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REPORT ON PILOT PLANT FLOTATION OF URANIUM ORE FROM DENISON MINES LIMITED

by

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EXTRACTION METALLURGY DIVISION

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SUMMARY

A flotation pilot plant for the recovery of uranium was operated for 32 days during April and June, 1964, with the object of producing a uranium concentrate suitable for further treatment by leaching.

By flotation, the best recoveries obtained ranged from 85 to 94 per cent of the uranium with ratios of concentration of from 2.4 to 2.0:1 respectively. By gravity concentration and flotation, the best recovery obtained was from 85 to 90 per cent of the uranium with ratios of concentration of 3.4 to 2.5:1 respectively. The principal uranium collector used was an emulsion of Acintol FA-1. The total cost of reagents would be in the order of from 21 to 31 cents per ton ore.

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INTRODUCTION

Over the past several years, the Extraction Metallurgy Division has conducted considerable test work investigating the use of flotation applied to the Elliot Lake, Ontario uranium ores. The purpose of the work was to develop a technique whereby a satisfactory uranium-bearing preconcentrate could be produced by flotation for subsequent treatment by hydrometallurgical methods. For this process to be economically acceptable the combined cost of flotation and leaching the preconcentrate plus the value of the uranium lost to the flotation tailings would have to be less than the cost of leaching the whole ore as is presently done.

The flotation test work done up to 1959 was reported in Mines Branch Technical Bulletin TB-2 (1) in which the results of a series of tests on various Elliot Lake ores are given. The average recovery obtained was 92 per cent of the uranium with a ratio of concentration of 1.8:1. The procedure used involved desliming of the ore using sodium hydroxide and sodium silicate as dispersants followed by a bulk uranium float using Acintol FA-1 or Acintol FA-2 as the collector. These latter products are tall oil fatty acids and were obtained from Charles Albert Smith, Limited of Montreal, Quebec. Although these results were encouraging, they were not entirely acceptable since the ratio of concentration was too low, and the value of uranium lost in the flotation tailings was considered to be too high.

Subsequent to 1959, further extensive laboratory work based on the method described in TB-2 was conducted. In this work, which was done on ore from Denison Mines Limited, plant practice was simulated in the laboratory by cyclic testing in which products such as the cleaner tailing and the scavenger concentrate from one test were retreated along with new feed in the following test. In one series of cyclic tests the sulphides were bulk-floated along with the uranium using Acintol FA-1, while in another series, the sulphides were removed using xanthate-type collectors before desliming and uranium flotation. In both series the slimes and final flotation concentrates were combined to form the preconcentrate. Both of these methods produced preconcentrates containing about 85 per cent of the uranium with a ratio of concentration of 2.3:1. Comparing these results with those reported in TB-2, it can be seen that the cyclic tests produced a lower uranium recovery but a higher ratio of concentration than did the bulk float. However, there remained the possibility that continuous recycling of the cleaner tailing and the scavenger concentrate in a plant operation would provide better results than those obtained by simulating continuous operation with laboratory equipment.

During the period 1959-1962, work was done by other investigators to study the flotation of uranium minerals from the Elliot Lake ores (2) (3). The results of these investigations showed, as did the Mines Branch work, that to obtain a high uranium recovery, a low ratio of concentration had to be accepted. In these studies the collectors used for uranium flotation were oleic acid (2) or organic phosphates (3) both of which are more expensive than the Acintol FA-1 or FA-2 used in the Mines Branch work.

In view of the laboratory work it was decided at the 1963 fall meeting of the Canadian Uranium Producers' Metallurgical Committee to conduct a flotation pilot plant run on uranium ore from Denison Mines Limited to further test the procedures developed in the laboratory. It was agreed that the pilot plant test was desirable on the grounds that it would produce more reliable data than the laboratory tests on which an assessment of the use of flotation in treating the Elliot Lake ores in the future could be based.

At this meeting it was pointed out that, although the loss of uranium in the flotation tailing was comparatively high, there was now a possibility of recovering the uranium in the tailing by heap leaching with bacteria. Bacterial leaching of uranium in the underground workings does, in fact, take place and if this phenomenon could be applied to the flotation tailing, the need for a high recovery in flotation would become less significant.

In summary then it was decided to conduct a pilot plant run on ore from Denison Mines Limited with the following three main objectives in mind:

1. to determine if the result of the laboratory work could be equalled or improved by continuous operation;

2. to provide a flotation preconcentrate for hydrometallurgical studies so that the combined value of flotation followed by various leaching techniques could be assessed;

3. to provide a quantity of flotation tailing for bacterial leaching test work.

PROCEDURE

A 25 ton shipment of run-of-mine ore was received from Denison Mines Limited, in March, 1964 and a further shipment of 25 tons was received in May, 1964. The pilot plant runs were started soon after the shipments arrived in order to minimize changes by oxidation. The samples were typical of the quartz-pebble conglomerate of the area. Brannerite, uraninite and monazite occur in the matrix of the conglomerate and fine grained pyrite

is associated with the radioactive minerals. The brannerite is fine-grained and often intergrown with other minerals, notably rutile and anatase.

The run-of-mine ore, which ranged from fines to pieces one foot or more in diameter, was crushed at the Mines Branch by means of a 12 x 6 in. jaw crusher, a 20 in. Symons cone and a 4 x 1/4 in. Hazemag Impact crusher to produce minus 4 mesh feed for the grinding circuit. Crushing was done daily so that surface oxidation of the flotation feed would be kept to a minimum. The ore was received in two shipments and Table 1 shows the head analyses, and the screen analyses of the crushed products. The analyses indicate that the two shipments were similar in composition with reference to the significant elements. The grinding and flotation circuit was operated about 5 1/2 hours per day at approximately 500 lb/hour.

In the first mill run the ore, crushed to all minus 4 mesh, was fed to a 20 in. dia x 30 in. rod mill in closed circuit with a 20 mesh Sweco screen. The oversize from the screen was returned to the rod mill while the undersize was fed to a ceramic P-50 Dorrclone in closed circuit with a 30 in. dia x 48 in. ball mill. The overflow from the cyclone was the feed to the flotation circuit. The rod mill feed rate was controlled by a Milltronics sonic control system designed to maintain a constant load on the mill. A Moore Slurry Density Control unit was used to maintain the Dorrclone feed at a constant pulp density of about 47 per cent solids.

TABLE 1

Head Assays and Screen Analyses on Crushed Composite Feed Samples for Run 1 and 2 and Uranium Distribution for Run 2

Mesh size	Run 1	Run 2			Head Assays (%)		
	Wt (%)	Wt (%)	U_3O_8 (%)	Dist (%)	Elements	Run 1	Run 2
+ 6	-	6.8	0.10	4.1	U_3O_8	0.17	0.18
+ 8	34.3	14.3	0.13	11.2	S	4.10	3.62
+ 10	16.3	13.3	0.12	9.7	Fe	4.42	5.1
+ 14	10.4	13.3	0.14	11.2	P	0.05	0.06
+ 20	-	9.6	0.13	7.5			
+ 28	14.0	9.2	0.14	7.8			
+ 35	5.2	6.7	0.17	6.9			
+ 48	4.6	6.4	0.18	6.9			
+ 65	3.4	4.4	0.22	5.8			
+ 100	3.1	4.2	0.27	6.8			
+ 150	2.1	2.3	0.37	5.1			
+ 200	1.5	2.0	0.36	4.3			
- 200	5.1	7.5	0.28	12.7			
	100.0	100.0	0.17	100.0			

The rod mill and Dorrclone were eliminated from the grinding circuit for the second mill run and a 50 mesh Sweco screen was added in place of the Dorrclone in the ball mill circuit. With this arrangement, the crushed ore was fed to the ball mill and the ball mill discharge was fed directly to a 20 mesh Sweco screen. The plus 20 mesh fraction from the screen was returned to the ball mill and the undersize was sent to the 50 mesh screen. The undersize from the 50 mesh screen was flotation feed while the oversize was returned to the ball mill. The Milltronics control unit was used to control the loading on the ball mill while the Moore Slurry Density Control was used to keep the feed to the 50 mesh Sweco screen at a constant pulp density of 50 per cent solids.

Flotation

The general procedure for the first series of tests was to remove the sulphides first, to separate the slimes from the sulphide tailings by a settling tank, and to make a rougher uranium concentrate which was cleaned and recleaned. A scavenger uranium concentrate was removed after the rougher float and this concentrate was returned along with the cleaner and recleaner tailings to the head of the circuit. The final tailing was passed over a Humphrey spiral to determine if any of the unfloated uranium was recoverable by gravity, and to provide some visual indication of the efficiency of the flotation step.

The overflow from the grinding circuit cyclone was fed to a 2 cu ft conditioner which gave a 5 minute retention time at 25 per cent solids. From the conditioner the pulp went to four No. 7 Denver cells having a total of 9 minutes retention time for sulphide flotation. The final concentrate was removed from the first cell, while the combined concentrate from the last three cells was returned to the first cell for cleaning. The sulphide tailings were pumped to a settling tank which removed the slimes in the overflow. The underflow from the settling tank was fed at about 40 per cent solids to a 6 cu ft conditioner with a retention time of about 20 minutes.

After conditioning, the pulp was diluted to 20 or 25 per cent solids and passed to fourteen No. 7 Denver cells. The first five cells were used for the uranium rougher float, and the last nine for the uranium scavenger float. The rougher concentrate went to ten No. 5 Denver cells for cleaning and recleaning. The scavenger concentrate and the cleaner tailings were returned to the first rougher cell.

On the basis of results obtained during the campaign, various changes were made in the circuit in order to effect improvements. Figure 1 is a diagram of the final flowsheet used for run No. 1, showing the reagent addition points. The recycle products (scavenger float and cleaner tailings) amounted to from 25 to 50 per cent of the feed weight depending on the amount of reagent used.

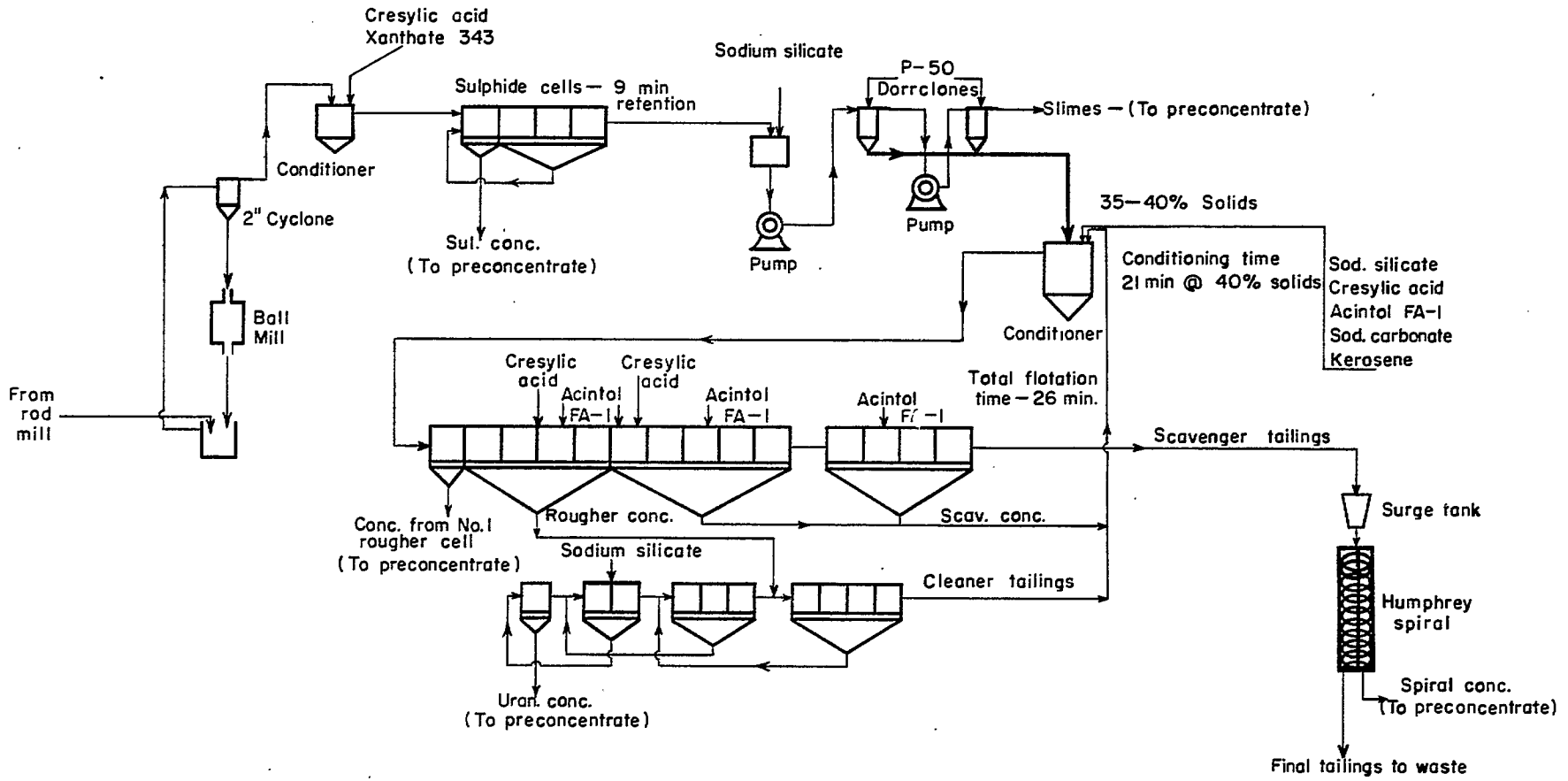

FIGURE 1 FLOWSHEET FOR FLOTATION PILOT PLANT TESTS 12 TO 19

The circuit used in run No. 2 was basically similar to that used in run No. 1 except that a Humphrey spiral was used in place of the sulphide float during the latter half of the run as shown in the flowsheet in Figure 2.

Reagents

For the sulphide float, the reagents were cresylic acid and Cyanamid's Xanthate 343. In the uranium minerals float, Acintol FA-1 was used as the collector for the uranium minerals, and was added to the conditioner and also stage-added to the rougher and scavenger cells. The Acintol was used as an emulsion consisting of 8 per cent Acintol FA-1 and 0.1 per cent sodium hydroxide in water. Other reagents added to the uranium conditioner ahead of the uranium flotation circuit were sodium silicate as a gangue depressant, cresylic acid and kerosene as frothers and sodium carbonate as a pH regulator. Cresylic acid was also stage-added to the rougher and scavenger cells as required. Sodium silicate was added to the cleaner circuit as a gangue depressant. The quantities of these reagents used in each test are given in Tables 2 and 6.

Sampling

A composite feed sample was made by taking one pound of sample every half hour from the belt of the dry ore feeder to the rod mill. The samples were used for head analyses and screen tests. Daily composite samples were made up from timed cuts (from 10 to 30 seconds, depending on the flow) of flotation feed, sulphide concentrate, slimes, rougher concentrate, scavenger concentrate, cleaner tailing, spiral concentrate, final uranium concentrate and final tailing. These samples were taken at half hour intervals commencing about two hours after the start-up for the day in order to allow stable operating conditions to be established before sampling was started, and were used for chemical analyses, metallurgical weight balances and screen analyses.

RESULTS

The mill run on the first shipment of ore was designated as Mill Run No. 1 and consisted of 18 days operation of the pilot plant. The results are given in Table 2.

At the start of the run the grinding circuit consisted of the rod mill, ball mill and settling tank for desliming. On the basis of the laboratory studies, the grind desired was about 65% minus 200 mesh. As the grind for the first two days(Tests 2 and 3) was too fine at 85% minus 200 mesh, 600 lb of steel balls were removed from the ball mill. The resultant grind was then too

TABLE 2

Operational Record - Denison Flotation Mill Run No. 1

Test No.	Flotation feed			Sulphide conc.			Slimes			U_3O_8 Conc.			Spiral conc. from tailing		
	-200 M (%)	Wt (lb/hr)	U_3O_8 (%)	Wt. (%)	U_3O_8 (%)	Dist. (%)	Wt (%)	U_3O_8 (%)	Dist. (%)	Wt (%)	U_3O_8 (%)	Dist. (%)	Wt (%)	U_3O_8 (%)	Dist. (%)
2	84.6	447	.12	3.8	.20	6.8	-	-	-	19.7	.30	53.0	-	-	-
3	84.5	462	.13	8.2	.26	16.5	23.2	.13	23.4	26.8	.23	47.8	-	-	-
4	62.9	467	.14	14.1	.27	24.8	1.4	.13	1.2	35.5	.24	55.5	-	-	-
5	45.6	497	.15	4.6	.16	6.2	5.0	.14	5.8	9.9	.44	36.2	-	-	-
6	50.9	467	.17	7.9	.24	13.6	6.4	.16	7.3	10.7	.57	43.7	-	-	-
7	65.8	470	.14	7.2	.26	15.1	7.4	.13	7.8	9.4	.62	47.1	-	-	-
8	54.0	500	.15	7.4	.21	11.5	12.8	.16	15.2	9.0	.67	44.6	-	-	-
9	68.3	500	.15	7.6	.22	11.6	4.6	.16	5.1	8.0	.50	27.8	-	-	-
10	61.9	484	.16	8.0	.17	10.2	12.0	.16	14.4	16.0	.38	45.6	-	-	-
11	64.4	489	.16	4.3	.12	4.5	13.5	.16	18.7	18.8	.36	58.7	-	-	-
12	62.2	491	.15	5.5	.15	5.8	13.0	.14	12.8	13.4	.40	37.6	-	-	-
										15.0	.26	27.5			
13	69.4	492	.16	5.6	.23	8.7	14.7	.16	15.8	9.0	.44	26.8	2.4	.09	1.5
										18.3	.33	40.5			
14	71.5	500	.16	5.8	.16	6.4	19.2	.17	22.4	6.0	.65	26.8	4.4	.04	1.2
										19.6	.29	39.1			
15	67.9	520	.16	5.0	.15	5.2	28.5	.16	31.4	9.4	.32	20.7	1.3	.11	1.0
										18.7	.28	36.1			
16	74.3	495	.16	3.2	.15	3.1	17.0	.18	19.6	11.3	.44	32.1	3.6	.15	3.5
										12.3	.39	30.9			
17	65.7	493	.16	4.5	.15	5.0	14.2	.18	19.0	17.5	.24	31.3	2.3	.10	1.7
										11.0	.40	32.5			
18	63.7	492	.16	4.3	.15	4.6	14.2	.16	16.1	6.0	.54	23.0	1.9	.37	5.0
										3.8	.97	26.2			
19	65.8	490	.15	4.9	.16	4.7	18.0	.15	16.4	5.5	1.43	47.9	2.6	.44	6.9

Note: Under "U_3O_8 concentrate" for Tests 12 to 19 two sets of data are given. The upper set is U_3O_8 recovery and grade in the concentrate from the first of the five rougher cells. The lower set is the U_3O_8 recovery and grade in the cleaner concentrate

Table 2 (Continued)

Test No.	Tailing			U_3O_8 Recovery	Ratio of Conc.	Uran. Float pH	Reagents (lb per ton ore)							
							Sul. Float		Deslime	Uran. Float				
	Wt (%)	U_3O_8 (%)	Dist (%)	(%)			Xanth. 343	Cres. Acid	Na_2SiO_3	Acintol	Cres. Acid	Na_2SiO_3	Kerosene	Na_2CO_3
2	72.0	0.057	36.8	65.2	3.6	8.1	.2	.04	.5	2.25	.2	.4	-	-
3	41.8	0.038	12.3	87.7	1.7	8.1	.2	.12	.5	2.2	.2	.5	-	-
4	49.0	0.058	18.5	81.5	2.0	8.4	.16	.10	.4	1.9	.08	.4	-	-
5	80.5	0.077	51.8	48.2	5.1	8.4	.16	.05	.4	2.0	.07	.7	-	-
6	75.0	0.066	35.4	64.6	4.0	9.0	.16	.05	.3	2.1	.07	.5	-	.6
7	76.0	0.049	30.0	70.0	4.2	9.1	.16	.05	.3	1.9	.07	.6	.08	.6
8	70.8	0.055	28.7	71.3	3.4	9.1	.16	.06	.4	2.2	.07	.6	.08	.6
9	79.8	0.10	55.5	44.5	4.9	9.2	.15	.05	.4	2.1	.09	.9	.08	.6
10	64.0	0.062	29.8	70.2	2.8	9.1	.15	.06	.4	2.6	.08	.6	.08	.6
11	63.4	0.033	18.1	81.9	2.7	8.8	.15	.06	.4	2.2	.10	.7	.08	.6
12	52.8	0.044	16.3	83.7	2.1	9.0	.15	.06	.4	1.9	.09	.6	.08	.6
13	50.0	0.020	6.7	93.3	2.0	9.1	.15	.06	.4	2.0	.12	.6	.08	.6
14	45.0	0.013	4.1	95.9	1.8	9.0	.15	.06	.4	2.1	.12	.6	.08	.6
15	37.1	0.022	5.6	94.4	1.6	9.1	.15	.06	.4	2.0	.12	.7	.08	.6
16	52.5	0.032	10.8	89.2	2.1	9.1	.15	.06	.4	2.2	.12	.7	.08	.6
17	50.5	0.028	10.5	89.5	2.0	9.1	.15	.06	.4	2.0	.12	.7	.08	.6
18	64.8	0.043	19.8	80.2	2.8	9.1	.15	.06	.4	1.7	.12	.72	.08	.6
19	63.3	0.049	18.9	81.1	2.7	9.1	.15	.06	.3	1.8	.12	.72	.08	.6

coarse at 45 to 50% minus 200 mesh (Tests 5 and 6) and additional balls were gradually added until in Test 9 the grind was satisfactory at about 65% minus 200 mesh. Due to adjustments to the grind and other operating difficulties Tests 2 to 10 did not give significant results except to indicate that excessively coarse or fine grinds gave poor flotation results.

The results of Test 11 were close to those obtained in the laboratory work. In this test the preconcentrate contained 81.9% of the uranium and the weight of the preconcentrate represented a ratio of concentration of 2.7 to 1. The amounts of flotation reagents used were similar also to those used in the laborarory scale testing.

Since it was observed in Tests 2 to 11 that the froth from the first rougher cell carried dark coloured minerals, this concentrate was removed separately as a finished product in Test 12, and only the concentrate from the last four rougher cells was cleaned. This procedure was followed for the remainder of Run 1. Comparing the results of Test 12 with those of Test 11 shows the effect of this flow sheet change was to lower the ratio of concentration appreciably since a higher weight reported in the uranium concentrate without a corresponding increase in extraction.

Since the desliming tank used up to Test 12 tended to surge, it was replaced by a P-50 Dorrclone for Test 13 in an attempt to provide a more even and efficient desliming operation. The uranium recovery in this test was 93%, 10% higher than that obtained in Test 12 where the desliming tank was used. These results suggest that the desliming efficiency is of considerable significance in determining the flotation results.

Test 14 and 15 were done using the same flow sheet and conditions as were used for Test 13. Although the uranium recoveries in these two tests were similar to that obtained in Test 13, the ratio of concentration was considerably lower. This was due to the high weights of slimes removed in these two tests, which, in turn, was because the P-50 cyclone was difficult to control. Consequently, for Test 16, a second P-50 cyclone was installed to retreat the overflow from the first cyclone. This change resulted in a reduction in the weight of slimes removed but at the same time the uranium recovery fell to about 89%; a drop of 4% as compared to Test 13 where only one cyclone was used for desliming. It can be seen from Table 2 that this drop is due to a lower recovery in the uranium float. A possible explanation for the lower recovery in the uranium concentrate is the fact that in Tests 16 and 17, the amount of sodium silicate added to the uranium float was 0.1 lb/ton more than was used in Test 13.

In Tests 18 and 19 the scavenger concentrates and cleaner tailings, instead of going directly to the conditioner as shown in Figure 1, were sent to the desliming cyclones for dewatering, in an effort to produce a higher pulp density in the conditioner. This resulted in a ratio of concentration of 2.7,

but the recovery dropped to 81%. However, at least part of this drop in recovery may reflect the reduction in the amount of Acintol FA-1 used in these tests as compared with that used in Tests 16 and 17.

The results of some screen analyses done on various plant products are given in Tables 3, 4 and 5. From Table 3, in which the size distribution is given for flotation feeds ground to 65%, 70% and 85% minus 200 mesh (Tests 17, 13 and 2 respectively), it can be seen that the uranium concentrates in the minus 200 mesh fractions. Table 3 also shows that the amount of minus 10 micron material produced is similar for all three grinds, although there is a spread of 20% in the amount of minus 200 mesh material present in these ground products. This would suggest that it would be difficult to control the amount of minus 10 micron material produced in the grinding of the Elliot Lake ores.

TABLE 3

Screen Analyses and Uranium Distribution of Several Flotation Feeds from Run No. 1

Test No.	13			2	17
Mesh size	Wt (%)	U_3O_8 (%)	Dist. (%)	Wt (%)	Wt (%)
+ 48	0.8	0.054	0.3	-	1.0
+ 65	2.1	0.061	0.8	-	2.6
+100	5.6	0.065	2.3	-	7.2
+150	14.1	0.081	7.4	5.9	11.6
+200	8.7	0.11	6.2	9.5	13.3
+ 56μ	6.2	0.33	13.3	8.2	5.9
+ 40μ	14.3	0.15	13.9	20.0	15.9
+ 28μ	11.4	0.16	11.8	18.8	13.2
+ 20μ	9.7	0.16	10.0	11.9	7.9
+ 14μ	6.5	0.17	7.2	5.9	3.5
+ 10μ	4.2	0.17	4.6	3.8	2.8
- 10μ	16.4	0.21	22.2	16.0	15.1
	100.0	0.16	100.0	100.0	100.0

TABLE 4

Screen Analyses and Uranium Distribution of Final Tailings from Test No. 13

Mesh size	Wt (%)	U_3O_8 (%)	Dist. (%)
+ 48	3.8	0.029	5.3
+ 65	5.1	0.023	5.8
+100	12.7	0.017	10.6
+150	15.5	0.015	11.1
+200	20.7	0.016	15.8
+ 56μ	7.1	0.026	8.7
+ 40μ	17.5	0.015	12.5
+ 28μ	9.8	0.016	7.7
+ 20μ	3.8	0.023	4.3
+ 14μ	1.7		
+ 10μ	0.9	0.067	8.1
- 10μ	1.4	0.15	10.1
	100.0	0.021	100.0

Table 4 shows the screen analysis of the final tailing from Test 13 along with the uranium distribution in the various size fractions. It can be shown from the data given in Tables 2, 3 and 4 that the uranium recovery from the plus 200 mesh fraction is about 80%, while that from the minus 200 mesh fraction is about 95%. It can be concluded from these results that as the grind becomes coarser the recovery of uranium by flotation will decrease.

Table 5 gives the results of sizing analyses, made using the Haultain Infrasizer, of slime products from Test 13, in which one P-50 Dorrclone was used for desliming, and Tests 16 and 17 in which two P-50 Dorrclones were used. The data show that whether one or two Dorrclones were used, the minus 20 micron fraction comprised 90% or more of the slime fraction. However, as already mentioned, the use of two Dorrclones in series resulted in more even operation, and as shown in Table 5, reduced slightly the amount of plus 20 micron material in the slime product.

Using the data with reference to Test 13 in Tables 2, 3, 4 and 5 it can be calculated that the minus 20 micron material in the flotation feed is distributed 7% to the final tailings and 50% to the slime fraction. It follows that the remaining 43% reported to the flotation concentrates; but mainly to the

TABLE 6

Operational Record - Denison Flotation Mill Run No. 2

Test No.	Flotation feed			Sulphide conc.			Slimes			U_3O_8 Conc.			Spiral conc. from tailing		
	-200M (%)	Wt (lb/hr)	U_3O_8 (%)	Wt. (%)	U_3O_8 (%)	Dist. (%)	Wt (%)	U_3O_8 (%)	Dist. (%)	Wt (%)	U_3O_8 (%)	Dist. (%)	Wt (%)	U_3O_8 (%)	Dist. (%)
20	45.9	508	.16	7.0	.22	10.5	16.5	.15	16.8	3.7	1.62	40.7	3.8	.51	13.2
21	44.9	488	.15	7.2	.19	9.4	17.4	.13	15.5	1.9	1.33	17.3	5.5	.79	29.8
22	45.5	577	.15	7.1	.19	9.1	17.0	.11	12.7	8.8	.83	49.4	5.9	.44	17.6
23	54.6	502	.14	7.9	.24	14.2	13.7	.12	12.2	5.5	.80	32.9	4.4	.49	16.1
24	56.1	510	.15	7.3	.26	14.5	11.9	.13	11.8	6.3	.62	29.8	2.4	.87	15.9
25	54.8	513	.16	7.6	.26	13.9	11.1	.13	10.1	20.3	.40	56.9	2.5	.41	7.2
				Spiral conc.											
26	56.3	505	.17	9.7	.44	37.3	8.8	.12	9.3	15.9	.24	33.4	5.1	.13	5.8
27	52.5	531	.14	10.9	.62	48.7	12.4	.11	10.0	13.8	.26	26.3	3.6	.09	2.3
28	54.3	500	.16	9.2	.65	44.3	11.6	.13	11.2	17.4	.25	32.3	3.2	.06	1.3
29	56.8	520	.16	8.8	.89	48.2	14.6	.13	11.6	17.5	.28	29.9	0.6	.09	0.3
30	56.1	582	.15	7.7	.82	45.4	8.9	.13	8.3	19.8	.23	32.7	0.9	.15	1.0
31	57.1	500	.15	8.8	.89	49.3	12.8	.14	11.3	12.2	.33	25.4	0.8	.18	0.9
32	58.7	503	.16	9.3	.84	52.2	7.6	.13	6.6	12.5	.31	26.0	1.0	.12	0.8

Table 6 (Continued)

Test No.	Tailing			U_3O_8 Recovery (%)	Ratio of Conc.	Uran. Float pH	Reagents (lb per ton ore)								Remarks
							Sul. Float		Deslime	Uran. Float					
	Wt (%)	U_3O_8 (%)	Dist. (%)				Xanth. 343	Cres. Acid	Na_2SiO_3	Acintol	Cres. Acid	Na_2SiO_3	Kerosene	Na_2CO_3	
20	69.0	.04	18.8	81.2	3.2	9.3	.16	.06	.1	2.2	.15	.6	.08	.6	
21	68.0	.06	28.0	72	3.1	9	.16	.06	.1	2.3	.13	.75	.08	.6	
22	61.2	.027	11.2	88.8	2.6	9	.16	.06	-	2.3	.21	.45	.07	.6	
23	68.5	.048	24.6	75.4	3.2	9	.16	.06	.1	2.0	.14	.75	.05	.6	
24	72.1	.051	28.0	72.0	3.6	9.1	.16	.06	-	2.1	.18	.62	.02	.6	
25	58.5	.029	11.9	88.1	2.4	8.9	.16	.06	-	2.1	.14	.57	.08	.6	Sodium in meta
26	60.5	.027	14.2	85.8	2.5	8.9	-	-	-	2.0	.11	.45	.11	.6	silicate used in
27	59.5	.029	12.7	87.3	2.5	8.9	-	-	-	1.9	.13	.47	.09	.6	place of sodium
28	58.6	.025	10.9	89.1	2.4	8.8	-	-	-	2.2	.10	.60	.08	.6	silicate
29	58.5	.028	10.0	90.0	2.4	8.8	-	-	-	2.1	.10	.55	.09	.6	
30	62.7	.028	12.6	87.4	2.7	8.7	-	-	-	2.2	.11	.55	.09	.6	Gelatin used in
31	65.4	.032	13.1	86.9	2.9	8.9	-	-	-	1.9	.09	.67	.09	.6	uranium flotation
32	69.6	.031	14.4	85.6	3.3	8.9	-	-	-	2.0	.08	.67	.09	.6	in amount of 0.05 lb/ton

uranium concentrate since the sulphide concentrate contains a total of only 5% of the final weight. This indicates that the flotation procedure used floats nearly all of any minus 20 micron material present in the flotation cells with no selectivity for the uranium-bearing particles.

TABLE 5

Infrasizer Analyses and Uranium Distribution of Slimes from Tests 13, 16 and 17

Size (microns)	16			13	17
	2 P-50-Dorrclone			1 P-50-Dorrclone	2 P-50-Dorrclones
	Wt (%)	U_3O_8 (%)	Dist. (%)	Wt (%)	Wt (%)
+ 56	2.1			0.1	0.1
+ 40	0.6	0.24	4.0	0.8	0.3
+ 28	0.4			3.0	1.4
+ 20	3.7	0.07	1.3	6.7	3.2
+ 14	20.6	0.14	15.4	19.8	22.8
+ 10	18.6	0.19	18.8	25.6	31.1
-10	54.0	0.21	60.5	44.0	41.1
	100.0	0.19	100.0	100.0	100.0

By the end of Test 19, the supply of the first shipment of ore had been exhausted. The mill run on the second shipment of ore was designated as Mill Run No. 2 and consisted of 13 days operation of the pilot plant. The results are given in Table 6.

Since it was considered that the amount of 20 micron material produced in grinding in Run No. 1 was excessive, the rod mill was eliminated from the grinding circuit for Run No. 2 in an attempt to minimize over-grinding. At the same time, a 50 mesh screen was used for classifying the ball mill discharge instead of the P-50 Dorrclone used in Run No. 1. To show the type of grind produced by this grinding circuit, the size analysis of the ground product for Test 22 is given in Table 7. Comparing these results with those given for Test 13, Table 3, it can be seen that the modifications made in the grinding circuit for Run No. 2 resulted in the weight of the minus 200 mesh fraction being reduced from 69% minus 200 mesh to 45% minus 200 mesh. At the same time the minus 20 micron fraction was reduced from 27.1% to 15.5%, and the minus 10 micron fraction was reduced from 16.4% to 11.3%. This grind was used in Tests 20 to 22, the first three tests in Run No. 2. For Tests 23 to 25, the grind was coarsened to 55% minus 200 mesh by lowering the pulp density in the ball mill.

TABLE 7

Screen Analyses and Uranium Distribution for Test 22 Flotation Feed

Mesh	Wt (%)	U_3O_8 (%)	Dist (%)
+ 48	0.3		
+ 65	8.8	0.074	4.8
+100	21.0	0.10	15.0
+150	13.5	0.14	13.5
+200	13.3	0.15	14.2
+ 56μ	5.8	0.44	18.1
+ 40	11.1	0.13	10.3
+ 28	6.3	0.11	4.9
+ 20	4.4	0.12	3.8
+ 14	2.8		
+ 10	1.4	0.14	4.2
- 10μ	11.3	0.14	11.2
	100.0	0.14	100.0

The recovery of the uranium in the flotation concentrate in Tests 20 to 25, in which the coarser grinds were used, was unsatisfactory, the highest being 80% in Test 25. This confirmed the results of Run No. 1 which indicated that coarse grinding was detrimental to uranium flotation.

The results of Tests 20 to 25 given in Table 6 show that much of the uranium which did not report in the uranium concentrate was recovered in the gravity concentrate from the Humphrey spiral used on the final flotation tailing. This suggested that, with the coarser grind, a combination of gravity concentration and uranium flotation might be advantageous. Consequently for Test 26, the sulphide flotation section was replaced by a Humphrey spiral as shown in Figure 2.

The results of Tests 27, 28 and 29 given in Table 6 show the effect of this flow sheet change, and indicate that gravity concentration in place of sulphide flotation is beneficial. In these three tests the uranium recovery averaged 89.5% with a ratio of concentration of about 2.4:1. The gravity concentrate contained 45 to 50% of the uranium in about 10% of the feed weight. The balance of the uranium recovered was distributed about 10% to the slime fraction and about 30% in the flotation concentrate in about 12% and 17% of the feed weight respectively. The net effect of using gravity concentration in place of sulphide flotation was to increase the ratio of concentration by about 20% without lowering the recovery appreciably from that obtained by the all flotation procedure.

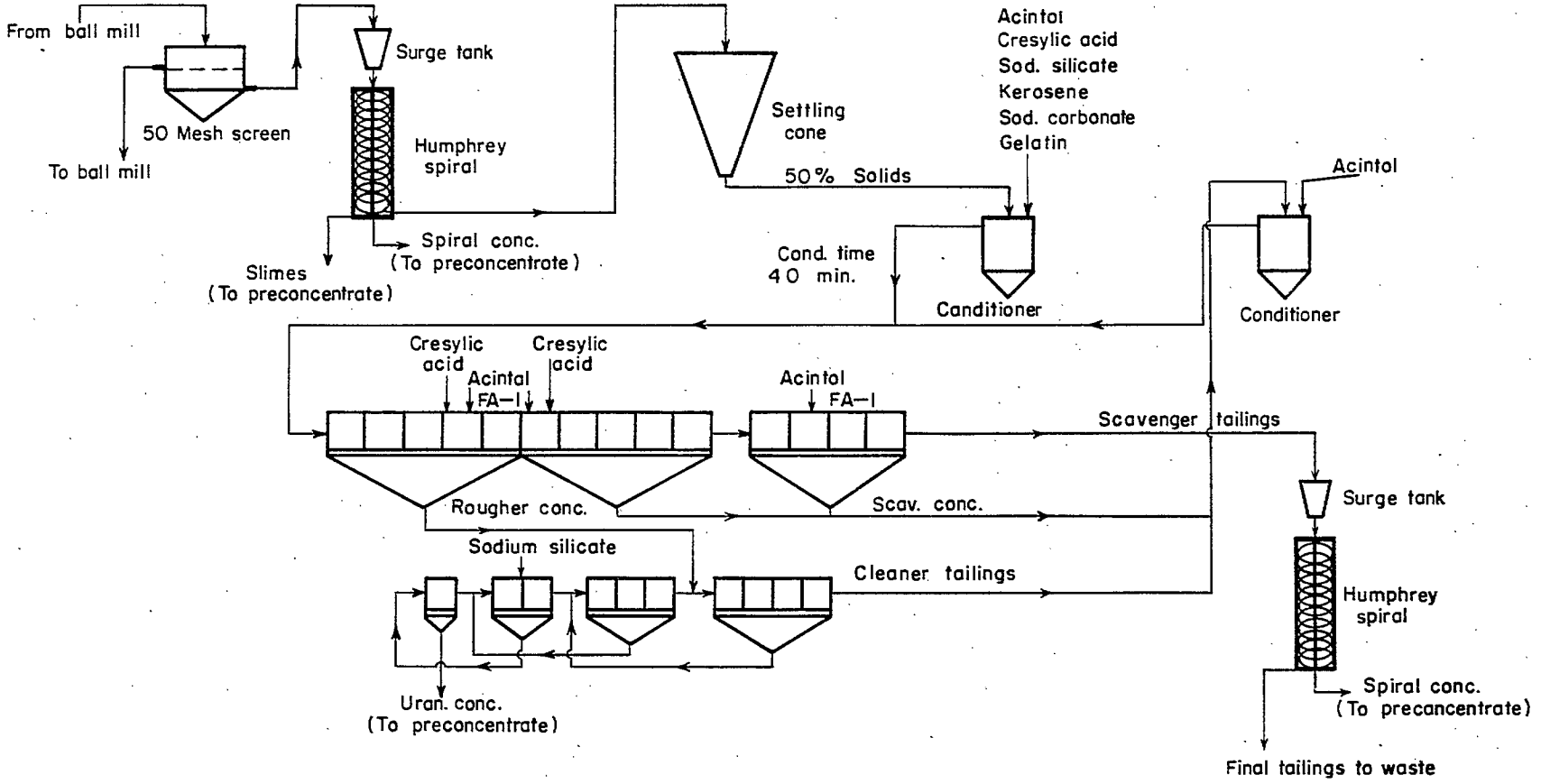

FIGURE 2 FLOWSHEET FOR FLOTATION PILOT PLANT TESTS 26 TO 30

In Test 30, 0.05 lb gelatin/ton of feed was added to the uranium flotation circuit as a gangue depressant. Comparing the results of this test with those of Tests 28 and 29 (Table 6), shows that the use of gelatin did not increase the U_3O_8 grade of the uranium concentrate. The drop in recovery and increase in the ratio of concentration obtained in this test was in fact due to a drop in the weight of slimes removed.

For Tests 31 and 32, the tailing from the rougher cells was reground before being fed to the scavenger cells. The use of gelatin was continued in these tests. The results of these two tests (Table 6) suggest that with this circuit the operation could be adjusted to produce a uranium recovery of close to 90% at a ratio of concentration of 3.0:1. This conclusion is based on the assumption that, with time, the plant could have been operated to produce the gravity concentrate made in Test 32 along with the slime recovery made in Test 31. It was not possible to operate the plant to actually obtain these results as the bulk ore sample was completely used up by the end of Test 32.

The tailing from Test 29 was sized and the uranium distribution in the size fraction determined. This data, given in Table 8 can be compared with that of Test 13 (Table 4), in which a finer grind and an all flotation circuit was used. A comparison of these two sets of data shows that gravity concentration in place of sulphide flotation resulted in a lowering of the uranium analyses of the coarser fractions in the tailings. On the other hand the finer sizes in the tailing from Test 29 contained more uranium than the corresponding sizes in the Test 13 tailing. Also there was considerably more minus 20 micron material in the tailing from Test 29. The higher uranium analyses of the fines and the greater proportion of slimes in the Test 29 tailing might have been due to the desliming operation effected by the Humprey spiral (Figure 2) being less efficient than the cyclones used in Test 13.

In Test 25 to 28, sodium meta silicate was used in place of sodium silicate in uranium flotation. A comparison of the results of Test 28 with those of Test 29 (Figure 6) suggest that these reagents are equally effective.

TABLE 8

Screen Analyses with Uranium Distribution of Final Tailings
Test No.29

Mesh Size	Wt (%)	U_3O_8 (%)	Dist. (%)
+ 48	0.1		
+ 65	9.7	0.019	7.8
+100	25.0	0.019	19.6
+150	17.5	0.020	14.3
+200	15.0	0.017	10.6
+ 56μ	5.9	0.023	5.7
+ 40	10.0	0.015	6.1
+ 28	5.4	0.021	4.5
+ 20	2.5	0.037	3.7
+ 14	2.9	0.048	5.7
+ 10	2.9	0.059	6.9
- 10	3.1	0.12	15.1
	100.0	0.025	100.0

DISCUSSION

The tests in the two runs can be classified into five series as follows:

(a) the series from Test 2 to Test 10 was considered a break-in period and consequently little significance was placed on the results;

(b) the series from Test 11 to Test 19 was carried out with a fine grind of 65-75% minus 200 mesh using the flowsheet shown in Figure 1, which involved sulphide flotation followed by desliming and uranium flotation;

(c) the series from Test 20 to Test 25 was carried out with a coarser grind of 45 to 55% minus 200 mesh, using the flowsheet shown in Figure 1;

(d) the series from Test 26 to 32 was carried out using the same grind as for (c) and employing the flowsheet shown in Figure 2, which involved gravity concentration of the sulphides and coarse uranium particles using a Humphrey spiral, followed by uranium flotation.

(e) Tests 31 and 32 in which the circuit used was similar to that used in (d) but with regrinding of the rougher tailing ahead of scavenger uranium flotation.

The results of series (b) (c) (d) and (e) are shown graphically in Figure 3. To obtain the data for the graphs on Figure 3, the uranium and the weight recovered in the gravity concentrate from the Humphrey spiral operating on the final flotation tailing was calculated as part of the tailing loss. This was done so that the data reflect only the results obtained in the main pre-concentration operations. Figure 3 shows that if the ratios of concentration are plotted against recoveries, the points obtained for each of the four series follow straight line trends of similar slope in which the ratio of concentration decreases as the uranium recovery in the preconcentrate increases.

Figure 3 shows that the poorest results obtained were those from series (b) and (c) in which flotation was used throughout (Graphs 1 and 2). Since the only major difference in procedure between series (b) and (c) was the fineness of grind it appears that Graph 2 is simply an extension of Graph 1 and consequently the results of these two series can be considered together. A study of Graphs 1 and 2 shows that generally, with the all flotation circuit, the uranium recovery is highest and the ratios of concentration lowest with the finer grinds. Considering only tests in which reagent additions were similar, it can be seen that Tests 13, 14, 16 and 17, all done on grinds between 65% and 75% minus 200 mesh, had recoveries ranging from 87% to 94% with ratios of concentration ranging from 2.3 to 2.0; while Tests 20, 21, 23 and 24 done on grinds of from 45 to 55% minus 200 mesh had recoveries ranging from 43% to 69% with ratios of concentration of from 3.9 to 3.6.

A further study of Graphs 1 and 2 along with the data on Tables 2 and 6 indicate other operating variables that will affect the uranium recovery and ratio of concentration in the all-flotation circuit. For example, Tests 18 and 19 on Graph 1 have low recoveries and high ratios of concentration because the weight and uranium recovered in the uranium flotation concentrates are lower than in the other tests of the series, and this in turn is probably due to the lower amount of Acintol FA-1 used in these two tests. In Test 15, the ratio of concentration is lower than might be expected because of the high weight of slimes recovered in this test. In Test 12, a low uranium recovery in the slime fraction caused the results of this test to be off the trend line, while in Test 11 the weight recovery in the uranium flotation concentrates was lower than in most of the other tests in series (b), (Graph 1). In Graph 2, the reasons for Tests 22 and 25 resulting in higher

recoveries than were obtained in Tests 20, 21, 23 and 24 are not clear although it is possible that the low amount of sodium silicate used in Test 22 and the meta sodium silicate used in Test 25 were contributing factors.

In summary, the results obtained from series (b) and (c), (Graphs 1 and 2, Figure 3) show that with a circuit involving flotation of the sulphides, followed by desliming and flotation of the uranium minerals,a uranium recovery of about 90 to 92% is possible with a ratio of concentration of 2.1 to 2.0:1. To obtain these results, the ore would have to be ground to at least 65% minus 200 mesh.

Graph 3, Figure 3 shows the results of Tests 26 to 30 in which sulphide flotation was replaced by gravity concentration using a Humphrey spiral. In these tests, the grind used was about 55% minus 200 mesh. The gravity tailings were deslimed and the deslimed fraction treated by flotation for recovery of the uranium minerals. As shown by Graph 3, this flow sheet produced a higher recovery for a given ratio of concentration than was obtained using all flotation. Of these tests, only the results of Test 31 did not fall reasonably close to the trend line, and it can be seen from Table 6 that in Test 31, the weight and uranium recovered in the slime fraction was lower than in the other tests in this series. It can be seen from Graph 3, that using gravity methods followed by uranium flotation with a grind of 55% minus 200 mesh, a uranium recovery of 90% can be expected with a ratio of concentration of 2.5:1.

Graph 4, Figure 3 represents the results of Tests 31 and 32. These tests were done using the same circuit as was used for Tests 26 to 30 except that in these two tests the rougher flotation tailing was reground ahead of scavenger flotation. Although there are only two points for this graph, Graph 4 was drawn on the assumption that it is parallel to the other graphs and that it would be possible to operate the plant to produce the slime recovery of Test 31 along with the gravity concentrate and flotation concentrate of Test 32. On this basis it can be seen from Graph 4, that this flow sheet was the best of the three tested and could be expected to produce a uranium recovery of 90% at a ratio of concentration of 2.9:1.

Regarding the effect of reagents in the uranium flotation step, it was observed that an increase in the Acintol and cresylic acid effects an increase in recovery and weight floated. The effect of sodium silicate on the other hand is to reduce the recovery and weight floated. The variations in the amounts of these reagents used in these tests was actually small. The amount of Acintol ranged from 1.7 to 2.3 lb per ton, cresylic acid ranged from 0.08 to 0.21 lb per ton and sodium silicate from 0.45 to 0.75 lb per ton. These variations, although small, produced significant changes in the ratio of concentration and recovery.

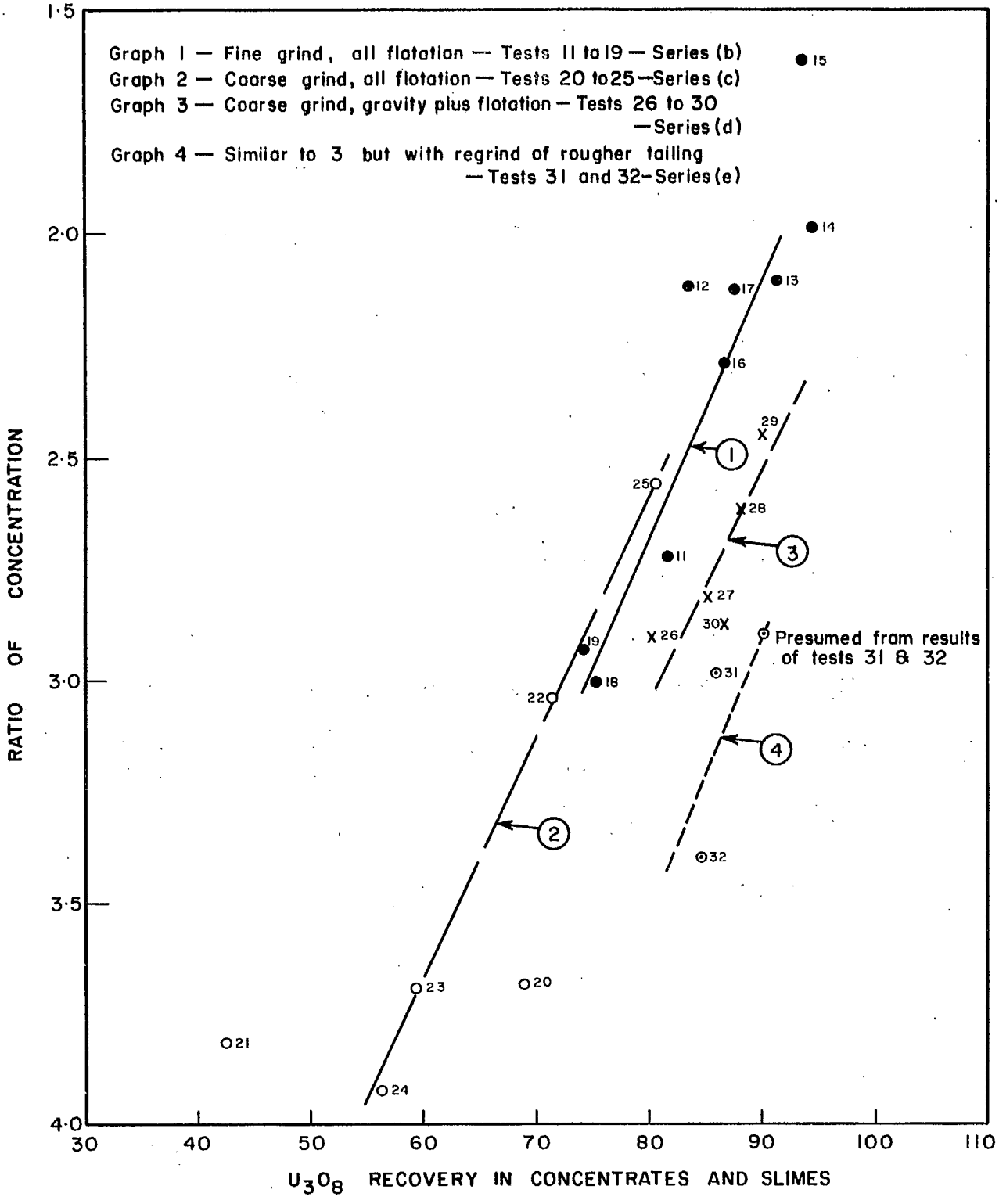

FIGURE 3 RESULTS OF TESTS 1 TO 32

The reagents used during the pilot plant operation and their approximate cost is given in Table 9.

TABLE 9

Reagent Consumption and Cost

Reagent	Cost/lb (cents)	All Flotation		Gravity+Uranium Flotation	
		Cspt. (lb/ton)	Cost (cents)	Cspt. (lb/ton)	Cost (cents)
Xanthate 343	19	0.15	3.04	-	-
Cresylic Acid	15	0.14-0.27	2.10- 4.05	0.08-0.13	1.2 - 1.95
Acintol FA-1	9	1.7 -2.3	15.30-20.70	1.9 -2.2	17.1 -19.8
Na_2SiO_3	1.5	0.45-1.12	0.67- 1.68	0.55-0.67	0.82- 1.0
Na_2CO_3	2	0.6	1.20	0.6	1.20
Kerosene	4	0.08	0.32	0.09	0.36
			22.63-30.99		20.68-24.31

In the final analysis the value of the preconcentration procedures tested in this work will depend on the relative costs of preconcentration followed by leaching, and of leaching the whole ore; considered along with the overall recoveries obtained by these two approaches. Preliminary leach tests of the preconcentrate produced by the gravity-flotation circuit has indicated that about 94% of the uranium in the preconcentrate could be recovered from the preconcentrate using current plant leaching procedures. With the 90% uranium recovery in preconcentration the overall recovery in preconcentration and leaching would be 84.6% from an ore containing 3.4 lb U_3O_8/ton. These tests also showed that the cost per ton of leaching the preconcentrate would be similar to that involved in the present leach plants. With this data, the following evaluation of preconcentration and leaching can be made.

Leaching Only

Mining cost/ton	$4.50
Milling cost/ton	3.50
Total	$8.00
Value of uranium recovered at $5.00/lb = 3.4 lb U_3O_8 x .94 x 5.00	$16.00
Net per ton or ore mined	$8.00

Preconcentration and Leaching

Mining cost/ton	$4.50
Crushing and grinding cost/ton	0.70
Flotation cost/ton (reagent cost x 3)	0.70
Hydrometallurgy cost/ton (0.34 tons at $2.80)*	0.95
Total	$6.85
Value of uranium recovered at $5.00/lb = 3.4 lb U_3O_8 x .846 x 5.00	$14.35
Net per ton of ore mined	$7.50

* Milling costs experienced in present leaching plants less grinding cost.

It can be seen from these figures that the loss of uranium to the tailing outweighs the advantages gained by preconcentration if the present practice is used to leach the preconcentrate. However if the uranium in the tailing can be recovered cheaply by bacterial leaching, and if more efficient hydrometallurgical techniques can be developed for treating the preconcentrate, the preconcentration techniques described in this report would be of considerable interest. This pilot plant study has provided products on which these leaching investigations can be conducted.

CONCLUSIONS

This pilot plant work has shown that, on ore similar to that of Denison Mines Limited, it is possible by using flotation and gravity methods to recover 90% of the uranium at ratios of concentration ranging from 2.1 to 2.9:1. If the ore is ground to 65% minus 200 mesh followed by sulphide flotation, desliming of the sulphide tailing and uranium flotation of the deslimed material, the lower ratio of concentration is obtained. If the ore is ground to 55% minus 200 mesh followed by a gravity concentration step, desliming of the gravity tailing and uranium flotation of the deslimed material, the higher ratio of concentration is obtained.

Since the success of a preconcentration step depends on obtaining a high ratio of concentration the gravity-flotation technique is superior to the all-flotation procedure. In addition, the gravity-flotation circuit would result in lower reagent costs and lower grinding cost than would be involved in the all flotation circuit. The reagent cost for the gravity-flotation circuit would be less than $0.25/ton of mill feed.

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