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Mineralogy and Benefication of Repparfjord Copper Ore

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MINISTRY OF TECHNOLOGY

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MINERALOGY AND BENEFICIATION OF REPPARFJORD COPPER ORE

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SUMMARY

This report describes work done on a copper ore from Repparfjord, Norway, to develop a flotation process and to provide additional mineralogical and grinding data. A two stage grinding and flotation process is described which results in an 86% recovery of the sulphide copper minerals at a concentrate grade of 33% copper.

INTRODUCTION

The sponsors, West's (Manchester) Limited, requested that work be done on the production of a copper concentrate in which a grade of between 25-40% Cu is obtained at a recovery of 88%.

Approximately 1½ tons of Repparfjord was received at minus 15 cm size and suitable hand specimens were selected for further mineralogical study. The balance of the ore was reduced to minus 2.5 cm and then further subdivided and crushed to provide an adequate number of 1 kg samples at minus 10 mesh B S S (1.68 m m).

After the grinding characteristic examination and liberation studies had been completed, a series of primary flotation tests using xanthates and also mercaptobenzothiazole were carried out. Based on the results of the laboratory scale tests, bulk (20 kg) floats were then carried out to provide suitable quantities of rougher concentrates for subsequent re-grinding and re-flotation examination.

Some of the products of both the primary and secondary floats were examined mineralogically and finally a Bond grindability test performed on the head ore.

Head Sample

1 kg of head ore at minus 10 mesh B S S (1.68 m m) was sent for partial chemical analysis and the data obtained are given in Table 1.

TABLE 1. - Head Ore Analysis (Partial)

Constituent	Assay %
Total Copper	0.49
Oxide Copper	0.09
*Sulphide Copper	0.40
Sulphur	0.15

*Obtained by difference: Total Copper minus Oxide Copper

It is to be noted at this stage that the total copper assay of 0.49% is much less than the reputed 0.8-0.9% Copper, a fact which was reported to the sponsors before further work was continued.

Mineralogical Composition

Examination showed the ore to be a metamorphosed intermediate igneous rock composed predominantly of silicate minerals - quartz, felspar and ferromangnesian silicates. The principal minor constituents appeared to be iron minerals - complex intergrowths of magnetite, hematite and goethite - and copper minerals.

Examination of polished briquettes showed that the pattern of the copper sulphide mineralisation was more complex than simple chalcocite with minor bornite occurrences. Both blue (neodigenite) and grey chalcocite occur, the blue variety normally rimming both bornite and chalcopyrite. The bornite frequently contains relict cores and oriented ex-solution lamellae of chalcopyrite. Minor covellite is also present (see Plate 1).

A full mineralogical modal analysis of the ore had not been requested and was not included in the proposals but a rough estimate of the relative abundance of the copper sulphide minerals can be made by examination of the assay results (Table 1).

The mineralogical examination suggested that the only sulphide minerals present were all copper or copper bearing minerals. On this basis, the sulphur and copper assays combine to give a copper sulphide mineral with an average copper content of 72.7%. This indicates that the predominate copper mineral must be chalcocite with a chalcocite to bornite ratio of about 1.4 to 1.0.

The uniform dispersion of the copper minerals and the thinness of some of the layers indicates that very fine grinding might be required for complete release.

Oxide copper minerals formed about 20% of the total copper minerals present; two major components were immediately apparent. Light blue chrysocolla was concentrated along jointing planes, which constituted primary weaknesses in the rock, while a dark green mineral, tentatively identified as malachite also occurred along the jointing planes but showed a greater tendency to be associated with the copper sulphide minerals. A lack of response to normal sulphidisation procedures in the initial flotation tests called this identification as malachite into question and on further examination the dark green constituent was judged to be the rather rare copper silicate mineral dioptase. This identification has now been confirmed by X-ray diffraction studies.

Beneficiation Studies

Assuming that the grade of the sample received was typical of the ore-body it appeared to be more than usually desirable to restrict the processing costs to an absolute minimum. Grinding forms the largest single cost element in a mineral processing flowsheet and if very fine grinding is required for adequate release of many of the smaller sulphide particles this cost becomes particularly large.

These considerations led to the adoption of a two stage process. In the first stage, grinding would proceed to the point where vigorous scavenging would produce a satisfactory rougher recovery which for the sulphide copper minerals should be in the region of 90-92%.

Because it was envisaged that the rougher concentrate would contain large quantities of tightly locked sulphide particles it would be most unsuitable for immediate cleaning. The second phase of the process would therefore consist of subjecting the rougher concentrate to fine grinding before attempting flotation to meet the requirements set out by the sponsor.

Because much of the oxide copper occurred along jointing planes, it was originally hoped that these minerals would tend to be preferentially liberated at an early stage in the grinding process to the extent that they could be satisfactorily recovered immediately after the primary sulphide float. However, as noted earlier, it was soon discovered that the non-sulphide copper minerals occurred in the form of silicates which are normally unresponsive to simple flotation techniques.

The presence of dioptase instead of the reported malachite, coupled with the fact that the head assay was only 0.5% total copper and not the reputed 0.8-0.9%, cast doubt upon the validity of the sample and the whole matter was referred to the sponsors. They stated that;

(a) leaching processes were not acceptable and the presence of copper silicates should therefore be ignored.

(b) the deposit was variable and the sample being examined was probably reasonably representative of some sections of the ore body.

Accordingly they directed that the investigation should be continued upon the sample that was sent and that work should be concentrated solely upon the recovery of sulphide minerals.

However since it is apparent from the tests reported later in Table 5, that a significant degree of association occurs between copper sulphide and copper silicate minerals even down to fine sizes, a series of oxide floats were performed to obtain the maximum overall sulphide recovery.

PRIMARY GRINDING AND PRIMARY FLOTATION

Grinding Characteristics

The initial step in the flotation process consisted in quantifying the liberation characteristics and determining the optimum mesh of grind.

Three of the 1 kg minus 10 mesh head charges were separately ground in a stainless steel mill for 800, 1600 and 3600 revolutions respectively.

The operating conditions were as follows:

Mill Dimension	Length: 30.48 cm Diameter: 15.24 cm
Grinding Media	Ten 2.54 cm diameter stainless steel rods
Operating Speed	80% critical
Pulp Density	60% solids by weight.

The resultant products were subjected to full size analyses and the results are shown in Table 2.

TABLE 2. - Grinding Characteristics of Head Ore

	800 rev. grind		1600 rev. grind		3600 rev. grind	
Size of Fraction Mesh B S S	Weight %	Cumulative % Wt finer	Weight %	Cumulative % Wt finer	Weight %	Cumulative % Wt finer
+ 36	0.32	99.68	-	100.00	-	100.00
- 36 + 52	2.26	97.42	0.02	99.98	-	100.00
- 52 + 72	13.71	83.71	0.60	99.38	0.01	99.99
- 72 + 100	25.71	58.00	6.50	92.88	0.06	99.93
- 100 + 150	15.90	42.10	19.92	77.96	0.89	99.04
- 150 + 200	9.64	32.46	18.60	54.36	4.45	94.59
- 200 + 300	8.86	23.60	17.67	36.69	18.54	76.05
- 300	23.60	-	36.69	-	76.05	-
	100.00		100.00		100.00	

The characteristics are shown in Fig. 1 and appear to be normal. To assist in the rapid correlation between mill revolutions and resultant product size distribution the data given in Table 2 have been re-plotted in Fig. 2.

Liberation Studies

Lock test series

The dynamic liberation characteristics of the copper sulphide minerals were of obvious interest and accordingly a lock test series was undertaken. In this series a number of 1 kg head charges were separately ground for 500, 1200, 1900, 2600 and 3300 revolutions. Reference to Fig. 2 will show that the span of product sizes ranged from approximately 90% minus 52 mesh to 75% minus 300 mesh. Each of the products was deslimed twice in a 2.54 cm diameter hydrocyclone under conditions which were designed to effect a d_{50} split at 12 microns for sp. gr. 2.65 nominal spheres. This desliming operation was necessary since the presence of slimes seriously impairs heavy liquid centrifuge operations. The slime fractions were analysed for copper directly but representative sub-samples of the cyclone underflow products were first subjected to successive heavy liquid separations at specific gravities of 3.0 and 3.7 and the density products were submitted for copper assays.

Unfortunately the metallurgical balances from the initial test series produced several serious discrepancies in the sulphide copper balances and examination of the products showed that the copper sulphide minerals had been attacked by the Clerici solution (thallium formate/malonate) which was used to effect the sp. gr. 3.7 separation. It was therefore necessary to repeat the tests using a single sp. gr. 3.0 separation in methylene iodide to which copper sulphide minerals are inert. The results are given in Table 3.

TABLE 3. - Results of Lock Test Series

Product	Weight, %	Total Copper		Oxide Copper		Sulphide Copper	
		Assay, %	Dist. %	Assay, %	Dist. %	Assay, %	Dist. %
	500 Rev Grind						
S G 3.0 Floats	89.10	0.20	37.2	0.06	60.9	0.14	31.9
S G 3.0 Sinks	2.35	11.50	56.4	0.62	16.6	10.88	65.3
Slimes	8.55	0.36	6.4	0.23	22.5	0.13	2.8
Head (Calc)	100.00	(0.479)	100.0	(0.088)	100.0	(0.392)	100.0
Head (Assay)		0.49		0.09		0.40	
	1200 Rev Grind						
S G 3.0 Floats	88.20	0.16	31.4	0.06	57.4	0.10	24.7
S G 3.0 Sinks	1.98	13.9	61.2	0.69	14.9	13.21	73.1
Slimes	9.82	0.34	7.4	0.26	27.7	0.08	2.2
Head (Calc)	100.00	(0.450)	100.0	(0.092)	100.0	(0.358)	100.0
Head (Assay)		0.49		0.09		0.40	
	1900 Rev Grind						
S G 3.0 Floats	84.81	0.13	22.0	0.06	52.5	0.07	14.7
S G 3.0 Sinks	2.34	14.9	69.6	0.70	16.9	14.20	82.1
Slimes	12.85	0.33	8.4	0.23	30.6	0.10	3.2
Head (Calc)	100.00	(0.502)	100.0	(0.097)	100.0	(0.405)	100.0
Head (Assay)		0.49		0.09		0.40	
	2600 Rev Grind						
S G 3.0 Floats	82.09	0.12	20.6	0.05	46.0	0.07	14.8
S G 3.0 Sinks	2.20	15.0	68.9	0.69	17.0	14.31	80.8
Slimes	15.71	0.32	10.5	0.21	37.0	0.11	4.4
Head (Calc)	100.00	(0.479)	100.0	(0.089)	100.0	(0.390)	100.0
Head (Assay)		0.49		0.09		0.40	
	3300 Rev Grind						
S G 3.0 Floats	80.16	0.13	20.9	0.05	43.1	0.08	15.8
S G 3.0 Sinks	2.19	15.5	68.1	0.72	17.0	14.78	79.9
Slimes	17.65	0.31	11.0	0.21	39.9	0.10	4.3
Head (Calc)	100.00	(0.498)	100.0	(0.093)	100.0	0.405	100.0
Head (Assay)		0.49		0.09		(0.40)	

The general trend of the results is that finer grinding, particularly beyond 1000 revs (30% passing 300 mesh), produces relatively small changes in the grades of the products. The grade of oxide copper in all products is virtually unaltered and the decrease in grade of sulphide copper in the floats is the mirror image of increases of sulphide copper grade in the slimes. The major effects are changes in weight split between the floats and the slimes.

A likely explanation of these results is that most of the sulphide copper is fairly readily liberated, but about 15% of the sulphide copper and most of the oxide copper is finely disseminated in the gangue.

The combined effects are best illustrated by using the data of Fig. 1. to convert grind times into percentages of material passing 300 mesh, and presenting the results as a lock test chart. This is shown in Fig. 3. It will be seen that maximum release of total copper occurs at around 45% minus 300 mesh. Finer grinds would definitely lead to unacceptable slime losses, whereas owing to the possibility of flotation of locked particles with some exposed mineral faces, reasonable flotation recovery although at lower grade might be expected from an appreciably coarser grind.

Flotation/Lock Test Correlation

In order to obtain some correlation between the lock test results - which give results based on particle density differences only - a series of flotation tests was carried out for a limited number of different batch grind products.

The basic procedure employed was as follows:

Grind:	For a specified number of revolutions in the laboratory rod mill.
Flotation:	A Wemco 'Mineralmaster' batch flotation cell was used.
pH:	Natural approximately 8.0.
Stage 1:	Add 0.025 kg/ton of K.A.X.(Potassium Amyl Xanthate) and condition for 2 minutes. Add 0.025 kg/ton of M.B.T.(Mercaptobenzothiazole) and 2 drops of 10% Aerofroth 65. Float until froth is clean.
Stage 2:	Repeat Stage 1 procedure with smaller collector additions of 0.0125 kg/ton of both K.A.X. and M.B.T.
Assays:	All products were assayed separately for total and oxide copper.

Although subsequent test results suggested that at coarser sizes, slightly higher recoveries might be attained by the use of K.A.X. alone, the K.A.X./M.B.T. mixture used in these tests was quite satisfactory and yielded a rapid float with good froth texture. The number of mill revolutions and flotation times used in each individual test are given in Table 4.

TABLE 4. - Flotation Series: Operating Variables

Test No	Number of Mill Revolutions	Flotation Times, min Stage 1	Stage 2
14	560	5	7
15	750	4	5
16	1000	4	4
17	1250	4	3
18	1500	4	3
19	1750	4	2
20	2000	4	2
21	2600	4	2

As the series progressed and the feeds became finer, there was an apparent tendency for more rapid flotation of the values, which is reflected in the reduced flotation times for the finer grinds. It will be remembered that flotation was continued until the froth became white and barren.

The metallurgical balances from these tests are given in Table 5.

The overwhelming bulk of the copper recovery was always made in the first flotation stage, while the effectiveness of the scavenger step also tended to diminish as the product size became finer. The compositions of the combined concentrates from each test are summarised in Table 6.

For comparison with the lock test results these overall sulphide copper recoveries have been plotted on Fig. 4. After some uncertainty at the coarser sizes - and excluding the obviously anomalous result of Test 20 - the pattern was quite similar to that of the sp. gr. 3.0 sink line. The differences were that the maximum rougher sulphide recovery was higher - at 95 per cent as compared to 85 per cent - and was attained at a slightly coarser grind - at about 80 per cent passing 120 mesh compared to 80 per cent passing 150 mesh.

Thus it appeared that two thirds of the sulphide copper contained in the sp. gr. 3.0 floats could be recovered in the rougher flotation stage. At a rough approximation this extra yield would mostly represent particles containing from about two to five per cent by volume of sulphide mineral.

It will be noted that, although the weight recoveries to concentrate varied from 5 to 8 per cent, the oxide copper recoveries ranged between 25 and 33 per cent. This concentration of oxide copper in the froth product - which persists down to 80 per cent minus 200 mesh - suggests that there must be a significant degree of association between the copper silicate and the copper sulphide minerals.

Table 5. - Flotation Series : Metallurgical Balances

Product	Weight %	Total Copper		Oxide Copper		Sulphide Copper	
		Assay %	Dist. %	Assay %	Dist. %	Assay %	Dist. %
				Test 14 : 560 Revs			
Conc 1	5.22	5.7	59.2	0.42	23.4	5.28	67.4
Conc 2	2.41	1.62	7.8	0.29	7.5	1.33	7.8
Tailing	92.37	0.18	33.0	0.07	69.1	0.11	24.8
Head (Calc)	100.00	(0.503)	100.0	(0.94)	100.0	(0.409)	100.0
Head (Assay)		0.49		0.09		0.40	
				Test 15 : 750 Revs			
Conc 1	3.63	10.0	75.7	0.46	18.8	9.54	88.6
Conc 2	2.12	0.61	2.7	0.28	6.7	0.33	1.8
Tailing	94.25	0.11	21.6	0.07	74.5	0.04	9.6
Head (Calc)	100.00	(0.480)	100.0	(0.087)	100.0	(0.391)	100.0
Head (Assay)		0.49		0.09		0.40	
				Test 16 : 1000 Revs			
Conc 1	5.99	6.0	76.3	0.38	27.6	5.62	86.7
Conc 2	1.20	1.55	4.0	0.33	4.9	1.22	3.8
Tailing	92.81	0.10	19.7	0.06	67.5	0.04	9.5
Head (Calc)	100.00	(0.470)	100.0	(0.082)	100.0	(0.388)	100.0
Head (Assay)		0.49		0.09		0.40	
				Test 17 : 1250 Revs			
Conc 1	4.12	9.9	79.5	0.45	20.7	9.45	92.0
Conc 2	2.31	0.50	2.3	0.25	6.4	0.25	1.4
Tailing	93.57	0.10	18.2	0.07	72.9	0.03	6.6
Head (Calc)	100.00	(0.513)	100.0	(0.090)	100.0	(0.423)	100.0
Head (Assay)		0.49		0.09		0.40	
				Test 18 : 1500 Revs			
Conc 1	6.41	6.0	80.7	0.34	24.0	5.66	94.0
Conc 2	1.60	0.57	1.9	0.29	5.1	0.28	1.2
Tailing	91.99	0.09	17.4	0.07	70.9	0.02	4.8
Head (Calc)	100.00	(0.477)	100.0	(0.091)	100.0	(0.386)	100.0
Head (Assay)		0.49		0.09		0.40	
				Test 19 : 1750 Revs			
Conc 1	5.53	7.6	82.1	0.45	26.3	7.15	94.7
Conc 2	1.51	0.54	1.6	0.30	4.9	0.24	0.8
Tailing	92.96	0.09	16.3	0.07	68.8	0.02	4.5
Head (Calc)	100.00	(0.512)	100.0	(0.095)	100.0	(0.418)	100.0
Head (Assay)		0.49		0.09		0.40	
				Test 20 : 2000 Revs			
Conc 1	3.91	10.0	76.5	0.50	21.9	9.50	88.0
Conc 2	1.10	0.58	1.2	0.29	3.6	0.29	0.8
Tailing	94.99	0.12	22.3	0.07	74.5	0.05	11.2
Head (Calc)	100.00	(0.511)	100.0	(0.089)	100.0	(0.422)	100.0
Head (Assay)		0.49		0.09		0.40	
				Test 21 : 2600 Revs			
Conc 1	4.80	8.2	79.5	0.47	24.3	7.73	92.2
Conc 2	2.60	0.34	1.8	0.21	5.9	0.13	0.9
Tailing	92.60	0.10	18.7	0.07	69.8	0.03	6.9
Head (Calc)	100.00	(0.495)	100.0	(0.093)	100.0	(0.402)	100.0
Head (Assay)		0.49		0.09		0.40	

TABLE 6. - Flotation Series: Composition of Combined Concentrates

Test No	Weight, %	Total Copper Assay, %	Total Copper Dist. %	Oxide Copper Assay, %	Oxide Copper Dist. %	Sulphide Copper Assay, %	Sulphide Copper Dist. %
14	7.63	4.41	67.0	0.38	30.9	4.03	75.2
15	5.75	6.54	78.4	0.39	25.5	6.14	90.4
16	7.19	5.26	80.3	0.37	32.5	4.88	90.5
17	6.43	6.52	81.8	0.38	27.1	6.14	93.4
18	8.01	4.92	82.6	0.33	29.1	4.59	95.2
19	7.04	6.09	83.7	0.42	31.2	5.67	95.5
20	5.01	7.93	77.7	0.45	25.5	7.48	88.8
21	7.40	5.44	81.3	0.38	30.2	5.06	93.1

Initial Flotation Test Work

An initial set of exploratory tests was carried out prior to and during the flotation lock test series. For these tests a relatively coarse grind (560 revs) was chosen in the expectation that a regrind circuit would eventually need to be adopted.

Test 1 was concerned with establishing the level of collector addition that would be required and successive 0.01 kg/ton additions of sodium secondary butyl xanthate (S.Sec B.X.) were made to the pulp in conjunction with Aerofroth 65 frother. The first additions produced only indiscriminate slime flotation and it appeared that at least 0.05 kg/ton of reagent would be required. Because maximum rougher recovery was the objective it was decided to standardise on an initial addition of 0.05 kg/ton followed by 0.025 kg/ton as a scavenger addition.

In Test 2, this sulphide flotation procedure was followed by an attempted oxide flotation stage. After an addition of 0.25 kg/t of sodium sulphide, which raised the pH to 9.2, no response was obtained until after 8 minutes aeration when a black coloured mineral rose into the froth. Two further additions of sodium sulphide (0.1 kg/ton each) produced very little extra response. All oxide concentrates were examined before combining them for assay and the absence of any visible green mineral led to the discovery that there was no malachite but instead a rare copper silicate - dioptase.

The results of this test are shown in Table 7.

TABLE 7. - Test 2. S. Sec B.X with Attempted Oxide Flotation

Product	Weight, %	Total Copper		Oxide Copper		Sulphide Copper	
		Assay, %	Dist. %	Assay, %	Dist. %	Assay, %	Dist. %
Sulphide Conc.	4.70	6.7	66.2	0.47	23.3	6.23	77.0
Oxide Conc.	3.22	2.13	14.4	0.55	18.6	1.58	13.4
Tailing	92.08	0.10	19.4	0.06	58.1	0.04	9.6
Head (Calc)	100.00	(0.476)	100.0	(0.095)	100.0	(0.381)	100.0
Head (Assay)		0.49		0.09		0.40	

Comparison with later results showed that, although the effect was not outstanding, the non-sulphide copper did, to some extent, respond to the sulphidisation procedure. Also the additional 13.4 per cent sulphide copper recovery in this stage - which raised the total sulphide recovery to 90.4 per cent - was a feature of interest which subsequently inspired some further experiments with NaHS.

Comparison of Xanthates

It will be convenient at this point to consider the results achieved by a test series designed to compare the effectiveness of sodium secondary butyl xanthate (S.Sec.B.X.), sodium iso-propyl xanthate (S.I.P.X.) and potassium amyl xanthate (K.A.X.). The standard two stage procedure was employed and the operating conditions are summarised below in Table 8.

TABLE 8. - Xanthate Comparisons - Operating Conditions

Test No	Type of Collector	Stage 1				Stage 2		
		Collector, kg/t	Cond. time, min	Aerofroth 65, Drops of 10%	Float Time, min	Collector, kg/t	Drops of 10% AF65	Float Time, min
7	S.Sec.B.X.	0.05	2	6	9	0.025	6	7
8	S.I.P.X.	0.05	2	6	6	0.025	6	7
9	K.A.X.	0.05	2	8	8	0.025	4	4

With amyl xanthate rather more frother was required to produce a satisfactory froth in the first stage. The results are given in Table 9.

A visual impression that K.A.X. was the best of these collectors was fully confirmed by the results. It was clearly superior to the S.Sec.B.X., which in turn appeared to be marginally better than the S.I.P.X.

TABLE 9. - Comparison of Xanthates: Metallurgical Balances

Product	Weight, %	Total Copper		Oxide Copper		Sulphide Copper	
		Assay, %	Dist. %	Assay, %	Dist. %	Assay, %	Dist. %
				Test 7: S.Sec.B.X.			
Conc. 1	2.91	10.5	62.5	0.50	16.7	10.00	72.3
Conc. 2	1.90	1.13	4.4	0.29	6.4	0.84	4.0
Tailing	95.19	0.17	33.1	0.07	76.9	0.10	23.7
Head (Calc)	100.00	(0.489)	100.0	(0.087)	100.0	(0.402)	100.0
Head (Assay)		0.49		0.09		0.40	
				Test 8: S.I.P.X.			
Conc. 1	3.63	7.7	56.2	0.43	16.0	7.27	66.1
Conc. 2	2.52	1.55	7.9	0.28	7.2	1.27	8.0
Tailing	93.85	0.19	35.9	0.08	76.8	0.11	25.9
Head (Calc)	100.00	(0.50)	100.0	(0.098)	100.0	0.399	100.0
Head (Assay)		0.49		0.09		(0.40)	
				Test 9: K.A.X.			
Conc. 1	3.90	9.5	69.8	0.46	18.0	9.04	81.7
Conc. 2	1.70	1.10	3.5	0.33	5.7	0.77	3.0
Tailing	94.40	0.15	26.7	0.08	76.3	0.07	15.3
Head (Calc)	100.00	(0.531)	100.0	(0.099)	100.0	(0.432)	100.0
Head (Assay)		0.49		0.09		0.40	

Use of K.A.X. with Various Auxiliary Collectors and Frothers

In the next few tests attempts were made to reinforce the action of the K.A.X. with various auxiliary collectors and frothers. The same K.A.X. additions as in Test 9 were made at each stage (except in Test 22) but the following variations were introduced

Test 10 In this test cresol (4 drops and 2 drops of pure reagent) was substituted for the Aerofroth 65.

Test 11 In this test additions of 0.05 kg/t and 0.025 kg/t of Aerosol 18 were made to the two stages. Two drops of 10 per cent Aerofroth 65 were also necessary in each stage.

Test 12 In this test additions of 0.05 kg/t and 0.025 kg/t of an Aerosol 22/ fuel oil mixture (in an 8:1) ratio were made. Four drops of Aerofroth 65 were found to be necessary in each stage.

Test 22 In this test the K.A.X. addition to the first stage was reduced to 0.025 kg/t while 0.06 kg/t additions of palm oil were made to both stages. Although reputedly self-frothing Aerofroth 65 (six drops and 4 drops of 10 per cent solution) was required in both stages.

The results of these tests are given in Table 10.

TABLE 10. - Use of K.A.X. with Auxiliary Collectors and Frothers

Product	Weight, %	Total Copper		Oxide Copper		Sulphide Copper	
		Assay, %	Dist. %	Assay, %	Dist. %	Assay, %	Dist. %
	Test 10. K.A.X. with Cresol Frother						
Conc. 1	3.50	7.7	53.3	0.46	18.3	7.24	60.6
Conc. 2	2.90	2.02	11.6	0.22	7.3	1.80	12.5
Tailing	93.60	0.19	35.1	0.07	74.4	0.12	26.9
Head (Calc)	100.00	(0.506)	100.0	(0.088)	100.0	(0.418)	100.0
Head (Assay)		0.49		0.09		0.40	
	Test 11. K.A.X. with Aerosol 18						
Conc. 1	2.50	10.2	52.7	0.62	15.5	9.58	62.5
Conc. 2	3.20	1.85	12.2	0.31	9.8	1.54	12.9
Tailing	94.30	0.18	35.1	0.08	74.7	0.10	24.6
Head (Calc)	100.00	(0.483)	100.0	(0.100)	100.0	(0.383)	100.0
Head (Assay)		0.49		0.09		0.40	
	Test 12. K.A.X. with Aerosol 22/Fuel Oil						
Conc. 1	4.21	7.4	62.9	0.51	22.3	6.89	72.7
Conc. 2	4.01	0.71	5.8	0.27	11.3	0.44	4.4
Tailing	91.78	0.17	31.3	0.07	66.4	0.10	22.9
Head (Calc)	100.00	(0.496)	100.0	(0.097)	100.0	(0.401)	100.0
Head (Assay)		0.49		0.09		0.40	
	Test 22. K.A.X. with Palm Oil						
Conc. 1	1.10	17.60	37.8	0.60	7.3	17.00	44.3
Conc. 2	1.50	7.60	22.2	0.36	6.0	7.24	25.7
Tailing	97.40	0.21	40.0	0.08	86.7	0.13	30.0
Head (Calc)	100.00	(0.512)	100.0	(0.090)	100.0	(0.422)	100.0
Head (Assay)		0.49		0.09		0.40	

In comparison with Test 9 the results produced by the introduction of cresol, Aerosol 18 and Aerosol 22 appear to be definitely inferior.

The K.A.X./palm oil combination used in Test 22 is of particular interest, because this is the reagent system used at the Ailik mine - a low grade (0.5 per cent Total Cu) Swedish copper operation situated north of the Arctic Circle which also employs a relatively coarse grind. The results of this test, however, were somewhat inconclusive. The total K.A.X. addition of 0.05 kg/t has clearly been less effective at recovering the values than the similar addition during the first stage of Test 9. On the other hand the selectivity of the system appears to be excellent and further reagent additions might have shown an overall improvement in flotation efficiency.

The Use of Mercaptobenzothiazole Mercaptobenzothiazole (M.B.T.) is frequently employed as a collector for copper sulphide minerals and was used - in conjunction with K.A.X. - throughout the grinding/flotation sequence reported in Tables 5 and 6.

The use of this reagent as the sole collector was explored in Test 13, in which the reagent additions were the usual 0.05 kg/t and 0.025 kg/t. Four drops of ten per cent AF 65 solution were required at each stage while the flotation times were 3 minutes and 5 minutes respectively. The results are given in Table 11.

TABLE 11. - Test 13. Use of M.B.T. as Sole Collector

Product	Weight, %	Total Copper		Oxide Copper		Sulphide Copper	
		Assay, %	Dist. %	Assay, %	Dist. %	Assay, %	Dist. %
Conc. 1	2.79	11.0	59.7	0.64	19.3	10.36	68.6
Conc. 2	2.59	1.08	5.4	0.33	9.3	0.75	4.6
Tailing	94.62	0.19	34.9	0.07	71.4	0.12	26.8
Head (Calc)	100.00	(0.515)	100.0	(0.093)	100.0	(0.424)	100.0
Head (Assay)		0.49		0.09		0.40	

This reagent, used either alone or with K.A.X. (as in Test 14), produced attractive and well-mineralised froths with rapid flotation times - which led to its selection for the Test 14 to 21 series. However, subsequent comparison with Test 9 (see Table 9) and Test 14 (see Table 5) showed that better recoveries were attained by the use of K.A.X. alone, and that M.B.T or M.B.T./K.A.X. mixtures produced similarly inferior results.

The Effects of Lime, Flocculation and Alkaline pH

During an unrelated investigation on another siliceous sulphide ore excellent flotation results were regularly obtained over a wide range of operating conditions - except when the natural strong flocculation of the ore was broken by the addition of sodium silicate.

The Repparfjord ore is rather unusual in being naturally totally dispersed and - with the results of this earlier investigation in mind - it appeared reasonable to explore whether the flotation selectivity might be improved by the preliminary addition of flocculating agents.

The initial exploration, conducted during the earlier stages of the investigation when S.Sec.B.X. was still being used as the collector, consisted of the following tests.

Test 4 In this test the pH of the pulp was raised to 9.7 with lime immediately prior to the addition of the Stage 1 collector increment.

Test 5 In this test 0.25 kg/t of lime was added to the pulp and conditioned for 5 minutes before the machine was switched off and the pulp permitted to settle for 40 minutes at a pH of 10.8. Visual check then showed that flocculation was complete - as demonstrated by the presence of a considerable depth of clear supernatant liquor. The machine was now switched on and the test continued according to the normal sequence.

Test 6 In this test - in an attempt to differentiate between those effects due to calcium ions and those merely due to higher pH - the Test 5 procedure was repeated using 0.25 kg/t of calcium chloride. Reasonable flocculation was noted after the 40 minute settlement period and flotation took place at a pH of 8.2.

Test 7 This was the standard comparison test using S.Sec.B.X. without any modifications.

The flotation times were as follows: Test 4 - 6 min and 6 min. Test 5 - 8 min and 6 min. Test 6 - 5 min and 6 min. Test 7 - 9 min and 7 min. The results are shown in Table 12.

Further Tests with K.A.X. and lime

In view of the effect of lime addition when using S.Sec.B.X. further tests were undertaken with lime in conjunction with K.A.X. and K.A.X./M.B.T. mixtures.

Test 23 In this test 0.25 kg/ton of lime was added to the pulp and conditioned for 5 minutes immediately prior to the first K.A.X. addition. Flotation pH was 10.7 and the flotation times were 8 min and 4 min.

Test 24 A similar procedure was employed in this test - in which a mixture of K.A.X./M.B.T. was used as the collector and the flotation times were again 8 min and 4 min.

Test 25 In this test the lime was added to the rod mill charge rather than the cell. K.A.X. was again the sole collector and flotation times were 8 min and 4 min.

The results of these tests are given in Table 13.

TABLE 12. - Tests 4 to 7: S.Sec.B.X. with pH Modification

Product	Weight, %	Total Copper		Oxide Copper		Sulphide Copper	
		Assay, %	Dist. %	Assay, %	Dist. %	Assay, %	Dist. %
	Test 4 - At pH 9.7 with Lime						
Conc. 1	3.23	8.8	58.9	0.78	23.7	8.02	68.9
Conc. 2	1.86	2.5	9.6	0.29	5.1	2.21	10.9
Tailing	94.91	0.16	31.5	0.08	71.2	0.08	20.2
Head (Calc)	100.00	(0.483)	100.0	(0.106)	100.0	(0.376)	100.0
Head (Assay)		0.49		0.09		0.40	
	Test 5 - 0.25 kg/t Lime at pH 10.8						
Conc. 1	4.01	8.5	71.2	0.40	16.4	8.10	85.0
Conc. 2	1.66	1.51	5.2	0.34	5.9	1.17	5.1
Tailing	94.33	0.12	23.6	0.08	77.7	0.04	9.9
Head (Calc)	100.00	(0.479)	100.0	(0.097)	100.0	(0.382)	100.0
Head (Assay)		0.49		0.09		0.40	
	Test 6 - 0.25 kg/t $CaCl_2$ at pH 8.2						
Conc. 1	3.29	9.5	64.6	0.54	17.9	8.96	76.7
Conc. 2	1.45	1.94	5.8	0.36	5.2	1.58	6.0
Tailing	95.26	0.15	29.6	0.08	76.9	0.07	17.3
Head (Calc)	100.00	(0.484)	100.0	(0.099)	100.0	(0.384)	100.0
Head (Assay)		0.49		0.09		0.40	
	Test 7 - Unmodified Standard Float						
Conc. 1	2.91	10.5	62.5	0.50	16.7	10.00	72.3
Conc. 2	1.90	1.13	4.4	0.29	6.4	0.84	4.0
Tailing	95.19	0.17	33.1	0.07	76.9	0.10	23.7
Head (Calc)	100.00	(0.489)	100.0	(0.087)	100.0	(0.402)	100.0
Head (Assay)		0.49		0.09		0.40	

In comparison with Test 9 the overall sulphide recoveries in both Tests 23 and 25 were slightly inferior. However, these tests were useful in confirming the general superiority of K.A.X. over the collector systems previously employed. Similarly the recovery in Test 24 with the K.A.X./M.B.T. mixture was lower than in the comparable Test 14.

These further tests showed no advantage from the addition of lime, and suggest that the improvements obtained in Tests 5 and 6 during the earlier series were probably due to the 40 minute quiescent period which enabled proper formation of the flocs. To exploit this advantage in practice, therefore, might entail the introduction of a thickener between the milling and flotation sections - which would probably be a rather unwelcome complication in what is visualised as a cheap and

TABLE 13. - Tests 23 - 25. Further Tests with Lime

Product	Weight, %	Total Copper		Oxide Copper		Sulphide Copper	
		Assay, %	Dist. %	Assay, %	Dist. %	Assay, %	Dist. %
		Test 23. K.A.X. with Lime to Cell					
Conc. 1	6.18	5.2	67.0	0.37	24.7	4.83	77.2
Conc. 2	1.80	1.63	6.1	0.30	5.8	1.33	6.2
Tailing	92.02	0.14	26.9	0.07	69.5	0.07	16.6
Head (Calc)	100.00	(0.480)	100.0	(0.093)	100.0	(0.378)	100.0
Head (Assay)		0.49		0.09		0.40	
		Test 24. K.A.X./M.B.T. with Lime to Cell					
Conc. 1	5.85	4.63	57.5	0.37	23.5	4.26	65.8
Conc. 2	2.00	1.24	5.3	0.30	6.5	0.94	5.0
Tailing	92.15	0.19	37.2	0.07	70.0	0.12	29.2
Head (Calc)	100.00	(0.471)	100.0	(0.092)	100.0	(0.379)	100.0
Head (Assay)		0.49		0.09		0.40	
		Test 25. K.A.X. with Lime to Mill					
Conc. 1	4.46	6.7	61.8	0.39	16.0	6.31	75.0
Conc. 2	3.56	1.06	7.8	0.24	7.8	0.82	7.8
Tailing	91.98	0.16	30.4	0.09	76.2	0.07	17.2
Head (Calc)	100.00	(0.484)	100.0	(0.109)	100.0	(0.375)	100.0
Head (Assay)		0.49		0.09		0.40	

simple flowsheet. Alternatively floc formation might be stimulated - under dynamic conditions - by increasing the lime addition. However, this would increase the cost and might also raise the pH of the effluent water to inconvenient levels.

This data was unavailable when the further experiments with hydrosulphide - described in the next section - were undertaken.

Further Tests with NaHS

In a review of the earlier test results the significantly increased copper sulphide recovery obtained - after sulphidisation - during the 'oxide' flotation stage of Test 2 attracted attention. Accordingly two further tests, Nos. 26 and 27, featuring hydrosulphide were undertaken. These tests were designed as replicates and the conditions were therefore identical.

Grind 560 revolutions with 0.25 kg/t of lime

Stage 1 Natural pH 10.6
Add 0.05 kg/t of commercial grade NaHS (pH ⟶ 11.2)
Condition for 5 minutes

Add 0.05 kg/t of K.A.X. and condition for 2 minutes

Add 10 per cent Aerofroth 65 solution (9 drops in two stages of 5 and 4 drops) and float for a total of 8 minutes - mineralised froth only started to appear after 4 to 5 minutes of aeration.

Stage 2 Add 0.025 kg/t of K.A.X. and condition for 2 minutes

Add 4 drops of additional 10 per cent AF 65 and float for 4 minutes.

The results are given in Table 14.

TABLE 14. - Tests 26 and 27. K.A.X. with Lime and NaHS

Product	Weight, %	Total Copper		Oxide Copper		Sulphide Copper	
		Assay, %	Dist. %	Assay, %	Dist. %	Assay, %	Dist. %
	Test 26 - Balance						
Conc. 1	7.38	3.5	54.0	0.39	31.6	3.11	59.2
Conc. 2	3.37	2.3	16.2	0.26	9.7	2.04	17.8
Tailing	89.25	0.16	29.8	0.06	58.7	0.10	23.0
Head (Calc)	100.00	(0.479)	100.0	(0.091)	100.0	(0.388)	100.0
Head (Assay)		0.49		0.09		0.40	
	Test 27 - Balance						
Conc. 1	5.65	5.0	54.7	0.54	30.0	4.46	60.7
Conc. 2	3.46	2.04	13.6	0.22	7.5	1.82	15.2
Tailing	90.89	0.18	31.7	0.07	62.5	0.11	24.1
Head (Calc)	100.00	(0.517)	100.0	(0.102)	100.0	(0.415)	100.0
Head (Assay)		0.49		0.09		0.40	

Although the weight recovery increased in both tests (and visual examination showed many locked sulphide particles) the results, which are directly comparable to those of Test 25 (see Table 13), were distinctly inferior. The overall copper sulphide recovery was reduced by 6 to 7 per cent while the concentrate grade also deteriorated. However, it is still possible that a smaller NaHS addition during the scavenger stage - after the recovery of the bulk of the liberated sulphide - might have yielded an overall performance improvement.

As replicates these two tests appeared to be reasonably satisfactory.

Bulk Flotation Tests without Oxide Float

On the basis of the primary flotation results it was decided that bulk floats (20 kg charges) would be carried out using the K.A.X./Aerofroth 65 combination of Test 9 and the primary grinding would be somewhere between 750 to 1250 revolutions of the mill. This is a somewhat less than indicated by the lock test work in Fig. 3 but that refers to completely liberated values whilst in flotation it is obviously possible to float locked sulphide particles (i.e.) at a coarser size produced by less grinding. Since the rougher concentrate is to be reground there is every advantage from the primary grinding cost point of view in floating the sulphides at as coarse a size as possible concomitant with maximum sulphide recovery.

The bulk sulphide float were carried out on feeds which had been ground for 750, 1000 and 1250 revolutions of the mill. In the first bulk float using a unit Wemco cell, the apparently inferior flotation performance that resulted was diagnosed as being due to too slow an impeller speed. The speed was therefore raised from 410 rev/min to 670 rev/min.

Along with the first bulk concentrate which was produced with the incorrect impeller speed after a 750 rev grind three other bulk concentrates were produced from primary grinds of 750, 1000 and 1250 revolutions of the mill respectively.

The following general conditions were used for the floats.

Grind 20 kg of -10 mesh ore for differing grind times at 60% solids

Flotation Natural pH approximately 8.0
0.22 kg/ton K.A.X.
Condition 2 minutes
Add 10 ml 10% Aerofroth 65
Float 10 minutes
Add 0.11 kg K.A.X.
Condition 2 minutes
Add 5 ml 10% Aerofroth 65
Float 8-14 minutes (until froth is barren).

The concentrates from the four bulk floats were combined to form feed for the re-grinding and re-flotation tests in Table 15. The results of the four floats (Tests 30-33) are given as follows:

TABLE 15. - (Balances for Tests 30-33)

Test 30

Grind: 20 kg of -10 mesh ground for 750 revs at 60% solids.

Float: Wemco cell with impeller speed too low for ideal floatation
(410 r.p.m.) 0.33 kg/t
Potassium Amyl Xanthate (K.A.X.)
Aerofroth 65 as 10% solution 15.0 mls.

		Assay %			Distribution %		
Product	% Wt	Total Cu.	Oxide Cu.	Sulphide Cu.	Total Cu.	Oxide Cu.	Sulphide Cu.
Conc.	6.34	4.470	0.290	4.18	58.2	21.4	66.1
Tail.	93.66	0.217	0.072	0.145	41.8	78.6	33.9
	100.00	(0.487)	(0.086)	(0.401)	100.0	100.0	100.0

Test 31

A new motor pulley was fitted which raised the impeller speed to 670 r.p.m. and resulted in a 19% increase in sulphide recovery.

Grind: 20 kg of -10 mesh ground for 750 revs of the mill at 60% solids.

Reagents: As in Test 30.

The following results were obtained:

		Assay %			Distribution %		
Product	% Wt	Total Cu.	Oxide Cu.	Sulphide Cu.	Total Cu.	Oxide Cu.	Sulphide Cu.
Conc.	18.07	2.27	0.255	2.015	78.2	47.6	85.1
Tail.	81.93	0.14	0.062	0.078	21.8	52.4	14.9
	100.00	(0.53)	(0.097)	(0.43)	100.0	100.0	100.0

Test 32

Grind: 20 kg of -10 mesh ground for 1000 revs at 60% solids.

Reagents: As in Test 30.

The following results were obtained:

		Assay %			Distribution %		
Product	% Wt	Total Cu.	Oxide Cu.	Sulphide Cu.	Total Cu.	Oxide Cu.	Sulphide Cu.
Conc.	13.93	4.500	0.251	2.249	81.0	36.7	93.6
Tail.	86.07	0.095	0.070	0.025	19.0	63.3	6.4
	100.00	(0.43)	(0.095)	(0.335)	100.0	100.0	100.0

Test 33

Grind: 20 kg of -10 mesh ground for 1250 revs at 60% solids.

Reagents: As in Test 30.

The following results were obtained:

		Assay %			Distribution %		
Product	% Wt	Total Cu.	Oxide Cu.	Sulphide Cu.	Total Cu.	Oxide Cu.	Sulphide Cu.
Conc.	14.97	2.47	0.28	2.19	77.5	45.5	85.2
Tail.	85.03	0.126	0.059	0.067	22.5	54.5	14.8
	100.00	(0.48)	(0.092)	(0.385)	100.0	100.0	100.0

The concentrates were bulked (from Tests 30-33) and the following table (Table 16) expresses the bulk percentage recoveries of total, oxide and sulphide copper into the concentrates.

TABLE 16

		Assay %			Distribution %		
Product	% Wt	Total Cu.	Oxide Cu.	Sulphide Cu.	Total Cu.	Oxide Cu.	Sulphide Cu.
F30 Conc	1.57	4.47	0.29	4.18	14.5	4.9	16.8
Tail	23.35	0.22	0.07	0.15	10.7	17.8	9.0
F31 Conc	4.77	2.27	0.26	2.02	22.4	13.5	24.6
Tail	21.64	0.14	0.06	0.08	6.3	14.1	4.4
F32 Conc	3.49	2.50	0.25	2.25	18.1	9.5	20.1
Tail	21.60	0.10	0.07	0.03	4.5	16.4	1.7
F33 Conc	3.53	2.47	0.28	2.19	18.1	10.7	19.8
Tail	20.05	0.13	0.06	0.07	5.4	13.1	3.6
Head Ore	100.00	(0.48)	(0.09)	(0.39)	100.0	100.0	100.0

The overall recovery of Total Copper in concs (Tests 30-33) is 73.1% at 2.64% Cu.
The overall recovery of Sulphide Copper in these concs is 81.3% at 2.38% Cu.

Because the operation of the impeller undoubtedly affected the flotation results of test 30 it is possible to calculate a hypothetical total copper and sulphide copper recovery which leaves out the lower recovery figure of Test 30 when the wrong impeller speed was used. The calculations are given in Table 17.

TABLE 17

Product	% Wt	% Total Cu	% Sulphide Cu	Dist. % Total Cu	Dist. % Sul. Cu
F31 Concentrate	6.3	2.27	2.015	29.9	33.1
Tail.	28.8	0.14	0.078	8.4	5.9
F32 Concentrate	4.7	2.50	2.249	24.6	27.6
Tail.	28.8	0.095	0.025	5.7	1.9
F33 Concentrate	4.7	2.47	2.19	24.3	26.8
Tail.	26.7	0.126	0.067	7.1	4.7
	100.0	(0.48)	(0.38)	100.0	100.0

The three concentrates would then represent a total copper recovery of 78.8% at 2.40% Cu and a total Sulphide copper recovery of 87.5% at 2.12% Cu.

Re-grind and re-flotation tests

In the first two tests (Tests 35 and 36) the level of K.A.X. addition was settled and although there is not a great difference in the grade and recoveries of these two tests it was finally decided that an addition rate of 0.22 kg/ton K.A.X. would be standard. The first results given in Table 18 are for concentrates ground at 50% solids for 560 revs.

TABLE 18

Test 35 (0.11 kg/t K.A.X.) 560 revs

		Assay %			Distribution %		
Product	% Wt	Total Cu.	Oxide Cu.	Sulphide Cu.	Total Cu.	Oxide Cu.	Sulphide Cu.
Concentrate	7.57	35.0	1.53	33.47	93.0	41.8	98.5
Rougher Tail	74.31	0.176	0.14	0.036	4.6	37.5	1.0
Cleaner Tail 1	13.41	0.309	0.28	0.029	1.4	13.6	0.2
" " 2	3.45	0.54	0.40	0.14	0.7	5.0	0.2
" " 3	1.26	0.76	0.47	0.29	0.3	2.1	0.1
	100.00	(2.85)	(0.28)	(2.57)	100.0	100.0	100.0

Test 36 (0.22 kg/t K.A.X.) 560 revs

		Assay %			Distribution %		
Product	% Wt	Total Cu.	Oxide Cu.	Sulphide Cu.	Total Cu.	Oxide Cu.	Sulphide Cu.
Concentrate	7.10	36.0	1.14	34.86	91.7	33.7	97.2
Rougher Tail	76.33	0.20	0.14	0.06	5.5	44.5	1.8
Cleaner Tail 1	12.14	0.37	0.27	0.10	1.6	13.7	0.5
" " 2	3.36	0.65	0.42	0.23	0.8	5.9	0.3
" " 3	1.07	1.07	0.49	0.58	0.4	2.2	0.2
	100.00	(2.79)	(0.24)	(2.55)	100.0	100.0	100.0

The remaining tests were done with the following grinding conditions:

(i) No regrind
(ii) 200 revs
(iii) 380 "
(iv) 750 "

The results of these re-grind, re-float tests are given in Table 19.

TABLE 19

		Assay %			Distribution %		
Product	% Wt	Total Cu.	Oxide Cu.	Sulphide Cu.	Total Cu.	Oxide Cu.	Sulphide Cu.
Test 37 (No regrind)							
Concentrate	12.77	20.0	0.90	19.10	91.8	42.2	97.2
Rougher Tail	62.77	0.18	0.13	0.05	4.0	30.0	1.2
Cleaner Tail 1	16.69	0.26	0.26	0.10	2.2	15.9	0.7
" " 2	5.62	0.65	0.40	0.25	1.3	8.3	0.6
" " 3	2.15	0.86	0.45	0.41	0.7	3.6	0.3
	100.00	(2.78)	(0.27)	(2.51)	100.0	100.0	100.0
Test 38 (200 rev regrind)							
Concentrate	9.66	26.0	1.15	24.85	91.7	42.2	96.9
Rougher Tail	72.36	0.21	0.14	0.07	5.5	38.5	2.1
Cleaner Tail 1	11.99	0.33	0.25	0.08	1.4	11.4	0.4
" " 2	4.27	0.55	0.36	0.19	0.9	5.9	0.3
" " 3	1.72	0.73	0.30	0.43	0.5	2.0	0.3
	100.00	(2.74)	(0.26)	(2.48)	100.0	100.0	100.0
Test 39 (380 rev regrind)							
Concentrate	9.42	26.0	0.91	25.09	89.3	36.0	94.4
Rougher Tail	69.10	0.32	0.14	0.18	8.0	40.7	5.0
Cleaner Tail 1	14.65	0.28	0.22	0.06	1.5	13.5	0.3
" " 2	5.69	0.42	0.33	0.09	0.9	7.9	0.2
" " 3	1.14	0.63	0.40	0.23	0.3	1.9	0.1
	100.00	(2.74)	(0.24)	(2.50)	100.0	100.0	100.0
Test 40 (750 rev regrind)							
Concentrate	7.83	33.0	1.10	31.90	92.0	35.1	97.4
Rougher Tail	66.72	0.21	0.13	0.08	5.0	35.3	2.1
Cleaner Tail 1	17.93	0.28	0.25	0.03	1.8	18.2	0.2
" " 2	5.70	0.40	0.35	0.05	0.8	8.1	0.1
" " 3	1.82	0.66	0.44	0.22	0.4	3.3	0.2
	100.00	(2.81)	(0.25)	(2.56)	100.0	100.0	100.0

The cleaner concentrates produced from Tests 36-40 inclusively and also the original re-grind feed were dry screened and the results are presented in Table 20 and also graphically in Figure 5.

TABLE 20. - Size Analyses of feed for regrind tests and also of concentrates produced in tests 36-40

Grind Conditions	Regrind Head		No Regrind		200 Rev Grind	
Mesh Size B S S	Wt %	Cumulative % Finer	Wt %	Cumulative % Finer	Wt %	Cumulative % Finer
+ 52	0.40	99.60	0.30	99.70	0.02	99.98
- 52 + 72	3.34	96.26	6.64	93.06	0.10	99.88
- 72 + 100	7.77	88.49	21.01	72.05	0.10	99.78
- 100 + 150	7.33	81.16	18.04	54.01	10.48	89.30
- 150 + 200	6.91	74.25	12.12	41.89	17.35	71.95
- 200 + 300	10.02	64.23	10.27	31.62	19.72	52.23
- 300	64.23		31.62		52.23	
	100.00		100.00		100.00	

Grind Conditions	380 Rev		560 Rev		750 Rev	
Mesh Size B S S	Wt %	Cumulative % Finer	Wt %	Cumulative % Finer	Wt %	Cumulative % Finer
+ 52	0.025	99.975	0.03	99.97	-	100.00
- 52 + 72	0.025	99.95	0.06	99.91	-	100.00
- 72 + 100	0.06	99.89	0.37	99.54	0.03	99.97
- 100 + 150	5.10	94.79	2.20	97.34	0.89	99.08
- 150 + 200	11.97	82.82	7.96	89.38	4.32	94.76
- 200 + 300	19.15	63.67	20.26	69.12	15.18	79.58
- 300	63.67		69.12		79.58	
	100.00		100.00		100.00	

Using the sulphide copper recovery figures from Table 18 (for Test 36) and Table 19 (for Tests 37-40) it is possible to construct a family of grade-recovery curves to find out where the optimum regrind conditions lie. The curves are constructed by calculating the recovery-grade figures for the concentrate alone, concentrate plus cleaner tail 1, concentrate plus cleaner tail 1 plus cleaner tail 2 and finally the concentrate plus the three cleaner tails. The curves are shown in Figure 6 and it can be seen that regrinding of the rougher concentrates using either the 560 of 750 rev. grind will produce concentrates with the optimum overall grade-recovery relationships. From Table 20, it will be apparent that these grind conditions gave concentrates with a size distribution of between 69 and 79% minus 300 mesh.

It is now possible at this stage to estimate a final recovery and grade for the first two stage processes involving bulk flotation without oxide flotation.

The best recovery for Tests 31-33 (i.e.) those using the correct impeller speed has been estimated at 87.5% sulphide copper recovery. In the regrind tests, tests 35 and 36 and 40 have given reasonable sulphide recoveries and grades and the best possible results would be

Test 35: 87.5 x 98.5 = 86.2% sulphide recovery at 33.5% copper
" 36: 87.5 x 97.2 = 85.1% " " at 34.9% "
" 40: 87.5 x 97.4 = 85.2 " " at 31.9% "

Using the actual sulphide recovery figure of 81.3% (cf) Table 16 Tests 30-33, these overall sulphide recovery figures would be as follows:

Test 35: 81.3 x 98.5 = 80.1% Sulphide recovery at 33.5. copper
" 36: 81.3 x 97.2 = 79.0% " " at 34.9% "
" 40: 81.3 x 97.4 = 79.2% " " at 31.9% "

Bulk floats with oxide concentrate float

As it was earlier noticed there seems to be some significant degree of association between oxide copper minerals and the sulphide copper minerals even down to finely ground feed sizes. There is therefore every good reason to attempt an oxide float after the bulk rougher sulphide float in order to obtain the maximum sulphide recovery at this stage.

In the bulk rougher float, 20 kg of minus 10 mesh head ore was ground for 1250 revs at 60% solids with 1.1 kg/ton lime. The lime was added to help flocculation because as it was earlier pointed out this seems to aid sulphide recovery. The K.A.X. addition of 0.33 kg/t was the same as in the earlier bulk sulphide float. After a sulphide rougher concentrate had been taken, an oxide concentrate was floated off after first conditioning the pulp with 0.22 kg/ton commercial sodium sulphide for 5 minutes and then adding 0.11 kg/ton K.A.X. and Aerofroth 65. A further addition of 0.11 kg/ton Sodium Sulphide was made towards the end of the float. The results of Test 42 and also for Test 45 (which is identical to Test 42 except that in the former case the bulk of the lime is added to the mill whilst in Test 45 lime is added to the cell directly) are given in Table 21.

Mineralogical Examination of Products

Representative products from Tests 32, 36 and 42 have been examined mineralogically as polished sections and the following observations were made.

Sulphide and Oxide Concentrates

Grains in both samples range from 10-150 μm in diameter, the copper sulphides tending to be uniformly small (10-50 μm), the iron minerals larger (50-100 μm) and the silicates spanning the entire range.

TABLE 21

Test 42

Product	% Wt	Assay % Total Cu.	Oxide Cu.	Sulphide Cu.	Distribution % Total Cu.	Oxide Cu.	Sulphide Cu.
Sulphide Conc.	5.56	5.60	0.39	5.21	67.0	24.6	77.0
Oxide Conc.	3.49	1.00	0.34	0.66	7.5	13.5	6.1
Tail	90.95	0.13	0.06	0.07	25.5	61.9	16.9
	100.00	(0.46)	(0.088)	(0.376)	100.0	100.0	100.0

Test 45

Product	% Wt	Assay % Total Cu.	Oxide Cu.	Sulphide Cu.	Distribution % Total Cu.	Oxide Cu.	Sulphide Cu.
Sulphide Conc.	9.62	2.81	0.29	2.52	57.9	29.2	65.3
Oxide Conc.	4.13	1.42	0.38	1.04	12.6	16.5	11.5
Tail	86.25	0.16	0.06	0.10	29.5	55.3	23.2
	100.00	(0.47)	(0.096)	(0.37)	100.0	100.0	100.0

In both concentrates, over 90% of the copper mineral is bornite ⟶ chalcocite with subsidiary covellite and chalcopyrite (c.a. 8% covellite and 2% chalcopyrite). The oxide concentrate contained more surface exposed bornite than the sulphide concentrate.

The iron minerals are magnetite (up to equal proportions in oxide conc. but absent in the sulphide conc.) and hematite. Small amounts of goethite alteration product are visible.

The bulk of both products is quartz, up to 95% in the case of the oxide concentrate and 90% in the sulphide concentrate.

Rougher Tails Test 42 (Moderate Sulphide Recovery)

A sink/float using methylene iodide was carried out and the following observations made.

Sp.gr. 3.3 Sinks. Mainly hematite and magnetite often in composition with quartz. Some grains of magnetite with small amounts of attached sulphide. Hematite partially altered to goethite. Some dioptase, mainly liberated.

Sp.gr. 3.0 Floats. Mainly quartz often with chloritic material in composition small amounts of dioptase/quartz. No trace of chrysocolla.

Rougher Tails Test 32

An example of good primary sulphide recovery. No visible sulphides. Large amounts of hematite/goethite and magnetite. Chloritic composites with quartz. Some liberated and some composite dioptase. Mainly quartz.

Regrind Rougher Tails Test 36

Sp.gr. 3.3 Sinks. Mainly hematite and magnetite. No sulphides visible. Some dioptase (liberated). Hematite with little alteration to goethite.

3.3 Floats. Some chlorite/quartz composition. No trace of chrysocolla. Some small amounts of composite dioptase/quartz.

The main conclusions to be drawn from the brief examination of these products are:

(1) Sulphide flotation very efficient except for those composites of magnetite/sulphide which may be physically too heavy for flotation.
(2) Dioptase did not respond to flotation.
(3) Chrysocolla appears to have gone entirely with slimes.
(4) Main heavy minerals in tails are iron oxides.

RE-GRIND AND RE-FLOTATION TESTS

The bulked oxide and sulphide concentrates were then treated further by regrinding for 750 revs. of the mill at 50% solids. Recleaner concentrates for both the sulphide and oxide stages were obtained and the results for Test 43 are shown in Table 22.

The general conditions for the float were:

Sulphide Rougher Float: 0.22 kg/ton K.A.X.
10 drops 10% AF 65 solution. Clean concentrate three times and combine tails with oxide float feed.

Oxide Rougher Float: 0.22 kg/ton. Commercial sodium sulphide followed by 0.1 kg/t K.A.X. after conditioning.
Float Rougher oxide concentrate 1.
Add further 0.1 kg/ton sodium sulphide.
Float Rougher oxide concentrate 2.
Clean combined concs. once.

The oxide concentrate was examined under a binocular microscope and was found to contain a fair proportion of magnetic material which was probably magnetite.

TABLE 22

Test 43

		Assay %			Distribution %		
Product	% Wt	Total Cu.	Oxide Cu.	Sulphide Cu.	Total Cu.	Oxide Cu.	Sulphide Cu.
Sulphide Conc.	13.10	31.0	1.48	29.52	94.8	54.1	98.6
Oxide Conc.	3.03	2.51	1.49	1.02	1.8	12.6	0.8
Oxide Cl. Tail	7.36	0.30	0.27	0.03	0.5	5.5	0.1
Oxide Rougher Tail	76.51	0.16	0.13	0.03	2.9	27.8	0.5
	100.00	(4.28)	(0.36)	(3.92)	100.0	100.0	100.0

The total sulphide recovery for the oxide and sulphide concentrates in Test 43 is 99.4%. Using the rougher concentrate recovery in Test 42 of 83.1% the total sulphide recovery is therefore:

99.4 x 83.1% = 82.6% at 24.2% sulphide copper.

If the sulphide concentrate of Test 43 is taken alone then the overall recovery becomes:

98.6 x 83.12 = 81.9% at 29.5% sulphide copper.

In this series of float tests the grade of concentrate is barely at the minimum required level whilst the recovery is somewhat below the minimum and as in the series of tests 36-40 there is no problem in the re-grind and cleaning stages. The success of processing this ore obviously rests on an adequate (90-92%) bulk sulphide rougher concentrate recovery.

CONCLUSIONS

It is apparent from the results of the secondary processing of this ore that acceptable grades and stage recoveries are no major problem. However in the bulk rougher processing stage, the recoveries are somewhat disappointing especially since the initial bulk floats of Tests 30-33 were not very well replicated in bulk tests 42 and 45.

In contrast to the earlier remarks made on the beneficial use of lime as a flocculant prior to flotation, it has now been noticed that one significant difference between the conditions of Tests 30-33 and Tests 42 and 45 was that in the former no lime was added and the best average total sulphide recovery was

87.5% (Table 17) whilst in the latter tests where lime was added the sulphide recovery was about 77%. It did however result in a higher rougher sulphide conc. grade and must have some effect on selectivity presumably by flocculation of the slimes. On these results there seems to be no strong basis for lime addition in the grinding stage because of the possible deleterious effect on rougher sulphide recovery.

In bulk floats incorporating an oxide rougher float it has been found necessary to retain such a step in the operations if maximum sulphide copper recovery is to be maintained. From the results obtained in the regrind and reflotation tests, it is obvious that an oxide concentrate step is not necessary. Because so called 'oxide copper' in this ore is predominantly in silicate form, it would not be beneficial to float off a 'silicate oxide concentrate' for the reason that in normal pyrometallurgical operations the bulk of the feed to the reverberatory furnace is sulphide in nature. Most losses occur into the slag either as metal inclusions or in the formation of non-reducible silicate copper complexes of the type that would already exist in the oxide concentrate. If an ore contains economically extractable amounts of oxide, the commonest type of flowsheet would simply involve the production of a high grade sulphide copper concentrate for smelting and a low grade oxide concentrate for acid leaching.

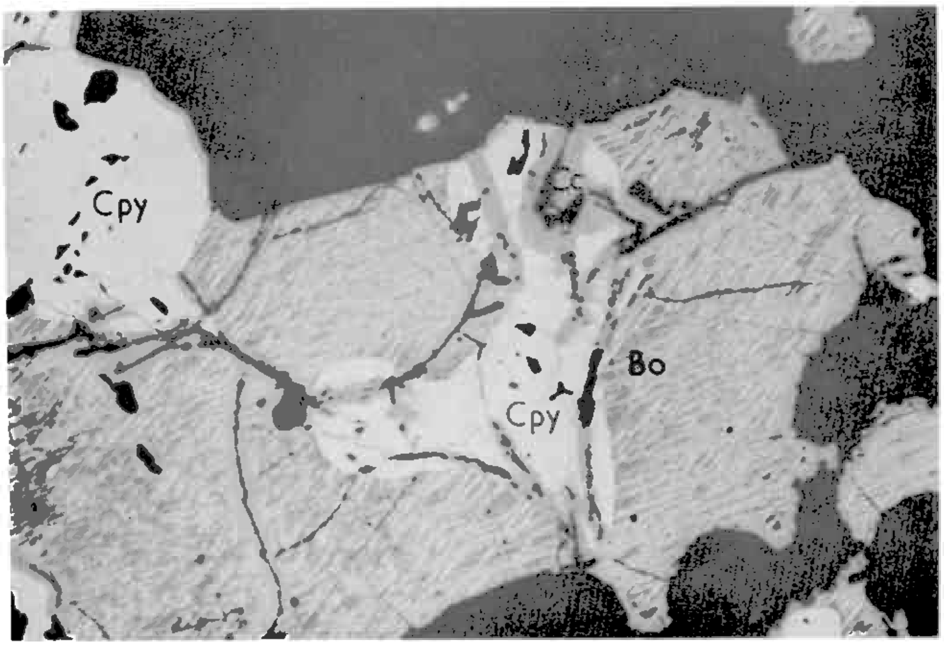
It is worthwhile pointing out at this stage that the sponsors requirements were originally based on an ore with a copper head grade of 0.8-0.9% which is fairly low and that the results that are described in this report are based on the even lower figure of 0.4% sulphide copper. It is to be expected that under normal circumstances i.e. more amenable mineralogy, that a higher grade feed would lead to easier accomplishment of higher concentrate grades and recoveries.

Additional Information

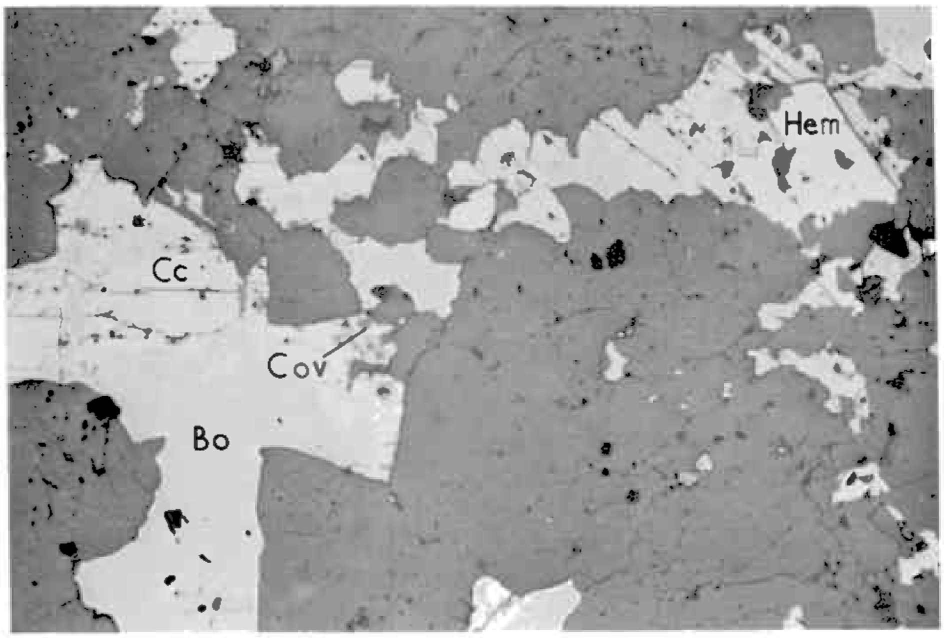
Bond Grindability test

This was done on a representative sample of the Repparfjord head ore at 390 microns.

The Work Index figure was 14.31 kwh/short ton.



Bornite (Bo) with core and ex-solution lamellae of chalcopyrite (Cpy). Blue chalcocite (Cc) rims both the bornite and chalcopyrite and has also developed along cracks. Magn: X140



Bornite (Bo) rimmed by blue chalcocite (Cc). Small patches of covellite (Cov) occur in the chalcocite. Note the hematite grain. Magn: X140

PLATE 1 - Typical occurrences of copper sulphide minerals.

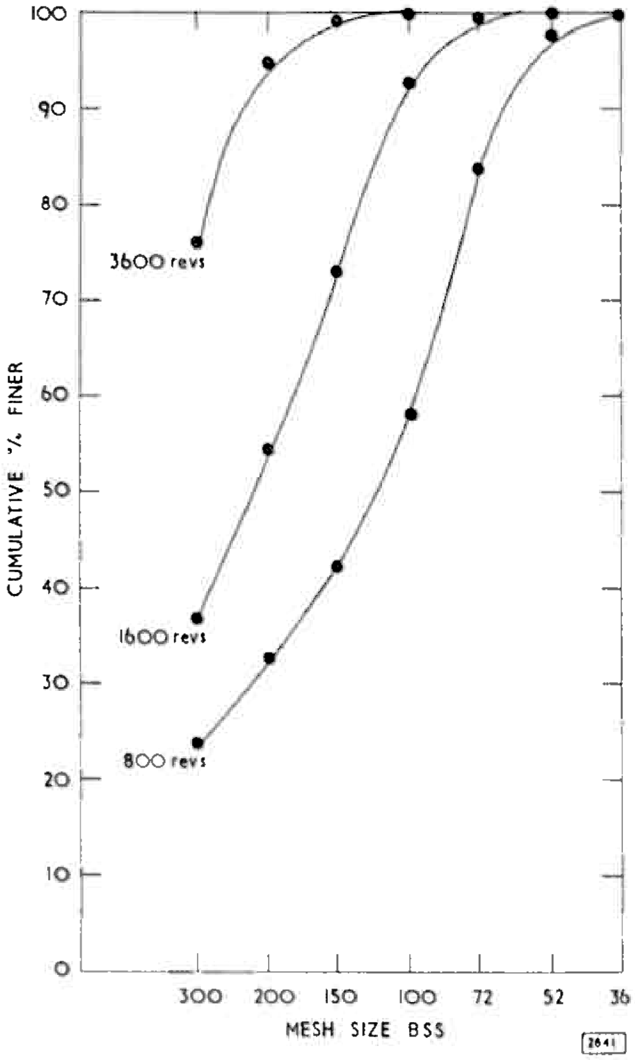


FIG.1 SIZE ANALYSES OF GROUND HEAD ORE FOR DIFFERENT MILL REVOLUTIONS

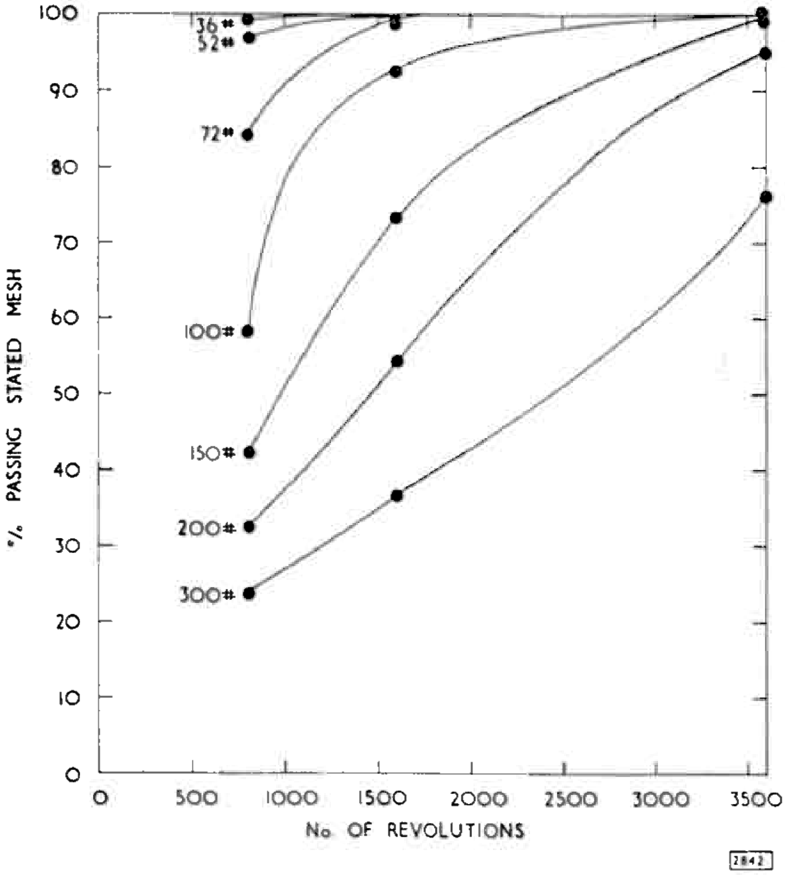


FIG.2 PRODUCT SIZE vs No. OF REVOLUTIONS

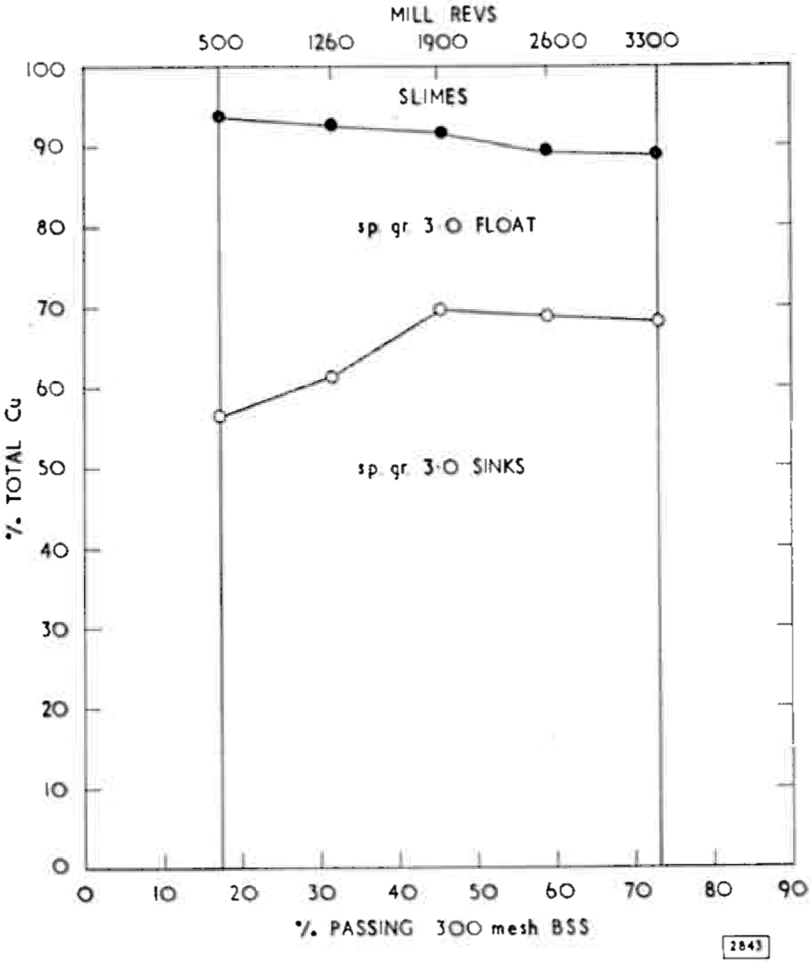


FIG.3 REPPARFJORD ORE LOCK TEST CHART

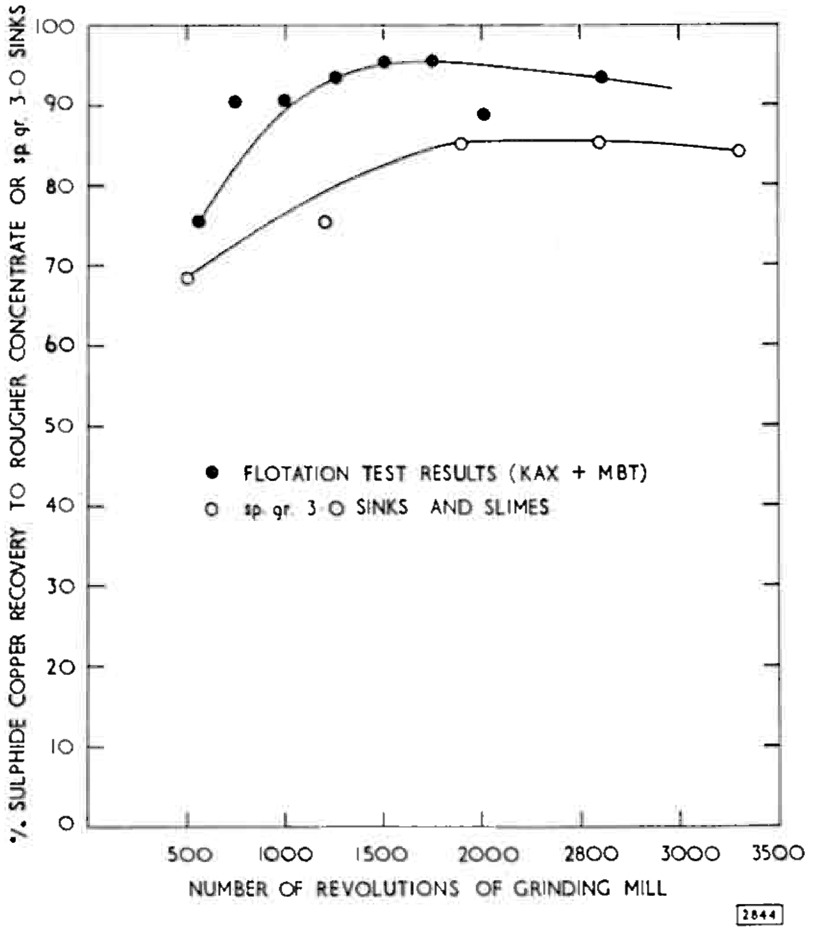


FIG.4 CORRELATION BETWEEN FLOTATION AND HEAVY LIQUID SEPARATION

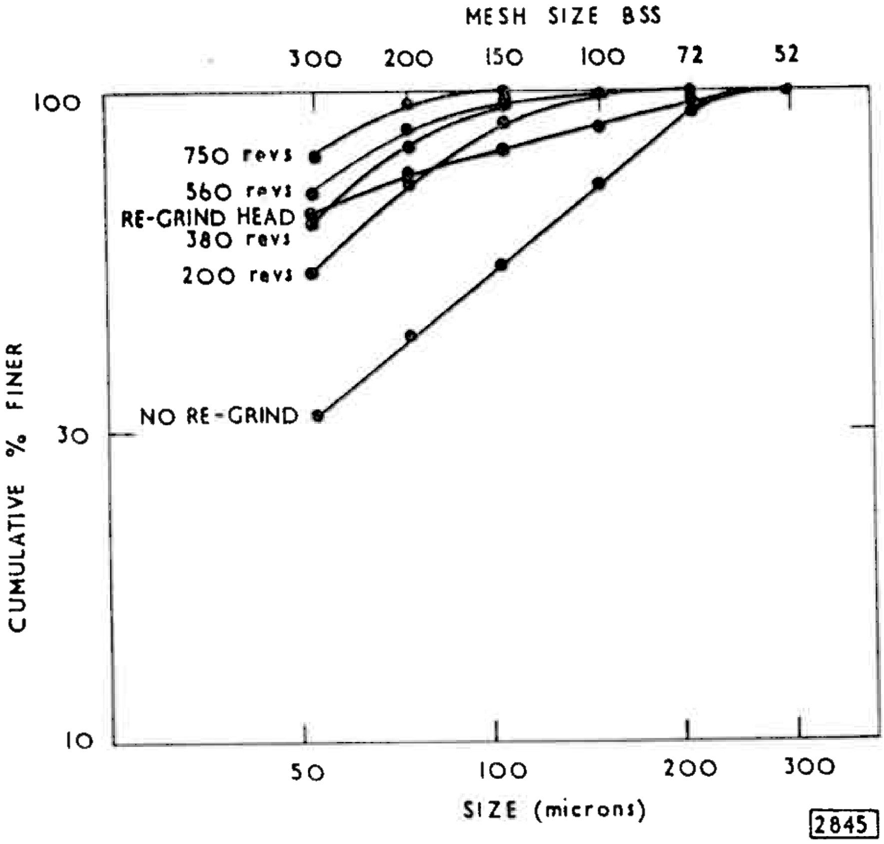


FIG. 5 RE-GRIND CONCENTRATE SIZE ANALYSIS

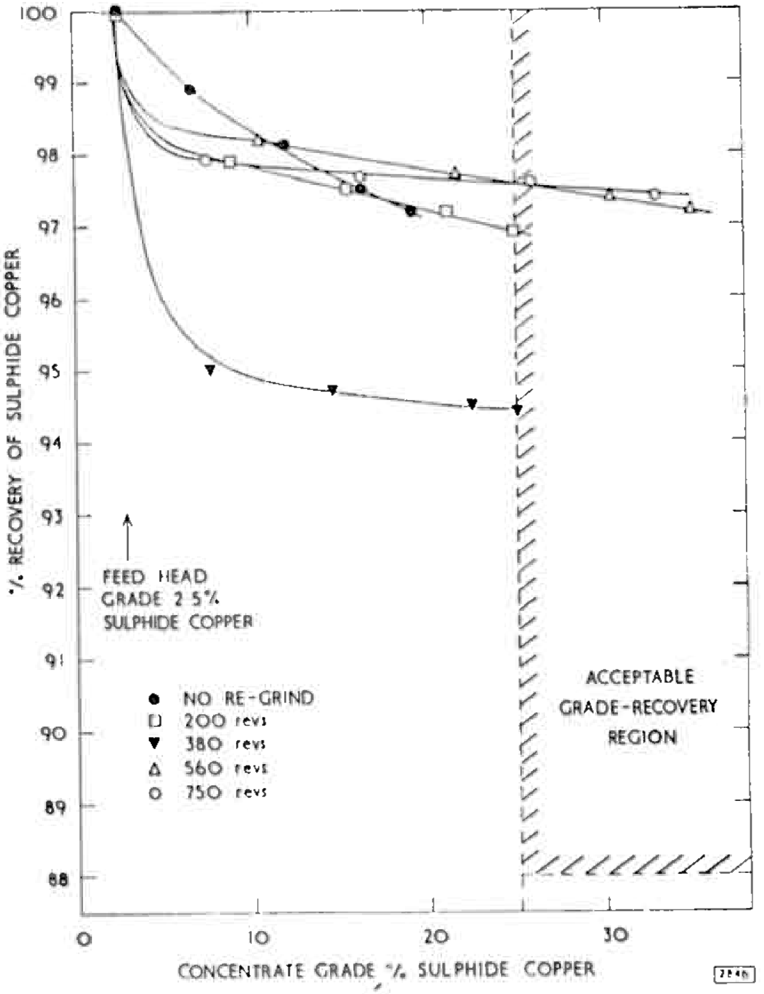


FIG.6 GRADE-RECOVERY FOR SULPHIDE COPPER IN RE-GRIND AND RE-FLOAT TESTS 36-40

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