O T T A W A May 28th, 1943.

R E P O R T

of the

ORE DRESSING AND METALLURGICAL LABORATORIES.

Investigation No. 1376.

1376

Concentration of a Scheelite Ore from the Emerald Tungsten Project of Wartime Metals Corporation at Salmo, British Columbia.

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(Copy No.___.)

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Shipments:

Six separate sacks of ore, having a total weight of 393 pounds, were received on October 29th, 1943. One other sack, weight 110 pounds, was received on January 14th, 1943. These shipments were consigned to these Laboratories by Mr. Dennis W. L. Fairbairn, Emerald Tungsten Project, Salmo, British Columbia, at the request of Dr. B. S. W. Buffam, of the office of the Metals Controller, Department of Munitions and Supply, Ottawa, Ontario.

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Location of the Property:

The property of the Emerald Tungsten Project, which is presently being operated by the Wartime Metals Corporation, is situated near Salmo, in the Nelson mining division of British Columbia.

Where Various Samples were Taken:

Dr. Buffam's letter of October 20th, 1942, stated that Samples Nos. 1, 2, 3, and 4 are all from the highly oxidized, high-grade North section of the main ore zone. Samples Nos. 5 and 6 contain low-grade ore from the skarn bands.

Sack No. 1. - Quartz ore - north end of ore zone.

Sack No. 2. - Oxidized ore - from pit, over an average depth of 8 feet.

Sack No. 3. - Soft oxidized ore - from two trenches about centre of ore section, immediately north of sulphide trench (Sack No. 4).

Sack No. 4. - From 10-foot sulphide trench.

Sack No. 5. - Skarn ore from above - Hedley Cottage.

Sack No. 6. - Skarn ore from road, 500 feet south of stable.

Sack No. 7. - South ore body - typical run-of-mine ore.

Sampling and Analysis:

After crushing, cutting and grinding by standard methods, representative samples of each sack were obtained which assayed as follows:

Sack No.	Weight, pounds	ASSAYS, per cent					
		WO_3	S	P	Mo	Fe	CaO
1	70	2.92	0.31	0.18	0.06	4.74	-
2	97	3.68	0.63	0.07	0.02	10.20	
3	64	2.05	3.64	0.20	0.05	28.85	
4	83	4.08	21.60	0.17	0.02	33.30	
5	39	0.12	0.07	0.065	0.01	4.74	
6	40	0.13	0.25	0.09	0.09	4.21	
7	110	0.98	11.68	0.08	0.05	-	11.19

(Continued on next page)

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(Sampling and Analysis, cont'd) -

On Composite Sample A, consisting of equal quantities of Sacks Nos. 1, 2 and 3, the analysis was as follows:

		Per cent
WO_3	-	3.11
S	-	1.28
P	-	0.14
Fe	-	13.35
Mo	-	0.04

Characteristics of the Ore:

Twelve polished sections, two from each of Samples Nos. 1 to 6 were prepared and examined microscopically for the purpose of determining the character of the ore.

Sample No. 1 -

Gangue material, which comprises almost the whole of the polished surfaces, consists essentially of milky-white to black fine-textured quartz which exhibits a few local, light-brown stains of iron oxides, especially along fractures. Unevenly distributed throughout the quartz are irregular grains and subhedral crystals which, under a "mineralight", fluoresce with the characteristic glow of scheelite. The glowing particles range from mere pin-points of light up to grains about 1½ millimetres in size and, in places, they are roughly concentrated into small fluorescent patches several millimetres across. When examined by means of the binocular microscope, however, these glowing patches are seen to be composed of individual grains which probably average about 150 microns (100 Tyler mesh) in size.

Metallic mineralization is very sparse and is represented by pyrite and "limonite" as rare, small, disseminated grains and crystals which are often associated.

(Continued on next page)

(Characteristics of the Ore, cont'd) -

Sample No. 2 -

The polished sections are composed predominantly of gangue which consists of milky-white quartz deeply stained with iron oxides, especially along fractures. When viewed under ultra-violet radiation it is seen to contain scheelite in the same modes of occurrence and grain sizes as in Sample No. 1.

"Limonite" is the only metallic mineral visible in the sections. In addition to the rust stains mentioned above it is common as irregular grains and small masses in gangue.

Sample No. 3 -

The two polished sections representing this sample are entirely different in appearance. One polished surface is composed of an intimate admixture of more or less massive pyrrhotite and pyrite with rare small grains of chalcopyrite. Scattered throughout the metallic mass are numerous inclusions of quartz, coarse to fine in size. The scheelite content of this section is almost nil, as when viewed megascopically under a "mineralight" no fluorescence can be seen, but by using a binocular microscope five or six tiny points of light are visible.

The other polished surface consists entirely of milky-white quartz with reddish brown stains of iron oxides along fractures. These stains, together with one small mass and a few irregular grains of "limonite", represent the only metallic mineralization present. Under an ultra-violet lamp one small patch, two or three millimetres wide, fluoresces with the characteristic colour of scheelite. As in Sample No. 1, however, when examined by means of a binocular microscope, this glowing patch is seen to be composed of a

(Characteristics of the Ore, cont'd) -

group of individual grains ranging in size from about one millimetre down to a pin-point of light.

Sample No. 4 -

The two sections of this sample are similar to the first one described under Sample No. 3. Although gangue forms the minor amount of the polished surfaces, it is abundantly distributed throughout the massive sulphides as irregular grains and small patches. For the most part it appears to be highly quartzitic but, as in Sample No. 1, it contains irregular grains and subhedral crystals of a softer mineral which fluoresces like scheelite under a "mineralight". These fluorescent grains probably average about 750 microns (-28+20 Tyler mesh) in size and are abundant in one section, almost absent in the other.

Sample No. 5 -

Metallic minerals are scarce and were observed only in one section where occasional tiny grains of pyrite, pyrrhotite, and "limonite" occur in gangue. The latter material is composed of a fine-grained mixture of rather hard, rock minerals which contain abundant finely disseminated carbonate. The polished surface of one section presents a mottled appearance of white to almost black material, while in the other section it has a roughly banded appearance composed of alternate layers of different colours.

A comparatively small amount of scheelite appears to be present in the sections when examined by means of a "mineralight". It occurs as small, scattered, fluorescent grains up to about one-half millimetre in size.

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(Characteristics of the Ore, cont'd) -

Sample No. 6 -

Pyrrhotite and pyrite are common in one section as small intimately admixed masses and coarse to fine grains disseminated in gangue. Molybdenite is visible in small amount in the other section as small curved plates (up to about 1 mm. in length) in gangue. The latter is essentially the same as in Sample No. 5.

When examined in ultra-violet radiation the section in which no molybdenite was observed shows scattered grains of scheelite up to about $\frac{5}{4}$ millimetre in size. While no fluorescence which the writer could attribute to scheelite was visible in the other section, specks of fluorescent light of a golden yellow colour are to be seen in and around the plates of molybdenite, indicating the probable presence of the secondary mineral Powellite (($Ca(Mo,W)O_4$)).

Investigative Work:

In his letter of October 20th, 1942, Dr. Buffam requested that a separate analysis of each sack be made and stated that Sacks Nos. 1, 2 and 3 could then be combined to form a composite sample (Lot A) for the test work.

Dr. Buffam also requested that on Sack No. 4 separate tests be made; that the ore in Sack No. 4 be then combined with the composite sample of Sacks Nos. 1, 2 and 3, in the proportion of 5 per cent of Sack No. 4 to 95 per cent of the composite sample, to form a further lot for test work (Lot B); and that Sacks Nos. 5 and 6 of the Skarn ore be combined for test work (Lot C).

On Sack No. 7, the shipment of January 14th, 1943,

(Investigative Work, cont'd) -

separate mill tests were made.

Sacks Nos. 1 to 4, inclusive, were all highly oxidized and consisted mostly of surface material from the North section of the main ore zone. Sacks Nos. 5 and 6, consisting of Skarn ore, were also surface material, somewhat oxidized. Sack No. 7 consisted of typical run-of-mine material from the South ore body and apparently was freshly broken underground and showed little oxidation. The North End ore body (Sacks Nos. 1, 2, 3 and 4) is quite small and the bulk of the ore is reported to be in the South end. The Skarn ore body (Sacks Nos. 5 and 6) has a vast tonnage of low-grade material.

The test work on the four different lots consisted of flotation and table concentrations on Lots A, B, and C, and flotation, table and magnetic concentration of Lot No. 7.

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Results of Test Work:

On Lot A, table concentration of the screened products gave concentrates assaying 60 per cent WO_3. By flotation concentration, concentrates assaying 60 per cent WO_3 also were obtained, and a table concentration of this flotation concentrate gave a concentrate assaying 72 per cent WO_3. Recoveries by table concentration gave about 40 per cent of total WO_3 in the concentrate and an additional 21 per cent in the table middling. In the flotation concentration, 69 per cent recovery was obtained in the cleaner concentrate with an additional 9 per cent in the middling product.

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(Results of Test Work, cont'd) -

On Sack No. 4, a flotation cleaner concentrate assaying 49.4 per cent WO_3 was obtained with a recovery of 89.3 per cent. 4.2 per cent of the WO_3 was in the middling product.

On Lot B, a flotation concentrate assaying 56 per cent WO_3 was secured with a recovery of 82 per cent.

On Lot C, a flotation cleaner concentrate was made assaying 7.38 per cent WO_3 giving a recovery of 55 per cent with a further 45 per cent of the WO_3 remaining in the middling product.

On Sack No. 7, a flotation concentrate assaying 36 per cent WO_3 with a recovery of 79 per cent was obtained. 19.3 per cent of the WO_3 was in the middling product. On this lot of ore, table concentration followed by flotation of the table tailings and magnetic separation and retabling of the primary table concentrates gave overall recoveries of about 90 per cent of the WO_3. The grade of concentrates was 45 per cent WO_3 in the final table concentrate and 31 per cent WO_3 in the cleaner flotation concentrate. When a super-panner machine replaced the table in the final cleaning operation, the grade of concentrate was raised to 68 per cent WO_3.

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In the details of the test work which follow

Lot A - Composite sample of Sacks Nos. 1, 2 and 3.

Lot No. 4 - Sack No. 4.

Lot B - 5 per cent of Sack No. 4 and 95 per cent of Lot A.

Lot C - Composite sample of Sacks Nos. 5 and 6.

Lot No. 7 - Sack No. 7.

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Details of Tests:

LOT A (COMPOSITE OF SACKS NOS. 1, 2 AND 3).

Test No. 1. - Flotation and Table Concentration.

In this test a portion of the composite sample was ground to pass through a 28-mesh screen and the pulp transferred to a flotation machine. A sulphide concentrate was then obtained. This concentrate was cleaned in a smaller flotation machine. The flotation tailings from the rougher flotation concentrate were then screened into three parts, plus 48 mesh, plus 100 mesh and minus 100 mesh, and concentrated separately on a Wilfley table. The tailings from these operations were then combined and floated.

Results:

Product	Weight, per cent	Assays, per cent WO_3	Assays, per cent S	Assays, per cent P	Distribution: of WO_3, per cent	Ratio of concentration
Sulphide Flotation.						
Sulphide cleaner conc.	1.34	0.49	38.28	-	0.20	75:1.
" " tailing	2.74	1.25	2.77	-	1.00	
" rougher conc.	4.08	-				
Table Concentration.						
Table conc., +48 mesh	0.48	58.07	1.45	0.06	8.2	208:1.
" " , +100 "	0.98	61.49	1.97	0.05	17.6	102:1.
" " , -100 "	0.77	62.44	3.20	0.05	14.1	130:1.
Table middling, +48	2.36	8.22	2.51	0.13	5.7	
" " , +100	1.82	8.24	4.27	0.21	4.4	
" " , -100	2.70	14.18	4.14	0.25	11.2	
Flotation of Combined Table Tailing.						
WO_3 conc. No. 1	1.71	23.93	2.08	1.81	12.0	58.5:1.
" " No. 2	2.38	9.43	1.97	0.46	6.6	42:1.
Final tailing	82.72	0.79	0.45	0.09	19.0	

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(Test No. 1, cont'd) -

Sulphide Float			WO3 Float		
Grind	= -28 mesh.		Grind	= -100 mesh.	
Amyl xanthate	= 0.30	lb./ton.	Cresylic acid	= 0.05	lb./ton.
Sodium silicate	= 1.00	"	Emulsol X-1	= 0.10	"
Cresylic acid	= 0.25	"	Orso	= 0.30	"
pH	= 5.4		Orso	= 0.10	"

Summary of Test No. 1:

		WO3 assays, per cent	Distribution of WO3, per cent
Combined table concentrate	=	61.1	39.9
" " middling	=	10.6	21.3
Combined flotation conc.	=	15.5	18.6
Tailing	=	0.8	19.0

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Test No. 2 (A, B, and C). - Flotation Concentration.

In this test, portions of Lot A were ground in a ball mill to different grinds as stated. As the pulp was acid, it was then well washed, filtered, and transferred to a flotation machine. Rougher flotation scheelite concentrates were then obtained. These concentrates were cleaned in a smaller machine.

Results:

Test No. 2-A.

Product	Weight, per cent	Assays, per cent WO3	P	S	Distribution of WO3, per cent	Ratio of concentration
Feed	100.00	3.06	-		100.0	
Flot. conc.	3.59	58.70	1.02	1.04	68.8	27.8:1.
" middling	3.82	7.14	-	2.46	8.9	
" tailing	92.59	0.74	-	1.03	22.3	

The flot. conc. assayed 0.17 per cent Mo.

• Calculated.

(Continued on next page)

(Test No. 2, cont'd) -

Test No. 2-B.

Product	Weight, Per cent	Assays, per cent WO3	P	S	Distribution of WO3, per cent	Ratio of concentration
Feed	100.00	3.02*	-	-	100.0	
Flot. conc.	3.37	62.91	-	3.69	70.3	29.6:1.
" middling	3.11	4.18	-	2.16	4.3	
" tailing	93.52	0.82	-	-	25.4	

Test No. 2-C.

Feed	100.00	2.94*	-		100.0	
Flot. conc.	3.64	60.00	1.05	3.22	74.2	27.5:1.
" middling	5.45	2.90	-	1.64	5.4	
" tailing	90.91	0.66	-	3.22	20.4	

The flot. conc. assayed 0.60 per cent Mo.

* Calculated.

Reagents Added to Cell (lb./ton ore).

Test No.	Grind, per cent -200 mesh	Soda ash	Sodium silicate	Sodium cyanide	Pine oil	Emulsol: X-1	Orso	pH
2-A	87.6	12.0	1.0	0.25	0.05	0.10	0.50	7.2
2-B	62.0	25.0	2.0	0.35	0.07	0.15	1.10	8.1
2-C	62.0	30.0	1.0	0.20	0.10	0.20	1.20	8.5

A microscopic examination of the tailing showed free grains of scheelite over 100 microns in diameter.

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Test No. 3. - Table Plus Flotation Concentration.

A portion of the composite sample at minus 28 mesh grind was concentrated on a Wilfley table. The table tailings were filtered, sampled, repulped, and transferred to a flotation machine. The resultant rougher flotation concentrate was cleaned on a smaller machine.

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B

(Test No. 3, cont'd) -

Results:

Table Concentration

Product	Weight, per cent	WO_3 assay, per cent	Distribution of WO_3, per cent	Ratio of concentration
Feed	100.00	3.15*	100.0	
Table conc.	7.57	25.40	60.9	13.2:1.
" middling	5.42	17.98	30.8	
" tailing	87.01	0.30	8.3	

The table conc. assayed 0.17 per cent P.

Flotation Concentration of Table Tailing.

Feed	100.00	0.39*	100.0	
Flot. conc.	0.69	32.80	57.8	145:1.
" middling	2.49	2.37	15.0	
" tailing	96.82	0.11	27.2	

The flot. concentrate assayed 2.00 per cent P.

* Calculated.

Reagents to Flotation (lb./ton ore).

Soda ash	Sodium silicate	Sodium cyanide	Pine oil	Emulsol X-1	Orso	pH
20.00	1.00	0.20	0.10	0.20	1.0	8.6

A screen test on the flotation tailing gave 27.6 per cent minus 200 mesh. A microscopic examination of the flotation tailing showed no large particles of scheelite.

Summary of Test No. 3:

	Per cent
WO_3 recovered by table concentration -	91.7
" " " flotation " -	5.4

In this calculation all the table middling and 50 per cent of the flotation middlings were included in the recoveries.

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(Details of Tests, cont'd) -

Test No. 4. - Flotation and Table Concentration.

A comparatively large portion of the composite sample was crushed to pass minus 28 mesh and the scheelite concentrated by flotation as in the previous tests. This concentrate was then passed over a Wilfley table, with the following results:

Table Concentration of Flotation Concentrate.

Product	Weight, per cent	WO 3 assay per cent	Distribution of WO3, per cent
Feed	100.00	62.11*	100.0
Table conc.	18.93	72.75	22.2
" tailing	81.07	59.63	77.8

* Calculated.

LOT NO. 4.

Test No. 5 (A and B). - Flotation Concentration.

Portions of Lot No. 4 were ground in a ball mill to different degrees of fineness. The acid pulp was then well washed, filtered, and transferred to a flotation machine. A rougher flotation concentrate was obtained. This concentrate was cleaned in a smaller machine.

Results:

Test No. 5-A.

Product	Weight, per cent	Assays, per cent WO3	S	P	Distribution of WO3, per cent	Ratio of concentration
Feed	100.00	4.05*			100.0	
Flot. conc.	6.26	45.34	10.57	-	70.3	16:1.
" middling	7.54	7.40	-	-	13.8	
" tailing	86.20	0.75	-	-	15.9	

The flot. concentrate assayed 0.10 per cent Mo.

* Calculated.

(Continued on next page)

(Test No. 5, cont'd) -

Test No. 5-B.

Product	Weight, per cent	Assays, per cent WO_3	S	P	Distribution of WO_3, per cent	Ratio of concentration
Feed	100.00	4.98		-	100.0	
Flot. conc.	9.00	49.44	8.58	1.42	89.3	11:1.
" middling	2.75	7.61	21.75	-	4.2	
" tailing	8.25	0.37	-	-	6.5	

The flot. concentrate assayed 0.17 per cent Mo.

* Calculated.

Reagents to Flotation (lb./ton ore).

Test No.	Grind, per cent -200 mesh	Soda ash	Sodium silicate	Sodium cyanide	Pine oil	Emulsol X-1	Orso	pH
5-A	63.4	21.0	1.0	0.5	0.05	0.20	0.6	8.9
5-B	75.4	25.0	2.0	0.5	0.10	0.20	1.2	9.3

In Test No. 5-B, 0.2 pound of NaCN per ton was added to cleaner cell.

LOT B (5 PER CENT OF SACK NO. 4 AND 95 PER CENT OF COMPOSITE SAMPLE A).

Test No. 6 (A and B). - Flotation Concentration.

Portions of the ore of Lot B were ground in a ball mill, the acid pulp was washed and filtered, and a rougher concentrate was obtained by flotation. This concentrate was cleaned in a smaller flotation machine.

Results:

Test No. 6-A.

Product	Weight, per cent	Assays, per cent WO_3	S	Distribution of WO_3, per cent	Ratio of concentration
Feed	100.00	3.32*	-	100.0	
Flot. conc.	3.58	64.10	3.16	69.2	28:1.
" middling	3.15	2.82	3.36	2.7	
" tailing	93.27	1.00		28.1	

The flot. concentrate assayed 1.49 per cent P and 0.13 per cent Mo.

* Calculated.

(Continued on next page)

(Test No. 6, cont'd) -

Test No. 6-B.

Product	Weight, per cent	Assays, per cent WO_3	Assays, per cent S	Distribution of WO_3, per cent	Ratio of concentration
Feed	100.00	3.16*	-	100.0	
Flot. conc.	4.63	55.95	6.00	81.8	21.5:1.
" middling	3.78	2.11	3.10	2.5	
" tailing	91.59	0.54		15.7	

* Calculated.

Reagents to Flotation Cell (lb./ton ore).

Test No.	Grind, per cent -200 mesh	Soda ash	Sodium silicate	Sodium cyanide	Pine oil	Emulsol X-1	Orso	pH
6-A	62.0	24.0	1.0	0.30	0.05	0.15	0.65	8.9
6-B	62.0	27.0	2.0	0.40	0.15	0.20	1.30	9.1

LOT C (COMPOSITE OF SACKS NOS. 5 AND 6).

Test No. 7 (A, B and C). - Flotation Concentration.

Portions of the ore of Lot C were ground in a ball mill to pass 77.2 per cent minus 200 mesh. The pulp was then transferred to a flotation machine and a rougher flotation concentrate obtained. This concentrate was cleaned in a smaller machine.

In Test No. 7-B, a preliminary sulphide concentrate was made.

Test No. 7-A.

Product	Weight, per cent	Assays WO_3, per cent	Distribution of WO_3, per cent	Ratio of concentration
Feed	100.00	0.12*	100.0	
Flot. conc.	0.94	7.38	54.9	106:1.
" middling	6.06	0.94	45.1	
" tailing	93.00	None detected	-	

The flot. concentrate assayed 1.65 per cent P and 1.32 per cent Mo.

* Calculated.

(Continued on next page)

B

(Test No. 7, cont'd) -

Reagents to Cell (lb./ton ore).

Grind, per cent -200 mesh	Soda ash	Sodium silicate	Sodium cyanide	Emulsol X-1	Orso	pH
77.2	5.0	2.0	0.20	0.10	0.40	9.6

Test No. 7-B.

Product	Weight, per cent	Assays, per cent WO3	S	P	Mo	Distribution of WO3, per cent	Ratio of concentration
Feed	100.00	0.13◆	-	-	-	100.0	
Sulphide conc.	0.84	0.04	5.36	-	1.29		119:1.
Tungsten "	2.78	4.30	0.80	1.33	0.06	90.1	36:1.
" middling	3.37	0.38	0.45	0.17	Trace	9.6	
Final tailing	93.01	-	0.08	-	-		

Test No. 7-C.

Feed	100.00	-	-	-		-	
Sulphide conc.	1.22	0.27	-	-	1.07	-	82:1.
Tungsten "	4.95	0.98	-	0.49	0.05	-	20:1.
Final tailing	93.83	-	-	-	-	-	

◆ Calculated.

Reagents to Grind (lb./ton ore).

Test No.	Amyl xanthate	Sodium silicate	Soda ash	Coal oil
7-B	0.02	1.00	-	-
7-C	0.02	1.00	2.0	0.10

Reagents to Cell (lb./ton ore):

Sulphide Flotations.

Test No. 7-B - Cresylic acid 0.09
Test No. 7-C - Pine oil 0.14

Tungsten Flotation.

Test No.	Soda ash	Emulsol X-1	Orso
7-B	3.0	0.10	0.3
7-C	-	0.05	0.1

F

(Details of Tests, cont'd) -

LOT NO. 7 (SACK NO. 7, FROM SOUTH ORE BODY).

Test No. 8 (A and B). - Flotation Concentration.

Straight flotation of Lot No. 7. In Test No. 8-A, no quebracho was used. In Test No. 8-B, quebracho was added to the cell. The ore was ground in a ball mill, the pulp floated, and the rougher concentrate cleaned as in the previous flotation tests.

Test No. 8-A.

Product	Weight, per cent	Assays, per cent WO_3	Assays, per cent S	Distribution of WO_3, per cent	Ratio of concentration
Feed	100.00	0.86*	-	100.0	
Flot. conc.	27.88	2.88	-	93.0	3.6:1.
" middling	9.10	0.25	-	2.6	
" tailing	63.02	0.06	-	4.4	

Test No. 8-B.

Feed	100.00	0.66	-	100.0	
Flot. conc.	1.89	25.70	0.53	73.9	53:1.
" middling	5.89	2.62	3.55	23.4	
" tailing	92.22	0.02	-	2.7	

The flot. concentrate assayed 1.46 per cent P and 0.14 per cent Mo.

* Calculated.

Reagents to Cell (lb./ton ore).

Test No.	Grind, per cent -200 mesh	Soda ash	Sodium silicate	Sodium cyanide	Emulsol: X-1	Orso	Quebracho	pH
8-A	77.4	10.0	2.0	0.50	0.20	1.1	-	9.0
8-B	77.4	11.0	2.0	0.50	0.20	0.7	0.20	9.3

In Test No. 8-B an additional 0.20 pound NaCN and 0.05 pound quebracho per ton were used in the cleaner cell.

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(Lot No. 7, cont'd) -

Test No. 9 (A and B). - Flotation Concentration.

In these tests, a sulphide flotation was made prior to the flotation of the scheelite. Conditions otherwise were as in the previous test.

Test No. 9-A.

Product	Weight, per cent	Assays, per cent WO_3	Assays, per cent S	Distribution, per cent WO_3	Distribution, per cent S	Ratio of concentration
Feed	100.00	0.76*	8.95*	100.0	100.0	
Sulphide conc.	16.90	0.09	14.65	2.0	27.6	5.9:1.
Scheelite "	1.16	21.96	0.77	33.3	0.1	86:1.
" middling	6.98	3.81	1.63	35.1	1.3	
Final tailing	74.96	0.30	8.48	29.6	71.0	

Test No. 9-B.

Feed	100.00	0.72*	12.73*	100.0	100.0	
Sulphide conc.	8.34	0.02	33.57	0.2	22.0	12:1.
Scheelite "	1.55	36.60	0.65	78.9	0.1	64.5:1.
" middling	5.91	2.39	4.92	19.7	2.3	
Final tailing	84.20	0.01	11.44	1.2	75.6	

The sulphide conc. assayed 0.02 per cent P.

* Calculated.

Reagents to Cell (lb./ton ore):

Sulphide Flotation.

Test No.	Grind, per cent -200 mesh	Soda ash	Sodium silicate	Amyl xanthate	Pine oil
9-A	79.0	2.0	2.0	0.15	0.10
9-B	58.6	4.0	2.0	0.10	0.07

Scheelite Flotation.

Test No.	Soda ash	Sodium silicate	Sodium cyanide	Emulsol: X-1	Orso	Quebracho	pH
9-A	3.0	-	0.20	0.05	0.40	0.35	9.1
9-B	3.0	-	0.20	0.10	0.80	0.20	9.6

In Test No. 9-A, 0.10 pound quebracho per ton was added to cleaner cell.

-

F

(Lot No. 7, cont'd) -

Test No. 10. - Table and Flotation Concentration.

The ore was ground to minus 35 mesh and concentrated on a Wilfley table. The table tailings were filtered, sampled, and the remaining scheelite concentrated by flotation.

Table Concentration of Minus 35 Mesh Ore.

Product	Weight, per cent	Assays, per cent WO_3	Assays, per cent S	Distribution of WO_3, per cent	Ratio of concentration
Feed	100.00	1.02*	-	100.0	
Table conc.	5.54	6.88	35.10	37.3	18:1.
" middling	9.36	2.30		21.1	
" tailing	85.10	0.50		41.6	

Flotation Concentration of Table Tailing.

Feed	100.00	0.28*	-	100.0	
Flot. conc.	2.76	5.92	0.40	59.2	36:1.
" middling	6.73	1.00	2.52	24.4	
" tailing	90.51	0.05	--	16.4	

* Calculated.

Reagents to Flotation (lb./ton ore).

Grind, per cent -200 mesh	Soda ash	Sodium silicate	Sodium cyanide	Emulsol X-1	Orso	pH
36.6	10.0	2.0	0.40	0.10	0.50	9.3

0.10 pound NaCN per ton was used in the cleaning of the flotation concentrate.

-

Test No. 11 (A, B, and C). - Magnetic, Table and Flotation Concentration.

These tests combined magnetic concentration of the pyrrhotite in the ore with table and flotation concentration. In all three tests the ore was ground to pass minus 35 mesh.

Test No. 11-A.

The ore at minus 35 mesh was concentrated on a Ball-Norton magnetic machine and a pyrrhotite concentrate

(Test No. 11-A, cont'd) -

obtained. The non-magnetic portion was concentrated on a Wilfley table and the remaining scheelite in the table tailings was concentrated by flotation.

Magnetic Concentration.

Product	Weight, per cent	Assays, per cent WO_3	Assays, per cent S	Distribution, per cent WO_3	Distribution, per cent S	Ratio of concentration
Feed	100.00	0.92*	10.89*	100.0	100.0	
Magnetic conc.	22.12	0.09	30.37	2.2	61.7	4.5:1.
" tailing	77.88	1.16	5.36	97.8	38.3	

Table Concentration of Non-Magnetic.

Feed	100.00	1.16*	5.36*	100.0	100.0	
Table conc.	4.47	12.87	29.41	49.7	24.5	22.3:1.
" middling	7.81	3.30	15.68	22.3	22.8	
" tailing	87.72	0.37	3.23	28.0	52.7	

Flotation of Table Tailing.

Feed	100.00	0.25*	3.51*	100.0	100.0	
Flot. conc.	0.88	11.49	2.66	40.9	0.7	114:1.
" middling	3.17	1.57	3.16	20.3	2.8	
" tailing	95.95	0.10	3.56	38.8	96.5	

* Calculated.

Reagents to Cell (lb./ton ore).

Grind, per cent -200 mesh	Soda ash	Sodium silicate	Sodium cyanide	Emulsol: X-1	Orso	Quebracho	pH
46.4	4.0	2.0	0.20	0.10	1.0	0.30	9.6

Summary of Test No. 11-A:

	Per cent
WO_3 in magnetic conc. -	2.2
WO_3 in table conc. plus middling -	70.4
WO_3 in flotation conc. plus middling -	16.8
WO_3 in flotation tailing -	10.6
	100.0

(Lot No. 7, cont'd) -

Test No. 11-B.

In this test, a portion of the ore at minus 35 mesh was passed over a Wilfley table and a rough table concentrate obtained. This concentrate was roasted and the calcine concentrated on a Ball-Norton magnetic machine. The non-magnetic portion was re-tabled and the primary table tailings concentrated by flotation.

First Table Concentration.

Product	Weight, per cent	Assays, per cent WO_3	Assays, per cent S	Distribution of WO_3, per cent	Ratio of concentration
Feed	100.00	0.98		100.0	
Table conc.	31.50	2.48*		79.7	3.2:1.
Table tailing	68.50	0.29	5.11	20.3	

The table concentrate was roasted at a temperature of 800° C. The calcine showed a 4.0 per cent loss in weight. This calcine was then concentrated on a Ball-Norton magnetic machine with a result of 3.3 per cent of the weight reporting in the magnetic concentrate and assaying 0.27 per cent WO_3. The non-magnetic portion was then re-tabled, as follows:

Second Table Concentration.

Feed	100.00	2.23*		100.0	
Table conc.	11.00	11.60	0.62	57.1	9:1.
" middling	22.00	1.92	1.55	18.9	
" tailing	67.00	0.80		24.0	

The table conc. assayed 0.13 per cent P.

* Calculated.

The primary table tailing was concentrated in a flotation machine, as follows:

Feed	100.00	0.29*		100.0	
Flot. conc.	0.28	15.32	2.86	14.8	257:1.
" middling	2.12	4.45	3.96	32.5	
" tailing	97.60	0.17		52.7	

The flotation conc. assayed 0.65 per cent P.

* Calculated.

(Continued on next page)

(Test No. 11-B, cont'd) -

Reagents Added to Cell (lb./ton ore).							
Grind, per cent -200 mesh:	Soda ash:	Sodium silicate:	Sodium cyanide:	Emulsol X-1	Orso:	Quebracho	pH
51.0	4.0	2.0	0.30	0.05	0.60	0.30	9.3

0.10 pound quebracho and 0.10 NaCN per ton were added to the cleaner cell.

Summary of Test No. 11-B:

	Per cent
WO_3 in magnetic conc. -	0.4
WO_3 in table conc. plus middling -	60.3
WO_3 in table tailing -	19.0
WO_3 in flotation conc. plus middling -	9.6
WO_3 in flotation tailing -	10.7
	100.0
WO_3 in rougher table concentrates -	79.3
WO_3 in rougher table tailings -	20.7
	100.0

-

Test No. 11-C.

A portion of the ore ground to minus 35 mesh was passed over a Wilfley table and a rough table concentrate obtained. This table concentrate was concentrated on the Ball-Norton magnetic machine and the magnetic portion discarded. The non-magnetics were re-tabled and the resultant table concentrate roasted. The primary flotation tailing was concentrated by flotation.

(Continued on next page)

(Test No. 11-C, cont'd) -

Rougher Table Concentration.

Product	Weight, per cent	Assays, per cent WO_3	Assays, per cent S	Distribution of WO_3, per cent	Ratio of concentration
Feed	100.00	0.98		100.00	
Table conc.	25.85	3.22◆		84.9	4:1.
" tailing	74.15	0.20	4.83	15.1	

Magnetic Concentration of Table Concentrate.

Feed	100.00	2.16◆		100.0	
Magnetic conc.	70.70	0.05	37.34	1.6	1.4:1.
Non-magnetic tailing	29.30	7.25		98.4	

Table Concentration of Non-Magnetics.

Feed	100.00	7.25◆		100.0	
Table conc.	25.21	22.74	28.65	79.0	
" middling	24.86	4.10	24.26	14.1	4:1.
" tailing	49.93	1.00	17.49	6.9	

The above table concentrate was roasted; the calcine produced assayed 28.10 per cent WO_3, 0.84 per cent S, 6.77 per cent CaO, and 0.11 per cent P.

Flotation of Primary Table Tailing.

Feed	100.00	0.19◆		100.0	
Flot. conc.	0.55	6.60	3.29	19.4	182:1.
" middling	3.75	2.23	4.04	44.7	
" tailing	95.70	0.07	4.85	35.9	

◆ Calculated.

Reagents Added to Cell (lb./ton ore).

Grind, per cent -200 mesh	Soda ash	Sodium silicate	Sodium cyanide	Emulsol X-1	Orso	Quebracho	pH
39.4	3.0	2.0	0.30	0.10	0.80	0.20	8.7

0.10 pound quebracho and 0.10 NaCN per ton were added to cleaner cell.

(Continued on next page)

(Test No. 11-C, cont'd) -

Summary of Test No. 11-C:

	Per cent
WO_3 in magnetic conc. =	1.4
WO_3 in table conc. plus middling =	77.7
WO_3 in table tailing =	5.8
WO_3 in flotation conc. plus middling -	9.7
WO_3 in flotation tailing -	5.4
	100.0
WO_3 in rougher table concentrate -	84.9
WO_3 in rougher table tailing -	15.1
	100.0

Test No. 12 (A and B). - Magnetic, Table, Superpanner and Flotation Concentration.

In this test, an attempt was made to determine the amounts of high-grade and lower-grade concentrates that could be shipped direct to a steel plant or to a chemical treatment plant following the ore-dressing procedure. A portion of the ore was ground through a set of rolls to pass 100 per cent minus 35 mesh. The pulp was then transferred to a Denver flotation cell and a pyrite concentrate obtained. The flotation tailings were then passed over a Wilfley table and a rougher table concentrate secured. This concentrate, which still contained some pyrrhotite, was then concentrated on a Ball-Norton magnetic machine and the magnetic portion discarded. The non-magnetic portion was then reconcentrated on a Wilfley table, in Test No. 12-A, and on a Haultain superpanner, in Test No. 12-B. The tailings from these operations were then combined with the original table tailings, reground, and floated. This flotation concentrate constituted the material to be shipped to the chemical treatment plant, while the table and superpanner concentrate would be the high-grade portion for

(Test No. 12 (A and B), cont'd) -

shipment to a steel plant. In order to remove the excess phosphorus, both of these final concentrates were subjected to a leaching operation with muriatic acid. The solution used was made as follows: 318 grams of 20° Bé muriatic acid was mixed with 682 grams of water. Two parts of solution were added to one part of concentrate and the mixture agitated in the cold for 45 minutes. It was then filtered, washed, and assayed for phosphorus.

Test No. 12-A:

Pyrite Flotation.

Product	Weight, per cent	WO_3 assay, per cent	Distribution of WO_3, per cent	Ratio of concentration
Feed	100.00	0.98	100.0	
Pyrite conc.	16.76	N.D.	-	6:1.
Flot. tailing	83.24	1.18	100.0	

Table Concentration of Flotation Tailing.

Feed	100.00	1.18	100.0	
Table conc.	11.95	8.23	83.4	8/4:1.
" tailing	89.05	0.22	26.6	

Magnetic Concentration of Table Concentrate.

Feed	100.00	8.23	100.0	
Mag. conc.	77.08	N.D.	-	1.3:1.
Non-mag. tailing	22.92	35.91	100.0	

Table Concentration of Non-Magnetics.

Feed	100.00	35.91	100.0	
Table conc.	28.19	45.72	35.9	3.5:1.
" tailing	71.81	32.06	64.1	

N.D. - None detected.

This table concentrate assayed, trace of phosphorus. In practice, the table tailing, which is a middling product containing 37.5 per cent of the values in the original feed, would be recirculated with an estimated additional recovery of 14 per cent, or a total recovery of 53 per cent.

(Continued on next page)

(Test No. 12-A, cont'd) -

Flotation of Combined and Reground Table Tailing.

Product	Weight, per cent	WO_3 assay, per cent	Distribution of WO_3, per cent	Ratio of concentration
Feed	100.00	0.31	100.0	
Pyrite conc.	3.62	N.D.	-	27.6:1.
Scheelite conc.*	0.29	31.64	20.2	345:1.
" middling	7.35	3.00	48.5	
Flot. tailing	88.73	0.16	31.3	

* This scheelite flotation concentrate assayed 0.03 per cent phosphorus.

Reagents added (lb./ton feed):

To the Grind -

$NaCO_3$	-	3.0

To the Pyrite Concentrate -

Amyl xanthate	-	0.10
Pine oil	-	0.15

To the WO_3 Concentrate -

$NaCO_3$	-	2.0
Quebracho	-	0.20
Emulsol X-1	-	0.10
Orso	-	0.80

Final pH, 9.3.
Time of flotation, 8 minutes.

Grind -

Mesh		Weight, per cent
+ 65	-	1.0
- 65+100	-	4.9
-100+150	-	12.7
-150+200	-	16.7
-200	-	64.7

Summary of Test No. 12-A:

	Per cent
WO_3 recovered as high-grade table conc. -	53.0
WO_3 recovered as low-grade flotation concentrate and flotation middling -	32.3
WO_3 remaining in final flotation tailing -	14.7
	100.0

(Lot No. 7, cont'd) -

Test No. 12-B.

In this test the Haultain superpanner replaced the Wilfley table in the concentration of the non-magnetics. Conditions otherwise were similar to those of Test No. 12-A. The initial pyrite concentrate and the magnetic concentrate were assayed for WO_3 and reported as "none detected" in each case.

The panner concentration of the non-magnetics was as follows:

Product	Weight, per cent	WO_3 assay, per cent	Distribution of WO_3, per cent	Ratio of concentration
Feed	100.00	48.56	100.0	
Panner conc.	22.00	67.91	30.8	4.5:1.
" tailing	78.00	43.10	69.2	

This panner concentrate assayed, traces of phosphorus.

The middling disposition will be similar to that in Test No. 12-A.

Flotation of Combined and Reground Table Tailing and Panner Tailing.

Product	Weight, per cent	WO_3 assay, per cent	Distribution of WO_3, per cent	Ratio of concentration
Feed	100.00	0.48	100.0	
Flot. conc.	2.06	19.10	80.8	48.5:1.
" middling	9.60	0.10	1.9	
" tailing	88.34	0.09	17.3	

This flotation concentrate assayed, trace of phosphorus.

Reagents added (lb./ton feed):

To the Grind -

Soda ash	- 3.0

To the Concentrate -

Soda ash	- 2.0
NaCN	- 0.30
Quebracho	- 0.20
Emulsol X-1	- 0.10
Orso	- 0.80

(Continued on next page)

(Test No. 12-B, cont'd) -

Summary of Test No. 12-B:

	Per cent
WO_3 recovered as high-grade panner conc. -	49.6
WO_3 recovered as low-grade flotation concentrate and flotation middling -	41.7
WO_3 remaining in flotation tailing -	8.7
	100.0

SUMMARY:

On the shipments from the North Ore Body, which consist of badly oxidized material, it was not found possible by straight flotation of the ore to produce a flotation tailing that could be discarded. By table concentration, followed by flotation of the table tailings, a final flotation tailing of 0.11 per cent WO_3 was obtained, with an overall recovery of 97 per cent of the WO_3 in the ore (Test No. 3). The grade of concentrates produced was as high as 62.9 per cent WO_3 in flotation concentration (Test No. 2-B) and 62 per cent in the screened table concentration (Test No. 1). When the flotation concentrates were passed over a Wilfley table, a grade of 72.7 per cent WO_3 was obtained (Test No. 4). These different table and flotation concentrates were generally high in sulphur and phosphorus. If, as in Test No. 1, a sulphide flotation is made prior to the scheelite concentration, the bulk of the molybdenite in the ore should report in this sulphide concentrate. In all the flotation tests on these North Ore Body shipments it was necessary to wash the pulp well prior to flotation, because of the acid condition of the badly oxidized ore.

On the composite Skarn Ore shipments, assaying

(Summary, cont'd) -

0.12 per cent WO_3, a cleaned flotation concentrate was obtained assaying 7.38 per cent WO_3 at a ratio of concentration of 106:1. This concentrate contained 54.9 per cent of the WO_3 in the ore with an additional 45.1 per cent in the middling product.

The shipment representing the South Ore Body, from which the main ore supply will be obtained, assayed 0.98 per cent WO_3. On this shipment, cleaned flotation concentrates were obtained assaying 25.7 per cent and 36.6 per cent WO_3 with sulphur contents of 0.53 and 0.65 per cent. The recovery of WO_3 was over 90 per cent in each case (Tests Nos. 8-B and 9-B). In these tests the use of quebrache reagent proved effective in depressing the calcite. By table concentration prior to flotation of the table tailings, a recovery of over 90 per cent of the WO_3 was obtained as a low-grade concentrate (Test No. 10). In Test No. 11-C it was shown that 70 per cent of the rougher table concentrate can be discarded, with a loss of only 1.6 per cent of the scheelite, by magnetic separation of the pyrrhotite in the concentrate. However, the concentrate obtained by retabling the non-magnetics still contained much sulphur in the form of pyrite. Similarly, in Test No. 11-A it was shown that magnetic separation will permit 22 per cent of the ore to be discarded, prior to concentration, with a loss of 2.2 per cent of the WO_3 in the ore.

As regards recovering the small amount of molybdenite that is present in the ore, it was not possible to determine definitely in the small-scale test work whether this could be accomplished in the form of a molybdenite concentrate.

The grade of concentrates produced from these different lots of ore was generally lower than the 65 to 70 per

B

(Summary, cont'd) -

cent WO_3 required for shipment to a steel plant. These results, however, were considered inherent to the small-scale test work and it was shown in Test No. 4 that a 72 per cent WO_3 grade can be made from the 62 per cent WO_3 feed used in that test.

An examination of the results obtained in Tests Nos. 12-A and 12-B disclosed the following:

Flotation of the pyrite followed by table concentration of the flotation tailing recovers 83.4 per cent of the WO_3 in the feed. The table tailing is high (0.22 per cent WO_3). Microscopic examination of this product showed that most of the losses were in particles of fine scheelite freed from gangue. As the test was made on an unclassified table feed, improved recoveries in practice can be anticipated when the flotation tailing has been sized in a hydraulic sizer prior to table concentration. Also, it was not possible with the laboratory-size table to obtain a clean concentrate, owing to the small quantity of scheelite in the product and also to the large amount of pyrrhotite remaining after flotation. With a full-sized table a much better cut could be obtained.

Magnetic separation removes this pyrrhotite, raising the grade from 8.23 per cent to 35.91 per cent with no recorded loss of WO_3.

Microscopic examination of the non-magnetics showed that the greater part of the scheelite was freed, the impurities consisting of gangue and some pyrite.

More efficient flotation at the initial stage of treatment probably would remove this pyrite.

A retabling of this product raised the grade to 45.72 per cent WO_3 in Test No. 12-A, and 67.91 per cent WO_3 in

(Summary, cont'd) -

Test 12-B where the superpanner was used. These results show that a grade of 65 per cent or better can be expected when full-sized equipment is used, and when a circulating load of free scheelite-gangue middling can be built up on the table.

The table tailing, 32.06 per cent WO_3 in Test No. 12-A and 43.10 per cent WO_3 in Test No. 12-B, consisted mainly of freed particles of scheelite and gangue. This again shows the inefficiency of the laboratory-sized equipment on a product of this nature.

In practice, with a primary flotation removing pyrite and considerable pyrrhotite, followed by wet magnetic separation to remove pyrrhotite prior to hydraulic sizing, followed by table concentration, a product of 65 per cent WO_3 or better should be obtained by table concentration.

These two tests (12-A and 12-B) indicate that 50 per cent recovery of the WO_3 can be expected with a grade of concentrate acceptable to Atlas Steels Limited.

Leaching of the concentrate to reduce the phosphorus content also is indicated.

The tests also indicate that if the table tailing is of a grade to warrant it, a further recovery in the form of a low-grade concentrate can be obtained by flotation. Under laboratory test conditions an additional 40 per cent is recorded.

..

The investigation shows two possible methods for the treatment of the ore from the South Ore Body. The first is outlined above (see, also, suggested Flow-Sheet No. 1 in the appendix to this report). The second method (see Flow-Sheet No. 2 in the appendix) would consist of a 65-mesh grind with flotation of the pyrite and most of the pyrrhotite

(Summary, cont'd) -

followed by a scheelite flotation. This flotation concentrate can then be followed by a further removal of pyrrhotite by a magnetic separator and a table concentration of the non-magnetics. This table concentrate then could be leached to remove phosphorus and shipped to Atlas Steels Limited. The lower-grade products would be treated chemically.

However, the results obtained from the present investigation are such that no conclusion as to the proper method of treatment can be arrived at from the small-scale laboratory tests. A pilot-plant run on from 20 to 40 tons is indicated, in order to obtain data on the performance of the ore under investigation.

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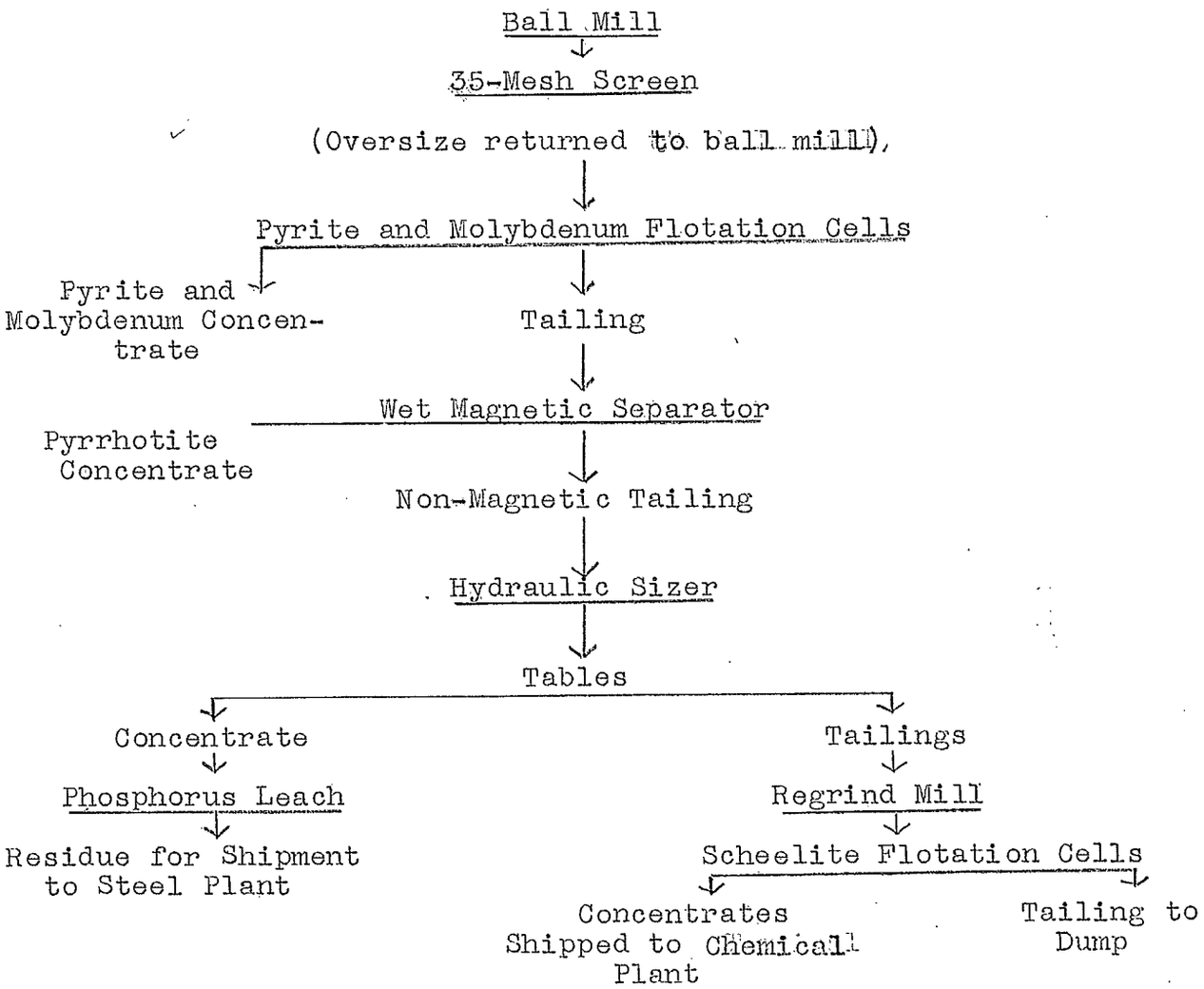
A P P E N D I X

FLOW-SHEETS OUTLINED IN THE SUMMARY OF REPORT OF INVESTIGATION NO. 1376.

FLOW-SHEET NO. 1.



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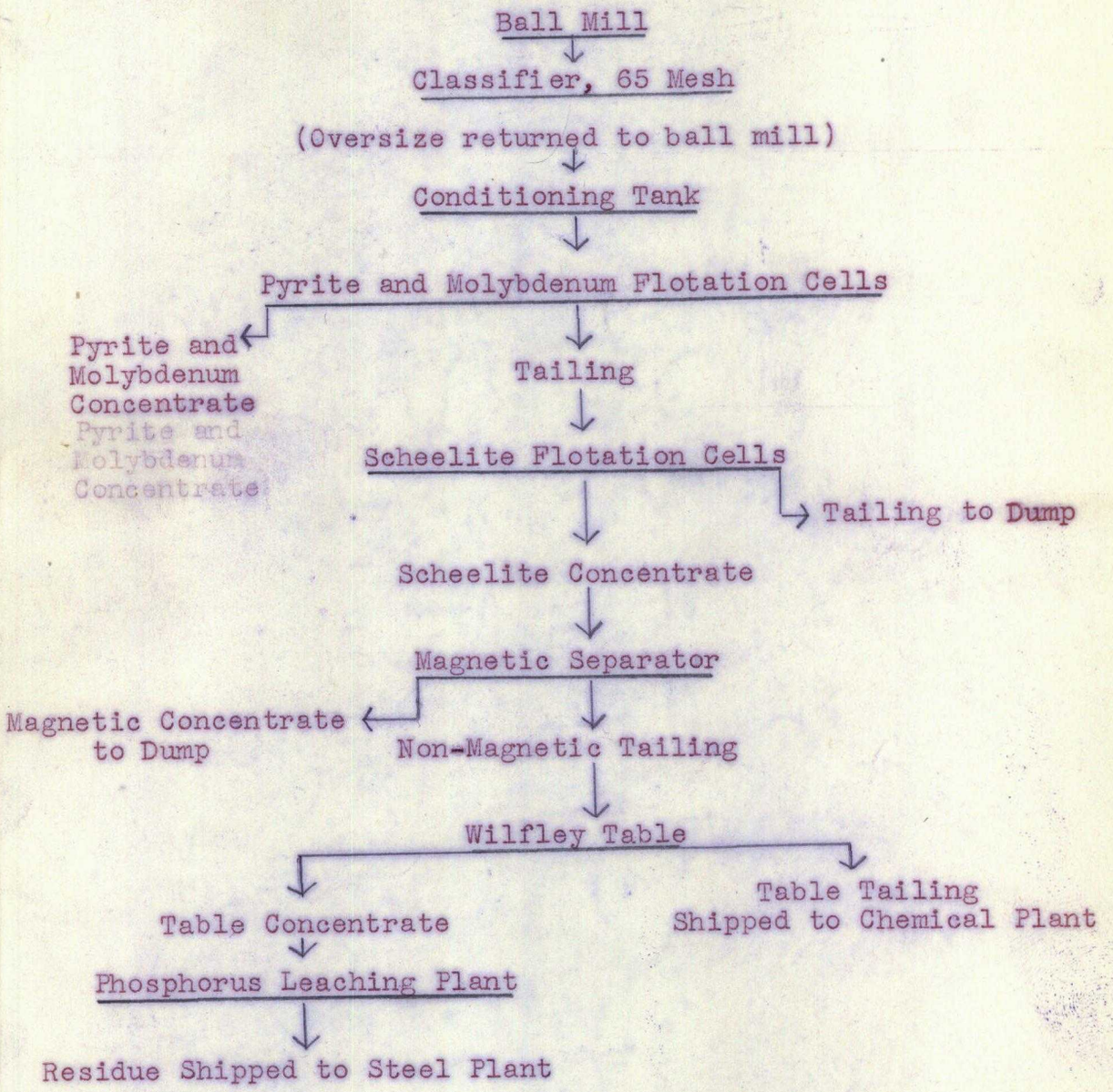

June 10, 1943.
HLB:GHB.

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