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CANADA

DEPARTMENT OF MINES AND TECHNICAL SURVEYS

OTTAWA

MINES BRANCH INVESTIGATION REPORT IR 58-139

PILOT PLANT LEACH TESTS ON ORE FROM THE KITTS PROPERTY OF BRITISH NEWFOUNDLAND EXPLORATION LTD., NEWFOUNDLAND

by

H. H. McCREEDY and W. A. GOW

RADIOACTIVITY DIVISION

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H.H. McCreedy* and W.A. Gow**

ABSTRACT

Continuous pilot plant, acid leaching at atmospheric pressure and ambient temperature under various acid and oxidizing conditions showed that extractions of over 95% could be obtained within 48 hours of contact time. Acid consumption values ranged from approximately 85 to 145 lb of 93% H_2SO_4 per ton of ore. Operating data indicated that no abnormal adverse factor would be involved in treating an ore of this type using conventional hydrometallurgical processing equipment.

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(39 pages, 19 tables, 3 illus.)

INTRODUCTION

British Newfoundland Exploration Ltd., operating under Atomic Energy Control Board Exploration Permit MX 25/57, requested, in letters dated March 21, 1958 and April 3, 1958 from Dr. A.P. Beavan, General Manager, that the Mines Branch carry out pilot plant testing of the conventional acid leach process on uranium ore from the company's Kitts property located near Makkovik, Labrador, Newfoundland. Dr. Beavan wrote from the company's head office located at 1900 Sherbrooke St. West, Montreal, Quebec.

Since previous test work on a laboratory scale on ore from this property[1] had shown the ore to be amenable to acid leaching, it was agreed that the Radioactivity Division would carry out the work requested. Subsequently, a bulk sample of approximately 34,200 lb was received at the Mines Branch on April 16, 1958. This sample was said to be from the Kitts property, in a letter dated April 14, 1958 from Dr. Beavan.

Because of the unexpectedly high uranium content of the first sample it was thought best to dilute this with waste rock to a value nearer to that expected at the mine. Hence, a 17,335 lb sample of waste rock was received on May 6, 1958. The original high grade sample was given Radioactivity Division No. 4/58-12 and was known to the company as K4, while the waste rock was catalogued as No. 5/58-4A. The waste rock sample was mixed with the high grade ore in a 1 to 1 ratio and thereby produced a composite ore of about

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(1) See reference on Page 22.

17 tons. This composite sample was designated as No. 5/58-4.

The first two pilot plant runs, LPP 85 and 86, were made on the composite lower grade ore (No. 5/58-4), while the last run, LPP 87, was made on the high grade ore (No. 4/58-12).

Operational data from the leach pilot plant are included in this report, while mineralogy of the head samples and ion exchange, solvent extraction and precipitation pilot plant studies will be covered in separate reports. Bench scale tests on previously submitted ore samples are also reported separately.

SUMMARY OF RESULTS

Table 1 is a summary of the average conditions and results of the pilot plant test work. It can be seen from this table that, on the samples tested, uranium extraction of over 97% was possible under the following conditions:

Grind	57 to 64% - 200 mesh
Leaching Time	48 hr
pH	1.8, controlled
Pulp density	60% solids
Sodium chlorate	5.0 lb/ton ore
Acid consumption	92 lb of 93% H_2SO_4/ton ore
Temperature	30°C - no heat added.

Soluble loss in the pilot plant was less than 1%, using three stages of washing.

Where high grade feed was used, residue assays were comparatively high although extraction was good. Additional uranium may be extracted from these residues by increased leaching after regrinding, or by increased leaching with an additional 5.0 lb $NaClO_3$/ton ore added. Whether the resulting 3% increase in extraction warrants the additional grinding or reagent cost will depend on economic factors which are not yet clear.

TABLE 1

Summary of Brinex Pilot Plant Results

Run No.	LPP 85	LPP 86	LPP 87
Days of continuous operation	9	12	10
Dry feed, lb, including start up batch	9,064	10,345	8,730
Feed assay, % U_3O_8, (Run Comp.)	0.60	0.63	1.29
Grind, % minus 200 mesh	57.0	64.4	64.0
pH controlled at:	1.5[a] 1.8[b]	1.5[d] 1.2[e]	1.5[g]
$NaClO_3$ addition, lb/ton ore	5.0[c]	3.0[d] 5.0[e, f]	5.0[g]
Final residue, % U_3O_8 (Av.)	0.013[a] 0.017[b]	0.025[d] 0.016[e]	0.035[g]
Rewashed final residue, % U_3O_8 (Av.)	0.009[a] 0.014[b]	0.021[d] 0.011[e]	0.029[g]
Uranium extraction, %	98.5[a] 97.7[b]	96.7[d] 98.3[e]	97.8[g]
Uranium recovery in solution, %	97.8[a] 97.2[b]	96.0[d] 97.5[e]	97.3[g]
No. 1 filter filtrate, U_3O_8 g/1 (Run Comp.)	6.99	7.64	14.22
No. 2 filter filtrate, U_3O_8 g/1 (Run Comp.)	1.31	2.00	2.75
No. 3 filter filtrate, U_3O_8 g/1 (Run Comp.)	0.16	0.41	0.45
Pregnant solution, U_3O_8 g/1 (Run Comp.)	3.17	3.43	5.95
Average total 93% H_2SO_4 consumed in leaching, lb/ton ore.	92[a] 85[b]	114[d] 139[e]	145

(a) May 22 to 25, (b) May 26 to 30, (c) May 22 to 30, (d) June 2 to 5, (e) June 6 to 13, (f) June 6 to 9 - $NaClO_3$ added dry to feed belt, June 10 to 13 - $NaClO_3$ added as aqueous solution to No. 2 Agitator, (g) June 16 to 25.

ANALYSES

Table 2 shows the chemical and radiometric analyses of the two ore samples used as pilot plant feed. Table 3 shows the results of the quantitative spectrographic analysis of the head sample of the high grade ore No. 4/58-12 .

TABLE 2

Head Sample Analyses of Ore Samples No. 5/58-4 and No.4/58-12

	Radioactivity No. 5/58-4	Radioactivity No. 4/58-12
U_3O_8 chemical	0.66%	1.37%
U_3O_8 (secondary)*	0.031%	-
ThO_2	<0.002%	-
As	0.03%	0.06%
P_2O_5	0.039%	0.13%
Fe (total)	8.38%	16.3%
CO_2 (evol)	0.59%	1.10%
CO_2 (comb)	9.12%	10.01%
S (comb)	3.19%	3.04%
V_2O_5	0.078%	0.14%
Mo	0.044%	0.095%
TiO_2	0.72%	
F	0.10%	
Mn		0.068%
Ag	-	0.02 oz/ton
Au	-	none detected
Radiometric calc. U_3O_8	0.66%	
Equivalent gamma U_3O_8	0.637%	
Equivalent beta U_3O_8	0.634%	
$(RE)_2O_3$	0.06%	

Note: Sample No. 5/58-4 was a 1:1 composite of barren waste rock and sample, No. 4/58-12.
Sample No. 4/58-12 was the original high grade sample submitted (British Newfoundland Exploration Ltd.No.K-4).

* A sample is leached for 10 minutes in a hot 10% solution of Na_2CO_3. The uranium dissolved is taken as an indication of the secondary uranium present.

TABLE 3

Quantitative Spectrographic Analyses of Sample No. 4/58-12

Element	%	Element	%
Si	P.C.	Cu	0.3
Al	10	Ti	0.5
Fe	15	Zr	0.01
Na	15	Ni	0.03
Mg	5	Co	0.01
Pb	1	B	0.003
Ca	4	Cr	none detected
Ba	0.1	Be	<0.001
Mn	0.4	U	0.8
V	0.08	Y	0.008
Mo	0.15		

P.C. - Principal constituent

DETAILS OF TEST WORK

Ore Sample Handling and Crushing

The first high grade ore sample (No. 4/58-12) was crushed to minus 4 mesh and stored in covered 45-gallon barrels. The second smaller shipment of waste rock (No. 5/58-4A) was crushed separately to minus 4 mesh and stored in similar 45-gallon barrels. The two samples were then mixed in a ratio of 1 to 1 until the smaller waste sample was used up. The result was a composite ore of approximately 17 1/2 tons and a remaining high grade sample of approximately 8 1/2 tons.

A flowsheet of the crushing plant used is shown in Figure 1. The screen analysis and uranium distribution of the high grade crushed ore before grinding are shown in Table 4. The screen analysis

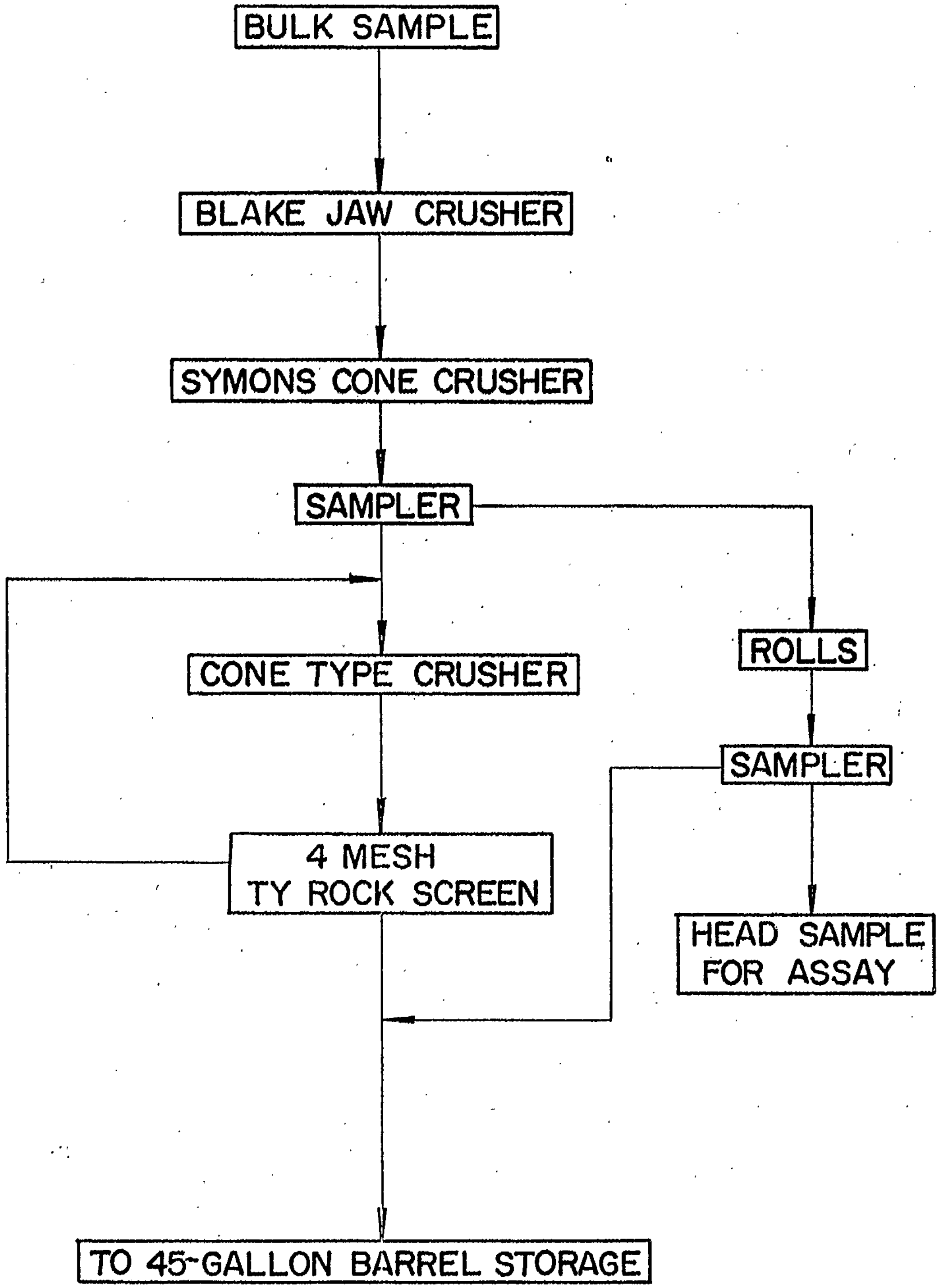

FIGURE I

CRUSHING PLANT FLOWSHEET

LPP 85 TO 87

of the lower grade composite would be similar to that shown in Table 4.

TABLE 4

Screen Analysis of Crushed Ore before Grinding, Sample No. 4/58-12

Mesh size	Weight, grams	Weight, %	U_3O_8 Assay, % (gamma)	U_3O_8 Dist., %
-4+10	819	55.0	1.13	44.7
-10+14	141	9.5	1.65	11.3
-14+35	223	15.0	1.94	20.9
-35+100	108	7.3	2.09	11.0
-100	197	13.2	1.27	12.1
TOTAL	1488	100.0	1.39	100.0

Grinding Circuit

A 30" by 18" Denver ball mill, in closed circuit with a 14" Dorr Duplex rake-classifier, was used to grind the minus 4 mesh ore. Steel balls were used as the grinding medium. The classifier overflow was pumped by a one-inch Denver vertical sand pump to a 3 ft by 2 ft drum filter.

The filter cake was stored in open-top steel boxes for a minimum amount of time (not more than 24 hr) before being used in the leach circuit. To keep the ore from drying out, and keep the possible formation of polythionates to a minimum, the ground moist ore was covered with a plastic sheet.

The grinding circuit is shown in Figure 2.

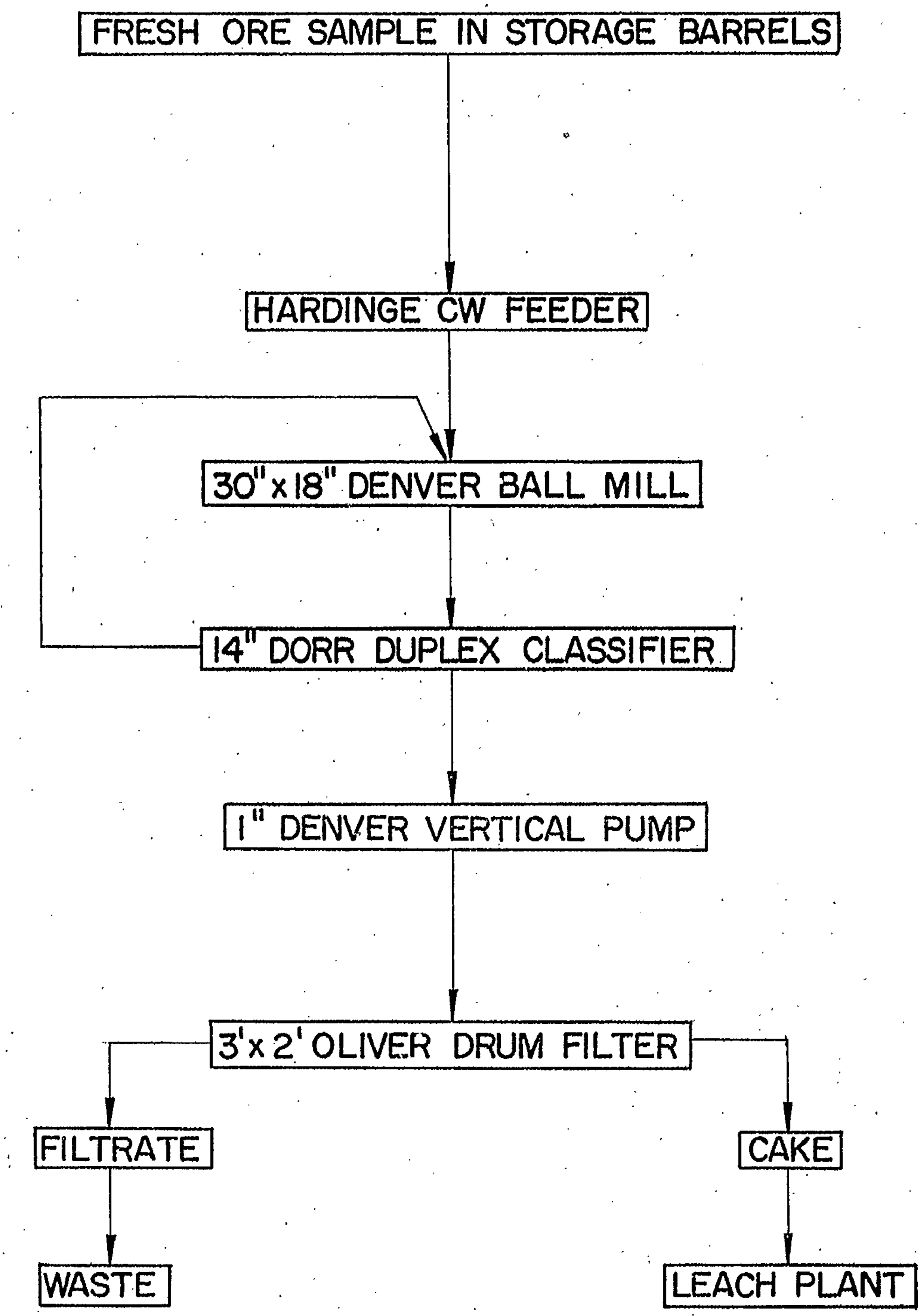

FIGURE 2

GRINDING CIRCUIT FLOWSHEET

LPP 85 TO 87

Leaching Circuit

The following nominal conditions were common in all pilot plant leaching operations:

Temperature	No heat added
Contact time, theoretical, hr	48
% solids in leach agitators (proposed)	60
Three stage continuous filtration:	
No. 1 filter wash soln.	
1/4% H_2SO_4	0.3 W*
lb/hr	11.4
No. 1 filter repulping soln.	
1/4% H_2SO_4	0.34 W*
lb/hr	12.9
No. 2 filter wash soln.	
1/4% H_2SO_4	0.3W*
lb/hr	11.4
No. 2 filter cake repulping soln.	
1/4 % H_2SO_4	0.34 W*
lb/hr	12.9
No. 3 filter wash soln.	
Water	0.3W*
lb/hr	11.4
Filtering aids:	Jaguar MDC or Separan as required on No. 1 filter only.

* W = dry ore weight

A flowsheet of the leach pilot plant is shown in Figure 3.

The ground ore was placed on a continuous rubber-belt feeder in batches every half hour. The feed rate, dry weight, was 38 lb per hour. Except for a period of four days, June 10 to 13 inclusive, the sodium chlorate was added dry to the feed belt. During this four day period in LPP 86, the chlorate was added, by a mechanical reagent feeder, as a 50% solution to the No. 2 agitator. The ore from the slow moving belt discharged into a 2 ft diameter by 7 ft feed repulper where water was added to produce a pulp of 60% solids by

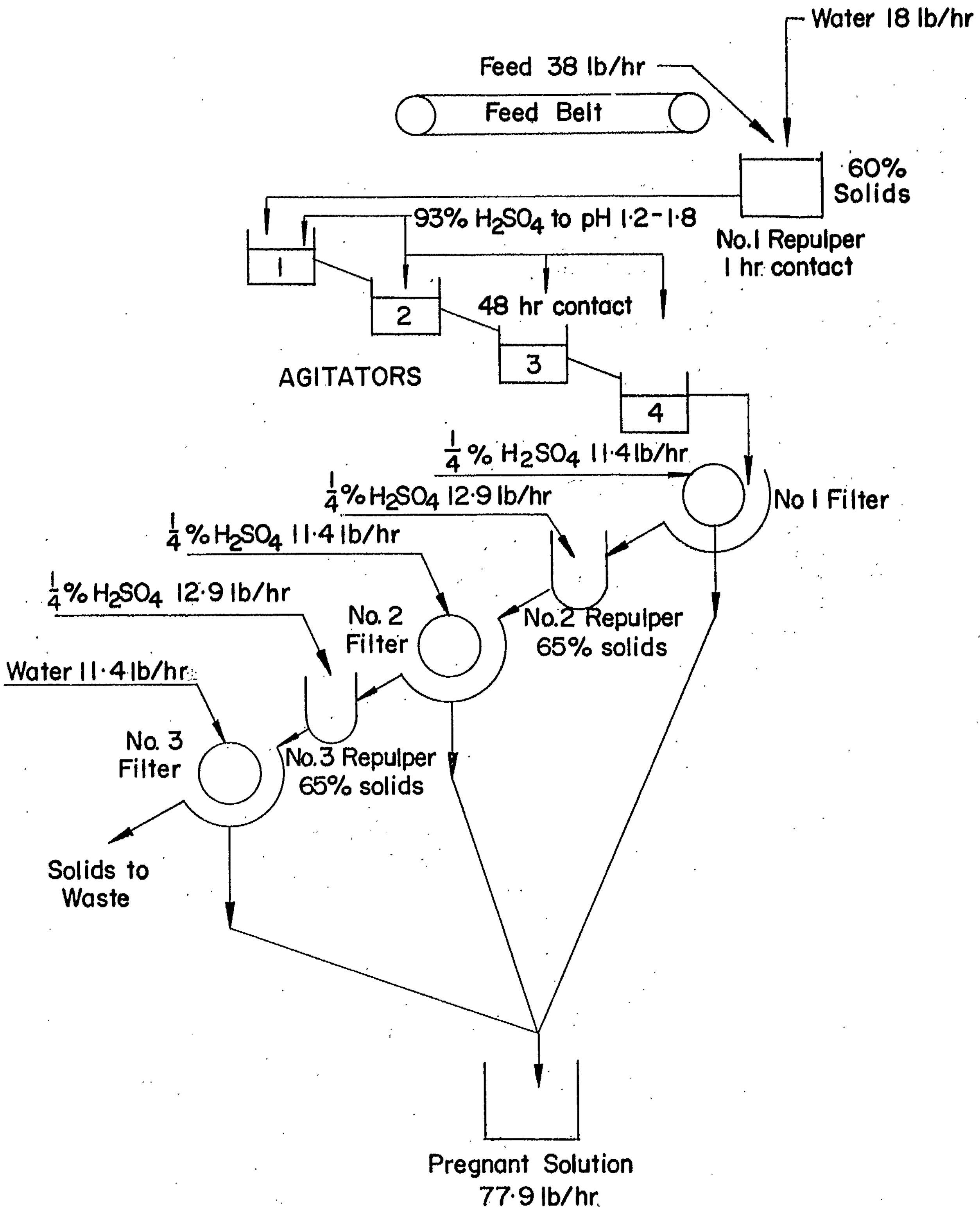

FIGURE 3

PILOT PLANT FLOWSHEET

LPP 85 - 87

weight. The feed pulp was pumped to the No. 1 agitator by a 1 1/2 in. O.D.S. pump. The pulp flowed by gravity to the other three agitator tanks which were in series. The leach tanks were 2 ft diameter by 3 ft deep of mild steel construction with Linatex linings. Pulp agitation was effected by 1/2 H.P. Greey Lightnin' Mixers with 7 in. marine-type impellers of type 316 stainless steel.

The 93% sulphuric acid, used for pH control, was added automatically to the No. 1 agitator by means of an air controlled valve connected to a Beckman, Model R, pH indicator in conjunction with a Brown Electronik recorder-controller. Manual additions of sulphuric acid were made every hour to the remaining three agitators in the series to maintain the required pH. No heat was added to the circuit. Pulp samples were taken from each agitator once per shift. The solids residues of these samples were washed with 1% sulphuric acid and water and assayed on a daily composite basis for U_3O_8 to check the rate of extraction throughout the leaching circuit. The leach liquors from the filtered pulp samples were assayed each shift for reducing power and total iron.

<u>Filtration Circuit</u>

A three-stage filtering circuit was used on the pulp which was gravity fed from the last agitator. The filters were 18 in. diameter by 12 in. rotary-type, drum filters constructed of stainless steel. All filters had a scraper-type discharge. The cakes formed on No. 1 and No. 2 filters were washed with 1/4% sulphuric acid and the filter cakes repulped with 1/4% acid in both cases. The cake formed on the third filter was washed with water and the cake was weighed and sampled before being discarded. The resultant mixed solution was pregnant liquor feed for subsequent treatment by ion exchange, solvent extraction or direct precipitation. The pregnant solution was clarified by allowing it to settle about eight hours before passing the decant through a Sethco clarifier. Cotton ST 19 filter cloth was used on all three filters. Jaguar MDC and Separan 2610 were used, individually, as required, for a filtering aid on the No. 1 filter only. A portion of the final residue sample was rewashed with 1% sulphuric acid and water. The difference between the U_3O_8 content of the final residue and the rewashed final residue was taken as the soluble loss in filtering.

Start-Up of Leach Circuit

For the start-up of the first run, pulp at 60% solids was pumped to agitators No. 4, 3, 2 and 1, respectively, at 12-hour intervals. 93% sulphuric acid was added manually to maintain pH 1.5. After 48 hours of contact time on the first batch in No. 4 agitator, continuous feeding to No. 1 agitator began.

At the end of each run 0.4 lb Jaguar MDC per ton ore was added to the pulp in each of the agitators and the agitation mechanism turned off until the beginning of the next run, some 3 days later. Only the filter boots and launders were cleaned out between runs. Continuous feeding to No. 1 tank and commencement of the agitation mechanisms were all that were required to start up the leaching circuit for the second and third runs.

The batch leach start-up data are shown in Table 5.

TABLE 5

Batch Leach Start-Up Data, LPP 85

	Batch No. 1	Batch No. 2	Batch No. 3	Batch No. 4
Grind, %-200 mesh.	65.5	-	62.8	-
Feed assay, % U_3O_8	0.38	0.47	0.46	0.54
Residue assay, % U_3O_8				
at 8 hr	0.012	0.018	0.019	0.016
at 12 hr	-	-	-	0.022
at 16 hr	0.006	0.012	0.020	-
at 24 hr	0.021	0.012	0.014	-
at 32 hr	0.012	0.012	-	-
at 36 hr	-		-	-
at 40 hr	0.009	-	-	-
at 48 hr	0.010	-	-	-
Acid Consumption, lb 93% H_2SO_4/ton ore				
at 0 hr	50	52	50	50
at 8 hr	78	78	80	82
at 12 hr	-	-	83	-
at 24 hr	82	80	85	-
at 36 hr	89	90	-	-
at 48 hr	95	-	-	-
Leach contact time, hr	48	36	24	12

Results and Discussion

The detailed leaching conditions and results are shown on Tables 6 and 7. Table 8 shows the run composite analyses of the various pilot plant products. Table 9 shows the screen analyses of run composite feed and residue samples.

TABLE 6

Leaching Conditions

Run No.	Date	Feed: Dry Feed, lb	Feed: H_2O, %	Feed: U_3O_8, %	Feed: $NaClO_3$ Added, lb/ton	93% Sulphuric Acid: Total Added, lb/ton	93% Sulphuric Acid: Total Added, lb	93% Sulphuric Acid: To No.1 Agit., lb	93% Sulphuric Acid: To No.2 Agit., lb	93% Sulphuric Acid: To No.3 Agit., lb	93% Sulphuric Acid: To No.4 Agit., lb	Solids in Agit., %	Temp in Agit., °C	pH: No.1 Agit.	pH: No.2 Agit.	pH: No.3 Agit.	pH: No.4 Agit.	pH: No.1 Repulp.	pH: No.2 Repulp.	Remarks
85	May 20	400	13.8	0.38	5.0															No.1 Batch
	20	400	13.8	0.47	5.0															No.2 Batch
	21	400	13.8	0.46	5.0															No.3 Batch Start-up
	21	400	13.8	0.54	5.0															No.4 Batch
	22	606	11.6	0.50	5.0		31.0	18.6	4.3	4.7	3.4		28	1.56	1.55	1.57	1.57			Continuous operation for 2/3 day.
	23	904	11.9	0.67	5.0	92	47.0	35.5	4.0	4.3	3.2	58	31	1.49	1.57	1.58	1.55	6.3	1.8	Scrubbed No.1 filter.
	24	896	12.3	0.69	5.1	90	46.0	35.4	3.8	3.8	3.0	58	30	1.48	1.55	1.55	1.53	6.3	1.8	Guar started at 0.02 lb/ton to No.1 filter.
	25	908	11.7	0.65	5.0	93	48.0	35.7	4.8	4.1	3.4	58	30	1.51	1.55	1.55	1.53	6.2	1.7	
	26	833	11.8	0.72	5.0	101	42.0	32.8	3.3	3.3	2.6	58	29	1.51	1.58	1.58	1.58	5.9	1.8	Changed No.1 cloth. Feed off 3 hr.
	27	905	11.3	0.69	5.0	73	33.0	30.6	0.8	1.1	0.5	58	26	1.79	1.74	1.72	1.65	5.9	1.8	Feed off 1/2 hr.
	28	916	11.3	0.65	5.0	83	38.0	29.5	3.0	3.2	2.3	58	26	1.78	1.84	1.84	1.81	6.0	1.7	Guar raised to 0.05 lb/ton to No. 1 filter.
	29	911	11.3	0.62	5.0	83	38.0	28.8	3.7	3.3	2.2	58	26	1.79	1.84	1.87	1.80	6.6	1.8	Scrubbed No.1 filter.
	30	585	11.9	0.58	5.0	85	25.0	18.7	2.8	2.2	1.3	58	27	1.81	1.83	1.82	1.80	6.8	1.8	Continuous operation for 2/3 day.
86	June 2	827	12.0	0.65	3.1	128	53.0	36.3	5.7	3.8	7.2	60	28	1.60	1.88	1.65	1.91	6.3	1.8	Changed to pH 1.5 and 3 lb $NaClO_3$/ton.
	3	898	13.1	0.65	3.0	125	56.0	36.7	6.0	6.5	6.8	59	30	1.52	1.55	1.56	1.55	6.7	1.6	
	4	913	12.1	0.49	3.0	94	43.0	28.5	4.8	5.6	4.1	60	27	1.49	1.53	1.54	1.52	6.5	1.7	Guar 0.1 lb/ton to No.1 filter. Scrubbed No. 1 cloth.
	5	905	11.9	0.57	3.0	110	50.0	34.1	5.6	6.1	4.2	59	29	1.57	1.54	1.55	1.50	6.1	1.7	
	6	884	11.1	0.54	5.0	172	76.0	44.0	10.5	11.1	10.4	59	31	1.20	1.26	1.27	1.27	6.6	1.6	Changed to pH 1.2 and 5 lb $NaClO_3$/ton.
	7	795	11.7	0.62	5.0	118	53.0	34.2	6.7	6.1	6.0	58	30	1.20	1.21	1.20	1.20	6.7	1.4	Guar 0.5 lb/ton. Changed No.1 cloth.
	8	918	11.6	0.64	5.0	119	62.0	44.7	5.9	6.3	5.1	59	30	1.18	1.23	1.24	1.23	6.7	1.4	Poor filtering on No. 1 with Guar.
	9	910	12.1	0.66	5.0	130	59.0	39.4	7.0	7.3	5.3	59	31	1.20	1.22	1.22	1.21	6.4	1.5	Changed to Separan 0.05 lb/ton.
	10	914	11.9	0.61	5.0	116	53.0	46.0	6.6	5.5	4.9	60	25	1.21	1.22	1.21	1.20	6.4	1.5	$NaClO_3$ changed to liquid in No.2 agitator.
	11	896	12.0	0.61	5.0	151	67.5	42.6	10.7	7.8	6.4	61	31	1.20	1.24	1.22	1.21	6.5	1.6	Separan raised to 0.1 lb/ton to No. 1.
	12	893	12.7	0.59	5.0	146	65.0	43.5	10.1	6.4	5.0	60	32	1.20	1.24	1.21	1.20	6.4	1.5	
	13	592	12.4	0.63	5.0	159	47.0	32.3	6.8	4.4	3.5	60	31	1.21	1.24	1.21	1.20	6.5		Continuous operation for 2/3 day.
87	June 16	792	10.6	1.06	5.0	164	65.0	44.0	8.4	6.9	5.7	61	28	1.57	1.56	1.53	1.53			High grade sample.
	17	898	11.4	1.15	5.1	131	59.0	49.0	3.3	4.0	2.7	60	29	1.48	1.51	1.52	1.50	6.9	1.7	New No.1 filter cloth.
	18	902	10.5	1.12	5.0	144	65.0	52.1	4.5	5.0	3.4	58	30	1.45	1.52	1.51	1.50	6.7	1.7	Guar 0.05 lb/ton to No.1 filter.
	19	739	11.7	1.06	5.0	152	56.0	44.7	3.8	4.4	3.1	59	30	1.46	1.56	1.53	1.51	6.6	1.7	
	20	817	9.5	1.29	4.9	132	54.0	40.1	5.0	4.7	4.4	58	30	1.40	1.53	1.53	1.52	6.7	1.6	Scrubbed No.1 filter cloth.
	21	939	9.0	1.38	4.9	149	70.0	54.9	5.4	5.9	3.8	59	32	1.47	1.53	1.54	1.51	6.9	1.7	New No.1 filter cloth. No guar.
	22	914	10.7	1.27	5.0	144	66.0	53.1	4.8	4.5	3.6	59	30	1.48	1.52	1.51	1.52	6.8	1.6	
	23	917	10.5	1.24	5.0	144	66.0	51.7	5.2	4.7	4.4	60	30	1.50	1.52	1.52	1.51	6.8	1.7	Scrubbed No.1 filter cloth.
	24	908	10.7	1.42	5.0	143	65.0	50.4	6.2	5.2	3.2	60	31	1.50	1.52	1.51	1.50	6.6	1.6	Guar added intermittently to No.1 filter.
	25	904	11.5	1.41	5.0	148	67.0	50.6	6.2	5.5	4.7	60	30	1.51	1.53	1.53	1.52	6.9	1.7	

TABLE 7

Leaching Results

L.P.P. No.	Date	Washed Solids Liquors																Residues, % U_3O_8					
		No.1 Agitator				No.2 Agitator				No.3 Agitator				No. 4 Agitator									
		S.G. of Pulp	pH	Red Power, g/l	Fe^{+3}, g/l	S.G. of Pulp	pH	Red Power, g/l	Fe^{+3}, g/l	S.G. of Pulp	pH	Red Power, g/l	Fe^{+3}, g/l	S.G. of Pulp	pH	Red Power, g/l	Fe^{+3}, g/l	No.1 Agit.	No.2 Agit.	No.3 Agit.	No.4 Agit.	No.1 Filter	No.2 Filter
	May																						
	20																						
	20																						
	21																						
	21																						
	22	1.63	-	-	-	1.64	-	-	-	1.68	-	-	-	1.67	-	-	-						
	23	1.65	1.70	5.7	5.2	1.64	1.70	8.3	4.1	1.64	1.70	11.2	3.2	1.67	1.68	12.8	3.0	0.016	0.013	0.009	0.008	0.081	0.043
	24	1.63	1.40	4.7	4.8	1.63	1.55	9.0	4.0	1.65	1.55	11.6	3.0	1.67	1.60	12.9	3.2	0.030	0.014	0.011	0.009	0.094	0.028
85	25	1.62	1.57	4.8	4.4	1.61	1.59	8.6	3.0	1.64	1.67	11.4	2.6	1.65	1.70	12.6	1.8	0.040	0.016	0.012	0.010	0.076	0.037
	26	1.65	1.57	5.1	4.2	1.62	1.58	8.6	2.5	1.64	1.58	11.0	2.6	1.67	1.58	12.5	2.5	0.036	0.016	0.013	0.012	0.084	0.032
	27	1.65	1.81	4.2	4.0	1.64	1.77	7.0	3.7	1.66	1.69	10.3	2.5	1.64	1.65	11.7	2.9	0.046	0.020	0.015	0.013	0.071	0.037
	28	1.62	1.82	3.6	3.7	1.62	1.90	7.0	2.8	1.67	1.88	9.9	2.4	1.65	1.84	11.3	2.2	0.046	0.024	0.019	0.017	0.084	0.034
	29	1.65	1.84	3.4	4.3	1.62	1.80	6.6	2.9	1.65	1.83	9.2	1.9	1.64	1.82	10.7	2.6	0.051	0.028	0.020	0.018	0.097	0.035
	30	1.69	1.96	3.8	3.3	1.65	1.84	6.7	2.0	1.67	1.85	9.2	2.1	1.68	1.81	10.9	1.7	0.051	0.024	0.017	0.018	0.091	0.035
	June																						
	2	1.65	1.51	6.2	3.3	1.65	1.65	10.2	1.9	1.68	1.67	12.0	3.0	1.68	1.70	13.7	2.0	0.044	0.034	0.029	0.028	0.045	0.077
	3	1.65	1.57	6.4	3.1	1.65	1.55	9.5	2.1	1.66	1.47	12.1	2.0	1.67	1.57	14.0	1.9	0.050	0.032	0.024	0.026	0.011	0.044
	4	1.65	1.55	4.9	3.3	1.68	1.57	7.9	2.7	1.68	1.57	11.6	2.1	1.69	1.58	12.4	2.0	0.035	0.025	0.022	0.024	0.011	0.039
	5	1.66	1.57	4.6	3.0	1.65	1.54	7.2	2.0	1.66	1.50	10.1	1.9	1.68	1.58	11.9	2.0	0.036	0.017	0.017	0.016	0.084	0.037
	6	1.63	1.37	5.0	3.9	1.65	1.38	8.4	2.2	1.69	1.37	10.5	2.1	1.68	1.41	11.8	2.0	0.026	0.017	0.017	0.015	0.071	0.038
	7	1.65	1.40	4.6	4.8	1.64	1.33	8.5	2.9	1.68	1.30	10.9	3.1	1.66	1.23	13.0	2.4	0.021	0.014	0.014	0.014	0.092	0.052
86	8	1.65	1.25	4.8	5.1	1.64	1.27	8.4	3.4	1.68	1.22	11.0	2.7	1.68	1.21	12.9	3.1	0.024	0.012	0.013	0.011	0.13	0.034
	9	1.68	1.20	4.8	5.8	1.67	1.17	8.8	4.0	1.70	1.22	11.8	2.9	1.68	1.18	13.5	3.6	0.023	0.013	0.013	0.010	0.14	0.049
	10	1.68	1.21	5.5	3.7	1.68	1.22	6.9	4.9	1.72	1.21	11.5	3.5	1.69	1.20	12.4	3.2	0.041	0.015	0.011	0.011	0.15	0.027
	11	1.67	1.22	7.6	1.6	1.67	1.25	5.9	5.7	1.73	1.23	9.4	5.2	1.70	1.20	12.0	4.0	0.025	0.017	0.011	0.010	0.17	0.026
	12	1.66	1.24	8.2	1.1	1.66	1.27	6.3	6.1	1.73	1.22	9.6	5.5	1.70	1.20	12.1	4.6	0.032	0.024	0.013	0.013	0.18	0.045
	13	1.66	1.25	8.1	1.1	1.64	1.32	6.0	6.2	1.68	1.30	9.6	5.0	1.68	1.22	11.9	5.3	0.030	0.019	0.013	0.014	0.16	0.034
	June																						
	16	1.68				1.69				1.72				1.68									
	17	1.68	1.52	6.8	4.3	1.67	1.50	10.4	2.6	1.73	1.57	13.6	2.5	1.69	1.53	15.4	2.6						
	18	1.65	1.43	6.7	4.7	1.66	1.51	9.9	2.9	1.71	1.53	12.5	2.7	1.70	1.50	14.2	3.3	0.046	0.03	0.03	0.022		
	19	1.67	1.50	7.1	3.6	1.64	1.55	10.6	2.5	1.70	1.61	12.8	2.4	1.70	1.56	14.1	2.5	0.062	0.035	0.035	0.022	0.13	0.059
87	20	1.65	1.60	7.6	3.7	1.66	1.55	10.8	3.6	1.71	1.55	13.4	3.0	1.69	1.52	15.0	2.7	0.083	0.037	0.029	0.023	0.20	0.075
	21	1.65	1.50	7.6	3.6	1.64	1.52	10.6	3.4	1.70	1.50	13.3	2.4	1.70	1.50	14.3	2.4	0.069	0.042	0.029	0.026	0.12	0.11
	22	1.68	1.55	7.3	4.0	1.67	1.57	9.7	2.7	1.68	1.52	12.4	3.2	1.70	1.50	13.9	3.7	0.087	0.046	0.053	0.028	-	-
	23	1.67	1.53	7.1	3.7	1.66	1.50	10.0	2.7	1.69	1.48	12.1	2.8	1.68	1.50	13.6	2.5	0.086	0.070	0.045	0.045	0.24	0.077
	24	1.69	1.51	7.4	4.5	1.68	1.52	9.2	5.2	1.70	1.49	11.4	4.4	1.70	1.50	12.8	4.1	0.089	0.049	0.050	0.047	0.20	0.10
	25	1.70	1.53	8.0	4.2	1.66	1.57	10.8	3.2	1.72	1.52	12.4	3.6	1.70	1.50	13.8	4.1	0.120	0.058	0.047	0.045	0.22	0.068

Note: Reducing power is expressed as grams ferrous iron per litre.

(Continued) -

Leaching Results

L.P.P.	Date	Pregnant: Wt, lb.	Pregnant: U_3O_8, g/l	Pregnant: Red Power, g/l	Pregnant: Fe^{+3}, g/l	Pregnant: pH	Pregnant: S.G.	Final Residue: Dry wt, lb	Final Residue: H_2O, %	Final Residue: U_3O_8, %	Final Residue: Rewashed Residue, U_3O_8 %	Screen Analysis: Feed, %-200	Screen Analysis: Residue, %-200	Ext'n, %	Recovery, %	Soluble Loss, %
	May															
	20											65.5				
	20															
	21											62.8				
	21															
	22	142	-	-	-	-	-	-	-	-	-	61.4	-	-	-	-
	23	1625	2.70	5.2	1.2	1.72	1.036	742	14.9	0.012	0.009	57.6	66.8	98.3	97.8	0.5
	24	1612	2.38	4.9	0.7	1.75	1.032	1010	13.5	0.013	0.009	55.3	61.0	98.2	97.4	0.8
85	25	1758	2.67	5.2	0.8	1.70	1.034	865	13.0	0.013	0.010	54.1	59.8	98.5	98.1	0.4
	26	1624	3.06	4.9	1.0	1.72	1.033	735	11.7	0.015	0.012	55.3	56.5	98.3	97.8	0.5
	27	1662	3.12	5.0	0.9	1.80	1.033	849	12.0	0.013	0.013	55.2	55.4	98.0	98.0	0.0
	28	1681	2.97	4.7	1.0	1.96	1.029	865	11.7	0.018	0.015	55.0	55.1	97.9	97.5	0.4
	29	1633	3.22	4.6	0.7	1.90	1.028	806	11.0	0.018	0.017	57.2	55.7	97.3	97.1	0.2
	30	1231	2.96	-	-	1.90	1.022	525	12.5	0.018	0.016				96.9	
		12968						6397								
	June															
	2	1324	3.25	4.9	2.0	1.95	1.028	586	12.8	0.025	0.023	61.5	57.4			
	3	1753	3.10	5.2	1.1	1.80	1.028	802	12.5	0.028	0.021	63.7	51.9			
	4	1503	3.38	5.4	1.5	1.80	1.032	864	13.5	0.025	0.024	65.3	58.8		96.2	
	5	1610	3.49	5.2	1.2	1.70	1.033	870	13.0	0.021	0.017	66.3	61.7	97.4	96.8	0.6
	6	1561	3.55	5.5	1.2	1.4	1.035	703	13.8	0.017	0.014	67.3	63.3	97.1	96.5	0.6
	7	1438	3.19	5.5	1.2	1.4	1.040	721	15.5	0.016	0.011	63.8	62.0	98.1	97.2	0.9
86	8	1615	3.47	5.8	1.0	1.5	1.044	856	16.9	0.015	0.008	61.6	64.9	98.5	97.2	1.3
	9	1576	3.44	4.9	1.9	1.5	1.042	839	17.1	0.019	0.011	62.0	63.6	98.2	96.9	1.3
	10	1587	3.37	5.7	1.2	1.5	1.040	839	17.7	0.016	0.011	65.7	62.8	98.3	97.5	0.8
	11	1644	3.42	3.9	2.8	1.6	1.039	876	16.7	0.013	0.009	64.4	62.8	98.6	98.0	0.6
	12	1721	3.48	5.2	1.8	1.5	1.040	903	17.1	0.016	0.010			98.4	97.4	1.0
	13	1189	3.23					601	17.7	0.015	0.010				97.5	
		18581						9460								
	June															
	16	1166	3.56	6.1	1.1	1.5	1.046	511	17.1	0.017	0.010	-	-	98.3	97.1	1.2
	17	1489	3.71	6.7	1.2	1.6	1.044	845	13.9	0.026	0.017	69.1	65.3	97.3	95.9	1.4
	18	1592	4.78	6.4	0.9	1.8	1.042	910	13.3	0.028	0.021	66.0	64.8	98.0	97.4	0.6
	19	1484	5.31	5.7	2.2	1.8	1.038	721	15.3	0.026	0.023	67.9	69.5	98.0	97.7	0.3
	20	1385	5.64	5.4	1.2	1.9	1.038	664	16.0	0.032	0.024	63.6	68.8	97.9	97.1	0.8
87	21	1729	5.94	5.4	1.2	1.7	1.038	898	15.3	0.036	0.032	59.1	66.2	97.0	96.6	0.4
	22	1755	6.37	5.1	1.1	1.7	1.037	857	17.0	0.044	0.034	61.6	66.5	97.4	96.6	0.8
	23	1710	6.87	6.4	0.4	1.7	1.038	817	15.4	0.042	0.040	60.4	65.4	97.1	97.0	0.1
	24	1720	6.65	5.0	0.9	1.7	1.035	848	15.1	0.045	0.035	62.2	64.9	97.2	96.5	0.7
	25	1758	6.98					876	15.2	0.054	0.054			95.6	95.6	0.0
		15788						7947								

Note: Reducing power is expressed as grams ferrous iron per litre.

(Concluded) -

TABLE 7 (Concluded)

Leaching Results

L.P.P.	Date	Filtrate Data																	
		No. 1 Filtrate						No. 2 Filtrate						No. 3 Filtrate					
		Wt, lb	U_3O_8, g/l	Red Power, g/l	Fe^{+3}, g/l	pH	S.G.	Wt, lb	U_3O_8, g/l	Red Power, g/l	Fe^{+3}, g/l	pH	S.G.	Wt, lb	U_3O_8, g/l	Red Power, g/l	Fe^{+3}, g/l	pH	S.G.
	May 20																		
	20																		
	21																		
	21																		
	22	101	-	-	-	-	-	41	-	-	-	-	-	nil	-	-	-	-	-
	23	592	5.93	11.1	2.2	1.7	1.07	556	1.37	2.9	0.5	1.9	1.02	477	0.22	0.7	0.1	2.1	1.006
	24	606	6.50	11.1	2.3	1.7	1.07	549	1.31	2.9	0.3	1.8	1.02	457	0.18	0.7	0.1	2.0	1.006
85	25	664	6.79	10.8	1.9	1.6	1.07	584	1.20	2.8	0.3	1.8	1.02	510	0.16	0.4	0.1	1.9	1.004
	26	604	6.82	10.4	1.9	1.7	1.06	489	1.26	2.2	0.1	1.8	1.02	531	0.19	0.6	0.1	2.0	1.005
	27	625	7.42	10.4	2.1	1.8	1.06	566	1.35	2.2	0.3	1.8	1.02	471	0.20	0.6	0.2	2.0	1.004
	28	644	7.73	9.9	2.1	1.9	1.06	541	1.27	2.0	0.3	1.8	1.01	496	0.17	0.6	0.1	1.9	1.005
	29	613	7.54	9.2	2.0	1.9	1.05	516	1.11	1.7	0.3	1.8	1.01	504	0.16	0.5	nil	1.9	1.004
	30	436	7.70	-	-	1.8	1.06	411	1.18	-	-	1.9	1.01	384	0.10			2.1	1.002
		4885						4253						3830					
	June 2	512	7.58	11.9	2.9	2.0	1.06	446	1.28	1.7	1.0	2.0	1.01	366	0.13	0.2	0.3	2.0	1.002
	3	727	8.43	9.9	2.6	1.6	1.05	543	1.34	2.7	1.0	1.8	1.006	483	0.13	0.6	0.2	1.9	1.004
	4	523	8.76	11.9	1.2	1.6	1.08	533	1.51	3.0	0.3	1.7	1.020	507	0.19	0.6	0.1	1.8	1.004
	5	534	8.43	11.6	3.3	1.6	1.07	562	1.59	2.9	0.3	1.6	1.020	514	0.17	0.6	0.1	1.7	1.004
	6	513	8.10	12.1	1.0	1.2	1.08	548	1.73	3.5	0.9	1.4	1.022	500	0.22	0.7	0.0	1.5	1.006
	7	487	7.58	13.2	2.4	1.2	1.09	472	1.44	3.2	0.8	1.5	1.022	479	0.28	0.9	0.0	1.7	1.005
86	8	595	7.02	11.6	4.0	1.4	1.08	524	1.82	3.8	0.8	1.5	1.024	496	0.37	1.0	0.1	1.8	1.006
	9	508	7.81	12.7	2.2	1.3	1.09	576	1.64	3.2	1.1	1.5	1.024	492	0.41	0.6	0.2	1.7	1.006
	10	471	8.40	12.8	3.7	1.3	1.09	648	1.91	3.6	1.1	1.6	1.026	468	0.34	0.8	0.1	1.7	1.006
	11	462		11.3	5.4	1.4	1.09	701	1.93	3.1	1.1	1.5	1.022	481	0.27	0.7	0.1	1.75	1.004
	12	486	8.64	11.0	5.8	1.3	1.09	679	2.17	4.8	1.3	1.5	1.027	556	0.37	0.9	0.3	1.65	1.006
	13	338	8.56					478	1.91					373	0.41				
		6156						6710						5715					
	June 16	441						390						335					
	17	527	10.26	15.7	1.2	1.5	1.102	490	2.08	3.6	0.1	1.7	1.024	472	0.33	0.9	0.1	1.8	1.006
	18	537	12.52	13.9	1.2	1.8	1.092	553	2.47	3.0	0.4	1.8	1.021	502	0.34	0.6	0.2	1.9	1.004
	19	519	11.89	11.3	2.6	1.6	1.072	516	2.79	2.8	0.8	1.8	1.020	449	0.44	0.7	0.1	1.9	1.004
	20	535	15.22	13.5	2.9	1.7	1.090	440	2.60	2.6	0.6	1.8	1.018	410	0.39	0.5	0.3	1.9	1.002
87	21	659	13.78	11.6	2.1	1.5	1.080	547	2.83	2.6	1.0	1.7	1.018	523	0.39	0.6	0.2	1.8	1.004
	22	685	14.43	11.2	1.2	1.5	1.076	543	3.09	2.7	0.6	1.7	1.018	527	0.50	0.6	0.2	1.8	1.004
	23	666	15.36	10.8	2.8	1.6	1.074	520	3.25	2.4	1.0	1.8	1.018	524	0.47	0.5	0.2	1.8	1.004
	24	683	15.47	10.8	2.6	1.5	1.076	531	3.01	2.5	0.7	1.7	1.018	506	0.53	0.6	0.1	1.7	1.005
	25	678	15.60	-	-	-	-	599	3.23	-	-	-	-	481	0.48	-	-	-	-
		5930						5129						4729					

Note: Reducing power is expressed as grams ferrous iron per litre.

TABLE 8

Run Composite Assay

Analysed for	U_3O_8	* U_3O_8 (secondary)	ThO_2	Fe (total)	Red Power,	Fe^{+3}	S	TiO_2	Mo	CO_2 (evol)	As	P_2O_5	V_2O_5	F	S/SO_4	Free Acid (H_2SO_4)
LPP 85																
Feed, %	0.60	0.030	<0.001	8.25			3.48	0.77	0.043	0.63	0.03	0.042	0.05	0.06,		
Final residue, %	0.014	0.004	<0.001								0.03	0.054		0.06		
Rewashed final residue, %	0.011															
No.1 filter filtrate, g/l	6.99			10.81	10.28	0.53										
No.2 filter filtrate, g/l	1.31			2.23	2.21	0.02										
No.3 filter filtrate, g/l	0.16			0.50	0.17	0.33										
Pregnant solution, g/l	3.17		0.01	5.10	4.80	0.30		0.07	0.0001		<0.01	0.087	0.05	0.07	7.05	1.05
LPP 86																
Feed, %	0.63	0.039	0.003	8.13			3.40	0.93	0.046	0.63	0.03	0.063		0.05		
Final residue, %	0.019	0.007	0.002								0.03	0.05		0.009		
Rewashed final residue, %	0.012															
No.1 filter filtrate, g/l	7.64			13.0	11.8	1.2										
No.2 filter filtrate, g/l	2.00			3.3	2.9	0.4										
No.3 filter filtrate, g/l	0.41			0.58	0.02	0.56										
Pregnant solution, g/l	3.43		0.003	5.6	5.0	0.6		0.07	0.0002		<0.01	0.29	0.03	0.06	9.11	3.1
LPP 87																
Feed, %	1.29	0.069	0.002	8.28			3.32	0.88	0.048	1.08	0.057	0.09	0.024	0.05		
Final residue, %	0.035	0.010	0.001								0.049	0.06		0.04		
Rewashed final residue, %	0.029															
No.1 filter filtrate, g/l	14.22			12.0	11.6	0.4										
No.2 filter filtrate, g/l	2.75			2.50	2.46	0.04										
No.3 filter filtrate, g/l	0.45			0.44	0.42	<0.02										
Pregnant solution, g/l	5.95		0.002	5.4	4.6			0.04	0.0002		0.01	0.16	0.038	0.05	8.74	0.85

* U_3O_8 dissolved by boiling in 10% sodium carbonate solution without an oxidizing agent for 30 minutes. This is presumed to be indicative of the amount of secondary uranium present.

TABLE 9

Uranium and Size Distribution in Feed and Final Residue Products

Feed	LPP 85			LPP 86			LPP 87		
Mesh Size	Weight, %	U_3O_8, %	U_3O_8 Dist., %	Weight, %	U_3O_8, %	U_3O_8 Dist., %	Weight, %	U_3O_8, %	Dist., %
+ 48	10.6	0.27	4.9	6.4	0.28	2.9	6.3	0.60	2.9
-48+ 65	8.1	0.46	6.3	5.5	0.48	4.3	6.3	1.06	5.1
-65+100	5.9	0.58	5.8	5.1	0.58	4.3	4.6	1.06	3.7
-100+150	13.9	0.67	15.8	13.5	0.56	12.4	15.3	2.27	26.5
-150+200	4.4	0.72	5.4	5.1	0.50	4.2	3.5	1.01	2.7
-200	57.1	0.64	61.8	64.4	0.68	71.9	64.0	1.21	59.1
Total	100.0	0.59	100.0	100.0	0.61	100.0	100.0	1.31	100.0
Run Composite Assay		0.60			0.63			1.29	
Final Residue									
+ 48	9.6	0.025	15.9	7.9	0.032	14.1	4.6	0.059	7.1
-48+ 65	7.5	0.030	14.9	6.2	0.032	11.0	5.3	0.062	8.6
-65+100	6.0	0.022	8.8	5.6	0.027	8.4	4.3	0.072	8.1
-100+150	14.5	0.015	14.5	14.0	0.019	14.8	15.6	0.046	18.7
-150+200	4.6	0.012	3.7	5.2	0.014	4.1	3.7	0.040	3.8
-200	57.8	0.011	42.2	61.1	0.014	47.6	66.5	0.031	53.7
Total	100.0	0.015	100.0	100.0	0.018	100.0	100.0	0.038	100.0
Run Composite Assay		0.014			0.019			0.036	

The screen analyses of the feeds (Table 9) show that the ore for the first pilot plant run was coarser than that used in the other two. However, previous bench scale tests showed that this variation would not affect extraction to any degree. About 10% more of the total uranium of the feed was in the minus 200 mesh size fraction in LPP 86 than in LPP 85 or LPP 87. This might tend to improve extraction, but since the same conditions were not used in each run, the data do not show the effect of this variable.

The screen analysis of the final residue (Table 9) showed that LPP 87 had about 5% more weight in the minus 200 mesh fraction than LPP 86, which in turn had about 3% more of this size than the residue from LPP 85. Variations of the same order may be noted in the analysis of the uranium distribution in the minus 200 mesh fractions of the final residues.

Data showed that leaching at pH 1.5 with 5 lb $NaClO_3$ per ton ore gave essentially the same extraction as pH 1.2 with 5 lb $NaClO_3$. However, the former conditions required 47 lb 100% H_2SO_4 per ton ore less than the latter (Table 1). Leaching at pH 1.8 with 5 lb $NaClO_3$ per ton ore reduced the acid consumption by about 7 lb 100% H_2SO_4 per ton ore but also reduced the uranium extracted by about 0.1 lb per ton solids. At an acidity of pH 1.2 the filtering characteristics were very poor and the cloth on the first filter blinded very quickly.

With both grades of ore samples leached at pH 1.5, about 93% of the extraction took place in the first agitator, 3 or 4% in the second agitator, and 1 or 2% over the last two agitators (Table 6). At pH 1.2, using the low grade ore, about 96% of the leaching took place in the first agitator, another 2% in the second, and approximately 0.5% extraction over the last two agitators.

Changing the point of the chlorate addition to the second agitator did not affect the overall extraction but the amount extracted in the first agitator was reduced slightly (Table 7, page 15).

Daily acid consumption values varied as much as 34 lb of 93% H_2SO_4 per ton ore in LPP 86, while maintaining pH 1.5. This was exclusive of planned acid level changes which required increased acid consumption (Table 6, page 14).

Bench scale leach tests gave similar results to pilot plant runs when identical samples were used. Laboratory techniques showed that high tailing residues in LPP 87 could be brought down appreciably with the addition of more chlorate and/or much finer grinding along with longer contact time (Table 11)*. Tailings assaying 0.052% U_3O_8 were reduced to 0.010% U_3O_8 in 48 hr of extended leaching when 5 lb $NaClO_3$ per ton of tailings was added. Grinding the tailings to 90% minus 200 mesh and leaching for 48 hours brought the tailings down to 0.009% U_3O_8.

The pilot plant sample reacted somewhat differently than previous ore samples from the same property, in that no hydrogen sulphide gas was evolved in leaching. An attempt was made to determine if it was the high graphite content of the pilot plant sample that prevented the gas from forming. This was tested by removing the graphite by flotation

* Page 27, Appendix 2

before leaching (Appendix 2). Although approximately one-half of the graphite was floated, about one-half of the sulphides also floated. No changes in the leaching characteristics were noted. A mineralogical investigation of the heavy fractions of the pilot plant and previous samples did not disclose any significant differences that would explain this variation in chemical reaction (Appendix 3).

The E.M.F. readings, taken at various locations in the leach circuit, were found to be positive in nearly all cases and, by local convention, the pulp was considered to be in an oxidizing state (Appendix 4).

Polythionate analysis of solutions from various locations in the leach circuit showed that this compound was first formed in the grinding circuit but was largely removed by the neutral filtration step prior to leaching. The second marked concentration of polythionates appeared in the third leach agitator (Appendix 5). One bench scale leach test showed that polythionate content could be increased fourfold by aeration of the acid pulp during leaching (Appendix 1).

Tailings neutralization tests with Alcan hydrated lime (Appendix 6) indicated that consumptions of about 23 to 30 lb of reagent per ton of ore could be expected. This range was considered normal for acid leach operations of this type.

Settling tests carried out on neutral and acid pulps from the pilot plant indicated that no trouble would be encountered, with ore of this type, in this solids-liquids separation step (Appendix 7). An addition of 0.05 lb of flocculating agent per ton of ore reduced the area required for settling acid or neutral pulps to about 0.2 to 0.3 sq ft per ton per day. This is well below that of any thickener design used in practice.

REFERENCE

1. H. H. McCreedy, Preliminary Extraction Tests on Uranium-bearing Ore Samples from the Kitts Property of British Newfoundland Explorations Ltd., Newfoundland, Reference Numbers 12/57-9, 12/57-20, 2/58-14 and 3/58-16. Mines Branch Investigation Report IR 58-150, Department of Mines and Technical Surveys, Ottawa, August 8, 1958 (Industrial Confidential).

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(Appendices 1 to 7 follow,)
(on pages 23 to 39.)

APPENDIX 1

Bench Scale Leach Test Details

Two leach tests were run on the low grade sample (No.5/58-4) using 1000-gram charges ground in Abbé porcelain laboratory ball mills with steel balls (4 A and 4 B). Another two tests were run on ore ground for the pilot plant (4 C and 4 D). In test 4 D, barren effluent from the ion exchange plant was used for pulp make-up. All samples were leached at 60% solids for 48 hours at room temperature. Results are shown in Table 10.

Several small scale leach tests were carried out on the high grade pilot plant feed and tailings (No. 4/58-12). Three tests (Nos.12A, 12B and 12C) were carried out on laboratory ground ore. Intermediate residue samples were taken to check the leaching rate. No heat was added to any of the tests.

The feed for one test (12 B) was made up of the rougher and cleaner tailings products from a flotation test carried out to remove the graphite before leaching. The flotation details are described in Appendix 2.

To study possible polythionate formation, one test (12 D) was tried with air being bubbled through the pulp during leaching.

Five tests (Nos. 12E to 12J inclusive) were performed on the final leached solids from the pilot plant (LPP 87). In one case (No.12G), extra chlorate was added to the repulped tailings at pH 1.5. In another test (No. 12J) the tailings were reground to 90% minus 200 mesh and

TABLE 10

Bench Leach Test Data, Low Grade Sample No. 5/58-4

	4A	4B	4C	4D
Grind, %-200 mesh	65	65	57	60.0
Sodium chlorate added, lb/ton	5	5	5	5
pH controlled at	1.5	1.0	1.5	1.5
Acid consumption, lb 100% H_2SO_4/ton ore,				
at 0 hr	46	56	44	42
at 8 hr	64	84	-	70
at 24 hr	72	122	82	84
at 32 hr	75	126	-	-
at 48 hr	80	130	96	94
Residue assay, % U_3O_8				
at 8 hr	0.018	.0.013	0.040	0.041
at 24 hr	0.016	0.012	0.022	0.016
at 32 hr	0.010	0.011	0.014	0.013
at 48 hr	0.007	0.009	0.010	0.010
Pregnant:				
ml	360	440	320	310
U_3O_8, g/l	9.95	7.43	10.63	9.62
U_3O_8, % Dist.	56.1	57.9	58.0	54.9
$NaClO_3$, g/l		nil	nil	nil
Reducing power, g/l	12.6	15.5	21.8	21.0
Total Fe, g/l	12.7	19.1	21.8	21.0
Wash:				
ml	775	780	790	810
U_3O_8, g/l	2.54	1.94	3.00	2.91
U_3O_8, % Dist.	42.9	40.6	40.4	43.4
$NaClO_3$, g/l	nil	nil	nil	nil
Reducing power, g/l	3.2	4.2	6.7	6.4
Total Fe, g/l	3.2	5.1	6.7	6.5
Final residue, g	913	925	962	897
% U_3O_8	0.007	0.009	0.010	0.010
U_3O_8, % Dist.	1.0	1.5	1.6	1.7
Calc. feed assay, % U_3O_8	0.64	0.61	0.61	0.61
Extraction based on calculated feed assay, %	99.0	98.5	98.4	98.3
Remarks:	Laboratory ground feed	Laboratory ground feed	Pilot plant ground feed	Barren effluent

leached for another 48 hours under the same conditions as used in the leach pilot plant (No. 87). The combination of regrinding along with increased oxidizing agent was used in still another test on the tailings (No. 12H).

Slower rate of addition of sodium chlorate, along with increased leach contact time to the high grade feed sample, was also tried as a method of increasing uranium extraction (12 K to 12 L).

Experimental data indicated that leaching the lower grade ore at pH 1.0 produced the same extraction as when pH 1.5 was used (4 A and 4B, Table 10). The higher acid content resulted in increased pulp filtering difficulties. On the samples of pilot plant feed, bench scale tests gave similar results to those obtained in the pilot plant. An extraction of 98.4% was realized in the bench scale (No. 4C), compared to a 98.5% extraction average for LPP 85 (Table 1). An acid consumption of 96 lb of 100% H_2SO_4 per ton ore in the bench scale test (No. 4C) compared to the average equivalent of 85.5 lb of 100% H_2SO_4 per ton of ore in the pilot plant i.e. 92 lb of 93% H_2SO_4/ton ore.

The use of ion exchange barren effluent for initial pulp make-up appeared to save only 2 lb of H_2SO_4 per ton ore to produce the same extraction (4 D compared to 4 C). This small difference is well within the variation of acid consumption caused by sample variation.

Sample variation was more prevalent in the high grade ore sample (No. 4/58-12) and therefore caused more variation in extraction values.

Leaching at pH 1.2 gave similar extraction to pH 1.5 but with increased acid consumption. The pH 1.2 test (12 A) gave an extraction of 98.7% with an acid consumption of 148 lb 100% H_2SO_4 per ton ore, compared to 97.7% extraction and an acid consumption of 126 lb 100% H_2SO_4 at pH 1.5 (12 A and 12 C, Table 11).

Removal of about half of the graphite and half of the sulphides by flotation did not materially affect the leaching characteristics of the ore (12 B). Acid consumption and extraction were essentially the same.

Aeration of the pulp during leaching did not affect the overall extraction but increased the polythionate content of the leach liquor approximately four-fold (12 D compared to 12 C).

Leaching tests on tailings solids indicated that increased contact time alone under pilot plant conditions would not result in a significant increase in extraction (12 E and 12 F). However, the addition of 5 lb $NaClO_3$ per ton tailings extracted approximately 80% of the uranium in the tailings. This would represent an increase in extraction from the original feed of about 3% (12 G).

Regrinding the tailings to 90% minus 200 mesh, and maintaining pilot plant conditions for another 48 hours without any further oxidizing agent, gave an extraction of about 82% of the uranium in the tailings. Regrinding, together with 5 lb $NaClO_3$ per ton tailings, raised the extraction of uranium from the tailings to about 86%.

TABLE 11

Bench Leach Test Data - High Grade Sample No. 4/58-12

Test No.	Feed						Tails				
	12 A	12 B	12 C	12 D	12 K	12 L	12 E	12 F	12 G	12 H	12 J
Grind, %-200 mesh	65	65	60	60	60	60	65	65	65	90	90
Sodium chlorate added, lb/ton	5	5	5	5	8*	9*	0	0	5	5	0
pH controlled at:	1.2	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5
Acid consumption, lb. 100% H_2SO_4/ton ore, at:											
0 hr	94	69	60	58	64	64	10	0	8	36	18
8 hr	106	-	92	94	94	100	-	-	26	52	44
24 hr	128	118	110	110	116	120	122	12	36	64	58
32 hr	-	-	114	-	-	-	-	-	-	-	-
48 hr	148	129	126	122	134	134	38	22	40	72	72
96 hr	-	-	-	-	160	162	-	-	-	-	-
Residue assay, % U_3O_8, at:	-	-	-	-	160	162	-	-	-	-	-
8 hr	0.06	0.11	0.040	0.068	0.17		0.037	-	0.018	0.009	0.010
24 hr	0.037	0.098	0.024	-	0.083	0.061	0.036	0.044	0.012	0.007	0.009
32 hr	-	-	0.024	-	-	-	-	-	-	0.006	-
48 hr	0.023	0.027	0.023	0.023	0.027	0.017	0.037	0.043	0.010	0.007	0.009
96 hr	-	-	-	—	0.012	0.012	0.012	-	-	-	-
Pregnant:											
ml	495	405	450	410	390	405	370	410	550	380	460
U_3O_8, g/l	16.56	15.36	13.5	16.7	18.28	20.80	-	19.7	0.55	0.60	0.52
U_3O_8, % dist.	68.3	73.0	66.5	69.2	62.4	68.6	-	63.3	64.4	49.0	51.0
Reducing power, g/l	15.3	7.3	14.1	14.4	-	23.8	-	21.4	10.9	19.3	19.1
Total Fe, g/l	15.5	7.3	0.6	14.4	-	24.1	-	22.0	15.1	21.4	20.1
Wash:											
ml	790	790	770	765	745	770	790	800	740	750	730
U_3O_8, g/l	4.62	2.68	3.71	3.72	5.62	4.86	-	5.33	0.10	0.23	0.20
U_3O_8, % dist.	30.4	24.8	31.2	28.7	36.6	30.5	-	33.4	15.8	37.1	31.2
Reducing power, g/l	4.1	1.7	3.9	3.2	-	5.2	-	4.7	1.9	7.6	nil
Total Fe, g/l	4.2	1.7	0.1	3.2	-	5.3	-	4.7	2.5	8.1	6.1
Final Residue,											
g	885	697	904	903	919	926	905	964	932	920	927
% U_3O_8	0.023	0.027	0.023	0.023	0.012	0.012	0.037	0.043	0.012	0.007	0.009
U_3O_8, % dist.	1.3	2.2	2.3	2.1	1.0	0.9	-	3.3	19.8	13.9	17.8
Calc. feed assay, % U_3O_8	1.36	1.22	1.01	1.10	1.24	1.33	-	1.28	0.050	0.051	0.051
Extraction based on calculated feed assay, %	98.7	97.8	97.7	97.9	99.0	99.1	-	96.7	80.2	86.1	82.2
Remarks:		Feed from float tails.	Preg. Poly. content 0.39 g/l.	Conditions similar to 12 C but with aeration. Preg. Poly content 1.62 g/l.	*$NaClO_3$ added at 2 lb per 24 hr for 4 days.	* $NaClO_3$ added at 3 lb per ton per 24 hr for 3 days.	Repulped pilot plant tailings. No material balance made.	No. 4 agitator pulp.	Repulped pilot plant tailings plus extra $NaClO_3$.	Repulped reground tailings plus extra chlorate.	Repulped reground tailings; no chlorate.

When the total chlorate was added at the rate of 2 lb per ton ore per 24 hr for 4 days the uranium extraction from the original ore was 99% in 96 hours (12 K). Chlorate at the rate of 3 lb per ton ore per 24 hr for 3 days gave essentially the same extraction in 96 hours, that is, 99 % (12L).

APPENDIX 2

Flotation Test Details - High Grade Ore

The purpose of the test was to produce a graphite-free ore sample for leaching. A sample of ore weighing 1150 grams was ground with 500 ml of water for 20 minutes. This gave a product containing approximately 65% minus 200 mesh. The pulp was filtered and 1000 grams, dry weight, were repulped to about 35% solids. The pH of the pulp was 7.6 but was adjusted to 9.1 with the addition of one pound of Na_2CO_3 per ton solids. One pound of kerosene per ton of solids and 0.05 lb of pine oil, per ton of solids were added. After a conditioning period of three minutes, flotation was carried out for five minutes. The rougher concentrate was returned to a flotation cell and cleaned for five minutes without any further reagent addition. The cleaner tailing and the rougher tailing products made up the acid leach feed.

The results of the flotation test are shown in Table 12 and the results of the leach test on the flotation tailings are shown in Table 11 (12B).

TABLE 12

Flotation Test Results - High Grade Ore 4/58-12

Product	Wt, %	U_3O_8, %	Dist. U_3O_8, %	S, %	Dist. S, %	C, %	Dist. C, %
Cleaner Conc.	18.5	0.18	3.1	9.30	55.7	29.54	51.0
Cleaner Tailing	19.5	0.56	10.1	3.89	24.6	10.90	19.9
Rougher Tailing	62.0	1.51	86.8	0.98	19.7	5.02	29.1
Total	100.0	1.08	100.0	3.1	100.0	10.7	100.0

APPENDIX 3

A Memorandum Comparing Superpanner Concentrates of Uranium Ore Samples Nos. 2/58-14 and 4/58-12 from British Newfoundland Exploration Ltd.

The purpose of this investigation was to compare the mineralogical composition of gravity concentrates of two samples of ore from British Newfoundland Exploration Ltd. as a possible indication of the difference in behaviour when subjected to acid leaching*. Superpanner concentrates of the -65+100 mesh fractions from equal weights of ore samples Nos. 2/58-14 and 4/58-12 were prepared by Mr. W.R.Honeywell and submitted for mineralogical examination: they weighed 2.412 grams and 5.630 grams respectively.

The mineral composition of each superpanner concentrate was determined by two methods. After riffling in two, one half of each concentrate was separated magnetically and the fractions analysed by grain counting with a low power microscope. A polished section was made of the other half of each concentrate and the mineral composition determined with an ore microscope and a Swift Point Counter. The results are shown in Table 13. Grains of gangue mineral were not included in the determination. Also,grains of pitchblende which contain gangue minerals (feldspar and quartz) were considered to be pure pitchblende,since an accurate estimate could not be made of the intimately intergrown gangue. The values for pitchblende are consequently high but can be used for purposes of comparison.

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* After acid addition, the pulp, using sample No. 2/58-14, had a reducing potential and considerable H_2S was given off, while pulps using sample No. 2/58-12 had an oxydizing potential and little H_2S was observed.

TABLE 13

Mineralogical Composition of Superpanner Concentrates -65+100 Mesh

Mineral	Sample No. 2/58-14			Sample No. 4/58-12		
	(1) Wt%	(2) Wt%	(3) Wt%	(1) Wt%	(2) Wt%	(3) Wt%
Pyrrhotite	58.2	53.0	3.03	75.8	67.1	8.00
Pitchblende	35.6	30.9	1.81	23.4	28.0	2.89
Pyrite	5.6	13.1	0.51	0.2	3.6	0.21
Chalcopyrite	0.3	1.1	0.04	0.2	0.6	0.04
Arsenopyrite	0.3	1.9	0.06	0.4	0.7	0.06
Total	100.0	100.0	5.45	100.0	100.0	11.20

Column (1) Composition of superpanner concentrate determined from magnetic fractions.
(2) Composition of superpanner concentrate determined by Swift Point Counter.
(3) Average of (1) and (2) expressed as wt % of the -65+100 mesh sample

From Table 13 it can be seen that a considerable increase in the amount of pyrrhotite, approximately 2 1/2 times, in sample No.4/58-12 compared to sample No. 2/58-14 is the chief reason for the larger superpanner concentrate obtained from the former sample. Pitchblende to a lesser extent also contributes to the larger superpanner concentrate from sample No. 4/58-12. The other metallic minerals, pyrite, chalcopyrite, and arsenopyrite occur in very small amounts and have little effect on the size of the superpanner concentrates.

APPENDIX 4

E.M.F. Measurements

E.M.F. readings were taken with a platinum-calomel electrode combination connected to a Beckman pH meter. Values were recorded on a number of days during each run when various conditions were in effect. Data were taken from the neutral No. 1 feed repulper and the four agitators.

With the platinum electrode connected to the upper positive terminal of the Beckman pH meter, all readings in the agitators were positive and therefore by local convention considered to be oxidizing in nature. In only one spot reading was a negative value obtained (-40) and that was in the neutral No. 1 feed repulper during LPP 86 (June 3). The No. 1 agitator had the highest positive reading (+380) during LPP 86 (June 9).

Similar trends were witnessed with both types of ore and no apparent irregularities detrimental to the leaching mechanism were noted.

The data are reported in Table 14.

TABLE 14

E.M.F. Measurements at Various Locations in Pilot Plant Runs, mv

Acid and Chlorate Conditions	LPP 85			LPP 86									LPP 87		
	pH 1.5, 5 lb $NaClO_3$	pH 1.8, 5 lb $NaClO_3$		pH 1.5, 3 lb $NaClO_3$			pH 1.2, 5 lb $NaClO_3$			pH 1.2, 5 lb $NaClO_3$ (Liquid to No.2)			pH 1.5, 5 lb $NaClO_3$		
	May 25	May 26	May 27	June 3	June 4	June 5	June 6	June 7	June 9	June 10	June 11	June 12	June 18	June 19	June 22
No.1 Repulper				-40			+25	0	0	0	0	0	+30	+25	+70
No.1 Agitator	+372	+368	+340	+350	+350	+360	+370	+360	+380	+315	+290	+260	+335	+340	+305
No.2 Agitator	+339	+328	+335	+300	+335	+350	+330	+330	+350	+310	+378	+355	+292	+310	+290
No.3 Agitator	+325	+316	+325	+290	+310	+320	+310	+310	+330	+340	+368	+330	+290	+290	+270
No.4 Agitator	+313	+305	+315	+260	+290	+300	+282	+290	+325	+320	+350	+328	+280	+275	+255

APPENDIX 5

Polythionate Content of Leach Solutions

Pulp samples were taken from the leach agitators, and the filtrates assayed for polythionates to note any build-up of these compounds. The pregnant solution and agitator liquors were sampled simultaneously and both solutions analysed for polythionate content (Table 15).

The data show that there was a marked increase in polythionate content in the grinding circuit and in the third agitator. There was no appreciable difference in polythionate concentration in liquors produced from the high and low grade samples.

TABLE 15

Polythionate Content of Leach Solutions
(Assays in g/l)

	LPP 86 (Start-up) (June 2)	LPP 86 (June 9)	LPP 87 (June 24)
No.1 Agitator liquor	0.01	0.005	0.004
No.2 Agitator liquor	0.03	0.008	0.009
No.3 Agitator liquor	0.17	0.17	0.20
No.4 Agitator liquor	0.20	0.29	0.32
Pregnant solution	0.14	0.18	0.18
Neutral grinding filtrate	0.28	-	-

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APPENDIX 6

Pilot Plant Tailings Neutralization Tests

Two tests were run on the final residue solids from the pilot plant to determine the amount of hydrated lime required for neutralization. The neutralization tests were carried out on the low grade tailings from pilot plant LPP 86 and the high grade tailings from pilot plant LPP 87. Both of the samples had been leached at pH 1.5.

Four thousand grams of tailings, dry weight, was mixed with 2, 545 ml of barren effluent at pH 1.75. Alcan hydrated lime was added to the slurry until pH 7.5 was reached. A contact time of one hour was allowed.

The consumption of neutralizing agent was 23 lb/ton ore and 30 lb/ton ore for the tailings produced from low grade and high grade feed respectively.

APPENDIX 7

Settling Test Details

Each test was carried out on approximately a 1000 ml sample of diluted pulp. The acid pulp was taken from the last agitator in the leaching circuit. The neutral pulp was made up from the neutral filter cake discharged in the grinding circuit. The pulp was diluted to 30-35% solids for testing. The pulp was agitated end over end in a graduated cylinder, at room temperature, after the settling agents had been added. All reagents were made up as 0.5% solutions. The height of the pulp was recorded as it settled.

The settling data are shown in Tables 16, 17 and 18. The screen analyses of samples, believed to be representative of the tests, are shown in Table 19.

The thickening characteristics of the ore appeared to be similar to other uranium ores tested previously. With neutral pulps of the low grade composite ore tested during LPP 86, it was found that adjusting the neutral pulp with approximately 1 lb H_2SO_4 per ton ore gave better clarity to the supernatant liquor but did not materially affect the rate of settling, with or without the addition of a flocculating agent (Test 86 C compared to 86 G, and 86 A compared to 86 B). The neutral pulp required an area of 1.5 sq ft/ton/day to settle without any reagents, and 0.5 sq ft/ton/day after the addition of 0.025 lb Jaguar MDC per ton ore (Tests 86 A and 86 D). Jaguar MDC, Separan 2610, and S-3171, at a concentration of 0.0125 lb reagent per ton ore, resulted in area requirements

TABLE 16

Settling Test Details - Pilot Plant LPP 85
Acid Pulp

Test No.	85 A	85 B	85 C	85 D	85 E	85 F	85 G	85 H	85 J
Net Wt, g	1300	1304	1303	1302	1303	1303	1300	1313	1306
Pulp,									
Vol, ml	1000	1005	1000	1000	1000	1000	1000	1000	1000
Specific gravity	1.300	1.298	1.303	1.302	1.303	1.303	1.300	1.313	1.306
% Solids	35.9	35.7	36.2	36.1	36.2	36.2	35.9	37.1	36.5
Reagent added,									
Type	Nil	Jaguar C	Separan C	SE Glue	Jaguar C	Polyox	Separan C	Separan	Separan
lb/ton	-	0.025	0.025	0.1	0.05	0.05	0.05	0.05	0.025
Clarity of SNL*	Extra Cloudy	V Hazy	Extra Cloudy	Cloudy	Hazy	Cloudy	Extra Cloudy	Hazy	Cloudy
Rate of settling, ft/hr	-	4.57	1.48	1.27	5.28	1.48	1.41	2.81	2.11
Initial liquids to solids ratio	1.79	1.80	1.76	1.77	1.76	1.76	1.79	1.70	1.74
Final liquids to solids ratio	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.67
Required settling area, sq ft/ton/day (by Coe-Clevenger)		0.33	0.98	1.15	0.27	0.98	1.06	0.49	0.67
Height, ml, at:									
0 min	1000	1000	1000	1000	1000	1000	1000	1000	1000
2 min		870	955	-	850	965	960	920	940
5 min		740	895	910	680	910	900	810	860
10 min		600	800	835	540	815	810	650	765
15 min		538	705	780	505	740	730	565	695
20 min		510	670	735	485	690	680	540	630
25 min		495	650	690	470	665	650	520	585
30 min		485	625	655	460	630	620	510	555
19 hr									

* Supernatant liquid.

TABLE 17

Settling Test Details - Pilot Plant LPP 86
Neutral Pulp

Test No.	86 A	86 B	86 C	86 D	86 E	86 F	86 G
Net Wt, g	1233	1240	1245	1242	1243	1233	1240
Pulp,							
Vol, ml	1000	1000	1000	1000	1000	1000	1000
Specific Gravity	1.233	1.240	1.245	1.242	1.243	1.233	1.240
% Solids	29.4	30.1	30.6	30.3	30.4	29.4	30.1
Reagent Added,							
Type	Nil	H_2SO_4	Jaguar C	Jaguar C	Separan	S 3171	H_2SO_4
lb/ton		1	0.0125	0.025	0.0125	0.0125	1
Type							Jaguar C
lb/ton							0.0125
Initial liquids to solids ratio	2.44	2.32	2.27	2.30	2.29	2.44	2.32
Clarity of SNL	Cloudy	Hazy	Very Hazy	Clear	Very Hazy	Very Hazy	Clear
Rate of settling, ft/hr	1.59	1.43	3.34	4.75	3.34	3.52	3.87
pH of SNL	7.8	6.2	7.8	7.8	7.8	7.6	6.2
Final liquids to solids ratio	0.67	0.67	0.67	0.67	0.67	0.67	0.67
Required area, sq ft/ton/day (by Coe-Clevenger)	1.48	1.53	0.64	0.46	0.65	0.67	0.57
Height, ml, at:							
0 min	1000	1000	1000	1000	1000	1000	1000
2 min	980	980	905	865	905	900	890
5 min	945	945	755	705	750	740	735
10 min	895	880	620	525	595	570	575
15 min	845	815	510	460	500	490	485
20 min	795	750	465	430	460	445	435
25 min	735	680	435	415	440	425	420
30 min	680	610	420	405	425	410	405
35 min	625	550		400	405		
60 min	430		380	375	380		
19 hr			355	360	355	340	345

TABLE 18

Settling Test Details - Pilot Plant LPP 87

	Neutral Pulp					Acid Pulp				
Test No.	87 A	87 B	87 C	87 D	87 E	87 F	87 G	87 H	87 J	87 K
Net Wt, g	1250	1249	1253	1253	1248	1260	1256	1256	1255	1262
Pulp,										
Vol, ml	1000	1000	1000	1000	1000	1000	1000	1000	1000	1000
Specific Gravity	1.250	1.249	1.253	1.253	1.248	1.260	1.256	1.256	1.255	1.262
% Solids	31.1	31.0	31.4	31.4	30.9	32.1	31.7	31.7	31.6	32.3
Reagents added,										
Type	Nil	H_2SO_4	Jaguar C	Separan	S-3171	Jaguar C	Jaguar C	Separan	S-3171	Separan
lb/ton		1	0.0125	0.0125	0.0125	0.025	0.05	0.05	0.05	0.025
Clarity of SNL	Cloudy	Hazy	Hazy	Hazy	Hazy	Hazy	Hazy	Hazy	Hazy	Hazy
Rate of settling, ft/hr	0.95	0.88	3.52	5.10	6.86	7.03	7.74	5.45	5.10	4.87
pH of SNL	7.6	6.2	7.6	7.6	7.6	2.5	2.5	2.5	2.5	2.5
Initial liquids to solids ratio	2.22	2.23	2.18	2.18	2.24	2.12	2.15	2.15	2.16	2.10
Final liquids to solids ratio	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.67
Required area, sq ft/ton/day (by Coe-Clevenger)	2.17	2.36	0.57	0.39	0.29	0.27	0.25	0.36	0.39	0.39
Height, ml, at:										
0 min	1000	1000	1000	1000	1000	1000	1000	1000	1000	1000
2 min	975	985	900	855	805	800	780	845	855	895
5 min	935	950	750	665	600	630	555	650	680	770
10 min	870	885	615	490	445	500	450	480	490	615
15 min	800	820	505	430	410		420			
20 min	730	750	445	405	390	425	400	420	425	430
25 min						410	390	405	410	410
30 min	590	610	410	380	360	400	380	400	400	395
35 min	525	550	400	370	355					
60 min	370	380	370	350	350	370	365	380	380	400
19 hr	330	330	350	335	340	355	360	370	370	380

TABLE 19

Representative Screen Analyses of Settling Test Samples

Pulp Type	Acid	Neutral	Neutral	Acid
Mesh Size	Tests 85A to J	Tests 86A to G	Tests 87A to E	Tests 87F to K
Wt %	Wt %	Wt %	Wt %	Wt %
+ 48	9.3	5.6	3.6	5.0
- 48+ 65	8.1	5.3	5.0	6.3
- 65+100	7.1	4.8	4.0	4.7
-100+150	15.3	14.7	15.2	15.5
-150+200	4.8	4.3	3.1	3.6
-200	55.4	65.3	69.1	64.9
Total	100.0	100.0	100.0	100.0

of about the same value (Tests 86 C, 86 E and 86 F). The clarity of the supernatant liquor was, more or less, the same under all these conditions.

The neutral pulp from the high grade ore tested in LPP 87 required a settling area of approximately 2.2 sq ft/ton/day. The addition of 0.0125 lb S-3171 per ton ore gave the lowest calculated area required of the three reagents used at this concentration. This lowest value was 0.29 sq ft/ton/day. Since the neutral pulp settled easily with this reagent concentration no higher amount was tried.

The acid pulp tested in LPP 85 from the low grade feed, produced a very cloudy interface between the solids and liquids and therefore the rate of settling and required area of settling could not be calculated. Jaguar MDC appeared to give the lowest area required for settling, as well as the best clarity of overflow, when used in acid pulp. 0.05 lb Jaguar MDC gave a value of 0.27 sq ft/ton/day with low grade ore (Test 85 E). When the acid pulp of high grade ore was used, the addition of 0.05 lb Jaguar MDC gave a required settling area of 0.25 sq ft/ton/day (Test 87 G). The clarity of all supernatant liquors produced from high grade acid pulp appeared to be about the same when reagents were added.

It can be seen that the addition of a comparatively small quantity of flocculating agent will appreciably reduce the area required for settling acid or neutral pulp of a type similar to that used in these tests.

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HHM/WAG/dm