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FLOTATION OF URANIUM FROM ELLIOT LAKE ORES

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ABSTRACT

This paper describes flotation test work conducted over the past several years at the Mines Branch, Ottawa, on the radioactive quartz-pebble conglomerate ores of the Elliot Lake area of Ontario. The purpose of this work has been to produce a uranium flotation concentrate for subsequent leaching operations. The results of earlier investigations are summarized and correlated with the results of mineralogical studies of the ores and flotation products. The results of recent pilot-plant investigations by flotation, as well as by gravity concentration followed by flotation of the gravity tailings, are given. The principal collector used for uranium flotation was a tall oil fatty acid, and the indicated total cost of all reagents ranged from 21 to 31 cents per ton of ore treated.

Introduction

I N Canada, uranium has been extracted from its ores by either an acid or a basic leach process. However, from the commencement of work on the treatment of uranium ores it was realized that ore preconcentration offered possibilities of savings in the capital cost of the plant and in processing costs. Various preconcentration techniques, such as sink-float, gravity and flotation, were tested. Flotation appeared to offer the best possibilities having regard to recovery and ratio of concentration.

In the intervening years, the Mines Branch staff and others [1][2] have done a considerable amount of laboratory and pilot-plant flotation work on Elliot Lake ores. This paper is a summary and review of the results of the Mines Branch work.

For the Elliot Lake ore, a study of the mineralogy of the deposits and the flotation results shows that a ratio of concentration of from 2.5 to 3.5 to 1 is possible with satisfactory recovery, whereas the production of a high-grade concentrate of, say, 10 or 15 per cent U_3O_8 is probably not feasible.

Mineralogy of Elliot Lake Uranium Ores

The Matinenda formation, the lowermost unit of the Huronian sediments of the Elliot Lake district of Ontario, is composed of coarse-grained clastic rocks including grit, quartzite, arkose and quartz-pebble conglomerate. Economic concentrations of uranium minerals occur in the quartz-pebble conglomerate at or near the base of the formation.

The mineralogical characteristics of the ore conglomerates of the district are remarkably uniform. The ratio of pebbles to matrix has been estimated to be 65:35, and the composition of the conglomerates of the Quirke Lake Trough has been estimated to be as follows:

Quartz-pebbles	40 — 65%
Sulphides	2 — 8%
Sericite	5 — 20%
Matrix quartz	15 — 25%

Variations in the proportions of the radioactive minerals are shown by the wide variation in the ratios of thorium to uranium: Th/U varies from bed to bed, ranging from 0.1:1 to 5:1. However, the uranium content of the orebodies is quite constant and averages between 2.0 and 2.5 lbs. U_3O_8 per ton. In typical ore conglomerate, the quartz pebbles are closely sized and range from 0.5 to 1.5 or 2 inches in diameter. They are usually moderately rounded to sub-angular and are loosely to tightly packed. The pebbles consist mainly of silica; white and grey quartz pebbles are predominant, but pebbles of pink quartz, chert and jasper also occur, as well as occasional pebbles of feldspar and granite. The matrix consists of poorly sorted granules and silt-sized particles of quartz, sericite, feldspar and some chlorite. In addition, pyrite, radioactive minerals and other minerals are present here. Except for rare occurrences in fine pyritized fractures in the pebbles, most of the radioactivity is present in the matrix, and the radioactive minerals usually occur in the more richly pyritized areas of the matrix.

Brannerite, uraninite and monazite are responsible for most of the radioactivity in the conglomerate. Brannerite, a metamict uranium titanate, is believed to be the most abundant ore mineral in the conglomerates, although either uraninite or monazite may predominate locally. The brannerite occurs as discrete, irregular aggregates of fine prismatic crystals that are often intergrown with quartz as well as with anatase and/or rutile. The average size of the aggregate is approximately 65 to 100 mesh and the maximum size 20 to 28 mesh. Some of the brannerite also occurs as fine stringers and disseminations in the quartz-sericite matrix.

Uraninite is present as subhedral grains, brecciated grains and remnant crystals that range in size from 48 to 325 mesh, with an average size of about 150 mesh. Some intergrowths of uraninite and brannerite are present.

Monazite, a rare-earth phosphate containing thorium and a minor amount of uranium, occurs as rounded grains with an average size of approximately 65 mesh and typically contains very fine inclusions of pyrite. The monazite is believed to account for most of the thorium and rare-earth contents of the radioactive conglomerate ores.

Several other radioactive minerals have been noted in the conglomerate ores, including zircon, hydrocarbon ("thucholite"), coffinite, pitchblende and uranothorite, but these minerals contribute only a very small proportion of the total radioactivity of the ores.

Pyrite is the main sulphide mineral in the matrix, pyrrhotite is less abundant and other sulphides are rare. The pyrite is present as euhedral to subhedral grains and in aggregates; the individual masses are usually between 14 and 100 mesh in size. The concentration of pyrite in the matrix varies considerably from place to place. Other sulphides that have been identified in the conglomerates, but which are present in minor quantities, include pyrrhotite, chalcopyrite, galena, marcasite, sphalerite and molybdenite.

A trace amount of gold is present; in one ore, fine inclusions of gold were observed microscopically in pyrite.

Anatase and rutile commonly are present in close association with brannerite. Anatase occurs as a fine-grained alteration product of brannerite.

The mineralogy of Elliot Lake ore shows that the uranium minerals, uraninite and brannerite, are associated with pyrite and pyrrhotite and that the fine-grained brannerite is often intergrown with rutile and anatase. The most suitable collectors for the uranium minerals proved to be fatty acid compounds. It is known from the literature that these compounds are collectors for many of the associated minerals, such as sericite, rutile, anatase, zircon and chlorite, and will also readily float the pyrite and pyrrhotite. Thus, in the flotation of the uranium minerals, these associated minerals also float to contaminate the uranium concentrate. Also, as all the minerals floated by this collector occur in the matrix, which amounts to about 35 per cent of the ore, it is apparent that a ratio of concentration of better than 2.5 to 3.0 to 1 will be difficult to obtain with satisfactory recovery.

Figure 1 is a photomicrograph of a polished section of ore showing the disseminated intergrowth of brannerite-anatase in siliceous gangue adjacent to grains of pyrite. *Figure 2* shows streaks of brannerite in siliceous gangue. This intergrowth and fine dissemination of the uranium minerals makes it difficult to obtain a high ratio of concentration because it is necessary to float some of the gangue minerals to obtain a satisfactory recovery.

Early Test Work

In all flotation test work, there are a number of variables that must be investigated for each individual ore. Some of the main variables are the effects of grind, desliming, water, the pH of the pulp and, of course, the kind and amount of the flotation reagents used. Before the Elliot Lake leach plants were designed, a considerable amount of flotation test work was carried out at the Mines Branch on some of the Elliot Lake ores to investigate the effects of these variables. As a starting point in the investigation, a variety of collector reagents were tried in various combinations. Some of these were Turkey red oil, sulphonated castor oil, sulphonated fish oil, sulphonated oleic acid, sulphonated sperm oil, sulphonated whale oil and linseed fatty acid. The sulphonated products were tried because they had proved to be more or less successful on Australian ores containing davidite, a mineral similar to brannerite. In conjunction with these, Cyanamid Reagents 721, 723 and 730 and Armour's D-142 and D-242 were used. Cresylic acid was used as a frother and fuel oil as a froth modifier.

The best result obtained in this work was a recovery of 92 per cent in 50 per cent of the weight. The reagents used were sulphonated sperm oil (2.0 lbs/ton), Cyanamid 721 (1.0 lb/ton), cresylic acid (0.25 lb/ton) and fuel oil (4.0 lbs/ton). With the object of testing these reagents under continuous operation to allow cleaning of the concentrate, a pilot-plant test at the rate of 90-100 lbs per hour was conducted. Although recoveries of 92.5 per cent were obtained, the float weight was only reduced to 44.6 per cent of the feed weight — or a ratio of concentration of 2.2:1.
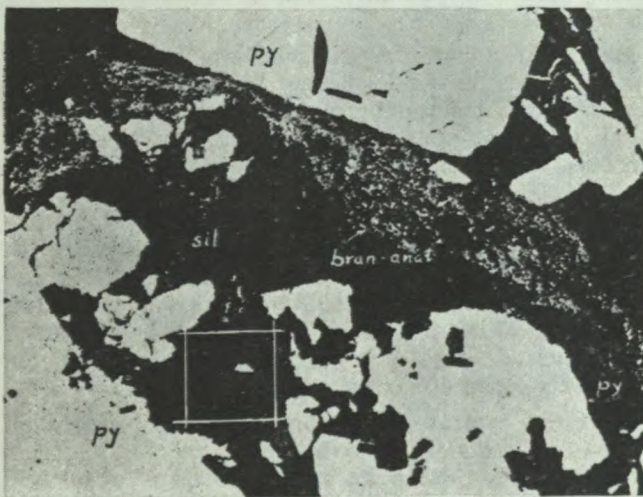

Figure 1.—Photomicrograph of polished section of ore, showing disseminated intergrowth of brannerite in siliceous gangue.
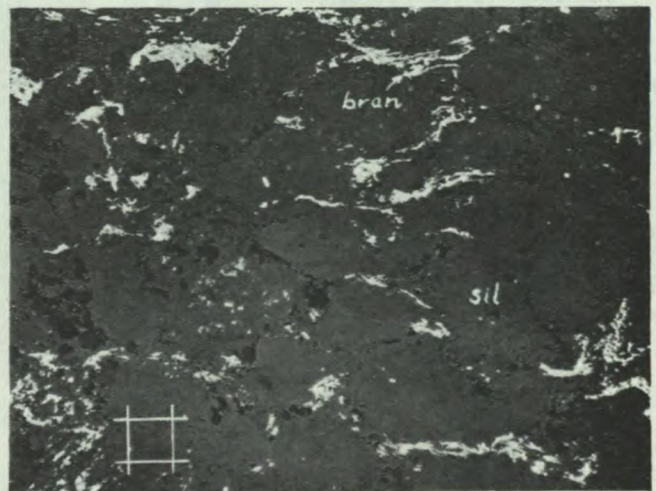(A 200-mesh opening is outlined on the photo)

Figure 2.—Polished section of ore, showing streaks of brannerite in siliceous gangue.
(A 200-mesh opening is outlined on the photo)

Samples of the cleaner tailing and final tailing were examined by the mineralogist, who observed some locked uranium particles and some free radio-active grains. As the grind was 92 per cent minus 200 mesh, finer grinding did not seem practical because an excessive amount of slimes would be produced. It was observed that the use of sodium silicate as a gangue depressant in the cleaning step resulted in a higher tailing loss, probably due to depression of the middling particles. Sodium silicate was found to be an excellent depressant of quartz gangue, but the quantity used in relation to the amount of collector was quite critical. The cost of reagents was in the order of 50 cents per ton of ore and the ratio of concentration was not considered satisfactory.

A further laboratory study was carried out a year later using mono-di-iso-octyl acid orthophosphate as a collector. The possibility of using an organic phosphate was suggested by the affinity of these reagents for uranium in solvent extraction processes. The results showed that when using 2 lbs of the orthophosphate reagent and 1.5 lbs of diesel fuel oil per ton of ore, recoveries of 90 per cent were obtained in 40 to 45 per cent of the weight; however, the cost of the reagent, which was in the order of $1.05/ton, was considered to be too high.

During the same period, a second method was developed in which the reagents were added directly to the ball mill. The reagents used were sulphonated whale oil (1.5 lbs/ton), linseed fatty acid (2.5 lbs/ton), diesel fuel oil (16.0 lbs/ton) and cresylic acid (0.3 lb/ton). The reagent cost for this method was about 85 cents per ton, with a recovery and ratio of concentration similar to the previous method. A disadvantage was that an excessive amount of froth was produced and was difficult to control. The next step was to find reagents that would be less costly and would not produce such a volume of froth.

Later Laboratory Studies

Previous testing had indicated that fatty acids were suitable collectors, although they were rather expensive at a cost of 17 to 19 cents a pound. In 1959, a series of laboratory tests was carried out using Acintol on five uranium ores from the Elliot Lake area. Acintol is a relatively inexpensive crude tall oil fatty acid, costing about 9 cents per pound. When using this reagent, it was found that the froth could be controlled by the use of sodium silicate in the correct amount, and that the Acintol should be stage-added to minimize excessive frothing. To obtain best results, it was necessary to deslime the flotation feed and to use a pH of 8 to 9.5 in the flotation step. Recoveries of from 92 to 95 per cent in 55 per cent of the original weight were obtained with a reagent cost of from 25 to 30 cents per ton.

Pilot-Plant Studies

In the spring of 1964, a flotation pilot plant was operated at the Mines Branch for a period of about 33 days. The pilot plant was operated with three main objects in mind:

1. — To determine whether results of the recent laboratory work using the Acintol reagents could be equalled or improved by continuous operation.

2. — To provide a flotation preconcentrate for hydrometallurgical studies for assessment of the com-bined value of flotation followed by various leaching techniques.

3. — To provide a quantity of flotation tailing for bacterial leaching tests, because, if the flotation tailing was amenable to bacterial leaching, the need for a high recovery by flotation would be of less importance.

The ore was received from an Elliot Lake mine in two 25-ton shipments about one month apart, and the pilot-plant runs were started soon after the shipments arrived in order to minimize changes by oxidation. The sample was typical of the quartz-pebble conglomerate of the area, as previously described.

Procedure

The mine ore was crushed to -4 mesh. In the first mill run, it was fed to a 20- by 20-in. rod mill in closed circuit with a 20-mesh screen. The oversize was returned to the rod mill, and the undersize was fed to a P-50 ceramic Dorrclone in closed circuit with a 30- by 48-in. ball mill. The overflow from the cyclone was the feed to the flotation circuit.

In the second mill run, the rod mill and the Dorrclone were eliminated from the grinding circuit. The Dorrclone was replaced by a 50-mesh Sweco screen and the undersize from it was the flotation feed. These runs were operated at the rate of 500 pounds per hour.

The general procedure for the first mill run was to remove the sulphides first, to separate the slimes from the sulphide tailings and to make a rougher uranium flotation concentrate that was cleaned and re-cleaned. A scavenger uranium concentrate was removed after the rougher float and this concentrate was returned along with the cleaner tailings to the head of the circuit. The final tailings were 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 feed to the flotation circuit was the cyclone overflow, which was fed to a 2-cu-ft conditioner giving a 5-minute retention time at 25 per cent solids. From the conditioner, the pulp went to four No. 7 Denver cells for the sulphide float, which had a 9-minute float time. The sulphide tailings were pumped to a settling tank which removed the slimes in the overflow. The underflow was fed at 40 per cent solids to a 6-cu-ft conditioner which gave a retention time of about 20 minutes. The pulp from the conditioner was diluted to 20 to 25 per cent solids and passed to fourteen No. 7 Denver cells. The first five were uranium roughers and the last nine were uranium scavengers. The rougher concentrate went to six No. 5 Denver cleaner cells, where it was cleaned and re-cleaned. The scavenger concentrate and the cleaner tailings were returned to the head of the circuit.

Figure 3 is a diagram of the final flowsheet used for the first mill run, showing the reagent addition points. The recycle products (scavenger float and cleaner tailings) were 25 to 50 per cent of the feed weight, depending on the amount of reagent used. The circuit used in the second mill run was basically similar to that used in the first run except that a Humphrey spiral was used in place of the sulphide float toward the latter half of the run, as shown in the flowsheet in *Figure 4*.
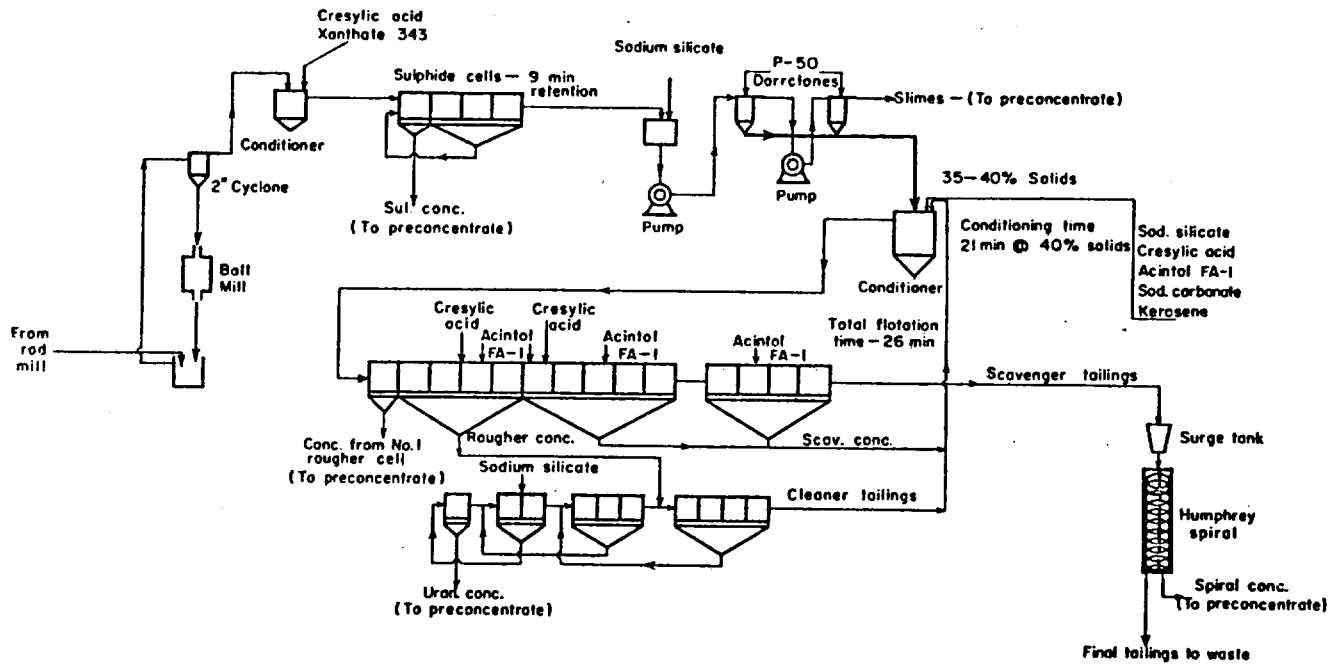
Cresylic acid
Xanthate 343

Sulphide cells — 9 min retention

Sodium silicate

P-50 Dorrclones

Slimes — (To preconcentrate)

Conditioner

2" Cyclone

Sul. conc. (To preconcentrate)

Pump

Pump

35—40% Solids

Ball Mill

Conditioning time 21 min @ 40% solids

Sod. silicate
Cresylic acid
Acintol FA-1
Sod. carbonate
Kerosene

Conditioner

From rod mill

Cresylic acid   Cresylic acid
Acintol FA-1   Acintol FA-1   Acintol FA-1

Total flotation time — 26 min

Scavenger tailings

Conc. from No. 1 rougher cell (To preconcentrate)
Rougher conc.
Sodium silicate
Scav. conc.

Cleaner tailings

Surge tank

Humphrey spiral

Uran. conc. (To preconcentrate)

Spiral conc. (To preconcentrate)

Final tailings to waste

Figure 3.—Flowsheet for Flotation Pilot-Plant Tests 12 to 19.

Table I—Operational Record — Flotation Mill Run No. 1

Test No.	Flotation Feed −200 M (%)	Flotation Feed Wt. (lb/hr)	Flotation Feed U₃O₈ (%)	Sulphide Conc. Wt. (%)	Sulphide Conc. U₃O₈ (%)	Sulphide Conc. Dist. (%)	Slimes Wt. (%)	Slimes U₃O₈ (%)	Slimes Dist. (%)	U₃O₈ Conc. Wt. (%)	U₃O₈ Conc. U₃O₈ (%)	U₃O₈ Conc. Dist. (%)	Spiral Conc. Wt. (%)	Spiral Conc. U₃O₈ (%)	Spiral Conc. 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	26.8	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 / 15.0	.40 / .26	37.6 / 27.5	—	—	—
13	69.4	492	.16	5.6	.23	8.7	14.7	.16	15.8	9.0 / 18.3	.44 / .33	26.8 / 40.5	2.4	.09	1.5
14	71.5	500	.16	5.8	.16	6.4	19.2	.17	22.4	6.0 / 19.6	.65 / .29	26.8 / 39.1	4.4	.04	1.2
15	67.9	520	.16	5.0	.15	5.2	28.5	.16	31.4	9.4 / 18.7	.32 / .28	20.7 / 36.1	1.3	.11	1.0
16	74.3	495	.16	3.2	.15	3.1	17.0	.18	19.6	11.3 / 12.3	.44 / .39	32.1 / 30.9	3.6	.15	3.5
17	65.7	493	.16	4.5	.15	5.0	14.2	.18	19.0	17.5 / 11.0	.24 / .40	31.3 / 32.5	2.3	.10	1.7
18	63.7	492	.16	4.3	.15	4.6	14.2	.16	16.1	6.0 / 3.8	.54 / .97	23.0 / 26.2	1.9	.37	5.0
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: "U₃O₈ concentrate" for Tests 12 to 19, two sets of data are given. The upper set is U₃O₈ recovery and grade in the concentrate from the first of the five rougher cells; the lower set is the U₃O₈ recovery and grade in the cleaner concentrate.

Test No.	Tailing Wt. (%)	Tailing U₃O₈ (%)	Tailing Dist. (%)	U₃O₈ Recovery (%)	Ratio of Conc.	Uran. Float pH	Sul. Float Xanth. 343	Sul. Float Cres. Acid	Deslime Na₂SiO₃	Uran. Float Acintol	Uran. Float Cres. Acid	Uran. Float N₂aSiO₃	Uran. Float Kerosene	Uran. Float N₂aCO₃
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

Reagents

For the sulphide float, the reagents were cresylic acid and Cyanamid's Xanthate 343.

In the uranium minerals float, Acintol FA-1 (supplied by Charles Albert Smith Ltd., Montreal, Que.) was used as the collector; it 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 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 are given in Tables I and II.
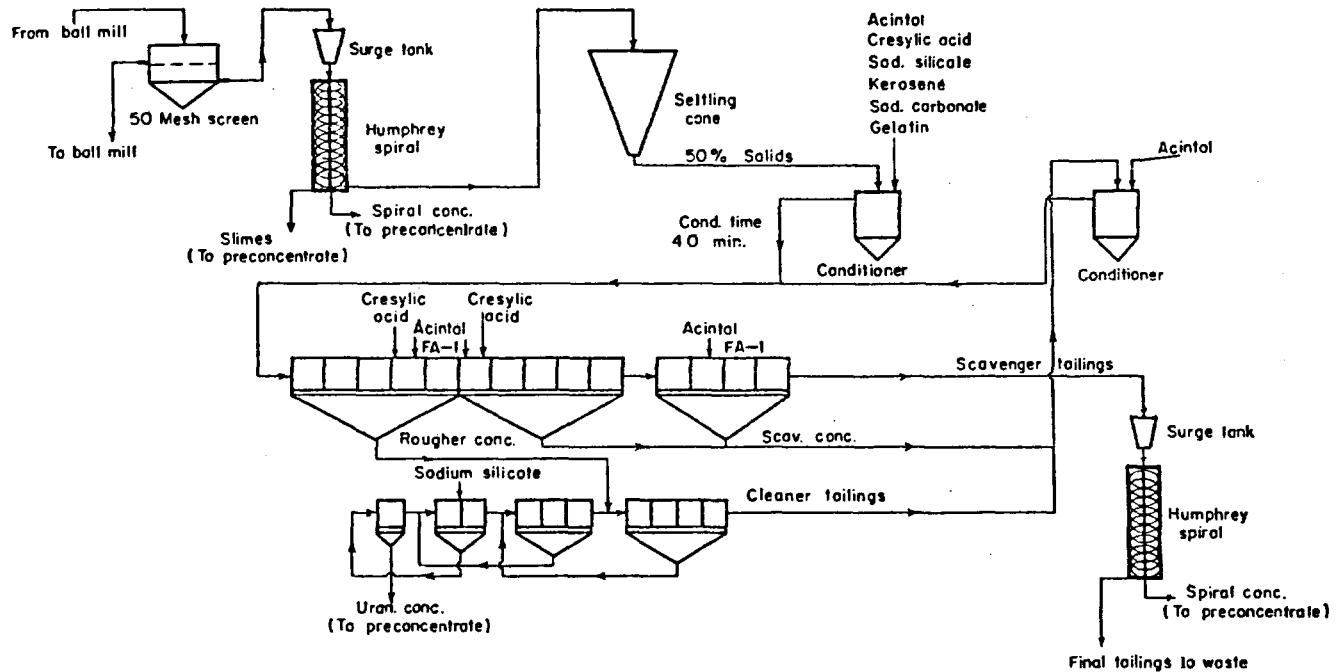

Figure 4.—Flowsheet for Flotation Pilot-Plant Tests 26 to 30.

Table II—Operational Record — Flotation Mill Run No. 2

Test No.	Flotation Feed			Sulphide Conc.			Slimes			U₃O₈ Conc.			Spiral Conc from Tailing		
	−200 M (%)	Wt. (lb/hr)	U₃O₈ (%)	Wt. (%)	U₃O₈ (%)	Dist. (%)	Wt. (%)	U₃O₈ (%)	Dist. (%)	Wt. (%)	U₃O₈ (%)	Dist. (%)	Wt. (%)	U₃O₈ (%)	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	17.3
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	.40
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

Test No.	Tailing			U₃O₈ Recovery (%)	Ratio of Conc.	Uran. Float pH	Reagents (lb per ton ore)								Remarks
							Sul. Float		Deslime	Uran. Float					
	Wt. (%)	U₃O₈ (%)	Dist. (%)				Xanth. 343	Cres. Acid	Na₂SiO₃	Acintol	Cres. Acid	Na₂SiO₃	Kerosene	Na₂CO₃	
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-silicate used in
26	60.5	.027	14.2	85.8	2.5	8.9	—	—	—	2.0	.11	.45	.11	.6	place of sodium silicate.
27	59.5	.029	12.7	87.3	2.5	8.9	—	—	—	1.9	.13	.47	.09	.6	
28	58.6	.025	10.9	89.1	2.4	8.8	—	—	—	2.2	.10	.60	.08	.6	
29	58.5	.028	10.0	90.0	2.4	8.8	—	—	—	2.1	.10	.55	.09	.6	Gelatin used in
30	62.7	.028	12.6	87.4	2.7	8.7	—	—	—	2.2	.11	.55	.09	.6	uranium flotation
31	65.4	.032	13.1	86.9	2.9	8.9	—	—	—	1.9	.09	.67	.09	.6	in amount of
32	69.6	.031	14.4	85.6	3.3	8.9	—	—	—	2.0	.08	.67	.09	.6	0.05 lb/ton.

Sampling

A composite feed sample was made by taking 1 pound of sample every half hour from the belt of the dry ore feeder to the rod mill. 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. To ensure sampling under stable conditions, these samples were taken about two hours after the start-up for the day, and at half-hour intervals thereafter. They were used for chemical analyses, metallurgical weight balances and screen analyses. Samples were also taken at various points to establish the per cent solids in the circuit.

Results of Pilot-Plant Study

The first mill run consisted of 18 days operation of the pilot plant; the second run consisted of 14 days operation. The results are given in Tables I and II.

The tests in the two runs can be classified into four series:

(a) — The series from Test 2 to 10 was considered a break-in period.

(b) — The series comprising Tests 11 to 19 was carried out with a grind of 65-75 per cent minus 200 mesh, using the flowsheet shown in *Figure 3*. This involved sulphide flotation followed by desliming and uranium flotation.

(c) — The series comprising Tests 20 to 25 was carried out with a coarser grind of 45 to 55 per cent minus 200 mesh, also using the flowsheet shown in *Figure 3*.
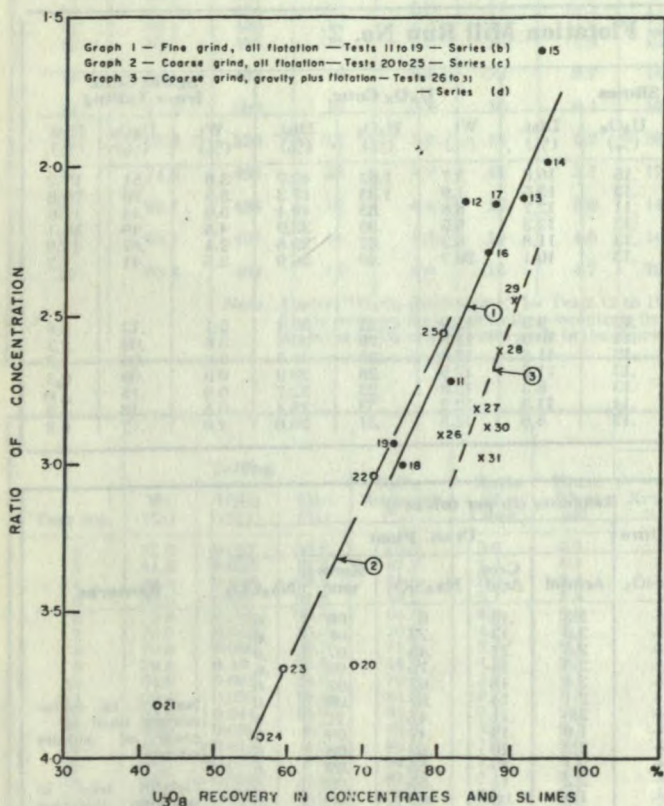

Figure 5.—Results of Tests 11 to 31.

(d) — The series comprising Tests 26 to 31 was carried out using the same grind as for (c) and employing the flowsheet shown in *Figure 4*. This involved gravity concentration of the sulphides and coarse uranium particles by Humphrey spiral, followed by uranium flotation. Desliming prior to flotation was also carried out by the Humphrey spiral.

The results of Series (b), (c) and (d) are shown graphically in *Figure 5*. To obtain the data for the graphs, the uranium recovered in the gravity concentrate from the Humphrey spiral operating on the final tailing was calculated as part of the tailing loss, so that the data would reflect only the results obtained in the main preconcentration operations. *Figure 5* shows that if the ratios of concentration are plotted against recoveries, the points obtained for each of the three series follow straight-line trends of similar slope in which the ratio of concentration decreases as the uranium recovery in the preconcentrate increases.

A study of Graph 1 and 2, *Figure 5*, shows that, with the all-flotation circuit, the uranium recovery is generally highest and the ratio 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, with grinds of between 65 and 75 per cent minus 200 mesh, had recoveries ranging from 87 to 94 per cent with ratios of concentration ranging from 2.3:1 to 2.0:1. Tests 20, 21, 23 and 24, with grinds of from 45 to 55 per cent minus 200 mesh, had recoveries ranging from 43 to 69 per cent with ratios of concentration of from 3.9:1 to 3.6:1.

Graph 3, *Figure 5*, shows the results of Tests 26 to 31, in which sulphide flotation was replaced by gravity concentration using a Humphrey spiral.

In these tests, the grind used was about 55 per cent minus 200 mesh. The gravity tailings were deslimed and the deslimed fraction treated by flotation for recovery of the uranium minerals. This flowsheet produced a higher recovery for a given ratio of concentration than was obtained using flotation only. Graph 3 shows that, by using gravity methods followed by flotation with a grind of 55 per cent minus 200 mesh, a uranium recovery of 90 per cent can be expected with a ratio of concentration of 2.5:1.

It was observed that an increase in Acintol and cresylic acid in the uranium flotation step effected an increase in recovery and weight floated. The effect of sodium silicate, on the other hand, was to reduce the recovery and weight floated. The variations in the amounts of these reagents used in these tests were actually small. The amount of Acintol ranged from 1.7 to 2.3 lbs 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.

As a means of depressing the gangue, meta-sodium silicate was tested as a substitute for sodium silicate, but no significant improvement was noted. Gelatin, however, when used in conjunction with sodium silicate, resulted in an improvement in the ratio of concentration (Tests 30 and 31).

The reagents used during the pilot-plant operation, and their approximate costs, are given in Table III.

Table III—Reagent Consumption and Cost

Reagent	Cost per lb. cents	All-Flotation		Gravity + Uranium Flotation	
		Consumption lb/ton	Cost cents	Consumption 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

Mineralogical Study of Pilot-Plant Products

Mineralogical examinations were carried out for four samples from all-flotation pilot-plant products, namely sulphides, slimes, U_3O_8 concentrate and tailings, and four samples from products produced by the Humphrey spiral and flotation of the gravity tailings —sulphide spiral concentrate, slimes, flotation U_3O_8 concentrates and tailings.

The uranium content of these products is given in Table IV.

To investigate the occurrence of the uranium-bearing minerals in the plant products, polished sections of the products other than the slimes were examined with an ore microscope. The slime fractions were too fine grained for microscopic examination. Autoradiographs were prepared of each polished section on alpha emulsion plates in order to locate the radioactive grains. All mineral identifications were confirmed by X-ray powder diffraction analysis of grains gouged from the polished sections.

Flotation Pilot-Plant Products

Mineralogical descriptions of the three products of the all-flotation pilot-plant test follow:

(1) Sulphide Concentrate

The U_3O_8 content of the sulphide product was approximately equal to that of a head sample of the ore. Of the few radioactive grains located by means of an autoradiograph, one was identified as uraninite. It occurred as a dense aggregate of irregular grey particles intergrown in quartz. Also present was a quartz grain containing a dense aggregate of fine grey lath-like particles showing strong straw-yellow internal reflection under crossed nicols. This grey lath-like aggregate was probably an intergrowth of brannerite and anatase.

(2) U_3O_8 Concentrate

This fraction contained about twice the content of U_3O_8 that was present in the head sample. Brannerite was the most abundant uranium-bearing mineral, occurring as aggregates of lath-like particles which in some grains were loosely packed in a quartz matrix (*Figure 6*) and in other grains were so densely packed as to form almost solid particles (*Figure 7*). Rust-brown to straw-yellow internal reflection, particularly noticeable in *Figure 6*, indicated the presence of anatase. The internal reflection was usually more intense in the outer portions of the grains. Fine, irregular

Table IV—Uranium Content of Pilot-Plant Products

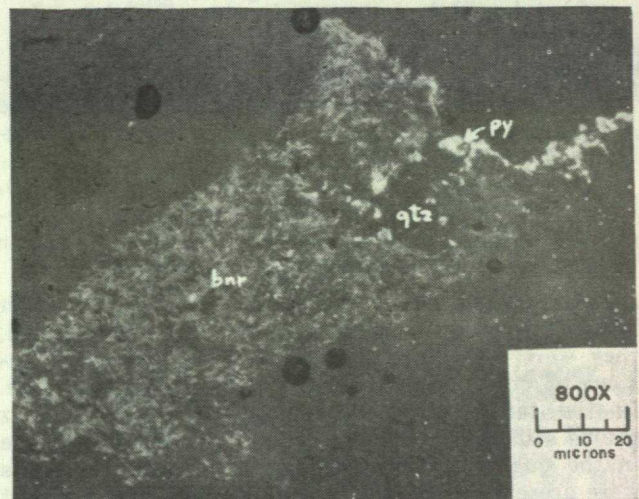| Pilot-Plant Test	Product	U_3O_8 %
Flotation	(1) Sulphides	0.16
	(2) Slimes	0.17
	(3) U_3O_8 concentrate	0.3
	(4) Tailings	0.023
Gravity and Flotation	(1) Sul. spiral conc.	0.7
	(2) Slimes	0.15
	(3) Flot. U_3O_8 conc.	0.25
	(4) Tailings	0.025

Figure 6.—U_3O_8 concentrate product showing brannerite in a quartz matrix.
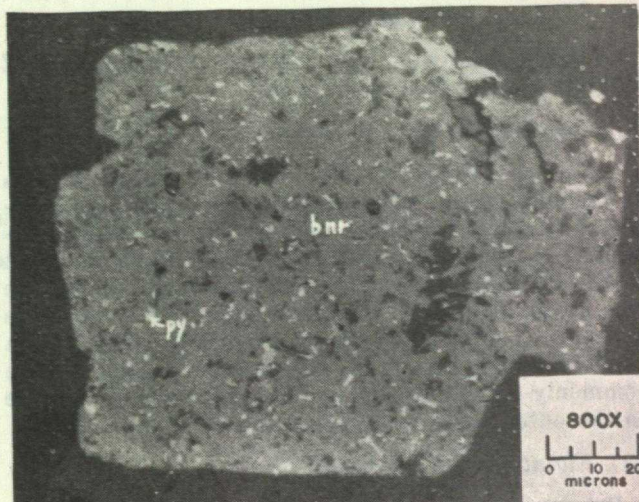

Figure 7.—U_3O_8 concentrate product showing solid particle of brannerite with pyrite inclusions.
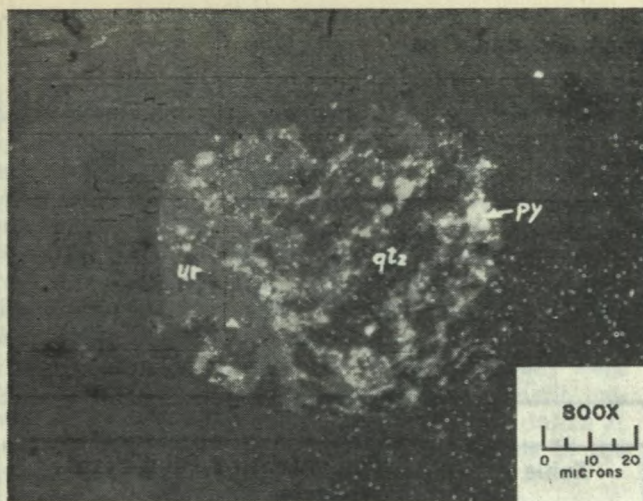
Figure 8.—U_3O_8 concentrate product showing relict crystal of altered uraninite.

Figure 9.—Sulphide spiral concentrate product showing brecciated relict crystals of uraninite in a quartz matrix.

lath-like particles of pyrite occurred in some of the brannerite grains. Occasionally, inclusions of hematite were also present in the brannerite.

Altered crystals of uraninite were present in the U_3O_8 concentrate. They often appeared to be brecciated, and in many cases only relics remained (*Figure 8*). Inclusions of pyrite and of other metallic minerals were less common than in the grains of brannerite.

(3) *Tailing*

The mode of occurrence of uranium-bearing minerals in the tailing product was similar to that of the U_3O_8 concentrate. However, brannerite was commonly sparser in brannerite-quartz grains, and only relics of uraninite crystals were present. The lower U_3O_8 content of the tailing as compared to that of the concentrate could be accounted for mainly by the smaller proportion of radioactive grains in the tailing.

It is evident from these descriptions that, even when the uranium-bearing minerals were concentrated, they were often intergrown with or contained inclusions of other minerals that diluted the concentrate. In the tailing product, although brannerite-quartz grains still reported, the brannerite was quite sparse in the quartz and only relics of uraninite crystals were present, indicating that these were very difficult to float because they were masked by the quartz.

Gravity-Flotation Pilot-Plant Products

Mineralogical descriptions of the three products of the second pilot-plant test follow:

(1) *Sulphide Spiral Concentrate*

There was approximately four times as much U_3O_8 in the sulphide spiral concentrate as in the head sample of the ore. Uraninite appeared to be more abundant than brannerite in this concentrate, in contrast to the other pilot-plant products in which brannerite was more abundant. Uraninite occurred as massive subhedral crystals and brecciated relict crystals in a quartz matrix (*Figure 9*). The brannerite grains were commonly quite massive, and the presence of anatase in the outer parts was indicated.

(2) *Flotation U_3O_8 Concentrate*

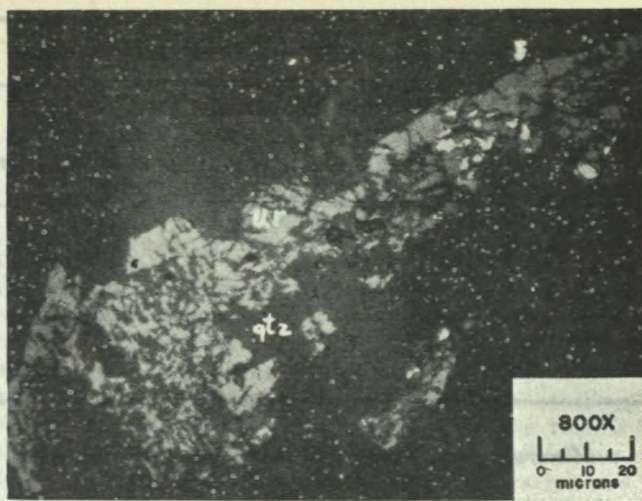Brannerite was the most abundant uranium-bearing mineral in the flotation U_3O_8 concentrate. Of particu-

lar interest in this concentrate were two types of grains not previously described. One was a grain of brannerite and rutile having a mosaic appearance. The other was a very weakly radioactive grain containing finely disseminated particles of uraninite in quartz.

(3) *Tailing*

Very few grains containing uranium-bearing minerals were present in the tailing product. One of these grains consisted of a few irregular grey particles of uraninite in a grain of quartz. The lath-like appearance of some of the particles in this grain indicates the presence of brannerite, but only uraninite was confirmed by X-ray diffraction methods.

From a study of the mineralogical descriptions of the gravity-flotation pilot-plant products, it was evident that the spirals were quite effective in recovering the uraninite crystals and the more massive brannerite crystals. It might be quite important to recover these crystals as soon as possible, as they might have a tendency to be partially broken down in the flotation cells. In the tailing product, it was again noticed that the uranium-bearing minerals consisted of irregular grey particles of uraninite and/or brannerite in quartz grains. As this was a coarser grind, this was understandable; if the mesh of grind was reduced, however, the uranium crystals would be destroyed and more slimes would be produced, thereby hindering recovery.

Discussion

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, compared with leaching the whole ore, and on the over-all recoveries obtained by these two approaches. Preliminary leach tests of the preconcentrate produced by the gravity-flotation circuit have indicated that about 94 per cent of the uranium in the preconcentrate could be recovered from the preconcentrate using current plant leaching procedures. With 90 per cent uranium recovery in preconcentration, the over-all recovery in preconcentration and leaching would be 84.6 per cent from an ore containing 3.4 lbs 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 these data, an evaluation of preconcentration and leaching, as shown in Table V, can be made.

Table V

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 lbs. U_3O_8 x .94 x 5.00 =	$16.00
Net per ton of 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 ton at $2.80)*	0.95
Total	$6.85
Value of uranium recovered at $5.00/lb = 3.4 lbs 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 preconcentrate is leached by the present method. However, if the uranium in the tailing can be recovered cheaply by bacterial leaching, and if more efficient hydro-metallurgical 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.

The best result obtained was a recovery of 90 per cent with a ratio of concentration of 2.5:1; this was obtained by gravity concentration followed by flotation. This recovery and the ratio of concentration are somewhat lower than desired. The mineralogy, however, indicates that a higher recovery is unlikely. Possibly one way to increase the ratio of concentration would be to float without desliming, as the untreated slime fraction accounts for nearly one-third of the preconcentrate. Further flotation testing with slimes present might yield useful results.

Conclusions

On uranium ore similar to that tested in these studies, it is possible, by using flotation and gravity methods, to recover 90 per cent of the uranium at ratios of concentration ranging from 2.1:1 to 2.9:1. If the ore is ground to 65 per cent 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 per cent 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.

As 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 costs 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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