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OTTAWA

MINES BRANCH INVESTIGATION REPORT IR 58-96

BENEFICIATION TEST WORK ON URANIUM-BEARING ORE FROM RAYROCK MINES LTD., YELLOWKNIFE, N. W. T.

Reference No. 12/57-19

by

W. R. HONEYWELL

RADIOACTIVITY DIVISION

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W. R. Honeywell*

SUMMARY OF RESULTS

Preconcentration by sink-float, jigging, tabling and flotation methods was tried on the sample. A discardable tailing could not be produced by jigging or tabling. By sink-float, a recovery of 78.7% in 44.9% of the weight was obtained at a grade of 0.27% U_3O_8. By flotation, a recovery of from 80 to 85% can be obtained in from 35 to 40% of the weight at a grade of from 0.40 to 0.35% U_3O_8.

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*Scientific Officer, Radioactivity Division, Mines Branch, Department of Mines and Technical Surveys, Ottawa, Canada.

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(15 tables, 0 illus.)

INTRODUCTION

A shipment of uranium-bearing ore, total weight approximately 931 pounds, was received at the Mines Branch, Ottawa, on December 26, 1957, from Rayrock Mines Limited, Yellowknife, N. W. T. Mr. F. B. Brien, consulting metallurgist for the company, in a letter dated January 31, 1958, and written from 2552 Roanoke Drive, Seattle 2, Washington, stated that the sample was representative of lower-grade ore from the Rayrock property. In the same letter, he requested that flotation and other preconcentration test work be carried out on this sample. As received, the sample was in coarse lump form.

LOCATION OF PROPERTY

The mine is located in the Marian River area, Yellowknife Mining Division, N. W. T. The head office address of the company is Suite 509, 25 Adelaide Street West, Toronto 1, Ontario. The property is covered by A. E. C. B. Mining Permit MP 13/57, issued March 6, 1957.

CHEMICAL AND SPECTROGRAPHIC ANALYSES

A head sample, cut from the material, was assayed chemically and radiometrically for uranium. Assays for other significant elements were also made and the results were as shown in Table 1.

TABLE 1

Chemical and Radiometric Analyses on Head Sample

Chem. Lab Sample No. RD-4259	%
U_3O_8 chemical	0.20
ThO_2 chemical	0.004
U_3O_8 secondary*	0.057
CO_2 evolution	0.15
CO_2 combustion	0.42
Fe (total)	3.05
S (total)	0.21
P_2O_5	0.065
As	<0.01
U_3O_8 radiometric	0.20
U_3O_8 gamma equivalent	0.174
U_3O_8 beta equivalent	0.185
Gold	not detectable
Silver	0.01 oz/ton
Specific Gravity	2.64

* A sample is leached for 30 minutes in a hot, 10% solution of Na_2CO_3. The uranium dissolved is taken as an indication of the secondary uranium present.

A semi-quantitative spectrographic analysis of the same head sample gave the results shown in Table 2.

TABLE 2

Spectrographic Analysis of Head Sample

Element	%	Element	%
Si	P. C.	Pb	0.06
Al	8.0	V	0.009
Fe	3.0	Cu	0.09
Mg	2.0	Zr	0.007
Na	1.5	Ti	0.06
Ca	0.8	Ni	0.01
As ?	0.2	Y	0.005
Ba	0.05	Yb ?	0.001
Mn	0.25	La ?	0.008

P. C. = principal constituent
? = not positively identified

GENERAL SUMMARY

Investigation into the preconcentration of the sample by sink-float, jigging, tabling and flotation methods was carried out.

1. On crushing the sample to -1 inch and treating at 2.64 S.G. by sink-float, the overall recovery would be 78.7% of the U_3O_8 in a concentrate containing 44.9% of the weight and 0.27% U_3O_8. (The uranium assays in this work were by gamma equivalent, which is about 10% lower than by the chemical method of assaying, for this ore).

2. On a sample crushed to -20 mesh, tabling was carried out with poor results. Since a low tailing was not produced, the ore is not amenable to gravity concentration by tabling.

3. Jigging tests, carried out on -4 + 14 mesh material, did not give satisfactory results. The tailings produced by the jigging, tabling and superpanning are all very similar in grade (about 0.12% U_3O_8). This further indicates that this sample is not amenable to preconcentration by these methods.

4. By flotation, using Acintol FA-2 as a promoter, a recovery of 79.0% of the uranium in a concentrate containing 35.5% of the weight and 0.37% $U3O_8$ was obtained. When using double-distilled oleic acid, the recovery was 84.6% of the uranium in a concentrate containing 39.68% of the weight and 0.40% U_3O_8.

SAMPLING

At the Mines Branch, the sample was first crushed to -1 inch and screened on a 4 mesh screen. After testing the - 1 inch product by sink-float methods, the sample was crushed to - 1/2 inch and riffled into quarters. One quarter was divided in half, and one half crushed to - 10 mesh for a head sample, and for flotation test samples. The other half was crushed to - 20 mesh for tabling tests.

MINERALOGY

The mineralogist* reports that the sample consists of a light-coloured, fine- to medium-grained rock, and is composed mostly of quartz. Metallic minerals include traces of specular hematite and the copper-iron sulphides, chalcopyrite and bornite. The uranium-bearing mineral is pitchblende.

The pitchblende is finely disseminated, or occurs as fine veinlets. The average area of pitchblende is around 65 mesh; the veinlets, however, are much finer.

DETAILS OF TEST WORK

1. Sink-float Concentration Tests

On a sample of the material, crushed to - 1 inch + 4 mesh, a sink-float test was carried out using the heavy liquid, acetylene

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* Hughson, M. R., "Mineralogy of a Bulk Sample of Uranium Ore from Rayrock Mines Ltd., Yellowknife Mining Division, N. W. T., Reference No. 12/57-19", Mines Branch Investigation Report IR 58-32, Department of Mines and Technical Surveys, Ottawa, Canada, March 4, 1958.

tetrabromide, diluted with carbon tetrachloride to give a specific gravity of 2.64. The float obtained at this medium density was retreated at a specific gravity between 2.61 and 2.62. Retreatment of the float was previously tried at 2.62 S.G. and only 2.5% sink was obtained. The float was also tried at 2.60 S.G. and all the pieces sank. The overall metallurgy for this test is given in Table 3.

TABLE 3

Sink-Float Results after Crushing to -1 inch

Products	Weight, %	γ Assay, % U_3O_8	% Dist, U_3O_8
-1 in. + 4M, Fl. at 2.615 S.G.	35.55	0.038	8.7
-1 in. + 4M, S. at 2.615, Fl. at 2.64 S.G.	19.48	0.10	12.6
-1 in. + 4M, S. at 2.64 S.G.	11.99	0.48	37.1
-4M, Untreated	32.98	0.196	41.6
Original Ore	100.00	0.16	100.0

If the sink at 2.64 S.G. is combined with the -4 mesh untreated material, the U_3O_8 recovery in the resulting preconcentrate would be 78.7% in 44.9% of the weight at a grade of 0.27% U_3O_8.

The sample was then crushed to - 1/2 inch and screened on 4, 10, 35, 65, 100 and 200 mesh screens. The three coarsest sizes were then treated by sink-float methods, as previously outlined. The four finer sizes were treated by the superpanner.

The screen analysis and distribution of uranium is given in Table 4 and the sink-float and superpanner results are given in Tables 5, 6 and 7.

TABLE 4

Screen Analysis and Uranium Distribution after Crushing to -1/2 in.

Size	Weight, %	Assay, * % U_3O_8	% Dist, U_3O_8
-1/2 in. + 4 mesh	57.94	0.159	51.7
-4 + 10 mesh	19.37	0.171	18.6
-10 + 35 mesh	11.55	0.181	11.8
-35 + 65 mesh	6.12	0.217	7.5
-65 + 100 mesh	1.13	0.301	1.9
-100 + 200 mesh	1.02	0.29	1.7
-200 mesh	2.87	0.425	6.8
	100.00	0.18	100.0

* The assays are taken from the calculated assays given in Table 5.

The results of the screen analysis (Table 4) show a slight tendency towards the preferential crushing of the uranium mineral, with the resultant concentration of uranium in the fine sizes.

The results given in Table 5 show that from 60 to 80% of the uranium was recovered in the sink at these sizes.

TABLE 5

Results of Sink-float and Superpanner Tests after Crushing to -1/2 in.

Size Fraction, Products	Wt %, Fraction	Wt %, Overall	Assay*, % U_3O_8	% Dist. U_3O_8 Fraction	% Dist. U_3O_8 Overall
Sink-float					
-1/2 in. + 4 Mesh					
Float at 2.64 S.G.	85.25	49.39	0.074	39.6	20.5
Sink at 2.64 S.G.	14.75	8.55	0.65	60.4	31.2
Fraction	100.00	57.94	0.159	100.0	51.7
-4 + 10 Mesh					
Float at 2.64 S.G.	84.30	16.33	0.063	31.1	5.8
Sink at 2.64 S.G.	15.70	3.04	0.75	68.9	12.8
Fraction	100.00	19.37	0.171	100.0	18.6
-10 + 35 Mesh					
Float at 2.64 S.G.	80.51	9.30	0.045	20.1	2.4
Sink at 2.64 S.G.	19.49	2.25	0.74	79.9	9.4
Fraction	100.00	11.55	0.181	100.0	11.8
Superpanning					
-35 + 65 Mesh					
Tip	0.72	0.044	7.43	24.8	1.9
1st Middling	6.03	0.369	0.66	18.0	1.4
2nd Middling	16.64	1.018	0.15	11.3	0.8
Tailing	76.61	4.689	0.13	45.9	3.4
Fraction	100.00	6.120	0.217	100.0	7.5
-65 + 100 Mesh					
Tip	1.25	0.014	12.99	52.9	1.0
1st Middling	2.76	0.031	0.80	5.9	0.1
2nd Middling	14.85	0.168	0.19	8.8	0.2
Tailing	81.14	0.917	0.12	32.4	0.6
Fraction	100.00	1.130	0.301	100.0	1.9
-100 + 200 Mesh					
Tip	0.90	0.009	16.38	50.0	0.9
1st Middling	10.11	0.103	0.37	13.3	0.2
2nd Middling	12.02	0.123	0.19	6.7	0.1
Tailing	76.97	0.785	0.12	30.0	0.5
Fraction	100.00	1.020	0.29	100.0	1.7
-200 Mesh					
Tip	1.46	0.042	9.90	34.4	2.4
1st Middling	7.49	0.215	0.53	9.0	0.6
2nd Middling	15.28	0.438	0.19	6.6	0.4
Tailing	57.76	1.658	0.14	18.9	1.3
Slimes	18.01	0.517	0.73	31.1	2.1
Fraction	100.00	2.870	0.425	100.0	6.8
Original Ore		100.00	0.18		100.0

*Assays for sink-float products were by gamma equivalents; superpanner assays were by chemical analysis.

TABLE 6

Metallurgical Balance for Sink-float and Superpanner Tests
Preconcentrate 15% of Feed Weight

Size Fraction, Products	Wt. %, Overall	Assay, % U_3O_8	% Dist., U_3O_8
Preconcentrate			
-1/2 in. + 4M, Sink at 2.64	8.55	0.65	31.2
-4 + 10M, Sink at 2.64	3.04	0.75	12.8
-10 + 35M, Sink at 2.64	2.25	0.74	9.4
-35 + 65M, Superpan Tip	0.044	7.43	1.9
-35 + 65M, Superpan 1st Middling	0.369	0.66	1.4
-65 + 100M, Superpan Tip	0.014	12.99	1.0
-65 + 100M, Superpan 1st Middling	0.031	0.80	0.1
-100 + 200M, Superpan Tip	0.009	16.38	0.9
-100 + 200M, Superpan 1st Middling	0.103	0.37	0.2
-200M, Superpan Tip	0.042	9.90	2.4
-200M, Superpan 1st Middling	0.215	0.53	0.6
-200M, Superpan Slimes	0.517	0.73	2.1
Preconcentrate	15.184	0.75	64.0
Tails			
-1/2 in. + 4M, Fl. at 2.64	49.39	0.074	20.5
-4 + 10M, Fl. at 2.64	16.33	0.063	5.8
-10 + 35M, Fl. at 2.64	9.30	0.045	2.4
-35 + 65M, Superpan 2nd Middling	1.018	0.15	0.8
-35 + 65M, Superpan Tailing	4.689	0.13	3.4
-65 + 100M, Superpan, 2nd Middling	0.168	0.19	0.2
-65 + 100M, Superpan, Tailing	0.917	0.12	0.6
-100 + 200M, Superpan, 2nd Middling	0.123	0.19	0.1
-100 + 200M, Superpan, Tailing	0.785	0.12	0.5
-200M, Superpan, 2nd Middling	0.438	0.19	0.4
-200M, Superpan, Tailing	1.658	0.14	1.3
Tailing	84.816	0.076	36.0
Original Ore	100.000	0.18	100.0

TABLE 7

Metallurgical Balance for Sink-float and Superpanner Tests
Preconcentrate 25% of Feed Weight

Size Fraction, Products	Wt. %, Overall	Assay, % U_3O_8	% Dist., U_3O_8
Preconcentrate			
-1/2 in. + 4M, Sink at 2.64	8.55	0.65	31.2
-4 + 10M, Sink at 2.64	3.04	0.75	12.8
-10 + 35M, Sink at 2.64	2.25	0.74	9.4
-35 + 65M, Superpan Tip	0.044	7.43	1.9
-35 + 65M, Superpan 1st Middling	0.369	0.66	1.4
-35 + 65M, Superpan 2nd Middling	1.018	0.15	0.8
-35 + 65M, Superpan Tailing	4.689	0.13	3.4
-65 +100M, Superpan Tip	0.014	12.99	1.0
-65 +100M, Superpan 1st Middling	0.031	0.80	0.1
-65 +100M, Superpan 2nd Middling	0.168	0.19	0.2
-65 +100M, Superpan Tailing	0.917	0.12	0.6
-100+200M, Superpan Tip	0.009	16.38	0.9
-100+200M, Superpan 1st Middling	0.103	0.37	0.2
-100+200M, Superpan 2nd Middling	0.123	0.19	0.1
-100+200M, Superpan Tailing	0.785	0.12	0.5
-200M, Superpan Tip	0.042	9.90	2.4
-200M, Superpan 1st Middling	0.215	0.53	0.6
-200M, Superpan 2nd Middling	0.438	0.19	0.4
-200M, Superpan Tailing	1.658	0.14	1.3
-200M, Superpan Slimes	0.517	0.73	2.1
Preconcentrate	24.980	0.51	71.3
Tails			
-1/2 in. + 4M, Fl. at 2.64	49.39	0.074	20.5
-4 + 10M, Fl. at 2.64	16.33	0.063	5.8
-10 + 35M, Fl. at 2.64	9.30	0.045	2.4
Tailing	75.02	0.068	28.7
Original Ore	100.00	0.18	100.0

The metallurgical balances, given in Tables 6 and 7, indicate the optimum grades of preconcentrates which might be expected to be obtained from gravity concentration methods. An acceptable tailing was not obtained, however, by the superpanner.

2. Gravity Concentration by Tabling

On a sample which had been crushed to -20 mesh and screened on a 65 mesh screen, tabling was carried out on a Wilfley laboratory table. The results of this work, as well as the metallurgical balance, are given in Table 8.

It will be noted that the results obtained by tabling were poor. The tailing grades, however, were similar to the tailing grades obtained by superpanning.

3. Gravity Concentration by Jigging

A sample, which had been crushed to -1/2 inch, was screened on 4, 14, and 35 mesh screens. The -4 + 14 mesh fraction was treated on a 4"x6" Denver mineral jig. The -14 + 35 mesh fraction was tabled and the -35 mesh fraction was untreated, as the grade of this fraction was approximately preconcentrate grade.

The results of the complete treatment of the various size fractions, by sink-float, jigging and tabling, are given in Table 9. The metallurgical balance, giving the preconcentrate grade and the recovery, is given in Table 10.

TABLE 8

Results of Tabling after Crushing to -20 Mesh

Size Fraction, Products	Wt. %, Fraction	Wt. %, Overall	Chem. Assay, % U_3O_8	% Dist. U_3O_8 Fraction	% Dist. U_3O_8 Overall
-20 + 65 Mesh					
Concentrate	9.64	7.14	0.52	35.4	23.4
Circulating*	0.57	0.42	0.29	1.1	0.8
1st Tailing	89.63	66.40	0.10	63.3	41.8
2nd Tailing	0.16	0.12	0.13	0.2	0.1
Fraction	100.00	74.08	0.142	100.0	66.1
-65 Mesh					
Concentrate	0.19	0.05	14.47	13.4	4.5
Middling (Actual)	8.82	2.29	0.49	20.8	7.0
1st Tailing	41.89	10.85	0.12	24.1	8.2
2nd Tailing	22.42	5.81	0.15	16.1	5.5
Fines	26.68	6.92	0.20	25.6	8.7
Fraction	100.00	25.92	0.21	100.0	33.9
Original Ore		100.00	0.16		100.0

Metallurgical Balance

-20 + 65M, Concentrate	7.14	0.52	23.4
-20 + 65M, Circulating*	0.42	0.29	0.8
-65M, Concentrate	0.05	14.47	4.5
-65M, Middling	2.29	0.49	7.0
-65, Fines	6.92	0.20	8.7
Preconcentrate	16.82	0.48	44.4
-20 + 65M, 1st Tailing	66.40	0.10	41.8
-20 + 65M, 2nd Tailing	0.12	0.13	0.1
-65 M, 1st Tailing	10.85	0.12	8.2
-65M, 2nd Tailing	5.81	0.15	5.5
Tailing	83.18	0.11	55.6
Original Ore	100.00	0.16	100.0

* At these sizes, the middling fraction was recirculated over the table and this portion represents the circulating material left at end of run.

TABLE 9

Results of Sink-float, Jigging and Tabling, after Crushing to -1/2 in.

Size Fraction, Products	Wt. %, Fraction	Wt. %, Overall	Assay*, % U_3O_8	% Dist. U_3O_8 Fraction	% Dist. U_3O_8 Overall
Sink-Float					
-1/2 in. + 4 Mesh					
Float at 2.64 S.G.	85.25	49.39	0.074	39.6	20.4
Sink at 2.64 S.G.	14.75	8.55	0.65	60.4	31.1
Sink-Float Fraction	100.00	57.94	0.159	100.0	51.5
Jigging					
-4 + 14 Mesh					
Top Bed	6.26	1.80	0.11	4.1	1.1
Bottom Bed	5.85	1.68	0.47	16.2	4.4
Hutch Concentrate	17.55	5.03	0.25	25.9	7.0
Tailing	70.34	20.17	0.13	53.8	14.6
Jigged Fraction	100.00	28.68	0.17	100.0	27.1
Tabling					
-14 + 35 Mesh					
Concentrate	0.08	0.002	20.28	9.1	0.2
Middling	29.13	0.609	0.35	47.7	1.2
1st Tailing	70.79	1.479	0.13	43.2	1.1
Tabled Fraction	100.00	2.090	0.21	100.0	2.5
-35 Mesh					
Untreated Fraction		11.29	0.30	100.0	18.9
Original Ore		100.00	0.18		100.0

* Assays for sink-float products in Tables 9 and 10 were by gamma equivalents, and the jigging and tabling assays were by chemical analysis.

TABLE 10

Metallurgical Balance of Sink-float, Jigging and Tabling after Crushing to -1/2 in.

Size Fraction, Products	Wt. %, Overall	Assay*, % U_3O_8	% Dist., U_3O_8, Overall
-1/2 in. + 4M, S. at 2.64 S.G.	8.55	0.65	31.1
-4 + 14M, Jig Bottom Bed	1.68	0.47	4.4
-4 + 14M, Jig Hutch Conc.	5.03	0.25	7.0
-14 + 35M, Table Concentrate	0.002	20.28	0.2
-14 + 35M, Table Middling	0.609	0.35	1.2
-14 + 35M, Table 1st Tailing	1.479	0.13	1.1
-35M, Untreated	11.29	0.30	18.9
Preconcentrate	28.640	0.40	63.9
-1/2 + 4M, Fl at 2.64 S.G.	49.39	0.074	20.4
-4 + 14M, Jig Top Bed	1.80	0.11	1.1
-4 + 14M, Jig Tailing	20.17	0.13	14.6
Tailing	71.36	0.09	36.1
Original Ore	100.000	0.18	100.0

4. Preferential Grinding

On some ores, the uranium mineral, being friable, grinds down into the fine sizes and the ground, coarser material may be left low in uranium. Three preferential grinding tests were carried out on a sample crushed to -4 mesh. The charge was 2000 g of pebbles in an Abbe mill grinding 1000 g of ore with 350 g water, or 74% solids. The samples were ground for 20, 30 and 40 minutes.

The results are given in Table 11.

TABLE 11

Results of Preferential Grinding Tests

	Wt, %	γ Assay, % U_3O_8	% Dist., U_3O_8
(a) 20 min grind			
Mesh			
+ 10	55.5	0.12	41.6
-10 + 20	13.5	0.17	14.4
-20 + 48	7.8	0.16	7.8
-48	23.2	0.25	36.2
Original Ore	100.0	0.16	100.0
(b) 30 min grind			
+ 10	53.3	0.12	40.3
-10 + 20	12.7	0.15	12.0
-20 + 48	6.5	0.15	6.2
-48	27.5	0.24	41.5
Original Ore	100.0	0.16	100.0
(c) 40 min grind			
+ 10	55.0	0.14	45.8
-10 + 20	10.9	0.14	9.1
-20 + 48	4.8	0.18	5.1
-48	29.3	0.23	40.0
Original Ore	100.0	0.17	100.0

There is a slight tendency for the uranium mineral to grind preferentially but it is not sufficiently pronounced for grinding to be used as a preconcentration method.

5. Flotation Tests

A number of flotation tests were carried out. A charge of 1150 g of ore was used giving a pulp density, at the start of the test, of about 28 percent in the flotation cell. A 40 minute grind was used in the first test and a 30 minute grind was used in all subsequent tests. These grinding times resulted in products containing 91.8 and 66.0% -200 mesh, respectively.

All of the tests, except one, were deslimed. Reagents added were NaOH, 1 lb/ton, and $Na_2 SiO_3$, 0.5 lb/ton. The slurry was stirred with the above reagents, allowed to settle 10 minutes, and deslimed. After this initial desliming, the slurry was stirred again, allowed to settle for another 10 minute period, and deslimed a second time. No additional reagents were added in the second desliming step.

Flotation With Acintol D*

Four tests were carried out using Acintol D, with variations as follows:

(a) 40 min grind, deslimed
(b) 30 min grind, deslimed (duplicate)
(c) 30 min grind, not deslimed

The flotation procedure which was used in these tests and all subsequent flotation tests, using Acintol products, was as follows:

Reagents Added	lb/ton
Rougher Float	
Na_2SiO_3	1.0
Acintol D	1.5
Conditioned - 3 minutes	
Float - 3 to 4 minutes (pH 8.0)	
Scavenger Float	
Acintol D	1.5
Dowfroth 250	0.04
Conditioned - 3 minutes	
Float - 4 minutes	

* The Acintol products are tall oil products supplied by Charles Albert Smith of Montreal and produced by Arizona Chemicals, New York.

The results of these tests are given in Tables 12 and 13. The test without desliming gave unsatisfactory results as nearly all of the solids floated. The tailings which were left were not as low grade as in the deslimed tests and, accordingly, all the subsequent tests were deslimed.

TABLE 12

Results of Flotation with Acintol D.
(40 min grind; 91.8% -200 mesh)

Products	Wt., %	Chem. Assay, % U_3O_8	% Dist., U_3O_8
Slimes	17.2	0.28	28.3
Rougher Concentrate	11.4	0.58	38.8
Scavenger Concentrate	41.5	0.10	24.4
Rougher Tailing	29.9	0.048	8.5
Original Ore	100.0	0.17	100.0

TABLE 13

Results of Flotation with Acintol D.
(30 min grind; 66.0% -200 mesh)

Products	Wt., %	Chem. Assay, % U_3O_8	% Dist., U_3O_8
Slimes	12.8	0.33	25.0
Rougher Concentrate	10.9	0.40	25.8
Cleaner Scavenger Concentrate*	8.3	0.47	23.1
Cleaner Scavenger Tailing	18.4	0.10	10.9
Rougher Tailing	49.6	0.052	15.2
Original Ore	100.0	0.17	100.0

* No additional reagent added to clean the rougher scavenger concentrate.

In Table 13, if the slimes, rougher float and cleaner scavenger float are combined, the U_3O_8 recovery would be 73.9%, in 32.0% of the weight, at a grade of 0.39% U_3O_8.

Flotation With Acintol FA-1 and FA-2

Two tests were conducted next to compare Acintol FA-1 and Acintol FA-2. In both tests, 3.0 lb of the tall oil per ton ore was used, and the procedure was similar to that used in the Acintol D tests. The Acintol FA-2 gave slightly better results than the Acintol FA-1. The results of the better test are given in Table 14.

TABLE 14

Results of Flotation with Acintol FA-2

Products	Wt., %	Chem. Assay, % U_3O_8	% Dist., U_3O_8
Slimes	13.25	0.30	24.1
Rougher Concentrate	6.19	0.69	25.8
Scavenger Concentrate	16.04	0.30	29.1
Rougher Tailing	64.52	0.054	21.0
Original Ore	100.00	0.17	100.0

If the slimes, rougher float and scavenger float are combined, the U_3O_8 recovery would be 79.0%, in 35.5% of the weight, at a grade of 0.37% U_3O_8.

Flotation Tests Using Oleic Acid

A series of four tests was carried out with double-distilled oleic acid as the promoter. Three tests were run with the pH about 8.3 and these gave the best results of all the tests carried out. In the fourth test, sodium silica fluoride (Na_2 Si F_6) was used instead of the sodium silicate used in all the other tests. The Na_2 Si F_6 resulted in a pulp pH of 6.2 during flotation. The results with the acid pH were not as satisfactory as with the basic pH.

All the tests were deslimed as previously outlined.

The method and results (Table 15) for the best test follow:

Reagents Added	lb/ton
Rougher Float	
$Na_2 SiO_3$	1.0
Oleic acid, double-distilled	1.0
Conditioned - 3 minutes	
Float - 3 minutes (pH 8.3)	
Scavenger Float	
Oleic acid, double-distilled	1.5
Conditioned - 3 minutes	
Float - 3 to 4 minutes	
Cleaned scavenger float - 5 minute.	

TABLE 15

Results of Best Test Using Oleic Acid

Products	Wt., %	Chem. Assay, % U_3O_8	% Dist., U_3O_8
Slimes	12.13	0.32	20.7
Rougher Concentrate	3.54	0.60	11.3
Cleaner Scavenger Concentrate	12.05	0.70	44.9
Cleaner Scavenger Tailing	11.96	0.12	7.7
Rougher Tailing	60.32	0.048	15.4
Original Ore	100.00	0.19	100.0

If the slimes, rougher float and cleaner scavenger float are combined, the recovery would be 76.9% in 27.72% of the weight at a grade of 0.52% U_3O_8. If the cleaner scavenger tails are also added to the above, the recovery would be 84.6% in 39.68% of the weight at a grade of 0.40% U_3O_8.

Tests using either octyl-phosphoric acid or Aero Promoter 710 gave unsatisfactory results.

DISCUSSION

The present sample is not amenable to preconcentration by jigging or tabling. Sink-float will effect some concentration, but difficulty may be experienced since a small change in the medium density has a marked effect on the amount of material that will sink or float.

By flotation, a recovery of from 80 to 85 percent can be obtained in from 35 to 40 percent of the weight at a grade of from 0.40 to 0.35% U_3O_8. The reagent cost would be about 50¢ per ton ore, plus freight.

It would seem that flotation would be the best method to preconcentrate this ore.

WRH/eew