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MINES BRANCH INVESTIGATION REPORT IR 58-191

INVESTIGATION OF URANIUM-BEARING ORE FROM NORTHSPAN URANIUM MINES LIMITED, LACNOR PROPERTY, ELLIOT LAKE, ONTARIO

by

W. R. HONEYWELL and H. H. McCREEDY

RADIOACTIVITY DIVISION

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Mines Branch Investigation Report IR 58-191

INVESTIGATION OF URANIUM-BEARING ORE FROM NORTHSPAN URANIUM MINES LIMITED, LACNOR PROPERTY, ELLIOT LAKE, ONTARIO
(Ref. No. 4/58-16)

by

W.R. Honeywell* and H. H. McCreedy*

SUMMARY OF RESULTS

Electronic sorting, sink-float, flotation, tabling and acid leaching tests were carried out on the sample.

By electronic sorting on minus 10 inch material, 26.8% of the weight could be discarded with a loss of 7.2% of the uranium.

By sink-float on minus 6 inch material, at a separating gravity of 2.66-2.67, from 30 to 40% of the weight could be discarded with a loss of from 5 to 9% of the uranium. On minus 1 inch material, from 35 to 50% of the weight could be discarded with a loss of from 3 to 8% of the uranium.

By flotation, from 32 to 55% of the weight could be eliminated with a loss of from 6 to 10% of the uranium.

Tabling did not give satisfactory results.

Acid leaching tests on the whole ore and on the sink-float preconcentrate resulted in 92-94% extraction of the U_3O_8 in each case. However, preconcentration followed by leaching effected an overall acid saving of 30 lb H_2SO_4/ton ore.

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

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(7 Tables, 1 Figure, 15 pages)

INTRODUCTION

At the Lacnor* property, a band of waste usually occurs in the middle of the ore bed. In mining the ore, the whole bed is mined and consequently this waste material is fed to the mill.

The object of the present work was to try to preconcentrate the mine-ore by eliminating this waste, using electronic sorting, sink-float, flotation or tabling methods, and thus effect an increase in leaching capacity, and a saving in overall leaching reagent cost.

Consequently, a shipment of uranium-bearing ore, weighing approximately 6000 pounds was received April 23, 1958 from the Lacnor Property of Northspan Uranium Mines Limited, Elliot Lake, Ontario. The shipment was designated as Radioactivity Division Sample No. 4/58-16. The request for the test work was contained in a letter dated March 17, 1958 from Mr. R. P. Ehrlich, Consulting Metallurgist for the Rio Tinto Management Services Limited, 335 Bay Street, Toronto, Ontario. In subsequent conferences with Mr. Ehrlich and with Dr. D. R. Derry, Vice-President of Exploration of the same company, it was decided that sink-float, electronic sorting, flotation and some leaching tests would be carried out on the sample.

The sample was received in fifteen drums and consisted of mine-ore ranging in size from about 12 inches down to fines.

* This mine was formerly named Lake Nordic Property.

LOCATION OF PROPERTY

The sample was shipped from the Lacnor mine, one of four mines operated by Northspan Uranium Mines Limited. The mine is located in the Elliot Lake area of northern Ontario.

The property is covered by A. E. C. B. Mining Permit MP 7/56 issued November 20, 1956.

CHEMICAL AND SPECTROGRAPHIC ANALYSES

A head sample from the material was assayed for uranium and other significant elements by chemical, radiometric and spectrographic methods. The results are shown in Tables 1 and 2.

TABLE 1

Head Sample Analyses
(Chemical and Radiometric)

Chemical Lab. No. RE-2724
Specific gravity - 2.67

Chemical Analysis	%
U_3O_8 (chemical)	0.11
U_3O_8 (secondary*)	0.015
CO_2 (evolution)	0.07
CO_2 (combustion)	0.55
Fe	3.95
S	2.59
Radiometric Analysis	
U_3O_8	0.10
U_3O_8 (beta equivalent)	0.102
U_3O_8 (gamma equivalent)	0.104

*A sample is leached for 30 minutes in a hot 10% solution of Na_2CO_3, without added oxidizing agent. The uranium dissolved is taken as an indication of the secondary uranium present.

TABLE 2

Head Sample Analysis
(Semi-quantitative Spectrographic)
(Assays in per cent)

Si	-	P. C.	Mn	-	0.03	B	-	0.003
Al	-	9.0	Ba	-	0.07	Zr	-	0.006
Fe	-	12.0	Mo	-	0.05	Cu	-	0.05
Ca	-	0.07	V	-	0.01(?)	Co	-	0.004
Mg	-	0.20	Cr	-	0.01	Y	-	0.01
Na	-	N. D.	Ti	-	0.08	Yb(?)	-	0.003
Zn	-	N. D.	Ni	-	0.05	La	-	0.01
Pb	-	0.1						

P. C. = principal constituent
N. D. = not detectable
? = identification not positive

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SAMPLING PROCEDURE

The sample as received consisted of mine-ore ranging in size from about 12 inches to fines. Electronic sorting was carried out on minus 10 inch ore from eight of the fifteen drums. The products from these tests were crushed separately to minus 1 inch. One-quarter of each product was then crushed to minus 10 mesh and a composite head sample was cut from these products.

MINERALOGY

The mineralogist* reports that brannerite, uraninite, and monazite occur in the matrix of a quartz-pebble conglomerate, and in thin bands in quartzite. Fine-grained, subhedral pyrite is usually associated with the radioactive minerals. Crystals of uraninite appear

*M. R. Hughson, Mineralogy Section. See reference on page 15

to be more numerous, but smaller, than aggregates of extremely fine-grained brannerite. The average diameter of the uraninite crystals is about 1/10 mm, whereas the aggregates of brannerite are usually about 1/3 mm. Although monazite is fairly common, it is less abundant than uraninite or brannerite. Monazite grains are usually around 1/4 mm in diameter.

DETAILS OF TEST WORK

Lapointe Picker Tests

Material from 8 of the 15 drums was crushed to minus 10 inch and sorted into minus 10 plus 6 inch and minus 6 plus 4 inch fractions for picker tests. The minus 4 inch fraction was untreated.

In carrying out the electronic sorting tests, the sensitivity of the equipment was adjusted so that the rejected material represented 25 - 30 % of the total ore sample (i. e. including the untreated minus 4 inch fraction).

The results of this work are given in Table 3.

TABLE 3

Overall Metallurgy for Lapointe Picker Tests

Products	Wt (%)	U_3O_8 Assay (%)	U_3O_8 Dist. (%)
-10 + 6 in. Concentrate	27.6	0.14	39.8
- 6 + 4 in. Concentrate	13.6	0.12	16.8
- 4 in. (Untreated)	32.0	0.11	36.2
Preconcentrate	73.2	0.123	92.8
-10 + 6 in. Waste	22.1	0.03	6.8
- 6 + 4 in. Waste	4.7	0.008	0.4
Waste	26.8	0.026	7.2
Original Ore	100.0	0.097	100.0

Sink-Float Concentration Tests

From the seven drums of ore not used for the Lapointe Picker tests, one drum was selected at random, and crushed to minus 6 inches. The minus 2 inch material was screened out and was set aside. The minus 6 plus 2 inch material was treated by static sink-float methods using heavy liquids with specific gravities of 2.65, 2.66, 2.68 and 2.71. Acetylene tetrabromide, adjusted to the required specific gravity with carbon tetrachloride, was used as the heavy medium in this work.

The results of these tests are given in Table 4 and are also shown in graphic form in Figure 1.

In order to obtain further data on the sink-float characteristics of the ore, the picker test products were crushed to minus 1 inch and a sample was riffled out for sink-float tests. It was screened on a 4 mesh screen and the minus 1 inch plus 4 mesh fraction was treated at specific gravities of 2.65, 2.67 and 2.71. The minus 4 mesh fraction was not treated.

The results of this test work are shown in Table 5 and Figure 1.

Gravity Concentration by Tabling

A sample was crushed to minus 20 mesh and tabled on a laboratory Wilfley table. The table feed was deslimed, but not sized, prior to tabling.

The results are given in Table 6.

TABLE 4

Sink-float Results after Crushing to Minus 6 Inches

Products and sp gr of Separation	Wt (%)	U_3O_8 Assay (%)	U_3O_8 Dist. (%)
2.65 sp gr			
-6 + 2 in. Float	6.17	0.015	1.3
-6 + 2 in. Sink	69.95	0.0613	59.2
- 2 in. (Untreated)	23.88	0.12	39.5
Original Ore	100.00	0.073	100.0
Preconcentrate*	93.83	0.076	98.7
2.66 sp gr			
-6 + 2 in. Float	34.32	0.014	6.8
-6 + 2 in. Sink	41.80	0.093	53.7
- 2 in. (Untreated)	23.88	0.12	39.5
Original Ore	100.00	0.073	100.0
Preconcentrate	65.68	0.103	93.2
2.68 sp gr			
-6 + 2 in. Float	25.94	0.025	7.1
-6 + 2 in. Sink	39.72	0.12	51.8
- 2 in. (Untreated)	34.34	0.11	41.1
Original Ore**	100.00	0.092	100.0
Preconcentrate	74.06	0.115	92.9
2.71 sp gr			
-6 + 2 in. Float	55.97	0.024	18.8
-6 + 2 in. Sink	20.15	0.15	41.7
- 2 in. (Untreated)	23.88	0.12	39.5
Original Ore	100.00	0.073	100.0
Preconcentrate	44.03	0.134	81.2

* The preconcentrate, in each case, is the sink and -2 in. (untreated) fractions combined.

**Another drum of ore was treated in this test.

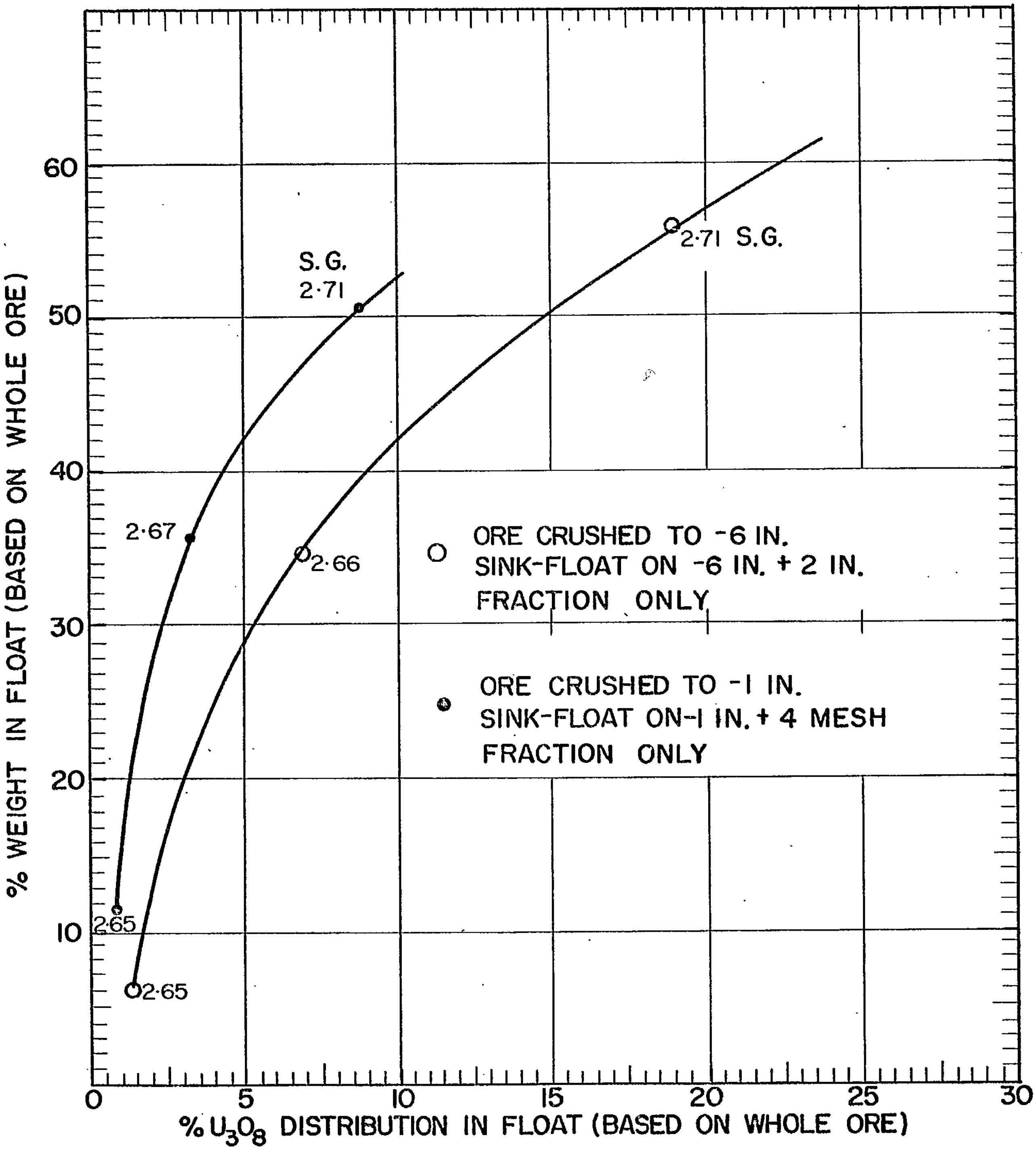

FIGURE 1

% U_3O_8 DISTRIBUTION BY SINK-FLOAT METHOD

TABLE 5

Sink-float Results after Crushing to Minus 1 Inch

Products and Specific Gravity of Separation	Wt (%)	U_3O_8 Assay (%)	U_3O_8 Dist. (%)
2.65 sp gr			
-1 in. + 4 mesh Float	11.88	0.006	0.8
-1 in. + 4 mesh Sink	60.62	0.096	61.3
-4 mesh (Untreated)	27.50	0.13	37.9
Original Ore	100.00	0.095	100.0
Preconcentrate*	88.12	0.106	99.2
2.67 sp gr			
-1 in. + 4 mesh Float	35.62	0.0087	3.3
-1 in. + 4 mesh Sink	36.88	0.151	58.8
-4 mesh (Untreated)	27.50	0.13	37.9
Original Ore	100.00	0.095	100.0
Preconcentrate	64.38	0.142	96.7
2.71 sp gr			
-1 in. + 4 mesh Float	50.58	0.016	8.8
-1 in. + 4 mesh Sink	21.92	0.23	53.3
-4 mesh (Untreated)	27.50	0.13	37.9
Original Ore	100.00	0.095	100.0
Preconcentrate	49.42	0.174	91.2

*The preconcentrate, in each case, is the sink and minus 4 mesh (untreated) fractions combined.

TABLE 6

Overall Metallurgical Results of Tabling Test

Products	Wt (%)	U_3O_8 Assay (%)	U_3O_8 Dist. (%)
Concentrate	6.42	0.32	19.5
Middlings	25.25	0.16	38.4
1st Tailing	46.28	0.052	22.9
2nd Tailing	13.31	0.098	12.4
Slimes	8.74	0.082	6.8
Original Ore	100.00	0.11	100.0

Acid Leaching of Feed and Preconcentration Products

A number of sulphuric acid leach tests were carried out on the whole ore and preconcentration products from sink-float tests. Sodium chlorate was used as the oxidant.

Sample A preconcentrate represented the sink product, at sp gr 2.67, of the minus 1 inch plus 4 mesh size fraction of the feed along with the minus 4 mesh untreated feed (Table 5). This combined product amounted to 64.38% of the total weight and contained 96.7% of the total uranium.

Sample B preconcentrate was made up of the sink product, at sp gr 2.71, of the minus 1 inch plus 4 mesh size fraction of the feed and the minus 4 mesh untreated feed (Table 5). This combined product amounted to 49.42% of the total feed weight and contained 91.2% of the total uranium.

The ore sample for each test was ground for 25 minutes in an Abbé porcelain laboratory ball mill with steel balls. Oxidant addition and acid level were varied in the tests. The leach conditions and results are listed in Table 7.

A higher leach residue resulted when the oxidant was omitted. In general, higher acid content, and increased oxidant concentration gave improved extraction. Preconcentrate A produced a saving in acid of about 30 lb/ton original ore for about the same extraction as the direct leaching of the whole ore.

TABLE 7

Leach Test Conditions and Results on Preconcentration Products

Sample	Composite Feed				Preconcentrate "A"					Preconcentrate "B"		Tailings "A"
Test No.	A	B	F	H	C	D	E	J	K	L	M	G
Grind, % -200 mesh	70	-	70	56	71	-	69	77	70	72	73	68
% wt of original ore treated	100	100	100	100	64.4	64.4	64.4	64.4	64.4	49.4	49.4	35.6
Sodium chlorate added, lb/ton original ore	2.5	nil	3.2	3.2	2.1	2.1	3.2	2.1	2.1	1.6	1.6	1.1
Acid addition, lb 100% H_2SO_4/ton												
original ore	140	100	90	95	64	80	61	61	95	47	95	28
leach feed	140	100	90	95	100	125	95	95	148	95	192	80
Residue assay, % U_3O_8 at:												
0 hr	0.10	-	0.10	0.11	-	-	-	0.16	0.15	0.18	0.19	0.011
24 hr	0.012	0.027	0.010	0.009	0.012	0.011	0.009	0.014	0.010	0.019	0.012	0.002
48 hr	0.004	0.010	0.007	0.005	0.008	0.006	0.007	0.010	0.006	0.012	0.008	0.001
Leach Solution												
Volume, ml	550	415	555	555	520	625	465	585	650	680	627	490
U_3O_8, g/l	1.13	1.16	1.21	1.12	1.87	1.52	2.12	1.91	1.76	2.31	2.03	0.13
U_3O_8, % Dist.	67.0	52.8	75.0	72.9	63.2	70.6	65.7	76.7	80.0	79.6	80.1	72.1
Reducing Power,* g/l	2.08	4.61	2.08	2.40	3.31	2.90	4.02	3.36	2.97	3.63	3.47	0.38
Total Fe, g/l	4.74	5.08	4.88	5.52	5.47	5.20	7.04	6.25	5.90	6.00	6.60	0.85
Free acid, g/l	70.2	51.6	46.0	48.0	46.0	54.4	43.8	44.4	75.0	38.0	100.0	51.5
Wash Solution												
Volume, ml	765	790	710	705	800	750	770	715	617	750	680	750
U_3O_8, g/l	0.35	0.42	0.22	0.26	0.61	0.45	0.58	0.34	0.37	0.38	0.35	0.02
U_3O_8, % Dist.	28.8	36.4	17.4	21.5	31.7	25.0	29.7	16.7	16.0	14.5	15.0	17.0
Reducing Power,* g/l	0.63	1.63	0.43	tr	1.08	0.78	1.12	-	-	-	-	0.12
Total Fe, g/l	1.32	1.77	0.98	-	1.81	1.30	1.90	-	-	-	-	0.38
Final Residue												
Weight, g	975	980	970	950	970	981	976	962	960	972	968	960
U_3O_8, %	0.004	0.010	0.007	0.005	0.008	0.006	0.007	0.010	0.006	0.012	0.008	0.001
U_3O_8, % Dist.	4.2	10.8	7.6	5.6	5.0	4.4	4.6	6.6	4.0	5.9	4.9	10.9
Calc. leach feed assay, % U_3O_8	0.095	0.093	0.092	0.089	0.16	0.14	0.15	0.15	0.15	0.20	0.16	0.0092
Extraction, %, based on calc. leach feed assay	95.8	89.2	92.4	94.4	95.0	95.6	95.4	93.4	96.0	94.1	95.1	89.1
U_3O_8 content, % of original ore	100	100	100	100	96.7	96.7	96.7	96.7	96.7	91.2	91.2	3.3
Extraction, %, based on original ore	95.8	89.2	92.4	94.4	91.9	92.4	92.3	90.3	92.8	85.8	86.7	2.9

Note: The leach temperature in all tests was controlled at 45°C.

* Titrated by standard potassium dichromate and reported as ferrous iron.

Flotation Test Work

Several flotation tests were carried out. Most of the tests were done on material ground for 20 minutes in an Abbé mill with a charge of steel balls. This grinding time gave a product of 51.3% minus 200 mesh, which is fairly close to the grind obtained in the operating mill.

A test, run without desliming, gave rather indifferent results, so all subsequent tests were run on deslimed material. The desliming was carried out by adding 1.0 lb NaOH and 0.5 lb Na_2SiO_3/ton of ore and then allowing the slurry to settle for 10 minutes before decanting the slimes. The operation was then repeated without the addition of more reagent.

The conditions and results of some of the better tests are given in the following pages.

Flotation Test No. 1

Reagents Added: lb/ton

NaOH	1.0
Na_2SiO_3	0.5
Deslimed	
Sulphonated whale oil	0.5
Linseed fatty acid	0.5
Diesel fuel oil	0.5
Cresylic acid	0.05

Conditioned - 4 minutes - (pH - 6.8)
Rougher float - 5 minutes
Cleaned rougher float - 4 to 5 minutes

Results, Flotation Test No. 1

Products	Wt (%)	U_3O_8 Assay (%)	U_3O_8 Dist. (%)
Slimes	12.63	0.10	11.1
Cleaner Float	27.72	0.32	78.4
Cleaner Tailing	21.58	0.016	3.1
Rougher Tailing	38.07	0.022	7.4
Original Ore	100.00	0.113	100.0

Note: Combining the slimes and cleaner float results in 89.5% recovery in 40.4% of the weight at a grade of 0.25% U_3O_8.

Flotation Test No. 2

Reagents Added:	lb/ ton
NaOH	1.0
Na_2SiO_3	0.5
Deslimed	
Na_2SiO_3	1.0
Acintol FA-2	1.5
Conditioned - 3 minutes - (pH - 7.9)	
Rougher float - 4 minutes	
Acintol FA-2	0.5
Conditioned - 3 minutes	
Scavenger float - 2 to 3 minutes	

The rougher float and scavenger float were combined and cleaned for 4 minutes.

Results, Flotation Test No. 2

Products	Wt (%)	U_3O_8 Assay (%)	U_3O_8 Dist. (%)
Slimes	14.16	0.092	15.0
Cleaner Float	21.06	0.27	65.9
Cleaner Tailing	32.74	0.034	12.8
Rougher Tailing	32.04	0.017	6.3
Original Ore	100.00	0.086	100.0

Note: By eliminating the rougher tailing, 93.7% recovery is obtained in 67.9% of the weight at a grade of 0.12% U_3O_8.

Flotation Test No. 3

Reagents Added:	lb/ton
NaOH	1.0
Na_2SiO_3	0.5
Deslimed	
Na_2SiO_3	1.0
Acintol FA-1	0.5
Conditioned - 3 minutes - (pH - 8.2)	
Rougher float - 3 minutes	
Acintol FA-1	0.5
Conditioned - 3 minutes	
1st Scavenger float - 4 minutes	
Acintol FA-1	0.5
Cresylic acid	0.05
Conditioned - 3 minutes	
2nd Scavenger float - 4 to 5 minutes	

Results, Flotation Test No. 3

Products	Wt (%)	U_3O_8 Assay (%)	U_3O_8 Dist. (%)
Slimes	13.24	0.091	12.3
Rougher Float	7.94	0.27	21.9
1st Scavenger Float	7.15	0.60	44.0
2nd Scavenger Float	16.77	0.068	11.7
Rougher Tailing	54.90	0.018	10.1
Original Ore	100.00	0.098	100.0

Note: By eliminating the rougher tailing 89.9% recovery is obtained in 45.1% of the weight at a grade of 0.19 % U_3O_8.

DISCUSSION

From the test work carried out on the sample submitted, it appears that sink-float would be the best preconcentration procedure to employ. On minus 1 inch material it was particularly effective. As well as effecting a saving of about 30 lb acid/ ton original ore, the leach plant throughput could be reduced by about 35 per cent and still process the same original tonnage, or conversely the leach plant capacity could be increased by 35 per cent.

REFERENCE

Hughson, M.R., Mineralogical Report on Uranium Ore from Northspan Uranium Mines, Ltd., Lake Nordic Property, Blind River Area, Ontario. Mines Branch Investigation Report IR 58-134, Department of Mines and Technical Surveys, Ottawa, Canada, August 12, 1958.

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