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TREATMENT OF HATCHES CREEK TUNGSTEN CONCENTRATE

(Amdel Report No.515)

by

E.E. Moskovits.

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* of the Australian Mineral Development Laboratories, Adelaide.

INTRODUCTION

Following a boom in production caused by the high tungsten prices induced by the Korean War, production at Hatches Creek declined. Though the mine and plant of the largest producer (Pioneer Mines N.L.) was maintained for a number of years afterwards, regular production virtually ceased in 1957.

The Pioneer mill produced a gravity tungsten concentrate containing copper bismuth and gold. A penalty was incurred for the presence of the first two elements.

Future development of the field may depend on operation of the small deposits of the area by a single mining company with a central mill and residential and administrative centre possibly based on the existing Pioneer Mine installations. Such an operation would probably be necessary to ensure a low level of costs. However, some deposits carry much higher proportions of copper than the Pioneer deposits themselves. The treatment plant required for a scheme such as that described above would be rather more elaborate than the present gravity mill.

The object of the research carried out at Amdel was to demonstrate that it was practical to produce separate copper, bismuth and wolfram concentrates. It was hoped that this would assist in the re-establishment of the field as a productive area by way of showing that the value of mine production could be based on a penalty free tungsten concentrate and some revenue from copper bismuth and gold.

The report has been included in the Bureau's Record series so that it may enjoy the wider distribution offered by the "Open File" system. It has not been altered in any way.

2/1/8 November, 1966

BUREAU OF MINERAL RESOURCES

Amdel Report

No. 515

TREATMENT OF HATCHES CREEK TUNGSTEN CONCENTRATE

by

E. E. Moskovits

Investigated by: Metallurgy Section
Officer in Charge: P. K. Hosking

P. A. Young. Director

THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES
Adelaide South Australia

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SUMMARY

History

Gravity concentration of Hatches Creek NT ore as previously practised gave a wolfram concentrate containing bismuth and copper minerals which carry penalties on sale.

Should the area be re-established for wolfram production returns will be increased if separate concentrates of wolfram, bismuth and copper can be marketed.

Objective

The objective of the investigation was to separate a mixed gravity concentrate into separate wolfram, bismuth and copper concentrates. The specification for bismuth into a wolfram concentrate was stated to be less than 1%.

Summary of Work Done

Anionic flotation using potassium amyl xanthate as collector effectively separated wolfram from the bismuth and copper minerals. A tungsten concentrate was produced representing 79.6% of the original weight and having an analysis of 66.8% WO_3, 0.83% Bi and 0.20% Cu. The recoveries were 90.0%, 12.5% and 12.7% respectively.

Efforts were made to produce separate bismuth and copper concentrates using a combination of flotation, magnetic and high tension techniques. Marked trends in separation were achieved and it is believed that a workable process could be developed. However, only two complete tests could be conducted within the financial allocations available and results were not consistent between tests apparently due to experimental difficulties.

As an alternative to physical beneficiation, leaching using ferric chloride was given a preliminary examination and this indicated that almost all bismuth can be extracted without difficulty and probably copper could be also.

Conclusions and Recommendations

The mixed gravity concentrate can be treated to give a high grade wolfram product by a simple xanthate flotation step. It is considered highly probable that further investigation would lead to the development of a workable process for separation of separate bismuth and copper concentrates.

It is recommended that further work be undertaken to:

a. establish conditions for physical concentration of bismuth and copper concentrates along the lines already pursued.

b. define conditions for recovery of bismuth and copper using ferric chloride leaching.

c. compare the economics of the respective processes.

1. INTRODUCTION

The aim of this investigation was to upgrade gravity tungsten concentrates from Hatches Creek, NT,[1] containing bismuth and copper minerals which carry penalties on sale. Another objective was to produce separate bismuth and copper concentrates.

The gravity concentrate contained gold, but an investigation to recover gold was not made, as sufficient quantity of sample for such a programme was not available.

2. MATERIAL EXAMINED

A 26 lb sample of a mixed concentrate was received from Melbourne University Ore Dressing Laboratory in August, 1964. The sample was stated to be a gravity concentrate produced at Hatches Creek. The material was thoroughly mixed, and portions riffled out for test work.

The sample contained on analysis:

		%	oz/long ton
Copper,	Cu	1.07	-
Bismuth,	Bi	4.99	-
Tungstic Oxide,	WO_3	64.5	-
Molybdenite,	MoS_2	0.23	-
Gold,	Au	-	0.920
Silver,	Ag	-	0.90

A mineralogical report of this material is given in Appendix A.

3. EQUIPMENT

The following equipment was used:

Stainless steel rod mill;	10.x 7 in. ID 1" dia. rods.
Denver flotation cell;	1 litre operating at 1500 rpm.

British Standard Screens (BSS).

Haultain Superpanner.

Stearns Disc-type Dry Magnetic Separator.

Carpco Laboratory Ocodel High Tension Separator.

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[1] JENSEN, E.B., (1966), "The Hatches Creek Tungsten and Bismuth Field," Aus. I.M.M., Proceedings No. 216, pages 17-30.

4. ANCILLARY MATERIALS

4.1 Collectors

The following reagents were used as collectors:

Reagent	Supplier	Approx. cost/lb(a)
Aero xanthate 325 (Sodium ethyl)	Cyanamid Australia Pty Limited, Melbourne.	0.274 c
Aero xanthate 350 (Potassium amyl)	As above.	0.34 c
Armac SD (Soya amine acetate)	Armour Chemical Division, Chicago, Illinois.	-
Duomeen S (an aliphatic diamine converted to acetate in the laboratory by adding the stoicheiometric amount of acetic acid).	As above.	0.60 c

(a) Adelaide prices.

4.2 Frother

Where additional frothing power was required Aerofroth 65 was added. This is a non-promoting water soluble polyglycol marketed by Cyanamid Australia Pty Limited, Melbourne.

5. EXPERIMENTAL PROCEDURE AND RESULTS

5.1 Screening

A screen analysis of a head sample, "as received", is shown in Table 1.

5.2 Grinding

The gravity concentrate was wet ground stagewise in the laboratory stainless steel rod mill to pass 52-mesh. Mill charges of 500-g of solids and 350 ml of water were used initially and the grinding time was 3 minutes for each stage.

5.3 Desliming

Desliming was done by decantation using three successive settling periods of 10 minutes on minus 52-mesh material. The initial 500-g charge was diluted to a volume of 10.5 litres in a bucket before decantation. The height of the liquid column was 23 cm. The volume was made up after each decantation. To prevent flocculation, sodium hydroxide at the rate of 0.5 lb per ton of feed was added to the first decantation stage. Separation was made at nominally 18 microns based on the specific gravity of quartz. Results are shown in Table 2.

5.4 Flotation

5.4.1 Anionic Flotation

The ore was conditioned for 1 minute with the reagents in the flotation cell, and then a flotation concentrate was removed over a 3-4 minute period. Additional flotation reagents were added to the flotation cell and a further concentrate removed over a 3-4 minute period. This procedure was repeated as necessary. Rougher and scavenger products were bulked and refloated. The cleaner and recleaner flotation stages lasted 3 minutes each.

In the first flotation stage, potassium amyl xanthate was used as collector. In the second flotation stage, sodium ethyl xanthate was used to float molybdenite and then flotation was continued using potassium amyl xanthate.

Aerofroth 65 was added as necessary to maintain adequate frother conditions.

In all flotation tests demineralised water was used and the temperature was maintained at 22^{0}C.

A Denver laboratory cell of 1 litre capacity was used and the pulp density was approximately 20% solids.

In primary (collective) flotation tests NaOH was used to adjust pH and in secondary (differential) flotation (reflotation of primary flotation concentrate) CaO was used. The pH was maintained between 9-11 in all tests.

The sample received was stagewise ground to pass 52 mesh. Primary flotation concentrates were washed with hot water (45^{0}C) before reflotation.

In Test 3 flotation was conducted on minus 52-mesh sample, in Tests 6 and 8 on deslimed minus 52-mesh material. In Test 8, 2.4 pound per ton of $CuSO_45H_2O$ was added and conditioning time was 10 minutes. Test results are shown in Table 3.

In Test 15 an anionic flotation concentrate produced from minus 52-mesh material was refloated after addition of CaO and KCN; pH was 11.7. Results are shown in Table 4.

5.4.2 Cationic Flotation

Two cationic flotation tests were conducted. In Test 7 Duomeen and in Test 12 Armac SD were used.

In both tests demineralised water was used and the temperature was kept at 22^{0}C.

In Test 7 deslimed minus 52-mesh material and in Test 12 undeslimed minus 52-mesh material were used as flotation feed. In Test 7 Duomeen S acetate was added as a 1% aqueous solution in a single addition. In Test 12 Armac SD was added as a 1% aqueous solution in two equal stage-additions. In Test 7, 3.2 pound per ton and in Test 12, 2.0 pound per ton of Na_2S were added to the cleaner and recleaner stages and the conditioning time was 10 minutes in each case.

Results are shown in Table 5.

5.5 Magnetic Separation

Magnetic separation of various flotation products was conducted using the Sterns disc-type separator. The magnets were operated at 0.8 amp. or 2800 gauss in Test 19 and 1.3 amp or 3600 gauss. In Tests 19, 23, 26, 27 and 29 leading and trailing Pole gaps were $\frac{1}{16}$-in.

Results are given in Section 5.7.

5.6 High Tension Separation

High tension separation of various flotation products were conducted using the Carpco Rotor type separator. Test conditions were as follows:

	Test 19	Test 23	Tests 26, 27 and 29
Voltage, KV	30	16	17
Splitter Position	60/30	58/34	58/34
Rotor Speed (a) Position	60	60	60

(a) Position 60 corresponds to 160 rpm.

Results are shown in Section 5.7

5.7 Combined Methods of Flotation, Magnetic and High Tension Separations

Experiments were conducted using flotation, magnetic and high tension separations. Flotation conditions were similar to those used in Anionic flotation (Section 5.4.1), magnetic separation and high tension separation were carried out under conditions described in 5.5 and 5.6.

In Test 29 a recleaner concentrate was produced by the use of potassium amyl xanthate (0.22 g/litre). The recleaner concentrate was magnetically separated. Results are shown in Table 6.

In Test 26 a recleaner concentrate was produced using potassium amyl xanthate (0.33 g/litre). The recleaner concentrate was submitted to magnetic separation. The non-magnetic fraction served as feed for high tension separation. Results are shown in Table 7.

In Test 23 a recleaner concentrate was produced by the use of potassium amyl xanthate (0.44 g/litre). Recleaner, cleaner and rougher tailings were bulked. The recleaner concentrate was separated magnetically and the non-magnetic fraction was used for high tension separation. The magnetic fraction (mainly wolframite) was added to the bulked tailing from Test 23. Results are shown in Table 8. In test 24 the non-magnetic fraction (mainly scheelite, bismuthinite and chalcopyrite) was refloated after conditioning for 10 minutes with CaO (1 g/litre) and KCN (0.34 g/litre). Results are shown in Table 9.

In Test 19 a recleaner concentrate was produced with 0.34 g per litre of potassium amyl xanthate. In Test 20 this product was magnetically separated. The non-magnetic fraction was used as feed for high tension separation. The non-conductor fraction was mainly scheelite and the conductor minerals were bismuthinite and chalcopyrite. The bismuthinite and chalcopyrite concentrate was floated in a highly alkaline (1 g/litre CaO) pulp. Flotation was preceded by conditioning for 10 minutes with 0.34 g/litre KCN.

Results are shown in Tables 10 and 11. A flowsheet is presented in Figure 1.

In Test 27 a recleaner concentrate was produced with potassium amyl xanthate (0.44 g/litre). Selective flotation of this product was conducted by using KCN (0.34 g/litre) and CaO (1 g/litre). Magnetic and high tension separations were conducted on concentrate 2 and second cleaner concentrate from selective flotation. Molybdenite was floated away from primary recleaner concentrate with sodium ethyl xanthate before flotation of bismuthinite with potassium amyl xanthate.

Results are shown in Table 12 and Flowsheet B in Figure 2.

5.8 Leaching with Ferric Chloride

Two leaching tests were carried out using hot ferric chloride.

Test 28 consisted of leaching thrown product from high tension separation of the relceaner concentrate of Test 26.

In Test 30 gravity concentrate as received (-52 mesh) was leached with hot ferric chloride.

In both cases 10 g were taken and added in a three neck round bottomed glass flask with heating mantel. 400 ml 20% $FeCl_3$ was added and the leaching time was 6 hours in each case. The temperature was kept at approximately 105°C. Results are tabulated below:

		Analysis			
		Wt g	Bi %	Cu %	WO_3 %
Test 28:	Feed	10.0	35.9	8.6	-
	Residue	6.0	1.05	9.35	-
	Extraction	-	98.2	35.0	-
Test 30:	Feed	10.0	(a)	(a)	-
	Residue	8.0	0.019	0.57	64.60
	Extraction	-	96.8	58.6	-

(a) Not assayed.

The leaching residue assayed 4.85 oz of gold per long ton.

5.9 Gravity Concentration

A rougher concentrate enriched in copper from flotation with potassium amyl xanthate, as collector was used as feed for gravity separation using the Haultain Superpanner. The feed was washed with hot water to remove reagent coating.

Results are shown in Table 13.

6. DISCUSSION

6.1 Screening

A screen analysis (Table 1) has shown that tungsten concentration is higher in the coarser particle size range. In contrast to this, copper grade is higher in the finer fractions, and is richest at the range of minus 25 plus 150-mesh.

6.2 Grinding

A mineragraphic examination (Appendix A) has shown that the distribution of wolframite, bismuthinite and chalcopyrite is similar in the minus 52 plus 150, minus 150 plus 300, and minus 300-mesh particle size ranges. Therefore the ore was ground to pass 52-mesh to reduce sliming.

6.3 Desliming

The results of desliming have not shown any marked differences in the relative friability of the minerals, except that wolframite is perhaps the most friable of the three economic minerals (wolframite, bismuthinite, chalcopyrite).

6.4 Flotation

Experiments have shown the superiority of anionic flotation over cationic flotation. Consequently anionic reagents were generally used throughout the work. It became possible to reduce the bismuth and copper content of the tungsten concentrate below the specified 1% for each metal. A typical test result (Test 26, Table 7) has shown that a tungsten concentrate (rougher tailing in anionic flotation) can easily be produced with potassium amyl xanthate, as collector. In 79.6% weight the tungsten concentrate gave on analysis 66.8% WO_3, 0.83% Bi and 0.20% Cu.

6.5 Magnetic Separation

It was possible with magnetic separation to remove from flotation concentrate fine wolframite at the expense of some bismuthinite and chalcopyrite (e.g. Test 20, Table).

6.6 High Tension Separation

High tension separation of products containing mainly scheelite, bismuthinite and chalcopyrite was not satisfactory, though marked trends of scheelite (non-conductor) concentration on one side and bismuthinite and chalcopyrite (conductors) concentration were achieved.

6.7 Combined Methods of Flotation, Magnetic and High Tension Separations

Tungstic acid content of flotation concentrate was reduced by magnetic separation of wolframite and by high tension separation of scheelite. Molybdenite was easily floated away from bismuth-copper concentrate with sodium ethyl xanthate. Selective flotation of bismuthinite from chalcopyrite was fairly effective. However, the bismuth separation would seem to be affected to a large degree by composite particles as described in Appendix A.

In Test 27 experiments were aimed at arranging all wet flotation stages together and then to follow up with dry-way cleaning stages. This work was not successful due apparently to experimental difficulties and these unfortunately were not recognised before the work had advanced too far to be repeated within the allocated funds. In earlier work flotation results were far superior and there is no apparent reason why such results should not be reproducible.

Because of the low efficiency in the later flotation work the final overall results do not prove that the dry-way processes can be placed after all the flotation steps.

6.8 Leaching with Ferric Chloride

Leaching with ferric chloride indicated that almost all bismuth and some copper can be extracted from the tungsten in the original sample as received and in flotation concentrate. However, it can be expected that on finer material copper extraction will be higher. The recovery of bismuth and copper from the solutions was not attempted, but this should be possible by hydrolysis and pH control and concentration.

6.9 Gravity Concentration

The results of the Haultain superpanner test showed a tendency for some wolframite and bismuthinite to concentrate in the heavy fraction giving a light fraction enriched in copper. However separations are not sufficiently selective for plant practice.

7. CONCLUSION AND RECOMMENDATIONS

It was possible to produce a high-grade tungsten concentrate from the mixed gravity concentrate by removing bismuth and copper minerals by the use of flotation. The bismuth-copper concentrate probably could be separated in bismuth and copper concentrates, but the development of a workable process needs further investigations.

It is recommended that further work should be undertaken with the following aims:

a. to establish optimum conditions for selective flotation of bismuthinite and chalcopyrite,

b. to define conditions for recovery of bismuth and copper using ferric chloride leaching,

c. to compare the economics of the respective processes.

8. ACKNOWLEDGEMENTS

The assistance given by Messrs P. N. Harris and G. P. Lewis in carrying out experiments is acknowledged.

APPENDIX A

MINERAGRAPHIC EXAMINATION OF WOLFRAMITE ORE, HATCHES CREEK

Representative portions of these samples were briquetted for examination. The proportions of the minerals present and the degree of liberation were estimated by point count.

TABLE 1: ESTIMATED PROPORTION OF MINERAL IN FRACTIONS
Volume: %

Constituents	Particle Size		
	-52 +150(a)	-150 +300(a)	-300(a)
Wolframite	31	39	22
Bismuthinite	15	10	16
Chalcopyrite	8	9	15
Pyrite	7	1	5
Bismuth	2	3	trace
Gangue	37	38	41
Total	100	100	100

(a) Mesh BSS.

<u>Minus 52 Plus 150-Mesh Sample</u>. Seventy-seven per cent of the bismuthinite in this sample is free. Twenty-three per cent occurs as composites with wolframite, chalcopyrite, bismuth and gangue. Seventy-eight per cent of the chalcopyrite is free whilst twenty-two per cent occurs as composites with covellite or gangue.

<u>Minus 150 Plus 300-Mesh Sample</u>. Sixty-one per cent of the bismuthinite in this sample is free. Thirty-nine per cent occurs as composites with bismuth, wolframite or chalcopyrite. Seventy-eight per cent of the chalcopyrite is free whilst twenty-two per cent occurs as composites with covellite, gangue or rarely bismuth.

<u>Minus 300-Mesh</u>. Seventy-five per cent of the bismuthinite in this sample is free. Twenty-five per cent occurs as composites with wolframite, bismuth, gangue or chalcopyrite. Seventy per cent of the chalcopyrite is free whilst thirty per cent occurs as composites with covellite or wolframite.

Investigation and Report by: D. E. Ayres.
Officer in Charge, Mineralogy Section; H. W. Fander.

APPENDIX B

MINERALOGICAL EXAMINATION
HATCHES CREEK BISMUTH

Two samples were examined. One was a flotation concentrate (ML/A9480) and the other residue from a ferric chloride leach (ML/A9496).

Both samples were briquetted and examined mineragraphically.

ML/A9480 : PS9351. This was found to consist of the following minerals:

Bismuth and bismuthinite, often as composites,

Chalcopyrite,

Pyrite and pyrite/chalcopyrite composites,

Digenite and covellite,

Molybdenite.

Marcasite and marcasite/chalcopyrite composites,

Possible traces of cassiterite in marcasite, and possible wolframite.

Composites of chalcopyrite/bismuthinite, marcasite/ chalcopyrite, sphalerite in chalcopyrite and chalcopyrite/covellite are also occasionally seen.

However, the majority of grains were mono-mineralic.

ML/A9496 : PS9352. This consists of molybdenite, chalcopyrite, pyrite, marcasite, and sphalerite in chalcopyrite.

Investigation and Report by: H. W. Fander.

Officer in Charge, Mineralogy Section: H. W. Fander.

TABLES 1 TO 13

FIGURES 1 AND 2

TABLE 1: SCREEN ANALYSIS OF HEAD SAMPLE

Nominal Aperture μ.	Mesh BSS	Weight %		Analysis %		Distribution %	
				Cu	WO_3	Cu	WO_3
+3350	+ 5	trace					
-3350 +2400	- 5 + 7	10.1)	68.9	0.76	67.45	51.2	72.6
-2400 +1680	- 7 + 10	14.0)					
-1680 +1200	- 10 + 14	8.2)					
-1200 + 850	- 14 + 18	10.6)					
- 850 + 600	- 18 + 25	10.1)					
- 600 + 420	- 25 + 36	8.1)					
- 420 + 300	- 36 + 52	7.8)					
- 300 + 210	- 52 + 72	7.8)	20.2	2.11	54.4	39.4	17.1
- 210 + 150	- 72 +100	5.7)					
- 150 + 105	-100 +150	6.7)					
- 105 + 75	-150 +200	2.2)	4.4	0.985	49.5	4.2	4.0
- 75 + 53	-200 +300	2.2)					
- 53	-300	6.5	6.5	0.845	62.5	5.2	6.3
		100.0	100.0	1.02	64.2	100.0	100.0

TABLE 2: SAND SLIME SEPARATION OF GROUND FEED

Product	Weight %	Assay %			Distribution %		
		Bi	Cu	WO_3	Bi	Cu	WO_3
Slime	4.9	4.2	0.81	63.4	4.1	3.7	4.8
Sand	95.1	5.0	1.08	64.3	95.9	96.3	95.2
Feed	100.0	4.9	1.07	64.3	100.0	100.0	100.0

TABLE 3: ANIONIC FLOTATION

Test No.	Sample	Fractions	Weight	Analysis %			Distribution %		
				Bi	Cu	WO_3	Bi	Cu	WO_3
3	Minus 52-mesh 100%	Recleaner concentrate	31.3	15.65	3.08	43.7	91.1	88.1	21.2
		Recleaner tailing) Cleaner tailing)	4.1) 5.2)	3.70	0.45	66.0	6.3	3.6	9.5
		Rougher tailing	59.4	0.24	0.16	74.0	2.6	8.3	69.3
		Feed	100.0	5.3	1.09	64.3	100.0	100.0	100.0
6	Deslimed minus 52-mesh 100%	Recleaner concentrate(a)) Recleaner tailing) Cleaner tailing)	18.0	23.2(b)	5.78(b)	35.0(b)	83.2	96.0	9.8
		Rougher tailing	82.0	1.01	0.054	70.7	16.8	4.0	90.2
		Feed	100.0	4.9(c)	1.07(c)	64.3(c)	100.0	100.0	100.0
8	Deslimed minus 52-mesh 100%	Rougher concentrate	23.7	8.7	4.2	54.2	40.7	91.0	19.6
		Rougher tailing	76.3	3.92	0.13	67.7	59.3	9.0	80.4
		Feed	100.0	5.0	1.09	64.4	100.0	100.0	100.0

a. This product contained 20.8 per cent Molybdenum.

b. Calculated by difference.

c. Head assay.

TABLE 4: ANIONIC FLOTATION - TEST 15

Product	Weight %	Analyses %			Distribution %		
		Bi	Cu	WO_3	Bi	Cu	WO_3
Concentrate	31.7	23.6	1.82	44.9	61.3	21.0	27.8
Tailing	68.3	6.92	3.17	53.8	38.7	79.0	72.2
Feed	100.0	12.2	2.7	51.0	100.0	100.0	100.0

TABLE 5: CATIONIC FLOTATION

Test No.	Product	Weight %		Analysis %			Distribution %		
				Bi	Cu	WO_3	Bi	Cu	WO_3
7	Recleaner concentrate	2.2		16.0	10.1	16.75	7.7	24.4	0.6
	Recleaner tailing	2.1		7.95	1.72	58.1	3.5	3.8	1.8
	Cleaner tailing	13.1		1.89	0.26	70.0	5.3	3.7	14.2
	Rougher tailing	82.6		4.65	0.76	65.0	83.5	68.1	83.4
	Feed	100.0		4.6	0.9	64.3	100.0	100.0	100.0
12	Recleaner concentrate	7.5	10.2	14.3	5.63	35.4	30.6	58.5	6.0
	Recleaner tailing	2.7							
	Cleaner tailing	16.9		9.0	0.83	52.7	31.7	14.2	14.6
	Rougher tailing	72.9		2.46	0.37	65.9	37.7	27.3	79.4
	Feed	100.0		4.7	1.0	61.0	100.0	100.0	100.0

TABLE 6: FLOTATION WITH MAGNET SEPARATION OF CONCENTRATE
Test 29

Product	Weight %		Analysis %			Distribution %		
			Bi	Cu	WO_3	Bi	Cu	WO_3
Re-clnr conc	23.6)	28.9	15.28	3.24	44.6	77.4	84.4	21.2
Re-clnr tail	5.3)							
Clnr tail	37.0)	71.1	1.80	0.26	67.1	22.6	15.6	78.8
Ro tail	34.1)							
Feed	100.0	100.0	5.7	1.12	60.8	100.0	100.0	100.0
Magnetic Separation of Recleaner Concentrate								
Magnetic	57.0		6.10	1.70	58.6	22.9	29.8	74.7
Non-magnetic	43.0		27.5	5.30	26.3	77.1	70.2	25.3
Feed	100.0		15.3	3.2	44.6	100.0	100.0	100.0

TABLE 7: FLOTATION WITH MAGNETIC AND HIGH-TENSION SEPARATION OF CONCENTRATE
Test 26

Product	Weight %	Analysis %			Distribution %		
		Bi	Cu	WO_3	Bi	Cu	WO_3
Re-clnr conc	20.4	22.33	5.42	26.12	87.5	87.3	9.1
Re-clnr tail) Clnr tail) Ro tail)	79.6	0.83	0.20	66.8	12.5	12.7	90.9
Feed	100.0	5.2	1.3	58.3	100.0	100.0	100.0
<u>Magnetic and High Tension Separation of Recleaner Concentrate</u>							
Magnetic	25.6	4.85	1.64	54.3	5.6	7.7	52.1
Pinned	29.0	16.9	3.80	38.0	21.9	20.3	41.3
Thrown	45.4 (a)	35.9	8.60	3.8	72.5	72.0	6.6
Feed	100.0	22.3	5.4	26.6	100.0	100.0	100.0

(a) Used as feed for ferric chloride leaching.

TABLE 8: ANIONIC FLOTATION

Test 23

Product	Weight %	Analysis %			Distribution %		
		Bi	Cu	WO_3	Bi	Cu	WO_3
Slimes	3.6	6.45	1.23	63.8	4.8	4.7	3.4
Recleaner concentrate	7.6	37.4 (a)	8.43(a)	5.55(a)	62.2	68.6	6.2
Bulked tailing	88.8(b)	1.7	0.28	71.5	33.0	26.7	90.4
Feed	100.0	4.6	0.9	66.2	100.0	100.0	100.0

(a) Calculated by difference.

(b) Magnetic and High Tension pinned products added.

TABLE 9: FLOTATION OF CONCENTRATE FROM TEST 23
Test 24

Product	Weight %	Analysis %			Distribution %		
		Bi	Cu	WO_3	Bi	Cu	WO_3
MoS_2 concentrate	3.5(a)	-	-	-	-	-	-
Cleaner concentrate	32.7	49.3	6.95	3.58	44.8	28.0	21.8
Cleaner tailing	33.4	47.0	9.65	2.82	43.2	39.4	17.7
Rougher tailing	30.4	17.0	8.73	10.7	12.0	32.6	60.5
Feed	100.0	37.4	8.4	5.5	100.0	100.0	100.0

(a) Assayed only for MoS_2, it contained 32.9% MoS_2.

TABLE 10: ANIONIC FLOTATION
Test 19

Product	Weight %		Analysis %			Distribution %		
			Bi	Cu	WO_3	Bi	Cu	WO_3
Recleaner concentrate	18.1	18.1	16.65[(a)]	6.13[(a)]	36.7[(a)]	75.5	90.8	10.1
Recleaner tailing	6.2)	81.9	1.18	0.14	72.0	24.5	9.2	89.9
Cleaner tailing	17.8)							
Rougher tailing	57.9)							
Feed	100.0	100.0	4.0	1.2	65.6	100.0	100.0	100.0

(a) Calculated by difference.

TABLE 11: SUMMARISED RESULTS OF TEST 20

Product	Weight %	Analysis %			Distribution %		
		Bi	Cu	WO_3	Bi	Cu	WO_3
Magnetic Separation							
Magnetic	28.3	4.22	1.34	61.4	7.1	6.2	47.2
High-Tension Separation							
Held	29.3	12.7	2.66	44.1	22.2	12.7	35.6
Flotation							
Concentrate 1(a)	1.5	-	-	-	-	-	-
Concentrate 2 recleaner concentrate	10.6	59.2	4.9	3.6	37.8	8.5	1.0
Concentrate 2 recleaner tailing Rougher tailing	30.3	18.0	14.7	20.4	32.9	72.6	16.2
Feed	100.0	16.7	6.1	36.7	100.0	100.0	100.0

(a) Assayed for MoS_2 only. It gave on analysis 34.1% MoS_2.

TABLE 12: RESULTS OF EXPERIMENTS FOLLOWING "FLOWSHEET B"
Test 27

Reagent	pH	Product	Weight %	Analysis %			Distribution %		
				Bi	Cu	WO_3	Bi	Cu	WO_3
Potassium amyl xanthate	11.2	Clnr conc	25.0	20.52	4.12	35.0	90.6	88.5	16.3
		Re-clnr tail) Clnr tail) Ro tail)	75.0	0.71	0.18	66.6	9.4	11.5	83.7
		Feed	100.0	5.6	1.1	59.8	100.0	100.0	100.0
		Reflotation of Second Cleaner Concentrate							
Sodium ethyl xanthate	11.3	Conc 1	0.8	(a)(b)	(a)	(a)	-	-	-
		Conc 2	72.2	25.60	3.89	34.41	90.7	67.2	70.8
		Tail	27.8	6.97	4.98	36.78	9.3	32.8	29.2
		Feed	100.0	20.5	4.2	35.0	100.0	100.0	100.0
		Magnetic and High Tension Separations - Concentrate 2 of Reflotation of Second Cleaner Concentrate							
		Magnetic	29.6	5.40	1.20	61.4	6.3	9.0	52.9
		Pinned	40.7	22.10	3.50	38.1	35.2	36.6	45.3
		Thrown	29.7	50.1	7.10	2.05	58.6	54.4	1.8
		Feed	100.0	25.6	3.9	34.4	100.0	100.0	100.0
		Rougher Tailing of Reflotation of Second Cleaner Concentrate							
		Magnetic	37.8	2.15	0.85	59.5	11.6	6.4	61.2
		Pinned	25.5	3.78	2.0	50.4	13.8	10.2	35.4
		Thrown	36.7	14.30	11.4	3.5	74.6	83.4	3.4
		Feed	100.0	7.0	5.0	37.0	100.0	100.0	100.0

(a) Not assayed.
(b) On analysis it have 76.2% MoS_2.

TABLE 13: SUPERPANNING OF FLOTATION CONCENTRATE

Product	Weight %	Analysis %			Distribution %		
		Bi	Cu	WO_3	Bi	Cu	WO_3
Heavy fraction	22.0	9.7	0.34	22.0	34.7	1.6	29.5
Middling	37.5	6.0	2.93	52.4	36.7	23.3	40.4
Light fraction	40.5	4.33	8.80	36.0	28.6	75.1	30.1
Feed	100.0	6.2	4.7	48.2	100.0	100.0	100.0

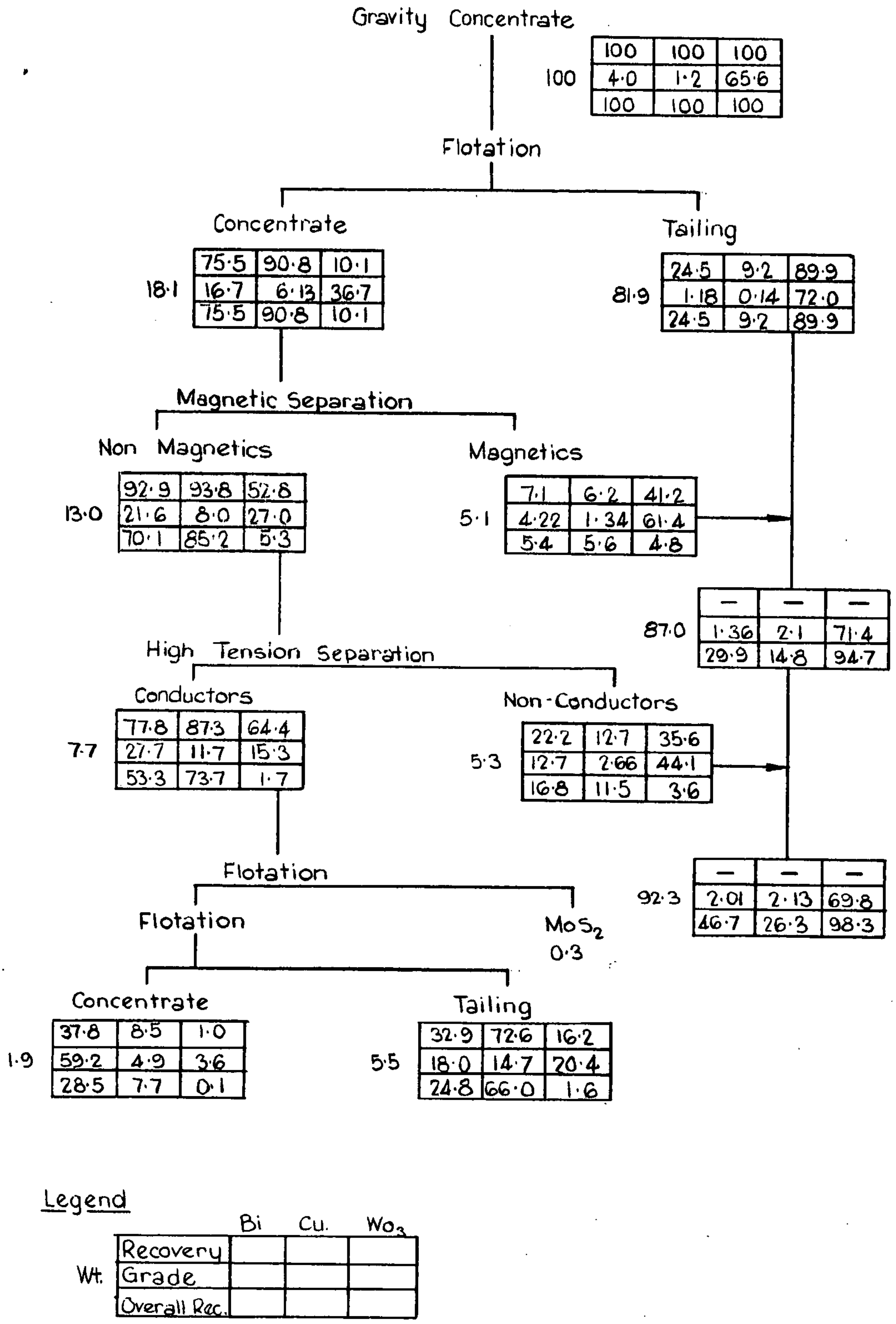

FIG. 1: FLOWSHEET SHOWING RESULTS OF TESTS 19 AND 20

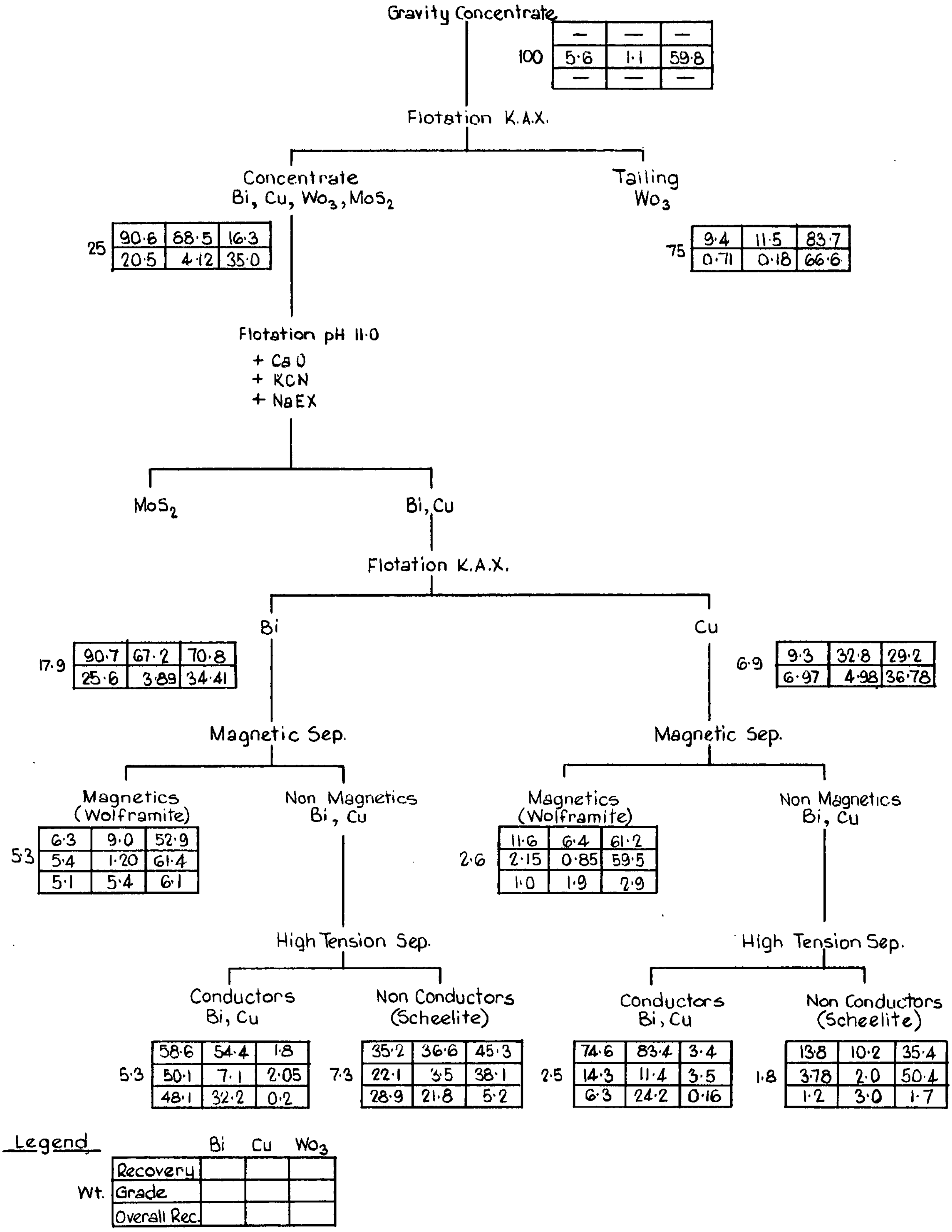

FIG. 2: FLOWSHEET B SHOWING RESULTS OF TEST 27