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PRELIMINARY REPORT

SAMPLING AND METALLURGICAL REPORT

BEAUROX MINES LIMITED

ROXMARK MINES LIMITED

GERALDTON-BEARDMORE AREA ONTARIO

TASHOTA-NIPIGON DEPOSIT

TOMBILL-BANKFIELD DEPOSIT

LITTLE LONG LAC DEPOSIT

By

DAVID MALOUF
General Manager

February 14, 1991

PRELIMINARY SUMMARY REPORT

1. As outlined in the President's letter of the 1989 Roxmark Mines Limited. Annual Report, Roxmark has secured the right to evaluate several gold tailings deposits in the Geraldton Beardmore Camp. Of the 3,000,000 Tons under agreement, preliminary testing has shown that a thorough evaluation is warranted and that the deposits could contain up to 135,000 ounces of gold for an average grade of plus or minus .045 ounces per ton. Assuming a 65% recovery and reprocessing costs in the order of $8.00 per ton on a scale of plus 1000 tons per day this project could generate the much needed exploration and development funds needed to develop the recent discoveries in the camp specifically Roxmark's Benedict Zone, the Hardrock Discovery Zone , etc. - It is also believed that capital costs required would be a fraction of the cost required to implement a conventional mine, mill scenario possible 25-30% - and that this equipment could later be supplemented to handle mine-run ore.

2. There are three properties of prime interest:

   A). Bankfield - Tombill with a common tailings pond
   B). Little Long Lac property of Algoma Steel.
   C). Tashota Nipigon.

3. All properties are in the Beardmore Geraldton Mining Division in the District of Thunder Bay.

   A) Bankfield-Tombill
   Located on the North side of Trans Canada Highway #11-8.25Km west of the turn off to Geraldton in the Western half of Errington Twp.

   B) Little Long Lac
   Located on either side of Hwy 584 approximately 3 Km. north of Hwy 11 south of the bridge that enters the town of Geraldton in Errington and Ashmore Twps.

   C) Tashota
   The Mine is located North west of Onamen Lake and south of Obashkegan Township, between Onamen Lake and Onamen River. It is accessible via the Camp 40 Road north to the Con Lake Road. Preceed north-east from the Con Lake intersection on the Mine Road for approximately 8 miles to the mine site.

4\. A) Bankfield-Tombill Claims
3 Patented Mining Claims
TB 10213
TB 110201
TB 10545

B) Little Long Lac
8 Patented Mining Claims
TB 10887
TB 10621
TB 10560
TB 10561
TB 10562
TB 10563
TB 10886
TB 10566

C) Tashota Nipigon
2 Patented Mining Claims
MK573 & KK524

5\. A) The Bankfield Tombill project is subject to a 25% NPI in favor of Bankfield and Tombill Mines.

B) The Little Long Lac project is subject to a 5% N.S.R. to Lac Minerals and annual payments of approximately $40,000 to Algoma subject to a positive production decision.

C) The Tashota Nipigon project is subject to a 4% NSR and an additional payment of $5,000.00

6\. Regional Geology

S.E. Malouf Consulting Geologists Limited entered into agreements on the above properties for Roxmark Mines Limited. Roxmark did the initial work involving research sampling and preliminary metallurgical work involving $15,000 in 1989 and early 1990 - Roxmark has agreed to give their subsidiary company Beauron Mines Limited (at present a private corporation) a chance to earn a 25% interest in the tailings project for doing a proper evaluation, metallurgical testing, feasibility study, and a further 25% interest for funding through to production.

7\. Current Status

A). Bankfield Tombill - Initial sampling with a Sonic Soil Sample @ 60 holes drilled indicated appreciable tonnage @ .051 Oz. per Ton. with isolated tonnage of high grade.

B) Little Long Lac - Investigation of production history indicates excessives losses in the mill with two periods of tailings retreatment. Initial sampling favorable - 1,780,000 Ton potential.

C). Tashota Nipigon - Report on sampling and metallurgy from Lakefield Research done by Lynx - Canada in 1978 indicates reserves of 50,000 tons of .088 Oz Gold per ton with indicated recoveries of 70.4%

8. Recommended Work Project

A program involving the expenditure of $200,000.00 is warranted. Grids will be established on all properties. The Tashota will be drilled and sampled on 25 foot centre because of the relatively small size of the deposit and high grade nature. The Bankfield Tombill will be drilled and sampled on 50 foot centers - Little Long Lac will be drilled and sampled on 100 foot centers with later definition at 50 feet. This should involve 10,000 to 11,000 feet of drilling and approximately 2400 assays - Sample results and locations will be plotted and grade contour lines established to locate economic reserves. The grid, drilling and sampling will cost $100,000.00 - Sample composites of reject samples will assembled and sent to Lakefield Research for metallurgical studies on the three representative bulk sample.

The metallurgical work should cost $40,000 - If this stage gives favourable results, it will be followed with a $50,000 environmental study and then a feasibility study.

9. The project began in August of 1990

10. The project will take nine months to complete or 200 days.

11. Work Completed

The proposed program was carried out at a cost of $165,911.54 - Grids were establish on each property and the drilling was done with a Sonic Soil Sampling machine - "BQ" Rods were used to drill down through the tailings and into organic material. Samples were taken at each five foot section.

A total of 11,000 feet were drilled and 2,621 samples taken. The assay results were plotted on assay plans. These results were then contoured to show areas averaging .03 ounces of gold per ton and better. Once these areas were known the sample rejects representing these areas were made into composites for each deposit and prepared for bulk metallurgical work at Lakefield. Supplementary bulk samples were taken with the use of a back hoe which cut five trenches on each of the Bankfield and the Little Long Lac deposits. The trenches were 50 feet long, the depth of the tailings and a two ton sample representative sample compiled from each deposit. There was sufficient material for testing on the Tashota property as each hole was double drilled.

All composites properly identified were shipped to Lakefield research and arrived December 27th, 1990.

Metallurgical work began in early January with preliminary investigations on gravity, flotation, bottle cyanide tests, 30 elements scans etc. After a review of initial results a decision was made to do heap leach column tests on all three ores and to do a combination of gravity (Falcon concentrator) and column flotation in a continues circuit on the Bankfield and Little Long Lac material. These tests have been paid for but are still in progress.

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When the results have been compiled a complete report will be presented to the Ministry. This will be accomplished by March 25 or earlier. We have included a copy of the preliminary metallurgical work done and underway, as well as assay plans of the respective properties.

11. Completion date is February 15, 1991.

12. <u>Reports Available</u>:

A) "Ontario Geological Survey"
Open file Report 5630 - 1986
Volume I - Pg. 87 thru 96
Pg. 329 Thru 338

Volume II - Pg. 582 thru 584
Pg. 571 item 8 Economic Features

B) "An Investigation of the Recovery of Gold"
From Tailings sample submitted by Tashota-Nipigon Mines Ltd.
Progress Report No. 1

Project No. L.R. 2190 - Lakefield Research.

13. Preliminary Metallurgical Reports.

14. Assay Maps with preliminary reserves.

Respectfully,

David Malouf
General Manager and Director
Roxmark Mines Limited
Beaurox Mines Limited

LAKEFIELD RESEARCH
A DIVISION OF FALCONBRIDGE LIMITED

FACSIMILE NO. (705) 652-6365

February 6, 1991

Mr. D. Malouf
Beaurox Mines Limited
Suite 801
80 Richmond Street West
Toronto, Ontario
M5H 2A4

Dear Mr.Malouf:

Re: LR4095 - Summary Letter No. 1
The Recovery of Gold from Low Grade Tailing Samples

Please find enclosed a summary of all testwork completed to date on the Bankfield, Bankfield-2, Tashota and Little Long Lac samples.

The testwork performed thus far includes head assay analyses, gravity concentration, flotation and direct cyanidation.

(I) Head Assay Analyses

Representative head samples were cut from each individual tailing sample and assayed for Au, Fe and S(total). The results are summarized as follows:

Table No. 1: Head Assays

	Bankfield	Bankfield-2	Tashota	Little Long Lac
Au, g/t	2.90	1.29	3.03	1.60
Fe, %	6.33	6.40	10.1	3.80
S(total), %	2.28	2.26	2.51	0.19

Size fraction analyses for Au and S(total) were also performed on the same head samples. Table 2 summarizes these results.

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Table 2: Size Fraction Analyses

(i) Bankfield

Mesh	Weight		Assays, g/t or %		Distribution, %	
Tyler	g	%	Au	S	Au	S
65	131.5	12.3	1.75	1.53	5.7	8.3
200	306.5	28.7	2.69	0.83	20.4	10.5
400	211.1	19.8	3.23	3.04	16.9	26.4
-400	417.3	39.1	5.52	3.19	57.0	54.8
Feed(calc)	1066.4	100.0	3.79	2.26	100.0	100.0

(ii) Tashota

Mesh	Weight		Assays, g/t or %		Distribution, %	
Tyler	g	%	Au	S	Au	S
65	54.4	5.0	6.04	2.84	10.6	6.3
200	388.5	35.4	4.52	1.65	56.6	26.1
400	248.2	22.6	1.65	2.45	13.2	24.8
-400	405.2	37.0	1.50	2.59	19.6	42.8
Feed(calc)	1096.3	100.0	2.83	2.24	100.0	100.0

(iii) Little Long Lac

Mesh	Weight		Assays, g/t or %		Distribution, %	
Tyler	g	%	Au	S	Au	S
65	27.6	2.6	3.45	0.19	6.5	3.1
200	374.4	35.8	1.13	0.07	28.8	15.4
400	245.0	23.4	1.26	0.21	21.0	30.2
-400	397.8	38.1	1.62	0.22	43.8	51.4
Feed(calc)	1044.8	100.0	1.41	0.16	100.0	100.0

(iv) Bankfield-2

Mesh	Weight		Assays, g/t or %		Distribution, %	
Tyler	g	%	Au	S	Au	S
65	3.9	0.7	1.73	1.54	1.0	0.5
200	130.4	24.4	1.02	0.39	19.3	4.5
400	142.2	26.6	1.46	2.19	30.1	27.8
-400	257.7	48.2	1.33	2.92	49.7	67.1
Feed(calc)	534.2	100.0	1.29	2.10	100.0	100.0

A 24 element ICP scan was also conducted and these results are summarized in Table 3.

Table 3: Size Fraction Analyses

	Bankfield	Tashota	Little Long Lac	Bankfield-2
As, %	1.39	0.004	0.12	0.85
Ba, %	0.04	0.02	0.05	0.04
Be, %	<0.0001	<0.0001	0.0001	0.0001
Ca, %	3.25	2.73	2.84	2.97
Cd, %	<0.003	<0.001	<0.0005	0.0007
Co, %	0.003	0.007	0.003	0.003
Cr, %	0.01	0.009	0.004	0.01
Cu, %	0.008	0.03	0.008	0.008
Fe, %	6.39	9.57	4.04	3.27
Mg, %	1.34	2.12	1.57	1.46
Mn, %	0.05	0.08	0.06	0.06
Mo, %	<0.01	<0.008	<0.01	<0.01
Na, %	3.24	1.21	1.71	3.34
Ni, %	0.006	0.006	0.006	0.006
P, %	0.04	0.03	0.05	0.04
Pb, %	<0.02	<0.01	<0.02	<0.01
S, %	2.28	2.15	0.22	2.01
Sb, %	0.001	<0.001	0.008	0.001
Se, %	<0.005	<0.005	<0.005	<0.0005
Sn, %	<0.002	<0.002	<0.002	<0.002
Te, %	<0.001	<0.001	<0.001	<0.001
Th, %	0.001	0.003	<0.001	<0.001
U, %	<0.001	<0.001	<0.001	<0.001
Zn, %	0.005	0.01	0.009	0.007

(ii) Gravity Concentration

A series of tests were conducted to investigate the recovery of gold by gravity concentration. 2 kg charges were passed over an 1/8 Wilfley shaking table with the table concentrate being upgraded on a Mozley separator. The results are summarized in Table 4.

Table No. 4: Summary of Gravity Concentration Testwork

Test #	Sample	Product	Weight %	Assay, g/t, % Au	S	Distribution, % Au	S
4	Tashota	Mozley Conc	0.1	141		2.9	
		Mozley Tail	2.7	5.44		4.6	
		+ 28 mesh Table Conc	1.1	1.61		0.5	
		Table Conc(calc)	3.9	6.70	20.5	8.0	31.4
		Table Tail	96.2	3.06	1.79	92.0	68.6
		Feed(calc)	100.0	3.22	2.51	100.0	100.0
5	Bankfield	Mozley Conc	0.1	297		8.1	
		Mozley Tail	3.8	7.67		10.3	
		+ 28 mesh Table Conc	1.7	1.00		0.6	
		Table Conc(calc)	5.5	9.66	10.7	19.0	25.8
		Table Tail	94.5	2.40	1.79	81.0	74.2
		Feed(calc)	100.0	2.80	2.28	100.0	100.0
6	Little Long Lac	Mozley Conc	0.1	116		5.5	
		Mozley Tail	5.2	3.45		14.8	
		+ 28 mesh Table Conc	0.1	1.12		0.1	
		Table Conc(calc)	5.4	4.59	1.24	20.4	35.3
		Table Tail	94.6	1.03	0.13	79.6	64.7
		Feed(calc)	100.0	1.22	0.19	100.0	100.0

This type of gravity concentration was not successful since only an average of 16% of the gold and 31% of the sulphur was recovered by the Wilfley table.

Further gravity concentration tests were performed using a Falcon concentrator. Table 5 summarizes these results.

Table 5: Summary of Falcon Concentrator Results

Test #	Sample	Product	Weight %	Assay, g/t, % Au	S	Distribution, % Au	S
7	Tashota	+ 28 mesh	3.9	1.53	4.34	1.9	8.2
		Falcon Conc	7.9	9.69	2.55	23.5	9.6
		Falcon Tail	88.2	2.75	1.98	74.7	82.3
		Feed(calc)	100.0	3.25	2.10	100.0	100.0
8	Little Long Lac	+ 28 mesh	1.0	4.57	0.38	3.3	2.2
		Falcon Conc	5.0	6.57	0.59	23.6	16.9
		Falcon Tail	94.0	1.08	0.15	73.1	80.9
		Feed(calc)	100.0	1.39	0.17	100.0	100.0

As seen in Table 5, an average of 24% of the gold and 13% of the sulphur was recovered in the Falcon concentrate. The higher recovery of gold over sulphur indicates that free gold displaced sulphide bearing minerals during the operation of the Falcon concentrator until the centrifugal bowl became full (~750 g).

(iii) Direct Cyanidation

Representative 1 kg charges were prepared from the gold tailing samples for cyanidation testing. All samples were leached for 120 hours at 50% solids using 1.0 g/L NaCN and pH 10.5 - 11.0. Aliquots were removed every 24 hours in order to determine the rate of extraction of the gold. The results are summarized as follows.

Table 6: Summary of Direct Cyanidation Results

Test #	Sample	Reagent Consumption kg/t		Au Extraction, %					Residue, g/t Au	Feed(calc), g/t Au
		NaCN	CaO	24h	48h	72h	96h	120h		
1	Tashota	3.28	10.4	68	78	78	78	78.3	0.69	3.30
2	Bankfield	0.91	4.10	79	79	79	79	79.0	0.62	3.24
3	Little Long Lac	0.51	0.79	56	55	55	55	55.1	0.67	1.53

Maximum extraction of gold was reached after 48 hours of leaching.

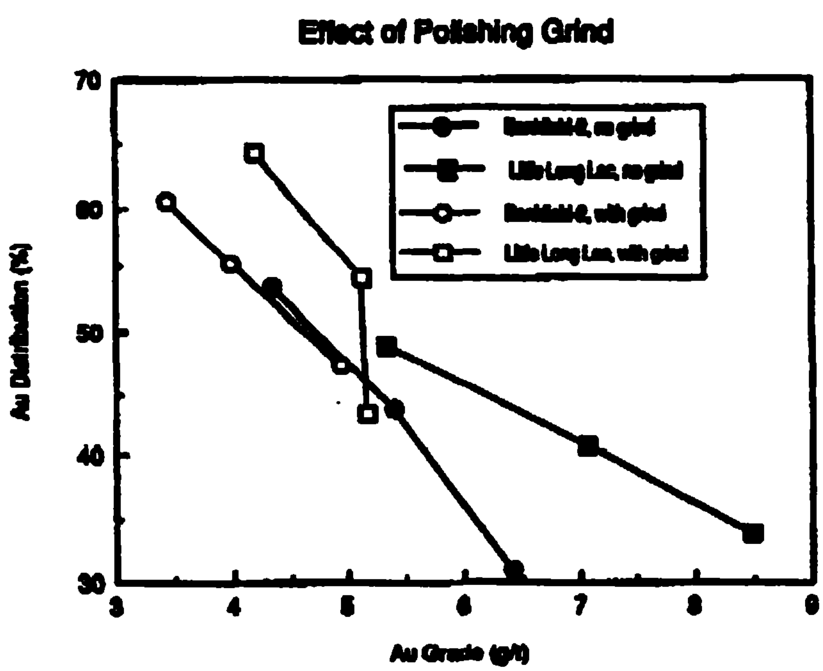
(iv) Flotation

A series of bench scale flotation tests was conducted to investigate the recovery of gold into a sulphide concentrate. Flotation tests were performed on either "as is" material or lightly reground material. The purpose of the light regrind was to polish the surfaces and remove any oxidized layers which may have formed. Aero 412 and PAX were used as collectors, Dowfroth 250 as the frother, $CuSO_4$ and Na_2S as activators. Table 7 summarizes the flotation conditions and metallurgical results.

Table 7: Summary of Flotation Conditions and Metallurgical Results

Test #	Sample	Conditions	Product	Weight %	Assay, g/t or % Au	S	Distribution, % Au	S
F1	Little Long Lac	no grind	Ro Conc 1	0.5	34.0	7.52	13.2	18.0
		200 g/t A350	Ro Conc 1+2	1.0	38.2	7.85	27.5	38.8
		120 g/t AF25	Ro Conc 1-3	5.3	10.9	2.27	43.4	55.4
		1000 g/t Na2S	Ro Conc 1-4	11.4	6.27	1.45	53.2	75.7
		400 g/t CuSO4	Ro Tail	88.6	0.71	0.22	46.8	24.3
			Head(calc)	-	1.34	0.22	-	-
F2	Bankfield-2	no grind	Ro Conc 1	6.4	6.43	19.9	30.9	61.4
		150 g/t A350	Ro Conc 1+2	10.8	5.39	15.9	43.9	83.1
		90 g/t R412	Ro Conc 1-3	16.3	4.34	11.5	53.4	90.7
		500 g/t CuSO4	Ro Tail	83.7	0.74	0.23	46.6	9.3
		18 g/t DF250	Head(calc)	-	1.33	2.07	-	-
F3	Little Long Lac	no grind	Ro Conc 1	5.5	8.47	1.94	33.9	60.1
		150 g/t A350	Ro Conc 1+2	7.8	7.06	1.57	40.8	69.5
		90 g/t R412	Ro Conc 1-3	12.6	5.32	1.13	48.9	80.2
		500 g/t CuSO4	Ro Tail	87.4	0.80	0.04	51.1	19.8
		20 g/t DF250	Head(calc)	113.3	1.37	0.18	-	-
F4	Bankfield-2	grind	Ro Conc 1	12.7	4.93	13.0	47.4	79.8
		150 g/t A350	Ro Conc 1+2	18.3	3.98	9.90	55.2	87.5
		90 g/t R412	Ro Conc 1-3	23.2	3.45	8.21	60.4	91.8
		500 g/t CuSO4	Ro Tail	76.8	0.68	0.22	39.6	8.2
		20 g/t DF250	Head(calc)	-	1.32	2.07	-	-
F5	Little Long Lac	grind	Ro Conc 1	11.2	5.17	0.90	43.5	49.8
		150 g/t A350	Ro Conc 1+2	14.1	5.11	0.82	54.2	57.4
		90 g/t R412	Ro Conc 1-3	20.4	4.20	0.68	64.2	68.5
		500 g/t CuSO4	Ro Tail	79.6	0.60	0.08	35.8	31.5
		20 g/t DF250	Head(calc)	-	1.33	0.20	-	-

As seen in Figure 1, the polishing grind helped improve the overall recovery of the gold, but with a lower grade of concentrate.



-Figure 1-

Appended to this summary are the test details. Column leaches on the Tashota, Little Long Lac and Bankfield-2 are underway.

If you have any questions, please do not hesitate to contact us.

Yours truly,
LAKEFIELD RESEARCH

K.W. Sbutt.

K.W. Sarbutt
Manager - Mineral Processing

D Evans

D. Evans
Project Metallurgist

KWS/DE:bjs
Enclosures - 9

DETAILS OF TESTWORK

Test No.	Sample	Test Description
1	Tashota	Cyanidation
2	Bankfield	Cyanidation
3	Little Long Lac	Cyanidation
F1	Little Long Lac	Flotation
F2	Bankfield-2	Flotation with no grind
F3	Little Long Lac	Flotation with grind
F4	Bankfield-2	Flotation with grind
F5	Little Long Lac	Flotation with grind

Test 1 Project: 4095 Date: Jan/8/91 Operator: KoS

Purpose: To evaluate Au extraction by direct cyanidation.

Procedure: The ore was pulped in a 2L bottle and agitated on mechanical rolls. NaCN and lime were added and maintained at described levels and cyanidation was carried out in 1 x 120 hour stage. Pregnant sub-samples were removed at 24,48,72, and 96h, with bottles being weighed before and after sampling. At end of test, pulp was filtered and washed, with all products submitted for assay.

Feed: 1000 g minus 28 mesh Tashota

Solution Volume: 1000 mL **Pulp Density:** 50 % Solids

Sol'n Composition: 1.0 g/L NaCN

pH Range: 10.5-11.0 Ca(OH)2

Reagent Consumption (kg/t of cyanide feed) NaCN: 3.28 CaO: 10.4

24h NaCN Consumption: 1.58
48h NaCN Consumption: 2.09
72h NaCN Consumption: 2.53
96h NaCN Consumption: 2.84
120h NaCN Consumption: 3.28

Time	Added, Grams				Residual		Consumed		pH
	Actual		Equivalent		Grams		Grams		
Hours	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	
0 - 1	1.05	8.29	1.00	6.30	0.51	-	0.49	-	10.5-9.5
1 - 3	0.52	1.22	0.49	0.93	0.68	-	0.32	-	10.6-10.0
3 - 5	0.34	0.90	0.32	0.68	0.96	-	0.04	-	11.0-10.3
5 - 8	0.04	0.81	0.04	0.62	0.77	-	0.23	-	11.3-10.5
8 - 24	0.24	0.66	0.23	0.50	0.59	-	0.41	-	11.3-10.5
24 - 32	0.43	0.37	0.41	0.28	0.78	-	0.22	-	11.1-10.6
32 - 48	0.23	0	0.22	0	0.71	-	0.29	-	10.6-10.4
48 - 72	0.31	0.34	0.29	0.26	0.58	-	0.42	-	11.0-10.4
72 - 96	0.44	0.17	0.42	0.13	0.70	-	0.30	-	10.8-10.3
96 - 120	0.32	0.31	0.30	0.24	0.58	0.06	0.42	9.87	10.9-10.7
Total	3.92	13.07	3.72	9.93	0.58	0.06	3.12	9.87	

Metallurgical Balance

Product	Weight mL, g	Assays Au, mg/L, g/t	Distribution, % Au	Estimated Extraction, % Au
+ 28 mesh	29.5	1.49	1.4	
24h Preg	25.0	1.31	1.0	68
48h Preg	25.0	1.54	1.2	78
72h Preg	25.0	1.55	1.2	78
96h Preg	25.0	1.57	1.2	78
120h Preg	1271	1.51	59.6	-
120h Wash	747	0.61	14.1	78.3
Residue	948.5	0.69	20.3	
Feed(calc)	978.0	3.30	100.0	

Test 2 Project: 4095 Date: Jan/9/91 Operator: KcS

Purpose: To evaluate Au extraction by direct cyanidation.

Procedure: The ore was pulped in a 2L bottle and agitated on mechanical rolls. NaCN and lime were added and maintained at described levels and cyanidation was carried out in 1 x 120 hour stage. Pregnant sub-samples were removed at 24,48,72, and 96h, with bottles being weighed before and after sampling. At end of test, pulp was filtered and washed, with all products submitted for assay.

Feed: 1000 g minus 28 mesh Bankfield

Solution Volume: 1000 mL **Pulp Density:** 50 % Solids

Sol'n Composition: 1.0 g/L NaCN

pH Range: 10.5-11.0 $Ca(OH)_2$

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.91 CaO: 4.10

24h NaCN Consumption: 0.60
48h NaCN Consumption: 0.71
72h NaCN Consumption: 0.78
96h NaCN Consumption: 0.78
120h NaCN Consumption: 0.91

Time	Added, Grams				Residual		Consumed		pH
	Actual		Equivalent		Grams		Grams		
Hours	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	
0 - 1	1.05	2.09	1.00	1.59	0.52	-	0.48	-	10.5-9.4
1 - 3	0.51	0.72	0.48	0.55	0.91	-	0.09	-	10.6-9.9
3 - 5	0.34	0.90	0.32	0.68	1.00	-	0	-	11.3-10.5
5 - 8	0	0.19	0	0.14	1.00	-	0	-	10.8-10.5
8 - 24	0	0.35	0	0.27	1.00	-	0	-	11.1-10.3
24 - 32	0	0.18	0	0.14	1.00	-	0	-	10.9-10.5
32 - 48	0	0	0	0	0.89	-	0.11	-	10.5-10.3
48 - 72	0.12	0.24	0.11	0.18	0.94	-	0.06	-	11.0-10.3
72 - 96	0.06	0.18	0.06	0.14	1.00	-	0	-	10.8-10.3
96 - 120	0	0.23	0	0.17	0.87	0.06	0.13	3.80	10.9-10.5
Total	2.06	5.08	1.96	3.86	0.87	0.06	0.85	3.80	

Metallurgical Balance

Product	Weight mL, g	Assays Au, mg/L, g/t	Distribution, % Au	Estimated Extraction, % Au
+ 28 mesh	53.1	1.73	2.9	
24h Preg	25.0	1.80	1.4	79
48h Preg	25.0	1.79	1.4	79
72h Preg	25.0	1.77	1.4	79
96h Preg	25.0	1.72	1.4	79
120h Preg	1085	1.65	58.3	-
120h Wash	750	0.73	17.2	79.0
Residue	928.1	0.62	18.1	
Feed(calc)	981.2	3.24	100.0	

Test 3 Project: 4095 Date: Jan/8/91 Operator: KcS

Purpose: To evaluate Au extraction by direct cyanidation.

Procedure: The ore was pulped in a 2L bottle and agitated on mechanical rolls. NaCN and lime were added and maintained at described levels and cyanidation was carried out in 1 x 120 hour stage. Pregnant sub-samples were removed at 24,48,72, and 96h, with bottles being weighed before and after sampling. At end of test, pulp was filtered and washed, with all products submitted for assay.

Feed: 1000 g minus 28 mesh Little Long Lac

Solution Volume: 1000 mL **Pulp Density:** 50 % Solids

Sol'n Composition: 1.0 g/L NaCN

pH Range: 10.5-11.0 Ca(OH)2

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.51 CaO: 0.79

24h NaCN Consumption:	0.21
48h NaCN Consumption:	0.24
72h NaCN Consumption:	0.33
96h NaCN Consumption:	0.45
120h NaCN Consumption:	0.51

Time	Added, Grams				Residual		Consumed		pH
	Actual		Equivalent		Grams		Grams		
Hours	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	
0 - 1	1.05	0.77	1.00	0.59	0.93	-	0.07	-	11.0-10.7
1 - 3	0.07	0.00	0.07	0	1.00	-	0	-	10.7-10.8
3 - 5	0	0.00	0	0	1.00	-	0	-	10.8-10.7
5 - 8	0	0.00	0	0	1.00	-	0	-	10.7-10.5
8 - 24	0	0.17	0	0.13	0.85	-	0.15	-	11.2-10.9
24 - 32	0.16	0	0.15	0	1.00	-	0	-	10.9-10.9
32 - 48	0	0	0	0	0.97	-	0.03	-	10.9-10.7
48 - 72	0.03	0	0.03	0	0.91	-	0.09	-	10.7-10.5
72 - 96	0.09	0	0.09	0	0.88	-	0.12	-	10.5-10.4
96 - 120	0.13	0.15	0.12	0.11	0.94	0.06	0.06	0.77	11.0-10.8
Total	1.53	1.09	1.45	0.83	0.94	0.06	0.50	0.77	

Metallurgical Balance

Product	Weight mL, g	Assays Au, mg/L, g/t	Distribution, % Au	Estimated Extraction, % Au
+ 28 mesh	5.2	4.29	1.5	
24h Preg	25.0	0.64	1.1	55
48h Preg	25.0	0.67	1.1	55
72h Preg	25.0	0.63	1.1	55
96h Preg	25.0	0.63	1.1	55
120h Preg	973	0.61	39.7	-
120h Wash	1104	0.15	11.1	55.1
Residue	969.0	0.67	43.4	
Feed(calc)	974.2	1.53	100.0	

Test No. F1 | Project No. 4095 | 1/28/91 | Operator: DE

Purpose: To perform preliminary rougher flotation test.

Procedure: As stated below.

Feed: ~2kg of -28 mesh Litte Long Lac

Grind: N/A

Stage	Reagents added, grams per tonne							Time, minutes			
	A350	AF25	Na2S	CuSO4	DF-250			Grind	Cond.	Froth	pH
Rougher 1	50	40							1	1	8.2
Condx				400					5		
Rougher 2	50	40							1	2	7.6
Condx			500						5		
Rougher 3	50	40							1	5	9.2
Condx			500						5		
Rougher 4	50								1	10	9.5

Stage	Ro										
Flotation Cell	D-1										
Speed: r.p.m.	1800										
% Solids	35										

Metallurgical Balance

Product	Weight g	Weight %	Assays, g/t, % Au	Assays, g/t, % S	Distribution, % Au	Distribution, % S
1. Ro Conc 1	10.2	0.5	34.0	7.52	13.2	18.0
2. Ro Conc 2	9.7	0.5	38.5	8.19	14.3	18.6
3. Ro Conc 3	84.2	4.3	4.95	0.95	15.9	18.8
4. Ro Conc 4	118.3	6.1	2.18	0.73	9.8	20.3
5. Ro Tailing	1727.5	88.6	0.71	0.06	46.8	24.3
Feed(calc)	1949.9	100.0	1.34	0.22	100.0	100.0

Combined Products

Product	Weight g	Weight %	Au	S	Au	S
Ro Conc 1+2	19.9	1.0	36.2	7.85	27.5	36.6
Ro Conc 1-3	104.1	5.3	10.9	2.27	43.4	55.4
Ro Conc 1-4	222.4	11.4	6.27	1.45	53.2	75.7

Test No. F2 | Project No. 4095 | 1/31/91 | Operator: JMD

Purpose: To perform preliminary rougher flotation test.

Procedure: As stated below.

Feed: ~2kg of -28 mesh Bankfield # 2

Grind: N/A

Stage	A350	R412	Na2S	CuSO4	DF-250			Grind	Cond.	Froth	pH
	Reagents added, grams per tonne							Time, minutes			
Codx 1			500						2		8.0
Condx 2				400					5		7.5
Rougher 1	50	30			10				1	5	7.6
Rougher 2	50	30							1	10	
Rougher 3	50	30			7.5				1	10	

Stage	Ro										
Flotation Cell	D-1										
Speed: r.p.m.	1800										
% Solids	35										

Metallurgical Balance

Product	Weight g	Weight %	Assays, g/t, % Au	Assays, g/t, % S	Distribution, % Au	Distribution, % S
1. Ro Conc 1	124.8	6.4	6.43	19.9	30.9	61.4
2. Ro Conc 2	86.1	4.4	3.89	10.2	12.9	21.7
3. Ro Conc 3	107.8	5.5	2.29	2.85	9.5	7.6
4. Ro Tailing	1633.9	83.7	0.74	0.23	46.6	9.3
Feed(calc)	1952.6	100.0	1.33	2.07	100.0	100.0

Combined Products

Product	Weight g	Weight %	Au	S	Au	S
Ro Conc 1+2	210.9	10.8	5.39	15.9	43.9	83.1
Ro Conc 1-3	318.7	16.3	4.34	11.5	53.4	90.7

Test No. F3 Project No. 4095 2/4/91 Operator: JMD

Purpose: To perform preliminary rougher flotation test.

Procedure: As stated below.

Feed: ~2kg of -28 mesh Little Long Lac

Grind: N/A

	Reagents added, grams per tonne							Time, minutes			
Stage	A350	R412	Na2S	CuSO4	DF-250			Grind	Cond.	Froth	pH
Codx 1			500						2		9.8
Condx 2				400					5		9.3
Rougher 1	50	30			10				1	5	
Rougher 2	50	30			2.5				1	10	
Rougher 3	50	30			7.5				1	15	

Stage	Ro										
Flotation Cell	D-1										
Speed: r.p.m.	1800										
% Solids	35										

Metallurgical Balance

Product	Weight g	Weight %	Assays, g/t, % Au	Assays, g/t, % S	Distribution, % Au	Distribution, % S
1. Ro Conc 1	107.6	5.5	8.47	1.94	33.9	60.1
2. Ro Conc 2	46.2	2.4	3.77	0.71	6.5	9.4
3. Ro Conc 3	93.0	4.7	2.44	0.40	8.4	10.7
4. Ro Tailing	1716.8	87.4	0.80	0.04	51.1	19.8
Feed(calc)	1963.6	100.0	1.37	0.18	100.0	100.0

Combined Products

Product	Weight g	Weight %	Assays, g/t, % Au	Assays, g/t, % S	Distribution, % Au	Distribution, % S
Ro Conc 1+2	153.8	7.8	7.06	1.57	40.4	69.5
Ro Conc 1-3	246.8	12.6	5.32	1.13	48.9	80.2

Test No. F4 Project No. 4095 2/4/91 Operator: JMD

Purpose: To perform preliminary rougher flotation test.

Procedure: As stated below.

Feed: ~2kg of -28 mesh Bankfield-2

Grind: 8minutes/2kg @ 65% solids in laboratory rod mill

	Reagents added, grams per tonne							Time, minutes			
Stage	A350	R412	Na2S	CuSO4	DF-250			Grind	Cond.	Froth	pH
Codx 1			500						2		9.2
Condx 2				400					5		7.8
Rougher 1	50	30			10				1	5	
Rougher 2	50	30			2.5				1	10	
Rougher 3	50	30			7.5				1	15	

Stage	Ro										
Flotation Cell	D-1										
Speed: r.p.m.	1800										
% Solids	35										

Metallurgical Balance

Product	Weight		Assays, g/t, %		Distribution, %	
	g	%	Au	S	Au	S
1. Ro Conc 1	249.6	12.7	4.93	13.0	47.4	79.8
2. Ro Conc 2	110.0	5.6	1.83	2.86	7.8	7.7
3. Ro Conc 3	95.4	4.9	1.43	1.85	5.3	4.3
4. Ro Tailing	1509.0	76.8	0.68	0.22	39.6	8.2
Feed(calc)	1964.0	100.0	1.32	2.07	100.0	100.0

Combined Products

Ro Conc 1+2	359.6	18.3	3.98	9.90	55.2	87.5
Ro Conc 1-3	455.0	23.2	3.45	8.21	60.4	91.8

Test No. F5 | Project No. 4095 | 2/4/91 | Operator: JMD

Purpose: To perform preliminary rougher flotation test.

Procedure: As stated below.

Feed: ~2kg of -28 mesh Little Long Lac

Grind: 5 minutes/2kg @ 65% solids in laboratory rod mill

	Reagents added, grams per tonne							Time, minutes			
Stage	A350	R412	Na2S	CuSO4	DF-250			Grind	Cond.	Froth	pH
Condx 1			500						2		9.7
Condx 2				400					5		9.0
Rougher 1	50	30			10				1	5	
Rougher 2	50	30			2.5				1	10	
Rougher 3	50	30			7.5				1	15	

Stage	Ro										
Flotation Cell	D-1										
Speed: r.p.m.	1800										
% Solids	35										

Metallurgical Balance

Product	Weight		Assays, g/t, %		Distribution, %	
	g	%	Au	S	Au	S
1. Ro Conc 1	220.0	11.2	5.17	0.90	43.5	49.8
2. Ro Conc 2	57.7	2.9	4.89	0.52	10.8	7.6
3. Ro Conc 3	122.6	6.2	2.12	0.36	9.9	11.1
4. Ro Tailing	1563.5	79.6	0.60	0.08	35.8	31.5
Feed(calc)	1963.8	100.0	1.33	0.20	100.0	100.0

Combined Products

Ro Conc 1+2	277.7	14.1	5.11	0.82	54.2	57.4
Ro Conc 1-3	400.3	20.4	4.20	0.68	64.2	68.5

LAKEFIELD RESEARCH
A DIVISION OF FALCONBRIDGE LIMITED

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March 5, 1991

Mr. D. Malouf
Beaurox Mines Limited
Suite 801
80 Richmond Street West
Toronto, Ontario
M5H 2A4

by FAX and Courier

Dear Mr.Malouf:

Re: LR4095 - Summary Letter No. 2
The Recovery of Gold from Low Grade Tailing Samples

Please find enclosed a summary of all testwork completed since the publication of Summary Letter No. 1.

The testwork performed during this segment includes direct cyanidation with lead nitrate addition, heap leach simulation, and column flotation and gravity separation pilot plant testing.

(i) Direct Cyanidation with Lead Nitrate

A series of tests was performed to investigate the effect of lead nitrate addition on the cyanidation response of the tailing samples. All tests involved the used of 1 kg charges diluted to 50% solids. A dosage of 500 g/t lead nitrate was added in each test. Cyanidation conditions and metallurgical results are summarized in Table 1.

185 CONCESSION STREET, POSTAL BAG 4300, LAKEFIELD, ONTARIO, CANADA KOL 2HO

(ii) Heap Leach Simulation

The feed for the column leach testwork was agglomerated with 5 kg/t of Portland No. 2 cement, the amount on lime consumed in bench scale cyanidation tests and approximately one half of the amount of sodium cyanide consumed in the preliminary bottle roll tests.

The column leach apparatus was set up using 15 cm diameter plexiglass conduit of about 1.5 metres in height. The bottoms of the columns were fitted with a steel mesh to retain the solids. The columns were leached with flowrates of about 5 mL per minute (15 L/h/m^2). The pregnant solution discharge from the column was passed through a cartridge containing 15 grams of Calgon GRC-22 pre-attritioned carbon. Flowrates were monitored daily and adjusted as required to maintain an uniform flow. The loaded carbons were changed periodically to monitor gold extraction. Barren solutions were regularly sample to determine NaCN and CaO concentrations. At time of issuing this report, the Bankfield-2 and Little Long Lac column tests were complete while Tashota column was entering Day 26. Table 3 summarizes the results of the column leach tests. The results for the Tashota sample are based on the direct head assay.

Table 3: Summary of Column Leach Results

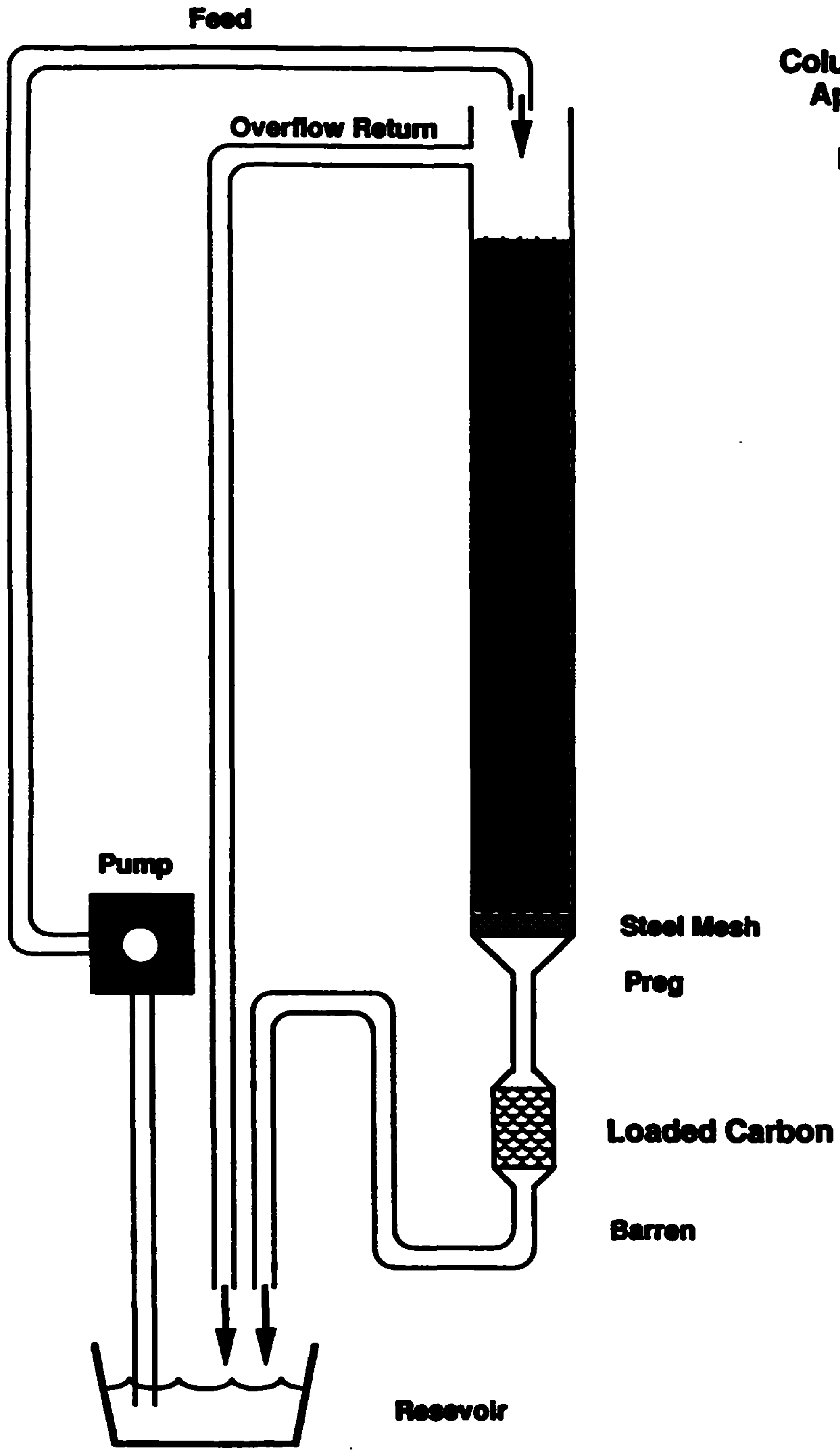
Standard Conditions: 0.5 g/L NaCN
pH 10.0 - 11.0
carbon changes on days 1, 2, 4, 7, 14 and 21
~5 mL/min flowrate
5 kg/t Portland No. 2 cement

Test #	Sample	Reagent Consumption kg/t		Au Extraction, %	Residue, g/t Au	Feed(calc), g/t Au
		NaCN	CaO			
10	Tashota	3.34	N/A	73.4*	N/A	N/A
11	Little Long Lac	1.00	0.97	50.0	0.68	1.40
12	Bankfield-2	1.75	4.75	56.0	0.57	1.30

* after 21 days of leaching

Figure 1 shows the column leach apparatus while Figure 2 depicts the recovery of gold with respect to time.

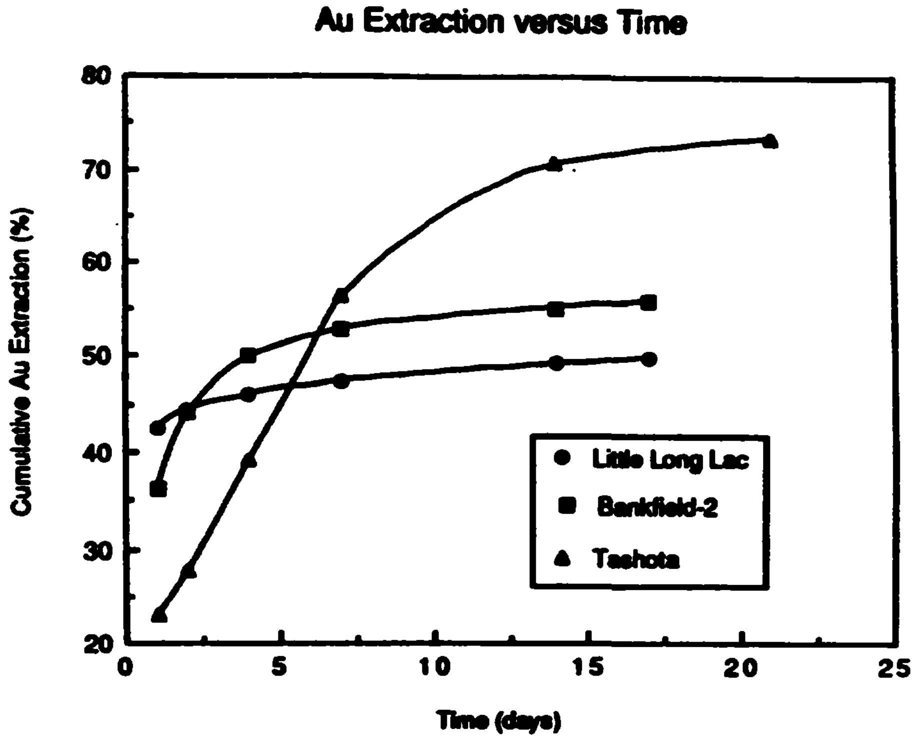
The metallurgical results, available to date, for Little Long Lac and Tashota column leaches confirms the results obtained from the preliminary bottle roll cyanidations (without the addition of lead nitrate). Again, it is not possible to compare the Bankfield and Bankfield-2 results since these samples are not representative of each other.



Column Leach Apparatus

Figure 1

Figure 2

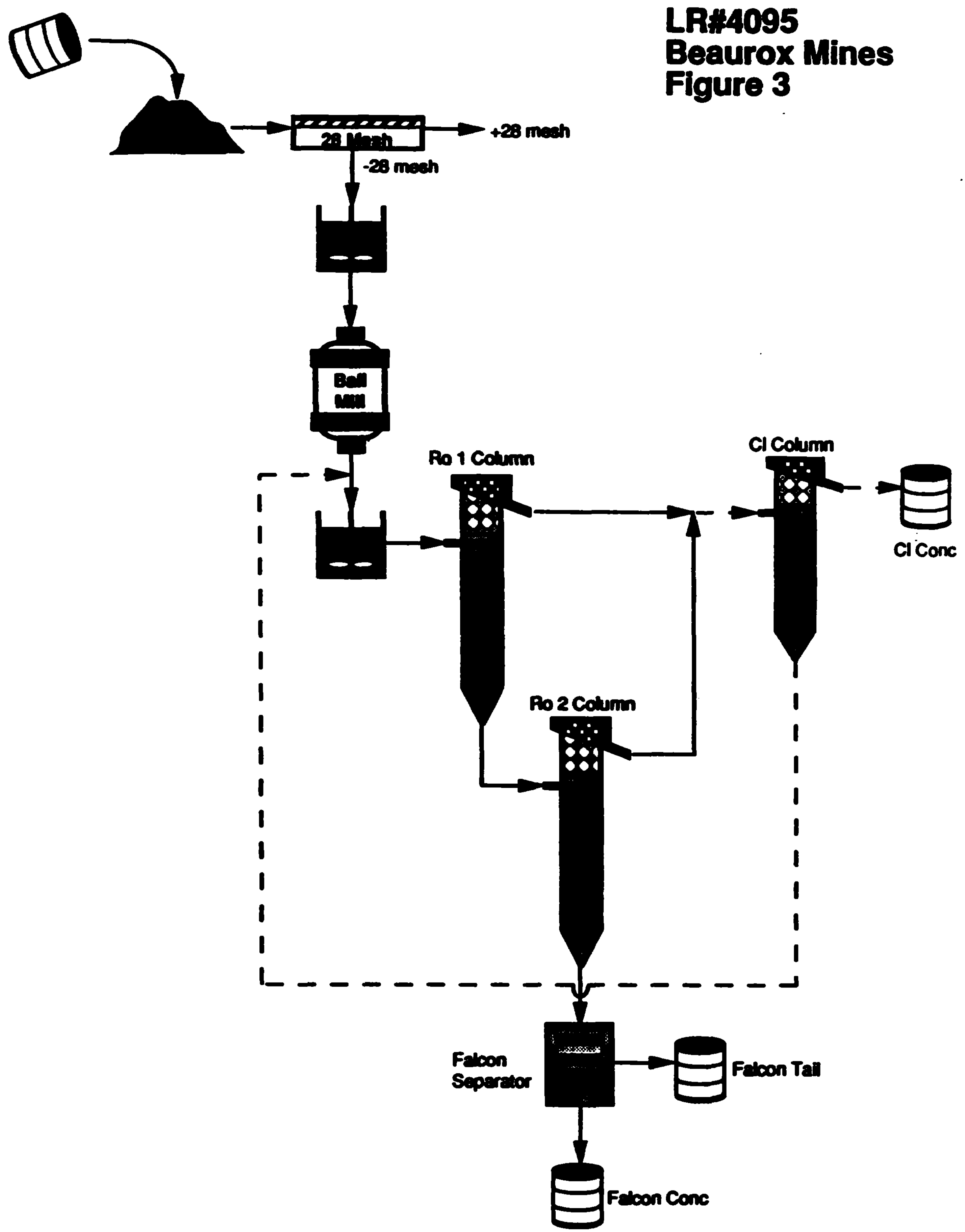


(iii) Column Flotation

The pilot plant testing of the Beaurox Mine samples utilized two 6 inch column, except for Test No. PP-1, which had a 4 inch cleaner column added.

The column were controlled by a process control computer using pressure transducers installed near the bottom of the columns. The pressure transducer senses the pulp level by transmitting a 0 to 100 mV signal to a µMAC6000 controller. These signals are digitized and used in a multi-PID alogrithm, along with manually selected setpoints, to prepare 4-20 mA signals to control the speed of the tailing paristaltic pumps. Air was injected into each column through a sintered stainless steel sparger. Feed to the column cells was controlled with manually set variable speed paristaltic pumps. A general arrangement of the flowsheet used is shown in Figure 3.

The purpose of the pilot plant testing was to determine if column flotation could produce a recovery the same as or greater than that achieved int the bench scale mechanical flotation tests.


LR#4095
Beaurox Mines
Figure 3
28 Mesh
+28 mesh
-28 mesh
Ball
Mill
Ro 1 Column
Cl Column
Cl Conc
Ro 2 Column
Falcon
Separator
Falcon Tail
Falcon Conc

All tests used two 6 inch diameter, 2.5 cu. ft. columns as roughers. The tailing from the first 6 inch column fed the second 6 inch column. The two rougher columns combined gave an average retention time of approximately 50 minutes. Concentrates from the rougher columns were collected together as a combined rougher concentrate. The tailing from the second rougher column was fed to a Falcon separator. This was done in an attempt to recover any free gold which was not floated. Test No. PP-1 also made use of a 4 inch cleaner column. The combined rougher concentrate was fed to this column. The cleaner concentrate was collected and the cleaner tailing was recirculated back to the first rougher column. Tables 4, 5, and 6 provide summaries of flotation conditions, column key variables and metallurgical results, respectively.

Table 4: Summary of Pilot Plant Flotation Conditions

Test #	Sample	Feed Rate kg/h	BMF Grind %-400 mesh	BMD Grind %-400 mesh	Reagents, g/t				
					Na2S	A350	R412	CuSO4	DF250
PP-1	Bankfield	79	42.2	49.4	661	159	115	684	17
PP-2	Bankfield	68	42.1	49.4	644	247	156	618	26
PP-3	Little Long Lac	81	40.8	49.0	600	182	113	593	33
PP-4	Little Long Lac	64	40.0	49.5	778	246	150	609	49

Analysis of the data shows that the columns were operated within normal ranges for wash rate, gas rate and gas holdup. The limited scope of the test program did not provide an opportunity to optimize conditions or to evaluate conditions outside the normal ranges.

Additional testing would be required to examine ways to improve gold recovery. Specifically, reduced wash water flow and shallower froth bed might improve recovery, but the present test results suggest that improvements in recovery might well be at the expense of grade. Additional bench scale tests may also be required to evaluate alternative collectors.

Table 5: Summary of Column Key Variables

Test #	Sample	Rougher#1 Gas Velocity cm/sec	Rougher#1 Wash Velocity cm/sec	Rougher#1 Feed Velocity cm/sec	Rougher#1 NRT minutes	Rougher#1 Flow Bias %	Rougher#1 Gas Hold Up %	Rougher#2 Gas Velocity cm/sec	Rougher#2 Wash Velocity cm/sec	Rougher#2 Feed Velocity cm/sec	Rougher#2 NRT minutes	Rougher#2 Flow Bias %	Rougher#2 Gas Hold Up %
PP-1	Bankfield	0.92	0.20	0.28	27	27	16	0.63	0.10	0.28	25	26	17
PP-2	Bankfield	1.13	0.20	0.34	22	32	22	0.66	0.10	0.33	20	30	16
PP-3	Little Long Lac	0.74	0.20	0.33	23	31	13	0.74	0.10	0.32	21	31	13
PP-4	Little long Lac	0.74	0.20	0.25	30	23	17	0.74	0.10	0.25	28	23	17

Table 6: Summary of Pilot Plant Metallurgical Results

Test #	Sample	Product	Weight %	Assay, g/t or % Au	Assay, g/t or % S	Distribution, % Au	Distribution, % S
PP-1	Bankfield	Combined Cl Conc	4.0	11.7		30.0	
		Falcon Conc	0.2	24.0		3.1	
		Falcon Tailing	96.0	1.08		67.0	
		Feed(calc)	100.0	1.53		100.0	
		Feed(assay)		1.53			
PP-2	Bankfield	Combined Ro Conc	8.6	8.48	27.1	47.2	69.6
		Falcon Conc	0.4	12.5	17.2	3.2	2.1
		Falcon Tailing	91.0	0.84	1.04	49.6	28.4
		Feed(calc)	100.0	1.54	3.34	100.0	100.0
		Feed(assay)		1.54	3.23		
PP-3	Little Long Lac	Combined Ro Conc	2.3	11.8	6.16	21.3	40.6
		Falcon Conc	0.2	30.3	9.43	4.7	7.2
		Falcon Tailing	97.5	0.98	0.14	74.0	52.2
		Feed(calc)	100.0	1.29	0.30	100.0	100.0
		Feed(assay)		1.29	0.24		
PP-4	Little Long Lac	Combined Ro Conc	1.7	13.5	7.06	20.7	44.5
		Falcon Conc	0.4	20.4	1.04	7.2	1.6
		Falcon Tailing	97.9	0.84	0.15	72.1	53.9
		Feed(calc)	100.0	1.14	0.27	100.0	100.0
		Feed(assay)		1.14	0.25		

(iv) Gravity Concentration

Two concentration tests were conducted using a Reichert Mark VII spiral and a Falcon separator. Figure 4 depicts the flowhseet layout and Table 7 summarizes the metallurgical results.

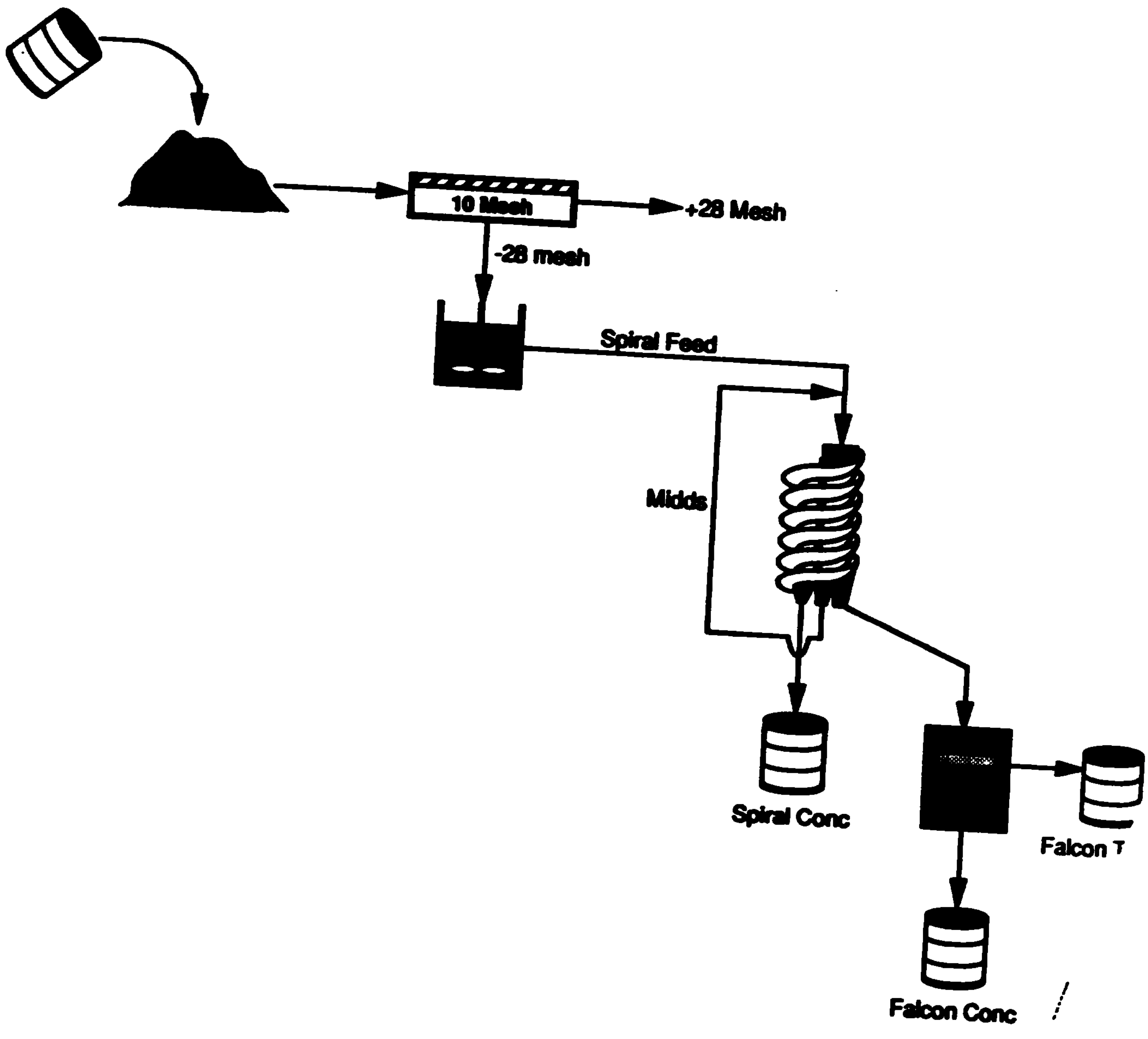
Table 7: Summary of Spiral Metallurgical Results

Test #	Sample	Product	Weight %	Assay, g/t Au	Distribution, % Au
PP-5	Bankfield	Spiral Conc	6.9	4.81	23.0
		Falcon Conc	0.2	70.5	9.7
		Falcon Tailing	92.9	1.05	67.3
		Feed(calc)	100.0	1.45	100.0
		Feed(assay)		1.45	
PP-6	Little Long Lac	Spiral Conc	3.2	6.57	17.9
		Falcon Conc	0.4	40.2	13.5
		Falcon Tailing	96.4	0.85	68.6
		Feed(calc)	100.0	1.19	100.0
		Feed(assay)		1.19	

The metallurgical results from the spiral tests are comparable with earlier gravity concentration testwork. An average of 31% of the gold was recovered using a spiral/Falcon combination in comparison to 24% gold recovery when only the Falcon separator was used.

There appears to be no advantage to using gravity concentration as a means of recovering free gold or gold bearing sulphides.

LR#4095
Beaurox Mines
Figure 4



(v) Conclusions

1. The addition of lead nitrate helped reduce the cyanide consumption by roughly half.
2. The extraction of gold from the column leaches was comparable to that obtained in the preliminary bottle roll tests.
3. Column flotation of the Bankfield and Little Long Lac bulk samples produced higher grade concentrates at lower recoveries in comparison to the earlier mechanical bench scale tests.
4. The recovery of gold using a spiral and Falcon separator was not successful. The metallurgical results obtained confirmed earlier gravity concentration testwork where only 24% of the gold was recovered.

(vi) Recommendations

1. To perform diagnostic leaching tests, in order to determine the location and association of the gold.
2. To conduct a mineralogical examination on feed and tailing sample. This will help identify gangue components, liberation, association and potential recovery of the gold. Centrifuge and heavy liquid tests would be conducted to concentrate the free gold and gold bearing sulphides.
3. To conduct further bench scale flotation tests to investigate new reagent schemes and optimize flotation conditions.
4. To conduct C.I.L. tests to determine if preg robbing or re-precipitation of the gold during leaching is affecting the final extraction of gold.

Appended to this summary are the test details. The Tashota 4 inch column leach will be completed on March 7th. The large scale Tashota column leach is underway and has been leaching for approximately one week.

If you have any questions, please do to hesitate to contact us.

Yours truly,
LAKEFIELD RESEARCH

K.W. Sarbutt
Manager - Mineral Processing

D. Evans
Project Metallurgist

KWS/DE:bjs
Enclosures - 41

Mr. D. Malouf
March 5, 1991

DETAILS OF TESTWORK

Test No.	Sample	Test Description
7	Tashota	cyanidation with lead nitrate
8	Bankfield-2	cyanidation with lead nitrate
9	Little Long Lac	cyanidation with lead nitrate
10	Tashota	column leach
11	Little Long Lac	column leach
12	Bankfield-2	column leach
PP-1	Bankfield	column flotation + gravity separation
PP-2	Bankfield	column flotation + gravity separation
PP-3	Little Long Lac	column flotation + gravity separation
PP-4	Little Long Lac	column flotation + gravity separation
PP-5	Bankfield	gravity concentration
PP-6	Little Long Lac	gravity concentration

Test 7 | Project: 4095 | Date: Feb/4/91 | Operator: JH

Purpose: To evaluate Au extraction by direct cyanidation.

Procedure: The ore was pulped in a 2L bottle and agitated on mechanical rolls. NaCN and lime were added and maintained at described levels and cyanidation was carried out in 1 x 72 hour stage. Lead nitrate was added at the being of the leach. At the end of the test, pulp was filtered and washed, with all products being submitted for assay.

Feed: 1000 g minus 28 mesh Tashota

Solution Volume: 1000 mL **Pulp Density:** 50 % Solids

Sol'n Composition: 0.5 g/L NaCN

pH Range: 10.5-11.0 $Ca(OH)_2$

$Pb(NO_3)_2$: 500 g/t

Reagent Consumption (kg/t of cyanide feed) NaCN: 1.47 CaO: 9.67

	NaCN	CaO
24h NaCN and CaO Consumption:	0.96	9.09
48h NaCN and CaO Consumption:	1.25	9.41
72h NaCN and CaO Consumption:	1.47	9.67

Time	Added, Grams				Residual		Consumed		pH
	Actual		Equivalent		Grams		Grams		
Hours	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	
0 - 1	0.53	8.13	0.50	6.18	0.10	-	0.40	-	10.9-9.0
1 - 2	0.42	1.30	0.40	0.99	0.50	-	0	-	10.9-9.8
2 - 20	0	0.76	0	0.58	0.20	-	0.30	-	10.9-9.5
20 - 24	0.32	0.58	0.30	0.43	0.35	0.02	0.15	8.15	10.8-10.0
24 - 48	0.16	0.40	0.15	0.30	0.25	0.02	0.25	0.28	10.9-9.9
48 - 72	0.26	0.36	0.25	0.27	0.30	0.04	0.20	0.23	10.8-10.0
Total	1.69	11.5	1.61	8.75	0.30	0.04	1.32	8.67	

Metallurgical Balance

Product	Weight mL, g	Assays Au, mg/L, g/t	Distribution, % Au
+ 28 mesh	87.5	2.14	6.5
Preg+Wash	2090	0.96	69.4
Residue	896.2	0.78	24.2
Feed(calc)	983.7	2.94	100.0

Test 8 Project: 4095 Date: Feb/4/91 Operator: JH

Purpose: To evaluate Au extraction by direct cyanidation.

Procedure: The ore was pulped in a 2L bottle and agitated on mechanical rolls. NaCN and lime were added and maintained at described levels and cyanidation was carried out in 1 x 72 hour stage. Lead nitrate was added at the being of the leach. At the end of the test, pulp was filtered and washed, with all products being submitted for assay.

Feed: 1000 g minus 28 mesh Bankfield-2

Solution Volume: 1000 mL **Pulp Density:** 50 % Solids

Sol'n Composition: 0.5 g/L NaCN

pH Range: 10.5-11.0 $Ca(OH)_2$

$Pb(NO_3)_2$: 500 g/t

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.37 CaO: 2.03

	NaCN	CaO
24h NaCN and CaO Consumption:	0.36	1.73
48h NaCN and CaO Consumption:	0.37	1.89
72h NaCN and CaO Consumption:	0.37	2.03

Time	Added, Grams				Residual		Consumed		pH
	Actual		Equivalent		Grams		Grams		
Hours	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	
0 - 1	0.53	1.38	0.50	1.05	0.40	-	0.10	-	11.2-9.8
1 - 2	0.11	0.36	0.10	0.27	0.50	-	0	-	10.9-10.1
2 - 20	0	0.24	0	0.18	0.30	-	0.20	-	10.8-9.7
20 - 24	0.21	0.26	0.20	0.20	0.45	0	0.05	1.70	10.7-10.1
24 - 48	0.05	0.21	0.05	0.16	0.50	0	0	0.16	11.0-10.0
48 - 72	0	0.21	0.00	0.16	0.50	0.02	0	0.14	10.9-10.1
Total	0.90	2.66	0.86	2.02	0.50	0.02	0.36	2.00	

Metallurgical Balance

Product	Weight mL, g	Assays Au, mg/L, g/t	Distribution, % Au
+ 28 mesh	2.3	0.43	0.1
Preg+Wash	2090	0.36	56.4
Residue	983.7	0.59	43.5
Feed(calc)	986.0	1.35	100.0

Test 9 Project: 4095 Date: Feb/4/91 Operator: JH

Purpose: To evaluate Au extraction by direct cyanidation.

Procedure: The ore was pulped in a 2L bottle and agitated on mechanical rolls. NaCN and lime were added and maintained at described levels and cyanidation was carried out in 1 x 72 hour stage. Lead nitrate was added at the being of the leach. At the end of the test, pulp was filtered and washed, with all products being submitted for assay.

Feed: 1000 g minus 28 mesh Little Long Lac

Solution Volume: 1000 mL **Pulp Density:** 50 % Solids

Sol'n Composition: 0.5 g/L NaCN

pH Range: 10.5-11.0 Ca(OH)2

Pb(NO3)2: 500 g/t

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.12 CaO: 0.89

24h NaCN and CaO Consumption:	0.12	0.67
48h NaCN and CaO Consumption:	0.12	0.73
72h NaCN and CaO Consumption:	0.12	0.89

Time	Added, Grams				Residual		Consumed		pH
	Actual		Equivalent		Grams		Grams		
Hours	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	
0 - 1	0.53	0.65	0.50	0.49	0.40	-	0.10	-	11.2-10.5
1 - 2	0.11	0.11	0.10	0.08	0.50	-	0	-	10.9-10.6
2 - 20	0	0.05	0	0.04	0.50	-	0	-	10.8-10.3
20 - 24	0	0.06	0	0.05	0.50	0.01	0	0.65	10.7-10.4
24 - 48	0	0.08	0	0.06	0.50	0	0	0.06	11.0-10.3
48 - 72	0	0.25	0	0.19	0.50	0.04	0	0.15	11.5-10.9
Total	0.64	1.20	0.61	0.91	0.50	0.04	0.12	0.86	

Metallurgical Balance

Product	Weight mL, g	Assays Au, mg/L, g/t	Distribution, % Au
+ 28 mesh	11.9	1.16	1.0
Preg+Wash	2090	0.33	49.1
Residue	973.3	0.72	49.9
Feed(calc)	985.2	1.43	100.0

Test 10 Project: 4095 Date: Mar/04/91 Operator: JH

Purpose: To evaluate Au extraction by direct cyanidation.

Procedure: Approximately 10 kg of agglomerated sample was loaded into a plastic column 102 mm in diameter, to a height of 106 cm. A piece of steel mesh was placed at the bottom of the column and a piece of burlap on top to help disperse the solution. Approximately 5 L of 0.5 g/L NaCN solution was percolated through the column at a rate of 5 mL per minute. The pregnant solution was passed through carbon column where the Au in solution was removed. The reagents were replenished as required during the test. The loaded carbon was changed after 1, 2, 4, 7, 14, and 21 days and replaced with fresh carbon.

Feed: 10000 g minus 28 mesh Tashota

Solution Volume: 5000 mL Pulp Density: 33 % Solids

Sol'n Composition: 0.5 g/L NaCN

pH Range: 10.0-11.0 Ca(OH)2

Reagent Consumption (kg/t of cyanide feed) NaCN: 3.34 CaO: N/A

Time	Added, Grams				Residual		Consumed		pH
	Actual		Equivalent		Grams		Grams		
Days	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	
Aggmolerate	17.0	139	16.2	106	-	-	-	-	
0 - 1	2.00	0	1.90	0	1.91	-	16.1	-	8.9-9.2
1 - 2	0.60	0	0.57	0	1.25	-	1.25	-	9.2-9.4
2 - 3	1.32	0.16	1.25	0.122	1.50	-	1.00	-	10.3-9.5
3 - 4	1.05	1.00	1.00	0.76	1.50	-	1.00	-	11.5-9.8
4 - 5	1.05	1.00	1.00	0.76	1.58	-	0.92	-	11.6-10.1
5 - 6	1.05	0	1.00	0	1.75	-	0.75	-	10.1-9.8
6 - 7	0.79	0.50	0.75	0.38	1.50	-	1.00	-	11.2-9.7
7 - 8	1.05	0.50	1.00	0.38	1.50	-	1.00	-	11.2-9.6
8 - 9	1.05	0.50	1.00	0.38	1.25	-	1.25	-	11.0-9.6
9 - 10	1.32	0.50	1.25	0.38	1.25	-	1.25	-	11.4-9.6
10 - 13	1.32	1.00	1.25	0.76	1.25	-	1.25	-	11.5-9.3
13 - 14	1.32	1.00	1.25	0.76	1.78	-	0.72	-	11.9-9.5
14 - 16	0.76	0.70	0.72	0.53	1.10	-	1.40	-	11.1-9.2
16 - 17	1.46	0.70	1.39	0.53	1.30	-	1.20	-	11.4-9.4
17 - 21	1.26	1.00	1.20	0.76	0.75	-	1.75	-	11.7-8.2
21 - 23	1.84	2.00	1.75	1.52	1.00	-	1.50	-	11.9-9.3
Total	36.2	150	34.4	114	1.00	-	33.4	-	

Metallurgical Balance

Product	Weight mL, g	Assays Au, mg/L, g/t	Distribution, % Au	Cum. to Date, % Au
+ 28 mesh	821	2.17	5.9	
Day 1 Loaded Carbon	15.0	463	22.9	22.9
Day 2 Loaded Carbon	13.9	105	4.8	27.7
Day 4 Loaded Carbon	14.0	246	11.4	39.1
Day 7 Loaded Carbon	15.6	338	17.4	56.5
Day 14 Loaded Carbon	16.9	254	14.2	70.6
Day 21 Loaded Carbon	17.9	46.3	2.7	73.4
Feed(assay) *	10000	3.03	-	

* BASED ON DIRECT HEAD ASSAY ONLY

Test 11 Project: 4095 Date: Feb/22/91 Operator: JH

Purpose: To evaluate Au extraction by direct cyanidation.

Procedure: Approximately 10 kg of agglomerated sample was loaded into a plastic column 102 mm in diameter, to a height of 100 cm. A piece of steel mesh was placed at the bottom of the column and a piece of burlap on top to help disperse the solution. Approximately 5 L of 0.5 g/L NaCN solution was percolated through the column at a rate of 5 mL per minute. The pregnant solution was passed through carbon column where the Au in solution was removed. The reagents were replenished as required during the test. The loaded carbon was changed after 1, 2, 4, 7, 14, and 17 days and replaced with fresh carbon.

Feed: 10000 g minus 28 mesh Little Long Lac

Solution Volume: 5000 mL Pulp Density: 33 % Solids

Sol'n Composition: 0.5 g/L NaCN

pH Range: 10.0-11.0 $Ca(OH)_2$

Reagent Consumption (kg/t of cyanide feed) NaCN: 1.00 CaO: 0.97

Time	Added, Grams				Residual		Consumed		pH
	Actual		Equivalent		Grams		Grams		
Days	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO	
Aggmolerate	3.00	10.5	2.85	7.98	-	-	-	-	
0 - 1	2.00	0	1.90	0	1.00	-	3.75	-	8.9-9.6
1 - 2	1.60	0	1.50	0	1.75	-	0.75	-	9.6-9.8
2 - 3	1.84	0	1.75	0	2.50	-	0	-	9.8-9.9
3 - 4	0	0	0	0	1.75	-	0.75	-	9.9-9.9
4 - 5	0.79	0.60	0.75	0.46	2.00	-	0.50	-	11.6-10.1
5 - 6	0.53	0	0.50	0	2.25	-	0.25	-	10.1-10.0
6 - 7	0.26	0	0.25	0	2.00	-	0.50	-	10.0-10.1
7 - 8	0.53	0	0.50	0	2.50	-	0	-	10.1-10.1
8 - 9	0	0.50	0	0.38	2.50	-	0	-	11.5-10.3
9 - 10	0	0	0	0	2.25	-	0.25	-	10.3-10.2
10 - 13	0.26	0	0.25	0	1.75	-	0.75	-	10.2-10.2
13 - 14	0.79	0	0.75	0	1.70	-	0.80	-	10.2-10.1
14 - 16	0.84	0.50	0.80	0.38	1.80	-	0.70	-	11.0-10.2
16 - 17	0.74	0.60	0.70	0.46	1.73	0.11	0.77	9.54	11.5-11.1
Total	13.2	12.7	12.5	9.65	1.73	0.11	9.77	9.54	

Metallurgical Balance

Product	Weight mL, g	Assays Au, mg/L, g/t	Distribution, % Au
+ 28 mesh	133	2.33	2.2
Day 1 Loaded Carbon	14.4	405	42.4
Day 2 Loaded Carbon	14.8	19.5	2.1
Day 4 Loaded Carbon	14.5	14.5	1.5
Day 7 Loaded Carbon	15.7	13.4	1.5
Day 14 Loaded Carbon	16.0	14.3	1.7
Day 17 Loaded Carbon	15.4	5.20	0.6
Barren Solution	3750	<0.002	0.1
Barren Wash	3760	<0.002	0.1
Residue	9676	0.68	47.8
Feed(calc)	9809	1.40	100.0

Test 12 Project: 4095 Date: Feb/22/91 Operator: JH

Purpose: To evaluate Au extraction by direct cyanidation.

Procedure: Approximately 10 kg of agglomerated sample was loaded into a plastic column 102 mm in diameter, to a height of 116 cm. A piece of steel mesh was placed at the bottom of the column and a piece of burlap on top to help disperse the solution. Approximately 5 L of 0.5 g/L NaCN solution was percolated through the column at a rate of 5 mL per minute. The pregnant solution was passed through carbon column where the Au in solution was removed. The reagents were replenished as required during the test. The loaded carbon was changed after 1, 2, 4, 7, 14, and 17 days and replaced with fresh carbon.

Feed: 10000 g Bankfield-2

Solution Volume: 5000 mL Pulp Density: 33 % Solids

Sol'n Composition: 0.5 g/L NaCN

pH Range: 10.0-11.0 Ca(OH)2

Reagent Consumption (kg/t of cyanide feed) NaCN: 1.75 CaO: 4.76

Time	Added, Grams				Residual		Consumed		pH
	Actual		Equivalent		Grams		Grams		
Days	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	
Agglomerate	5.00	55	4.75	41.8	-	-	-	-	
0 - 1	2.00	0	1.90	0	0.75	-	5.90	-	8.6-9.4
1 - 2	1.84	0	1.75	0	1.25	-	1.25	-	9.4-9.6
2 - 3	1.32	1.00	1.25	0.76	2.00	-	0.50	-	11.5-10.0
3 - 4	0.53	0	0.50	0	2.00	-	0.50	-	10.0-9.7
4 - 5	0.53	0.50	0.50	0.38	2.25	-	0.25	-	11.9-9.8
5 - 6	0.26	0.50	0.25	0.38	1.50	-	1.00	-	11.1-10.0
6 - 7	1.05	0	1.00	0	1.75	-	0.75	-	10.0-9.8
7 - 8	0.79	0.50	0.75	0.38	2.00	-	0.50	-	10.7-9.9
8 - 9	0.53	0.50	0.50	0.38	1.50	-	1.00	-	11.5-9.8
9 - 10	1.05	0.80	1.00	0.61	1.75	-	0.75	-	10.8-10.0
10 - 13	0.79	0.50	0.75	0.38	1.00	-	1.50	-	10.8-9.6
13 - 14	1.58	0.50	1.50	0.38	1.20	-	1.30	-	11.1-9.8
14 - 16	1.37	0.50	1.30	0.38	1.50	-	1.00	-	11.2-9.6
16 - 17	0.89	0.85	0.85	0.65	1.63	0	0.87	46.5	11.1-10.3
Total	19.5	61.2	18.6	46.5	1.63	0	17.1	46.5	

Metallurgical Balance

Product	Weight mL, g	Assays Au, mg/L, g/t	Distribution, % Au
Day 1 Loaded Carbon	14.6	313	36.1
Day 2 Loaded Carbon	14.5	69.0	7.9
Day 4 Loaded Carbon	15.6	49.8	6.1
Day 7 Loaded Carbon	15.6	23.0	2.8
Day 14 Loaded Carbon	16.7	17.2	2.3
Day 17 Loaded Carbon	16.7	5.20	0.7
Barren Solution	3540	<0.002	0.1
Barren Wash	3840	<0.002	0.1
Residue	9762	0.57	44.0
Feed(calc)	9762	1.30	100.0

LB#4095-Beaurox Mines Limited

Test#PP-1

Purpose: To recovery a gold bearing sulphide concentrate using column flotation and gravity concentration.

Procedure: A bulk sample was prepared by pulping it to approximately 65% solids and removing the +28 mesh material prior to storage in a 1500 L conditioner. The -28 mesh fraction was than lightly reground in a Denver ball mill before being further diluted to approximately 35% solids in a small 10 L conditioner. This comprised the feed to the first of two 6 inch column flotation cells. The first column was fed at a rate of 3.7 L/minute slurry. The rougher tail from the first column was used to feed the second column. Rougher concentrates from both 6 inch columns were combined to feed a 4 inch cleaner column cell. Tailing from the cleaner column was recirculated to the head of the flotation circuit while the cleaner concentrate was collected. Tailing from the second 6 inch column was passed through a Falcon separator. A Falcon concentrate and tailing were collected and assayed.

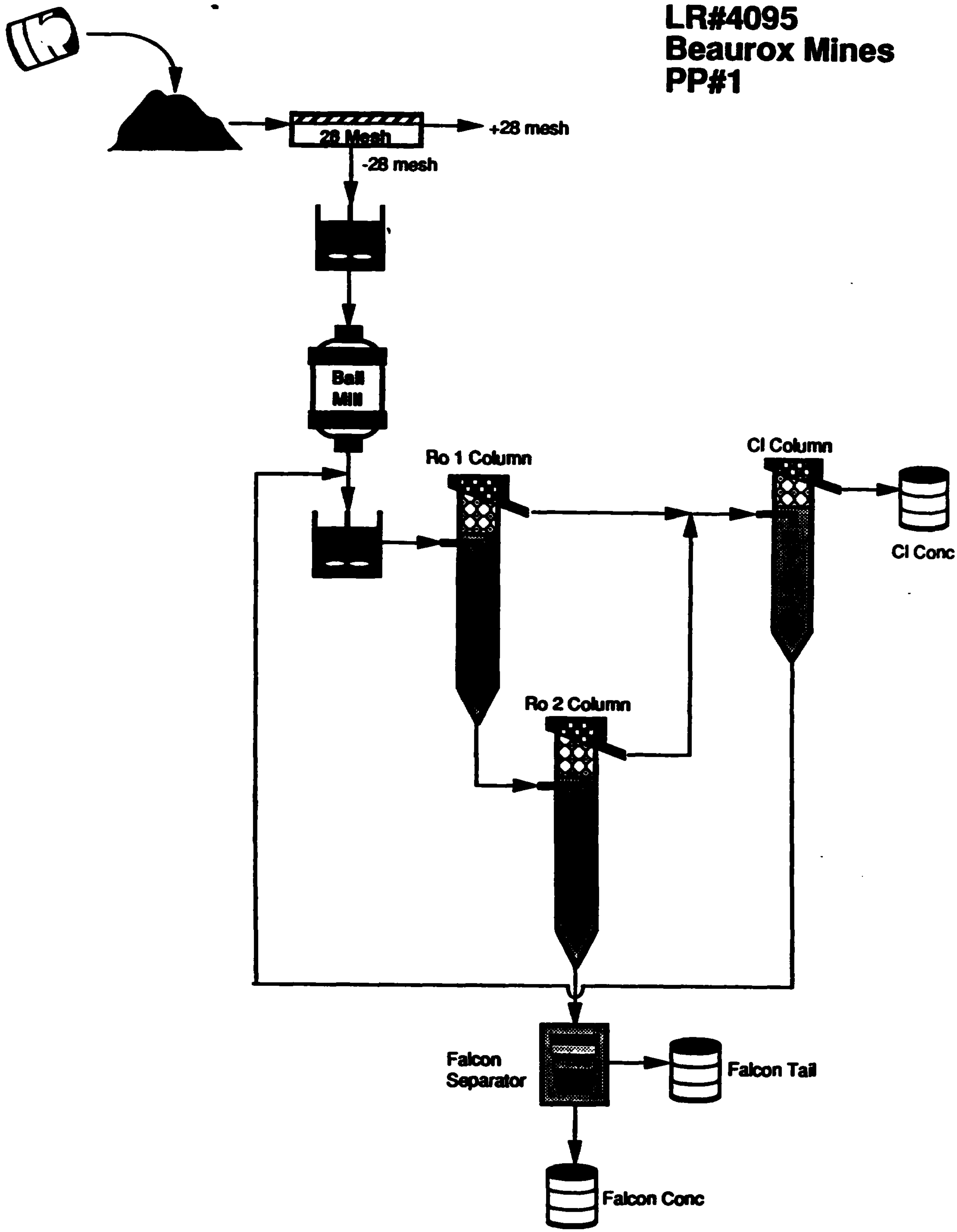
Sample: Bankfield Bulk Sample (-28 mesh)

Flowsheet: See next page.

Equipment List:
2 X 150 mm flotation columns, 57 and 72 litres in capacity
100 mm flotation column, 28 litres in capacity
Denver ball mill, 305 mm X 610 mm, 1.5 kW
Denver conditioner, 1200 mm X 1520 mm, 1500 L
760 mm diameter Sweco Vibro Energy Separator, 28 mesh deck
Falcon Separator, Model B-5

Results:

LR#4095
Beaurox Mines
PP#1



Reagents

Point of Addition	Reagent Name	Solution Strength %	Rate mL/min or drops/min	Rate g/t	Feed Rate t/h
Ball Mill Feed	Na2S	10	8.7	661	0.079
Flotation Conditioner	A350	2	10.5	159	
	R412	100	20.0	115	
	CuSO4	10	9.0	684	
	DF250	2	1.1	17	

Column Flotation Conditions

(i) Rougher 1 Column

(i) Column Cell Data

Diameter	150 mm
X-sectional Area	177 cm2
Total Height	272 cm
Total Volume	80 L

(ii) Operating Parameters

Gas	9.8 L/min
Wash	2.12 L/min
Feed Rate	3.01 L/min
Level	75 cm
PXD	41 mV
Operating Volume	72 L

(iii) Key Variables

Gas Velocity	0.92 cm/sec
Wash Velocity	0.20 cm/sec
Feed Velocity	0.28 cm/sec
NRT	27 minutes
Flow Bias	27 %
Gas Hold Up	16 %

Column Flotation Conditions (continued)

(ii) Rougher 2 Column

(i) Column Cell Data

Diameter	150 mm
X-sectional Area	177 cm2
Total Height	396 cm
Total Volume	72 L

(ii) Operating Parameters

Gas	6.7 L/min
Wash	1.06 L/min
Feed Rate	2.94 L/min
Level	30 cm
PXD	53 mV
Operating Volume	57 L

(iii) Key Variables

Gas Velocity	0.63 cm/sec
Wash Velocity	0.10 cm/sec
Feed Velocity	0.28 cm/sec
NRT	25 minutes
Flow Bias	26 %
Gas Hold Up	17 %

Assays

Stream	Product	Au, g/t	S, %
Ball Mill Discharge	BMD	1.55	
Rougher 1 Feed	R1F	1.63	3.55
Rougher 1 Conc	R1C	11.3	
Rougher 2 Conc	R2C	10.4	
Combined Rougher Conc	CRC	10.2	33.2
Cleaner Conc	CLC	11.7	
Rougher 1 Tailing	R1T	1.39	
Rougher 2 Tailing	R2T	1.13	1.69
Cleaner Tailing	CLT	7.30	
Falcon Conc	FLC	24.0	16.2
Falcon Tailing	FLT	1.08	1.66

Metallurgical Balance (2 product formula)

Product	Weight	Assay, g/t or %		Distribution, %	
	%	Au	S	Au	S
CLC	4.0	11.7		30.0	
R2T	96.0	1.13		70.0	
BMD(calc)	100.0	1.55		100.0	
BMD(assay)		1.55			
BMD	100.0	1.55		100.0	
CLT	1.4	7.30		6.6	
R1F(calc)	101.4	1.63		106.6	
R1F(assay)		1.63			
R1C	2.5	11.3		17.9	
R1T	99.0	1.39		88.7	
R1F(calc)	101.4	1.63		106.6	
R1F(assay)		1.63			
R2C	2.8	10.4		18.6	
R2T	96.2	1.13		70.1	
R1T(calc)	99.0	1.39		88.7	
R1T(assay)		1.39			
R1C	2.5	11.3		17.9	
R2C	2.8	10.4		18.6	
CRC(calc)	5.2	10.8		36.5	
CRC(assay)		10.2			
CLC	4.0	11.7		30.0	
CLT	1.4	7.30		6.6	
CRC(calc)	5.4	10.5		36.6	
CRC(assay)		10.2			
FLC	0.2	24.0	16.2	3.1	
FLT	96.0	1.08	1.66	67.0	
R2T(calc)	96.2	1.13	1.69	70.1	
R2T(assay)		1.13	1.69		

Overall Metallurgical Balance

Product	Weight %	Assay, g/t or % Au	Assay, g/t or % S	Distribution, % Au	Distribution, % S
CLC	4.0	11.7		30.0	
FLC	0.2	24.0		3.1	
FLT	96.0	1.06		67.0	
R1F(calc)	100.2	1.55		100.0	
R1F(assay)		1.55			

Screen Analyses

(i) Ball Mill Feed

Mesh	Weight g	% Weight Individual	% Weight Cumulative	% Weight Passing
65	0.2	0.1	0.1	99.9
100	1.9	1.0	1.1	98.9
150	18.4	9.6	10.7	89.3
200	36.9	19.3	30.1	69.9
270	32.8	17.2	47.2	52.8
400	20.1	10.5	57.8	42.2
-400	80.6	42.2	100.0	-
Total	190.9	100.0	-	-

(ii) Ball Mill Discharge

Mesh	Weight g	% Weight Individual	% Weight Cumulative	% Weight Passing
65	0.0	0.0	0.0	100.0
100	0.6	0.4	0.4	99.6
150	7.8	4.6	5.0	95.0
200	25.0	14.8	19.8	80.2
270	30.3	17.9	37.7	62.3
400	21.8	12.9	50.6	49.4
-400	83.4	49.4	100.0	-
Total	168.9	100.0	-	-

(iii) Cleaner Conc

Mesh	Weight g	% Weight Individual	% Weight Cumulative	% Weight Passing
65	0.0	0.0	0.0	100.0
100	0.1	0.1	0.1	99.9
150	0.7	0.3	0.4	99.6
200	5.3	2.2	2.6	97.4
270	34.5	14.4	17.0	83.0
400	77.2	32.2	49.1	50.9
-400	122.2	50.9	100.1	-
Total	240.0	100.1	-	-

Screen Analyses (continued)

(iv) Rougher 1 Conc

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.0	0.0	0.0	100.0
100	0.1	0.1	0.1	99.9
150	0.2	0.3	0.4	99.6
200	1.4	1.8	2.2	97.8
270	7.5	9.6	11.7	88.3
400	19.4	24.7	36.5	63.5
-400	49.8	63.5	100.0	-
Total	78.4	100.0	-	-

(v) Rougher 2 Conc

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.0	0.0	0.0	100.0
100	0.1	0.3	0.3	99.7
150	0.4	1.1	1.4	98.6
200	1.6	4.5	5.9	94.1
270	5.2	14.6	20.5	79.5
400	8.3	23.3	43.8	56.2
-400	20.0	56.2	100.0	-
Total	35.6	100.0	-	-

LB#4095-Beaurox Mines Limited

Test#PP-2

Purpose: To recovery a gold bearing sulphide concentrate using column flotation and gravity concentration.

Procedure: A bulk sample was prepared by pulping it to approximately 65% solids and removing the +28 mesh material prior to storage in a 1500 L conditioner. The -28 mesh fraction was than lightly reground in a Denver ball mill before being further diluted to approximately 35% solids in a small 10 L conditioner. This comprised the feed to the first of two 6 inch column flotation cells. The first column was fed at a rate of 3.7 L/minute slurry. The rougher tail from the first column was used to feed the second column. Rougher concentrates from both 6 inch columns were combined and collected for assay. Tailing from the second 6 inch column was passed through a Falcon separator. A Falcon concentrate and tailing were collected and assayed.

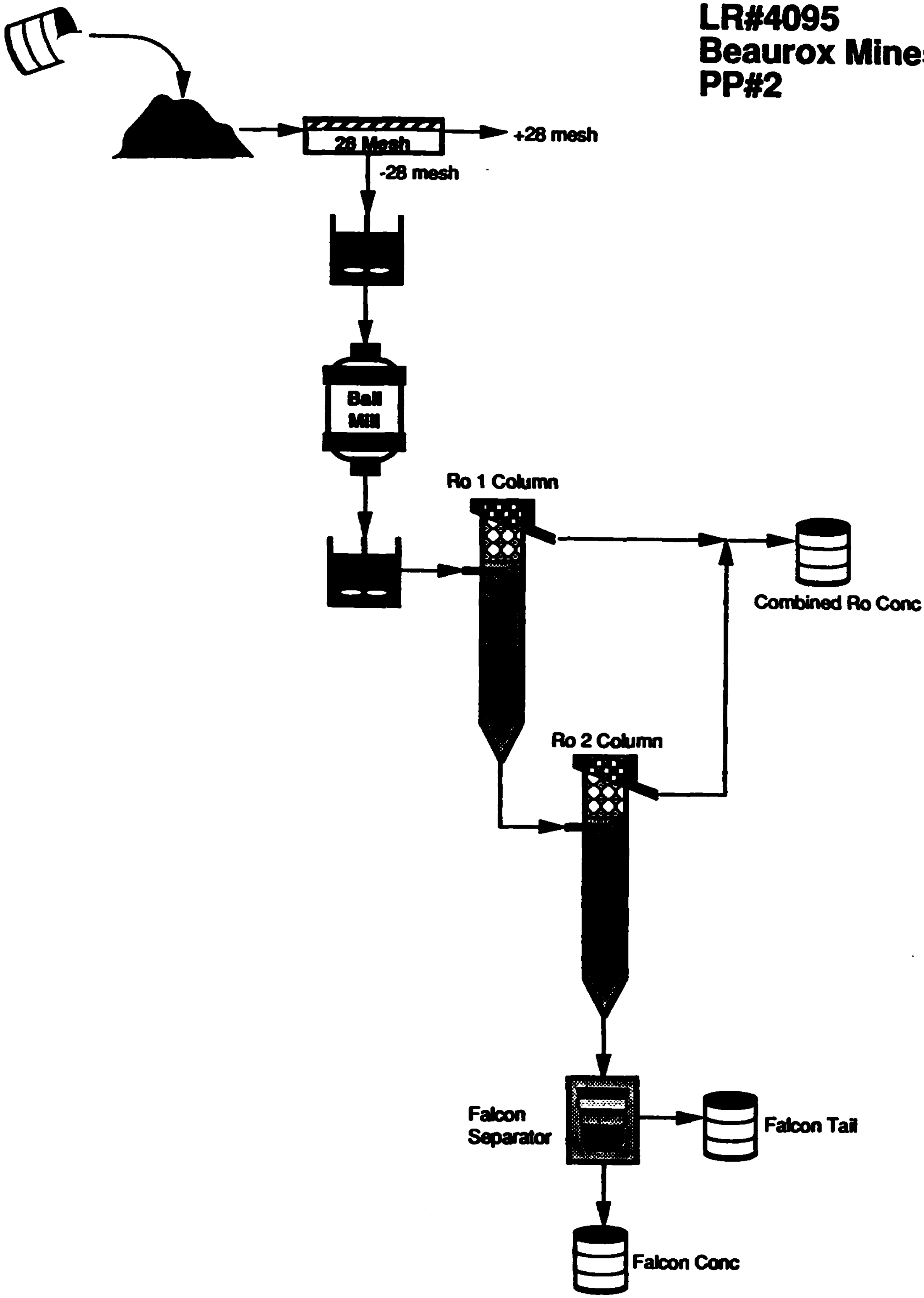
Sample: Bankfield Bulk Sample (-28 mesh)

Flowsheet: See next page.

Equipment List:
2 X 150 mm flotation columns, 57 and 72 litres in capacity
Denver ball mill, 305 mm X 610 mm, 1.5 kW
Denver conditioner, 1200 mm X 1520 mm, 1500 L
760 mm diameter Sweco Vibro Energy Separator, 28 mesh deck
Falcon Separator, Model B-5

Results:

LR#4095
Beaurox Mines
PP#2



Reagents

Point of Addition	Reagent Name	Solution Strength %	Rate mL/min or drops/min	Rate g/t	Feed Rate t/h
Ball Mill Feed	Na2S	10	7.3	644	0.068
Flotation Conditioner	A350	2	14.0	247	
	R412	100	23.3	156	
	CuSO4	10	7.0	618	
	DF250	2	1.5	26	

Column Flotation Conditions

(i) Rougher 1 Column

(i) Column Cell Data

Diameter	150 mm
X-sectional Area	177 cm2
Total Height	272 cm
Total Volume	80 L

(ii) Operating Parameters

Gas	12.0 L/min
Wash	2.12 L/min
Feed Rate	3.62 L/min
Level	53 cm
PXD	41 mV
Operating Volume	72 L

(iii) Key Variables

Gas Velocity	1.13 cm/sec
Wash Velocity	0.20 cm/sec
Feed Velocity	0.34 cm/sec
NRT	22 minutes
Flow Bias	32 %
Gas Hold Up	22 %

Column Flotation Conditions (continued)

(ii) Rougher 2 Column

(i) Column Cell Data

Diameter	150 mm
X-sectional Area	177 cm2
Total Height	396 cm
Total Volume	72 L

(ii) Operating Parameters

Gas	7.0 L/min
Wash	1.06 L/min
Feed Rate	3.52 L/min
Level	35 cm
PXD	53 mV
Operating Volume	57 L

(iii) Key Variables

Gas Velocity	0.66 cm/sec
Wash Velocity	0.10 cm/sec
Feed Velocity	0.33 cm/sec
NRT	20 minutes
Flow Bias	30 %
Gas Hold Up	16 %

Assays

Stream	Product	Au, g/t	S, %
Ball Mill Discharge	BMD	1.54	3.23
Rougher 1 Conc	R1C	9.49	
Rougher 2 Conc	R2C	7.95	
Combined Rougher Conc	CRC	8.97	27.1
Rougher 1 Tail	R1T	1.30	
Rougher 2 Tail	R2T	0.89	1.11
Falcon Conc	FLC	12.5	17.2
Falcon Tail	FLT	0.84	1.04

Metallurgical Balance (2 product formula)

Product	Weight	Assay, g/t or %		Distribution, %	
	%	Au	S	Au	S
R1C	2.9	9.49		18.1	
R1T	97.1	1.30		81.9	
BMD(calc)	100.0	1.54		100.0	
BMD(assay)		1.54			
R2C	5.6	7.95		29.1	
R2T	91.4	0.89		52.8	
R1T(calc)	97.1	1.30		81.9	
R1T(assay)		1.30			
R1C	2.9	9.49		18.1	
R2C	5.6	7.95		29.1	
CRC(calc)	8.6	8.48		47.2	
CRC(assay)		8.97			
CRC	8.6	8.48	27.1	47.2	69.6
R2T	91.4	0.89	1.11	52.8	30.4
BMD(calc)	100.0	1.54	3.34	100.0	100.0
BMD(assay)		1.54	3.23		
FLC	0.4	12.5	17.2	3.2	2.1
FLT	91.0	0.84	1.04	49.6	28.4
R2T(calc)	91.4	0.89	1.11	52.8	30.4
R2T(assay)		0.89	1.11		

Overall Metallurgical Balance

CRC	8.6	8.48	27.1	47.2	69.6
FLC	0.4	12.5	17.2	3.2	2.1
FLT	91.0	0.84	1.04	49.6	28.4
BMD(calc)	100.0	1.54	3.34	100.0	100.0
BMD(assay)		1.54	3.23		

Screen Analyses

(i) Ball Mill Feed

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.2	0.1	0.1	99.9
100	1.5	0.9	1.0	99.0
150	13.8	7.9	8.8	91.2
200	33.0	18.8	27.6	72.4
270	31.7	18.1	45.7	54.3
400	21.4	12.2	57.9	42.1
-400	74.0	42.1	100.0	-
Total	175.6	100.0	-	-

(ii) Rougher 1 Conc

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.0	0.0	0.0	100.0
100	0.0	0.0	0.0	100.0
150	0.3	0.7	0.7	99.3
200	1.2	2.9	3.6	96.4
270	4.4	10.5	14.1	85.9
400	8.9	21.3	35.4	64.6
-400	27.0	64.6	100.0	-
Total	41.8	100.0	-	-

(iii) Combined Rougher Conc

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.0	0.0	0.0	100.0
100	0.1	0.3	0.3	99.7
150	0.4	1.1	1.4	98.6
200	1.6	4.5	5.9	94.1
270	5.2	14.6	20.5	79.5
400	8.3	23.3	43.8	56.2
-400	20.0	56.2	100.0	-
Total	35.6	100.0	-	-

LB#4095-Beaurox Mines Limited

Test#PP-3

Purpose: To recovery a gold bearing sulphide concentrate using column flotation and gravity concentration.

Procedure: A bulk sample was prepared by pulping it to approximately 65% solids and removing the +28 mesh material prior to storage in a 1500 L conditioner. The -28 mesh fraction was than lightly reground in a Denver ball mill before being further diluted to approximately 35% solids in a small 10 L conditioner. This comprised the feed to the first of two 6 inch column flotation cells. The first column was fed at a rate of 3.7 L/minute slurry. The rougher tail from the first column was used to feed the second column. Rougher concentrates from both 6 inch columns were combined and collected for assay. Tailing from the second 6 inch column was passed through a Falcon separator. A Falcon concentrate and tailing were collected and assayed.

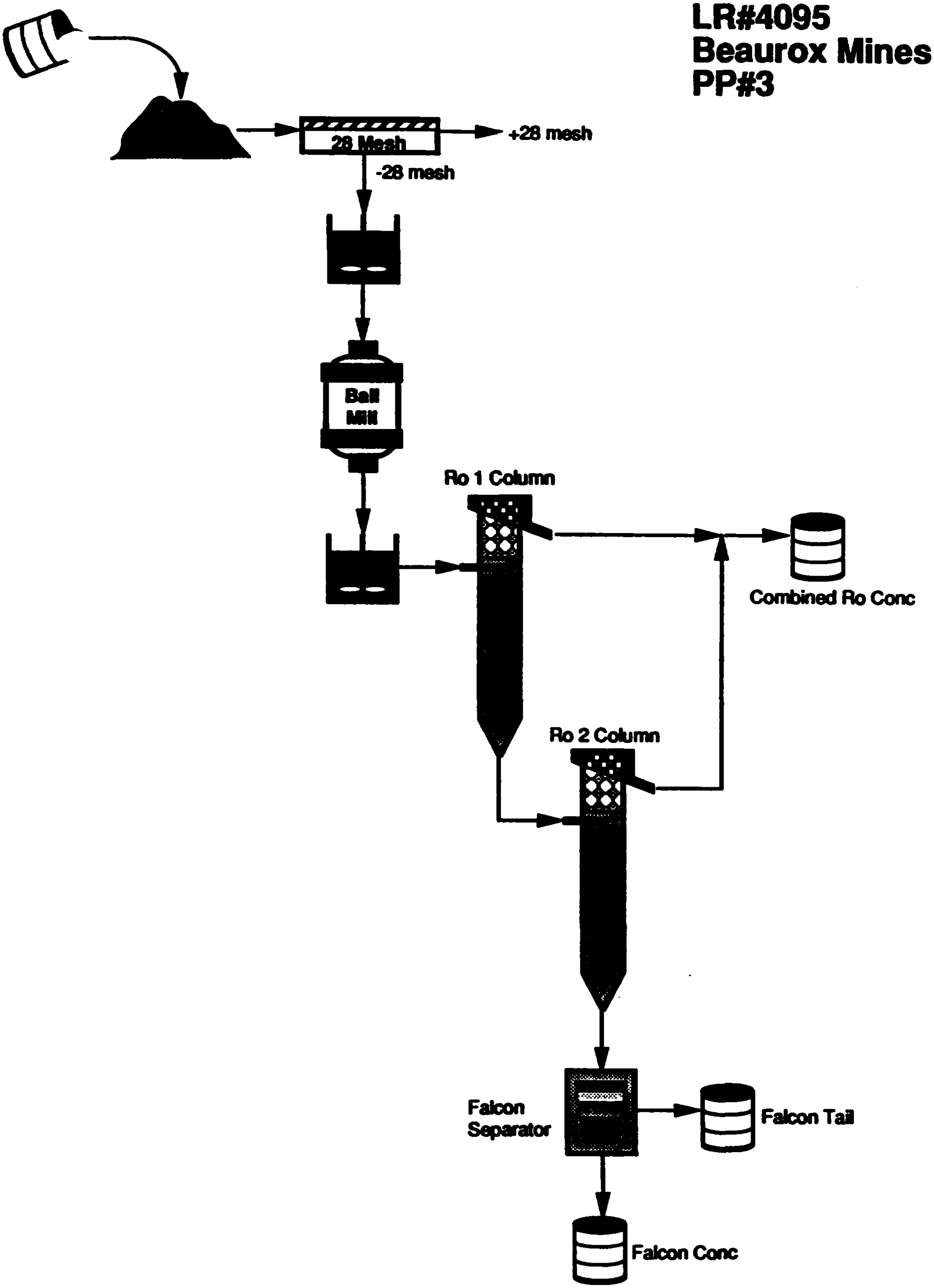
Sample: Little Long Lac Bulk Sample (-28 mesh)

Flowsheet: See next page.

Equipment List: 2 X 150 mm flotation columns, 57 and 72 litres in capacity
Denver ball mill, 305 mm X 610 mm, 1.5 kW
Denver conditioner, 1200 mm X 1520 mm, 1500 L
760 mm diameter Sweco Vibro Energy Separator, 28 mesh deck
Falcon Separator, Model B-5

Results:

LR#4095
Beaurox Mines
PP#3



Reagents

Point of Addition	Reagent Name	Solution Strength %	Rate mL/min or drops/min	Rate g/t	Feed Rate t/h
Ball Mill Feed	Na2S	10	8.1	600	0.081
Flotation Conditioner	A350	2	12.3	182	
	R412	100	20.0	113	
	CuSO4	10	8.0	593	
	DF250	2	2.2	33	

Column Flotation Conditions

(i) Rougher 1 Column

(i) Column Cell Data

Diameter	150 mm
X-sectional Area	177 cm2
Total Height	272 cm
Total Volume	80 L

(ii) Operating Parameters

Gas	7.9 L/min
Wash	2.12 L/min
Feed Rate	3.47 L/min
Level	45 cm
PXD	44 mV
Operating Volume	72 L

(iii) Key Variables

Gas Velocity	0.74 cm/sec
Wash Velocity	0.20 cm/sec
Feed Velocity	0.33 cm/sec
NRT	23 minutes
Flow Bias	31 %
Gas Hold Up	13 %

Column Flotation Conditions (continued)

(ii) Rougher 2 Column

(i) Column Cell Data

Diameter	150 mm
X-sectional Area	177 cm2
Total Height	396 cm
Total Volume	72 L

(ii) Operating Parameters

Gas	7.9 L/min
Wash	1.06 L/min
Feed Rate	3.40 L/min
Level	18 cm
PXD	56 mV
Operating Volume	57 L

(iii) Key Variables

Gas Velocity	0.74 cm/sec
Wash Velocity	0.10 cm/sec
Feed Velocity	0.32 cm/sec
NRT	21 minutes
Flow Bias	31 %
Gas Hold Up	13 %

Assays

Stream	Product	Au, g/t	S, %
Ball Mill Discharge	BMD	1.29	0.24
Rougher 1 Conc	R1C	12.1	6.64
Rougher 2 Conc	R2C	10.2	4.03
Combined Rougher Conc	CRC	12.9	5.93
Rougher 1 Tail	R1T	1.08	0.25
Rougher 2 Tail	R2T	1.04	0.16
Falcon Conc	FLC	30.3	9.43
Falcon Tail	FLT	0.98	0.14

Metallurgical Balance (2 product formula)

Product	Weight %	Assay, g/t or % Au	Assay, g/t or % S	Distribution, % Au	Distribution, % S
R1C	1.9	12.1	6.64	17.9	34.0
R1T	98.1	1.08	0.25	82.1	66.0
BMD(calc)	100.0	1.29	0.37	100.0	100.0
BMD(assay)		1.29	0.24		
R2C	0.4	10.2	4.03	3.4	6.6
R2T	97.7	1.04	0.16	78.7	59.4
R1T(calc)	98.1	1.08	0.18	82.1	66.0
R1T(assay)		1.08	0.25		
R1C	1.9	12.1	6.64	17.9	34.0
R2C	0.4	10.2	4.03	3.4	6.6
CRC(calc)	2.3	11.8	6.16	21.3	40.6
CRC(assay)		12.9	5.93		
CRC	2.3	11.8	6.16	21.3	40.6
R2T	97.7	1.04	0.16	78.7	59.4
BMD(calc)	100.0	1.29	0.30	100.0	100.0
BMD(assay)		1.29	0.24		
FLC	0.2	30.3	9.43	4.7	7.2
FLT	97.5	0.98	0.14	74.0	52.2
R2T(calc)	97.7	1.04	0.16	78.7	59.4
R2T(assay)		1.04	0.16		

Overall Metallurgical Balance

Product	Weight %	Assay Au	Assay S	Distribution Au	Distribution S
CRC	2.3	11.8	6.16	21.3	40.6
FLC	0.2	30.3	9.43	4.7	7.2
FLT	97.5	0.98	0.14	74.0	52.2
BMD(calc)	100.0	1.29	0.30	100.0	100.0
BMD(assay)		1.29	0.24		

Screen Analyses

(i) Ball Mill Feed

Mesh	Weight g	% Weight Individual	% Weight Cumulative	% Weight Passing
65	3.4	2.2	2.2	97.8
100	13.3	8.6	10.8	89.2
150	23.1	14.9	25.6	74.4
200	22.8	14.7	40.3	59.7
270	17.8	11.5	51.8	48.2
400	11.5	7.4	59.2	40.8
-400	63.3	40.8	100.0	-
Total	155.2	100.0	-	-

(ii) Ball Mill Discharge

Mesh	Weight g	% Weight Individual	% Weight Cumulative	% Weight Passing
65	0.8	0.7	0.7	99.3
100	4.9	4.1	4.8	95.2
150	11.5	9.6	14.4	85.6
200	15.5	13.0	27.4	72.6
270	16.0	13.4	40.8	59.2
400	12.2	10.2	51.0	49.0
-400	58.6	49.0	100.0	-
Total	119.5	100.0	-	-

Screen Analyses (continued)

(iii) Rougher 1 Conc

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.0	0.0	0.0	100.0
100	0.0	0.0	0.0	100.0
150	0.6	0.6	0.6	99.4
200	1.6	1.5	2.1	97.9
270	3.6	3.4	5.5	94.5
400	5.6	5.3	10.8	89.2
-400	94.3	89.2	100.0	-
Total	105.7	100.0	-	-

(iv) Rougher 2 Conc

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.0	0.0	0.0	100.0
100	0.0	0.0	0.0	100.0
150	1.2	1.1	1.1	98.9
200	2.7	2.6	3.7	96.3
270	5.1	4.8	8.5	91.5
400	6.7	6.3	14.9	85.1
-400	90.0	85.1	100.0	-
Total	105.7	100.0	-	-

LB#4095-Beaurox Mines Limited

Test#PP-4

Purpose: To recovery a gold bearing sulphide concentrate using column flotation and gravity concentration.

Procedure: A bulk sample was prepared by pulping it to approximately 65% solids and removing the +28 mesh material prior to storage in a 1500 L conditioner. The -28 mesh fraction was than lightly reground in a Denver ball mill before being further diluted to approximately 35% solids in a small 10 L conditioner. This comprised the feed to the first of two 6 inch column flotation cells. The first column was fed at a rate of 3.7 L/minute slurry. The rougher tail from the first column was used to feed the second column. Rougher concentrates from both 6 inch columns were combined and collected for assay. Tailing from the second 6 inch column was passed through a Falcon separator. A Falcon concentrate and tailing were collected and assayed.

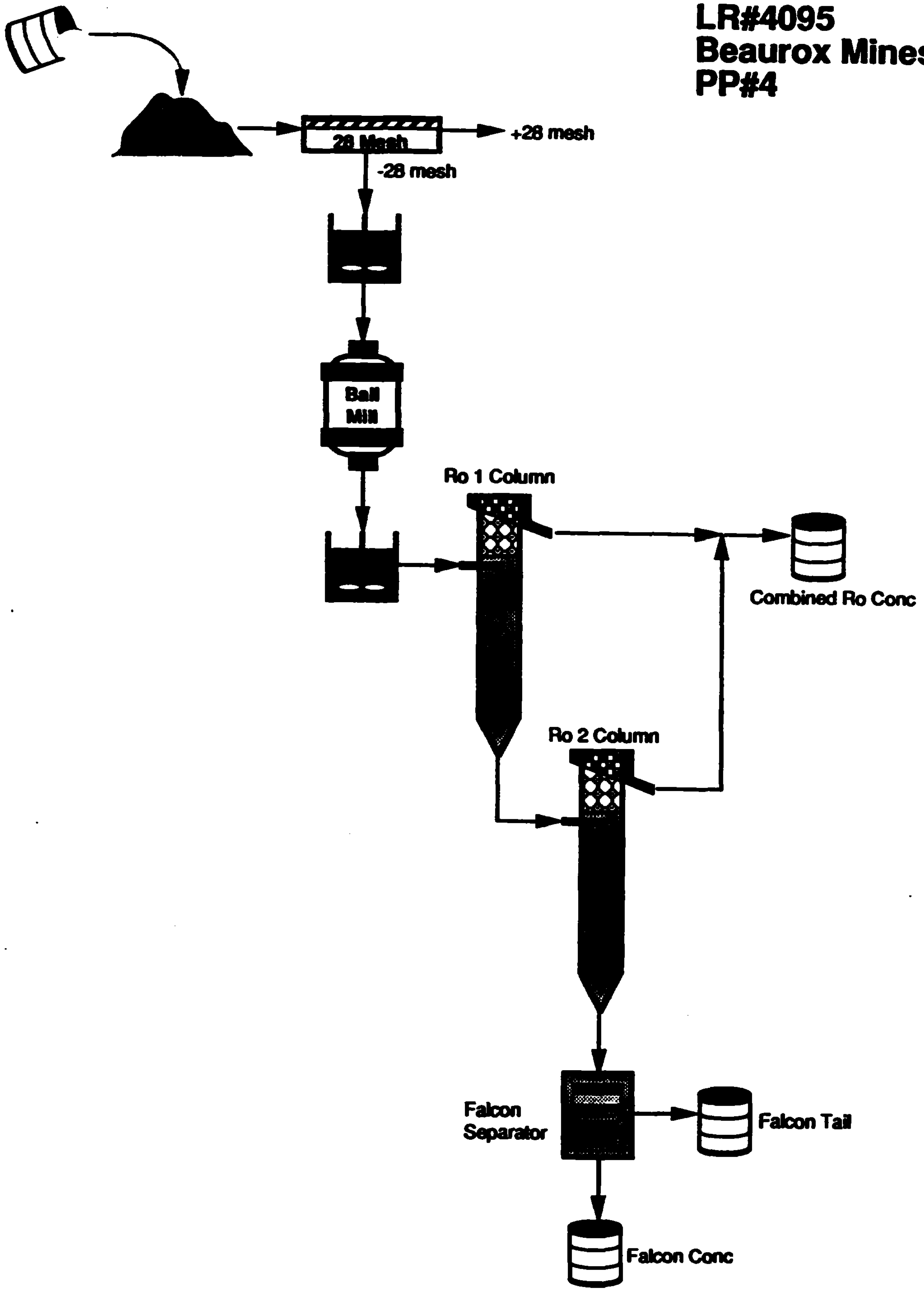
Sample: Little Long Lac Bulk Sample (-28 mesh)

Flowsheet: See next page.

Equipment List: 2 X 150 mm flotation columns, 57 and 72 litres in capacity
Denver ball mill, 305 mm X 610 mm, 1.5 kW
Denver conditioner, 1200 mm X 1520 mm, 1500 L
760 mm diameter Sweco Vibro Energy Separator, 28 mesh deck
Falcon Separator, Model B-5

Results:

LR#4095
Beaurox Mines
PP#4



Reagents

Point of Addition	Reagent Name	Solution Strength %	Rate mL/min or drops/min	Rate g/t	Feed Rate t/h
Ball Mill Feed	Na2S	10	8.3	778	0.064
Flotation Conditioner	A350	2	13.1	246	
	R412	100	21.0	150	
	CuSO4	10	6.5	609	
	DF250	2	2.6	49	

Column Flotation Conditions

(i) Rougher 1 Column

(i) Column Cell Data

Diameter	150 mm
X-sectional Area	177 cm2
Total Height	272 cm
Total Volume	80 L

(ii) Operating Parameters

Gas	7.9 L/min
Wash	2.12 L/min
Feed Rate	2.64 L/min
Level	45 cm
PXD	44 mV
Operating Volume	72 L

(iii) Key Variables

Gas Velocity	0.74 cm/sec
Wash Velocity	0.20 cm/sec
Feed Velocity	0.25 cm/sec
NRT	30 minutes
Flow Bias	23 %
Gas Hold Up	17 %

Column Flotation Conditions (continued)

(ii) Rougher 2 Column

(i) Column Cell Data

Diameter	150 mm
X-sectional Area	177 cm2
Total Height	396 cm
Total Volume	72 L

(ii) Operating Parameters

Gas	7.9 L/min
Wash	1.06 L/min
Feed Rate	2.62 L/min
Level	18 cm
PXD	56 mV
Operating Volume	57 L

(iii) Key Variables

Gas Velocity	0.74 cm/sec
Wash Velocity	0.10 cm/sec
Feed Velocity	0.25 cm/sec
NRT	28 minutes
Flow Bias	23 %
Gas Hold Up	17 %

Assays

Stream	Product	Au, g/t	S, %
Ball Mill Discharge	BMD	1.14	0.25
Rougher 1 Conc	R1C	17.5	10.0
Rougher 2 Conc	R2C	9.73	4.29
Combined Rougher Conc	CRC	10.1	4.58
Rougher 1 Tail	R1T	1.00	0.20
Rougher 2 Tail	R2T	0.92	0.15
Falcon Conc	FLC	20.4	1.04
Falcon Tail	FLT	0.84	0.15

Metallurgical Balance (2 product formula)

Product	Weight	Assay, g/t or %		Distribution, %	
	%	Au	S	Au	S
R1C	0.8	17.5	10.0	13.0	30.0
R1T	99.2	1.00	0.20	87.0	70.0
BMD(calc)	100.0	1.14	0.28	100.0	100.0
BMD(assay)		1.14	0.24		
R2C	0.9	9.73	4.29	7.7	14.5
R2T	98.3	0.92	0.15	79.3	55.5
R1T(calc)	99.2	1.00	0.19	87.0	70.0
R1T(assay)		1.00	0.20		
R1C	0.8	17.5	10.0	13.0	30.0
R2C	0.9	9.73	4.29	7.7	14.5
CRC(calc)	1.7	13.5	7.06	20.7	44.5
CRC(assay)		10.1	4.58		
CRC	1.7	13.5	7.06	20.7	44.5
R2T	98.3	0.92	0.15	79.3	55.5
BMD(calc)	100.0	1.14	0.27	100.0	100.0
BMD(assay)		1.14	0.25		
FLC	0.4	20.4	1.04	7.2	1.6
FLT	97.9	0.84	0.15	72.1	53.9
R2T(calc)	98.3	0.92	0.15	79.3	55.5
R2T(assay)		0.92	0.15		

Overall Metallurgical Balance

CRC	1.7	13.5	7.06	20.7	44.5
FLC	0.4	20.4	1.04	7.2	1.6
FLT	97.9	0.84	0.15	72.1	53.9
BMD(calc)	100.0	1.14	0.27	100.0	100.0
BMD(assay)		1.14	0.25		

Screen Analyses

(i) Ball Mill Feed

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	4.2	2.5	2.5	97.5
100	13.6	8.1	10.6	89.4
150	24.4	14.5	25.1	74.9
200	23.4	13.9	39.0	61.0
270	20.3	12.1	51.1	48.9
400	15.0	8.9	60.0	40.0
-400	67.3	40.0	100.0	-
Total	168.2	100.0	-	-

(ii) Ball Mill Discharge

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.8	0.5	0.5	99.5
100	6.1	4.0	4.5	95.5
150	17.2	11.3	15.8	84.2
200	19.9	13.1	28.9	71.1
270	19.9	13.1	42.0	58.0
400	13.0	8.5	50.5	49.5
-400	75.3	49.5	100.0	-
Total	152.2	100.0	-	-

Screen Analyses (continued)

(iii) Rougher 1 Conc

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.0	0.0	0.0	100.0
100	0.0	0.0	0.0	100.0
150	0.7	1.9	1.9	98.1
200	1.6	4.3	6.2	93.8
270	3.6	9.7	15.9	84.1
400	4.5	12.1	28.0	72.0
-400	26.7	72.0	100.0	-
Total	37.1	100.0	-	-

(iv) Rougher 2 Conc

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.0	0.0	0.0	100.0
100	0.0	0.0	0.0	100.0
150	1.1	1.2	1.2	98.8
200	2.5	2.8	4.1	95.9
270	5.3	6.0	10.0	90.0
400	6.4	7.2	17.3	82.7
-400	73.3	82.7	100.0	-
Total	88.6	100.0	-	-

LR#4095-Beaurox Mines Limited

Test#PP-5

Purpose: To recovery a gold bearing sulphide concentrate using gravity concentration.

Procedure: A bulk sample was prepared by pulping it to approximately 65% solids and removing the +28 mesh material prior to storage in a 1500 L conditioner. The -28 mesh fraction was diluted to 35% solids and simultaneously pumped to the head of the spiral. The spiral concentrate was collected for assay purposes while the spiral midds were combined with fresh feed and pumped back to the head of the spiral. The spiral tail was passed through a Falcon separator where a Falcon concentrate and tail were collected and assayed.

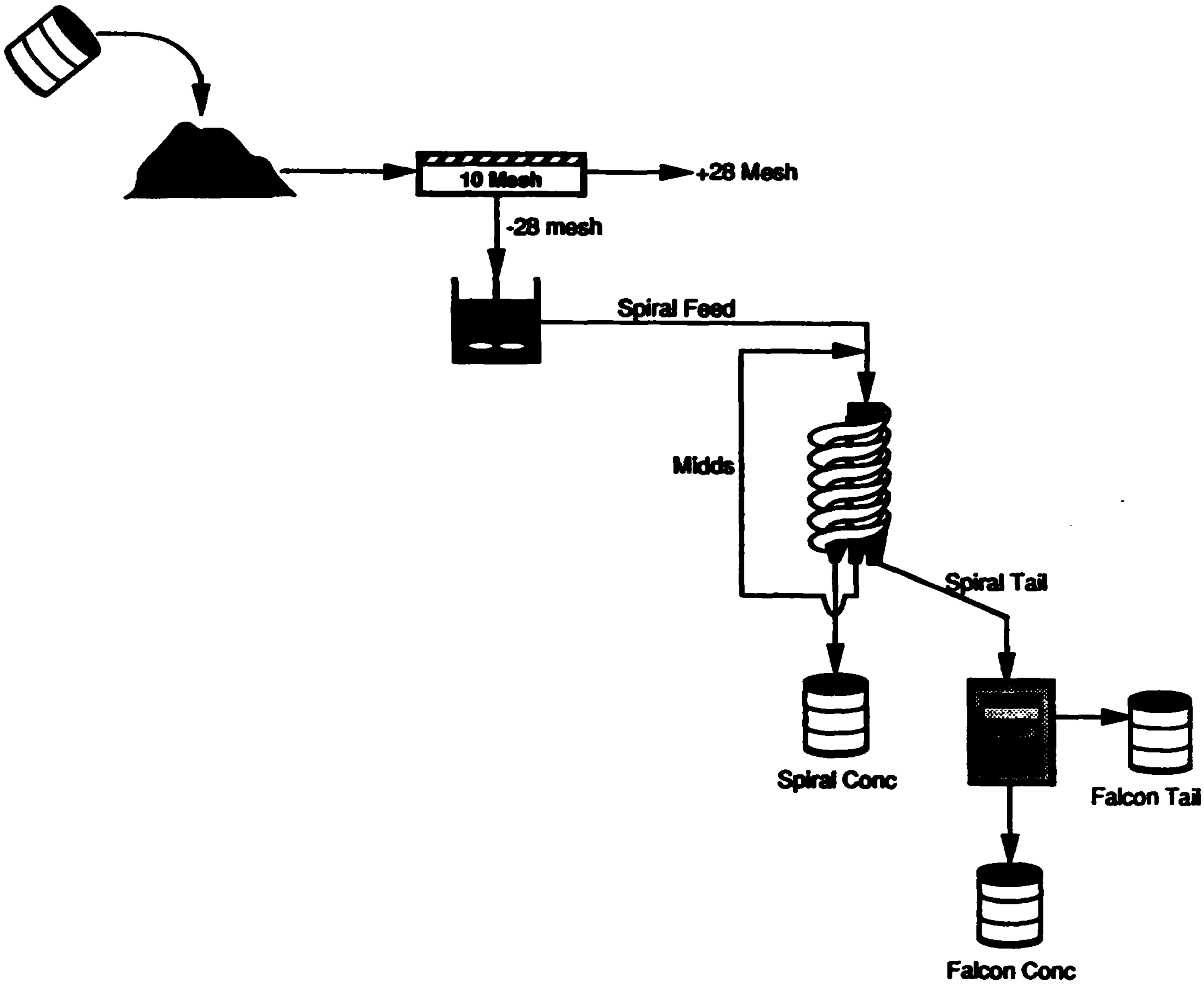
Sample: Bankfield Bulk Sample (-28 mesh)

Flowsheet: See next page.

Equipment List: Denver conditioner, 1200 mm X 1520 mm, 1500 L
Falcon Separator, Model B-5
Reichert Mark VII Spiral

Results:

LR#4095
Beaurox Mines
PP-5



Aassys

Stream	Product	Au, g/t
Spiral Feed	SPF	1.45
Spiral Conc	SPC	4.81
Spiral Tailing	SPT	1.20
Falcon Conc	FLC	70.5
Falcon Tailing	FLT	1.05

Metallurgical Balance (two product formula)

Product	Weight %	Assay, g/t Au	Distribution Au
SPC	6.9	4.81	23.0
SPT	93.1	1.20	77.0
SPF(calc)	100.0	1.45	100.0
SPC(assay)		1.45	
FLC	0.2	70.5	9.7
FLT	92.9	1.05	67.3
SPT(calc)	93.1	1.20	77.0
SPT(assay)		1.20	

Overall Metallurgical Balance

SPC	6.9	4.81	23.0
FLC	0.2	70.5	9.7
FLT	92.9	1.05	67.3
SFD(calc)	100.0	1.45	100.0
SFD(assay)		1.45	

LB#4095-Beaurox Mines Limited

Test#PP-6

Purpose: To recovery a gold bearing sulphide concentrate using gravity concentration.

Procedure: A bulk sample was prepared by pulping it to approximately 65% solids and removing the +28 mesh material prior to storage in a 1500 L conditioner. The -28 mesh fraction was diluted to 35% solids and simultaneously pumped to the head of the spiral. The spiral concentrate was collected for assay purposes while the spiral midds were combined with fresh feed and pumped back to the head of the spiral. The spiral tail was passed through a Falcon separator where a Falcon concentrate and tail were collected and assayed.

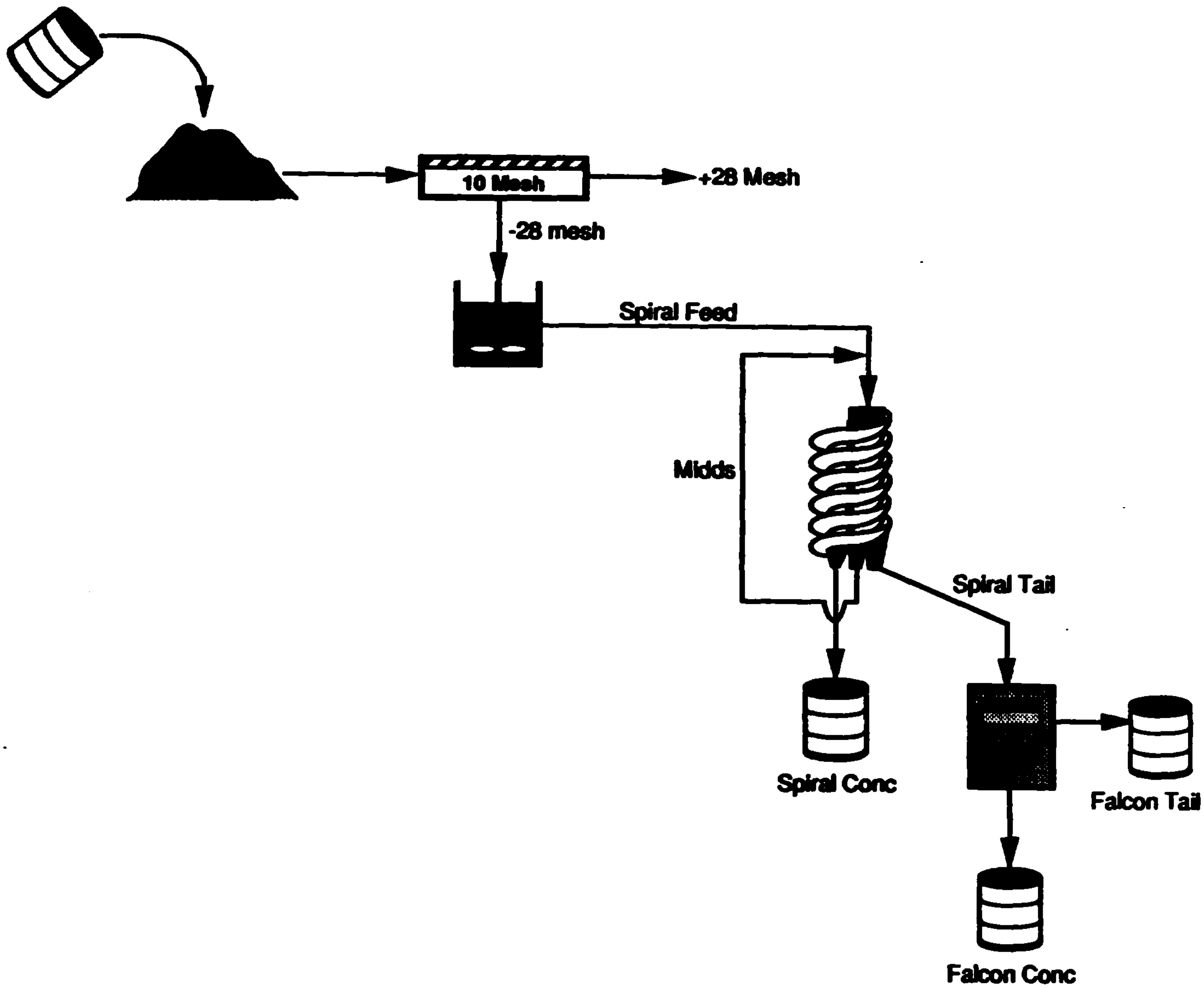
Sample: Little Long Lac Bulk Sample (-28 mesh)

Flowsheet: See next page.

Equipment List: Denver conditioner, 1200 mm X 1520 mm, 1500 L
Falcon Separator, Model B-5
Reichert Mark VII Spiral

Results:

LR#4095
Beaurox Mines
PP-6



Aassys

Stream	Product	Au, g/t
Spiral Feed	SPF	1.19
Spiral Conc	SPC	6.57
Spiral Tailing	SPT	1.01
Falcon Conc	FLC	40.2
Falcon Tailing	FLT	0.85

Metallurgical Balance (two product formula)

Product	Weight %	Assay, g/t Au	Distribution Au
SPC	3.2	6.57	17.9
SPT	96.8	1.01	82.1
SPF(calc)	100.0	1.19	100.0
SPC(assay)		1.19	
FLC	0.4	40.2	13.5
FLT	96.4	0.85	68.6
SPT(calc)	96.8	1.01	82.1
SPT(assay)		1.01	

Overall Metallurgical Balance

SPC	3.2	6.57	17.9
FLC	0.4	40.2	13.5
FLT	96.4	0.85	68.6
SFD(calc)	100.0	1.19	100.0
SFD(assay)		1.19	

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THE RECOVERY OF GOLD

from low grade tailing samples

submitted by

BEAUROX MINES LIMITED

Progress Report No. 3

Project No. L.R. 4095

NOTE:

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LAKEFIELD RESEARCH
A DIVISION OF FALCONBRIDGE LIMITED
March 15, 1991

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INTRODUCTION

This report contains the results of testwork conducted on four different Beaurox Mines tailing samples as required by Mr. D. Malouf of Malouf Consulting. The purpose of the testwork was to investigate the recovery of gold by means of cyanidation, flotation and gravity concentration techniques.

The results were forwarded to Mr. Malouf as the testwork proceeded.

LAKEFIELD RESEARCH

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SUMMARY

1. Head Analyses

Representative head samples were cut from each individual tailing sample and assayed for Au, Fe and S(total). The results are summarized as follows:

Table No. 1: Head Assays

	Bankfield	Bankfield-2	Tashota	Little Long Lac
Au, g/t	2.90	1.29	3.03	1.60
Fe, %	6.33	6.40	10.1	3.80
S(total), %	2.28	2.26	2.51	0.19

Size fraction analyses for Au and S(total) were also performed on the same head samples. Table 2 summarizes these results.

Table 2: Size Fraction Analyses

(i) Bankfield

Mesh	Weight		Assays, g/t or %		Distribution, %	
Tyler	g	%	Au	S	Au	S
65	131.5	12.3	1.75	1.53	5.7	8.3
200	306.5	28.7	2.69	0.83	20.4	10.5
400	211.1	19.8	3.23	3.04	16.9	26.4
-400	417.3	39.1	5.52	3.19	57.0	54.8
Feed(calc)	1066.4	100.0	3.79	2.28	100.0	100.0

(ii) Tashota

Mesh	Weight		Assays, g/t or %		Distribution, %	
Tyler	g	%	Au	S	Au	S
65	54.4	5.0	6.04	2.84	10.6	6.3
200	388.5	35.4	4.52	1.65	56.6	26.1
400	248.2	22.6	1.65	2.45	13.2	24.8
-400	405.2	37.0	1.50	2.59	19.6	42.8
Feed(calc)	1096.3	100.0	2.83	2.24	100.0	100.0

(iii) Little Long Lac

Mesh	Weight		Assays, g/t or %		Distribution, %	
Tyler	g	%	Au	S	Au	S
65	27.6	2.6	3.45	0.19	6.5	3.1
200	374.4	35.8	1.13	0.07	28.8	15.4
400	245.0	23.4	1.26	0.21	21.0	30.2
-400	397.8	38.1	1.62	0.22	43.8	51.4
Feed(calc)	1044.8	100.0	1.41	0.16	100.0	100.0

(iv) Bankfield-2

Mesh	Weight		Assays, g/t or %		Distribution, %	
Tyler	g	%	Au	S	Au	S
65	3.9	0.7	1.73	1.54	1.0	0.5
200	130.4	24.4	1.02	0.39	19.3	4.5
400	142.2	26.6	1.46	2.19	30.1	27.8
-400	257.7	48.2	1.33	2.92	49.7	67.1
Feed(calc)	534.2	100.0	1.29	2.10	100.0	100.0

A 24 element ICP scan was also conducted and these results are summarized in Table 3.

Table 3: Semi-Quantitative Analyses

	Bankfield	Tashota	Little Long Lac	Bankfield-2
As, %	1.39	0.004	0.12	0.85
Ba, %	0.04	0.02	0.05	0.04
Be, %	<0.0001	<0.0001	0.0001	0.0001
Ca, %	3.25	2.73	2.84	2.97
Cd, %	<0.003	<0.001	<0.0005	0.0007
Co, %	0.003	0.007	0.003	0.003
Cr, %	0.01	0.009	0.004	0.01
Cu, %	0.008	0.03	0.008	0.008
Fe, %	6.39	9.57	4.04	3.27
Mg, %	1.34	2.12	1.57	1.46
Mn, %	0.05	0.08	0.06	0.06
Mo, %	<0.01	<0.008	<0.01	<0.01
Na, %	3.24	1.21	1.71	3.34
Ni, %	0.006	0.006	0.006	0.006
P, %	0.04	0.03	0.05	0.04
Pb, %	<0.02	<0.01	<0.02	<0.01
S, %	2.28	2.15	0.22	2.01
Sb, %	0.001	<0.001	0.008	0.001
Se, %	<0.005	<0.005	<0.005	<0.0005
Sn, %	<0.002	<0.002	<0.002	<0.002
Te, %	<0.001	<0.001	<0.001	<0.001
Th, %	0.001	0.003	<0.001	<0.001
U, %	<0.001	<0.001	<0.001	<0.001
Zn, %	0.005	0.01	0.009	0.007

2. Bench Scale Testwork

2.1 Gravity Concentration

A series of tests were conducted to investigate the recovery of gold by gravity concentration. 2 kg charges were passed over an 1/8 Wilfley shaking table with the table concentrate being upgraded on a Mozley separator. The results are summarized in Table 4.

Table No. 4: Summary of Gravity Concentration Testwork

Test #	Sample	Product	Weight %	Assay, g/t, % Au	S	Distribution, % Au	S
4	Tashota	Mozley Conc	0.1	141		2.9	
		Mozley Tail	2.7	5.44		4.6	
		+ 28 mesh Table Conc	1.1	1.61		0.5	
		Table Conc(calc)	3.9	6.70	20.5	8.0	31.4
		Table Tail	96.2	3.08	1.79	92.0	68.6
		Feed(calc)	100.0	3.22	2.51	100.0	100.0
5	Bankfield	Mozley Conc	0.1	297		8.1	
		Mozley Tail	3.8	7.67		10.3	
		+ 28 mesh Table Conc	1.7	1.00		0.6	
		Table Conc(calc)	5.5	9.66	10.7	19.0	25.8
		Table Tail	94.5	2.40	1.79	81.0	74.2
		Feed(calc)	100.0	2.80	2.28	100.0	100.0
6	Little Long Lac	Mozley Conc	0.1	116		5.5	
		Mozley Tail	5.2	3.45		14.8	
		+ 28 mesh Table Conc	0.1	1.12		0.1	
		Table Conc(calc)	5.4	4.59	1.24	20.4	35.3
		Table Tail	94.6	1.03	0.13	79.6	64.7
		Feed(calc)	100.0	1.22	0.19	100.0	100.0

This type of gravity concentration was not successful since only an average of 16% of the gold and 31% of the sulphur was recovered by the Wilfley table.

Further gravity concentration tests were performed using a Falcon concentrator. Table 5 summarizes these results.

Table 5: Summary of Falcon Concentrator Results

Test #	Sample	Product	Weight %	Assay, g/t, % Au	S	Distribution, % Au	S
13	Tashota	+ 28 mesh	3.9	1.53	4.34	1.9	8.2
		Falcon Conc	7.9	9.69	2.55	23.5	9.6
		Falcon Tail	88.2	2.75	1.96	74.7	82.3
		Feed(calc)	100.0	3.25	2.10	100.0	100.0
14	Little Long Lac	+ 28 mesh	1.0	4.57	0.38	3.3	2.2
		Falcon Conc	5.0	6.57	0.59	23.6	16.9
		Falcon Tail	94.0	1.08	0.15	73.1	80.9
		Feed(calc)	100.0	1.39	0.17	100.0	100.0

As seen in Table 5, an average of 24% of the gold and 13% of the sulphur was recovered in the Falcon concentrate. The higher recovery of gold over sulphur indicates that free gold displaced sulphide bearing minerals during the operation of the Falcon concentrator until the centrifugal bowl became full (~750 g).

2.2 Direct Cyanidation

Representative 1 kg charges were prepared from the gold tailing samples for cyanidation testing. All samples were leached for 120 hours at 50% solids using 1.0 g/L NaCN and pH 10.5 - 11.0. Aliquots were removed every 24 hours in order to determine the rate of extraction of the gold. The results are summarized as follows.

Table 6: Summary of Direct Cyanidation Results

Test #	Sample	Reagent Consumption kg/t		Au Extraction, %					Residue, g/t Au	Feed(calc), g/t Au
		NaCN	CaO	24h	48h	72h	96h	120h		
1	Tashota	3.28	10.4	68	78	78	78	78.3	0.69	3.30
2	Bankfield	0.91	4.10	79	79	79	79	79.0	0.62	3.24
3	Little Long Lac	0.51	0.79	55	55	55	55	55.1	0.67	1.53

Maximum extraction of gold was reached after 48 hours of leaching.

2.3 Direct Cyanidation with Lead Nitrate

A series of tests was performed to investigate the effect of lead nitrate addition on the cyanidation response of the tailing samples. All tests involved the use of 1 kg charges diluted to 50% solids. A dosage of 500 g/t lead nitrate was added in each test. Cyanidation conditions and metallurgical results are summarized in Table 7.

Table 7: Summary of Cyanidation Conditions and Results

Standard Conditions: 1 kg charge
72 hour retention time
pH 10.5 - 11.0
0.5 g/L NaCN
500 g/t $Pb(NO_3)_2$

Test #	Sample	Reagent Consumption kg/t		Au Extraction, % 72h	Residue, g/t Au	Feed(calc), g/t Au
		NaCN	CaO			
7	Tashota	1.47	9.67	69.4	0.78	2.94
8	Bankfield-2	0.37	2.03	56.4	0.59	1.35
9	Little Long Lac	0.12	0.89	49.1	0.72	1.43

In comparison to earlier cyanidation testwork, the addition of the lead nitrate did help the cyanide consumption. Table 8 provides a comparison of the different sets of testwork. The Bankfield and Bankfield-2 test results are not included since they are not representative of each other.

Table 8: Comparison of Cyanidation Testwork

Test #	Sample	Reagent Consumption kg/t		Au Extraction, %	Residue, g/t Au	Feed(calc), g/t Au
		NaCN	CaO			
1	Tashota	3.28	10.4	78.3	0.69	3.30
2		1.47	9.67	69.4	0.78	2.94
3	Little Long Lac	0.51	0.79	55.1	0.67	1.53
9		0.12	0.89	49.1	0.72	1.43

As seen in Table 8, there was an average decrease of approximately 7% in the overall extraction of gold. This decrease may be accounted for by the lower calculated feed assays.

2.4 Heap Leach Simulation

The feed for the column leach testwork was agglomerated with 5 kg/t of Portland No. 2 cement, the amount of lime consumed in bench scale cyanidation tests and approximately one half of the amount of sodium cyanide consumed in the preliminary bottle roll tests.

The column leach apparatus was set up using 15 cm diameter plexiglass conduit of about 1.5 metres in height. The bottoms of the columns were fitted with a steel mesh to retain the solids. The columns were leached with flowrates of about 5 mL per minute (15 L/h/m^2). The pregnant solution discharge from the column was passed through a cartridge containing 15 grams of Calgon GRC-22 pre-attritioned carbon. Flowrates were monitored daily and adjusted as required to maintain a uniform flow. The loaded carbons were changed periodically to monitor gold extraction. Barren solutions were regularly sampled to determine NaCN and CaO concentrations. Table 9 summarizes the results of the column leach tests.

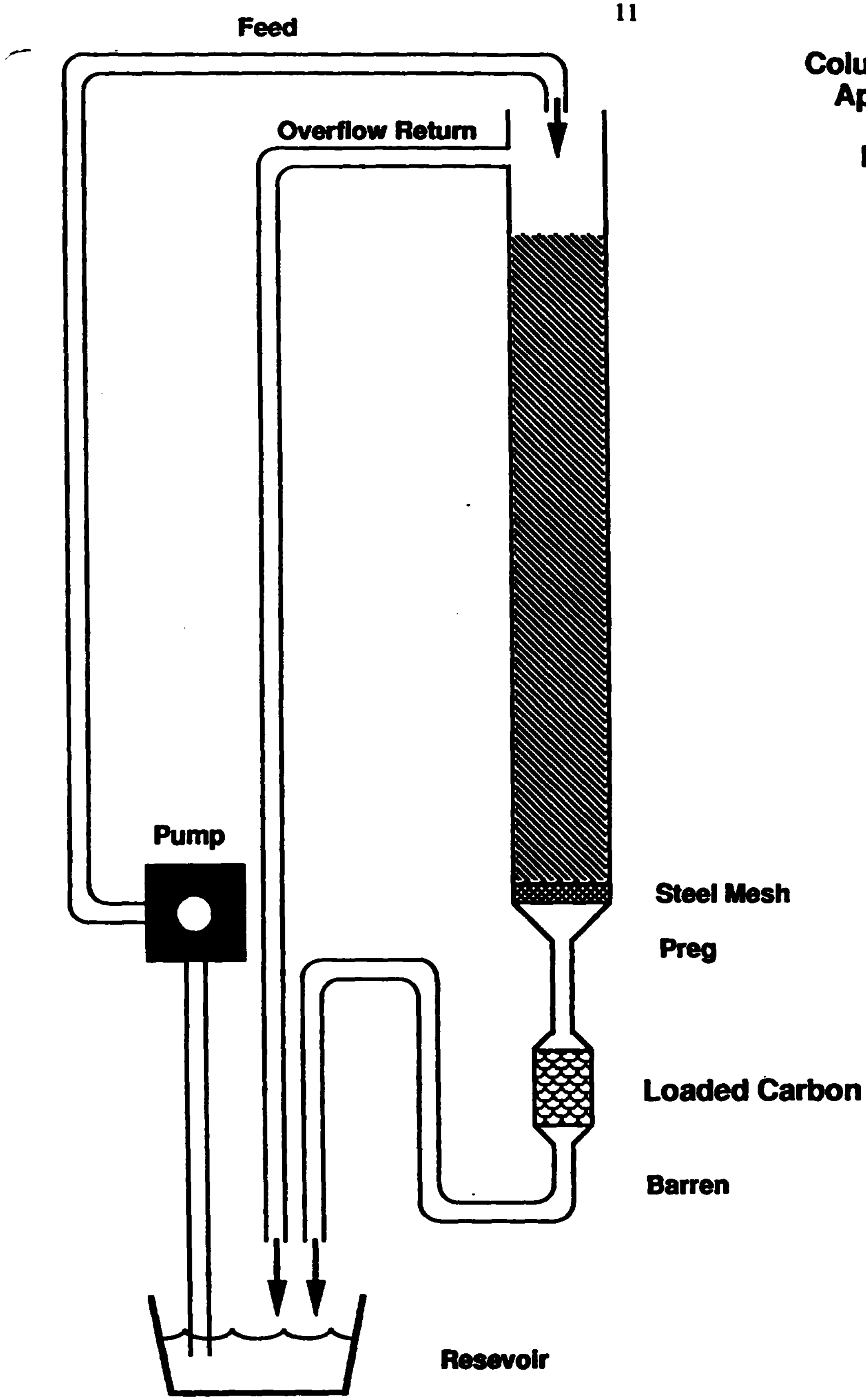
Table 9: Summary of Column Leach Results

Standard Conditions: 0.5 g/L NaCN
pH 10.0 - 11.0
carbon changes on days 1, 2, 4, 7, 14 and 21
~5 mL/min flowrate
5 kg/t Portland No. 2 cement

Test #	Sample	Reagent Consumption kg/t		Au Extraction, %	Residue, g/t Au	Feed(calc), g/t Au
		NaCN	CaO			
10	Tashota	3.67	11.8	75.7	0.60	3.00
11	Little Long Lac	1.00	0.97	50.0	0.68	1.40
12	Bankfield-2	1.75	4.75	56.0	0.57	1.30

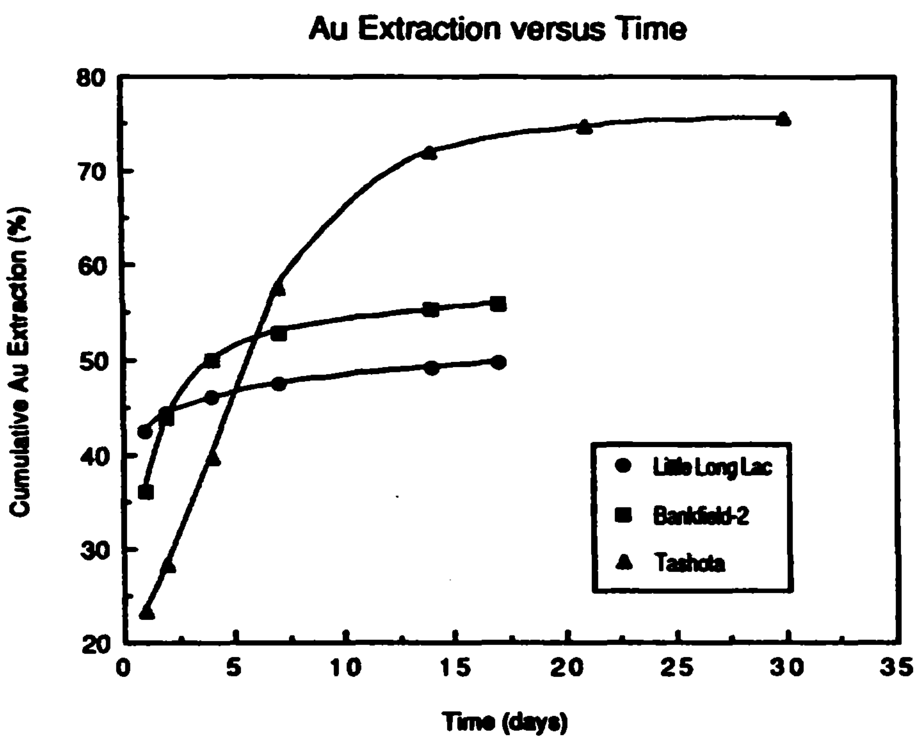
Figure 1 shows the column leach apparatus while Figure 2 depicts the recovery of gold with respect to time.

The metallurgical results for Little Long Lac and Tashota column leaches confirms the results obtained from the preliminary bottle roll cyanidations (without the addition of lead nitrate). Again, it is not possible to compare the Bankfield and Bankfield-2 results since these samples are not representative of each other.



Column Leach Apparatus

Figure 1



-Figure 2-

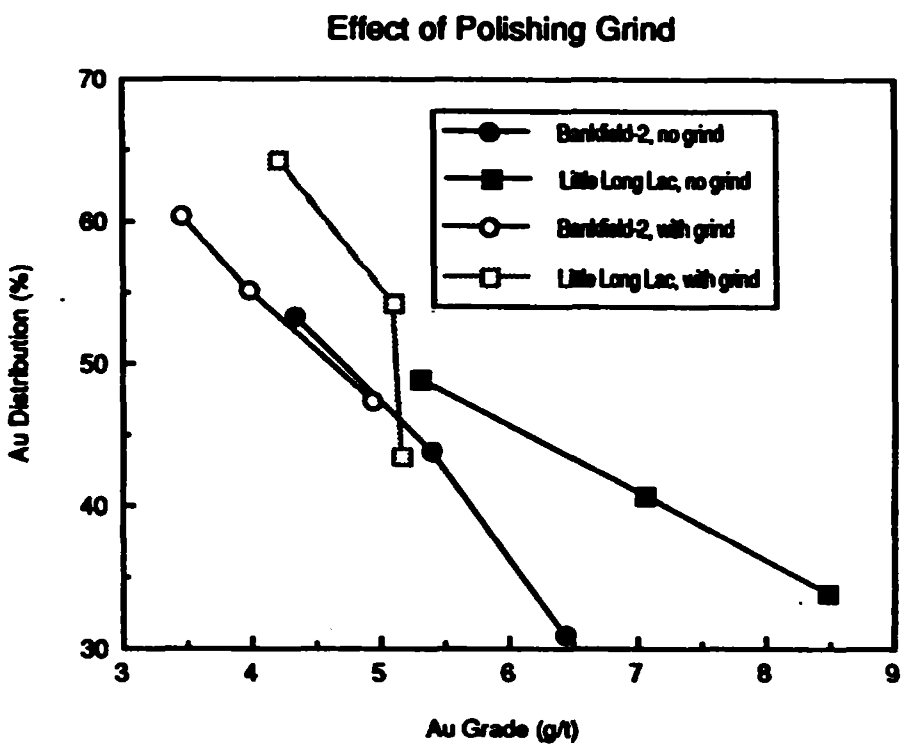
2.5 Flotation

A series of bench scale flotation tests was conducted to investigate the recovery of gold into a sulphide concentrate. Flotation tests were performed on either "as is" material or lightly reground material. The purpose of the light regrind was to polish the surfaces and remove any oxidized layers which may have formed. Aero 412 and PAX were used as collectors, Dowfroth 250 as the frother, $CuSO_4$ and Na_2S as activators. Table 10 summarizes the flotation conditions and metallurgical results.

Table 10: Summary of Flotation Conditions and Metallurgical Results

Test #	Sample	Conditions	Product	Weight %	Assay, g/t or % Au	S	Distribution, % Au	S
F1	Little Long Lac	no grind	Ro Conc 1	0.5	34.0	7.52	13.2	18.0
		200 g/t A350	Ro Conc 1+2	1.0	36.2	7.85	27.5	36.6
		120 g/t AF25	Ro Conc 1-3	5.3	10.9	2.27	43.4	55.4
		1000 g/t Na2S	Ro Conc 1-4	11.4	6.27	1.45	53.2	75.7
		400 g/t CuSO4	Ro Tail	88.6	0.71	0.22	46.8	24.3
			Head(calc)	-	1.34	0.22	-	-
F2	Bankfield-2	no grind	Ro Conc 1	6.4	6.43	19.9	30.9	61.4
		150 g/t A350	Ro Conc 1+2	10.8	5.39	15.9	43.9	83.1
		90 g/t R412	Ro Conc 1-3	16.3	4.34	11.5	53.4	90.7
		500 g/t CuSO4	Ro Tail	83.7	0.74	0.23	46.6	9.3
		18 g/t DF250	Head(calc)	-	1.33	2.07	-	-
F3	Little Long Lac	no grind	Ro Conc 1	5.5	8.47	1.94	33.9	60.1
		150 g/t A350	Ro Conc 1+2	7.8	7.06	1.57	40.8	69.5
		90 g/t R412	Ro Conc 1-3	12.6	5.32	1.13	48.9	80.2
		500 g/t CuSO4	Ro Tail	87.4	0.80	0.04	51.1	19.8
		20 g/t DF250	Head(calc)	113.3	1.37	0.18	-	-
F4	Bankfield-2	grind	Ro Conc 1	12.7	4.93	13.0	47.4	79.8
		150 g/t A350	Ro Conc 1+2	18.3	3.98	9.90	55.2	87.5
		90 g/t R412	Ro Conc 1-3	23.2	3.45	8.21	60.4	91.8
		500 g/t CuSO4	Ro Tail	76.8	0.68	0.22	39.6	8.2
		20 g/t DF250	Head(calc)	-	1.32	2.07	-	-
F5	Little Long Lac	grind	Ro Conc 1	11.2	5.17	0.90	43.5	49.8
		150 g/t A350	Ro Conc 1+2	14.1	5.11	0.82	54.2	57.4
		90 g/t R412	Ro Conc 1-3	20.4	4.20	0.68	64.2	68.5
		500 g/t CuSO4	Ro Tail	79.6	0.60	0.08	35.8	31.5
		20 g/t DF250	Head(calc)	-	1.33	0.20	-	-

As seen in Figure 3, the polishing grind helped improve the overall recovery of the gold, but with a lower grade of concentrate.



-Figure 3-

3. Pilot Plant Testwork

3.1 Column Flotation

The pilot plant testing of the Beaurox Mine samples utilized two 6 inch columns, except for Test No. PP-1, which had a 4 inch cleaner column added.

The columns were controlled by a process control computer using pressure transducers installed near the bottom of the columns. The pressure transducer senses the pulp level by transmitting a 0 to 100 mV signal to a μMAC6000 controller. These signals are digitized and used in a multi-PID alogrithm, along with manually selected setpoints, to prepare 4-20 mA signals to control the speed of the tailing peristaltic pumps. Air was injected into each column through a sintered stainless steel sparger. Feed to the column cells was controlled with manually set variable speed peristaltic pumps. A general arrangement of the flowsheet used is shown in Figure 4.

The purpose of the pilot plant testing was to determine if column flotation could produce a recovery the same as or greater than that achieved in the bench scale mechanical flotation tests.

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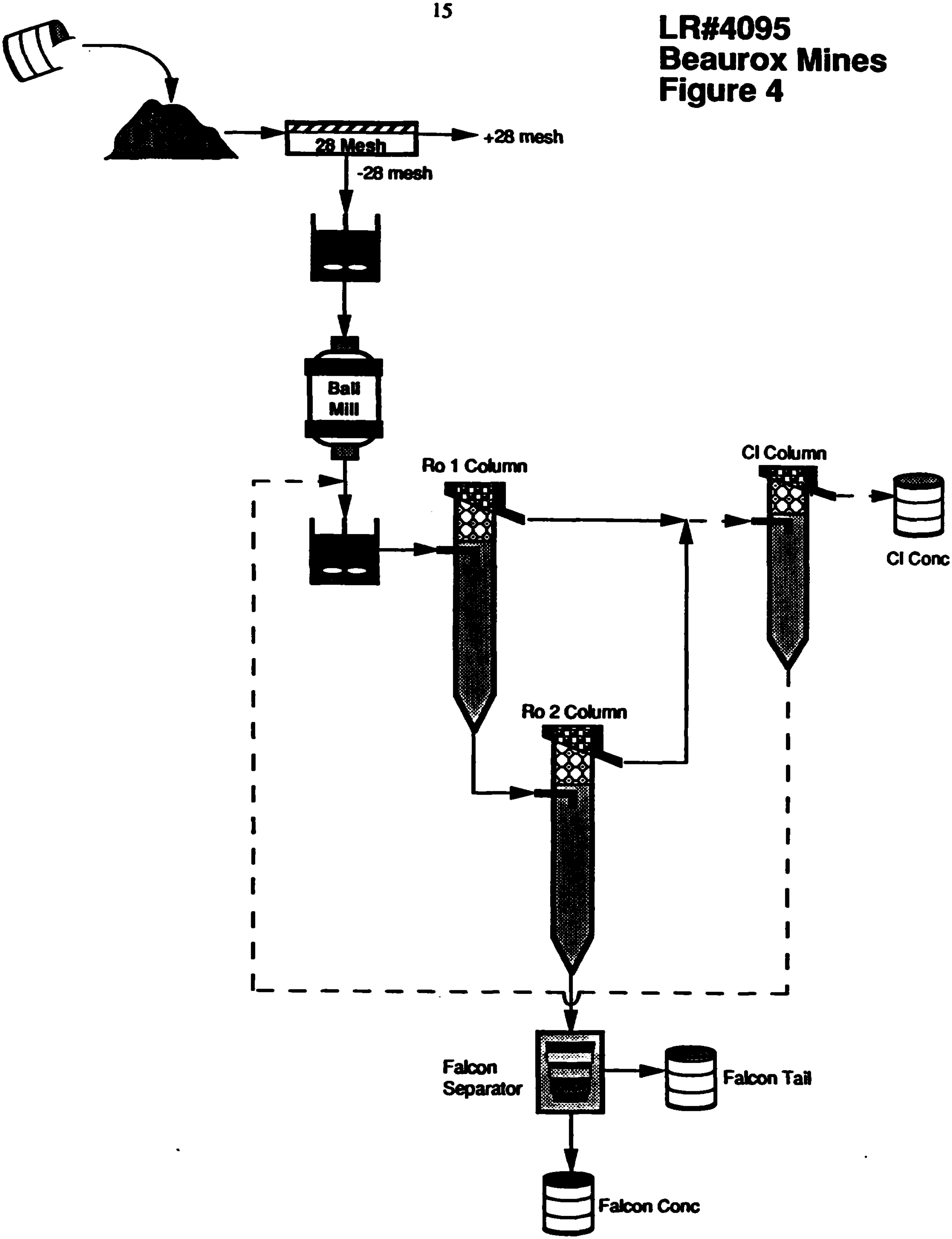
Figure 4



All tests used two 6 inch diameter, 2.5 cu. ft. columns as roughers. The tailing from the first 6 inch column fed the second 6 inch column. The two rougher columns combined gave an average retention time of approximately 50 minutes. Concentrates from the rougher columns were collected together as a combined rougher concentrate. The tailing from the second rougher column was fed to a Falcon separator. This was done in an attempt to recover any free gold which was not floated. Test No. PP-1 also made use of a 4 inch cleaner column. The combined rougher concentrate was fed to this column. The cleaner concentrate was collected and the cleaner tailing was recirculated back to the first rougher column. Tables 11, 12 and 13 provide summaries of flotation conditions, column key variables and metallurgical results, respectively.

Table 11: Summary of Pilot Plant Flotation Conditions

Test #	Sample	Feed Rate kg/h	BMF Grind %-400 mesh	BMD Grind %-400 mesh	Reagents, g/t				
					Na2S	A350	R412	CuSO4	DF250
PP-1	Bankfield	79	42.2	49.4	661	159	115	684	17
PP-2	Bankfield	68	42.1	49.4	644	247	156	618	26
PP-3	Little Long Lac	81	40.8	49.0	600	182	113	593	33
PP-4	Little Long Lac	64	40.0	49.5	778	246	150	609	49

Analysis of the data shows that the columns were operated within normal ranges for wash rate, gas rate and gas holdup. The limited scope of the test program did not provide an opportunity to optimize conditions or to evaluate conditions outside the normal ranges.

Additional testing would be required to examine ways to improve gold recovery. Specifically, reduced wash water flow and shallower froth bed might improve recovery, but the present test results suggest that improvements in recovery might well be at the expense of grade. Additional bench scale tests may also be required to evaluate alternative collectors.

Table 12: Summary of Column Key Variables

Test #	Sample	Rougher#1 Gas Velocity cm/sec	Rougher#1 Wash Velocity cm/sec	Rougher#1 Feed Velocity cm/sec	Rougher#1 NRT minutes	Rougher#1 Flow Bias %	Rougher#1 Gas Hold Up %	Rougher#2 Gas Velocity cm/sec	Rougher#2 Wash Velocity cm/sec	Rougher#2 Feed Velocity cm/sec	Rougher#2 NRT minutes	Rougher#2 Flow Bias %	Rougher#2 Gas Hold Up %
PP-1	Bankfield	0.92	0.20	0.28	27	27	16	0.63	0.10	0.28	25	26	17
PP-2	Bankfield	1.13	0.20	0.34	22	32	22	0.66	0.10	0.33	20	30	16
PP-3	Little Long Lac	0.74	0.20	0.33	23	31	13	0.74	0.10	0.32	21	31	13
PP-4	Little long Lac	0.74	0.20	0.25	30	23	17	0.74	0.10	0.25	28	23	17

Table 13: Summary of Pilot Plant Metallurgical Results

Test #	Sample	Product	Weight %	Assay, g/t or % Au	Assay, g/t or % S	Distribution, % Au	Distribution, % S
PP-1	Bankfield	Combined Cl Conc	4.0	11.7		30.0	
		Falcon Conc	0.2	24.0		3.1	
		Falcon Tailing	96.0	1.08		67.0	
		Feed(calc)	100.0	1.53		100.0	
		Feed(assay)		1.53			
PP-2	Bankfield	Combined Ro Conc	8.6	8.48	27.1	47.2	69.6
		Falcon Conc	0.4	12.5	17.2	3.2	2.1
		Falcon Tailing	91.0	0.84	1.04	49.6	28.4
		Feed(calc)	100.0	1.54	3.34	100.0	100.0
		Feed(assay)		1.54	3.23		
PP-3	Little Long Lac	Combined Ro Conc	2.3	11.8	6.16	21.3	40.6
		Falcon Conc	0.2	30.3	9.43	4.7	7.2
		Falcon Tailing	97.5	0.98	0.14	74.0	52.2
		Feed(calc)	100.0	1.29	0.30	100.0	100.0
		Feed(assay)		1.29	0.24		
PP-4	Little Long Lac	Combined Ro Conc	1.7	13.5	7.06	20.7	44.5
		Falcon Conc	0.4	20.4	1.04	7.2	1.6
		Falcon Tailing	97.9	0.84	0.15	72.1	53.9
		Feed(calc)	100.0	1.14	0.27	100.0	100.0
		Feed(assay)		1.14	0.25		

3.2 Gravity Concentration

Two concentration tests were conducted using a Reichert Mark VII spiral and a Falcon separator. Figure 5 depicts the flowsheet layout and Table 14 summarizes the metallurgical results.

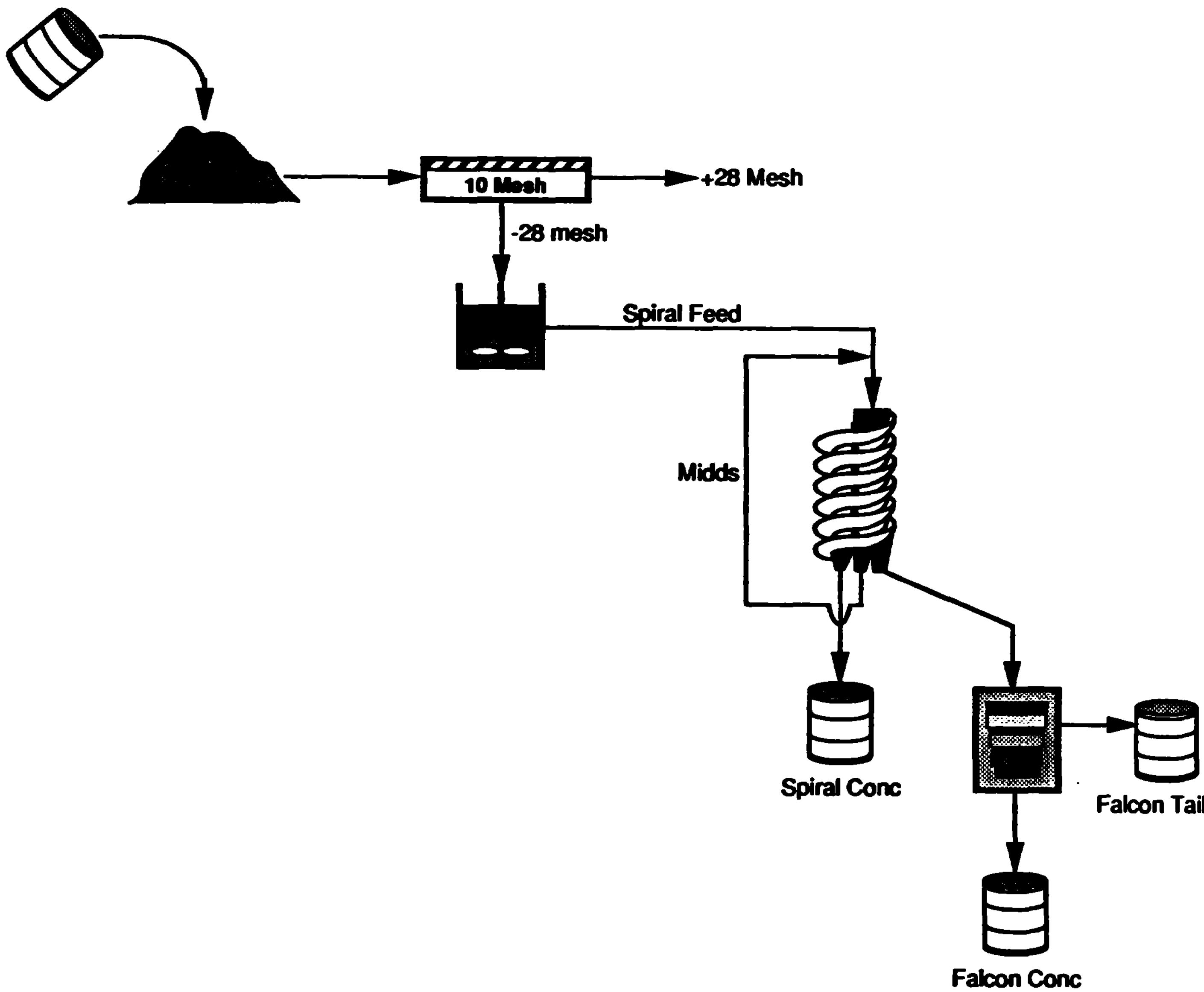
Table 14: Summary of Spiral Metallurgical Results

Test #	Sample	Product	Weight %	Assay, g/t Au	Distribution, % Au
PP-5	Bankfield	Spiral Conc	6.9	4.81	23.0
		Falcon Conc	0.2	70.5	9.7
		Falcon Tailing	92.9	1.05	67.3
		Feed(calc)	100.0	1.45	100.0
		Feed(assay)		1.45	
PP-6	Little Long Lac	Spiral Conc	3.2	6.57	17.9
		Falcon Conc	0.4	40.2	13.5
		Falcon Tailing	96.4	0.85	68.6
		Feed(calc)	100.0	1.19	100.0
		Feed(assay)		1.19	

The metallurgical results from the spiral tests are comparable with earlier gravity concentration testwork. An average of 31% of the gold was recovered using a spiral/Falcon combination in comparison to 24% gold recovery when only the Falcon separator was used.

There appears to be no advantage to using gravity concentration as a means of recovering free gold or gold bearing sulphides.

LR#4095
Beaurox Mines
Figure 5



CONCLUSIONS

1. The addition of lead nitrate helped reduce the cyanide consumption by roughly half.
2. The extraction of gold from the column leaches was comparable to that obtained in the preliminary bottle roll tests.
3. Column flotation of the Bankfield and Little Long Lac bulk samples produced higher grade concentrates at lower recoveries in comparison to the earlier mechanical bench scale tests.
4. The recovery of gold using a spiral and Falcon separator was not successful. The metallurgical results obtained confirmed earlier gravity concentration testwork where only 24% of the gold was recovered.

RECOMMENDATIONS

1. To perform diagnostic leaching tests, in order to determine the location and association of the gold.
2. To conduct a mineralogical examination on feed and tailing sample. This will help identify gangue components, liberation, association and potential recovery of the gold. Centrifuge and heavy liquid tests would be conducted to concentrate the free gold and gold bearing sulphides.
3. To conduct further bench scale flotation tests to investigate new reagent schemes and optimize flotation conditions.
4. To conduct C.I.L. tests to determine if preg robbing or re-precipitation of the gold during leaching is affecting the final extraction of gold.

SAMPLE PREPARATION AND DISPOSITION

On December 8, 1990, January 8 and January 28, 1991, three individual shipments of ore were received at Lakefield Research and given the designation numbers LR9035861, LR9135893 and LR91306035, respectively.

The first shipment contained samples which were used for bench scale testwork. These samples were riffled with 3/4 of the original being stored for subsequent testwork. The 1/4 riffled samples had a series of 1 kg and 10 kg charges prepared from it.

The second shipment received was a series of bulk samples representative of the first shipment, which were used for large column leaches and pilot plant testwork.

The third shipment was a replacement shipment for the original Bankfield since it was decided that this sample was not representative of the overall tailing sample. The new sample was designated as Bankfield-2.

DETAILS OF TESTWORK

Test No.	Sample	Test Description
1	Tashota	Cyanidation
2	Bankfield	Cyanidation
3	Little Long Lac	Cyanidation
4	Tashota	Gravity concentration
5	Bankfield	Gravity concentration
6	Little Long Lac	Gravity concentration
7	Tashota	Cyanidation with lead nitrate
8	Bankfield-2	Cyanidation with lead nitrate
9	Little Long Lac	Cyanidation with lead nitrate
10	Tashota	Column leach
11	Little Long Lac	Column leach
12	Bankfield-2	Column leach
13	Tashota	Gravity concentration
14	Little Long Lac	Gravity concentration
F1	Little Long Lac	Flotation
F2	Bankfield-2	Flotation
F3	Little Long Lac	Flotation
F4	Bankfield-2	Flotation with grind
F5	Little Long Lac	Flotation with grind
PP-1	Bankfield-2	Column flotation pilot plant
PP-2	Bankfield-2	Column flotation pilot plant
PP-3	Little Long Lac	Column flotation pilot plant
PP-4	Little Long Lac	Column flotation pilot plant
PP-5	Bankfield-2	Gravity concentration pilot plant
PP-6	Little Long Lac	Gravity concentration pilot plant

Test 1 Project: 4095 Date: Jan/8/91 Operator: KcS

Purpose: To evaluate Au extraction by direct cyanidation.

Procedure: The ore was pulped in a 2L bottle and agitated on mechanical rolls. NaCN and lime were added and maintained at described levels and cyanidation was carried out in 1 x 120 hour stage. Pregnant sub-samples were removed at 24,48,72, and 96h, with bottles being weighed before and after sampling. At end of test, pulp was filtered and washed, with all products submitted for assay.

Feed: 1000 g minus 28 mesh Tashota

Solution Volume: 1000 mL **Pulp Density:** 50 % Solids

Sol'n Composition: 1.0 g/L NaCN

pH Range: 10.5-11.0 $Ca(OH)_2$

Reagent Consumption (kg/t of cyanide feed) NaCN: 3.28 CaO: 10.4

24h NaCN Consumption: 1.56
48h NaCN Consumption: 2.09
72h NaCN Consumption: 2.53
96h NaCN Consumption: 2.84
120h NaCN Consumption: 3.28

Time	Added, Grams				Residual		Consumed		pH
	Actual		Equivalent		Grams		Grams		
Hours	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	
0 - 1	1.05	8.29	1.00	6.30	0.51	-	0.49	-	10.5-9.5
1 - 3	0.52	1.22	0.49	0.93	0.68	-	0.32	-	10.6-10.0
3 - 5	0.34	0.90	0.32	0.68	0.96	-	0.04	-	11.0-10.3
5 - 8	0.04	0.81	0.04	0.62	0.77	-	0.23	-	11.3-10.5
8 - 24	0.24	0.66	0.23	0.50	0.59	-	0.41	-	11.3-10.5
24 - 32	0.43	0.37	0.41	0.28	0.78	-	0.22	-	11.1-10.6
32 - 48	0.23	0	0.22	0	0.71	-	0.29	-	10.6-10.4
48 - 72	0.31	0.34	0.29	0.26	0.58	-	0.42	-	11.0-10.4
72 - 96	0.44	0.17	0.42	0.13	0.70	-	0.30	-	10.8-10.3
96 - 120	0.32	0.31	0.30	0.24	0.58	0.06	0.42	9.87	10.9-10.7
Total	3.92	13.07	3.72	9.93	0.58	0.06	3.12	9.87	

Metallurgical Balance

Product	Weight mL, g	Assays Au, mg/L, g/t	Distribution, % Au	Estimated Extraction, % Au
+ 28 mesh	29.5	1.49	1.4	
24h Preg	25.0	1.31	1.0	68
48h Preg	25.0	1.54	1.2	78
72h Preg	25.0	1.55	1.2	78
96h Preg	25.0	1.57	1.2	78
120h Preg	1271	1.51	59.6	-
120h Wash	747	0.61	14.1	78.3
Residue	948.5	0.69	20.3	
Feed(calc)	978.0	3.30	100.0	

Test 2 **Project: 4095** **Date: Jan/8/91** **Operator: KcS**

Purpose: To evaluate Au extraction by direct cyanidation.

Procedure: The ore was pulped in a 2L bottle and agitated on mechanical rolls. NaCN and lime were added and maintained at described levels and cyanidation was carried out in 1 x 120 hour stage. Pregnant sub-samples were removed at 24,48,72, and 96h, with bottles being weighed before and after sampling. At end of test, pulp was filtered and washed, with all products submitted for assay.

Feed: 1000 g minus 28 mesh Bankfield

Solution Volume: 1000 mL **Pulp Density:** 50 % Solids

Sol'n Composition: 1.0 g/L NaCN

pH Range: 10.5-11.0 Ca(OH)2

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.91 CaO: 4.10

24h NaCN Consumption: 0.60
48h NaCN Consumption: 0.71
72h NaCN Consumption: 0.78
96h NaCN Consumption: 0.78
120h NaCN Consumption: 0.91

Time	Added, Grams				Residual		Consumed		pH
	Actual		Equivalent		Grams		Grams		
Hours	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	
0 - 1	1.05	2.09	1.00	1.59	0.52	-	0.48	-	10.5-9.4
1 - 3	0.51	0.72	0.48	0.55	0.91	-	0.09	-	10.6-9.9
3 - 5	0.34	0.90	0.32	0.68	1.00	-	0	-	11.3-10.5
5 - 8	0	0.19	0	0.14	1.00	-	0	-	10.8-10.5
8 - 24	0	0.35	0	0.27	1.00	-	0	-	11.1-10.3
24 - 32	0	0.18	0	0.14	1.00	-	0	-	10.9-10.5
32 - 48	0	0	0	0	0.89	-	0.11	-	10.5-10.3
48 - 72	0.12	0.24	0.11	0.18	0.94	-	0.06	-	11.0-10.3
72 - 96	0.06	0.18	0.06	0.14	1.00	-	0	-	10.8-10.3
96 - 120	0	0.23	0	0.17	0.87	0.06	0.13	3.80	10.9-10.5
Total	2.08	5.08	1.98	3.86	0.87	0.06	0.85	3.80	

Metallurgical Balance

Product	Weight mL, g	Assays Au, mg/L, g/t	Distribution, % Au	Estimated Extraction, % Au
+ 28 mesh	53.1	1.73	2.9	
24h Preg	25.0	1.80	1.4	79
48h Preg	25.0	1.79	1.4	79
72h Preg	25.0	1.77	1.4	79
96h Preg	25.0	1.72	1.4	79
120h Preg	1085	1.65	56.3	-
120h Wash	750	0.73	17.2	79.0
Residue	928.1	0.62	18.1	
Feed(calc)	981.2	3.24	100.0	

Test 3 | Project: 4095 | Date: Jan/8/91 | Operator: KcS

Purpose: To evaluate Au extraction by direct cyanidation.

Procedure: The ore was pulped in a 2L bottle and agitated on mechanical rolls. NaCN and lime were added and maintained at described levels and cyanidation was carried out in 1 x 120 hour stage. Pregnant sub-samples were removed at 24,48,72, and 96h, with bottles being weighed before and after sampling. At end of test, pulp was filtered and washed, with all products submitted for assay.

Feed: 1000 g minus 28 mesh Little Long Lac

Solution Volume: 1000 mL **Pulp Density:** 50 % Solids

Sol'n Composition: 1.0 g/L NaCN

pH Range: 10.5-11.0 Ca(OH)2

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.51 CaO: 0.79

24h NaCN Consumption: 0.21
48h NaCN Consumption: 0.24
72h NaCN Consumption: 0.33
96h NaCN Consumption: 0.45
120h NaCN Consumption: 0.51

Time	Added, Grams				Residual		Consumed		pH
	Actual		Equivalent		Grams		Grams		
Hours	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	
0 - 1	1.05	0.77	1.00	0.59	0.93	-	0.07	-	11.0-10.7
1 - 3	0.07	0.00	0.07	0	1.00	-	0	-	10.7-10.8
3 - 5	0	0.00	0	0	1.00	-	0	-	10.8-10.7
5 - 8	0	0.00	0	0	1 00	-	0	-	10.7-10.5
8 - 24	0	0.17	0	0.13	0 85	-	0.15	-	11.2-10.9
24 - 32	0.16	0	0.15	0	1 00	-	0	-	10.9-10.9
32 - 48	0	0	0	0	0.97	-	0.03	-	10.9-10.7
48 - 72	0.03	0	0.03	0	0 91	-	0.09	-	10.7-10.5
72 - 96	0.09	0	0.09	0	0.88	-	0.12	-	10.5-10.4
96 - 120	0.13	0.15	0.12	0.11	0.94	0.06	0.06	0.77	11.0-10.8
Total	1.53	1.09	1.45	0.83	0.94	0.06	0.50	0.77	

Metallurgical Balance

Product	Weight mL, g	Assays Au, mg/L, g/t	Distribution, % Au	Estimated Extraction, % Au
+ 28 mesh	5.2	4.29	1.5	
24h Preg	25.0	0.64	1.1	55
48h Preg	25.0	0.67	1.1	55
72h Preg	25.0	0.63	1.1	55
96h Preg	25.0	0.63	1.1	55
120h Preg	973	0.61	39.7	-
120h Wash	1104	0.15	11.1	55.1
Residue	969.0	0.67	43.4	
Feed(calc)	974.2	1.53	100.0	

Test No. 4 LR#4095 Feb.14/91 Operator: KcS

Gravity Concentration

Purpose: To investigate the recovery of a gold bearing sulphide concentrate by gravity concentration using a Wilfley 1/8 table and further upgrading of the gravity concentrate with the Mozley separator.

Procedure: The sample was passed over a Wilfley table through an open circuit where the concentrate and tailing were collected.
The concentrate was then transferred to the Mozley separator for further cleaning. The Mozley concentrate and tailing were decanted and submitted for assay.

Feed: 2 kg of Tashota composite.

Grind: As is.

Metallurgical Results

Product	Weight	Assay, g/t, %		Distribution, %	
	%	Au	S	Au	S
Mozley Conc	0.1	141		2.9	
Mozlry Tail	2.7	5.44		4.6	
+ 28 mesh Table Conc	1.1	1.61		0.5	
Table Conc(calc)	3.9	6.70	20.5	8.0	31.4
Table Tail	96.2	3.08	1.79	92.0	68.8
Feed(calc)	100.0	3.22	2.51	100.0	100.0

Test No. 5 LR#4095 Feb.14/91 Operator: KcS

Gravity Concentration

Purpose: To investigate the recovery of a gold bearing sulphide concentrate by gravity concentration using a Wilfley 1/8 table and further upgrading of the gravity concentrate with the Mozley separator.

Procedure: The sample was passed over a Wilfley table through an open circuit where the concentrate and tailing were collected.
The concentrate was then transferred to the Mozley separator for further cleaning. The Mozley concentrate and tailing were decanted and submitted for assay.

Feed: 2 kg of Bankfield comopsite.

Grind: As is.

Metallurgical Results

Product	Weight	Assay, g/t, %		Distribution, %	
	%	Au	S	Au	S
Mozley Conc	0.1	297		8.1	
Mozlry Tail	3.8	7.67		10.3	
+ 28 mesh Table Conc	1.7	1.00		0.6	
Table Conc(calc)	5.5	9.66	10.7	19.0	25.8
Table Tail	94.5	2.40	1.79	81.0	74.2
Feed(calc)	100.0	2.80	2.28	100.0	100.0

Test No. 6 LR#4095 Feb.14/91 Operator: KcS

Gravity Concentration

Purpose: To investigate the recovery of a gold bearing sulphide concentrate by gravity concentration using a Wilfley 1/8 table and further upgrading of the gravity concentrate with the Mozley separator.

Procedure: The sample was passed over a Wilfley table through an open circuit where the concentrate and tailing were collected.
The concentrate was then transferred to the Mozley separator for further cleaning. The Mozley concentrate and tailing were decanted and submitted for assay.

Feed: 2 kg of Little long Lac composite.

Grind: As is.

Metallurgical Results

Product	Weight %	Assay, g/t, % Au	S	Distribution, % Au	S
Mozley Conc	0.1	116		5.5	
Mozlry Tail	5.2	3.45		14.8	
+ 28 mesh Table Conc	0.1	1.12		0.1	
Table Conc(calc)	5.4	4.59	1.24	20.4	35.3
Table Tail	94.6	1.03	0.13	79.6	64.7
Feed(calc)	100.0	1.22	0.19	100.0	100.0

Test 7 Project: 4095 Date: Feb/4/91 Operator: JH

Purpose: To evaluate Au extraction by direct cyanidation.

Procedure: The ore was pulped in a 2L bottle and agitated on mechanical rolls. NaCN and lime were added and maintained at described levels and cyanidation was carried out in 1 x 72 hour stage. Lead nitrate was added at the being of the leach. At the end of the test, pulp was filtered and washed, with all products being submitted for assay.

Feed: 1000 g minus 28 mesh Tashota

Solution Volume: 1000 mL Pulp Density: 50 % Solids

Sol'n Composition: 0.5 g/L NaCN

pH Range: 10.5-11.0 $Ca(OH)_2$

$Pb(NO_3)_2$: 500 g/t

Reagent Consumption (kg/t of cyanide feed) NaCN: 1.47 CaO: 9.67

24h NaCN and CaO Consumption:	0.98	9.09
48h NaCN and CaO Consumption:	1.25	9.41
72h NaCN and CaO Consumption:	1.47	9.67

Time	Added, Grams				Residual		Consumed		pH
	Actual		Equivalent		Grams		Grams		
Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO	
0 - 1	0.53	8.13	0.50	6.18	0.10	-	0.40	-	10.9-9.0
1 - 2	0.42	1.30	0.40	0.99	0.50	-	0	-	10.9-9.8
2 - 20	0	0.78	0	0.58	0.20	-	0.30	-	10.9-9.5
20 - 24	0.32	0.56	0.30	0.43	0.35	0.02	0.15	8.15	10.8-10.0
24 - 48	0.16	0.40	0.15	0.30	0.25	0.02	0.25	0.28	10.9-9.9
48 - 72	0.26	0.36	0.25	0.27	0.30	0.04	0.20	0.23	10.8-10.0
Total	1.69	11.5	1.61	8.75	0.30	0.04	1.32	8.67	

Metallurgical Balance

Product	Weight mL, g	Assays Au, mg/L, g/t	Distribution, % Au
+ 28 mesh	87.5	2.14	6.5
Preg+Wash	2090	0.96	69.4
Residue	896.2	0.78	24.2
Feed(calc)	983.7	2.94	100.0

Test 8 Project: 4095 Date: Feb/4/91 Operator: JH

Purpose: To evaluate Au extraction by direct cyanidation.

Procedure: The ore was pulped in a 2L bottle and agitated on mechanical rolls. NaCN and lime were added and maintained at described levels and cyanidation was carried out in 1 x 72 hour stage. Lead nitrate was added at the being of the leach. At the end of the test, pulp was filtered and washed, with all products being submitted for assay.

Feed: 1000 g minus 28 mesh Bankfield-2

Solution Volume: 1000 mL **Pulp Density:** 50 % Solids

Sol'n Composition: 0.5 g/L NaCN

pH Range: 10.5-11.0 $Ca(OH)_2$

$Pb(NO_3)_2$: 500 g/t

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.37 CaO: 2.03

24h NaCN and CaO Consumption:	0.36	1.73
48h NaCN and CaO Consumption:	0.37	1.89
72h NaCN and CaO Consumption:	0.37	2.03

Time	Added, Grams				Residual		Consumed		pH
	Actual		Equivalent		Grams		Grams		
Hours	NaCN	$Ca(OH)_2$	NaCN	CaO	NaCN	CaO	NaCN	CaO	
0 - 1	0.53	1.38	0.50	1.05	0.40	-	0.10	-	11.2-9.8
1 - 2	0.11	0.36	0.10	0.27	0.50	-	0	-	10.9-10.1
2 - 20	0	0.24	0	0.18	0.30	-	0.20	-	10.8-9.7
20 - 24	0.21	0.28	0.20	0.20	0.45	0	0.05	1.70	10.7-10.1
24 - 48	0.05	0.21	0.05	0.16	0.50	0	0	0.16	11.0-10.0
48 - 72	0	0.21	0.00	0.16	0.50	0.02	0	0.14	10.9-10.1
Total	0.90	2.66	0.86	2.02	0.50	0.02	0.36	2.00	

Metallurgical Balance

Product	Weight mL, g	Assays Au, mg/L, g/t	Distribution, % Au
+ 28 mesh	2.3	0.43	0.1
Preg+Wash	2090	0.36	56.4
Residue	983.7	0.59	43.5
Feed(calc)	986.0	1.35	100.0

Test 9 Project: 4095 Date: Feb/4/91 Operator: JH

Purpose: To evaluate Au extraction by direct cyanidation.

Procedure: The ore was pulped in a 2L bottle and agitated on mechanical rolls. NaCN and lime were added and maintained at described levels and cyanidation was carried out in 1 x 72 hour stage. Lead nitrate was added at the being of the leach. At the end of the test, pulp was filtered and washed, with all products being submitted for assay.

Feed: 1000 g minus 28 mesh Little Long Lac

Solution Volume: 1000 mL **Pulp Density:** 50 % Solids

Sol'n Composition: 0.5 g/L NaCN

pH Range: 10.5-11.0 $Ca(OH)_2$

$Pb(NO_3)_2$: 500 g/t

Reagent Consumption (kg/t of cyanide feed) NaCN: 0.12 CaO: 0.89

	NaCN	CaO
24h NaCN and CaO Consumption:	0.12	0.67
48h NaCN and CaO Consumption:	0.12	0.73
72h NaCN and CaO Consumption:	0.12	0.89

Time	Added, Grams				Residual		Consumed		pH
	Actual		Equivalent		Grams		Grams		
Hours	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	
0 - 1	0.53	0.65	0.50	0.49	0.40	-	0.10	-	11.2-10.5
1 - 2	0.11	0.11	0.10	0.08	0.50	-	0	-	10.9-10.6
2 - 20	0	0.05	0	0.04	0.50	-	0	-	10.8-10.3
20 - 24	0	0.06	0	0.05	0.50	0.01	0	0.65	10.7-10.4
24 - 48	0	0.08	0	0.06	0.50	0	0	0.06	11.0-10.3
48 - 72	0	0.25	0	0.19	0.50	0.04	0	0.15	11.5-10.9
Total	0.64	1.20	0.61	0.91	0.50	0.04	0.12	0.86	

Metallurgical Balance

Product	Weight mL, g	Assays Au, mg/L, g/t	Distribution, % Au
+ 28 mesh	11.9	1.16	1.0
Preg+Wash	2090	0.33	49.1
Residue	973.3	0.72	49.9
Feed(calc)	985.2	1.43	100.0

Test 10 Project: 4095 Date: Mar/04/91 Operator: JH

Purpose: To evaluate Au extraction by direct cyanidation.

Procedure: Approximately 10 kg of agglomerated sample was loaded into a plastic column 102 mm in diameter, to a height of 106 cm. A piece of steel mesh was placed at the bottom of the column and a piece of burlap on top to help disperse the solution. Approximately 5 L of 0.5 g/L NaCN solution was percolated through the column at a rate of 5 mL per minute. The pregnant solution was passed through carbon column where the Au in solution was removed. The reagents were replenished as required during the test. The loaded carbon was changed after 1, 2, 4, 7, 14, 21, and 30 days and replaced with fresh carbon.

Feed: 10000 g minus 28 mesh Tashota

Solution Volume: 5000 mL Pulp Density: 33 % Solids

Sol'n Composition: 0.5 g/L NaCN Cement: 5 kg/t

pH Range: 10.0-11.0 $Ca(OH)_2$

Reagent Consumption (kg/t of cyanide feed) NaCN: 3.67 CaO: 11.8

Time	Added, Grams				Residual		Consumed		pH
	Actual		Equivalent		Grams		Grams		
Days	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	
Aggmolerate	17.0	139	16.2	106	-	-	-	-	
0 - 1	2.00	0	1.90	0	1.91	-	16.1	-	8.9-9.2
1 - 2	0.60	0	0.57	0	1.25	-	1.25	-	9.2-9.4
2 - 3	1.32	0.16	1.25	0.122	1.50	-	1.00	-	10.3-9.5
3 - 4	1.05	1.00	1.00	0.76	1.50	-	1.00	-	11.5-9.8
4 - 5	1.05	1.00	1.00	0.76	1.58	-	0.92	-	11.6-10.1
5 - 6	1.05	0	1.00	0	1.75	-	0.75	-	10.1-9.8
6 - 7	0.79	0.50	0.75	0.38	1.50	-	1.00	-	11.2-9.7
7 - 8	1.05	0.50	1.00	0.38	1.50	-	1.00	-	11.2-9.6
8 - 9	1.05	0.50	1.00	0.38	1.25	-	1.25	-	11.0-9.6
9 - 10	1.32	0.50	1.25	0.38	1.25	-	1.25	-	11.4-9.6
10 - 13	1.32	1.00	1.25	0.76	1.25	-	1.25	-	11.5-9.3
13 - 14	1.32	1.00	1.25	0.76	1.78	-	0.72	-	11.9-9.5
14 - 16	0.76	0.70	0.72	0.53	1.10	-	1.40	-	11.1-9.2
16 - 17	1.46	0.70	1.39	0.53	1.30	-	1.20	-	11.4-9.4
17 - 21	1.26	1.00	1.20	0.76	0.75	-	1.75	-	11.7-8.2
21 - 23	1.84	2.00	1.75	1.52	1.00	-	1.50	-	11.9-9.3
23 - 27	1.58	2.00	1.50	1.52	1.00	-	1.50		10.6-9.1
27 - 30	1.58	3.00	1.50	2.28	1.00	0	1.50	117	10.5-9.9
Total	39.4	155	37.4	117	1.00	0	36.4	117	

Test 10 Project: 4095 Date: Mar/04/91 Operator: JH

Metallurgical Balance

Product	Weight mL, g	Assays Au, mg/L, g/t	Distribution, % Au	Cum. Extraction, % Au
+ 28 mesh	821	2.17	6.0	
Day 1 Loaded Carbon	15.0	463	23.3	23.3
Day 2 Loaded Carbon	13.9	105	4.9	28.2
Day 4 Loaded Carbon	14.0	246	11.6	39.8
Day 7 Loaded Carbon	15.6	338	17.7	57.5
Day 14 Loaded Carbon	16.9	254	14.4	71.9
Day 21 Loaded Carbon	17.9	46.3	2.8	74.7
Day 30 Loaded Carbon	16.3	17.1	0.9	75.6
Barren Solution	4100	<0.002	0	75.6
Bareen Wash	8880	<0.002	0.1	75.7
Residue	9098	0.60	18.3	
Feed(assay) *	9919	3.00	100.0	

Test 11 Project: 4095 Date: Feb/22/91 Operator: JH

Purpose: To evaluate Au extraction by direct cyanidation.

Procedure: Approximately 10 kg of agglomerated sample was loaded into a plastic column 102 mm in diameter, to a height of 100 cm. A piece of steel mesh was placed at the bottom of the column and a piece of burlap on top to help disperse the solution. Approximately 5 L of 0.5 g/L NaCN solution was percolated through the column at a rate of 5 mL per minute. The pregnant solution was passed through carbon column where the Au in solution was removed. The reagents were replenished as required during the test. The loaded carbon was changed after 1, 2, 4, 7, 14, and 17 days and replaced with fresh carbon.

Feed: 10000 g minus 28 mesh Little Long Lac

Solution Volume: 5000 mL Pulp Density: 33 % Solids

Sol'n Composition: 0.5 g/L NaCN Cement: 5 kg/t

pH Range: 10.0-11.0 $Ca(OH)_2$

Reagent Consumption (kg/t of cyanide feed) NaCN: 1.00 CaO: 0.97

Time	Added, Grams				Residual		Consumed		pH
	Actual		Equivalent		Grams		Grams		
Days	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	
Aggmolerate	3.00	10.5	2.85	7.98	-	-	-	-	
0 - 1	2.00	0	1.90	0	1.00	-	3.75	-	8.9-9.6
1 - 2	1.60	0	1.50	0	1.75	-	0.75	-	9.6-9.8
2 - 3	1.84	0	1.75	0	2.50	-	0	-	9.8-9.9
3 - 4	0	0	0	0	1.75	-	0.75	-	9.9-9.9
4 - 5	0.79	0.60	0.75	0.46	2.00	-	0.50	-	11.6-10.1
5 - 6	0.53	0	0.50	0	2.25	-	0.25	-	10.1-10.0
6 - 7	0.26	0	0.25	0	2.00	-	0.50	-	10.0-10.1
7 - 8	0.53	0	0.50	0	2.50	-	0	-	10.1-10.1
8 - 9	0	0.50	0	0.38	2.50	-	0	-	11.5-10.3
9 - 10	0	0	0	0	2.25	-	0.25	-	10.3-10.2
10 - 13	0.26	0	0.25	0	1.75	-	0.75	-	10.2-10.2
13 - 14	0.79	0	0.75	0	1.70	-	0.80	-	10.2-10.1
14 - 16	0.84	0.50	0.80	0.38	1.80	-	0.70	-	11.0-10.2
16 - 17	0.74	0.60	0.70	0.46	1.73	0.11	0.77	9.54	11.5-11.1
Total	13.2	12.7	12.5	9.65	1.73	0.11	9.77	9.54	

Test 11 Project: 4095 Date: Feb/22/91 Operator: JH

Metallurgical Balance

Product	Weight mL, g	Assays Au, mg/L, g/t	Distribution, % Au	Cum. Extraction, % Au
+ 28 mesh	133	2.33	2.2	
Day 1 Loaded Carbon	14.4	405	42.4	42.4
Day 2 Loaded Carbon	14.8	19.5	2.1	44.5
Day 4 Loaded Carbon	14.5	14.5	1.5	46.0
Day 7 Loaded Carbon	15.7	13.4	1.5	47.6
Day 14 Loaded Carbon	16.0	14.3	1.7	49.2
Day 17 Loaded Carbon	15.4	5.20	0.6	49.8
Barren Solution	3750	<0.002	0.1	49.9
Barren Wash	3760	<0.002	0.1	50.0
Residue	9676	0.68	47.8	
Feed(calc)	9809	1.40	100.0	

Test 12 Project: 4095 Date: Feb/22/91 Operator: JH

Purpose: To evaluate Au extraction by direct cyanidation.

Procedure: Approximately 10 kg of agglomerated sample was loaded into a plastic column 102 mm in diameter, to a height of 116 cm. A piece of steel mesh was placed at the bottom of the column and a piece of burlap on top to help disperse the solution. Approximately 5 L of 0.5 g/L NaCN solution was percolated through the column at a rate of 5 mL per minute. The pregnant solution was passed through carbon column where the Au in solution was removed. The reagents were replenished as required during the test. The loaded carbon was changed after 1, 2, 4, 7, 14, and 17 days and replaced with fresh carbon.

Feed: 10000 g Bankfield-2

Solution Volume: 5000 mL Pulp Density: 33 % Solids

Sol'n Composition: 0.5 g/L NaCN Cement: 5 kg/t

pH Range: 10.0-11.0 Ca(OH)2

Reagent Consumption (kg/t of cyanide feed) NaCN: 1.75 CaO: 4.76

Time	Added, Grams				Residual		Consumed		pH
	Actual		Equivalent		Grams		Grams		
Days	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	
Aggmolerate	5.00	55	4.75	41.8	-	-	-	-	
0 - 1	2.00	0	1.90	0	0.75	-	5.90	-	8.6-9.4
1 - 2	1.84	0	1.75	0	1.25	-	1.25	-	9.4-9.6
2 - 3	1.32	1.00	1.25	0.76	2.00	-	0.50	-	11.5-10.0
3 - 4	0.53	0	0.50	0	2.00	-	0.50	-	10.0-9.7
4 - 5	0.53	0.50	0.50	0.38	2.25	-	0.25	-	11.9-9.8
5 - 6	0.26	0.50	0.25	0.38	1.50	-	1.00	-	11.1-10.0
6 - 7	1.05	0	1.00	0	1.75	-	0.75	-	10.0-9.8
7 - 8	0.79	0.50	0.75	0.38	2.00	-	0.50	-	10.7-9.9
8 - 9	0.53	0.50	0.50	0.38	1.50	-	1.00	-	11.5-9.8
9 - 10	1.05	0.80	1.00	0.61	1.75	-	0.75	-	10.8-10.0
10 - 13	0.79	0.50	0.75	0.38	1.00	-	1.50	-	10.8-9.6
13 - 14	1.58	0.50	1.50	0.38	1.20	-	1.30	-	11.1-9.8
14 - 16	1.37	0.50	1.30	0.38	1.50	-	1.00	-	11.2-9.6
16 - 17	0.89	0.85	0.85	0.65	1.63	0	0.87	46.5	11.1-10.3
Total	19.5	61.2	18.6	46.5	1.63	0	17.1	46.5	

Metallurgical Balance

Product	Weight mL, g	Assays Au, mg/L, g/t	Distribution, % Au	Cum. Extraction, % Au
Day 1 Loaded Carbon	14.6	313	36.1	36.1
Day 2 Loaded Carbon	14.5	69.0	7.9	44.0
Day 4 Loaded Carbon	15.6	49.8	6.1	50.1
Day 7 Loaded Carbon	15.6	23.0	2.8	52.9
Day 14 Loaded Carbon	16.7	17.2	2.3	55.2
Day 17 Loaded Carbon	16.7	5.20	0.7	55.9
Barren Solution	3540	<0.002	0.1	56.0
Barren Wash	3840	<0.002	0.1	56.0
Residue	9762	0.57	44.0	
Feed(calc)	9762	1.30	100.0	

Test No. 13 LR#4095 Feb.4/91 Operator: JH

Gravity Concentration

Purpose: To investigate the recovery of a gold bearing sulphide concentrate by gravity concentration using a Falcon Concentrator.

Procedure: The sample was passed through a Falcon concnetrator where the concentrate and tailing were collected.
The feed was screened at 28 mesh prior to treatment.
Water was added to the feed to dilute it to approximately 10% solids by weight.
Final products were assayed for gold and sulphur.

Feed: ~10 kg of -28 mesh Tashota composite.

Grind: As is.

Metallurgical Results

Product	Weight	Assay, g/t, %		Distribution, %	
	%	Au	S	Au	S
+ 28 mesh	3.9	1.53	4.34	1.9	8.2
Falcon Conc	7.9	9.69	2.55	23.5	9.6
Falcon Tail	88.2	2.75	1.96	74.7	82.3
Feed(calc)	100.0	3.25	2.10	100.0	100.0

Test No. 14 | LR#4095 | Feb.4/91 | Operator: JH

Gravity Concentration

Purpose: To investigate the recovery of a gold bearing sulphide concentrate by gravity concentration using a Falcon Concentrator.

Procedure: The sample was passed through a Falcon concnetrator where the concentrate and tailing were collected.
The feed was screened at 28 mesh prior to treatment.
Water was added to the feed to dilute it to approximately 10% solids by weight.
Final products were assayed for gold and sulphur.

Feed: ~10 kg of -28 mesh Little Long Lac composite.

Grind: As is.

Metallurgical Results

Product	Weight %	Assay, g/t, % Au	Assay, g/t, % S	Distribution, % Au	Distribution, % S
+ 28 mesh	3.9	1.53	4.34	1.9	8.2
Falcon Conc	7.9	9.69	2.55	23.5	9.6
Falcon Tail	88.2	2.75	1.96	74.7	82.3
Feed(calc)	100.0	3.25	2.10	100.0	100.0

Test No. F1 | Project No. 4095 | 1/29/91 | Operator: DE

Purpose: To perform preliminary rougher flotation test.

Procedure: As stated below.

Feed: ~2kg of -28 mesh Litte Long Lac

Grind: N/A

	Reagents added, grams per tonne							Time, minutes			
Stage	A350	AF25	Na2S	CuSO4	DF-250			Grind	Cond.	Froth	pH
Rougher 1	50	40							1	1	8.2
Condx				400					5		
Rougher 2	50	40							1	2	7.6
Condx			500						5		
Rougher 3	50	40							1	5	9.2
Condx			500						5		
Rougher 4	50								1	10	9.5

Stage	Ro										
Flotation Cell	D-1										
Speed: r.p.m.	1800										
% Solids	35										

Metallurgical Balance

Product	Weight g	Weight %	Assays, g/t, % Au	Assays, g/t, % S	Distribution, % Au	Distribution, % S
1. Ro Conc 1	10.2	0.5	34.0	7.52	13.2	18.0
2. Ro Conc 2	9.7	0.5	38.5	8.19	14.3	18.6
3. Ro Conc 3	84.2	4.3	4.95	0.95	15.9	18.8
4. Ro Conc 4	118.3	6.1	2.18	0.73	9.8	20.3
5. Ro Tailing	1727.5	88.6	0.71	0.06	46.8	24.3
Feed(calc)	1949.9	100.0	1.34	0.22	100.0	100.0

Combined Products

Product	Weight g	Weight %	Au	S	Au	S
Ro Conc 1+2	19.9	1.0	36.2	7.85	27.5	36.6
Ro Conc 1-3	104.1	5.3	10.9	2.27	43.4	55.4
Ro Conc 1-4	222.4	11.4	6.27	1.45	53.2	75.7

Test No. F2 | Project No. 4095 | 1/31/91 | Operator: JMD

Purpose: To perform preliminary rougher flotation test.

Procedure: As stated below.

Feed: ~2kg of -28 mesh Bankfield # 2

Grind: N/A

	Reagents added, grams per tonne							Time, minutes			
Stage	A350	R412	Na2S	CuSO4	DF-250			Grind	Cond.	Froth	pH
Codx 1			500						2		8.0
Condx 2				400					5		7.5
Rougher 1	50	30			10				1	5	7.6
Rougher 2	50	30							1	10	
Rougher 3	50	30			7.5				1	10	

Stage	Ro
Flotation Cell	D-1
Speed: r.p.m.	1800
% Solids	35

Metallurgical Balance

Product	Weight g	Weight %	Assays, g/t, % Au	Assays, g/t, % S	Distribution, % Au	Distribution, % S
1. Ro Conc 1	124.8	6.4	6.43	19.9	30.9	61.4
2. Ro Conc 2	86.1	4.4	3.89	10.2	12.9	21.7
3. Ro Conc 3	107.8	5.5	2.29	2.85	9.5	7.6
4. Ro Tailing	1633.9	83.7	0.74	0.23	46.6	9.3
Feed(calc)	1952.6	100.0	1.33	2.07	100.0	100.0

Combined Products

Product	Weight g	Weight %	Assays, g/t, % Au	Assays, g/t, % S	Distribution, % Au	Distribution, % S
Ro Conc 1+2	210.9	10.8	5.39	15.9	43.9	83.1
Ro Conc 1-3	318.7	16.3	4.34	11.5	53.4	90.7

Test No. F3 | Project No. 4095 | 2/4/91 | Operator: JMD

Purpose: To perform preliminary rougher flotation test.

Procedure: As stated below.

Feed: ~2kg of -28 mesh Little Long Lac

Grind: N/A

Stage	Reagents added, grams per tonne							Time, minutes			pH
	A350	R412	Na2S	CuSO4	DF-250			Grind	Cond.	Froth	
Codx 1			500						2		9.8
Condx 2				400					5		9.3
Rougher 1	50	30			10				1	5	
Rougher 2	50	30			2.5				1	10	
Rougher 3	50	30			7.5				1	15	

Stage	Ro
Flotation Cell	D-1
Speed: r.p.m.	1800
% Solids	35

Metallurgical Balance

Product	Weight g	Weight %	Assays, g/t, % Au	Assays, g/t, % S	Distribution, % Au	Distribution, % S
1. Ro Conc 1	107.6	5.5	8.47	1.94	33.9	60.1
2. Ro Conc 2	46.2	2.4	3.77	0.71	6.5	9.4
3. Ro Conc 3	93.0	4.7	2.44	0.40	8.4	10.7
4. Ro Tailing	1716.8	87.4	0.80	0.04	51.1	19.8
Feed(calc)	1963.6	100.0	1.37	0.18	100.0	100.0

Combined Products

Product	Weight g	Weight %	Au	S	Dist. Au	Dist. S
Ro Conc 1+2	153.8	7.8	7.06	1.57	40.4	69.5
Ro Conc 1-3	246.8	12.6	5.32	1.13	48.9	80.2

Test No. F4 Project No. 4095 2/4/91 Operator: JMD

Purpose: To perform preliminary rougher flotation test.

Procedure: As stated below.

Feed: ~2kg of -28 mesh Bankfield-2

Grind: 5minutes/2kg @ 65% solids in laboratory rod mill

	Reagents added, grams per tonne							Time, minutes			
Stage	A350	R412	Na_2S	$CuSO_4$	DF-250			Grind	Cond.	Froth	pH
Codx 1			500						2		9.2
Condx 2				400					5		7.8
Rougher 1	50	30			10				1	5	
Rougher 2	50	30			2.5				1	10	
Rougher 3	50	30			7.5				1	15	

Stage	Ro
Flotation Cell	D-1
Speed: r.p.m.	1800
% Solids	35

Metallurgical Balance

Product	Weight g	Weight %	Assays, g/t, % Au	Assays, g/t, % S	Distribution, % Au	Distribution, % S
1. Ro Conc 1	249.6	12.7	4.93	13.0	47.4	79.8
2. Ro Conc 2	110.0	5.6	1.83	2.86	7.8	7.7
3. Ro Conc 3	95.4	4.9	1.43	1.85	5.3	4.3
4. Ro Tailing	1509.0	76.8	0.68	0.22	39.6	8.2
Feed(calc)	1964.0	100.0	1.32	2.07	100.0	100.0

Combined Products

Product	Weight g	Weight %	Assays, g/t, % Au	Assays, g/t, % S	Distribution, % Au	Distribution, % S
Ro Conc 1+2	359.6	18.3	3.98	9.90	55.2	87.5
Ro Conc 1-3	455.0	23.2	3.45	8.21	60.4	91.8

Test No. F5 | Project No. 4095 | 2/4/91 | Operator: JMD

Purpose: To perform preliminary rougher flotation test.

Procedure: As stated below.

Feed: ~2kg of -28 mesh Little Long Lac

Grind: 5 minutes/2kg @ 65% solids in laboratory rod mill

	Reagents added, grams per tonne							Time, minutes			
Stage	A350	R412	Na2S	CuSO4	DF-250			Grind	Cond.	Froth	pH
Codx 1			500						2		9.7
Condx 2				400					5		9.0
Rougher 1	50	30			10				1	5	
Rougher 2	50	30			2.5				1	10	
Rougher 3	50	30			7.5				1	15	

Stage	Ro										
Flotation Cell	D-1										
Speed: r.p.m.	1800										
% Solids	35										

Metallurgical Balance

Product	Weight		Assays, g/t, %		Distribution, %	
	g	%	Au	S	Au	S
1. Ro Conc 1	220.0	11.2	5.17	0.90	43.5	49.8
2. Ro Conc 2	57.7	2.9	4.89	0.52	10.8	7.6
3. Ro Conc 3	122.6	6.2	2.12	0.36	9.9	11.1
4. Ro Tailing	1563.5	79.6	0.60	0.08	35.8	31.5
Feed(calc)	1963.8	100.0	1.33	0.20	100.0	100.0

Combined Products

Ro Conc 1+2	277.7	14.1	5.11	0.82	54.2	57.4
Ro Conc 1-3	400.3	20.4	4.20	0.68	64.2	68.5

LB#4095-Beaurox Mines Limited

Test#PP-1

Purpose: To recovery a gold bearing sulphide concentrate using column flotation and gravity concentration.

Procedure: A bulk sample was prepared by pulping it to approximately 65% solids and removing the +28 mesh material prior to storage in a 1500 L conditioner. The -28 mesh fraction was than lightly reground in a Denver ball mill before being further diluted to approximately 35% solids in a small 10 L conditioner. This comprised the feed to the first of two 6 inch column flotation cells. The first column was fed at a rate of 3.7 L/minute slurry. The rougher tail from the first column was used to feed the second column. Rougher concentrates from both 6 inch columns were combined to feed a 4 inch cleaner column cell. Tailing from the cleaner column was recirculated to the head of the flotation circuit while the cleaner concentrate was collected. Tailing from the second 6 inch column was passed through a Falcon separator. A Falcon concentrate and tailing were collected and assayed.

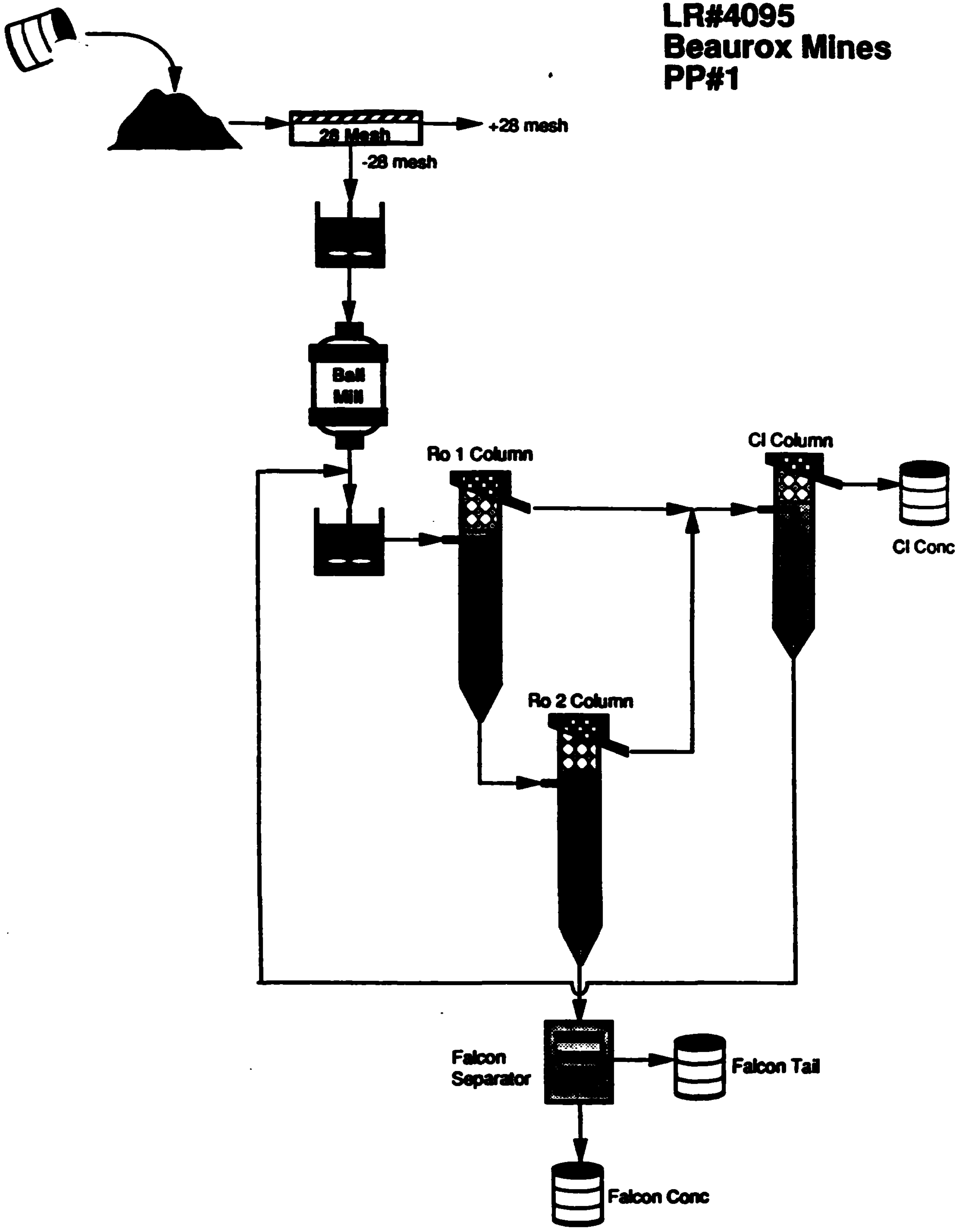
Sample: Bankfield Bulk Sample (-28 mesh)

Flowsheet: See next page.

Equipment List: 2 X 150 mm flotation columns, 57 and 72 litres in capacity
100 mm flotation column, 28 litres in capacity
Denver ball mill, 305 mm X 610 mm, 1.5 kW
Denver conditioner, 1200 mm X 1520 mm, 1500 L
760 mm diameter Sweco Vibro Energy Separator, 28 mesh deck
Falcon Separator, Model B-5

Results:

LR#4095
Beaurox Mines
PP#1



Reagents

Point of Addition	Reagent Name	Solution Strength %	Rate mL/min or drops/min	Rate g/t	Feed Rate t/h
Ball Mill Feed	Na2S	10	8.7	661	0.079
Flotation Conditioner	A350	2	10.5	159	
	R412	100	20.0	115	
	CuSO4	10	9.0	684	
	DF250	2	1.1	17	

Column Flotation Conditions

(i) Bougher 1 Column

(i) Column Cell Data

Diameter	150 mm
X-sectional Area	177 cm2
Total Height	272 cm
Total Volume	80 L

(ii) Operating Parameters

Gas	9.8 L/min
Wash	2.12 L/min
Feed Rate	3.01 L/min
Level	75 cm
PXD	41 mV
Operating Volume	72 L

(iii) Key Variables

Gas Velocity	0.92 cm/sec
Wash Velocity	0.20 cm/sec
Feed Velocity	0.28 cm/sec
NRT	27 minutes
Flow Bias	27 %
Gas Hold Up	16 %

Column Flotation Conditions (continued)

(ii) Rougher 2 Column

(i) Column Cell Data

Diameter	150 mm
X-sectional Area	177 cm2
Total Height	396 cm
Total Volume	72 L

(ii) Operating Parameters

Gas	6.7 L/min
Wash	1.06 L/min
Feed Rate	2.94 L/min
Level	30 cm
PXD	53 mV
Operating Volume	57 L

(iii) Key Variables

Gas Velocity	0.63 cm/sec
Wash Velocity	0.10 cm/sec
Feed Velocity	0.28 cm/sec
NRT	25 minutes
Flow Bias	26 %
Gas Hold Up	17 %

Assays

Stream	Product	Au, g/t	S, %
Ball Mill Discharge	BMD	1.55	
Rougher 1 Feed	R1F	1.63	3.55
Rougher 1 Conc	R1C	11.3	
Rougher 2 Conc	R2C	10.4	
Combined Rougher Conc	CRC	10.2	33.2
Cleaner Conc	CLC	11.7	
Rougher 1 Tailing	R1T	1.39	
Rougher 2 Tailing	R2T	1.13	1.69
Cleaner Tailing	CLT	7.30	
Falcon Conc	FLC	24.0	16.2
Falcon Tailing	FLT	1.08	1.66

Metallurgical Balance (2 product formula)

Product	Weight %	Assay, g/t or % Au	S	Distribution, % Au	S
CLC	4.0	11.7		30.0	
R2T	96.0	1.13		70.0	
BMD(calc)	100.0	1.55		100.0	
BMD(assay)		1.55			
BMD	100.0	1.55		100.0	
CLT	1.4	7.30		6.6	
R1F(calc)	101.4	1.63		106.6	
R1F(assay)		1.63			
R1C	2.5	11.3		17.9	
R1T	99.0	1.39		88.7	
R1F(calc)	101.4	1.63		106.6	
R1F(assay)		1.63			
R2C	2.8	10.4		18.6	
R2T	96.2	1.13		70.1	
R1T(calc)	99.0	1.39		88.7	
R1T(assay)		1.39			
R1C	2.5	11.3		17.9	
R2C	2.8	10.4		18.6	
CRC(calc)	5.2	10.8		36.5	
CRC(assay)		10.2			
CLC	4.0	11.7		30.0	
CLT	1.4	7.30		6.6	
CRC(calc)	5.4	10.5		36.6	
CRC(assay)		10.2			
FLC	0.2	24.0	16.2	3.1	
FLT	96.0	1.08	1.66	67.0	
R2T(calc)	96.2	1.13	1.69	70.1	
R2T(assay)		1.13	1.69		

Overall Metallurgical Balance

Product	Weight %	Assay, g/t or % Au	Assay, g/t or % S	Distribution, % Au	Distribution, % S
CLC	4.0	11.7		30.0	
FLC	0.2	24.0		3.1	
FLT	96.0	1.08		67.0	
R1F(calc)	100.2	1.55		100.0	
R1F(assay)		1.55			

Screen Analyses

(i) Ball Mill Feed

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.2	0.1	0.1	99.9
100	1.9	1.0	1.1	98.9
150	18.4	9.6	10.7	89.3
200	36.9	19.3	30.1	69.9
270	32.8	17.2	47.2	52.8
400	20.1	10.5	57.8	42.2
-400	80.6	42.2	100.0	-
Total	190.9	100.0	-	-

(ii) Ball Mill Discharge

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.0	0.0	0.0	100.0
100	0.6	0.4	0.4	99.6
150	7.8	4.6	5.0	95.0
200	25.0	14.8	19.8	80.2
270	30.3	17.9	37.7	62.3
400	21.8	12.9	50.6	49.4
-400	83.4	49.4	100.0	-
Total	168.9	100.0	-	-

(iii) Cleaner Conc

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.0	0.0	0.0	100.0
100	0.1	0.1	0.1	99.9
150	0.7	0.3	0.4	99.6
200	5.3	2.2	2.6	97.4
270	34.5	14.4	17.0	83.0
400	77.2	32.2	49.1	50.9
-400	122.2	50.9	100.1	-
Total	240.0	100.1	-	-

Screen Analyses (continued)

(iv) Rougher 1 Conc

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.0	0.0	0.0	100.0
100	0.1	0.1	0.1	99.9
150	0.2	0.3	0.4	99.6
200	1.4	1.8	2.2	97.8
270	7.5	9.6	11.7	88.3
400	19.4	24.7	36.5	63.5
-400	49.8	63.5	100.0	-
Total	78.4	100.0	-	-

(v) Rougher 2 Conc

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.0	0.0	0.0	100.0
100	0.1	0.3	0.3	99.7
150	0.4	1.1	1.4	98.6
200	1.6	4.5	5.9	94.1
270	5.2	14.6	20.5	79.5
400	8.3	23.3	43.8	56.2
-400	20.0	56.2	100.0	-
Total	35.6	100.0	-	-

LB#4095-Beaurox Mines Limited

Test#PP-2

Purpose: To recovery a gold bearing sulphide concentrate using column flotation and gravity concentration.

Procedure: A bulk sample was prepared by pulping it to approximately 65% solids and removing the +28 mesh material prior to storage in a 1500 L conditioner. The -28 mesh fraction was than lightly reground in a Denver ball mill before being further diluted to approximately 35% solids in a small 10 L conditioner. This comprised the feed to the first of two 6 inch column flotation cells. The first column was fed at a rate of 3.7 L/minute slurry. The rougher tail from the first column was used to feed the second column. Rougher concentrates from both 6 inch columns were combined and collected for assay. Tailing from the second 6 inch column was passed through a Falcon separator. A Falcon concentrate and tailing were collected and assayed.

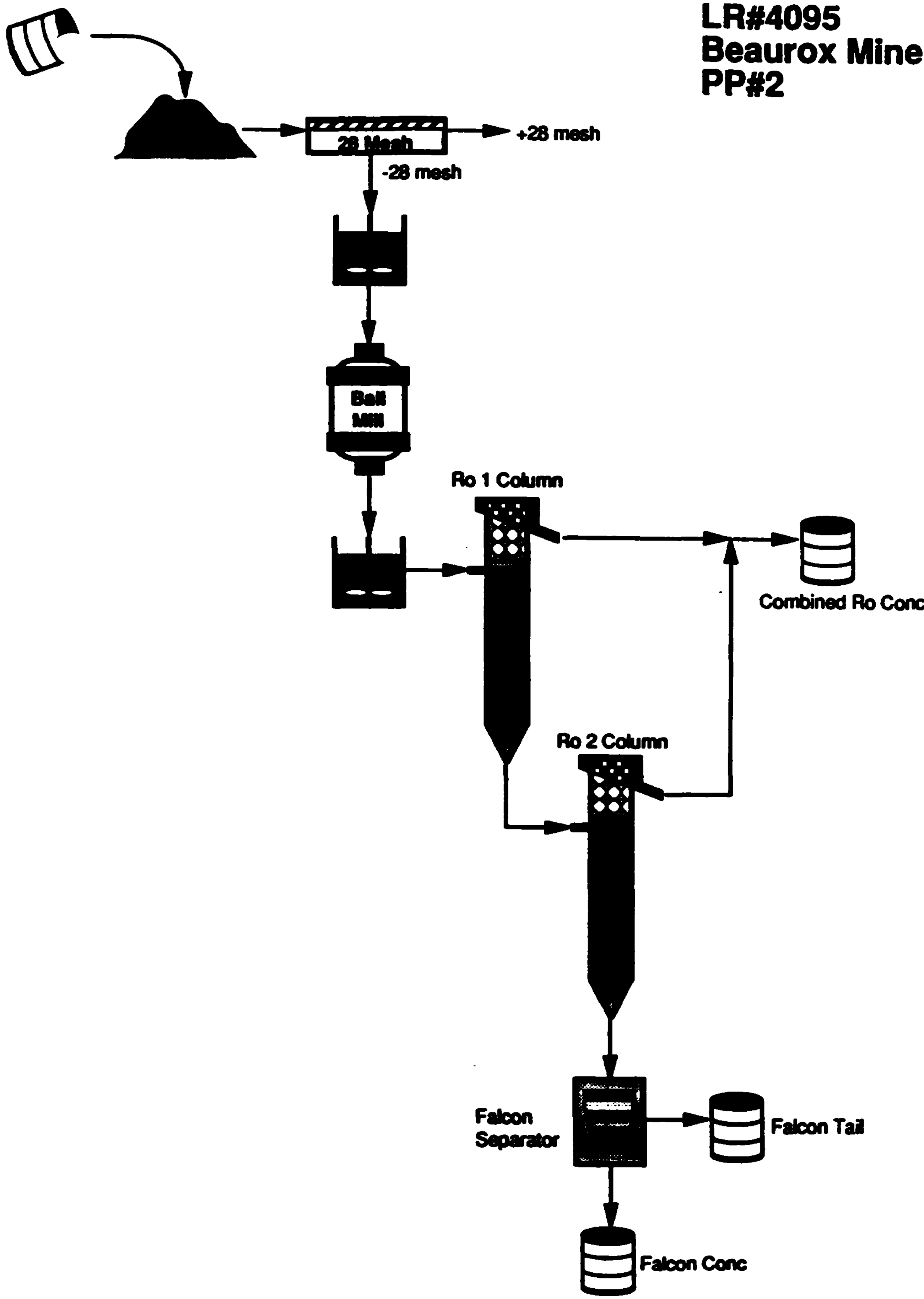
Sample: Bankfield Bulk Sample (-28 mesh)

Flowsheet: See next page.

Equipment List: 2 X 150 mm flotation columns, 57 and 72 litres in capacity
Denver ball mill, 305 mm X 610 mm, 1.5 kW
Denver conditioner, 1200 mm X 1520 mm, 1500 L
760 mm diameter Sweco Vibro Energy Separator, 28 mesh deck
Falcon Separator, Model B-5

Results:

LR#4095
Beaurox Mines
PP#2



Reagents

Point of Addition	Reagent Name	Solution Strength %	Rate mL/min or drops/min	Rate g/t	Feed Rate t/h
Ball Mill Feed	Na2S	10	7.3	644	0.068
Flotation Conditioner	A350	2	14.0	247	
	R412	100	23.3	156	
	CuSO4	10	7.0	618	
	DF250	2	1.5	26	

Column Flotation Conditions

(i) Rougher 1 Column

(i) Column Cell Data

Diameter	150 mm
X-sectional Area	177 cm2
Total Height	272 cm
Total Volume	80 L

(ii) Operating Parameters

Gas	12.0 L/min
Wash	2.12 L/min
Feed Rate	3.62 L/min
Level	53 cm
PXD	41 mV
Operating Volume	72 L

(iii) Key Variables

Gas Velocity	1.13 cm/sec
Wash Velocity	0.20 cm/sec
Feed Velocity	0.34 cm/sec
NRT	22 minutes
Flow Bias	32 %
Gas Hold Up	22 %

Column Flotation Conditions (continued)

(ii) Rougher 2 Column

(i) Column Cell Data

Diameter	150 mm
X-sectional Area	177 cm2
Total Height	396 cm
Total Volume	72 L

(ii) Operating Parameters

Gas	7.0 L/min
Wash	1.06 L/min
Feed Rate	3.52 L/min
Level	35 cm
PXD	53 mV
Operating Volume	57 L

(iii) Key Variables

Gas Velocity	0.66 cm/sec
Wash Velocity	0.10 cm/sec
Feed Velocity	0.33 cm/sec
NRT	20 minutes
Flow Bias	30 %
Gas Hold Up	16 %

Assays

Stream	Product	Au, g/t	S, %
Ball Mill Discharge	BMD	1.54	3.23
Rougher 1 Conc	R1C	9.49	
Rougher 2 Conc	R2C	7.95	
Combined Rougher Conc	CRC	8.97	27.1
Rougher 1 Tail	R1T	1.30	
Rougher 2 Tail	R2T	0.89	1.11
Falcon Conc	FLC	12.5	17.2
Falcon Tail	FLT	0.84	1.04

Metallurgical Balance (2 product formula)

Product	Weight	Assay, g/t or %		Distribution, %	
	%	Au	S	Au	S
R1C	2.9	9.49		18.1	
R1T	97.1	1.30		81.9	
BMD(calc)	100.0	1.54		100.0	
BMD(assay)		1.54			
R2C	5.6	7.95		29.1	
R2T	91.4	0.89		52.8	
R1T(calc)	97.1	1.30		81.9	
R1T(assay)		1.30			
R1C	2.9	9.49		18.1	
R2C	5.6	7.95		29.1	
CRC(calc)	8.6	8.48		47.2	
CRC(assay)		8.97			
CRC	8.6	8.48	27.1	47.2	69.6
R2T	91.4	0.89	1.11	52.8	30.4
BMD(calc)	100.0	1.54	3.34	100.0	100.0
BMD(assay)		1.54	3.23		
FLC	0.4	12.5	17.2	3.2	2.1
FLT	91.0	0.84	1.04	49.6	28.4
R2T(calc)	91.4	0.89	1.11	52.8	30.4
R2T(assay)		0.89	1.11		

Overall Metallurgical Balance

Product	Weight %	Au	S	Au	S
CRC	8.6	8.48	27.1	47.2	69.6
FLC	0.4	12.5	17.2	3.2	2.1
FLT	91.0	0.84	1.04	49.6	28.4
BMD(calc)	100.0	1.54	3.34	100.0	100.0
BMD(assay)		1.54	3.23		

Screen Analyses

(i) Ball Mill Feed

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.2	0.1	0.1	99.9
100	1.5	0.9	1.0	99.0
150	13.8	7.9	8.8	91.2
200	33.0	18.8	27.6	72.4
270	31.7	18.1	45.7	54.3
400	21.4	12.2	57.9	42.1
-400	74.0	42.1	100.0	-
Total	175.6	100.0	-	-

(ii) Rougher 1 Conc

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.0	0.0	0.0	100.0
100	0.0	0.0	0.0	100.0
150	0.3	0.7	0.7	99.3
200	1.2	2.9	3.6	96.4
270	4.4	10.5	14.1	85.9
400	8.9	21.3	35.4	64.6
-400	27.0	64.6	100.0	-
Total	41.8	100.0	-	-

(iii) Combined Rougher Conc

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.0	0.0	0.0	100.0
100	0.1	0.3	0.3	99.7
150	0.4	1.1	1.4	98.6
200	1.6	4.5	5.9	94.1
270	5.2	14.6	20.5	79.5
400	8.3	23.3	43.8	56.2
-400	20.0	56.2	100.0	-
Total	35.6	100.0	-	-

LR#4095-Beaurox Mines Limited

Test#PP-3

Purpose: To recovery a gold bearing sulphide concentrate using column flotation and gravity concentration.

Procedure: A bulk sample was prepared by pulping it to approximately 65% solids and removing the +28 mesh material prior to storage in a 1500 L conditioner. The -28 mesh fraction was than lightly reground in a Denver ball mill before being further diluted to approximately 35% solids in a small 10 L conditioner. This comprised the feed to the first of two 6 inch column flotation cells. The first column was fed at a rate of 3.7 L/minute slurry. The rougher tail from the first column was used to feed the second column. Rougher concentrates from both 6 inch columns were combined and collected for assay. Tailing from the second 6 inch column was passed through a Falcon separator. A Falcon concentrate and tailing were collected and assayed.

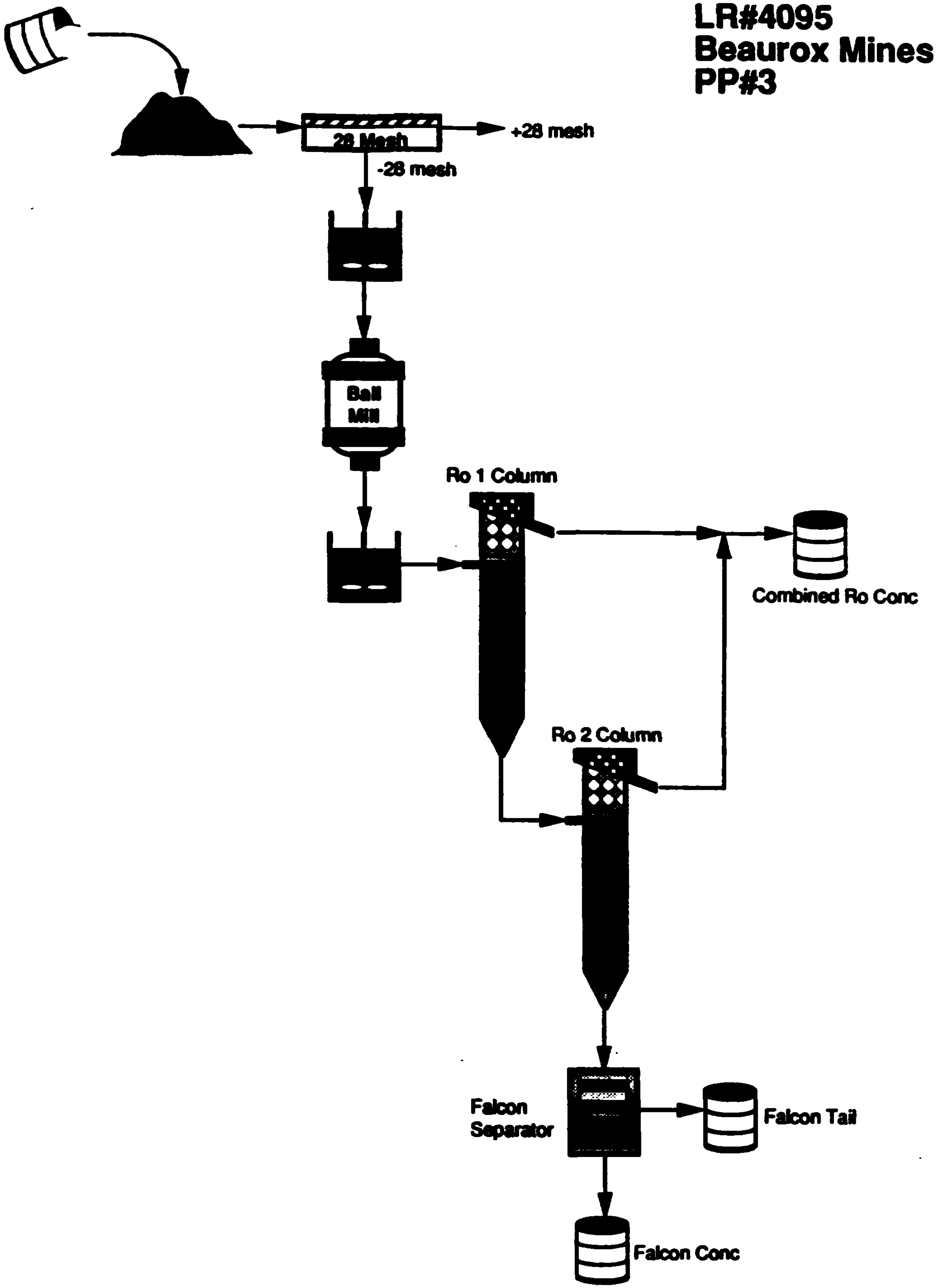
Sample: Little Long Lac Bulk Sample (-28 mesh)

Flowsheet: See next page.

Equipment List: 2 X 150 mm flotation columns, 57 and 72 litres in capacity
Denver ball mill, 305 mm X 610 mm, 1.5 kW
Denver conditioner, 1200 mm X 1520 mm, 1500 L
760 mm diameter Sweco Vibro Energy Separator, 28 mesh deck
Falcon Separator, Model B-5

Results:

LR#4095
Beaurox Mines
PP#3



Reagents

Point of Addition	Reagent Name	Solution Strength %	Rate mL/min or drops/min	Rate g/t	Feed Rate t/h
Ball Mill Feed	Na2S	10	8.1	600	0.081
Flotation Conditioner	A350	2	12.3	182	
	R412	100	20.0	113	
	CuSO4	10	8.0	593	
	DF250	2	2.2	33	

Column Flotation Conditions

(i) Rougher 1 Column

(i) Column Cell Data

Diameter	150 mm
X-sectional Area	177 cm2
Total Height	272 cm
Total Volume	80 L

(ii) Operating Parameters

Gas	7.9 L/min
Wash	2.12 L/min
Feed Rate	3.47 L/min
Level	45 cm
PXD	44 mV
Operating Volume	72 L

(iii) Key Variables

Gas Velocity	0.74 cm/sec
Wash Velocity	0.20 cm/sec
Feed Velocity	0.33 cm/sec
NRT	23 minutes
Flow Bias	31 %
Gas Hold Up	13 %

Column Flotation Conditions (continued)

(ii) Rougher 2 Column

(i) Column Cell Data

Diameter	150 mm
X-sectional Area	177 cm2
Total Height	396 cm
Total Volume	72 L

(ii) Operating Parameters

Gas	7.9 L/min
Wash	1.06 L/min
Feed Rate	3.40 L/min
Level	18 cm
PXD	56 mV
Operating Volume	57 L

(iii) Key Variables

Gas Velocity	0.74 cm/sec
Wash Velocity	0.10 cm/sec
Feed Velocity	0.32 cm/sec
NRT	21 minutes
Flow Bias	31 %
Gas Hold Up	13 %

Assays

Stream	Product	Au, g/t	S, %
Ball Mill Discharge	BMD	1.29	0.24
Rougher 1 Conc	R1C	12.1	6.64
Rougher 2 Conc	R2C	10.2	4.03
Combined Rougher Conc	CRC	12.9	5.93
Rougher 1 Tail	R1T	1.08	0.25
Rougher 2 Tail	R2T	1.04	0.16
Falcon Conc	FLC	30.3	9.43
Falcon Tail	FLT	0.98	0.14

Metallurgical Balance (2 product formula)

Product	Weight %	Assay, g/t or % Au	S	Distribution, % Au	S
R1C	1.9	12.1	6.64	17.9	34.0
R1T	98.1	1.08	0.25	82.1	66.0
BMD(calc)	100.0	1.29	0.37	100.0	100.0
BMD(assay)		1.29	0.24		
R2C	0.4	10.2	4.03	3.4	6.6
R2T	97.7	1.04	0.16	78.7	59.4
R1T(calc)	98.1	1.08	0.18	82.1	66.0
R1T(assay)		1.08	0.25		
R1C	1.9	12.1	6.64	17.9	34.0
R2C	0.4	10.2	4.03	3.4	6.6
CRC(calc)	2.3	11.8	6.16	21.3	40.6
CRC(assay)		12.9	5.93		
CRC	2.3	11.8	6.16	21.3	40.6
R2T	97.7	1.04	0.16	78.7	59.4
BMD(calc)	100.0	1.29	0.30	100.0	100.0
BMD(assay)		1.29	0.24		
FLC	0.2	30.3	9.43	4.7	7.2
FLT	97.5	0.98	0.14	74.0	52.2
R2T(calc)	97.7	1.04	0.16	78.7	59.4
R2T(assay)		1.04	0.16		

Overall Metallurgical Balance

Product	Weight %	Au	S	Au	S
CRC	2.3	11.8	6.16	21.3	40.6
FLC	0.2	30.3	9.43	4.7	7.2
FLT	97.5	0.98	0.14	74.0	52.2
BMD(calc)	100.0	1.29	0.30	100.0	100.0
BMD(assay)		1.29	0.24		

Screen Analyses

(i) Ball Mill Feed

Mesh	Weight g	% Weight Individual	% Weight Cumulative	% Weight Passing
65	3.4	2.2	2.2	97.8
100	13.3	8.6	10.8	89.2
150	23.1	14.9	25.6	74.4
200	22.8	14.7	40.3	59.7
270	17.8	11.5	51.8	48.2
400	11.5	7.4	59.2	40.8
-400	63.3	40.8	100.0	-
Total	155.2	100.0	-	-

(ii) Ball Mill Discharge

Mesh	Weight g	% Weight Individual	% Weight Cumulative	% Weight Passing
65	0.8	0.7	0.7	99.3
100	4.9	4.1	4.8	95.2
150	11.5	9.6	14.4	85.6
200	15.5	13.0	27.4	72.6
270	16.0	13.4	40.8	59.2
400	12.2	10.2	51.0	49.0
-400	58.6	49.0	100.0	-
Total	119.5	100.0	-	-

Screen Analyses (continued)

(iii) Rougher 1 Conc

Mesh	Weight g	% Weight Individual	% Weight Cumulative	% Weight Passing
65	0.0	0.0	0.0	100.0
100	0.0	0.0	0.0	100.0
150	0.6	0.6	0.6	99.4
200	1.6	1.5	2.1	97.9
270	3.6	3.4	5.5	94.5
400	5.6	5.3	10.8	89.2
-400	94.3	89.2	100.0	-
Total	105.7	100.0	-	-

(iv) Rougher 2 Conc

Mesh	Weight g	% Weight Individual	% Weight Cumulative	% Weight Passing
65	0.0	0.0	0.0	100.0
100	0.0	0.0	0.0	100.0
150	1.2	1.1	1.1	98.9
200	2.7	2.6	3.7	96.3
270	5.1	4.8	8.5	91.5
400	6.7	6.3	14.9	85.1
-400	90.0	85.1	100.0	-
Total	105.7	100.0	-	-

LR#4095-Beaurox Mines Limited

Test#PP-4

Purpose: To recovery a gold bearing sulphide concentrate using column flotation and gravity concentration.

Procedure: A bulk sample was prepared by pulping it to approximately 65% solids and removing the +28 mesh material prior to storage in a 1500 L conditioner. The -28 mesh fraction was than lightly reground in a Denver ball mill before being further diluted to approximately 35% solids in a small 10 L conditioner. This comprised the feed to the first of two 6 inch column flotation cells. The first column was fed at a rate of 3.7 L/minute slurry. The rougher tail from the first column was used to feed the second column. Rougher concentrates from both 6 inch columns were combined and collected for assay. Tailing from the second 6 inch column was passed through a Falcon separator. A Falcon concentrate and tailing were collected and assayed.

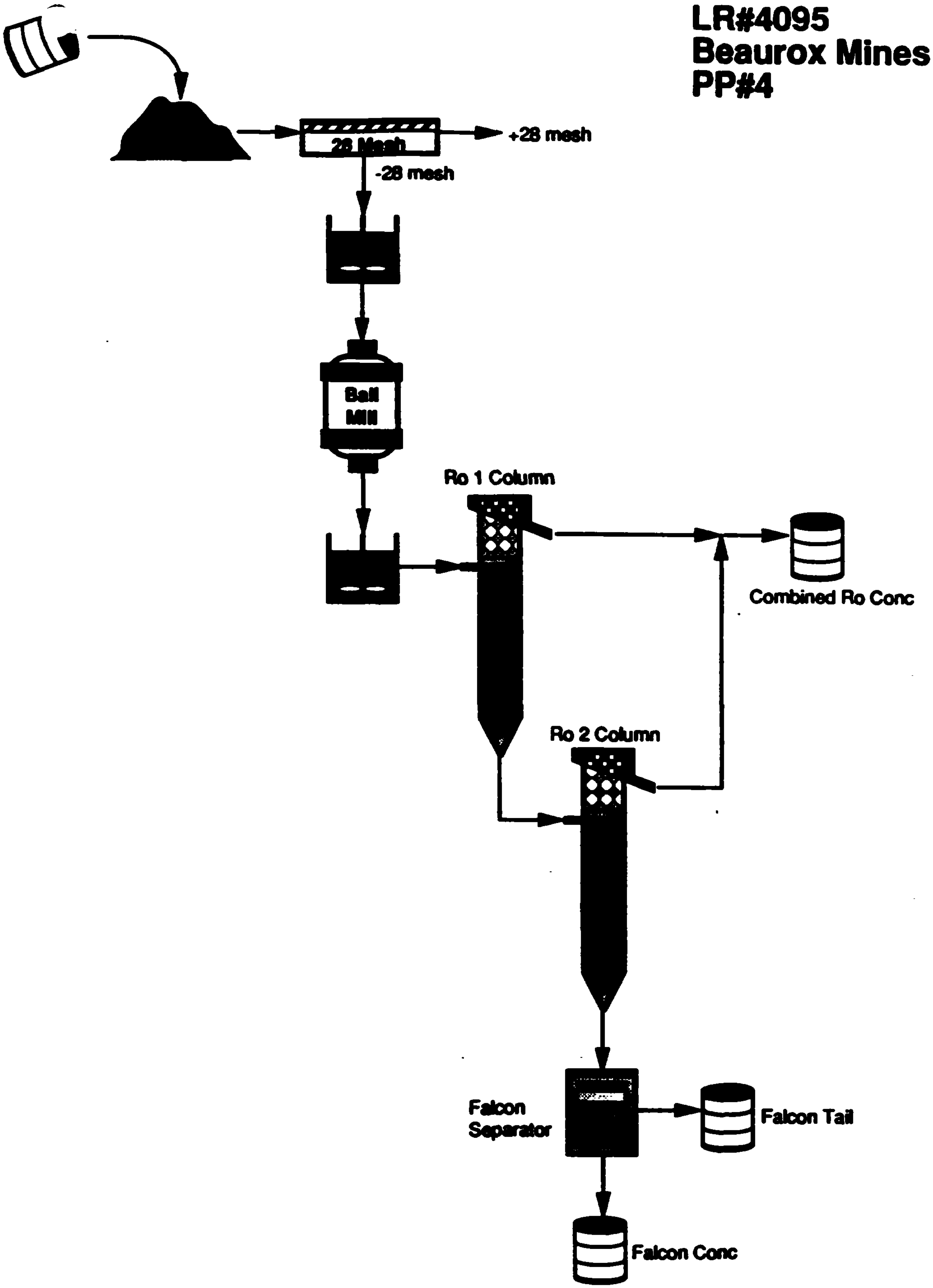
Sample: Little Long Lac Bulk Sample (-28 mesh)

Flowsheet: See next page.

Equipment List:
2 X 150 mm flotation columns, 57 and 72 litres in capacity
Denver ball mill, 305 mm X 610 mm, 1.5 kW
Denver conditioner, 1200 mm X 1520 mm, 1500 L
760 mm diameter Sweco Vibro Energy Separator, 28 mesh deck
Falcon Separator, Model B-5

Results:

LR#4095
Beaurox Mines
PP#4



Reagents

Point of Addition	Reagent Name	Solution Strength %	Rate mL/min or drops/min	Rate g/t	Feed Rate t/h
Ball Mill Feed	Na2S	10	8.3	778	0.064
Flotation Conditioner	A350	2	13.1	246	
	R412	100	21.0	150	
	CuSO4	10	6.5	609	
	DF250	2	2.6	49	

Column Flotation Conditions

(i) Rougher 1 Column

(i) Column Cell Data

Diameter	150 mm
X-sectional Area	177 cm2
Total Height	272 cm
Total Volume	80 L

(ii) Operating Parameters

Gas	7.9 L/min
Wash	2.12 L/min
Feed Rate	2.64 L/min
Level	45 cm
PXD	44 mV
Operating Volume	72 L

(iii) Key Variables

Gas Velocity	0.74 cm/sec
Wash Velocity	0.20 cm/sec
Feed Velocity	0.25 cm/sec
NRT	30 minutes
Flow Bias	23 %
Gas Hold Up	17 %

Column Flotation Conditions (continued)

(ii) Rougher 2 Column

(i) Column Cell Data

Diameter	150 mm
X-sectional Area	177 cm2
Total Height	396 cm
Total Volume	72 L

(ii) Operating Parameters

Gas	7.9 L/min
Wash	1.06 L/min
Feed Rate	2.62 L/min
Level	18 cm
PXD	56 mV
Operating Volume	57 L

(iii) Key Variables

Gas Velocity	0.74 cm/sec
Wash Velocity	0.10 cm/sec
Feed Velocity	0.25 cm/sec
NRT	28 minutes
Flow Bias	23 %
Gas Hold Up	17 %

Assays

Stream	Product	Au, g/t	S, %
Ball Mill Discharge	BMD	1.14	0.25
Rougher 1 Conc	R1C	17.5	10.0
Rougher 2 Conc	R2C	9.73	4.29
Combined Rougher Conc	CRC	10.1	4.58
Rougher 1 Tail	R1T	1.00	0.20
Rougher 2 Tail	R2T	0.92	0.15
Falcon Conc	FLC	20.4	1.04
Falcon Tail	FLT	0.84	0.15

Metallurgical Balance (2 product formula)

Product	Weight %	Assay, g/t or % Au	S	Distribution, % Au	S
R1C	0.8	17.5	10.0	13.0	30.0
R1T	99.2	1.00	0.20	87.0	70.0
BMD(calc)	100.0	1.14	0.28	100.0	100.0
BMD(assay)		1.14	0.24		
R2C	0.9	9.73	4.29	7.7	14.5
R2T	98.3	0.92	0.15	79.3	55.5
R1T(calc)	99.2	1.00	0.19	87.0	70.0
R1T(assay)		1.00	0.20		
R1C	0.8	17.5	10.0	13.0	30.0
R2C	0.9	9.73	4.29	7.7	14.5
CRC(calc)	1.7	13.5	7.06	20.7	44.5
CRC(assay)		10.1	4.58		
CRC	1.7	13.5	7.06	20.7	44.5
R2T	98.3	0.92	0.15	79.3	55.5
BMD(calc)	100.0	1.14	0.27	100.0	100.0
BMD(assay)		1.14	0.25		
FLC	0.4	20.4	1.04	7.2	1.6
FLT	97.9	0.84	0.15	72.1	53.9
R2T(calc)	98.3	0.92	0.15	79.3	55.5
R2T(assay)		0.92	0.15		

Overall Metallurgical Balance

Product	Weight %	Au	S	Au	S
CRC	1.7	13.5	7.06	20.7	44.5
FLC	0.4	20.4	1.04	7.2	1.6
FLT	97.9	0.84	0.15	72.1	53.9
BMD(calc)	100.0	1.14	0.27	100.0	100.0
BMD(assay)		1.14	0.25		

Screen Analyses

(i) Ball Mill Feed

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	4.2	2.5	2.5	97.5
100	13.6	8.1	10.6	89.4
150	24.4	14.5	25.1	74.9
200	23.4	13.9	39.0	61.0
270	20.3	12.1	51.1	48.9
400	15.0	8.9	60.0	40.0
-400	67.3	40.0	100.0	-
Total	168.2	100.0	-	-

(ii) Ball Mill Discharge

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.8	0.5	0.5	99.5
100	6.1	4.0	4.5	95.5
150	17.2	11.3	15.8	84.2
200	19.9	13.1	28.9	71.1
270	19.9	13.1	42.0	58.0
400	13.0	8.5	50.5	49.5
-400	75.3	49.5	100.0	-
Total	152.2	100.0	-	-

Screen Analyses (continued)

(iii) Rougher 1 Conc

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.0	0.0	0.0	100.0
100	0.0	0.0	0.0	100.0
150	0.7	1.9	1.9	98.1
200	1.6	4.3	6.2	93.8
270	3.6	9.7	15.9	84.1
400	4.5	12.1	28.0	72.0
-400	26.7	72.0	100.0	-
Total	37.1	100.0	-	-

(iv) Rougher 2 Conc

Mesh	Weight g	Individual	% Weight Cumulative	Passing
65	0.0	0.0	0.0	100.0
100	0.0	0.0	0.0	100.0
150	1.1	1.2	1.2	98.8
200	2.5	2.8	4.1	95.9
270	5.3	6.0	10.0	90.0
400	6.4	7.2	17.3	82.7
-400	73.3	82.7	100.0	-
Total	88.6	100.0	-	-

LR#4095-Beaurox Mines Limited

Test#PP-5

Purpose: To recovery a gold bearing sulphide concentrate using gravity concentration.

Procedure: A bulk sample was prepared by pulping it to approximately 65% solids and removing the +28 mesh material prior to storage in a 1500 L conditioner. The -28 mesh fraction was diluted to 35% solids and simultaneously pumped to the head of the spiral. The spiral concentrate was collected for assay purposes while the spiral midds were combined with fresh feed and pumped back to the head of the spiral. The spiral tail was passed through a Falcon separator where a Falcon concentrate and tail were collected and assayed.

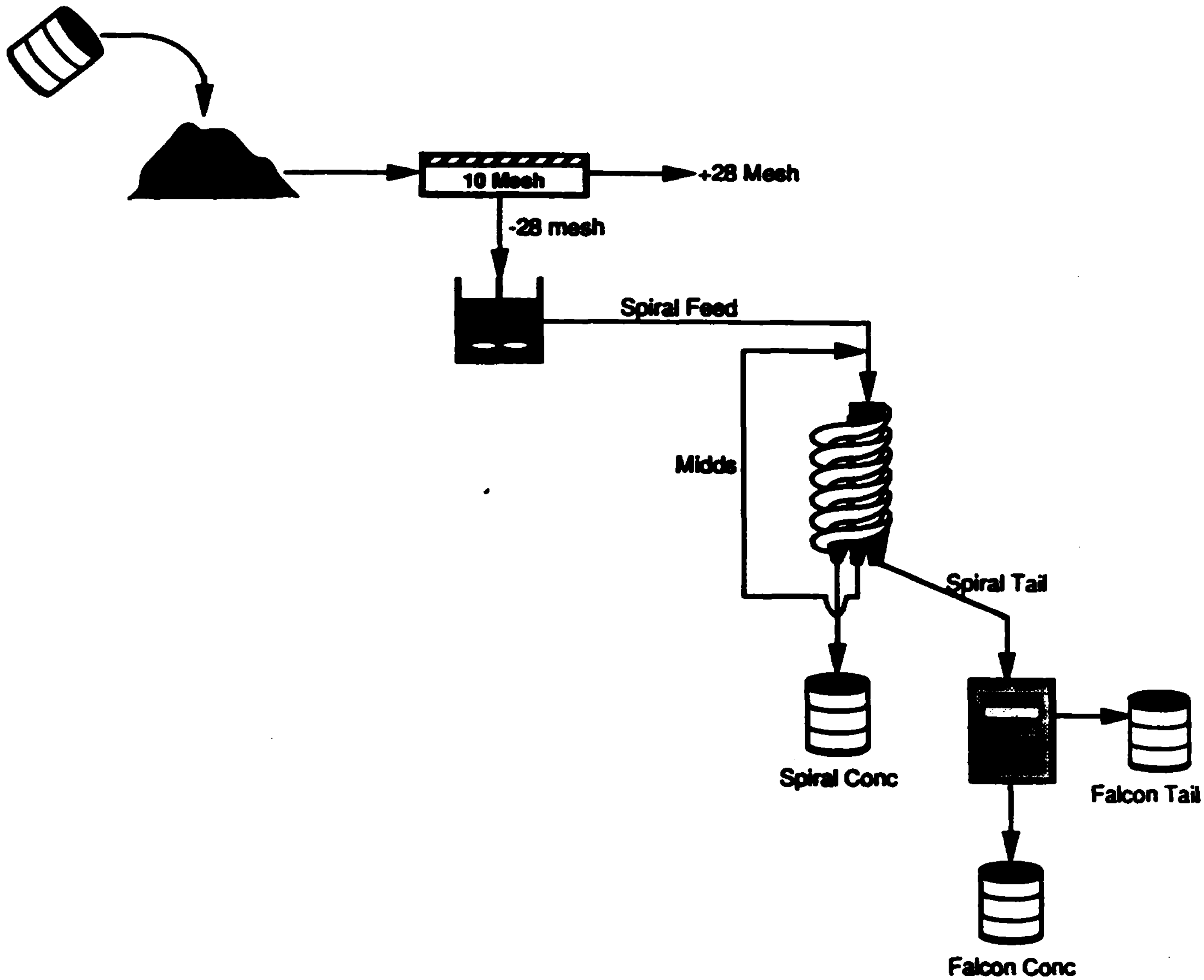
Sample: Bankfield Bulk Sample (-28 mesh)

Flowsheet: See next page.

Equipment List: Denver conditioner, 1200 mm X 1520 mm, 1500 L
Falcon Separator, Model B-5
Reichert Mark VII Spiral

Results:

LR#4095
Beaurox Mines
PP-5



Assays

Stream	Product	Au, g/t
Spiral Feed	SPF	1.45
Spiral Conc	SPC	4.81
Spiral Tailing	SPT	1.20
Falcon Conc	FLC	70.5
Falcon Tailing	FLT	1.05

Metallurgical Balance (two product formula)

Product	Weight %	Assay, g/t Au	Distribution Au
SPC	6.9	4.81	23.0
SPT	93.1	1.20	77.0
SPF(calc)	100.0	1.45	100.0
SPC(assay)		1.45	
FLC	0.2	70.5	9.7
FLT	92.9	1.05	67.3
SPT(calc)	93.1	1.20	77.0
SPT(assay)		1.20	

Overall Metallurgical Balance

SPC	6.9	4.81	23.0
FLC	0.2	70.5	9.7
FLT	92.9	1.05	67.3
SFD(calc)	100.0	1.45	100.0
SFD(assay)		1.45	

LB#4095-Beaurox Mines Limited

Test#PP-6

Purpose: To recovery a gold bearing sulphide concentrate using gravity concentration.

Procedure: A bulk sample was prepared by pulping it to approximately 65% solids and removing the +28 mesh material prior to storage in a 1500 L conditioner. The -28 mesh fraction was diluted to 35% solids and simultaneously pumped to the head of the spiral. The spiral concentrate was collected for assay purposes while the spiral midds were combined with fresh feed and pumped back to the head of the spiral. The spiral tail was passed through a Falcon separator where a Falcon concentrate and tail were collected and assayed.

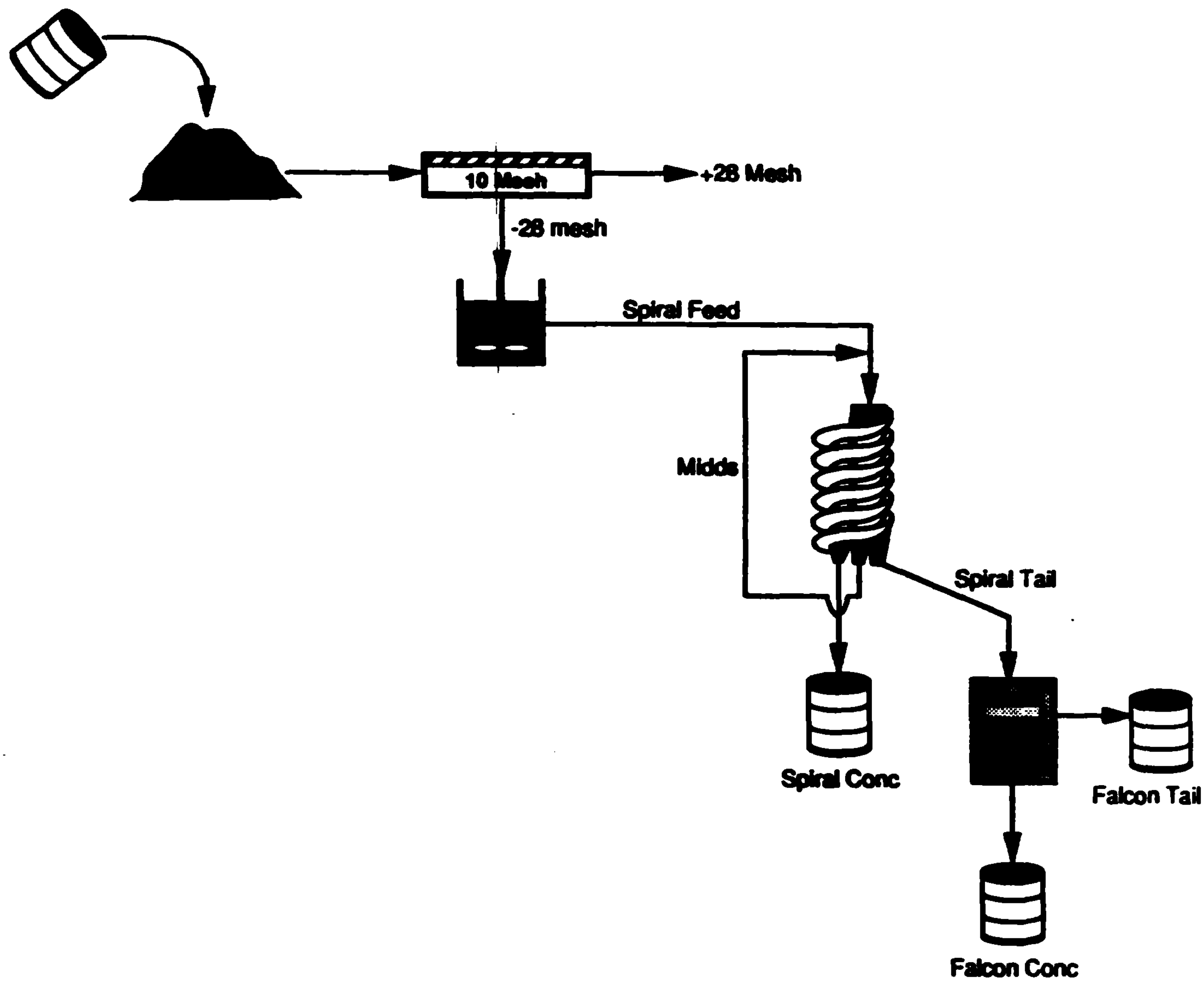
Sample: Little Long Lac Bulk Sample (-28 mesh)

Flowsheet: See next page.

Equipment List: Denver conditioner, 1200 mm X 1520 mm, 1500 L
Falcon Separator, Model B-5
Reichert Mark VII Spiral

Results:

LR#4095
Beaurox Mines
PP-6



Assays

Stream	Product	Au, g/t
Spiral Feed	SPF	1.19
Spiral Conc	SPC	6.57
Spiral Tailing	SPT	1.01
Falcon Conc	FLC	40.2
Falcon Tailing	FLT	0.85

Metallurgical Balance (two product formula)

Product	Weight %	Assay, g/t Au	Distribution Au
SPC	3.2	6.57	17.9
SPT	96.8	1.01	82.1
SPF(calc)	100.0	1.19	100.0
SPC(assay)		1.19	
FLC	0.4	40.2	13.5
FLT	96.4	0.85	68.6
SPT(calc)	96.8	1.01	82.1
SPT(assay)		1.01	

Overall Metallurgical Balance

SPC	3.2	6.57	17.9
FLC	0.4	40.2	13.5
FLT	96.4	0.85	68.6
SFD(calc)	100.0	1.19	100.0
SFD(assay)		1.19	

FEB - 3 1991

PROPOSAL

GOLD RECOVERY by COLUMN FLOTATION and GRAVITY SEPARATION for Beaurox Mines Limited

This proposal was prepared at the request of Mr. Dave Malouf of Beaurox Mines Limited.

Lakefield Research will undertake to perform the work described for the stated cost, given that the cost estimate is deemed accurate to within 20 percent, and provided that the program can be completed within the 1991 calendar year.

This proposal and cost estimate is submitted in confidence to Beaurox Mines Limited.

Jim Furey

James T. Furey
Senior Engineer

K. W. Sarbutt.

Keith W. Sarbutt
Manager - Mineral Processing

Lakefield Research
A Division of Falconbridge Limited

February 6th, 1991

TECHNICAL PROPOSAL

Introduction

It is proposed to conduct a preliminary evaluation of column flotation in order to determine its potential to improve precious metal grade and recovery. The proposal includes a cost estimate for a continuous pilot scale evaluation that is intended to supplement the on-going bench scale test program currently underway.

Current bench scale testing indicates that gravity separation by tabling recovered about 20% of the gold in 5% of the weight, at a grade of 5 to 10 g/t Au. Sulphide recovery was 25% to 35%. Falcon separator tests showed slightly higher gold recoveries of about 25%, at similar grades, but sulphide recovery was significantly lower at 10% to 15%. Flotation testing showed that up to 53% of the gold was recovered into 15% of the weight, at grades of about 5 g/t Au. Sulphide recoveries were significantly higher at 90%.

In discussions with Lakefield Research staff, it was concluded that the collection bowl in the Falcon Separator will fill efficiently with gold and sulphides, but that once the bowl is full, trading of gold for sulphides will be somewhat reduced, and gold losses may occur. The Falcon Separator might, therefore, perform more efficiently if sulphides were removed first. In flotation, gold recovery was more efficient that sulphide recovery, and it is therefore recommended that the sample should be treated first by flotation, to recover gold and sulphides, and that the flotation tailings then be treated in a Falcon Separator to recover incremental gold lost to flotation.

Bench scale flotation tests to compare performance with and without a polishing regrind indicated that recovery was improved to +60% with the regrind, and it is recommended that a light polishing regrind be included in any further testing.

Equipment and Facilities

Lakefield Research has available a variety of column flotation cells ranging in size from 50 mm diameter (9 liters) to 300 mm diameter (350 liters). All column cells operate with computer level control using a µMAC-6000 process control computer. In addition, the full range of conventional mineral processing can be provided, including gravity concentration by jig, spiral, shaking table and centrifugal jig (Falcon, Knelson and Kelsey).

In column flotation it is generally recommended that, for generation of adequate scale-up parameters, tests should be conducted in cells of at least 100 mm diameter.

Professional and Technical Staff

The core of column flotation expertise at Lakefield Research includes Dr. Bert J. Huls, P.Eng, Manager - Technology, James T. Furey, P.Eng, Senior Engineer, Steve R. Williams, Senior Project Engineer and Maria Falutsu, Metallurgist. The knowledge, skills and experience of this team covers virtually all aspects of column flotation, from pure research at the university level, through applied and industrial research to design, commissioning and operation of commercial column flotation installations.

Continuous Test Program

It is recommended to treat 2 x 2 ton samples at 150 kg/h in a continuous circuit consisting of a polishing regrind followed by sulphide flotation in a 150 mm diameter column cell of 60 liters capacity, with column cell tailings being treated in a Falcon Separator. A Reichart No. 7 Spiral will be included in the plant set up. This will provide for a short run evaluation of the Reichart spiral as a replacement for the column flotation stage. All products will be assayed for Au only, with multiple determinations on concentrates. In addition, the final concentrates will be submitted for multi-element scan by ICP (Inductively Coupled Plasma) spectrometer.

Project Schedule

It is anticipated that the continuous treatment program will require about two weeks to complete, followed by an additional one to two weeks for preparation of the final technical report.

Project Control

The project team will be managed by a Project Engineer, who will retain overall responsibility for the technical aspects of the program, will supervise individual tests and will ensure diligent and astute analysis of all test results. Ore preparation, sampling of the column cell and sample preparation will be the responsibility of a technician trained in all aspects of pilot scale column cell operation and sampling. Analytical results will be scrutinized by senior analytical staff as well as the metallurgical staff to ensure quality control.

Project Security

It is the policy of Lakefield Research that all analyses, data and reports paid for by the client are the property of the client, and will be maintained in strictest confidence.

PRICE PROPOSAL

The cost estimate detailed in the following pages is based on the following analytical and labour rates;

Professionals	$75.00 / hour
Technologists	$65.00 / hour
Technicians	$55.00 / hour

This proposal is based on our best estimate of the amount of work that should be required to complete the project, and the cost estimate is deemed accurate to within 20%. Invoices will reflect actual analytical service and labour usage required by the program, and any savings resulting from early success in the test program will be passed along to the client. Any changes in scope dictated by prior testing results will be made only after consultation with the client and subject to his approval. Under no circumstances will the client be invoiced for problems or delays which are beyond his ability to control. Lakefield Research will retain the right to schedule testing so as to maximize the efficiency of the Lakefield Research facilities, and will undertake to ensure that the needs of the project will be diligently served.

Cost Estimate - Continuous Testing

Based on 150 kg/h throughput using 150 mm column cell and Falcon separator.

Sample preparation		$ 1,040
Circuit Preparation		2,080
Circuit Operation		
Metallurgist	$ 600	
Column Cell Operator	520	
Product Handler / Sampler	440	
Sample Assay Preparation	260	
Assays (10 x Au only)	120	
Screen Analyses (1 each)	50	
Cost per Test	$ 1,990	
Total of Four Tests (2 samples, 2 tests each)		7,960
Circuit Dismantling		1,040
Multi-element ICP Scan on circuit concentrates		100
Reporting and Supervision		1,800
Total Estimated Cost (+/- 20%)		**$ 14,020**

Lakefield Research
A Division of Falconbridge Ltd
Lakefield, Ontario
February 6th, 1991 / jtf

INVOICE

LAKEFIELD RESEARCH
A DIVISION OF FALCONBRIDGE LIMITED
Postal Bag 4300, 185 Concession St., Lakefield, Ontario K0L 2H0
Phone: (705) 652-3341 Telex No. 06 962842
Fax No. (705) 652-6365

No.: 32110

DATE February 6 19 91

G.S.T. NUMBER R101733426

TO: Beaurox Mines Limited
801, 80 Richmond Street, West
TORONTO, Ontario
M5H 1A4

Mr. Dave Malouf

Our Project L.R. 4095 - Ongoing Testwork

Re: Column Flotation and Gravity Separation

Sample Preparation		$ 1,040.00
Circuit Preparation		2,080.00
Circuit Operation		7,960.00
Circuit Dismantling		1,040.00
Multi-element ICP Scan on circuit concentrates		100.00
Reporting and Supervision		1,800.00
		$ 14,020.00
	G.S.T.	981.40
	Total	$ 15,001.40

BEAUROX MIN
Suite 801, 80 Richmond St. West, Toronto, Ontario M
Fax (416) 360-7355

FINAL REPORT

PHASE I

SAMPLING AND METALLURGICAL REPORT

BEAUROX MINES LIMITED

GERALDTON-BEARDMORE AREA ONTARIO

TASHOTA-NIPIGON DEPOSIT

TOMBILL-BANKFIELD DEPOSIT

LITTLE LONG LAC DEPOSIT

By

DAVID MALOUF
General Manager

March 19, 1991

FIELD OFFICE: P.O. Box 730, Geraldton, Ontario, P0T 1M0 Telephone: (807) 854-0441
Fax (807) 854-0997

INDEX

Report Beaurox Mines Limited

by

David Malouf

March 1991

Proposal Metallurgical Work by Lakefield Research

Location Maps

Assay Maps with Preliminary Reserve (in back packet)

LAKEFIELD RESEARCH

THE RECOVERY OF GOLD

from low grade tailings samples

submitted by

BEAUROX MINES LIMITED

Progress Report No. 3

Bound and submitted separately.

BEAUROX MINES LIMITED

Suite 801, 80 Richmond St. West, Toronto, Ontario M5H 2A4 Telephone: (416) 860-1636 or 860-1701
Fax (416) 360-7355

SUMMARY REPORT

Introduction

1. As outlined in the President's letter of the 1989 Roxmark Mines Limited. Annual Report, Roxmark has secured the right to evaluate several gold tailings deposits in the Geraldton Beardmore Camp. Of the 3,000,000 Tons under agreement, preliminary testing has shown that a thorough evaluation is warranted and that the deposits could contain up to 135,000 ounces of gold for an average grade of plus or minus .045 ounces per ton. Assuming a 65% recovery and reprocessing costs in the order of $8.00 per ton on a scale of plus 1000 tons per day this project could generate the much needed exploration and development funds needed to develop the recent discoveries in the camp specifically Roxmark's Benedict Zone, the Hardrock Discovery Zone , etc. - It is also believed that capital costs required would be a fraction of the cost required to implement a conventional mine, mill scenario possible 25-30% - and that this equipment could later be supplemented to handle mine-run ore.

2. There are three properties of prime interest: .

   A). Bankfield - Tombill with a common tailings pond
   B). Little Long Lac property of Algoma Steel.
   C). Tashota Nipigon.

3. All properties are in the Beardmore Geraldton Mining Division in the District of Thunder Bay.

   A) <u>Bankfield-Tombill</u>
   Located on the North side of Trans Canada Highway #11-8.25Km west of the turn off to Geraldton in the Western half of Errington Twp.

   B) <u>Little Long Lac</u>
   Located on either side of Hwy 584 approximately 3 Km. north of Hwy 11 south of the bridge that enters the town of Geraldton in Errington and Ashmore Twps.

   C) <u>Tashota</u>
   The Mine is located North west of Onamen Lake and south of Obashkegan Township, between Onamen Lake and Onamen River. It is accessible via the Camp 40 Road north to the Con Lake Road. Proceed north-east from the Con Lake intersection on the Mine Road for approximately 8 miles to the mine site.

/...2

FIELD OFFICE: P.O. Box 730, Geraldton, Ontario, P0T 1M0 Telephone: (807) 854-0441
Fax (807) 854-0997

4\. A) <u>Bankfield-Tombill Claims</u>
3 Patented Mining Claims
TB 10213
TB 110201
TB 10645

B) <u>Little Long Lac</u>
8 Patented Mining Claims
TB 10887
TB 10621
TB 10560
TB 10561
TB 10562
TB 10563
TB 10886
TB 10566

C) <u>Tashota Nipigon</u>
2 Patented Mining Claims
KK523 & KK524

5\. A) The Bankfield Tombill project is subject to a 25% NPI in favor of Bankfield and Tombill Mines.

B) The Little Long Lac project is subject to a 5% N.S.R. to Lac Minerals and annual payments of approximately $40,000 to Algoma subject to a possitive production decision.

C) The Tashota Nipigon project is subject to a 4% NSR and an additional payment of $5,000.00

6\. <u>Regional Geology</u>

S.E. Malouf Consulting Geologists Limited entered into agreements on the above properties for Roxmark Mines Limited. Roxmark did the initial work involving research sampling and preliminary metallurgical work involving $15,000 in 1989 and early 1990 - Roxmark has agreed to give their subsidiary company Beaurox Mines Limited (at present a private corporation) a chance to earn a 25% interest in the tailings project for doing a proper evaluation, metallurgical testing, feasibility study, and a further 25% interest for funding through to production.

7\. <u>Current Status</u>

A). Bankfield Tombill - Initial sampling with a Sonic Soil Sample @ 60 holes drilled indicated appreciable tonnage @ .051 Oz. per Ton. with isolated tonnage of high grade.

B) Little Long Lac - Investigation of production history indicates excessives losses in the mill with two periods of tailings retreatment. Initial sampling favorable - 1,780,000 Ton potential.

C). Tashota Nipigon - Report on sampling and metallurgy from Lakefield Research done by Lynx - Canada in 1978 indicates reserves of 50,000 tons of .088 Oz Gold per ton with indicated recoveries of 70.4%

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8. Recommended Work and Scope of Project

A program involving the expenditure of $200,000.00 is warranted. Grids will be established on all properties. The Tashota will be drilled and sampled on 25 foot centre because of the relatively small size of the deposit and high grade nature. The Bankfield Tombill will be drilled and sampled on 50 foot centers - Little Long Lac will be drilled and sampled on 100 foot centers with later definition at 50 feet. This should involve 10,000 to 11,000 feet of drilling and approximately 2400 assays - Sample results and locations will be plotted and grade contour lines established to locate economic reserves. The grid, drilling and sampling will cost $100,000.00 - Sample composites of reject samples will assembled and sent to Lakefield Research for metallurgical studies on the three representative bulk sample.

The metallurgical work should cost $40,000 - If this stage gives favourable results, it will be followed with a $50,000 environmental study and then a feasibility study.

9. The project began in August of 1990

10. The project will take nine months to complete or 200 days.

11. Work Completed

The proposed program was carried out at a cost of $165,911.54 - Grids were establish on each property and the drilling was done with a Sonic Soil Sampling machine - "BQ" Rods were used to drill down through the tailings and into organic material. Samples were taken at each five foot section.

A total of 11,000 feet were drilled and 2,621 samples taken. The assay results were plotted on assay plans. These results were then contoured to show areas averaging .03 ounces of gold per ton and better. Once these areas were known the sample rejects representing these areas were made into composites for each deposit and prepared for bulk metallurgical work at Lakefield. Supplementary bulk samples were taken with the use of a back hoe which cut five trenches on each of the Bankfield and the Little Long Lac deposits. The trenches were 50 feet long, the depth of the tailings and a two ton sample representative sample compiled from each deposit. There was sufficient material for testing on the Tashota property as each hole was double drilled.

All composites properly identified were shipped to Lakefield research and arrived December 27th, 1990.

Metallurgical work began in early January with preliminary investigations on gravity, flotation, bottle cyanide tests, 30 elements scans etc. After a review of initial results a decision was made to do heap leach column tests on all three ores and to do a combination of gravity (Falcon concentrator) and column flotation in a continues circuit on the Bankfield and Little Long Lac material.

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12. Reserves

Reserve calculations on accompanying maps are preliminary and cannot be properly addressed until the metallurgical results of future work demonstrate what the economical cut off grade will be. An attempt was made to evaluate each occurrence completely. However the tonnages sampled relate to historical production in the following manner:

	Historical	Sampled
Little Longlac	1,782,516	1,360,000 Tons or 76.3%
Bankfield-Tombill	419,631	265,190 Tons or 63.3%
Tashota	51,250	32,423 Tons or 63.25%

We would expect the sampled tailings to be 10 to 15% lower because the shallower fringe areas were not sampled or because of normal migration into water courses. In our tonnage calculations we used an overall tonnage factor of 19 cubic feet per ton. This may have been light especially on the Tashota-Nipigon and the combined Bankfield-Tombill where iron content was 10.1% and 6.4% respectively. Years of compaction and settling could also explain the difference.

Specific gravity tests will be done in an attempt to explain these differences and increase tonnages.

13. Preliminary Reserves

Preliminary reserves from known information:

	Tons	Grade	Ounces
1. Little Longlac			
A) at no cut off	1,360,000	.031	42,160 Oz.
B) at .025 cut off	928,000	.037	34,336 Oz.
C) at .03 cut off	532,000	.0425	22,610 Oz.
2. Bankfield-Tombill			
D) at no cut off	265,190	.0384	10,183.30 Oz
E) at .03 cut off	206,190	.044	9,072.40 Oz
3. Tashota-Nipigon			
F) at no cut off	32,423	.078	2,529 Oz
SUMMARY			
A+D+F	1,657,613	.033	54,872.3
B+E+F	1,164,613	.0394	45,937.
C+E+F	770,613	.0444	34,211.4

Note:

Beaurox Mines Ltd. plans to computarize all known assay data to be able to print out assay plans readily with varying grade contours.

14. Results and Conclusions

See Lakefield Research, Metallurgical Progress Report No. 3 and Proposal dated February 20th, 1991.

Tashota-Nipigon

The Heap Leach approach to reprocessing the Tashota-Nipigon tailings appears to be viable with preconditioning and agglomeration. Work will continue on the feasibility study and will include the variables of project costs, permits, environmental concerns etc. An intermediate step of a field pilot test will be examined to eliminate possible variances that could occur between test work and production experience, and to train personnel.

Bankfield and Little Long Lac

Results to date on the Bankfield and Little Long Lac deposits have been encouraging and informative but point out that further laboratory work is required to optimize recoveries. See Technical Proposal Lakefield, February 20, 1991; Bench scale testing indicates that gravity separation by tabling recovered about 20% of the gold in 5% of the weight at a grade of 5 to 10 g/t Au - Sulphide recovery was 25% to 35% Falcon separator tests showed slightly higher gold recoveries of about 25% at similar grades, but sulphide recovery was significantly lower at 10% to 15% - Flotation testing showed that up to 53% of the gold was recovered into 15% of the weight at grades of above 5 g/t Au - sulfides recoveries were significantly higher at 90% - Flotation tests to compare performance with and without a polishing regrind indicate that recovery was improved to + 60% with a regrind.

It was therefore reasonable to assume that a combination of flotation followed by the Falcon concentrator or viceversa could attain the targeted recoveries of + 70%

A continuous test was designed involving column flotation and the falcon concentrator. Column flotation was chosen over conventional flotation because of lower capital cost requirements, lower maintenance and overhead costs and the possibility of producing a higher grade concentrate. The test failed to achieve the results obtained in conventional flotation; possibly because of the characteristics of the ore, the limited scope of the test work, the fineness of the gold or because of the incompatibility of the ore with column flotation.

We propose to return to conventional flotation coupled with gravity concentration as this appears to be the logical next step. However as a prerequisite further bench scale flotation tests will be conducted to investigate new reagents schemes and optimize floatation conditions.

Results and Conclusions (cont.)

Heap leach tests on the Bankfield and Little Lac material gave 56% and 50.0% respectfully. Investigative work will be carried out on the rejects of this material to see if target recoveries can be attained through a combination heap leach followed by concentration of sulfides and heavy elements.

Specific gravity tests will be conducted on all three ores in an attempt to relate sampled tonnates with historical production records.

The proposal includes a mineralogical examination on feed and tailings samples. This will help identify gaugue components, liberation, association and potential recovery of the gold.

15. Future Program PHASE II

Once additional Metallurgical work is complete, we hope to demonstrate a viable project.

Expected Cost PHASE II-A

-Metallurgical Work - Lakefield proposal		$ 34,964.40
-Additional sample material required (representative) incl. extraction, freight and assays		20,000.00
-Environmental impact study		50,000.00
-Engineering and plant design		50,000.00
	SUB TOTAL:	$154,964.40

PHASE II-B

Trial production - Three tests with lots of 20,000 tons each mined and processed at an estimated cost of $10.00 per ton		$600,000.00
	TOTAL:	$754,964.40

16. Reports Available:

A) "Ontario Geological Survey"
Open file Report 5630 - 1986
Volume I - Pg. 87 thru 96
Pg. 329 Thru 338

Volume II - Pg. 582 thru 584
Pg. 571 item 8 Economic Features

Reports Available (cont.)

B) "An Investigation of the Recovery of Gold" from Tailings sample submitted by Tashota-Nipigon Mines Ltd. Progress Report No. 1

Project No. L.R. 2190 - Lakefield Research.

17. Metallurgical Report

"The Recovery of Gold from Low Grade Tailings Samples", submitted by Beaurox Mines Limited - Progress Report No. 3

18. Assay Maps with preliminary reserves (in back packet):

1 copy	Tombill	scale 1"- 50'
1 "	Bankfield	scale 1" - 100'
3 "s	Little Long Lac	scale 1" - 100' with varying cut off grades

19. Proposal by Lakefield Research February 20, 1991

20. Location Maps.

Respectfully,

David Malouf
General Manager and Director
Roxmark Mines Limited
Beaurox Mines Limited

PROPOSAL

GOLD RECOVERY
by GRAVITYSEPARATION and FLOTATION
for
BEAUROX MINES LIMITED

This proposal was prepared at the request of Mr. Dave Malouf of Beaurox Mines Limited.

Lakefield Research will undertake to perform the work described for the stated cost, given that the cost estimate is deemed accurate to within 20 percent, and provided that the program can be completed within the 1991 calendar year.

This proposal and cost estimate is submitted in confidence to Beaurox Mines Limited.

K.W. Sarbutt

Keith Sarbutt
Manager - Mineral Processing

Rene Jackman

Rene Jackman
Senior Project Metallurgist

Lakefield Research
A Division of Falconbridge Limited
February 20th, 1991

TECHNICAL PROPOSAL

Introduction

This proposal covers testwork to recover gold from the Bankfield and Little Long Lac tailing samples.

Based on the results of preliminary testwork reported in Progress Report No. 3, it is proposed to conduct further laboratory gravity separation and flotation testwork. The gravity separation tests would investigate the application of the Kelsey Jig to recover gold from the Bankfield and Little Long Lac samples directly as well as from the heap leach tailings from the same samples. Additional bench scale flotation tests would be conducted to try to improve the gold recovery achieved in the initial testwork.

Subject to the results of this laboratory testwork, a continuous pilot plant scale evaluation has been proposed incorporating gravity separation and conventional flotation.

Gravity Separation Testwork

Gold recovery using the Kelsey Jig will be investigated. Testwork on the Bankfield tailing has been proposed as part of an on-going Lakefield Research program. Costs for this work will be borne by Lakefield Research. If the results show promise, similar testwork will be conducted on the Little Long Lac tailing sample. In addition, Kelsey Jig tests will be conducted on the heap leach tailings after a slight regrind to break down the agglomerates.

Samples presently in storage will be used for this testwork.

Flotation Testwork

Bench scale flotation tests will be performed to investigate the effect of alternative collectors, modifiers, and pH on the recovery of gold in a bulk sulphide concentrate.

Samples presently in storage will be used for this testwork.

Continuous Pilot Plant Testwork

It is proposed to treat 2 tonne samples of the Bankfield and Little Long Lac tailings in a continuous circuit at a feedrate of 100 - 150 kg/h. The circuit will include a polishing regrind and conventional flotation. Gravity separation will be incorporated either on the flotation feed or flotation tailing using a Falcon Separator. Products will be assayed for gold and sulphur. Additional analyses may be conducted on the final concentrates. Also, multi-element ICP scans may be conducted on the final tailing water and solids for environmental considerations.

A minimum of a 2 tonne sample of each of the Bankfield and Little Long Lac samples will be required. It is proposed that a bench scale flotation test first be conducted on the piot plant feed samples to compare the response to the samples used in the laboratory testwork.

Project Schedule

The laboratory test program will take approximately 3 weeks to complete. It is anticipated that the continuous pilot scale program will require 2 weeks to complete. The final report will follow two weeks after the testwork is complete.

COST ESTIMATE

The cost estimate detailed in the following pages is based on our Analytical Fee Schedule for Metallurgical Testwork, and on the following labour rates;

Professionals	$100.00 / hour
Technologists	$70.00 / hour
Technicians	$55.00 / hour

These rates are firm for 1991, but are subject to revision in 1992. The proposal is based on our best estimate of the amount of work that should be required to complete the project. Invoices will reflect actual analytical service and labour usage required by the program. Any changes in scope dictated by prior testing results will be made only after consultation with the client and subject to his approval.

COST ESTIMATE

LABORATORY TESTWORK

Sample Preparation	**$615**
Specific Gravity Determinations - 3	**$26**
XRD on Head Samples to determine gangue minerals	**$150**
Flotation Testwork estimated 8 tests at $560 each	**$4480**
Kelsey Jig Testwork	
examine variables using Little Long Lac sample	**$948**
2 tests on heap leach residues	**$1314**
Supervision	**$700**
Project Management	**$400**
Report Preparation	**$560**
Miscellaneous, contingency	**$700**
TOTAL (GST not included)	**$10053**

CONTINUOUS PILOT PLANT TESTWORK

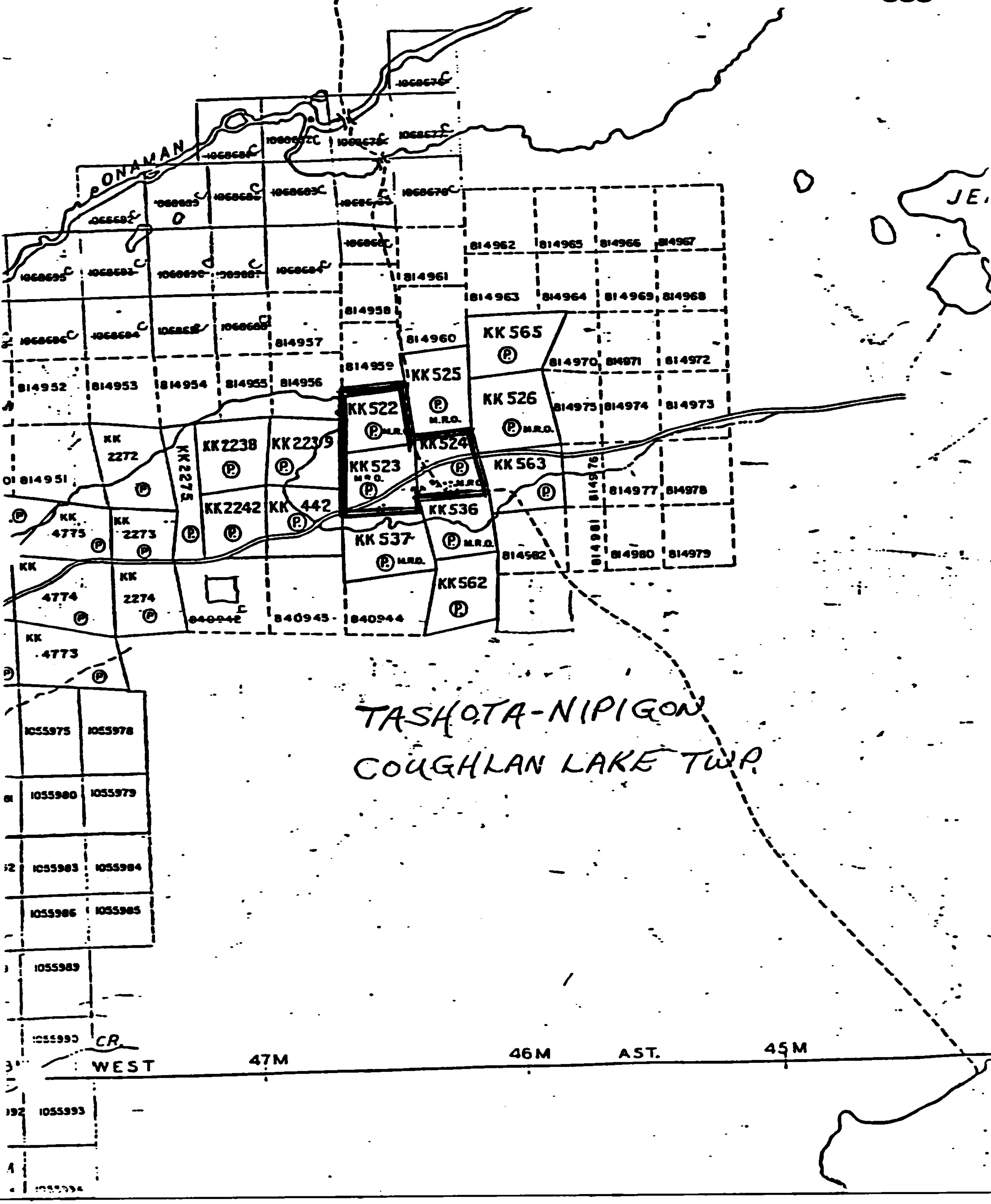
Sample Preparation			**$2240**
	2 tonnes per sample		
Laboratory Tests			**$800**
	to confirm flotation results on pilot plant feed		
Tailings Disposal			**$600**
	$150/t x 4		
Circuit Preparation			**$2240**
Circuit Clean-up			**$2000**
Circuit Operation			
	Metallurgist	$800	
	Foreman	$560	
	Operator	$440	
	Sample preparation	$260	
	Assays - 7 x Au, S	$196	
	Screen analyses	$50	
	Cost per test	$2306	
	4 tests		**$9224**
Additional Concentrate Analyses			**$200**
Equipment Depreciation			**$1200**
Environmental			
	tailing and tailing water analyses		**$500**
Project Management			**$1000**
Report Preparation			**$1120**
Miscellaneous, contingency			**$1500**
TOTAL (GST not included)			**$22624**

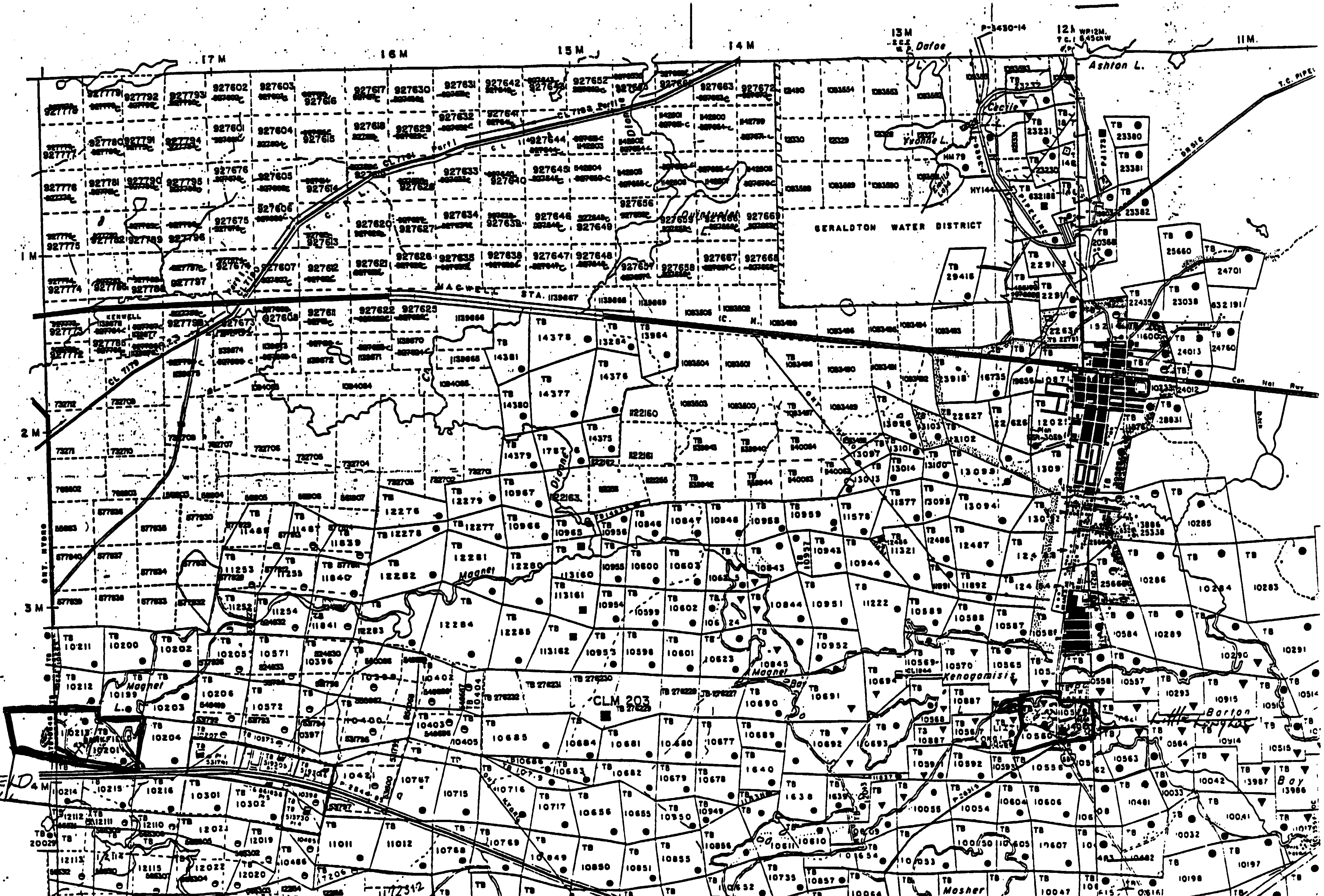
OVERALL ESTIMATE

Laboratory Testwork	**$10053**
Pilot Plant Testwork	**$22624**
TOTAL (GST not included)	**$32677**

42E10NW0113 63.6072 ASHMORE

900





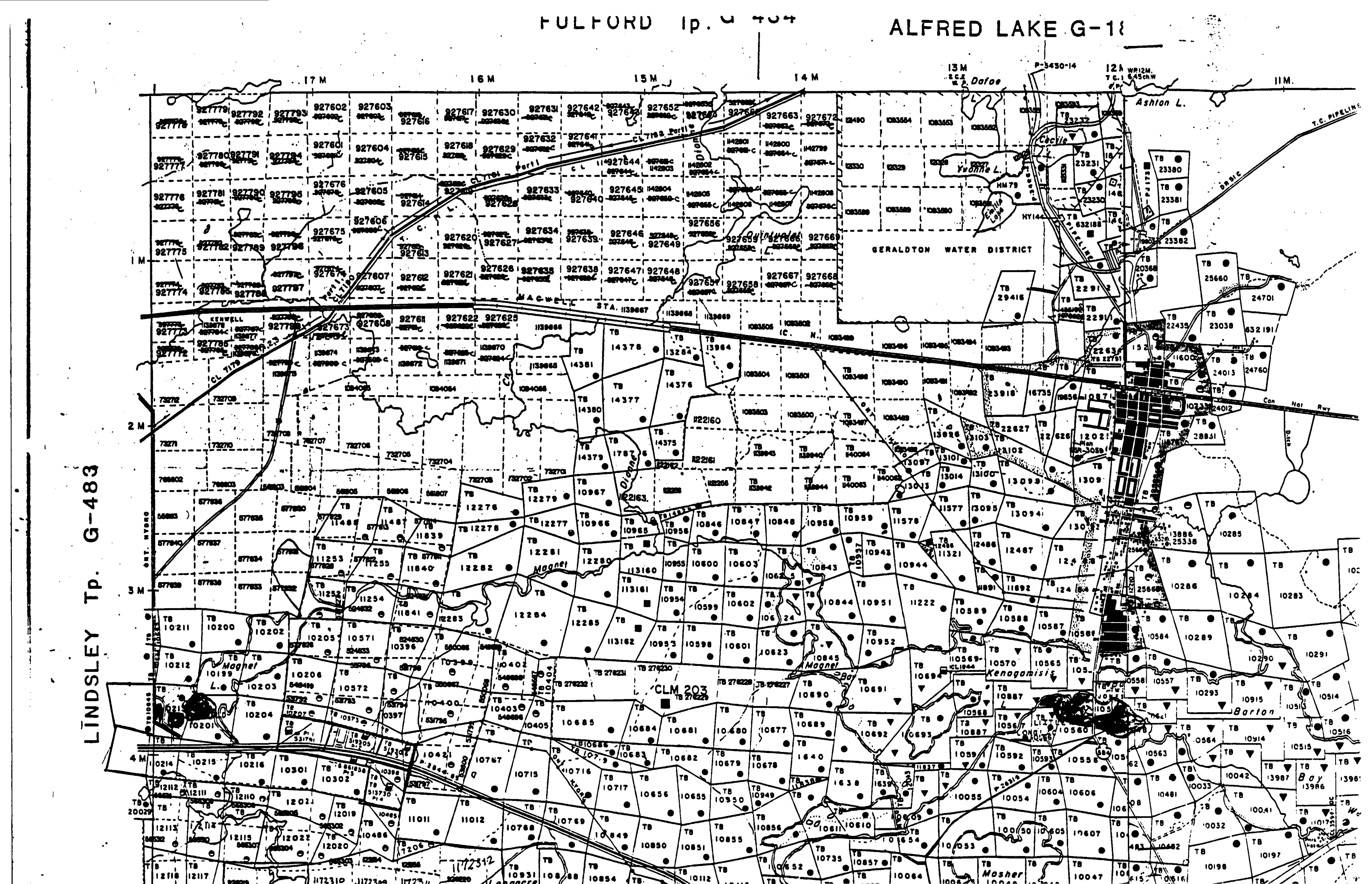
LINDSLEY Tp. G-483

Little Long Lac Tailings

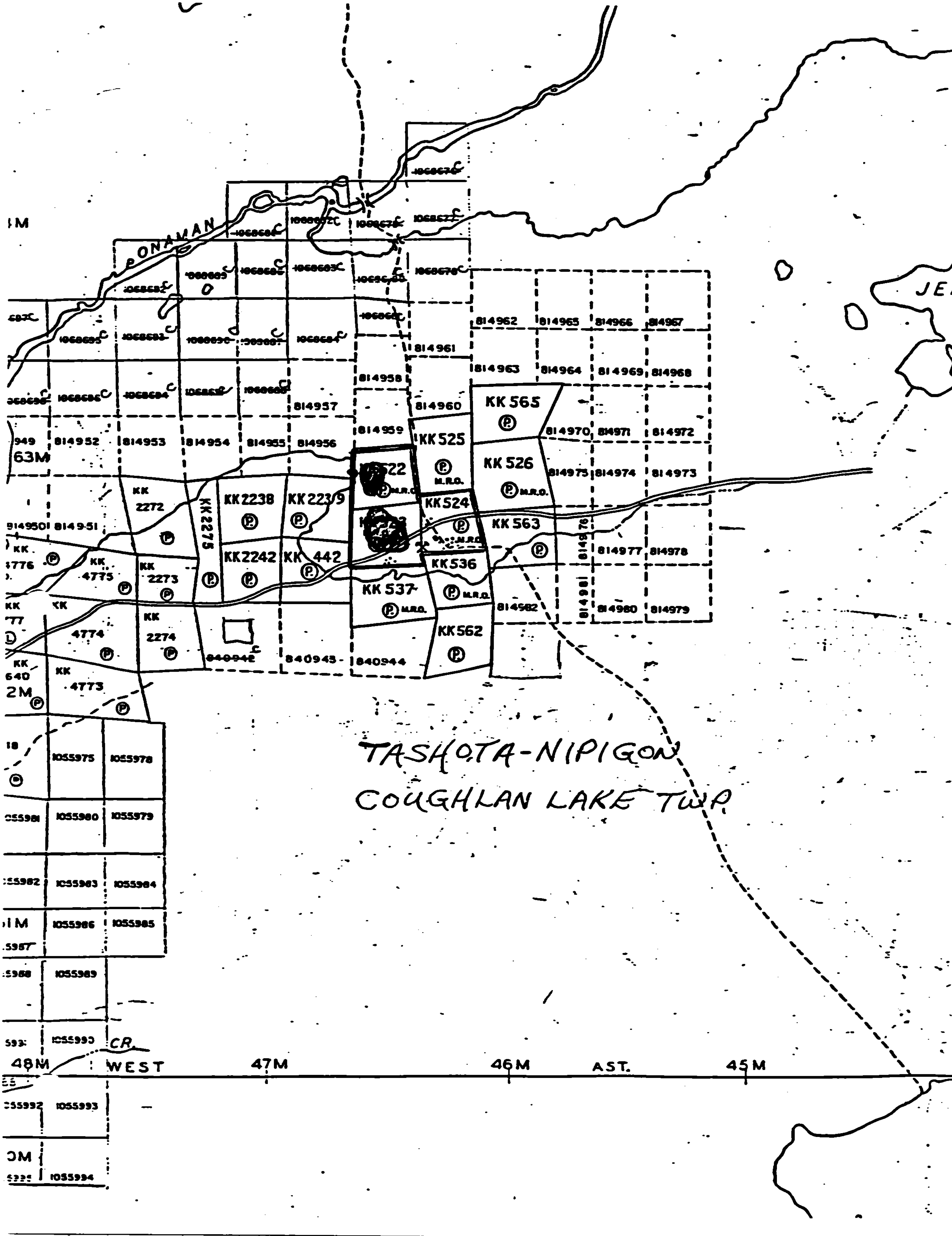
TERRILL RANKFIELD TAILINGS

LITTLE LONG LAC PROJECT

BANKFIED-TOMBILL PROJECT



FULFORD Tp.
ALFRED LAKE G-1
LINDSLEY Tp. G-483
GERALDTON WATER DISTRICT
CLM 203
Ashton L.
Kenogamisis
Magnet L.



TASHOTA-NIPIGON
COUGHLAN LAKE TWP.
PONAMAN
KK 565
KK 525
KK 526
KK 524
KK 563
KK 536
KK 537
KK 562
KK 2238
KK 2239
KK 2242
KK 442
KK 2272
KK 2275
KK 2273
KK 2274
KK 4775
KK 4774
KK 4773
M.R.O.
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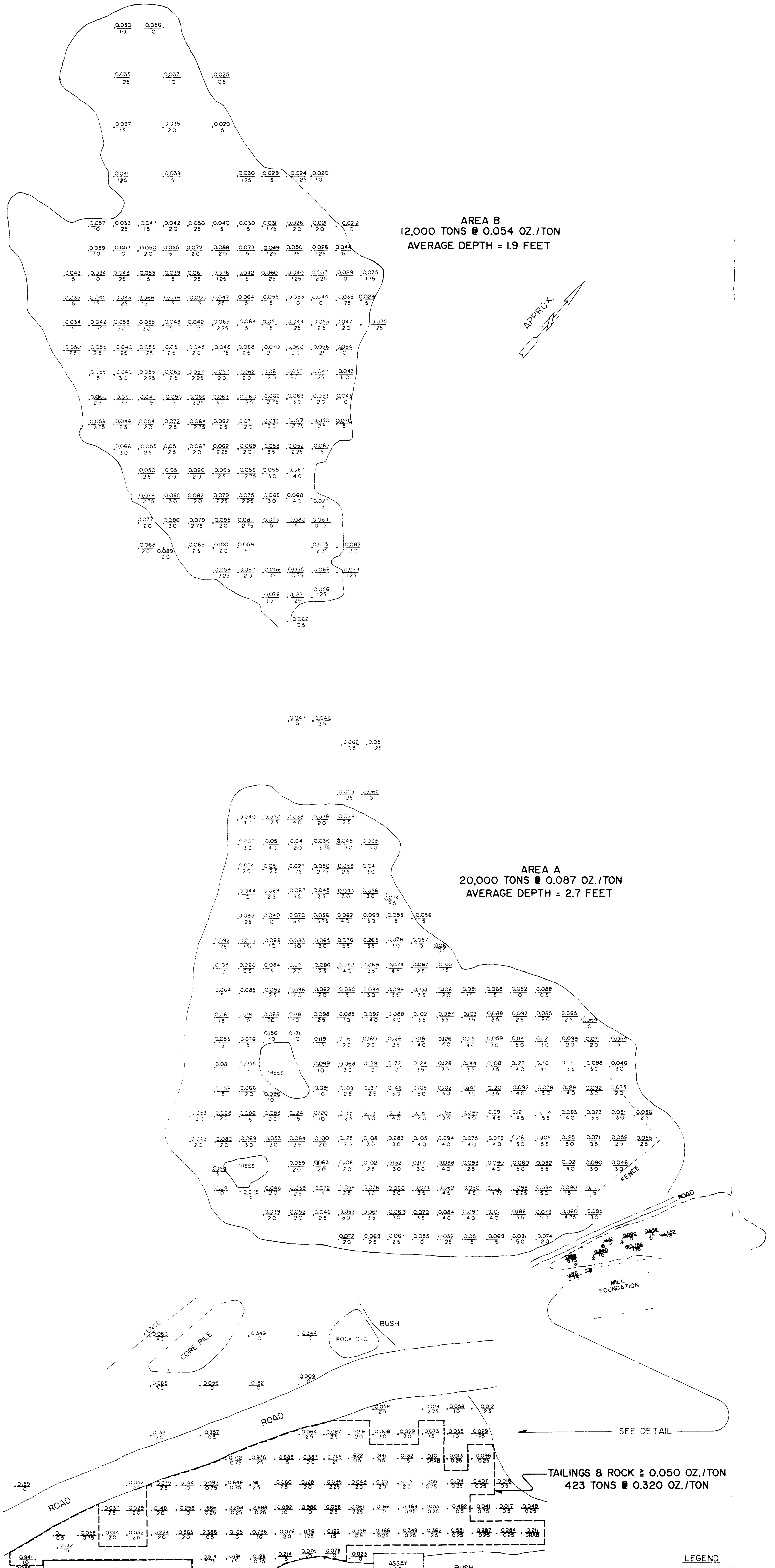
1055994
63M
48M
WEST
47M
46M
AST.
45M
CR.



SCALE 1" = 20'

SUMMARY OF RESERVE POTENTIAL

AREA A: 20,000 TONS @ 0.087 OZ./TON

AREA B: 12,000 TONS @ 0.054 OZ./TON

TOTAL : 32,000 TONS @ 0.075 OZ./TON

LEGEND

0.098 — OZ./TON
5.25 — DEPTH OF TAILINGS (FEET)

63.6072

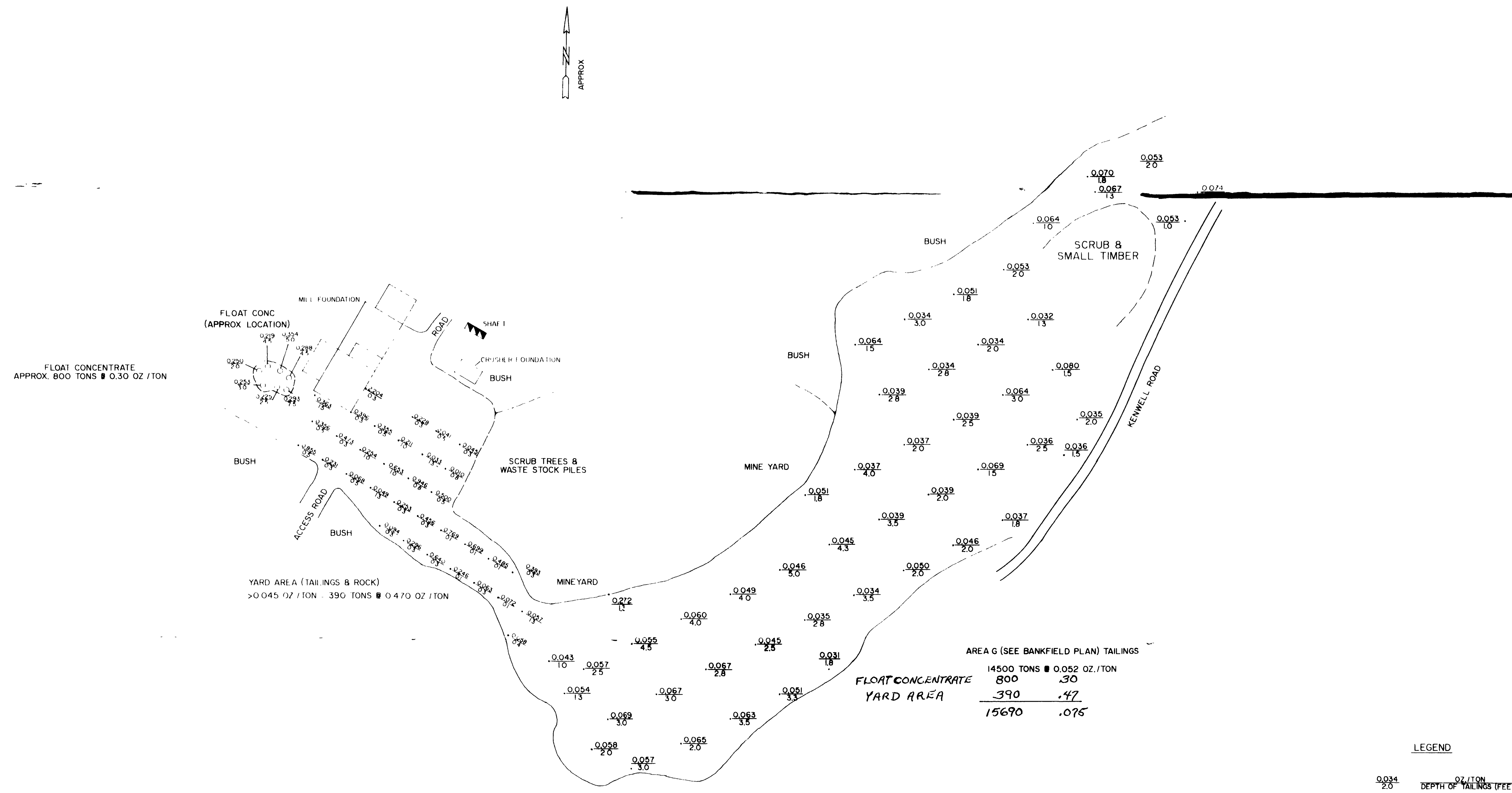
BEAUROX MINES LTD.

TASHOTA TAILINGS PROJECT

RESERVE POTENTIAL & ASSAY PLAN

SCALE 1" = 50	DATE 07/11/90	DRAWING NUMBER
DRAWN BY C. KEATS	APPROVED	REV

200

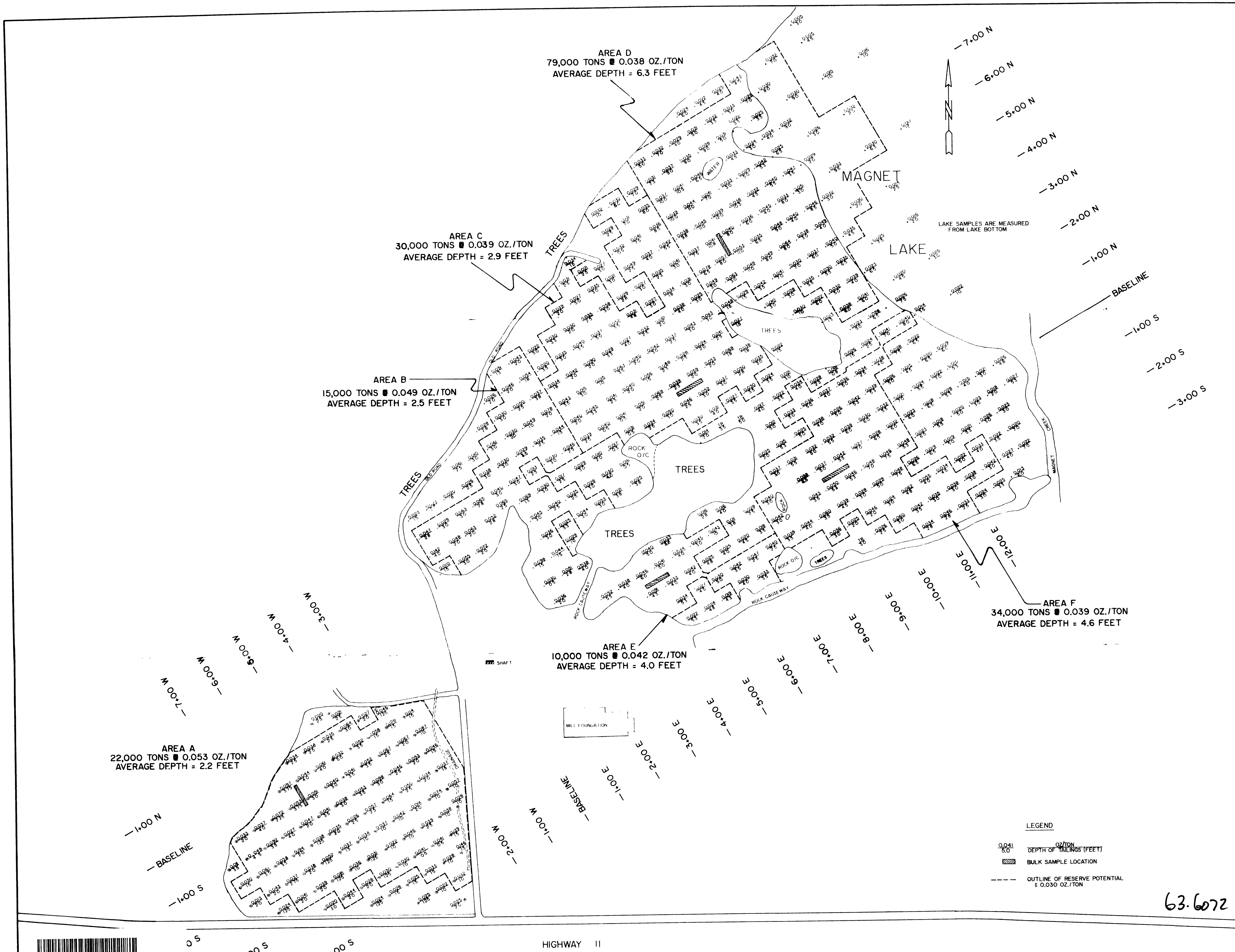


63.6072

BEAUROX MINES LTD.		
TOMBILL TAILINGS PROJECT		
ASSAY PLAN		
SCALE 1" = 50	DATE 13/11/90	DRAWING NUMBER
DRAWN BY C KEATS	APPROVED	REV

42E10NW0113 63.6072 ASHMORE

210



SUMMARY OF RESERVE POTENTIAL

TOTAL SAMPLED TAILINGS :

264,000 TONS @ 0.037 OZ./TON
OR 265,190 0.384 *

RESERVES ≥ 0.030 OZ./TON

AREA: A 22,000 TONS @ 0.053 OZ./TON
B 15,000 @ 0.049
C 30,000 @ 0.039
D 79,000 @ 0.038
E 10,000 @ 0.042
F 34,000 @ 0.039
(TOMBILL) G 14,500 @ 0.052
1190 0.356 * HIGH GRADE

TOTAL 205,000 TONS @ 0.042 OZ./TON
OR 206190 0.044 *

LEGEND

$\frac{0.041}{5.0}$ OZ/TON / DEPTH OF TAILINGS (FEET)

BULK SAMPLE LOCATION

OUTLINE OF RESERVE POTENTIAL ≥ 0.030 OZ./TON

63.6072

BEAUROX MINES LTD.		
BANKFIELD TAILINGS PROJECT		
RESERVE POTENTIAL & ASSAY PLAN		
SCALE 1" = 100'	DATE 08/11/1990	DRAWING NUMBER
DRAWN BY C KEATS	APPROVED	REVISED


KENOGAMISIS LAKE
LAKE SAMPLES ARE MEASURED FROM LAKE BOTTOM
AREA B
111,000 TONS @ 0.037 OZ./TON
AVERAGE DEPTH = 8.1 FEET
AREA C
58,000 TONS @ 0.058 OZ./TON UNCUT
58,000 TONS @ 0.044 OZ./TON CUT TO 0.1
AVERAGE DEPTH = 3.5 FEET
PRIVATE PROPERTY
SMALL SCRUB & TAG ALDERS
AREA A
20,000 TONS @ 0.030 OZ./TON
AVERAGE DEPTH = 6.0 FEET
TREES
BASELINE
ARENA ROAD
HIGHWAY 584
APPROX.
AREA D
739,000 TONS @ 0.035 OZ./TON
AVERAGE DEPTH = 7.8 FEET
ACCESS
WATER METER (WELL)
WATER LINE
TWP. LINE
SWAMP
SCRUB
TOTAL RESERVES ≥ 0.025 OZ./TON
928,000 TONS @ 0.037 OZ./TON UNCUT
0.036 OZ./TON CUT TO 0.1
(AREA A+B+C+D)
LEGEND
OZ./TON
DEPTH OF TAILINGS (FEET)
HIT OBSTRUCTION
INDICATES APPROXIMATE LOCATION
BULK SAMPLE LOCATION
63.6072
BEAUROX MINES LTD.
LITTLE LONGLAC TAILINGS PROJECT
ASSAY PLAN (SELECTED GRADE)
C. KEATS
230

LITTLE LONGLAC WEST:

SAMPLED AREA: 284,000 TONS OF 0.036 OZ./TON UNCUT
284,000 TONS OF 0.032 OZ./TON CUT TO 0.1
AVERAGE DEPTH: 5.8 FEET

ESTIMATED TOTAL WEST SIDE TONNAGE: 500,000 TONS

LITTLE LONGLAC EAST - 1,075,000 TONS OF 0.030 OZ./TON
AVERAGE DEPTH: 7.6 FEET

COMBINED TOTAL SAMPLED LITTLE LONGLAC TAILINGS
1,360,000 TONS OF 0.030 OZ./TON CUT TO 0.1 OZ./TON. (0.031 UNCUT)

LEGEND:
0.033 / 4.5 — OZ./TON / DEPTH OF TAILINGS (FEET)
• HIT OBSTRUCTION
INDICATES APPROXIMATE LOCATION
BULK SAMPLE LOCATION

63.6072

BEAUROX MINES LTD.
LITTLE LONGLAC TAILINGS PROJECT
ASSAY PLAN (TOTAL DEPTH)

DATE 10/12/90	SCALE 1" = 100'	DRAWING NO.
DRAWN BY C. KEATS	APPROVED	REVISIONS

240

TOTAL RESERVES ≥ 0.030 OZ./TON
532,000 TONS @ 0.043 OZ./TON UNCUT
0.041 OZ./TON CUT TO 0.1
(AREA A+B+C+D)

LEGEND

$\frac{0.033}{4.5}$ OZ./TON / DEPTH OF TAILINGS (FEET)

· HIT OBSTRUCTION

--- INDICATES APPROXIMATE LOCATION

▨ BULK SAMPLE LOCATION

63.6072

BEAUROX MINES LTD.		
LITTLE LONGLAC TAILINGS PROJECT		
ASSAY PLAN (SELECTED GRADE)		
DATE 16/12/90	SCALE 1" = 100'	DRAWING NO.
DRAWN BY E. KEATS	APPROVED	REVISIONS

250