DEPARTMENT OF MINES.

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-RESEARCH AND DEVELOPMENT BRANCH-

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CROCKER'S WELL ORE - FLOTATION INVESTIGATIONS.

PART 1:

Removal of acid-consuming minerals from flotation concentrates produced from high grade ore.

PART 2:

Flotation of low grade ore.

by

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CROCKER'S WELL ORE - FLOTATION INVESTIGATIONS.

PART 1 - Removal of acid-consuming mineral from flotation concentrate produced from high-grade ore.

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- Abstract -

Various types of flotation treatment were investigated with a view to improving the concentrates from Crocker's Well to make them more suitable for leaching.

1. SUMMARY.

Leaching tests on Crocker's Well flotation concentrates have shown that:

(a) The consumption of acid is high because of large amounts of mica minerals and apatite in the concentrates.

(b) The excessive oiliness of the concentrates interferes with the normal leaching process.

Laboratory tests show that most of the mica mineral can be removed by flotation in acid circuit prior to flotation of the uranium minerals. Small losses of uranium occur in the mica fraction but this is compensated by an appreciable increase in the grade of the final concentrates and a considerable reduction in acid consumption during leaching.

The inclusion of two new surface active agents in the normal four-component reagent mixture known as the Radium Hill mixture reduces the amount of oil retained by the concentrates.

Attempts to remove the phosphate mineral, apatite, were not successful.

2. MATERIAL EXAMINED.

The following samples were used for testing:

(1) Ore obtained from shaft sinking in November 1956 and received as "Lot No. 25" assaying 12.7 lbs U_3O_8 per ton.

(2) Ore assaying 6 - 7 lbs U_3O_8 per ton prepared by diluting "Lot No.25" ore with low grade country rock.

3. EQUIPMENT.

The equipment used in the tests included:

Fagergren laboratory batch flotation cell - 500 gram capacity.

Laboratory steel ball mill - 500 gram capacity.

4. EXPERIMENTAL PROCEDURE and RESULTS.

In the following tests the four component or Radium Hill mixture consisted of:

1.5 lb. Peltogen/ton.
2.0 lb. Linseed fatty acid/ton.
6.0 lb. Fuel Oil/ton.
0.5 lb. Cresylic acid/ton.

4.1 Apatite separation.

Tests were carried out on ore assaying 12 lbs. U_3O_8 per ton to float apatite preferentially from the uranium minerals using caustic starch and oleic acid as the major reagents.

Three tests were carried out in the following manner:

1. 500 grams of the ore ground to 50 percent minus 200 mesh (Tyler series) in ball mill at 60 percent solids.
2. Pulp transferred to float cell and deslimed using 1.0 lb NaOH/ton and 0.2 lb Na_2SiO_3/ton as gangue dispersants.
3. Pulp was then conditioned in float cell for 10 minutes with 1.0, 2.0 and 3.0 lb. caustic starch/ton for the three tests.
4. Oleic acid equivalent to 2.0 lbs per ton of ore was added in four equal stages to each of the three tests.

Results of this series are shown in Table 1.

TABLE 1.

Apatite Separation Test 1-3.

Product.	Percent Weight.	U_3O_8 lb/ton.	P_2O_5 Percent.	U_3O_8 Dist. Percent.	P_2O_5 Dist. Percent.
Test 1: 1.0 lb. starch.					
Slime.	7.6	14.1	0.53	7.9	9.4
Apatite Conc.	3.1	6.0	0.75	1.4	5.4
Tailings.	89.3	13.7	0.41	90.7	85.2
FEED.	100.0	13.5	0.43	100.0	100.0
Test 2: 2.0 lb. starch.					
Slime.	10.2	14.6	0.54	11.9	12.5
Apatite Conc.	1.6	5.4	0.21	0.7	0.8
Tailing.	88.2	12.4	0.43	87.4	86.7
FEED.	100.0	12.5	0.44	100.0	100.0
Test 3: 3.0 lb. starch.					
Slime.	9.3	13.4	0.53	10.0	13.2
Apatite.	1.3	7.3	0.18	0.8	0.6
Tailing.	89.4	12.4	0.36	89.2	86.2
FEED.	100.0	12.4	0.37	100.0	100.0

Test No.4

This test was carried out under the following conditions:

1. 500 gram ore ground and deslimed as in previous tests.
2. Pulp conditioned in an attrition conditioner for 20 minutes at 50 percent solids with 5.0 lb. of Radium Hill reagent/ton plus 0.25 lb of ethylene oxide condensate per ton. The latter chemical produces concentrates which are less oily than concentrates produced by using the Radium Hill reagent alone.

3. The rougher concentrate was cleaned once and conditioned in float cell with 3.0 lb. caustic starch/ton for 10 minutes.

4. Oleic acid was added in stages of 0.5 lb/ton up to a total of 2.0 lb/ton. The amount floating was very small and was consequently bulked as "apatite concentrate". The mineral not floated in this step was cleaned as "absite concentrate".

The results of Test 4 are shown in Table 2.

TABLE 2.

Apatite Separation - Test No. 4.

Product.	Weight Percent.	U_3O_8 lb/ton.	P205 Percent.	U_3O_8 Dist. Percent.	P_2O_5 Dist. Percent.
Slime.	8.2	14.1	0.55	10.2	10.9
Apatite conc.	2.5	156.8	7.60	34.6	46.3
Absite conc.	3.7	161.3	2.65	52.6	23.8
1st cleaner tail.	11.8	1.23	0.10	1.3	2.9
Rougher tail.	73.8	0.2	0.09	1.3	16.1
FEED.	100.0	11.4	0.41	100.0	100.0

Test No.5.

In this test the ore was ground and deslimed as in previous tests. Pulp was conditioned with 1 lb. caustic starch/ton for 10 minutes in the float cell. Oleic acid added (2.0 lb/ton) and an apatite concentrate removed. Tailing from this step was conditioned at 50 percent solids in an attrition conditioner for 20 minutes with 5 lb. Radium Hill reagent/ton, plus 0.25 lb. ethylene oxide condensate per ton. The rougher float was cleaned once.

Results of Test No.5 are shown in Table 3.

TABLE 3.

Apatite Separation - Test No. 5.

Product.	Weight Percent.	U_3O_8 lb/ton.	P_2O_5 Percent.	U_3O_8 Distrib. Percent.	P_2O_5 Distrib. Percent.
Slime.	8.0	12.8	0.57	7.2	10.4
Apatite conc.	2.4	9.3	0.30	1.6	1.7
Absite conc.	10.0	125.0	3.43	87.4	78.4
1st cleaner tail.	20.7	1.37	0.03	1.9	1.4
Rougher tail.	58.9	0.47	0.06	1.9	8.1
FEED.	100.0	14.3	0.44	100.0	100.0

4.2 Separation of the Mica minerals.

Preliminary laboratory tests showed that mica minerals in acid circuit and on deslimed feed responded to flotation with ARMAC 12D (Laurylamine acetate) and the following five tests were carried out to confirm results.

Test No. 1.

600 grams of ore were ground and deslimed as in the tests aimed at apatite removal.

The sands were conditioned for 1 minute with 5 ml of 10 percent H_2SO_4 after which the "ARMAC 12D" was added as a 1 percent solution in stages of 1 ml, using a total of 2.5 ml and the mica float taken off immediately after each addition.

The tailing from the mica float was filtered and the solids conditioned for 15 minutes with:

0.75 lb. Peltogen.
1.00 lb. Linseed fatty acids.
3.00 lb. Fuel oil.
0.25 lb. Naphthenic acid.
0.11 lb. Sorbitan monoleate.
0.14 lb. Ethylene oxide condensate,

per ton of ore. The reagents were mixed before being added to the circuit.

The uranium minerals were floated and the rougher concentrate cleaned twice. Flocculation occurred in the rougher flotation stage and a lot of quartz floated. This was remedied by adding 5 ml of 10 percent H_2SO_4 to the first cleaner, which effectively dispersed the "flocs" and dropped out the quartz.

The results of Test No.1 are shown in Table 4.

TABLE 4.

Mica Separation - Test No. 1.

Product.	Weight Percent.	U_3O_8 lb/ton.	P_2O_5 Percent.	U_3O_8 Distrib. Percent.	P_2O_5 Distrib. Percent.
Slime.	8.7	14.0	0.46	9.2	12.0
Mica float.	12.0	5.9	0.27	5.4	10.0
Recleaner conc.	3.0	300.0	5.77	68.1	54.0
" tail.	4.0	26.2	0.54	7.9	6.5
Cleaner tail.	30.3	3.6	0.17	8.2	15.5
Rougher tail.	42.0	0.34	0.02	1.2	2.0
FEED.	100.0	13.2	0.34	100.0	100.0

Test No.2 and 3.

Two further tests were carried out under similar conditions, the only change being in point of addition of the acid to disperse flocs in uranium flotations, viz:

Test No.2 5 ml. 10% H_2SO_4 added to second cleaner.
Test No.3 " " " " "conditioner.

Results of Test Nos. 2 and3 are shown in Tables 5 and 6 respectively.

TABLE 5.
Mica Separation - Test No.2.

Product.	Weight Percent.	U_3O_8 lb/ton.	P_2O_5 Percent.	U_3O_8 Dist. Percent.	P_2O_5 Dist. Percent.
Slime fraction.	9.2	13.3	0.49	9.3	13.7
Mica float.	13.0	5.3	0.36	5.3	14.3
Recleaner conc.	3.0	290.0	5.51	66.4	50.0
" tail.	7.7	28.2	0.86	16.6	20.0
Cleaner tail.	5.1	3.8	0.13	1.4	1.0
Rougher tail.	62.0	0.2	0.1	1.0	1.0
FEED.	100.0	13.1	0.33	100.0	100.0

TABLE 6.

Mica Separation - Test No. 3.

Product.	Weight Percent.	U_3O_8 lb/ton.	P_2O_5 Percent.	U_3O_8 Dist. Percent.	P_2O_5 Dist. Percent.
Slime fraction.	8.2	11.5	0.43	7.8	13.6
Mica float.	6.4	7.9	0.31	4.2	7.8
Recleaner conc.	3.2	310.0	5.76	82.7	71.2
" tail. } Cleaner tail. }	2.7	10.4	0.41	2.3	4.3
Rougher tail.	79.5	0.45	0.01	3.0	3.1
FEED.	100.0	12.0	0.26	100.0	100.0

Test No.4

A further test was made using 6 lb. U_3O_8 feed was follows:

4 x 500 gram charges were ground and deslimed as before.

The deslimed sands were conditioned with 20 ml. of 10 percent H_2SO_4 and the mica was floated using 0.2 lb/ton "ARMAC 12D" as in previous tests.

The tailing from the mica float was filtered and washed once to counteract effects of residual amine from mica stage, then conditioned for 15 minutes with the reagent mixture for uranium flotation and 5 ml. 10 percent H_2SO_4.

The rougher concentrate was cleaned 3 times with additions of 5 ml. of 10 percent H_2SO_4 to the first and second cleaners respectively.

Results of Test No.4 are shown in Table 7.

TABLE 7.

Mica Separation - Test No. 4.

Product.	Weight Percent.	U_3O_8 lb/ton.	P_2O_5 Percent.	U_3O_8 Dist. Percent.	P_2O_5 Distrib. Percent.
Slime fraction.	9.5	6.6	0.12	8.8	3.7
Mica float.	6.2	2.8	0.27	2.5	5.5
Concentrate.	2.6	215.0	8.9	78.6	76.8
3rd cleaner tail.	0.8	21.3	0.96	2.4	2.5
2nd cleaner tail.	3.5	8.3	0.34	4.0	4.3
1st cleaner tail.	14.2	1.1	0.07	2.1	3.0
Rougher tail.	63.2	0.18	0.02	1.6	4.2
FEED.	100.0	7.1	0.3	100.0	100.0

Test No. 5.

In order to demonstrate the effect of preliminary removal of mica, a further test was carried out as far as possible in conditions similar to Test No.4 but the mica was not removed prior to flotation of the uranium minerals.

The procedure adopted was:

4 x 500 gram charges of ore were ground and deslimed as in Test No. 4.

The deslimed sands were conditioned with the uranium reagent mixture and 5 cc. 10 percent H_2SO_4 for 15 minutes.

The rougher concentrate was cleaned three times with additions of 5 cc. 10 percent H_2SO_4 to the first and second cleaners.

The results of Test No.5 are shown in Table 8.

TABLE 8.

Mica Separation - Test No. 5.

Product.	Weight Percent.	U_3O_8 lb/ton.	P_2O_5 Percent.	U_3O_8 Dist. Percent.	P_2O_5 Dist. Percent.
Slime fraction.	6.4	6.5	0.36	5.6	8.5
Concentrate.	8.1	79.3	2.9	89.3	87.8
3rd cleaner tail.	1.3	4.4	0.18	0.9	0.7
2nd cleaner tail.	5.0	2.2	0.04	1.5	0.7
1st cleaner tail.	8.5	0.52	0.01	0.6	0.0
Rougher tail.	70.7	0.22	0.01	2.1	2.3
FEED.	100.0	7.2	0.27	100.0	100.0

5. OBSERVATIONS and CONCLUSIONS - PART 1.

The separation of apatite from the uranium minerals will obviously require more selective reagents, as the tests results indicate a similarity in flotation behaviour between the phosphate and uranium minerals.

On the other hand separation of the mica minerals is fairly complete and it may be possible to reduce the loss of uranium in the mica fraction by cleaning or floating at lower pH. Most of the uranium in the slime fraction would probably be recovered by recombining this fraction with the tailing from mica flotation. This may result in a lowering of grade of the final concentrates but more work would have to be done along these lines.

Leach tests on a sample of "mica-free" concentrates produced in the laboratory were reported as being satisfactory and acid consumption was considerably reduced.

The inclusion of two surface-active agents, namely, sorbitan mono-oleate and "Nonion P100% to the Radium Hill reagent mixture produces concentrates which are much less oily in nature than concentrates obtained using the normal four-component Radium Hill mixture.

PART II - Flotation of Low Grade Ore.

-Abstract-

Flotation tests carried out on ore assaying 0.75 lb. U_3O_8 per ton indicate that recovery of approximately eighty percent of the uranium with concentrate grades of 20 - 25 lb. U_3O_8 per ton should be possible.

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1. SUMMARY.

Laboratory and pilot plant tests were carried out using ore from a number of shaft samples received at Thebarton over the last twelve months. These parcels of ore were derived from the Mining Branch investigations and varied in grade from 0.1 to 2.0 lb. U_3O_8/ton.

Flotation tests show that the grade of concentrate obtained in laboratory cyclic batch tests falls considerably as the middling fraction tends to report with the concentrate.

2. MATERIAL EXAMINED.

Six samples of ore were tested, five of which were selected from the shaft sample lots having a U_3O_8 content ranging from 0.25 lb to 2.0 lb. per ton. These five samples were representative of Lot Nos. 1, 5, 8, 9 and 10.

The sixth sample was a composite blended from all shaft sample lots received and this sample assayed 0.75 lb U_3O_8/ton. This was considered by the Mining Branch to be the average grade of ore from the section being investigated.

3. EQUIPMENT USED.

The equipment used in the tests was as follows:

1. Laboratory batch ball mill 500 grams capacity.
2. Laboratory Fagergren flotation cells.
3. Pilot Plant equipment consists of ball mill, classifier twelve Agitair flotation cells and ancillary equipment such as pumps, feeders, etc.

4. EXPERIMENTAL PROCEDURE and RESULTS.

4.1 Flotation tests on samples of varying U_3O_8 content.

Five samples were selected at U_3O_8 contents of 0.25, 0.5, 1.0, 1.5 and 2.0 lb/ton and laboratory tests carried out to determine the probable concentrate grade for each ore type.

The results of these tests are shown in Table 9.

The following conditions were standard for all tests.

1. Grinding.

   50 percent minus 200 mesh (Tyler series).

2. Flotation times.

   Roughing - 10 minutes.
   1st cleaning - 6 minutes.
   2nd " - 3 minutes.
   3rd " - 1½ minutes.

3. Pulp Density Rougher float.

   34 percent solids.

4. Conditioning with reagents in float cell for 15 minutes.

5. Reagents.

   5.0 lb of the following mixture per ton:

Peltogen.	0.6 part.
Linseed fatty acids.	0.8 "
Fuel Oil.	3.5 "
Naphthenic acid.	0.5 "
Sorbitan mono-oleate (S.M.O.)	0.09 "
Octylphenol oxyethylene condensate (P100).	0.23 "

TABLE 9.

Batch Flotation Tests on Ore Samples of Varying Grade.

Test	Feed Grade lb. U_3O_8/ton	Products.	Weight Percent	U_3O_8 lb/ton.	U_3O_8 Distrib. Percent.
1.	0.25	Concentrate.	1.5	5.4	42.7
		3rd cleaner tail.) 2nd cleaner tail.)	5.5	1.3	36.0
		1st cleaner tail.	10.5	0.2	12.4
		Rougher tail.	82.5	0.02	8.9
		FEED.	100.0	0.20	100.0
2.	0.5	Concentrate.	1.6	17.2	53.4
		3rd cleaner tail.) 2nd cleaner tail.)	5.6	2.7	28.6
		1st cleaner tail.	16.9	0.47	15.1
		Rougher tail.	75.9	0.02	2.9
		FEED	100.0	0.52	100.0
3.	1.0	Concentrate.	1.2	58.2	73.5
		3rd cleaner tail.) 2nd cleaner tail.)	2.8	4.7	14.1
		1st cleaner tail.	16.3	0.4	6.7
		Rougher tail.	79.7	0.07	5.7
		FEED.	100.0	0.95	100.0
4.	1.5	Concentrate.	1.4	70.8	76.3
		3rd cleaner tail.) 2nd cleaner tail.)	4.6	4.4	15.2
		1st cleaner tail.	15.0	0.63	7.3
		Rougher tail.	79.0	0.02	1.2
		FEED.	100.0	1.30	100.0
5.	2.0	Concentrate.	1.4	105.3	70.4
		3rd cleaner tail.) 2nd cleaner tail.)	4.0	6.9	13.5
		1st cleaner tail.	18.3	1.16	10.2
		Rougher tail.	76.3	0.16	5.9
		FEED.	100.0	2.08	100.0

The concentrates from each of the five tests were submitted for mineralogical examination and results are reported in Table 10.

TABLE 10.

Mineralogical analyses of flotation concentrates from Tests 1 to 5 inclusive.

Minerals.	Test 1. Weight Percent.	Test 2. Weight Percent.	Test 3. Weight Percent.	Test 4. Weight Percent.	Test 5. Weight Percent.
Opaque Brannerite.	0.75	3.2	11.6	15.4	20.4
Non-opaque "	0.75	1.2	6.0	7.0	9.0
Apatite.	29.7	62.5	52.0	33.6	33.6
Rutile.	14.4	14.8	18.3	27.6	18.3
Quartz + Felspar.	11.3	5.5	2.1	6.8	4.7
Ferruginous clay.	17.4	-	-	-	-
Micas, chlorite.	10.8	4.8	tr.	tr.	tr.
Fluorite, zircon.	tr.	tr.	tr.	tr.	tr.
Monazite, orthite etc.	tr.	tr.	tr.	tr.	tr.
Hematite, limonite.	6.2	2.8	5.5	5.2	5.0
Sulphides.	3.9	3.7	4.5	4.4	6.4
Magnetite.	4.8	1.5	tr.	tr.	2.6
Ilmenite.	tr.	tr.	tr.	tr.	tr.
FEED.	100.0	100.0	100.0	100.0	100.0

The above concentrates differed somewhat from the earlier Crocker's Well concentrates in having mostly the black opaque form of brannerite present and not the distinctive yellow form of thorium-brannerite (Absite) usually visible. A study of the mineralogical analyses shows marked decrease in the micaceous minerals as the grade of concentrate improves. In consequence of this, experimental work on the separation of mica minerals from the concentrates proposed in Part 1 of this report, was not necessary for this ore.

4.2 Pilot Plant test - 2.0 lbs U_3O_8/ton feed.

Approximately forty pounds of concentrates were produced in the 100 lb/hour pilot plant for leaching tests by the Chemical Research Section.

The shaft sample (Lot No.10) of 2.0 lbs U_3O_8/ton grade was used as feed.

Reagent quantity and type was as used in the laboratory tests.

The concentrate was given two cleanings and the final product assayed 12.5 lb. U_3O_8/ton for a recovery of 93.0 percent of the uranium.

A sample of the Pilot Plant concentrate was submitted for mineralogical examination and the result is reported in Table 11.

TABLE 11.

Mineralogical Analysis of Concentrate. Produced in Pilot Plant Tests.

Mineral.	Percent Weight.
Opaque (Magnetite, hematite brannerite).	16.7
Non-opaque brannerite.	3.3
Apatite.	4.2
Rutile.	7.0
Micas, chlorite.	13.5
Quartz, felspar.	47.5
Clay material.	7.8
Fluorite.	tr.
Zircon.	tr.
Tourmaline.	tr.
	100.0

NOTE:

An attempt to eliminate the quartz-felspar gangue from the concentrate by recleaning a portion in the laboratory float cell was not successful.

4.3 Flotation tests on ore of fixed grade (0.75 lb. U_3O_8/ton).

In September, 1957 advice was received to the effect that the probable grade of the Crocker's Well deposit would be in the order of 0.75 to 1.0 lb U_3O_8/ton.

A representative sample of approximately six tons was blended from the shaft sample lots and this sample was used for all further laboratory and pilot plant tests. The sample assayed 0.75 lb U_3O_8/ton.

4.3.1 Test Procedure.

Tests were carried out using 5.0, 4.5, 4.0, 3.5, 3.0 and 2.5 lb. of reagent per ton. All other conditions of tests were similar to those used for the first series. Test results are shown in Table 12.

TABLE 12.

Batch Flotation Tests on 0.75 lb. U_3O_8 feed with Varying amounts of reagent.

Test.	Reagent lb/ton.	Products.	Weight Percent.	U_3O_8 lb/ton.	U_3O_8 Distrib. Percent.
1.	5.0	Concentrate.	2.8	24.4	85.2
		Recleaner tail.	3.5	0.69	3.0
		Cleaner tail.	15.5	0.40	7.4
		Rougher tail.	78.2	0.05	4.4
		HEADS.	100.0	0.80	100.0
2.	4.5	Concentrate.	2.0	31.4	82.0
		Recleaner tail.	2.0	1.66	4.3
		Cleaner tail.	11.8	0.6	9.2
		Rougher tail.	84.2	0.04	4.5
		HEADS.	100.0	0.77	100.0
3.	4.0	Concentrate.	2.1	23.5	73.0
		Recleaner tail.	1.9	3.4	9.0
		Cleaner tail.	7.3	0.47	5.0
		Rougher tail.	88.7	0.11	13.0
		HEADS.	100.0	0.68	100.0
4.	3.5	Concentrate.	1.7	29.8	69.8
		Recleaner tail.	2.3	2.0	6.3
		Cleaner tail.	10.0	1.14	15.6
		Rougher tail.	86.0	0.07	8.3
		HEADS.	100.0	0.73	100.0
5.	3.0	Concentrate.	1.8	24.6	63.0
		Recleaner tail.	1.8	4.9	12.5
		Cleaner tail.	9.0	0.67	8.4
		Rougher tail.	87.4	0.13	16.2
		HEADS.	100.0	0.70	100.0
6.	2.5	Concentrate.	1.4	28.2	47.6
		Recleaner tail.	1.8	4.9	10.6
		Cleaner tail.	8.0	1.6	15.4
		Rougher tail.	88.8	0.25	26.4
		HEADS.	100.0	0.83	100.0

As the results of the tests shown in Table 12 showed that 4.5 lb. of reagent per ton was the minimum amount required to give optimum results, a laboratory cyclic test was carried out using this amount of reagent. The result of this test is shown in Table 13.

TABLE 13.

Cyclic Test Using 4.5 lb/ton of Reagents.

Product.	Weight Percent.	U_3O_8 lb/ton.	U_3O_8 Distrib. Percent.
Concentrate.	7.7	8.4	93.0
Recleaner tail.	1.9	0.25	0.7
Cleaner tail.	3.0	0.34	1.3
Rougher tail.	87.4	0.04	5.0
HEADS.	100.0	0.70	100.0

It will be observed from these results that when the middlings products are recycled they tend to report with the concentrate, this increasing recovery but substantially reducing the grade. In plant operation however, it should be possible to produce a higher grade concentrate, similar to batch tests, at the expense of recovery.

4.3.2 Pilot Plant Test 0.75 lb. U_3O_8/ton feed.

Approximately 50 lb. of concentrates were produced in the 100 lb/hour pilot plant under similar conditions to those used in the laboratory cyclic test.

In this pilot scale test it was hoped to produce concentrate of a grade similar to that obtained in the laboratory batch tests. Because of the difficulty in altering conditions in the cleaning circuit of the pilot scale equipment, this aim was not achieved since the concentrate produced assayed only 7.5 lbs U_3O_8 per ton representing a recovery of 92.0 percent of the uranium.

5. OBSERVATIONS and CONCLUSIONS.

The nature of the diluent in the pilot plant concentrates is not known, as the concentrates contain about 80 percent of material finer than 200 mesh (Tyler series), which makes mineralogical estimation very difficult. It is likely, however, that the main contamination is due to slimed apatite and felspar and the use of depressants such as acids should be investigated.

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