Dr. J. Convey

[redacted]

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DEPARTMENT OF MINES AND TECHNICAL SURVEYS

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MINES BRANCH INVESTIGATION REPORT IR 64-58

A LABORATORY AND PILOT PLANT INVESTIGATION ON A TIN ORE FROM CHARLOTTE COUNTY, NEW BRUNSWICK, FOR MOUNT PLEASANT MINES LIMITED

by

G. O. HAYSLIP

MINERAL PROCESSING DIVISION

44p

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SUMMARY OF RESULTS

Preliminary test work on samples, which varied considerably, showed that the ore is a very complex one, requiring detailed study of all phases of concentration.

Preliminary laboratory tests showed that about 60% of the tin could be recovered in table concentrates and middlings of different grades.

Pilot plant tests made to produce concentrate for smelting tests gave lower recoveries in tabling due to more tin being tied-up in flotation concentrates.

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INTRODUCTION

Location of Property

The property of Mount Pleasant Mines Limited is located about 28 miles northeast of St. Stephen, New Brunswick in Charlotte County, at latitude 45° 26'N and longitude 66°49'W. The property consists of 131 claims. There are two ore-bearing areas, known as the North zone and the South or Fire Tower zone. Diamond drilling has been done on both zones and an adit has been driven for over 4000 ft in the North zone.

Shipments

The first samples of ore from Mount Pleasant Mines Limited were received on October 30th, 1961 and consisted of two boxes of heavily-weathered surface material, and several bags of diamond drill core samples. These diamond drill core samples were labelled with diamond drill hole numbers and the footage from which they were taken. Chemical and spectrographic analyses were given for some of the samples and a general description of the mineralization, e.g., tin ore--low sulphide ore, high sulphide ore, low copper, was included. No test work was done on the weathered surface material from this shipment. Two composite samples were made from some of the drill core. Sample No. 1 was from diamond drill hole 45 and was said to be tin ore with no sulphides. Sample No. 2 was from diamond drill holes 18, 58, 60 and 61 and was said to be tin ore with some sulphides.

Sample No. 3 was received on November 27th, 1961. This sample, weighing about 1000 lb, consisted of heavily-weathered surface material.

Eight bags of ore, weighing 650 lb, were received on May 10th, 1962. This ore was said to have been taken from a mineralized area encountered in the adit being driven to the main ore zone. This ore was designated as Sample No. 4.

On October 19, 1962 and on November 16th, 1962, two carloads of ore, weighing 57 1/2 and 62 1/2 tons respectively, were received for pilot plant testing. These shipments were said to be from the main ore zone.

Purpose of Investigation

Mount Pleasant Mines Limited had obtained a property in New Brunswick which showed mineralization containing the elements: tin, zinc, copper, lead, tungsten, and molybdenum, as well as traces of many other elements. Preliminary investigations had shown that the mineralogy of the ore was very complex.

Dr. J.E. Riddell, President, Mount Pleasant Mines Limited, 30 The Driveway, Ottawa 4, Ontario, requested an investigation of the ore to determine if the valuable constituents, particularly the tin minerals, were amenable to concentration. Later a request was made for pilot plant testing to produce concentrates for subsequent smelting tests.

Sampling and Analysis

From the samples designated for testing, representative fragments of ore and gangue were selected for microscopic examination. Each sample was then crushed and sampled according to standard procedures to obtain a head sample for chemical analysis, the remainder of each sample being used for test work.

The two carload shipments of ore used in the pilot plant tests were not sampled. From daily grinding circuit feed samples, a composite analysis was calculated for each shipment.

The head analyses of different samples of ore are as follows:

TABLE 1

Analyses of Samples

	No. 1*	No. 2*	No. 3**	No. 4***
Au	trace	trace	-	-
Ag	0.04 oz/ton	0.52 oz/ton	-	-
Sn	0.50 %	1.00 %	0.56%	0.61%
Cu	-	0.50 %	0.10%	0.24%
Pb	-	0.80 %	-	0.10%
Zn	-	3.27 %	2.30%	1.68%
S	0.34 %	3.35 %	-	1.75%
Fe	-	-	-	10.53%
As	-	-	-	0.17%
Sol Sn	-	-	-	0.04%

* From Internal Report MS-AC-62-74.

** From Internal Report MS-AC-62-129.

*** From Internal Report MS-AC-62-687.

Methods of Tin Concentration

Tin concentration has been done for hundreds of years. The methods used now are similar to those used in the beginning with the addition of mechanical refinements. The basic method used to recover tin is gravity concentration.

For large mineral grains or aggregates jigging can be used. For finer grained material shaking tables are used after the material has been sized, usually by hydraulic classification. A recent change from the shaking tables is the use of Humphreys spiral concentrators. For finer sizes slime tables are used, also vanners, round tables, frames and Buckman concentrators.

Some sulphides are usually present in tin ores. If the amount is small they are usually concentrated with the tin and then removed from the gravity concentrates by flotation. If the amount of sulphides present is large, as in the Mount Pleasant ore, the usual practice is to float the sulphides away before sizing and tabling.

Many methods have been proposed for the flotation of cassiterite itself but so far no method has been commercially successful in producing high grade concentrates with good recovery. For this reason it was decided to use the conventional gravity methods of concentration in this investigation.

MINERALOGICAL EXAMINATION

Several mineralogical examinations of different ore samples and test products were made on the ore. The major studies were reported in Mines Branch Investigation Report IR 62-16, "A Mineralogical Investigation of the Ore from Mount Pleasant Mines Limited, New Brunswick", by W. Petruk, Mineral Sciences Division, April 11, 1962, and, Mines Branch Investigation Report IR 63-15, "Mineralogical Investigation of Samples from the Mount Pleasant Tin Deposit in New Brunswick", by W. Petruk, Mineral Sciences Division, February 8, 1963. Copies of these reports were submitted to all concerned.

In addition, several internal reports were made on specific problems encountered during the investigation of the ore. These will be referred to later in the discussion of the results.

RESULTS OF INVESTIGATION

The investigation done on this ore points out the complexity of the association of the different minerals and the difficulty of recovering these minerals in their respective concentrates. This investigation covered the treatment of this ore in a very general way and, points out the need for a detailed study of the different operations involved in treating this ore.

Concentration of tin in bench testing gave recoveries of about 60% of the tin in a relatively low grade concentrate. No attempt was made to produce premium grade concentrates.

Bulk flotation of the zinc-copper-lead minerals in bench tests gave recoveries of over 90% of these minerals but the separation into their respective concentrates was poor. Flotation of zinc, from a sample low in copper, gave rougher recoveries up to 90% of the zinc but it was difficult to make a suitable grade of concentrate. Only a limited number of flotation tests were made.

Pilot plant investigations on concentrating the tin produced results that were poorer than those obtained in bench tests. Best results gave a recovery of only 44% of the tin. A large amount of the tin was entrapped in the sulphide concentrates; due to the higher metal sulphide content of the pilot plant sample the proportion of sulphide concentrate was much larger than in the bench tests.

Flotation of sulphides in the pilot plant investigation gave results as good as in bench testing. Zinc concentrate produced in the pilot plant was upgraded in bench testing to over 50% zinc.

DETAILS OF INVESTIGATION

After a few preliminary tests were done on Sample No. 2, Dr. Riddell requested that work be done on Sample No. 3. Most of the sulphide flotation investigation was done on this sample as well as a few tabling tests for tin concentration. After the sample of fresh ore, called Sample No. 4, arrived, the work of tabling the tin mineral after removing the sulphides was concentrated on this sample.

In the flotation tests the frother used was Dowfroth 250 and the sulphide collectors most commonly used were Reagent 303, a potassium ethyl xanthate and Reagent 325, a sodium ethyl xanthate. These were used as being convenient and their use does not constitute endorsement by the Mines Branch.

Following considerable bench scale testing, a pilot plant investigation was made on two carload lots of ore. The purpose of this was to provide tin concentrates for smelting tests and, to confirm the results from the laboratory investigation.

Flotation of Sulphides

Several tests were done on material from No. 2 sample. The procedure was to make a differential sulphide float by first floating off the copper and lead minerals, followed by zinc flotation, and finally a scavenger float of the remaining sulphides. Details of the test were as follows:

Test 1

Reagents and Conditions

Operation	Reagents - lb/ton		Time, min	pH
Grind (62.3% -200 m)	Lime	- 1.0	15	
	Sodium cyanide	- 0.2		
	Zinc sulphate	- 0.4		
Cu-Pb conditioning	*Reagent 303	- 0.065	3	9.5
Cu-Pb flotation (1st stage)	Dowfroth 250	- 0.03	3	
	Reagent 303	- 0.035		
Cu-Pb flotation (2nd stage)	Reagent 303	- 0.05	4	
Zn conditioning	Lime	- 1.0	5	11.1
	Copper sulphate	- 0.6		
	Reagent 303	- 0.1		
Zn flotation (1st stage)	Dowfroth 250	- 0.03	3	
Zn flotation (2nd stage)	Reagent 303	- 0.05	2	
Scavenger conditioning	H_2SO_4		2	7
Scavenger flotation	Reagent 303	- 0.2	5	
	Dowfroth 250	- 0.03		

*Potassium ethyl xanthate.

TABLE 2

Results of Test 1

Product	Weight %	Analysis %				Distribution %			
		Cu	Pb	Zn	Sn	Cu	Pb	Zn	Sn
Feed (calcd)	100.0	0.50	0.73	3.00	1.05	100.0	100.0	100.0	100.0
Cu-Pb conc	5.0	5.52	12.87	17.90	2.88	55.5	87.7	29.9	13.7
Zn conc	4.8	3.28	0.34	36.90	2.00	31.6	2.7	59.1	9.2
Scavenger conc	3.8	1.00	0.44	4.56	1.72	7.7	2.7	5.8	6.2
Flotation tailing	86.4	0.03	0.058	0.18	0.86	5.2	6.9	5.2	70.9

Test 2

Reagents were changed slightly in this test in an attempt to improve the separation between the copper and zinc minerals.

Reagents and Conditions

Operation	Reagents - lb/ton		Time, min	pH
Grind (62.3% -200 m)	Lime	0.5	15	
	Zinc sulphate	0.5		
	Sodium cyanide	0.1		
	Sodium sulphite	1.0		
Cu conditioning	Reagent 303	0.05	5	7.2
Cu flotation	Frother 70	0.06	5	
Zn conditioning	Lime	1.0	5	10.3
	Copper sulphate	0.6		
	Reagent 303	0.1		
Zn flotation	Frother 70	0.03	5	
Scavenger conditioning	Reagent 303	0.20	2	9.9
Scavenger flotation	Frother 70	0.03		

TABLE 3

Results of Test 2

Product	Weight %	Analysis % Cu	Zn	Sn	Distribution % Cu	Zn	Sn
Feed (calcd)	100.0	0.50	3.17	1.07	100.0	100.0	100.0
Cu conc	5.7	5.46	23.17	3.00	62.2	41.7	15.9
Zn conc	6.0	2.40	27.28	1.95	28.8	51.7	11.2
Scavenger conc	3.2	0.74	3.60	1.65	4.8	3.8	4.7
Flotation tailing	85.1	0.025	0.11	0.86	4.2	2.8	68.2

Test 3

A bulk copper-lead-zinc concentrate was made followed by a scavenger concentrate. The bulk copper-lead-zinc concentrate was filtered, reground and refloated in an attempt to separate the copper-lead minerals from the zinc mineral.

Reagents and Conditions

Operation	Reagent - lb/ton			Time, min	pH
Grind (62.3% -200 m)	Lime	-	1.5	15	
Conditioning	Copper sulphate	-	0.3	5	10.2
Bulk flotation	Reagent 303	-	0.1	5	
	Dowfroth 250	-	0.03		
After 5 min	Reagent 303	-	0.1	5	
	Dowfroth 250	-	0.03		
Scavenger flotation	Reagent 350	-	0.2	5	
	Dowfroth 250	-	0.03		
Concentrate regrind	Zinc sulphate	-	1.0	10	
	Sodium sulphite	-	1.0		
	Sodium cyanide	-	0.4		
Flotation	Reagent 303	-	0.015	2	
	Dowfroth 250	-	0.007		
After 2 min	Reagent 303	-	0.015	2	
	Dowfroth 250	-	0.007		

TABLE 4

Results of Test 3

Product	Weight %	Analysis %				Distribution %			
		Cu	Pb	Zn	Sn	Cu	Pb	Zn	Sn
Feed (calcd)	100.0	0.45	0.80	3.16	0.98	100.0	100.0	100.0	100.0
Cu-Pb conc	2.7	6.70	18.60	22.91	2.54	40.0	62.5	19.6	7.1
Cu-Pb tailing (Zn conc)	7.9	2.52	2.11	26.26	1.97	44.5	21.3	65.5	16.3
Scavenger conc	4.3	1.20	0.96	7.03	2.07	11.1	5.0	9.5	9.2
Flotation tailing	85.1	0.02	0.11	0.20	0.78	4.4	11.2	5.4	67.4

Test 4

A bulk copper-lead-zinc concentrate and a scavenger concentrate were made as in the previous test and the concentrates obtained were passed over a Jones high-intensity, wet, magnetic separator set at 15 amps.

TABLE 5

Results of Test 4

Product	Weight %	Analysis %				Distribution %			
		Cu	Pb	Zn	Sn	Cu	Pb	Zn	Sn
Feed (calcd)	100.0	0.49	0.72	3.06	0.89	100.0	100.0	100.0	100.0
Bulk mag conc	1.8	3.65	3.36	33.20	1.76	14.3	8.3	19.6	3.6
Bulk middlings	4.2	4.28	6.59	24.42	1.66	36.7	38.9	33.7	7.9
Bulk non-mag tailing	3.7	3.70	7.57	22.05	1.45	28.6	38.9	26.8	6.1
Scavenger mag conc	0.8	1.36	0.96	18.91	2.18	2.0	1.4	4.9	1.9
Scavenger middling	2.0	1.68	0.74	8.52	2.49	6.1	1.4	5.5	5.6
Scavenger non-mag tailing	2.0	1.80	0.64	7.48	2.38	8.2	1.4	4.9	5.4
Flotation tailing	85.5	0.02	0.08	0.16	0.72	4.1	9.7	4.6	69.5

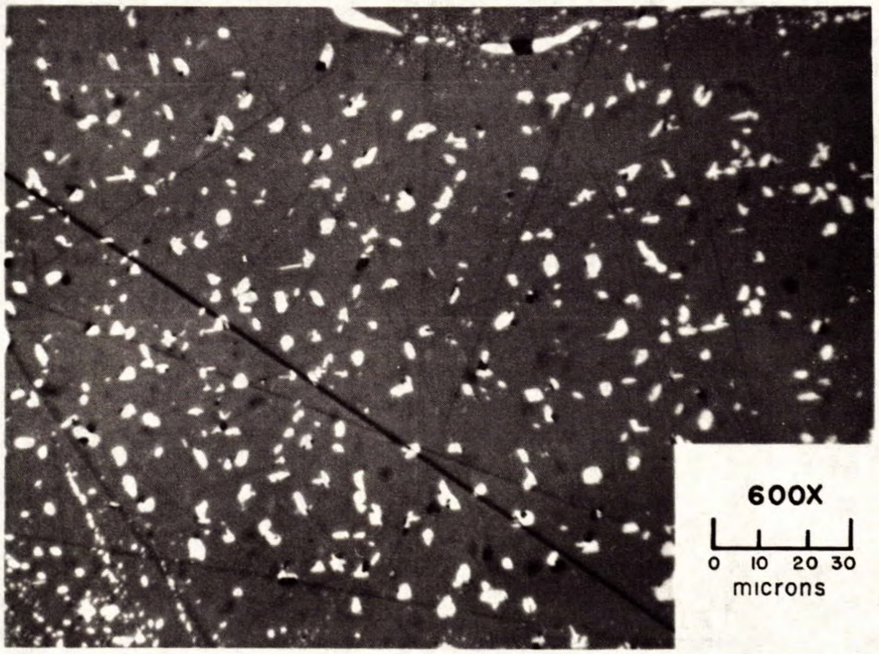

Figure 1.* Photomicrograph of a polished section showing a field of sphalerite (grey) containing tiny inclusions of chalcopyrite (white dots).

From this photomicrograph it is seen that the sphalerite contains many tiny inclusions of chalcopyrite, less than 5 microns in diameter, which cannot be liberated. With this type of ore a compromise must be made between the degree of separation of the minerals and their recoveries in their respective concentrates. It was felt that this would be a major investigation in itself and it was decided to concentrate the work on tin recovery after making a bulk sulphide float.

Test 5

Test work was changed to No. 3 sample. As the chief sulphide mineral was sphalerite, it was decided not to try to separate the small amount of chalcopyrite present and a bulk sulphide concentrate was made. The flotation tailing was screened on 200 mesh and the plus 200 mesh fraction was passed over a shaking table. The minus 200 mesh material was passed over a blanket table.

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*Figure 3 of Mines Branch Investigation Report IR 62-16.

Reagents and Conditions

Operation	Reagents - lb/ton			Time, min	pH
Grind (62.3% -200 m)	Lime	-	1.0	15	
Conditioning	Copper sulphate	-	0.3	5	7.5
	Reagent 325*	-	0.1		
Bulk flotation	Dowfroth 250	-	0.06	4	
	Reagent 325	-	0.05	3	
	Reagent 325	-	0.05	3	
	Dowfroth 250	-	0.03		
Cleaner flotation	Lime	-	1.0	2	11.5

TABLE 6

Flotation Results of Test 5

Product	Weight %	Analysis % Zn	Distribution % Zn
Feed (calcd)	100.0	2.09	100.0
Zn cl conc	3.9	36.50	68.0
Zn cl tailing	3.1	11.46	17.2
Rougher flotation tailing	93.0	0.33	14.8

The results of gravity concentration of the flotation tailing are shown in Table 7.

*Sodium ethyl xanthate.

TABLE 7

Gravity Concentration Results of Test 5

Product	Weight %	Analysis % Sn	Distribution % Sn
Feed (flot tailing calcd)	100.0	0.54	100.0
+200 m Table conc	3.4	3.63	22.2
+200 m Table tailing	44.4	0.15	13.0
-200 m Blanket conc	6.3	2.88	33.3
-200 m Blanket tailing	45.9	0.36	31.5

Test 6

Test 6 was similar to Test 5 except that a coarser grind was used and the zinc rougher concentrate was reground before cleaning. The rougher flotation tailing was screened on 200 mesh and concentrated as in the previous test.

Reagents and Conditions

Operation	Reagents - lb/ton			Time, min	pH
Grind (47.1% -200 m)	Lime	-	1.5	10	
Conditioning	Copper sulphate	-	0.3	5	9.4
	Reagent 325	-	0.1		
Rougher flotation	Dowfroth 250	-	0.06	4	
	Reagent 325	-	0.05	3	
	Reagent 325	-	0.05	3	
	Dowfroth 250	-	0.03		
Conc regrind	Lime	-	1.0	10	
Cleaner flotation	Dowfroth 250	-	0.015	3	12

TABLE 8

Flotation Results of Test 6

Product	Weight %	Analysis % Zn	Distribution % Zn
Feed (calcd)	100.0	2.20	100.0
Zn cl conc	2.8	39.29	50.0
Zn cl tailing	4.6	15.82	33.3
Rougher flotation tailing	92.6	0.40	16.7

TABLE 9

Gravity Concentration Results of Test 6

Product	Weight %	Analysis % Sn	Distribution % Sn
Feed (flot tailing calcd)	100.0	0.56	100.0
+200 m Table conc	8.2	2.58	37.9
+200 m Table tailing	55.8	0.10	10.0
-200 m Blanket conc	3.4	4.00	24.3
-200 m Blanket tailing	32.6	0.48	27.8

Test 7

A bulk concentrate was produced using the same reagents and conditions as in Test 5. The rougher flotation concentrate was reground and passed over a Jones wet magnetic separator to concentrate the sphalerite magnetically.

TABLE 10

Magnetic Concentration of Flotation Concentrate, Test 7

Product	Weight %	Analysis % Zn	Distribution % Zn
Feed (calcd)	100.0	2.17	100.0
Mag conc	1.7	44.74	35.0
Mag middling	3.2	20.00	29.5
Non-mag tailing	3.4	13.79	21.7
Rougher flotation tailing	91.7	0.33	13.8

The flotation tailing was decanted through a 325 mesh screen. The plus 325 mesh fraction was tabled and the minus 325 mesh fraction was passed over a blanket table.

TABLE 11

Gravity Concentration Results of Test 7

Product	Weight %	Analysis % Sn	Distribution % Sn
Feed (flot tailing calcd)	100.0	0.56	100.0
Table conc	3.8	3.92	26.5
Table tailing	50.5	0.12	10.8
Blanket conc	7.2	2.55	32.7
Blanket tailing	38.5	0.44	30.0

Test 8

A sulphide concentrate was floated off as in the previous tests. To make a high grade of concentrate the pH was kept at 12.0. The rougher concentrate was cleaned once at a pH of 12.0 and then filtered to remove excess reagents. The filtered concentrate then was reground with lime, sodium cyanide and water and floated again at a pH of 12.0.

Reagents and Conditions

Operation	Reagents - lb/ton		Time, min	pH
Grind (62.3% -200 m)	Lime	- 5.0	15	
Conditioning	Copper sulphate	- 0.35		12.0
	Reagent 325	- 0.1		
Rougher flotation	Pine oil	- 0.06	5	
	Reagent 325	- 0.05	5	
Cleaner flotation	Lime	- 0.5	5	12.0
Regrind	Sodium cyanide	- 0.03	15	
Recleaner flotation	Lime	- 0.2	4	12.0

TABLE 12

Flotation Results of Test 8

Product	Weight %	Analysis % Zn	Distribution % Zn
Feed (calcd)	100.0	2.13	100.0
Zn Recl conc	1.6	45.02	33.8
Zn Recl tailing	4.7	25.86	57.3
Zn cl tailing	1.5	0.90	0.4
Rougher flotation tailing	92.2	0.19	8.5

The rougher flotation tailing was deslimed on a 325 mesh screen. The plus 325 mesh fraction was tabled to produce a concentrate, middling and tailing. The minus 325 mesh fraction was elutriated in a Wade hydraulic separator to remove the minus 20 micron fraction. The minus 325 mesh plus 20 micron fraction was passed over a blanket table with the blanket concentrate being repassed to make a final concentrate and a middling product. The results of tabling and blanketing are shown in Table 13.

TABLE 13

Gravity Concentration of Flotation Tailing, Test 8

Product	Weight %	Analysis % Sn	Distribution % Sn
Feed (rougher flot tailing)	92.2	0.53*	88.0
Sands (+325 m)	67.8	0.54*	65.7
Slimes (-325 m)	24.4	0.51*	22.3
Table conc	6.3	4.06	45.7
Table middling	7.9	0.47	6.6
Table tailing	53.6	0.14	13.4
Wade overflow (-20 microns)	15.2	0.49	13.2
Blanket conc	0.4	2.88	2.1
Blanket middling	0.6	0.83	0.9
Blanket tailing	8.2	0.41	6.1

*Calculated.

As the table concentrate was low in grade, it was decided to try to upgrade it by other methods of concentration. After drying, a sample of table concentrate was passed over a Carpco High Tension Separator. The concentrate produced was passed over a Stearns High Intensity Magnetic Separator. The non-magnetic tailing was screened on 48 mesh.

TABLE 14

High Tension Concentration of Table Concentrate, Test 8

Product	Weight %	Analysis % Sn	Distribution % Sn
Feed (table conc)	6.3	4.06	45.7
Non-mag tailing (-48 m)	0.6	29.47	31.6
Non-mag tailing (+48 m)	0.1	1.56	0.4
Mag conc	0.2	3.34	1.2
High tension tailing	5.4	1.30	12.5

The table middling from Test 8 was similarly treated on a Carpco High Tension Separator.

TABLE 15

High Tension Concentration of Table Middling, Test 8

Product	Weight %	Analysis % Sn	Distribution % Sn
Feed (table middling)	7.9	0.47	6.6
High tension conc	0.1	11.00*	2.0
High tension middling	0.2	1.23	0.3
High tension tailing	7.6	0.32	4.3

*Calculated.

Test 9

A sample of minus 10 mesh ore was treated in a Denver Mineral Jig to make a concentrate and a tailing. The jig tailing was screened on a 35 mesh screen and the oversize was ground to minus 35 mesh. The minus 35 mesh product was tabled to produce a coarse concentrate and a tailing. The table tailing was screened on 200 mesh and the minus 200 mesh material was repassed over the table to produce a fine concentrate and a tailing.

TABLE 16

Jigging and Tabling Results of Test 9

Product	Weight %	Analysis % Sn	Distribution % Sn
Feed (calcd)	100.0	0.51	100.0
Jig conc	8.7	1.97	33.7
Coarse table conc	1.9	1.68	6.3
Coarse table tailing	29.2	0.27	15.5
Fine table conc	7.3	1.86	26.8
Fine table tailing	52.9	0.17	17.7

The remaining small scale tests were done on Sample No. 4, which was taken from a mineralized zone encountered in the adit being driven to the main ore zone.

Test 10

As this sample contained a large amount of chlorite, an iron-bearing, magnesium aluminum silicate, it was thought that a separation might be made using a Jones high intensity magnetic separator. A sample of ore was stage ground through 100 mesh and the sulphides were removed by flotation. The flotation tailing was then passed through the Jones separator to make a magnetic concentrate, a middling, and a non-magnetic tailing.

Reagents and Conditions for Flotation

Operation	Reagents - lb/ton		Time, min	pH
Conditioning	Copper sulphate	- 0.35	10	6.8
Flotation	Reagent 325	- 0.1	6	
	Dowfroth 250	- 0.03		

TABLE 17

Results of Test 10

Product	Weight %	Analysis % Sn	Distribution % Sn
Feed (calcd)	100.0	0.55	100.0
Sulphide conc	5.7	0.88	9.1
Magnetic conc	35.6	0.39	25.2
Middling	31.0	0.61	34.3
Non-magnetic tailing	27.7	0.62	31.4

Test 11

Approximately 12,000 g of ore was stage ground, in 2000 g batches, to minus 50 mesh and the sulphides were floated off using reagents and conditions identical to Test 10. The flotation tailing was then sized hydraulically into 8 different size fractions. Each fraction was tabled on a laboratory size Deister shaking table. The 5 coarser fractions were tabled using a sand deck; the 3 finest fractions were tabled using a slime deck.

Screen tests were done on the 4 coarser sand fractions.

TABLE 18

Size Distribution Table Tailings, Test 11

Mesh	No. 1 Fraction	No. 2 Fraction	No. 3 Fraction	No. 4 Fraction
+ 65	19.8	0.3	0.1	-
-65 + 100	50.9	3.7	0.2	-
-100 + 150	25.6	48.5	4.2	-
-150 + 200	3.3	41.5	57.4	5.0
-200 + 325	0.4	5.7	37.2	78.0
-325	-	0.3	0.9	17.0
Total	100.0	100.0	100.0	100.0

TABLE 19

Results of Test 11

Product	Weight %	Analysis % Sn	Distribution % Sn
Feed (calcd)	100.00	0.52	100.0
Sulphide conc	6.23	0.98	11.8
No. 1 Table conc	0.47	27.24	24.5
No. 1 Table middling	3.35	2.75	17.6
No. 1 Table tailing	28.23	0.14	7.6
No. 2 Table conc	0.06	37.60	4.4
No. 2 Table middling	1.05	2.86	5.7
No. 2 Table tailing	13.36	0.10	2.5
No. 3 Table conc	0.07	24.32	3.3
No. 3 Table middling	0.19	1.89	0.8
No. 3 Table tailing	7.07	0.08	1.1
No. 4 Table conc	0.11	6.02	1.3
No. 4 Table tailing	3.93	0.13	1.0
No. 5 Table conc	0.29	3.77	2.1
No. 5 Table tailing	6.99	0.10	1.3
No. 6 Table conc	0.20	9.80	3.8
No. 6 Table middling	0.08	0.64	0.2
No. 6 Table tailing	4.37	0.14	1.1
No. 7 Table conc	0.22	1.46	0.6
No. 7 Table middling	0.61	0.98	1.1
No. 7 Table tailing	10.12	0.18	3.4
No. 8 Table conc	0.18	0.50	0.2
No. 8 Table middling	0.55	0.36	0.4
No. 8 Table tailing	12.27	0.18	4.2

Test 12

About 60 lb of ore was ground continuously in a small laboratory rod mill. The mill discharge was screened on a 50 mesh screen with the screen oversize being returned to the mill intermittently. After all of the ore was ground it was floated in batches at approximately 35% solids. Three stages of flotation were used to make a copper-lead float, a zinc float and a scavenger float. All rougher concentrates were cleaned once.

The rougher flotation tailing was screened successively on 100, 150 and 200 mesh screens. The minus 200 mesh material was classified hydraulically into 5 successively finer fractions. All size fractions, except the finest one, were tabled.

Reagents and Conditions for Flotation

Operation	Reagents - lb/ton		Time, min	pH
Cu-Pb conditioning	Reagent 325	- 0.05	2	6.9
Cu-Pb rougher flotation	Dowfroth 250	- 0.05	2 1/2	
Cu-Pb cleaner flotation	Nil		1 1/2	
Zn conditioning	Copper sulphate Reagent 325	- 0.4 - 0.1	3	6.9
Zn rougher flotation	Dowfroth 250	- 0.03	4	
Zn cleaner flotation	Lime	- 1.5	2	10.3
Scavenger conditioning	Reagent 350*	- 0.1	2	7.1
Scavenger rougher flotation	Pine oil	- 0.03	5	
Scavenger cleaner flotation	Nil		3	

*Potassium amyl xanthate.

TABLE 20

Results of Test 12

Product	Weight %	Analysis % Sn*	Cu	Zn	Pb	Distribution % Sn
Feed	100.0	0.56	0.24	1.68	0.10	100.0
Flotation						
No. 1 sulphide cl conc	0.45	0.58	13.55	12.95	0.59	0.5
No. 1 sulphide cl tailing	1.13	0.56	5.75	13.01	0.73	1.1
No. 2 sulphide cl conc	1.03	0.77	3.40	38.76	0.14	1.4
No. 2 sulphide cl tailing	1.93	0.66	1.48	7.42	0.17	2.2
No. 3 sulphide cl conc	1.62	1.44	3.50	24.48	0.12	4.1
No. 3 sulphide cl tailing	1.09	1.62	1.38	4.85	0.12	3.1
Tailing						
+100 m table conc	0.22	38.86	-	-	-	15.2
+100 m table midd. A	0.49	10.75	-	-	-	9.3
+100 m table midd. B	2.27	1.28	-	-	-	5.2
+100 m table tailing	38.98	0.15	-	-	-	10.4
+150 m table conc	0.03	44.36	-	-	-	2.4
+150 m table midd. A	0.09	7.52	-	-	-	1.2
+150 m table midd. B	1.56	0.31	-	-	-	0.8
+150 m table tailing	4.30	0.09	-	-	-	0.7
+200 m table conc	0.05	48.27	-	-	-	4.3
+200 m table midd. A	0.18	6.66	-	-	-	2.1
+200 m table midd. B	0.14	0.76	-	-	-	0.2
+200 m table tailing	8.52	0.09	-	-	-	1.4
No. 1 fine table conc	0.10	48.42	-	-	-	8.6
No. 1 fine table middling	0.15	4.93	-	-	-	1.3
No. 1 fine table tailing	2.91	0.18	-	-	-	0.9
No. 2 fine table conc	0.04	44.70	-	-	-	3.2
No. 2 fine table middling	0.08	12.25	-	-	-	1.7
No. 2 fine table tailing	6.34	0.18	-	-	-	2.0
No. 3 fine table conc	0.04	10.70	-	-	-	0.8
No. 3 fine table middling	0.11	3.04	-	-	-	0.6
No. 3 fine table tailing	2.84	0.27	-	-	-	1.4
No. 4 fine table conc	0.43	1.96	-	-	-	1.5
No. 4 fine table tailing	2.35	0.17	-	-	-	0.7
No. 5 fine fraction	20.53	0.32	-	-	-	11.7

*Calculated.

Test 13

About 75 lb of ore was ground to minus 80 mesh in a laboratory rod mill using the same procedure as in Test 12. After grinding the ore, batch flotation was done at 35 per cent solids to make a copper-lead concentrate, a zinc concentrate and a scavenger concentrate. The rougher copper-lead and zinc concentrates were cleaned twice in an attempt to decrease the amount of tin in the final flotation concentrates.

The rougher flotation tailing was classified hydraulically into 6 fractions. The 5 coarser fractions were tabled.

TABLE 21

Results of Test 13

Product	Weight %	Analysis % Total Sn	Cu	Zn	Sol Sn	Distribution % Total Sn
Feed	100.0	0.59*	0.24	1.68	0.04	100.0
Flotation						
No. 1 sulphide conc	0.22	0.87	12.50	16.47	0.46	0.3
No. 1 sulphide recl tail	0.17	0.76	4.20	11.22	0.14	0.2
No. 1 sulphide cl tail	0.36	0.32	0.23	2.19	0.03	0.2
No. 2 sulphide conc	4.18	1.44	-	33.92	0.75	10.1
No. 2 sulphide recl tail	0.29	0.78	-	6.89	0.40	0.4
No. 2 sulphide cl tail	0.79	0.43	-	1.53	0.05	0.6
No. 3 sulphide conc	1.41	3.08	-	-	0.34	7.3
No. 3 sulphide cl tail	1.23	0.70	-	-	0.05	1.5
Tabling						
No. 1 table conc	0.21	47.88	-	-	-	16.9
No. 1 table middling	0.78	6.08	-	-	-	8.0
No. 1 table tailing	11.61	0.23	-	-	-	4.5
No. 2 table conc	0.28	6.58	-	-	-	3.1
No. 2 table middling	0.53	1.60	-	-	-	1.4
No. 2 table tailing	8.10	0.13	-	-	-	1.8
No. 3 table conc	0.15	24.68	-	-	-	6.2
No. 3 table middling	0.88	0.60	-	-	-	0.9
No. 3 table tailing	12.15	0.09	-	-	-	1.8
No. 4 table conc	0.11	36.66	-	-	-	6.8
No. 4 table middling	0.62	1.51	-	-	-	1.6
No. 4 table tailing	14.73	0.09	-	-	-	2.2
No. 5 table conc	0.63	3.71	-	-	-	3.9
No. 5 table tailing	5.60	0.09	-	-	-	0.8
No. 6 slimes fraction	34.97	0.33	-	-	-	19.5

*Calculated.

PILOT PLANT INVESTIGATION

Two shipments of ore weighing 57 1/2 and 62 1/2 tons, were received for pilot plant tests. Each shipment was treated separately but, except for minor changes, the same flowsheet was used in both tests.

The general flowsheet was to grind the ore in a rod mill, sizing it by means of a Dutch State Mines (DSM) curved screen which was in closed circuit with the rod mill. Most of the sulphides were floated off in three separate rougher concentrates and the rougher flotation tailing was sized by means of a 5-compartment Dorr Hydrosizer. The sizer overflow was fed to a series of 3 settling cones of increasing diameter with the overflow of the third cone going to two cyclones in series. The overflow from the second cyclone went to waste. All flotation products, sizer products, settling cone and cyclone sized products were stored separately for further treatment.

The above flowsheet was followed for the second test except that the sizer overflow went directly to 3 cyclones in series instead of to the settling cones. Each cyclone produced a sized product and the overflow from the third cyclone went to waste. No adjustments were made to the cyclones to obtain their best operating conditions. The main objective was to keep the system in balance.

The product from the DSM screen made at the beginning of the first test was too coarse, and, after a few days operation, a finer screen was installed. This screen made a 50 mesh product and was used for the remainder of the investigation.

During each test the ore was ground, the sulphides were floated, and the tailings were sized on a continuous basis. The daily runs were made for periods of 8 to 12 hours until all of the ore had been processed.

The coarser size fractions from each test were treated separately on a No. 14 Deister Diagonal Deck shaking table equipped with a sand deck. Tabling operations were done on a continuous basis, each fraction usually taking several hours. Some of the fractions required several days to complete.

The six coarser sized fractions from the first pilot plant test were tabled to produce a concentrate, a middling, and a tailing. The tailing was allowed to go to waste as the amount of material involved was too large to retain.

During the tabling of the first coarse size fraction, grab samples of the table tailing were taken and assayed. Although the results were high, they corresponded to the values obtained in the laboratory tests. It was thought that the higher losses in the coarser fractions were due to unlocked particles of cassiterite lost in the gangue. Due to the rush to get the tabling done before the second shipment of ore arrived, all of the other tailing samples were collected and assayed at the end of the tests. It was found then that the tailing values on these finer fractions were much higher than in the laboratory tests.

After floating and sizing the ore from the second shipment, and before tabling the fractions, several tests were made to improve the table operation. Minor changes in operating techniques were made and tests were done on the sized fractions retained when the coarse DSM screen was used at the start of the test work on the first shipment. After some adjustments were made the results obtained checked those obtained in laboratory tests.

To be as certain as possible of obtaining the best results when tabling the fractions from the second shipment, a sample of a size fraction from the first lot mentioned above was tabled and an assay of the tailing obtained. When the result was satisfactory, the corresponding size fraction made on the second shipment was tabled without any change in the adjustments. In spite of these precautions, the results obtained from tabling the fractions from the second shipment were poorer than those from the first shipment.

Since the table used was equipped with a sand deck, it was felt that it could not be used to treat any of the finer material. Accordingly, a pilot plant model of a Buckman concentrator was made. This machine is a type of blanket table covered with a rubberized cloth having a waffle texture which forms a large number of pockets in which the heavy minerals are deposited. The model made was hand operated but commercial models are automated to work on a feed on-feed off-wash-discharge cycle. These machines are presently being made commercially to recover fine heavy minerals.

Tests were done, using this machine on fraction 7 of each shipment of ore. Blanket table results on the ore from the first shipment were better than those on the ore from the second shipment.

The remaining size fractions were considered to be too fine for practical purposes and so were sampled only to make a metallurgical balance.

Pilot Plant Test 1

The ore was crushed to minus 1/4 in. and fed to the rod mill at an average rate of 1000 lb per hour. This feed rate varied between 960 and 1020 lb per hour. Total time of operation was 108 hours for a total calculated feed weight of 110,040 lb.

The rod mill discharge went to a DSM screen with the screen oversize retreated by a rake classifier in an attempt to wash out any residual fines not removed by the screen. The classifier sands were returned to the rod mill. The combined screen undersize and the classifier overflow having a size distribution as shown in Table 21 went to a small conditioner and then to flotation at 30% solids.

TABLE 22

Size Distribution of Flotation Feed, Pilot Plant Test 1

Total Feed		-200 Mesh Fraction	
Mesh	Wt %	Micron	Wt %
+ 48	1.3	+ 56	7.2
-48 + 65	6.9	-56 + 40	10.0
-65 + 100	12.1	-40 + 28	8.3
-100 + 150	11.3	-28 + 20	6.7
-150 + 200	9.4	-20 + 14	4.9
-200	59.0	-14 + 10	3.6
		-10	18.3

Flotation was divided into 3 stages. In the first stage an attempt was made to float off copper and lead with starvation quantities of reagents. In the second stage the pulp was conditioned for zinc flotation and most of the zinc was floated off. In the third stage a scavenger float was made in an attempt to remove the remaining sulphides. All three rougher concentrates were cleaned once without additional reagents.

Reagents and Conditions

Operation	Reagents - lb/ton		Time, min	pH
Grind (59.0% -200 m)	Sodium cyanide	- 0.1		
	Zinc sulphate	- 0.5		
Cu-Pb conditioning	Reagent 325	- 0.05	4	6.8
Cu-Pb flotation	Dowfroth 250	- 0.005	4	
Zn conditioning	Copper sulphate	- 0.5	10	6.9
	Reagent 325	- 0.1		
Zn flotation	Lime	- 1.5		9.7
	Dowfroth 250	- 0.01		
Scavenger flotation	Reagent 350	- 0.1		
	Dowfroth 250	- 0.003		

TABLE 23

Results of Sulphide Flotation, Pilot Plant Test 1

Weight, lb	Product	Weight %	Analysis %				Distribution %			
			Sn	Cu	Pb	Zn	Sn	Cu	Pb	Zn
91,380	Feed (calcd)	100.0	0.66	0.24	0.49	2.46	100.0	100.0	100.0	100.0
2,605	Cu-Pb conc	2.9	1.40	3.65	14.58	23.56	6.0	43.1	85.8	27.7
4,856	Zn conc	5.3	1.86	2.12	0.62	29.78	14.9	45.5	6.7	64.1
845	Scavenger conc	0.9	2.35	1.10	0.61	8.16	3.3	4.1	1.0	3.0
83,074	Flotation tailing	90.9	0.55	0.02	0.035	0.14	75.8	7.3	6.5	5.2

The chemical analyses given in Table 23 were made on composite samples of the different products from each daily run. The feed weight was obtained by taking timed samples of the mill feed and calculating the total weight for each run. Concentrate weights were obtained by collecting all of each concentrate in barrels, weighing each barrel and correcting for tare and moisture content. Tailing weight was obtained by difference.

An attempt was made to upgrade the zinc concentrate. A sample of the final zinc concentrate from one of the daily mill runs was filtered and divided into two parts, A and B. Part A was ground through 200 mesh and then cleaned twice with lime being added before each cleaning stage to keep the pH at 12. Part B was ground to minus 200 mesh with sufficient lime to keep the pH at 12. The ground pulp was then cleaned twice with an addition of 0.01 lb of sodium cyanide per ton of cleaner feed before each cleaning stage.

TABLE 24

Laboratory Cleaning of Zn Concentrates

Product	Weight %	Analysis %			Distribution %		
		Zn	Cu	Sn	Zn	Cu	Sn
Feed A (calcd)	100.0	28.25	1.76	2.07	100.0	100.0	100.0
Recleaner conc	42.7	48.66	2.94	1.08	73.5	71.1	22.3
Recleaner tailing	10.5	38.10	2.75	1.54	14.2	16.4	7.8
Cleaner tailing	46.8	7.42	0.47	3.09	12.3	12.5	69.9
Feed B (calcd)	100.0	28.84	1.78	2.01	100.0	99.9	100.0
Recleaner conc	31.7	50.50	2.88	0.96	55.5	51.4	15.2
Recleaner tailing	10.4	44.12	2.94	1.28	15.9	17.2	6.6
Cleaner tailing	57.9	14.23	0.96	2.71	28.6	31.3	78.2

The flotation tailing was sized into 10 fractions; 5 from the Dorr Hydrosizer, 3 from the settling cones, and 2 from cyclones. The 5 fractions from the sizer and the underflow from the first settling cone were tabled starting with the coarsest size on the Deister table. The table tailings were sampled and a screen test was made on each sample.

TABLE 25

Size Distribution of Table Tailings, Pilot Plant Test 1

Mesh	Weight %					
	No. 1 Fraction	No. 2 Fraction	No. 3 Fraction	No. 4 Fraction	No. 5 Fraction	No. 6 Fraction
+ 35	1.8	-	-	-	-	-
-35 + 48	14.3	5.3	2.8	-	-	-
-48 + 65	47.8	48.9	21.9	10.8	4.8	-
-65 + 100	31.1	39.5	44.3	37.3	25.1	0.4
-100 + 150	3.9	5.9	25.0	38.1	41.1	1.0
-150 + 200	0.6	0.4	5.2	12.9	26.6	12.8
-200	0.5	-	0.8	0.9	2.4	85.8
Total	100.0	100.0	100.0	100.0	100.0	100.0

TABLE 26

Results from Tabling Size Fractions of Flotation Tailing, Pilot Plant Test 1

Product	Weight lb	Weight %	Analysis % Sn	Distribution % Sn
Feed (calcd)	51,151.5	55.98	0.60	50.6
No. 1 Table conc	49.2	0.05	37.29	2.9
No. 1 Table middling	203.0	0.22	2.65	0.9
No. 1 Table tailing	6,255.0	6.84	0.31	3.2
No. 2 Table conc	25.2	0.03	45.03	2.1
No. 2 Table middling	101.0	0.11	8.79	1.5
No. 2 Table tailing	5,098.0	5.58	0.19	1.7
No. 3 Table conc	36.4	0.04	48.00	2.9
No. 3 Table middling	112.0	0.12	6.93	1.2
No. 3 Table tailing	8,294.0	9.08	0.24	3.3
No. 4 Table conc	26.9	0.03	54.17	2.4
No. 4 Table middling	101.0	0.11	6.19	1.1
No. 4 Table tailing	5,115.0	5.60	0.21	1.8
No. 5 Table conc	42.8	0.05	49.36	3.8
No. 5 Table middling A	161.0	0.18	5.25	1.4
No. 5 Table tailing A	7,414.0	8.11	0.22	2.7
No. 5 Table middling B	145.0	0.16	8.62	2.1
No. 5 Table tailing B	3,971.0	4.35	0.24	1.5
No. 6 Table conc	130.0	0.14	24.80	5.3
No. 6 Table middling	695.0	0.76	3.18	3.6
No. 6 Table tailing A	1,153.0	1.26	0.58*	1.1
No. 6 Table tailing B	5,159.0	5.65	0.20	1.7
No. 6 Table tailing C	6,864.0	7.51	0.21	2.4

*Caused by concentrate leaking into tailing launder.

The No. 7 slime fraction was treated on the pilot plant model of the Buckman concentrator. Several tests were made using different operating conditions. Variables checked were slope of table, rate of feed, dilution, and length of feed period. Best results were obtained by feeding the concentrator at the rate of 400 g per foot of width per minute for a 5 minute period. Pulp density was 15 per cent solids and the slope of the table was 2 inches per foot.

Several batches of pulp were passed over the table and the rougher concentrates were combined and retreated with the results as shown in Figure 2.

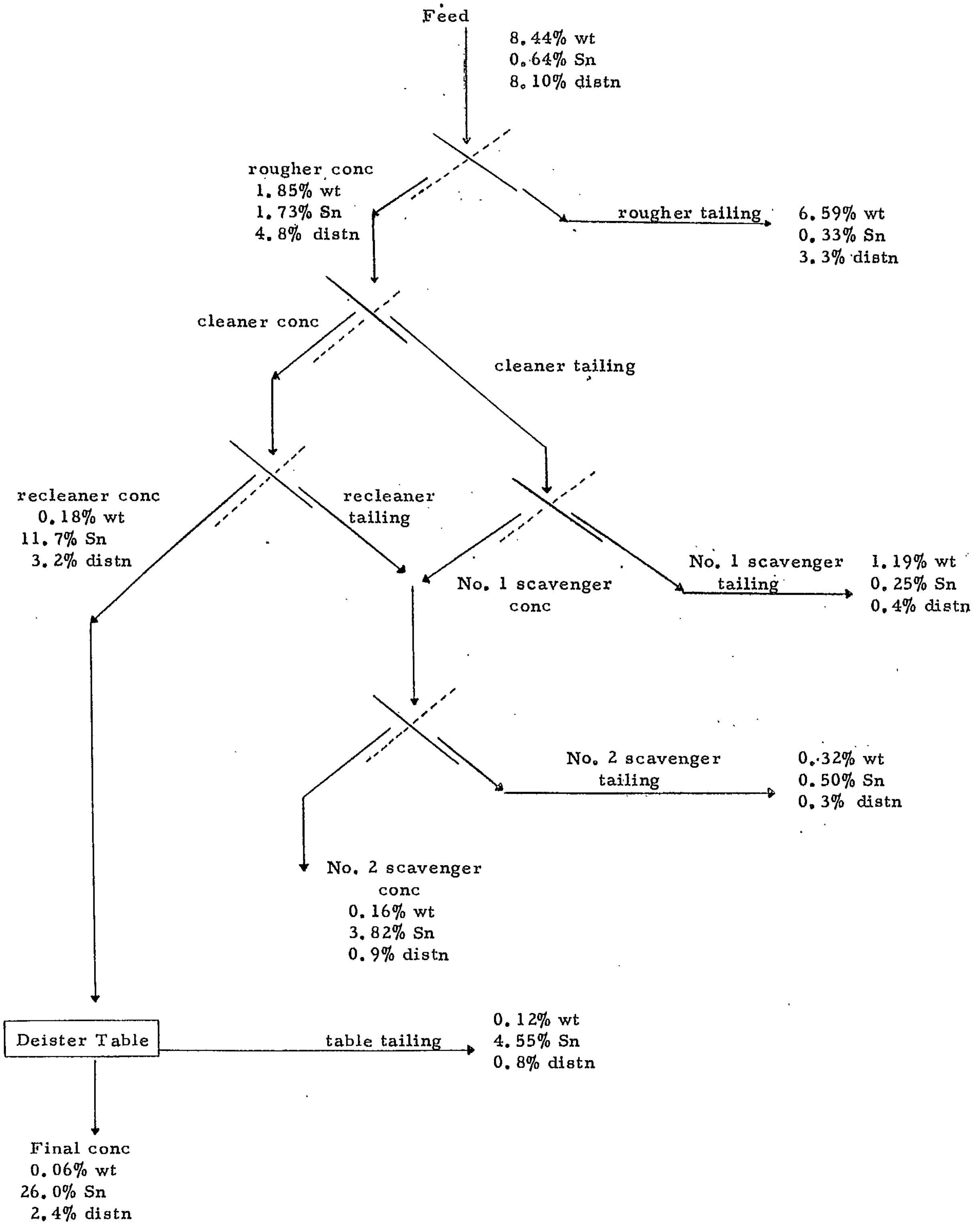

Figure 2. Treatment of slimes by Buckman concentrator.

The remaining size fractions were weighed, sampled and assayed.

TABLE 27

Slimes Distribution, Pilot Plant Test 1

Product	Weight %	Analysis % Sn	Distribution % Sn
No. 8 fraction	14.1	0.49	10.4
No. 9 fraction	2.3	0.60	2.1
No. 10 fraction	1.2	0.44	0.8
Final slimes	8.9	0.28	3.8
Total	26.5	0.43	17.1

A summary of the results of the pilot plant test, made by combining all of the concentrates, middlings, and tailings follows:

TABLE 28

Summary of Results, Pilot Plant Test 1

Product	Weight %	Analysis % (calcd) Sn	Distribution % Sn
Feed (calcd)	100.0	0.66	100.0
Sulphide conc	9.1	1.74	24.2
Table conc	0.4	36.0	21.8
Table middlings	1.9	4.59	13.5
Table tailings	62.1	0.25	23.4
Slimes	26.5	0.43	17.1

As mentioned above, after the first pilot plant test, size fractions of the ore from the coarse grind were tabled until satisfactory results were obtained. Assuming that the weights of the table products would be almost unchanged, Table 25 was recalculated using the new tailing values obtained. The concentrate and middling products were combined since the distribution of the tin released from the tailings could not be predicted. A new calculated distribution of the tin between the combined concentrate and middling and the tailing was thus obtained (Table 29).

TABLE 29

Calculated Recovery of Tin from Pilot Plant Test 1

Product	Weight %	Analysis % Sn	Distribution % Sn
Feed (calcd)	55.98	0.60	50.6
No. 1 Table conc + middling	0.27	11.85	4.9
No. 1 Table tailing	6.84	0.21	2.1
No. 2 Table conc + middling	0.14	17.14	3.6
No. 2 Table tailing	5.58	0.19	1.7
No. 3 Table conc + middling	0.16	23.75	5.7
No. 3 Table tailing	9.08	0.12	1.7
No. 4 Table conc + middling	0.14	26.00	4.2
No. 4 Table tailing	5.60	0.12	1.1
No. 5 Table conc + middling	0.39	15.90	9.4
No. 5 Table tailing	12.46	0.11	2.1
No. 2 Table conc + middling	0.90	8.11	11.1
No. 6 Table tailing	14.42	0.14	3.0

These overall results are summarized in Table 30.

TABLE 30

Summary of Calculated Results from Pilot Plant Test 1

Product	Weight %	Analysis % Sn	Distribution % Sn
Feed	100.0	0.66	100.0
Sulphides	9.1	1.74	24.2
Table conc + middlings	2.3	12.14	43.0
Table tailings	62.1	0.15	15.7
Slimes	26.5	0.43	17.1

Pilot Plant Test 2

The second test was made using the same grinding and flotation circuit as in Pilot Plant Test 1 with the same reagent balance. Although 3 separate concentrates were produced they were combined, for storage purposes, at the request of Mount Pleasant Mines Limited. The flotation tailing was sized in the Dorr sizer as before but the sizer overflow was then fed to 3 cyclones in series. These operations were carried out for a total period of 88 hours at an average feed rate of 975 pounds per hour.

By flotation, 15,393 lb of concentrate or 17.75 per cent of the feed was produced. This material had a composite analysis of tin 2.31%, copper 3.30%, lead 5.08%, zinc 28.97%, and soluble tin 0.42%. Typical analyses of a daily run are given in Table 31.

TABLE 31

Analyses of Flotation Products, Pilot Plant Test 2

Product	Analysis %			
	Sn	Cu	Pb	Zn
Flotation feed	1.20	0.53	0.63	4.13
Cu-Pb conc	1.77	4.80	11.12	33.05
Zn conc	2.37	2.60	1.05	30.80
Scavenger conc	3.39	1.28	0.83	8.36
Flotation tailing	1.07	0.03	0.036	0.22

The 5 size fractions from the Dorr sizer and the underflow of the first cyclone were treated on a Deister table as in the previous test.

TABLE 32

Results from Tabling Size Fractions of Flotation Tailing, Pilot Plant Test 2

Product	Weight lb	Weight %	Analysis % Sn	Distribution % Sn
Feed (calcd)	54,152.7	62.46	1.14	58.0
No. 1 Table conc	35.0	0.04	42.95	1.4
No. 1 Table middling	241.0	0.28	12.61	2.8
No. 1 Table tailing	3,283.0	3.79	0.73	2.3
No. 2 Table conc	42.7	0.05	39.52	1.6
No. 2 Table middling	224.0	0.26	6.87	1.5
No. 2 Table tailing	2,748.0	3.17	0.55	1.4
No. 3 Table conc	45.0	0.05	42.77	1.7
No. 3 Table middling	305.0	0.35	4.55	1.3
No. 3 Table tailing	4,660.0	5.37	0.30	1.3
No. 4 Table conc	57.50	0.07	47.38	2.7
No. 4 Table middling	857.0	0.99	3.01	2.4
No. 4 Table tailing A	3,940.0	4.54	0.28	1.1
No. 4 Table tailing B	990.0	1.14	0.33	0.3
No. 5 Table conc	79.50	0.09	51.51	3.8
No. 5 conc from middling	20.0	0.02	41.40	0.6
No. 5 middling from middling	284.0	0.33	4.47	1.2
No. 5 tailing from middling	1,425.0	1.64	0.66	0.9
No. 5 rougher tailing A	3,180.0	3.67	0.24	0.7
No. 5 rougher tailing B	4,535.0	5.23	0.25	1.1
No. 6 Table conc	187.0	0.22	48.00	8.6
No. 6 conc from middling	60.0	0.07	62.25	3.6
No. 6 middling from middling	619.0	0.71	12.58	7.3
No. 6 tailing from middling A	985.0	1.14	0.86	0.8
No. 6 tailing from middling B	1,110.0	1.28	0.96	1.0
No. 6 rougher tailing	24,240.0	27.96	0.29	6.6

The underflow from each of the cyclones, and the overflow from the final cyclone were sized by sedimentation.

TABLE 33

Size Distribution of Cyclone Products, Pilot Plant Test 2

Microns	No. 1 Cyclone Underflow	No. 2 Cyclone Underflow	No. 3 Cyclone Underflow	No. 3 Cyclone Overflow
+ 40	44.5	0.9	1.2	0.8
-40 + 29	33.2	3.7	7.6	2.9
-29 + 18	9.0	17.5	6.2	2.9
-18 + 10	5.7	33.0	26.6	3.4
-10 + 6	3.3	11.7	19.0	8.6
-6	4.3	33.2	39.4	81.4
Total	100.0	100.0	100.0	100.0

The No. 7 fraction, the underflow from the second cyclone, was treated on the Buckman concentrator. Several tests were made to determine the best conditions. The results of the best test are given in Table 34.

TABLE 34

Results of Buckman Concentrator Test on No. 7 Fraction

Product	Weight %	Analysis % Sn	Distribution % Sn
Feed (calcd)	3.28	0.76	2.1
Conc	0.73	1.44	0.9
Tailing	2.55	0.56	1.2

The No. 8 fraction was not treated.

The results of the second pilot plant test are summarized in Table 35. All gravity products are grouped according to the amount of contained tin.

TABLE 35

Summary of Results of Pilot Plant Test 2

Product	Weight %	Analysis % Sn	Distribution % Sn
Feed	100.00	1.23	100.0
Sulphides	17.75	2.31	33.4
Table conc	0.61	48.36	24.0
Table middlings	7.71	3.20	20.1
Table tailings	57.42	0.34	16.0
Slimes	16.51	0.48	6.5

A sample of table tailing from No. 5 fraction of the second pilot plant test was submitted for microscopic examination to determine the cause of the high tailing losses. The sample was separated into fractions by means of heavy liquids at specific gravities of 2.96, 3.33 and 3.70 and each fraction was examined under binocular and petrographic microscopes. The results of the investigation are reported in Table 36*.

*From Mineral Sciences Division Test Report M-63 16, "Tailing from a Gravity Separation of Mount Pleasant Ore", by W. Petruk, March 26, 1963.

TABLE 36

Mineralogy of Fractions from Table Tailing No. 5 Fraction, Pilot Plant Test 2

Fraction		Weight per cent of fraction	Per cent Sn	Minerals
Float	2.96	92.81	0.11	Quartz, with traces of chlorite, mica and sulphides. A few cassiterite inclusions are present in quartz, but no free cassiterite was observed.
Float	3.33	5.93	1.11	Chlorite, quartz, fluorite and some sphalerite and mica. Small cassiterite inclusions are present in quartz, chlorite and fluorite, but no free cassiterite was observed.
Float	3.70	1.25	3.65	Quartz, topaz, fluorite, chlorite, sphalerite and traces of siderite. Minute grains of cassiterite are present in topaz and chlorite, and larger grains are present in quartz, fluorite and chlorite. No free cassiterite was observed.
Sink	3.70	0.01		Pyrite, sphalerite, chalcopyrite, ferberite, cassiterite inclusions in quartz and fluorite.

An additional separation was made at a specific gravity of 2.80 in which 83.5% of the material reported in the float fraction and which assayed 0.10% tin.

As shown in Table 36, all of the tin observed occurs as cassiterite enclosed within gangue minerals. Oil immersion mounts of the fractions show that the cassiterite in the 2.96 float fraction occurs as minute isolated grains in quartz (see Figure 3), and that the cassiterite in the heavier fractions occurs either as larger inclusions in gangue minerals or as clusters of minute grains enclosed in gangue (see Figure 4).

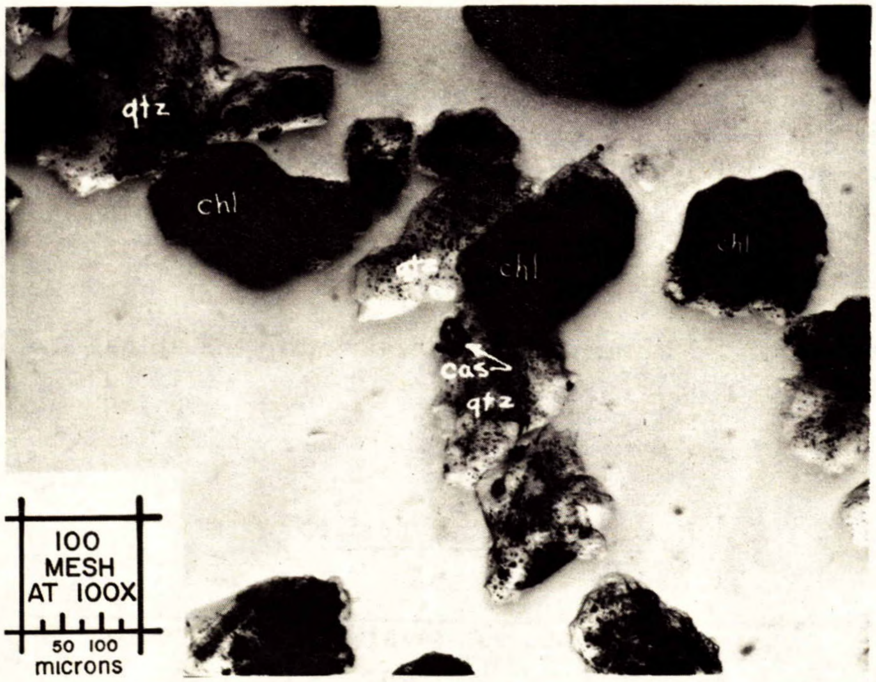

Figure 3. Photomicrograph of an oil immersion mount prepared from a 2.96 float fraction showing a cassiterite inclusion (cas) in quartz (qtz). The dark grains marked (chl) are chlorite.

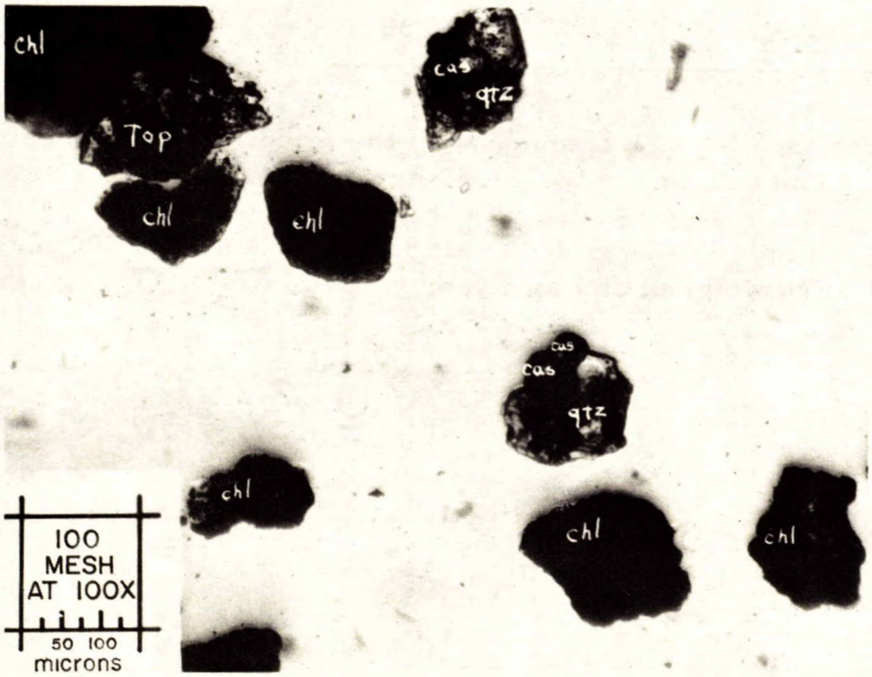

Figure 4. Photomicrograph of an oil immersion mount prepared from a 3.70 float fraction showing cassiterite inclusions (cas) in quartz (qtz). The grains marked (chl) and (top) are chlorite and topaz respectively.

Spiral Concentration Tests

The suggestion was made by Mr. A.C. King of Mount Pleasant Mines Limited that the tables might be replaced by Humphrey spirals and, under the supervision of Mr. King a series of tests were made using a spiral. The remainder of the ore was used in these spiral tests.

Spiral Test 1

Unclassified flotation tailing was fed to the spiral at a calculated feed rate of 904 lb per hour.

TABLE 37

Results of Spiral Test 1

Product	Weight %	Analysis % Sn	Distribution % Sn
Feed (calcd)	100.0	1.02	100.0
Spiral conc	15.2	3.39	51.0
Spiral middling	11.6	0.65	7.8
Spiral tailing	73.2	0.58	41.2

A similar test was made with the middling product being recirculated. The tailing assayed 0.39 per cent Sn.

A sample of the spiral tailing was screened on 200 mesh and the fractions were weighed and assayed.

TABLE 38

Size Distribution of Tailing, Spiral Test 1

Product	Weight %	Analysis % Sn	Distribution % Sn
Tailing (calcd)	100.0	0.55	100.0
+200 m	37.7	0.19	12.7
-200 m	62.3	0.77	87.3

Spiral Tests 2, 3 and 4

These tests were run on unclassified ground ore without flotation at feed rates of 829, 1613, and 1534 lb per hour respectively. Test 2 at 829 lb per hour gave the best results both in recovery and grade.

TABLE 39

Results of Spiral Test 2

Product	Weight %	Analysis % Sn	Distribution % Sn
Feed (calcd)	100.0	1.28	100.0
Spiral conc	23.8	3.08	57.0
Spiral middling	11.2	0.93	7.8
Spiral tailing	65.0	0.69	35.2

Spiral Test 5

The sized products from the Dorr sizer were combined and fed to the spiral at a calculated feed rate of 903 lb per hour.

TABLE 40

Results of Spiral Test 5

Product	Weight %	Analysis % Sn	Distribution % Sn
Feed (calcd)	100.0	1.09	100.0
Spiral conc	7.4	9.13	62.4
Spiral tailing	92.6	0.44	37.6

Spiral Test 6

A sample of the No. 3 spigot product, 48 mesh grind, from the first carload of ore was passed over the spiral. The tailing from this test contained 0.19% Sn compared to a tailing value of 0.13% Sn obtained by conventional tabling.

DISCUSSION OF RESULTS

The test work done on Mount Pleasant ore can only be considered as being indicative of the results which may be obtainable in plant operation. The test work was a preliminary study of the ore and was principally concerned with the recovery of tin. The chief purpose of the pilot plant tests was to provide concentrates for smelting tests. From the work done, it is readily apparent that the ore is very complex and requires a good deal of intensive study.

From the test work and the mineralogical investigations, it is obvious that the chalcopyrite and sphalerite are intimately associated. It was not possible to make a complete separation of these two minerals in preliminary tests and conclusive results on the recovery of copper and zinc from this ore can be obtained only after a detailed investigation.

The sphalerite was dark in colour, indicating a high iron content, however, the iron content was not as high as expected. A mineralogical study indicated that the iron content of the pure mineral was 10.9% Fe with a theoretical maximum grade of 54.9% Zn. The best grade obtained in batch cleaning of pilot plant concentrate was 50.5% Zn with a recovery of 35.5 per cent of the total zinc. With further test work it should be possible to increase the recovery.

Although the iron content was fairly high, it was not possible to separate the sphalerite by high intensity magnetic concentration. Less than half of the zinc in a flotation concentrate could be recovered by this method.

Galena appears to be free from chalcopyrite and sphalerite and with further study it should be possible to make a separate lead concentrate.

There are trace amounts of several other minerals present in the ore but the amounts vary from sample to sample and no work was done on the recovery of any of them.

The flotation of the sulphides resulted in the removal of a considerable amount of tin. In the preliminary bench tests this amounted to about 12 per cent of the total tin. In the pilot plant tests this amount increased to over 24 per cent in the first test and to over 33 per cent in the second test. A microscopic study and grain count of the different concentrates produced in the first pilot plant test* showed that over 50 per cent of the cassiterite was present as free grains, indicating poor cleaning of the flotation concentrates. It is possible that concentration of the sulphides and tin together, followed by removal of the sulphides would result in better separation of cassiterite from sulphides. This alternative step should be studied more fully.

Since the fractions from the second shipment were tabled in the same manner and under identical conditions as in the second part of the first test, it is logical to assume that the differences in the results must be caused by a difference in the distribution of the cassiterite in the second shipment.

It was stated by company officials that the second shipment was not blended in the same manner as the first shipment. To make up the tonnage of ore shipped, an extra amount of high grade ore had been included in the shipment. It has also been stated that in high grade areas there is a greater diffusion of cassiterite into the neighbouring wall rock. These conditions might account for the presence of a greater amount of middling products in the higher grade shipment and the resulting higher tailing losses. A detailed study of the distribution of cassiterite might indicate the amenability to concentration of the cassiterite in different parts of the orebody.

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*Mineral Sciences Division Internal Report MS-63-7, "Degree of Liberation of Cassiterite in Concentrates of Mount Pleasant Ore", by W. Petruk, February 6, 1963.

The production of slimes caused a considerable loss of tin, especially in the first pilot plant test. The use of cyclones instead of settling cones in the second test appeared to concentrate more material and more tin into the underflow of the first cyclone and made this fraction more amenable to concentration. More extensive work on obtaining the optimum conditions for cycloning might improve the results.

The Buckman concentrator was fairly successful in concentrating some of the fine slime from the first test but was not successful when used on finer products after the first cycloning in the second test.

No attempt was made to upgrade the table concentrates. This is usually done by floating off the residual sulphides, followed by magnetic separation of magnetic minerals. A study of the different impurities would be necessary to determine the final treatment required. No upgrading was done because it was necessary to dry and weigh the concentrates produced to obtain a metallurgical balance. Drying the concentrates resulted in oxidation of the sulphides and agglomeration of the cassiterite grains. Also, the company wished to have products of different grades so they could be blended to make any grade of concentrate desired.

The failure in the pilot plant tests to obtain results as good as were obtained in small scale laboratory tests can be attributed chiefly to the loss of tin in the sulphide concentrates, caused by lack of cleaning. An intensive study of all phases of the concentration procedure would probably show that the recovery of tin from this ore can be increased.

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Sedimentation size analyses were done by Mr. T.B. Sundara, trainee under the Colombo Plan for India.

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