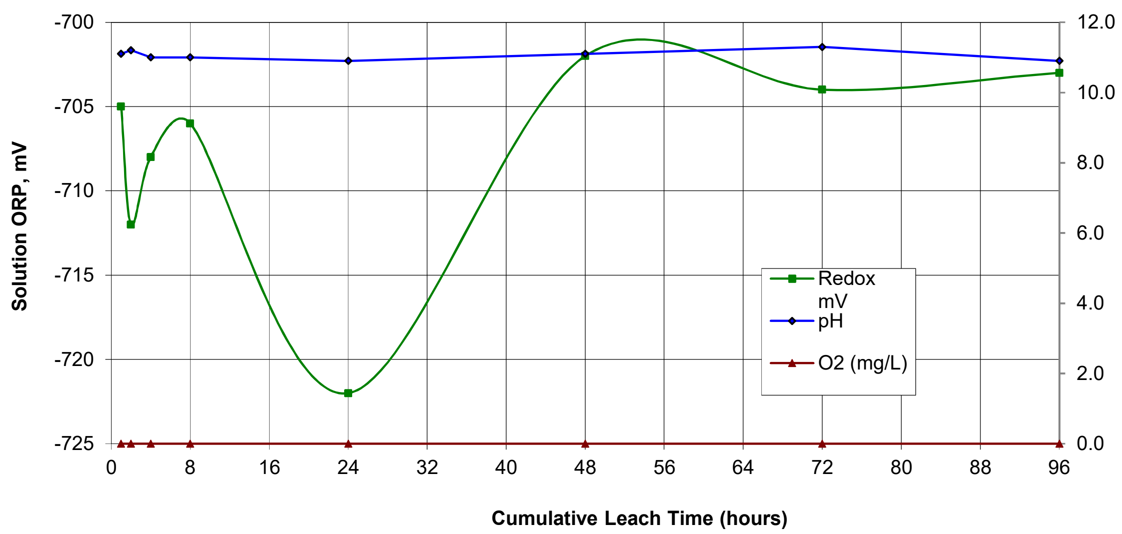
Product	Cumulative	Vol or	Units	Free CN	Free NaCN	mol WAD/	Assay - g/tonne (ppm)			Distribution - percent		
	Time - Hrs	Mass		mg/l, calc'd	mg/l	mol Cu	Au	Ag	Cu	Au	Ag	Cu
Cyanide Liquor (1 hr)	1	2000	mL	2579.9	4,860	10167.38	0.16	0.03	0.62	21.74%	8.02%	1.30%
Cyanide Liquor (2 hr)	2	2000	mL	2412.2	4,544	2458.88	0.32	0.04	2.40	43.69%	10.78%	5.03%
Cyanide Liquor (4 hr)	4	2000	mL	2717.6	5,119	5535.57	0.45	0.04	1.20	61.79%	10.89%	2.57%
Cyanide Liquor (8 hr)	8	2000	mL	2898.7	5,460	3831.08	0.56	0.05	1.85	77.34%	13.67%	3.96%
Cyanide Liquor (24hr)	24	2000	mL	3107.2	5,853	1605.19	0.71	0.09	4.74	98.48%	24.50%	10.04%
Cyanide Liquor (48hr)	48	2000	mL	2882.1	5,429	1261.08	0.74	0.09	5.60	103.52%	24.74%	11.94%
Cyanide Liquor (72hr)	72	2000	mL	3159.1	5,951	852.87	0.68	0.11	9.09	96.37%	30.33%	19.35%
Cyanide Liquor (96hr)	96	2000	mL	2692.6	5,072	651.30	0.66	0.12	10.16	94.58%	33.30%	21.78%
Cyanidation Tails	-	997	g				0.080	0.50	75	5.42%	66.70%	78.22%
Calculated Feed, gm/t		997	g				1.476	0.75	96	100.00%	100.00%	100.00%
Head Assay, gm/t		1000					0.950	1.00	70			

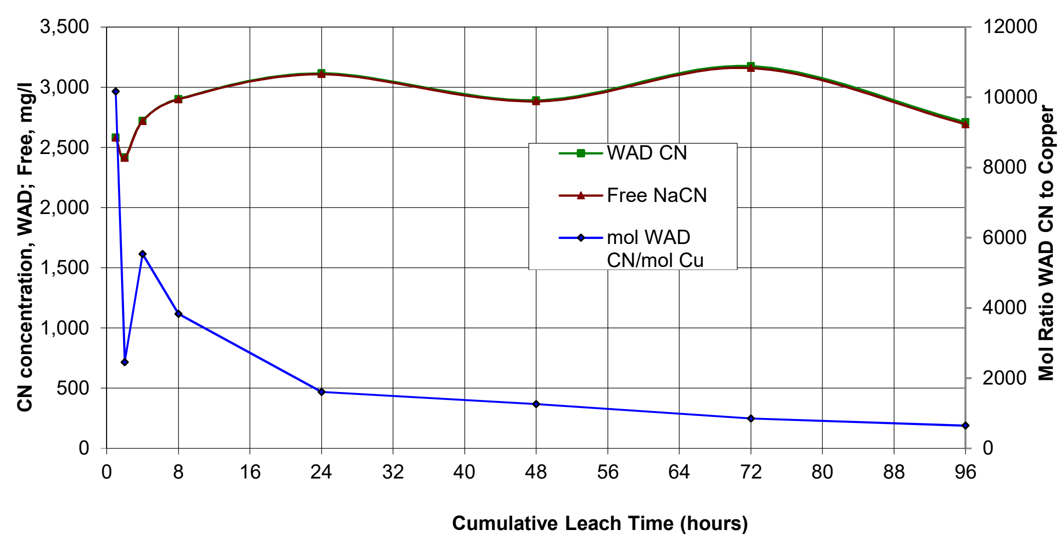
Cyanide Leach Kinetic Curves



Flowsheet Schematic







Test No:	BL0295-03
Date:	31-Jul-18
Test Type:	Rougher Test.
Test Objective:	Preliminary Rougher Test.
Sample:	2 kg of Master Composite Milling Split
Nominal Sizing:	75µm K_{80}

Stage	Reagents - g/tonne			Time	Electrochemistry	
	PAX		MIBC	Minutes	pH	Eh-mV
Primary Grind				20	8.5	200
Rougher 1	20		8	2	8.5	130
Rougher 2	20		8	2	8.6	120
Rougher 3	20		15	2	8.5	80
Rougher 4	20		15	2	8.5	20

Primary Grind	
Mill	Mild Steel Mill
Media	20kg Mild Steel Rods
Water Addn:	1500ml

Flotation Information	Rougher
Flotation Device:	D12
Cell Volume - Litres:	4.4
Impellar Speed - rpm:	800
Flotation Gas:	Air
Water Type:	Kamloops Tap


BL295-03 Rougher Con 1


BL295-03 Rougher Con 2


BL295-03 Rougher Con 3


BL295-03 Rougher Con 4

BL0295-03 Master Composite Milling Split

Metallurgical Balance

Product	Weight		Assay - percent or g/t		Distribution - percent	
	%	grams	Au	Ag	Au	Ag
Ro Con 1	2.9	58.2	19.7	3.0	69.2	14.9
Ro Con 2	2.7	54.1	4.16	0.5	13.6	2.3
Ro Con 3	2.3	44.9	0.35	0.5	1.0	1.9
Ro Con 4	1.7	33.5	0.17	0.5	0.3	1.4
Ro Tl Pan Con	0.3	5.3	14.5	2.0	4.6	0.9
Ro Tl Pan Tl	2.4	48.2	1.28	1.0	3.7	4.1
Rougher Tail	87.8	1750.5	0.07	0.5	7.5	74.5
Recalc. Feed	100.0	1994.7	0.83	0.6	100	100
Measured Feed			0.95	1.0		

BL0295-03 Master Composite Milling Split

Cumulative Balance

Product	Weight		Assay - percent or g/t		Distribution - percent	
	%	grams	Au	Ag	Au	Ag
Product 1	2.9	58.2	19.7	3.0	69.2	14.9
Products 1 to 2	5.6	112.3	12.2	1.8	82.8	17.2
Products 1 to 3	7.9	157.2	8.82	1.4	83.8	19.1
Products 1 to 4	9.6	190.7	7.30	1.3	84.1	20.5
Products 1 to 5	9.8	196.0	7.49	1.3	88.8	21.4
Products 1 to 6	12.2	244.2	6.26	1.2	92.5	25.5
Product 6	87.8	1750.5	0.07	0.5	7.5	74.5
Recalc. Feed	100.0	1994.7	0.83	0.6	100	100

Test No:	BL0295-04
Date:	31-Jul-18
Test Type:	Rougher Test.
Test Objective:	Investigate Gravity on Front End.
Sample:	2 kg of Master Composite Milling Split
Nominal Sizing:	75µm K_{80}

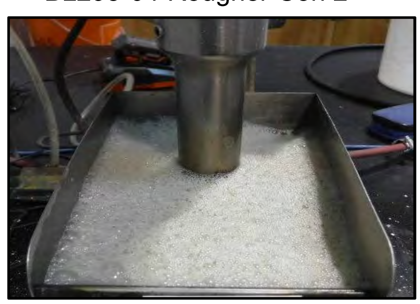
Stage	Reagents - g/tonne			Time Minutes	Electrochemistry	
	PAX		MIBC		pH	Eh-mV
Primary Grind				28		
Knelson						
Panning						
Rougher 1	20		15	2	8.5	226
Rougher 2	20		8	2	8.4	45
Rougher 3	20			2	8.4	22
Rougher 4	20			2	8.5	20

Primary Grind		Flotation Information	Rougher
Mill	Mild Steel Mill	Flotation Device:	D12
Media	20kg Mild Steel Rods	Cell Volume - Litres:	4.4
Water Addn:	1500ml	Impellar Speed - rpm:	800
		Flotation Gas:	Air
		Water Type:	Kamloops Tap


BL295-04 Rougher Con 1


BL295-04 Rougher Con 2


BL295-04 Rougher Con 3


BL295-04 Rougher Con 4

BL0295-04 Master Composite Milling Split

Metallurgical Balance

Product	Weight		Assay - percent or g/t		Distribution - percent	
	%	grams	Au	Ag	Au	Ag
Pan Con	0.2	4.7	206	14.0	79.6	5.3
Ro Con 1	2.9	58.9	1.11	3.0	5.4	14.3
Ro Con 2	2.2	43.5	0.85	1.0	3.0	3.5
Ro Con 3	1.9	37.1	0.32	0.5	1.0	1.5
Ro Con 4	1.4	28.8	0.22	0.5	0.5	1.2
Rougher Tail	91.3	1825.3	0.07	0.5	10.5	74.1
Recalc. Feed	100.0	1998.3	0.61	0.6	100	100
Measured Feed			0.95	1.0		

BL0295-04 Master Composite Milling Split

Cumulative Balance

Product	Weight		Assay - percent or g/t		Distribution - percent	
	%	grams	Au	Ag	Au	Ag
Product 1	0.2	4.7	206	14.0	79.6	5.3
Products 1 to 2	3.2	63.6	16.24	3.8	84.9	19.7
Products 1 to 3	5.4	107.1	9.99	2.7	88.0	23.2
Products 1 to 4	7.2	144.2	7.50	2.1	88.9	24.7
Products 1 to 5	8.7	173.0	6.29	1.8	89.5	25.9
Product 6	91.3	1825.3	0.07	0.5	10.5	74.1
Recalc. Feed	100.0	1998.3	0.61	0.6	100	100

Test No:	BL295-05
Date:	01-Aug-18
Test Type:	Standard bottle roll procedure.
Test Objective:	Repeat Test 01 at 75um.
Sample:	Master Comp
Grind:	75µm
Target Maintenance NaCN:	1000 ppm

	Au, gm/t	Ag, gm/t	Cu, %
Head Assay, Ave:	0.950	1.00	0.01%
Head Assay 1:	0.95	1	0.01%
Head Assay 2:	0.95	1	0.01%
Head Assay 3:	0.95	1	0.01%

Total Mass, Initial: 6240.3 grams including bottle

	Au, gm/t	Ag, gm/t	Cu, %
Tails Assay, Ave:	0.015	0.5	0.005%
Tail Assay 1:	0.010	0.5	0.005%
Tail Assay 2:	0.020	0.5	0.005%

Sample Size 20 ml
Calculated Head

	Au	Ag	Cu
Head, mg:	1.900	2.000	140
Tails, mg:	0.030	1.000	99
Calculated Head, mg:	2.570	1.409	127

Cyanidation Leaching @ pH 10.5 (lime), 1000ppm NaCN, O2 sparged,

Parameter	Time	Added (g)		WAD CN	pH		Dissolved	Redox	Temp
	Cum	NaCN	Lime	mg/l CN	Measured	Adjusted	O_2 (mg/L)	mV	Deg. C
Natural	-	-	-		8.7	-	2.8		
Leach 0	0	4.00	0.99		10.5	10.5	2.8		
Leach 1	1	0.20	-	534	11.1	-	11.3	-637	23.9
Leach 2	2	0.20	-	506	11.2	-	18.2	-629	24.2
Leach 3	6	-	-	536	11.1	-	>20	-609	24.2
Leach 4	8	-	-	548	11.0	-	>20	-613	24.6
Leach 5	24	0.24	-	478	11.0	-	>20	-602	23.5
Leach 6	48	0.12	-	498	11.1	-	>20	-624	22.5
Leach 7	72	-	-	524	11.0	-			
Leach 8	96	-	-	520	11.0	-	>20	-625	24.1
Total	96	4.76	0.99		-	-	-	-	
CaO Titration									

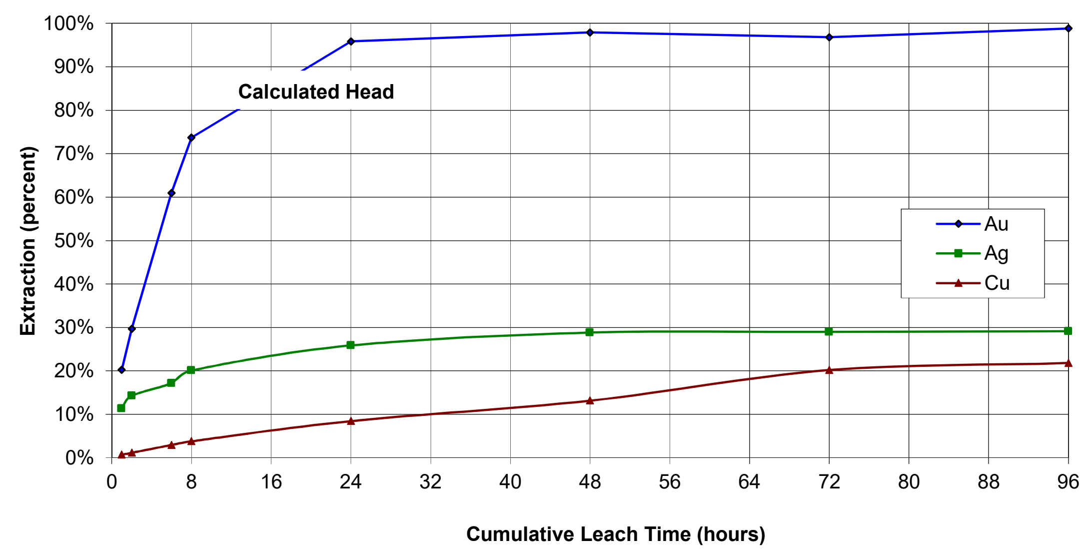
Mass of Sample	2000
Volume of Water	4000
Pulp Density	33

NaCN Consumption	0.46	kg/tonne
Lime Consumption	0.50	kg/tonne

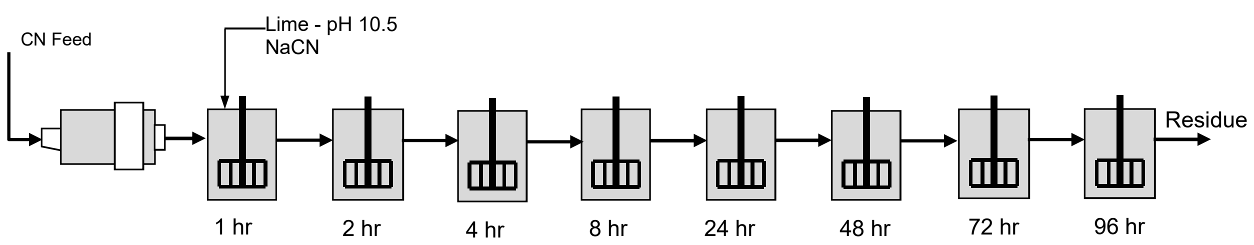
Cumulative Metallurgical Balance

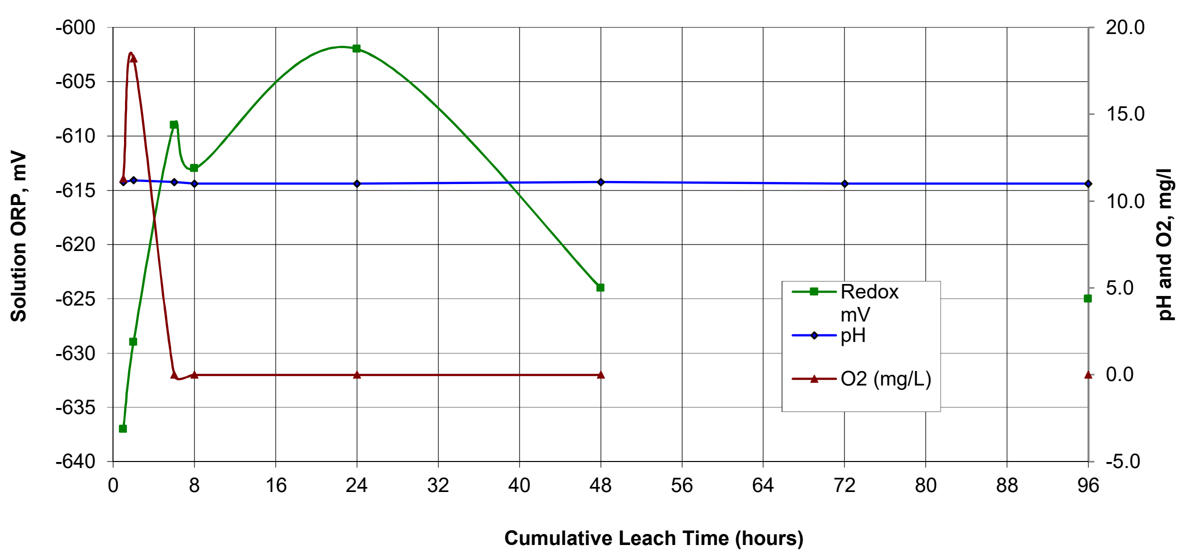
Product	Cumulative	Vol or	Units	Free CN	Free NaCN	mol WAD/	Assay - g/tonne (ppm)			Distribution - percent		
	Time - Hrs	Mass		mg/l, calc'd	mg/l	mol Cu	Au	Ag	Cu	Au	Ag	Cu
Cyanide Liquor (1 hr)	1	4000	mL	534.0	1,006	5932.89	0.13	0.04	0.22	20.23%	11.35%	0.69%
Cyanide Liquor (2 hr)	2	4000	mL	505.3	952	3432.66	0.19	0.05	0.36	29.67%	14.25%	1.14%
Cyanide Liquor (4 hr)	6	4000	mL	534.9	1,008	1408.94	0.39	0.06	0.93	60.94%	17.16%	2.94%
Cyanide Liquor (8 hr)	8	4000	mL	546.2	1,029	1125.26	0.47	0.07	1.19	73.69%	20.08%	3.78%
Cyanide Liquor (24hr)	24	4000	mL	474.0	893	439.25	0.61	0.09	2.66	95.85%	25.86%	8.44%
Cyanide Liquor (48hr)	48	4000	mL	491.3	925	293.80	0.62	0.10	4.14	97.88%	28.82%	13.15%
Cyanide Liquor (72hr)	72	4000	mL	513.8	968	201.55	0.61	0.10	6.35	96.80%	28.97%	20.19%
Cyanide Liquor (96hr)	96	4000	mL	509.0	959	185.95	0.62	0.10	6.83	98.83%	29.11%	21.81%
Cyanidation Tails	-	1998	g				0.015	0.50	50	1.17%	70.89%	78.19%
Calculated Feed, gm/t		1998	g				1.286	0.71	63	100.00%	100.00%	100.00%
Head Assay, gm/t		2000					0.950	1.00	70			

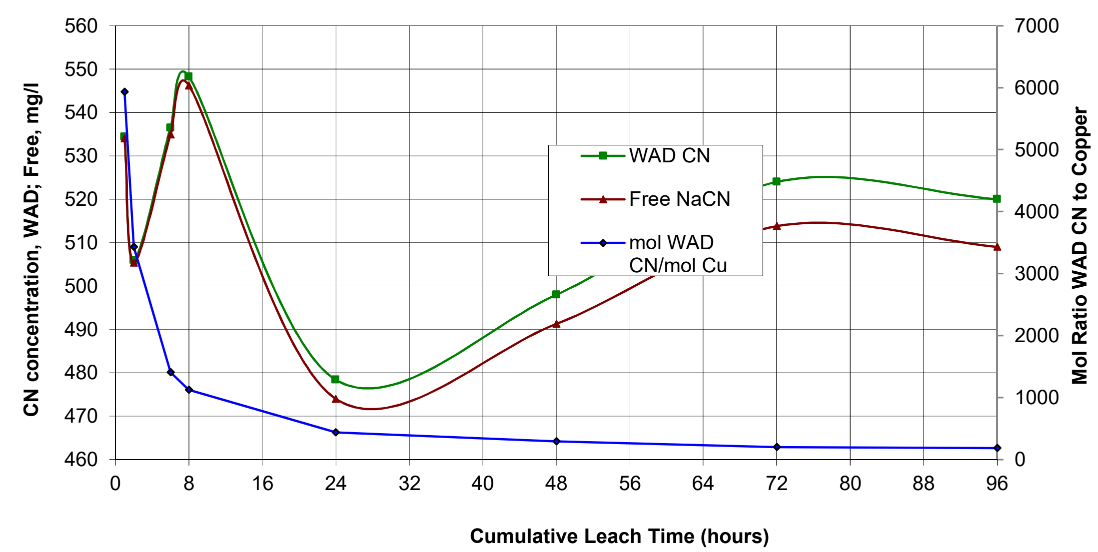
Cyanide Leach Kinetic Curves



Flowsheet Schematic







Test No:	BL295-06		Au, gm/t	Ag, gm/t	Cu, %	
Date:	01-Aug-18	Head Assay, Ave:	0.950	1.00	0.01%	
Test Type:	Standard bottle roll procedure.	Head Assay 1:	0.95	1	0.01%	
Test Objective:	Repeat Test 05 (duplicate).	Head Assay 2:	0.95	1	0.01%	
Sample:	Master Comp	Head Assay 3:	0.95	1	0.01%	
Grind:	75µm					
Target Maintenance NaCN:	1000 ppm	Total Mass, Initial:	6242.2	grams including bottle		

	Au, gm/t	Ag, gm/t	Cu, %
Tails Assay, Ave:	0.008	0.5	0.00%
Tail Assay 1:	0.010	0.5	0.005%
Tail Assay 2:	0.005	0.5	0.005%
Sample Size	20 ml		
Calculated Head			

	Au	Ag	Cu
Head, mg:	1.900	2.000	140
Tails, mg:	0.015	1.000	98
Calculated Head, mg:	2.273	1.486	129

Cyanidation Leaching @ pH 10.5 (lime), 1000ppm NaCN, O2 sparged,

Parameter	Time	Added (g)		WAD CN	pH		Dissolved	Redox	Temp
	Cum	NaCN	Lime	mg/l CN	Measured	Adjusted	O_2 (mg/L)	mV	Deg. C
Natural	-	-	-		8.6	-			
Leach 0	0	4.00	0.99		10.5	10.5	4.8		
Leach 1	1	0.44	-	467	11.1	-	17.1	-622	23.8
Leach 2	2	-	-	532	11.3	-	16.2	-635	24.2
Leach 3	6	-	-	533	11.1	-	>20	-612	24.6
Leach 4	8	-	-	554	11.1	-	>20	-604	24.7
Leach 5	24	0.20	-	545	11.1	-	>20	-594	23.5
Leach 6	48	0.16	-	490	11.1	-	>20	-620	22.4
Leach 7	72	-	-	532	11.1	-			
Leach 8	96	-	-	530	11.1	-	>20	-626	24.2
Total	96	4.80	0.99		-	-	-	-	
CaO Titration									

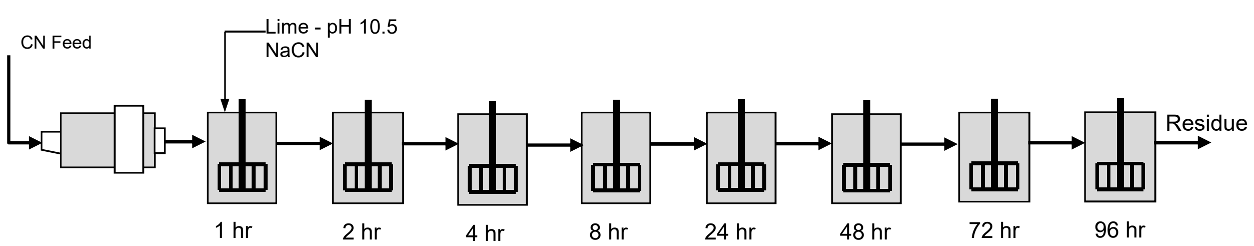
Mass of Sample	2000
Volume of Water	4000
Pulp Density	33

NaCN Consumption	0.45	kg/tonne
Lime Consumption	0.50	kg/tonne

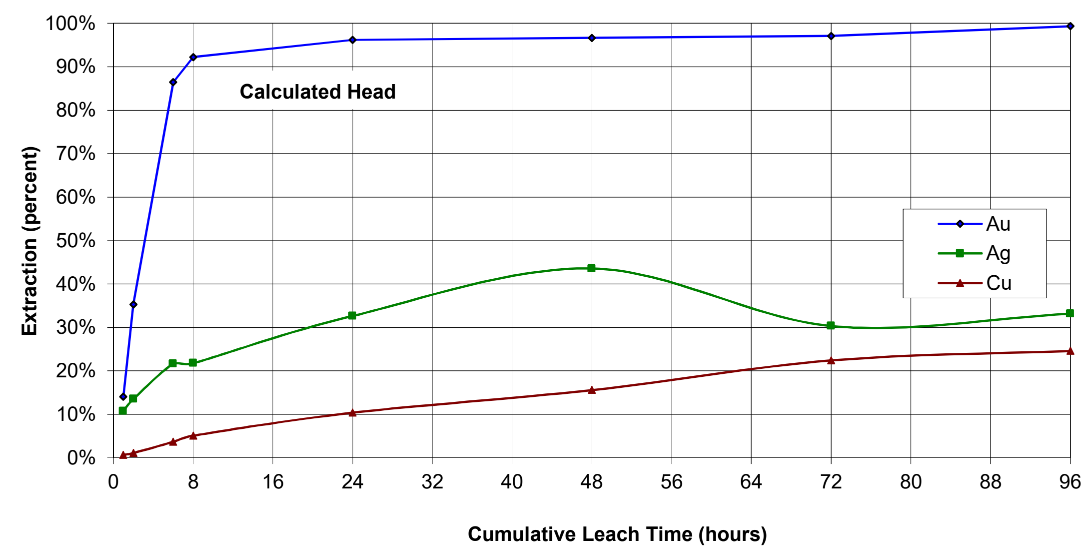
Cumulative Metallurgical Balance

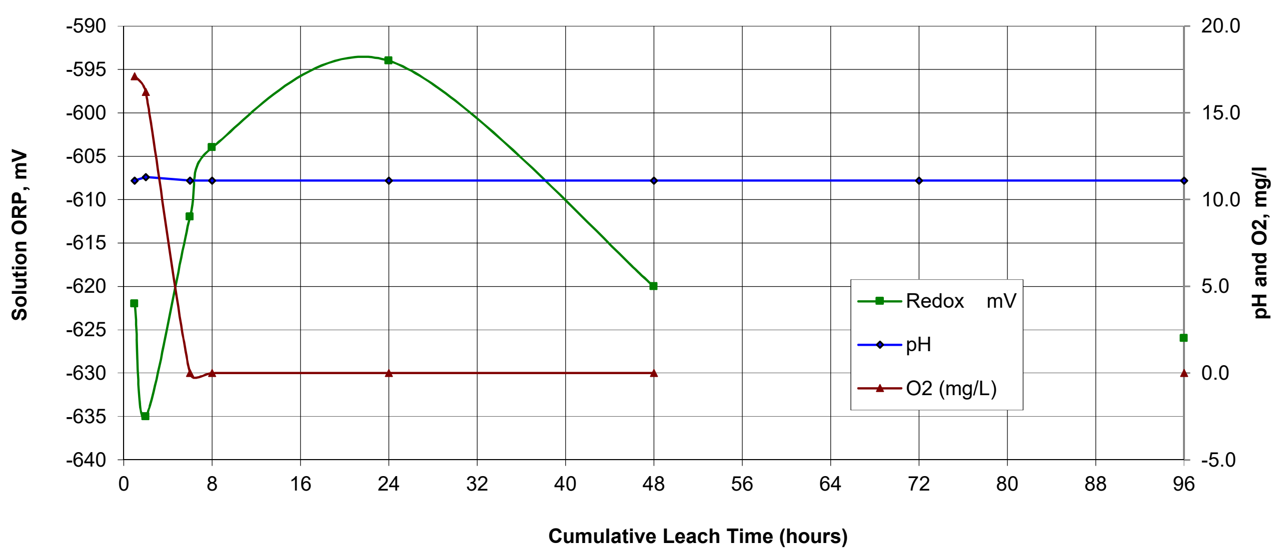
Product	Cumulative	Vol or	Units	Free CN	Free NaCN	mol WAD/	Assay - g/tonne (ppm)			Distribution - percent		
	Time - Hrs	Mass		mg/l, calc'd	mg/l	mol Cu	Au	Ag	Cu	Au	Ag	Cu
Cyanide Liquor (1 hr)	1	4000	mL	466.6	879	5430.66	0.08	0.04	0.21	14.08%	10.77%	0.65%
Cyanide Liquor (2 hr)	2	4000	mL	531.9	1,002	3715.93	0.20	0.05	0.35	35.26%	13.52%	1.09%
Cyanide Liquor (4 hr)	6	4000	mL	531.0	1,000	1103.25	0.49	0.08	1.18	86.47%	21.66%	3.67%
Cyanide Liquor (8 hr)	8	4000	mL	551.0	1,038	829.81	0.52	0.08	1.63	92.18%	21.77%	5.08%
Cyanide Liquor (24hr)	24	4000	mL	539.9	1,017	400.01	0.54	0.12	3.33	96.16%	32.64%	10.37%
Cyanide Liquor (48hr)	48	4000	mL	481.9	908	239.84	0.54	0.16	4.99	96.63%	43.58%	15.57%
Cyanide Liquor (72hr)	72	4000	mL	520.5	980	181.22	0.54	0.11	7.17	97.11%	30.33%	22.41%
Cyanide Liquor (96hr)	96	4000	mL	517.4	975	165.11	0.55	0.12	7.84	99.34%	33.17%	24.59%
Cyanidation Tails	-	1986	g				0.008	0.50	49	0.66%	66.83%	75.41%
Calculated Feed, gm/t		1986	g				1.145	0.75	65	100.00%	100.00%	100.00%
Head Assay, gm/t		2000					0.950	1.000	70			

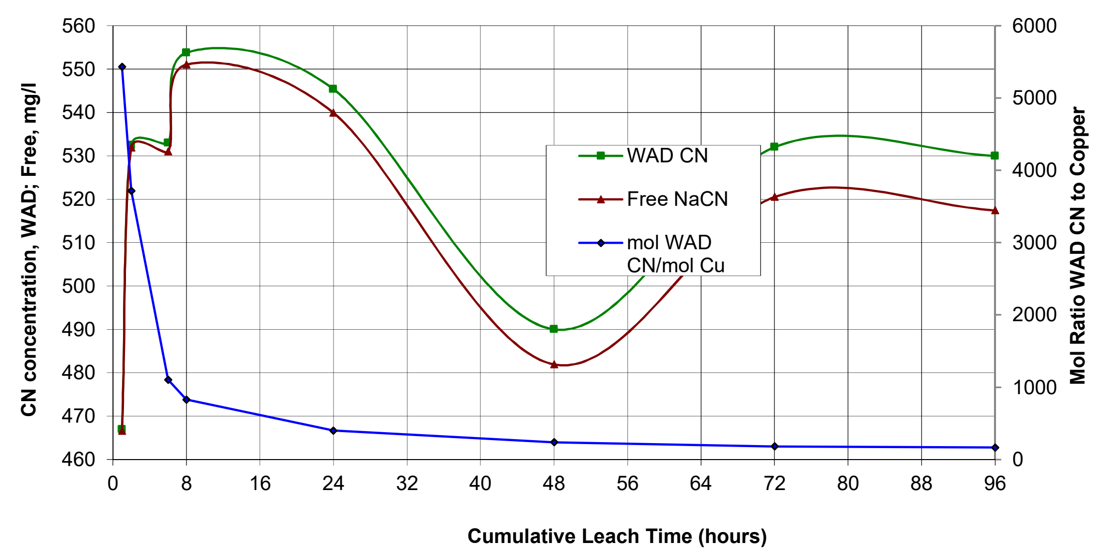
Flowsheet Schematic



Cyanide Leach Kinetic Curves







Test No:	BL295-07			Au, gm/t	Ag, gm/t	Cu, %
Date:	01-Aug-18		Head Assay, Ave:	0.950	1.00	0.01%
Test Type:	Standard bottle roll procedure.		Head Assay 1:	0.95	1	0.01%
Test Objective:	Evaluate grind series with gravity.		Head Assay 2:	0.95	1	0.01%
Sample:	Master Comp		Head Assay 3:	0.95	1	0.01%
Grind:	150µm					
Target Maintenance NaCN:	1000	ppm	Total Mass, Initial:		grams including bottle	

	Au, gm/t	Ag, gm/t	Cu, %
Tails Assay, Ave:	0.010	0.5	0.00%
Tail Assay 1:	0.010	0.5	0.005%
Tail Assay 2:	0.010	0.5	0.005%

	Au	Ag	Cu
Head, mg:	1.889	1.989	139
Tails, mg:	0.020	0.993	95
Calculated Head, mg:	2.429	1.279	111

Sample Size: 20 ml

Calculated Head:

Cyanidation Leaching @ pH 10.5 (lime), 1000ppm NaCN, O2 sparged,

Parameter	Time	Added (g)		WAD CN	pH		Dissolved	Redox	Temp
	Cum	NaCN	Lime	mg/l CN	Measured	Adjusted	O_2 (mg/L)	mV	Deg. C
Natural	-	-	-		8.6	-	6.8	-25	22.0
Leach 0	0	4.00	0.60		10.5	10.5	6.8	-665	22.0
Leach 1	1	0.44	-	488	10.9	-	20.0	-605	22.1
Leach 2	2	0.16	-	500	10.9	-	>20	-605	25.2
Leach 3	4	-	-	534	10.9	-	>20	-609	23.2
Leach 4	8	0.12	-	504	10.8	-	>20	-613	23.5
Leach 5	24	0.08	-	510	10.8	-	>20	-621	22.3
Leach 6	48	-	-	524	10.8	-	>20	-625	22.5
Leach 7	72	-	-	522	10.8	-	>20	-626	24.2
Total	72	4.80	0.60		-	-	-	-	

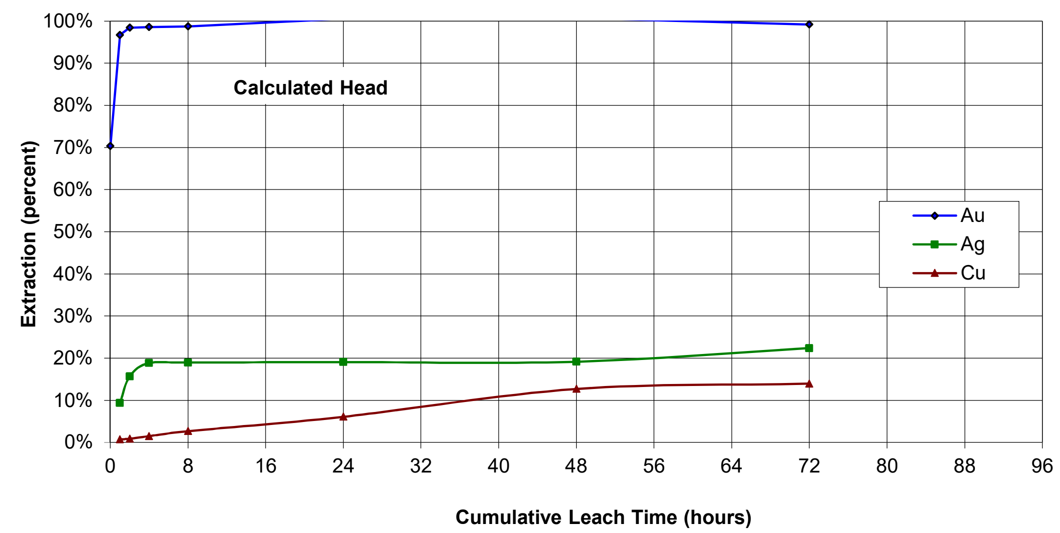
Cumulative Metallurgical Balance

Product	Cumulative	Vol or	Units	Free CN	Free NaCN	mol WAD/	Assay - g/tonne (ppm)			Distribution - percent		
	Time - Hrs	Mass		mg/l, calc'd	mg/l	mol Cu	Au	Ag	Cu	Au	Ag	Cu
Pan Con	0	3.5	g				488			70.34%	0.00%	0.00%
Cyanide Liquor (1 hr)	1	4000	mL	487.6	918	5675.71	0.16	0.03	0.21	96.69%	9.38%	0.76%
Cyanide Liquor (2 hr)	2	4000	mL	499.5	941	4884.83	0.17	0.05	0.25	98.47%	15.69%	0.91%
Cyanide Liquor (4 hr)	4	4000	mL	533.3	1,004	3105.36	0.17	0.06	0.42	98.61%	18.89%	1.53%
Cyanide Liquor (8 hr)	8	4000	mL	502.8	947	1663.48	0.17	0.06	0.74	98.75%	18.99%	2.69%
Cyanide Liquor (24hr)	24	4000	mL	507.3	956	741.45	0.18	0.06	1.68	100.53%	19.08%	6.10%
Cyanide Liquor (48hr)	48	4000	mL	518.4	976	365.66	0.18	0.06	3.50	100.68%	19.17%	12.70%
Cyanide Liquor (48hr)	72	4000	mL	516.4	973	269.44	0.17	0.07	3.83	99.18%	22.39%	13.96%
Cyanidation Tails	-	1985	g				0.010	0.50	48	0.82%	77.61%	86.04%
Calculated Feed, gm/t		1989	g				1.222	0.64	56	100.00%	100.00%	100.00%
Head Assay, gm/t		2000					0.950	1.00	70			

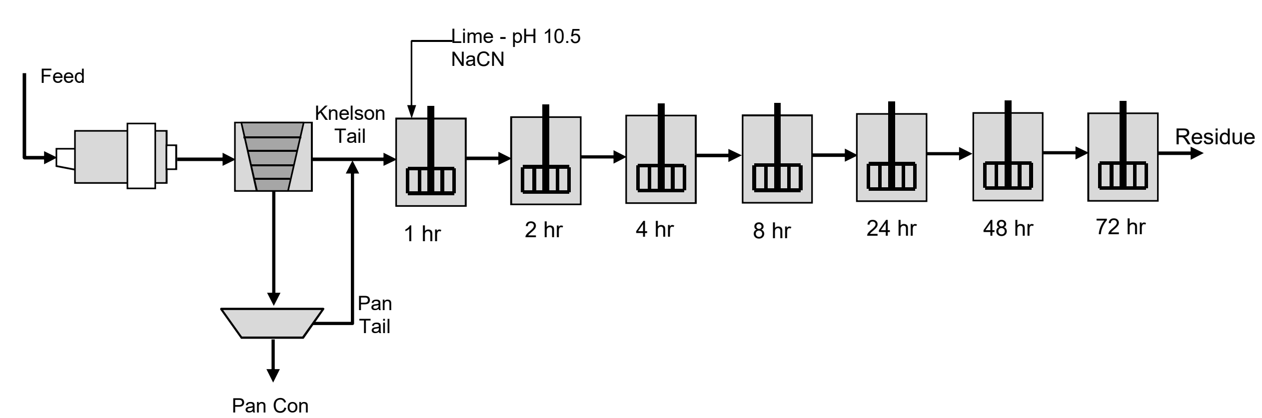
Mass of Sample	2000
Volume of Water	4000
Pulp Density	33

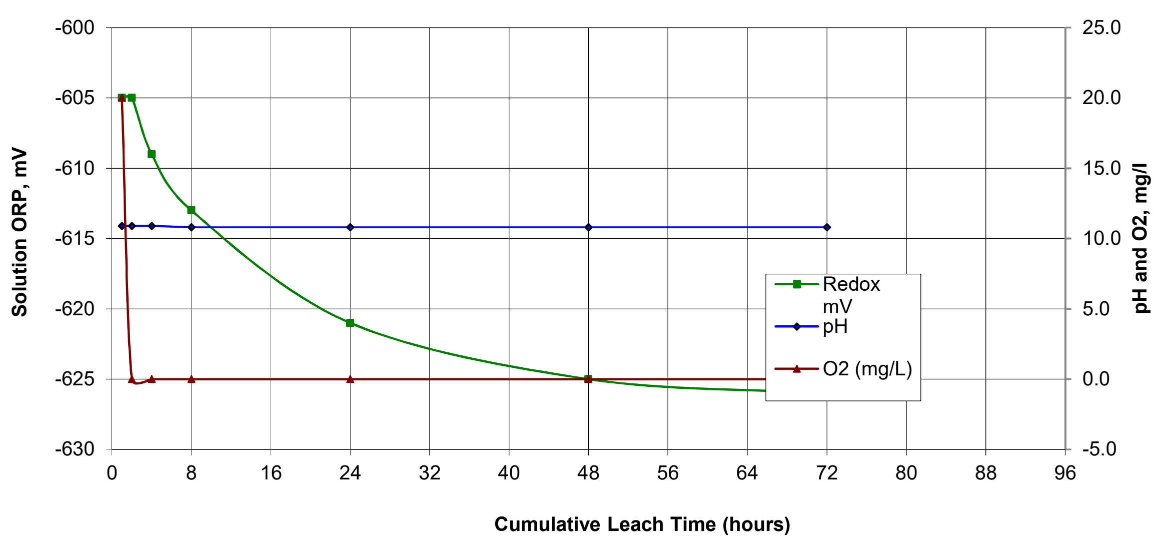
NaCN Consumption	0.45	kg/tonne
Lime Consumption	0.30	kg/tonne

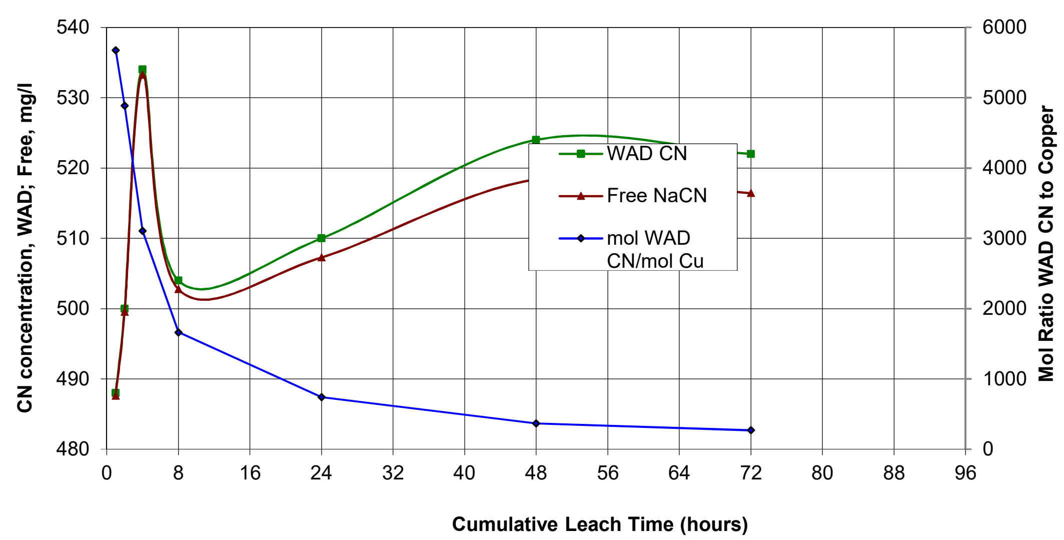
Cyanide Leach Kinetic Curves



Flowsheet Schematic







Test No:	BL295-08		Au, gm/t	Ag, gm/t	Cu, %		
Date:	01-Aug-18	Head Assay, Ave:	0.950	1.00	0.01%		
Test Type:	Standard bottle roll procedure.	Head Assay 1:	0.95	1	0.01%		
Test Objective:	Evaluate grind series with gravity.	Head Assay 2:	0.95	1	0.01%		
Sample:	Master Comp	Head Assay 3:	0.95	1	0.01%		
Grind:	106µm						
Target Maintenance NaCN:	1000 ppm	Total Mass, Initial:		grams including bottle			

	Au, gm/t	Ag, gm/t	Cu, %
Tails Assay, Ave:	0.020	0.5	0.01%
Tail Assay 1:	0.020	0.5	0.005%
Tail Assay 2:	0.020	0.5	0.005%
Sample Size	20	ml	
Calculated Head			

	Au	Ag	Cu
Head, mg:	1.887	1.986	139
Tails, mg:	0.040	0.991	100
Calculated Head, mg:	2.712	1.280	119

Gravity Cyanidation Leaching @ pH 10.5 (lime), 1000ppm NaCN, O2 sparged,

Parameter	Time	Added (g)		WAD CN	pH		Dissolved	Redox	Temp
	Cum	NaCN	Lime	mg/l CN	Measured	Adjusted	O_2 (mg/L)	mV	Deg. C
Natural	-	-	-		8.6	-	6.0	-25	22.0
Leach 0	0	4.00	0.59		10.5	10.5	6.0	-665	22.0
Leach 1	1	0.32	-	492	10.8	-	>20	-607	22.0
Leach 2	2	0.24	-	500	10.7	-	>20	-601	22.5
Leach 3	4	-	-	530	10.8	-	>20	-613	23.3
Leach 4	8	-	-	524	10.7	-	>20	-615	23.6
Leach 5	24	0.12	-	502	10.7	-	>20	-621	22.2
Leach 6	48	-	-	530	10.7	-	>20	-620	23.5
Leach 7	72	-	-	524	10.8	-	>20	-626	24.2
Total	72	4.68	0.59		-	-	-	-	
CaO Titration									

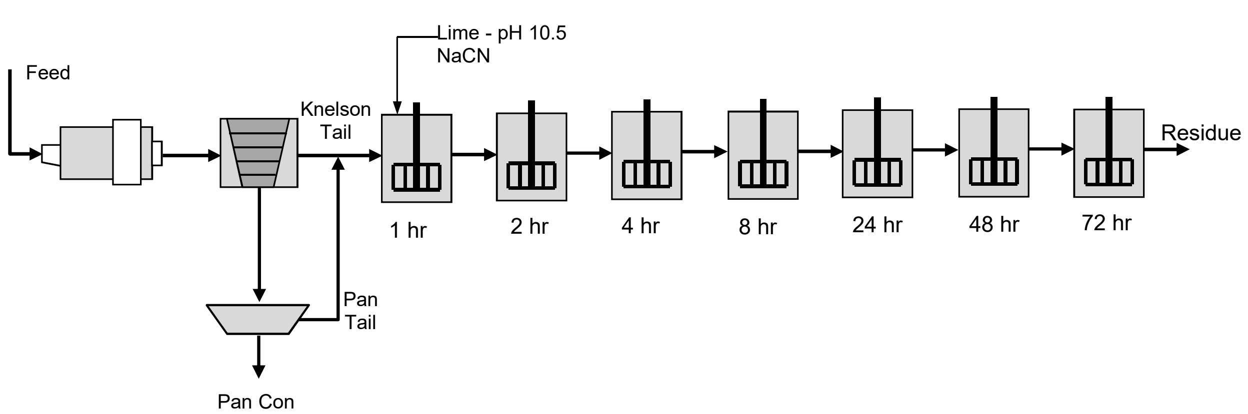
Cumulative Metallurgical Balance

Product	Cumulative	Vol or	Units	Free CN	Free NaCN	mol WAD/	Assay - g/tonne (ppm)			Distribution - percent		
	Time - Hrs	Mass		mg/l, calc'd	mg/l	mol Cu	Au	Ag	Cu	Au	Ag	Cu
Pan Con		4.2	g				460			71.27%	0.00%	0.00%
Cyanide Liquor (1 hr)	1	4000	mL	491.5	926	4291.67	0.16	0.04	0.28	94.87%	12.50%	0.94%
Cyanide Liquor (2 hr)	2	4000	mL	499.3	940	2978.55	0.15	0.07	0.41	93.51%	21.94%	1.38%
Cyanide Liquor (4 hr)	4	4000	mL	528.9	996	2087.87	0.15	0.09	0.62	93.62%	28.31%	2.09%
Cyanide Liquor (8 hr)	8	4000	mL	522.2	984	1207.38	0.16	0.09	1.06	95.21%	28.45%	3.58%
Cyanide Liquor (24hr)	24	4000	mL	498.3	939	540.13	0.18	0.08	2.27	98.27%	25.46%	7.65%
Cyanide Liquor (48hr)	48	4000	mL	523.1	985	297.58	0.18	0.06	4.35	98.41%	19.33%	14.67%
Cyanide Liquor (72hr)	72	4000	mL	516.4	973	269.44	0.18	0.07	4.75	98.54%	22.55%	16.08%
Cyanidation Tails	-	1982	g				0.020	0.50	51	1.46%	77.45%	83.92%
Calculated Feed, gm/t		1986	g				1.366	0.64	60	100.00%	100.00%	100.00%
Head Assay, gm/t		2000					0.950	1.00	70			

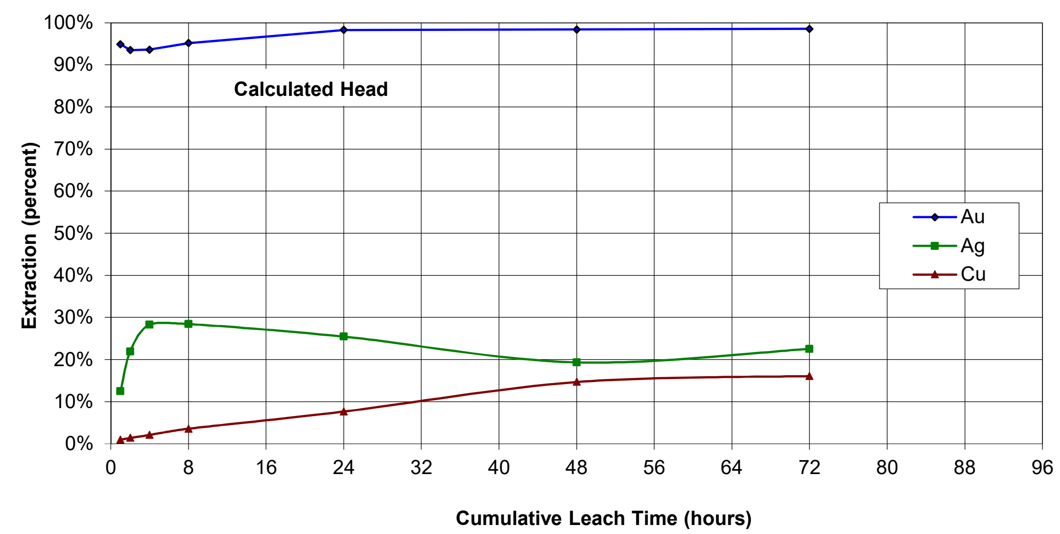
Mass of Sample	2000
Volume of Water	4000
Pulp Density	33

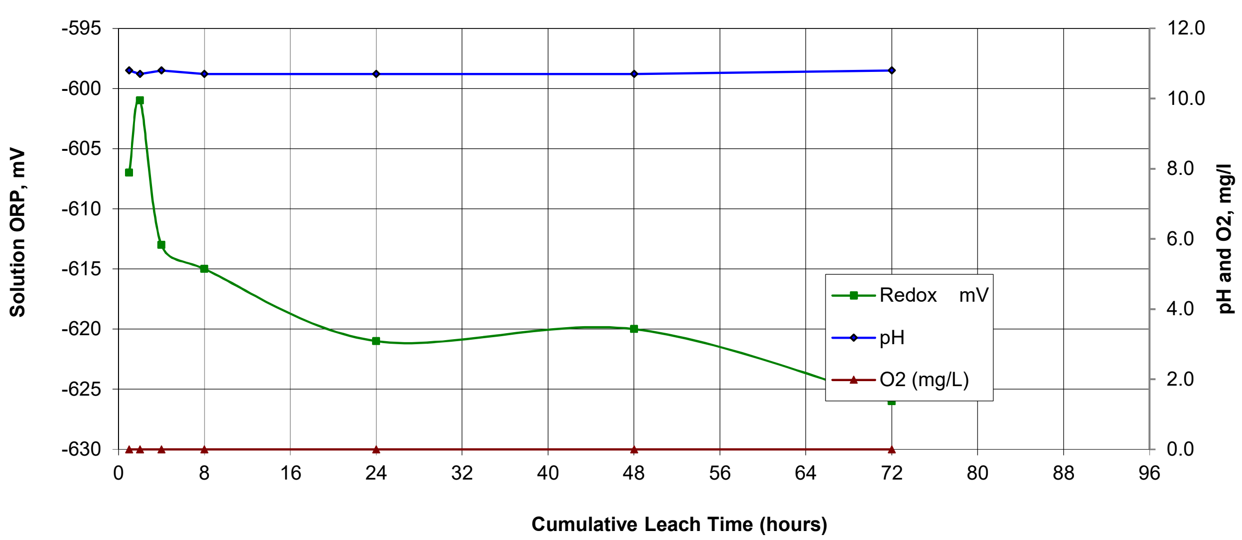
NaCN Consumption	0.39	kg/tonne
Lime Consumption	0.30	kg/tonne

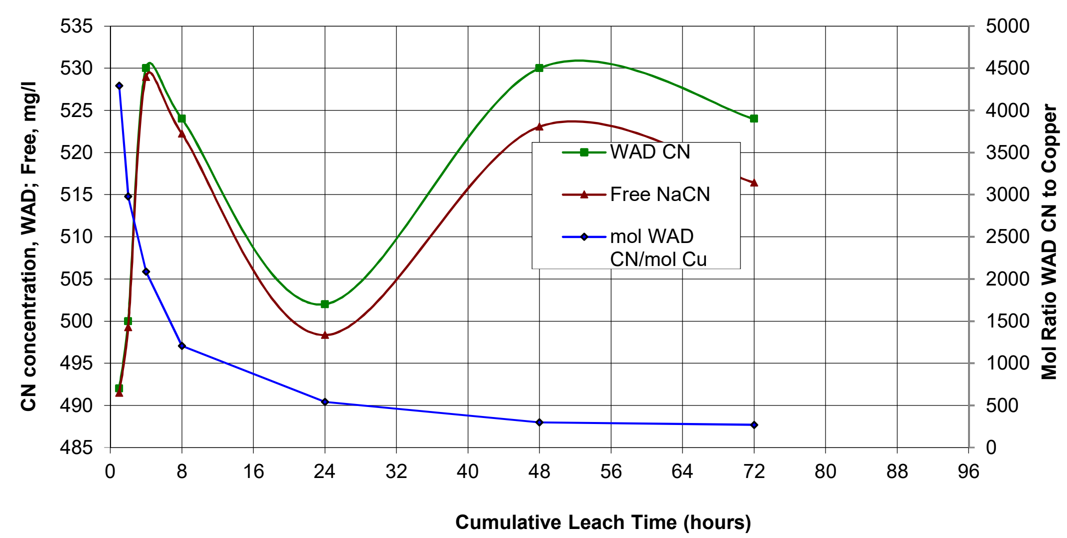
Flowsheet Schematic



Cyanide Leach Kinetic Curves







Test No:	BL295-09			Au, gm/t	Ag, gm/t	Cu, %
Date:	01-Aug-18		Head Assay, Ave:	0.950	1.00	0.01%
Test Type:	Standard bottle roll procedure.		Head Assay 1:	0.95	1	0.01%
Test Objective:	Evaluate grind series with gravity.		Head Assay 2:	0.95	1	0.01%
Sample:	Master Comp		Head Assay 3:	0.95	1	0.01%
Grind:	75µm					
Target Maintenance NaCN:	1000	ppm	Total Mass, Initial:		grams including bottle	

	Au, gm/t	Ag, gm/t	Cu, %
Tails Assay, Ave:	0.015	0.5	0.01%
Tail Assay 1:	0.010	0.5	0.005%
Tail Assay 2:	0.020	0.5	0.006%
Sample Size	20	ml	
Calculated Head			

	Au	Ag	Cu
Head, mg:	1.888	1.988	139
Tails, mg:	0.030	0.993	108
Calculated Head, mg:	1.816	1.361	130

Gravity Cyanidation Leaching @ pH 10.5 (lime), 1000ppm NaCN, O2 sparged,

Parameter	Time	Added (g)		WAD CN	pH		Dissolved	Redox	Temp
	Cum	NaCN	Lime	mg/l CN	Measured	Adjusted	O_2 (mg/L)	mV	Deg. C
Natural	-	-	-		8.5	-	6.6	-25	22.0
Leach 0	0	4.00	0.59		10.5	10.5	6.6	-665	22.0
Leach 1	1	0.24	-	504	10.8	-	>20	-611	22.0
Leach 2	2	0.20	-	506	10.8	-	>20	-598	22.5
Leach 3	4	-	-	524	10.8	-	>20	-609	23.2
Leach 4	8	-	-	522	10.7	-	>20	-610	23.5
Leach 5	24	0.32	-	492	10.7	-	>20	-617	22.3
Leach 6	48	-	-	530	10.7	-	>20	-620	23.5
Leach 7	72	-	-	520	10.8	-	>20	-623	24.3
Total	72	4.76	0.59		-	-	-	-	

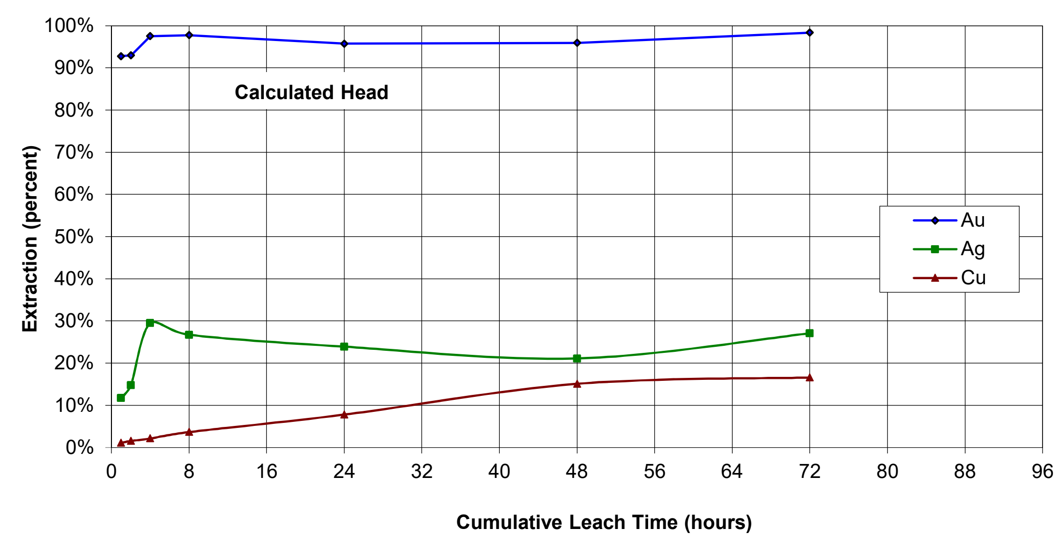
Cumulative Metallurgical Balance

Product	Cumulative	Vol or	Units	Free CN	Free NaCN	mol WAD/	Assay - g/tonne (ppm)			Distribution - percent		
	Time - Hrs	Mass		mg/l, calc'd	mg/l	mol Cu	Au	Ag	Cu	Au	Ag	Cu
Pan Con	0	2.3	g				437			55.33%	0.00%	0.00%
Cyanide Liquor (1 hr)	1	4000	mL	503.4	948	3419.38	0.17	0.04	0.36	92.77%	11.75%	1.11%
Cyanide Liquor (2 hr)	2	4000	mL	505.1	951	2471.72	0.17	0.05	0.50	92.95%	14.75%	1.54%
Cyanide Liquor (4 hr)	4	4000	mL	522.8	985	1854.82	0.19	0.10	0.69	97.55%	29.52%	2.14%
Cyanide Liquor (8 hr)	8	4000	mL	520.0	980	1080.46	0.19	0.09	1.18	97.76%	26.73%	3.66%
Cyanide Liquor (24hr)	24	4000	mL	487.9	919	476.85	0.18	0.08	2.52	95.76%	23.92%	7.80%
Cyanide Liquor (48hr)	48	4000	mL	522.2	984	265.26	0.18	0.07	4.88	95.96%	21.10%	15.10%
Cyanide Liquor (72hr)	72	4000	mL	511.5	963	237.84	0.19	0.09	5.34	98.36%	27.08%	16.59%
Cyanidation Tails	-	1985	g				0.015	0.50	55	1.64%	72.92%	83.41%
Calculated Feed, gm/t		1988	g				0.914	0.68	65	100.00%	100.00%	100.00%
Head Assay, gm/t		2000					0.950	1.00	70			

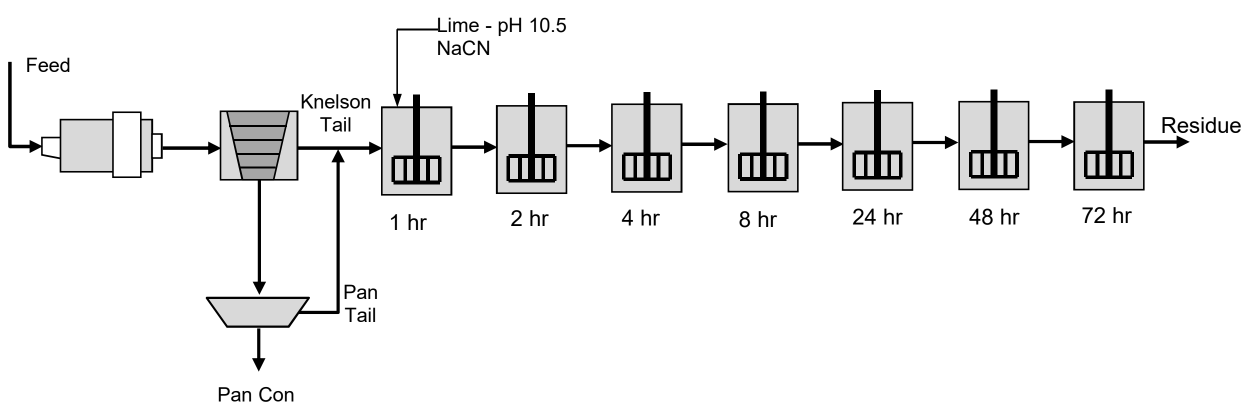
Mass of Sample	2000
Volume of Water	4000
Pulp Density	33

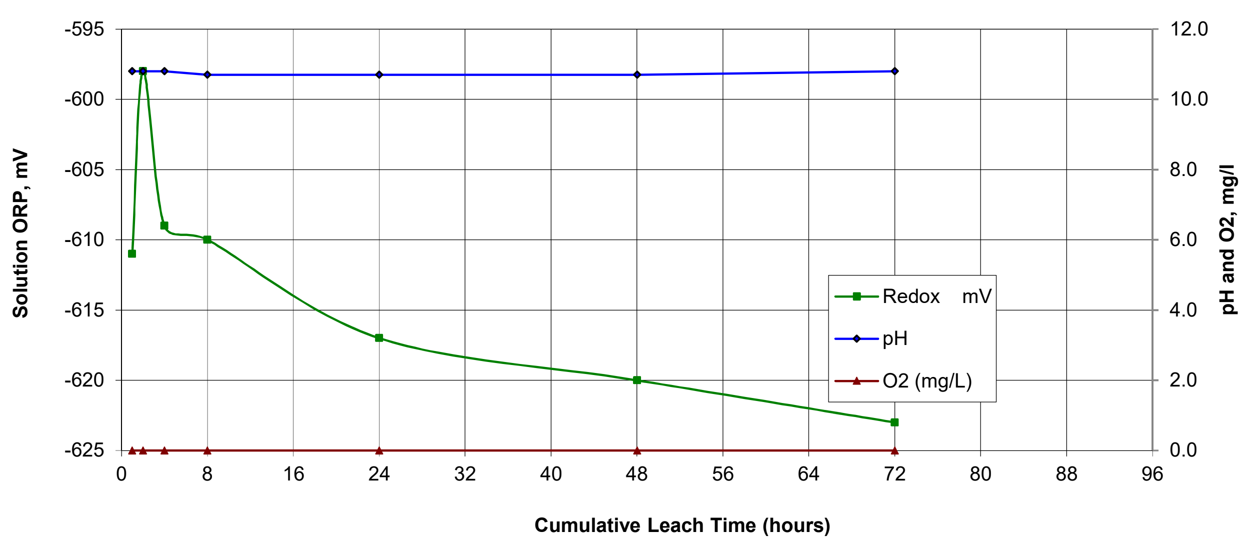
NaCN Consumption	0.45	kg/tonne
Lime Consumption	0.30	kg/tonne

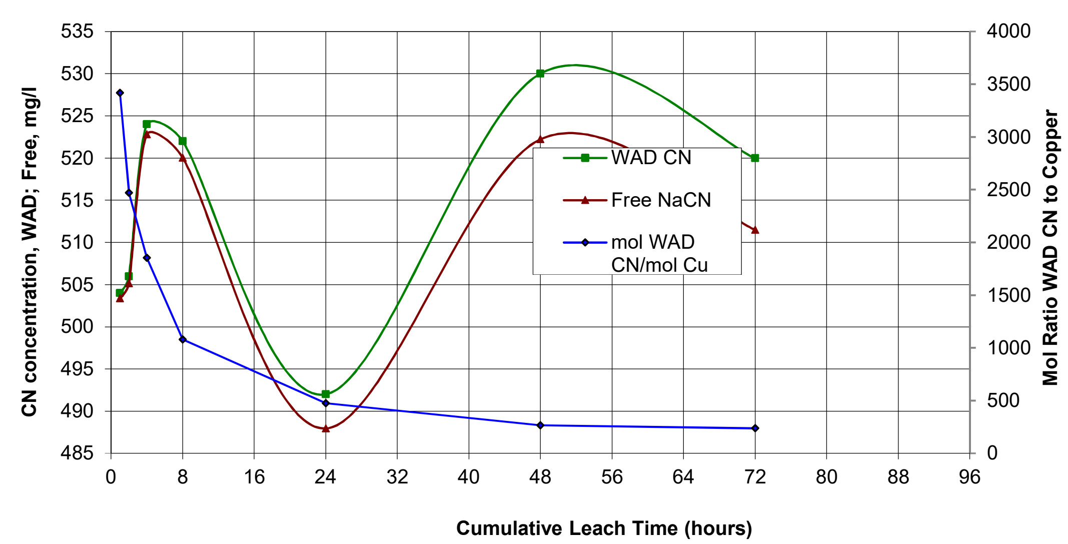
Cyanide Leach Kinetic Curves



Flowsheet Schematic







Test No:	BL295-10		Au, gm/t	Ag, gm/t	Cu, %
Date:	01-Aug-18	Head Assay, Ave:	0.950	1.00	0.01%
Test Type:	Standard bottle roll procedure.	Head Assay 1:	0.95	1	0.01%
Test Objective:	Evaluate grind series with gravity.	Head Assay 2:	0.95	1	0.01%
Sample:	Master Comp	Head Assay 3:	0.95	1	0.01%
Grind:	53µm				
Target Maintenance NaCN:	1000 ppm	Total Mass, Initial:		grams including bottle	

	Au, gm/t	Ag, gm/t	Cu, %
Tails Assay, Ave:	0.015	0.5	0.01%
Tail Assay 1:	0.010	0.5	0.005%
Tail Assay 2:	0.020	0.5	0.005%

Sample Size: 20 ml
Calculated Head:

	Au	Ag	Cu
Head, mg:	1.889	1.988	139.167
Tails, mg:	0.030	0.993	106.444
Calculated Head, mg:	2.003	1.326	131

Gravity Cyanidation Leaching @ pH 10.5 (lime), 1000ppm NaCN, O2 sparged,

Parameter	Time Cum	Added (g) NaCN	Added (g) Lime	WAD CN mg/l CN	pH Measured	pH Adjusted	Dissolved O_2 (mg/L)	Redox mV	Temp Deg. C
Natural	-	-	-		8.5	-	5.9	-25	22.0
Leach 0	0	4.00	0.60		10.5	10.5	5.9	-665	22.0
Leach 1	1	0.28	-	490	10.7	-	>20	-614	22.4
Leach 2	2	0.16	-	504	10.7	-	>20	-603	22.7
Leach 3	4	-	-	532	10.8	-	>20	-611	23.4
Leach 4	8	-	-	528	10.7	-	>20	-613	23.5
Leach 5	24	0.16	-	496	10.6	-	>20	-620	22.1
Leach 6	48	-	-	524	10.7	-	>20	-620	22.8
Leach 7	72	-	-	500	10.7	-	>20	-621	24.3
Total	72	4.60	0.60		-	-	-	-	

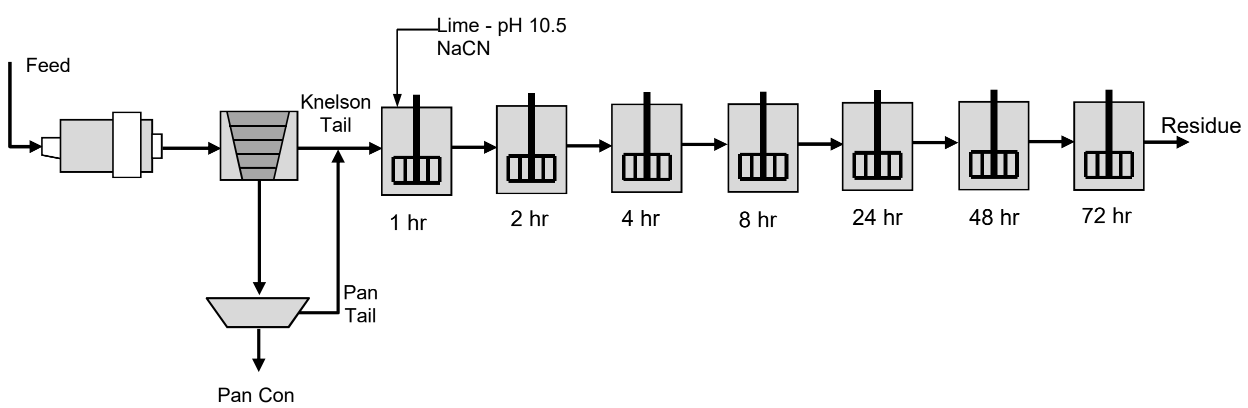
Cumulative Metallurgical Balance

Product	Cumulative Time - Hrs	Vol or Mass	Units	Free CN mg/l, calc'd	Free NaCN mg/l	mol WAD/ mol Cu	Assay - g/tonne (ppm) Au	Ag	Cu	Distribution - percent Au	Ag	Cu
Pan Con		2.2	g				560			61.49%	0.00%	0.00%
Cyanide Liquor (1 hr)	1	4000	mL	489.4	922	3419.38	0.11	0.07	0.35	83.46%	21.12%	1.06%
Cyanide Liquor (2 hr)	2	4000	mL	503.1	948	2322.60	0.14	0.08	0.53	89.56%	24.24%	1.62%
Cyanide Liquor (4 hr)	4	4000	mL	530.5	999	1565.50	0.25	0.24	0.83	111.66%	72.63%	2.54%
Cyanide Liquor (8 hr)	8	4000	mL	525.3	989	772.21	0.21	0.11	1.67	103.92%	33.77%	5.11%
Cyanide Liquor (24hr)	24	4000	mL	491.4	926	422.10	0.20	0.08	2.87	102.14%	24.89%	8.78%
Cyanide Liquor (48hr)	48	4000	mL	514.8	970	222.19	0.17	0.07	5.76	96.35%	21.99%	17.62%
Cyanide Liquor (72hr)	72	4000	mL	490.2	923	196.97	0.18	0.08	6.20	98.51%	25.11%	19.04%
Cyanidation Tails	-	1986	g				0.015	0.50	54	1.49%	74.89%	80.96%
Calculated Feed, gm/t		1988	g				1.008	0.67	66	100.00%	100.00%	100.00%
Head Assay, gm/t		2000					0.950	1.00	70			

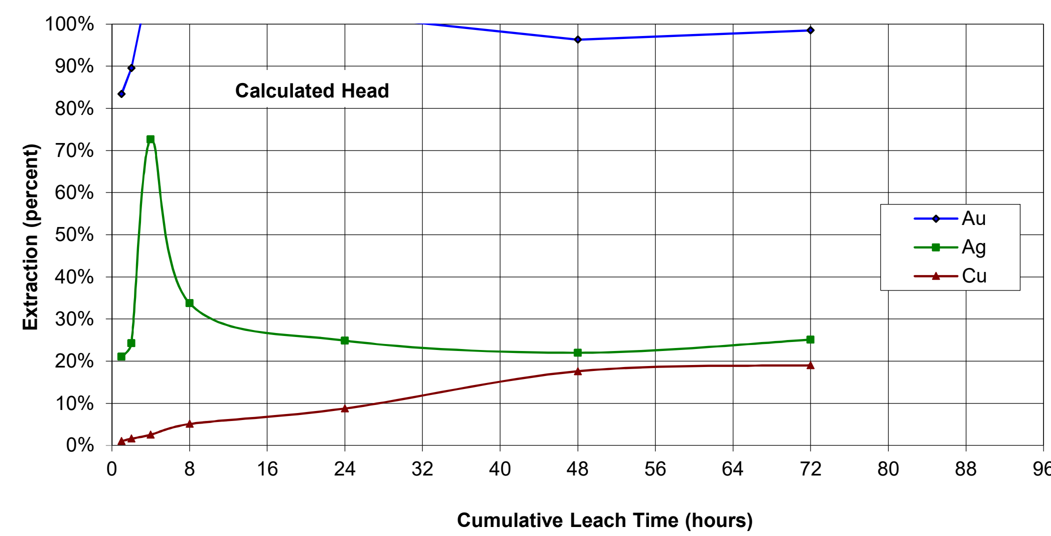
Mass of Sample	2000
Volume of Water	4000
Pulp Density	33

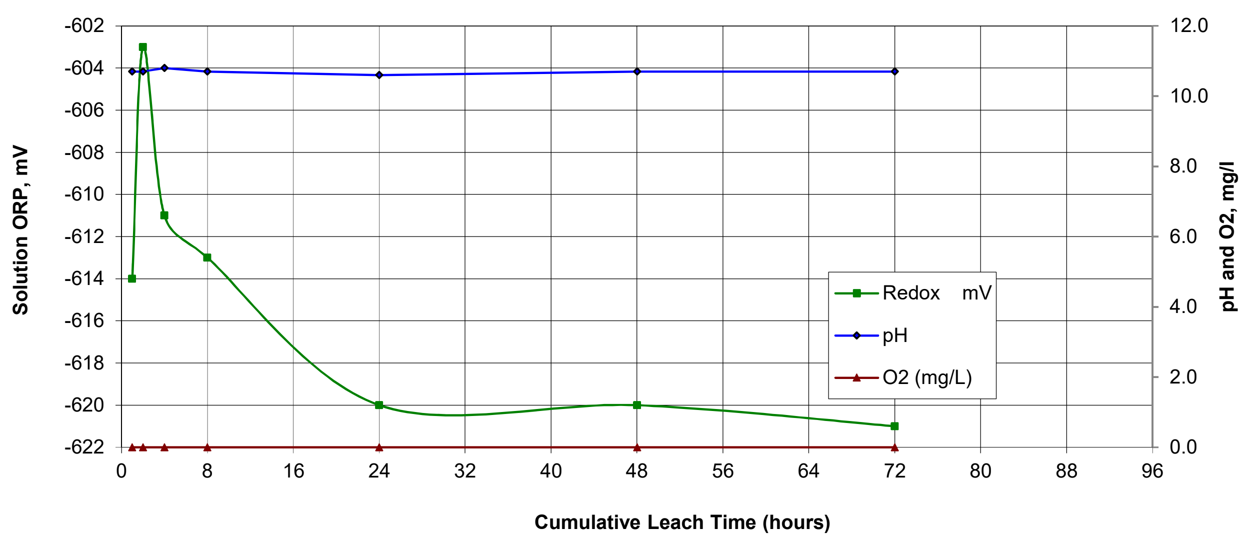
NaCN Consumption	0.45	kg/tonne
Lime Consumption	0.30	kg/tonne

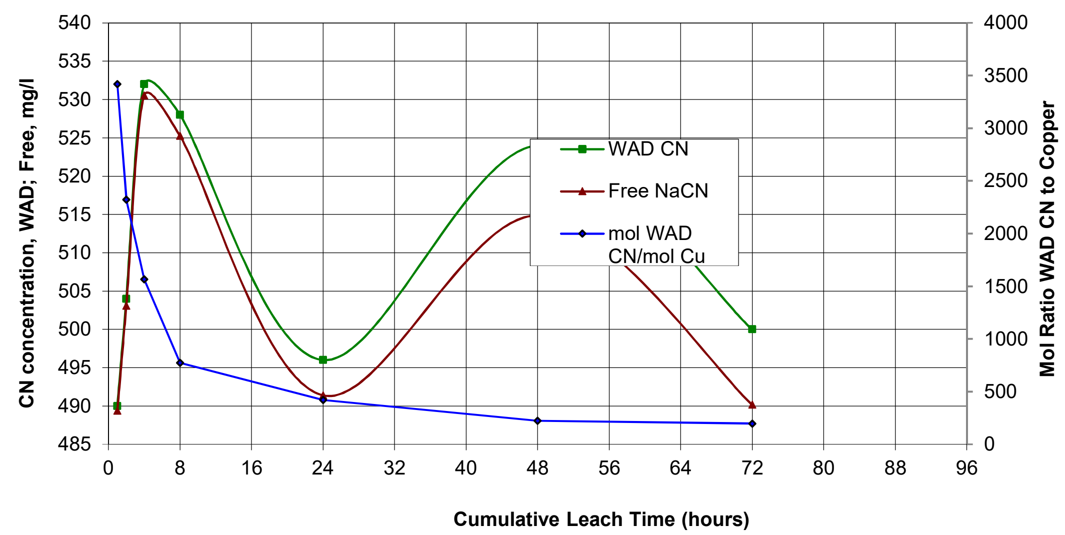
Flowsheet Schematic



Cyanide Leach Kinetic Curves

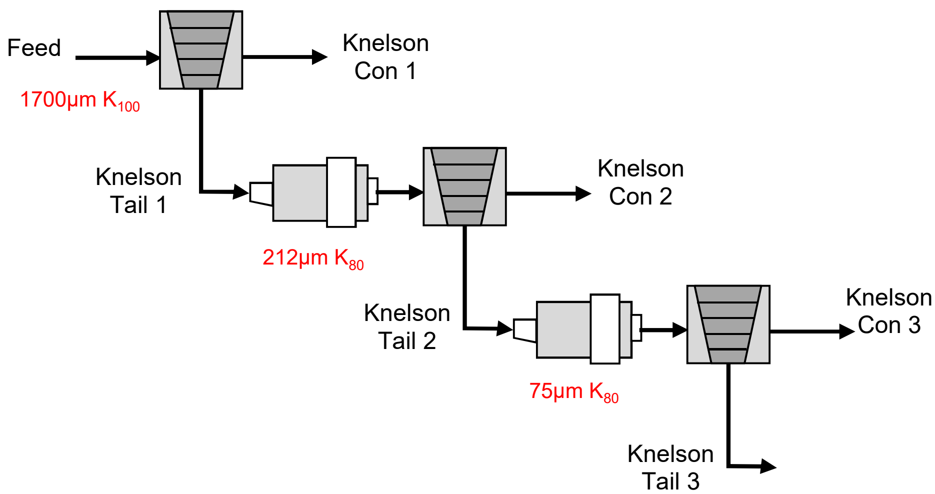






Test No: BL0295-11
Date: 13-Aug-18
Test Type: GRG
Test Objective: Evaluate gravity recoverable gold
Sample: Master Composite
Nominal Sizing: 1700µm K_{100}

Flowsheet Schematic



Knelson Concentration Conditions	
G's	60
Water pressure to bowl	13.8 kPa
Fluidisation Water Flowrate	4.5
Feed Mass - kg	19.0

BL0295-11 Master Composite
Metallurgical Balance

Sample ID	Product	Weight		Assay - g/t	Dist'n
		%	grams	Au	(%)
Master Composite	Knelson Con 1	0.6	96.3	60.2	28.7
	Knelson Con 2	0.4	67.1	120	39.8
	Knelson Con 3	0.4	60.8	46.2	13.9
	Knelson Tail 3	98.7	16812	0.21	17.5
Recalculated Feed			17036	1.18	
GRG (%)					82.5

BL0295-11 Master Composite Knelson Con 1
ASSAY BY SIZE

Sample ID	Size Fraction	Mass			Assay - g/tonne	Distr'n - %
	µm	g	%	Cum %	Gold	Gold
Knelson Con 1	1700	9.9	10.3	89.7	0.34	0.1
	1180	28.5	29.6	60.1	2.08	1.0
	850	17.8	18.5	41.6	8.40	2.6
	600	10.8	11.2	30.4	3.35	0.6
	425	7.1	7.4	23.1	0.17	0.0
	300	5.1	5.3	17.8	175	15.4
	212	4.0	4.2	13.6	138	9.5
	150	3.4	3.5	10.1	298	17.5
	106	2.8	2.9	7.2	249	12.0
	75	2.2	2.3	4.9	390	14.8
	53	1.6	1.7	3.2	352	9.7
	38	1.3	1.3	1.9	323	7.3
	25	1.0	1.0	0.8	323	5.6
	-25	0.8	0.8	0.0	282	3.9
Calc Head		96.3	100.0		60.2	100.0

BL0295-11 Master Composite Knelson Con 2
ASSAY BY SIZE

Sample ID	Size Fraction	Mass			Assay - g/tonne	Distr'n - %
	µm	g	%	Cum %	Gold	Gold
Knelson Con 2	1700		0.0	100.0		0.0
	1180		0.0	100.0		0.0
	850		0.0	100.0		0.0
	600	0.6	0.9	99.1	27.2	0.2
	425	0.5	0.7	98.4	27.2	0.2
	300	2.6	3.9	94.5	27.2	0.9
	212	6.8	10.1	84.4	65.1	5.5
	150	10.5	15.6	68.7	130	16.9
	106	10.9	16.2	52.5	150	20.4
	75	10.6	15.8	36.7	127	16.8
	53	8.3	12.4	24.3	117	12.1
	38	7.2	10.7	13.6	127	11.4
	25	4.4	6.6	7.0	121	6.6
	-25	4.7	7.0	0.0	154	9.0
Calc Head		67.1	100.0		120	100.0

Note: Fractions 600 to 300 combined for assay.

BL0295-11 Master Composite Knelson Con 3
ASSAY BY SIZE

Sample ID	Size Fraction	Mass			Assay - g/tonne	Distr'n - %
	µm	g	%	Cum %	Gold	Gold
Knelson Con 3	1700		0.0	100.0		0.0
	1180		0.0	100.0		0.0
	850		0.0	100.0		0.0
	600	0.1	0.2	99.8	12.4	0.0
	425	0.1	0.2	99.7	12.4	0.0
	300	0.3	0.5	99.2	12.4	0.1
	212	1.2	2.0	97.2	12.4	0.5
	150	4.8	7.9	89.3	26.8	4.6
	106	9.3	15.3	74.0	12.5	4.1
	75	13.4	22.0	52.0	18.8	9.0
	53	11.4	18.8	33.2	42.1	17.1
	38	9.5	15.6	17.6	59.5	20.1
	25	5.4	8.9	8.7	91.7	17.6
	-25	5.3	8.7	24.5	142	26.7
Calc Head		60.8	100.0		46.2	100.0

Note: Fractions 600 to 212 combined for assay.

BL0295-11 Master Composite Knelson Tail 3
ASSAY BY SIZE

Sample ID	Size Fraction	Mass			Assay - g/t	Distr'n - %
	µm	g	%	Cum %	Gold	Gold
Knelson Tail 3	1700		0	100		0.00
	1180		0.0	100.0		0.0
	850		0.0	100.0		0.0
	600		0.0	100.0		0.0
	425		0.0	100.0		0.0
	300		0.0	100.0		0.0
	212	0.1	0.0	100.0	0.17	0.0
	150	3.5	0.7	99.3	0.17	0.6
	106	21.3	4.3	95.0	0.13	2.6
	75	52.4	10.5	84.5	0.12	6.1
	53	72.0	14.4	70.1	0.14	9.3
	38	95.3	19.1	51.1	0.13	11.7
	25	157.2	31.4	19.6	0.12	17.9
	-25	98.2	19.6	50.5	0.56	51.9
Calc Head		500.0	100.0		0.21	100.0
K80		67				

Note: Fractions 212 to 150 combined for assay.

Duplicate Knelson Tails Assays - g/t	
Cut	Gold
1	0.13
2	0.13

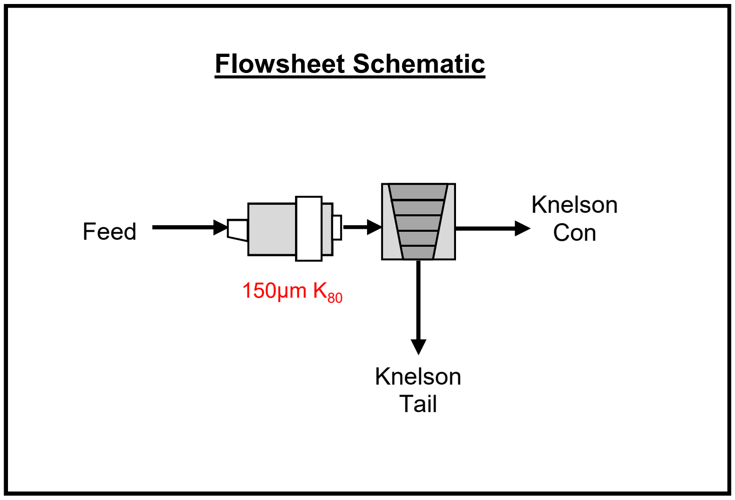
BL0295-11 Master Composite Feed
ASSAY BY SIZE

Sample ID	Size Fraction	Mass			Assay - g/t	Distr'n - %
	µm	g	%	Cum %	Gold	Gold
Master Composite - Feed	1700	31.5	3.2	96.8	0.10	0.5
	1180	180.2	18.1	78.7	0.59	15.8
	850	158.0	15.9	62.8	0.50	11.9
	600	124.3	12.5	50.3	0.67	12.5
	425	95.4	9.6	40.7	0.57	8.1
	300	75.2	7.6	33.1	0.61	6.9
	212	60.2	6.1	27.0	0.63	5.7
	150	46.0	4.6	22.4	0.47	3.2
	106	40.2	4.0	18.4	1.42	8.5
	75	31.6	3.2	15.2	1.16	5.5
	53	26.9	2.7	12.5	1.79	7.2
	38	27.4	2.8	9.7	1.06	4.3
	25	20.1	2.0	7.7	1.27	3.8
	-25	76.3	7.7	0.0	0.55	6.3
Calc Head		993	100.0		0.67	100.0
K80		1212				

Intermediate Knelson Tails Size Distributions

Size Fraction	Kn Tls 1			Kn Tls 2			Kn Tls 3		
µm	g	%	Cum %	g	%	Cum %	g	%	Cum %
1700	29.6	3.3	96.7	0.0	0.0	100.0	0.0	0.0	100.0
1180	163.9	18.2	78.6	0.0	0.0	100.0	0.0	0.0	100.0
850	146.3	16.2	62.3	0.0	0.0	100.0	0.0	0.0	100.0
600	116.4	12.9	49.4	0.1	0.1	99.9	0.0	0.0	100.0
425	87.1	9.7	39.8	0.9	0.9	99.0	0.0	0.0	100.0
300	67.9	7.5	32.3	3.3	3.3	95.7	0.0	0.0	100.0
212	56.0	6.2	26.1	6.5	6.5	89.2	0.1	0.0	100.0
150	45.6	5.1	21.0	8.9	8.9	80.3	3.5	0.7	99.3
106	41.1	4.6	16.4	10.7	10.7	69.6	21.3	4.3	95.0
75	33.4	3.7	12.7	10.8	10.8	58.8	52.4	10.5	84.5
53	28.8	3.2	9.6	9.9	9.9	48.9	72.0	14.4	70.1
38	25.1	2.8	6.8	9.5	9.5	39.4	95.3	19.1	51.1
25	18.8	2.1	4.7	9.5	9.5	29.9	157.2	31.4	19.6
-25	42.3	4.7	0.0	29.9	29.9	0.0	98.2	19.6	0.0
	902.3	100.0		100.0	100.0		500.0	100.0	
K80	1215			149			67		

Test No:	BL0295-12
Date:	18-Sep-18
Test Type:	Bulk Gravity
Test Objective:	Generate products for downstream evaluation
Sample:	Master Composite
Nominal Sizing:	150µm K_{80}



Knelson Concentration Conditions	
G's	120
Water pressure to bowl	13.8 kPa
Fluidisation Water Flowrate	4.5
Feed Mass - kg	29.1

BL0295-12 Master Composite
Metallurgical Balance

Sample ID	Product	Weight		Assay - g/t	Dist'n
		%	grams	Au	(%)
Master Composite	Knelson Con	0.2	59.0	278	65.5
	Knelson Tail	99.8	29050	0.30	34.5
Recalculated Feed			29109	0.86	

BL0295-12 Master Composite Knelson Tail
ASSAY BY SIZE

Sample ID	Size Fraction	Mass			Assay - g/tonne	Distr'n - %
	µm	g	%	Cum %	Gold	Gold
Master Composite	212	15.0	3.1	96.9	0.36	3.7
	150	67.1	13.9	83.0	0.33	15.5
	106	60.7	12.5	70.5	0.44	18.7
	75	55.8	11.5	59.0	0.28	10.9
	53	48.3	10.0	49.0	0.28	9.5
	38	56.8	11.7	37.2	0.13	5.2
	-38	180.2	37.2	0.0	0.29	36.5
Calc Head		483.9	100.0		0.30	100.0

BL0295-12 Master Composite Feed
ASSAY BY SIZE

Sample ID	Size Fraction	Mass			Assay - g/t	Distr'n - %
	µm	g	%	Cum %	Gold	Gold
Master Composite - Feed	212	8.2	1.6	98.4	0.47	0.8
	150	86.4	17.3	81.1	1.94	32.9
	106	61.6	12.3	68.8	1.72	20.8
	75	57.6	11.5	57.3	1.15	13.1
	53	48.5	9.7	47.6	0.89	8.5
	38	56.6	11.3	36.2	0.71	7.9
	-38	181.3	36.2	0.0	0.45	16.1
Calc Head		500	100.0		1.02	100.0
K80		145				

Test No:	BL295-13		Au, gm/t	Ag, gm/t	Cu, %
Date:	20-Sep-18	Head Assay, Ave:	0.300	1.00	0.01%
Test Type:	Standard bottle roll procedure.	Head Assay 1:	0.30	1	0.01%
Test Objective:	Evaluate air	Head Assay 2:	0.30	1	0.01%
Sample:	Test 12 Knelson Tails	Head Assay 3:	0.30	1	0.01%
Grind:	150µm as received				
Target Maintenance NaCN:	1000 ppm	Total Mass, Initial:		grams including bottle	

	Au, gm/t	Ag, gm/t	Cu, %
Tails Assay, Ave:	0.040	0.5	0.01%
Tail Assay 1:	0.040	0.5	0.006%
Tail Assay 2:	0.040	0.5	0.006%

* Ag estimated, assay value <ppm.

Sample Size: 20 ml

Calculated Head:

	Au	Ag	Cu
Head, mg:	0.299	0.997	69.755
Tails, mg:	0.040	0.498	56.801
Calculated Head, mg:	0.312	0.560	63

Gravity Cyanidation Leaching @ pH 10.5 (lime), 1000ppm NaCN, Air sparged,

Parameter	Time Cum	Added (g) NaCN	Added (g) Lime	WAD CN mg/l CN	pH Measured	pH Adjusted	Dissolved O_2 (mg/L)	Redox mV	Temp Deg. C
Natural	-	-	-		9.6	-	9.6	313	20.2
Leach 0	0	2.00	0.80		10.5	10.5	9.6	228	20.2
Leach 1	1	0.10	-		10.8	-	9.7	228	19.8
Leach 2	2	0.00	-		10.8	-	9.4	207	19.1
Leach 3	4	0.00	-		10.8	-	9.4	189	19.9
Leach 4	8	0.00	-		10.7	-	7.9	-85	20.0
Leach 5	24	0.10	-		10.7	-	7.9	153	19.8
Leach 6	48	0.06	-		10.7	-	7.7	188	21.0
Leach 7	72	-	-	534	10.6	-	7.8	193	20.4
Total	72	2.26	0.80		-	-	-	-	

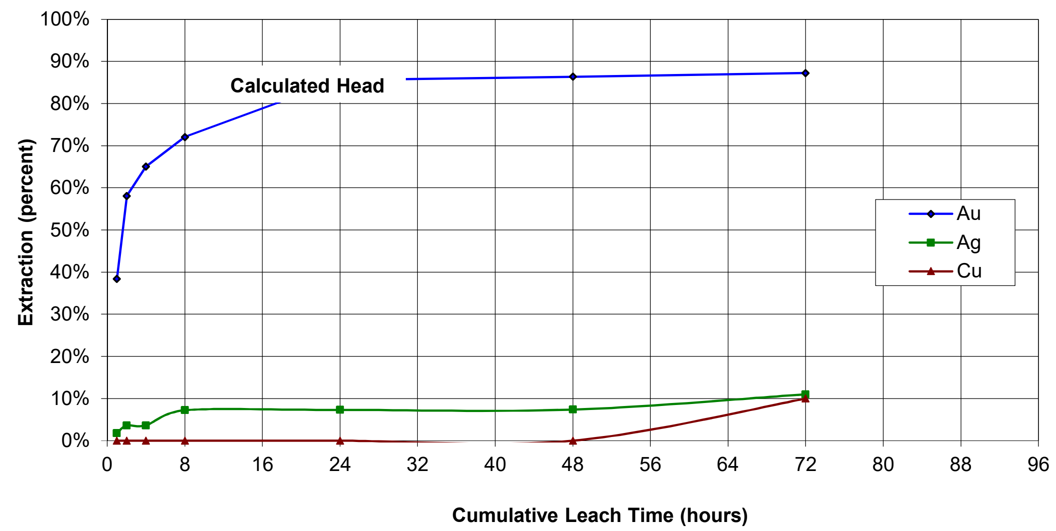
Cumulative Metallurgical Balance

Product	Cumulative Time - Hrs	Vol or Mass	Units	Free CN mg/l, calc'd	Free NaCN mg/l	mol WAD/ mol Cu	Assay - g/tonne (ppm) Au	Ag	Cu	Distribution - percent Au	Ag	Cu
Cyanide Liquor (1 hr)	1	2000	mL	0.0	0	0.00	0.06	0.01	0.00	38.43%	1.79%	0.00%
Cyanide Liquor (2 hr)	2	2000	mL	0.0	0	0.00	0.09	0.01	0.00	58.03%	3.59%	0.00%
Cyanide Liquor (4 hr)	4	2000	mL	0.0	0	0.00	0.10	0.01	0.00	65.01%	3.63%	0.00%
Cyanide Liquor (8 hr)	8	2000	mL	0.0	0	0.00	0.11	0.02	0.00	72.06%	7.23%	0.00%
Cyanide Liquor (24hr)	24	2000	mL	0.0	0	0.00	0.13	0.02	0.00	85.57%	7.30%	0.00%
Cyanide Liquor (48hr)	48	2000	mL	0.0	0	0.00	0.13	0.02	0.00	86.40%	7.38%	0.00%
Cyanide Liquor (72hr)	72	2000	mL	528.9	996	413.96	0.13	0.03	3.15	87.23%	11.02%	9.98%
Cyanidation Tails	-	997	g				0.040	0.50	57	12.77%	88.98%	90.02%
Calculated Feed, gm/t		997	g				0.313	0.56	63	100.00%	100.00%	100.00%
Head Assay, gm/t		1000										

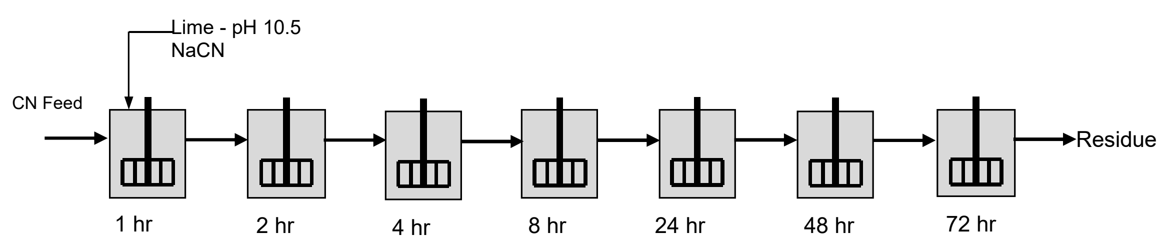
Mass of Sample	1000
Volume of Water	2000
Pulp Density	33

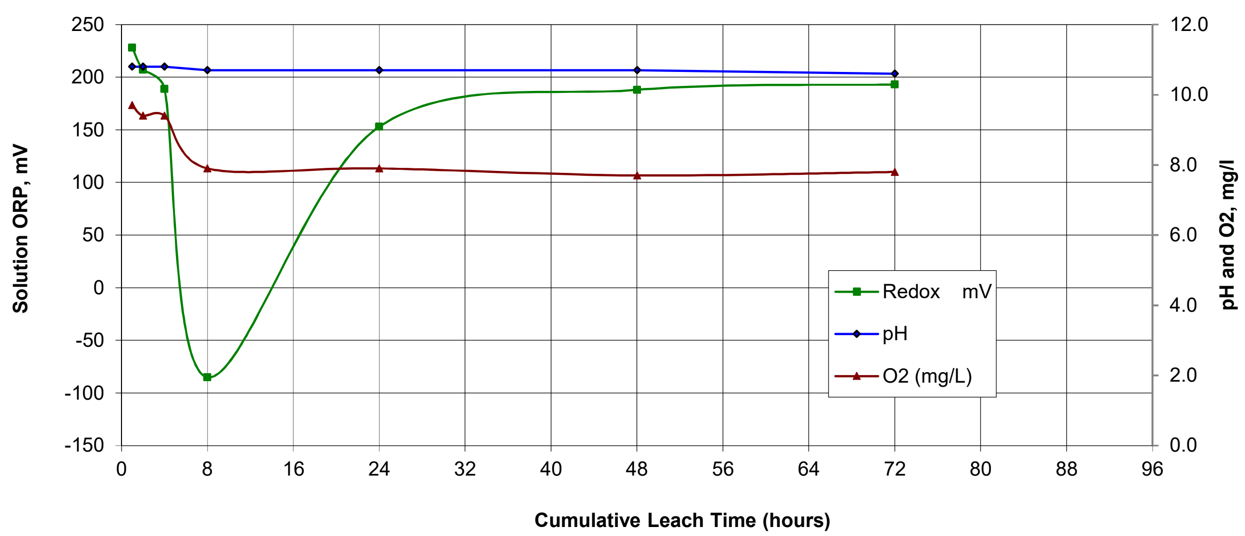
NaCN Consumption	0.27	kg/tonne
Lime Consumption	0.80	kg/tonne

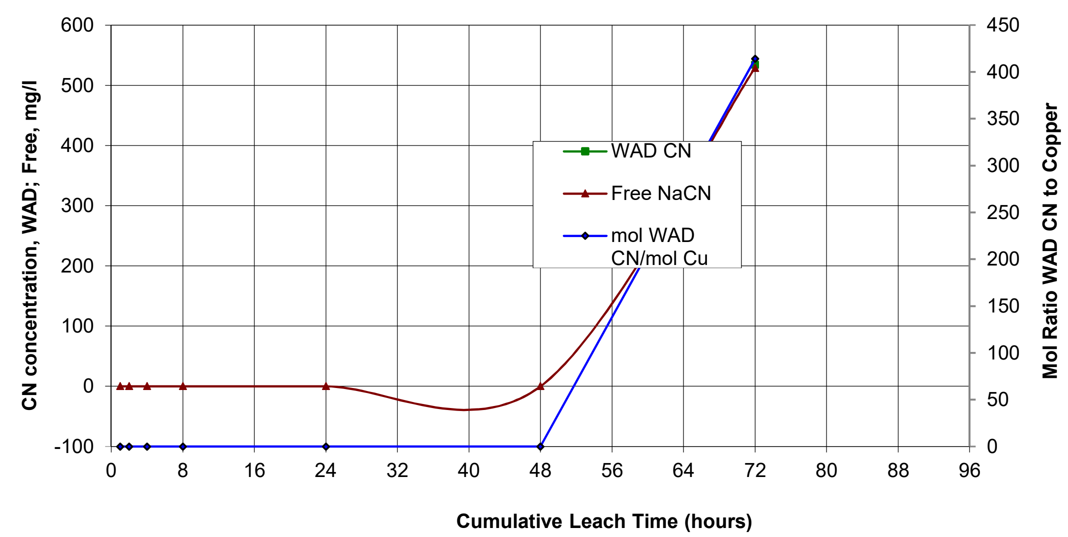
Cyanide Leach Kinetic Curves



Flowsheet Schematic







Test No: BL0295-14
Date: 21-Sep-18
Test Type: Rougher Test.
Test Objective: Evaluate Rougher of Bulk Gravity Tailings.
Sample: 2 kg of Test 12 Knelson Tails
Nominal Sizing: as is

Stage	Reagents - g/tonne			Time Minutes	Electrochemistry	
	PAX		MIBC		pH	Eh-mV
Primary Grind					7.7	245
Rougher 1	20			2	7.8	200
Rougher 2	20			2	7.8	200
Rougher 3	20			2	7.8	199
Rougher 4	20			2	7.8	195

Primary Grind	
Mill	Mild Steel Mill
Media	20kg Mild Steel Rods
Water Addn:	1500ml

Flotation Information	Rougher
Flotation Device:	D12
Cell Volume - Litres:	4.4
Impellar Speed - rpm:	800
Flotation Gas:	Air
Water Type:	Kamloops Tap

BL0295-14 Test 12 Knelson Tails

Metallurgical Balance

Product	Weight %	Weight grams	Assay - percent or g/t Au	Assay - percent or g/t Ag	Distribution - percent Au	Distribution - percent Ag
Ro Con 1	1.9	36.6	9.59	3.0	51.8	10.2
Ro Con 2	1.0	20.2	2.92	0.5	8.7	0.9
Ro Con 3	0.8	15.6	1.81	0.5	4.2	0.7
Ro Con 4	0.8	15.2	0.93	0.5	2.1	0.7
Rougher Tail	95.5	1872.9	0.12	0.5	33.3	87.4
Recalc. Feed	100.0	1960.5	0.35	0.5	100	100
Measured Feed			0.30	-		

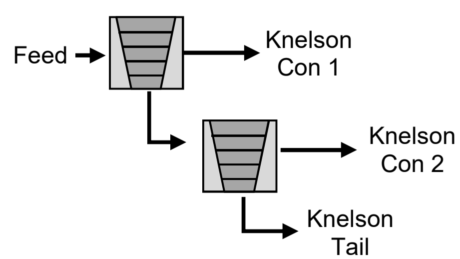
BL0295-14 Test 12 Knelson Tails

Cumulative Balance

Product	Weight %	Weight grams	Assay - percent or g/t Au	Assay - percent or g/t Ag	Distribution - percent Au	Distribution - percent Ag
Product 1	1.9	36.6	9.59	3.0	51.8	10.2
Products 1 to 2	2.9	56.8	7.22	2.1	60.5	11.2
Products 1 to 3	3.7	72.4	6.05	1.8	64.6	11.9
Products 1 to 4	4.5	87.6	5.17	1.5	66.7	12.6
Product 5	95.5	1872.9	0.12	0.5	33.3	87.4
Recalc. Feed	100.0	1960.5	0.35	0.5	100	100

Test No: BL0295-15
Date: 1-Oct-18
Test Type: Knelson Separation Test.
Test Objective: Evaluate additional gravity recoverable gold.
Sample: 2 kg of Test 12 Knelson Tails
Nominal Sizing: as is

Flowsheet Schematic



Stage	Inlet Pressure	G-Force	Flowrate Lpm	Time Minutes
Grind				
KN Separation 1	2.5	120	4	5
KN Separation 2	2.5	120	4	5

BL0295-15 Test 12 Knelson Tails

Metallurgical Balance

Product	Weight		Assay - g/t	Distribution - percent
	%	grams	Au	Au
Knelson Con 1	2.8	55.9	3.04	24.9
Knelson Con 2	2.9	57.8	1.11	9.4
Knelson Tail	94.2	1855.5	0.24	65.7
Recalc. Feed	100.0	1969.2	0.35	100
Measured Feed			0.30	

* Note - Ag assays estimated, assay values were <1 ppm.

BL0295-15 Test 12 Knelson Tails

Cumulative Balance

Product	Weight		Assay - g/t	Distribution - percent
	%	grams	Au	Au
Product 1	2.8	55.9	3.04	24.9
Products 1 to 2	5.8	113.7	2.06	34.3
Product 5	94.2	1855.5	0.24	65.7
Recalc. Feed	100.0	1969.2	0.35	100

Overall Cumulative Balance with Test 12 and 15

Cumulative Balance

Product	Weight		Assay - g/t	Distribution - percent
	%	grams	Au	Au
KC T12	0.2	59.0	278	62.2
KC T12 + KC1 T15	3.0	872.4	21.6	71.5
KC T12 + KC1 T15 + KC2 T15	6.0	1714.9	11.6	75.1
Knelson Tail T15	94.0	27394.2	0.24	24.9
Recalc Feed	100.0	29109.0	0.91	100

Test No:	BL295-16		Au, gm/t	Ag, gm/t	Cu, %
Date:	01-Oct-18	Head Assay, Ave:	0.300	1.00	0.01%
Test Type:	Standard bottle roll procedure.	Head Assay 1:	0.30	1	0.01%
Test Objective:	Repeat Test 13 with oxygen	Head Assay 2:	0.30	1	0.01%
Sample:	Test 12 Knelson Tails	Head Assay 3:	0.30	1	0.01%
Grind:	150µm as received				
Target Maintenance NaCN:	1000 ppm	Total Mass, Initial:		grams including bottle	

	Au, gm/t	Ag, gm/t	Cu, %
Tails Assay, Ave:	0.020	0.5	0.00%
Tail Assay 1:	0.020	0.5	0.004%
Tail Assay 2:	0.020	0.5	0.004%

* Ag estimated, assay value <ppm.

Sample Size	20 ml
Calculated Head	

	Au	Ag	Cu
Head, mg:	0.299	0.998	69.846
Tails, mg:	0.020	0.499	43.504
Calculated Head, mg:	0.376	0.562	50

Cyanidation Leaching @ pH 10.5 (lime), 1000ppm NaCN, O2 sparged,

Parameter	Time	Added (g)		WAD CN	pH		Dissolved	Redox	Temp
	Cum	NaCN	Lime	mg/l CN	Measured	Adjusted	O_2 (mg/L)	mV	Deg. C
Natural	-	-	-		8.4	-	7.7		20.0
Leach 0	0	2.00	0.57		10.5	10.5	7.7		20.0
Leach 1	1	0.22	-		10.6	-	>20		20.0
Leach 2	2	0.00	-		10.9	-	>20		19.0
Leach 3	4	0.00	-		10.7	-	>20		20.0
Leach 4	8	0.00	-		10.7	-	>20		20.0
Leach 5	24	0.16	-		10.5	-	>20		20.0
Leach 6	48	0.00	-		10.8	-	>20		21.0
Leach 7	72	0.00	-	547	10.5	-	>20		20.0
Total	72	2.38	0.57		-	-	-	-	

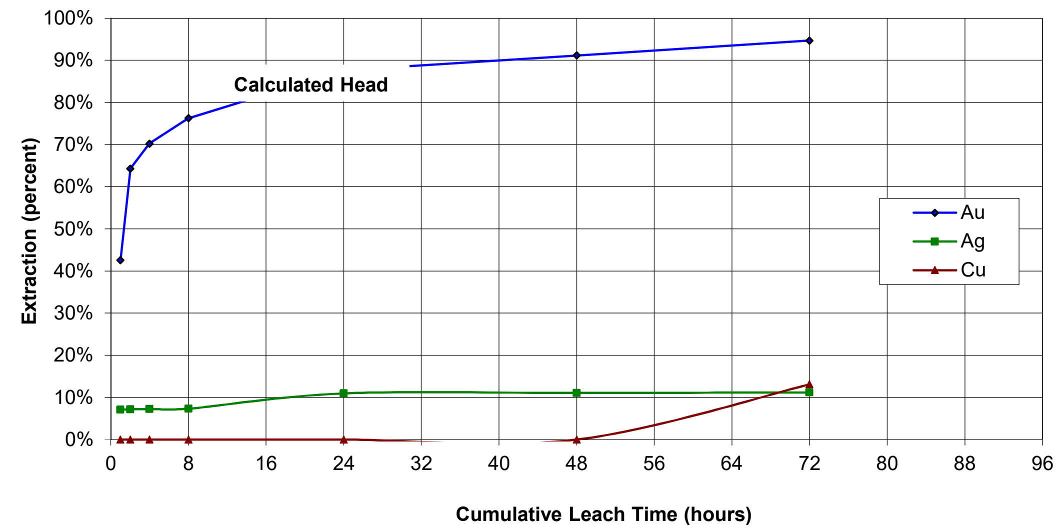
Cumulative Metallurgical Balance

Product	Cumulative	Vol or	Units	Free CN	Free NaCN	mol WAD/	Assay - g/tonne (ppm)			Distribution - percent		
	Time - Hrs	Mass		mg/l, calc'd	mg/l	mol Cu	Au	Ag	Cu	Au	Ag	Cu
Cyanide Liquor (1 hr)	1	2000	mL	0.0	0	0.00	0.08	0.02		42.57%	7.12%	0.00%
Cyanide Liquor (2 hr)	2	2000	mL	0.0	0	0.00	0.12	0.02		64.28%	7.19%	0.00%
Cyanide Liquor (4 hr)	4	2000	mL	0.0	0	0.00	0.13	0.02		70.24%	7.26%	0.00%
Cyanide Liquor (8 hr)	8	2000	mL	0.0	0	0.00	0.14	0.02		76.25%	7.33%	0.00%
Cyanide Liquor (24hr)	24	2000	mL	0.0	0	0.00	0.16	0.03		87.64%	10.97%	0.00%
Cyanide Liquor (48hr)	48	2000	mL	0.0	0	0.00	0.17	0.03		91.15%	11.07%	0.00%
Cyanide Liquor (72hr)	72	2000	mL	541.6	1,020	407.17	0.17	0.03	3.28	94.69%	11.18%	13.10%
Cyanidation Tails	-	998	g				0.020	0.50	44	5.31%	88.82%	86.90%
Calculated Feed, gm/t		998	g				0.377	0.56	50	100.00%	100.00%	100.00%
Head Assay, gm/t		1000										

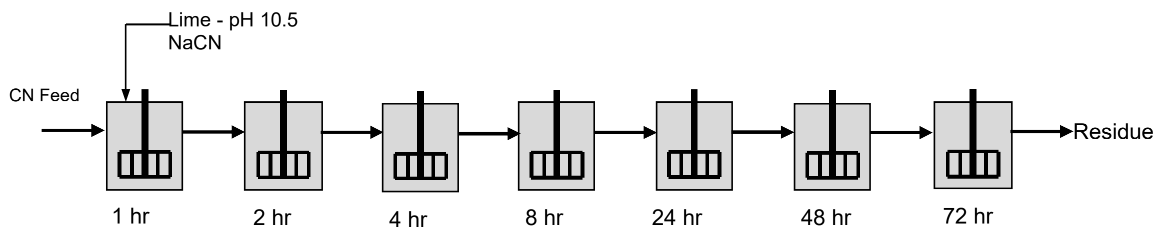
Mass of Sample	1000
Volume of Water	2000
Pulp Density	33

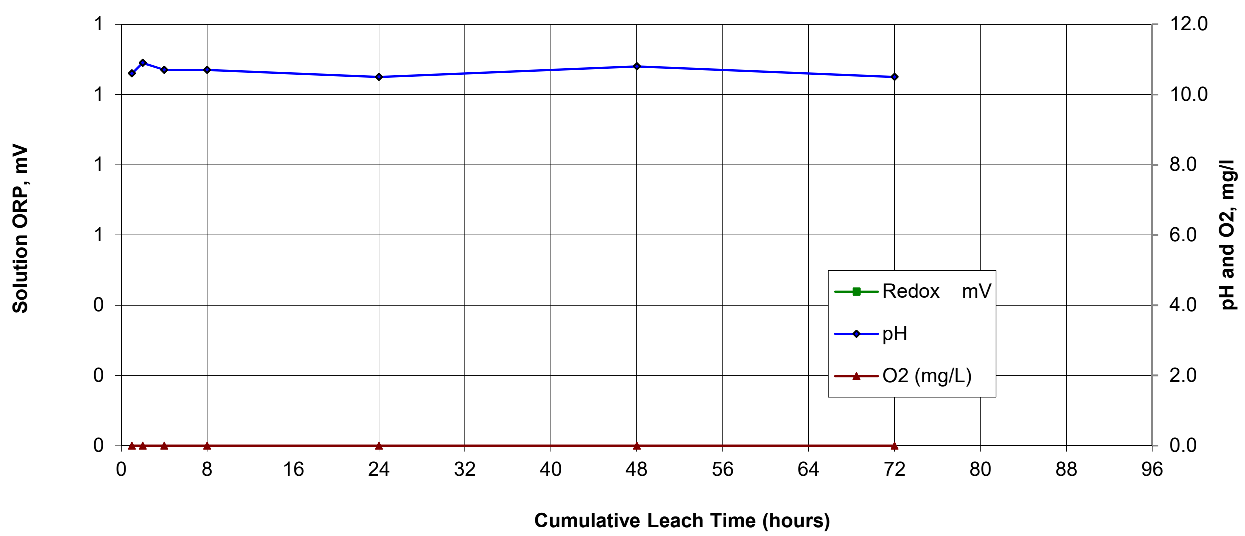
NaCN Consumption	0.34	kg/tonne
Lime Consumption	0.57	kg/tonne

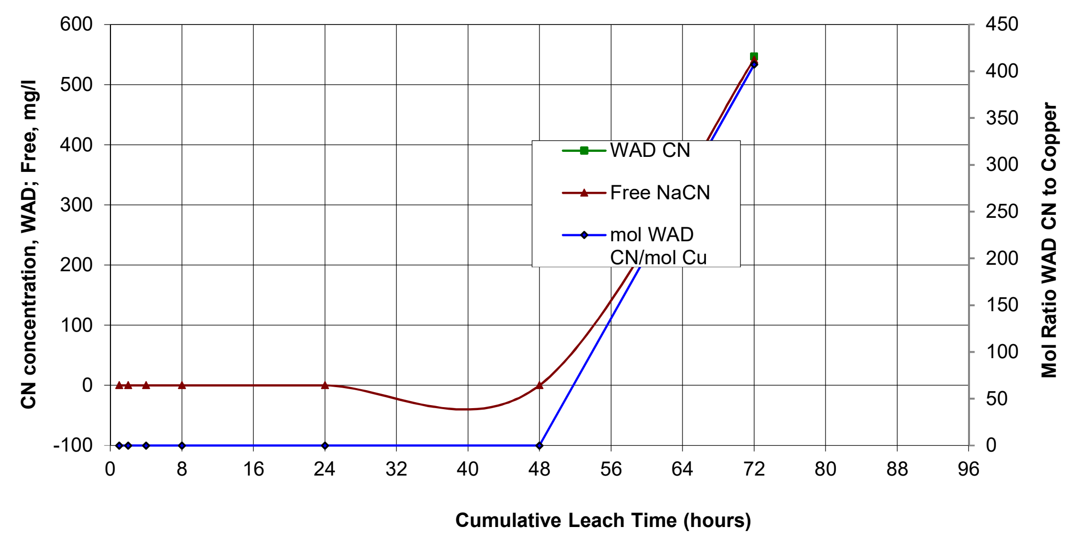
Cyanide Leach Kinetic Curves



Flowsheet Schematic







Test No:	BL295-17		Au, gm/t	Ag, gm/t	Cu, %
Date:	01-Oct-18	Head Assay, Ave:	0.300	1.00	0.01%
Test Type:	Standard bottle roll procedure.	Head Assay 1:	0.30	1	0.01%
Test Objective:	Repeat Test 16 at 750ppm NaCN	Head Assay 2:	0.30	1	0.01%
Sample:	Test 12 Knelson Tails	Head Assay 3:	0.30	1	0.01%
Grind:	150µm as received				
Target Maintenance NaCN:	750 ppm	Total Mass, Initial:		grams including bottle	

	Au, gm/t	Ag, gm/t	Cu, %
Tails Assay, Ave:	0.020	0.5	0.01%
Tail Assay 1:	0.020	0.5	0.005%
Tail Assay 2:	0.020	0.5	0.005%

* Ag estimated, assay value <ppm.

Sample Size 20 ml

Calculated Head

	Au	Ag	Cu
Head, mg:	0.297	0.989	69.251
Tails, mg:	0.020	0.495	51.048
Calculated Head, mg:	0.396	0.557	57

Cyanidation Leaching @ pH 10.5 (lime), 750ppm NaCN, O2 sparged,

Parameter	Time	Added (g)		WAD CN	pH		Dissolved	Redox	Temp
	Cum	NaCN	Lime	mg/l CN	Measured	Adjusted	O_2 (mg/L)	mV	Deg. C
Natural	-	-	-		8.3	-	7.9		20.0
Leach 0	0	1.50	0.55		10.5	10.5	7.9		20.0
Leach 1	1	0.18	-		10.8	-	>20		20.0
Leach 2	2	0.00	-		10.9	-	>20		19.0
Leach 3	4	0.00	-		10.7	-	>20		20.0
Leach 4	8	0.00	-		10.8	-	>20		20.0
Leach 5	24	0.20	-		10.7	-	>20		20.0
Leach 6	48	0.00	-		10.8	-	>20		21.0
Leach 7	72	0.00	-	449	10.5	-	>20		20.0
Total	72	1.88	0.55		-	-	-	-	

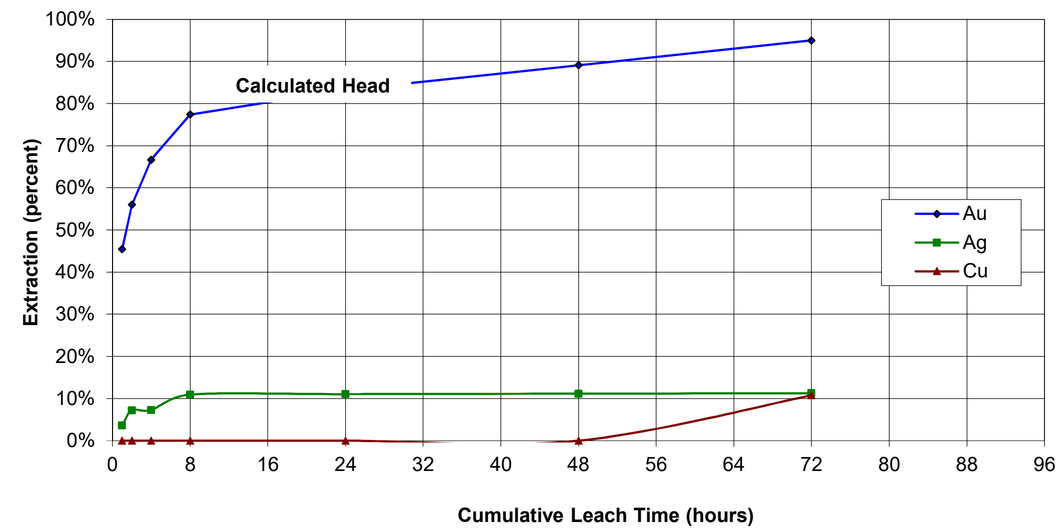
Cumulative Metallurgical Balance

Product	Cumulative	Vol or	Units	Free CN	Free NaCN	mol WAD/	Assay - g/tonne (ppm)			Distribution - percent		
	Time - Hrs	Mass		mg/l, calc'd	mg/l	mol Cu	Au	Ag	Cu	Au	Ag	Cu
Cyanide Liquor (1 hr)	1	2000	mL	0.0	0	0.00	0.09	0.01		45.46%	3.59%	0.00%
Cyanide Liquor (2 hr)	2	2000	mL	0.0	0	0.00	0.11	0.02		56.01%	7.21%	0.00%
Cyanide Liquor (4 hr)	4	2000	mL	0.0	0	0.00	0.13	0.02		66.67%	7.28%	0.00%
Cyanide Liquor (8 hr)	8	2000	mL	0.0	0	0.00	0.15	0.03		77.43%	10.94%	0.00%
Cyanide Liquor (24hr)	24	2000	mL	0.0	0	0.00	0.16	0.03		83.24%	11.05%	0.00%
Cyanide Liquor (48hr)	48	2000	mL	0.0	0	0.00	0.17	0.03		89.09%	11.16%	0.00%
Cyanide Liquor (72hr)	72	2000	mL	444.2	837	356.11	0.18	0.03	3.08	95.00%	11.27%	10.77%
Cyanidation Tails	-	989	g				0.020	0.50	52	5.00%	88.73%	89.23%
Calculated Feed, gm/t		989	g				0.400	0.56	58	100.00%	100.00%	100.00%
Head Assay, gm/t		1000										

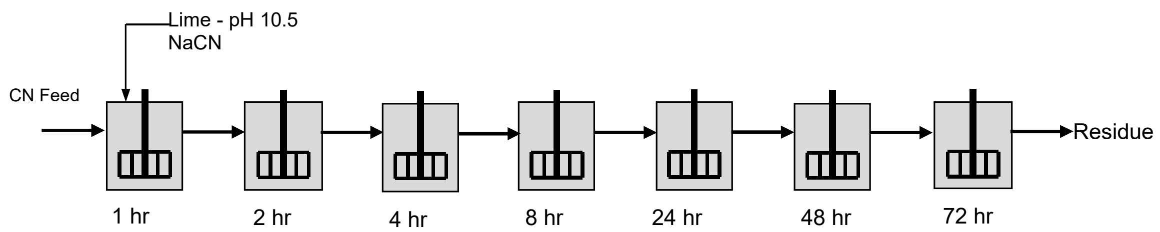
Mass of Sample	1000
Volume of Water	2000
Pulp Density	33

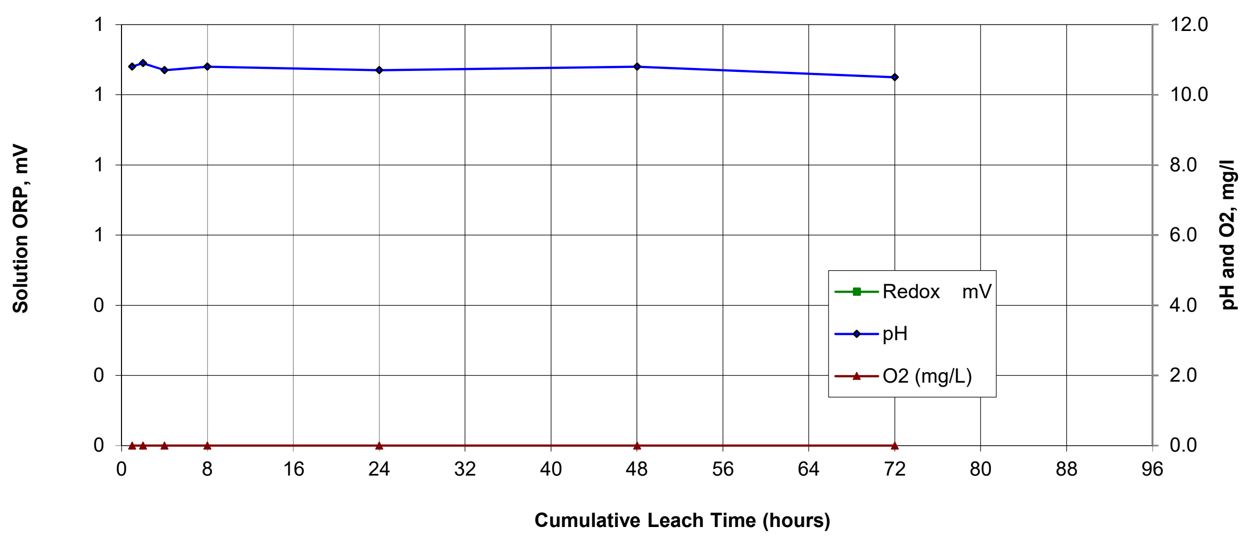
NaCN Consumption	0.21	kg/tonne
Lime Consumption	0.55	kg/tonne

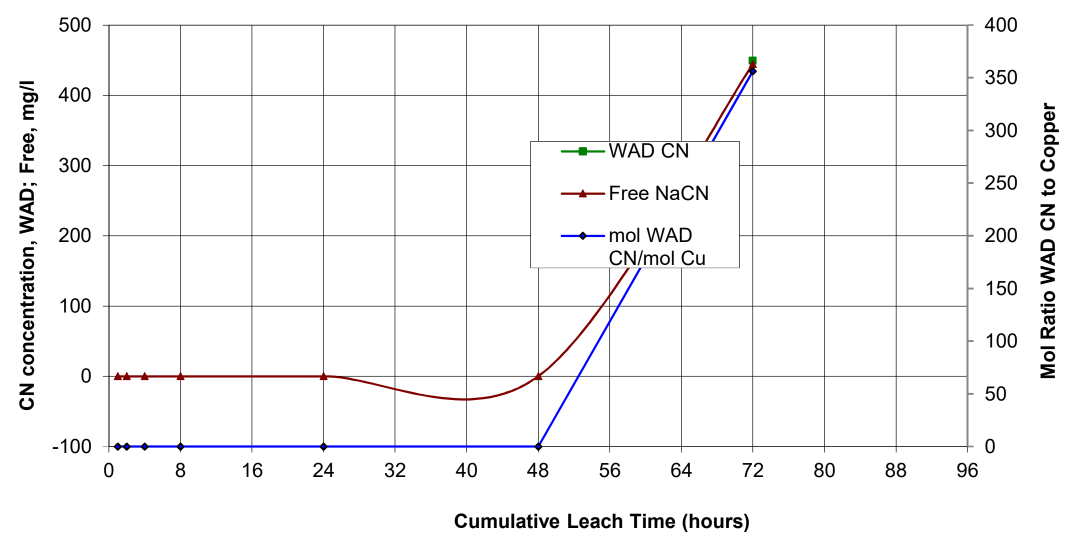
Cyanide Leach Kinetic Curves



Flowsheet Schematic







Test No:	BL295-18		Au, gm/t	Ag, gm/t	Cu, %
Date:	01-Oct-18	Head Assay, Ave:	0.300	1.00	0.01%
Test Type:	Standard bottle roll procedure.	Head Assay 1:	0.30	1	0.01%
Test Objective:	Repeat Test 17 at 500ppm NaCN	Head Assay 2:	0.30	1	0.01%
Sample:	Test 12 Knelson Tails	Head Assay 3:	0.30	1	0.01%
Grind:	150µm as received				
Target Maintenance NaCN:	500 ppm	Total Mass, Initial:		grams including bottle	

	Au, gm/t	Ag, gm/t	Cu, %
Tails Assay, Ave:	0.020	0.5	0.01%
Tail Assay 1:	0.020	0.5	0.005%
Tail Assay 2:	0.020	0.5	0.006%

* Ag estimated, assay value <ppm.

Sample Size: 20 ml

Calculated Head:

	Au	Ag	Cu
Head, mg:	0.298	0.994	69.587
Tails, mg:	0.020	0.497	52.290
Calculated Head, mg:	0.374	0.560	57

Cyanidation Leaching @ pH 10.5 (lime), 500ppm NaCN, O2 sparged,

Parameter	Time	Added (g)		WAD CN	pH		Dissolved	Redox	Temp
	Cum	NaCN	Lime	mg/l CN	Measured	Adjusted	O_2 (mg/L)	mV	Deg. C
Natural	-	-	-		8.4	-	8.0		20.0
Leach 0	0	1.00	0.63		10.5	10.5	8.0		20.0
Leach 1	1	0.00	-		10.7	-	>20		20.0
Leach 2	2	0.00	-		10.8	-	>20		19.0
Leach 3	4	0.00	-		10.6	-	>20		20.0
Leach 4	8	0.14	-		10.7	-	>20		20.0
Leach 5	24	0.00	-		10.6	-	>20		20.0
Leach 6	48	0.00	-		10.7	-	>20		21.0
Leach 7	72	0.00	-	301	10.3	-	>20		20.0
Total	72	1.14	0.63		-	-	-	-	

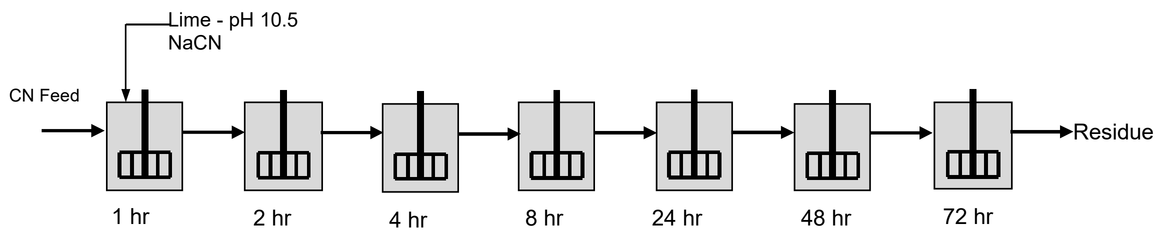
Cumulative Metallurgical Balance

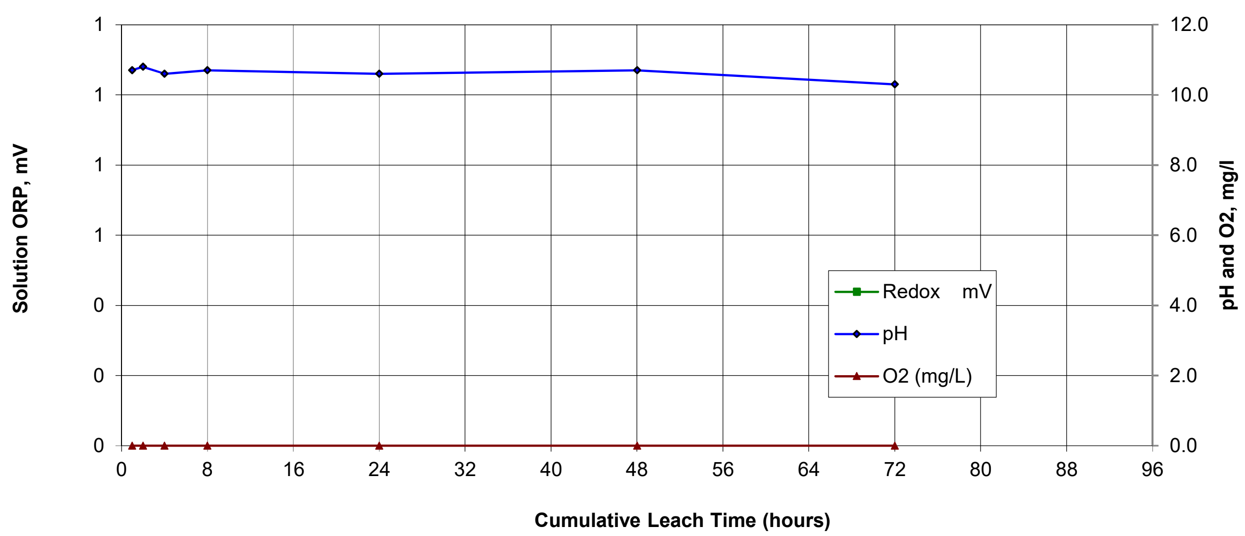
Product	Cumulative	Vol or	Units	Free CN	Free NaCN	mol WAD/	Assay - g/tonne (ppm)			Distribution - percent		
	Time - Hrs	Mass		mg/l, calc'd	mg/l	mol Cu	Au	Ag	Cu	Au	Ag	Cu
Cyanide Liquor (1 hr)	1	2000	mL	0.0	0	0.00	0.04	0.01		21.41%	3.57%	0.00%
Cyanide Liquor (2 hr)	2	2000	mL	0.0	0	0.00	0.08	0.02		43.03%	7.18%	0.00%
Cyanide Liquor (4 hr)	4	2000	mL	0.0	0	0.00	0.12	0.03		64.87%	10.82%	0.00%
Cyanide Liquor (8 hr)	8	2000	mL	0.0	0	0.00	0.14	0.03		76.21%	10.93%	0.00%
Cyanide Liquor (24hr)	24	2000	mL	0.0	0	0.00	0.15	0.03		82.32%	11.03%	0.00%
Cyanide Liquor (48hr)	48	2000	mL	0.0	0	0.00	0.16	0.03		88.47%	11.14%	0.00%
Cyanide Liquor (72hr)	72	2000	mL	296.5	558	285.58	0.17	0.03	2.57	94.68%	11.25%	8.95%
Cyanidation Tails	-	994	g				0.020	0.50	53	5.32%	88.75%	91.05%
Calculated Feed, gm/t		994	g				0.376	0.56	58	100.00%	100.00%	100.00%
Head Assay, gm/t		1000										

Mass of Sample	1000
Volume of Water	2000
Pulp Density	33

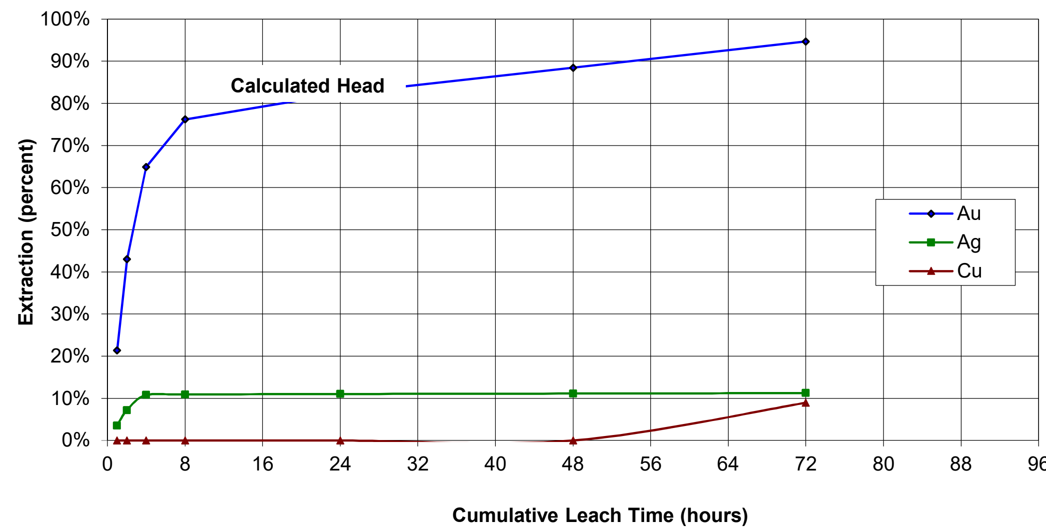
NaCN Consumption	0.02	kg/tonne
Lime Consumption	0.63	kg/tonne

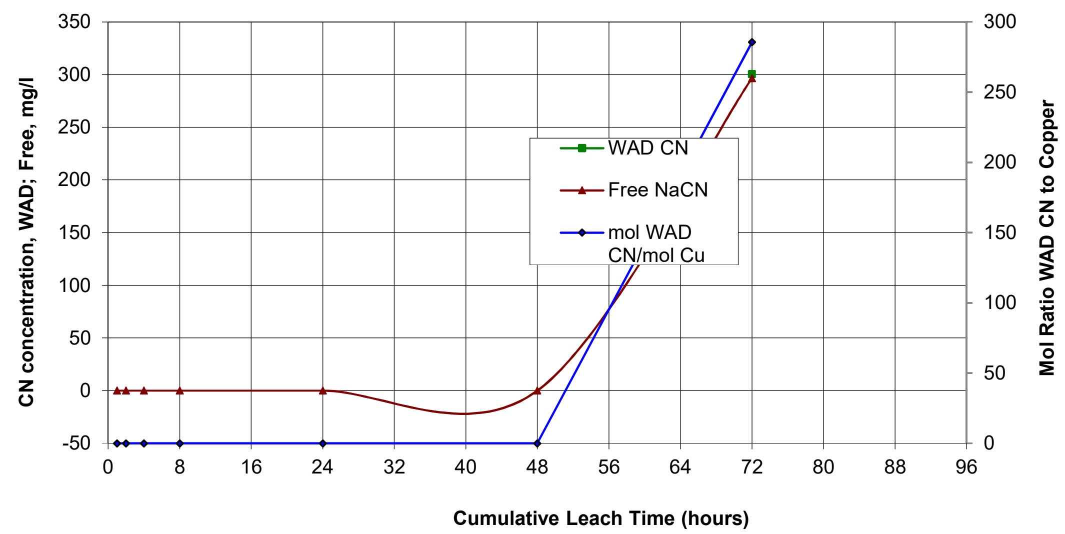
Flowsheet Schematic





Cyanide Leach Kinetic Curves





Test No:	BL295-19		Au, gm/t	Ag, gm/t	Cu, %	
Date:	01-Oct-18	Head Assay, Ave:	0.300	1.00	0.01%	
Test Type:	Standard bottle roll procedure.	Head Assay 1:	0.30	1	0.01%	
Test Objective:	Repeat Test 18 at 250ppm NaCN	Head Assay 2:	0.30	1	0.01%	
Sample:	Test 12 Knelson Tails	Head Assay 3:	0.30	1	0.01%	
Grind:	150µm as received					
Target Maintenance NaCN:	250 ppm	Total Mass, Initial:		grams including bottle		

	Au, gm/t	Ag, gm/t	Cu, %
Tails Assay, Ave:	0.020	0.5	0.00%
Tail Assay 1:	0.020	0.5	0.005%
Tail Assay 2:	0.020	0.5	0.005%

* Ag estimated, assay value <ppm.

Sample Size 20 ml

Calculated Head

	Au	Ag	Cu
Head, mg:	0.300	0.998	69.888
Tails, mg:	0.020	0.499	48.922
Calculated Head, mg:	0.312	0.561	54

Cyanidation Leaching @ pH 10.5 (lime), 250ppm NaCN, O2 sparged,

Parameter	Time	Added (g)		WAD CN	pH		Dissolved	Redox	Temp
	Cum	NaCN	Lime	mg/l CN	Measured	Adjusted	O_2 (mg/L)	mV	Deg. C
Natural	-	-	-		8.3	-	8.0		20.0
Leach 0	0	0.50	0.60		10.5	10.5	8.0		20.0
Leach 1	1	0.00	-		10.7	-	>20		20.0
Leach 2	2	0.00	-		10.8	-	>20		19.0
Leach 3	4	0.00	-		10.6	-	>20		20.0
Leach 4	8	0.00	-		10.6	-	>20		20.0
Leach 5	24	0.06	-		10.4	-	>20		20.0
Leach 6	48	0.00	-		11.0	-	>20		21.0
Leach 7	72	0.00	-	143	10.0	-	>20		20.0
Total	72	0.56	0.60		-	-	-	-	

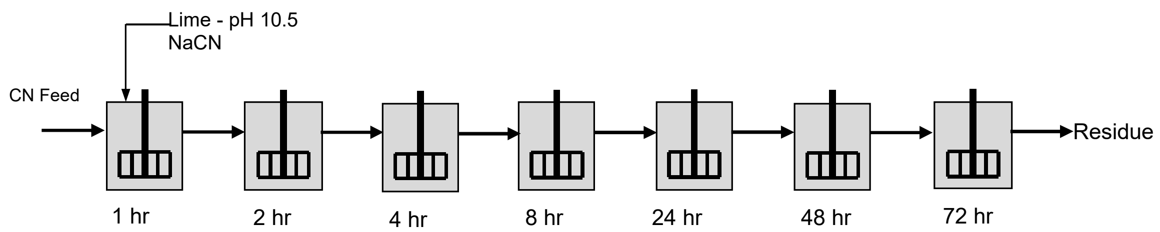
Mass of Sample	1000
Volume of Water	2000
Pulp Density	33

NaCN Consumption	0.04	kg/tonne
Lime Consumption	0.60	kg/tonne

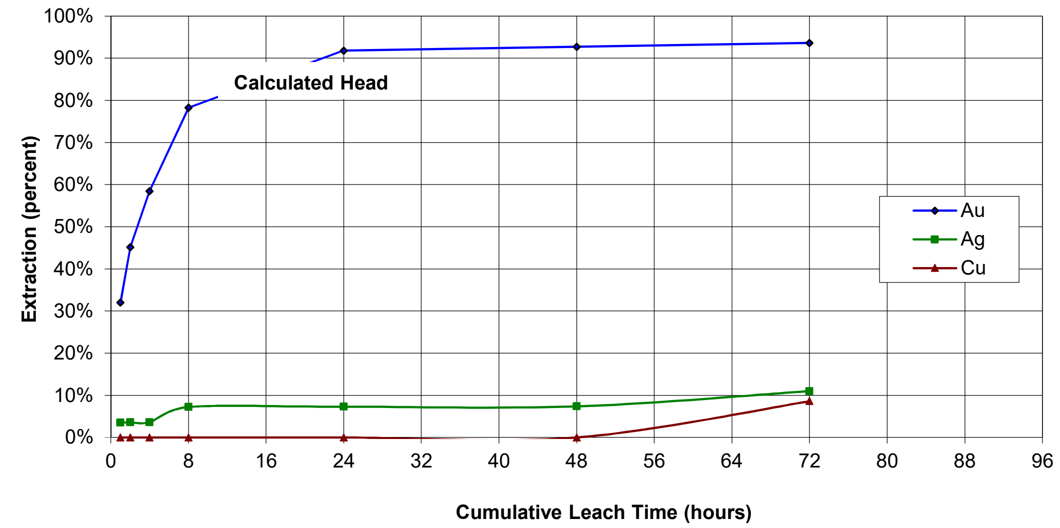
Cumulative Metallurgical Balance

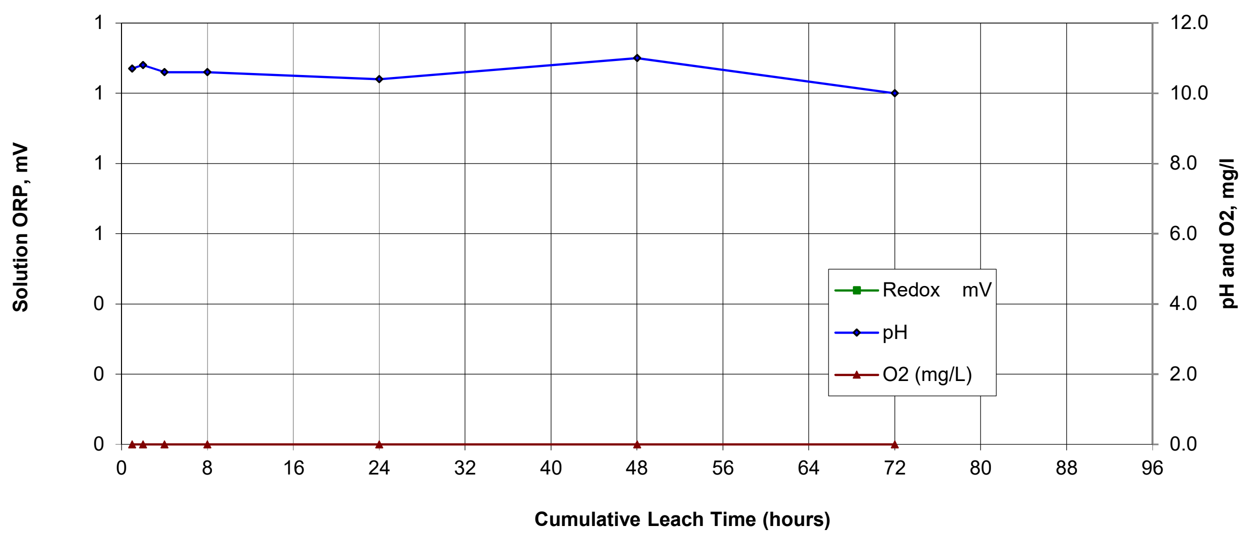
Product	Cumulative	Vol or	Units	Free CN	Free NaCN	mol WAD/	Assay - g/tonne (ppm)			Distribution - percent		
	Time - Hrs	Mass		mg/l, calc'd	mg/l	mol Cu	Au	Ag	Cu	Au	Ag	Cu
Cyanide Liquor (1 hr)	1	2000	mL	0.0	0	0.00	0.05	0.01		32.03%	3.57%	0.00%
Cyanide Liquor (2 hr)	2	2000	mL	0.0	0	0.00	0.07	0.01		45.17%	3.60%	0.00%
Cyanide Liquor (4 hr)	4	2000	mL	0.0	0	0.00	0.09	0.01		58.43%	3.64%	0.00%
Cyanide Liquor (8 hr)	8	2000	mL	0.0	0	0.00	0.12	0.02		78.23%	7.24%	0.00%
Cyanide Liquor (24hr)	24	2000	mL	0.0	0	0.00	0.14	0.02		91.81%	7.31%	0.00%
Cyanide Liquor (48hr)	48	2000	mL	0.0	0	0.00	0.14	0.02		92.71%	7.38%	0.00%
Cyanide Liquor (72hr)	72	2000	mL	139.1	262	151.36	0.14	0.03	2.30	93.60%	11.02%	8.59%
Cyanidation Tails	-	998	g				0.020	0.50	49	6.40%	88.98%	91.41%
Calculated Feed, gm/t		998	g				0.313	0.56	54	100.00%	100.00%	100.00%
Head Assay, gm/t		1000										

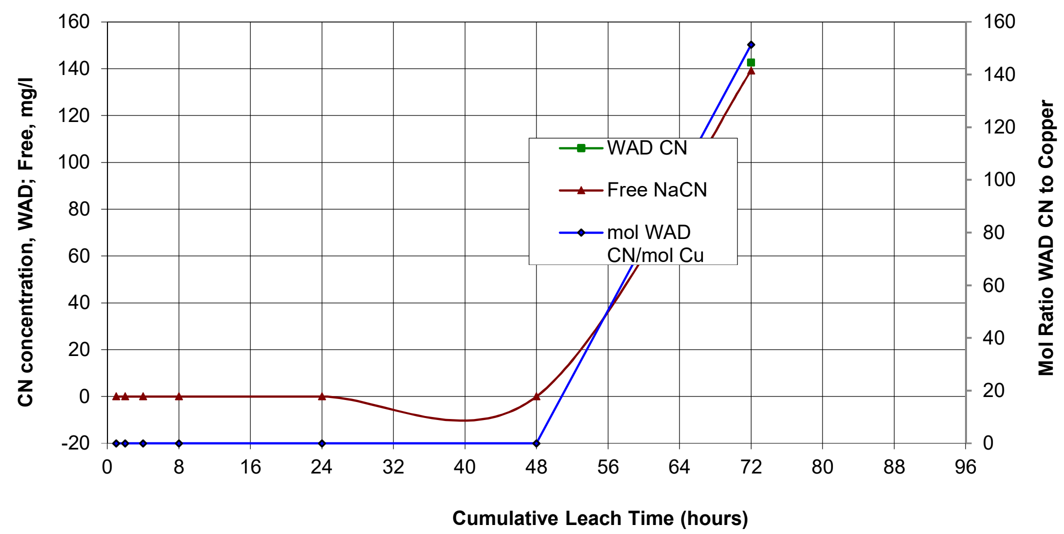
Flowsheet Schematic



Cyanide Leach Kinetic Curves







Test No:	BL295-20		Au, gm/t	Ag, gm/t	Cu, %	
Date:	01-Oct-18	Head Assay, Ave:	0.300	1.00	0.01%	
Test Type:	Standard bottle roll procedure.	Head Assay 1:	0.30	1	0.01%	
Test Objective:	Repeat Test 16 at 25% Solids	Head Assay 2:	0.30	1	0.01%	
Sample:	Test 12 Knelson Tails	Head Assay 3:	0.30	1	0.01%	
Grind:	150µm as received					
Target Maintenance NaCN:	1000 ppm	Total Mass, Initial:		grams including bottle		

Cyanidation Leaching @ pH 10.5 (lime), 1000ppm NaCN, O2 sparged,

	Au, gm/t	Ag, gm/t	Cu, %
Tails Assay, Ave:	0.030	0.5	0.01%
Tail Assay 1:	0.030	0.5	0.005%
Tail Assay 2:	0.030	0.5	0.005%

* Ag estimated, assay value <ppm.

Sample Size 20 ml

Calculated Head

	Au	Ag	Cu
Head, mg:	0.300	0.999	69.895
Tails, mg:	0.030	0.499	51.123
Calculated Head, mg:	0.400	0.592	59

Parameter	Time	Added (g)		WAD CN	pH		Dissolved	Redox	Temp
	Cum	NaCN	Lime	mg/l CN	Measured	Adjusted	O_2 (mg/L)	mV	Deg. C
Natural	-	-	-		8.3	-	8.1		20.0
Leach 0	0	3.00	0.85		10.5	10.5	8.1		20.0
Leach 1	1	0.21	-		11.0	-	>20		20.0
Leach 2	2	0.45	-		11.0	-	>20		19.0
Leach 3	4	0.00	-		10.9	-	>20		20.0
Leach 4	8	0.00	-		11.0	-	>20		20.0
Leach 5	24	0.00	-		10.9	-	>20		20.0
Leach 6	48	0.00	-		10.5	-	>20		21.0
Leach 7	72	0.00	-	576	10.8	-	>20		20.0
Total	72	3.66	0.85		-	-	-	-	

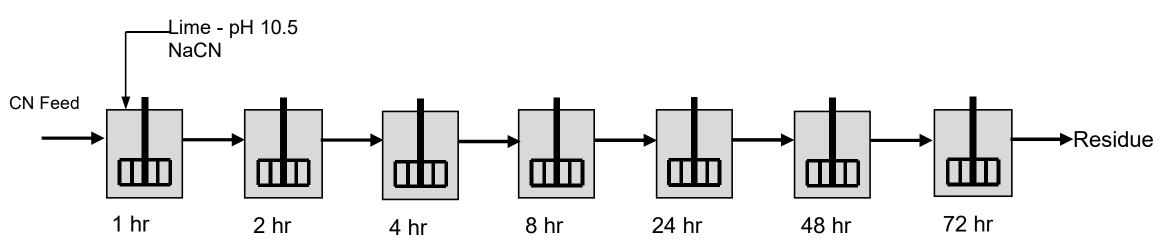
Mass of Sample	1000
Volume of Water	3000
Pulp Density	25

NaCN Consumption	0.43	kg/tonne
Lime Consumption	0.85	kg/tonne

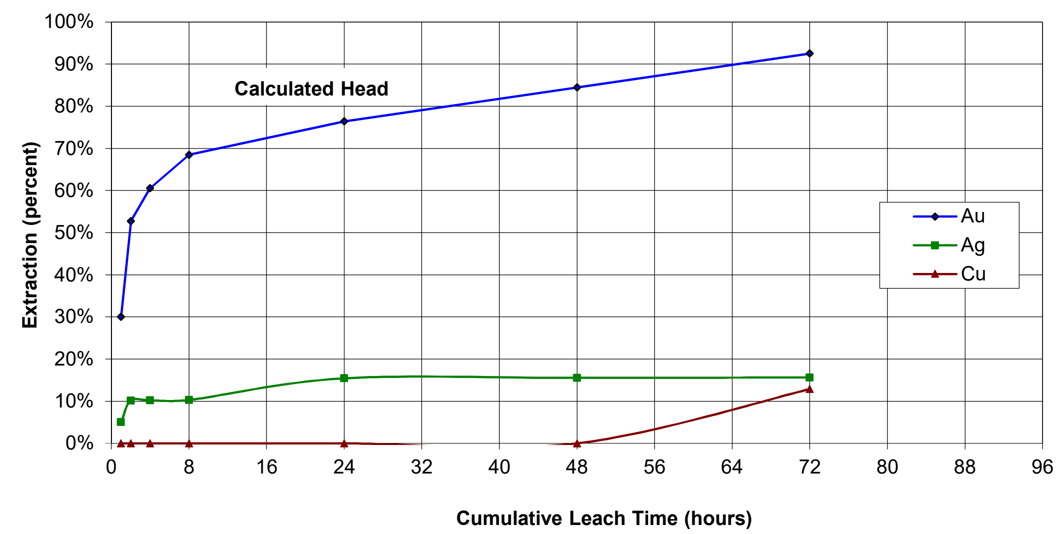
Cumulative Metallurgical Balance

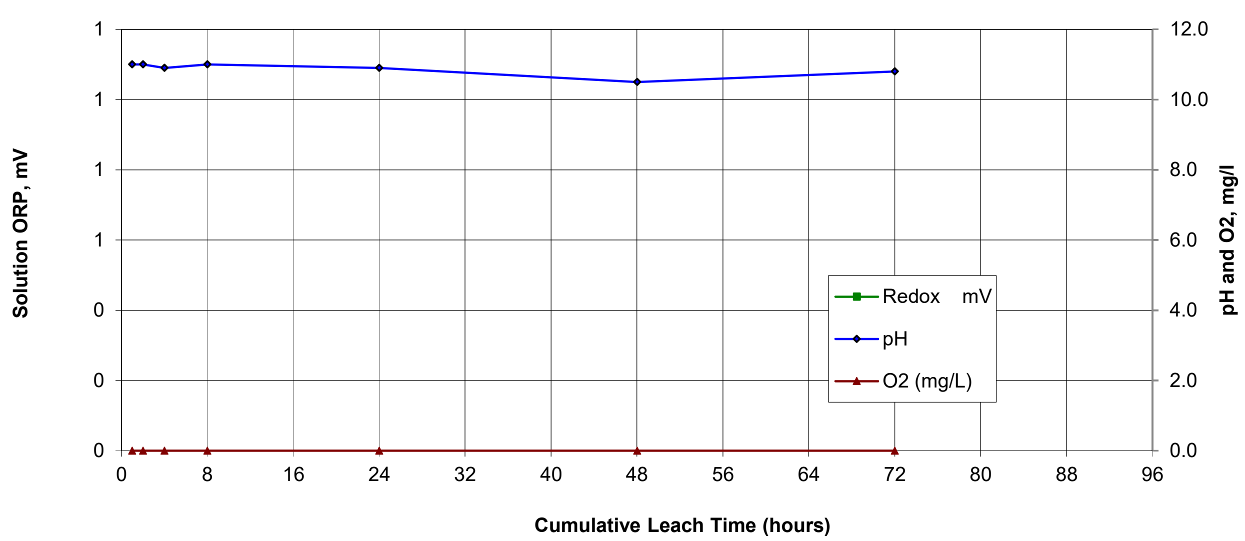
Product	Cumulative	Vol or	Units	Free CN	Free NaCN	mol WAD/	Assay - g/tonne (ppm)			Distribution - percent		
	Time - Hrs	Mass		mg/l, calc'd	mg/l	mol Cu	Au	Ag	Cu	Au	Ag	Cu
Cyanide Liquor (1 hr)	1	3000	mL	0.0	0	0.00	0.04	0.01		30.02%	5.07%	0.00%
Cyanide Liquor (2 hr)	2	3000	mL	0.0	0	0.00	0.07	0.02		52.73%	10.17%	0.00%
Cyanide Liquor (4 hr)	4	3000	mL	0.0	0	0.00	0.08	0.02		60.59%	10.24%	0.00%
Cyanide Liquor (8 hr)	8	3000	mL	0.0	0	0.00	0.09	0.02		68.49%	10.31%	0.00%
Cyanide Liquor (24hr)	24	3000	mL	0.0	0	0.00	0.10	0.03		76.45%	15.44%	0.00%
Cyanide Liquor (48hr)	48	3000	mL	0.0	0	0.00	0.11	0.03		84.45%	15.54%	0.00%
Cyanide Liquor (72hr)	72	3000	mL	571.5	1,076	557.79	0.12	0.03	2.52	92.51%	15.65%	12.88%
Cyanidation Tails	-	999	g				0.030	0.50	51	7.49%	84.35%	87.12%
Calculated Feed, gm/t		999	g				0.400	0.59	59	100.00%	100.00%	100.00%
Head Assay, gm/t		1000										

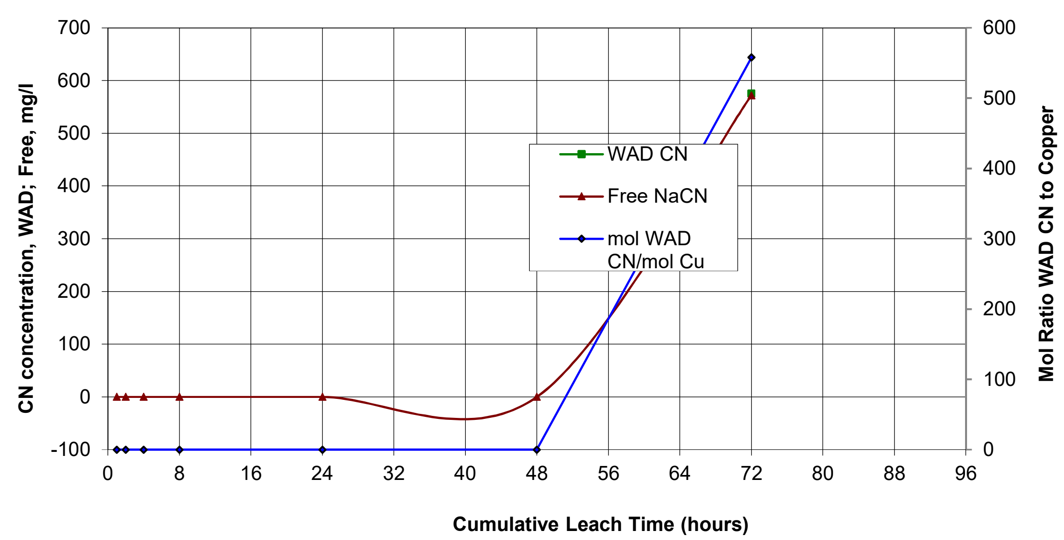
Flowsheet Schematic



Cyanide Leach Kinetic Curves







Test No:	BL295-21			Au, gm/t	Ag, gm/t	Cu, %
Date:	01-Oct-18		Head Assay, Ave:	0.300	1.00	0.01%
Test Type:	Standard bottle roll procedure.		Head Assay 1:	0.30	1	0.01%
Test Objective:	Repeat Test 20 at 40% Solids		Head Assay 2:	0.30	1	0.01%
Sample:	Test 12 Knelson Tails		Head Assay 3:	0.30	1	0.01%
Grind:	150µm as received					
Target Maintenance NaCN:	1000	ppm	Total Mass, Initial:		grams including bottle	

	Au, gm/t	Ag, gm/t	Cu, %
Tails Assay, Ave:	0.015	0.5	0.01%
Tail Assay 1:	0.010	0.5	0.005%
Tail Assay 2:	0.020	0.5	0.005%

* Ag estimated, assay value <ppm.

Sample Size 20 ml

Calculated Head

	Au	Ag	Cu
Head, mg:	0.299	0.996	69.713
Tails, mg:	0.015	0.498	52.783
Calculated Head, mg:	0.316	0.577	59

Gravity Cyanidation Leaching @ pH 10.5 (lime), 1000ppm NaCN, O2 sparged,

Parameter	Time	Added (g)		WAD CN	pH		Dissolved	Redox	Temp
	Cum	NaCN	Lime	mg/l CN	Measured	Adjusted	O_2 (mg/L)	mV	Deg. C
Natural	-	-	-		8.4	-	7.9		20.0
Leach 0	0	1.50	0.42		10.5	10.5	7.9		20.0
Leach 1	1	0.17	-		10.8	-	>20		20.0
Leach 2	2	0.00	-		10.7	-	>20		19.0
Leach 3	4	0.00	-		10.7	-	>20		20.0
Leach 4	8	0.07	-		10.7	-	>20		20.0
Leach 5	24	0.00	-		10.6	-	>20		20.0
Leach 6	48	0.63	-		10.7	-	>20		21.0
Leach 7	72	0.00	-	692	10.5	-	>20		20.0
Total	72	2.37	0.42		-	-	-	-	

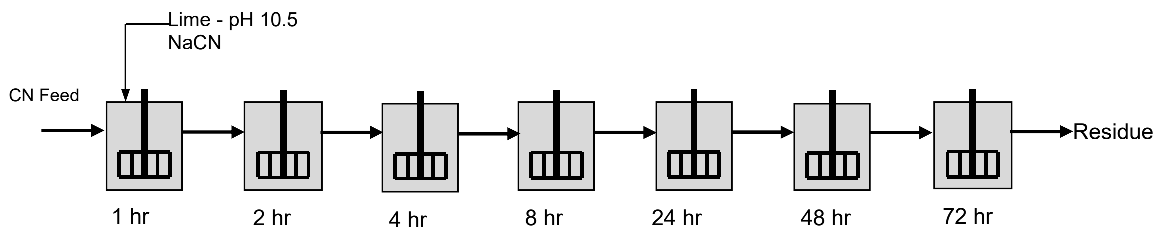
Mass of Sample	1000
Volume of Water	1500
Pulp Density	40

NaCN Consumption	0.43	kg/tonne
Lime Consumption	0.42	kg/tonne

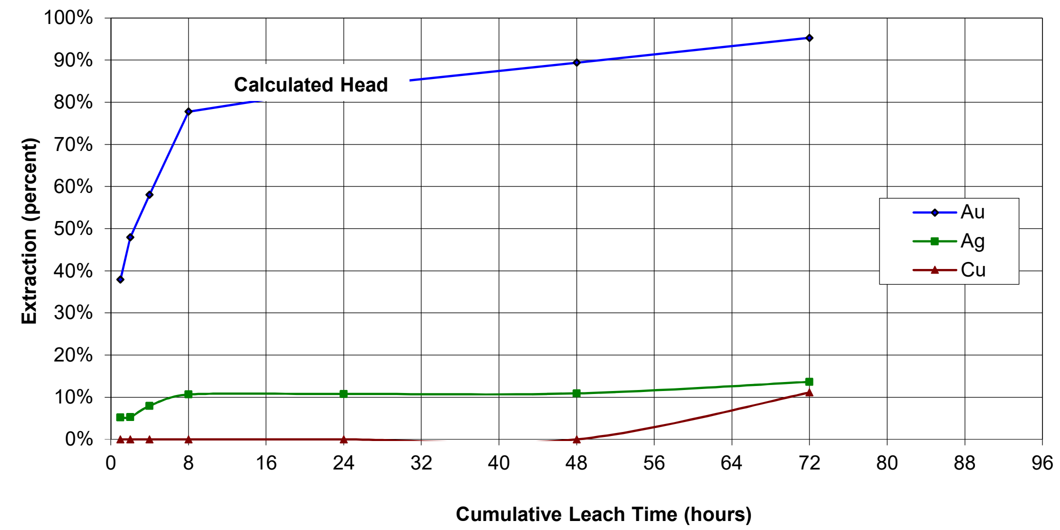
Cumulative Metallurgical Balance

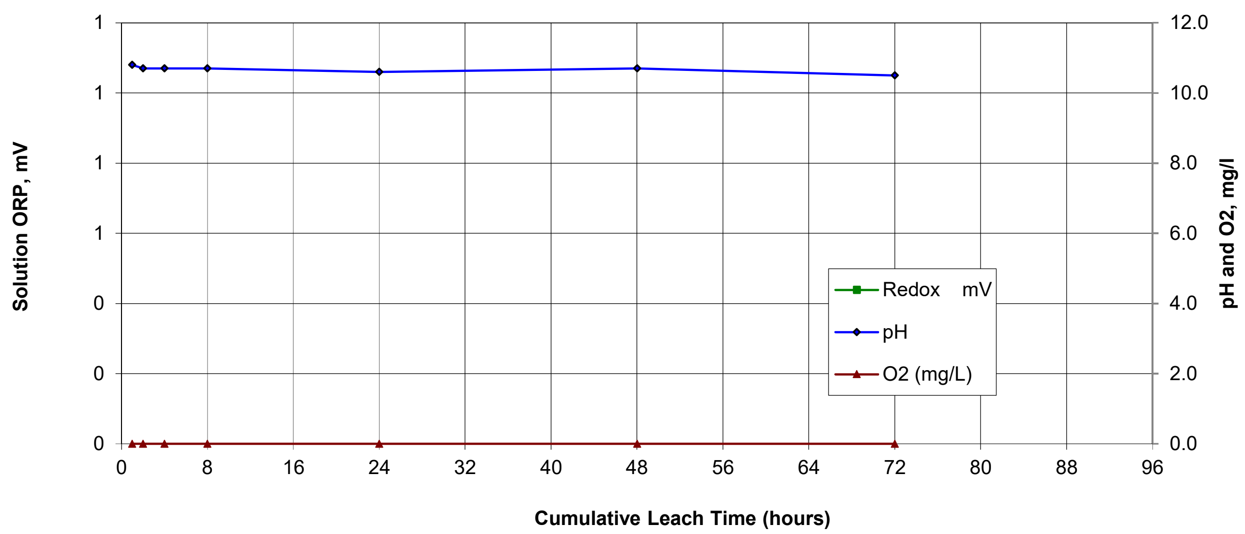
Product	Cumulative	Vol or	Units	Free CN	Free NaCN	mol WAD/	Assay - g/tonne (ppm)			Distribution - percent		
	Time - Hrs	Mass		mg/l, calc'd	mg/l	mol Cu	Au	Ag	Cu	Au	Ag	Cu
Cyanide Liquor (1 hr)	1	1500	mL	0.0	0	0.00	0.08	0.02		37.96%	5.20%	0.00%
Cyanide Liquor (2 hr)	2	1500	mL	0.0	0	0.00	0.10	0.02		47.95%	5.27%	0.00%
Cyanide Liquor (4 hr)	4	1500	mL	0.0	0	0.00	0.12	0.03		58.08%	7.94%	0.00%
Cyanide Liquor (8 hr)	8	1500	mL	0.0	0	0.00	0.16	0.04		77.81%	10.65%	0.00%
Cyanide Liquor (24hr)	24	1500	mL	0.0	0	0.00	0.17	0.04		83.57%	10.78%	0.00%
Cyanide Liquor (48hr)	48	1500	mL	0.0	0	0.00	0.18	0.04		89.39%	10.92%	0.00%
Cyanide Liquor (72hr)	72	1500	mL	685.0	1,290	384.16	0.19	0.05	4.40	95.27%	13.66%	11.11%
Cyanidation Tails	-	996	g				0.015	0.50	53	4.73%	86.34%	88.89%
Calculated Feed, gm/t		996	g				0.317	0.58	60	100.00%	100.00%	100.00%
Head Assay, gm/t		1000										

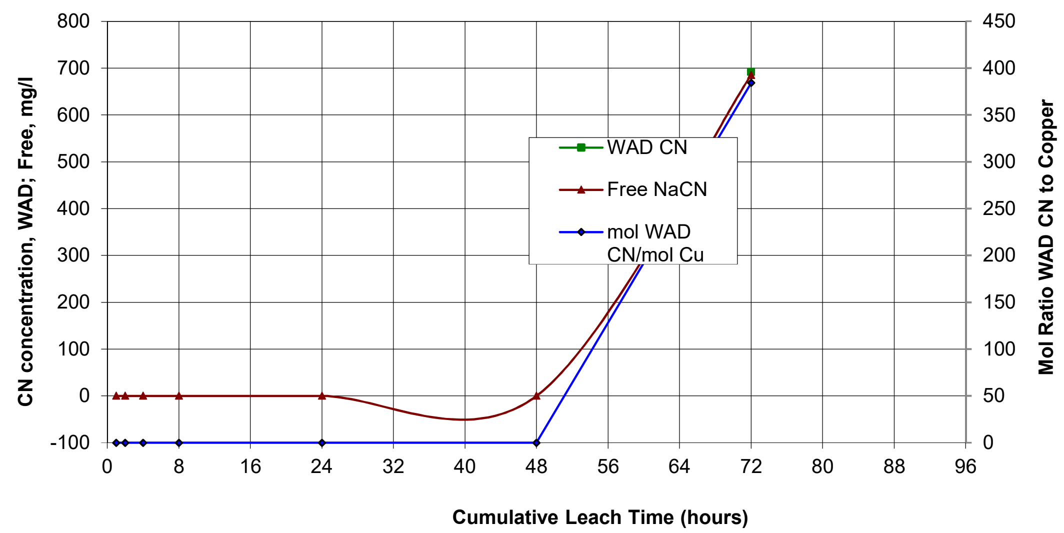
Flowsheet Schematic



Cyanide Leach Kinetic Curves







Test No:	BL295-22			Au, gm/t	Ag, gm/t	Cu, %
Date:	01-Oct-18		Head Assay, Ave:	0.300	1.00	0.01%
Test Type:	Standard bottle roll procedure.		Head Assay 1:	0.30	1	0.01%
Test Objective:	Repeat Test 16 with carbon (CIL).		Head Assay 2:	0.30	1	0.01%
Sample:	Test 12 Knelson Tails		Head Assay 3:	0.30	1	0.01%
Grind:	150µm as received					
Target Maintenance NaCN:	1000	ppm	Total Mass, Initial:		grams including bottle	

Cyanidation Leaching @ pH 10.5 (lime), 1000ppm NaCN, O2 sparged, CIL 50g/L

	Au, gm/t	Ag, gm/t	Cu, %
Tails Assay, Ave:	0.015	0.5	0.00%
Tail Assay 1:	0.020	0.5	0.005%
Tail Assay 2:	0.010	0.5	0.005%

* Ag estimated, assay value <ppm.

Sample Size 20 ml

Calculated Head

	Au	Ag	Cu
Head, mg:	0.298	0.992	69.426
Tails, mg:	0.015	0.496	48.003
Calculated Head, mg:	0.397	0.516	55

Parameter	Time	Added (g)		WAD CN	pH		Dissolved	Redox	Temp	Carbon
	Cum	NaCN	Lime	mg/l CN	Measured	Adjusted	O_2 (mg/L)	mV	Deg. C	g
Natural	-	-	-		8.5	-	7.9		20.0	
Leach 0	0	2.00	0.70		10.5	10.5	7.9		20.0	100.0
Leach 1	1	0.28	-		10.8	-	>20		20.0	
Leach 2	2	0.00	-		10.7	-	>20		19.0	
Leach 3	4	0.10	-		10.7	-	>20		20.0	
Leach 4	8	0.12	-		10.8	-	>20		20.0	
Leach 5	24	0.36	-		11.0	-	>20		20.0	
Leach 6	48	0.36	-		11.2	-	>20		21.0	
Leach 7	72	0.00	-	403	11.0	-	>20		20.0	
Total	72	3.22	0.70		-	-	-	-		

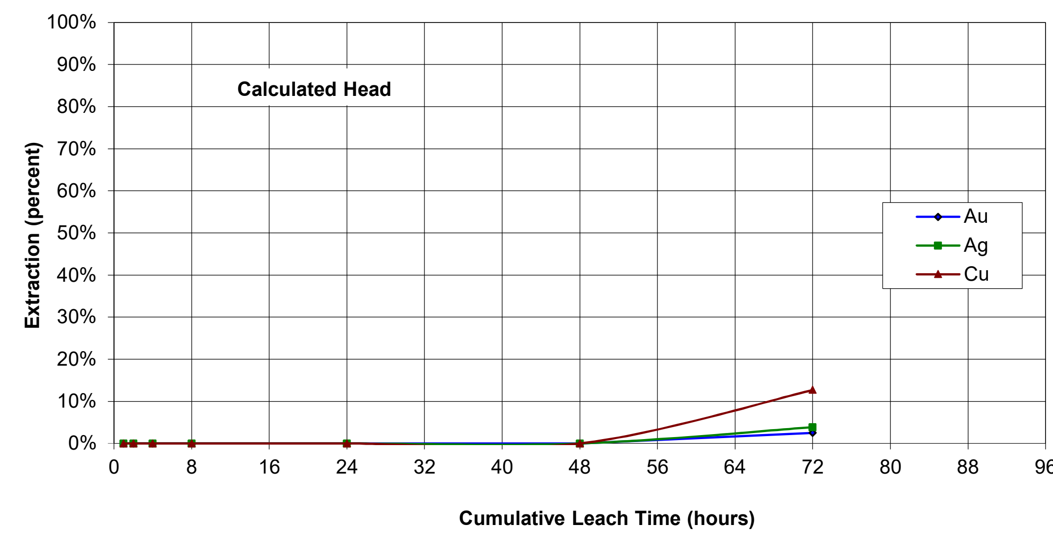
Cumulative Metallurgical Balance

Product	Cumulative	Vol or	Units	Free CN	Free NaCN	mol WAD/	Assay - g/tonne (ppm)			Distribution - percent		
	Time - Hrs	Mass		mg/l, calc'd	mg/l	mol Cu	Au	Ag	Cu	Au	Ag	Cu
Cyanide Liquor (1 hr)	1	2000	mL	0.0	0	0.00				0.00%	0.00%	0.00%
Cyanide Liquor (2 hr)	2	2000	mL	0.0	0	0.00				0.00%	0.00%	0.00%
Cyanide Liquor (4 hr)	4	2000	mL	0.0	0	0.00				0.00%	0.00%	0.00%
Cyanide Liquor (8 hr)	8	2000	mL	0.0	0	0.00				0.00%	0.00%	0.00%
Cyanide Liquor (24hr)	24	2000	mL	0.0	0	0.00				0.00%	0.00%	0.00%
Cyanide Liquor (48hr)	48	2000	mL	0.0	0	0.00				0.00%	0.00%	0.00%
Cyanide Liquor (72hr)	72	2000	mL	397.3	748	281.86	0.01	0.01	3.49	2.52%	3.88%	12.69%
Carbon (72 hr)		101.6	g				3.66	0.50	4.00	93.73%	96.12%	7.22%
Cyanidation Tails	-	992	g				0.015	0.50	48	3.75%	96.12%	87.31%
Calculated Feed, gm/t		992	g				0.400	0.52	55	6.27%	100.00%	100.00%
Head Assay, gm/t		1000										

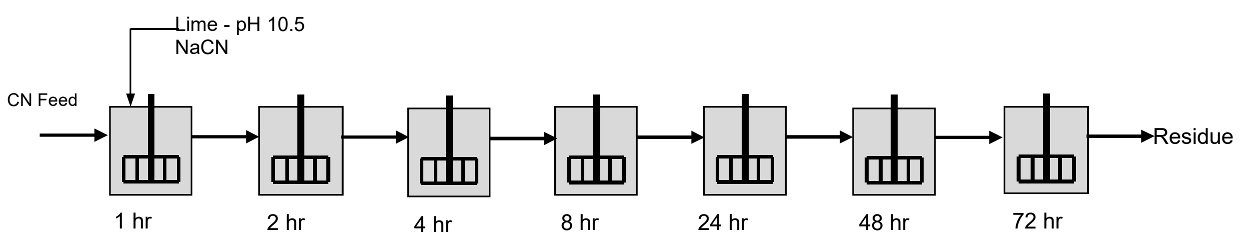
Mass of Sample	1000
Volume of Water	2000
Pulp Density	33

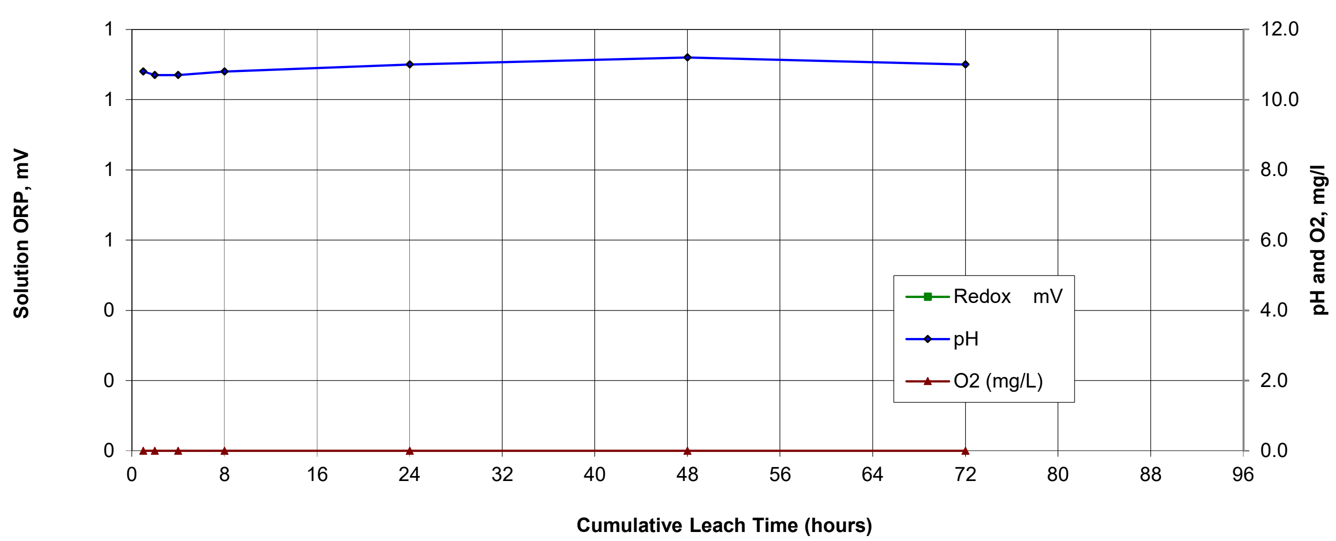
NaCN Consumption	1.72	kg/tonne
Lime Consumption	0.70	kg/tonne

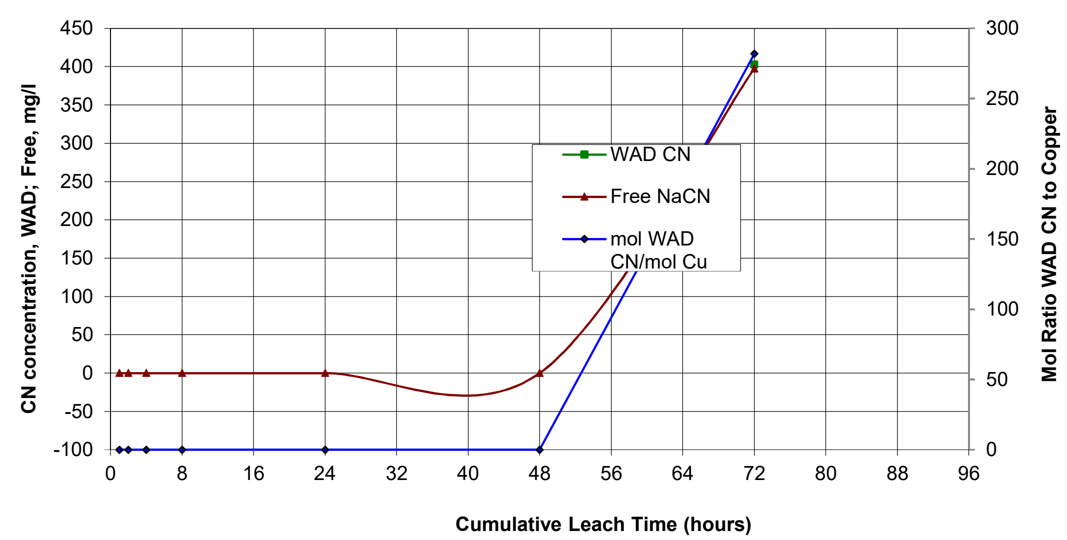
Cyanide Leach Kinetic Curves



Flowsheet Schematic





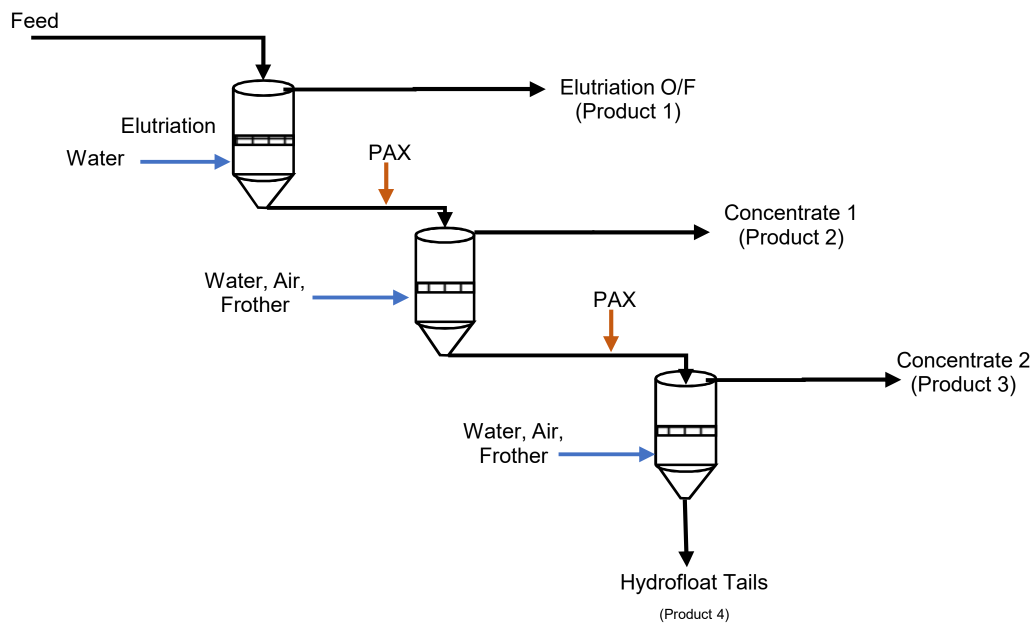


Test No: BL0295-23
Date: 5-Nov-18
Test Type: Hydrofloat test
Test Objective: Evaluate Hydrofloat processing of Bulk Gravity Tailings.
Sample: 17 kg of Test 12 Knelson Tails
Nominal Sizing: as is

Stage	Reagents - g/tonne			Air	Water Flow	
	PAX		W31	L/min	l/min	cm/sec
Elutriation			-	-	3.5	0.38
Concentrate 1	50		As needed	0.6	3.5	0.38
Concnetrate 2	50		As needed	0.8	4.5	0.49

Flotation Information	Rougher
Flotation Device:	HF1
Cell Diameter (in):	5.5
Flotation Gas:	Air
Water Type:	Kamloops Tap

Flowsheet Schematic



BL0295-23 Test 12 Knelson Tails
Metallurgical Balance

Product	Weight		Assay - percent or g/t		Distribution - percent	
	%	grams	Au	S	Au	S
Elutriation O/F	65.7	10909.4	0.24	0.05	44.6	49.9
Concentrate 1	5.2	859.7	2.23	0.44	32.7	36.7
Concentrate 2	3.5	584.3	0.82	0.16	8.2	8.9
HF Tailings	25.6	4260.6	0.20	0.01	14.5	4.5
Recalc. Feed	100.0	16614.0	0.35	0.06	100	100
Measured Feed			0.30	-		

BL0295-23 Test 12 Knelson Tails
Cumulative Balance

Product	Weight		Assay - percent or g/t		Distribution - percent	
	%	grams	Au	S	Au	S
Product 1	65.7	10909.4	0.24	0.05	44.6	49.9
Products 1 to 2	70.8	11769.1	0.39	0.08	77.3	86.5
Products 1 to 3	74.4	12353.4	0.41	0.08	85.5	95.5
Products 2 to 3	8.7	1444.0	1.66	0.33	40.8	45.6
Product 4	25.6	4260.6	0.20	0.01	14.5	4.5
Recalc. Feed	100.0	16614.0	0.35	0.06	100	100

BL0295-23 Test 12 Knelson Tails
Flotation only Balance

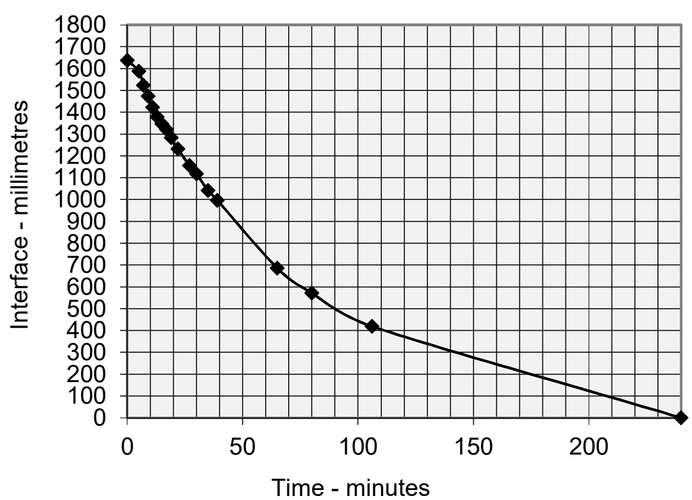
Product	Weight		Assay - percent or g/t		Distribution - percent	
	%	grams	Au	S	Au	S
Product 2	15.1	859.7	2.23	0.44	59.0	73.1
Products 2 to 3	25.3	1444.0	1.66	0.33	73.8	91.0
Product 4	74.7	4260.6	0.20	0.01	26.2	9.0
	100.0	5704.6	0.57	0.09	100	100

TABLE B-1
BL0295 PERCOLATION EVALUATION TEST
Master Composite -6.3mm Crush

PERCOLATION DATA

Elapsed Time (min)	Interface Height (mm)	Interface Height (mm)	Volume Percolated
	Liquid	Solids	mLs
0	1638	1600	0
5	1588	1588	
7	1524		
9	1473		335
11	1422		
13	1378		422
15	1346		
17	1321		
19	1283		579
22	1232		
27	1156		
30	1118		838
35	1041		
39	997		
65	686		1385
80	572		
106	419		
240	0	1568	2625

Percolation Interface



Test No:	CL-01	**Column Test - Locked Cycle**	Dry Wt Charge: 162.0 kg	Column ID: 149.2 mm	
Date:	31-Jul-18		Lime Addition: 0.6 kg/t	Column Area: 0.01748 m^2	
Test Type:	Locked Cycle Column Leach Test		Lime Addition: 97.2 gm	Target App. Rate: 12 $\ell/hr.m^2$	
Test Objective:	Determine Ultimate Extraction		water added: 4.5 L	Target App. Rate: 3.49669 ml/min	
Sample ID:	**BAM MC**			Target Leach NaCN: 0.6 gpl NaCN	
Crush Size:	P_{100} 31.5 mm			Target pH: 10-10.5	
				Barren Solution spg: 1.000	
				Preg. Solution spg: 1.000	Ore spg: 2.717
				Starting Column Height 5.98 m	
				Start Bulk Density: 1.55 t/m^3	
				Ending Column Height 5.96 m	
				End Bulk Density: 1.55 t/m^3	Porosity: 43%
				Slump: 0.3%	
				Drained Weight: 171.1 kg	174.79
				Dried Weight: 161.4 kg	
			Sample in Column: 7-31-18 13:00	Retained Moisture: 5.7%	5.3% calc'd retained moisture
					7.9% calc'd holdup w/w
				Barren Bucket, tare: gm	18.9% by volume
				Preg. Bucket, tare: 0.0 gm	44.2% Saturation

	Au, gm/t	Ag, gm/t	Cu, %		Au, gm/t	Ag, gm/t	Cu, %
Head Assay, Ave:	0.950	1.000	0.11	Tails Assay, Ave:	0.327	0.20	0.01%
Head Assay 1:	0.950	1	0.01%	Tail Assay 1:	0.327	0.20	0.01%
Head Assay 2:	0.950	1	0.01%	Tail Assay 2:	0.327	0.20	0.01%
Head Assay 3:	0.950	1	0.01%				

Preg Sample: 20.0 ml
Carbon Effluent Sample: 20.0 ml
Barren Sample: 20.0 ml

	Au	Ag	Cu				
					#DIV/0!		
Head, mg:	153.900	162	178,200				
Tails, mg:	52.73	32	10,557				
Calculated Head, mg:	121.557	37.515	11,366				
	0.7503489		-27%	0.566	0.78263	-21.6%	
				0.784		-0.2056325	24 hr Drain-down: 3720.0 ml
63.546	MW Cu			0.770			Active Holdup: 23.0 l/tonne
26.0177	MW CN		0.78984				Active Holdup: 35.7 l/m3
49.0075	MW NaCN						
1.8836215	NaCN/CN						

CL-01 - BAM MC - P100 31.5 mm - 0.6 gpl NaCN - 0.6 kg/t Lime - 12 l/hr.m2 Leach Rate

Soln. Start Date	Cycle (Days)	Preg. Solution Data: Temp. Deg. C	End wt, gms	Vol (mls)	pH	ORP mv	WAD CN mg/l CN	Au (ppm)	Ag (ppm)	Cu (ppm)	Carbon Effluent Data: Vol (mls)	pH	ORP mv	WAD CN mg/l CN	Au (ppm)	Ag (ppm)	Cu (ppm)	Carbon: Carbon No.	Carbon dry gms	Au gm/t	Ag gm/t
7-31-18 13:00	End 24 hr																				
7-31-18 13:00	0			In Column																	
8-3-18 14:15	3.05		1420	1420	11.6	-110	0	0.005	0.005	0.07	1,420	12	-122	100				1.1	167.5	368	34
8-5-18 9:00	4.83		8360	8360	11.8	-125	141.3	4.29	0.39	0.88	8,360	12	-122	100	0.04	0.02	0.67				
8-5-18 20:30	5.31		2060	2060	12.1	-125	236.4	1.60	0.12	0.87	2,060	12.1	-120	116.071	0.03	0.01	0.65				
8-6-18 22:15	6.39		4800	4800	12	-120	254.2	1.02	0.08	0.89	4,800	11.90	-119	128.215	0.01	0.01	0.68				
8-7-18 14:00	7.04		2960	2960	12.20	-115	266.4	0.69	0.04	0.73	2,960	12.10	-106	236.3	0.01	0.01	0.65				
8-8-18 11:00	7.92		4080	4080	12.50	-131	269.3	0.58	0.04	0.86	4,080	12.20	-100	225.0	0.01	0.01	1				
8-9-18 14:15	9.05		5340	5340	12.10	-117	346.5	0.40	0.03	2	5,340	12.00	-108	251.1	0.01	0.01	2				
8-10-18 11:40	9.94		3000	3000	12.40	-105	189.3	0.37	0.03	8	3,000	12.10	-100	254.2	0.01	0.01	5				
8-11-18 5:30	10.69		1000	1000	12.40	-103	213.7	0.38	0.03	9	1,000	12.20	-101	190.2	0.01	0.01	1				
8-12-18 5:00	11.67		6820	6820	11.70	-101	224.7	0.38	0.03	8	6,820	12.00	-61	7.2	0.01	0.01	9				
8-14-18 11:30	13.94		6360	6360	12.00	-102	235.2	0.23	0.02	5	6,360	11.60	-105	185.1	0.01	0.01	5				
8-15-18 5:00	14.67		3920	3920	12.10	-83	196.8	0.13	0.02	3	3,920	11.90	-65	265.0	0.01	0.01	4				
8-16-18 5:00	15.67		5200	5200	12.20	-107	230.5	0.13	0.01	4	5,200	11.70	-100	176.3	0.01	0.01	4				
8-17-18 5:00	16.67		5320	5320	11.50	-54	280.0	0.12	0.01	6	5,320	11.30	-64	214.9	0.01	0.01	8				
8-18-18 5:00	17.67		4660	4660	11.50	-77	239.9	0.09	0.01	8	4,660	11.30	-80	269.0	0.01	0.01	8				
8-19-18 12:30	18.98		7060	7060	11.50	-109	156.6	0.10	0.01	8	7,060	11.00	-118	178.5	0.01	0.01	8				
8-20-18 14:30	20.06		5760	5760	11.50	-107	215.9	0.09	0.01	8	5,760	11.30	-106	189.4	0.01	0.01	8				
8-21-18 6:15	20.72		3520	3520	11.50	-95	188.4	0.10	0.01	8	3,520	11.40	-84	211.8	0.01	0.01	8				
8-22-18 5:45	21.70		5020	5020	11.50	-108	205.0	0.07	0.01	10	5,020	11.00	-105	188.9	0.01	0.01	9				
8-23-18 5:30	22.69		5180	5180	11.30	-88	204.5	0.05	0.02	11	5,180	10.90	-92	186.3	0.01	0.01	11				
8-24-18 5:10	23.67		5080	5080	11.40	-82	206.6	0.01	0.01	12	5,080	11.10	-85	205.5	0.01	0.01	11				
8-25-18 5:00	24.67		5160	5160	11.30	-43	214.9	0.05	0.01	12	5,160	10.90	-62	174.3	0.03	0.01	12				
8-26-18 5:15	25.68		5080	5080	11.30	-44	220.1	0.04	0.01	12	5,080	11.00	-68	212.3	0.01	0.01	11				
8-27-18 5:15	26.68		5220	5220	11.20	-75	190.5	0.08	0.01	11	5,220	11.10	-73	186.3	0.01	0.01	11				
8-28-18 5:15	27.68		5300	5300	11.20	-44	202.9	0.07	0.01	12	5,300	10.80	-41	197.7	0.01	0.01	13				
8-29-18 5:15	28.68		5040	5040	11.10	-42	213.3	0.09	0.02	14	5,040	10.70	-41	175.4	0.01	0.01	14				
8-30-18 5:15	29.68		4780	4780	11.00	-41	186.3	0.11	0.01	14	4,780	10.70	-43	187.3	0.01	0.01	14				
8-31-18 5:15	30.68		6060	6060	11.00	-33	150.9	0.06	0.01	14	6,060	10.50	-31	197.2	0.01	0.01	14				
9-1-18 0:01	31.46		3640	3640	11.00	24	161.8	0.08	0.01	14	3,640	10.50	43	149.8	0.01	0.01	15				

Soln. Start	Cycle	Preg. Solution Data									Carbon Effluent Data							Carbon			
Date	(Days)	Temp.	End	Vol	pH	ORP	WAD CN	Au	Ag	Cu	Vol	pH	ORP	WAD CN	Au	Ag	Cu	Carbon	Carbon	Au	Ag
7-31-18 13:00	End 24 hr	Deg. C	wt, gms	(mls)		mv	mg/l CN	(ppm)	(ppm)	(ppm)	(mls)		mv	mg/l CN	(ppm)	(ppm)	(ppm)	No.	dry gms	gm/t	gm/t
9-2-18 17:50	33.20		9180	9180	11.20	-16	132.7	0.06	0.01	15	9,180										
9-3-18 13:20	34.01		4140	4140	11.10	-4	124.9	0.04	0.01	16	4,140	10.60	8	66.1	0.01	0.01	16				
9-4-18 6:30	34.73		3860	3860	10.70	8	141.0	0.04	0.01	16	3,860	10.20	26	108.2	0.01	0.01	15				
9-5-18 5:30	35.69		5000	5000	10.70	-73	120.7	0.05	0.01	16	5,000	10.10	-55	102.5	0.01	0.01	16				
9-6-18 5:30	36.69		5260	5260	11.00	126	137.9	0.04	0.01	15	5,260	10.40	143	120.7	0.01	0.01	16				
9-7-18 6:15	37.72		5060	5060	10.70	87	143.6	0.04	0.01	17	5,060	10.30	146	111.9	0.01	0.01	15	1.2	150.0		
9-8-18 5:15	38.68		4560	4560	10.80	98	147.8	0.06	0.01	16	4,560	10.30	119	104.6	0.01	0.01	14				
9-9-18 1:45	39.53		4140	4140	10.80	202	129.1	0.05	0.01	18	4,140	10.30	210	90.0	0.01	0.01	15				
9-10-18 7:00	40.75		5760	5760	10.80	169	154.5	0.05	0.01	17	5,760	10.20	193	80.7	0.01	0.01	16				
9-11-18 5:45	41.70		4660	4660	10.80	157	106.2	0.07	0.01	17	4,660	10.30	16	108.2	0.01	0.01	18				
9-12-18 5:15	42.68		4980	4980	10.80	46	109.3	0.05	0.01	16	4,980	10.20	62	64.5	0.01	0.01	16				
9-13-18 5:15	43.68		5160	5160	10.80	43	124.9	0.08	0.01	17	5,160	10.10	42	80.1	0.01	0.01	17				
9-14-18 5:15	44.68		5160	5160	10.30	179	94.2	0.07	0.01	17	5,160	10.00	191	77.5	0.01	0.01	16				
9-15-18 20:00	46.29		8260	8260	10.50	131	96.8	0.06	0.01	18	8,260	9.80	153	45.8	0.01	0.01	14				
9-16-18 11:00	46.92		3220	3220	10.60	121	91.1	0.05	0.01	18	3,220	10.30	135	67.6	0.01	0.01	19				
9-17-18 5:15	47.68		4180	4180	10.60	151	74.9	0.06	0.01	17	4,180	10.10	141	69.2	0.01	0.01	15				
9-18-18 6:30	48.73		5360	5360	10.50	118	77.0	0.03	0.01	17	5,360	10.10	132	45.3	0.01	0.01	16				
9-19-18 5:15	49.68		5000	5000	10.20	120	78.6	0.04	0.01	17	5,000	9.90	117	63.0	0.01	0.01	17				
9-20-18 6:30	50.73		5360	5360	10.40	151	34.9	0.04	0.01	18	5,360	9.70	157	45.0	0.01	0.01	15				
9-21-18 6:20	51.72		5220	5220	10.40	179	43.4	0.02	0.01	17	5,220	9.80	186	43.2	0.01	0.01	18				
9-22-18 0:05	52.46		3740	3740	10.20	155	45.6	0.02	0.01	18	3,740	9.90	166	33.2	0.01	0.01	23				
9-23-18 10:55	53.91		7720	7720	10.20	177	36.8	0.02	0.01	17	7,720	9.40	185	27.2	0.01	0.01	6				
9-24-18 6:00	54.71		4260	4260	10.10	125	22.4	0.01	0.01	17	4,260	9.70	132	31.2	0.01	0.01	15				
9-25-18 6:30	55.73		5080	5080	10.10	122	31.4	0.02	0.01	17	5,080	9.50	143	17.0	0.01	0.01	11				
9-26-18 7:00	56.75		4980	4980	10.00	160	27.8	0.02	0.01	16	4,980	9.40	161	14.6	0.01	0.01	10				
9-27-18 6:00	57.71		4660	4660	9.90	159	27.4	0.03	0.01	14	4,660	9.40	156	16.7	0.01	0.01	9				
9-28-18 5:15	58.68		5000	5000	9.80	136	26.5	0.02	0.01	15	5,000	9.50	150	15.4	0.01	0.01	6				
9-29-18 12:15	59.97		6360	6360	10.00	239	16.1	0.01	0.01	13	6,360	9.60	244	20.6	0.01	0.01	5				
9-30-18 12:45	60.99		5100	5100	9.60	244	15.4	0.01	0.01	12	5,100	9.50	217	12.9	0.01	0.01	5				
10-1-18 7:00	61.75		3060	3060	9.30	222	14.7	0.01	0.01	10	3,060	9.40	220	9.2	0.01	0.01	4				
10-2-18 6:55	62.75		2280	2280	9.20	218	14.8	0.01	0.01	11	2,280	9.10	220	12.9	0.01	0.01	2				
10-3-18 6:15	63.72		5220	5220	9.50	185	33.7	0.01	0.01	11	5,220	9.00	201	17.0	0.01	0.01	1				
10-4-18 6:05	64.71		5060	5060	9.50	254	12.6	0.01	0.01	11	5,060	9.00	256	30.7	0.01	0.01	1				
10-5-18 6:15	65.72		5060	5060	9.50	168	57.1	0.01	0.01	7	5,060	8.80	168	65.4	0.01	0.01	2				
10-6-18 8:40	66.82		5680	5680	9.60	110	252.6	0.02	0.01	6	5,680	8.70	117	12.5	0.01	0.01	7				
10-7-18 0:05	67.46		2040	2040	9.90	75	258.4	0.05	0.01	7	2,040	9.70	74	168.9	0.01	0.01	73				
10-8-18 0:20	68.47		5480	5480	9.90	148	180.4	0.02	0.01	5	5,480	9.70	111	80.5	0.01	0.01	38				
10-9-18 7:45	69.78		6740	6740	10.00	113	60.2	0.02	0.01	4	6,740	10.00	100	108.8	0.01	0.01	16				
10-10-18 8:40	70.82		5500	5500	10.00	87	148.8	0.03	0.01	40	5,500	10.10	73	79.1	0.04	0.01	6				
10-11-18 5:15	71.68		4520	4520	9.90	85	114.2	0.03	0.02	32	4,520	9.90	82	76.0	0.03	0.02	34				
10-12-18 7:00	72.75		4520	4520	9.60	151	107.7	0.02	0.01	16	4,520	9.50	154	78.1	0.01	0.01	29				
10-13-18 10:00	73.88		5740	5740	9.90	190	94.7	0.03	0.01	22	5,740	9.60	192	46.3	0.02	0.01	18				
10-14-18 12:00	74.96		5840	5840	10.10	190	118.1	0.03	0.01	20	5,840	9.70	172	55.2	0.02	0.01	22				
10-15-18 6:10	75.72		4300	4300	10.10	165	178.0	0.02	0.01	27	4,300	10.00	165	102.5	0.02	0.01	24				
10-16-18 7:20	76.76		5500	5500	10.20	167	149.3	0.03	0.01	22	5,500	10.00	162	113.4	0.02	0.01	28				
10-17-18 7:35	77.77		5540	5540	10.30	232	98.3	0.01	0.01	22	5,540	10.10	209	119.7	0.01	0.01	23				
10-18-18 7:15	78.76		3460	3460	10.10	190	87.9	0.01	0.01	22	3,460	9.90	186	72.9	0.01	0.01	23				
10-19-18 5:25	79.68		1100	1100	10.20	225	81.2	0.01	0.01	23	1,100	10.10	219	47.4	0.01	0.01	20				
10-20-18 0:05	80.46		3980	3980	10.40	184	85.3	0.01	0.01	24	3,980	9.50	191	51.0	0.01	0.01	7				
10-21-18 0:05	81.46		5240	5240	10.20	132	94.7	0.01	0.01	25	5,240	9.80	146	50.0	0.01	0.01	22				
10-22-18 5:30	82.69		6340	6340	10.30	130	60.9	0.01	0.01	24	6,340	9.90	140	63.5	0.01	0.01	25				
10-23-18 7:00	83.75		5360	5360	10.50	156	80.7	0.01	0.01	21	5,360	10.20	152	81.2	0.01	0.01	25				
10-24-18 6:55	84.75		5200	5200	10.50	113	69.2	0.01	0.01	22	5,200	10.10	115	59.8	0.01	0.01	22				
10-25-18 11:30	85.94		6100	6100	10.90	122	66.6	0.01	0.01	24	6,100	10.40	120	57.2	0.01	0.01	22				
10-26-18 7:05	86.75		4160	4160	10.50	141	70.2	0.01	0.01	25	4,160	10.20	130	56.2	0.01	0.01	23				
10-27-18 5:40	87.69		4720	4720	10.70	161	36.3	0.01	0.01	18	4,720	10.10	166	37.3	0.01	0.01	23				
10-28-18 0:01	88.46		3720	3720	10.90	176	5.9	0.01	0.01	3	3,720	10.20	160	32.9	0.01	0.01	18				
10-29-18 7:40	89.78		6400	6400	10.60	250	8.1	0.01	0.01	0	6,400	10.30	240	9.1	0.01	0.01	4				
10-30-18 7:20	90.76		3720	3720	10.40	190	3.8	0.01	0.01	0	3,720	10.30	191	3.2	0.01	0.01	1				
10-31-18 5:15	91.68			0							0										

Test No: **CL-02** **Column Test - Locked Cycle**

Date: 31-Jul-18

Test Type: Locked Cycle Column Leach Test

Test Objective: Determine Ultimate Extraction

Sample ID: **BAM MC**

Crush Size: P_{100} 6.3 mm

Dry Wt Charge: 149.8 kg

Lime Addition: 0.6 kg/t

Lime Addition: 89.9 gm

Column ID: 149.2 mm

Column Area: 0.01748 m^2

Target App. Rate: 12 ℓ/hr.m^2

Target App. Rate: 3.49669 ml/min

Target Leach NaCN: 0.6 gpl NaCN

Target pH: 10-10.5

Barren Solution spg: 1.000

Preg. Solution spg: 1.000

Ore spg: 2.717

Starting Column Height 5.98 m

Start Bulk Density: 1.43 t/m^3

Ending Column Height 5.93 m

End Bulk Density: 1.44 t/m^3 Porosity: 47%

Slump: 0.8%

Drained Weight: 156.9 kg 156.9

Dried Weight: 145.4 kg

Sample in Column: 8-2-18 11:00

Retained Moisture: 7.3% 4.5% calc'd retained moisture

4.7% calc'd holdup w/w

Barren Bucket, tare: gm 11.9% by volume

Preg. Bucket, tare: 0.0 gm 25.4% Saturation

	Au, gm/t	Ag, gm/t	Cu, %
Head Assay, Ave:	0.950	1.000	0.11
Head Assay 1:	0.950	1	0.01%
Head Assay 2:	0.950	1	0.01%
Head Assay 3:	0.950	1	0.01%

	Au, gm/t	Ag, gm/t	Cu, %
Tails Assay, Ave:	0.095	0.20	0.01%
Tail Assay 1:	0.095	0.20	0.01%
Tail Assay 2:	0.095	0.20	0.01%

Preg Sample: 20.0 ml

Carbon Effluent Sample: 20.0 ml

Barren Sample: 20.0 ml

	Au	Ag	Cu
Head, mg:	142.310	150	164,780
Tails, mg:	13.85	29	11,222
Calculated Head, mg:	126.762	37.507	12,329
	0.8462059		-12%

#DIV/0!

0.891 0.78263 10.8%

0.784 0.10257167

0.770

24 hr Drain-down: ml

Active Holdup: 0.0 l/tonne

Active Holdup: 0.0 l/m3

63.546 MW Cu

26.0177 MW CN 0.89074

49.0075 MW NaCN

1.8836215 NaCN/CN

CL-02 - BAM MC - P100 6.3 mm - 0.6 gpl NaCN - 0.6 kg/t Lime - 12 l/hr.m2 Leach Rate

43314.45833

Soln. Start	Cycle	Preg. Solution Data									Carbon Effluent Data							Carbon			
Date	(Days)	Temp.	End	VOl	pH	ORP	WAD CN	Au	Ag	Cu	VOl	pH	ORP	WAD CN	Au	Ag	Cu	Carbon	Carbon	Au	Ag
8-2-18 11:00	End 24 hr	Deg. C	wt, gms	(mls)		mv	mg/l CN	(ppm)	(ppm)	(ppm)	(mls)		mv	mg/l CN	(ppm)	(ppm)	(ppm)	No.	dry gms	gm/t	gm/t
	0																				
8-2-18 11:00	0			In Column																	
8-5-18 9:00	2.92		0	0							0										
8-6-18 22:20	4.47		1560	1560	11.9	-86	1.45	0.005	0.005	0.11	1,560							1.1	167.7	638	48
8-7-18 14:00	5.13		2620	2620	12.2	-90	4.30	4.18	0.12	0.12	2,620										
8-8-18 11:10	6.01		3060	3060	12.4	-83	93.0	10.3	1.11	1.05	3,060	11.8	-66	1.719	0.005	0.005	0.005				
8-9-18 14:15	7.14		4480	4480	12.1	-110	210.1	4.29	0.37	1.48	4,480	11.8	-85	256.4	0.0	0.0	0.6				
8-10-18 11:45	8.03		4380	4380	12.3	-98	365.1	1.85	0.15	2	4,380	12.1	-95	209.9	0.0	0.0	1.3				
8-11-18 5:35	8.77		1560	1560	12.2	-107	384.2	1.31	0.12	2	1,560	12.3	-103	233.4	0.01	0.01	2				
8-12-18 5:00	9.75		5380	5380	11.9	-124	295.2	1.13	0.11	4	5,380	12.2	-52	115.1	0.01	0.01	2				
8-14-18 11:30	12.02		5340	5340	12.0	-104	348.1	0.77	0.07	8	5,340	11.5	-87	296.5	0.01	0.01	4				
8-15-18 5:00	12.75		1240	1240	12.1	-104	272.6	0.69	0.06	9	1,240	11.9	-93	249.1	0.01	0.01	7				
8-16-18 5:00	13.75		3420	3420	12.2	-88	223.4	0.59	0.06	9	3,420	10.5	-66	95.5	0.01	0.01	7				
8-17-18 5:00	14.75		5380	5380	11.40	-65	253.9	0.35	0.03	14	5,380	10.50	-75	186.8	0.01	0.01	8				
8-18-18 5:00	15.75		4380	4380	11.50	-79	238.3	0.24	0.03	14	4,380	11.10	-75	233.6	0.01	0.01	10				
8-19-18 12:45	17.07		3900	3900	11.60	-106	229.0	0.22	0.02	14	3,900	10.80	-106	211.8	0.01	0.01	14				
8-20-18 14:40	18.15		5440	5440	11.60	-107	227.9	0.25	0.01	14	5,440	10.90	-99	154.0	0.01	0.01	13				
8-21-18 6:15	18.80		3220	3220	11.50	-90	236.8	0.21	0.01	14	3,220	11.20	-91	186.8	0.01	0.01	14				
8-22-18 5:45	19.78		4260	4260	11.50	-94	199.8	0.19	0.01	12	4,260	10.60	-74	178.0	0.01	0.01	15				
8-23-18 5:30	20.77		3920	3920	11.30	-88	218.0	0.18	0.02	13	3,920	10.70	-88	207.6	0.01	0.01	14				
8-24-18 5:10	21.76		3820	3820	11.40	-79	218.5	0.17	0.04	15	3,820	10.70	-80	185.2	0.01	0.01	13				
8-25-18 5:00	22.75		1740	1740	11.30	-85	195.1	0.22	0.02	14	1,740	10.80	-79	159.7	0.01	0.01	13				
8-26-18 5:15	23.76		3360	3360	11.40	-65	201.9	0.18	0.03	14	3,360	10.80	-74	208.7	0.01	0.01	10				
8-27-18 5:15	24.76		3700	3700	11.30	-70	189.9	0.26	0.02	15	3,700	10.50	-67	153.5	0.01	0.01	15				
8-28-18 5:15	25.76		1840	1840	11.10	-49	177.4	0.22	0.02	17	1,840	10.40	-50	158.2	0.01	0.01	15				
8-29-18 5:15	26.76		4340	4340	11.20	-50	189.4	0.27	0.01	17	4,340	10.10	-49	139.5	0.01	0.01	15				
8-30-18 5:15	27.76		4360	4360	11.20	-50	189.4	0.17	0.01	18	4,360	10.70	-43	159.2	0.01	0.01	17				
8-31-18 5:15	28.76		5060	5060	11.10	-44	173.3	0.15	0.01	22	5,060	10.60	-36	199.8	0.01	0.01	17				
9-1-18 0:01	29.54		2800	2800	11.10	-11	184.7	0.12	0.01	20	2,800	10.60	-5	135.8	0.01	0.01	18				
9-2-18 17:50	31.28		9320	9320	11.10	-24	172.2	0.09	0.01	20	9,320	10.10	42	83.3	0.01	0.01	20				
9-3-18 13:20	32.10		4500	4500	10.70	21	127.5	0.09	0.01	17	4,500	10.60	23	84.8	0.01	0.01	20				

Soln. Start	Cycle	Preg. Solution Data									Carbon Effluent Data							Carbon			
Date	(Days)	Temp.	End	VOl	pH	ORP	WAD CN	Au	Ag	Cu	VOl	pH	ORP	WAD CN	Au	Ag	Cu	Carbon	Carbon	Au	Ag
8-2-18 11:00	End 24 hr	Deg. C	wt, gms	(mls)		mv	mg/l CN	(ppm)	(ppm)	(ppm)	(mls)		mv	mg/l CN	(ppm)	(ppm)	(ppm)	No.	dry gms	gm/t	gm/t
9-4-18 6:30	32.81		3720	3720	10.70	-77	142.6	0.08	0.01	18	3,720	10.40	-57	105.6	0.01	0.01	21				
9-5-18 5:30	33.77		5240	5240	10.60	-63	140.5	0.08	0.01	17	5,240	10.10	-49	79.6	0.01	0.01	18				
9-6-18 5:15	34.76		5000	5000	10.80	119	142.1	0.06	0.01	20	5,000	10.30	147	123.8	0.01	0.01	19				
9-7-18 6:15	35.80		4860	4860	10.70	107	130.6	0.10	0.02	20	4,860	10.20	131	104.1	0.01	0.01	17	1.2	150.0		
9-8-18 5:15	36.76		4220	4220	10.70	108	132.7	0.10	0.01	23	4,220	10.20	147	96.3	0.01	0.01	14				
9-9-18 1:45	37.61		3820	3820	10.80	182	126.4	0.10	0.01	21	3,820	10.20	199	64.5	0.01	0.01	17				
9-10-18 7:00	38.83		4400	4400	10.70	173	114.5	0.12	0.01	22	4,400	10.00	186	54.6	0.01	0.01	18				
9-11-18 5:45	39.78		3260	3260	10.50	55	110.3	0.10	0.01	21	3,260	10.10	80	72.9	0.01	0.01	19				
9-12-18 5:15	40.76		3820	3820	10.70	43	126.4	0.11	0.01	20	3,820	10.00	66	52.6	0.01	0.01	18				
9-13-18 5:15	41.76		4000	4000	10.40	49	99.9	0.10	0.01	21	4,000	10.00	50	43.2	0.01	0.01	21				
9-14-18 5:15	42.76		3960	3960	9.80	190	84.8	0.12	0.01	21	3,960	9.60	188	53.1	0.01	0.01	19				
9-15-18 20:00	44.38		4760	4760	10.10	136	121.8	0.08	0.01	24	4,760	9.50	160	59.8	0.01	0.01	13				
9-16-18 11:00	45.00		800	800	10.00	136	80.1	0.11	0.01	23	800	10.00	143	66.1	0.01	0.01	24				
9-17-18 5:15	45.76		680	680	9.90	150	76.5	0.10	0.01	22	680	9.30	167	70.2	0.01	0.01	15				
9-18-18 6:30	46.81		620	620	9.60	144	62.4	0.11	0.01	21	620	9.00	154	40.1	0.01	0.01	9				
9-19-18 5:15	47.76		500	500	9.40	129	56.7	0.12	0.01	21	500	9.00	139	43.2	0.01	0.01	1				
9-20-18 6:30	48.81		1080	1080	9.90	153	33.6	0.08	0.01	24	1,080	8.90	169	25.8	0.01	0.01	0				
9-21-18 6:20	49.81		7220	7220	10.50	190	67.1	0.09	0.01	25	7,220	8.60	217	29.1	0.01	0.01	0				
9-22-18 0:01	50.54		4660	4660	10.40	150	56.5	0.08	0.01	20	4,660	9.30	149	20.4	0.01	0.01	23				
9-23-18 10:55	52.00		6180	6180	10.30	157	40.1	0.06	0.01	21	6,180	9.40	175	23.0	0.01	0.01	12				
9-24-18 6:00	52.79		2640	2640	10.30	118	26.7	0.04	0.01	18	2,640	9.60	127	31.0	0.01	0.01	21				
9-25-18 6:30	53.81		1700	1700	10.00	133	17.7	0.05	0.01	19	1,700	9.30	145	16.0	0.01	0.01	11				
9-26-18 7:00	54.83		980	980	9.50	157	29.1	0.05	0.01	20	980	8.70	170	19.8	0.01	0.01	0				
9-27-18 6:00	55.79		680	680	9.50	157	33.8	0.04	0.01	21	680	8.80	175	15.8	0.01	0.01	1				
9-28-18 6:45	56.82		580	580	9.40	145	33.4	0.03	0.01	22	580	8.80	159	19.6	0.01	0.01	0				
9-29-18 12:15	58.05		1340	1340	10.10	222	35.6	0.04	0.01	21	1,340	8.90	238	9.1	0.01	0.01	0				
9-30-18 12:45	59.07		1320	1320	10.00	212	42.3	0.03	0.01	22	1,320	8.70	228	10.7	0.01	0.01	0				
10-1-18 7:00	59.83		1320	1320	10.30	194	37.6	0.04	0.01	23	1,320	8.80	224	15.5	0.01	0.01	0				
10-2-18 6:55	60.83		1440	1440	10.10	175	39.7	0.04	0.01	22	1,440	8.60	200	14.8	0.01	0.01	1				
10-3-18 6:15	61.80		3040	3040	10.20	192	29.7	0.03	0.01	23	3,040	8.70	214	21.3	0.01	0.01	0				
10-4-18 6:05	62.80		2400	2400	10.30	200	34.6	0.02	0.01	24	2,400	9.10	220	17.4	0.01	0.01	1				
10-5-18 6:15	63.80		2460	2460	10.10	133	28.6	0.02	0.01	21	2,460	8.90	150	14.4	0.01	0.01	1				
10-6-18 8:40	64.90		2120	2120	9.90	90	26.4	0.01	0.01	19	2,120	8.90	108	25.0	0.01	0.01	1				
10-7-18 0:05	65.55		1220	1220	10.00	65	24.8	0.01	0.01	18	1,220	9.20	76	11.3	0.01	0.01	9				
10-8-18 0:20	66.56		1560	1560	10.00	103	22.4	0.01	0.01	17	1,560	8.80	127	19.3	0.01	0.01	0				
10-9-18 7:45	67.86		1780	1780	10.00	90	19.9	0.01	0.01	12	1,780	8.80	112	13.7	0.01	0.01	0				
10-10-18 8:40	68.90		960	960	9.80	73	43.2	0.01	0.01	10	960	9.10	90	5.2	0.01	0.01	1				
10-11-18 5:15	69.76		760	760	9.80	75	16.1	0.01	0.01	9	760	8.90	97	17.1	0.01	0.01	0				
10-12-18 7:00	70.83		2000	2000	9.80	134	18.0	0.01	0.01	9	2,000	8.60	156	8.3	0.01	0.01	0				
10-13-18 10:10	71.97		4600	4600	9.90	171	249.8	0.27	0.03	16	4,600	8.60	193	40.1	0.01	0.01	0				
10-14-18 12:10	73.05		3560	3560	9.60	163	232.1	0.21	0.02	23	3,560	9.10	175	109.8	0.02	0.01	77				
10-15-18 6:10	73.80		2260	2260	9.90	141	155.6	0.18	0.01	17	2,260	9.50	153	149.3	0.02	0.01	77				
10-16-18 7:20	74.85		3500	3500	9.80	141	101.5	0.12	0.01	10	3,500	9.40	153	35.9	0.02	0.01	42				
10-17-18 7:35	75.86		6180	6180	9.70	206	191.0	0.06	0.01	8	6,180	9.40	214	80.7	0.01	0.01	23				
10-18-18 7:15	76.84		4720	4720	9.80	174	166.5	0.06	0.01	7	4,720	9.70	172	121.2	0.01	0.01	16				
10-19-18 5:25	77.77		1680	1680	10.10	206	94.7	0.04	0.01	13	1,680	10.00	211	132.2	0.01	0.01	14				
10-20-18 0:05	78.55		860	860	9.70	165	138.4	0.05	0.01	30	860	9.80	161	71.3	0.01	0.01	14				
10-21-18 0:05	79.55		3940	3940	10.00	146	123.2	0.06	0.01	38	3,940	9.50	154	68.7	0.01	0.01	4				
10-22-18 5:30	80.77		4380	4380	10.10	140	106.2	0.10	0.01	43	4,380	9.50	146	66.2	0.01	0.01	28				
10-23-18 7:00	81.83		4780	4780	10.30	134	131.7	0.08	0.01	46	4,780	9.90	142	64.5	0.01	0.01	39				
10-24-18 6:55	82.83		2740	2740	10.00	124	121.8	0.07	0.01	27	2,740	9.60	123	97.8	0.01	0.01	49				
10-25-18 11:40	84.03		4560	4560	10.40	120	139.5	0.07	0.01	24	4,560	10.00	123	70.8	0.01	0.01	32				
10-26-18 7:05	84.84		3000	3000	10.00	113	105.1	0.06	0.01	20	3,000	9.80	119	102.5	0.01	0.01	32				
10-27-18 5:40	85.78		2080	2080	10.20	175	141.5	0.07	0.01	21	2,080	10.00	173	81.2	0.01	0.01	23				
10-28-18 0:01	86.54		980	980	9.90	164	153.5	0.07	0.01	24	980	9.90	162	102.0	0.01	0.01	21				
10-29-18 7:40	87.86		4280	4280	10.30	242	91.6	0.05	0.01	30	4,280	9.70	230	53.1	0.01	0.01	22				
10-30-18 7:20	88.85		3320	3320	9.90	220	104.1	0.04	0.01	31	3,320	9.70	209	91.6	0.01	0.01	28				
10-31-18 6:55	89.83		1860	1860	10.00	233	123.3	0.03	0.01	28	1,860	9.90	208	84.8	0.01	0.01	30				
11-1-18 7:07	90.84		1520	1520	9.70	229	109.3	0.02	0.01	39	1,520	9.40	211	41.6	0.01	0.01	28				
11-2-18 6:35	91.82		1100	1100	9.80	144	85.3	0.02	0.01	38	1,100	9.40	146	36.4	0.01	0.01	11				
11-3-18 0:01	92.54		540	540	9.40	164	84.3	0.03	0.01	39	540	9.20	164	36.9	0.01	0.01	21				
11-4-18 11:00	94.00		2120	2120	9.80	144	85.9	0.04	0.01	36	2,120	9.10	152	52.0	0.01	0.01	3				
11-5-18 7:30	94.85		1160	1160	10.00	132	110.8	0.04	0.01	36	1,160	9.50	139	35.0	0.01	0.01	29				
11-6-18 7:00	95.83		1720	1720	10.00	154	83.8	0.03	0.01	33	1,720	9.30	169	35.9	0.02	0.01	20				
11-7-18 10:00	96.96		4820	4820	10.10	127	81.2	0.07	0.01	34	4,820	9.10	153	35.7	0.01	0.01	19				
11-8-18 6:00	97.79		3780	3780	10.10	141	78.6	0.05	0.01	31	3,780	9.70	140	38.5	0.01	0.01	38				
11-9-18 7:20	98.85		3820	3820	10.10	129	53.4	0.05	0.01	30	3,820	9.70	128	63.5	0.01	0.01	26				
11-10-18 0:30	99.56		1260	1260	10.10	132	60.9	0.04	0.01	28	1,260	9.80	141	58.3	0.01	0.01	30				
11-11-18 0:01	100.54		1020	1020	9.60	130	57.2	0.03	0.01	29	1,020	9.30	135	65.0	0.01	0.01	24				
11-12-18 0:01	101.54		2800	2800	10.10	100	66.1	0.03	0.01	29	2,800	9.40	125	40.1	0.01	0.01	15				

Soln. Start	Cycle	Preg. Solution Data									Carbon Effluent Data							Carbon			
Date	(Days)	Temp.	End	VOl	pH	ORP	WAD CN	Au	Ag	Cu	VOl	pH	ORP	WAD CN	Au	Ag	Cu	Carbon	Carbon	Au	Ag
8-2-18 11:00	End 24 hr	Deg. C	wt, gms	(mls)		mv	mg/l CN	(ppm)	(ppm)	(ppm)	(mls)		mv	mg/l CN	(ppm)	(ppm)	(ppm)	No.	dry gms	gm/t	gm/t
11-13-18 7:15	102.84		5020	5020	10.40	76	60.4	0.04	0.01	27	5,020	9.50	119	65.6	0.01	0.01	23				
11-14-18 6:20	103.81		2920	2920	10.40	125	58.3	0.02	0.01	26	2,920	9.90	127	57.2	0.01	0.01	27				
11-15-18 7:24	104.85		3020	3020	10.40	79	55.2	0.01	0.01	28	3,020	9.50	118	17.8	0.01	0.01	10				
11-16-18 5:00	105.75		2920	2920	10.40	267	47.9	0.01	0.01	34	2,920	9.40	235	19.5	0.01	0.01	12				
11-17-18 10:35	106.98		2180	2180	10.20	264	72.9	0.02	0.01	32	2,180	8.90	270	8.7	0.01	0.01	4				
11-18-18 11:56	108.04		3240	3240	9.60	205	71.8	0.02	0.01	23	3,240	9.10	199	28.6	0.01	0.01	20				
11-19-18 7:30	108.85		1480	1480	10.00	170	73.9	0.02	0.01	31	1,480	9.30	180	16.7	0.01	0.01	8				
11-20-18 7:00	109.83		2000	2000	10.20	247	49.4	0.02	0.01	32	2,000	9.30	247	23.6	0.01	0.01	13				
11-21-18 7:10	110.84		3120	3120	11.20	104	42.7	0.01	0.01	31	3,120	9.80	136	22.8	0.01	0.01	17				
11-22-18 7:30	111.85		3340	3340	11.40	198	33.3	0.01	0.01	27	3,340	10.30	198	29.0	0.01	0.01	22				
11-23-18 6:05	112.80		4440	4440	11.40	122	30.1	0.01	0.01	25	4,440	10.50	152	29.3	0.01	0.01	22				
11-24-18 10:20	113.97		2220	2220	11.20	189	20.0	0.01	0.01	16	2,220	10.60	198	20.4	0.01	0.01	22				
11-25-18 0:01	114.54		500	500	10.70	174	17.3	0.01	0.01	13	500	10.50	178	20.7	0.01	0.01	14				
11-26-18 8:00	115.88		960	960	10.40	173	13.7	0.01	0.01	14	960	9.90	185	10.2	0.01	0.01	7				
11-27-18 7:30	116.85		460	460	9.80	217	17.4	0.01	0.01	14	460	9.60	266	13.9	0.01	0.01	7				
11-28-18 7:40	117.86		360	360	9.80	121	17.1	0.01	0.01	14	360	9.50	118	7.7	0.01	0.01	4				
11-29-18 10:30	118.98		220	220	9.60	127	12.0	0.01	0.01	10	220	9.60	127	5.1	0.01	0.01	3				
11-30-18 6:40	119.82		240	240	9.70	216	15.1	0.01	0.01	12	240	9.70	225	7.1	0.01	0.01	3	Fn	152.4		
12-1-18 11:40	121.03		220	220	9.50	138	8.0	0.01	0.01	11	220										
12-2-18 11:45	122.03		320	320	9.80	127	10.6	0.01	0.01	10	320										
12-3-18 7:40	122.86		600	600	10.30	118	8.5	0.01	0.01	8	600										
12-4-18 6:40	123.82		580	580	10.10	183	7.9	0.01	0.01	7	580										
12-5-18 5:05	124.75		2180	2180	10.80	185	7.1	0.01	0.01	6	2,180										
12-6-18 6:30	125.81		5340	5340	10.80	105	1.6	0.01	0.01	1	5,340										
12-7-18 6:30	126.81		4420	4420	11.00	142	0.7	0.01	0.01	0	4,420										
12-8-18 6:30	127.81																				
12-9-18 16:24	129.22		4240		10.80	168	0.4	0.01	0.01	0											
12-10-18 5:10	129.76		1200		10.00	184	1.0	0.01	0.01	0											
12-11-18 6:30	130.81		380					0.01	0.01	0											
12-12-18 6:30	131.81																				
12-13-18 6:30	132.81																				

APPENDIX C – ASSAYS

APPENDIX C
ASSAYS

Reference
Kemetco Research Analysis Report, August 8, 2018
Activation Laboratories Ltd. Certificate of Analysis A18-09455
Activation Laboratories Ltd. Certificate of Analysis A18-09915

TABLE C-1
HEAD ASSAYS

Sample	Au	Fe	S	Cu	Ag
	ppm	%	%	%	ppm
Master Comp Head 1	0.88	2.3	0.13	0.007	1
Master Comp Head 2	0.53	2.3	0.11	0.007	1
Master Comp Head 3	0.97				
Master Comp Head 4	0.60				
Master Comp Head 5	1.24				
Master Comp Head 6	2.10				
Master Comp Head 7	0.58				
Master Comp Head 8	0.66				
Average	0.95	2.3	0.12	0.007	1
Screen Metallics	0.84				

TABLE C-2A
ASSAY BY SIZE AND CLASS

Sample ID	Size Fraction µm	Mass g	Mass %	Mass Cum %	Assay - g/tonne Gold	Assay - g/tonne Silver	Assay - g/tonne Copper	Distribution - percent Gold	Distribution - percent Silver	Distribution - percent Copper
MC - Coarse Crush (-31.5mm) Feed	26500	3380.0	11.4	88.6	1.25	0.5	20	11.2	11.4	3.4
	19000	8740.0	29.5	59.1	1.50	0.5	70	34.6	29.5	30.7
	12500	5800.0	19.6	39.6	1.25	0.5	50	19.1	19.6	14.6
	9500	1900.0	6.4	33.2	1.27	0.5	90	6.4	6.4	8.6
	6300	2600.0	8.8	24.4	1.17	0.5	70	8.0	8.8	9.1
	1700	4060.0	13.7	10.7	0.97	0.5	80	10.4	13.7	16.3
	600	1480.0	5.0	5.7	0.72	0.5	80	2.8	5.0	5.9
	425	320.0	1.1	4.7	2.57	0.5	90	2.2	1.1	1.4
	300	260.0	0.9	3.8	1.16	0.5	90	0.8	0.9	1.2
	212	220.0	0.7	3.0	1.15	0.5	100	0.7	0.7	1.1
	150	200.0	0.7	2.4	2.23	0.5	130	1.2	0.7	1.3
	106	220.0	0.7	1.6	1.84	0.5	150	1.1	0.7	1.7
	-106	480.0	1.6		1.32	0.5	190	1.7	1.6	4.6
Calc Head		29660.0	100.0		1.28	0.5	67	100.0	100.0	100.0
Assay Head					0.95	1.0	70			
Calc K80	24047									

* Ag values of 0.1 are estimated, actual assay value is <0.2g/t.

Particle Size Distribution

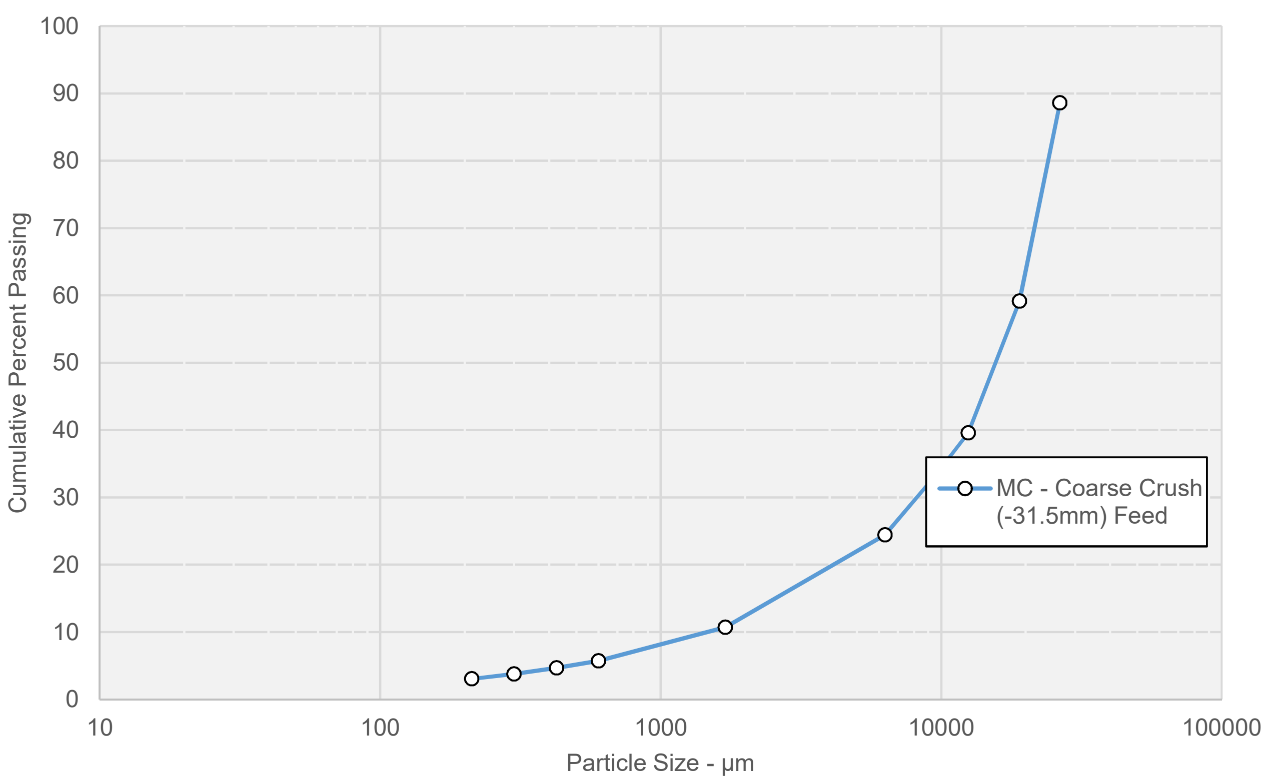
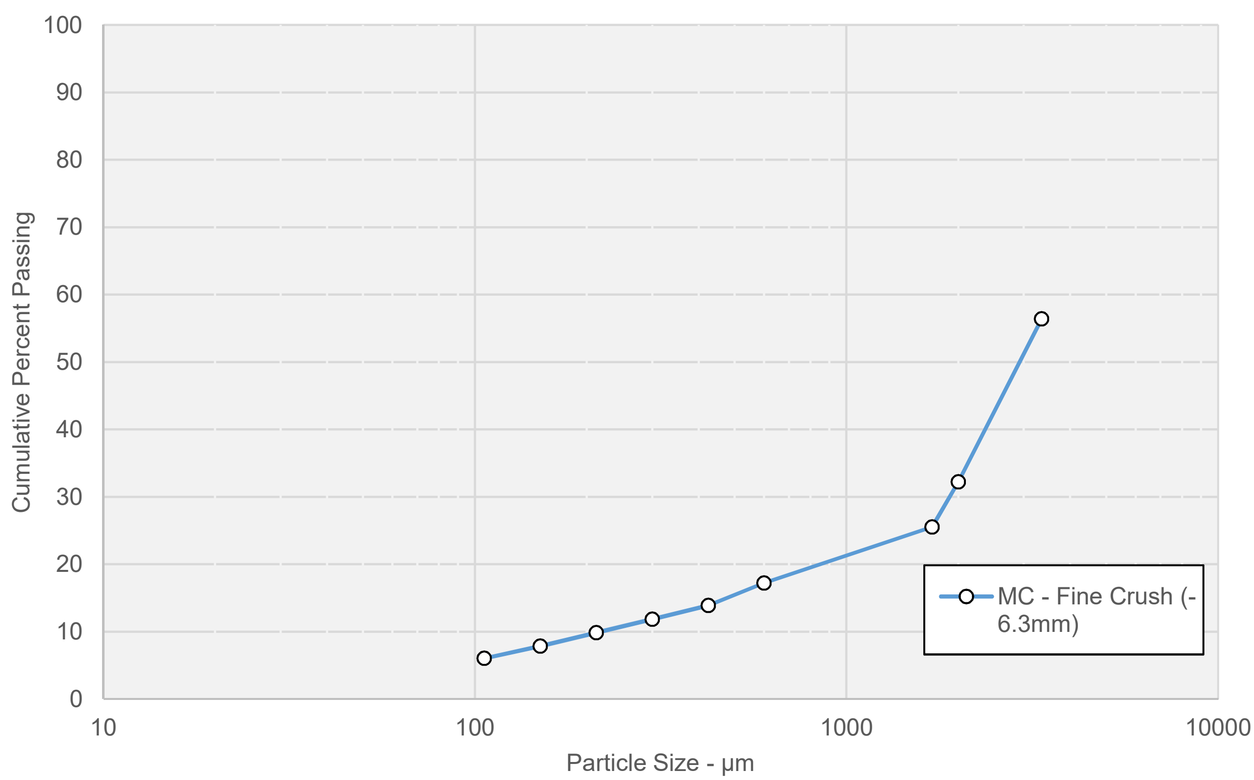


TABLE C-2B
ASSAY BY SIZE AND CLASS

Sample ID	Size Fraction	Mass			Assay - g/tonne			Distribution - percent		
	µm	g	%	Cum %	Gold	Silver	Copper	Gold	Silver	Copper
MC - Fine Crush (-6.3mm)	3350	3900.0	43.6	56.4	0.37	0.5	60	21.9	43.6	33.6
	2000	2160.0	24.2	32.2	1.33	0.5	80	43.9	24.2	24.8
	1700	600.0	6.7	25.5	0.59	0.5	60	5.4	6.7	5.2
	600	740.0	8.3	17.2	0.62	0.5	60	7.1	8.3	6.4
	425	300.0	3.4	13.9	0.46	0.5	80	2.1	3.4	3.5
	300	180.0	2.0	11.9	0.80	0.5	90	2.2	2.0	2.3
	212	180.0	2.0	9.8	0.48	0.5	100	1.3	2.0	2.6
	150	180.0	2.0	7.8	0.94	0.5	110	2.6	2.0	2.8
	106	160.0	1.8	6.0	1.86	0.5	140	4.5	1.8	3.2
	-106	540.0	6.0		1.09	0.5	200	9.0	6.0	15.5
Feed		8940.0	100.0		0.73	0.5	78	100.0	100.0	100.0
Assay Head					0.95	1.0	70			
Calc K80	4927									

* Ag values of 0.1 are estimated, actual assay value is <0.2g/t.

Particle Size Distribution



<u>TABLE C-2C</u>

<u>ASSAY BY SIZE AND CLASS</u>

Sample ID	Size Fraction µm	Mass g	Mass %	Mass Cum %	Assay - g/tonne Gold	Assay - g/tonne Silver	Assay - g/tonne Copper	Distribution - percent Gold	Distribution - percent Silver	Distribution - percent Copper
CL-1 MC - Coarse Crush (-31.5mm) Residue	26500	1160.0	11.7	88.3	0.56	0.2	87	20.1	11.7	15.5
	19000	2240.0	22.6	65.7	0.16	0.2	53	11.1	22.6	18.4
	12500	1500.0	15.2	50.5	0.19	0.2	40	8.8	15.2	9.3
	9500	740.0	7.5	43.0	0.74	0.2	40	16.9	7.5	4.6
	6300	860.0	8.7	34.3	0.72	0.2	50	19.1	8.7	6.6
	1700	1780.0	18.0	16.4	0.33	0.2	60	18.2	18.0	16.5
	600	720.0	7.3	9.1	0.16	0.2	67	3.6	7.3	7.4
	425	160.0	1.6	7.5	0.07	0.2	90	0.3	1.6	2.2
	300	140.0	1.4	6.1	0.12	0.2	107	0.5	1.4	2.3
	212	100.0	1.0	5.1	0.09	0.2	133	0.3	1.0	2.1
	150	220.0	2.2	2.8	0.08	0.2	167	0.5	2.2	5.7
	106	60.0	0.6	2.2	0.05	0.2	210	0.1	0.6	1.9
	-106	220.0	2.2		0.07	0.2	220	0.5	2.2	7.5
Calc Head		9900.0	100.0		0.33	0.2	65.43	100.0	100.0	100.0
Assay Head					0.95	1.0	70			
Calc K80	23461									

* Ag values of 0.2 are estimated, actual assay value is <1g/t.

<u>Particle Size Distribution</u>

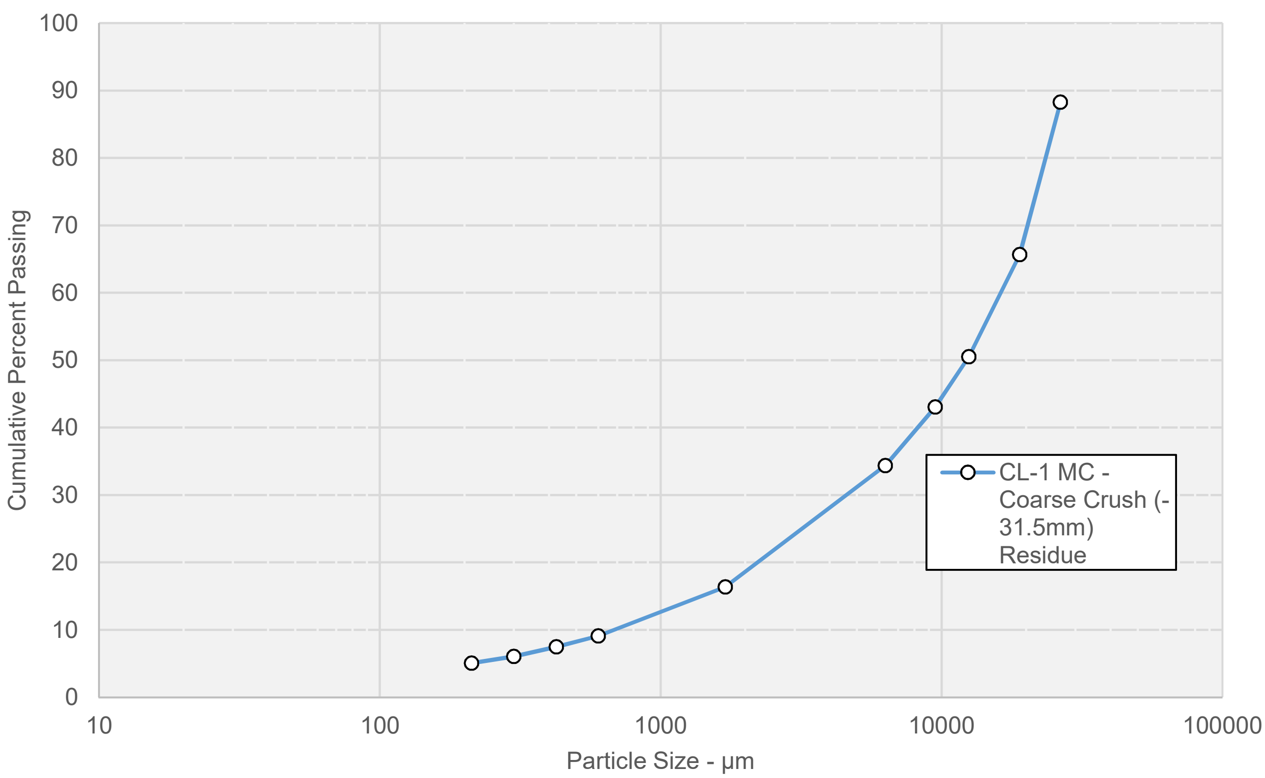


TABLE C-2D
ASSAY BY SIZE AND CLASS

Sample ID	Size Fraction	Mass			Assay - g/tonne			Distribution - percent		
	µm	g	%	Cum %	Gold	Silver	Copper	Gold	Silver	Copper
CL-2 MC - Fine Crush (-6.3mm) Residue	3350	3011.2	33.4	66.6	0.11	0.2	76	39.8	33.4	33.0
	2000	2276.5	25.3	41.3	0.09	0.2	52	23.7	25.3	16.9
	1700	416.3	4.6	36.6	0.31	0.2	54	15.1	4.6	3.2
	600	1215.0	13.5	23.2	0.07	0.2	71	9.9	13.5	12.4
	425	407.5	4.5	18.6	0.04	0.2	66	1.9	4.5	3.9
	300	299.3	3.3	15.3	0.06	0.2	84	2.1	3.3	3.6
	212	274.1	3.0	12.3	0.05	0.2	93	1.7	3.0	3.7
	150	167.7	1.9	10.4	0.05	0.2	120	0.9	1.9	2.9
	106	225.2	2.5	7.9	0.03	0.2	138	0.8	2.5	4.5
	-106	711.0	7.9		0.05	0.2	156	4.1	7.9	15.9
Calc Head		9003.8	100.0		0.10	0.2	77	100.0	100.0	100.0
Assay Head					0.95	1.0	70			
Calc K80	4457									

* Ag values of 0.2 are estimated, actual assay value is <1g/t.

Particle Size Distribution

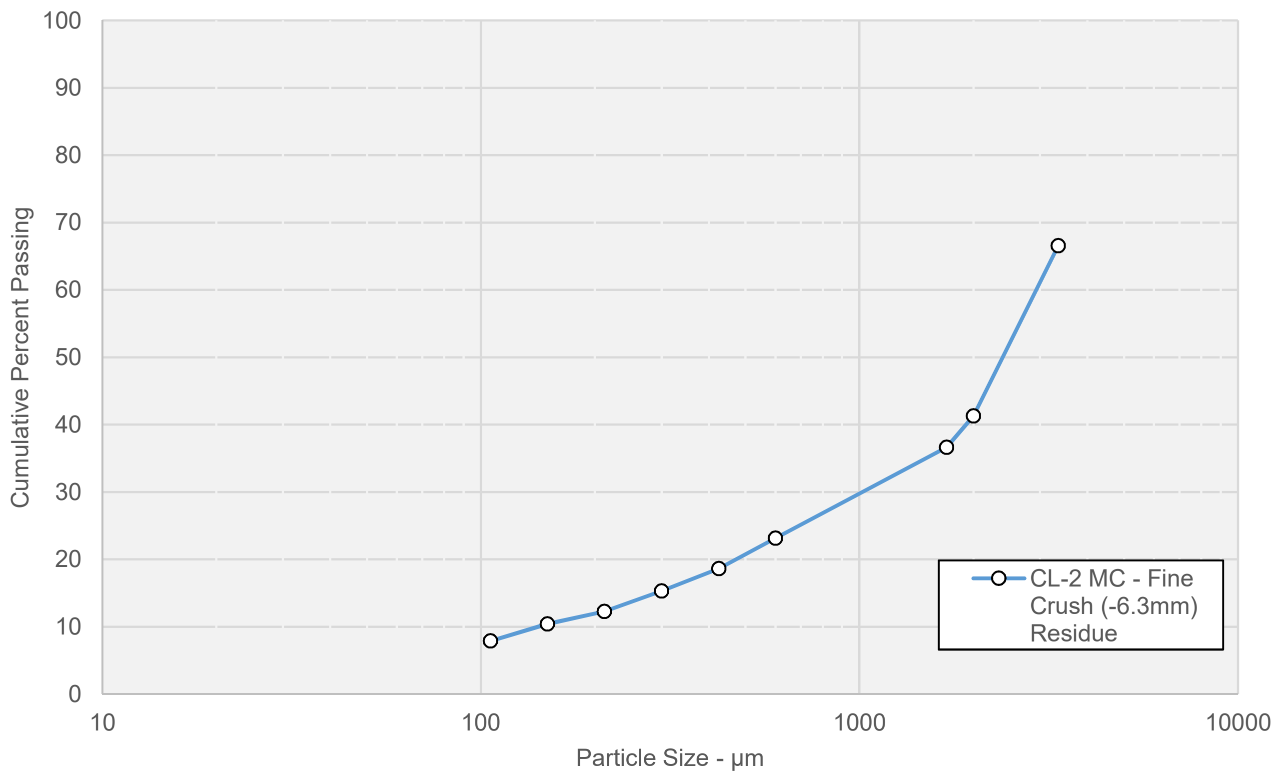


TABLE C-3
RECALCULATED HEAD ASSAYS
Master Composite

Test	Assays - g/tonne		
	Au	Ag	Cu
1	0.93	0.7	83
2	1.48	0.7	96
3	0.83	0.6	-
4	0.61	0.6	-
5	1.29	0.7	63
6	1.14	0.7	65
7	1.22	0.6	56
8	1.37	0.6	60
9	0.91	0.7	65
10	1.01	0.7	66
11	1.18	-	-
12	0.86	-	-
Average	1.07	0.7	69
Measured	0.95	1	70

ANALYSIS REPORT

Date of ICP Analysis: 02-Aug-18

DATE of Report: 08-Aug-18

Purchase Order

APPROVED BY:

KEMETCO Research

#150 – 13260 Delf Place
Richmond, BC V6V 2A2
Tel.: 604-273-3600 ext. 226

Base Metallurgical Laboratories Ltd.
4-1425 Cariboo Place
Kamloops, BC, V2C 5Z3, Canada
Office: 1 250 314 4046
CONTACT: Bradley Angove, brad@basemetlabs.com

COMMENTS: CN matrix samples

METHODS: ICP-OES Analysis of Total & Dissolved Metals in Water and Wastewater, 5240/5245 v.3.1

Kemetco ID	**DL**	180802D-01	180802D-02						
Client ID		Base Met-BL295-01-96hr	Base Met-BL295-02-96hr						
ELEMENTS	**mg/L**	mg/L	mg/L						
Ag Silver	0.1	<0.1	0.12						
Al Aluminium	0.4	10.8	6.69						
As Arsenic	0.4	<0.4	<0.4						
Au Gold	0.2	0.54	0.79						
B Boron	1	<1.	<1.						
Ba Barium	0.02	0.04	0.04						
Be Beryllium	0.02	<0.02	<0.02						
Bi Bismuth	0.5	<0.4	<0.4						
Ca Calcium	0.2	2.21	1.45						
Cd Cadmium	0.02	<0.02	<0.02						
Co Cobalt	0.1	0.11	0.15						
Cr Chromium	0.1	<0.1	0.11						
Cu Copper	0.2	7.99	9.68						
Fe Iron	0.2	14.5	77.8						
K Potassium	1	17.4	25.3						
Li Lithium	0.2	<0.2	<0.2						
Mg Magnesium	0.2	4.58	0.93						
Mn Manganese	0.02	0.05	0.74						
Mo Molybdenum	0.2	<0.2	<0.2						
Na Sodium	1	507	2473						
Ni Nickel	0.1	0.61	0.89						
Pb Lead	0.4	<0.4	<0.4						
Sb Antimony	0.4	<0.4	<0.4						
Se Selenium	0.4	<0.4	<0.4						
Si Silicon	0.5	12.1	11.1						
Sn Tin	0.4	<0.4	<0.4						
Sr Strontium	0.02	0.04	0.02						
Ti Titanium	0.2	<0.2	<0.2						
Tl Thallium	0.4	<0.4	<0.4						
U Uranium	1	<1.	<1.						
V Vanadium	0.2	0.21	0.30						
Zn Zinc	0.1	0.99	0.92						

Quality Analysis ... **Actlabs** *Innovative Technologies*

Date Submitted: 20-Jul-18
Invoice No.: A18-09455
Invoice Date: 13-Aug-18
Your Reference: 295

Base Metallurgical Laboratories Ltd.
4-1425 Cariboo Place
Kamloops BC
Canada

ATTN: Bradly Angove

CERTIFICATE OF ANALYSIS

1 Pulp samples were submitted for analysis.

The following analytical package(s) were requested: Code 4B (1-10) Major Elements Fusion ICP(WRA)

REPORT **A18-09455**

This report may be reproduced without our consent. If only selected portions of the report are reproduced, permission must be obtained. If no instructions were given at time of sample submittal regarding excess material, it will be discarded within 90 days of this report. Our liability is limited solely to the analytical cost of these analyses. Test results are representative only of material submitted for analysis.

Notes:

Total includes all elements in % oxide to the left of total.

Values which exceed the upper limit should be assayed for accurate numbers.

CERTIFIED BY:

Emmanuel Eseme , Ph.D.
Quality Control

ACTIVATION LABORATORIES LTD.
41 Bittern Street, Ancaster, Ontario, Canada, L9G 4V5
TELEPHONE +905 648-9611 or +1.888.228.5227 FAX +1.905.648.9613
E-MAIL Ancaster@actlabs.com ACTLABS GROUP WEBSITE www.actlabs.com

Date Submitted:	20-Jul-18
Invoice No.:	A18-09455
Invoice Date:	13-Aug-18
Your Reference:	295

Base Metallurgical Laboratories Ltd.
4-1425 Cariboo Place
Kamloops BC
Canada

ATTN: Bradly Angove

CERTIFICATE OF ANALYSIS

1 Pulp samples were submitted for analysis.

The following analytical package(s) were requested: Code 1E3-Kamloops Aqua Regia ICP(AQUAGEO)

REPORT **A18-09455**

This report may be reproduced without our consent. If only selected portions of the report are reproduced, permission must be obtained. If no instructions were given at time of sample submittal regarding excess material, it will be discarded within 90 days of this report. Our liability is limited solely to the analytical cost of these analyses. Test results are representative only of material submitted for analysis.

Notes:

Total includes all elements in % oxide to the left of total.

Values which exceed the upper limit should be assayed for accurate numbers.

CERTIFIED BY:

Emmanuel Eseme , Ph.D.
Quality Control

ACTIVATION LABORATORIES LTD.
9989 Dallas Drive, Kamloops, British Columbia, Canada, V2C 6T4
TELEPHONE +250 573-4484 or +1.888.228.5227 FAX +1.905.648.9613
E-MAIL Kamloops@actlabs.com ACTLABS GROUP WEBSITE www.actlabs.com

Analyte Symbol	Ag	Cd	Cu	Mn	Mo	Ni	Pb	Zn	Al	As	B	Ba	Be	Bi	Ca	Co	Cr	Fe	Ga	Hg	K	La	Mg
Unit Symbol	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	%	ppm	ppm	%	ppm	%
Lower Limit	0.2	0.5	1	5	1	1	2	2	0.01	2	10	10	0.5	2	0.01	1	1	0.01	10	1	0.01	10	0.01
Method Code	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP
BI295 Master Comp - 1.7mm	< 0.2	< 0.5	88	507	< 1	126	< 2	86	5.13	24	< 10	23	< 0.5	< 2	0.73	32	211	5.45	10	< 1	0.05	< 10	5.73

Analyte Symbol	Na	P	S	Sb	Sc	Sr	Ti	Th	Te	Tl	U	V	W	Y	Zr	SiO2	Al2O3	Fe2O3(T)	MnO	MgO	CaO	Na2O	K2O
Unit Symbol	%	%	%	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	%	%	%	%	%	%	%	%
Lower Limit	0.001	0.001	0.01	2	1	1	0.01	20	1	2	10	1	10	1	1	0.01	0.01	0.01	0.001	0.01	0.01	0.01	0.01
Method Code	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP
BI295 Master Comp - 1.7mm	0.018	0.026	0.09	3	18	4	0.01	< 20	< 1	< 2	< 10	151	< 10	2	3	52.00	17.33	8.29	0.084	11.70	1.21	0.27	1.03

Analyte Symbol	TiO2	P2O5	LOI	Total	Ba	Sr	Y	Sc	Zr	Be	V
Unit Symbol	%	%	%	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Lower Limit	0.001	0.01		0.01	2	2	1	1	2	1	5
Method Code	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP
BI295 Master Comp - 1.7mm	0.953	0.06	6.95	99.88	91	30	15	41	59	< 1	289

Analyte Symbol	Ag	Cd	Cu	Mn	Mo	Ni	Pb	Zn	Al	As	B	Ba	Be	Bi	Ca	Co	Cr	Fe	Ga	Hg	K	La	Mg
Unit Symbol	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	%	ppm	ppm	%	ppm	%
Lower Limit	0.2	0.5	1	5	1	1	2	2	0.01	2	10	10	0.5	2	0.01	1	1	0.01	10	1	0.01	10	0.01
Method Code	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP
DNC-1 Meas																							
DNC-1 Cert																							
GXR-6 Meas	0.2	0.5	67	935	1	23	84	116	6.75	190	< 10	880	0.8	< 2	0.17	11	75	5.47	20	3	1.10	< 10	0.40
GXR-6 Cert	1.30	1.00	66.0	1010	2.40	27.0	101	118	17.7	330	9.80	1300	1.40	0.290	0.180	13.8	96.0	5.58	35.0	0.0680	1.87	13.9	0.609
W-2a Meas																							
W-2a Cert																							
SY-4 Meas																							
SY-4 Cert																							
BIR-1a Meas																							
BIR-1a Cert																							
OREAS 904 (Aqua Regia) Meas	0.3	< 0.5	6100	414	1	31	8	25	1.83	87		69	7.7	< 2	0.04	88	25	6.10	< 10		0.88	40	0.20
OREAS 904 (Aqua Regia) Cert	0.366	0.0580	6300	410	2.02	36.6	8.49	22.4	1.25	91.0		68.0	6.54	3.74	0.0404	82.0	17.5	6.40	3.40		0.603	33.9	0.143
OREAS 922 (AQUA REGIA) Meas	1.0	0.7	2240	733	< 1	33	62	268	2.82	7		68	0.7	3	0.40	19	46	5.10	< 10		0.46	36	1.39
OREAS 922 (AQUA REGIA) Cert	0.851	0.28	2176	730	0.69	34.3	60	256	2.72	6.12		70	0.65	10.3	0.324	19.4	40.7	5.05	7.62		0.376	32.5	1.33
OREAS 923 (AQUA REGIA) Meas	1.5	< 0.5	4500	838	< 1	32	77	345	2.84	7		53	0.7	8	0.40	21	42	6.07	< 10		0.38	34	1.53
OREAS 923 (AQUA REGIA) Cert	1.62	0.40	4248	850	0.84	32.7	81	335	2.80	7.07		54	0.61	21.8	0.326	22.2	39.4	5.91	8.01		0.322	30.0	1.43
OREAS 907 (Aqua Regia) Meas	1.1	0.6	6320	320	4	5	33	150	1.18	35		204	1.1	10	0.28	44	9	7.81	20		0.36	38	0.23
OREAS 907 (Aqua Regia) Cert	1.30	0.540	6370	330	5.64	4.74	34.1	139	0.945	37.0		225	0.870	22.3	0.280	43.7	8.59	8.18	14.7		0.286	36.1	0.221
Oreas 621 (Aqua Regia) Meas	66.7	249	3570	511	12	25	> 5000	> 10000	1.78	75			0.6	3	1.42	29	35	3.38	< 10	5	0.37	18	0.45
Oreas 621 (Aqua Regia) Cert	68.0	278	3660	520	13.3	25.8	13600	51700	1.60	75.0			0.530	3.85	1.65	27.9	31.3	3.43	9.29	3.93	0.333	19.4	0.436
BI295 Master Comp - 1.7mm Orig	< 0.2	< 0.5	87	502	1	125	< 2	84	5.09	23	< 10	22	< 0.5	< 2	0.70	32	210	5.40	10	< 1	0.05	< 10	5.67
BI295 Master Comp - 1.7mm Dup	< 0.2	< 0.5	88	512	< 1	127	< 2	88	5.17	25	< 10	24	< 0.5	< 2	0.75	32	213	5.51	10	6	0.05	< 10	5.78
Method Blank	< 0.2	< 0.5	< 1	< 5	< 1	< 1	< 2	< 2	< 0.01	< 2	< 10	< 10	< 0.5	< 2	< 0.01	< 1	< 1	< 0.01	< 10	< 1	< 0.01	< 10	< 0.01
Method Blank																							

Analyte Symbol	Na	P	S	Sb	Sc	Sr	Ti	Th	Te	Tl	U	V	W	Y	Zr	SiO2	Al2O3	Fe2O3(T)	MnO	MgO	CaO	Na2O	K2O
Unit Symbol	%	%	%	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	%	%	%	%	%	%	%	%
Lower Limit	0.001	0.001	0.01	2	1	1	0.01	20	1	2	10	1	10	1	1	0.01	0.01	0.01	0.001	0.01	0.01	0.01	0.01
Method Code	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	AR-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP
DNC-1 Meas																47.40	18.31	9.84	0.140	10.06	11.29	1.91	0.23
DNC-1 Cert																47.15	18.34	9.97	0.150	10.13	11.49	1.890	0.234
GXR-6 Meas	0.102	0.031	0.01	2	17	28		< 20	< 1	< 2	< 10	159	< 10	3	7								
GXR-6 Cert	0.104	0.0350	0.0160	3.60	27.6	35.0		5.30	0.0180	2.20	1.54	186	1.90	14.0	110								
W-2a Meas																53.42	15.26	10.78	0.170	6.30	10.90	2.24	0.64
W-2a Cert																52.4	15.4	10.7	0.163	6.37	10.9	2.14	0.626
SY-4 Meas																51.59	20.43	6.14	0.110	0.50	8.00	6.86	1.65
SY-4 Cert																49.9	20.69	6.21	0.108	0.54	8.05	7.10	1.66
BIR-1a Meas																48.77	15.55	11.37	0.170	9.58	13.31	1.81	0.02
BIR-1a Cert																47.96	15.50	11.30	0.175	9.700	13.30	1.82	0.030
OREAS 904 (Aqua Regia) Meas		0.095	0.04	< 2	5	15		< 20		< 2	< 10	32		15									
OREAS 904 (Aqua Regia) Cert		0.0950	0.0340	0.780	3.83	16.5		7.56		0.150	5.20	21.7		17.2									
OREAS 922 (AQUA REGIA) Meas	0.030	0.062	0.37	2	4	13		< 20		< 2	< 10	35	< 10	15	22								
OREAS 922 (AQUA REGIA) Cert	0.021	0.063	0.386	0.57	3.15	15.0		14.5		0.14	1.98	29.4	1.12	16.0	22.3								
OREAS 923 (AQUA REGIA) Meas		0.061	0.69	2	4	11		< 20		< 2	< 10	35	< 10	14	29								
OREAS 923 (AQUA REGIA) Cert		0.061	0.684	0.58	3.09	13.6		14.3		0.12	1.80	30.6	1.96	14.3	22.5								
OREAS 907 (Aqua Regia) Meas	0.100	0.021	0.06	5	2	10	0.02	< 20	< 1	< 2	< 10	7	< 10	6	6								
OREAS 907 (Aqua Regia) Cert	0.0860	0.0240	0.0660	2.28	2.16	11.7	0.0170	8.04	0.230	0.120	2.15	5.12	0.980	6.52	43.7								
Oreas 621 (Aqua Regia) Meas	0.180	0.033	4.51	120	2	12		< 20		4	< 10	13	< 10	6	58								
Oreas 621 (Aqua Regia) Cert	0.160	0.0335	4.50	107	2.20	18.9		5.91		0.770	163	10.9	1.00	6.87	55.0								
BI295 Master Comp - 1.7mm Orig	0.018	0.027	0.08	2	18	4	0.01	< 20	3	< 2	< 10	149	< 10	2	3								
BI295 Master Comp - 1.7mm Dup	0.019	0.025	0.10	3	18	4	0.01	< 20	< 1	< 2	< 10	152	< 10	2	3								
Method Blank	0.013	< 0.001	< 0.01	< 2	< 1	< 1	< 0.01	< 20	< 1	< 2	< 10	< 1	< 10	< 1	< 1								
Method Blank																< 0.01	< 0.01	< 0.01	0.007	< 0.01	< 0.01	< 0.01	< 0.01

Analyte Symbol	TiO2	P2O5	Ba	Sr	Y	Sc	Zr	Be	V
Unit Symbol	%	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Lower Limit	0.001	0.01	2	2	1	1	2	1	5
Method Code	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP
DNC-1 Meas	0.480	0.09	106	143	15	31	32		151
DNC-1 Cert	0.480	0.070	118	144.0	18.0	31	38		148
GXR-6 Meas									
GXR-6 Cert									
W-2a Meas	1.070	0.14	192	197	19	35	82	< 1	269
W-2a Cert	1.06	0.140	182	190	24.0	36.0	94.0	1.30	262
SY-4 Meas	0.290	0.13	346	1182	114	< 1	530	3	6
SY-4 Cert	0.287	0.131	340	1191	119	1.1	517	2.6	8.0
BIR-1a Meas	0.980	0.04	11	111	13	44	15	< 1	332
BIR-1a Cert	0.96	0.021	6	110	16	44	18	0.58	310
OREAS 904 (Aqua Regia) Meas									
OREAS 904 (Aqua Regia) Cert									
OREAS 922 (AQUA REGIA) Meas									
OREAS 922 (AQUA REGIA) Cert									
OREAS 923 (AQUA REGIA) Meas									
OREAS 923 (AQUA REGIA) Cert									
OREAS 907 (Aqua Regia) Meas									
OREAS 907 (Aqua Regia) Cert									
Oreas 621 (Aqua Regia) Meas									
Oreas 621 (Aqua Regia) Cert									
Bl295 Master Comp - 1.7mm Orig									
Bl295 Master Comp - 1.7mm Dup									
Method Blank									
Method Blank	< 0.001	< 0.01	< 2	< 2	< 1	< 1	< 2	< 1	< 5

Quality Analysis ... **Actlabs** *Innovative Technologies*

Date Submitted:	25-Jul-18
Invoice No.:	A18-09915
Invoice Date:	09-Aug-18
Your Reference:	295

Base Metallurgical Laboratories Ltd.
4-1425 Cariboo Place
Kamloops BC
Canada

ATTN: Bradly Angove

CERTIFICATE OF ANALYSIS

1 Pulp samples were submitted for analysis.

The following analytical package(s) were requested: Code 1A4-1000 (150mesh) - Kamloops Au-Fire Assay-Metallic Screen-1000g

REPORT **A18-09915**

Notes:

A representative 1000 gram split is seived at 150 mesh (105 micron) with assays performed on the entire +150 mesh and 2 splits of the -150 mesh fraction. A final assay is calculated on the weight of each fraction.

CERTIFIED BY:

Emmanuel Eseme , Ph.D.
Quality Control

ACTIVATION LABORATORIES LTD.
9989 Dallas Drive, Kamloops, British Columbia, Canada, V2C 6T4
TELEPHONE +250 573-4484 or +1.888.228.5227 FAX +1.905.648.9613
E-MAIL Kamloops@actlabs.com ACTLABS GROUP WEBSITE www.actlabs.com

Analyte Symbol	Au + 150 mesh	Au - 150 mesh (A)	Au - 150 mesh (B)	Total Au	+ 150 mesh	- 150 mesh	Total Weight
Unit Symbol	g/mt	g/mt	g/mt	g/mt	g	g	g
Lower Limit	0.03	0.03	0.03	0.03			
Method Code	FA-MeT	FA-MeT	FA-MeT	FA-MeT	FA-MeT	FA-MeT	FA-MeT
BL295 MC	7.34	0.63	0.73	0.84	22.06	908.00	930.06

Analyte Symbol	Total Au
Unit Symbol	g/mt
Lower Limit	0.03
Method Code	FA-MeT
OxQ90 Meas	24.7
OxQ90 Cert	24.88
OXN117 Meas	7.70
OXN117 Cert	7.679

APPENDIX D – COMMINUTION

APPENDIX D
COMMINUTION TESTING

Reference
SGS Minerals "Sandard Bond Rod Mill Grindability Test", Report 16682-01, July 26, 2018
JKTech Pty "SMC Test Report", Report 18008/P5, July 2018

TABLE D-1
BOND ABRASION TEST
BL295 Master Composite

Original Paddle Weight: 95.31 g
Final Paddle Weight: 95.17 g
Abrasion Index, A_i: 0.14

Product Sizing µm	Weight g	Weight % Retained	Cumulative % Passing
16000	0	0.00	100.0
12500	205	12.81	87.2
11200	150	9.35	77.8
9500	305	19.09	58.7
6300	322	20.16	38.6
4750	70	4.40	34.2
3360	57	3.59	30.6
<3360	489	30.60	
TOTAL	**1598**		**

K80 = **11502**
Total Feed Weight (g) **1600**
Lost Weight (g) **2**

Particle Size Distribution Plot

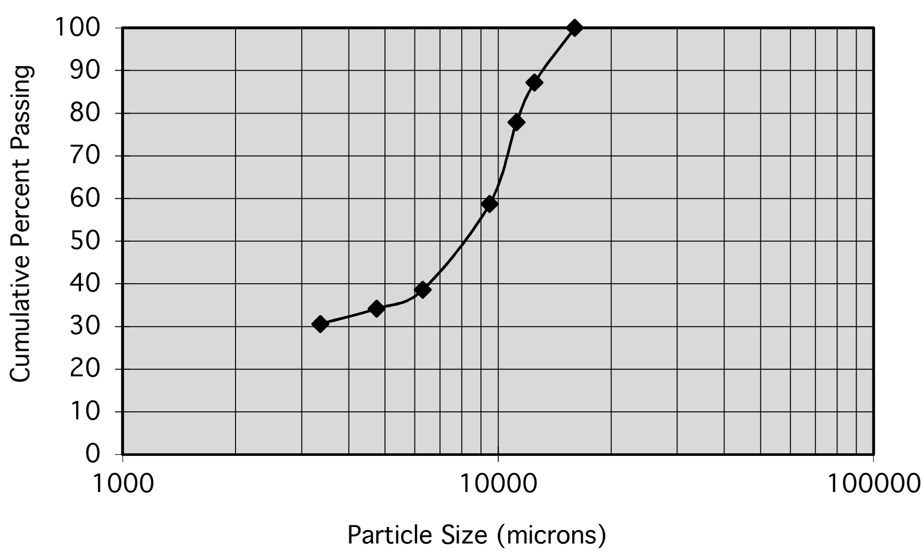


TABLE D-2A

BOND BALL MILL WORK INDEX DETERMINATION TEST

BL295 - Master Composite

Weight of 700 ml Sample	1394.8 g	Aperture Test Sieve :	106µm
1/3.5 of Sample Weight :	398.5 g	Percent Undersize :	21.7%

Cycle	Weight of New Feed	Number of Revolutions	Weight of Oversize	Weight of Undersize			
				Product	Feed	Net Product	Net/Rev
1	1394.8	150	977.8	417.0	302.0	115.0	0.77
2	417.0	402	886.2	508.6	90.3	418.3	1.04
3	508.6	277	964.5	430.3	110.1	320.2	1.16
4	430.3	264	985.2	409.6	93.2	316.4	1.20
5	409.6	259	1001.0	393.8	88.7	305.1	1.18

BOND'S WORK INDEX FORMULA

$$Wi = (44.5 \times 1.102) / (Pi^{.23} \times Gpb^{.82} \times (10/\sqrt{P} - 10/\sqrt{F}))$$

Pi = Sieve Size Tested	106	µm
Gbp = Net undersize produced per revolution of mill.	1.18	g
P = 80% Passing size of test product.	75	µm
F = 80% Passing size of test feed.	1838	µm

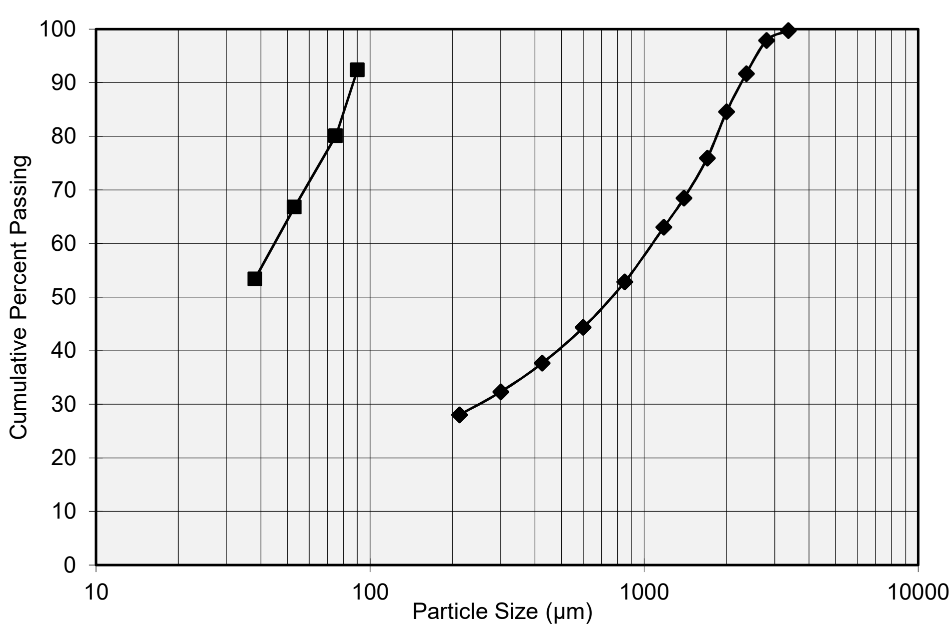
BOND BALL WORK INDEX (Wi) 15.9 kw-hr/tonne

TABLE D-2B
BOND BALL MILL WORK INDEX SIZINGS
BL295 - Master Composite

Particle Size		Feed to Cycle 1			Equilibrium Cycle Undersize		
mesh	µm	Weight (g) Retained	Weight % Retained	Cumulative % Passing	Weight (g) Retained	Weight % Retained	Cumulative % Passing
6 Mesh	3360	0.50	0.25	99.8	-	-	-
7 Mesh	2800	3.70	1.85	97.9	-	-	-
8 Mesh	2360	12.40	6.20	91.7	-	-	-
9 Mesh	2000	14.20	7.10	84.6	-	-	-
10 Mesh	1700	17.30	8.65	76.0	-	-	-
12 Mesh	1400	14.90	7.45	68.5	-	-	-
14 Mesh	1180	10.90	5.45	63.1	-	-	-
20 Mesh	850	20.40	10.20	52.9	-	-	-
28 Mesh	600	17.00	8.50	44.4	-	-	-
35 Mesh	425	13.30	6.65	37.7	-	-	-
48 Mesh	300	10.70	5.35	32.4	-	-	-
65 Mesh	212	8.60	4.30	28.1	-	-	-
100 Mesh	150	7.30	3.65	24.4	-	-	-
150 Mesh	106	5.50	2.75	21.7	-	-	-
170 Mesh	90	-	-	-	7.60	7.60	92.4
200 Mesh	75	-	-	-	12.30	12.30	80.1
270 Mesh	53	-	-	-	13.30	13.30	66.8
400 Mesh	38	-	-	-	13.40	13.40	53.4
TOTAL		200.0	100.00	**	100.0	100.00	**

K80 = 1838µm **K80 = 75µm**

Particle Size Distribution Plot



SGS Minerals Services

Standard Bond Rod Mill Grindability Test

Project No.:	16682-01	Date:	26-Jul-18
Sample.:	BL295 Master Comp	Laboratory:	Vancouver (Canada)

Purpose: To determine the rod mill grindability of the sample in terms of a Bond work index number.

Procedure: The equipment and procedure duplicate the Bond method for determining rod mill work indices.

Test Conditions:

Feed 100% Passing	0.5	inch
Mesh of grind:	14	mesh
Test feed weight (1250 mL):	2,535	grams
Equivalent to : 2,028	kg/m³	at Minus 1/2"
Weight % of the undersize material in the rod mill feed:	27.6%	
Weight of undersize product for 100% circulating load:	1,268	grams

Results: Gram per Rev Average for the Last Three Stages = **7.85 g**
Circulation load = **99%**

CALCULATION OF A BOND WORK INDEX

$$RWI = \frac{62}{P1^{0.23} \times Grp^{0.625} \times \left\{ \frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}} \right\}}$$

P1 = 100% passing size of the product	1,180	microns
Grp = Grams per revolution	7.85	grams
P_{80} = 80% passing size of product	912	microns
F_{80} = 80% passing size of the feed	9,592	microns

RWI = **14.7** kWh/ton (Imperial)

RWI = **16.2** kWh/tonne (metric)

Comments:

Stage No.	# of Revs	New Feed (grams)	Product in Feed (grams)	Material to Be Ground (grams)	Material Passing 14 mesh in Product (grams)	Net Ground Material (grams)	Material Ground Per Mill Rev (grams)
1	50	2,535	700	568	979	279	5.58
2	123	979	270	998	1,111	841	6.84
3	126	1,111	307	961	1,260	954	7.57
4	122	1,260	348	920	1,279	931	7.63
5	120	1,279	353	915	1,284	931	7.76
6	118	1,284	354	913	1,295	940	7.97
7	114	1,295	357	910	1,249	892	7.82
		Average for Last Three Stages =			1,276 g		7.85 g

SGS Minerals Services

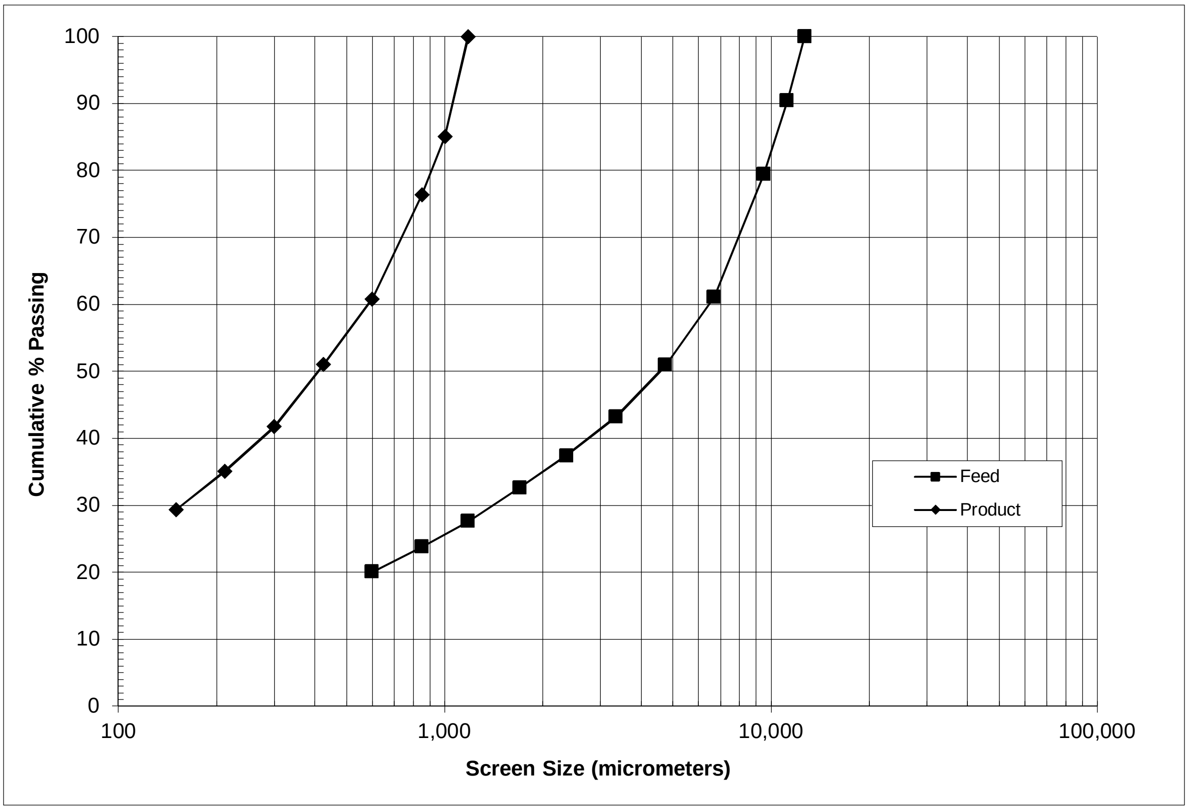
Standard Bond Rod Mill Grindability Test

Project No.: 16682-01
Sample.: BL295 Master Comp

Date: 26-Jul-18
Laboratory: Vancouver (Canada)

Feed Particle Size Analysis / **Product Particle Size Analysis**

Size		Feed Weight	Feed % Retained		Feed % Passing	Product Weight	Product % Retained		Product % Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative	grams	Individual	Cumulative	Cumulative
1/2"	12,700	0.0	0.0	0.0	100.0				
7/16"	11,200	154.3	9.6	9.6	90.4				
3/8"	9,500	177.0	11.0	20.6	79.4				
3	6,700	294.1	18.3	38.9	61.1				
4	4,750	163.5	10.2	49.1	50.9				
6	3,350	124.4	7.7	56.8	43.2				
8	2,360	93.2	5.8	62.6	37.4				
10	1,700	77.2	4.8	67.4	32.6				
14	1,180	80.4	5.0	72.4	27.6	0.0	0.0	0.0	100.0
18	1,000	-	-	-	-	52.7	14.9	14.9	85.1
20	850	61.3	3.8	76.2	23.8	30.8	8.7	23.7	76.3
28	600	60.3	3.8	80.0	20.0	54.8	15.5	39.2	60.8
35	425					34.4	9.8	49.0	51.0
48	300					32.7	9.3	58.2	41.8
65	212					23.5	6.7	64.9	35.1
100	150					20.3	5.8	70.7	29.3
Pan		322.2	20.0	100.0	-	103.5	29.3	100.0	-
Total	**-**	**1607.9**	**100.0**	**F_{80}:**	**9,592**	**352.7**	**100.0**	**P_{80}:**	**912**



SMC TEST® REPORT

Landore Resources

Tested by: Base Metallurgical Laboratories

Kamloops, BC, Canada

Prepared by: Matt Weier

JKTech Job No: 18008/P5
Testing Date: July 2018

TABLE OF CONTENTS

Page No

APPENDICES

Page No

LIST OF FIGURES

Page No

LIST OF TABLES

Page No

1 EXECUTIVE SUMMARY

1.1 SMC Results Summary

Table 1 - SMC Test® Results

Sample Designation	*DWi* (kWh/m³)	*DWi* (%)	*Mi* Parameters (kWh/t)			SG
			Mia	Mih	Mic	
Master Composite	5.1	30.0	15.8	11.1	5.7	2.67

Table 2 – Parameters derived from the SMC Test® Results

Sample Designation	A	b	A*b	t_a	SCSE (kWh/t)
Master Composite	30.5	1.71	52.2	0.51	8.74

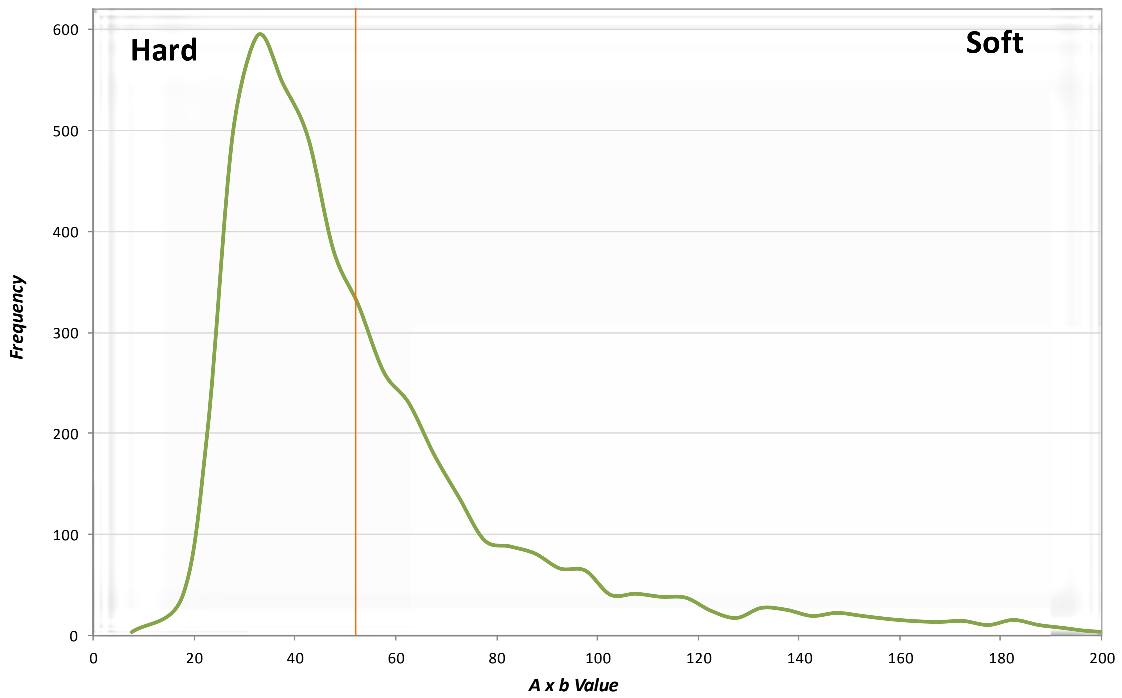


Figure 1 - Frequency Distribution of *A*b* in the JKTech Database

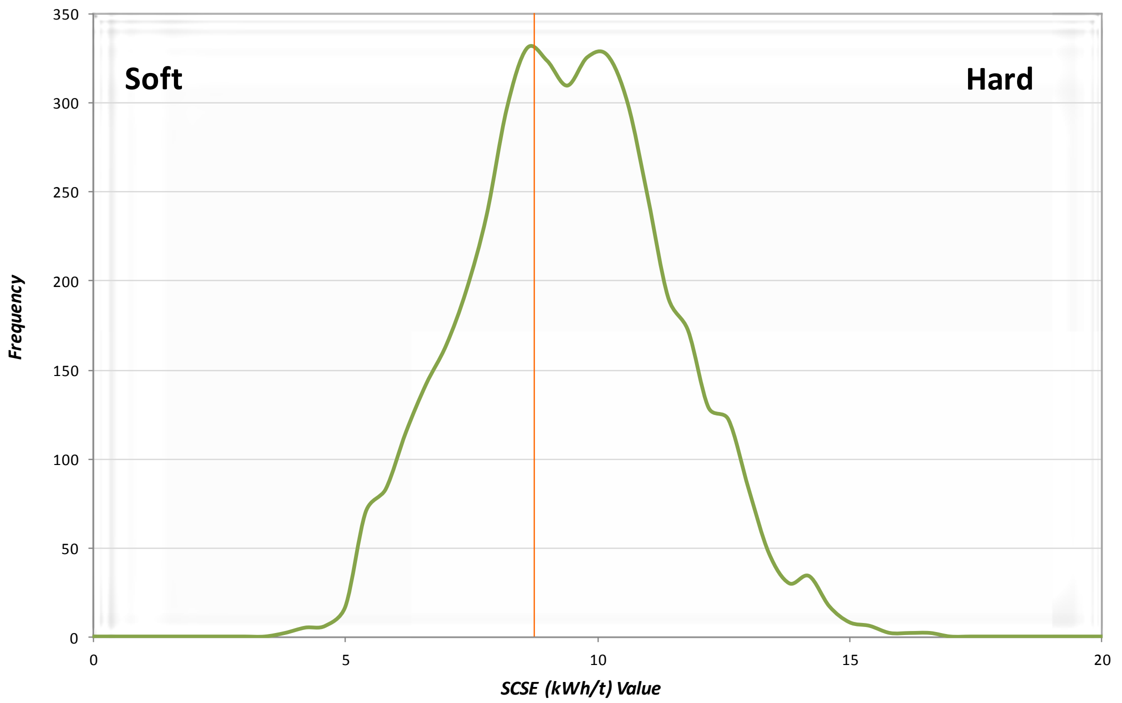


Figure 2 - Frequency Distribution of SCSE in the JKTech Database

2 INTRODUCTION

SMC data for one sample from BAM East Project were received from Base Metallurgical Laboratories on July 31, 2018, by JKTech for SMC test analysis. The sample was identified as Master Composite. The data were analysed to determine the JKSimMet and SMC Test comminution parameters. SMC Test results were forwarded to SMC Testing Pty Ltd for the analysis of the SMC Test data. Analysis and reporting were completed on August 03, 2018.

3 THE SMC TEST®

3.1 Introduction

The standard JK Drop-Weight test provides ore specific parameters for use in the JKSimMet Mineral Processing Simulator software. In JKSimMet, these parameters are combined with equipment details and operating conditions to analyse and/or predict SAG/autogenous mill performance. The same test procedure also provides ore type characterisation for the JKSimMet crusher model.

The SMC Test was developed by Steve Morrell of SMC Testing Pty Ltd (SMCT). The test provides a cost effective means of obtaining these parameters, in addition to a range of other power-based comminution parameters, from drill core or in situations where limited quantities of material are available. The ore specific parameters have been calculated from the test results and are supplied to Landore Resources in this report as part of the standard procedure

3.2 General Description and Test Background

The SMC Test® was originally designed for the breakage characterisation of drill core and it generates a relationship between input energy (kWh/t) and the percent of broken product passing a specified sieve size. The results are used to determine the so-called JK Drop-Weight index (DWi), which is a measure of the strength of the rock when broken under impact conditions and has the units kWh/m^3. The DWi is directly related to the JK rock breakage parameters A and b and hence can be used to estimate the values of these parameters as well as being correlated with the JK abrasion parameter - t_a. For crusher modelling the t_{10}-E_{cs} matrix can also be derived. This is done by using the size-by-size $A*b$ values that are used in the SMC Test® data analysis (see below) to estimate the t_{10}-E_{cs} values for each of the relevant size fractions in the crusher model matrix.

For power-based calculations, (see APPENDIX B), the SMC Test® provides the comminution parameters M_{ia}, M_{ih} and M_{ic}. M_{ia} is the work index for the grinding of coarser particles (> 750 µm) in tumbling mills such as autogenous (AG), semi-autogenous (SAG), rod and ball mills. M_{ih} is the work index for the grinding in High Pressure Grinding Rolls (HPGR) and M_{ic} for size reduction in conventional crushers.

The SMC Test® is a precision test, which uses particles that are either cut from drill core using a diamond saw to achieve close size replication or else selected from crushed material so that particle mass variation is controlled within a prescribed range. The particles are then broken at a number of prescribed impact energies. The high degree of control imposed on both the size of particles and the breakage energies used, means that the test is largely free of the repeatability problems associated with tumbling-mill based tests. Such tests usually suffer from variations in feed size (which is not closely controlled) and energy input, often assumed to be constant when in reality it can be highly variable (Levin, 1989).

The relationship between the DWi and the JK rock breakage parameters makes use of the size-by-size nature of rock strength that is often apparent from the results of full JK Drop-Weight tests. The effect is illustrated in Figure 3, which plots the normalized values of A*b against particle size. This figure also shows how the gradient of these plots varies across the full range of rock types tested. In the case of a conventional JK Drop-Weight test, these values are effectively averaged and a mean value of A and b is reported. The SMC Test® uses a single size and makes use of relationships such as that shown in Figure 3 to predict the A and b of the particle size that has the same value as the mean for a JK full Drop-Weight test.

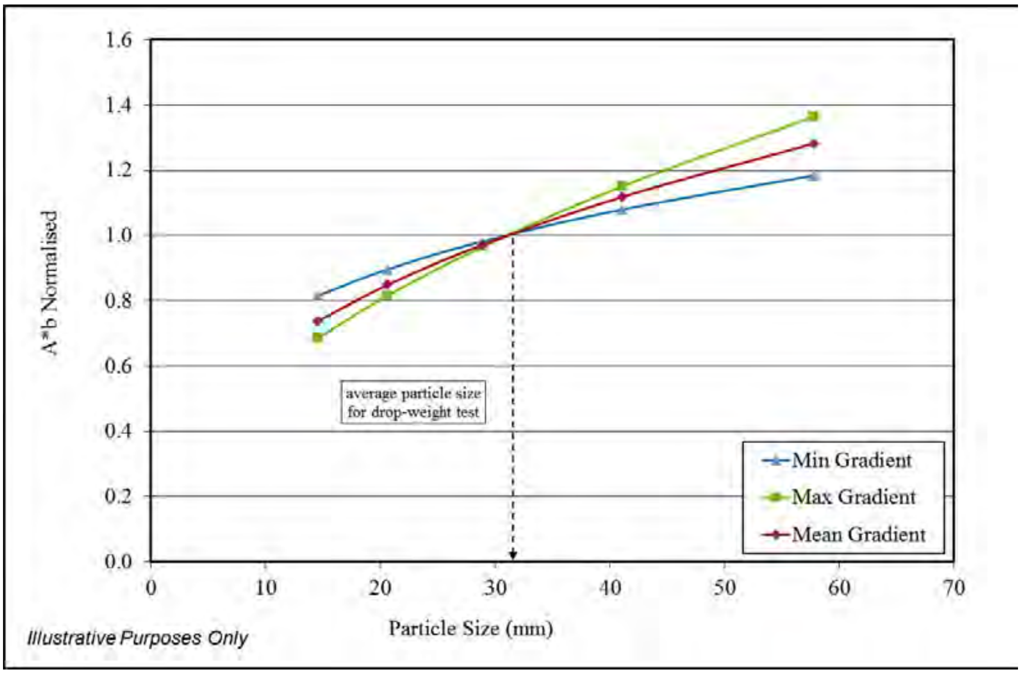


Figure 3 – Relationship between Particle Size and *A*b*

3.3 The Test Procedure

In the SMC Test®, five sets of 20 particles are broken, each set at a different specific energy level, using a JK Drop-Weight tester. The breakage products are screened at a sieve size selected to provide a direct measurement of the t_{10} value.

The test calls for a prescribed target average volume for the particles, with the target being chosen to be equivalent to the mean volume of particles in one of the standard JK Drop-Weight test size fractions.

The rest height of the drop-head (gap) is recorded after breakage of each particle to allow for a correction to the drop energy. After breaking all 20 particles in a set, the broken product is sieved at an aperture size, one tenth of the original particle size. Thus, the percent passing mass gives a direct reading of the t_{10} value for breakage at that energy level.

There are two alternative methods of preparing the particle sets for breakage testing: the particle selection method and the cut core method. The particle selection method is the most commonly used as it is generally less time consuming. The cut core method requires less material, so tends to be used as a fallback method, only when necessary to cope with restricted sample availability.

3.3.1 Particle Selection Method

For the particle selection method, the test is carried out on material in one of three alternative size fractions: -31.5+26.5, -22.4+19 or -16+13.2 mm. The largest size fraction is preferred but requires more material.

In the particle selection method, particles are chosen so that their individual masses lie within ±30% of the target mass and the mean mass for each set of 20 lies within ±10% of the target mass. A typical set of particles is shown in Figure 4.

Figure 4 – A Typical Set of Particles for Breakage (Particle Selection Method)

Before commencing breakage tests on the particles, the ore density is determined by first weighing a representative sample of particles in air and then in water.

3.3.2 Cut Core Method

The cut core method uses cut pieces of quartered (slivered) drill core. Whole core or half core can be used, but when received in this form it needs to be first quartered as a preliminary step in the procedure. Once quartered, any broken or tapered ends of the quartered lengths are cut, to square them off. Before the lengths of quartered core are cut to produce the pieces for testing, each one is weighed in air and then in water, to obtain a density measurement and a measure of its mass per unit length.

The size fraction targeted when the cut core method is used depends on the original core diameter. The target size fraction is selected to ensure that pieces of the correct volume will have “chunky” rather than “slabby” proportions.

Having measured the density of the core, the target volume can be translated into a target mass and with the average mass per unit length also known, an average cutting interval can be determined for the core.

Sufficient pieces of the quartered core are cut to generate 100 particles. These are then divided into the five sets of 20 and broken in the JK Drop-Weight tester at the five different energy levels. Within each set, the three possible orientations of the particles are equally represented (as far as possible, given that there are 20 particles). The orientations prescribed for testing are shown in Figure 5.

Figure 5 – Orientations of Pieces for Breakage (Cut Core Method)

The cut core method cannot be used for cores with diameters exceeding 70 mm, where the particle masses would be too large to achieve the highest prescribed energy level.

3.4 SMC Test® Results

The SMC Test® results for the Master Composite sample from BAM East Project are given in Table 3. This table includes the average rock density and the DWi (Drop-Weight index) that is the direct result of the test procedure. The values determined for the M_{ia}, M_{ih} and M_{ic} parameters developed by SMCT are also presented in this table. The M_{ia} parameter represents the coarse particle component (down to 750 µm), of the overall comminution energy and can be used together with the M_{ib} (fine particle component) to estimate the total energy requirements of a conventional comminution circuit. The use of these parameters is explained further in APPENDIX B. The derived estimates of parameters A, b and t_a that are required for JKSimMet comminution modelling are given in Table 4.

Also included in the derived results are the SAG Circuit Specific Energy (SCSE) values. The SCSE value is derived from simulations of a “standard” circuit comprising a SAG mill in closed circuit with a pebble crusher. This allows $A*b$ values to be described in a more meaningful form. SCSE is described in detail in APPENDIX A.

In the case of the Master Composite sample from BAM East Project, the A and b estimates are based on a correlation using the database of all results so far accumulated by SMCT.

Table 3 - SMC Test® Results

Sample Designation	*DWi* (kWh/m^3)	*DWi* (%)	*Mi* Parameters (kWh/t)			SG
			Mia	*Mih*	*Mic*	
Master Composite	5.11	30	15.8	11.1	5.7	2.67

For more details on how the M_{ia}, M_{ih} and M_{ic} parameters are derived and used, see APPENDIX B or go to the SMC Testing website at http://www.smctesting.com/about and click on the link to download Steve Morrell's paper on this subject.

Table 4 – Parameters derived from the SMC Test® Results

Sample Designation	*A*	*b*	t_a	*SCSE* (kWh/t)
Master Composite	30.5	1.71	0.51	8.74

The influence of particle size on the specific comminution energy needed to achieve a particular t_{10} value can also be inferred from the SMC Test® results. The energy requirements for five particle sizes, each crushed to three different t_{10} values, are presented in Table 5.

Table 5 – Crusher Simulation Model Specific Energy Matrix

Sample Designation	Particle Size (mm)														
	14.5			20.6			28.9			41.1			57.8		
	t_{10} Values (%) for Given Specific Energies in kWh/t														
	10	20	30	10	20	30	10	20	30	10	20	30	10	20	30
Master Composite	0.26	0.56	0.89	0.23	0.48	0.77	0.20	0.42	0.67	0.17	0.36	0.58	0.15	0.32	0.51

The SMC Test® database now contains over 40,000 test results on samples representing more than 1300 different deposits worldwide.

Around 99% of the DWi values lie in the range 0.5 to 14.0 kWh/m^3, with soft ores being at the low end of this range and hard ores at the high end.

A cumulative graph of DWi values from the SMC Test® Database is shown in Figure 6 below. This graph can be used to compare the DWi of the material from BAM East

Project, with the entire population of ores in the SMCT database. The figures on the y-axis of the graph represent the percentages of all ores tested that are softer than the x-axis (DWi) value selected.

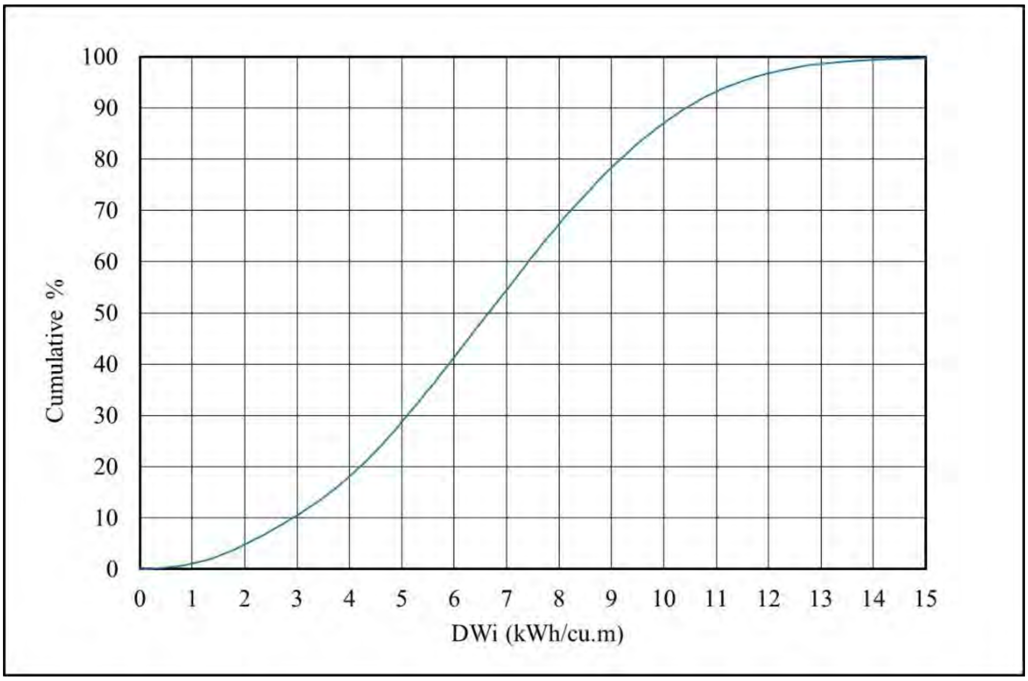


Figure 6 – Cumulative Distribution of DWi Values in SMCT Database

A further cumulative distribution graph is provided in Figure 7 to allow a comparison of the M_{ia}, M_{ih} and M_{ic} values obtained for the BAM East Project material, with the entire population of values for these parameters contained in the SMCT database.

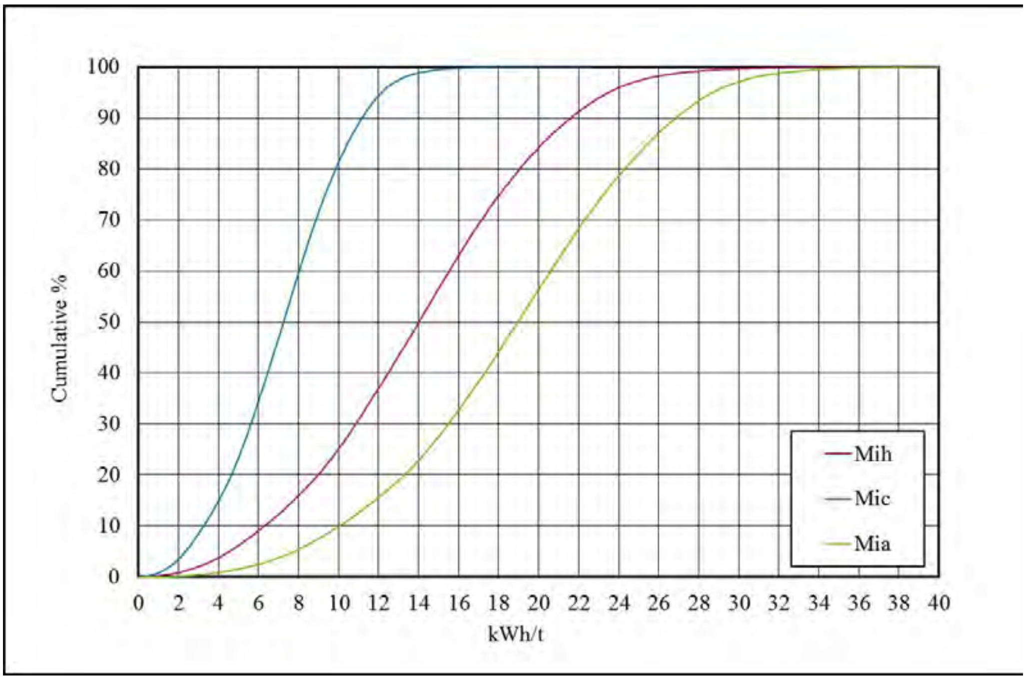


Figure 7 - Cumulative Distribution of M_{ia}, M_{ih} and M_{ic} Values in the SMCT Database

The value of $A*b$, which is also a measure of resistance to impact breakage, is calculated and presented in Table 6, which also gives a comparison to the population of samples in the JKTech database, with the percent of samples present in the JKTech database that are softer. Note that in contrast to the DWi, a high value of $A*b$ means that an ore is soft whilst a low value means that it is hard.

Table 6 – Derived Values for $A*b$, t_a and SCSE

Sample Designation	A*b		t_a		SCSE (kWh/t)	
	Value	%	Value	%	Value	%
Master Composite	52.2	40.1	0.51	41.4	8.74	37.3

In Figure 8 and Figure 9 below, histogram style frequency distributions for the $A*b$ values and for the SCSE values in the JKTech DW database are shown respectively.

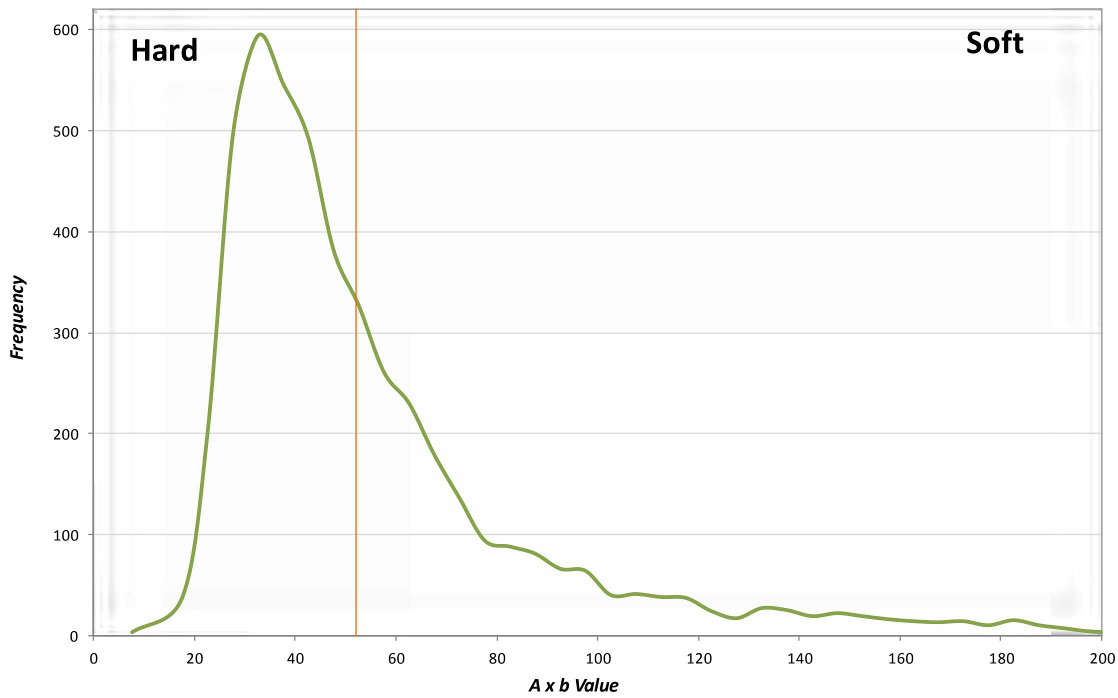


Figure 8 - Frequency Distribution of $A*b$ in the JKTech Database

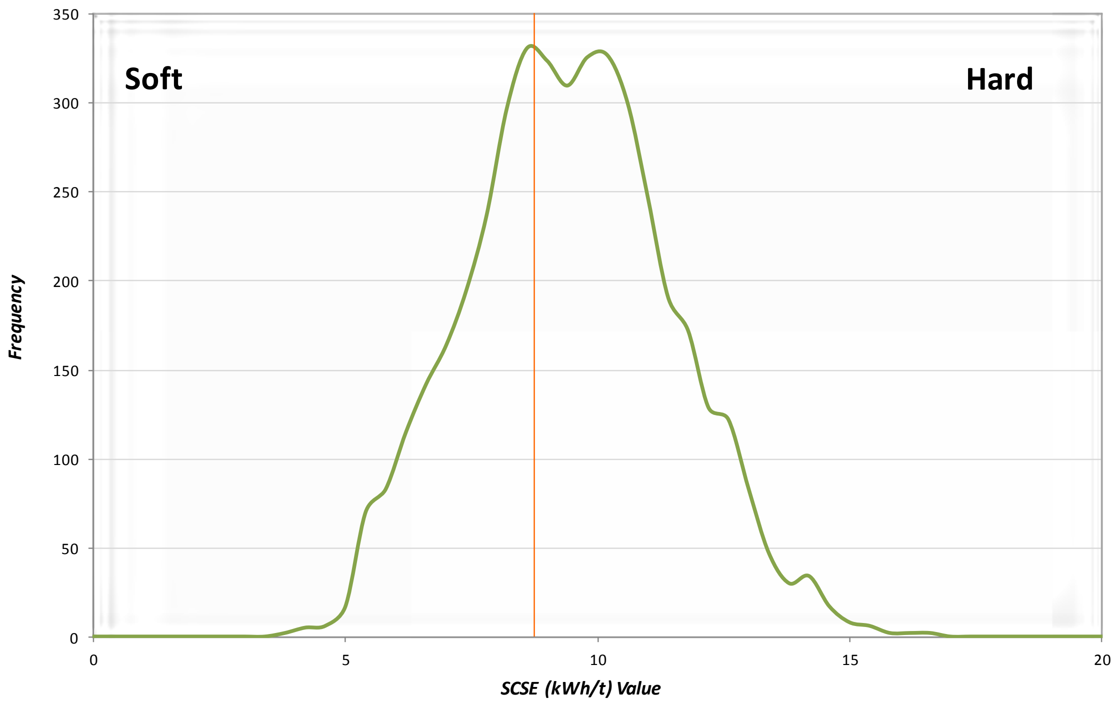


Figure 9 - Frequency Distribution of SCSE in the JKTech Database

4 REFERENCES

Andersen, J. and Napier-Munn, T.J., 1988, “Power Prediction for Cone Crushers", Third Mill Operators' Conference, Aus.I.M.M (Cobar, NSW), May 1988, pp 103 - 106

Bailey, C., *et al*, 2009. “What Can Go Wrong in Comminution Circuit Design?”, Proceedings of the Tenth Mill Operators’ Conference, (Adelaide, SA), pp. 143–149

Bond, F.C., 1961. "Crushing and Grinding Calculations Parts I and II", British *Chemical Engineering*, Vol 6, Nos 6 and 8

Leung, K. 1987. "An Energy-Based Ore Specific Model for Autogenous and Semi-Autogenous Grinding Mills." Ph.D. Thesis. University of Queensland (unpublished)

Leung, K., Morrison, R.D. and Whiten, W.J., 1987. "An Energy Based Ore Specific Model for Autogenous and Semi-autogenous Grinding", Copper *87*, Vina del Mar, Vol. 2, pp 71 - 86

Levin, J., 1989. Observation on the bond standard grindability test, and a proposal for a standard grindability test for fine materials. SAIMM 89 (1), 13-21.

Morrell, S. 1996. "Power Draw of Wet Tumbling Mills and Its Relationship to Charge Dynamics - Parts I and II", *Transaction Inst. Min. Metall.* (Sect C: Mineral Process Extr. Metall.), 105, 1996, pp C43-C62

Morrell, S., 2004[a]. *Predicting the Specific Energy of Autogenous and Semi-autogenous Mills from Small Diameter Drill Core Samples*. Minerals Engineering, Vol 17/3 pp 447-451

Morrell, S., 2004[b]. *An Alternative Energy-Size Relationship To That Proposed By Bond For The Design and Optimisation Of Grinding Circuits*. International Journal of Mineral Processing, 74, 133-141.

Morrell, S., 2006. *Rock Characterisation for High Pressure Grinding Rolls Circuit Design*, Proc International Autogenous and Semi Autogenous Grinding Technology,Vancouver, vol IV pp 267-278.

Morrell,S., 2008, *A method for predicting the specific energy requirement of comminution circuits and assessing their energy utilisation efficiency*, *Minerals Engineering*, Vol. 21, No. 3.

Shi, F. and Kojovic, T., 2007. Validation of a model for impact breakage incorporating particle size effect. Int. Journal of Mineral Processing, 82, 156-163.

Veillette, G., and Parker, B., 2005. Boddington Expansion Project Comminution Circuit Features and Testwork, Randol Gold Forum Proceedings.

5 DISCLAIMER

Warranty by JKTech

a. JKTech will use its best endeavours to ensure that all documentation, data, recommendations, information, advice and reports ("Material"), provided by JKTech to the client ("Recipient"), is accurate at the time of providing it.

Extent of Warranty by JKTech

b. JKTech does not make any representations as to any matter, fact or thing that is not expressly provided for in the Material.

c. JKTech does not give any warranty, nor accept any liability in connection with the Material, except to the extent, if any, required by law or specifically provided in writing by JKTech to the Recipient.

d. JKTech will not be liable to the Recipient for any claims relating to Material in any language other than in English.

e. If, apart from this Disclaimer, any warranty would be implied whether by law, custom or otherwise, that warranty is to the full extent permitted by law excluded.

f. The Recipient will promptly advise JKTech in writing of any losses, damages, compensation, liabilities, amounts, monetary and non-monetary costs and expenses ("Losses"), incurred or likely to be incurred by the Recipient or JKTech in connection with the Material, and any claims, actions, suits, demands or proceedings ("Liabilities") which the Recipient or JKTech may become liable in connection with the Material.

Indemnity and Release by the Recipient

g. The Recipient indemnifies, releases, discharges and saves harmless, JKTech against any and all Losses and Liabilities, suffered or incurred by JKTech, whether under the law of contract, tort, statutory duty or otherwise as a result of:

   i) the Recipient relying on the Material;

   ii) any liability for infringement of a third party's trade secrets, proprietary or confidential information, patents, registered designs, trademarks or names, copyright or other protected rights; and

   iii) any act or omission of JKTech, any employee, agent or permitted sub-contractor of JKTech in connection with the Material.

Limit of Liability

h. JKTech's liability to the Recipient in connection with the Material, whether under the law of contract, tort, statutory duty or otherwise, will be limited to the lesser of:

   i) the total cost of the job; or

   ii) JKTech providing amended Material rectifying the defect.

Exclusion of Consequential Loss

i. JKTech is not liable to the Recipient for any consequential, special or indirect loss (loss of revenue, loss of profits, business interruption, loss of opportunity and legal costs and disbursements), in connection with the Material whether under the law of contract, tort, statutory duty or otherwise.

Defects

j. The Recipient must notify JKTech within seven days of becoming aware of a defect in the Material. To the extent that the defect is caused by JKTech's negligence or breach of contract, JKTech may, at its discretion, rectify the defect.

Duration of Liability

k. After the expiration of one year from the date of first providing the Material to the client, JKTech will be discharged from all liability in connection with the Material. The Recipient (and persons claiming through or under the Recipient) will not be entitled to commence any action, claim or proceeding of any kind whatsoever after that date, against JKTech (or any employee of JKTech) in connection with the Material.

Contribution

l. JKTech's liability to the Recipient for any loss or damage, whether under the law of contract, tort, statutory duty or otherwise will be reduced to the extent that an act or omission of the Recipient, its employees or agents, or a third party to whom the Recipient has disclosed the Material, contributed to the loss or damage.

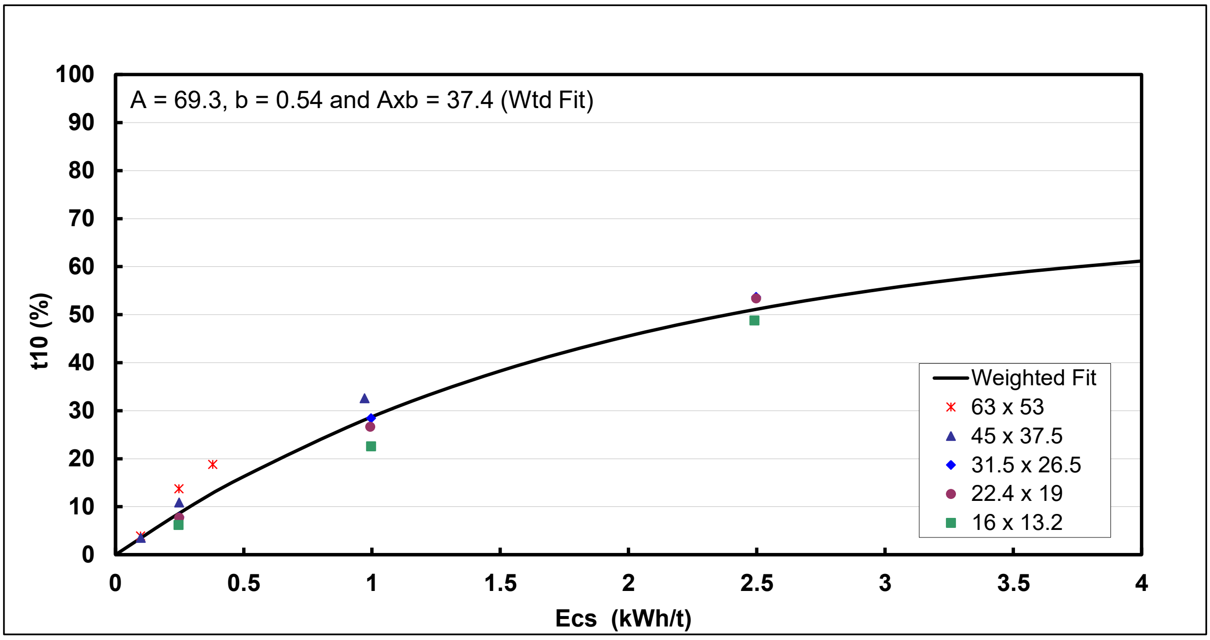
Severability

m. If any provision of this Disclaimer is illegal, void, invalid or unenforceable for any reason, all other provisions which are self-sustaining and capable of separate enforcement will, to the maximum extent permitted by law, be and continue to be valid and enforceable.

APPENDICES

APPENDIX A. SAG CIRCUIT SPECIFIC ENERGY (SCSE)

For a little over 20 years, the results of JK Drop Weight tests and SMC tests have been reported in part as A, b and t_a parameters. A and b are parameters which describe the response of the ore under test to increasing levels of input energy in single impact breakage. A typical t_{10} v Ecs curve resulting from a Drop Weight test is shown in App Figure 1.



App Figure 1 – Typical t_{10} v Ecs curve

The curve shown in App Figure 1 is represented by an equation which is given in Equation 1.

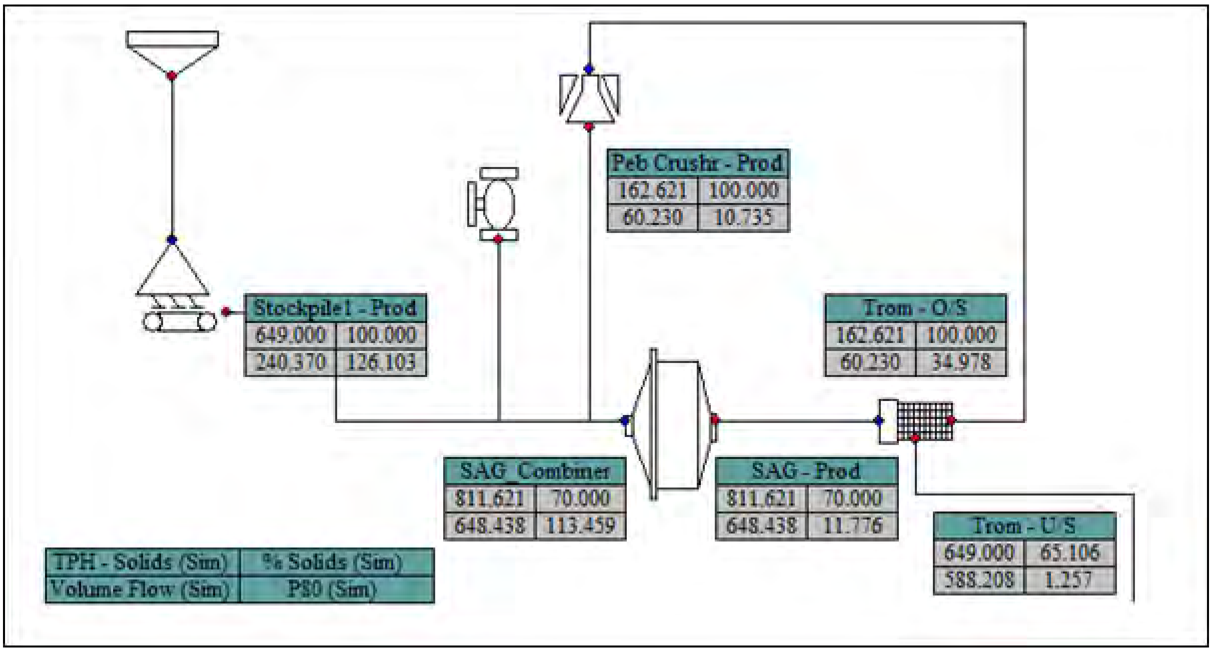
$$t_{10} = A(1 - e^{-b.Ecs}) \qquad \text{Equation 1}$$

The parameters A and b are generated by least squares fitting Equation 1 to the JK Drop Weight test data. The parameter t_a is generated from a tumbling test.

Both A and b vary with ore type but having two parameters describing a single ore property makes comparison difficult. For that reason the product of A and b, referred to as A*b, which is related to the slope of the t_{10} – E_{cs} curve at the origin, has been universally accepted as the parameter which represents an ore's resistance to impact breakage.

The parameters A, b and t_a have no physical meaning in their own right. They are ore hardness parameters used by the AG/SAG mill model in JKSimMet which permits prediction of the product size distribution and the power draw of the AG/SAG mill for a given feed size distribution and feed rate. In a design situation, the dimensions of the mill are adjusted until the load in the mill reaches 25 % by volume when fed at the required feed rate. The model predicts the power draw under these conditions and from the power draw and throughput the specific energy is determined. The specific energy is mainly a function of the ore hardness (A and b values), the feed size and the dimensions of the mill (specifically the aspect ratio) as well as to a lesser extent the operating conditions such as ball load, mill speed, grate/pebble port size and pebble crusher activity.

There are two drawbacks to the approach of using A*b as the single parameter to describe the impact resistance of a particular ore. The first is that A*b is inversely related to impact resistance, which adds unnecessary complication. The second is that A*b is related to impact resistance in a non-linear manner. As mentioned earlier this relationship and how it affects comminution machine performance can only be predicted via simulation modelling. Hence to give more meaning to the A and b values and to overcome these shortcomings, JKTech Pty Ltd and SMC Testing Pty Ltd have developed a “standard” simulation methodology to predict the specific energy required for a particular tested ore when treated in a “Standard” circuit comprising a SAG mill in closed circuit with a pebble crusher. The flowsheet is shown in App Figure 2 .



App Figure 2 – Flowsheet used for “Standard” AG/SAG circuit simulations

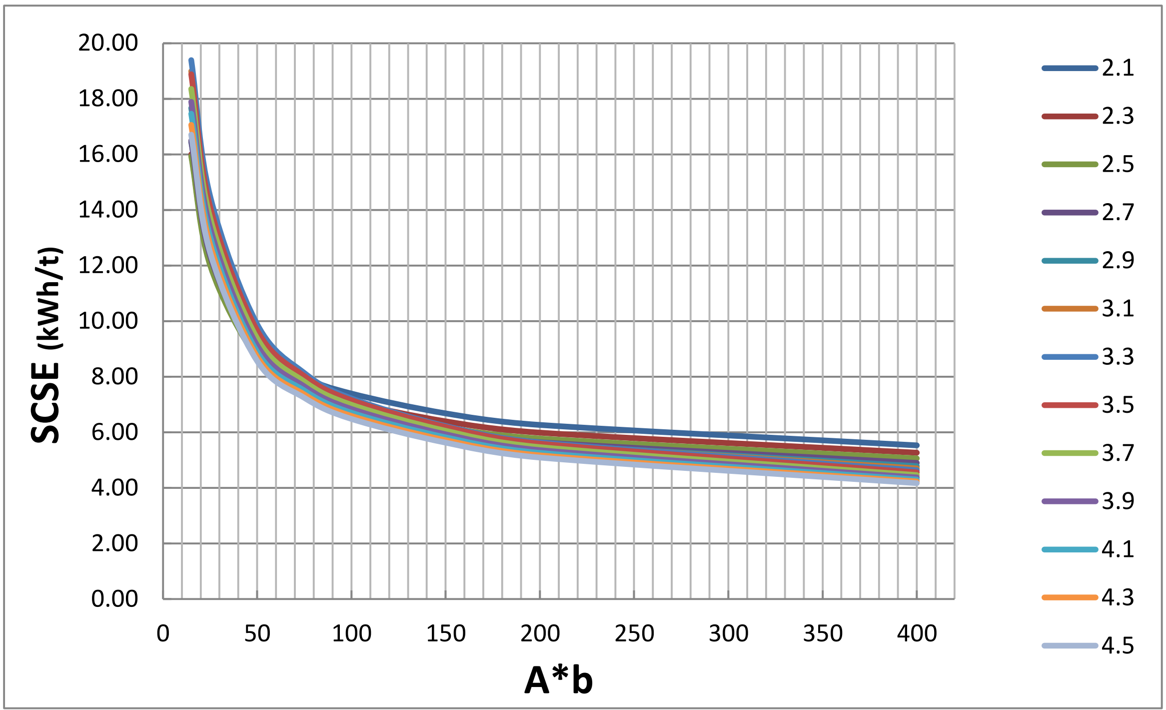
The specifications for the “standard” circuit are:

- SAG Mill
    - inside shell diameter to length ratio of 2:1 with 15 ° cone angles
    - ball charge of 15 %, 125 mm in diameter
    - total charge of 25 %
    - grate open area of 7 %
    - apertures in the grate are 100 % pebble ports with a nominal aperture of 56 mm
- Trommel
    - Cut Size of 12 mm
- Pebble Crusher
    - Closed Side Setting of 10 mm
- Feed Size Distribution
    - F_{80} from the t_a relationship given in Equation 2

The feed size distribution is taken from the JKTech library of typical feed size distributions and is adjusted to meet the ore specific 80 % passing size predicted using the Morrell and Morrison (1996) F_{80} – t_a relationship for primary crushers with a closed side setting of 150 mm given in Equation 2.

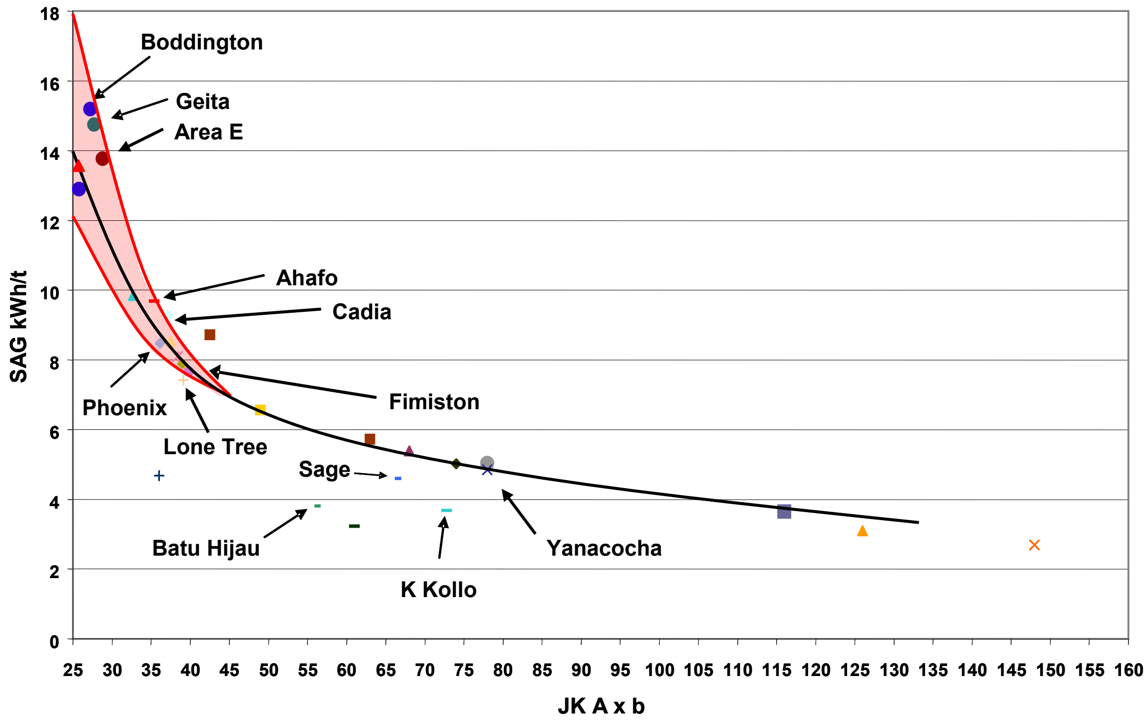
$$F_{80} = 71.3 - 28.4 * \ln(t_a) \qquad \text{Equation 2}$$

Simulations were conducted with A*b values ranging from 15 to 400, t_a values ranging from 0.145 to 3.866 and solids SG values ranging from 2.1 to 4.5. For each simulation, the feed rate was adjusted until the total load volume in the SAG mill was 25 %. The predicted mill power draw and crusher power draw were combined and divided by the feed rate to provide the specific energy consumption. The results are shown in App Figure 3.



App Figure 3 – The relationship between A*b and specific energy at varying SG for the “Standard” circuit.

It is of note that the family of curves representing the relationship between Specific energy and A*b for the “standard” circuit is very similar to the specific energy – A*b relationship for operating mills published in Veillette and Parker, 2005 and reproduced here in App Figure 4.



App Figure 4 – A*b vs SAG kWh/t for operating AG/SAG mills (after Veillette and Parker, 2005).

Of course, the SCSE quoted value will not necessarily match the specific energy required for an existing or a planned AG/SAG mill due to differences in the many operating and design variables such as feed size distribution, mill dimensions, ball load and size and grate, trommel and pebble crusher configuration. The SCSE is an effective tool to compare in a relative manner the expected behaviour of different ores in AG/SAG milling in exactly the same way as the Bond laboratory ball mill work index can be used to compare the relative grindability of different ores in ball milling (Bond, 1961 and Rowland and Kjos, 1980). However the originally reported A and b parameters which match the SCSE will be still be required in JKSimMet simulations of a proposed circuit to determine the AG/SAG mill specific energy required for that particular grinding task. Guidelines for the use of JKSimMet for such simulations were given in Bailey *et al*, 2009.

APPENDIX B. BACKGROUND AND USE OF THE SMC TEST®

B 1 Introduction

The SMC Test® was developed to provide a range of useful comminution parameters through highly controlled breakage of rock samples. Drill core, even quartered small diameter core is suitable. Only relatively small quantities of sample are required and can be re-used to conduct Bond ball work index tests.

The results from conducting the SMC Test® are used to determine the so-called drop-weight index (DW_i), which is a measure of the strength of the rock, as well as the comminution indices M_{ia}, M_{ih} and M_{ic} . The SMC Test® also estimates the JK rock breakage parameters A, b and t_a as well as the JK crusher model's $t10$-Ecs matrix, all of which are generated as part of the standard report output from the test.

In conjunction with the Bond ball mill work index the DW_i and the M_i suite of parameters can be used to accurately predict the overall specific energy requirements of circuits containing:

- AG and SAG mills.
- Ball mills
- Rod mills
- Crushers
- High Pressure Grinding Rolls (HPGR)

The JK rock breakage parameters can be used to simulate crushing and grinding circuits using JKTech's simulator – JKSimMet.

B 2 Simulation Modelling and Impact Comminution Theory

When a rock fragment is broken, the degree of breakage can be characterised by the "t_{10}" parameter. The t_{10} value is the percentage of the original rock mass that passes a screen aperture one tenth of the original rock fragment size. This parameter allows the degree of breakage to be compared across different starting sizes.

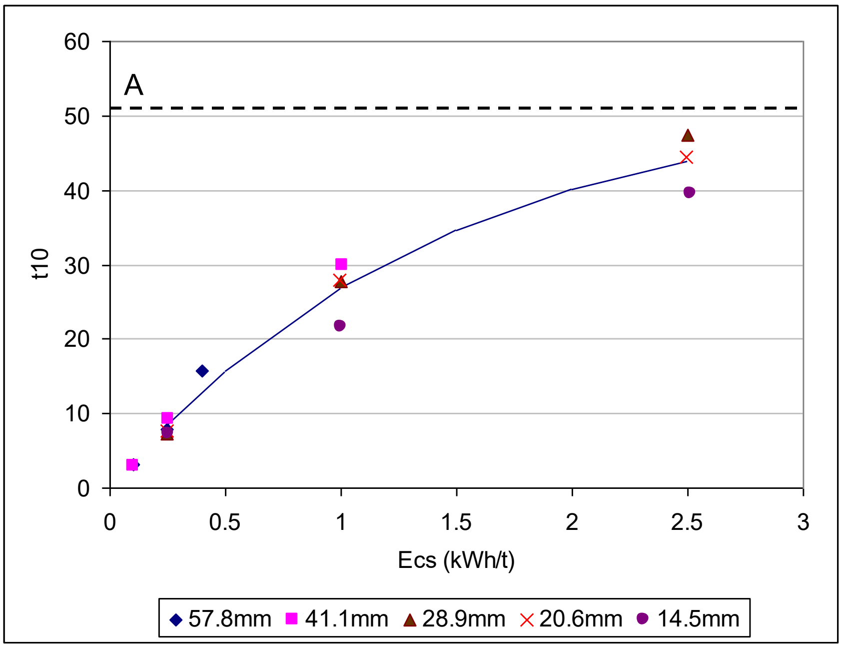
The specific comminution energy (Ecs) has the units kWh/t and is the energy applied during impact breakage. As the impact energy is varied, so does the t_{10} value vary in response. Higher impact energies produce higher values of t_{10}, which of course means products with finer size distributions.

The equation describing the relationship between the t_{10} and Ecs is given below.

$$t_{10} = A(1 - e^{-b.Ecs}) \qquad \text{Equation 1}$$

As can be seen from this equation, there are two rock breakage parameters A and b that relate the t_{10} (size distribution index) to the applied specific energy (Ecs). These parameters are ore specific and are normally determined from a full JK Drop-Weight test.

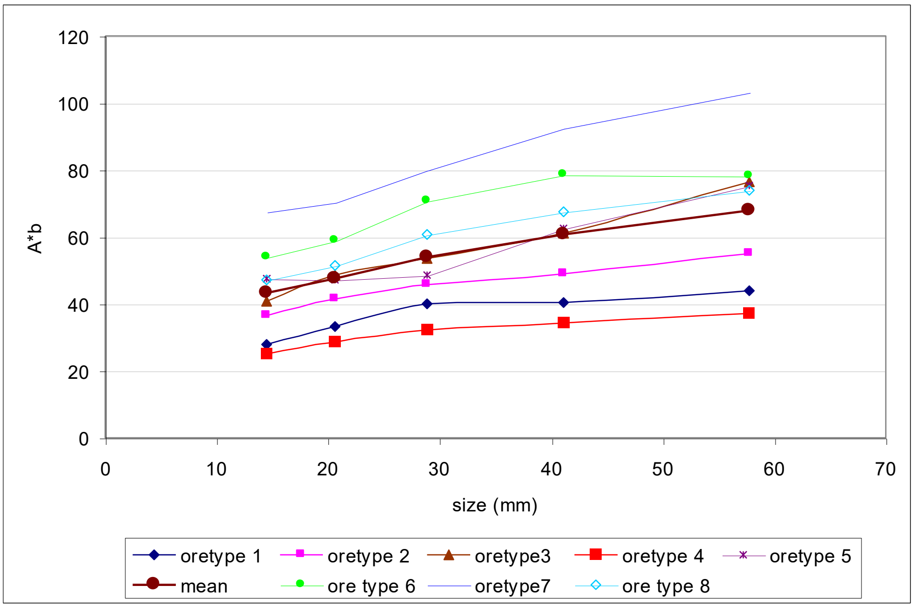
A typical plot of t_{10} vs Ecs from a JK Drop-Weight test is shown in App Figure 5. The relationship is characterised by the two-parameter equation above, where t_{10} is the dependent variable.



App Figure 5 - Typical t_{10} v Ecs Plot

The t_{10} can be thought of as a "fineness index" with larger values of t_{10} indicating a finer product size distribution. The value of parameter A is the limiting value of t_{10}. This limit indicates that at higher energies, little additional size reduction occurs as the Ecs is increased beyond a certain value. $A*b$ is the slope of the curve at 'zero' input energy and is generally regarded as an indication of the strength of the rock, lower values indicating a higher strength.

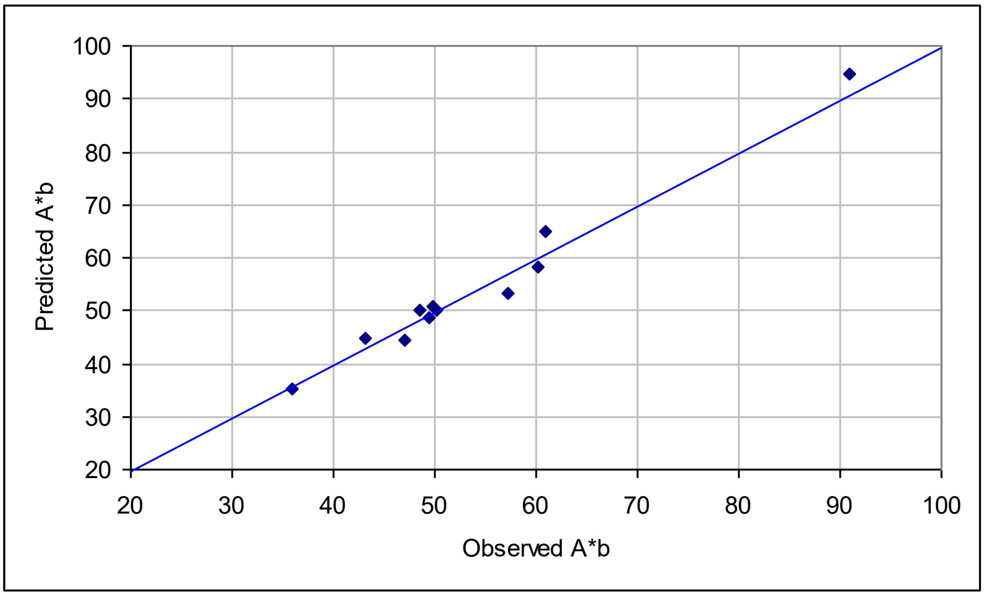
The SMC Test® is used to estimate the JK rock breakage parameters A and b by utilizing the fact that there is usually a pronounced (and ore specific) trend to decreasing rock strength with increasing particle size. This trend is illustrated in App Figure 6 which shows a plot of $A*b$ versus particle size for a number of different rock types.



App Figure 6 - Size Dependence of A*b for a Range of Ore Types

In the case of a conventional JK Drop-Weight test these values are effectively averaged and a mean value of A and b is reported. The SMC Test® uses a single size and makes use of relationships such as that shown in App Figure 6 to predict the A and b of the particle size that has the same value as the mean for a full JK Drop-Weight test.

An example of this is illustrated in App Figure 7, where the observed values of the product $A*b$ are plotted against those predicted using the DWi. Each of the data points in App Figure 7 is a result from a different ore type within an orebody.



App Figure 7 - Predicted v Observed A*b

The A and b parameters are used with Equation 1 and relationships such as illustrated in App Figure 6 to generate a matrix of Ecs values for a specific range of

t_{10} values and particle sizes. This matrix is used in crusher modelling to predict the power requirement of the crusher given a feed and a product size specification (Napier-Munn et al (1996)).

The A and b parameters are also used in AG/SAG mill models, such as those in JKSimMet, for predicting how the rock will break inside the mill. From this description the models can predict what the throughput, power draw and product size distribution will be (Napier-Munn et al (1996)). Modelling also enables a detailed flowsheet to be built up of the comminution circuit response to changes in ore type. It also allows optimisation strategies to be developed to overcome any deleterious changes in circuit performance predicted from differences in ore type. These strategies can include both changes to how mills are operated (eg ball load, speed etc) and changes to feed size distribution through modification of blasting practices and primary crusher operation (mine-to-mill).

B 3 Power-Based Equations

B 3.1 General

The DW_i, M_{ia}, M_{ih} and M_{ic} parameters are used in so-called power-based equations which predict the specific energy of the associated comminution machines. The approach divides comminution equipment into three categories:

- Tumbling mills, eg AG, SAG, rod and ball mills
- Conventional reciprocating crushers, eg jaw, gyratory and cone
- HPGRs

Tumbling mills are described using 2 indices: M_{ia} and M_{ib}
Crushers have one index: M_{ic}
HPGRs have one index: M_{ih}

For tumbling mills the 2 indices relate to "coarse" and "fine" ore properties plus an efficiency factor which represents the influence of a pebble crusher in AG/SAG mill circuits. "Coarse" in this case is defined as spanning the size range from a P80 of 750 microns up to the P80 of the product of the last stage of crushing or HPGR size reduction prior to grinding. "Fine" covers the size range from a P80 of 750 microns down to P80 sizes typically reached by conventional ball milling, ie about 45 microns. The choice of 750 microns as the division between "coarse" and "fine" particle sizes was determined during the development of the technique and was found to give the best overall results across the range of plants in SMCT's data base. Implicit in the approach is that distributions are parallel and linear in log-log space.

The work index covering grinding in tumbling mills of coarse sizes is labelled M_{ia}. The work index covering grinding of fine particles is labelled Mib (Morrell, 2008). M_{ia} values are provided as a standard output from a SMC Test® (Morrell, 2004a) whilst M_{ib} values can be determined using the data generated by a conventional Bond ball mill work index test (M_{ib} is NOT the Bond ball work index). M_{ic} and M_{ih} values are also provided as a standard output from a SMC Test® (Morrell, 2009).

The general size reduction equation is as follows (Morrell, 2004b):

$$W_i = M_i.4(x_2^{f(x_2)} - x_1^{f(x_1)})$$ Equation 3

where

M_i = Work index related to the breakage property of an ore (kWh/tonne); for grinding from the product of the final stage of crushing to a P80 of 750 microns (coarse particles) the index is labelled Mia and for size reduction from 750 microns to the final product P80 normally reached by conventional ball mills (fine particles) it is labelled M_{ib}. For conventional crushing M_{ic} is used and for HPGRs Mih is used.

Wi = Specific comminution (kWh/tonne)

x_2 = 80% passing size for the product (microns)

x_1 = 80% passing size for the feed (microns)

$f(x_j)$ = -(0.295 + x_j/1000000) (Morrell, 2006) Equation 4

For tumbling mills the specific comminution energy (Wi) relates to the power at the pinion or for gearless drives - the motor output. For HPGRs it is the energy inputted to the rolls, whilst for conventional crushers Wi relates to the specific energy as determined using the motor input power less the no-load power.

B 3.2 Specific Energy Determination for Comminution Circuits

The total specific energy (W_T) to reduce primary crusher product to final product size is given by:

$$W_T = W_a + W_b + W_c + W_h + W_s$$ Equation 5

where

W_a = specific energy to grind coarser particles in tumbling mills

W_b = specific energy to grind finer particles in tumbling mills

W_c = specific energy for conventional crushing

W_h = specific energy for HPGRs

W_s = specific energy correction for size distribution

Clearly only the W values associated with the relevant equipment in the circuit being studied are included in Equation 5.

B 3.2.1 Tumbling mills

For coarse particle grinding in tumbling mills Equation 3 is written as:

$$W_a = K_1 M_{ia}.4(x_2^{f(x_2)} - x_1^{f(x_1)})$$ Equation 6

where

K_1 = 1.0 for all circuits that do not contain a recycle pebble crusher and 0.95 where circuits do have a pebble crusher

x_1 = P_{80} in microns of the product of the last stage of crushing before grinding

x_2 = 750 microns

M_{ia} = Coarse ore work index and is provided directly by SMC Test®

For fine particle grinding Equation 3 is written as:

$$W_b = M_{ib}.4({x_3}^{f(x_3)} - {x_2}^{f(x_2)}) \qquad \text{Equation 7}$$

where
x_2 = 750 microns
x_3 = P_{80} of final grind in microns
M_{ib} = Provided by data from the standard Bond ball work index test using the following equation (Morrell, 2006):

$$M_{ib} = {18.18}\Big/{{P_1}^{0.295}(Gbp)({p_{80}}^{f(p_{80})} - {f_{80}}^{f(f_{80})})} \qquad \text{Equation 8}$$

where
Mib = fine ore work index (kWh/tonne)
P_1 = closing screen size in microns
Gbp = net grams of screen undersize per mill revolution
p_{80} = 80% passing size of the product in microns
f_{80} = 80% passing size of the feed in microns

Note that the Bond ball work index test should be carried out with a closing screen size which gives a final product P80 similar to that intended for the full scale circuit.

B 3.2.2 Conventional Crushers and HPGR

Equation 3 for conventional crushers is written as:

$$W_c = S_c K_2 M_{ic}.4({x_2}^{f(x_2)} - {x_1}^{f(x_1)}) \qquad \text{Equation 9}$$

where
S_c = coarse ore hardness parameter which is used in primary and secondary crushing situations. It is defined by Equation 10 with K_s set to 55.
K_2 = 1.0 for all crushers operating in closed circuit with a classifying screen. If the crusher is in open circuit, eg pebble crusher in a AG/SAG circuit, K_2 takes the value of 1.19.
x_1 = P_{80} in microns of the circuit feed
x_2 = P_{80} in microns of the circuit product
M_{ic} = Crushing ore work index and is provided directly by SMC Test®
The coarse ore hardness parameter (S) makes allowance for the decrease in ore hardness that becomes significant in relatively coarse crushing applications such as primary and secondary cone/gyratory circuits. In tertiary and pebble crushing circuits it is normally not necessary and takes the value of unity. In full scale HPGR circuits where feed sizes tend to be higher than used in laboratory and pilot scale machines the parameter has also been found to improve predictive accuracy. The parameter is defined by Equation 10.

$$S = K_s(x_1.x_2)^{-0.2} \qquad \text{Equation 10}$$

where
K_s = machine-specific constant that takes the value of 55 for conventional crushers and 35 in the case of HPGRs

x_1 = P_{80} in microns of the circuit feed
x_2 = P_{80} in microns of the circuit product

Equation 3 for HPGR's crushers is written as:

$$W_h = S_h K_3 M_{ih}.4(x_2{}^{f(x_2)} - x_1{}^{f(x_1)}) \qquad \text{Equation 11}$$

where
S_h = coarse ore harness parameter as defined by Equation 10 and with K_s set to 35
K_3 = 1.0 for all HPGRs operating in closed circuit with a classifying screen. If the HPGR is in open circuit, K3 takes the value of 1.19.
x_1 = P_{80} in microns of the circuit feed
x_2 = P_{80} in microns of the circuit product
M_{ih} = HPGR ore work index and is provided directly by SMC Test®

B 3.2.3 Specific Energy Correction for Size Distribution (Ws)

Implicit in the approach described in this appendix is that the feed and product size distributions are parallel and linear in log-log space. Where they are not, allowances (corrections) need to be made. By and large, such corrections are most likely to be necessary (or are large enough to be warranted) when evaluating circuits in which closed circuit secondary/tertiary crushing is followed by ball milling. This is because such crushing circuits tend to produce a product size distribution which is relatively steep when compared to the ball mill circuit cyclone overflow. This is illustrated in App Figure 8, which shows measured distributions from an open and closed crusher circuit as well as a ball mill cyclone overflow. The closed circuit crusher distribution can be seen to be relatively steep compared with the open circuit crusher distribution and ball mill cyclone overflow. Also the open circuit distribution more closely follows the gradient of the cyclone overflow. If a ball mill circuit were to be fed two distributions, each with same P80 but with the open and closed circuit gradients in App Figure 8, the closed circuit distribution would require more energy to grind to the final P80. How much more energy is required is difficult to determine. However, for the purposes of this approach it has been assumed that the additional specific energy for ball milling is the same as the difference in specific energy between open and closed crushing to reach the nominated ball mill feed size. This assumes that a crusher would provide this energy. However, in this situation the ball mill has to supply this energy and it has a different (higher) work index than the crusher (ie the ball mill is less energy efficient than a crusher and has to input more energy to do the same amount of size reduction). Hence from Equation 9, to crush to the ball mill circuit feed size (x_2) in open circuit requires specific energy equivalent to:

$$W_c = 1.19 * M_{ic}.4(x_2{}^{f(x_2)} - x_1{}^{f(x_1)}) \qquad \text{Equation 12}$$

For closed circuit crushing the specific energy is:

$$W_c = 1 * M_{ic}.4(x_2{}^{f(x_2)} - x_1{}^{f(x_1)}) \qquad \text{Equation 13}$$

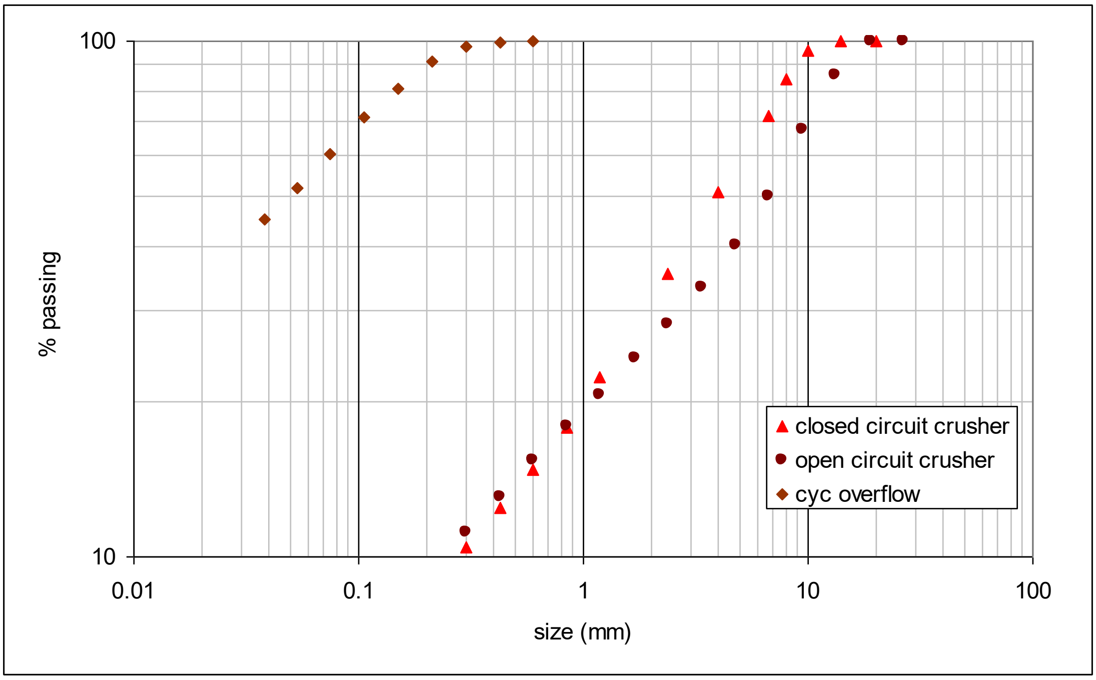
The difference between the two (Equation 12 and Equation 13) has to be provided by the milling circuit with an allowance for the fact that the ball mill, with its lower energy efficiency, has to provide it and not the crusher. This is what is referred to in Equation 5 as W_s and for the above example is therefore represented by:

$$W_s = 0.19 * M_{ia}.4(x_2^{f(x_2)} - x_1^{f(x_1)}) \qquad \text{Equation 14}$$

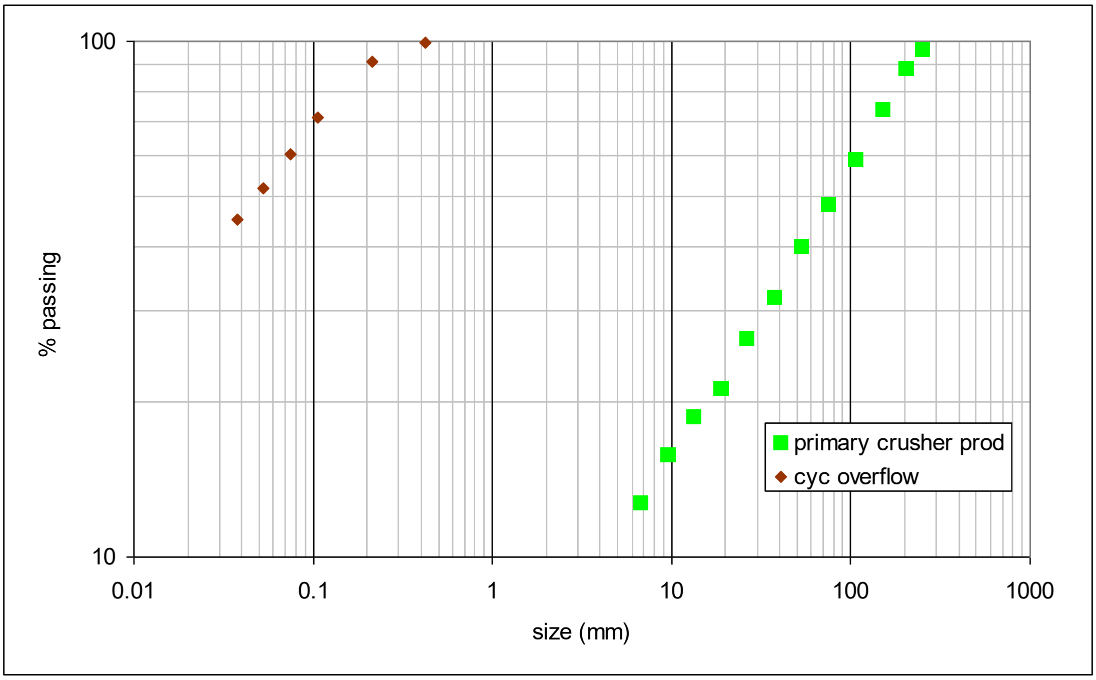
Note that in Equation 14 M_{ic} has been replaced with M_{ia}, the coarse particle tumbling mill grinding work index.

In AG/SAG based circuits the need for W_s appears to be unnecessary as App Figure 9 illustrates. Primary crusher feeds often have the shape shown in App Figure 9and this has a very similar gradient to typical ball mill cyclone overflows. A similar situation appears to apply with HPGR product size distributions, as illustrated in App Figure 10. Interestingly SMCT's data show that for HPGRs, closed circuit operation appears to require a lower specific energy to reach the same P80 as in open circuit, even though the distributions for open and closed circuit look to have almost identical gradients. Closer examination of the distributions in fact shows that in closed circuit the final product tends to have slightly less very fine material, which may account for the different energy requirements between the two modes of operation. It is also possible that recycled material in closed circuit is inherently weaker than new feed, as it has already passed through the HPGR previously and may have sustained micro-cracking. A reduction in the Bond ball mill work index as measured by testing HPGR products compared it to the Bond ball mill work index of HPGR feed has been noticed in many cases in the laboratory (see next section) and hence there is no reason to expect the same phenomenon would not affect the recycled HPGR screen oversize.

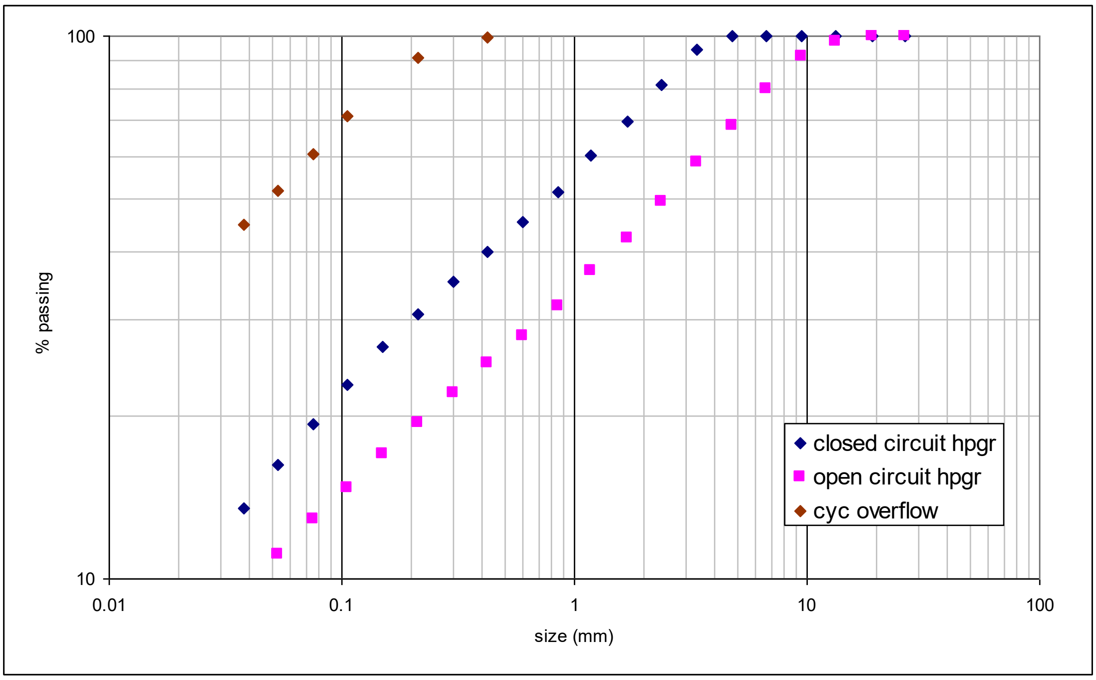
It follows from the above arguments that in HPGR circuits, which are typically fed with material from closed circuit secondary crushers, a similar feed size distribution correction should also be applied. However, as the secondary crushing circuit uses such a relatively small amount of energy compared to the rest of the circuit (as it crushes to a relatively coarse size) the magnitude of size distribution correction is very small indeed – much smaller than the error associated with the technique - and hence may be omitted in calculations.



App Figure 8 – Examples of Open and Closed Circuit Crushing Distributions Compared with a Typical Ball Mill Cyclone Overflow Distribution



App Figure 9 – Example of a Typical Primary Crusher (Open and Circuit) Product Distribution Compared with a Typical Ball Mill Cyclone Overflow Distribution



App Figure 10 – Examples of Open and Closed Circuit HPGR Distributions Compared with a Typical Ball Mill Cyclone Overflow Distribution

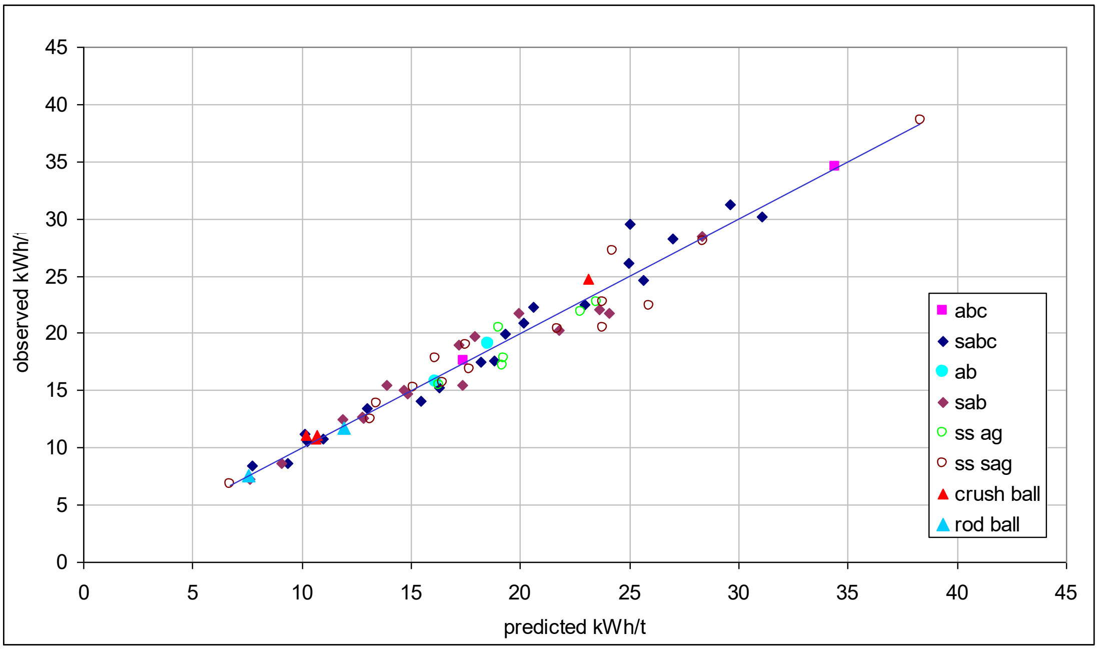
B 3.2.4 Weakening of HPGR Products

As mentioned in the previous section, laboratory experiments have been reported by various researchers in which the Bond ball work index of HPGR products is less than that of the feed. The amount of this reduction appears to vary with both material type and the pressing force used. Observed reductions in the Bond ball work index have typically been in the range 0-10%. In the approach described in this appendix no allowance has been made for such weakening. However, if HPGR products are available which can be used to conduct Bond ball work index tests on then M_{ib} values obtained from such tests can be used in Equation 7. Alternatively the M_{ib} values from Bond ball mill work index tests on HPGR feed material can be reduced by an amount that the user thinks is appropriate. Until more data become available from full scale HPGR/ball mill circuits it is suggested that, in the absence of Bond ball mill work index data on HPGR products, the M_{ib} results from HPGR feed material are reduced by no more than 5% to allow for the effects of micro-cracking.

B 3.3 Validation

B 3.3.1 Tumbling Mill Circuits

The approach described in the previous section was applied to over 120 industrial data sets. The results are shown in App Figure 11. In all cases, the specific energy relates to the tumbling mills contributing to size reduction from the product of the final stage of crushing to the final grind. Data are presented in terms of equivalent specific energy at the pinion. In determining what these values were on each of the plants in the data base it was assumed that power at the pinion was 93.5% of the measured gross (motor input) power, this figure being typical of what is normally accepted as being reasonable to represent losses across the motor and gearbox. For gearless drives (so-called wrap-around motors) a figure of 97% was used.



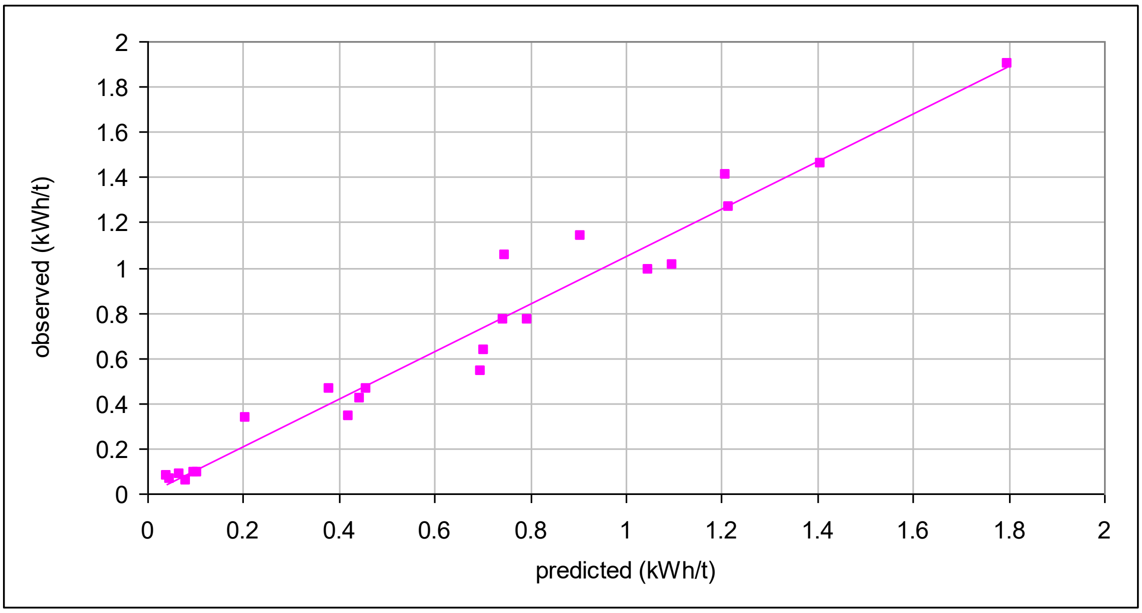
App Figure 11 – Observed vs Predicted Tumbling Mill Specific Energy

B 3.3.2 Conventional Crushers

Validation used 12 different crushing circuits (25 data sets), including secondary, tertiary and pebble crushers in AG/SAG circuits. Observed vs predicted specific energies are given in App Figure 12. The observed specific energies were calculated from the crusher throughput and the net power draw of the crusher as defined by:

Net Power = Motor Input Power – No Load Power Equation 15

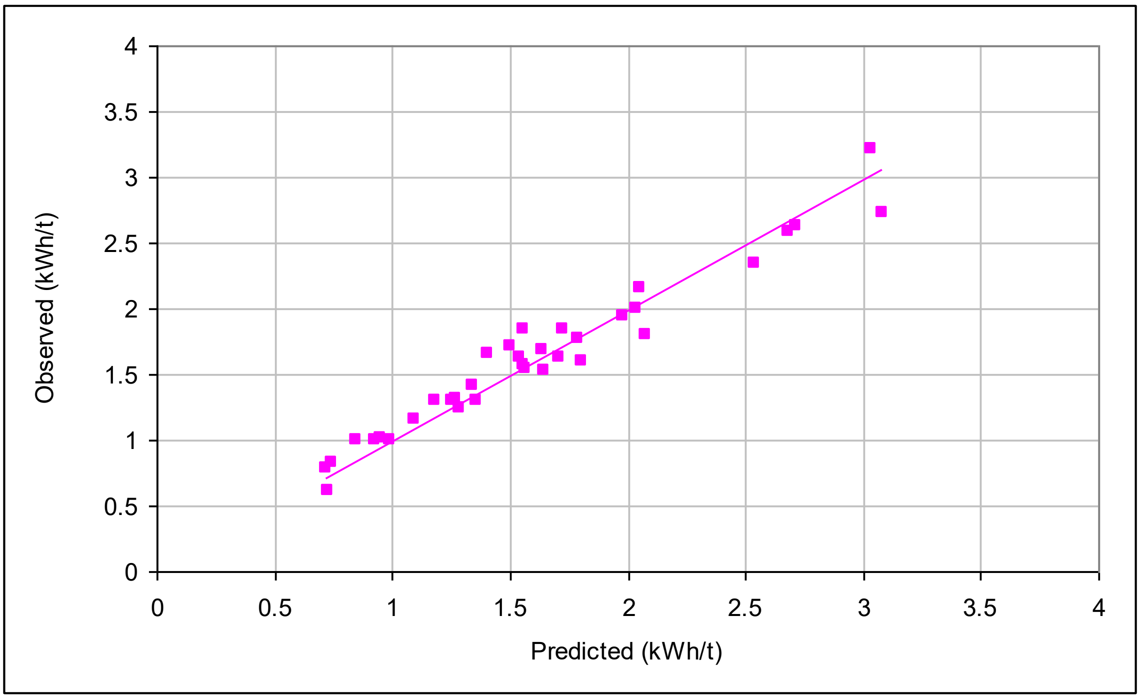
No-load power tends to be relatively high in conventional crushers and hence net power is significantly lower than the motor input power. From examination of the 25 crusher data sets the motor input power was found to be on average 20% higher than the net power.



App Figure 12 – Observed vs Predicted Conventional Crusher Specific Energy

B 3.3.3 HPGRs

Validation for HPGRs used data from 19 different circuits (36 data sets) including laboratory, pilot and industrial scale equipment. Observed vs predicted specific energies are given in App Figure 13. The data relate to HPGRs operating with specific grinding forces typically in the range 2.5-3.5 N/mm^2. The observed specific energies relate to power delivered by the roll drive shafts. Motor input power for full scale machines is expected to be 8-10% higher.



App Figure 13 – Observed vs Predicted HPGR Specific Energy

B 4 WORKED EXAMPLES

A SMC Test® and Bond ball work index test were carried out on a representative ore sample. The following results were obtained:

SMC Test®:
M_{ia} = 19.4 kWh/t
M_{ic} = 7.2 kWh/t
M_{ih} = 13.9 kWh/t
Bond test carried out with a 150 micron closing screen:
M_{ib} = 18.8 kWh/t

Three circuits are to be evaluated:

- SABC
- HPGR/ball mill
- Conventional crushing/ball mill

The overall specific grinding energy to reduce a primary crusher product with a P_{80} of 100 mm to a final product P_{80} of 106 μm needs to be estimated.

B 4.1 SABC Circuit

Coarse particle tumbling mill specific energy:

$$W_a = 0.95 * 19.4 * 4 * \left(750^{-(0.295+750/1000000)} - 100000^{-(0.295+100000/1000000)}\right)$$

= 9.6 kWh/t

Fine particle tumbling mill specific energy:

$$W_b = 18.8 * 4 * \left(106^{-(0.295+106/1000000)} - 750^{-(0.295+750/1000000)}\right)$$

= 8.4 kWh/t

Pebble crusher specific energy:

In this circuit, it is assumed that the pebble crusher feed P_{80} is 52.5mm. As a rule of thumb this value can be estimated by assuming that it is 0.75 of the nominal pebble port aperture (in this case the pebble port aperture is 70mm). The pebble crusher is set to give a product P_{80} of 12mm. The pebble crusher feed rate is expected to be 25% of new feed tph.

$$W_c = 1.19 * 7.2 * 4 * \left(12000^{-(0.295+12000/1000000)} - 52500^{-(0.295+52500/1000000)}\right)$$

= 1.12 kWh/t when expressed in terms of the crusher feed rate
= 1.12 * 0.25 kWh/t when expressed in terms of the SABC circuit new feed rate
= 0.3 kWh/t of SAG mill circuit new feed

Total net comminution specific energy:

W_T = 9.6 + 8.4 + 0.3 kWh/t
= 18.3 kWh/t

B 4.2 HPGR/Ball Milling Circuit

In this circuit primary crusher product is reduced to a HPGR circuit feed P_{80} of 35 mm by closed circuit secondary crushing. The HPGR is also in closed circuit and reduces the 35 mm feed to a circuit product P_{80} of 4 mm. This is then fed to a closed circuit ball mill which takes the grind down to a P_{80} of 106 µm.

Secondary crushing specific energy:

$$W_c = 1*55*(35000 * 100000)^{-0.2} * 7.2 * 4 * \left(35000^{-(0.295+35000/1000000)} - 100000^{-(0.295+100000/1000000)}\right)$$

= 0.4 kWh/t

HPGR specific energy:

$$W_h = 1*35*(4000 * 35000)^{-.2} * 13.9 * 4 * \left(4000^{-(0.295+4000/1000000)} - 35000^{-(0.295+35000/1000000)}\right)$$
= 2.4 kWh/t

Coarse particle tumbling mill specific energy:

$$W_a = 1*19.4*4*\left(750^{-(0.295+750/1000000)} - 4000^{-(0.295+4000/1000000)}\right)$$
= 4.5 kWh/t

Fine particle tumbling mill specific energy:

$$W_b = 18.8*4*\left(106^{-(0.295+106/1000000)} - 750^{-(0.295+750/1000000)}\right)$$
= 8.4 kWh/t

Total net comminution specific energy:

W_T = 4.5 + 8.4 + 0.4 + 2.4 kWh/t
= 15.7 kWh/t

B 4.3 Conventional Crushing/Ball Milling Circuit

In this circuit primary crusher product is reduced in size to P_{80} of 6.5 mm via a secondary/tertiary crushing circuit (closed). This is then fed to a closed circuit ball mill which grinds to a P80 of 106 µm.

Secondary/tertiary crushing specific energy:

$$W_c = 1*7.2*4*\left(6500^{-(0.295+6500/1000000)} - 100000^{-(0.295+100000/1000000)}\right)$$

= 1.7 kWh/t

Coarse particle tumbling mill specific energy :

$$W_a = 1 * 19.4 * 4 * \left(750^{-(0.295+750/1000000)} - 6500^{-(0.295+6500/1000000)}\right)$$

= 5.5 kWh/t

Fine particle tumbling mill specific energy:

$$W_b = 18.8 * 4 * \left(106^{-(0.295+106/1000000)} - 750^{-(0.295+750/1000000)}\right)$$

= 8.4 kWh/t

Size distribution correction;

$$W_s = 0.19 * 19.4 * 4 * \left(6500^{-(0.295+6500/1000000)} - 100000^{-(0.295+100000/1000000)}\right)$$

= 0.9 kWh/t

Total net comminution specific energy:

W_T = 5.5 + 8.4 + 1.7 + 0.9kWh/t
= 16.5 kWh/t

APPENDIX E – SIZINGS

APPENDIX E
SIZINGS

<u>TABLE E-1</u>
<u>GRIND CALIBRATION DATA</u>

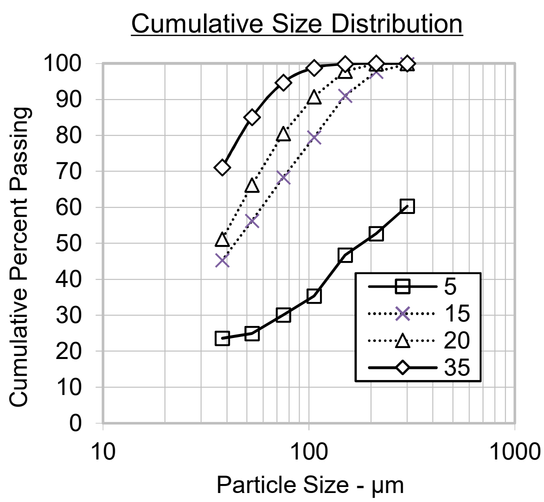
<u>Master Composite</u>

Sieve Size (μm)	Cumulative Percent Passing			
	Grind 1	Grind 2	Grind 3	Grind 4
300	60.3	99.8	100.0	100.0
212	52.7	97.7	99.9	100.0
150	46.7	91.0	97.9	99.8
106	35.3	79.5	90.8	98.8
75	30.1	68.3	80.5	94.7
53	24.9	56.2	66.2	85.1
38	23.6	45.3	51.2	71.1

Parameter	Grind Calibration Data			
	Grind 1	Grind 2	Grind 3	Grind 4
Grind Time - min	5	15	20	35
Sample - g	2000	2000	2000	2000
Water - mL	1000	1500	1500	1500
K_{80} - μm	676	108	74	47

Grinding Mill: BM

Grinding Media: 20kg MS



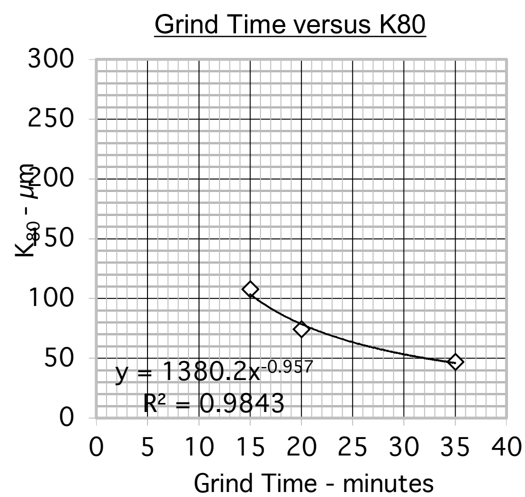


TABLE E-2

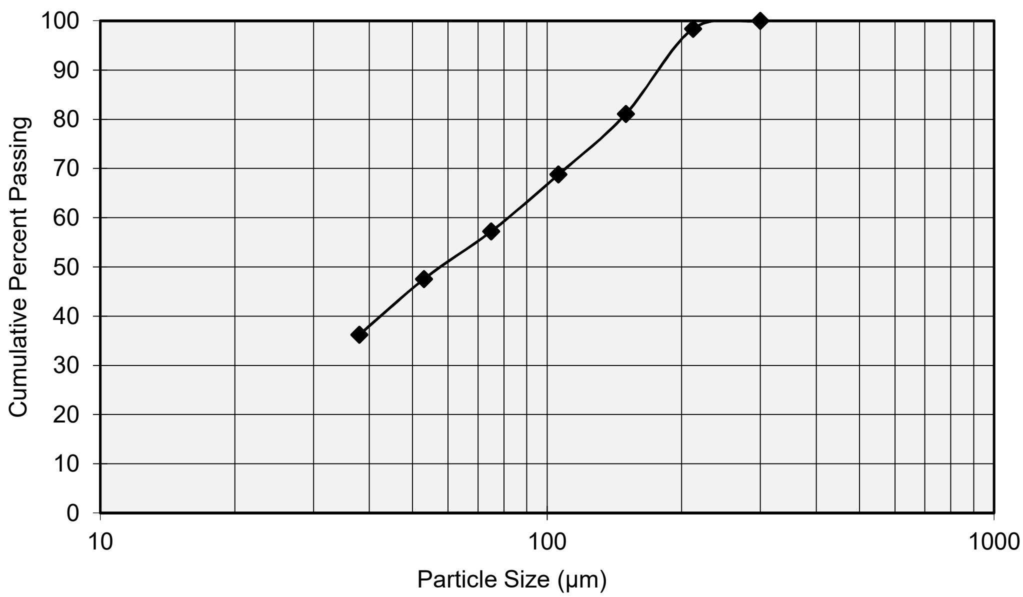
FEED SIZING

Master Composite @ -212μm Feed

Particle Size		Weight (g) Retained	Weight % Retained	Cumulative % Passing
mesh	μm			
48 Mesh	300	0.00	0.00	100.0
65 Mesh	212	8.20	1.64	98.4
100 Mesh	150	86.40	17.28	81.1
150 Mesh	106	61.60	12.32	68.8
200 Mesh	75	57.60	11.52	57.2
270 Mesh	53	48.50	9.70	47.5
400 Mesh	38	56.60	11.32	36.2
TOTAL		500.0	100.00	**

K80 = 146μm

Particle Size Distribution Plot



APPENDIX C

CAMP Report

CHARACTERIZATION
of Metallurgical Processing Products for the Landore-BAM Project
by Automated Mineralogy

Prepared for

Allard Engineering Services

November 9, 2018

Prepared by:

Gary Wyss
Center for Advanced Mineral & Metallurgical Processing (CAMP)
Montana Tech of the University of Montana
1301 West Park Street
Butte, MT 59701
(406) 496-4145
gwyss@mtech.edu

Table of Contents

List of Tables

List of Figures

EXECUTIVE SUMMARY

The Center for Advanced Mineral and Metallurgical Processing (CAMP) received six (6) samples for characterization by automated mineralogy on September 27, 2018. The samples were labeled:

- BL295-03 (Gravity Rougher Con and Tails),
- BL295-12 (Knelson Con and Tails), and
- BL295-14 (Rougher Con and Tails).

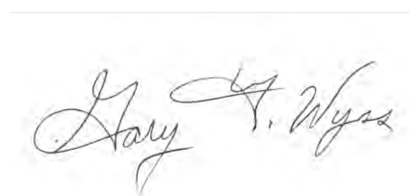
The objective of the study was to determine the occurrence of gold in the process samples to help determine potential paths for recovery optimization. The original samples were analyzed by the Tescan Integrated Mineralogical Analysis (TIMA). TIMA is a scanning electron microscopy-energy dispersive X-ray (SEM-EDS) - based automated mineralogical technique, similar to QEMSCAN and MLA. The samples were sieved into multiple size fractions and analyzed for overall mineralogy. The 100 X 200 mesh fraction was subjected to heavy liquid separation (HLS) using lithium metatungstate (LMT). Unfortunately, the Knelson Concentrate (BL295-12) reacted with the LMT and no "sink" fraction was recovered from that sample.

Only one gold-containing particle was recovered from the tailings samples, which was found in the gravity tails (BL295-03). The gold particle in the tails was trapped in an elongate chlorite particle. Numerous gold particles were found in the Knelson Concentrate, likely providing a reasonable assessment of the gold occurrence in the samples. Many particles were found associated with the major gangue mineral, chlorite. The gold grains in chlorite were mainly elongated, appearing to be flattened out by the same metamorphic forces (compression/shear) responsible for the formation of the phyllosilicate (sheet silicate), chlorite. Some of these elongate gold grains exceeded 100 µm in length with the width being only about a third (and less) of the length. A few smaller fragmented particles of gold were also observed in the chlorite. Gold also appeared to be associated with cobaltite (CoAsS). The cobaltite in the samples typically contained 5 to 10% nickel, as an isomorphous substitution for cobalt. Gold was observed as small blebs of 1 to 10 µm in the subhedral to euhedral cobaltite particles. Gold also occurred as seams at the cobaltite grain boundaries and as larger grains attached to the perimeter. Less commonly observed were inclusions of gold with and in quartz and as small grains with silver and bismuth tellurides. Several large liberated, relatively equidimensional gold particles were found, but offered little insight as to its occurrence. Nearly all of the gold particles examined were 95 to 100% Au according to spot EDS analysis with the lowest gold content grain still at greater than 90% Au.

Relatively few gold particles were found in the HLS "sink" material, so limited information was gained by the extra effort. Regardless, from the HLS it was clear that the relatively dense silicate, kyanite, which has a specific gravity of ~3.6 was enriched in the HLS "sink" material from the tailings samples and in the as-received Knelson Con sample. Chlorite (clinochlore) has a specific gravity ranges from 2.6 to 3.0, depending on its composition. Chlorite was slightly reduced in content in the HLS "sink" relative to the as-received samples but not dramatically, indicating that density separations will need to be monitored closely to remain selective. With that said, rejection of too much chlorite will likely result in gold losses due to its high degree of association with chlorite.

Collectively, silicates were the predominant mineral group at around 95% in all of the tailings samples and at 88 to 90% in the rougher con samples and being the lowest in the Knelson Con (BL295-12) where silicates were 73%. Phyllosilicates (sheet silicates) were 54 to 56% in the tailings samples, 52 to 54% in the rougher cons and 38% in the Knelson Con. Chlorite was the primary phyllosilicate, ranging from 45 to 47% in all of the samples except the Knelson Con where chlorite was still 31% of the overall mineralogy. Quartz, the next most abundant silicate, was 30 to 34% in the tailings and 19 to 24% in the concentrates. The miscellaneous silicates were a group of fine, variable composition silicates and aluminosilicates of variable composition and likely consisted of a mix of silicates and phyllosilicates. The miscellaneous silicates comprised 2 to 3% of the samples, except for the gravity con (BL295-03) where they were ~7% and the flotation con (BL295-14) where they were nearly 9%. Kyanite was 0.4 to 1% in the tailings and concentrates with the exception of the Knelson Con where it jumped to 7%. The primary sulfides were pyrite and pyrrhotite. Both were low in the tailings (<0.1%), slightly higher in the gravity and flotation concentrates (1-2%) but in the Knelson Con pyrite reached 7% and pyrrhotite was 5%.

If further study is warranted, it may be beneficial to perform HLS on finer sieve fraction material, focusing on the tailings samples.



Gary F. Wyss
Laboratory/Equipment Specialist
November 9, 2018

Qualifying Statement

This confidential report was prepared for Allard Engineering Services and their contractors and is based on information available at the time of the report preparation. It is believed the information, estimates, conclusions and recommendations contained herein are reliable under the conditions and subject to the qualifications set forth. Furthermore, the information, estimates, conclusions and recommendations are based on the experience of CAMP and data supplied by others, but the actual result of the work is dependent, in part, on factors over which CAMP has no control.

Scope of Work (Overview)

The purpose of the study was to determine the disposition (deportment and associations) of gold in the provided tailings and concentrates generated from gravity, flotation and Knelson separation studies.

The as-received samples were wet sieved into five size fractions with heavy liquid separation (HLS) performed on the 100 X 200 mesh fraction. Unfortunately, the heavy liquid reacted with the Knelson concentrate (BL295-12), preventing a definitive separation from that sample. Automated mineralogical (AM) analysis was performed on sized specimens from the original samples and five of the specimens created from HLS.

TIMA Analysis

Particle mounts were prepared from each sample consisting of five sieve fractions. Random particle mounts were created from the larger sieve fractions (i.e., +100 & 100 X 200) with transverse (cross-sectional) mounts from the finer sieve fractions (i.e., 200 X 325, 325 X 400 and -400). Random particle mounts were also prepared from the HLS "sink" material (100 X 200 mesh) with the exception of the Knelson Con (BL295-12).

TIMA data was obtained by the liberation method where the specimen is mapped in a grid-like fashion using backscattered electron (BSE) imaging and EDS X-ray analysis. The area for each grid is based on the assigned pixel spacing ranging from 3 μm to 15 μm depending on the particle size fraction, the smaller spacing for finer fractions and larger for larger particles. The BSE image is used to delineate phase boundaries and the X-ray spectrum data is used to determine mineral phase identity based on mineral identification protocol the surface area data in pixels is used for quantitative determination of the minerals identified.

Mechanical Sieve Analysis

The as-received samples were wet-sieved through 100, 200, 325 and 400 US mesh screens. The mass distribution results are shown in Table 1. It can be seen from the mass distribution results that the gravity and flotation separations generated concentrates that were finer than the tails, while the Knelson concentrate was coarser than the tailings.

Table 1. Mass distribution of as-received samples (Wt. %).

Sieve Fraction (US mesh)	**BL295-03 Gravity Tails**	**BL295-03 Gravity Con**	**BL295-12 Knelson Tails**	**BL295-12 Knelson Con**	**BL295-14 Rougher Tails**	**BL295-14 Rougher Con**
+100	2.5	0.9	24.5	48.6	23.4	19.7
100 X 200	20.3	5.8	22.6	28.5	22.8	9.4
200 X 325	23.5	10.9	13.3	12.1	13.8	6.2
325 X 400	3.1	3.6	3.8	2.6	3.5	2.6
-400	50.6	78.9	35.8	8.2	36.6	62.1
Total	100	100	100	100	100	100

TIMA Particle Size Distribution

The TIMA-calculated particle size distributions are shown in Figure 1. Due to the default colors it is difficult to discern some of the samples, but the Knelson Con contained the largest particles and the gravity Rougher Con (BL295-03) was the finest as indicated. The statistical median (P_{50}) and P_{80} are tabulated in Table 2 which show that the Knelson Con was coarsest with a P_{80} of 178 µm and the gravity Rougher Con (BL295-03) was the finest with a P_{80} of 24 µm.

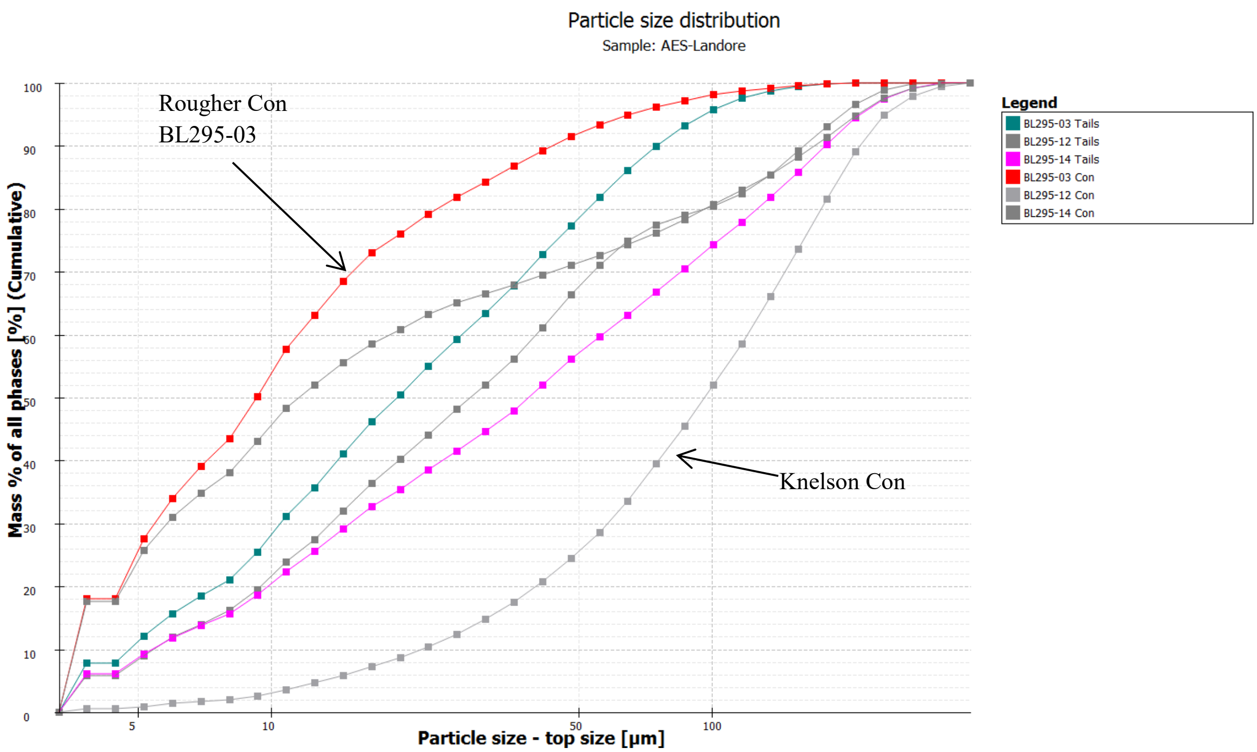


Figure 1. TIMA particle size distributions for the as-received samples.

Table 2. TIMA-calculated particle size statistics (µm).

Parameter	**BL295-03 Gravity Rougher Tails**	**BL295-03 Gravity Rougher Con**	**BL295-12 Knelson Tails**	**BL295-12 Knelson Con**	**BL295-14 Rougher Tails**	**BL295-14 Rougher Con**
Median	19	9	29	97	39	12
P_{80}	53	24	98	178	127	97

Modal Mineralogy

All of the samples contained a majority of silicates consisting mainly of chlorite and other phyllosilicates and quartz. Concentrate samples were all enriched in sulfides, tellurides and arsenides relative to their corresponding tailings samples and oxides were greatest in the Knelson Con.

Chlorite was the most abundant silicate and ranged from 45 to 47% in all the samples except for the Knelson Con (BL295-12) where it was 31%. Quartz was 30 to 33% in the tailings samples, 24% in the Knelson Con and 19 to 21% in the gravity and flotation con samples. The miscellaneous silicates consisted mainly of chlorite and quartz that were too fine and intimately associated to distinguish by automated mineralogy techniques. Garnet content is likely misclassified chlorite due to the similarity of the EDS patterns. Kyanite was present at 1% or less in the samples with the exception of the Knelson concentrate where it was 7% due to its high specific gravity relative to other silicates. The main oxides were hematite/magnetite and rutile which were also greatest in the Knelson Con at 3.1% and 1.6%, respectively. Interestingly, native iron (tramp iron?) was 2.8% in the Knelson Con. It should be noted that the Knelson Con sample was a "rusty" color which stood out from the other samples that were essentially gray to greenish-gray in appearance.

The predominant sulfides were pyrite and pyrrhotite. Pyrite was 1% in the gravity/flotation cons and 7% in the Knelson con and at trace levels in the tailings. Pyrrhotite trended similarly, reaching 5% in the Knelson con. The cobalt sulfide-arsenide, cobaltite (CoAsS) was 2.5% in the Knelson con and was a key mineral in the study due to the notable association with gold in the ore. Commonly, the cobaltite contained 5 to 8% nickel and was occasionally observed to be around 15% when manually examined by EDS. The base metal sulfide chalcopyrite was most prevalent in the cons, reaching 0.66% in the Knelson Con. Sphalerite and galena were present but only at trace levels.

Gold was found in the Knelson Con at 0.033% and at trace levels in the Rougher Con (BL295-14). Bright phase analysis was performed and detected more gold particles/grains and is discussed later in the report.

Table 3 is a comprehensive listing of minerals identified in the original samples.

Table 3. Modal Mineral concentrations (weight %).

Mineral	BL295-03 Tails	BL295-03 Con	BL295-12 Tails	BL295-12 Con	BL295-14 Tails	BL295-14 Con
Chlorite - Clinochlore	44.8	46.9	46.0	31.3	47.2	44.9
Quartz	33.7	20.8	31.3	24.2	30.5	18.7
Muscovite	8.38	8.53	8.11	5.32	8.53	6.17
Misc. Silicates	2.23	6.61	3.11	1.81	3.03	8.61
Garnet	2.04	3.00	1.81	0.58	1.83	2.87
Kyanite	1.03	0.54	1.06	7.19	0.99	0.39
Pyrite	P	1.20	0.04	7.15	0.01	1.25
Plagioclase	2.03	1.26	1.81	1.56	1.74	0.96
Pyrrhotite	P	0.92	0.09	5.04	0.02	1.83
Amphibole	0.13	1.47	0.39	0.48	0.33	4.61
Rutile	0.85	0.80	0.76	1.60	0.77	0.78
Hematite/Magnetite	0.10	0.27	0.21	3.12	0.16	0.63
Calcite	0.84	0.67	0.88	0.80	0.68	0.52
Kaolinite	0.48	0.56	0.47	0.73	0.41	0.43
Iron	0.09	0.11	0.01	2.78	0.01	0.04
Ankerite	0.39	0.45	0.42	0.46	0.35	0.74
Dolomite	0.32	0.49	0.40	0.39	0.30	0.88
Cobaltite	P	0.11	P	2.48	P	0.06
Chalcopyrite	P	0.21	0.02	0.66	P	0.37
Biotite	0.22	0.14	0.22	0.11	0.27	0.04
Pyroxene	P	0.20	0.02	0.01	P	0.52
Apatite	0.14	0.11	0.12	0.20	0.11	0.08
Cassiterite	0.09	0.15	0.05	0.05	0.06	0.10
Schorl	0.05	0.03	0.10	0.13	0.09	0.03
Ilmenite	0.02	0.03	0.01	0.15	0.01	0.02
Sphalerite	P	0.01	P	0.03	ND	0.08
K-Feldspar	0.02	0.04	0.01	0.01	P	0.01
Galena	P	P	ND	0.03	P	0.02
Scheelite	ND	ND	ND	0.04	ND	ND
Gold	**ND**	**ND**	**ND**	**0.033**	**ND**	**P**
Zircon	P	P	P	0.02	P	P
Pentlandite	ND	0.01	P	0.01	P	P
Monazite	P	P	P	0.02	P	P
Barite	ND	P	P	0.02	P	P
Arsenopyrite	ND	ND	ND	0.01	P	ND

P – mineral present, but found at less than 0.01%

ND – mineral not detected

Mineralogy of the original samples is presented by mineral groupings in Table 4 which shows that silicates were 94 to 95% in the tailings and 88 to 90% in the rougher con samples and only 73% in the Knelson Con. The phyllosilicates were 51 to 56% in the samples but only 38% in the Knelson Con. Sulfides and the associated tellurides and arsenides were higher in the cons and reached 15% in the Knelson Con. Oxides were between 1 and 1.5% in most of the samples also reaching a maximum in the Knelson Con at 5%. Carbonates were low and were 1 to 2% throughout. The phosphates, sulfates and others were collectively 0.1 to 0.2%, again with the exception of the Knelson Con where they were 3% and mainly due to the presence of iron.

Table 4. Content by mineral grouping in the original samples (weight %).

Mineral Group	**BL295-03 Tails**	**BL295-03 Con**	**BL295-12 Tails**	**BL295-12 Con**	**BL295-14 Tails**	**BL295-14 Con**
Total Silicates	95.2	90.1	94.4	73.4	95.0	88.2
Phyllosilicates	53.9	56.1	54.8	37.5	56.4	51.5
Other Silicates	41.3	34.0	39.6	36.0	38.5	36.7
Sulfides, Tellurides & Arsenides	0.01	2.47	0.15	15.4	0.03	3.63
Oxides	1.05	1.25	1.04	4.95	1.00	1.53
Carbonates	1.55	1.62	1.70	1.65	1.33	2.14
Phosphates, Sulfates & Others	0.22	0.22	0.14	3.02	0.12	0.12
Au/Ag Minerals	ND	ND	ND	0.033	ND	0.0001

ND – mineral not detected

TIMA-Calculated Composition

The TIMA-calculated bulk elemental content presented in Table 5 was derived from the TIMA modal mineralogy and the assigned chemistry found in the Appendix. Gold-containing minerals were found in the Knelson Con and the Rougher Con (BL295-14) but was only calculated at 0.031 in the Knelson Con and <0.001% in the Rougher Con, too low to be of quantitative value. Arsenic, a potential penalty element was calculated at nearly 1% in the Knelson Con.

Table 5. MLA-calculated elemental composition for the original samples (Wt. %)

Element	**BL295-03 Tails**	**BL295-03 Con**	**BL295-12 Tails**	**BL295-12 Con**	**BL295-14 Tails**	**BL295-14 Con**
Oxygen	49.0	46.3	48.6	39.4	48.7	45.7
Silicon	25.8	21.8	25.0	19.5	24.8	21.5
Iron	5.58	7.28	5.83	15.8	5.87	7.91
Magnesium	7.40	8.30	7.64	5.15	7.80	8.76
Aluminum	7.02	7.54	7.11	6.99	7.26	6.97
Sulphur	P	1.09	0.06	6.47	0.01	1.53
Potassium	0.83	0.93	0.83	0.54	0.87	0.78
Calcium	0.74	0.63	0.75	0.74	0.61	0.67
Hydrogen	0.64	0.68	0.66	0.46	0.68	0.65
Titanium	0.51	0.49	0.46	1.01	0.47	0.48
Carbon	0.19	0.20	0.21	0.20	0.16	0.26
Arsenic	**P**	**0.04**	**P**	**0.96**	**P**	**0.02**
Cobalt	P	0.02	P	0.51	ND	0.01
Copper	P	0.07	0.01	0.23	P	0.13
Tin	0.07	0.12	0.04	0.04	0.04	0.08
Sodium	0.06	0.08	0.06	0.05	0.06	0.09
Nickel	P	0.02	P	0.31	P	0.01
Fluorine	0.06	0.05	0.05	0.04	0.06	0.03
Phosphorus	0.02	0.02	0.02	0.04	0.02	0.01
Zinc	P	0.01	P	0.02	ND	0.05
Manganese	0.01	0.01	0.01	0.01	0.01	0.02
Lead	P	P	ND	0.03	P	0.02
Gold	**ND**	**ND**	**ND**	**0.031**	**ND**	**P**
Tungsten	ND	ND	ND	0.03	ND	ND
Boron	P	P	P	P	P	P
Barium	ND	P	P	0.01	P	P
Zirconium	P	P	P	0.01	P	P
Cerium	P	P	P	0.01	P	P
Antimony	ND	ND	ND	P	ND	P
Bismuth	ND	P	ND	P	ND	P
Tellurium	ND	P	ND	P	ND	P
Lanthanum	P	P	ND	P	P	P
Neodymium	P	P	ND	P	ND	P
Silver	**ND**	**ND**	**ND**	**P**	**ND**	**ND**

P – element present, but calculated at less than 0.01%

ND – mineral(s) containing this element were not encountered

TIMA Analysis of Pre-concentrated Sink Fractions

Heavy Liquid Separation (HLS)

Heavy liquid separation (HLS) was conducted on the 100 X 200 mesh sieve fraction from each sample. Lithium metatungstate (LMT) was the heavy liquid used which has a density of 2.9 g/ml. Unfortunately, the specimen from the Knelson Con (BL295-12) reacted with the LMT forming a dark blue solution/mixture rendering it inseparable. Therefore, no HLS "sink" specimen was prepared from the Knelson Con. The separation was successful for the tailings samples where 3% of the sample reported to the "sink" fraction and 9 to 13% of the concentrates reported to the "sink" fraction (with the exception of the Knelson Con) as shown in Table 6.

Table 6. Distribution of HLS "sink" material (%).

Sample ID	"Sink" Fraction (%)
BL295-03 Tails	3.1
BL295-12 Tails	3.3
BL295-14 Tails	3.3
BL295-03 Con	13.3
BL295-12 Con	--
BL295-14 Con	9.4

Mineral Content of Pre-concentrated Material

Silicates remained fairly high in the HLS "sink" material from the tailings. Kyanite was especially pronounced in the tailings and chlorite persisted but a lower content, relative to the original samples. In the concentrate "sink" material kyanite decreased markedly and chlorite remained high in the gravity Rougher Con from BL295-03 and then dropped in the BL295-14 Rougher Con. The sulfides, pyrrhotite and pyrite, were enriched slightly in the tailings but much more so in the concentrates at 23 to 24% in the BL295-03 gravity con and 33 to 34% in the BL295-14 Rougher Con. A thorough analysis of the "sink" mineralogy is shown below in Table 7.

Unfortunately, no gold-containing particles were found in the tailings HLS "sink" material, with the exception of one particle identified upon manual review of the BL295-03 gravity Tails. A few particles containing gold were identified in the BL295-03 Con and a handful in the BL295-14 Rougher Con. The gold-containing particles reviewed by manual inspection are detailed in the following section.

Table 7. Mineral content of the heavy liquid concentrate - sink fraction (weight %).

Mineral	BL295-03 Gravity Tails	BL295-12 Gravity Tails	BL295-14 Tails	BL295-03 Rougher Con	BL295-14 Rougher Con
Kyanite	42.3	37.8	43.8	3.15	0.83
Chlorite - Clinochlore	25.1	19.1	17.0	22.5	3.63
Pyrrhotite	0.01	2.18	0.93	23.8	32.9
Pyrite	0.03	2.07	0.17	22.8	33.9
Quartz	8.34	9.96	11.1	4.68	3.97
Misc-Silicates	4.34	4.73	4.28	2.67	1.40
Rutile	2.98	3.74	3.53	2.71	4.03
Muscovite	3.40	2.99	2.76	2.77	0.25
Kaolinite	3.31	3.61	3.96	0.55	0.12
Chalcopyrite	P	0.39	P	3.01	7.11
Hematite/Magnetite	0.43	0.84	1.11	1.79	2.81
Plagioclase	1.06	1.42	1.44	0.50	0.62
Biotite	1.14	1.06	0.99	0.47	0.09
Apatite	0.40	1.32	1.17	0.06	0.06
Garnet	0.58	0.52	0.36	0.74	0.10
Amphibole	0.20	0.29	0.31	0.79	0.50
Cobaltite	P	0.02	P	1.09	0.31
Calcite	0.04	0.19	0.17	0.13	0.81
Ankerite	0.09	0.16	0.13	0.20	0.38
Dolomite	0.05	0.11	0.12	0.27	0.25
Ilmenite	0.16	0.24	0.21	0.03	0.02
Schorl	0.18	0.15	0.13	0.16	0.03
Cassiterite	0.07	0.18	0.15	0.05	0.11
Sphalerite	ND	0.07	0.01	0.09	0.37

Pentlandite	ND	0.01	P	0.08	0.09
Titanite	0.03	0.04	0.03	0.01	0.01
Gold	**ND**	**ND**	**ND**	**0.042**	**0.034**
Barite	ND	0.01	P	0.01	0.04
Enstatite	P	P	P	0.04	0.02
Arsenopyrite	ND	ND	ND	0.01	0.04
Iron	P	0.01	0.01	P	0.02
Galena	ND	P	ND	P	0.04
Zircon	P	0.01	0.01	0.01	0.01
K-Feldspar	P	0.01	P	P	0.01
Diopside	P	P	P	P	P
Bi-Tellurides	ND	P	P	P	P
Monazite	P	P	P	P	P
Ullmannite	ND	ND	ND	P	P
Tennantite	ND	ND	ND	ND	P

P – mineral present, but found at less than 0.01%
ND – mineral not detected

Occurrence Gold/Silver-Containing Phases (Grain Size & associations)

Gold was found primarily associated with chlorite and cobaltite. Also, a few gold grains were found with the silver telluride, hessite, and associated with bismuth telluride which also occasionally contained silver. Typically, small gold grains of 1 to 15 µm were found as inclusions in cobaltite and larger seams and attached gold at the cobaltite grain boundaries. Larger elongate grains occurred interlayered with chlorite.

Numerous particles containing gold were identified in the study. Most of the gold-containing particles were encountered from the modal analysis of the Knelson Con (BL295-12). Several more particles were found in the analysis of the HLS “sink” specimens and all that were reviewed manually are presented in the following table. The approximate size of the manually observed gold/silver-containing grains is presented and for the elongated grains, two dimensions are shown. Several of the elongate gold grains were observed interbedded with the metamorphic mineral, chlorite, and speculated that the liberated elongate gold particles were likely liberated from chlorite. When a gold grain was present as an inclusion or shared much of its boundary with another mineral it is referred to as the host and when other minerals were present within the host mineral they are cited in the association column in Table 8.

The gold grains/particles found in the samples were extremely high in gold content. Spot EDS analysis determined that nearly all of the gold particles were 95 to 100% Au. All of the grains examined were greater than 90% Au.

Table 8. Gold/silver-containing particle deportment.

Particle No.	Sample	Au Phase	Approx. Size (μm) (%)	Host	Association
1	BL295-03 Tails	Gold	40 X 15	Chlorite	--
2	BL295-03 Con	Gold	30	Quartz	--
3	BL295-03 Con	Gold	5	Cobaltite	--
4	BL295-12 Con	Gold	1 - 5	Cobaltite	Bi-Telluride
5	BL295-12 Con	Gold	1 - 5	Cobaltite	--
6	BL295-12 Con	Gold	35 X 10	Quartz?	--
7	BL295-12 Con	Gold	3 - 15	Cobaltite	Chlorite-Rutile
8	BL295-12 Con	Gold	50 - 60	Liberated	--
9	BL295-12 Con	Gold	30 X 10		Chlorite
10	BL295-12 Con	Gold	35	Liberated	--
11	BL295-12 Con	Gold	5 X 50	Liberated	FeO
12	BL295-12 Con	Gold	1 - 5	Hessite	--
13	BL295-12 Con	Gold	40 X 150	Quartz/Chlorite	--
14	BL295-12 Con	Gold	5 X 55	Cobaltite	Bi-Telluride
15	BL295-12 Con	Gold	10 X 65	Chlorite	Cobaltite
16	BL295-12 Con	Gold	60 X 125	Liberated	--
17	BL295-12 Con	Gold	25 X 125	Liberated	Chlorite
18	BL295-12 Con	Gold	15 X 115	Chlorite	Pyrrhotite?
19	BL295-12 Con	Gold	65	Liberated	--
20	BL295-12 Con	Gold	25 - 30		Cobaltite/BiSb-Telluride
21	BL295-12 Con	Gold	15 X 50	Chlorite	--
22	BL295-12 Con	Gold	35 - 55	Liberated	Cobaltite
23	BL295-12 Con	Gold	25 - 100	Liberated	--
24	BL295-12 Con	Gold	50 - 60	Liberated	--
25	BL295-12 Con	Gold	10	Bi-Telluride	--
26	BL295-12 Con	Gold	3 - 10	Cobaltite	Bi-Telluride
27	BL295-12 Con	Gold	1 - 5	Cobaltite	--
28	BL295-12 Con	Gold	25		Cobaltite
29	BL295-12 Con	Gold	1 - 5	Cobaltite	Chalcopyrite
30	BL295-12 Con	Gold	10	Cobaltite	Altaite (Pb-Telluride)
31	BL295-12 Con	Gold	30	Liberated	--
32	BL295-12 Con	Gold	20	Liberated	--
33	BL295-14 Con	AgBiTe	20		Quartz
34	BL295-14 Con	Gold	5	Chlorite	Ag-Bi Telluride
35	BL295-14 Con	Gold	5 - 10	Quartz	--
36	BL295-14 Con	Hessite	20	Chlorite	--
37	BL295-14 Con	Hessite	20	Pyrrhotite	Bi-Telluride

The Knelson Con had the most and largest gold particles as categorized in Table 9 and graphically displayed in Figure 2. Generally, there is relative agreement in grain size between manual and TIMA-determined values; however, it appears that the TIMA-determined grain size may be smaller than observed. This may in part be due to the manner in which TIMA handles the geometry of the elongate particles.

Table 9. TIMA-determined gold grain size frequency.

Size range / Number of grains of Gold	Midpoint (µm)	BL295-03 Con	BL295-12 Con	BL295-14 Con
≥3.3<5.5 µm	4.3	3	0	3
≥5.5<9.3 µm	7.2	0	4	4
≥9.3<16 µm	12	2	6	1
≥16<26 µm	20	1	9	2
≥26<44 µm	34	2	5	1
≥44<73 µm	56	1	2	1
Total		**9**	**26**	**12**

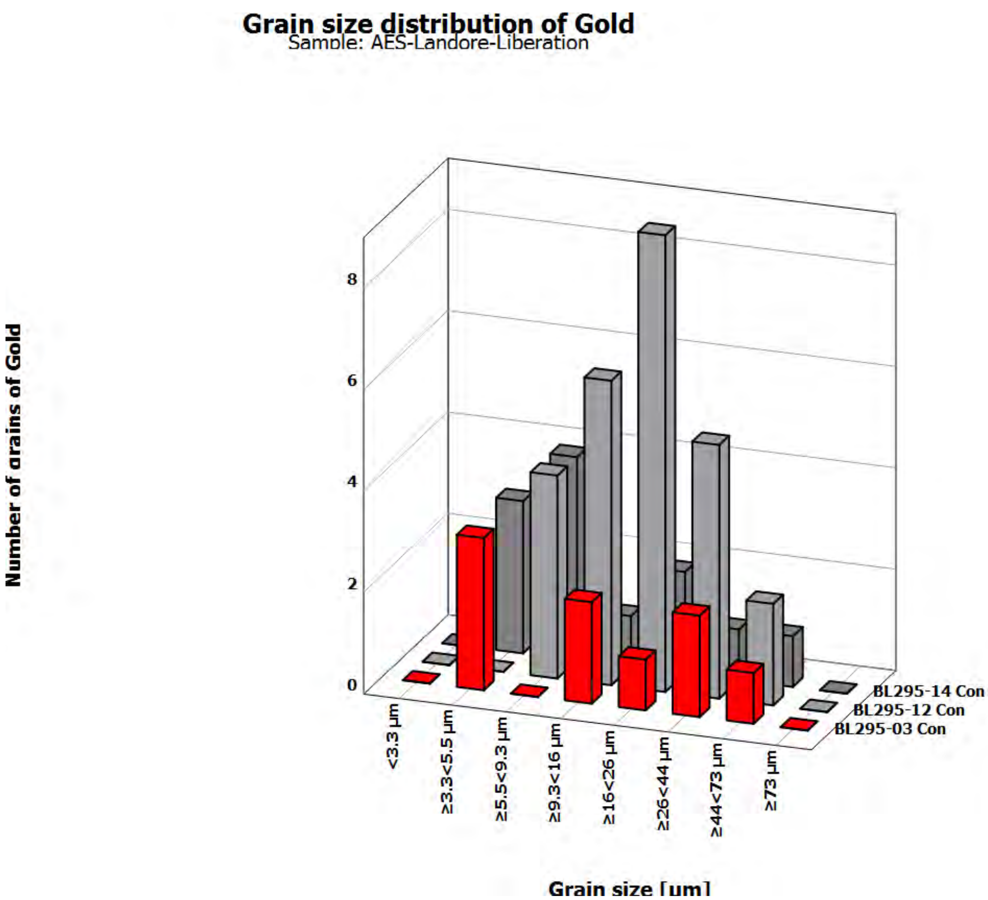


Figure 2. 3-D plot of the gold grains in the concentrate samples.

TIMA Images – Gravity Rougher Tails (BL295-03)

Figure 3 is a classified false color image from the Rougher Tails (BL295-03) sample. The sample was composed of mainly clinochlore (lt green) and quartz (lt pink). The associated backscattered electron (BSE) image is shown in Figure 4. The darker phase is chlorite and the brighter phase is pyrite in the circled particle.

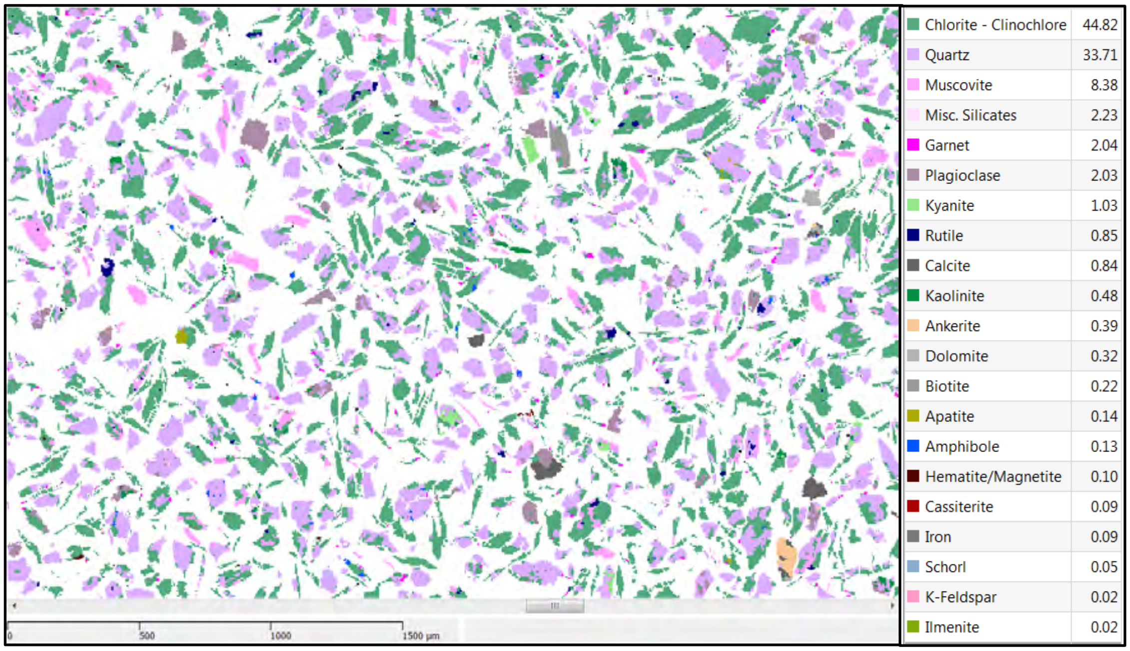


Figure 3. Classified TIMA image from BL295-03 Tails. Concentration palette values are in mass percentage.

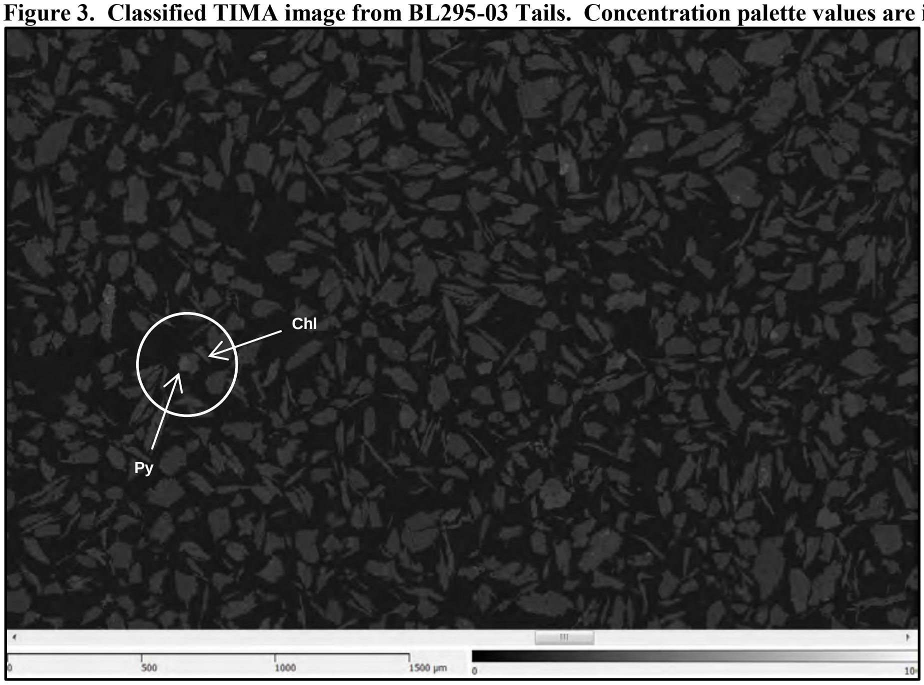


Figure 4. BSE image from sample BL295-03 Tails.

Kyanite was the main constituent of the HLS "sink" material from the BL295-03 Tails and is shown in light green, which is difficult to discern from chlorite in the false color image in Figure 5 as both are light shades of green with chlorite being a slighter darker hue. The morphology of kyanite is typically long, slender particles with angular cleavage. Both kyanite and chlorite are silicates and they appear darker than pyrite in the BSE image in Figure 6.

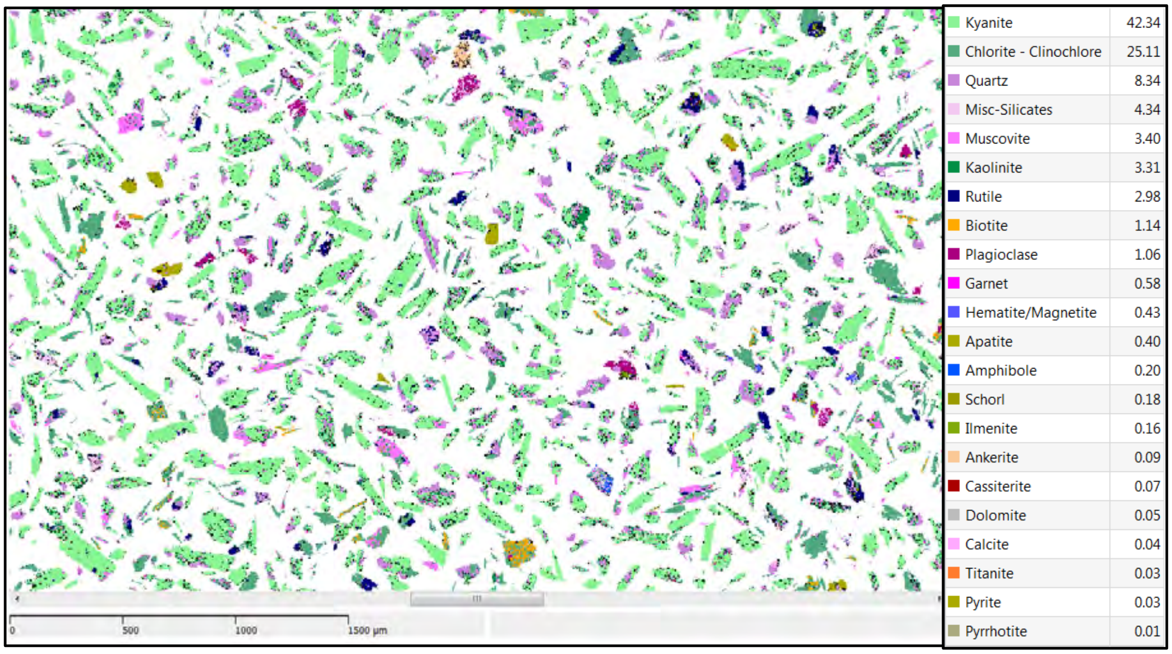


Figure 5. Classified TIMA image from BL295-03 Tails HLS "sink" fraction. Concentration palette values are in mass percentage

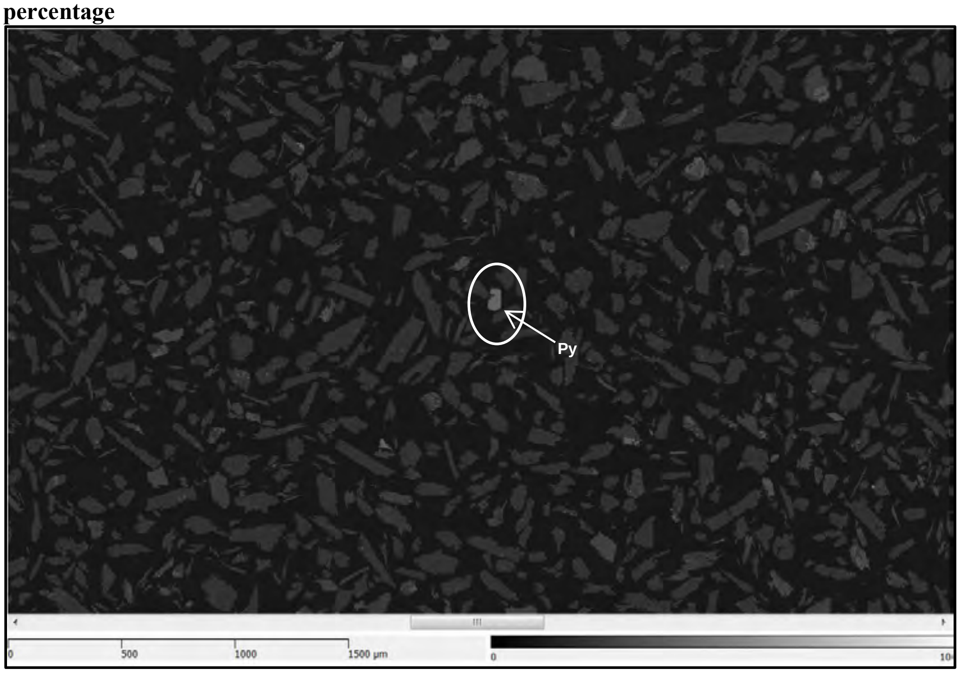


Figure 6. BSE image from sample BL295-03 Tails HLS "sink".

Gravity Rougher Tails (BL295-03) Particle Photomicrographs.

The only gold grain found in the tailings samples occurred, interlayered with chlorite. The elongate grain was approximately 40 X 15 µm (Figure 7).

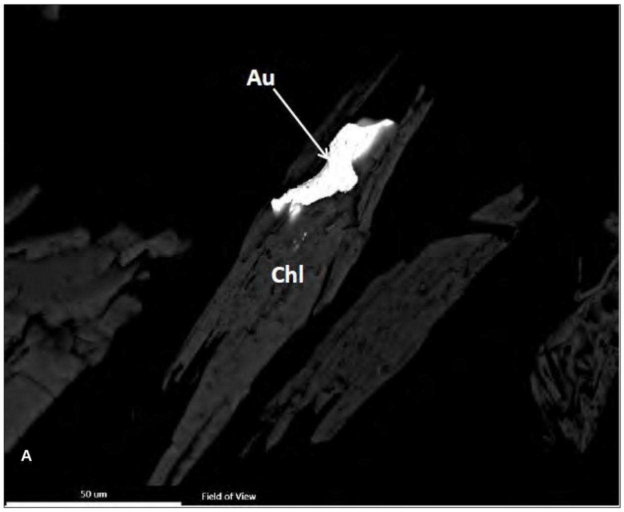


Figure 7. Gold grain trapped in chlorite from the gravity tails sample (BL295-03).

TIMA Images – Rougher Con (BL295-03)

The false color image in Figure 8 shows is mostly light green and pink due to chlorite and quartz and the orange particles are chalcopyrite. Chalcopyrite and pyrite are identified in the corresponding BSE image in Figure 9.

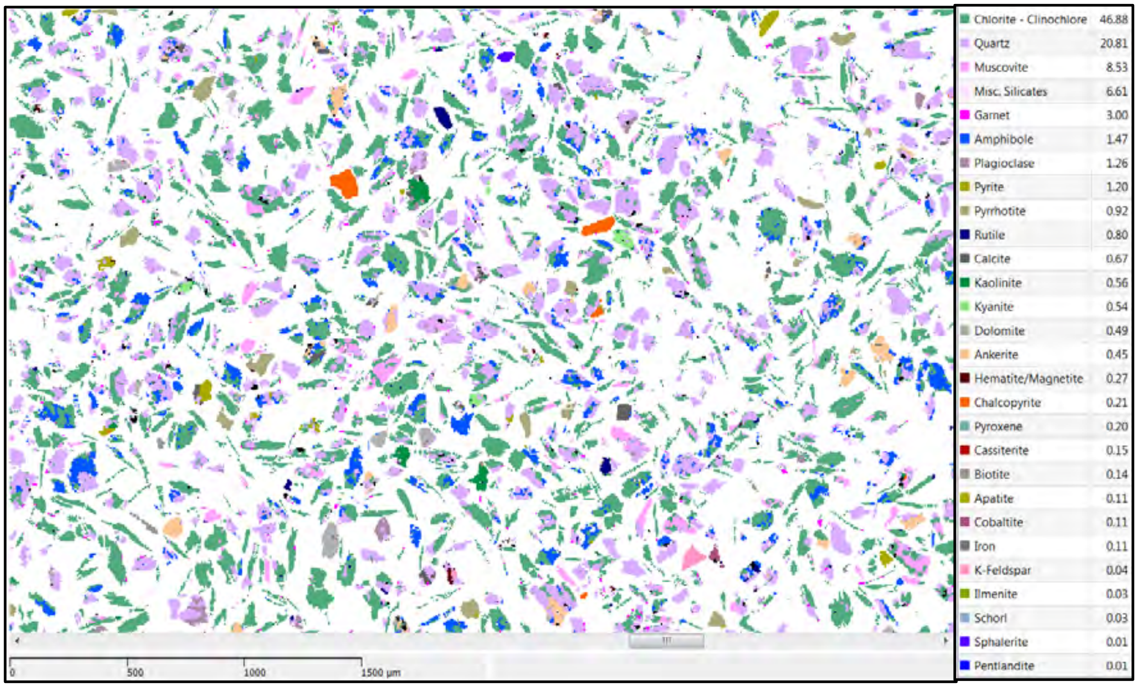


Figure 8. Classified TIMA image from BL295-03 Con. Concentration palette values are in mass percentage.

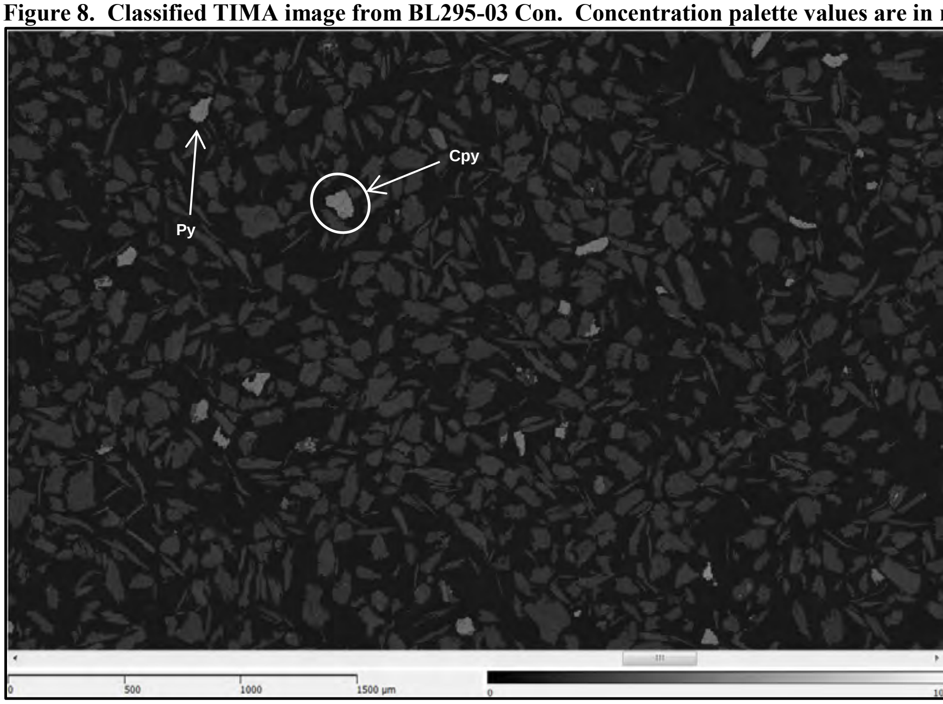


Figure 9. BSE image from sample BL295-03 Con.

Pyrrhotite (lt gray) and pyrite (olive) are the main constituents in the HLS "sink" fraction from BL295-03 Gravity Con in Figure 10. Gold is yellow in the false color image in Figure 10 and white in the BSE image in Figure 11.

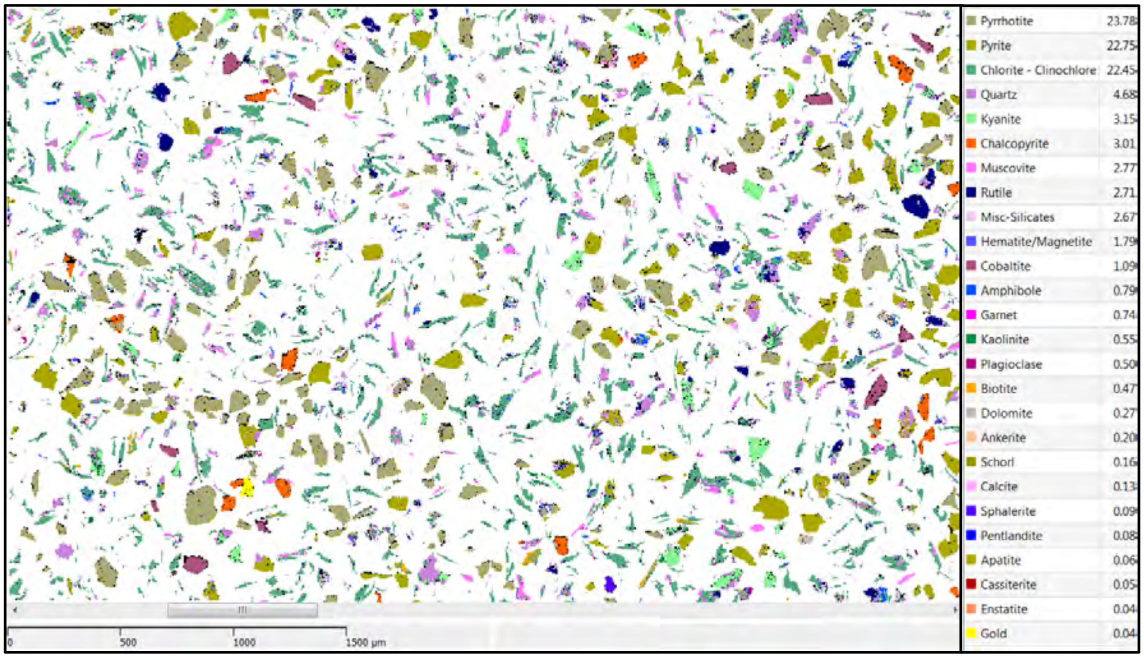


Figure 10. Classified TIMA image from BL295-03 Con HLS "sink" fraction. Concentration palette values are in mass percentage.

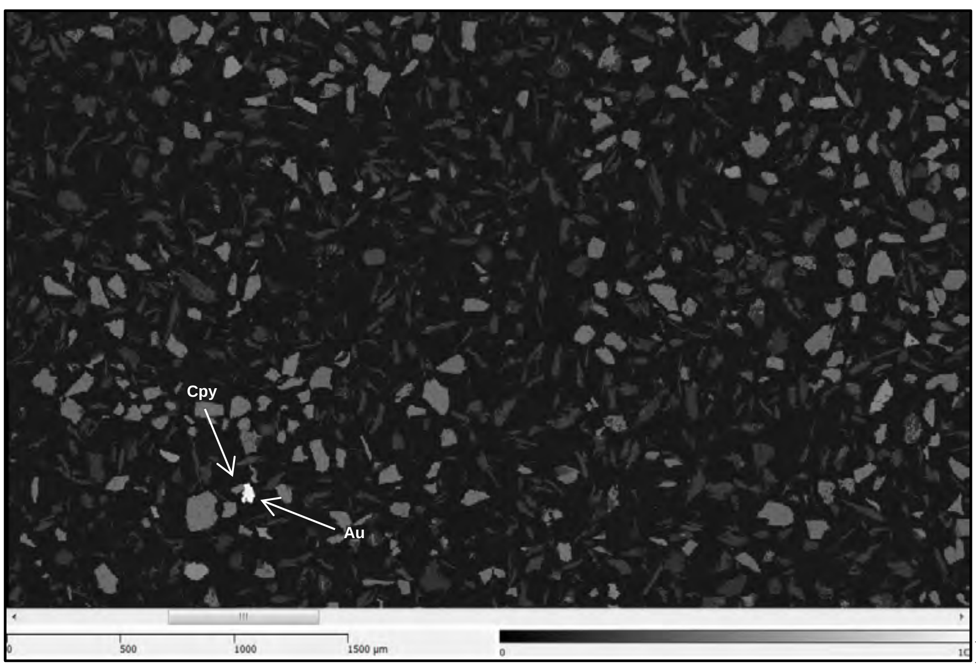


Figure 11

Figure 11. BSE image from sample BL295-03 Con HLS "sink" fraction.

Gravity Rougher Con (BL295-03) Gold-Containing Particle Photomicrographs.

A small 5 μm gold inclusion in cobaltite is shown in Figure 12 A and a large 30 μm gold grain is attached to quartz in Figure 12B.

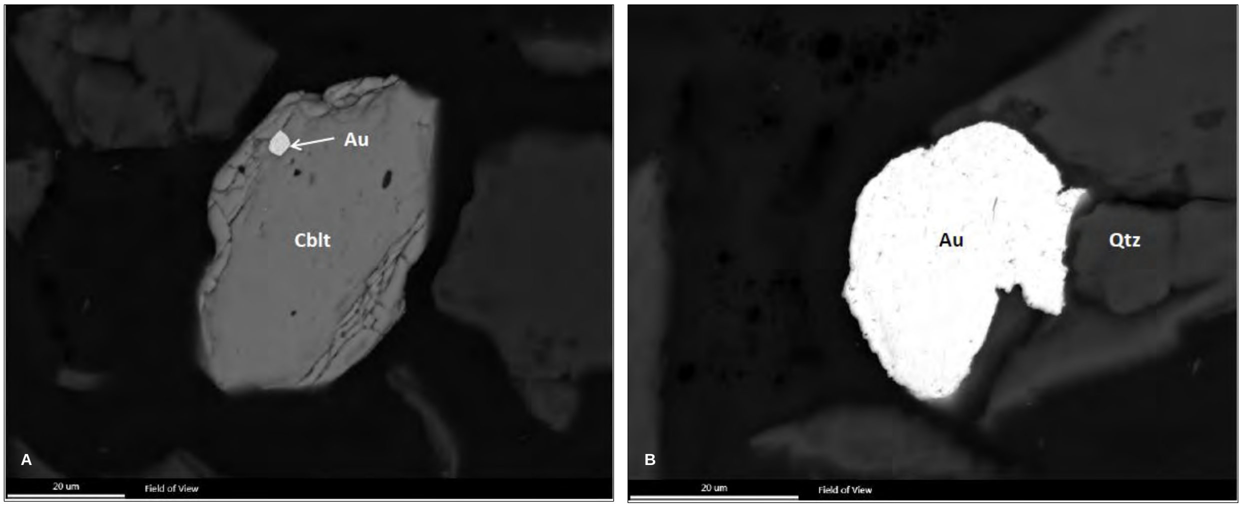


Figure 12. Small, locked gold inclusion in cobaltite (A) and large gold grain attached to quartz (B).

TIMA Images – Knelson Tails (BL295-12)

As seen in the rougher tails sample, the Knelson Tails (BL295-12) was composed of primarily chlorite (lt green) and quartz (lt pink) as shown in the false color image (Figure 13). Chalcopyrite is identified in the BSE image in Figure 14.

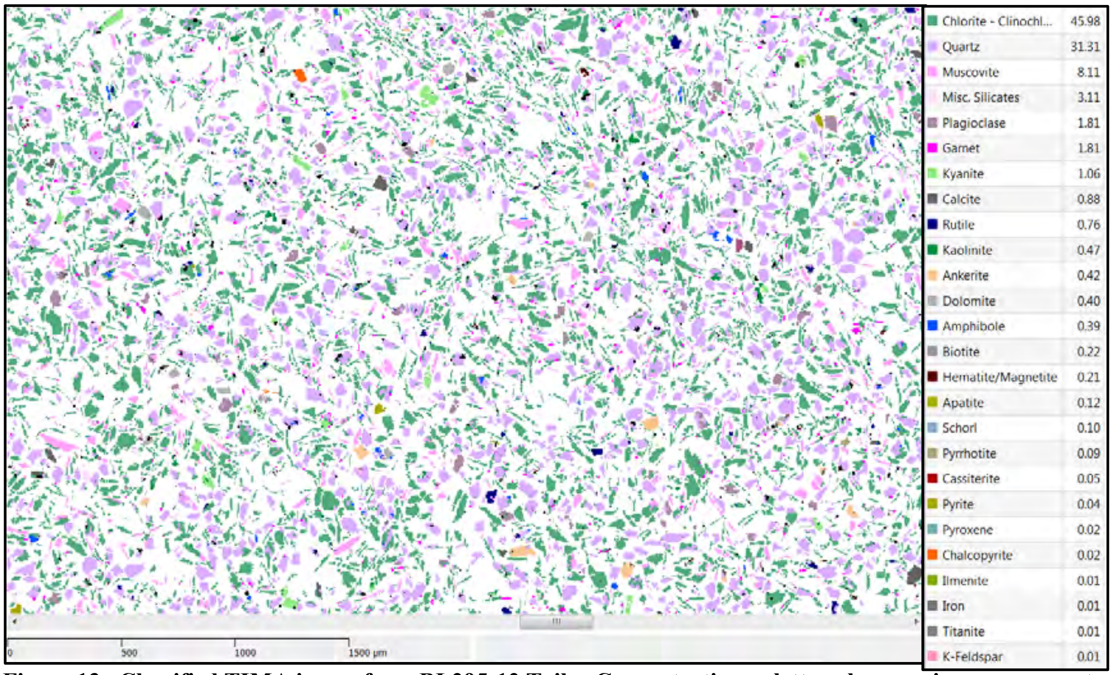


Figure 13. Classified TIMA image from BL295-12 Tails. Concentration palette values are in mass percentage.

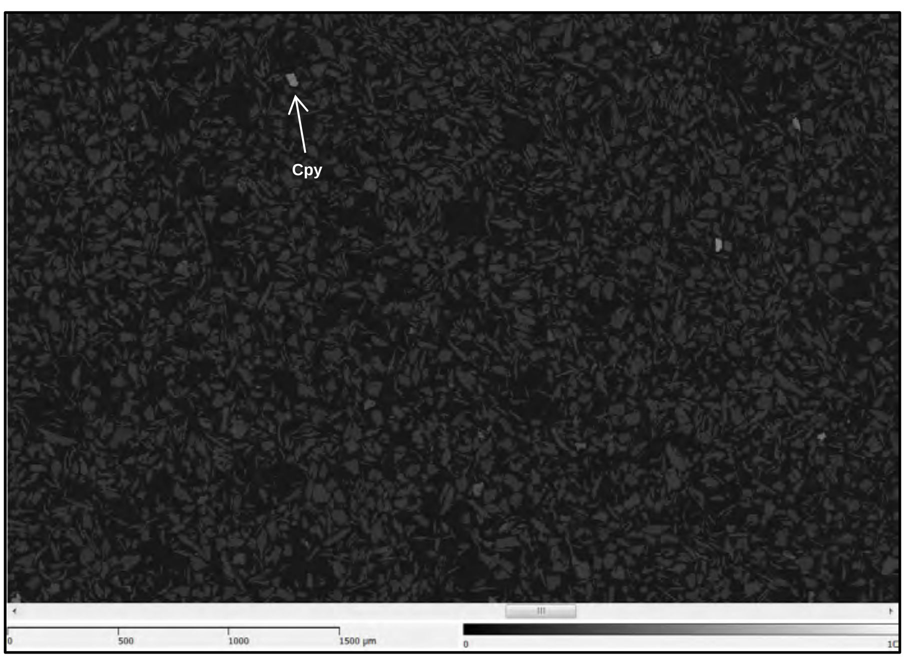


Figure 14. BSE image from sample BL295-12 Tails.

Pyrite (olive) stands out from the mostly green particles of kyanite and chlorite in the HLS "sink" material from the Knelson Tails in Figure 15. Pyrite is slightly brighter than the silicates as pointed out in the BSE image in Figure 16.

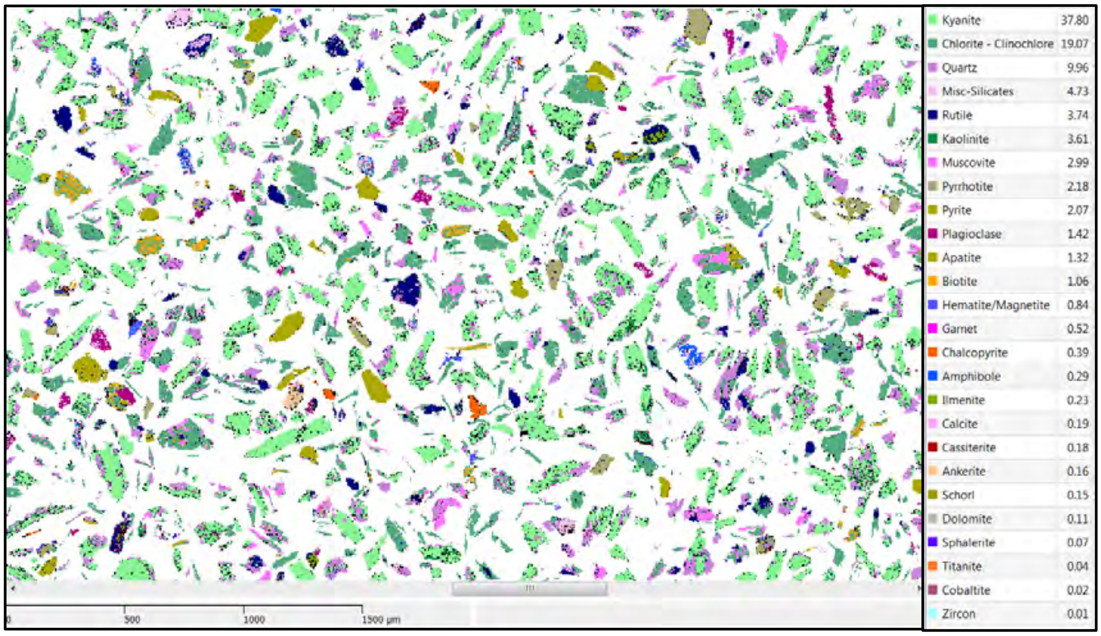


Figure 15. Classified TIMA image from BL295-12 Tails HLS "sink" fraction. Concentration palette values are in mass percentage.

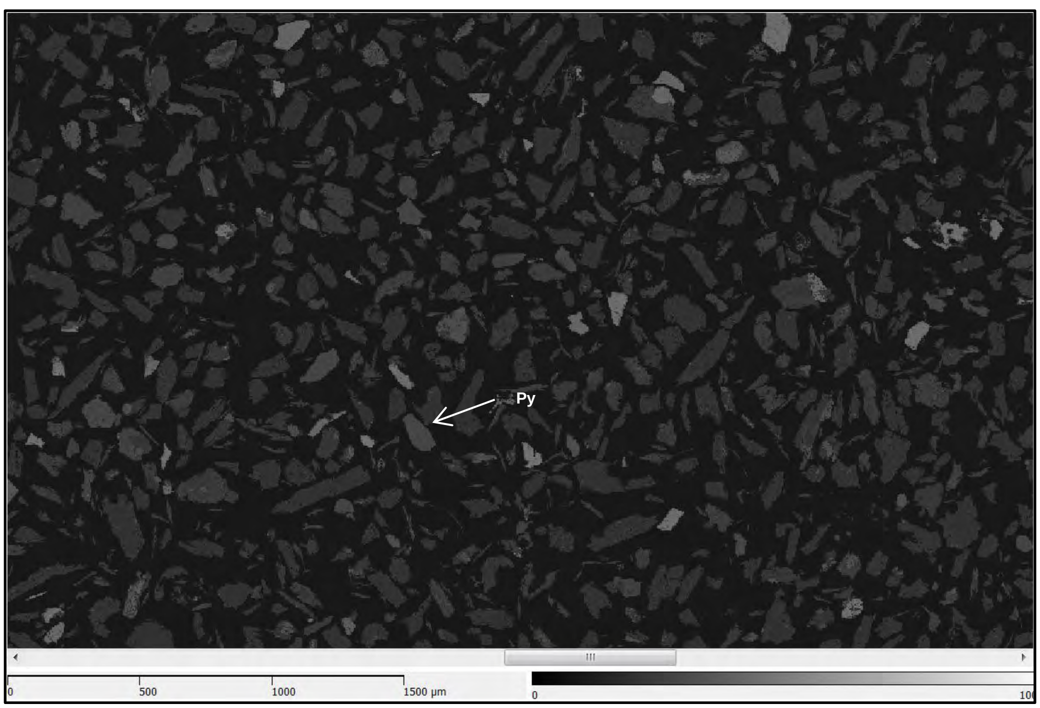


Figure 16. BSE image from sample BL295-12 Tails HLS "sink" fraction.

TIMA Images – Knelson Con (BL295-12)

Gold is yellow in the false color image (Figure 17) and difficult to see; however, it is easier to see in the BSE image (Figure 18) where it is white against the black background.

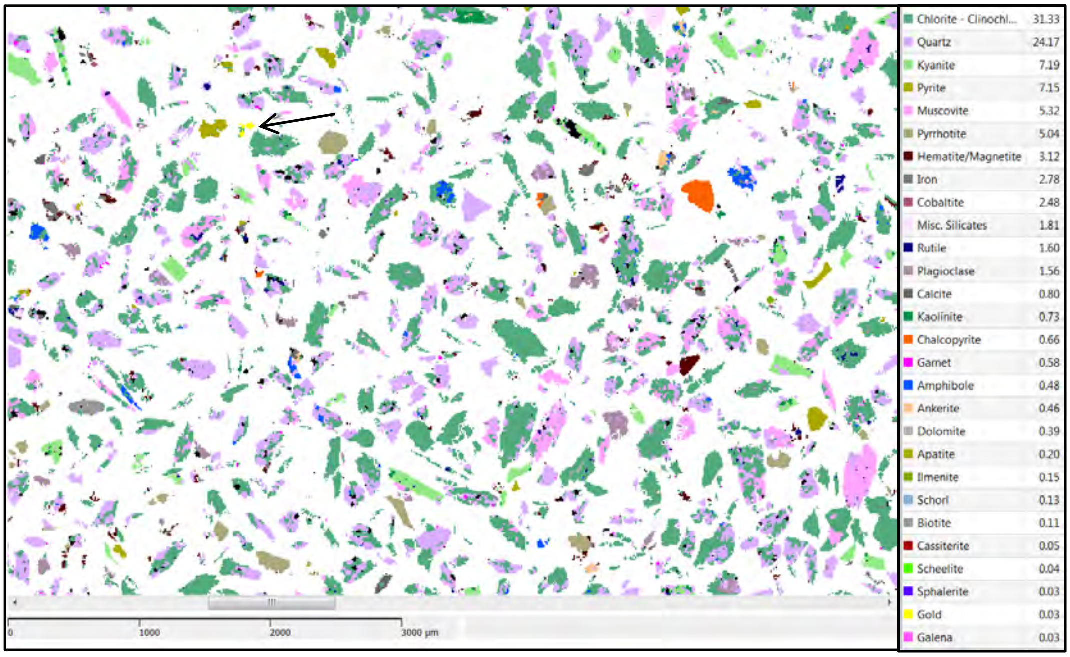


Figure 17. Classified TIMA image from BL295-12 Con. Concentration palette values are in mass percentage.

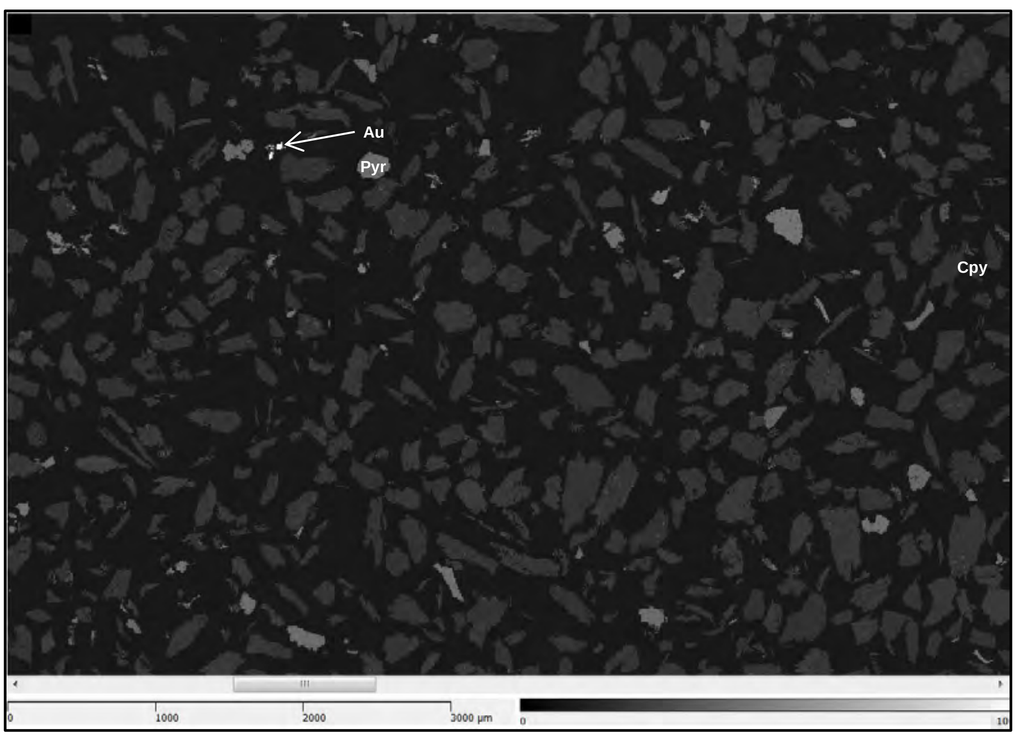


Figure 18. BSE image from sample BL295-12 Con.

Knelson Con Particle Photomicrographs.

Large elongate grains of over 100 µm in length but much less in width are shown in Figure 19A & B. The gold grain in Figure 19B is still attached to a mix of silicates including quartz, chlorite and muscovite. Two grains are shown in Figure 19C, one is "sandwiched" in chlorite and the other apparently liberated. Another gold grain is flattened out and separated from iron by a layer of chlorite with a chunk of cobaltite pressed into the outer edge, probably a remnant from grinding and preparation.

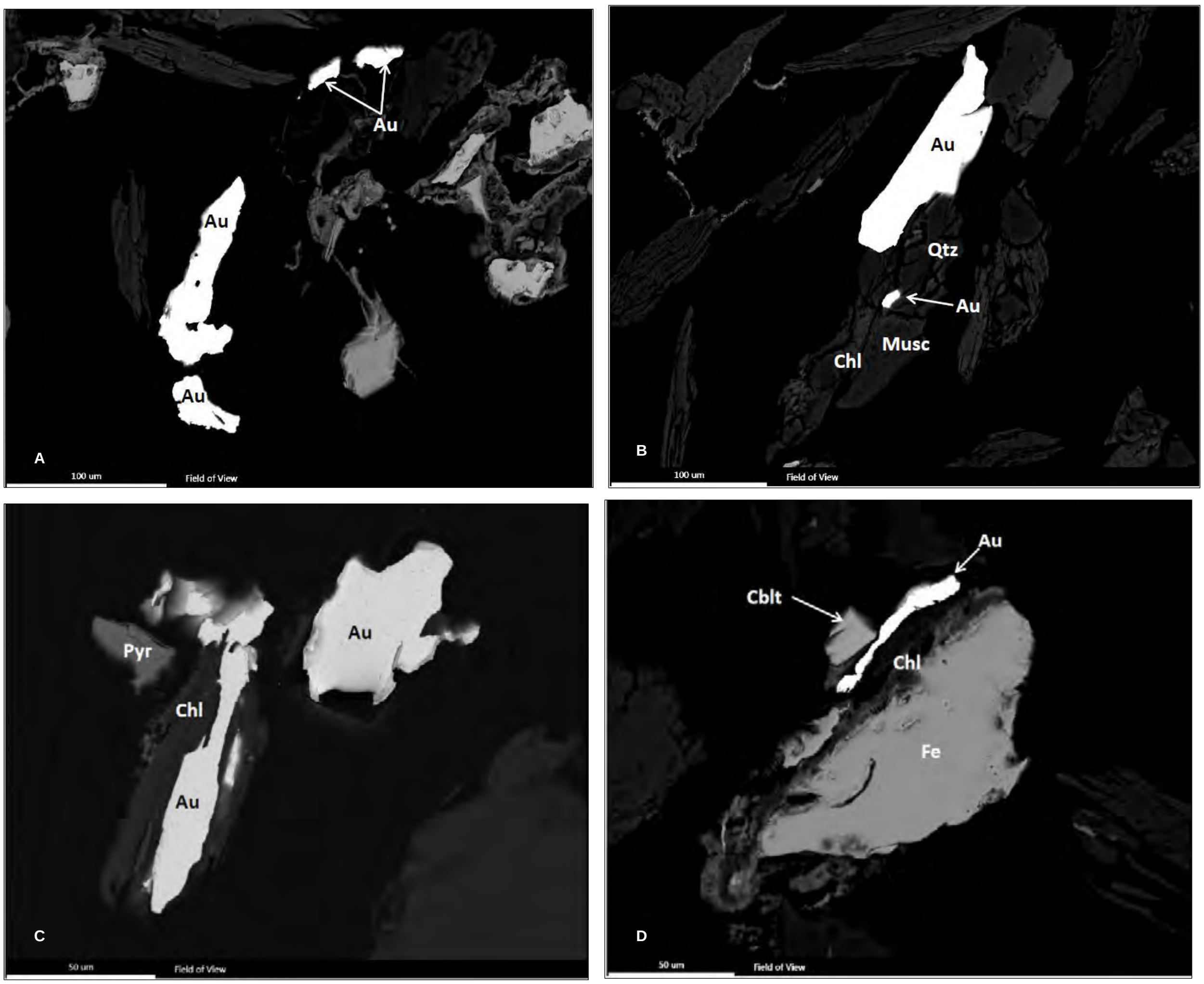


Figure 19. Gold with silicates, mainly chlorite.

Another relatively large gold grain (~50 µm lengthwise) is coated by chlorite (Figure 20A) and a grain pressed against a quartz particle in Figure 20B.

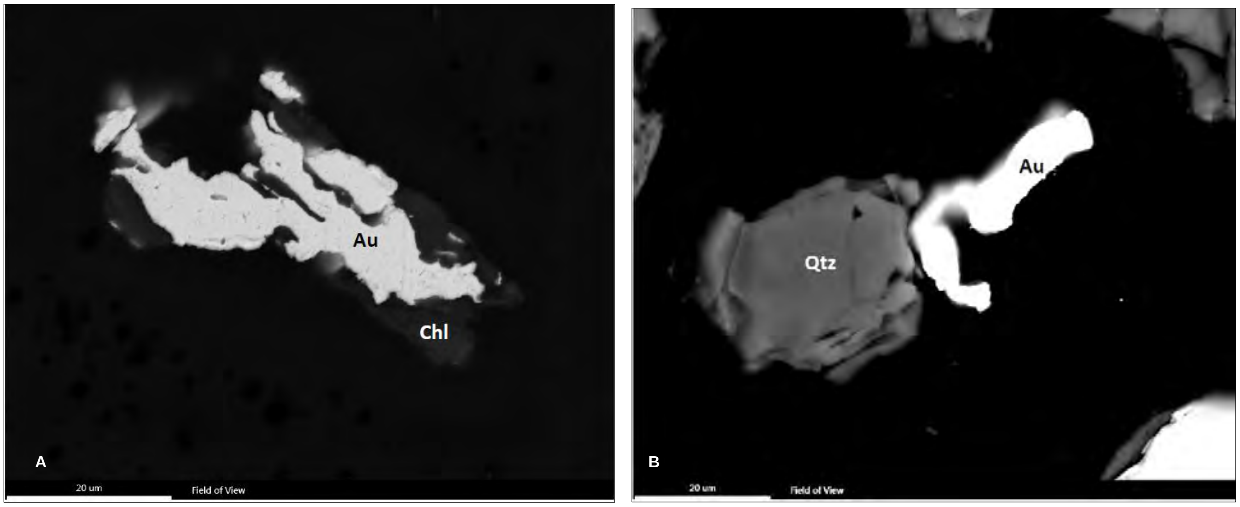


Figure 20. Gold with chlorite and attached? to quartz.

The following page (Figure 21) shows a number of examples of gold in cobaltite. Gold occurs as a small streamer or seam in A & C and small blebs of 1 to 5 µm in the other cobaltite particles. Numerous small blebs are seen in the cobaltite particle in Figure 21B, but only one is gold while the others are bismuth telluride. The 10 µm gold inclusion in the cobaltite particle in Figure 21E is associated with chlorite and rutile.

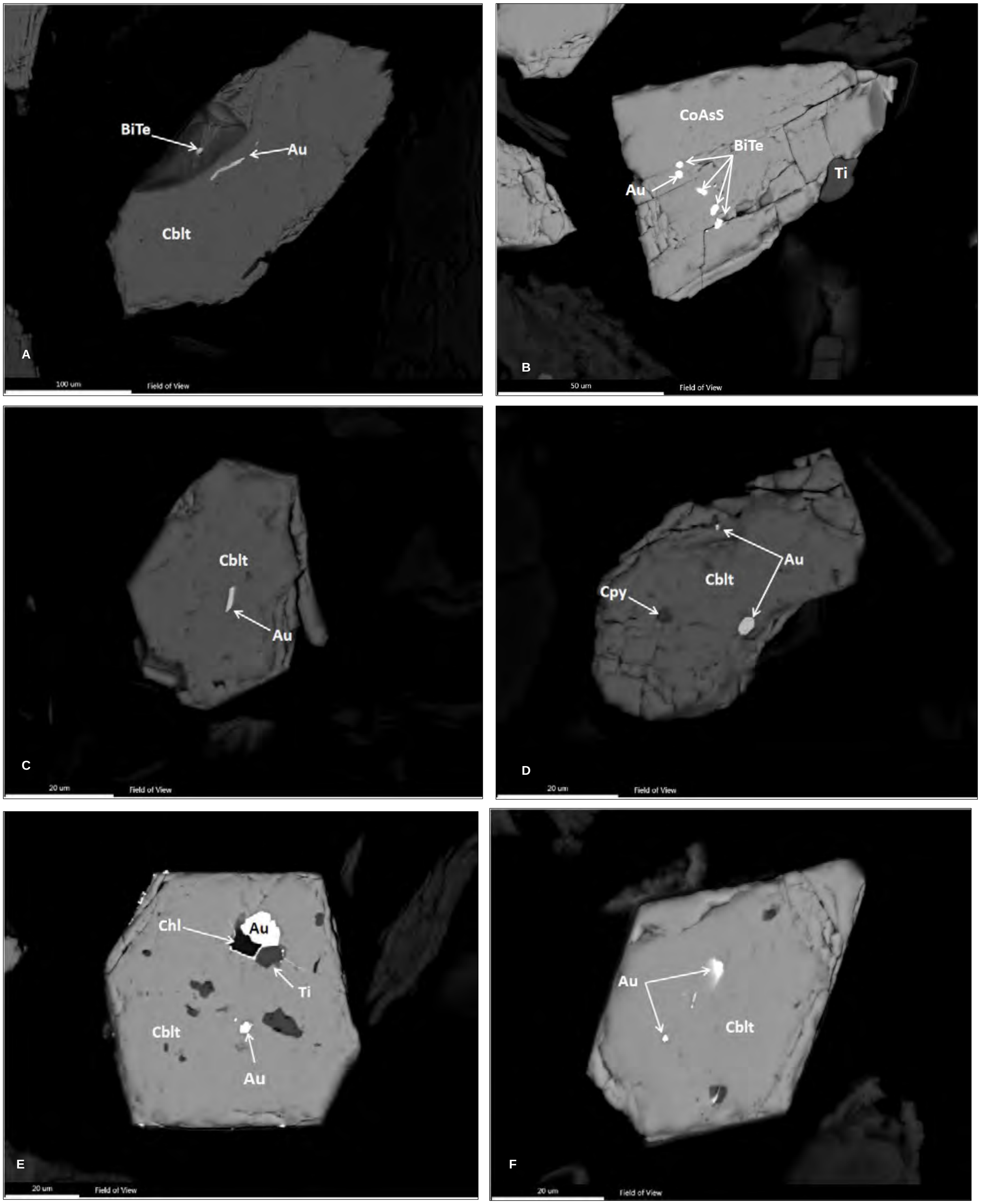


Figure 21. Gold inclusions in cobaltite from the Knelson Con (BL295-12).

Gold occurred at the cobaltite grain boundaries as shown in images in Figure 22. Smaller gold grains of nearly 10 µm are shown in Figure 22A and over 20 µm in Figure 22C & D. Tellurides of lead, bismuth and antimony are directly associated with gold in Figure 22B & D while gold and bismuth telluride are both inclusions in Figure 22A.

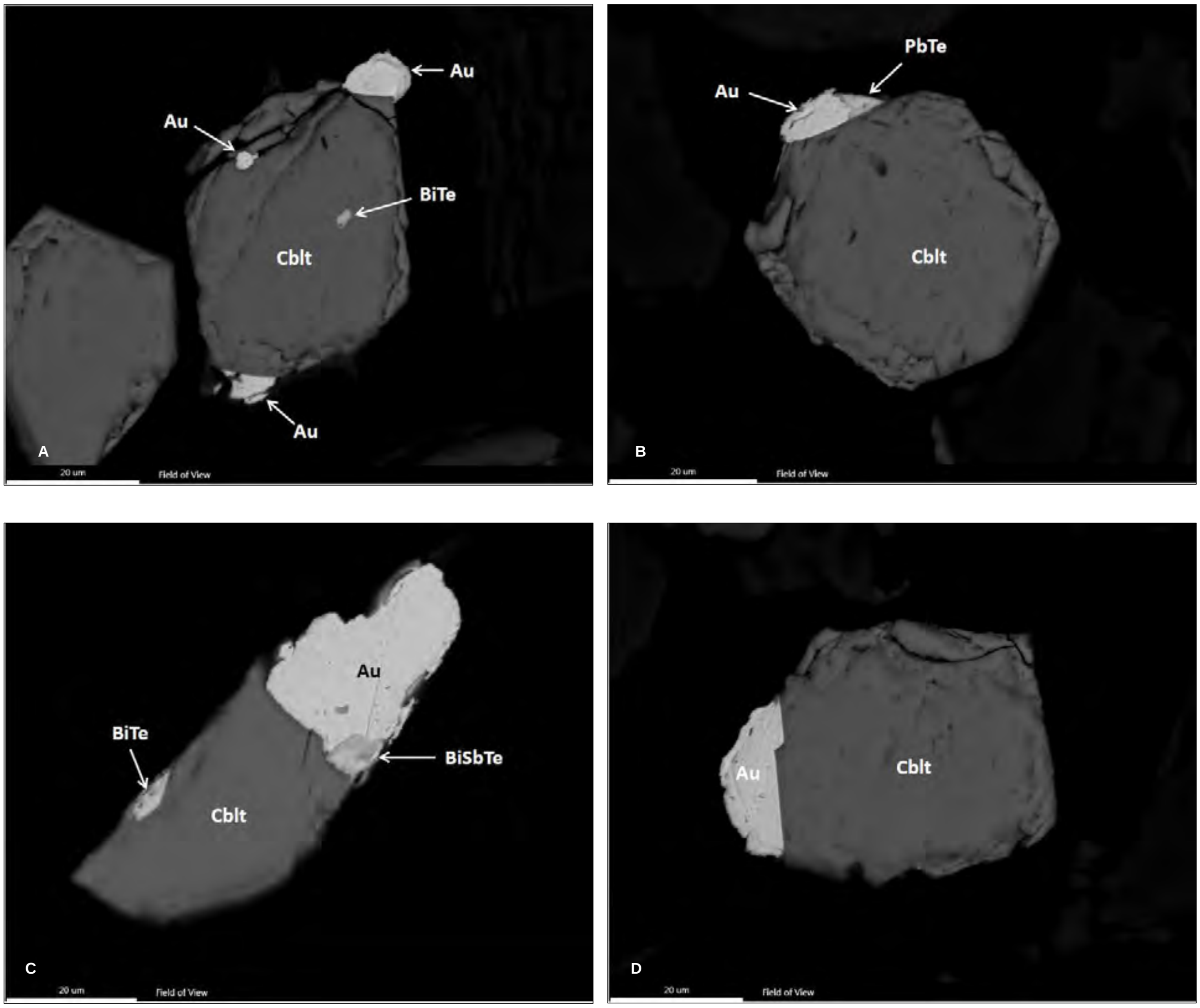


Figure 22. Gold occurring at the grain boundary of cobaltite.

Two small (1-5 µm) gold grains are located at the edge of a large hessite particle in Figure 23A. A large gold particle of about 50 µm retains a fragment of cobaltite on its perimeter in Figure 23B.

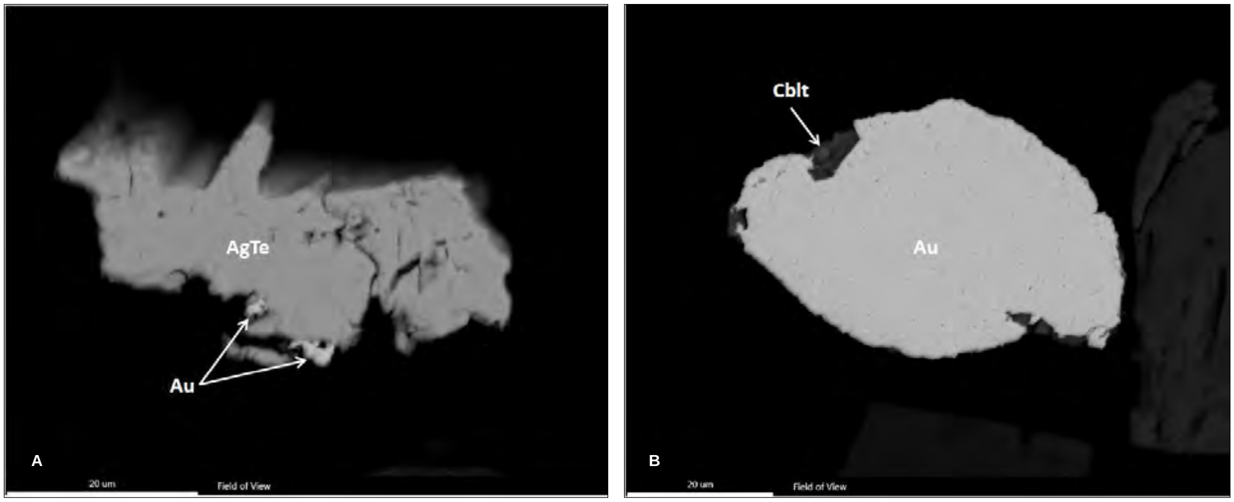


Figure 23. Gold with silver telluride (A) and remnant cobaltite on large gold particle (B).

Bismuth telluride contains a 10 µm gold inclusion in Figure 24A. A 15 to 20 µm gold particle may be attached to iron oxide in Figure 24B.

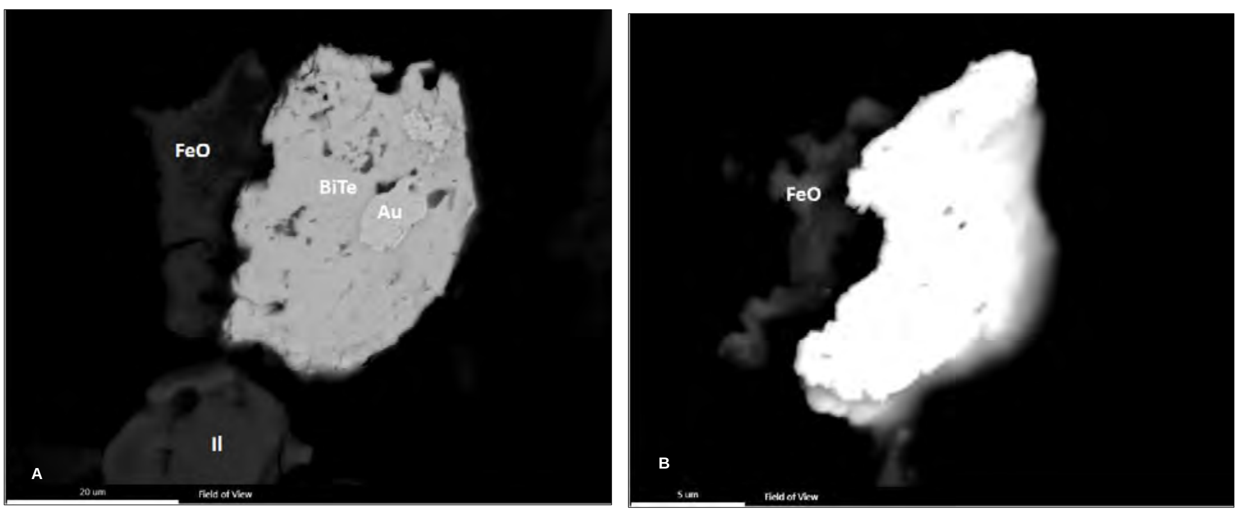


Figure 24. Gold inclusion in bismuth telluride (A) gold particle associated with iron oxide (B).

Large, liberated gold particles are shown in Figure 25. The gold particle in Figure 25A is nearly 140 µm in length and the particle in Figure 25B is approaching 100 µm.

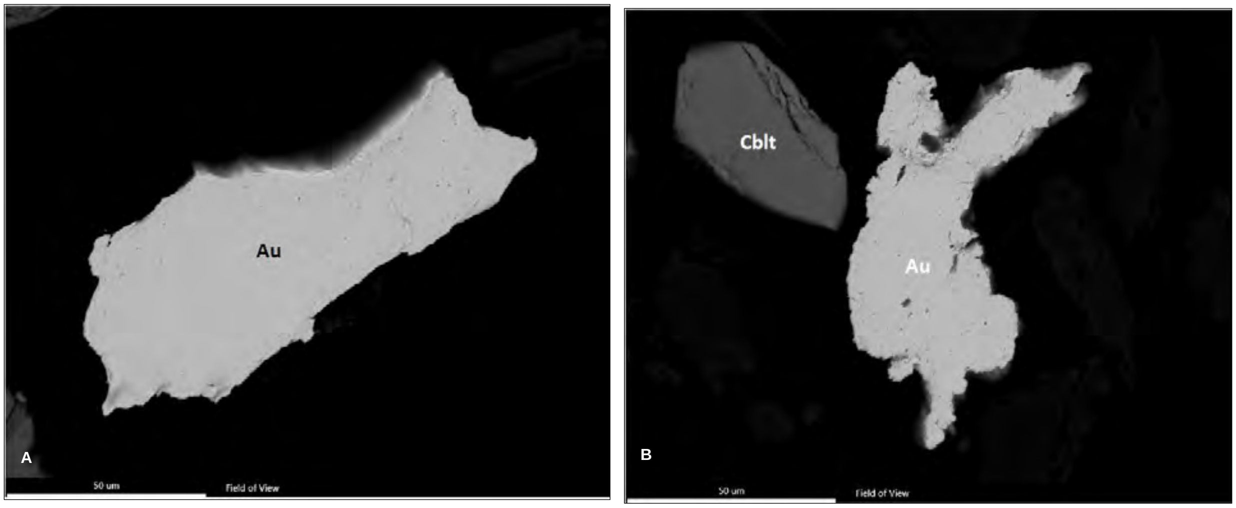


Figure 25. Large liberated gold particles.

The elongate gold particle in Figure 26A is likely liberated from between layers of chlorite. Two more liberated gold grains are shown in Figure 26B, one in the 50 µm range.

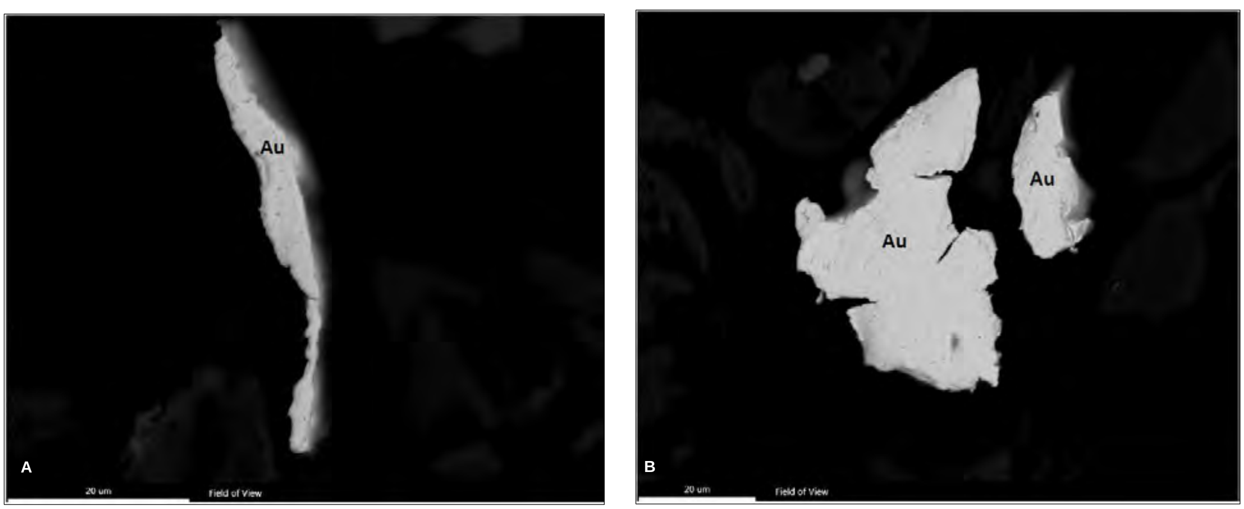


Figure 26. Elongate gold particle (A) and liberated gold particles (B).

More liberated gold grains are shown in Figure 27 that range in size from about 15 μm (Figure 27D) up to 60 μm (Figure 27A).

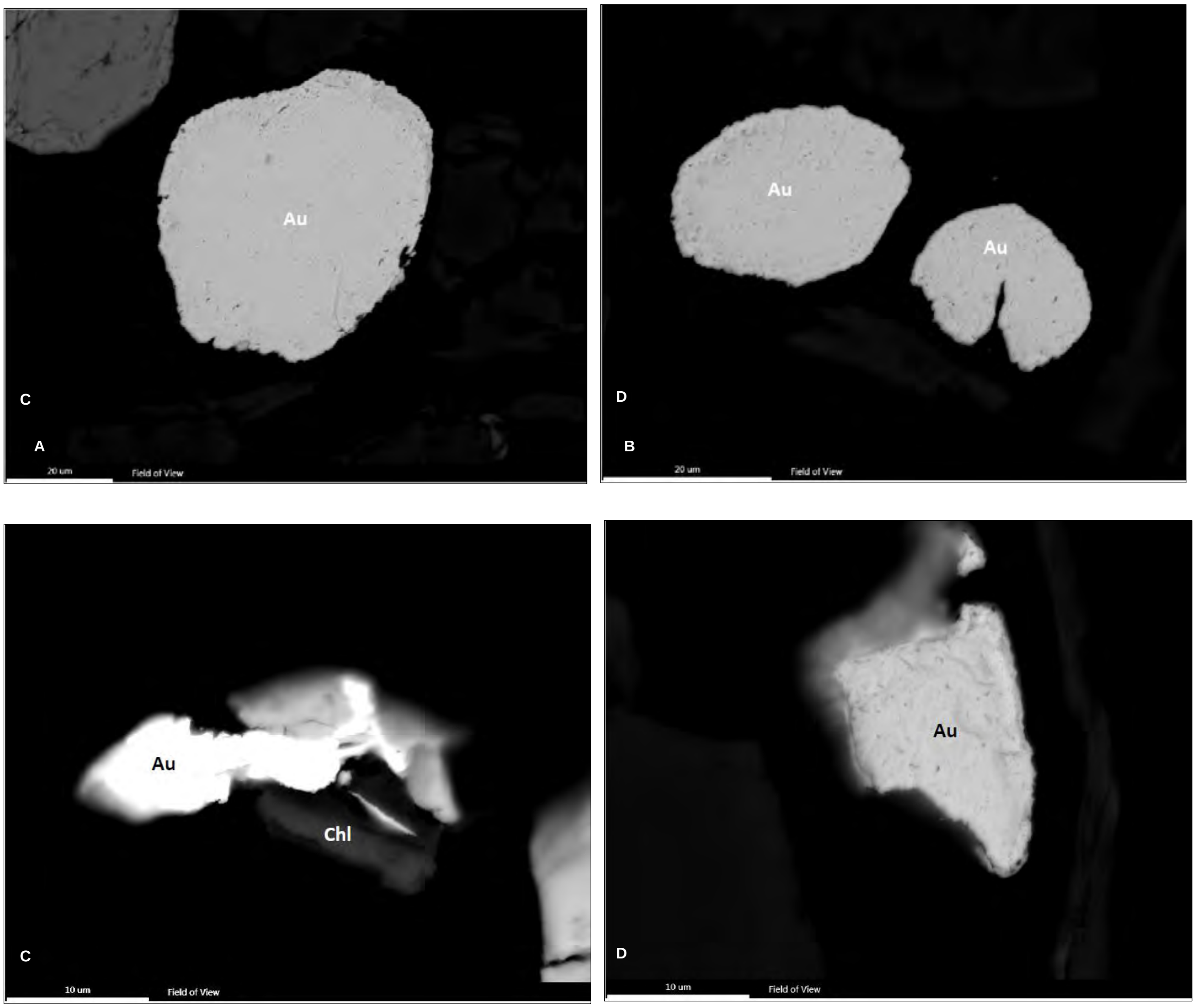


Figure 27. Liberated gold particles from Knelson Con (BL295-12).

TIMA Images – Rougher Tails (BL295-14)

The false color image for the Rougher Tails was mostly chlorite and quartz (Figure 28) and the bright particle in the BSE image (Figure 29) was barite.

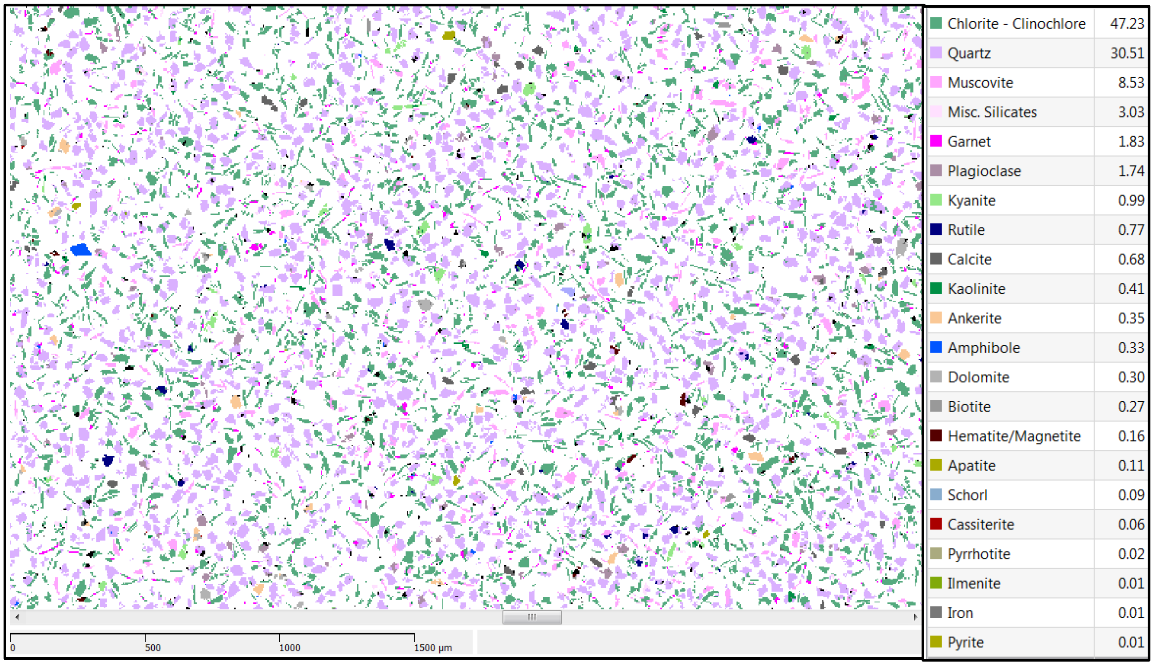


Figure 28. Classified TIMA image from BL295-14 Tails. Concentration palette values are in mass percentage.

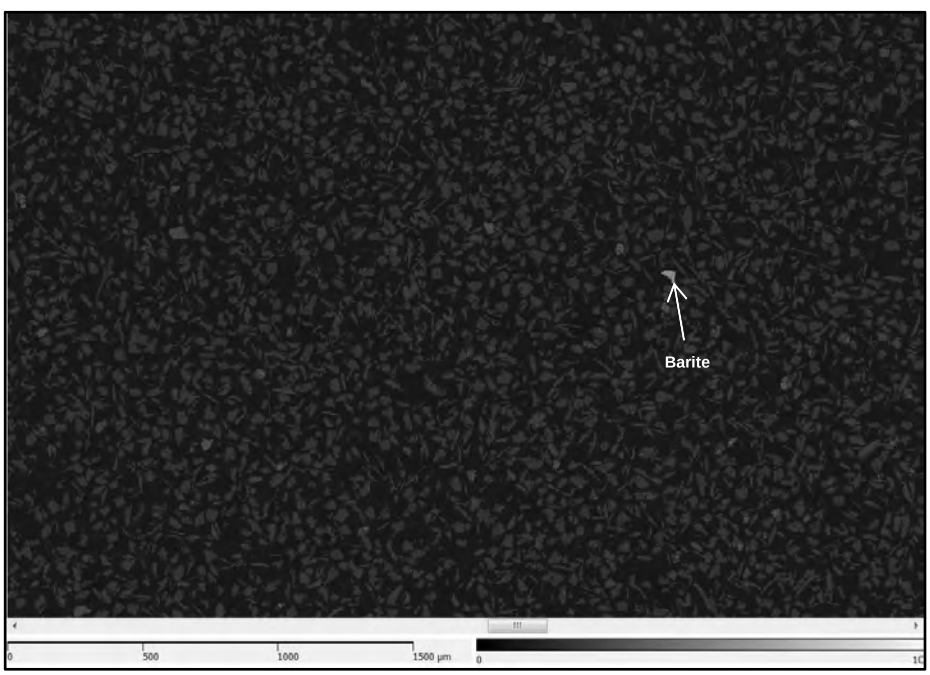


Figure 29. BSE image from sample BL295-14 Tails.

The HLS "sink" fraction from the BL295-14 Rougher tails was mostly kyanite and chlorite (Figure 30). Even though pyrrhotite and pyrite are both iron sulfides they can be distinguished by chemistry as well as grey level response in the BSE image where pyrrhotite is brighter than pyrite (Figure 31).

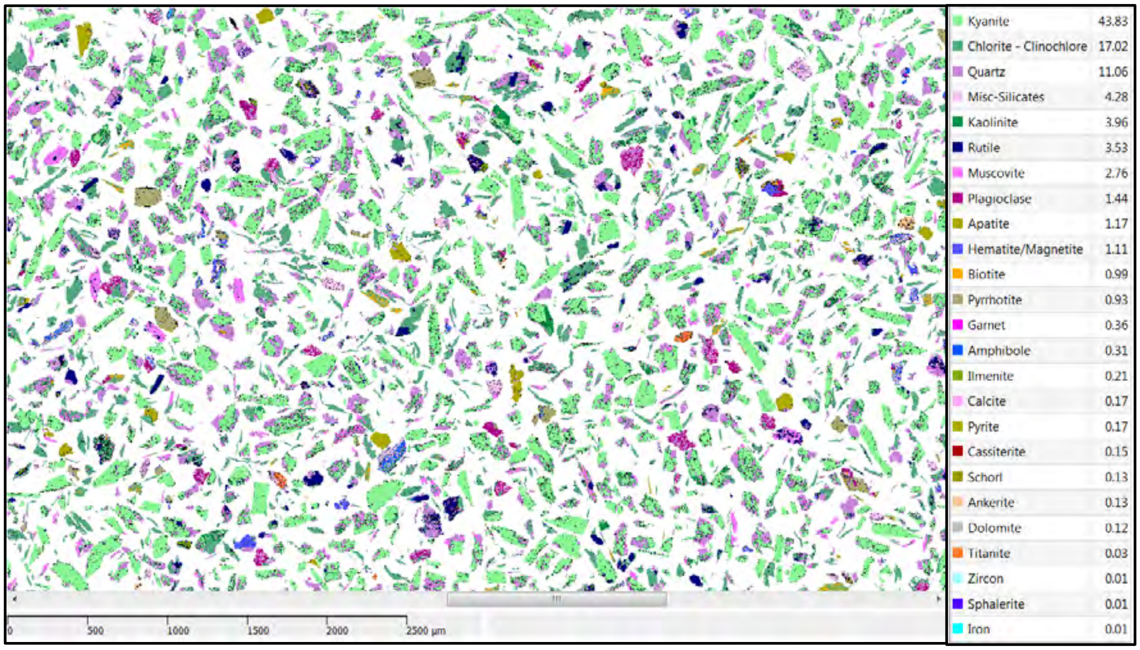


Figure 30. Classified TIMA image from BL295-14 Tails HLS "sink" fraction. Concentration palette values are in mass percentage.

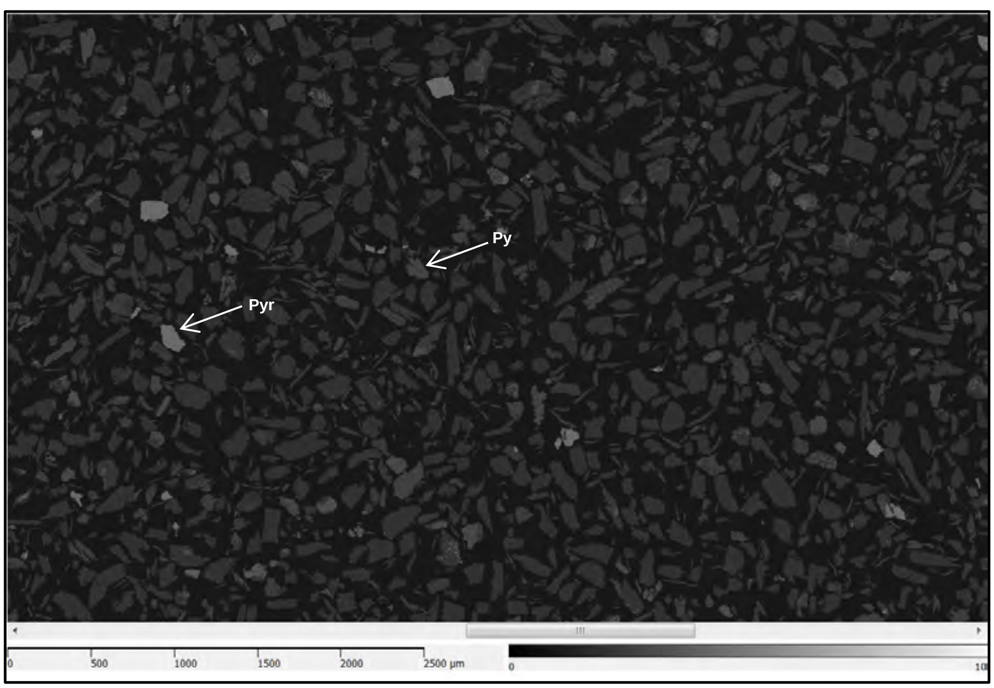


Figure 31. BSE image from sample BL295-14 Tails HLS "sink" fraction.

TIMA Images – Rougher Con (BL295-14)

The false color image of the Rougher Con in Figure 32 shows the sample is mostly chlorite and other silicates with amphibole in blue. Pyrite and pyrrhotite are the brighter particles in the BSE image in Figure 33.

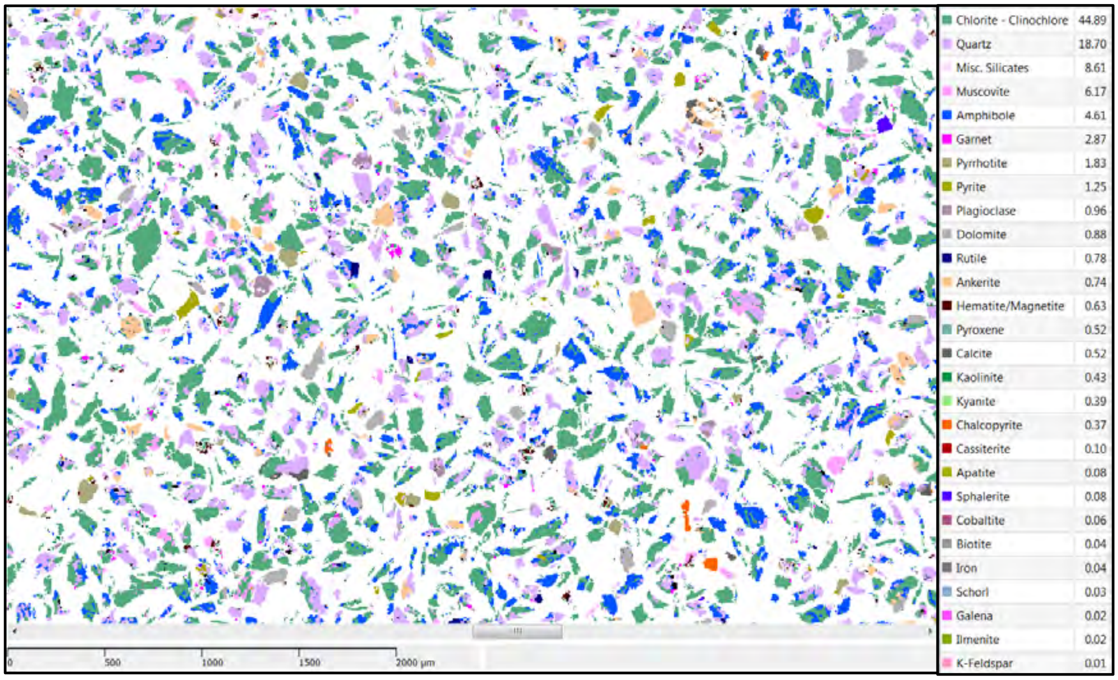


Figure 32. Classified TIMA image from BL295-14 Con. Concentration palette values are in mass percentage.

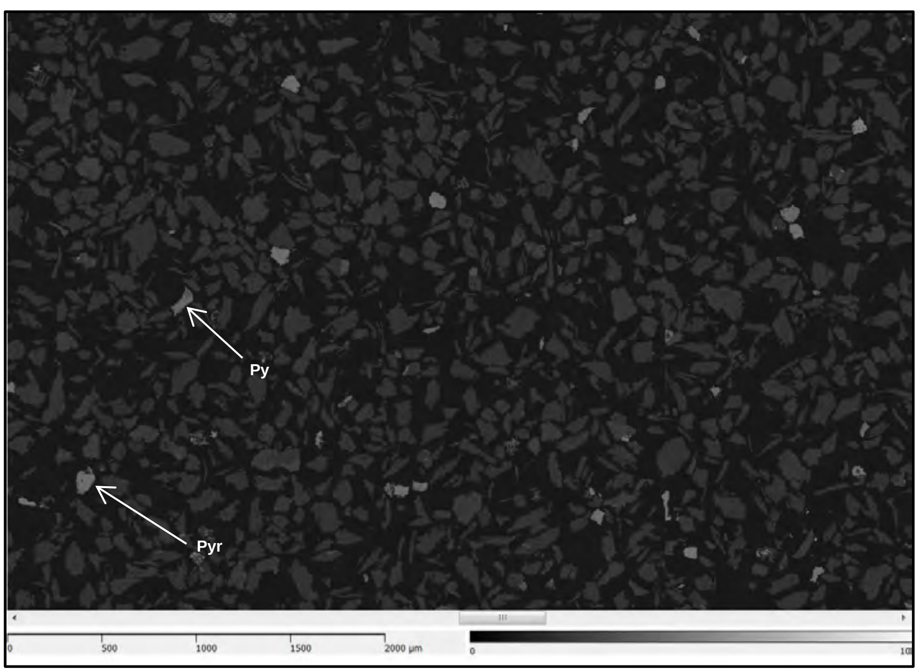


Figure 33. BSE image from sample BL295-14 Con.

The HLS "sink" material from the Rougher Con (BL295-14) was mainly pyrite and pyrrhotite as shown by olive and light gray in the false color image (Figure 34). A small gold grain was found in a complex silicate particle of kyanite, chlorite and quartz (Figure 35).

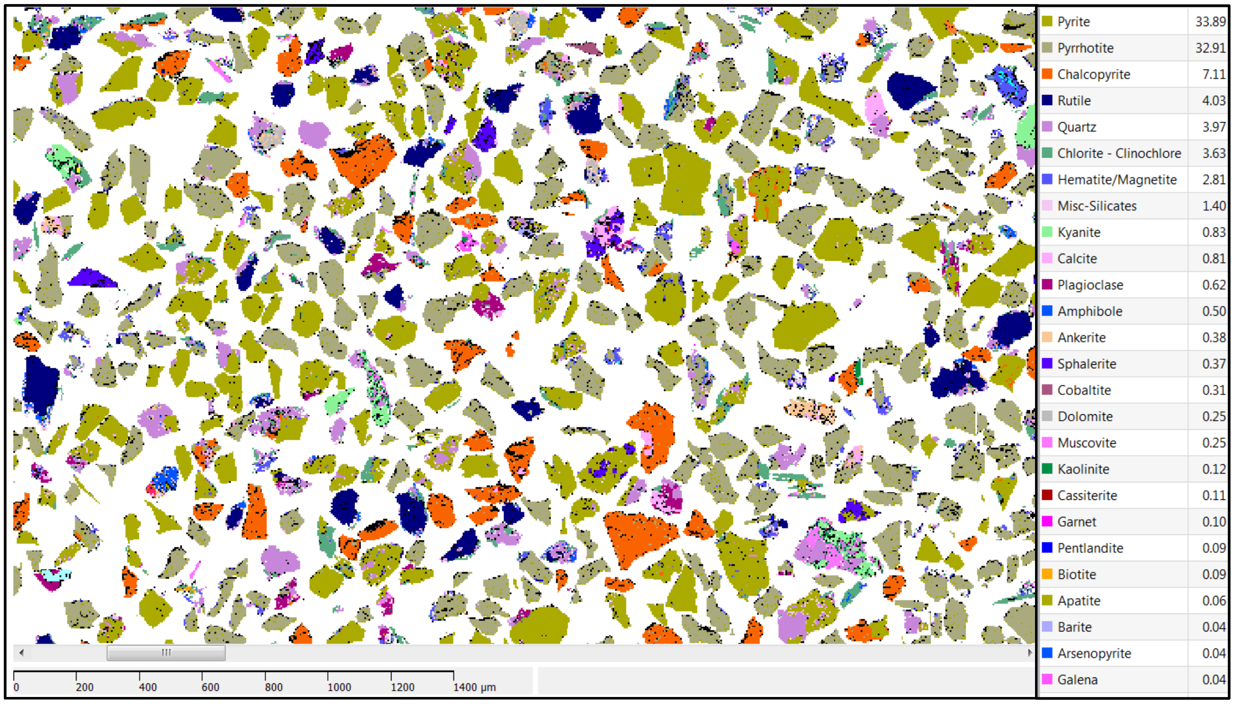


Figure 34. Classified TIMA image from BL295-14 Con HLS "sink" fraction. Concentration palette values are in mass percentage.

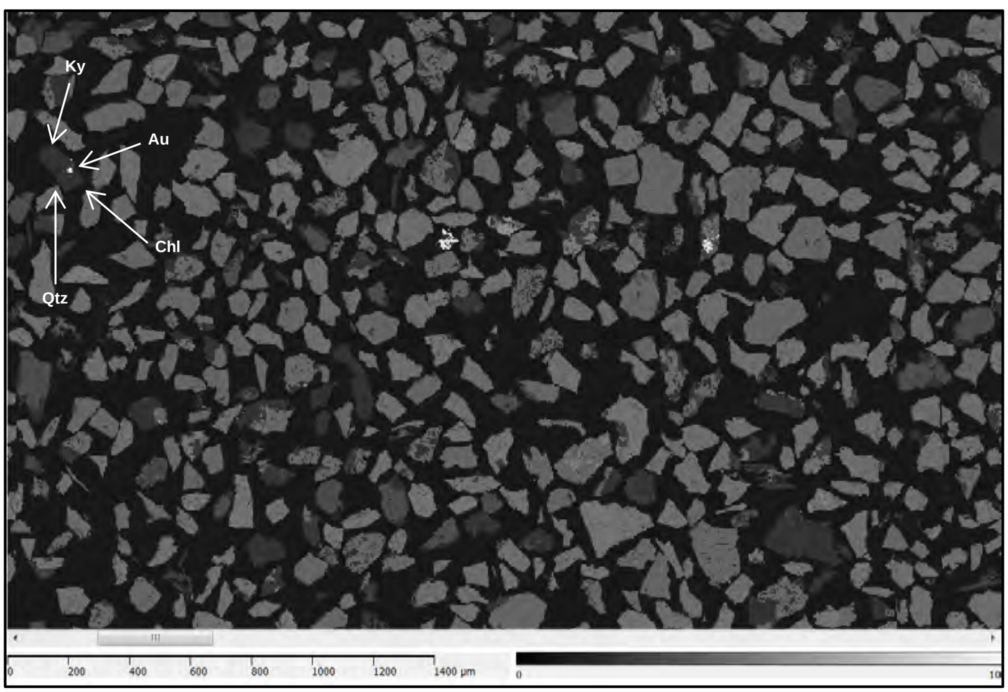


Figure 35. BSE image from sample BL295-14 Con HLS "sink" fraction.

Rougher Con (BL295-14) Particle Photomicrographs.

A small gold inclusion of 5 to 10 µm was locked in quartz in Figure 36A. The edge of a gold particle of around 20 µm is exposed adjacent to a chlorite particle in Figure 36B.

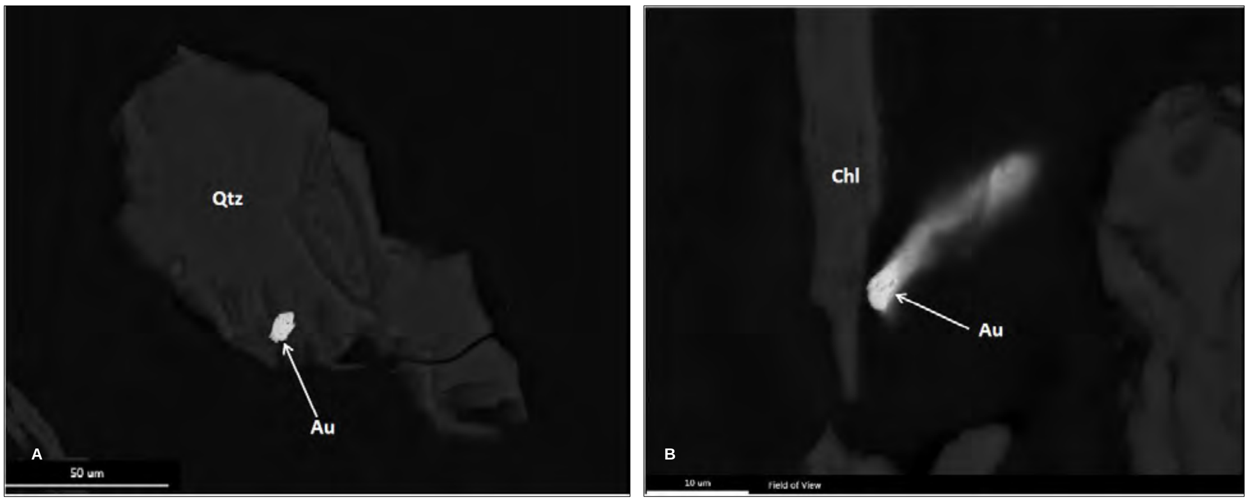


Figure 36. Gold inclusion in quartz (A) and liberated gold (B) in Rougher Con (BL295-14).

The silver telluride, hessite, is intermingled with bismuth telluride in a 20 µm inclusion in pyrrhotite (Figure 37). Hessite was found locked in chlorite and a silver-bismuth telluride inclusion had a small associated gold grain of around 5 µm (Figure 37B).

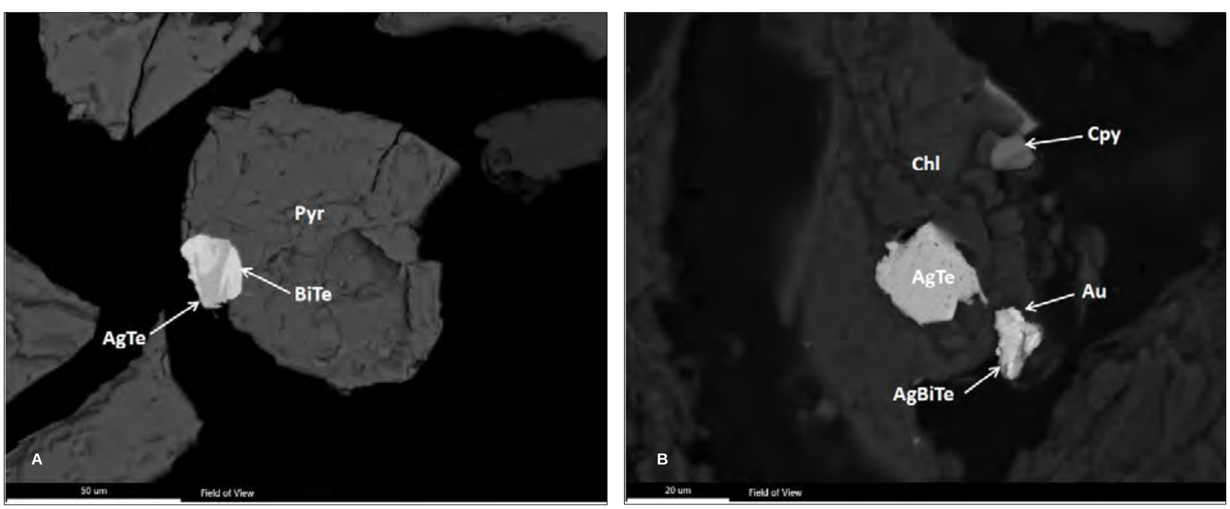


Figure 37. Silver and bismuth tellurides comingling in pyrrhotite (A) and associated with gold in chlorite (B).

Figure 38 shows a 20 to 30 µm grain of complex composition containing mixed silver-bismuth tellurides and bismuth selenide/tellurides attached to quartz.

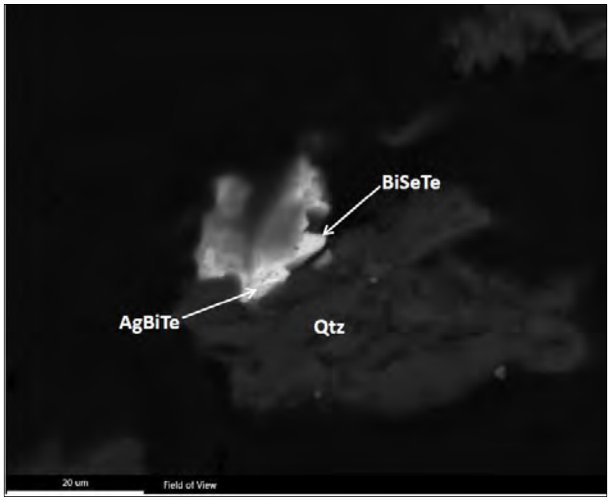


Figure 38. Silver-bismuth telluride and bismuth-selenium telluride attached to quartz.

Gold Associations

The mineral associations as determined using the TIMA analysis are presented in Table 10. The associations for Knelson Con (BL295-12) are shown since the gold occurring in this sample was poorly liberated; therefore, should yield the best mineral associations. Chlorite/Clinochlore displayed the strongest association with gold according to TIMA which is in concurs with manual observation. However, the association with cobaltite seems low, but may be the result of finer gold particles being found with cobaltite. The association with pentlandite seems somewhat erroneous since manual observation did not reveal any gold particles associated with this mineral.

Table 10. Mineral associations for Knelson Con (BL295-12).

Mineral	Chlorite - Clinochlore	Pyrrhotite	Quartz	Cobaltite	Gold	Bi-Tellurides	Pentlandite	Free surface
Gold	54.3	0.7	3.6	0.7	---	0.2	2.7	28.5
Bi-Tellurides	2.9	3.9	8.9	21.5	0.3	---	0.0	13.2

Appendix

Name	Composition	Density [g/cm³]
Garnet	O 47.6, Si 20.9, Mg 18.1, Al 13.4	3.74
Hornblende	O 34.4, Si 16.7, Fe 14.7, Ca 11.1, Mg 9.1, Al 8.7, Ti 4.7, Mn 0.54	2.9
Schorl	O 43.8, Al 20.1, Si 15.2, Fe 13.1, B 3.0, Mg 2.8, Na 1.7, Ca 0.42	3.15
K-Feldspar	O 46.0, Si 30.3, K 14.0, Al 9.7	2.56
Albite	O 48.7, Si 31.5, Al 10.8, Na 8.3, Ca 0.76	2.62
Diopside	O 44.3, Si 25.9, Ca 18.5, Mg 11.2	3.25
Kyanite	O 49.4, Al 33.3, Si 17.3	3.56
Garnet - Andradite	O 37.8, Ca 23.7, Fe 22.0, Si 16.6	3.7
Ankerite	O 46.5, Ca 19.4, Fe 16.2, C 11.6, Mg 3.5, Mn 2.7	2.97
Biotite	O 43.4, Si 19.4, Mg 14.0, K 9.0, Fe 6.4, Al 6.2, F 1.1, H 0.41	3.09
Chlorite - Clinochlore	O 48.4, Mg 15.3, Si 14.2, Fe 11.7, Al 9.1, H 1.4	2.65
Monazite	Ce 29.2, O 26.6, La 14.5, P 12.9, Nd 12.0, Th 4.8	5.15
Muscovite	O 47.4, Si 21.1, Al 20.3, K 9.8, F 0.95, H 0.46	2.77
Plagioclase	O 47.3, Si 31.1, Al 10.0, Ca 7.4, Na 4.2	2.68
Titanite	O 36.7, Ti 25.1, Ca 23.3, Si 12.3, Fe 1.6, Al 0.94, Mg 0.05, Mn 0.04	3.48
NiCoFeAsS	As 38.1, S 20.1, Ni 15.9, Co 15.6, Fe 10.4	6
Dolomite	O 52.1, Ca 21.7, Mg 13.2, C 13.0	2.84
Apatite	Ca 39.7, O 38.1, P 18.4, F 3.8	3.15
Calcite	O 48.0, Ca 40.0, C 12.0	2.71
Alunite	O 54.1, Al 19.5, S 15.5, K 9.4, H 1.5	2.59
Anorthite	O 38.5, Si 30.2, Al 18.5, Ca 12.3, K 0.53	2.74
Enstatite	O 47.8, Si 28.0, Mg 24.2	3.1
Kaolinite	O 55.8, Si 21.8, Al 20.9, H 1.6	2.6
Sphalerite	Zn 64.1, S 33.1, Fe 2.9	4.05
Misc-Silicates	O 51.7, Si 32.2, Al 11.0, K 2.9, Fe 1.4, Na 0.83	2.7
Al-Silicates-mixed	O 48.4, Mg 15.3, Si 14.2, Fe 11.7, Al 9.1, H 1.4	2.55
Galena	Pb 86.6, S 13.4	7.4
Scheelite	W 63.9, O 22.2, Ca 13.9	5.9
Amphibole	O 49.2, Si 28.8, Mg 21.8, H 0.26	2.85
Barite	Ba 58.8, O 27.4, S 13.7	4.48
Bornite	Cu 63.3, S 25.6, Fe 11.1	5.1
Chalcopyrite	S 34.9, Cu 34.6, Fe 30.4	4.2
Ilmenite	Fe 36.8, O 31.6, Ti 31.6	4.72
Zircon	Zr 43.1, O 33.6, Si 14.8, Hf 4.7, REE 3.8	4.65
Ullmannite	Sb 60.0, Ni 25.3, S 14.7	6.65
Pentlandite	Fe 35.5, Ni 33.8, S 30.7	4.6
BiPbTe	Bi 43.7, Te 29.5, Pb 16.6, Co 4.9, Fe 2.5, Ni 2.1, S 0.74	7

Cobaltite	As 38.8, Co 26.8, S 19.7, Ni 7.9, Fe 6.8	6.33
Altaite	Pb 69.5, Te 30.5	8.1
Calaverite	Te 52.7, Au 47.3	9.04
Anthophyllite	O 49.2, Si 28.8, Mg 21.8, H 0.26	2.85
Gersdorffite	As 45.9, Ni 22.6, S 16.1, Fe 10.8, Co 4.7	5.9
Cassiterite	Sn 78.8, O 21.2	6.8
Hessite	Ag 66.5, Te 33.5	7.2
Tsmuoite	Bi 65.2, Te 29.0, Fe 4.6, Se 1.2	8.16
Chromferide	Fe 89.0, Cr 11.0	0.1
Melonite	Te 81.5, Ni 18.5	7.3
Iron	Fe 100.0	7.3
Fluorite	Ca 51.3, F 48.7	3.13
Hematite/Magnetite	Fe 70.0, O 30.0	5.13
Pyrite	S 53.4, Fe 46.5	5.01
Pyrrhotite	Fe 62.3, S 37.7	4.58
Quartz	O 53.3, Si 46.7	2.62
Rutile	Ti 59.9, O 40.1	4.25
Gold	Au 95.0, Ag 5.0	16