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MINES BRANCH INVESTIGATION REPORT IR 62-73

PILOT PLANT INVESTIGATION ON SCHEELITE ORE FROM CANADA TUNGSTEN MINING CORPORATION LTD., FLAT RIVER, N. W. T.

by

T. F. BERRY & R. P. BAILEY

MINERAL PROCESSING DIVISION

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PILOT PLANT INVESTIGATION ON SCHEELITE ORE FROM CANADA TUNGSTEN MINING CORPORATION LTD., FLAT RIVER, N.W.T.

by

T. F. Berry* and R. P. Bailey**

SUMMARY OF RESULTS

About 34 tons of high-sulphide scheelite ore containing an average of 2.39% WO_3 were milled during the pilot plant run.

Most of the sulphides were removed by an initial bulk flotation with the loss of only 3% of the tungsten mineral. In subsequent treatment of the flotation tailing by sizing, successive tabling, sulphide scavenger flotation, drying and high intensity dry magnetic separation, approximately 70% of the tungsten was recovered in a final gravity concentrate assaying about 76% WO_3. An additional 15% of the tungsten was recovered by scheelite flotation of a slime fraction followed by acid-leaching to produce a concentrate assaying about 69% WO_3.

A satisfactory reduction of the impurities to less than specification maxima was achieved in the final gravity concentrate. However, additional work on a plant scale will be necessary to reduce the sulphur in the scheelite flotation concentrate to an acceptable level.

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INTRODUCTION

In October, 1959, at the request of Canada Tungsten Mining Corporation Ltd., Suite 502, 1200 West Pender Street, Vancouver 1, B.C., investigations were started by the Mines Branch in Ottawa on the recovery of scheelite from ore and drill core samples from the company's property in the Flat River area, N.W.T.

These investigations, which continued for about two and a half years, were reported in Mines Branch Investigation Reports IR 60-80(1), IR 61-107(2), and IR 61-108(3), and led to a decision by Canada Tungsten Mining Corporation Ltd., in January, 1962, to ask the Mines Branch to undertake a pilot plant investigation on a large shipment of the ore.

The pilot plant investigation was carried out under the general direction of Mr. R. W. Burton, Assistant Manager of Canada Tungsten Mining Corporation Ltd., in cooperation with officers of the Non-Ferrous Minerals Section, Mineral Processing Division, Mines Branch, Ottawa.

Mr. R. H. Ross, Metallurgist for American Metal Climax Inc., issued a report(4) dated November 30, 1961, in which he outlined a flowsheet based on the results of the previous investigational work at the Mines Branch and elsewhere. The object of the Mines Branch pilot plant investigation was to simulate the flowsheet presented by Mr. Ross and to produce as much gravity and flotation concentrate as possible for use by the company.

With the exception of one small pilot test involving about four tons of ore, described in Mines Branch Investigation Report IR 60-80(1), the earlier investigations were limited to small scale bench tests including both gravity and flotation concentration of the scheelite. The present pilot plant investigation was designed to make possible, for the first time on a relatively large scale, observations on the grinding and classification, sulphide flotation and copper recovery, and sizing and tabling during a continuous operation. Only the scheelite flotation was run on an intermittent basis because of the small amount of material available.

Soon after the pilot plant investigation began it became apparent that the results would be considerably different from those predicted by Mr. Ross(4). Because these results would have an important bearing on the operation of a full sized mill considerable time was spent testing modifications of the planned flowsheet while keeping in mind the type of equipment already acquired for the operation at Flat River.

Origin and Description of Ore

According to a preliminary report(5) by R. W. Burton, issued at the completion of this pilot plant investigation, the ore available was that stored at the Flat River property (about 16 tons), at Watson Lake, Y.T. (about 17 tons) and at Lakefield, Ontario (about 1 ton). All three lots were shipped to the Mines Branch in 744 jute bags and were treated as one shipment. Although approximately 10% of the ore was highly oxidized the remainder was firm lump ore showing only slight oxidation.

DETAILS OF PILOT PLANT INVESTIGATION

Description of Equipment

1. 8 in. x 12 in. Hadfield, Blake type, jaw crusher
2. 20 in. Symons (standard) cone crusher
3. 24 in. x 48 in. Sturtevant Moto-Vibro screen
4. Hardinge constant weight feeder
5. 30 in. x 48 in. rod mill
6. 12 in. 60° arc DSM Screen
7. 24 in. x 36 in. Denver conditioner
8. 10 No. 7 Denver Sub A flotation cells (bulk sulphide flotation)
9. 18 in. x 24 in. Denver conditioner
10. 10 No. 5 Denver Sub A flotation cells (copper flotation)
11. 3 in. 20° Dorrclone hydrocyclone
12. 4 compartment, each 50 mm, type M hydrocyclone
13. Southwest (SWECO) separator - 18 in. and 30 in. diam. holding at various times, 50 m, 80 m and 100 m screens
14. Standard size Wilfley table
15. 1/2 size Deister table
16. 1/4 size Wilfley table
17. Laboratory size Deister table
18. 3 ft. x 3 ft. pan filter
19. 24 in. x 10 ft. stainless steel belt dryer with 4 Calrod heating elements
20. 3 roll, 10 in. wide, Dings type dry high intensity magnetic separator
21. 3 cross belt, Wetherill type dry high intensity magnetic separator
22. Laboratory model Stearns " " " " "
23. Jones, wet high intensity magnetic separator
24. 5 No. 7 Denver Sub A flotation cells (scheelite flotation)
25. 8 No. 5 " " " " " (" ")

Crushing

The ore was crushed in an 8 in. x 12 in. Hadfield jaw crusher followed by a 20 in. Symons cone crusher in closed circuit with a Sturtevant Moto-Vibro screen giving an undersize product substantially -3/8 in. To reduce the possibility of the crushed ore becoming oxidized not more than 5 tons of crushed mill feed was kept on hand at any one time.

There were no problems in crushing the ore although there was considerable dust loss none of which was recovered for treatment.

A typical screen analysis of the mill feed delivered by the crushing plant may be seen in Table 1.

TABLE 1

Typical Screen Test on Mill Feed

Mesh Size	Weight % Retained	Grouped Weight % Retained
+ 3/8 inch	1.0	1.0
+3 mesh	35.8	75.0
+4 "	23.0	
+6 "	10.0	
+8 "	6.2	
+10 "	4.7	10.5
+14 "	2.8	
+20 "	3.0	
+28 "	2.1	7.4
+35 "	2.2	
+48 "	1.6	
+65 "	1.5	
+100 "	1.4	4.6
+150 "	1.0	
+200 "	0.9	
+325 "	1.3	
-325 "	1.5	1.5
Total	100.0	100.0

Grinding and Classification

Early in the investigation it was realized that, to prevent over-grinding, rod milling in closed circuit with a screen was necessary, and that at the grind (-65m) needed to liberate the scheelite, a DSM screen was the only practical one available.

Mr. Ross proposed in his report(4) that to prevent over-dilution of the sulphide flotation feed, two stages of cycloning be incorporated with the DSM screen in the classification of the rod mill discharge. However, early in the pilot plant investigation, it was found that, when the pulp density in the rod mill was maintained at or near 72% solids, the rod mill discharge could be diluted and pumped directly over the DSM screen without adversely affecting the density of the flotation feed. This procedure, which was followed throughout the investigation, obviated the need for a dewatering cyclone in the grinding and classification circuit.

During the pilot plant tests either a 0.35 mm or a 0.5 mm DSM screen was used with the former giving a better product for flotation by reducing to a minimum the amount of coarse sulphides. At a rod mill feed rate of 600 lb/hr, it was necessary to block off a portion of the screen surface of the 0.5 mm screen using plywood inserts since the DSM screen appeared to operate at its best when the screen deck was flooded with pulp. This modification of the screen deck was not necessary when the 0.35 mm screen was used.

The non-blinding characteristics claimed by the manufacturers of the 60° arc DSM screen were not realized in operation under the conditions of this investigation. Reversal of the pulp flow, as frequently as twice each hour, reduced the blinding but did not eliminate it.

Throughout the pilot plant investigation, the crushed ore was drawn from the storage bin by a Hardinge constant-weight feeder discharging at the rate of 600 lb/hr to a 30 in. x 48 in. rod mill. This mill, operating at a speed of 39 rpm, and having no liners carried a rod charge of between 1000 and 1100 lb of 44 in. H.C. rods having a maximum diameter of 2¼ in. and a minimum diameter of 7/8 in. A standard 12 in. DSM screen frame was mounted above the feed end of the rod mill. The discharge, diluted to about 45% solids, was raised by a bucket elevator to the feed box of this screen. The screen underflow was fed by gravity to a sulphide flotation conditioner with the oversize dropping directly into the feed scoop of the rod mill.

A satisfactory feed for sulphide flotation was produced by maintaining pulp densities in the ranges shown in the following table.

TABLE 2

Grinding Circuit Pulp Densities

Rod mill discharge	69-74% solids
Feed to DSM screen	41-45% "
DSM screen o'size	73-77% "
DSM screen u'flow	32-36% "

Screen tests typical of the grinding circuit products are shown in Tables 3 and 4. These were obtained by averaging the results of screen tests made during two phases of the pilot plant investigation, one with the 0.5 mm DSM screen in the circuit, the other using the 0.35 mm screen.

TABLE 3

Screen Tests on Grinding Circuit Products using 0.5 mm DSM Screen

Mesh Size	Rod Mill Discharge		DSM O'size		DSM U'flow	
	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing
+48 mesh	14.4	85.6	43.2	56.8	4.6	95.4
+65 "	13.0	72.6	20.7	36.1	9.4	86.0
+100 "	14.4	58.2	13.1	23.0	13.8	72.2
+150 "	11.3	46.9	6.2	16.8	12.5	59.7
+200 "	9.8	37.1	3.6	13.2	11.2	48.5
+325 "	14.0	23.1	5.8	7.4	16.9	31.6
-325 "	23.1	-	7.4	-	31.6	-
Total	100.0	-	100.0	-	100.0	-

TABLE 4

Screen Tests on Grinding Circuit Products using 0.35 mm DSM Screen

Mesh Size	Rod Mill Discharge		DSM O'size		DSM U'flow	
	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing
+48 mesh	15.3	84.7	34.3	65.7	1.8	98.2
+65 "	15.3	69.4	28.2	37.5	6.8	91.4
+100 "	16.1	53.3	20.4	17.1	12.3	79.1
+150 "	11.5	41.8	7.8	9.3	12.7	66.4
+200 "	8.4	33.4	3.0	6.3	10.8	55.6
+325 "	10.5	22.9	2.9	3.4	15.5	40.1
-325 "	22.9	-	3.4	-	40.1	-
Total	100.0	-	100.0	-	100.0	-

Earlier investigations indicated that a grind of 65% -200m would be necessary to liberate the scheelite. However, in this pilot plant investigation, satisfactory liberation in the feed to sulphide flotation was achieved at 55.6% -200m with the 0.35 mm DSM screen in the circuit. When the 0.5 mm screen was used, the flotation feed at 48.5% -200m was too coarse for good sulphide elimination.

Bulk Sulphide Flotation

The DSM screen underflow was fed by gravity at approximately 34% solids to a 24 in. x 36 in. Denver conditioner where samples representative of the mill head were taken. The following is typical of the results which were obtained.

TABLE 5

Typical Chemical Analysis(5) of Flotation Feed

Element	Per Cent
Tungsten trioxide (WO_3)	2.39
Sulphur (Total S)	15.05
Copper (Cu)	0.41
Bismuth (Bi)	0.08
Phosphorus (P)	0.06
Iron (Soluble Fe)	24.30
Molybdenum (Mo)	0.03

The conditioned pulp was fed to ten No. 7 Denver Sub A flotation cells from which a rougher concentrate was floated and pumped to four No. 7 cells for one stage of cleaning. A small conditioner having a capacity of 1.5 cu ft was placed at the head of the sulphide cleaner circuit in order to reduce surging in that circuit caused by an oversize centrifugal pump handling the sulphide rougher concentrate.

The flotation of the pyrrhotite and chalcopyrite in a bulk sulphide concentrate did not present any unusual problems. The single stage cleaning resulted in the loss of about 3% of the scheelite in the bulked concentrate. With two or possibly three stages of cleaning this scheelite loss might be reduced. While the total recovery of sulphides was about 90%, it is possible that considerable improvement might be expected with fresh unoxidized ore. The possibility of striking a balance between virtual elimination of the sulphides by flotation and possible increased scheelite losses must be considered.

During the sulphide flotation the reagent consumption which was slightly higher than anticipated remained relatively constant throughout the pilot plant test. Initially, Aerofloat 25 was used as the sulphide promoter but it resulted in a heavy unmanageable froth in which a large scheelite loss occurred. Potassium amyl xanthate was substituted for the Aerofloat 25 and used throughout the remainder of the investigation, although some Aerofloat 25 was used in the final two rougher cells. Staged additions of copper sulphate and potassium amyl xanthate were made about half way through the rougher circuit to stimulate the recovery of slow floating sulphides. When this was done, less of these reagents was fed at the head of the circuit. Sodium silicate was added to No. 1 cleaner cell to provide a slightly more brittle froth thereby rejecting trapped scheelite grains.

The following table shows the reagents which were used in the sulphide flotation circuit.

TABLE 6

Reagents Used in Bulk Sulphide Flotation

Reagent	lb/ton Ore	Point of Addition
Copper sulphate	0.9	Rod mill feed
Potassium amyl xanthate	0.3	Conditioner
Aerofroth No. 70	0.05	No. 1 rougher cell
Copper sulphate	0.4	No. 5 " "
Potassium amyl xanthate	0.1	No. 6 " "
Aerofloat 25*	0.02	No. 8 " "
Sodium silicate	0.03	No. 1 cleaner cell
Rougher circuit pH	6.7 to 6.9 at 50°F	

* Used as a scavenger to recover residual sulphides.

An examination of the sulphide flotation tailing indicated that the largest (+48m) and the finest (slime) sulphide particles tended to resist flotation. By replacing the 0.5 mm DSM screen in the grinding circuit with the 0.35 mm DSM screen, which reduced the proportion of +48m material going to sulphide flotation, a higher sulphide recovery was obtained. The ore contained limonite, due probably to pyrrhotite oxidation. This material became slimed during grinding along with partially oxidized pyrrhotite and this finely-divided material did not lend itself to recovery by flotation.

The following table shows a typical screen test on a sulphide flotation tailing obtained when a 0.35 mm DSM screen was used in the grinding circuit.

TABLE 7

Typical Screen Test on a Sulphide Flotation Tailing

Mesh Size	Weight % Retained
+48 mesh	3.1
+65 "	10.0
+100 "	15.6
+150 "	14.5
+200 "	12.0
+325 "	15.5
-325 "	29.3
Total	100.0

Although there was reasonably good agreement between the assayed and the calculated head throughout the pilot plant investigation, in virtually every test the assayed head was lower. This was probably due to the difficulty experienced in sampling the long and complex circuit. For purposes of calculating metallurgical balances the calculated head was used.

The following sulphide metallurgical balance is typical of the results which were obtained in this phase of the operation. The figures shown in Table 8 were taken from Test 19 in which an excellent overall metallurgical balance was obtained. The sulphide elimination in this test, however, was slightly lower than that obtained in other tests.

TABLE 8

Typical Sulphide Metallurgical Balance

Product	Weight %	Analyses %[*]			Distribution %		
		WO_3	S	Cu	WO_3	S	Cu
Sulphide cl conc	39.6	0.21	33.36	1.15	3.9	87.6	92.6
" rougher tail	60.4	3.88	3.09	0.06	96.1	12.4	7.4
Head (calcd)	100.0	2.43	15.08	0.49	100.0	100.0	100.0
Head (assayed)	-	2.26	14.31	0.45	-	-	-

[*] From Internal Report MS-AC-62-391.

Copper Flotation

The bulked sulphide cleaner concentrate flowed by gravity to a small Denver conditioner for treatment with sodium cyanide and lime to a pH of 9.7. It was then pumped, at about 35% solids, to a bank of No. 5 Denver cells for flotation of a rougher copper concentrate which was cleaned three times. Since a minimum of attention was given to the copper circuit which was operated for only two or three days, the optimum conditions for recovery and concentrate grade were not determined. Although the copper recovery was only 70%, further work on a plant scale will probably effect a marked increase.

The reagents used in the copper circuit based on the initial mill feed were as follows:

TABLE 9

Reagents Used in Copper Flotation

Reagent	lb/ton Ore	Point of Addition
Lime (CaO)	0.2	conditioner
Sodium cyanide	0.1	"
Aerofroth No. 7	0.03	end of rougher cells
Sodium silicate	0.03	cleaner circuit
Sodium cyanide	0.03	" "
Rougher circuit pH		9.6-9.8 at 54°F

The copper concentrate, although finely divided, was not slimy and filtered readily. The following table shows a typical screen test on a final copper concentrate.

TABLE 10

Typical Screen Test on a Copper Concentrate

Mesh Size	Weight % Retained
+100 mesh	0.3
+150 "	0.8
+200 "	1.1
+325 "	6.8
-325 "	91.0
Total	100.0

The final copper concentrate produced each day was of relatively low grade, having an analysis of about 16% Cu. The gold in this concentrate averaged 0.10 oz Au/ton.

Just before abandoning the copper circuit, about 10 lb of a final concentrate was made after repeated cleaning. A complete analysis may be seen in the following table. It is worth noting that the gold content of this concentrate was reduced. This is possibly due to the depressing effect on the gold of the additional lime used in the recleaning stages.

TABLE 11

Analysis of a Final Recleaned Copper Concentrate

Element	Analysis *
Copper (Cu)	27.34 per cent
Tungsten trioxide (WO_3)	0.19 " "
Sulphur (Total S)	34.24 " "
Bismuth (Bi)	0.15 " "
Arsenic (As)	0.03 " "
Molybdenum (Mo)	0.003 " "
Insoluble	0.68 " "
Gold (Au)	0.06 oz/ton
Silver (Ag)	0.86 "

* From Internal Report MS-AC-62-471.

Sizing and Gravity Concentration

Initially, both single and double cycloning were used in an attempt to remove the -325m material from the flotation tailing before tabling. A wide range of pulp densities and pressures were tried but the pulp flow was too small for the available pumps. The results of screen tests on the products from single and double stage cyclone classifications are shown in Tables 12 and 13. The daily results varied widely and the examples selected, although never reproduced, were the best obtained.

However, before introducing other methods of classification (i.e. screening, desliming), several table concentration tests were completed using single-stage cycloning. To avoid confusion these gravity circuit modifications, of which there were seven, will be described in chronological order.

TABLE 12

Screen Tests Showing the Results of Single-Stage Cycloning

Mesh Size	Sulphide Flot Tail		Cyclone O'flow		Spigot Discharge	
	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing
+48 mesh	3.6	96.4	-	-	5.1	94.9
+65 "	9.3	87.1	-	100.0	11.2	83.7
+100 "	13.5	73.6	0.2	99.8	14.2	69.5
+150 "	11.9	61.7	0.5	99.3	12.6	56.9
+200 "	10.0	51.7	1.0	98.2	10.7	46.2
+325 "	14.0	37.7	5.1	93.2	15.2	31.0
-325 "	37.7	-	93.2	-	31.0	-
Total	100.0	-	100.0	-	100.0	-

TABLE 13

Screen Tests Showing the Results of Double-Stage Cycloning

Mesh Size	Sulphide Flot Tail		No. 1 Cyclone				No. 2 Cyclone			
			O'flow		Spigot Discharge		O'flow		Spigot Discharge	
	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing
+48 mesh	2.8	97.2	-	-	5.4	94.6	-	-	4.8	95.2
+65 "	8.4	88.8	-	100.0	10.2	84.4	-	100.0	11.2	84.0
+100 "	14.4	74.4	0.4	99.6	11.6	72.8	0.4	99.6	17.6	66.4
+150 "	13.2	61.2	0.2	99.4	11.4	61.4	0.4	99.2	16.6	49.8
+200 "	7.0	54.2	0.2	99.2	11.0	50.4	0.2	99.0	12.6	37.2
+325 "	14.2	40.0	0.2	99.0	18.4	32.0	0.8	98.2	18.2	19.0
-325 "	40.0	-	99.0	-	32.0	-	98.2	-	19.0	-
Total	100.0	-	100.0	-	100.0	-	100.0	-	100.0	-

(a) Circuit Modification 1

In this test, illustrated by the gravity concentration flowsheet in Figure 1, the 0.5 mm DSM screen with its area reduced to 5.25 in. using plywood inserts was in operation in the grinding circuit.

The sulphide flotation tailing was pumped to single-stage cyclone classification with the cyclone overflow going to a settling cone for eventual scheelite flotation. The spigot discharge from the cyclone was tabled on a full-size Wilfley table from which a slime tailing was removed by setting and frequently checking the position of a splitter along the tailing edge of the table. This tailing was pumped to the thickening cone. A rougher scheelite concentrate was removed and retained for cleaning. The table middling was pumped to a Sweco separator equipped with a 100m screen. The +100m fraction was tabled on a small laboratory Deister table with the concentrate joining the original Wilfley table concentrate and the tailing going to waste. The -100m fraction was tabled on a $\frac{1}{4}$ deck Wilfley table with the concentrate from this table joining the first two concentrates. A sand tailing from this table was discarded and a slime tailing was pumped to the settling cone.

Table 14 shows a metallurgical balance for this test based on weights and assays rather than on assays only. Table 15 summarizes results of screen tests on the mill products.

TABLE 14

Metallurgical Balance - Modification 1

Product	Weight %	Analyses %*	Distn %
		WO_3	WO_3
Sulphide cl conc (1)	29.3	0.27	3.4
Cyclone o'flow (2)	8.6	2.89	10.7
Wilfley table conc (3)	8.9	17.60	67.8
" " tail (slime)(4)	3.5	2.96	4.5
Deister " conc (5)	0.3	1.44	0.2
" " tail (6)	23.6	0.22	2.2
$\frac{1}{4}$ deck Wilfley table conc (7)	4.2	2.36	4.3
$\frac{1}{4}$ " " " tail (sand)(8)	18.6	0.50	4.0
$\frac{1}{4}$ " " " " (slime)(9)	3.0	2.24	2.9
Head (calcd)	100.0	2.31	100.0

* From Internal Report MS-AC-62-330.

Summary of Metallurgical Balance

Gravity conc (3 + 5 + 7)	13.4	12.46	72.3
Scheelite flot feed (2 + 4 + 9)	15.1	2.78	18.1
Circuit tail (1 + 6 + 8)	71.5	0.31	9.6
Head (calcd)	100.0	2.31	100.0
" (assayed)	-	2.40	-

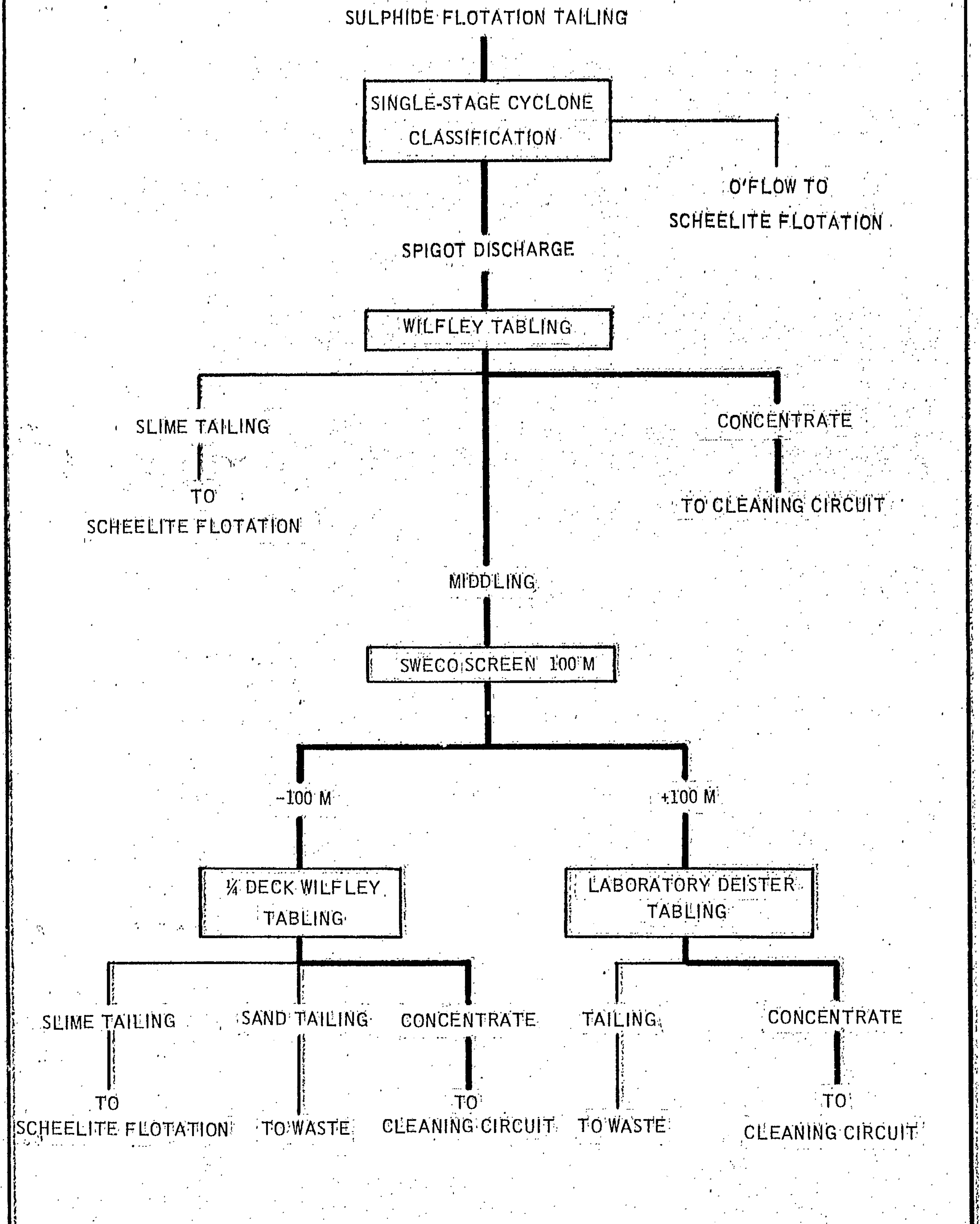

FIGURE 1 – GRAVITY CONCENTRATION CIRCUIT – MODIFICATION 1

TABLE 15

Screen Tests on Gravity Concentration Circuit - Modification 1

Mesh Size	Sulphide Flot Tail		Cyclone O'flow		Spigot Discharge	
	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing
+48 mesh	2.4	97.8	-	-	3.8	96.2
+65 "	7.2	90.4	-	100.0	10.3	85.9
+100 "	11.3	79.1	0.2	99.8	15.4	70.5
+150 "	12.2	66.9	0.2	99.6	13.6	56.9
+200 "	11.2	55.7	0.1	99.5	12.2	44.7
+325 "	16.8	38.9	0.3	99.2	16.8	27.9
-325 "	38.9	-	99.2	-	27.9	-
Total	100.0	-	100.0	-	100.0	-

TABLE 15 (cont'd)

Mesh Size	Wilfley Midd		Wilfley Tail		Deister Tail	
	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing
+48 mesh	2.8	97.2	1.2	98.8	7.0	93.0
+65 "	10.0	87.2	3.3	95.5	22.4	70.6
+100 "	15.8	71.4	3.2	92.3	37.8	32.8
+150 "	15.6	55.8	1.4	90.9	25.6	7.2
+200 "	14.6	41.2	0.9	90.0	4.6	2.6
+325 "	20.0	21.2	2.6	87.4	1.0	1.6
-325 "	21.2	-	87.4	-	1.6	-
Total	100.0	-	100.0	-	100.0	-

(b) Circuit Modification 2

In this test, illustrated by the flowsheet in Figure 2, the $\frac{1}{4}$ deck Wilfley table was replaced by a $\frac{1}{2}$ deck Deister table to treat the -100m material from the Sweco separator. The +100m material was discarded. The Deister slime tailing was pumped to the settling cone and the concentrate was retained for cleaning. The middling from the Deister table was treated on a $\frac{1}{4}$ deck Wilfley table from which the slime and sand tailing was pumped to waste and a concentrate was removed for cleaning.

A metallurgical balance is shown in Table 16. It will be noted that the recovery of scheelite was very high in this test with a corresponding decrease in the percentage going to scheelite flotation. The results of screen tests on some of the mill products are summarized in Table 17.

TABLE 16

Metallurgical Balance - Modification 2

Product	Weight %	Analyses % *	Distn %
		WO_3	WO_3
Sulphide cl conc (1)	36.8	0.17	2.0
Cyclone o'flow (2)	7.5	3.08	7.5
Wilfley table conc (3)	4.4	47.23	67.7
" " tail (slime)(4)	6.7	1.48	3.2
Sweco separator +100m (5)	28.6	0.39	3.6
$\frac{1}{2}$ deck Deister table conc (6)	1.0	36.80	12.0
$\frac{1}{2}$ " " " tail (7)	2.7	0.80	0.7
$\frac{1}{4}$ " Wilfley table conc (8)	0.8	5.60	1.5
$\frac{1}{4}$ " " " tail (sand + slime) (9)	11.5	0.49	1.8
Head (calcd)	100.0	3.06	100.0

* From Internal Report MS-AC-62-334.

Summary of Metallurgical Balance

Gravity conc (3 + 6 + 8)	6.2	40.01	81.2
Scheelite flot feed (2 + 4 + 7)	16.9	2.08	11.4
Circuit tail (1 + 5 + 9)	76.9	0.30	7.4
Head (calcd)	100.0	3.06	100.0
" (assayed)	-	2.13	-

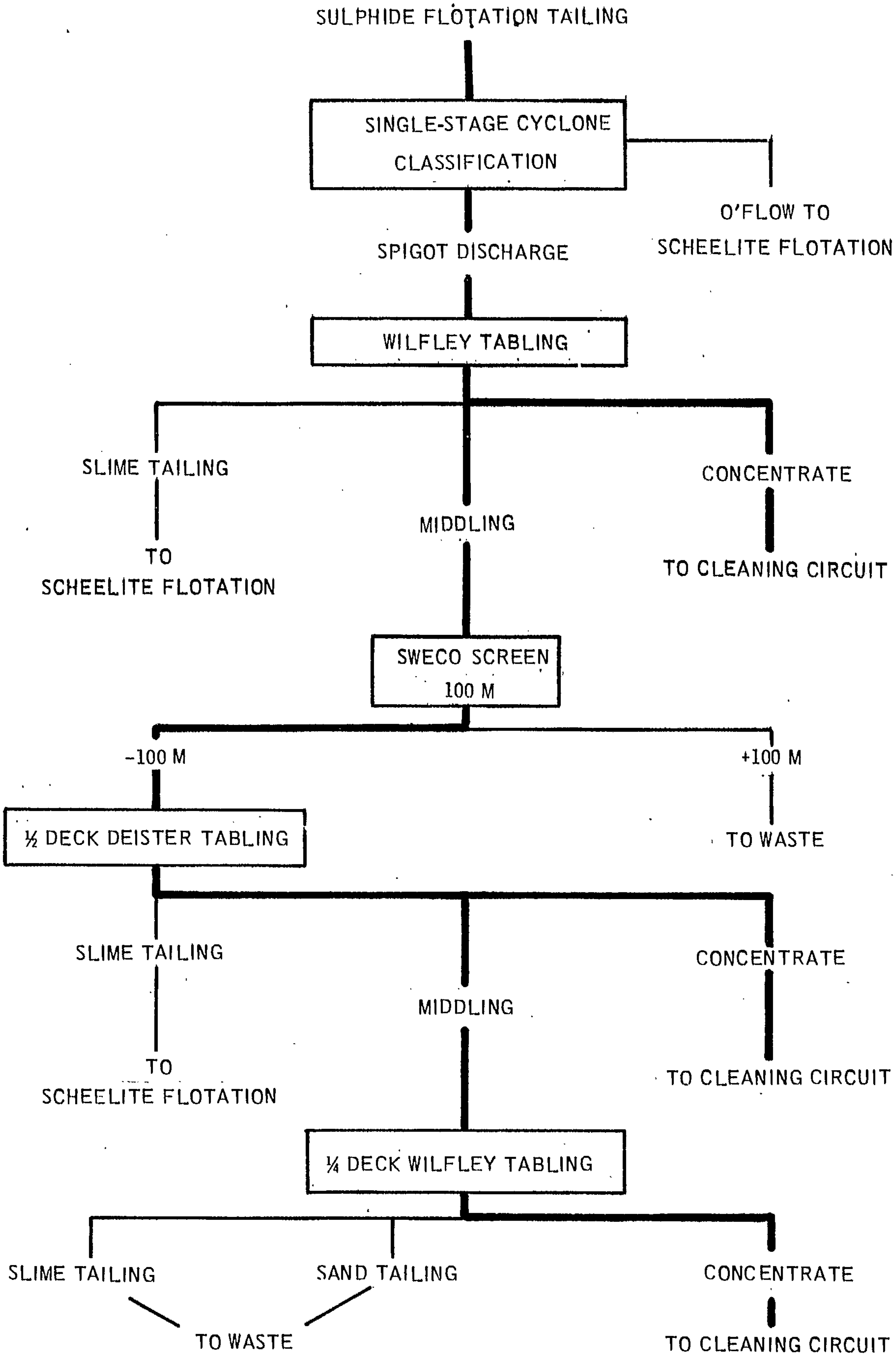

FIGURE 2 – GRAVITY CONCENTRATION CIRCUIT – MODIFICATION 2

TABLE 17

Screen Tests on Gravity Concentration Circuit - Modification 2

Mesh Size	Sulphide Flot Tail		Cyclone Spigot Discharge		Sweco Separator +100m		Sweco Separator -100m		½ deck Deister Midd	
	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing
+48 mesh	5.6	94.4	7.2	92.8	11.2	88.8	-	-	0.3	99.7
+65 "	10.3	84.1	12.6	80.2	23.7	65.1	-	100.0	0.6	99.1
+100 "	14.3	69.8	14.7	65.5	33.7	31.4	0.2	99.8	0.8	98.3
+150 "	12.3	57.5	12.4	53.1	24.0	7.4	7.7	92.1	10.1	88.2
+200 "	10.2	47.3	10.4	42.7	5.1	2.3	20.6	71.5	23.9	64.3
+325 "	14.0	33.3	15.3	27.4	1.3	1.0	34.6	36.9	33.8	30.5
-325 "	33.3	-	27.4	-	1.0	-	36.9	-	30.5	-
Total	100.0	-	100.0	-	100.0	-	100.0	-	100.0	-

(c) Circuit Modification 3

The flowsheet used in this test was identical to that shown in Figure 2 with the exception that an 80m screen replaced the 100m screen on the Sweco separator and the $\frac{1}{4}$ deck Wilfley table slime tailing was assayed separately and pumped to the settling cone.

TABLE 18

Metallurgical Balance - Modification 3

Product	Weight %	Analyses %*	Distn %
		WO_3	WO_3
Sulphide cl conc (1)	34.1	0.31	4.0
Cyclone o'flow (2)	7.8	2.96	8.8
Wilfley table conc (3)	3.5	40.00	53.2
" " tail (4)	3.5	2.71	3.6
Sweco separator +80m (5)	15.5	0.45	2.6
$\frac{1}{2}$ deck Deister table conc (6)	1.1	34.88	14.6
$\frac{1}{2}$ " " " tail (7)	7.1	0.73	2.0
$\frac{1}{4}$ " Wilfley " conc (8)	3.2	5.30	6.4
$\frac{1}{4}$ " " " tail (sand) (9)	22.9	0.47	4.2
$\frac{1}{4}$ " Wilfley table tail (slime) (10)	1.3	1.16	0.6
Head (calcd)	100.0	2.63	100.0

* From Internal Report MS-AC-62-368.

Summary of Metallurgical Balance

Gravity conc (3 + 6 + 8)	7.8	25.04	74.2
Scheelite flot (2 + 4 + 7 + 10)	19.7	1.98	15.0
Tailing (1 + 5 + 9)	72.5	0.39	10.8
Head (calcd)	100.0	2.63	100.0
" (assayed)	-	2.24	-

TABLE 19

Screen Tests on Gravity Concentration Circuit - Modification 3

Mesh Size	Sulphide Flot Tail		Sweco Separator +80m		Sweco Separator -80m		½ deck Deister Midd	
	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing
+48 mesh	4.4	95.6	15.3	84.7	-	-	-	100.0
+65 "	10.0	85.6	38.4	46.3	-	100.0	0.5	99.5
+100 "	13.2	72.4	35.3	11.0	10.8	89.2	10.5	89.0
+150 "	12.3	60.1	8.0	3.0	18.8	70.4	18.0	71.0
+200 "	10.7	49.4	1.5	1.5	16.8	53.6	17.6	53.4
+325 "	15.0	34.4	0.7	0.8	24.0	29.6	25.8	27.6
-325 "	34.4	-	0.8	-	29.6	-	27.6	-
Total	100.0	-	100.0	-	100.0	-	100.0	-

(d) Circuit Modification 4

During this test the cyclone was removed from the circuit, and the sulphide rougher tailing went directly to the large Wilfley table. The Sweco separator was equipped with an 80m screen and the $\frac{1}{4}$ deck Wilfley table slime was pumped to a settling cone. In all other respects, the flowsheet for this test remained as shown in Figure 2.

TABLE 20

Metallurgical Balance - Modification 4

Product	Weight %	Analyses % *	Distn %
		WO_3	WO_3
Sulphide cl conc (1)	36.5	0.14	1.9
Wilfley table conc (2)	4.6	39.60	67.1
" " tail (3)	15.6	2.18	12.5
Sweco separator +80m (4)	15.1	0.32	1.8
$\frac{1}{2}$ deck Deister table conc (5)	1.0	25.20	9.3
$\frac{1}{2}$ " " " tail (6)	3.2	1.31	1.5
$\frac{1}{4}$ " Wilfley " conc (7)	3.5	1.92	2.5
$\frac{1}{4}$ " " " tail (sand) (8)	15.4	0.44	2.5
$\frac{1}{4}$ " Wilfley table tail (slime) (9)	5.1	0.48	0.9
Head (calcd)	100.0	2.71	100.0

* From Internal Report MS-AC-62-368.

Summary of Metallurgical Balance

Gravity conc (2 + 5 + 7)	9.1	21.20	78.9
Scheelite flot feed (3 + 6 + 9)	23.9	1.70	14.9
Circuit tail (1 + 4 + 8)	67.0	0.25	6.2
Head (calcd)	100.0	2.71	100.0
" (assayed)	-	2.18	-

TABLE 21

Screen Tests on Gravity Concentration Circuit - Modification 4

Mesh Size	Sulphide Flot Tail		Sweco Separator +80m		Sweco Separator -80m		½ deck Deister Midd	
	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing
+48 mesh	5.6	94.4	15.4	84.6	-	-	-	100.0
+65 "	11.0	83.4	36.0	48.6	-	100.0	0.4	99.6
+100 "	15.0	68.4	37.3	11.3	9.0	91.0	10.3	89.3
+150 "	13.2	55.2	8.1	3.2	18.1	72.9	20.8	68.5
+200 "	10.5	44.7	1.6	1.6	18.0	54.9	18.5	50.0
+325 "	14.3	30.4	1.1	0.5	27.3	27.6	26.2	23.8
-325 "	30.4	-	0.5	-	27.6	-	23.8	-
Total	100.0	-	100.0	-	100.0	-	100.0	-

(o) Circuit Modification 5

In this test the grinding circuit was altered by substituting a 0.35 mm DSM screen, without plywood inserts, for the masked 0.5 mm screen used previously.

Figure 3 shows the gravity concentration circuit for this modification. As in the previous test, no cyclone classification was used, and the sulphide rougher flotation tailing was deslimed on the large Wilfley table with the slime fraction being pumped to the settling cone for scheelite flotation. The Sweco separator with an 80m screen was retained to treat the Wilfley table middling.

The $\frac{1}{4}$ deck Wilfley table was placed ahead of the $\frac{1}{2}$ deck Deister table for this test in the hope that the standard riffles on the former would effect better desliming than the extremely deep riffles on the latter.

A metallurgical balance is shown in Figure 22 and the results of screen tests on gravity circuit products are listed in Table 23.

TABLE 22

Metallurgical Balance - Modification 5

Product	Weight %	Analyses % *	Distn %
		WO_3	WO_3
Sulphide cl conc (1)	34.1	0.15	2.3
Wilfley table conc (2)	2.9	42.80	55.2
" " tail (3)	13.2	2.54	14.9
Sweco separator +80m (4)	12.7	0.24	1.4
$\frac{1}{4}$ deck Wilfley table conc (5)	0.4	18.32	3.2
$\frac{1}{4}$ " " " tail (6)	24.3	0.89	9.6
$\frac{1}{2}$ " Deister " conc (7)	1.5	16.20	10.8
$\frac{1}{2}$ " " " tail (sand)(8)	10.3	0.45	2.1
$\frac{1}{2}$ " " " " (slime) (9)	0.6	2.12	0.5
Head (calcd)	100.0	2.25	100.0

* From Internal Report MS-AC-62-388.

Summary of Metallurgical Balance

Gravity conc (2 + 5 + 7)	4.8	32.45	69.2
Scheelite flot feed (3 + 6 + 9)	38.1	1.48	25.0
Circuit tail (1 + 4 + 8)	57.1	0.22	5.8
Head (calcd)	100.0	2.25	100.0
" (assayed)	-	1.92	-

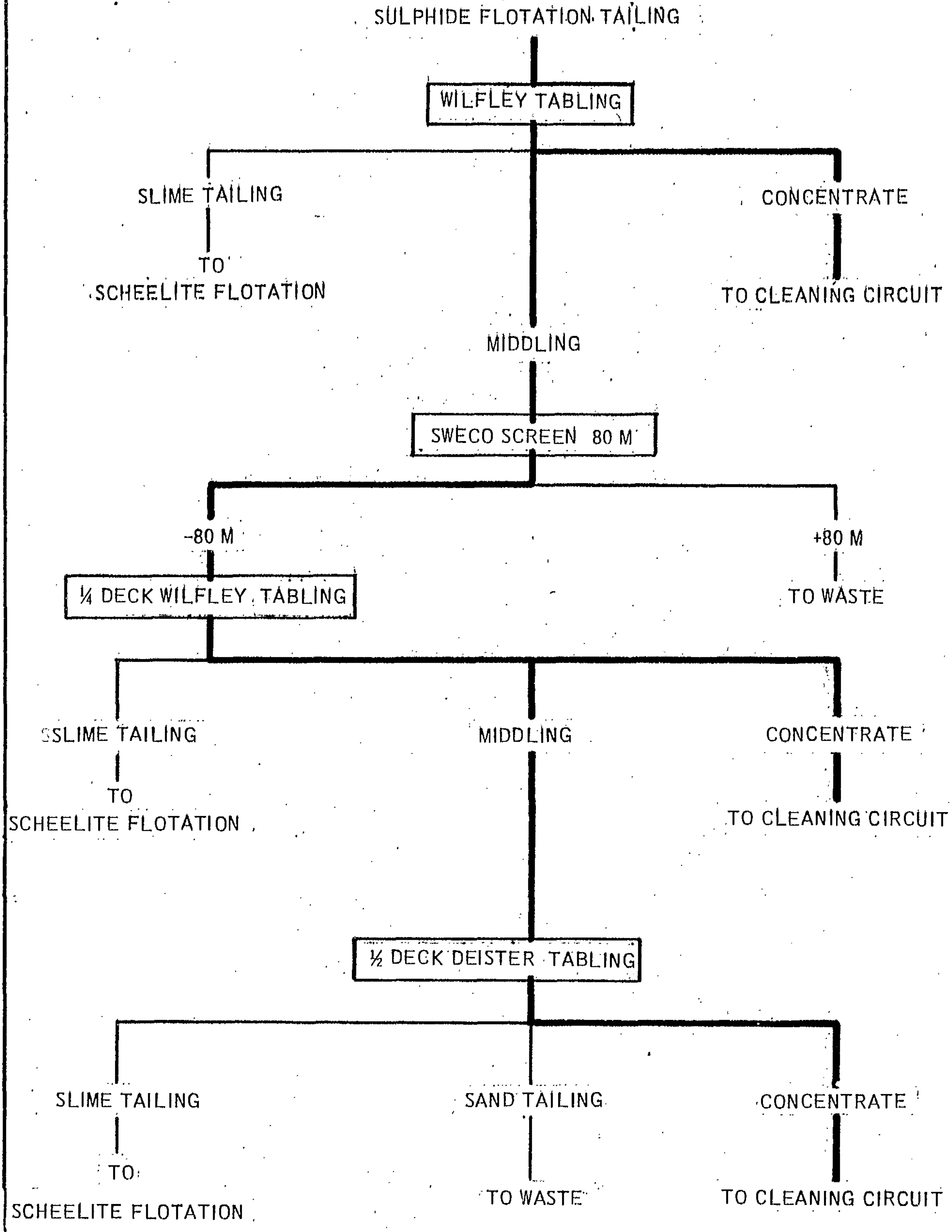

FIGURE 3 – GRAVITY CONCENTRATION CIRCUIT – MODIFICATION 5

TABLE 23

Screen Tests on Gravity Concentration Circuit - Modification 5

Mesh Size	Sulphide Flot Tail		Wilfley Table Midd		Sweco Separator +80m		Sweco Separator -80m	
	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing
+48 mesh	2.0	98.0	1.4	98.6	5.6	94.4	-	100.0
+65 "	8.6	89.4	7.4	91.2	30.4	64.0	0.2	99.8
+100 "	15.8	73.6	15.2	76.0	46.8	17.2	6.4	93.4
+150 "	14.5	59.1	15.8	60.2	10.0	7.2	16.8	76.6
+200 "	10.8	48.3	15.0	45.2	5.1	2.1	18.5	58.1
+325 "	14.2	34.1	20.6	24.6	1.8	0.3	27.4	30.7
-325 "	34.1	-	24.6	-	0.3	-	30.7	-
Total	100.0	-	100.0	-	100.0	-	100.0	-

(f) Circuit Modification 6

The only change in this circuit was the replacement of the 80m screen on the Sweco separator with a 100m screen. In all other respects this test was identical to the previous test shown in Figure 3. An attempt was made to recover a higher proportion of the scheelite in a gravity concentrate at a sacrifice in the grade.

TABLE 24

Metallurgical Balance - Modification 6

Product	Weight %	Analyses % *	Distn %
		WO_3	WO_3
Sulphide cl conc (1)	37.0	0.31	5.2
Wilfley table conc (2)	7.6	18.00	62.0
" " tail (3)	12.1	2.80	15.5
Sweco separator +100m (4)	14.6	0.20	1.3
¼ deck Wilfley table conc (5)	2.2	3.04	3.1
¼ " " " tail (6)	7.7	2.13	7.4
½ " Deister " conc (7)	1.3	3.12	1.8
½ " " " tail (sand) (8)	12.2	0.33	1.8
½ deck Deister table tail (slime) (9)	5.3	0.80	1.9
Head (calcd)	100.0	2.20	100.0

* From Internal Report MS-AC-62-391.

Summary of Metallurgical Balance

Gravity conc (2 + 5 + 7)	11.1	13.29	66.9
Scheelite flot feed (3 + 6 + 9)	25.1	2.17	24.8
Circuit tail (1 + 4 + 8)	63.8	0.29	8.3
Head (calcd)	100.0	2.20	100.0
" (assayed)	-	2.00	-

TABLE 25

Screen Tests on Gravity Concentration Circuit - Modification 6

Mesh Size	Sulphide Flot Tail		Sweco Separator -100m		¼ deck Wilfley Midd	
	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing
+48 mesh	0.6	99.4	-	-	-	100.0
+65 "	2.5	96.9	-	-	0.1	99.9
+100 "	5.6	91.3	-	100.0	0.3	99.6
+150 "	8.4	82.9	8.8	91.2	10.4	89.2
+200 "	10.5	72.4	18.0	73.2	23.6	65.6
+325 "	18.8	53.6	28.4	44.8	36.3	29.3
-325 "	53.6	-	44.8	-	29.3	-
Total	100.0	-	100.0	-	100.0	-

(g) Circuit Modification 7

With this test the gravity concentration circuit was stabilized and no further changes were made for the remainder of the pilot plant investigation. Figure 4 shows the flowsheet used in this test.

While gravity concentrates of high grades and recoveries were obtained in the previous tests, the results which were obtained in this test, which are shown in Table 27, came close to duplicating the results anticipated by Mr. R. H. Ross in his report.

In this test the $\frac{1}{2}$ deck Deister table was again interchanged with the $\frac{1}{4}$ deck Wilfley table. A small laboratory Deister table was added to the circuit to recover a small scavenger concentrate.

It was observed on the first Wilfley table that the upper part of the scheelite concentrate band was very uniform in size and, visually, quite free of contaminants. During $1\frac{1}{4}$ hours of steady operation, this portion of the table concentrate was collected separately for weighing and analysis. The results summarized in Table 26 show that, by this means, 28.6% of the scheelite in the ore can be recovered directly as a "select" concentrate requiring no further beneficiation.

TABLE 26

Recovery of "Select" Concentrate on First Wilfley Table

Product	Weight %	Analyses* %					Distribution % WO_3	
		WO_3	S	P	Bi	Mo	Wilfley Table	Overall
"Select" cut	30.0	79.0	0.065	0.003	0.01	0.004	50.0	28.6
Remainder	70.0	34.1	-	-	-	-	50.0	28.6
Total conc	100.0	47.6	-	-	-	-	100.0	57.2

* From Internal Report MS-AC-62-470.

TABLE 27

Metallurgical Balance - Modification 7

Product	Weight %	Analyses % *	Distn %
		WO_3	WO_3
Sulphide cl conc (1)	37.9	0.21	3.5
Wilfley table conc (2)	2.7	47.60	57.2
" " tail (3)	8.3	2.60	9.6
Sweco separator +100m (4)	21.8	0.32	3.1
½ deck Deister table conc (5)	0.6	44.40	11.9
½ " " " tail (6)	13.4	1.6	9.5
¼ " Wilfley " conc (7)	0.7	6.4	2.0
¼ " " " tail (8)	0.9	1.66	0.7
Laboratory Deister conc (9)	0.2	3.05	0.3
" " tail (10)	13.5	0.36	2.2
Head (calcd)	100.0	2.24	100.0

* From Internal Report MS-AC-62-391.

Summary of Metallurgical Balance

Gravity conc (2 + 5 + 7 + 9)	4.2	38.15	71.4
Scheelite flot feed (3 + 6 + 8)	22.6	1.97	19.8
Circuit tail (1 + 4 + 10)	73.2	0.27	8.8
Head (calcd)	100.0	2.24	100.0
" (assayed)	-	2.26	-

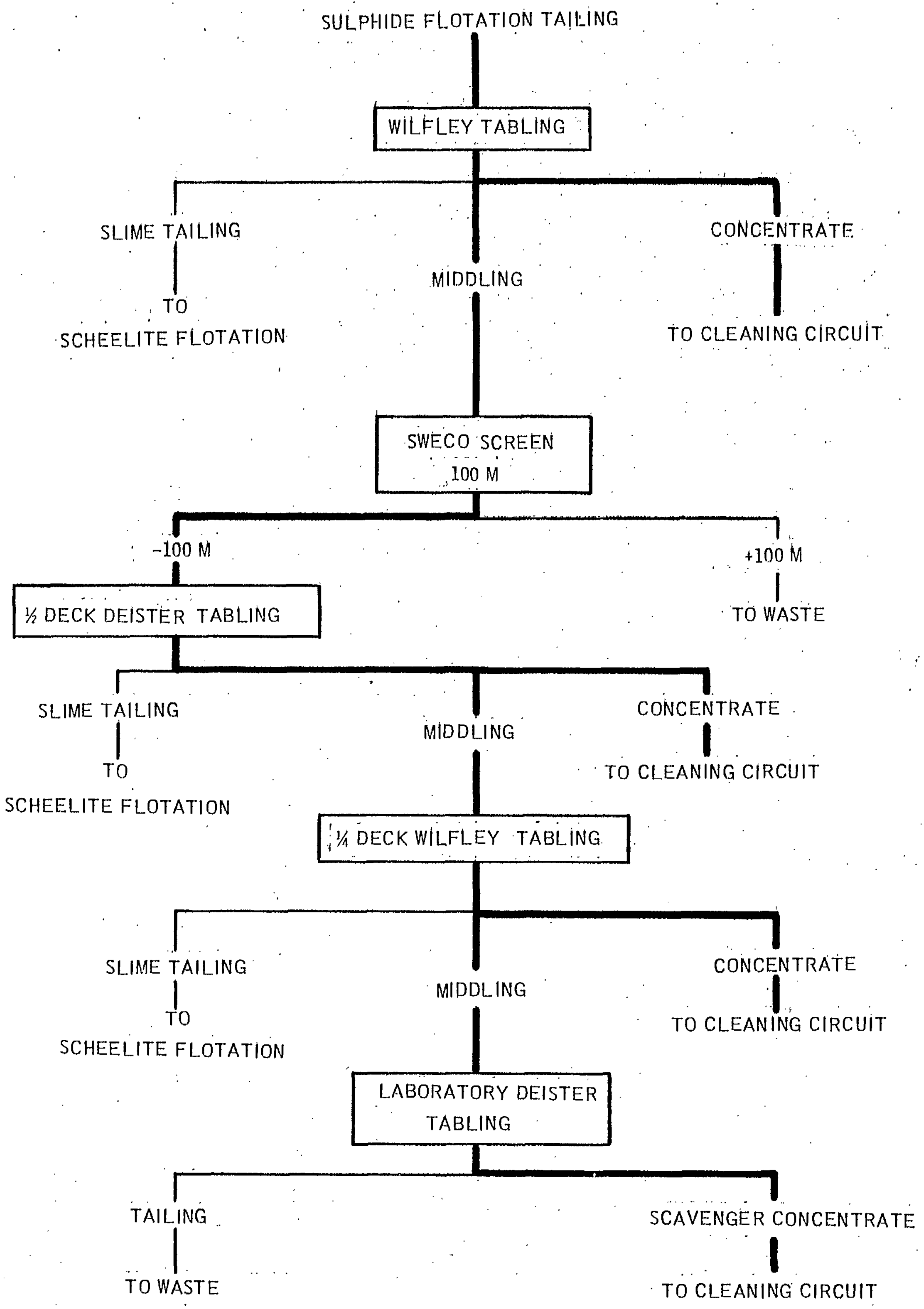

FIGURE 4 – GRAVITY CONCENTRATION CIRCUIT – MODIFICATION 7

TABLE 28

Screen Tests on Gravity Concentration Circuit - Modification 7

Mesh Size	Sulphide Flot Tail		Sweco Separator +100m		Sweco Separator -100m		Wilfley Table Midd	
	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing	% Retained	% Passing
+48 mesh	3.1	96.9	-	-	-	-	2.8	97.2
+65 "	10.0	86.9	-	-	-	-	9.8	87.4
+100 "	14.6	72.3	77.2	22.8	-	100.0	16.0	71.4
+150 "	13.5	58.8	22.8	-	6.8	93.2	15.0	56.4
+200 "	11.0	47.8	-	-	16.5	76.7	12.4	44.0
+325 "	14.4	33.4	-	-	27.8	48.9	17.2	26.8
-325 "	33.4	-	-	-	48.9	-	26.8	-
Total	100.0	-	100.0	-	100.0	-	100.0	-

Cleaning of Gravity Concentrates

(a) Tabling

All the scheelite rougher table concentrates, which contained coarse pyrrhotite, garnet, magnetite and a variety of ferro-magnesian minerals, were treated in a cleaning operation illustrated by the flowsheet in Figure 5. Without blending, the wet concentrates were fed at random to this cleaning circuit. Because of this lack of uniform feed, sampling was difficult and frequent adjustment of the tables was necessary to obtain maximum grade and recovery.

The wet concentrates were fed by a Com-Bin feeder to a Sweco separator fitted with a 50m screen. Because all available tables were in use, the +50m material was held for subsequent tabling.

The -50m fraction was treated on a large Wilfley table from which as clean a contrate as possible was taken. The table middling and table tailing were combined and pumped to a Sweco separator equipped with a 100m screen. The +100m fraction went to a laboratory Deister table from which a concentrate was recovered and a tailing discarded. The -100m fraction was treated on a $\frac{1}{2}$ deck Deister table to produce a concentrate, a slime tailing which was collected in the settling cone for subsequent scheelite flotation and a middling which was pumped to a $\frac{1}{4}$ deck Wilfley table. From the latter a concentrate was taken and the tailing was pumped to the settling cone.

The metallurgical balance for the -50m Sweco separator fraction is shown in Table 29. For simplicity, the combined tailings from the $\frac{1}{2}$ deck Deister and the $\frac{1}{4}$ deck Wilfley table are reported as scheelite flotation feed.

On completion of the -50m tabling operation, the Sweco screen oversize (+50m), very high in sulphides, was treated on a laboratory Deister table. This produced a coarse concentrate containing only about 5% WO_3 and a tailing which was held for return to the grinding circuit during clean-up of the mill circuit. No samples were taken during this coarse tabling operation.

All table concentrates from the gravity cleaning circuit were dried and combined for magnetic cleaning. The inclusion of the coarse (+50m), low-grade concentrate reduced the grade of the composite gravity concentrate to about 38% WO_3 with about 5% sulphur.

TABLE 29

Metallurgical Balance - Gravity Cleaning Circuit

Product	Weight %	Analyses % *	Distn %
		WO_3	WO_3
Sweco separator +50m (1)	20.0	3.52	5.7
Wilfley table conc (2)	12.9	63.40	66.4
½ deck Deister table conc (3)	2.8	69.20	15.7
¼ " Wilfley " " (4)	0.7	28.20	1.6
Laboratory Deister table conc (5)	0.9	22.36	1.6
" " " tail (6)	11.3	0.57	0.5
Scheelite flot feed (7)	51.4	2.00	8.5
Head (calcd)	100.0	12.31	100.0

* From Internal Report MS-AC-62-468, MS-AC-62-470.

Summary of Metallurgical Balance

Sweco separator (+50m) (1)	20.0	3.52	5.7
Gravity conc (2 + 3 + 4 + 5)	17.3	60.77	85.3
Scheelite flot feed (7)	51.4	2.00	8.5
Circuit tail (6) **	11.3	0.57	0.5
Head (calcd)	100.0	12.31	100.0

** The circuit tailing was discarded because of the small amount of contained scheelite.

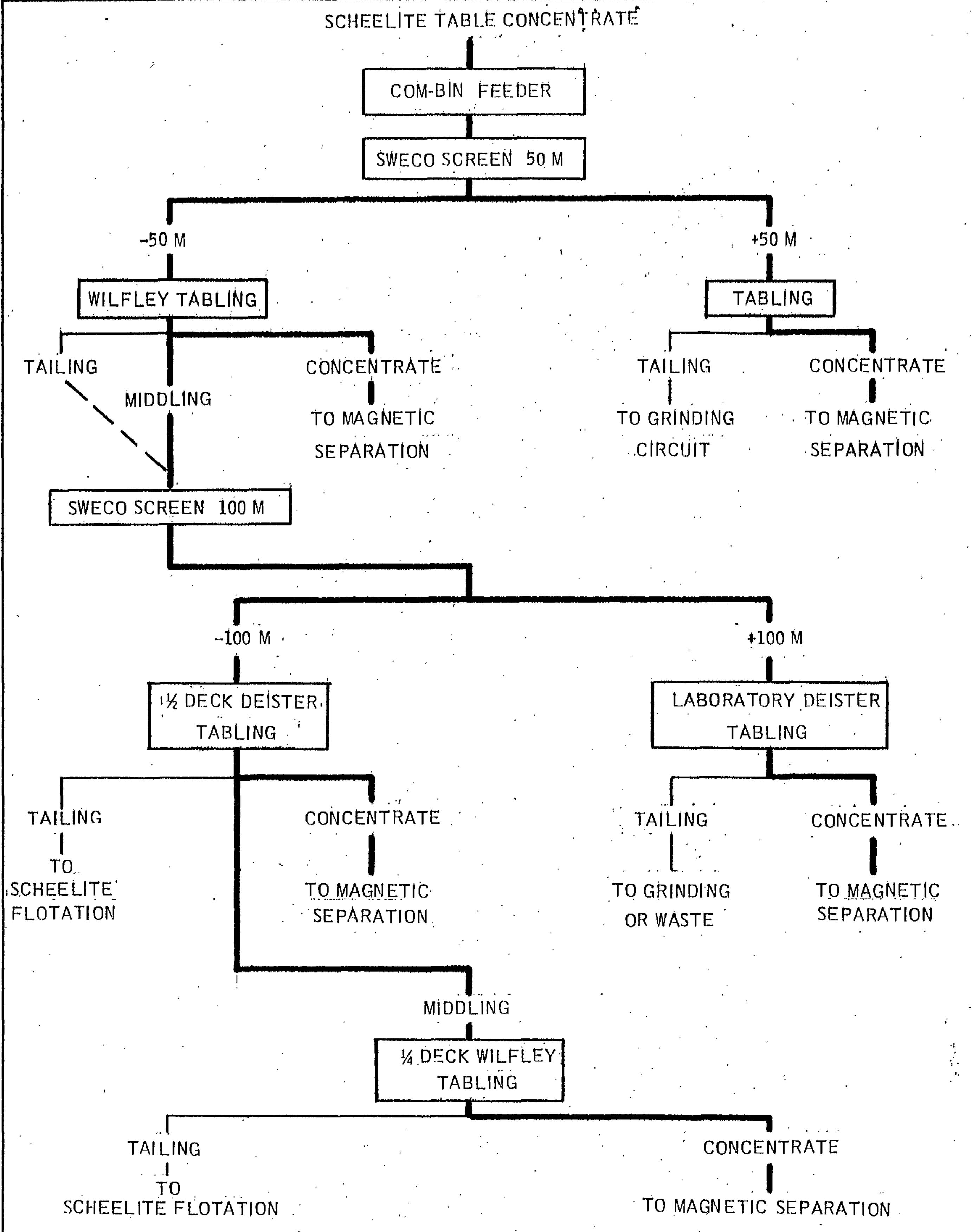

FIGURE 5 – GRAVITY CONCENTRATION CLEANING CIRCUIT

(b) Magnetic Treatment of Cleaner Table Concentrate

At all stages of gravity concentration, appreciable sulphides and garnet were observed in the scheelite concentrate band. Because a previous investigation(1) had shown that these impurities could be removed magnetically, several small-scale magnetic separation tests were made on concentrate samples taken during beneficiation in the gravity circuit. For dry tests, a Stearns high intensity magnetic separator was used; wet tests were done with a Jones high intensity machine.

In dry magnetic separation tests, the sulphur in the concentrates from the first Wilfley and the ½ deck Deister tables was reduced only to 1.13% and 1.51% respectively, as shown in Tables 30 and 31.

TABLE 30

Stearns Magnetic Cleaning of a Sample of the Wilfley Table Concentrate

Product	Weight %	Analysis % *					Distn %	
		WO_3	S	Bi	P	Mo	WO_3	S
Conc (non-mag)	74.7	73.0	1.13	0.02	0.022	0.006	98.9	20.4
Tail (mag)	25.3	2.4	12.93	-	-	-	1.1	79.6
Head (calc)	100.0	55.1	4.11	-	-	-	100.0	100.0

TABLE 31

Stearns Magnetic Cleaning of a Sample of the Deister Table Concentrate

Product	Weight %	Analysis % *					Distn %	
		WO_3	S	Bi	P	Mo	WO_3	S
Conc (non-mag)	42.6	76.6	1.51	0.03	0.041	0.007	95.9	11.6
Tail (mag)	57.4	2.45	8.54	-	-	-	4.1	88.4
Head (calcd)	100.0	34.0	5.54	-	-	-	100.0	100.0

* From Internal Report MS-AC-62-468.

TABLE 32

Jones High Intensity Wet Magnetic Cleaning of a Sample of the Wilfley Table Concentrate

Intensity	Product	Weight %	Analyses % *					Distn % WO_3
			WO_3	S	Bi	P	Mo	
10 amp	Conc (non-mag) Tail (mag)	91.2 8.8	73.28 27.60	0.45 6.28	0.04 -	0.01 -	0.008 -	96.5 3.5
	Head (calcd)	100.0	69.26	0.96	-	-	-	100.0
15 amp	Conc (non-mag) Tail (mag)	90.2 9.8	72.90 30.20	0.33 6.39	0.03 -	0.004 -	0.004 -	95.7 4.3
	Head (calcd)	100.0	68.72	0.92	-	-	-	100.0
30 amp	Conc (non-mag) Tail (mag)	89.7 10.3	77.00 30.40	0.27 6.72	0.04 -	0.006 -	0.008 -	95.7 4.3
	Head (calcd)	100.0	72.20	0.93	-	-	-	100.0

* From Internal Report MS-AC-62-470.

Because the proposed plant flowsheet was based on the use of dry, high-intensity magnetic separation, part of the final composite gravity concentrate was treated on the Dings magnetic separator. In this test, dust losses were high and considerable scheelite appeared in the magnetic tailing. Treatment of another portion on the Wetherill magnetic separator gave a less apparent loss of scheelite in the magnetics, with decreased dust losses.

Figure 6 shows the flowsheet of a test using both separators. Composite gravity concentrate was passed through the Wetherill separator and the non-magnetics produced were cleaned and re-cleaned in the Dings machine. As shown in Table 33, a high grade concentrate, low in impurities, was produced, but considerable scheelite was retained by the Dings magnetic fraction.

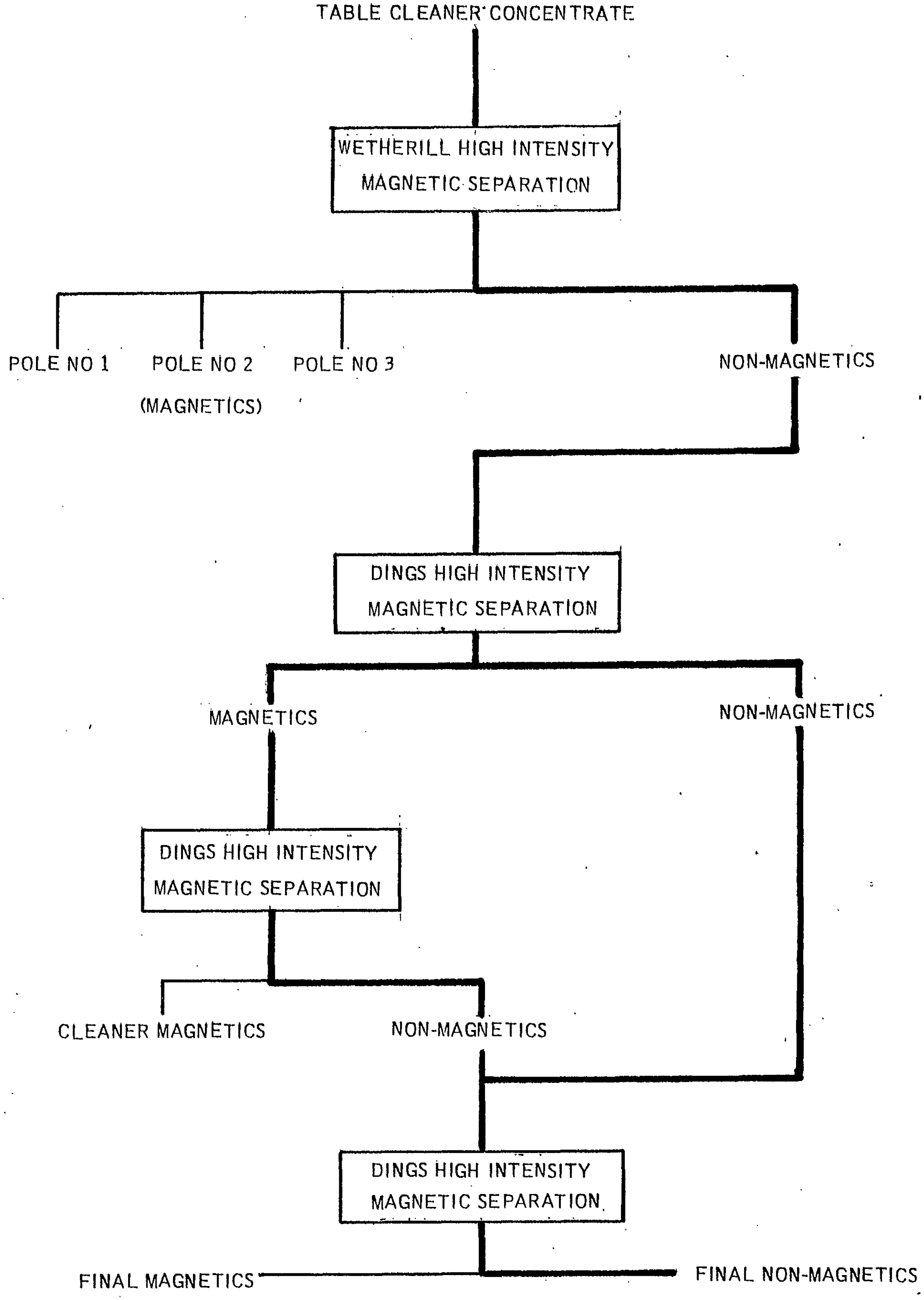

FIGURE 6 – HIGH INTENSITY DRY MAGNETIC SEPARATION

TABLE 33

Wetherill and Dings High Intensity Magnetic Separation

Product	Weight %	Analyses % *					Distn %
		WO_3	S	Bi	Mo	Cu	WO_3
Dings final conc (non-mag)	28.7	73.60	0.10	0.01	0.008	0.02	56.6
" " tail (mag)	21.6	55.20	-	-	-	-	31.9
" cl " (mag)	13.6	28.93	-	-	-	-	10.5
Wetherill " (pole No. 1 mag)	10.6	0.87	-	-	-	-	0.2
Wetherill tail (pole No. 2 mag)	12.8	1.47	-	-	-	-	0.5
Wetherill tail (pole No. 3 mag)	12.7	0.76	-	-	-	-	0.3
Head (calcd)	100.0	37.36	-	-	-	-	100.0

* From Internal Report MS-AC-62-470.

After discarding the Wetherill magnetic tailings, all other fractions were re-mixed with the unused portion of the composite gravity concentrate. All this material was then treated in the Wetherill separator adjusted to ensure near-maximum flux density. However, the non-magnetic concentrate, although of satisfactory WO_3 content, still contained 0.66% sulphur.

To check the efficiency of this Wetherill separation, a small sample representative of the feed was first treated at low intensity in the Ball-Norton separator to remove strongly magnetic material and then in the laboratory Stearns machine at maximum intensity. Even this treatment failed to reduce the sulphur content below 0.59%. Test data are summarized in Table 34.

TABLE 34

Stearns Magnetic Separation Test on a Sample of Wetherill Feed

Product	Weight %	Analyses % *						Distn %	
		WO_3	S	Bi	P	Mo	Cu	WO_3	S
Conc (non-mag)	72.7	77.1^c	0.59	0.01	0.011	0.007	0.016	97.28	9.9
Tail (mag)	26.8	5.76	14.39	-	-	-	-	2.69	88.7
Magnetics (B-N)	0.5	3.04	12.64	-	-	-	-	0.03	1.4
Head (calcd)	100.0	57.6	4.35^c	-	-	-	-	100.0	100.0

* From Internal Report MS-AC-62-506

^{c} Calculated

The foregoing Stearns test was repeated on an identical sample, after roasting it at 750°F for one hour in a muffle furnace while rabbling at 20 minute intervals. As shown in Table 35, this procedure reduced the sulphur content of the scheelite concentrate to 0.057%, but with some loss in WO_3 recovery.

TABLE 35

Stearns Magnetic Separation Test on a Sample of Wetherill Feed (Roasted)

Product	Weight %	Analyses % *						Distn %	
		WO_3	S	Bi	P	Mo	Cu	WO_3	S
Conc (non-mag)	67.1	77.1^c	0.057	0.01	0.009	0.002	0.008	89.8	1.0
Tail (mag)	22.2	24.53	2.97	-	-	-	-	9.4	18.7
Magnetics (B-N)	10.7	3.77	26.35	-	-	-	-	0.8	80.3
Head (calcd)	100.0	57.6	3.52^c	-	-	-	-	100.0	100.0

* From Internal Report MS-AC-62-506.

^{c} Calculated

(c) Flotation and Magnetic Cleaning of Cleaner Table Concentrate

A desire to avoid this high temperature treatment prompted a bench flotation test to remove the sulphides from the cleaned gravity concentrate. A 1000 g grab sample of a concentrate moderately high in sulphides was floated at a natural pH using the same reagents as in the rougher sulphide flotation ($CuSO_4$, Z-5 and Aerofroth 70). In this test, results of which are summarized in Table 36, the sulphur content of the scheelite concentrate was reduced to 0.20% and the bismuth to 0.03%.

TABLE 36

Results of Sulphide Scavenger Flotation

Product	Weight %	Analyses % *		Distn %	
		S	Bi	S	Bi
Conc	1.2	24.52	2.01	59.7	44.4
Tail	98.8	0.20	0.03	40.3	55.6
Head (calcd)	100.0	2.27	0.05	100.0	100.0

* From Internal Report MS-AC-62-468.

In view of the success of this small-scale test, all the concentrate (non-magnetics from the final pass through the Wetherill) was subjected to similar flotation in a bank of four No. 7 Denver cells. Most of the sulphides removed in the flotation froth were +100 mesh. Because of oxidation of these sulphides in previous drying, the flotation feed required long and vigorous scouring by agitation in the conditioner. No samples for chemical analysis were taken during this sulphide scavenger operation.

The scheelite concentrate (flotation tailing) was de-watered on a vacuum pan filter and dried electrically. To ensure removal of any magnetic contaminants which may have been picked up in the final stages of the cleaning, the dried concentrate was again passed through the Wetherill magnetic separator.

The effectiveness of the flotation-magnetic separation procedure is indicated by the 0.19% sulphur in the final cleaned gravity concentrate.

Chemical Analysis of Final Scheelite Gravity Concentrate

Identical samples cut from the final gravity concentrate, at -200m, were analyzed by three laboratories, with the results as shown in Table 37.

TABLE 37

Chemical Analysis of Final Scheelite Gravity Concentrate

Element	Mines Branch* Ottawa, Ont.	Climax Molybdenum Golden, Colo.	Coast Eldridge Ltd. Vancouver, B.C.
	Per cent	Per cent	Per cent
Tungsten trioxide (WO_3)	77.07	77.77	77.38
Molybdenum (Mo)	0.005	0.012	0.02
Bismuth (Bi)	0.010	nil	0.04
Phosphorus (P)	0.007 (a)	0.027	0.021
Sulphur (total S)	0.19 (b)	0.16	0.17
Copper (Cu)	0.02	0.032	0.022
Iron (total Fe)	0.32	0.32	0.42
Insoluble	1.06	1.00	1.19
Silica (HF vol)	-	not reported	0.55
Lead (Pb)	-	0.019	trace
Zinc (Zn)	-	0.018	"
Tin (Sn)	-	nil	"
Arsenic (As)	-	nil	nil
Antimony (Sb)	-	0.015	nil
Nickel (Ni)	-	0.012	trace
Columbium (Cb)	-	0.009	"
Selenium (Se)	-	nil	nil

* From Internal Report MS-AC-62-499.

a - probably low

b - by combustion method

Scheelite Flotation

Because of the small amount of material available (about 20% of the ore containing about 20% of the WO_3), this circuit, illustrated by Figure 7), was operated on an intermittent basis. The results which were obtained corresponded closely with those indicated by the small scale laboratory work.

The feed to the scheelite flotation circuit comprising the slime fraction from the sizing and the gravity concentration circuits, was pumped at 35% solids from the settling cone to a 24 in. x 36 in. Denver conditioner. The pulp, on discharge from the conditioner, was diluted to 25% solids and was fed to five No. 7 Denver cells where a scheelite rougher concentrate was removed. This concentrate was cleaned in five stages in No. 5 Denver cells, with the cleaner tailing being pumped back to the conditioner.

During flotation a narrow band of rather coarse scheelite was visible on a pilot table which handled the scheelite rougher flotation tailing. This coarse scheelite reported in the flotation feed because of the rather inefficient cyclone classification of the sulphide rougher tailing. To recover this scheelite, the rougher flotation tailing was pumped to $\frac{1}{4}$ deck Wilfley table from which a concentrate was removed. This concentrate was added to the bulk gravity concentrate during the cleaning operation.

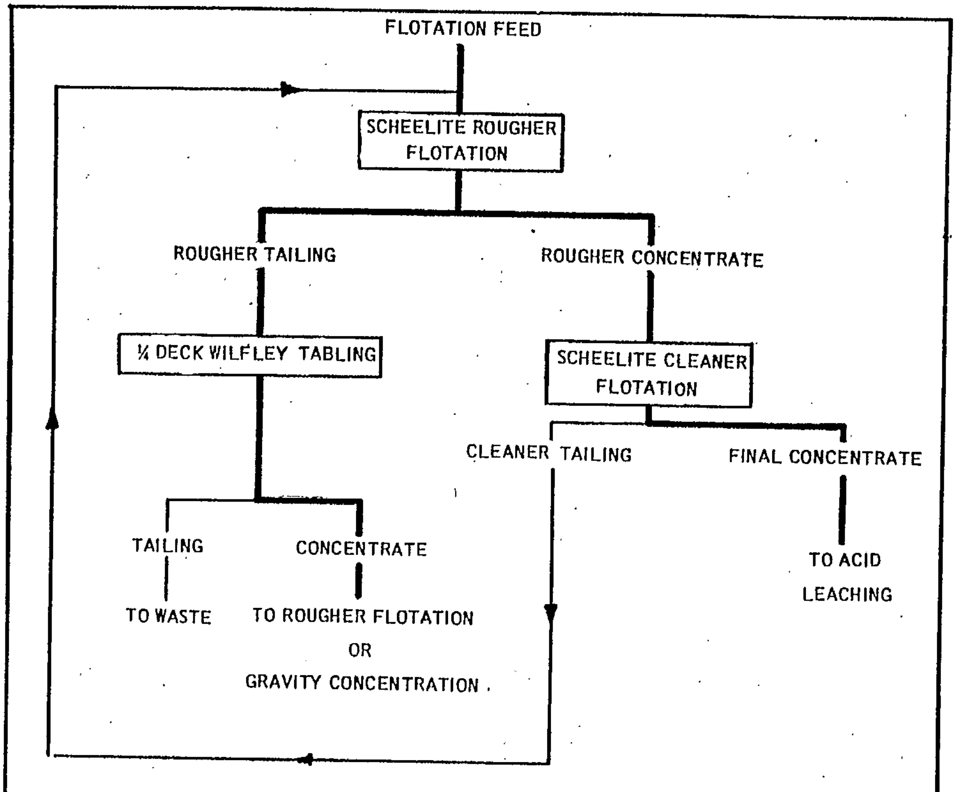

FIGURE 7 – SCHEELITE FLOTATION CIRCUIT

The pH control of the pulp was adjusted by varying the amount of sodium carbonate fed to the conditioner. The best flotation occurred between a pH of 9.5 and 10.1 and no advantage accrued from carrying a higher pH. It was necessary that the temperature of the pulp in the rougher circuit be maintained between 68°F and 72°F. On occasion, when the temperature of the pulp did rise above 80°F, an extremely poor scummy froth was obtained.

Generally the scheelite rougher froth was a dirty grey colour and of low grade, averaging about 17% WO_3 from an initial feed of about 1.7% WO_3. This grey colour was due to the presence of slimed sulphides in the flotation feed. To counteract this the writers feel that it might be advantageous to scavenge for the sulphides prior to scheelite flotation.

The reagents which were used during the flotation are shown in Table 38. The manufacture of Reagent 765 which is a fatty acid consisting of oleic and linoleic acids is being discontinued by American Cyanamid Company due to limited demand and a satisfactory substitute will have to be found.

TABLE 38

Reagents Used in Scheelite Flotation

Reagent	lb/ton Feed	Point of Addition
Sodium carbonate	3.50	Conditioner
" silicate	0.20	"
" cyanide	0.25	"
Quebracho	0.30	"
Reagent 765 (A.C.C.)	0.15	"
Aerofroth No. 70	0.04	No. 1 rougher cell
Reagent 765 (A.C.C.)	0.04	" " "
" " "	0.02	No. 3 " "
Sodium cyanide	0.05	No. 1 cleaner "
" silicate	0.01	No. 1-2 " "
Aerosol No. 18	0.04	No. 1-3-5 cleaner cell

Table 39 shows the results which were obtained during scheelite flotation. The scheelite cleaner concentrate was filtered to about 10% moisture and submitted to the Extraction Metallurgy Division for acid leaching.

TABLE 39

Metallurgical Balance of Scheelite Flotation Circuit

Product	Weight %	Analysis % *		Distn %
		WO3	S	WO3
Scheelite cl conc	2.50	50.88	5.12	73.6
$\frac{1}{4}$ deck Wilfley table conc	0.13	15.20	-	0.5
" " " " tail	97.37	0.46	-	25.9
Head (calcd)	100.0	1.73	-	100.0
" (assayed)	-	1.65	2.74	-

* From Internal Report MS-AC-62-468.

A small percentage of Bi, Cu and Mo was present in the scheelite cleaner concentrate.

Hydrochloric Acid Leaching of Flotation Cleaner Concentrate*

Introduction

Four buckets containing wet flotation concentrate weighing 118 lb (dry weight) were received from the Mineral Processing Division.

A meeting was held with Mr. R. W. Burton in which he outlined the conditions to be followed in the leaching of the flotation concentrate using hydrochloric acid, and in which the following specifications were given for the leached concentrate.

TABLE 40

Specifications for Acid Leached Concentrate

Element		Range %
Sulphur	(S)	0.25
Bismuth	(Bi)	0.05
Copper	(Cu)	0.02
Molybdenum	(Mo)	0.02
Phosphorus	(P)	0.05

Procedure

After discussion with Mr. Burton, it was decided in view of his and our past experience that the concentrate would be leached in two stages. Pulp density would be 65-75% solids. The first stage would be operated with no heat addition and concentrated hydrochloric acid would be added until effervescence stopped. At this time heat would be applied to bring the temperature up to 149°F. In the second stage, run immediately after the first stage with no liquor separation, a free-acid value of 5% would be maintained. Retention time would be one hour in the second stage.

The leach liquor would then be decanted while still hot. The leach residue would be repulped with hot water followed by decantation several times, then pan-filtered using hot water (around 120°F) until the pH of the wash water reached 7.

It was requested that the head and residue samples be analyzed for WO_3, S, P, Mo, Cu, Bi and CO_2, and the solutions for free HCl, WO_3 and P.

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* From Internal Report EMT 62-13 by B. H. Lucas, Extraction Metallurgy Division, Mines Branch, August 15, 1962.

The leach was carried out in plastic tanks with plastic-coated agitators. Steam was introduced through open plastic pipes. A rubber-lined pan filter and a rubber-lined filtrate receiver were used in the wash circuit.

Reagent grade hydrochloric acid was used, and free acid determinations were done using 1 N. NaOH and methyl orange indicator.

Results

The leach was done in two batches using a total of 96.5 lb (dry) of flotation concentrate at 70% solids initially. The solids content was reduced to 45-50% solids by the end of the leach due to acid and steam additions.

The temperature rose quickly to 153°F upon steam addition. The slurries were immediately cooled; but in 27 minutes (32 minutes from initial steam addition) a blue colouration, indicating soluble tungsten appeared. At this point the leaching was stopped.

The washing of the residue with water to pH 7 was very slow, but the addition of a small amount of sodium carbonate quickly brought the pH to 8. However, it caused frothing and a separation of a black slimy material. This slime caused blinding of the filter cloth; but after its removal the remaining residue filtered rapidly.

TABLE 41

Chemical Analyses* of Leach Feed

Element		Per Cent
Tungsten trioxide	(WO_3)	50.5
Phosphorus	(P)	0.61
Carbon dioxide	(CO_2 comb.)	8.80
" "	(CO_2 evol.)	6.92
Copper	(Cu)	0.005
Sulphur	(Total S)	1.33
Manganese	(Mn)	0.18
Molybdenum	(Mo)	0.0015
Bismuth	(Bi)	0.015

* From Report EMD No. 5/62-2.

TABLE 42

Leach Details

Elapsed Time (min)	Temp (°F)	HCl* Addition (lb cum)	Free Acid Conc (g/l)	Observations
0	32			
4	52	12	-	
11	64	24	-	
19	75	36	-	
24	81	48	-	strong yellow colour to leaching
33	-	49.6	49.2	steam on
38	131	50.0		steam off
65	117	stop		blue colour to leach liquor and H_2S odour
84				completed decant. of leach liquor wt leach liquor = 68 lb

* reagent grade HCl : 37% HCl (average)

TABLE 43

Wash Details

Elapsed Time (min)	Wt H_2O added (lb)	Temp (°F)	Wt Filtrate (lb)	pH	Observations
36	40	99			Repulping and decanting
	60	86			
106	60	79			
141	60	75			
	60	81	310	3.4	transferred to pan filter
	120			3.6	washing - 80°-90° H_2O
	5			8.0	added approx. 1/2 lb Na_2CO_3
			150		Na_2CO_3 caused fine precipitation
			92		
	Total wt = 552 lb				

TABLE 44

Chemical Analysis of Leach Products

Sample	Wt (lb)	S.G.	Vol (gal)	Analyses (g/l, %)						
				WO_3	P	S	Cu	Mo	Bi	Free HCl***
Leach liquor	68	1.25	5.44	0.95	7.8					45.0
Wash liquor	552	1.02	54.1	0.42	0.52					3.0
Residue*					0.003	1.13				
Final Residue**	68.5			68.95	0.005	1.46			0.01	

* Grab sample.

** The final residue was taken back to Mineral Processing Division and given further treatment. Assays are calculated values.

*** Acid is actually a mixture of H_3PO_4, H_2PO_4 and HCl.

Conclusions and Discussion

The low grade scheelite flotation concentrate was successfully leached in a two-stage operation using concentrated hydrochloric acid and heat.

After the addition of 380 lb 100% HCl/ton dry feed over a 33 min period the calcium carbonate was eliminated. The temperature averaged 89°F. The phosphorus was eliminated by leaching for a further 32 min at an average temperature of 138°F. It was necessary to add only 3.5 lb 100% HCl/ton dry feed in this second stage.

Longer second-stage retention time would have been required if the temperature had not been allowed to rise to 149°F. The washing of the leach residue to neutral pH by the quantitative addition of a neutralizing agent such as sodium carbonate will need further study. If processing of the residue to remove sulphur is contemplated, there would be no advantage in going beyond a pH of 3.5 to 4.

The tungsten content of the flotation concentrate was raised from 50.5% WO_3 to 68.95% WO_3 and with the exception of the sulphur, all the specifications listed in Table 40 were met.

Treatment of Acid Leached Flotation Concentrate

Acid-leaching of the scheelite cleaner concentrate was successful in removing or reducing to low levels the calcium and the phosphorus. However, the 1.46% sulphur remaining in the acid-leached concentrate made necessary the investigation of supplementary cleaning operations which are detailed below.

(a) Flotation

Several attempts were made to float the sulphides from the acid-leached concentrate with the same reagents which were used in the sulphide flotation circuit. Because of completely unmanageable frothing conditions, all such attempts proved unsuccessful.

(b) High Intensity Wet Magnetic Separation

The entire acid-leached concentrate was passed through a Jones high intensity wet magnetic separator operating at the maximum intensity of 30 amp. The magnetic fraction was floated using the previously mentioned sulphide flotation reagents. The results which may be seen in Table 45 show the rather poor elimination of sulphur which was obtained.

TABLE 45

Magnetic Separation of Acid-Leached Concentrate

Product	Weight g	Weight %	Analyses %[*]				Distn %	
			WO_3	S	P	Bi	WO_3	S
Conc (non-mag)	17,282	56.7	71.68	1.20	0.006	0.01	58.0	44.0
Midd	11,392	37.4	72.00	1.37	0.005	0.01	39.0	33.9
Sulphide tail	1,094	3.6	33.92	0.98	-	-	1.8	2.4
" conc	716	2.3	37.28	12.55	-	-	1.2	19.7
Head (calcd)	30,484	100.0	68.95	1.46	-	-	100.0	100.0

[*] From Internal Report MS-AC-62-528.

The high sulphur content (1.27%, calculated from the combined concentrate and middling shown in Table 45) served to confirm an earlier observation, mentioned in the section dealing with scheelite flotation, that the introduction of a scavenger sulphide flotation section, prior to scheelite flotation, might be advantageous.

(c) Preparation of Synthetic Concentrate

Because of the very low sulphur content of the gravity concentrate, equal portions of this and the high-sulphur acid-leached flotation concentrate were mixed by dry grinding to about 85% -325m. This produced a fine material of marketable grade representative of a flotation concentrate which might be produced by the application of successful cleaning techniques.

(d) Chemical Analysis of Synthetic Concentrate

Identical samples of this concentrate at all -200m were analysed by three laboratories with the results shown in Table 46.

TABLE 46

Chemical Analysis[*] of Final Acid-Leached (Synthetic) Concentrate

Element	Ledoux & Co. Ltd. Teaneck, N.J.	Coast Eldridge Ltd. Vancouver 1, B.C.	Canada Tungsten Vancouver 1, B.C.
	Percent	Percent	Percent
Tungsten trioxide (WO_3)	73.09	75.16	69.10
Molybdenum (Mo)	0.02	0.02	0.013
Bismuth (Bi)	0.09	0.03	0.016
Phosphorus (P)	0.02	0.028	0.008
Sulphur (total S)	0.69	0.43	0.43
Copper (Cu)	0.01	0.023	0.022
Iron (total Fe)	0.71	1.06	1.04
Insoluble	not reported	5.20	4.96
Silica (H.F. Vol)	2.38	4.15	not reported
Semi-quantitative			
Lead (Pb)	0.01	nil	nil
Zinc (Zn)	0.01	"	"
Tin (Sn)	nil	trace	trace
Arsenic (As)	0.01	"	"
Antimony (Sb)	nil	nil	nil
Nickel (Ni)	not reported	not detected	trace
Columbium (Cb)	" "	" "	"
Selenium (Se)	" "	trace	nil

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[*] Commercial laboratory analyses on shipped concentrate taken from correspondence received.

(e) Roasting of Acid-Leached Concentrate

Because of the successful removal of sulphur from the gravity concentrate by roasting and high-intensity magnetic separation, similar small-scale tests were made on samples of acid-leached scheelite flotation concentrate.

Two similar samples were roasted for one hour at 750°F in a muffle furnace and rabbled at 20-minute intervals. After removing one sample to cool, the furnace temperature was increased to 1470°F, held at that point for one hour and allowed to cool slowly to room temperature before the second sample was removed. A small portion of each sample was taken for analysis; the remainder of each was divided into two parts for dry and wet magnetic separation tests.

Roasting at 750°F lowered the sulphur content of the concentrate from 1.13% to 0.64%. Subsequent dry magnetic separation reduced this to 0.41%, while recovering 91.7% of the tungsten. Because approximately 50% of the sulphur in the roasted concentrate was present as water-soluble sulphate, wet magnetic separation removed both magnetic sulphide and soluble sulphate, leaving only 0.16% sulphur in the final non-magnetic concentrate with 95.8% of the tungsten. The results are summarized in Table 47.

TABLE 47

Sulphur Removal from Flotation Concentrate Roasted at 750°F

Product	Weight %	Analysis % *			Distn %	
		WO_3	S (tot)	S (SO_4)	WO_3	S (tot)
Conc (as rec'd)	-	73.47	1.13	-	-	-
" (roasted)	-	-	0.64	0.35	-	-
Dry mag separation						
Ball-Norton mag	0.8	n.d.	16.15	-	-	18.3
Stearns mag	10.0	61.2	2.11	-	8.3	29.9
" non-mag	89.2	75.5^{c}	0.41	-	91.7	51.8
Feed	100.0	73.47	0.70^{c}	-	100.0	100.0
Wet mag separation						
Jones mag	7.6	40.32	2.75	-	4.2	30.6
" non-mag	92.4	76.2^{c}	0.16	-	95.8	21.6
Feed	100.0	73.47	0.64	-	100.0	52.2**

* From Internal Reports MS-AC-62-578 and -600.

** 47.8% of the sulphur was water soluble.

c calculated

n.d. none detected

Roasting at 1470°F was less effective in eliminating sulphur but resulted in complete conversion from sulphide to sulphate, with only part of the latter in water-soluble form. Consequently, little sulphur was removed in magnetic form by either dry or wet magnetic separation. However, leaching of some soluble sulphate in the wet test reduced the sulphur content of the concentrate to 0.46%. Results of this test are shown in Table 48.

TABLE 48

Sulphur Removal from Flotation Concentrate Roasted at 1470°F

Product	Weight %	Analysis % *			Distn %	
		WO_3	S (tot)	S (SO_4)	WO_3	S (tot)
Head (analyzed)	-	73.47	1.13	-	-	-
Conc (roasted)	-	-	0.75	0.75	-	-
Dry mag separation						
Ball-Norton mag	nil	-	-	-	-	-
Stearns mag	1.9	46.4	1.10	-	1.2	2.7
" non-mag	98.1	74.0^{c}	0.77	-	98.8	97.3
Feed	100.0	73.47	0.78^{c}	-	100.0	100.0
Wet mag separation						
Jones mag	7.5	47.68	0.45	-	4.9	4.4
" non-mag	92.5	75.6^{c}	0.46	-	95.1	57.0
Feed	100.0	73.47	0.75	-	100.0	61.4**

* From Internal Reports MS-AC-62-578 and -600.

** 38.6% of the sulphur was water soluble.

c calculated

OVERALL RECOVERIES AND METALLURGICAL BALANCE

A condensed metallurgical balance, typical of the pilot mill operation, is shown in Table 49. These figures, extracted from the report of R. W. Burton(5), integrate the best results of the several experimental flowsheets and show the trend of the separation obtained rather than precise details of the several products recovered. The gravity (table) concentrate and the acid-leached flotation concentrate shipped to American Metal Climax Inc. were both higher in WO_3 content than the corresponding products shown in this table.

Although scheelite recovered by gravity concentration was somewhat greater than expected and that by flotation correspondingly lower, the overall recovery in this pilot plant investigation of approximately 85%, was the same as that obtained in test work elsewhere.

TABLE 49

Condensed Metallurgical Balance

Product	Weight %	Analyses %			Distn %		
		WO_3	Cu	S	WO_3	Cu	S
Sulphide conc	38.0	0.20	0.29	36.00	3.2	27.5	92.0
Copper "	1.3	0.10	22.20	31.90	-	70.0	3.0
Table "	2.2	76.50	-	0.27	70.0	-	3.0
Table tail	40.0	0.30	0.01	1.20	5.0	1.2	
Scheelite flot conc*	0.5	67.00	-	0.40	15.1	-	2.0
" " tail	18.0	0.90	0.02	2.30	6.7	1.3	
Head (calcd)	100.0	2.40	0.40	15.00	100.0	100.0	100.0

* Following acid-treatment.

CONCLUSIONS AND DISCUSSION

During the ten weeks of the pilot plant tests, 33.7 short tons of ore were milled. Despite losses of scheelite in sampling, by spillage and for experimental purposes, concentrates typical of marketable scheelite were produced in the following quantities: 1083 lb gravity concentrate and 90 lb flotation concentrate.

Because of the complexity of the flowsheet and the many changes made during the pilot plant investigation, the writers feel that the several sections of the circuit warrant separate emphasis, particularly since the results of the tests differed somewhat from those indicated by earlier small-scale tests.

Grinding

Throughout the investigation every effort was made to maintain the pulp density in the rod mill at or above 72% solids. Any appreciable reduction from this figure led to overgrinding and sliming of the scheelite and reduced the recovery in the gravity concentration circuit.

Classification

The DSM screen operated best when the screen deck was flooded with pulp. A tendency to blind after a short operation was overcome by fastening a solenoid vibrator to the screen frame. The 0.5 mm screen first used in the circuit permitted sulphides, too coarse for flotation, to get into the mill flow. These ultimately reported in the gravity concentrate. This condition was substantially corrected by the use of a 0.35 mm screen which gave a slightly finer overall grind but did not cause any large increase in the scheelite slime losses.

In his report(4) Mr. Ross felt that the use of a DSM screen alone would result in too much dilution of the feed to sulphide flotation and recommended that a de-watering cyclone be used in conjunction with the screen to avoid the need for primary thickening. However, during the pilot plant tests it was found that when the pulp density of the rod mill discharge was maintained at or above 72% solids no serious dilution was encountered and neither cycloning nor thickening was required at this stage.

Sulphide Flotation

No unusual problems were encountered in the sulphide flotation circuit. Because of the long flotation time, stage additions of reagents were essential to the adequate removal of the sulphides. Probably no more than about 95% of the sulphides can be eliminated in a primary flotation circuit. The 3% loss of WO_3 at this stage, mostly by mechanical entrainment of the scheelite grains in the heavy sulphide froth, might have been reduced if more than one stage of cleaning had been employed. Small additions of sodium silicate in the cleaner circuit produced a brittle froth which allowed part of the trapped scheelite to be recovered.

Copper Flotation

Although little attention was given to this part of the circuit, in the few days of its operation a copper recovery of 70% was obtained in a concentrate containing about 16% copper and 0.1 oz Au/ton. Super-cleaning of this concentrate increased the copper grade to 27.34% but reduced the gold to 0.06 oz/ton. This reduction in gold was probably caused by the depressing effect on the gold-bearing minerals of the extra lime used in the recleaning operation.

Sizing

Sizing of the feed to the gravity concentration circuit is essential to high recovery on the tables. In the pilot plant tests, cyclone sizing was unsatisfactory due to a relatively small pulp flow and oversize pumps. Therefore, the large Wilfley table was used in the dual role of desliming the sulphide flotation tailing and collecting a gravity concentrate.

If a DSM screen as fine as 0.35 mm is used for primary classification, then the sulphide flotation tailing, after removal of the -325m material by desliming or cycloning, might profitably be split into two fractions instead of three as proposed by Mr. Ross(4). The mesh size for this separation would best be determined under plant operating conditions.

Gravity Concentration

Of the several table circuit variations used in attempts to obtain a maximum recovery of scheelite, the flowsheet shown in Figure 4 gave results closely paralleling those predicted by Mr. Ross(4). This flowsheet produced a bulk gravity concentrate containing 38.15% WO_3 for a recovery of 71.4% of the tungsten in the ore. Sizing of the Wilfley table middling on a 100m screen resulted in the rejection of 21.8% of the ore containing 3.1% of the WO_3. The scheelite in this fraction was locked in the gangue and would require further grinding for liberation.

An interesting aspect of this flowsheet was that it was possible to cut from the Wilfley table a "select" concentrate requiring no further beneficiation. This concentrate, assaying 79.0% WO_3, represented 50.0% of the tungsten recovered by this table and 28.6% of the tungsten in the ore.

Cleaning of Gravity Concentrate

A moderately high grade scheelite concentrate (60.77% WO_3) was produced in the cleaner tabling operation illustrated by the flowsheet in Figure 5, but this did not remove garnet, nor did it reduce other impurities, particularly sulphur, to acceptable levels.

Although small-scale wet high-intensity magnetic cleaning reduced the sulphur content of the scheelite concentrate to 0.27%, the limited commercial experience with such wet separation prompted more attention to conventional dry magnetic techniques. However, even on a laboratory Stearns high-intensity machine, sulphur content could not be reduced below 0.59% compared to 0.66% obtained by large-scale treatment of the dried concentrate on a Wetherill separator. A Dings machine produced a very clean final concentrate containing only 0.10% sulphur, but low scheelite recovery and severe dust losses discouraged its further use. Only after low temperature roasting (at 750°F) to increase the magnetic susceptibility of the sulphides could a concentrate with less than 0.10% sulphur be produced by Stearns dry magnetic separation.

However, satisfactory removal of sulphur (to 0.19%) was achieved on a large scale by sulphide scavenger flotation followed by Wetherill magnetic separation. The effectiveness of this technique is further indicated by the low level of other impurities, including bismuth, shown by the chemical analysis of the final shipped concentrate (Table 37). A simpler procedure might be to send the cleaner table concentrate, or even the rougher table concentrate, directly to a scavenger flotation step to remove residual sulphides before magnetic cleaning.

Scheelite Flotation

Despite brief, intermittent operation in this phase of the pilot plant tests, the results were similar to those predicted from previous investigations. Scheelite flotation produced a cleaner concentrate of about 50% WO_3 with a recovery of 73% of the WO_3 in the feed, equivalent to about 15% of the WO_3 in the original ore. The circuit operated best at a pH between 9.5 and 10.1 and a pulp temperature of 68°F to 72°F.

The rougher froth at best was dirty and of low grade (about 17% WO_3). The writers feel that, if at all possible, some effort should be made to scavenge the scheelite flotation feed to remove sulphides which had resisted the initial bulk sulphide flotation. Such a scavenger operation would have the added advantage of removing some of the copper, bismuth and molybdenum which otherwise appear in the concentrate.

Acid Leaching of Flotation Concentrate

This was a relatively simple operation in which no unexpected problems were encountered. The low grade (50% WO_3) flotation cleaner concentrate was successfully leached to remove the calcium and the phosphorus in a two-stage system using concentrated hydrochloric acid and heat. The resulting concentrate containing 68.95% WO_3 and 1.46% S, met the specifications for all impurities except sulphur.

Treatment of Acid-Leached Concentrate

Because of interference caused by the reagents used in scheelite flotation, neither magnetic separation nor supplementary sulphide flotation reduced the sulphur in the acid-leached concentrate to an acceptable level. However, in a small-scale test, roasting of the concentrate at a relatively low temperature (750°F) eliminated about 40% of the sulphur and subsequent wet magnetic separation an equal amount leaving only 0.16% S. The WO_3 lost in the combined operation amounted to only 4.3%. Although this technique was not applied to the bulk of the acid-leached concentrate, further investigation of the procedure is recommended.

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