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RECOVERY OF TANTALUM-BEARING MINERALS FROM FINELY DISSEMINATED ORES IN A PEGMATITE DEPOSIT AT BERNIC LAKE, MANITOBA

by

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MINERAL PROCESSING DIVISION

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SUMMARY OF RESULTS

Three low-grade refractory ores, used in this investigation, were characterized by the presence of coarse-grained as well as finely disseminated tantalum-bearing minerals, along with various amounts of sulphide minerals, ilmenite, and a wide variety of gangue minerals.

The average assays of these ores were as follows:

Date of Ore Sample	% Ta_2O_5
November 18, 1971	- 0.14% Ta_2O_5
November 25, 1971	- 0.20% Ta_2O_5
June 1 and 2, 1972	- 0.15% Ta_2O_5

The following is a summary of the results that might be expected by supplementing the present gravity operation at the Bernic Lake concentrator with slime-deck tabling of fines and re-ground middlings, followed by rejection of sulphides by flotation:

Ores Investigated	Tantalum Concentrate % Ta_2O_5 Assay	Tantalum Concentrate % Ta_2O_5 Recovery
November 18, 1971	50.0	60.0
November 25, 1971*	51.12	66.0
June 1 and 2, 1972	52.65	55.5

* Rejection of sulphides not required

The rejection of sulphides by flotation has already been installed for this purpose at the Bernic Lake concentrator and it is operating satisfactorily.

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CONTENTS

LIST OF TABLES

FIGURES

INTRODUCTION

Location of Property

The Bernic Lake pegmatite deposit is under the northwestern portion of Bernic Lake in the Lac du Bonnet area, about 125 miles northeast of Winnipeg, Manitoba. The property is owned by Tantalum Mining Corporation of Canada Limited, a subsidiary of Chemalloy Minerals Limited. It is the only producer of tantalum concentrates in North America.

Previous Mines Branch Investigations on Tantalum Ores from Bernic Lake

A considerable amount of work has been done at the Mines Branch on tantalum-bearing ores from this deposit. Because some earlier work is closely connected with this investigation, the earlier investigations are hereunder summarized.

(a) Test Work in 1961

Laboratory test work(1) consisting of gravity concentration was carried out in 1961 on a Bernic Lake ore sample containing 0.58% Ta_2O_5. The conclusion of this test work was that 32.7% of the Ta_2O_5 in the ore could be recovered in a gravity concentrate, which would contain 41.25% Ta_2O_5.

(b) Laboratory and Pilot Plant Investigations in 1967

More extensive laboratory and pilot plant investigations(2,5) on the tantalum ores from this property were done in 1967. These investigations were used as a basis for mill design(3,4) of the newly-formed Tantalum Mining Corporation of Canada Limited.

The laboratory investigations were done on three samples of coarse-grain tantalite ores applying gravity concentration (sand-deck rougher and cleaner tabling) and high intensity magnetic separation. The ores contained between 0.4 and 0.2% Ta_2O_5. The rougher tabling of these ores, ground to 20and 28 mesh, produced rougher concentrates containing between 80 and 84% of the Ta_2O_5 in the ores.

Grinding the rougher concentrates to 65 mesh (secondary grind) followed by re-tabling (cleaner tabling) produced cleaner concentrates assaying 51 to 54% Ta_2O_5 and recovered 70 and 28 per cent respectively. Based on the rougher recovery, therefore, between 10 and 14% of the tantalum in the ore was lost to the cleaner-table tailings. Between 60 and 70% of the Ta_2O_5 in these cleaner tailings was in minus 270 mesh sizes.

Because the tantalum-bearing minerals are brittle and slime easily during grinding and the sand-deck tabling can not effectively recover fine material, some fine-ground tantalite remained in the tailings.

From the primary grind about minus 20 mesh, no significant amount of the fine tantalite was present in the rougher tailings. After grinding the rougher concentrate to minus 65 mesh followed by a two-stage tabling, between 60 and 70% of the tantalum in the cleaner tailings was in fine (minus 270-mesh) sizes. This fine tantalite from the cleaner tailings was recovered by a Jones high-intensity wet magnetic separator in concentrates assaying between 16.5% Ta_2O_5 and 23.8% Ta_2O_5. The tantalum in these high-intensity concentrates comprised about one half of the tantalum present in the cleaner tailings or between 5 and 7% of the Ta_2O_5 in the ore. The final tantalum concentrate, consisting of table cleaner concentrate and high-intensity concentrate combined, assayed between 44 and 50% Ta_2O_5 and recovered between 76 and 70 per cent respectively.

A pilot-plant investigation(3,4,5) was performed in 1967 on ore, similar to that used for the laboratory investigations, that contained coarse-grained tantalum-bearing minerals with practically no fine dissemination of these minerals.

The 50-ton ore sample assayed 0.31% Ta_2O_5. The pilot-plant flowsheet, the same as in the laboratory investigations, consisted of 20-mesh rougher grind, sand-deck rougher tabling, and grinding of rougher concentrate to about 65 mesh followed by sand-deck cleaner tabling. The optimum feed rate in pilot-plant tabling was found to be between 1500 and 1700 lb/hr per full-size operating table. This produced tantalum concentrates assaying between 48 and 56% Ta_2O_5 with about 78% recovery.

Additional recovery of 1.3 to 2.9% Ta_2O_5 was obtained by treating the fines from the cleaner tailings with a Jones high-intensity wet magnetic separator. The concentrates produced had 42.4 and 17.5% Ta_2O_5 grades respectively.

These two operations, combined, produced tantalum concentrates of 47.4 to 55.7% Ta_2O_5 grade with over-all recovery of 80.5 to 78.8 per cent respectively.

(c) Laboratory Investigations in 1969

Since the beginning of operation, the Bernic Lake concentrator has been discarding cleaner tailings assaying about 0.5% Ta_2O_5 and containing between 8 and 10% of the tantalum in the ore. Between 60 and 70% of the tantalum in these tailings is finer than 270 mesh. This fraction assayed about 4% Ta_2O_5 and contained between 6 and 7% of the Ta_2O_5 in the ore(6,7). Most of the tantalum-bearing minerals in the fines of the cleaner tailings are free.

At the company's request, a 2-week visit(6) to the concentrator was arranged and, as a result, an investigation(7) on the two samples of fines from the concentrator's cleaner tailings was made. The samples assayed 9% and 1.3% Ta_2O_5, i.e., one higher than average and the other lower than average. In this investigation, slime-deck tabling and wet high-intensity magnetic separation were applied to recover the fine tantalum-bearing minerals. The slime-deck table gave more favourable results and produced concentrates containing 56.20 to 61.7% Ta_2O_5 to recover 23.5 to 69.3% of the Ta_2O_5 contained in the total cleaner tailing or 2.1 to 6.2% of the Ta_2O_5 in the ore.

Current Operation at Bernic Lake Concentrator

The tantalum-bearing minerals are concentrated by the gravity method developed at the Mineral Processing Division of the Mines Branch[2,3,4,5].

The ore, assaying about 0.2% Ta_2O_5, is ground to about minus 28 mesh (primary grind) and tabled on triple sand-deck Deister tables (rougher tabling). The rougher middling from each table is returned to the head of the table. Each table treats about 2.25 tons of ore per hour including the returned middlings. These tables produce rougher concentrates, comprising between 4 and 5% of the ore by weight, assaying between 3 and 4% Ta_2O_5, and containing about 75% of the tantalum in the ore. The rougher tailings, containing about 25% of the tantalum in the ore, are discarded.

The table rougher concentrate is reground to minus 65 mesh and treated by a low-intensity wet magnetic separation to remove metallics and other magnetic material.

The ground rougher concentrate, free of magnetics, is then tabled on Deister sand-deck tables, and a final (cleaner) tantalum concentrate is obtained. The cleaner middlings are recycled to the head of each cleaner table.

The final tantalum concentrates produced at the Bernic Lake concentrator during 1969, 1970, 1971, and 1972 had the following Ta_2O_5 grades and recoveries[8]:

TABLE 1

Grades and Recovery from Bernic Lake Concentrator

Operating Results	Tantalum Concentrates			
	1969	1970	1971	1972
Grade, % Ta_2O_5	54.5	51.5	50.1	50.2*
Recovery	60.8	61.6	61.0	58.2

*calculated

Based on the tantalum losses to the rougher tailings (approximately 25% of the Ta_2O_5 in the ore) and the results from Table 1, the tantalum loss to the cleaner tailing is approximately 14% of the tantalum in the ore.

The size and tantalum distribution in the concentrator's rougher and cleaner tailings for the month of February, 1972[8] are recorded in Table 2.

TABLE 2

Size and Tantalum Distribution in Concentrator's Rougher and Cleaner Tailings for Month of February 1972

Tailings	Mesh	% Weight	% Ta_2O_5	
			Assay	Distn in Tailing
Rougher tailing, February, 1972	- 28+ 35	13.2	0.045	11.9
	- 35+ 48	12.3	0.037	9.1
	- 48+ 65	10.5	0.029	6.1
	- 65+100	11.5	0.025	5.7
	-100+200	19.1	0.021	8.0
	-200+270	5.2	0.019	2.0
	+270 (cum)	71.8	0.029	42.8
	-270	28.2	0.101	57.2
	Total ro tail	100.0	0.049	100.0
Cleaner tailing, February, 1972	- 28+ 35	20.8	0.182	7.5
	- 35+ 48	15.5	0.278	8.5
	- 48+ 65	14.8	0.248	7.2
	- 65+100	17.3	0.199	6.8
	-100+200	19.1	0.158	5.9
	-200+270	3.0	0.253	1.5
	+270 (cum)	90.5	0.280	37.4
	-270	9.5	3.35	62.6
	Total cl tail	100.0	0.575	100.0

The results from Table 2 show that 57.2% of the tantalum in the rougher tailing was in the minus 270-mesh (fines) fraction. This represents about 14.2% of the tantalum in the ore. Also, the minus 270-mesh fraction of the cleaner tailing contains 62.6% of the tantalum in this tailing which represents about 9% of the tantalum in the ore. The minus 270-mesh fraction of this cleaner tailing assays 3.35% Ta_2O_5, i.e., about the same as cleaner tailing from our laboratory investigation, done in 1969[7], which showed that this finely ground tantalite can be recovered by slime-deck tabling.

The concentrator's cleaner tailing was discarded until the beginning of 1972. It is now being separated into minus and plus 270-mesh fractions. The minus 270-mesh fraction is stockpiled for future treatment and the plus 270-mesh fraction of the cleaner tailings is discarded[8].

<u>Nature of the Investigation Requested</u>

In his letter of December 9, 1971, Mr. C. T. Williams, Manager of Tantalum Mining Corporation of Canada Ltd., wrote:

> "Although not apparent mineralogically, there has been a change in the ore as we mine downward and on the average the amount of very finely disseminated tantalite is increasing. Our recoveries are dropping, and also it becomes increasingly more difficult to make our 45% to 50% Ta_2O_5 grade specification. The grade of the ore being processed is lower than that used in your work, but we have in the past had normal recovery on grades as low, or lower. We also have occasional problems with intermediate gravity minerals, such as tourmaline, ilmenite, and occasional arsenopyrite which of course is nearly as dense as the wodgenite, but occurs at times locked with amphibolite gangue.

Our questions seem to be multiple. Firstly, will the so-called fine-grained ore we are presently encountering give the same response to the process you developed in 1967? If it doesn't, is it possible that changes might be made to the present flowsheet to maintain recovery and product grade at reasonable levels? If this is not possible, what other metallurgical process alternatives are there?".

Ore Shipments for this Investigation

Three ore samples, assaying from 0.15% to 0.2% Ta_2O_5 and containing various amounts of finely disseminated minerals, were received for this investigation. Dates when the ore samples were obtained at the concentrator and the contents of Ta_2O_5 are as follows:

Date of Ore Sample	% Ta_2O_5
November 18, 1971	0.153
November 25, 1971	0.205
June 1 and 2, 1972	0.159

The amount of finely disseminated tantalum minerals was fairly high in the ore samples of November 18, 1971 and June 1 and 2, 1972, whereas the ore sample of November 25, 1971 contained less of the finely disseminated minerals and less sulphides and ilmenite than the other two ore samples.

Analysis

All analyses for this investigation were done by the Tantalum Mining Corporation of Canada Limited at Bernic Lake, Manitoba.

OUTLINE OF INVESTIGATION

It was mentioned above that the initial investigations [2,3,5] were done on Bernic Lake ores that contained coarse-grained tantalum bearing minerals assaying 0.3 to 0.4% Ta_2O_5 and practically no finely disseminated tantalum minerals.

From the beginning of operation (1969) to 1971, the ore milled contained between 0.2 and 0.25% coarse Ta_2O_5 but little or no finely disseminated tantalum-bearing minerals. During this period, the company operated without major difficulties.

Since 1971, the ore has deteriorated in that grade is decreasing (occasionally down to 0.14% Ta_2O_5) and finely disseminated tantalum-bearing minerals, sulphides (particularly arsenopyrite), and ilmenite are increasing in the ore. These three negative factors in the ore adversely affect the grade of the concentrate and the over-all tantalum recovery[8].

Because the tantalum-bearing minerals are softer and easier to grind than most of the other minerals in the ore (silicates, arsenopyrite, pyrite, ilmenite, etc.), the coarse-grain tantalum-bearing minerals are ground finer than most of the other minerals in the ore. Part of the fines produced during the grinding operation, therefore, originates from the coarse-grain minerals and most of these tantalum fines are free grains of the tantalum-bearing minerals.

Both the sulphides (mainly arsenopyrite) and the ilmenite, being harder than the tantalum-bearing minerals, remain coarser than the tantalum-bearing minerals during grinding. As a result, most of these hard minerals report in the tantalum concentrate during the rougher and cleaner tabling to lower the tantalum grade of the concentrate.

Because of these factors, the characteristics of the ore received, and the company's request, this investigation consisted mainly of the following steps:

A - recovering most of the tantalum-bearing minerals without major changes in the current flowsheet;

B - regrinding the middling and cleaner tailing to minus 200 mesh, or finer if feasible, and recovering tantalum concentrate from this finely ground material by various methods; and

C - rejecting sulphides from the final tantalun concentrate by flotation.

DETAILS OF INVESTIGATION

Investigation on Ore of November 18, 1971

Preliminary Test Work

Most of the preliminary testing was done on the ore shipped. Various tabling schemes were investigated with the objective of producing a marketable tantalum concentrate at maximum recovery.

The procedures and the flowsheet used in some of the more pertinent tests will be described in detail.

Test 4 - This test consisted of a 100-mesh primary grind, rougher tabling on a Deister sand deck, followed by slime-deck tabling of the fines from the rougher tabling and reground cleaner tailing.

Details of this procedure were as follows:

(1) the ore, ground to minus 100 mesh, was tabled on a Deister sand deck and a low-grade rougher concentrate was obtained; the concentrate was retabled on a Deister sand deck to obtain a "coarse" cleaner concentrate;

(2) the cleaner tailing from (1) was ground to about minus 200 mesh and tabled on a Deister slime deck to obtain a slime deck concentrate; and

(3) the rougher tailing from (1) and the cleaner tailing from (2) were combined and screened on 400 mesh; the plus 400-mesh fraction was discarded and the minus 400-mesh fraction was fed to a Jones high-intensity wet magnetic separator to determine whether this separation would sufficiently concentrate such fine tantalum-bearing minerals.

The flowsheet of this test (without the Jones separator) is presented in Figure 1 and the results in Table 3.

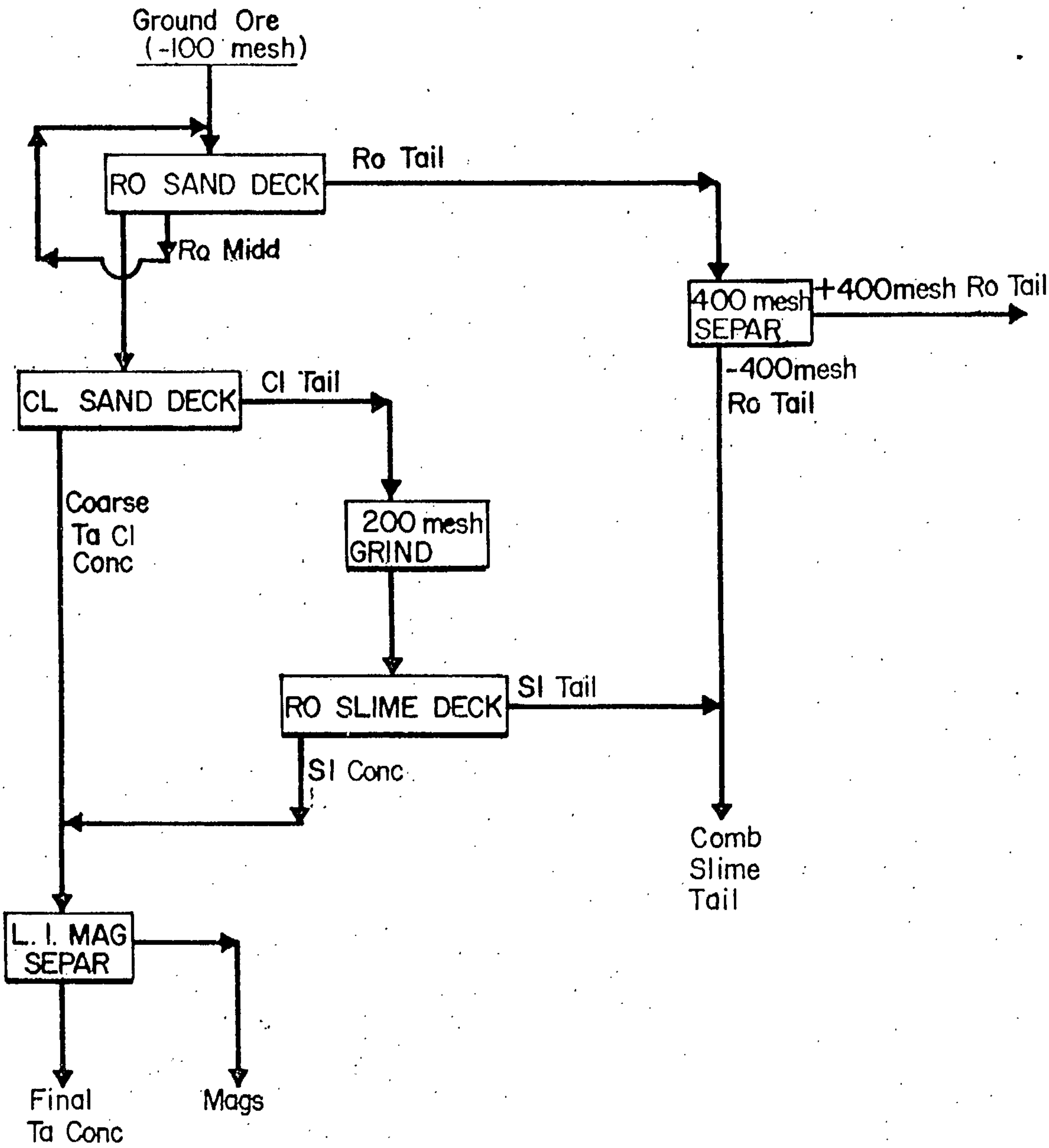

Figure 1. FLOWSHEET OF TEST 4
ORE: NOVEMBER 18/1971

TABLE 3

Tabling Results from Test 4; Ore: November 18, 1971

Grind Mesh	Product	% Weight	% Ta_2O_5 Assay	% Ta_2O_5 Distn
-100	"Coarse" Ta cl conc	0.246	31.30	58.4
-200	Slime Deck conc	0.030	18.00	4.1
	Final Ta conc (calcd)	0.276	30.00	62.5
	Ro slimes + cl tail	36.548	0.116	32.2
-100+400	Ro tail "sands"	62.776	0.010	4.7
+100	Mica from ro grind	0.400	0.18	0.6
-100	Feed (calcd)	100.000	0.132	100.0

When the slimes from the rougher and cleaner tailings were treated by a Jones separator, the concentrate obtained had a grade of only 3.69% Ta_2O_5. Therefore, the idea of upgrading the slimes by high-intensity wet magnetic separation was abandoned.

Test 5 - Shaking tables usually operate most effectively on closely sized feed. If the size range is large, the table feed should be separated into several size fractions and each fraction should be tabled separately. The separate tabling of the size fractions always improves the grade of concentrate and the recovery of the valuables.

In view of this, the major steps of the procedure for this test were as follows:

(1) ore was ground to minus 48 mesh and then separated into a plus 80-mesh (coarse) fraction and a minus 80-mesh (fine) fraction;

(2) the coarse and fine fractions were tabled separately on Deister sand decks; the rougher fines from tabling the fine fraction were saved and the sands from both fractions were discarded;

(3) the concentrate from the coarse fraction was ground to minus 65 mesh and combined with the concentrate from the fine fraction; this combined concentrate was then retabled on a Deister sand deck to obtain a "coarse" tantalum cleaner concentrate.

(4) the cleaner tail from (3) was ground to about 200 mesh and screened on a 270-mesh screen; the screen oversize was discarded and the under-size was combined with the fines from rougher tabling as explained in (2). These combined fines from rougher and cleaner tailings were tabled on a Deister slime deck to obtain a "fine" tantalum concentrate. The middlings from each tabling step of this procedure were recycled to the feed of each table.

The flowsheet of this procedure is presented in Figure 2 and the results in Table 4.

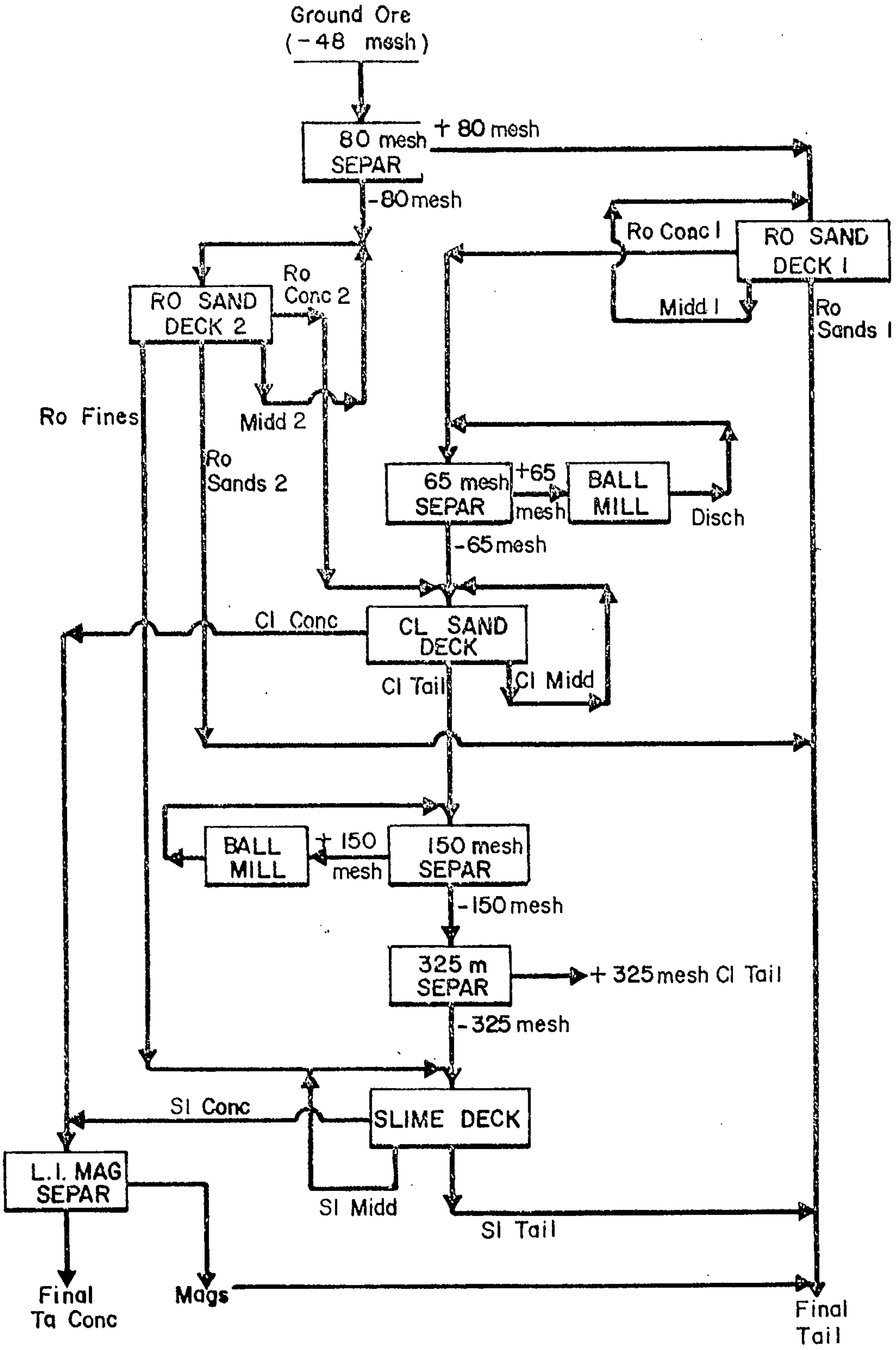

Figure 2. FLOWSHEET OF TEST 5.
ORE : NOVEMBER 18/1971

TABLE 4

Results from Test 5 Ore; November 18, 1971

Grind-Mesh			Product	% Weight	% Ta_2O_5	
Prim	Sec	Ter			Assay	Distn
	-65		Coarse cl conc	0.17	35.77	50.0
		-150	Fine cl conc	0.06	17.95	9.0
			Final conc (calcd)	0.23	31.30	59.0
			Slime tailing	23.61	0.108	21.3
			Slime middling	0.84	0.576	4.1
			Ground cl tailing (150 to 325-mesh)	2.71	0.387	8.6
			Rougher sands 1	26.50	TR	0.2
			Rougher sands 2	46.11	0.018	6.8
-48			Feed (calcd)	100.00	0.122	100.0

It can be expected that recyling of the "Slime Middlings" and the "Ground Cleaner Tailing" would recover about 50% of the tantalum values in these products, i.e., about 6% more tantalum would be recovered from the ore. The above procedure, therefore, should result in about 65% tantalum recovery from this ore in a concentrate containing about 31% Ta_2O_5.

It was observed that non-tantalum-bearing minerals in the tantalum concentrate lowered the tantalum grade of the concentrate. A mineralogical examination of the concentrates was done by the Mineral Sciences Division to identify these minerals; see "Mineralogical Composition of Tantalum Concentrates", page 19.

Test 9 - The results from fine and moderate grinds of Test 4 and 5 did not show much difference in the over-all grade and recovery of tantalum. Therefore, it was decided to grind the ore to minus 35 mesh, (only slightly finer than the current primary grind at Bernic Lake which is minus 28 mesh), proceed with the fine grinding of the middling products, and table the finely ground material on slime decks.

This procedure gave fairly satisfactory results from this unusually difficult ore, so it will be described in detail.

(a) Initially, the ore was ground to minus 35 mesh and tabled on a sand deck. Three products were obtained from this step:

(1) rougher concentrate,

(2) rougher middlings, and

(3) rougher tailing.

(b) Rougher concentrate was ground (secondary grind) to minus 65 mesh and tabled on a Deister sand deck as usual and the "coarse" tantalum cleaner concentrate obtained.

(c) The cleaner tailing from (b) and the rougher middlings (a-2) were combined and ground to minus 200 mesh (tertiary grind).

(d) The rougher tailing (a-3) was screened on 200 mesh, the plus 200-mesh fraction rejected, and the minus 200-mesh fraction combined with ground products from (c).

(e) The ground cleaner tailing, ground rougher middling, and the minus 200-mesh portion of the rougher tailing were combined and screened on a 325-mesh screen to obtain a plus and a minus 325-mesh fraction.

(f) The plus 325-mesh fraction from (e) was tabled on a Deister sand deck and the minus 325-mesh fraction was tabled on a Deister slime deck; two tantalum concentrates were obtained and combined with the "coarse" tantalum cleaner concentrate from (b).

(g) The combined tantalum concentrates (from "b" and "f") were treated by a low-intensity magnetic separator and magnetic portion rejected, leaving the final tantalum concentrate as non-magnetics.

The results of this procedure are presented in Table 5, and the flowsheet in Figure 3.

TABLE 5

Tabling Results from Test 9; Ore: November 18, 1971

Grind-Mesh			Product	% Weight	% Ta_2O_5	
Prim	Sec	Tert			Assay	Distn
	-65		Coarse cl conc (-65 mesh)	0.125	40.18	35.4
		-200	Scav conc #1 (+325 mesh)	0.050	31.45	11.1
		-200	Scav conc #2 (-325 mesh)	0.021	43.02	6.5
			Final conc (calcd)	0.196	38.26	53.0
		-200	Scav midd #1 (+325 mesh)*	0.126	3.76	3.3
		-200	Scav midd #2 (-325 mesh)*	0.196	8.10	11.3
			Comb middlings*	0.322	6.40	14.6
-35			Rougher tail (+270 mesh)	43.571	0.005	1.6
		-200	Scav tail #1 (+325 mesh)	22.501	0.005	0.8
		-200	Scav tail #2 (-325 mesh)	33.410	0.127	30.0
-35			Feed (calcd)	100.000	0.141	100.0

* circulating load

As a continuous recycling of both scavenger middlings to the table feed would recover about 50% of the tantalum contained in these middlings, it can be expected, therefore, that this procedure would recover between 60 and 61% of the Ta_2O_5 in 38% Ta_2O_5 concentrate.

The use of Bartles-Mozley tables(9) or Holman slime tables(9), or their combination, to treat the "Scavenger tailing #2 (-325 mesh)", assaying 0.127% Ta_2O_5 and containing 30% of the tantalum in the ore, could recover a low-grade concentrate that could be suitable for chemical treatments and thus increase the over-all tantalum recovery.

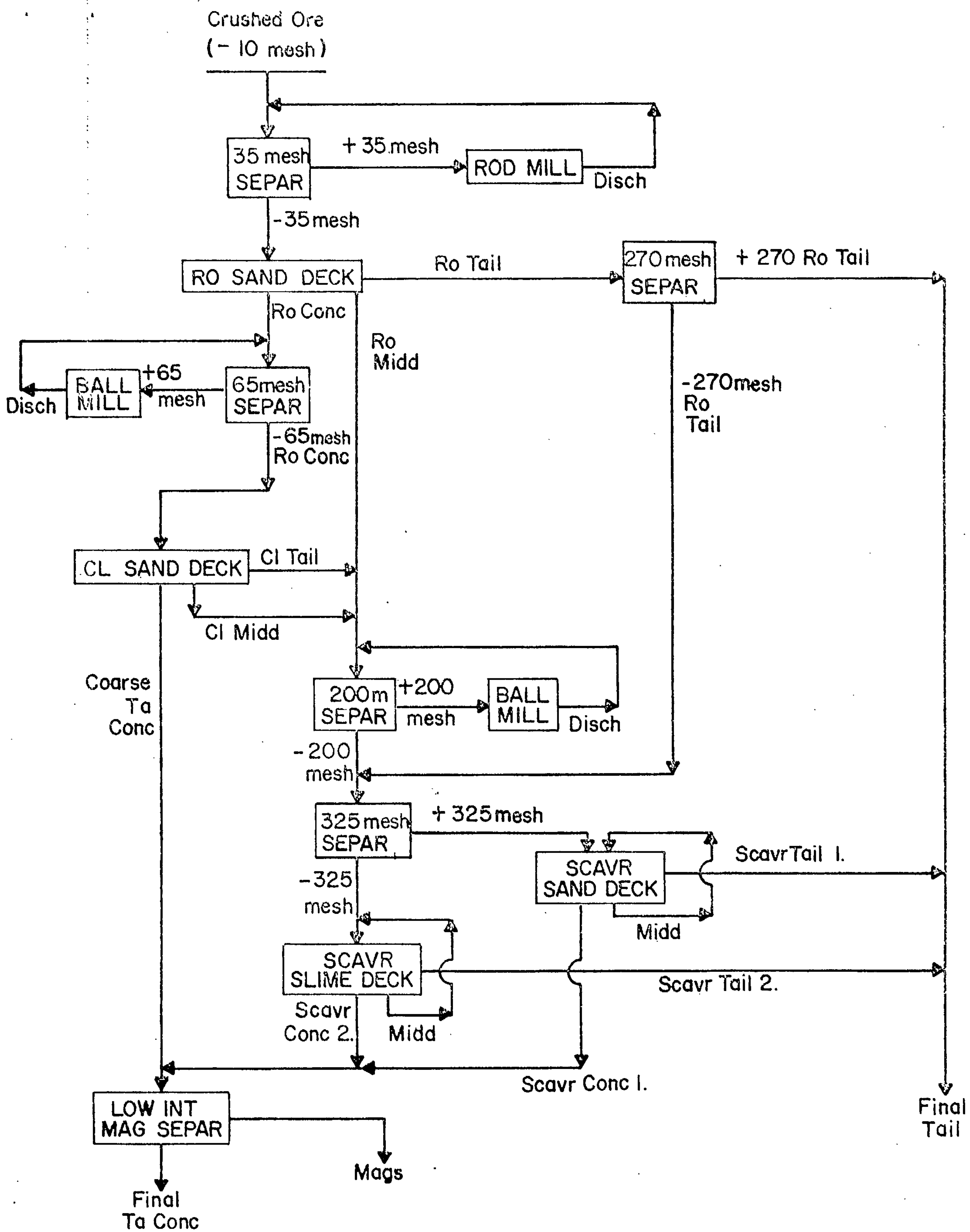

Figure 3. FLOWSHEET OF TEST 9.
ORE SAMPLE: NOVEMBER 18, 1971

Mineralogical Composition of Tantalum Concentrates

To identify other than tantalum-bearing minerals, a mineralogical examination* of the concentrates from Tests 5 and 9 (Tables 3 and 5) was done by Dr. D. C. Harris, Mineral Sciences Division, Mines Branch.

Test 5 - Tantalite, wodginite, and microlite are the principal tantalum-bearing minerals in this concentrate. Due to their similar optical properties under reflected light, these minerals could not be distinguished from each other. Microlite was rarely detected.

Wodginite is a distinct mineral that can be considered as a high-tin tantalite which, in earlier terminology, was called stanniferous tantalite.

Cassiterite is fairly abundant in the concentrate but, because it is of a colour similar to tantalite-wodginite, its relative proportion could not be determined. About half of the cassiterite grains contain up to 4.4% Ta. Tin also occurs in significant amounts in the wodginite grains.

Several sulphide minerals were identified in the concentrate of Test 5 and, based on grain counting, they constitute approximately 20% (Figure 4). The prinipal sulphides are arsenopyrite and pyrite with minor to trace amounts of pyrrhotite, chalcopyrite, galena, and native bismuth intergrown with an unidentified Ag-Pb-Bi sulphosalt, niccolite, native silver, and stannite.

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* From Mines Branch Investigation Reports IR 72-39 and IR 72-54.

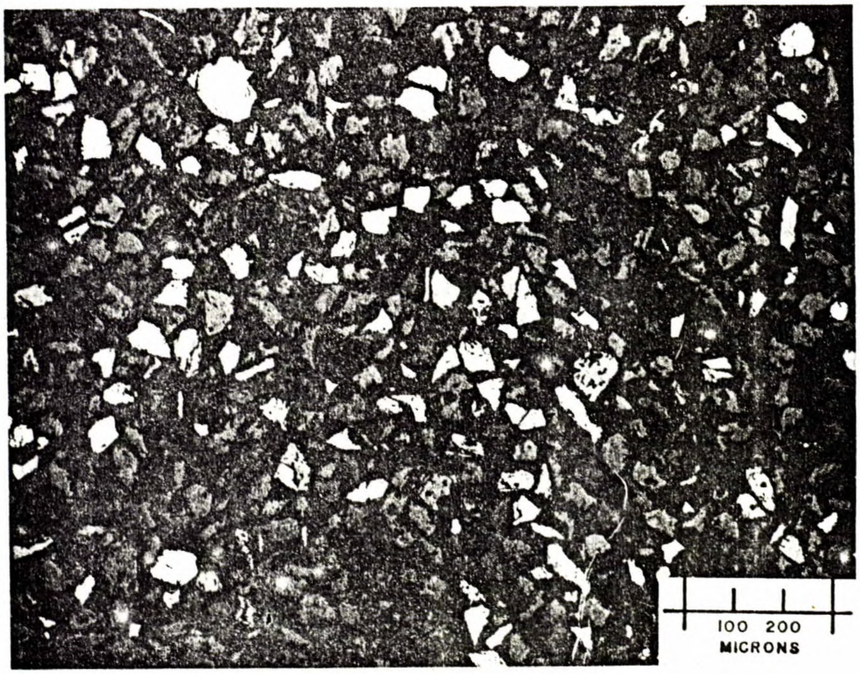

Figure 4. Photomicrograph of the coarse tantalum cleaner concentrate of Test 5 to illustrate the content of sulphides (white) and degree of liberation.

Test 9 - More extensive mineralogical examination was done on the tantalum concentrates from this test. As in the concentrate from Test 5, the principal tantalum-bearing minerals in the tantalum concentrates from Test 9 are tantalite, wodginite, and microlite. Microlite is rare in the ore. Cassiterite and wodginite are the principal tin minerals, with up to 17.8% SnO_2 being detected in the wodginite grains. The principal sulphide minerals are arsenopyrite and pyrite with minor amounts of pyrrhotite, chalcopyrite, galena, native bismuth, niccolite, and stannite.

The mineralogical conclusion obtained from each tantalum concentrate and some middlings are given in detail.

Coarse Tantalum Cleaner Concentrate

Many sulphide minerals were noted in this concentrate (Figure 5). Based on grain counting, the sulphides represent approximately 22% by volume, ilmenite 4%, with the tantalum-bearing phases and the gangue minerals accounting for the remainder. Cassiterite is the principal tin mineral, although most of the tantalite-wodginite grains contain tin; microprobe analyses of 13 random grains gave an average composition of Ta_2O_5 - 70.2%, Nb_2O_5 - 3.3%, SnO_2 - 13.3%, TiO_2 - 3.0%, MnO - 8.9%, FeO - 2.6%, total 101.3 wt %. Almost every tantalite-wodginite grain larger than 150 μ contains inclusions of gangue minerals.

Scavenger Tantalum Concentrate #1

In comparison to the coarse tantalum cleaner concentrate, this fraction contains a higher sulphide content, more ilmenite, iron spherules, and magnetite grains (Figure 6).

Removal of the magnetic fraction with a hand magnet shows that it represents 4.5 wt % of the sample. Based on grain counting, the proportion of the following fractions were noted: Ta minerals, 54.0%; ilmenite, 16.0%; sulphides, 20.0%; and gangue, 10.0%.

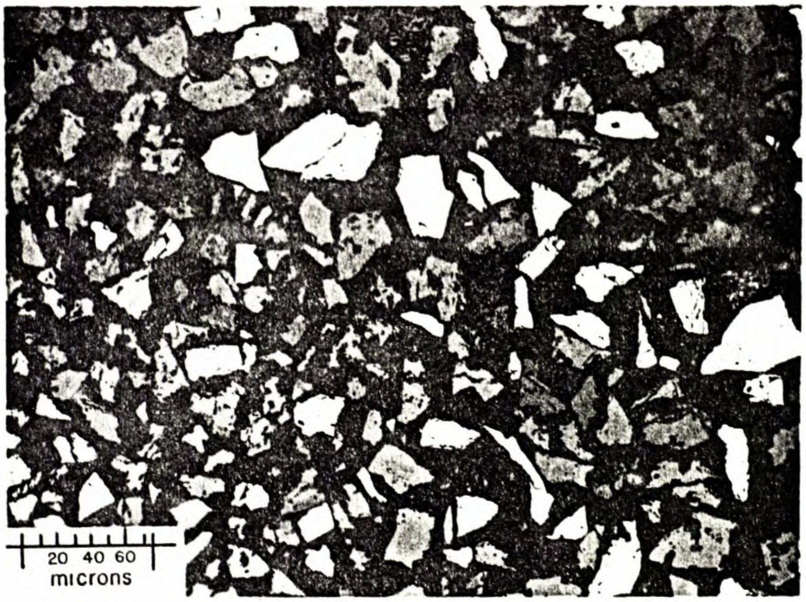

Figure 5. Photomicrograph of the sulphide grains (white) in the coarse Ta cleaner concentrate of Test 9.

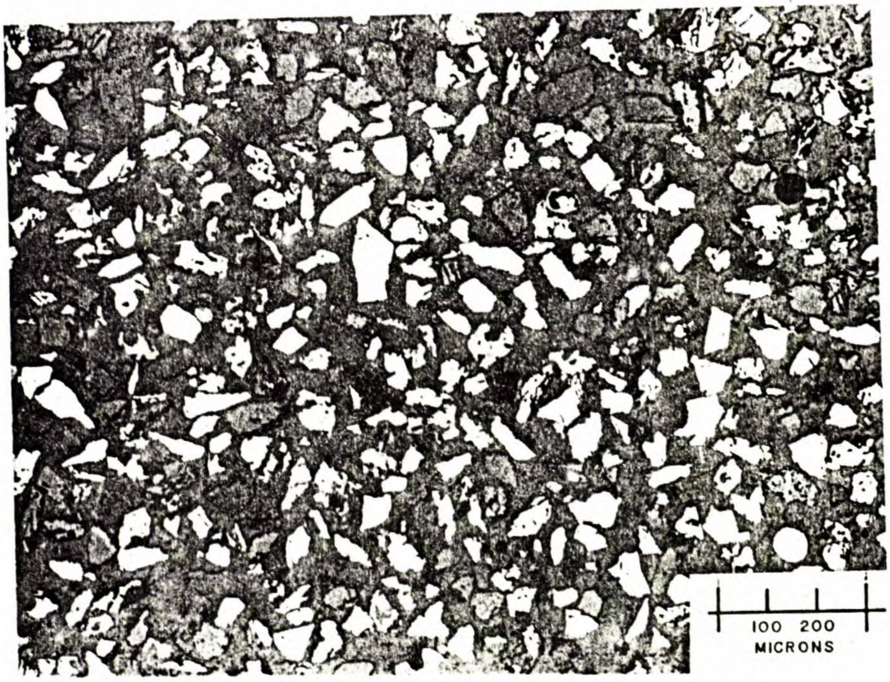

Figure 6. Photomicrograph of the sulphide grains (white) in the scavenger concentrate #1 of Test 9. An unusually high ilmenite content was noted in this sample.

Scavenger Tantalum Concentrate #2

This sample contains 12% sulphides, whereas no magnetic grains were noted (Figure 7). The sample is too fine-grained to permit examination in more detail.

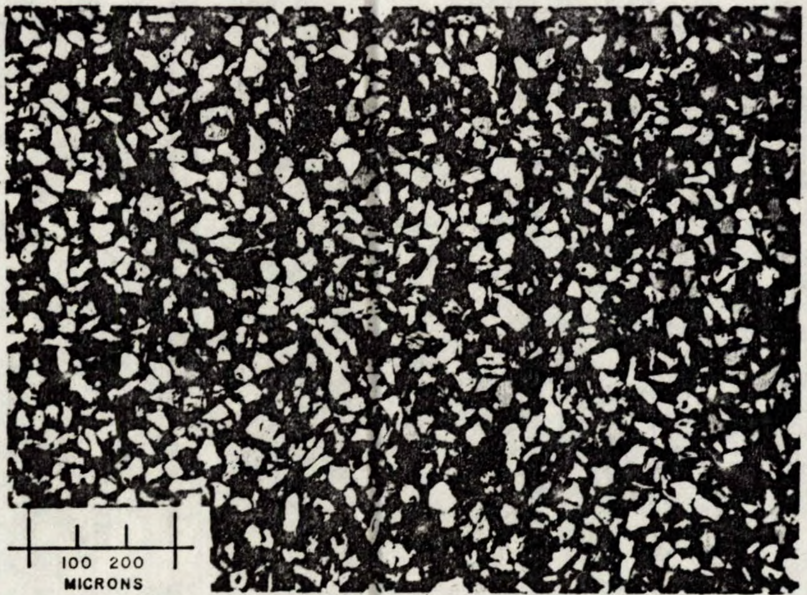

Figure 7. Photomicrograph of the sulphide grains (white) in the scavenger concentrate #2 of Test 9.

The tantalum grade of each concentrate with the amounts of sulphides and ilmenite present in these concentrates from Table 5 are given in Table 6.

TABLE 6

Sulphides and Ilmenite Contents in Tantalum Concentrates from Test 9; Ore: November 18, 1971

Product	Wt %	Ta_2O_5 %	Sulphides, Vol %	Ilmenite, Vol %
Coarse cl conc	0.125	40.18	22.0	4.0
Scav conc #1	0.050	31.45	20.0	16.0
Scav conc #2	0.021	43.02	12.0	3.0
Final conc (calcd)	0.196	38.26	20.4	8.9

Investigation on Ore of November 25, 1971

This ore was not as difficult to treat as the ore of November 18, 1971. Several tests were done by procedures similar to those used on the ore of November 18, 1971, i.e., rougher tabling of sized and un-sized ore, with and without recycling the rougher middlings. The rougher middlings and cleaner tailing were reground and tabled on slime decks. The procedures applied for each test will be described separately.

Test 6 - The major steps of the procedure for this test were as follows:

(a) The ore was ground to minus 48 mesh.

(b) The ground ore was separated into three sized fractions and tabled separately but without recycling of middlings (see "d"). The coarse and middle fractions were tabled on two sand decks and the fine fractions on a slime deck. Three rougher concentrates were obtained from these treatments.

(c) The rougher concentrates obtained from the coarse and middle fraction were combined and ground to minus 65 mesh and then tabled (cleaner tabling) to obtain a "coarse cleaner concentrate".

(d) The cleaner tailing from (c) and the rougher middlings from the coarse and middle fraction were ground to about minus 150 mesh and then separated into the plus 270-mesh fraction and the minus 270-mesh fraction. The plus 270-mesh fraction was rejected.

(e) The minus 270-mesh fraction from (d) was combined with the rougher concentrate obtained from the fine fraction of the ground ore (see "c") and the combined finely ground material was then tabled on a slime deck to obtain a "fine tantalum concentrate".

A detailed flowsheet of this procedure is illustrated in Figure 8 and the results in Table 7.

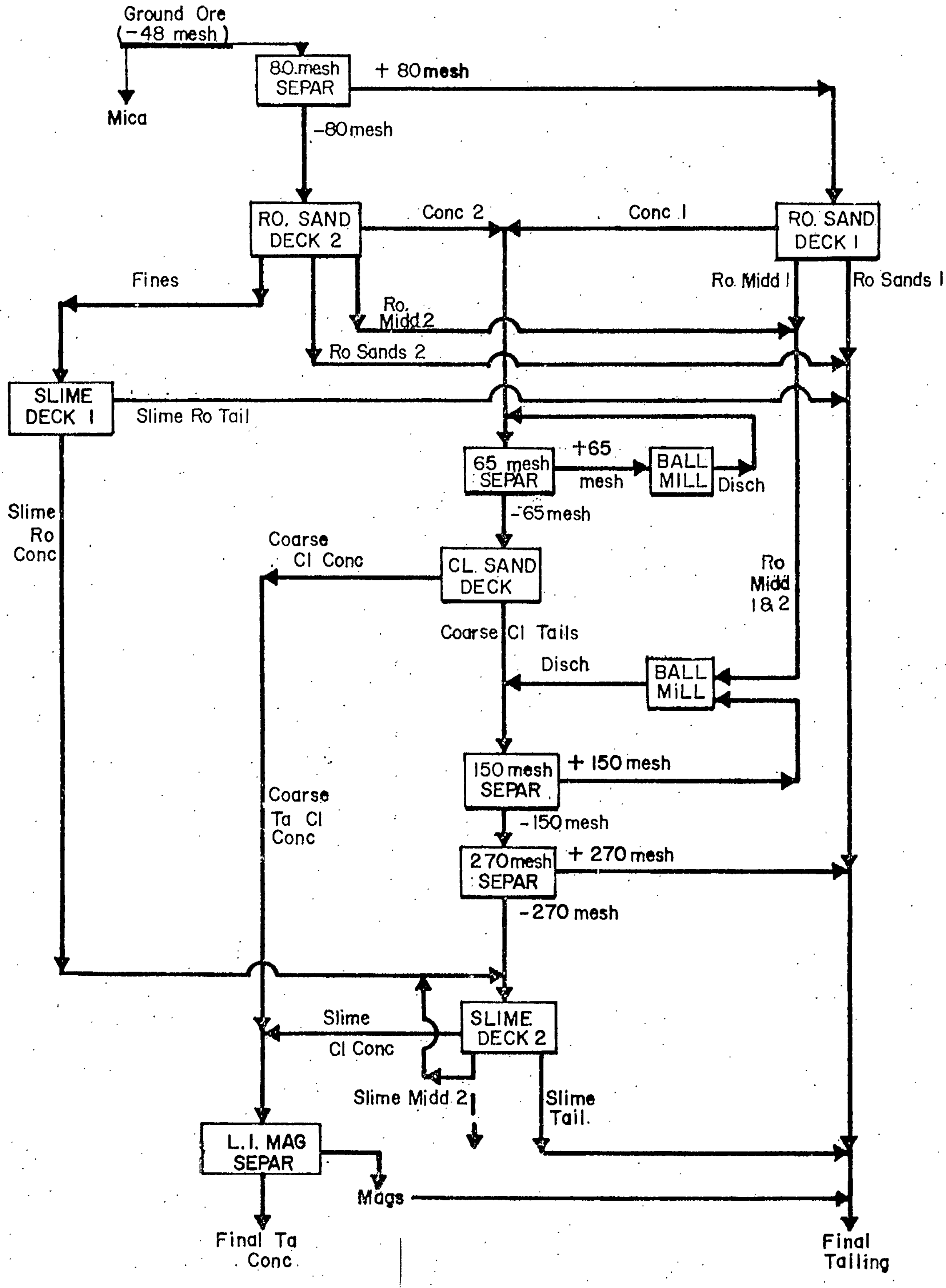

Figure 8. FLOWSHEET OF TEST 6.
ORE : NOVEMBER 25/1971

TABLE 7

Tabling Results from Test 6; Ore: November 25, 1971

Grind-Mesh			Product	%	% Ta O	
Prim	Sec	Ter		Weight	Assay	Distn
	-65		Coarse cl conc	0.18	50.88	49.0
		-150	Fine cl conc	0.05	45.61	12.2
			Final conc (calcd)	0.23	50.0	61.2
		-150 +270	+270 mesh from tert grind	7.70	0.203	8.3
			Slime deck tailing 1	11.28	0.107	6.5
		-270	Slime deck tailing 2	7.22	0.220	8.5
		-270	Slime deck middling	0.05	12.49	2.1
-48 +80			Sand deck tailing 1	37.41	0.026	5.2
-80			Sand deck tailing 2	36.10	0.038	7.2
			Mags from final conc	0.05	5.96	0.3
			Mica from all grinds	0.80	0.18	0.7
-48			Feed (calcd)	100.00	0.188	100.0

The continuous recycling of the slime deck tailing 2 and the slime deck middling 2, containing 10.6% of tantalum in the ore, should recover approximately 50% of the tantalum in these two products, which would result in an additional recovery of about 5.3% of the tantalum in the ore.

With this assumption, the above procedure should recover about 66% of the tantalum in the ore in a concentrate of about 50% Ta_2O_5 grade.

Test 8 - Procedure for this test was the same as that described in Test 9, (Figure 3 for the ore of November 18, 1971). The results obtained from Test 8 are recorded in Table 8.

TABLE 8

Tabling Results from Test 8; Ore: November 25, 1971

Grind-Mesh			Product	% Weight	% Ta_2O_5	
Prim	Sec	Tert			Assay	Distn
	-65		Coarse cl conc (-65 mesh)	0.128	53.81	36.5
		-200	Scav conc #1 (+325 mesh)	0.056	50.70	14.5
		-200	Scav conc #2 (-325 mesh)	0.055	45.37	13.4
			Final conc (calcd)	0.239	51.12	64.4
		-200	Scav midd #1 (+325 mesh)*	0.033	7.32	1.3
		-200	Scav midd #2 (-325 mesh)*	0.028	11.37	1.7
			Comb middlings*	0.061	9.2	3.0
-35			Rougher tail (+200 mesh)	55.385	0.010	2.9
		-200	Scav tail #1 (+325 mesh)	18.106	0.018	1.7
		-200	Scav tail #2 (-325 mesh)	26.209	0.202	28.0
-35			Feed (calcd)	100.000	0.189	100.0

* Circulating load.

It can be expected that an additional 1.5% of the Ta_2O_5 can be recovered from the middlings (circulating load) when in continuous recycling. It can be assumed, therefore, that this procedure can produce a final tantalum concentrate containing over 50% Ta_2O_5 to recover 65 to 66 per cent.

From the operating point of view, the following details concerning the various circuits of this test would be of interest:

(i) weight of the rougher concentrate was about 7% of the ore, i.e., slightly higher than usual

(ii) weight of rougher middlings was about 13.5% of the ore

(iii) weight of the minus 200-mesh fraction of the rougher tailing was about 27% of the ore; and

(iv) the feed for the tertiary grind (rougher middlings and cleaner tailing) was about 20% of the ore by weight.

No problems were experienced in obtaining good grades of the concentrates with reasonable tantalum recoveries, so a mineralogical examination of the concentrates obtained from this ore sample was not carried out.

The relationship between Ta_2O_5 grades and recoveries of the tantalum concentrates without rejection of sulphides is presented in Figure 9.

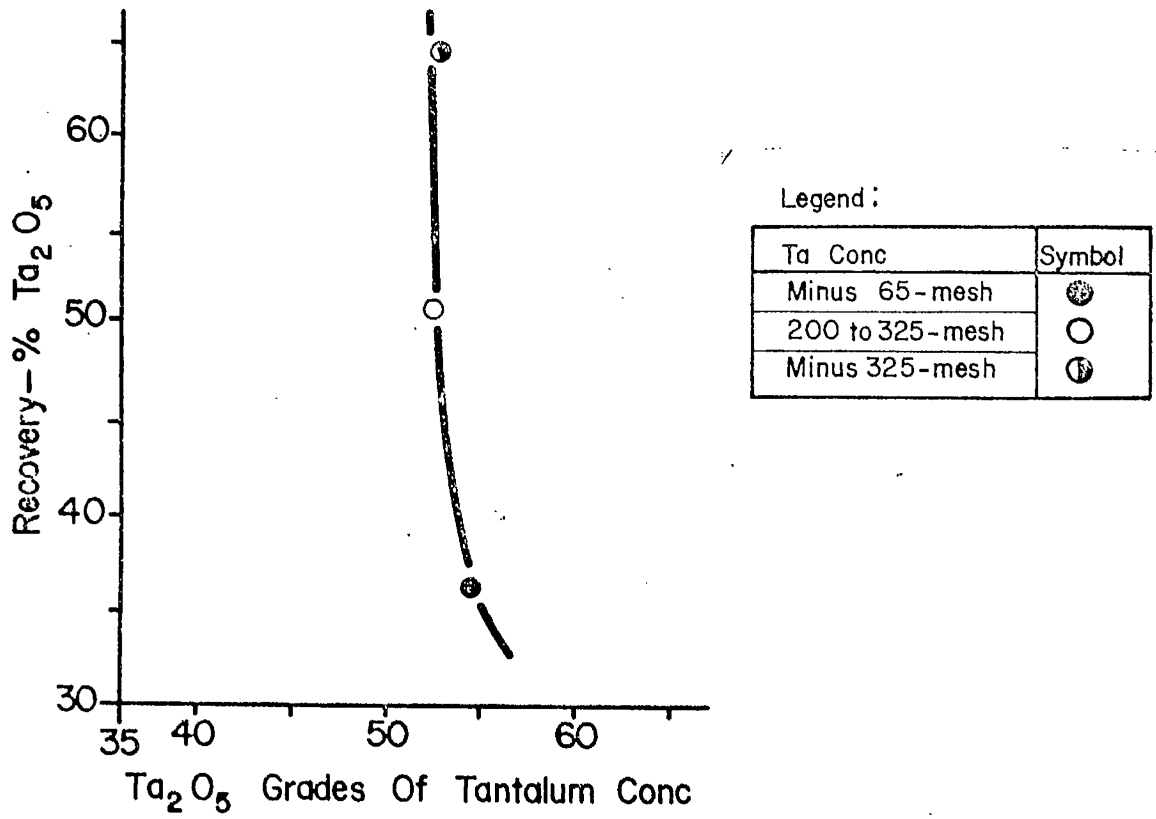

Figure 9 RELATIONSHIP BETWEEN Ta_2O_5 GRADE AND RECOVERIES OF TANTALUM CONCENTRATES. ORE: NOVEMBER 25, 1971, TEST 8

Investigation on Ore of June 1-2, 1972

The characteristics of this ore were similar to those of the two previous samples, i.e., it contained finely disseminated tantalum-bearing minerals with more sulphides than the ore of November 25, 1971 but not as much as the ore of November 18, 1971.

Test 10 - The procedure for Test 9 gave fairly satisfactory results, so this procedure (Figure 3) was applied to the ore of June 1 and 2, 1972. The results obtained are recorded in Table 9.

TABLE 9

Tabling Results from Test 10; Ore: June 1 and 2, 1972

Grind-Mesh			Product	%	% Ta_2O_5	
Prim	Sec	Tert		Weight	Assay	Distn
	-65		Coarse cl conc (-65 mesh)	0.128	49.40	31.8
		-200	Scav conc #1 (+325 mesh)	0.035	44.48	7.8
		-200	Scav conc #2 (-325 mesh)	0.041	53.11	10.9
			Final conc (calcd)	0.204	49.30	50.5
		-200	Scav midd #1 (+325 mesh)*	0.242	3.65	4.4
		-200	Scav midd #2 (-325 mesh)*	0.176	6.37	5.7
			Comb middlings (calcd)	0.418	4.80	10.1
-35			Rougher tail (+200 mesh)	50.108	0.055	13.8
		-200	Scav tail #1 (+325 mesh)	21.900	0.002	0.2
		-200	Scav tail #2 (-325 mesh)	27.370	0.185	25.4
-35			Feed (calcd)	100.000	0.20	100.0

* Circulating load.

With the same assumption as in the previous tests, approximately 5% of the tantalum should be recovered from the combined middlings when this material is recycled to the feed of each table. It can be concluded, therefore, that between 55 and 56% of the tantalum in this ore can be recovered in a concentrate assaying about 50% Ta_2O_5 by this procedure.

It was observed that the concentrates contained various amounts of sulphide minerals. Therefore, a mineralogical examination of these concentrates was made.

Mineralogical Composition of Tantalum Concentrates from Test 10

The three tantalum concentrates shown in Table 9 were submitted for a mineralogical examination. The summarized results obtained from this examination follow *.

Coarse Tantalum Cleaner Concentrate - In comparison with the coarse tantalum cleaner concentrate of Test 9, this sample contains fewer sulphides and appears to be finer-grained, with most of the grains being smaller than 50 microns in diameter (Figure 10). The larger Ta-fragment contains gangue inclusions but these are not as abundant as in Test 9. Grain counting gave the following proportions: Ta-minerals, 84%; sulphides, 6%; ilmenite, 1%; gangue, 9%.

Scavenger Tantalum Concentrate #1 - The main difference between this concentrate and the coarse tantalum cleaner concentrate is its higher sulphide content; see Figure 11.

Scavenger Tantalum Concentrate #2 - This fraction was so fine-grained that grain counting was impossible (Figure 12). Sulphides were usually estimated at between 5 and 10%. Several magnetite and metallic particles were noted. These were removed by hand magnet and represented 4.1 wt % of the sample.

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* From Mines Branch Investigation Report IR 72-54, October, 1972.

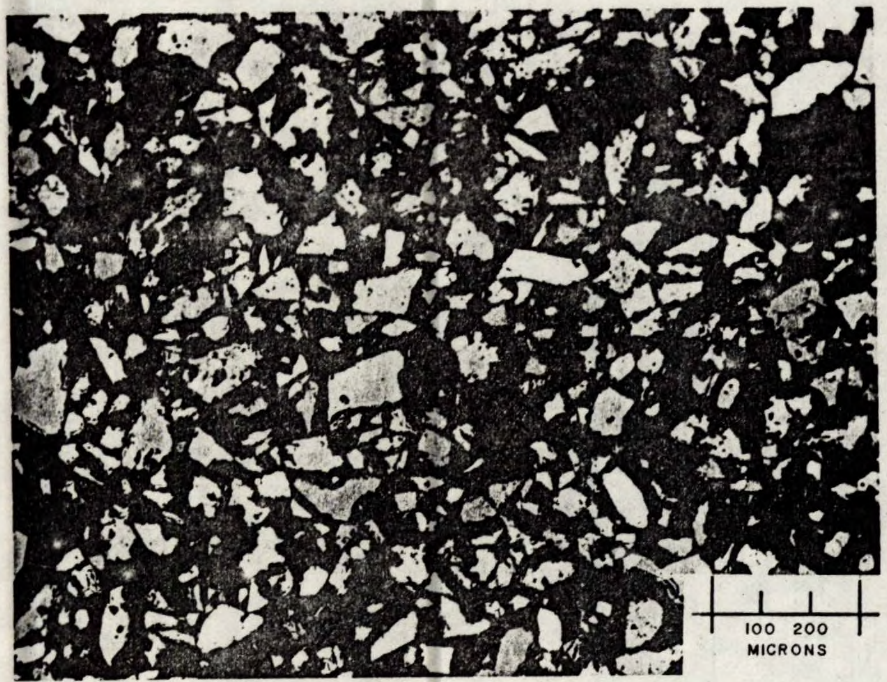


Figure 10. Photomicrograph of the sulphide grains (white) in Test 10 cleaner concentrate.

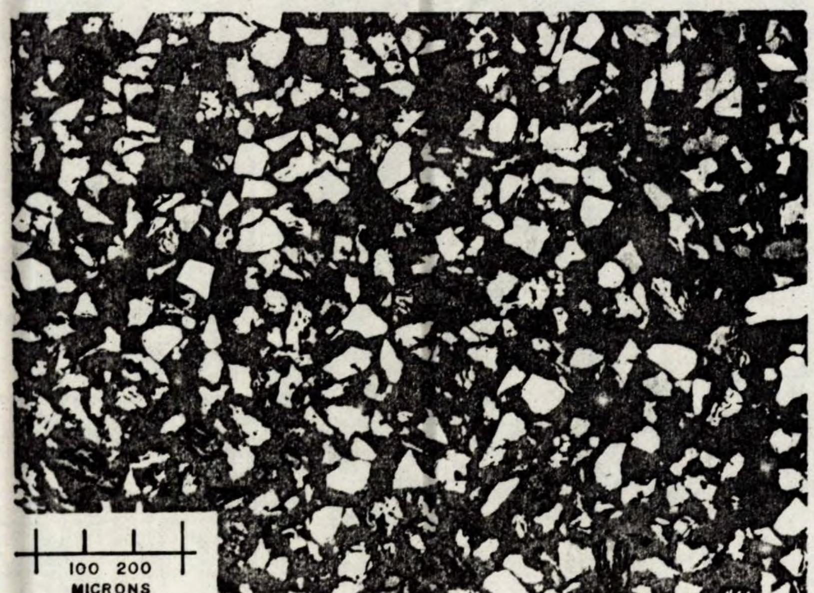


Figure 11. Photomicrograph of the sulphide grains (white) in Test 10 scavenger concentrate #1.

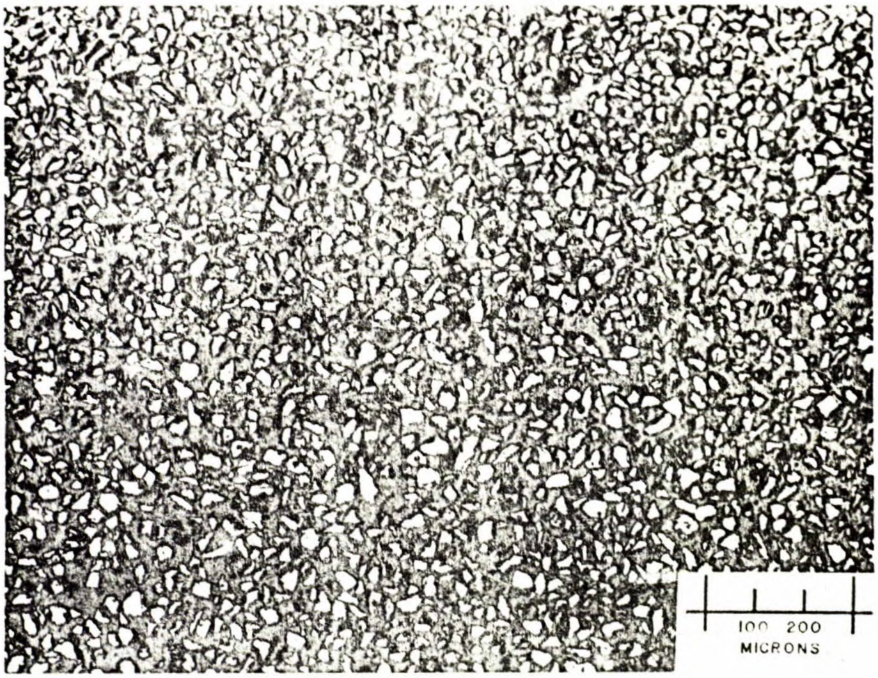


Figure 12. Photomicrograph of the sulphide grains (white) in the scavenger concentrate #2 of Test 10.

The tantalum grade of each concentrate with the amounts of sulphides and ilmenite present in these concentrates (Table 9) are given in Table 10.

TABLE 10

Sulphide and Ilmenite Content in Tantalum Concentrates from Test 10; Ore: June 1 and 2, 1972

Product	Weight %	Ta_2O_5 %	Sulphides Vol %	Ilmenite Vol %
Coarse cl conc	0.128	49.40	6.0	6.0
Scav conc #1	0.035	44.48	11.0*	N.A.
Scav conc #2	0.041	53.11	6.0*	N.A.
Final conc (calcd)	0.204	49.30	6.8	-

* Determined with the Quantimet by Dr. W. Petruk, Mineral Sciences Division.

Rejection of Sulphides and Ilmenite from Tantalum Concentrates by Flotation and Magnetic Separation

The mineralogical examination of the tantalum concentrates, obtained from the ore of November 18, 1971, and containing a lot of finely disseminated tantalum-bearing minerals, sulphides, and ilmenite, showed that 12 to 22% of these concentrates were composed of sulphides, mainly arsenopyrite, and 3 to 16% of ilmenite (Table 6, Test 9).

There was insufficient ore to produce enough material to do the flotation of sulphides and the magnetic separation of ilmenite, so it was suggested that the company conduct this test work at the mine on the fresh samples and remove sulphides by flotation using a xanthate and a frother as flotation reagents. This should not present any problem because the tantalum-bearing minerals do not respond to the collecting effect of xanthates. The removal of ilmenite from the concentrates is usually done by a high-intensity magnetic separator.

Most of the arsenopyrite and ilmenite remain in the cleaner middlings during the cleaner tabling operation at the Bernic Lake concentrator and thereby contaminate the final tantalum concentrate, so the company used these cleaner middlings, assaying 19.49% Ta_2O_5, as the initial material to conduct flotation and high-intensity magnetic separation tests for the rejection of arsenopyrite and ilmenite. The results[8] showed that about 87% of the arsenopyrite and about 78% of the ilmenite were rejected from the middlings by these two methods and that practically no tantalum was lost. Potassium amyl xanthate and methylisobutynol were used as flotation reagents and high-intensity magnetic or electrostatic separators for ilmenite rejection. Based on these results and the mineralogical results given in Table 6, Test 9 (Ore: November 18, 1971), the calculated grades of the tantalum concentrates from Table 5 of this test are given in Table 11.

TABLE 11

Grades of Tantalum Concentrates before and after Rejection of Sulphides and Ilmenite, Test 9: Ore: November 18, 1971

Product	% Weight		% Ta_2O_5 Assay		% Volume Sulphides		% Volume Ilmenite	
	Before	After	Before	After	Before	After*	Before	After**
Coarse cl conc	0.125	0.098	40.18	51.00	22.0	3.0	4.0	0.9
Scav cl conc #1	0.050	0.035	31.45	44.70	20.0	2.7	16.0	3.5
Scav cl conc #2	0.021	0.017	43.02	56.20	12.0	1.5	3.0	0.6
Final conc (calcd)	0.196	0.150	38.26	50.00	20.4	2.9	8.9	1.3
Sulphide conc		0.046						

* Calculated at 87% rejection.
** Calculated at 78% rejection.

The calculated results from Table 5 after rejection of the sulphides and ilmenite are recorded in Table 12.

TABLE 12

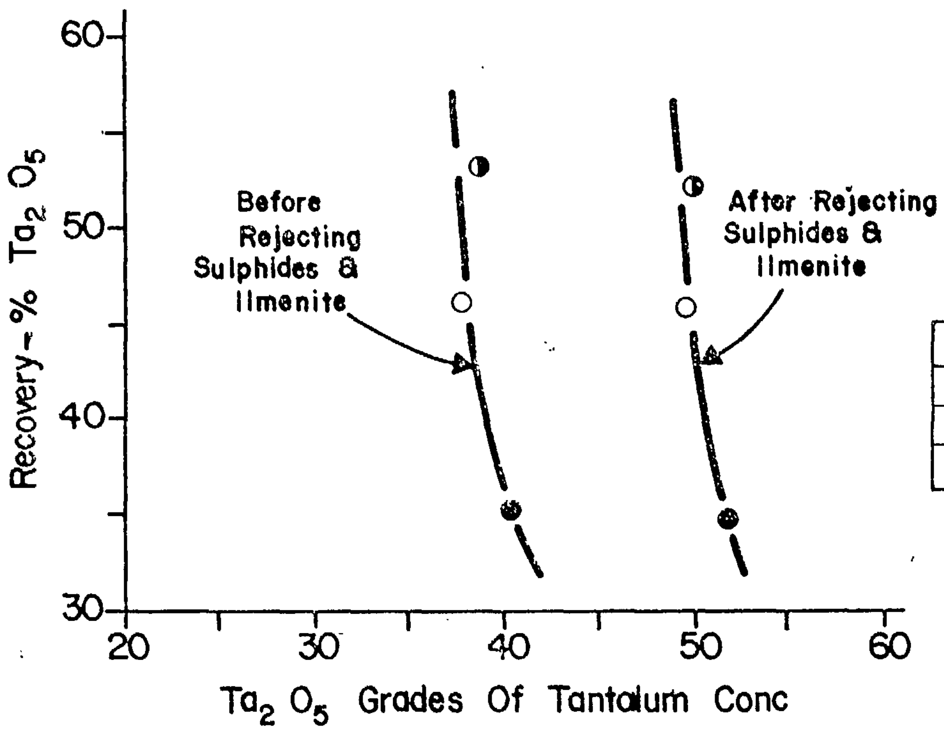
Calculated Results of Test 9 from Table 5
after Sulphide-Ilmenite Rejection,
Ore: November 18, 1971

Grind-Mesh			Product	%	% Ta_2O_5	
Prim	Sec	Tert		Weight	Assay	Distn
	-65		Coarse cl conc (-65 mesh)	0.098	51.00	35.4
		-200	Scav conc #1 (+325 mesh)	0.035	44.78	11.1
		-200	Scav conc #2 (-325 mesh)	0.017	56.20	6.5
			Final conc (calcd)	0.150	50.00	53.0
			Sulphide conc (calcd)	0.046	Tr.	-
		-200	Scav midd #1 (+325 mesh)*	0.126	3.76	3.3
		-200	Scav midd #2 (-325 mesh)*	0.196	8.10	11.3
			Comb middlings*	0.322	6.40	14.6
-35			Rougher tail (+200 mesh)	43.571	0.005	1.6
		-200	Scav tail #1 (+325 mesh)	22.501	0.005	0.8
		-200	Scav tail #2 (-325 mesh)	33.410	0.127	30.0
-35			Feed (calcd)	100.000	0.141	100.0

* Circulating load.

Table 11 and Table 12 show that a tantalum concentrate of 38.26% Ta_2O_5 grade from the refractory ore of November 18, 1971, containing high amounts of sulphides and ilmenite, could be upgraded by flotation and magnetic separation to about 50% Ta_2O_5 without tantalum losses.

The relationship between Ta_2O_5 grades and recoveries of the tantalum concentrates before and after the rejection of sulphides and ilmenite from the concentrates from the ore of November 18, 1971, are presented graphically in Figure 13.



Ta Conc	Symbol
Minus 65-mesh	●
200 to 325-mesh	○
Minus 325-mesh	◐

Figure 13. EFFECT OF SULPHIDES AND ILMENITE REJECTION ON Ta_2O_5 GRADE AND RECOVERY OF TANTALUM CONCENTRATE.
ORE: NOVEMBER 18, 1971, TEST 9

The sulphide and ilmenite contents in the tantalum concentrates from the ore of June 1 and 2, 1972, Table 10, Test 10, were much smaller than in the concentrates from the ore of November 18, 1971 (Table 6).

After rejection of sulphides by flotation (without the rejection of ilmenite), the calculated grades of the tantalum concentrates from the June ore of Test 10, Table 10, are given in Table 13.

TABLE 13

Grades of Tantalum Concentrates before and after
Rejection of Sulphides, Test 10, Ore: June 1 and 2, 1972

Product	% Weight		% Ta_2O_5 Assay		% Volume Sulphides	
	Before	After	Before	After*	Before	After*
Coarse cl conc	0.128	0.123	49.40	51.40	6.0	0.8
Scav conc #1	0.035	0.031	44.48	50.40	11.0	1.4
Scav conc #2	0.041	0.036	53.11	60.60	6.0	0.8
Final conc (calcd)	0.204	0.190	49.30	52.65	6.8	0.9
Sulphide conc (calcd)		0.014				

* Calculated at 87% rejection.

The calculated results of Test 10, after the rejection of the sulphides and based on the results from Table 13, are recorded in Table 14.

TABLE 14

Calculated Results of Test 10 from Table 9
after Sulphides Rejection; Ore - June 1 and 2, 1972

Grind-Mesh			Product	%	% Ta_2O_5	
Prim	Sec	Tert		Weight	Assay	Distn
	-65		Coarse cl conc (-65 mesh)	0.123	51.40	31.8
		-200	Scav conc #1 (+325 mesh)	0.031	50.40	7.8
		-200	Scav conc #2 (-325 mesh)	0.036	60.60	10.9
			Final conc (calcd)	0.190	52.65	50.5
			Sulphide conc (calcd)	0.014	Tr.	-
		-200	Scav midd #1 (+325 mesh)*	0.242	3.65	4.4
		-200	Scav midd #2 (-325 mesh)*	0.170	6.37	5.7
			Comb middlings*(calcd)	0.412	4.80	10.1
-35			Rougher tail (+200 mesh)	50.104	0.056	13.8
		-200	Scav tail #1 (+325 mesh)	21.910	0.002	0.2
		-200	Scav tail #2 (-325 mesh)	27.370	0.185	25.4
-35			Feed (calcd)	100.000	0.20	100.0

* Circulating load

Table 13 and Table 14 showed that the 49.30% Ta_2O_5 final concentrate was upgraded only to 52.65% Ta_2O_5 by rejection of sulphides because only a few sulphides were contained by the final tantalum concentrate.

The relationship between Ta_2O_5 grades and recoveries of the tantalum concentrates before and after the rejection of sulphides from the concentrates from the ore of June 1 and 2, 1972, are presented graphically in Figure 14.

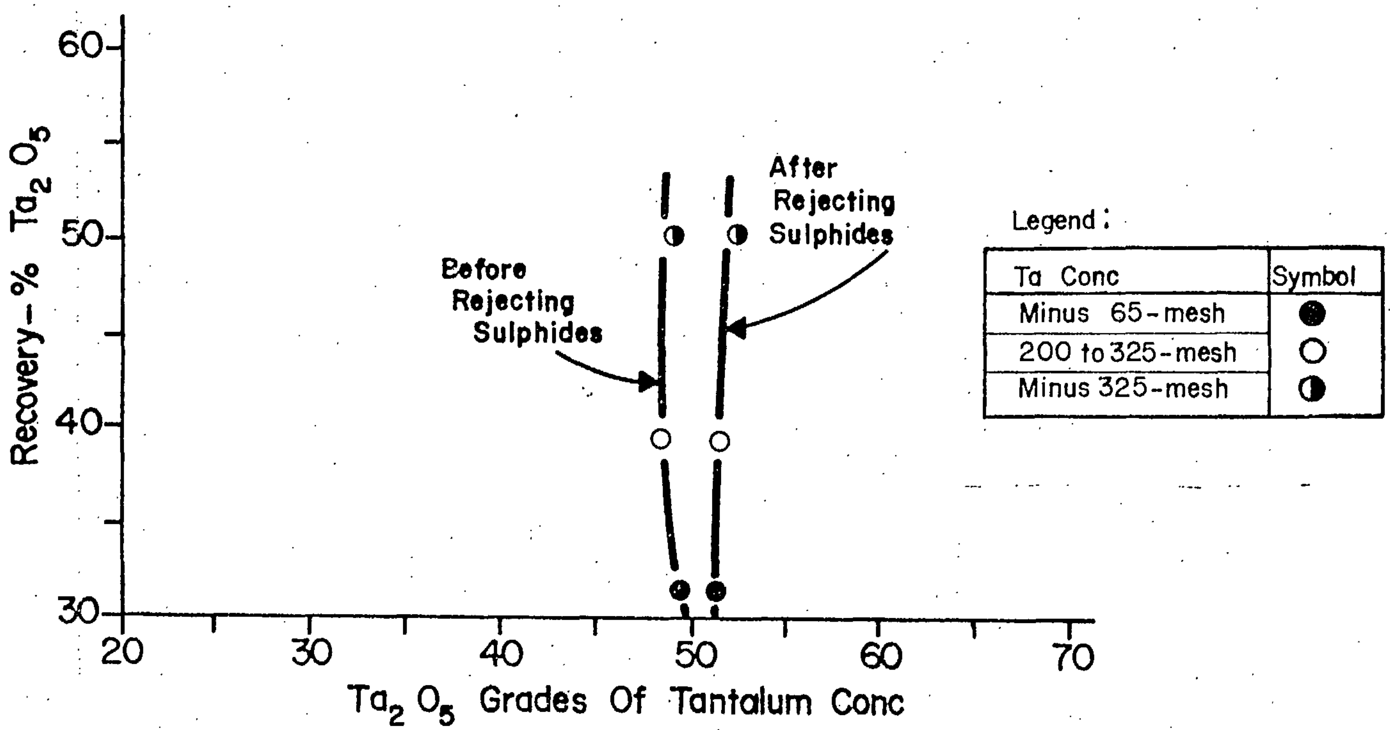


Figure 14. EFFECT OF SULPHIDES REJECTION ON Ta_2O_5 GRADE AND RECOVERY OF TANTALUM CONCENTRATES. ORE: JUNE 1 & 2, 1972, TEST 10

DISCUSSION

To recover finely disseminated minerals from any ore, a fine grind of the ore is required to liberate these minerals and make them amenable to concentration. The required fineness of grind depends on the degree of dissemination of the minerals in the ore.

Ore Sample: November 18, 1971

This was the most refractory ore for the recovery of tantalum minerals that has been received from the property.

The results of Test 4, in which this ore was ground to minus 100 mesh and tabled without sizing, as illustrated by the flowsheet in Figure 1, showed that 62.8% of the weight was rejected by the rougher tabling, with only 5.3% tantalum losses, to recover 94.7% of the tantalum in the rougher concentrate (Table 4).

However, upgrading of the rougher concentrate,by cleaner tabling and tabling of the fines produced during this single (100-mesh) grind, showed that 32.2% of the tantalum in the ore was lost to the Deister slime-deck tailing. This indicates that the Deister slime-decks could not effectively recover the slimes by tabling the un-sized ore.

The results of Test 5, in which the ore was ground to minus 48 mesh and split into two fractions on an 80-mesh screen for separate rougher table concentration, as illustrated by the flowsheet in Figure 2, showed that 72.6% of the weight was rejected with a loss of 7.0% of the tantalum and a rougher recovery of 93%. Upgrading the rougher concentrates by the procedure outlined in this flowsheet produced a final concentrate assaying 31.3% Ta_2O_5 with a 59% recovery. Here again the major loss of tantalum was in the slime-deck tailing which contained 21.3% of the tantalum in the ore.

In Test 9, the ore was ground to minus 35 mesh and treated on single-stage rougher and cleaner tables. This was similar to current Bernic Lake operation, except that the rougher middlings and cleaner tailings were reground to 200 mesh and separated into plus and minus 325-mesh fractions for separate slime-deck tabling as illustrated by the flowsheet in Figure 3 (p. 18). This procedure gave quite satisfactory results on this type of ore, recovering 60% of the tantalum in a 38.26% Ta_2O_5 concentrate (Table 5).

The tantalum concentrates obtained from this test, as well as from Test 4 and Test 5, contained high amounts of arsenopyrite and ilmenite (Table 6). If most of the sulphides were removed from the concentrate by flotation, a required grade of 50% Ta_2O_5 should be obtained from this ore with about 60% tantalum recovery.

It can be expected that the tantalum concentrates from Test 4 (30.0% Ta_2O_5 grade, 62.5% recovery) and Test 5 (31.30% Ta_2O_5 grade, 65% recovery) can be upgraded from 30.0% Ta_2O_5 to 43.8% and from 31.3% Ta_2O_5 to 45% Ta_2O_5 by flotation without an appreciable loss of tantalum.

From the comparison of the results from these three procedures (Tests 4, 5, and 9, Figures 1, 2, and 3), it appears that the procedure of Test 9, Figure 3, gave the most favourable result. The application of this procedure would not require any major changes in the present operation at the Bernic Lake concentrator, but it would need the following additional operations:

(1) flotation of sulphides and, occasionally, rejection of ilmenite by a high-intensity magnetic separation;

(2) fine grinding of middlings;

(3) sizing and slime-deck tabling of ground middlings and other fines.

Ore Sample: November 25, 1971

This ore sample contained less sulphides and ilmenite than the ore of November 18, 1971. The application of the procedure as outlined in Figure 3 but without the sulphides-ilmenite rejection gave good results from this ore (over 50% Ta_2O_5 concentrate with 65 to 66% recovery) despite the presence of some sulphides and ilmenite in the concentrate (see Test 8, Table 8). If the sulphides-ilmenite minerals are rejected, the grade of this concentrate would be increased slightly. If the original tantalum grade, however, were kept below 50% Ta_2O_5, to obtain slightly higher recovery, and then upgraded by flotation and high-intensity magnetic separation to the 50% Ta_2O_5, the over-all tantalum recovery could be increased slightly.

The procedure of Test 6, Figure 8, consisting of screening the minus 48-mesh ore into two fractions, tabling each fraction, grinding the rougher middlings and cleaner tailings to minus 150 mesh, and then slime tabling gave results similar to those of Test 8 with about 66% tantalum recovery in a 50.0% Ta_2O_5 concentrate.

Ore Sample: June 1 and 2, 1972

This ore sample contained less sulphides and ilmenite than in the ore sample of November 18, 1971, but more than the ore sample of November 25, 1971. Application of the procedure outlined in Figure 3 with the recycling of scavenger middlings, produced a 50% Ta_2O_5 concentrate but the tantalum recovery would range between 55 and 56 per cent. This concentrate contained less sulphides and ilmenite than the tantalum concentrate from the ore of November 18, 1971. The rejection of sulphides by flotation from this concentrate would increase the tantalum grade to 52.65% maintaining the same recovery of 55 to 56 per cent.

CONCLUSIONS

Tantalum concentrates assaying 50% Ta_2O_5 at 53 to 66% recovery can be obtained from these refractory ores by following the current gravity concentrating process used at the Bernic Lake concentrator but with the additional treatment illustrated in the flowsheet (Figure 3). This comprises slime-deck tabling of fines and reground middlings followed by the rejection of sulphides by flotation. The slime-deck tabling would increase the over-all tantalum recovery, whereas the rejection of sulphides by flotation would increase the tantalum grade of the final gravity concentrate.

The rejection of sulphides by flotation has been already installed for this purpose at the Bernic Lake concentrator and it is operating satisfactorily.

ACKNOWLEDGEMENTS

The authors acknowledge, as contributions to this investigation, the extensive mineralogical examinations of the tantalum concentrates conducted by Dr. H. C. Harris and Dr. W. Petruk, Mineral Sciences Division.

Acknowledgements are also extended to the members of the Ore Dressing Laboratory of the Mineral Processing Sub-Division for performing grinding, screening and sample preparation, and assisting in tabling operations.

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DR/RWB/am