APPENDIX-5 MINERAL DRESSING TEST ON ORE FROM KILEMBE COPPER MINE.

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1. Introduction

A mineral processing study was carried out to comprehend the actual milling process operation at the Kilembe concentrator and the processing characteristics of the ore from the Kilembe Mine. The writer believes that the results of this study would contribute to the improvement in the milling operation at the Kilembe Mine.

The study comprises analyses of size distribution and separation results on the mill products, a laboratory metallurgical test including microscopic observations of ore and product samples, and a floatation test.

Unfortunately, it was impossible to performe a study on the actual milling results such in detail as intended in advance, because the mill operation was suspended at the time of the observation.

However, these tests on the samples collected at the Kilembe Mine have provided a great deal of information in regard to the nature and the processing characteristics of the ore from the Kilembe Mine.

This study was executed at the Niihama Research Laboratory, Besshi Division of Sumitomo Metal Mining Co. Ltd., Japan.

2. Summary

The results of the study are summarized as follows;

(1) Sampling at all stages of the milling process was not carried out due to the suspension of the milling operation at the time of the observation. However, a great deal of information was obtained through the metallurgical and mineralogical tests on the samples of the ore and the mill products collected at the Kilembe Mine.

(2) A chemical and a screening analyses were conducted for the products such as the mill feed, copper concentrates, pyrite concentrates. and tailings. The results of the chemical analysis were close to those done at the Kilembe Concentrator.

(3) The cobalt/py-sulphur ratio in the ore was approximately 3×10^{-2} with a few exceptions.

(4) Sulphide minerals contained in the ore samples were pyrite, chalcopyrite and pyrrhotite in descending order of the amount,with a minor amount of sphalerite and lesser amounts of chalcocite and a cobalt-mineral.

(5) A chalcopyrite-pyrite ratio was calculated at one to two on the basis of the assay result of the mill feed sample. Pyrrhotite content was one-tenth of pyrite.

(6) A microscopic observation indicated that suplhide minerals were relatively coarse in grain size, and that pyrite and chalcopyrite were associated with each other in an appreciably simple manner. Accordingly, these two minerals seem to be easily separable in the flotation. Association of sulphides with gangue minerals was generally simple in its manner, though it was observed in parts that chalcopyrite, as fine as around 10 μ in grain size, occurred dispersedly in association with gangues.

(7) An EPMA (Electron Probe Micro-Analyzer) analysis indicated that the cobalt-mineral contained an appreciable amount of nickel and was more likely to be siegenite in chemical composition than linnaeite. Cobalt content in pyrite was variable ranging between 0.7 to 7% from point to point analized.

Most of cobalt is presumeably contained in pyrite, and only a minor amount in the Ni-Co mineral.

(8) The work index of the ore was measured at 15.4 KWH per metric ton.

(9) The results of the bulk flotation test on a lump ore sample are summerized as follows;

(9-a) The required bulk flotation time was no more than 15 minutes.

(9-b) The suitable grinding size for the bulk flotation was minus 200 mesh 50%. However, other flotation conditions and the grinding cost should be also taken into account.

(9-c) Pyrite was statisfactorily depressed in the bulk flotation by adding more than 2kg per ton of lime, while the copper content in the tailings was uninfluenced.

(10) Successive bulk-differential flotation tests were carried out on a mill feed sample. The results were as follows;

(10-a) Copper content in tailings of the bulk flotation was uninfluenced by adding more than 2kg per ton of lime as in the case (9-c).

The result, as well as that of the case (9-c), indicates that chalcopyrite can be easily separated from pyrite.

(10-b) In the differential floation, copper content in the copper concentrates increased easily up to approximately 30% by twice cleanings.

(10-c) The results of the bulk-differential flotation were uninfluenced by variable amounts of lime exceeding 2kg per ton.

Copper concentrates of 28.5% Cu were obtained with a copper recovery of 93.3%.

(10-d) The copper content in the pyrite concentrates increased by increasing amount of added lime, while the copper distribution was unchanged.

(10-e) 70% of copper in the pyrite concentrates was in the state of midlings with gangues.

(10-f) It will be desirable for performing satisfactory copper recovery and concentrate grade in the differential flotation to add a certain amount of lime in the bulk floatation.

However, addition of lime in the bulk flotation is unrecommended for maximizing the recovery of pyrite with an intention to recover cobalt as much as possible.

(10-g) A series of the products of the flotation test were analized for cobalt and sulpher.

No increase or decrease in cobalt/py-sulphur ratio was reconized in any products.

(10-h) The result of a screening test of the tailings indicated that copper content was higher in the coarser fractions than in the finer, and that copper minerals in the coarser fractions were mostly in the state of middlings with gangue.

It appears difficult to separate these meddlings by grinding then in more or less fine sizes. Therefore, a scheme to float the middlings should be innovated to reduce the copper content in the tailing to a lower level than that performed at the present concentrator.

(11) Similar tests as the above were carried out on a lump ore sample. As the result, copper concentrates of 28.5% Cu were obtain with a copper recovery of 95%. These figures appear to be a little better than those obtained in the above tests.

(12) A preliminary test was conducted to examine if the copper content in the tailings would be reduced by re-feeding the reground bulk middlings into the bulk flotators. However, the result indicated that the copper content in the tailing was a very little influenced by this process.

(13) A bulk flotation test by using kerosene was carried out with an intention to improve the metallurgical results by recovering copper in the state of middlings in the tailings.

The results were as follows;

(13-a) The copper content in the tailings decreased to approximately 0.075% and was substantially improved in comparison with that performed in the present process (0.11%).

(13-b) A screening test of the tailings indicated that the copper contents in the coarser fractions were much smaller than those obtain by the present process.

(13-c) Copper concentrates of 28.5% Cu were obtained with a copper recovery of 94.3% which was improved by 1% in comparison with that performed by the present process.

(13-d) Increasing amounts of collector and flotation time may lower the copper content in the tailings

Effect of regrinding the bulk middlings should be confirmed, before the procedures tested in this study is actually applied in the milling operation.

3. Sample

Upon recieving the following samples (Table-1) at the Niihama Research Laboratory on March 13th, 1978, a mineral processing test was carried out on these samples which had been collected by the Japanese Survey Team for the Development of the Kilembe Mine during the site examination.

Table-1 List of Samples

Type of Samples	Sample Location	Nos.of Samples	Weight (kg)
Copper Concentrates	Kasese Filter Plant	1	2.6
Pyrite Concentrates-A*[1]	Kasese Pyrite Dam	1	2.1
Pyrite Concentrates-B*[2]	Kasese Pyrite Dam	1	2.2
Tailings*[3]	Tailing Dam E	1	1.8
Tailing Sand*[4]	Tailing Dam E	1	1.0
Mill Feed	Fine Ore Silo	1	11.1
Lump Ore	Stock Pile	1	73.7
Lump Ore	Underground Stopes	6	8.1
Total		13	102.6

Note

*[1] Taken from a part of pyrite concentrates which appears to have been deposited in reatively recent years.

*[2] Taken from a part of pyrite concentrates which appears to have been deposited a few years ago.

*[3] Taken in the tailing dam about 15m away from the outlet of the tailing pipe.

*[4] Classified sand for embankment.

Table 2 Spectrum Analysis of Kilembe Concentrator Products

Sample	Mill Feed	Copper Concentrates	Pyrite Concentrates	Tailings
Cr	+5	(±)	+5	+5
Ca	+2	±	+3	+2
Ti	+	(±)	±	+2
Zn	+	+5	±	±
Na	+5	+5	+5	+5
Ag	+	+3	+	(±)
Cu	+5	+5	+5	+5
V	+3	(±)	±	+3
Mo	+	(±)	±	±
Al	+3	+	+3	+3
Bi	(±)	(±)	±	(±)
Ni	+5	+3	+5	+5
Co	+3	+2	+3	+3
Fe	+5	+5	+5	+5
Si	+5	+5	+5	+5
Mg	+5	+5	+5	+5
Sn	±	±	(±)	±
Pb	+	+	±	+
Mn	+5	+2	+3	+5
B	+3	±	+	+

The following elements are – .

W, Cd, Ge, Pt, Sb, Au, Se, Te, As, Rh, Ir, Ru, Pd, Be, Ba

Table 3 Chemical Analysis of Kilembe Concentrator Products

Component \ Sample	Mill Feed	Copper Concentrate	Pyrite Concentrate		Tailing
			A	B	
Cu	1.87	26.35	0.35	0.10	0.28
Sol. Cu *5)	0.01	1.22	-	-	0.01
S	7.66	32.10	41.55	41.93	4.40
Fe	10.95	28.26	36.44	35.48	8.20
Insol.	63.83	4.88	12.81	11.64	71.90
Co	0.21	0.18	1.27	1.30	0.14
Ni	-	0.04	0.14	-	
Mo	-	0.01	-	-	-
Pb	-	0.01	0.01	-	-
Zn	0.012	0.12	0.004	-	-
As	-	0.001	0.01	-	-
Sb	-	0.01	-	-	-
Bi	-	0.01	-	-	-
Cd	-	0.0005	-	-	-
Hg	-	0.0001	-	-	-
Sn	-	0.01	-	-	-
Se	-	0.01	0.01	-	-
Te	-	0.01	-	-	-
F	-	0.01	-	-	-
Mn	-	-	0.02	-	-
Water Sol. Cl	-	0.05	-	-	-
SiO_2	42.9	3.42	-	-	-
Al_2O_3	11.5	0.91	-	-	-
CaO	5.81	0.71	-	-	-
MgO	5.19	0.43	-	-	-
Na_2O	3.47	-	-	-	-
K_2O	0.96	-	-	-	-
Au g/T	0.2	2.2	0.2	-	0.1
Ag g/T	2	19	2	-	1
Ig. Loss *6)	1.22	-	-	-	-
Total	90.31	92.54	92.59	90.45	84.92

(Note)

*5 Sol. Cu ; Soluble copper dissolved when 100 ml of 5 wt. pct H_2SO_4 was added to 1 gram of sample and agitated for 1 hour.

*6 Ig. Loss: Ignition loss when heated at 600°C for 1 hour.

Table 4 Screen Analysis of Copper Concentrate

Size (micron)	Wt %		Assay %		Distribution %	
		Passing	Cu	S	Cu	S
105	2.4	97.6	17.47	25.72	1.5	1.8
74	5.9	91.7	23.10	30.30	4.9	5.3
44	24.9	66.8	26.35	34.40	23.6	25.5
20	30.8	36.0	28.18	35.16	31.2	32.2
-20	36.0		29.95	32.95	38.8	35.2
Total	100.0		27.80	33.66	100.0	100.0

Table 5 Screen Analysis of Pyrite Concentrate

Size (micron)	Wt %		Assay %			Distribution %		
		Passing	Cu	S	Co	Cu	S	Co
105	2.5	97.5	0.67	10.96	0.34	8.6	0.7	0.6
74	5.5	92.0	0.58	31.20	1.03	16.4	4.1	4.0
44	34.3	57.7	0.21	44.98	1.53	37.1	36.7	37.4
20	38.5	19.2	0.076	46.27	1.52	15.1	42.4	41.6
-20	19.2		0.23	35.31	1.20	22.8	16.1	16.4
Total	100.0		0.19	42.01	1.41	100.0	100.0	100.0

Table 6 Screen Analysis of Tailing

Size (micron)	Wt %		Assay %		Distribution %	
		Passing	Cu	S	Cu	S
297	7.5	92.5	0.33	0.70	10.7	1.3
210	16.9	75.6	0.27	0.78	19.8	3.1
149	19.0	56.6	0.24	1.39	19.7	6.3
105	18.4	38.2	0.21	3.40	16.7	14.9
74	12.0	26.2	0.20	8.10	10.4	23.1
44	12.3	13.9	0.20	9.74	10.7	28.5
-44	13.9		0.20	6.90	12.0	22.8
Total	100.0		0.23	4.20	100.0	100.0

4. Screening and Chemical Analyses of the Samples

4-1 Copper Concentrates

The results of the spectrum and chemical analyses on the copper concentrate sample are shown in Table 2 and 3, and those of the screening analyses, in Table 4 with assay results of the screened products.

4-2 Pyrite Concentrate

The results of the spectrum, chemical and screening analyses on the pyrite concentrate samples are given in Table 2, 3 and 5 respectvely.

The cobalt contents in two types of the pyrite concentrate samples were assayed at 1.27 and 1.30% as shown in Table-3, which were within the range of previous assay values of pyrite concentrates from 1.0 to 1.6% obtained at the Kilembe and other laboratories.

As shown in Table 5, the cobalt content of a pyrite concentrate sample is estimated at 1.41% on the basis of the assay results of its screened products. This value is a little higher than the above assay results. The discrepancy is presumably caused by disolution of sulfate in the sample in the course of the wet screening analysis which was applied for this study. The weight of the pyrite concentrate sample-A decreased by 7% as a result of the wet screening analysis.

The copper content in the pyrite concentrate sample-B was 0.10%, much lower than 0.35% Co in the sample-A which was apparently fresher than the sample B. Copper may have been oxidized and leached in some amount, while the pyrite concentrates have been deposited in an open-air for years.

4-3 Tailings and Tailing Sand

The results of the spectrum, chemical and screening analyses on the tailing sample are given in Table-2,3 and 6 respectively.

The result of the screening analysis indicated that the minus 200 mesh fraction occupied 26% of the total amount. This tailing size is apparently coarser than that of the bulk flotation recorded at the concentrator. The tailing sample may have been collected at the location where coarser fractions of tailings were deposited.

The result of the screening analysis of the tailing sand sample is shown in Table 7. The coarse fraction of tailings, after classified by cyclones, is being used for stope-filling in the underground. Size distribution of the stope-filling sand may be close to this result.

Table 7 Screening of Tailing Sand

Size (micron)	Weight (%)	Passing (%)
+297	9.9	90.1
+210	19.6	70.5
+149	24.1	46.4
+105	20.5	25.9
+ 74	10.2	15.7
+ 44	8.1	7.6
- 44	7.6	
Total	100.0	

4-4 Mill Feed and Lump Ore

The mill feed and lump ore samples were used mainly for the flotation test.

The mill feed samples, which had been crushed to the size less than 3/4 inch, was further crushed to the size minus 10 mesh for the flotation test.

The lump ore sample was devided by hand-picking into four groups, each of which was crushed to the size minus 10 mesh and assayed for Cu, S and Co. The assay results are given in Table 8.

Table8 Assay Result of Lump Ore (Hand Sorted)

Ore	Weight Kg	Assay % Cu	S	Co	Estimated Co/Py.S Ratio*7
High Grade	10.0	9.20	25.27	0.58	3.6×10^{-2}
Middle Grade	17.4	1.92	14.15	0.35	2.9×10^{-2}
Low Grade	23.4	1.45	5.10	0.09	2.5×10^{-2}
Poor Gangue	22.9	0.36	1.22	0.02	2.3×10^{-2}
Total	73.7	2.27	8.77	0.20	3.1×10^{-2}

*7 Estimated according to the following formula Co/Py.S Ratio = Co/ (S - 1.01 Cu)

A composite sample was prepared for the flotation test by blending the four groups of the samples so that it contained 1.8% Cu and 7.6% S. The composite sample is here-in-after termed the lump ore sample.

The assay results of the mill feed sample are given in Table-2 and 3.

4-5 Underground Ore Sample

The assay results of the six ore samples collected in the underground are shown in Table 9.

The cobalt/pyrite sulfur ratios, as shown in Table 8 and 9, were around 3×10^{-2} with an exception.

Table-9 Assay Result of Underground Samples

Sample No.	Sample Location	Assay (%) Cu	S	Co	Ni	Estimated Co/PyS Ratio
K02201	Lower Bukangama 4900L 21 x C	3.37	27.10	0.70	0.060	3.0×10^{-2}
K02202	"	1.12	7.15	0.14	0.025	2.3×10^{-2}
K02203	"	5.42	21.00	0.69	0.12	4.4×10^{-2}
K02204	"	1.85	19.94	0.46	0.082	2.5×10^{-2}
K02205	"	0.94	4.87	0.10	0.024	2.6×10^{-2}
K02206	Upper Bukangama 5200L 24 x C		3.83	0.011	0.017	0.3×10^{-2}

5. Microscopic Observation and EPMA Analysis

The lump ore and mill product samples were observed under the microscope and analized by the EPMA.

Pyrite is a major sulphide mineral in the ore samples, while chalcopyrite and pyrrhotite are subordinate.

Pyrrhotite content, varying greatly from place to place even in a single polished section, is one-tenth of pyrite or less in general.

The ratio of pyrite (including pyrrhotite) to chalcopyrite is estimated at approximately 2 to 1 on the basis of the assay results of the mill feed sample.

The ratio appears to be rather low for the ore deposits of the strata-bound sulphide deposit category to which the ore deposits of the Kilembe Mine belong.

The major copper mineral is chalcopyrite, although a very minor amount of chalcocite is rarely observed under the microscope. Such oxidized copper minerals as malachite and chrysocolla occur in parts of the ore deposits mear the surface.

Other than the above described, a small amount of sphalerite are observed as well as a minor amount of a cobalt-mineral.

Grain sizes of the sulphide minerals, ranging widely from more than 10 mm to the order of a few microns, are generally coarse.

Photo-1 shows a typical occurrance of coarse sluphide minerals. Association of pyrite and chalcopyrite is simple and loose as shown in the micro-photograph.

Photo-2 and 3 show a typical association of pyrite and chalcopyrite, which occur discretely or associated with each other in a easily separable state.

It is rarely observed that chalcopyrite grains, as fine as less than 10 microns in grain size are dispersed in a coarse pyrite grain, although none of these micro-photographs exhibits such an occurrance.

Photo-4 shows a relatively complex occurrance of sulphide minerals.

Fine chalcopyrite grains of approximately 10 microns in grain size are occasionally observed dispersedly in gangue minerals as shown in Photo-5.

Occurrances of pyrrhotite and a cobalt-mineral are shown in Photo-6 and 7. The cobalt mineral is closely associated with either pyrrhotite or pyrite.

Photo-8 is a micro-photograph of pyrite concentrates (a flotation test product), in which association of pyrrhotite and cobalt is observed.

Photo-9 is a micro-photograph of copper concentrate (a mill product sample), in which a very small amount of chalcocite is observed.

Photo-10 and 11 are EPMA scanning images for the same area as shown in Photo-7 and 8.

The results of the EPMA point analyses on the cobalt-mineral in Photo-7 and 8 are given in Table-10.

Table-10 Point Analysis on Linnaeite Partieles

Point No.	Origin of Particle	Assay (%) S	Fe	Co	Ni	Total
1	Lump Ore	37.7	11.3	17.8	25.6	92.4
2	Pyrite Conc.	32.8	2.8	34.4	14.4	84.4
3	"	32.0	6.8	25.2	18.5	82.5
4	"	34.0	2.9	31.1	20.6	88.6
5	"	35.9	2.1	37.8	16.4	92.2
6	"	36.7	2.1	35.5	18.4	92.7
7	"	32.7	2.7	31.4	19.2	86.0
8	"	35.2	1.8	34.9	19.1	91.0

The cobalt-mineral contains a substantial amount of nickel and is more likely to be siegenite in chemical composition than linaeite.

The results of the EPMA analyses on pyrite in the pyrite concentrate samples are shown in Table 11.

Cobalt content in pyrite varies in great deal between 0.9 and 7% from point to point. However, each value does not represent a whole grain of pyrite, because each point was analized by a point beam of 2 microns in diameter. Judging from the scanning images, average cobalt contents in pyrite grains appear to be less fluctuated than indivisual point values given in Table-11.

The amount of the cobalt mineral is apparently too small to explain the total cobalt content in pyrite concentrates (1.0 to 1.6% Co). Most of cobalt is presumably contained in pyrite.

Table-11 Point Analysis on Pyrite Particles

Point No.	Origin of Particle	Assay (%) S	Fe	Co	Ni	Total
1	Pyrite Conc.	49.3	35.9	7.1	0.1	92.3
2	"	50.8	41.8	3.2	0.1	95.8
3	"	51.6	43.8	2.4	-	97.8
4	"	50.7	44.2	1.8	-	96.7
5	"	50.8	41.7	4.3	-	96.8
6	"	51.5	45.6	0.9	-	98.0
7	"	51.2	41.9	4.9	-	98.0
8	"	50.3	44.9	1.6	-	96.8
9	"	49.4	44.0	1.5	-	94.9
10	"	49.6	42.8	1.5	-	93.9
11	"	52.4	44.5	1.9	-	98.8
12	"	49.7	42.5	2.9	-	95.1

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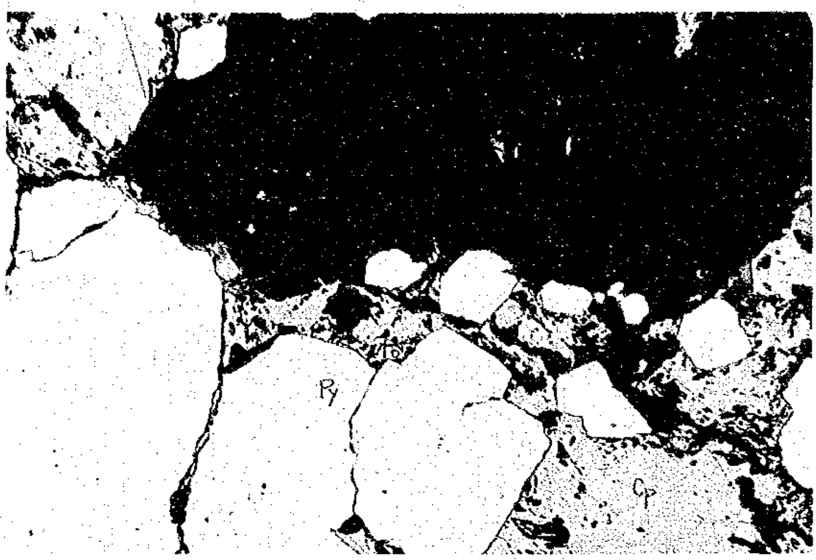
LEGEND FOR MICRO-PHOTOGRAPHS

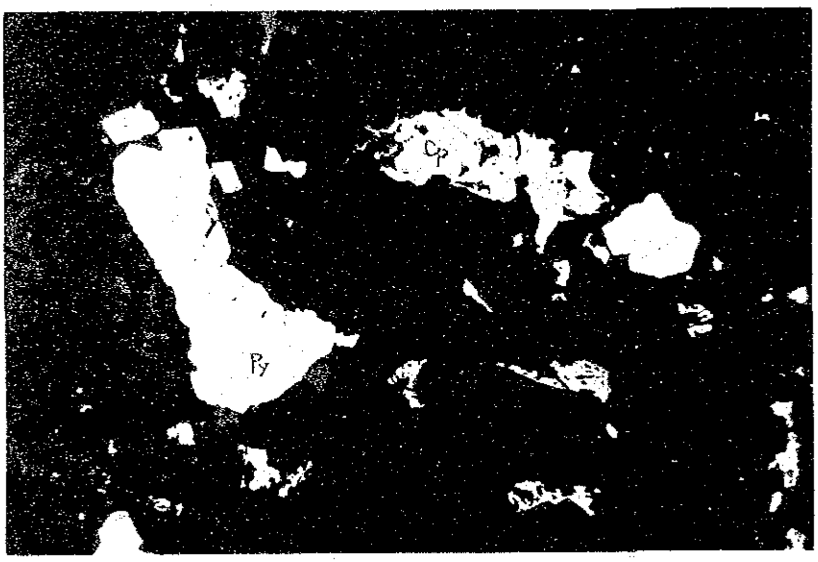
Symbol	Color	Name of Mineral
Cp	Thick Yellow	Chalcopyrite
Py	Pale Yellow	Pyrite
Po	Pale Pink	Pyrrhotite
Zn	Pale Blue	Sphalerite
Co	Blue	Chalcocite
Sg	Creamy White	Siegenite
Mg	Grayish Blue	Magnetite
G	Grayish Black Grayish Green	Gangue

Photo 1　　Lump Ore



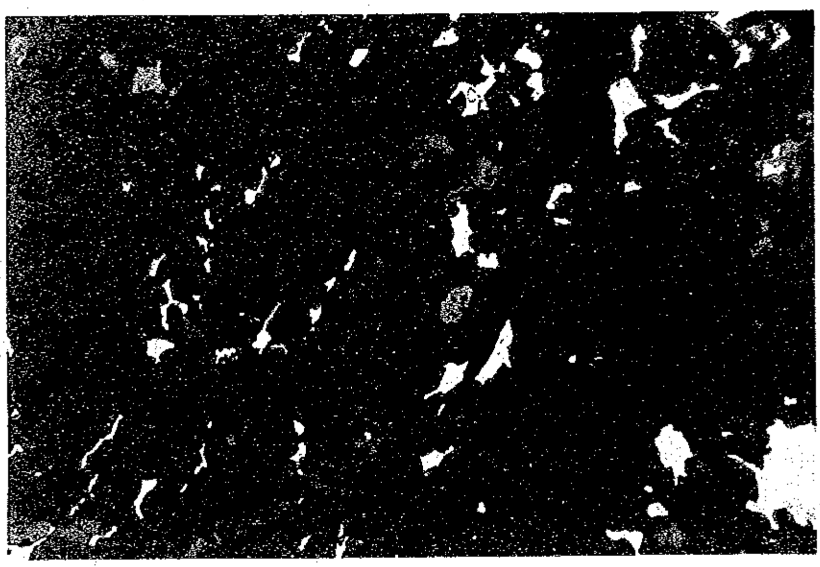
X 50

Photo 2　　Lump Ore



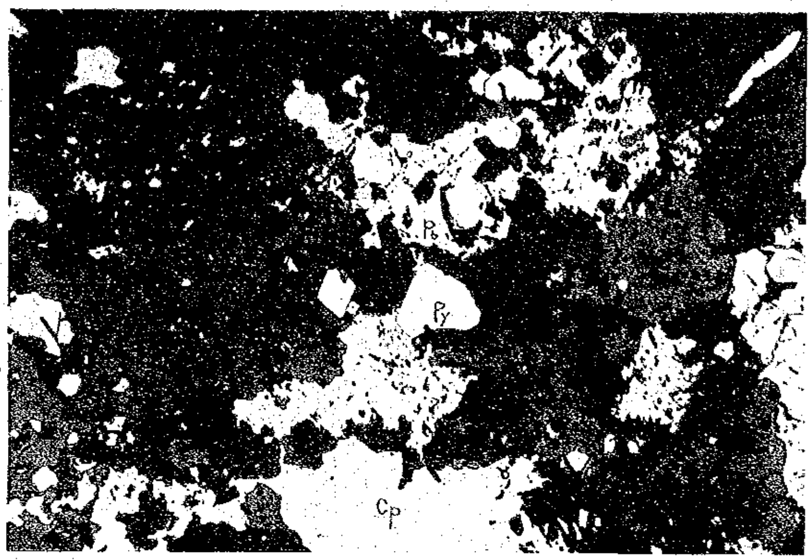
X 50

Photo 3　　Lump Ore



X 50

Photo 4 Lump Ore



X 50

Photo 5 Lump Ore

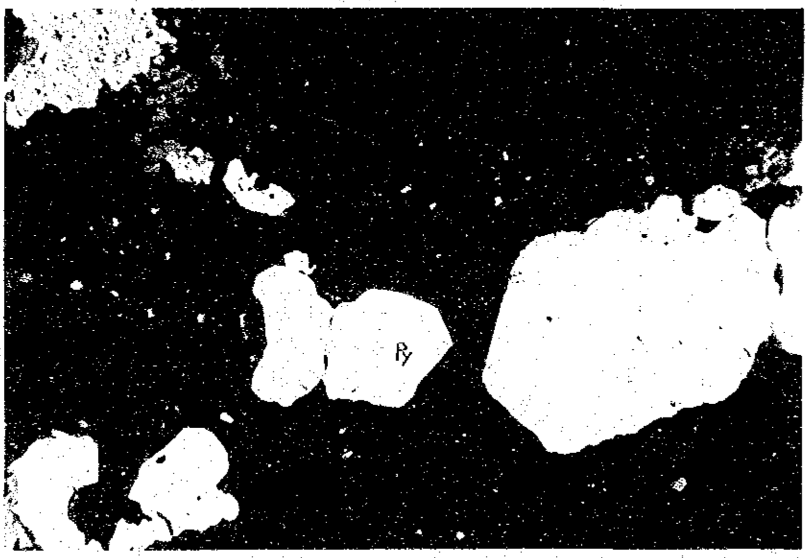


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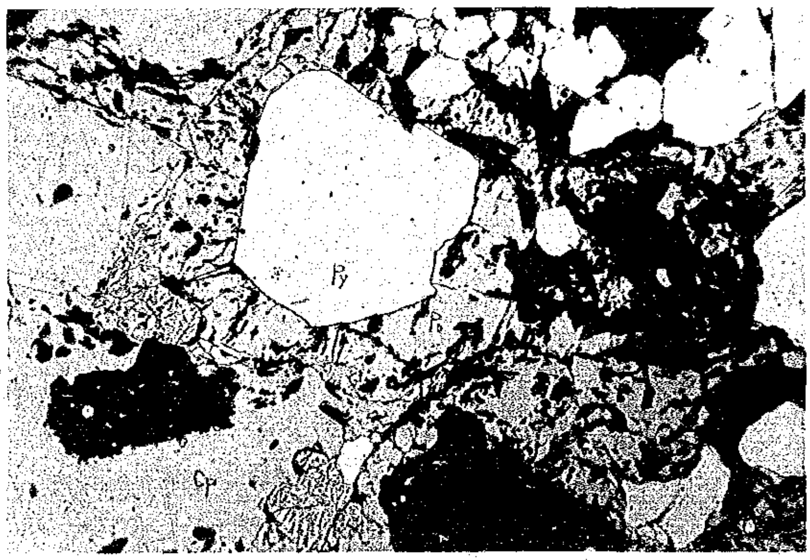
Photo 6 Lump Ore



X 50

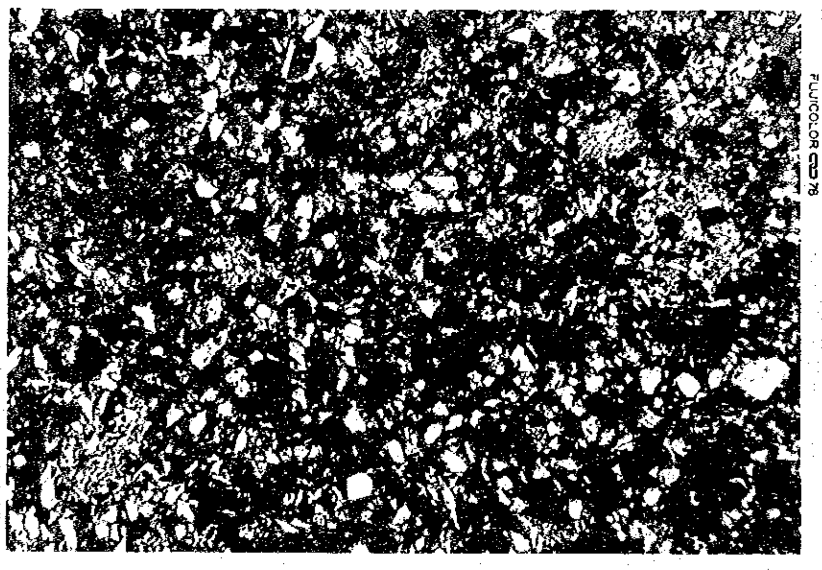
Photo 7 Lump Ore

x 110

Photo 8 Pyrite Concentrate

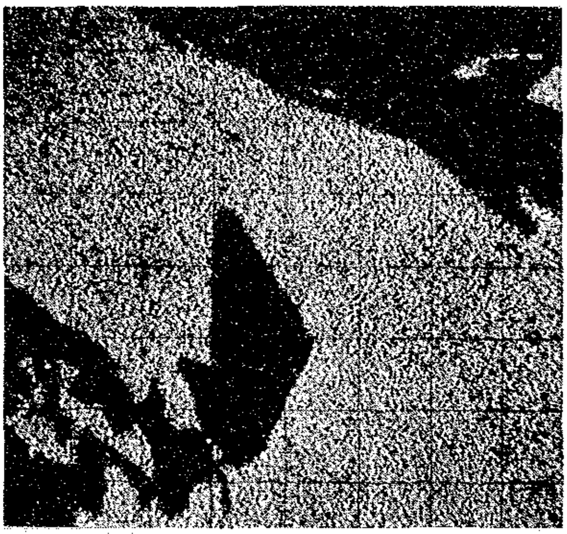
x 360

Photo 9 Copper Cocentrate



x 50

Photo 10 Scanning Emage on the Field of Photo 7
—— (1)



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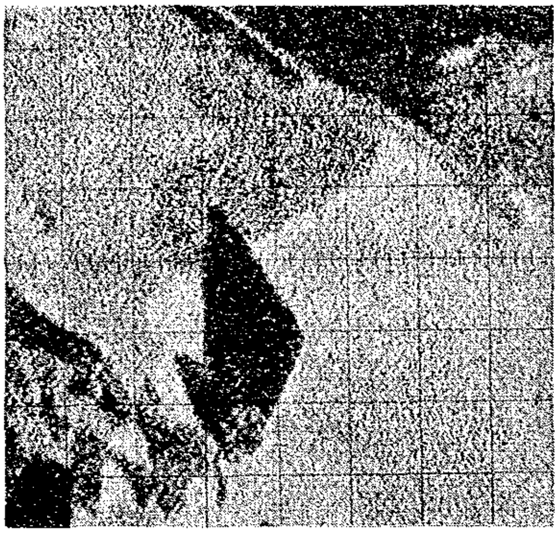
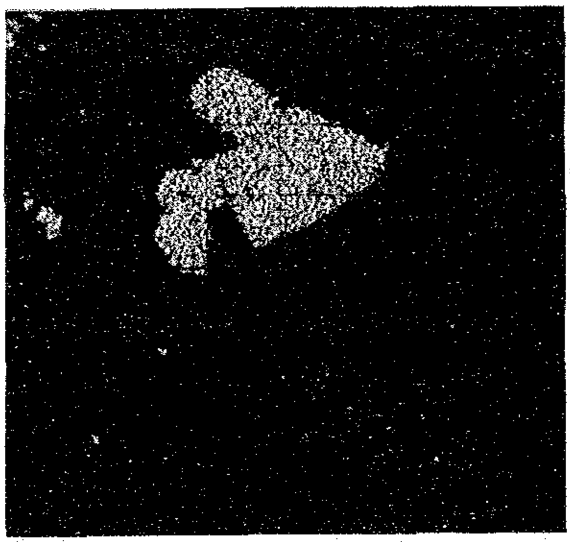
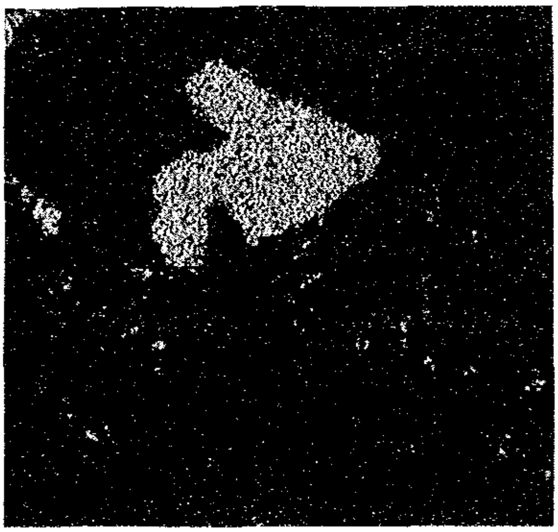


Fe



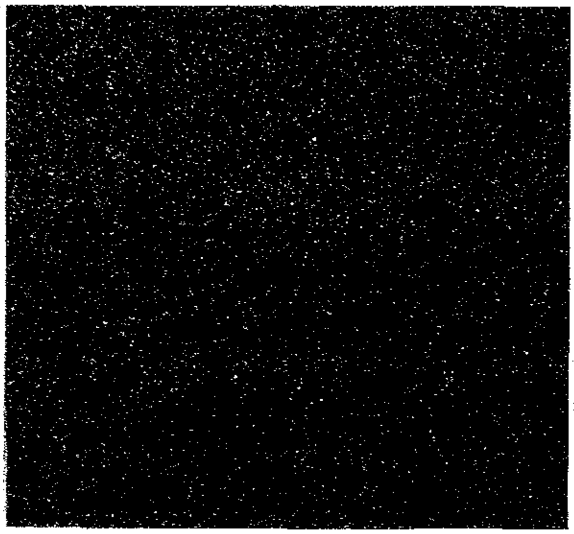
Co



Ni

Photo 10 ——— (2)

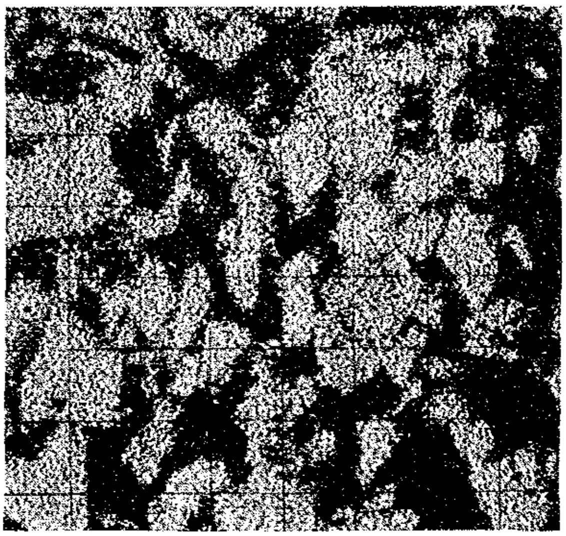
Cu



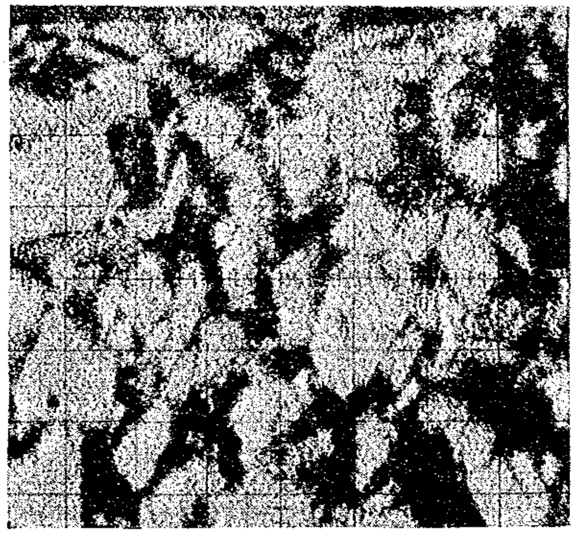
As
Pb

S.C.

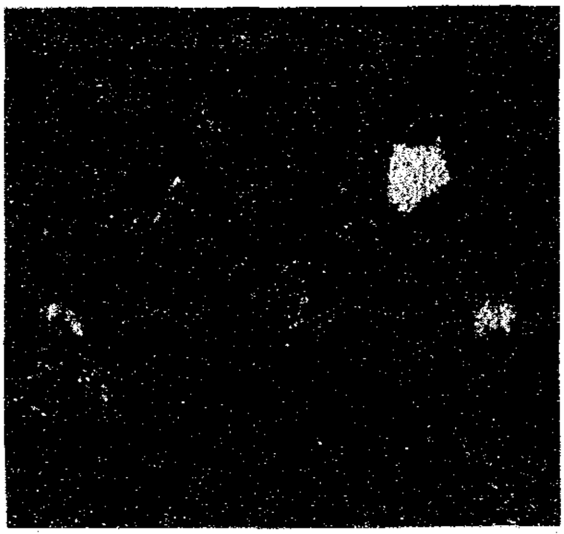
Photo 11 Scanning Emage on the Field of Photo 8 ——— (1)



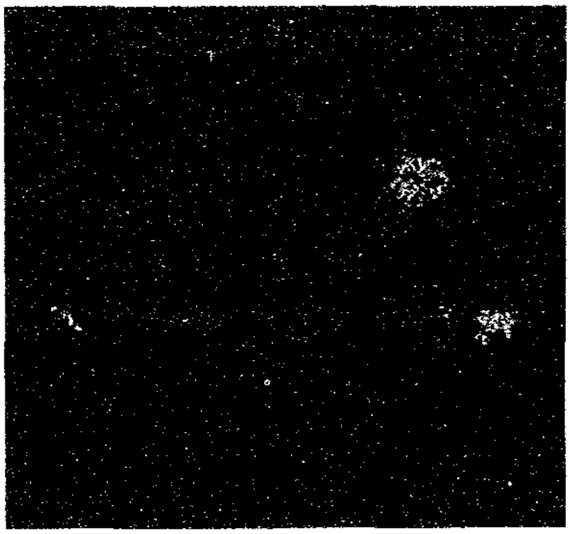
S



Fe

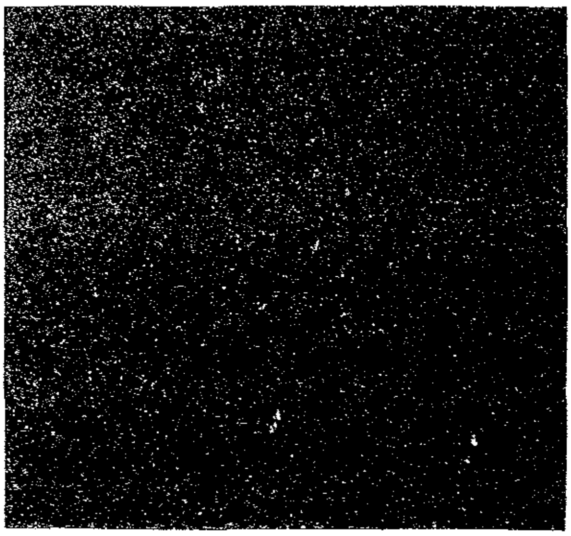


Co

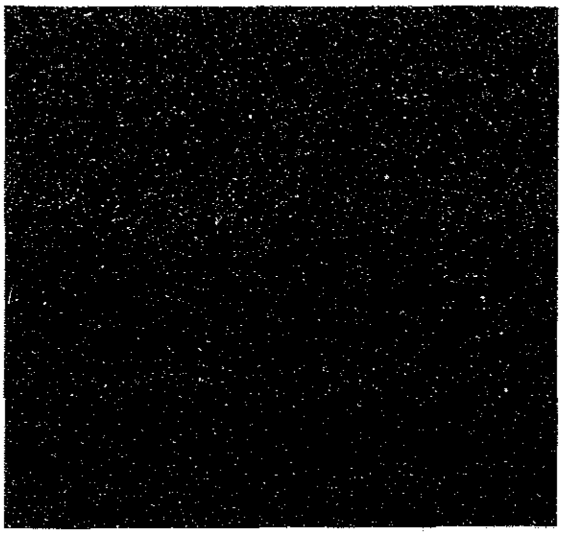


Ni

Photo 11 ———— (2)



Cu



As
Pb

S.C.

6. Metallurgical Test

6-1 Grinding Work Index

A grindability test was carried out the lump ore sample in accordance with the procedure specified in Japanese Industrial Standard M4002, 1969 (Testing Method of Grinding Work Index).

The following factors were determined or measured in this test;

F: 80% passing size (μ) of feed

P_1: Opening (μ) of the test sieve used
P_1 = 210 μ in this test

P: 80% passing size (μ) of the undersize ground product screened by the test sieve (P_1) when stabilized.

w: Weight (g) of 700 ml of feed after packing
w = 1,425g in this test

a: Quantity(g) of the oversize ground product screened by the test sieve(P_1) for each run.

U: Ratio of the undersize feed screened by the test sieve(P_1) to the total feed
U = 0.091 in this test

Gbp: Quantity (g) of the undersize ground product screened by the sieve (P1) per revolution of the test ball mill.

The results of the grindability test are shown in Table 12, 13 and Fig.1. The work index (Wi) is given by the following formula;

$$Wi = \frac{4.45}{P^{0.23} \times Gbp^{0.82} \times \left(\sqrt{\frac{10}{P}} - \sqrt{\frac{10}{F}}\right)} \times 1.1$$

Where: P_1 = 210,
Gbp = 1.76,
P = 162,
and F = 2,480, as the results of this test, the work index of the lump ore sample is computed at 15.4 KWH/metric ton.

Fig. 1 Size Distribution of Feed and Undersize

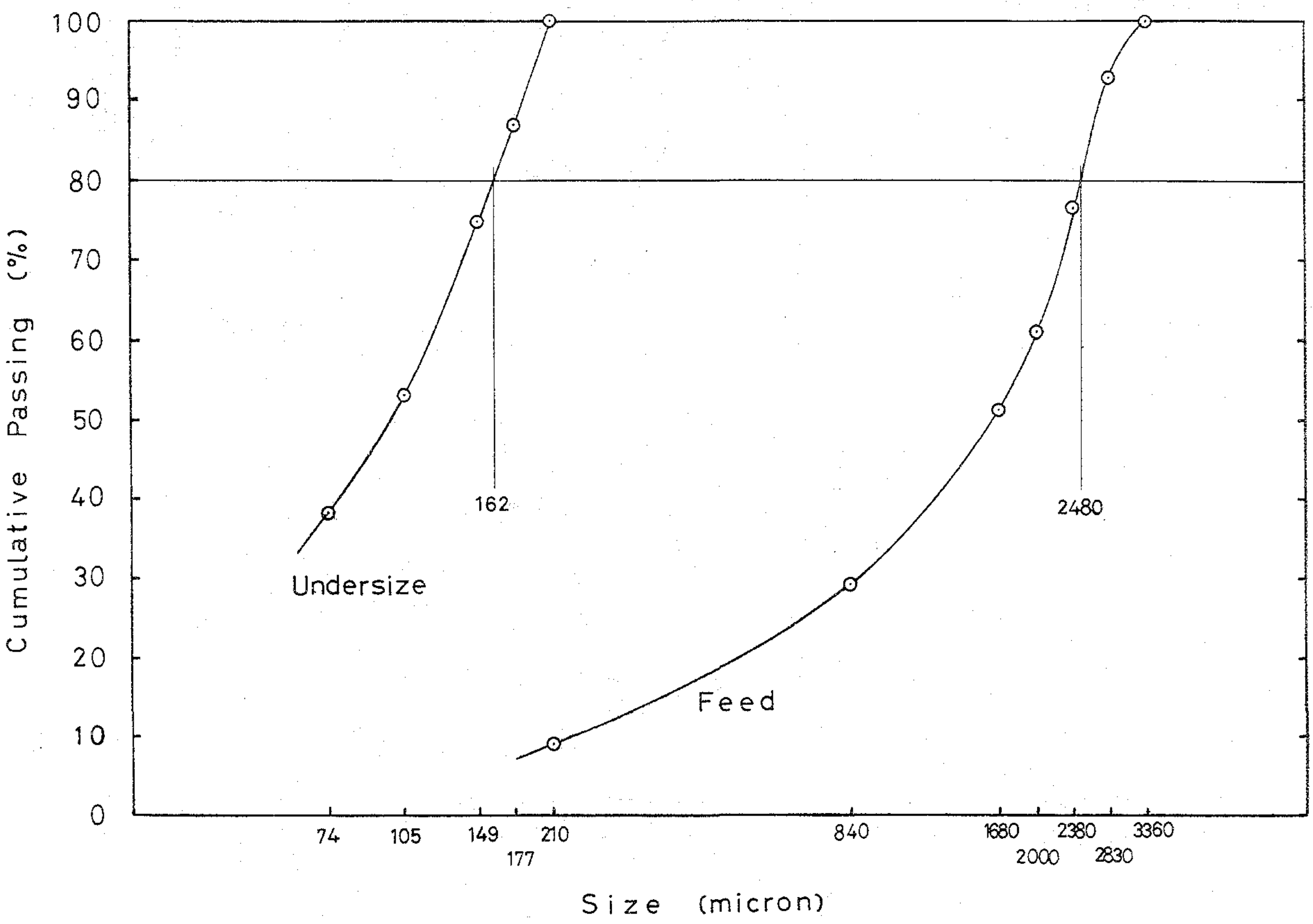


Table 12 Test Result of Grinding Work Index (1)

	(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)
Run No. {n}	Mill Revolution {N}	Product on P1 (g) {a} (measured)	New Feed (g) { W - a}	Undersize thru P1 Before Grinding (g) {(3) x U}	Produced Undersize thru P1 After Grinding (g) {(3) - (4)}	Aimed Value of (5) $\{\frac{W}{3.5} - (4)\}$	Circulation (%) $\{\frac{(2)}{(3)} \times 100\%\}$	Gbp {(5)/(1)}	Revoluation Next Run {(6)/(9)}
(1)	150	1083	342	130	212	376	317	1.41	267
(2)	267	1005	420	31	389	369	239	1.46	253
(3)	253	991	434	38	396	367	228	1.57	234
(4)	234	1003	422	40	382	369	238	1.63	226
(5)	226	1004	421	38	383	369	239	1.70	217
(6)	217	1011	414	38	376	369	244	1.73	213
(7)	213	1016	409	38	371	370	248	1.74	213
(8)	213	1008	417	37	380	369	242	1.78	207
(9)	207	1021	404	38	366		253	1.77	
Stabilized Average 7 - 9 Run							248	1.76	

Table 13 Test Result of Grinding Work Index (2)

Size (micron) JIS STD Sieve	Feed (-3360μ)		Undersize of P_1 (Stabilized 7-9 Run)	
	Weight %	Cumulative Passing %	Weight %	Cumulative Passing %
3.360		100.0		
2,830	7.3	92.7		
2,380	16.0	76.7		
2,000	15.7	61.0		
1,680	9.7	51.3		
840	22.0	29.3		
210	20.2	9.1		100.0
177	(-210) 9.1		13.1	86.9
149			12.1	74.8
105			21.6	53.2
74			15.0	38.2
-74			38.2	
Total	100.0		100.0	
80% Passing Size (μ) (*9)	2,480		162	

*9 Values read from the size distribution curve in Fig.1

Table 14 A measurement of Apparent Specific Gravity

Sample		Tailing		Tailing Sand (Deslimed)	
Specification		-200 mesh 50% +65 mesh 5% s = 4%		-200 mesh 10% +65 mesh 12%	
Specific Gravity Dt		2.93		2.91	
Volume of cake (cm^3)		(1)	(2)	(1)	(2)
		116.3	116.3	116.3	71.8
Weight of Cake (g)	Wet	217	215	191	116
	dry	176.5	178.5	162	100
moisture (%)		18.7	17.0	15.2	13.8
Apparent Specific gravity (wet) Dw		1.87	1.85	1.64	1.62
Apparent Specific gravity (dry) Dd.		1.52	1.53	1.39	1.39
		(av.) 1.53		(av.) 1.39	
Void Ratio 100%x(Dt-Dd)/Dt (%)		48		52	

6-2 Specific Gravity of Tailings

True and apparent specific gravities of tailings were measured to provide references for stope-filling in the underground. The tailings of the flotation were used for the estimation.

An apparent specific gravity was obtained by measuring volume and weight of the cake which was produced by filtering the slurry of tailings thorugh a small pan filter. The results are shown in Table 14.

6-3 Bulk Flotation Test

A ser es of bulk flotation tests (test 1-8) were carried out to examine effects of the flotation size, flotation time and added amount of lime to the metallurgical results. The lump ore sample was used for the test feed.

Eight test conditions were selected from the combinations of three levels of flotation sizes (minus 200 mesh 45, 50, 55%) and four levels of added amounts of lime (0, 1, 2, 3 kg/ton), as shown in Table-15.

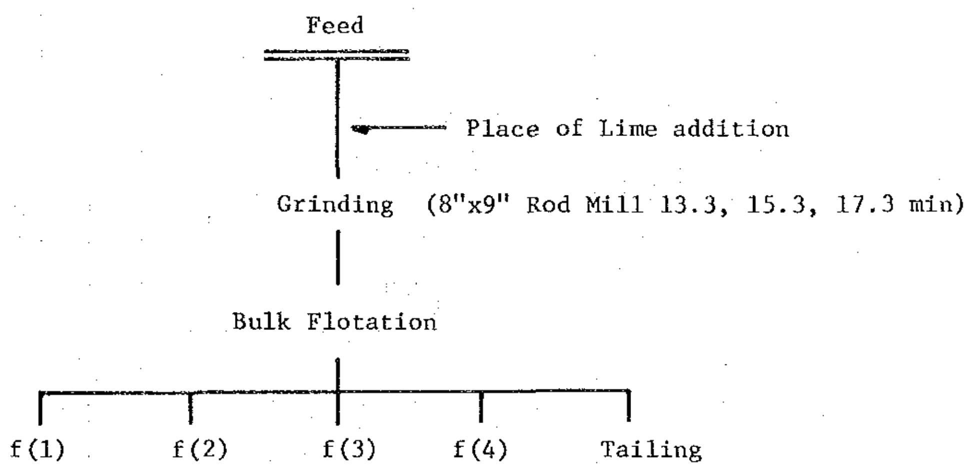
Other test conditions such as the flotation time and the amounts of collectors are indicated in Fig.2 togather with the test flowsheet.

The result of the screening analysis for the flotation feed with the size minus 200 mesh 50% is shown in Table-16.

The test results are given in Table 17 through Table 24.

The relations between flotation time, added amounts of lime, and flotation sizes, and Cu in tailings are shown in Fig.3 through Fig.5.

Fig 2 Test Flowsheet and Condition (Test 1--8)



	f(1)	f(2)	f(3)	f(4)	Total Addition
Flotation Time (min)	0 - 5	5 - 10	10 - 15	15 - 20	
Reagent (g/T)					
Mixed Xanthate	10	5	5	5	25
Z-200	5	-	-	-	5
DF-250	5	5	-	-	10

Table 15 Levels of Test Conditions (Test 1--8)

		Lime Addition kg/T			
		0	1	2	3
Flotation size -200 mesh (%)	45	Test 5			Test 7
	50	Test 1	Test 2	Test 3	Test 4
	55	Test 6			Test 8

Table 16 Screen Analysis of Flotation Feed (-200mesh 50%)

Size (micron)	Wt %		Assay %		Distribution %	
		Passing	Cu	S	Cu	S
297	1.75	98.23	0.14	0.65	0.11	0.15
210	2.99	95.26	0.19	1.66	0.31	0.65
149	6.23	89.03	0.67	3.46	2.27	2.80
105	19.70	69.33	1.56	7.63	16.72	19.54
74	18.45	50.88	1.85	9.67	18.57	23.19
44	19.45	31.43	1.98	9.52	20.95	24.07
20	13.47	17.96	2.44	8.73	17.88	15.29
-20	17.96		2.37	6.13	23.16	14.31
Total	100.00		1.84	7.69	100.00	100.00

Fig 3 Relationship between Flotation Time and Cu in Tailing

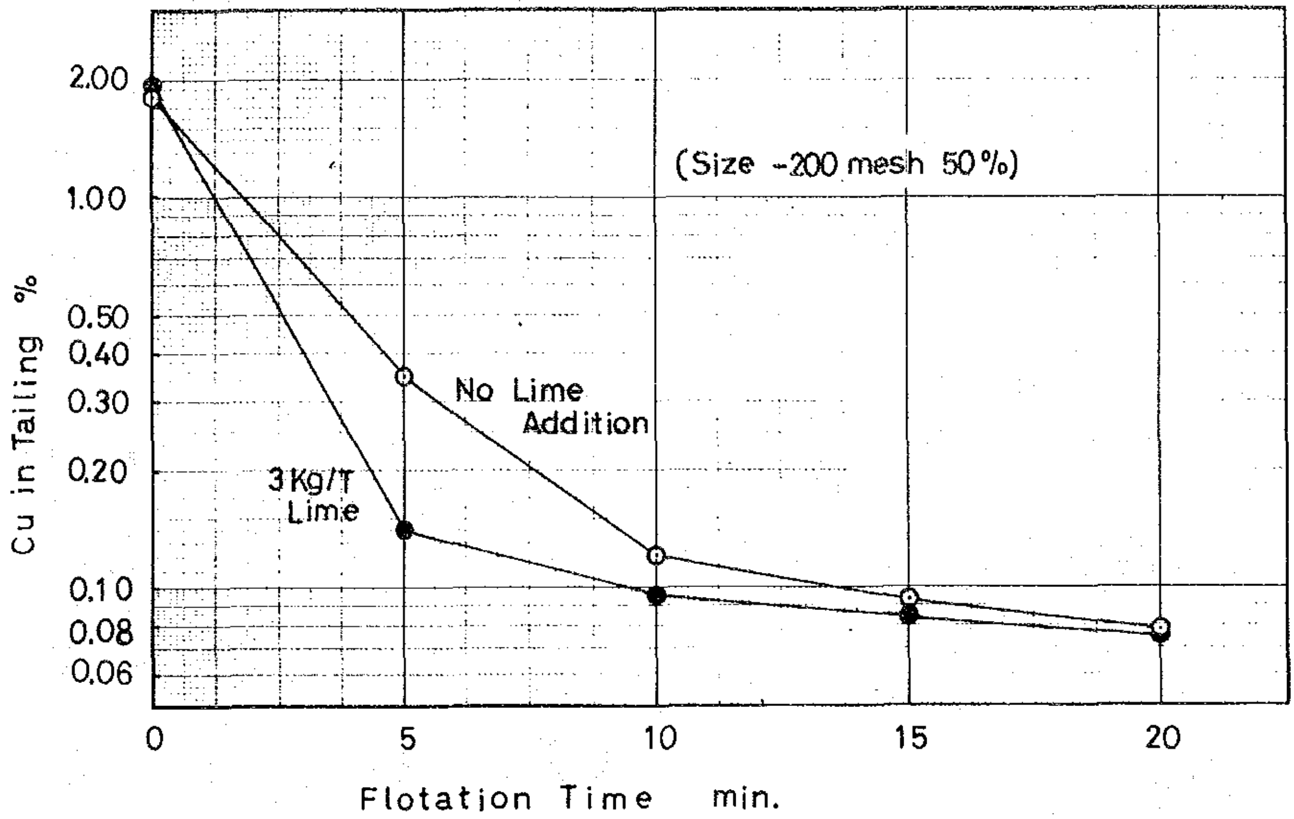


Fig. 4 Relationship between Lime Addition and Tailing Grade

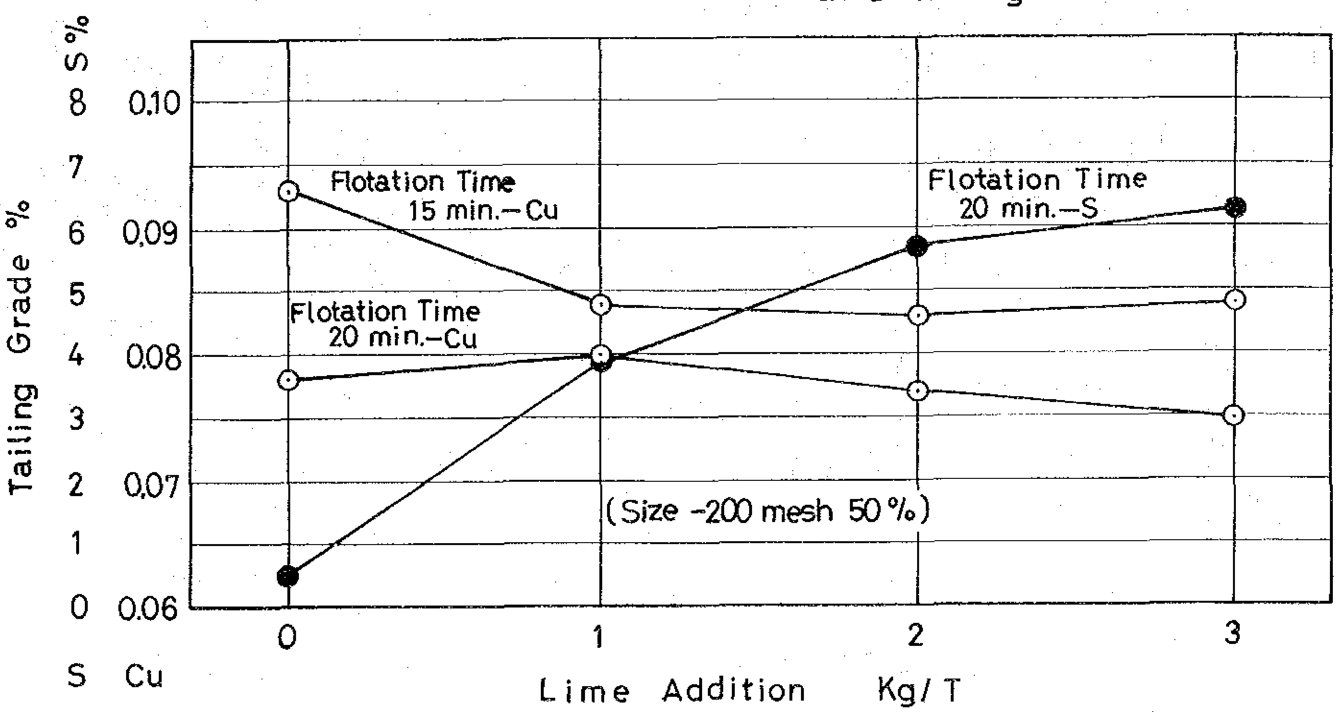
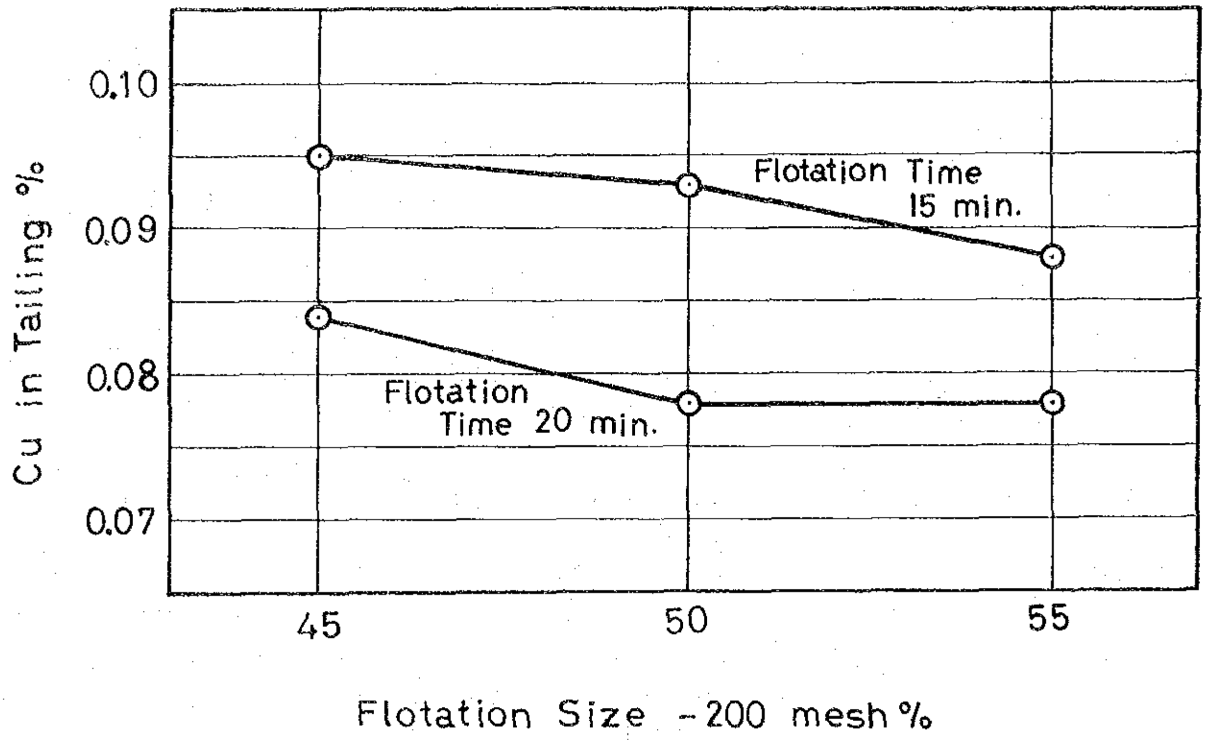


Fig. 5 Relationship between

Flotation Size and Copper in Tailing

—— (1) No Lime Addition



—— (2) Lime Addition 3 Kg/T

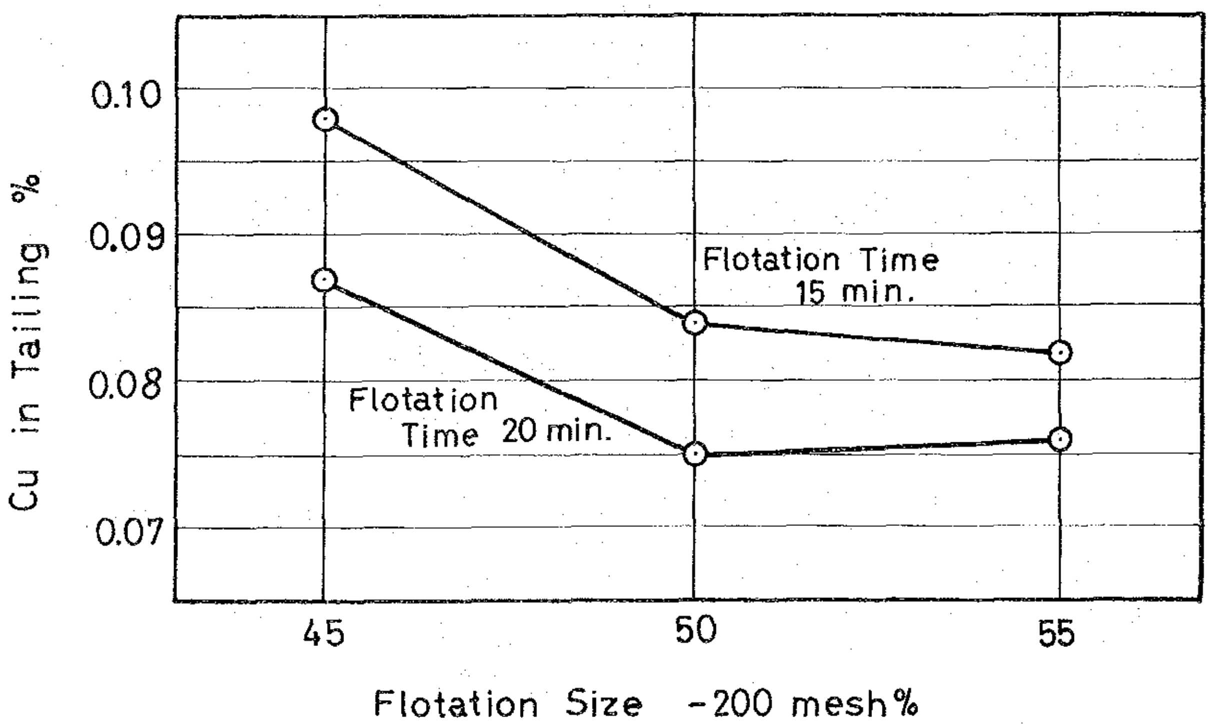


Table 17 Metallurgical Result (Test 1)

Product		Wt %	Assay %		Distribution %	
			Cu	S	Cu	S
Feed		100.00	1.82	7.69	100.00	100.00
(1)	Froth (1)	13.75	11.00	40.97	83.33	73.25
(2)	" (2)	2.88	7.00	41.52	11.11	15.55
(3)	" (3)	0.75	3.18	34.60	1.31	3.38
(4)	" (4)	1.12	1.21	15.88	0.75	2.31
(5)	Tailing	81.50	0.078	0.52	3.50	5.51
(1)+(2)		16.63	10.31	41.07	94.44	88.80
(1)+(2)+(3)		17.38	10.00	40.77	95.75	92.18
(1)+(2)+(3)+(4)		18.50	9.47	39.28	96.50	94.49
(5)+(4)		82.62	0.093	0.73	4.25	7.82
(5)+(4)+(3)		83.37	0.121	1.03	5.56	11.20
(5)+(4)+(3)+(2)		86.25	0.35	2.47	16.67	26.75

Table 18 Metallurgical Result (Test 2)

Product		Wt %	Assay %		Distribution %	
			Cu	S	Cu	S
Feed		100.00	1.82	7.77	100.00	100.00
(1)	Froth (1)	6.56	26.04	31.70	93.81	26.77
(2)	" (2)	1.12	2.58	35.54	1.59	5.13
(3)	" (3)	1.44	0.49	44.20	0.39	8.19
(4)	" (4)	2.44	0.25	48.60	0.33	15.27
(5)	Tailing	88.44	0.080	3.92	3.88	44.64
(1)+(2)		7.68	22.62	32.26	95.40	31.90
(1)+(2)+(3)		9.12	19.12	34.15	95.79	40.09
(1)+(2)+(3)+(4)		11.56	15.14	37.20	96.12	55.36
(5)+(4)		90.88	0.084	5.12	4.21	59.91
(5)+(4)+(3)		92.32	0.091	5.73	4.60	68.10
(5)+(4)+(3)+(2)		93.44	0.121	6.09	6.19	73.23

Table 19 Metallurgical Result (Test 3)

Product		Wt %	Assay %		Distribution %	
			Cu	S	Cu	S
Feed		100.00	1.83	7.65	100.00	100.00
(1)	Froth (1)	5.88	28.75	33.34	92.19	25.63
(2)	" (2)	1.12	4.87	19.97	2.98	2.92
(3)	" (3)	0.62	1.87	19.50	0.63	1.58
(4)	" (4)	0.50	1.25	19.50	0.34	1.28
(5)	Tailing	91.88	0.077	5.71	3.86	68.59
(1)+(2)		7.00	24.93	31.20	95.17	28.55
(1)+(2)+(3)		7.62	23.05	30.25	95.80	30.13
(1)+(2)+(3)+(4)		8.12	21.71	29.59	96.14	31.41
(5)+(4)		92.38	0.083	5.78	4.20	69.87
(5)+(4)+(3)		93.00	0.095	5.88	4.83	71.45
(5)+(4)+(3)+(2)		94.12	0.15	6.04	7.81	74.37

Table 20 Metallurgical Result (Test 4)

Product		Wt %	Assay %		Distribution %	
			Cu	S	Cu	S
Feed		100.00	1.95	8.12	100.00	100.00
(1)	Froth (1)	6.37	28.58	30.83	93.23	24.17
(2)	" (2)	0.88	4.87	19.34	2.19	2.10
(3)	" (3)	0.75	1.61	17.46	0.62	1.61
(4)	" (4)	0.94	0.95	14.00	0.46	1.62
(5)	Tailing	91.06	0.075	6.29	3.50	70.50
(1)+(2)		7.25	25.70	29.44	95.42	26.27
(1)+(2)+(3)		8.00	23.44	28.31	96.04	27.88
(1)+(2)+(3)+(4)		8.94	21.08	26.81	96.50	29.50
(5)+(4)		92.00	0.084	6.37	3.96	72.12
(5)+(4)+(3)		92.75	0.096	6.46	4.58	73.73
(5)+(4)+(3)+(2)		93.63	0.14	6.58	6.77	75.83

Table 21 Metallurgical Result (Test 5)

Product		Wt %	Assay %		Distribution %	
			Cu	S	Cu	S
Feed		100.00	1.82	7.39	100.00	100.00
(1)	Froth (1)	14.63	11.41	39.63	91.94	78.48
(2)	" (2)	1.94	2.82	42.00	3.01	11.03
(3)	" (3)	0.81	1.65	26.11	0.74	2.86
(4)	" (4)	1.00	0.97	13.91	0.53	1.88
(5)	Tailing	81.62	0.084	0.52	3.78	5.75
(1)+(2)		16.57	10.40	39.91	94.95	89.51
(1)+(2)+(3)		17.38	10.00	39.26	95.69	92.37
(1)+(2)+(3)+(4)		18.38	9.51	37.88	96.22	94.25
(5)+(4)		82.62	0.095	0.68	4.31	7.63
(5)+(4)+(3)		83.43	0.110	0.93	5.05	10.49
(5)+(4)+(3)+(2)		85.37	0.17	1.86	8.06	21.52

Table 22 Metallurgical Result (Test 6)

Product		Wt %	Assay %		Distribution %	
			Cu	S	Cu	S
Feed		100.00	1.80	7.53	100.00	100.00
(1)	Froth (1)	12.50	12.32	40.50	85.65	67.27
(2)	" (2)	3.38	4.98	44.35	9.36	19.92
(3)	" (3)	0.75	2.22	39.95	0.93	3.98
(4)	" (4)	0.69	1.23	20.84	0.47	1.91
(5)	Tailing	82.68	0.078	0.63	3.59	6.92
(1)+(2)		15.88	10.76	41.32	95.01	87.19
(1)+(2)+(3)		16.63	10.37	41.26	95.94	91.17
(1)+(2)+(3)+(4)		17.32	10.01	40.44	96.41	93.08
(5)+(4)		83.37	0.088	0.80	4.06	8.83
(5)+(4)+(3)		84.12	0.107	1.15	4.99	12.81
(5)+(4)+(3)+(2)		87.50	0.31	2.82	14.35	32.73

Table 23 Metallurgical Result (Test 7)

Product		Wt %	Assay %		Distribution %	
			Cu	S	Cu	S
Feed		100.00	1.88	7.91	100.00	100.00
(1)	Froth (1)	6.00	28.58	31.06	91.28	23.56
(2)	" (2)	1.19	5.30	17.93	3.36	2.70
(3)	" (3)	0.63	1.70	16.44	0.57	1.31
(4)	" (4)	1.12	0.96	13.05	0.57	1.85
(5)	Tailing	91.06	0.087	6.13	4.22	70.58
(1)+(2)		7.19	24.73	28.89	94.64	26.26
(1)+(2)+(3)		7.82	22.87	27.88	95.21	27.57
(1)+(2)+(3)+(4)		8.94	20.13	26.03	95.78	29.42
(5)+(4)		92.18	0.098	6.21	4.79	72.43
(5)+(4)+(3)		92.81	0.108	6.28	5.36	73.74
(5)+(4)+(3)+(2)		94.00	0.17	6.43	8.72	76.44

Table 24 Metallurgical Result (Test 8)

Product		Wt %	Assay %		Distribution %	
			Cu	S	Cu	S
Feed		100.00	1.77	7.22	100.00	100.00
(1)	Froth (1)	5.44	29.90	32.16	92.10	24.23
(2)	" (2)	0.87	6.12	20.76	3.01	2.50
(3)	" (3)	0.50	1.97	20.05	0.56	1.39
(4)	" (4)	0.50	1.19	18.17	0.34	1.26
(5)	Tailing	92.69	0.076	5.50	3.99	70.62
(1)+(2)		6.31	26.62	30.59	95.11	26.73
(1)+(2)+(3)		6.81	24.81	29.81	95.67	28.12
(1)+(2)+(3)+(4)		7.31	23.20	29.02	96.01	29.38
(5)+(4)		93.19	0.082	5.57	4.33	71.88
(5)+(4)+(3)		93.69	0.092	5.65	4.89	73.27
(5)+(4)+(3)+(2)		94.56	0.15	5.78	7.90	75.77

(Discussion)

The copper content in tailings declines sharply for the first ten minutes and gently for the subsequent ten minutes as seen in Fig.3. Therefore, 15 minutes of the flotation time appear to be adequate, although the longer the flotation time is, the better the result is expected. The test result only for the flotation size -200 mesh 50% is shown in Fig.3. However, the test results for other cases suggest that there appear to be the similar relations between the flotation time and the copper content in tailings for different flotation sizes.

Relations between flotation size and copper content in tailings are compared for two cases (Fig.5); in one case, no lime is added and in the other, 3 kg/ton of lime is added. In the former case, the copper content in tailings decreases from 0.095% to 0.88% in accordance with reduction of the flotation size from minus 200 mesh 45% to 55%. The similar tendency for the copper content in tailings is also observed in the latter case. Decrease in the copper content in tailings is around 0.01% for the both cases and negligibly small.

However, it must be noted that the coarser fraction of the test feed is smaller in the size distribution than that of the actual bulk flotation feed, and that the range of the size distribution of the test feed is narrower.

The copper content in tailings is higher in the coarser fraction as described in the later section. Copper in the coarser fraction is mostly in the state of middlings with gangues.

Accordingly, the difference of the copper contents in tailings between the different flotation sizes will become larger for the size distribution of the actual bulk flotation feed. The adequate flotation size may be around minus 200 mesh 50% for the present operation.

The optimum flotation size is affected by other flotation conditions and should be economically determined by taking into account of such factors as Cu-recovery and grinding cost. A more detail study is necessary to determine the optimum flotation size.

In Fig.4, the copper content in the tailings varies in a very small range from 0.075 to 0.080% for the different amounts of added lime from 0 to 3 kg/T (flotation time 20 minutes), or tends to slightly decrease by increasing amount of added lime.

On the other hand, the sulfur content in the tailings increases sharply from about 1% to nearly 6% by increasing amount of added lime up to 2 kg/T, and gently to a little over 6% by adding 3 kg/T of lime. This result indicates that addition of 2 kg/T of lime depresses most amount of pyrite.

With this results, a very small change in the copper content in the tailing suggests that chalcopyrite is loosely asseciated with pyrite and that these two minerals are easily separable in a flotation process.

Judging from all these results, it would be expedient for aiming only at recoverying copper to depress liberated pyrite in the bulk flotation stage and to reduce the load in the differential flotation stage.

However, a flotation process is necessary to be developed for maxmizing pyrite recovery, if one intends to recovery cobalt as much as possible.

In the series of these flotation tests, copper content in tailing is 0.09% in general, which is much lower than 0.13% Cu in the tailings of the present operation. It is probably a major reason that the test sample contained less amount of slimy material than the ore presently treated in the concentrator, although various other reasons are concievable.

6-4 Bulk and Differential Flotation Test

Successive bulk and differential flotation tests were conducted to compare over-all metallurgical results.

In test 9, 10, and 11, it was intended to compare metallurgical results for various amounts of added lime in the bulk flotation process. The mill feed sample was used for these test and the bulk flotation size was -200 mesh 50%.

Test 12 was done on the lump ore sample without adding lime. The bulk flotation size was -200 mesh 50% for this test. These test conditions were the same as those for the test 9. The purpose of the test was to compare the difference in the metallurgical results for the different types of the samples.

The test flowsheet, conditions and results for the series of these test are shown in Fig.6 and 7, and in Table 25 through 33. The relation between Cu-recovery and copper concentrate grade is shown in Fig.8.

Fig 6 Test Flowsheet (Test 9 and 12)

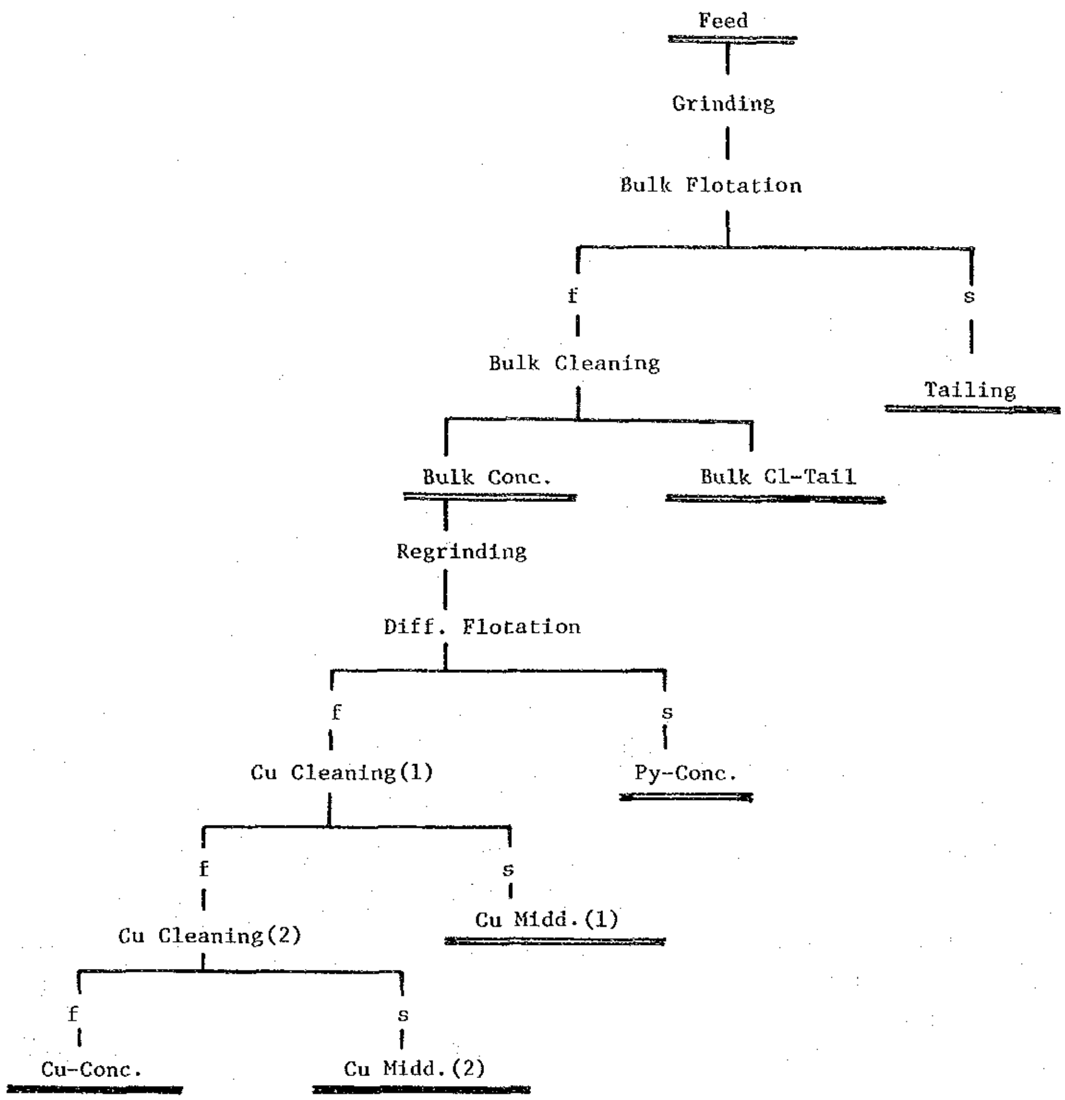


Table 25 Test Condition (Test 9)

	Bulk Fl.	Bulk Cl.	Diff. Fl.	Cu Cl. (1)	Cu Cl. (2)	Total
Test Feed	Mill Feed (800g)					
Grinding Mill	8"x9" RM		6"x7" RM			
Grinding Time (min)	10		5			
Size -200 mesh (%)	50					
Flotation Time (min)	15	5	10	5	5	
Pulp Density (%)	40	27	27	12	11	
Pulp pH	7.7	-	12.3	12.1	12	
Reagent (g/T)						
Lime	-	-	880	130	-	1010
Mixed Xanthate	25	3	-	-	-	28
Z-200	5	3	5	2		15
DF-250	15	-	3	-	-	18

Fig 7 Test Flowsheet (Test 10,11)

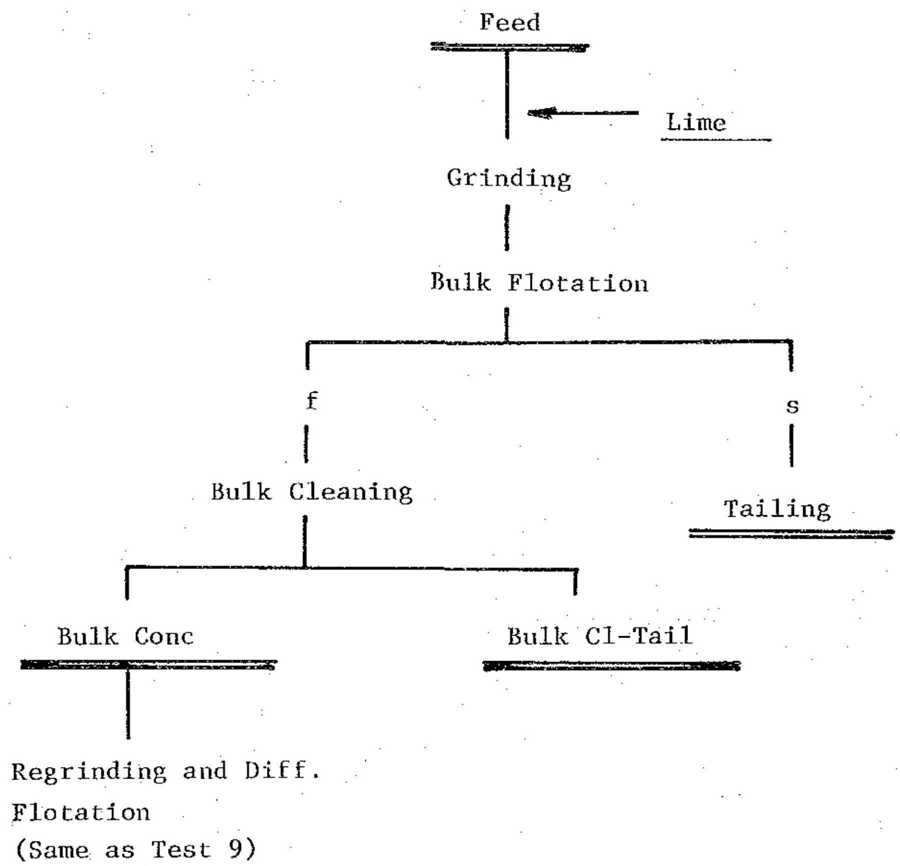


Table 26 Metallurgical Result (Test 9)

Product		Wt %	Assay %		Distribution %	
			Cu	S	Cu	S
Feed		100.00	1.87	7.57	100.00	100.00
(1)	Cu-Conc	5.61	30.70	34.30	91.62	25.41
(2)	Cu-Midd (2)	0.50	6.06	32.40	1.61	2.14
(3)	Cu-Midd (1)	1.02	1.52	33.11	0.82	4.46
(4)	Py-Conc	11.10	0.099	43.41	0.59	63.64
(5)	Bulk Cl-Tail	1.51	0.66	3.22	0.53	0.64
(6)	Tailing	80.26	0.113	0.35	4.83	3.71
(1)+(2)		6.11	28.68	34.08	93.23	27.55
(1)+(2)+(3)		7.03	25.15	31.98	94.05	29.69
(1)+(2)+(3)+(4)		18.23	9.76	39.73	94.64	95.65
(5)+(6)		81.77	0.123	0.40	5.36	4.35

Table 27 Screen Analysis of Tailing (Test 9)

Size (micron)	Wt %	Assay %		Distribution %	
		Cu	S	Cu	S
210	7.3	0.19	0.51	12.9	10.6
149	9.7	0.14	0.42	12.8	11.6
105	23.0	0.12	0.40	25.9	26.2
74	13.9	0.090	0.31	11.7	12.3
-74	46.1	0.085	0.30	36.7	39.3
Total	100.0	0.107	0.35	100.0	100.0

Table 28 Test Condition (Test 10)

	Bulk Fl.	Bulk Cl.	Diff. Fl.	Cu Cl. (1)	Cu Cl. (2)	Total
Test Feed	Mill Feed (800g)					
Grinding Mill	8"x9" RM		6"x7" RM			
Grinding Time (min)	10		4			
Size -200 mesh (%)	50					
Flotation Time (min)	15	5	10	5	5	
Pulp Density (%)	40					
Pulp pH	12.0-11.7	-	12.3	12	12	
Reagent (g/T)						
Lime	1500	-	780	130		2410
Mixed Xanthate	25	3	-	-	-	28
Z-200	5	3	5	2	-	15
DF-250	15	-	3	-	-	18

Table 29 Test Condition (Test 11)

	Bulk Fl.	Bulk Cl.	Diff. Fl.	Cu Cl. (1)	Cu Cl. (2)	Total
Test Feed	Mill Feed (800g)					
Grinding Mill	8"x9" RM		6"x7" RM			
Grinding Time (min)	10		3			
Size -200 mesh (%)	50					
Flotation Time (min)	15	5	10	5	5	
Pulp Density (%)	40					
Pulp pH	12.3-12.0	-	12.4	12	12	
Reagent (g/T)						
Lime	2000	-	630	180		2810
Mixed Xanthate	25	3	-	-	-	28
Z-200	5	3	5	2	-	15
DF-250	15	-	3	-	-	18

Table 30 Metallurgical Result (Test 10)

Product		Wt %	Assay % Cu	S	Co	Co/PyS Ratio	Dist'n % Cu	S	Co
Feed		100.00	1.91	7.85	0.21	$3.5x10^{-2}$	100.00	100.00	100.00
(1)	Cu-Conc	5.60	31.14	33.97	0.06	$2.4x10^{-2}$	91.10	24.24	1.58
(2)	Cu-Midd (2)	0.38	8.07	28.62	0.55	$2.7x10^{-2}$	1.60	1.38	0.99
(3)	Cu-Midd (1)	0.69	3.18	30.04	0.79	$2.9x10^{-2}$	1.15	2.64	2.57
(4)	Py-Conc	7.30	0.18	41.52	1.43	$3.5x10^{-2}$	0.69	38.62	49.22
(5)	Bulk Cl-Tail	1.38	0.83	3.70	0.14	$4.9x10^{-2}$	0.60	0.65	0.91
(6)	Tailing	84.65	0.110	3.01	0.112	$3.9x10^{-2}$	4.86	32.47	44.73
Sulfide in Tailing (*10)		5.45	0.37	41.25	1.45	$3.5x10^{-2}$	1.08	29.27	37.26
Gangue in Tailing (*10)		79.20	0.089	0.31	0.02	$9.1x10^{-2}$	3.78	3.20	7.47
(1)+(2)		5.98	29.67	33.63	0.09	$2.5x10^{-2}$	92.70	25.62	2.57
(1)+(2)+(3)		6.67	26.93	33.26	0.16	$2.6x10^{-2}$	93.85	28.26	5.14
(1)+(2)+(3)+(4)		13.97	12.95	37.58	0.83	$3.4x10^{-2}$	94.54	66.88	54.36
(5)+(6)		86.03	0.122	3.02	0.113	$3.9x10^{-2}$	5.46	33.12	45.64

*10 Sulfide in Tailing, Gangue in Tailing : Sulfide froth and sinks obtained by refloating the tailing with H_2SO_4 and xanthate after flotation test.

Table 31 Metallurgical Result (Test 11)

Product		Wt %	Assay % Cu	S	Distribution Cu	S
Feed		100.00	1.88	7.65	100.00	100.00
(1)	Cu-Conc	5.40	31.35	33.81	90.08	23.87
(2)	Cu-Midd (2)	0.38	10.88	28.47	2.20	1.42
(3)	Cu-Midd (1)	0.50	4.47	26.20	1.19	1.71
(4)	Py-Conc	3.83	0.31	30.90	0.63	15.47
(5)	Bulk- Cl-Tail	1.38	1.04	5.82	0.76	1.05
(6)	Tailing	88.51	0.109	4.88	5.14	56.48
(1)+(2)		5.78	30.00	33.46	92.28	25.29
(1)+(2)+(3)		6.28	27.97	32.88	93.47	27.00
(1)+(2)+(3)+(4)		10.11	17.49	32.13	94.10	42.47
(5)+(6)		89.89	0.123	4.89	5.90	57.53

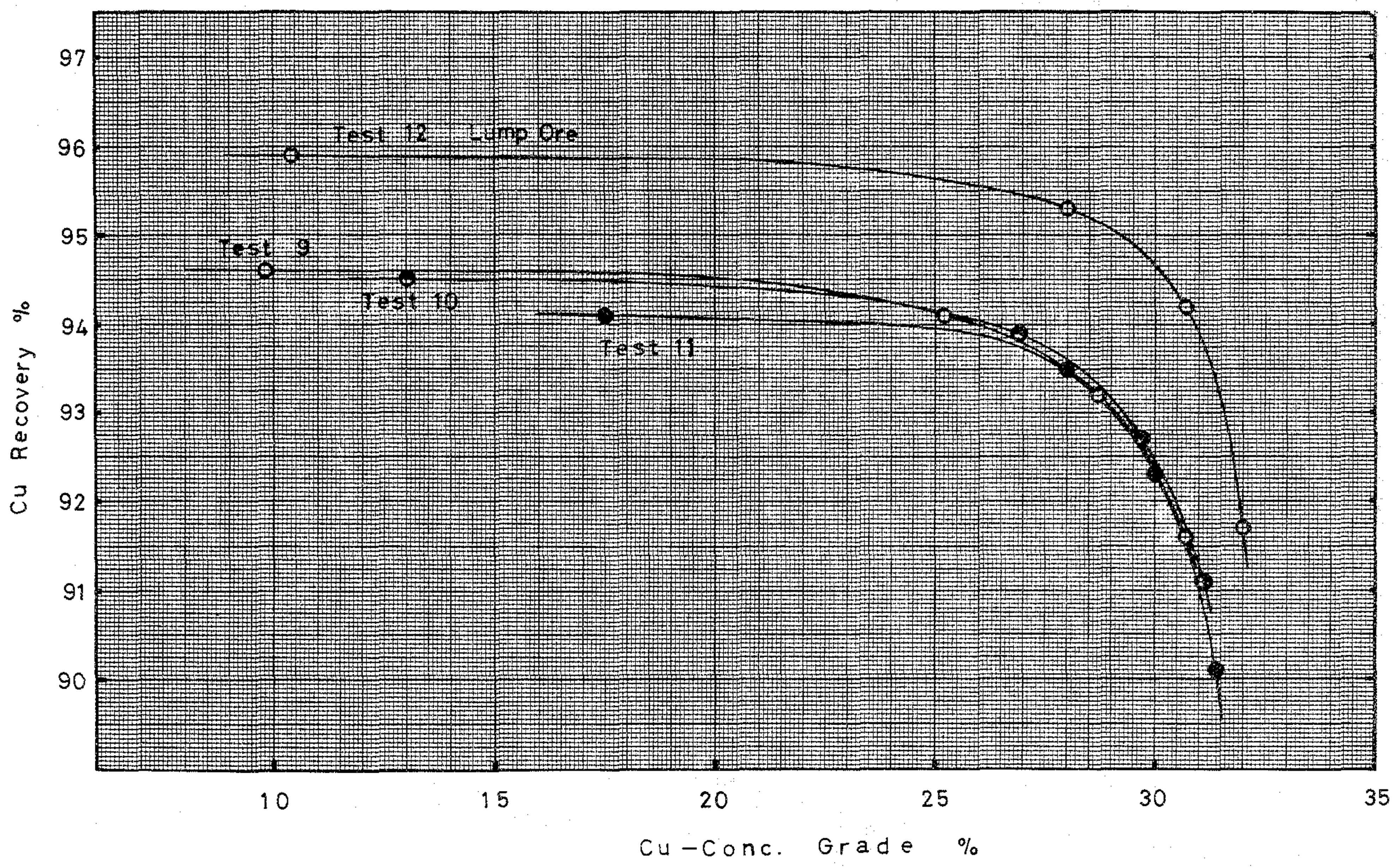
Table 32 Test Condition (Test 12)

Circuit	Bulk Fl.	Bulk Cl.	Diff. Fl.	Cu Cl. (1)	Cu Cl. (2)	Total
Test Feed	Lump Ore (800g)					
Grinding Mill	8"x9" RM		6"x7" RM			
Grinding Time (min)	15		4			
Size -200 mesh (%)	50					
Flotation Time (min)	8	5	10	5	5	
Pulp Density (%)	40					
Pulp pH	8.5	-	-	-	-	
Reagent						
Lime	-	-	880	130	-	1010
Mixed Xanthate	25	3	-	-	-	28
Z-200	5	3	5	2	-	15
DF-250	15	-	3	-	-	18

Table 33 Metallurgical Result (Test 12)

Product		Wt %	Assay %		Distribution %	
			Cu	S	Cu	S
Feed		100.00	1.88	7.93	100.00	100.00
(1)	Cu-Conc	5.39	32.00	34.52	91.66	23.47
(2)	Cu-Midd (2)	0.38	12.60	33.90	2.55	1.62
(3)	Cu-Midd (1)	0.63	3.28	36.72	1.10	2.92
(4)	Py-Conc	10.90	0.102	47.03	0.59	64.67
(5)	Bulk Cu-Tail	1.63	0.44	7.16	0.38	1.47
(6)	Tailing	81.07	0.086	0.57	3.72	5.85
(1)+(2)		5.77	30.72	34.48	94.21	25.09
(1)+(2)+(3)		6.40	28.02	34.70	95.31	28.01
(1)+(2)+(3)+(4)		17.30	10.37	42.47	95.90	92.68
(5)+(6)		82.70	0.093	0.70	4.10	7.32

Fig. 8 Cu Recovery as a function of Cu-Conc Grade —(1)



(Discussion)

The sulfur content in the tailings for the test 9, 10 and 11 increases from 0.35% to 4.8% by increasing amount of added lime (0, 1.5, 2.5 kg/T), while the copper content stays around 0.11%. These results are the same as those obtained in the test 1 through 8.

In the differential flotation test, copper concentrates exceed 30% in Cu grade were obtained by twice cleanings. The ore is easily up-graded in copper content by differential flotation, that is, chalcopyrite is easily separable from pyrite.

As a matter of course, the copper content in the bulk concentrates increases by increasing amount of added lime in the bulk flotation, and the differential flotation becomes easier.

The copper separation results were similar in all of these tests as shown in Fig.8. The copper concentrates of 28.5% Cu were obtained with a copper recovery of 93.3%.

The copper recovery is a little higher than that in the actual operation. However, it seems to be reasonable, taking into account that the actual mill feed contains some amount of wash slime.

The copper separation results are a little affected by varying the amount of added lime as indicated by the above tests. However, it is advisable that no lime is added, if one intends to recover pyrite as much as possible. Without adding lime, the load in the differential flotation and accordingly the re-grinding cost will increase, though lime consumption will decrease.

As the results of the screening test on the tailings in the test 9, the copper content in the coarser fractions is higher than in the finer fractions (Table-27). The microspic observation of the screened products indicates that copper contained in the coarser fractions than 105 microns is mostly in the state of middlings with gangues. Only a small quantity of liberated copper perticles are observed even in the finer fractions. The middling copper particles are too fine to be liberated only by further grinding the tailings to some extent. Accordingly, a scheme, which recovers middling copper particles as they are, should be innovated to decrease the copper content in tailings to a lower level than being performed in the present mill operation.

The copper content in the pyrite concentrate increases from 0.099% to 0.31% by increasing amount of added lime. However, its copper distribution stays almost unchanged, around 0.6%.

The microscopic observation of the pyrite concentrate obtained by the test 11 indicates that about 30% of copper particles in the pyrite concentrate are liberated and the rest, in a state of middlings mostly with gangues. In other words, chalcopyrite is closely associated with gangues in parts, while association of chalcopyrite and pyrite is loose. This occurrance explains the fact that the copper content in the pyrite concentrate increases without changing its copper distribution by increasing amount of added lime in the bulk flotation.

All the products of the test 10 were assayed for cobalt to examine the cobalt distribution (Table 30). The cobalt/pyrite-sulfur ratio in each product is around 3×10^{-2}, although there are some differences. The ratio is relatively high in the bulk cleaner tail and the gangue in the tailings, and low in the copper concentrate and the copper middling. These differences may be cause by cobalt content in gangues or the cobalt mineral, or by distribution of pyrrhotite. However, they are too small to influence cobalt distribution in the products.

The ratio in the pyrite concentrate is 3.5×10^{-2} and identical with that in the tailings.

At any event, most of cobalt is distributed in close association with pyrite.

As the result of test 12 on the lump ore sample, the copper concentrate of 28.5% Cu was obtained with a copper recovery of approximately 95%, which is about 2% higher than that obtained by the test on the mill feed sample.

7. The Effect of Circulating Load in Bulk Flotation Circuit.

The effect of circulating load in bulk flotation, which is being practiced in the actual milling operation, was preliminarily examine.

The test flowsheet, conditions and results are shown in Fig.9, Table 34 and 35.

Fig 9 Test Flowsheet (Test 13)

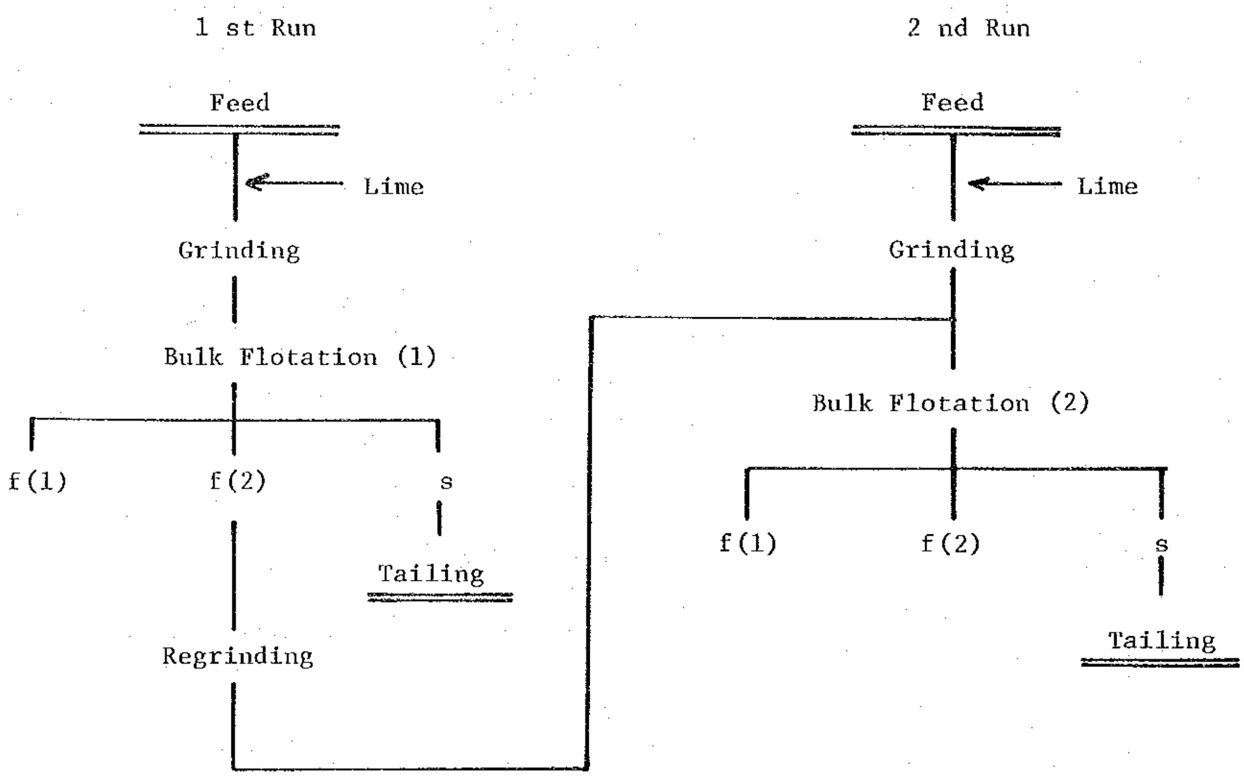


Table 34 Test Condition (Test 13)

Circuit	Bulk Fl. (1) and (2)		f(2) Regrinding	Total
	f(1)	f(2)		
Test Feed	Mill Feed (800g)			
Grinding Mill	8"x9" RM		6"x7" RM	
Grinding Time (min)	11		3	
Size -200 mesh (%)	50			
Flotation Time (min)	5	10		15
Pulp Density (%)	40			
Pulp pH	12.0	11.7		
Reagent (G/T)				
Lime	1500			1500
Mixed Xanthate	10	15		25
Z-200	5	-		5
DF-250	10	5		15

(Discussion)

The copper content in the tailings, obtained in the first run of the flotation of the new feed, was 0.110%, and that, obtained in the second run of the flotation of the feed added with the middling, was 0.116%.

The effect of circulating load in the bulk flotation circuit was very small.

8. Improvement in Metallurgical Results

Judging from the results of a number of the flotation test explained in the preceding sections, recovery of copper middling with gangue is important for improving the metallurgical results.

For this purpose, application of kerosene was proved to be effective after a few examinations.

It was intended to carry out an over-all test by using kerosene, and to compare the results with those obtained in the preceeding tests.

The test was done twice (test 14 and test 15), fundamentally in the same manner with minor differences in regrinding time and in the number of cleaning stages.

The test flowsheet, conditions and results are given in Fig.10 and in Tables 38 through 40.

The relations between copper content in the copper concentrates and copper recovery are shown in Fig.11 with the results obtained in the test 9.

(Discussion)

The copper content in the tailings was 0.074% in the test 15 and 0.075% in the test 16, which are considerably lower than approximately 0.11% Cu in the tailings obtained by the preceding tests 9 through 11.

In Fig.12, it is observed that the results of the two test are nearly the same and that the copper recovery is 94.3% where the copper concentrate grade is 28.5% Cu. The recovery is improved by about 1% in comparison with the results of the tests 9 through 11.

In the test 14, the regrinding time for the bulk froth was 3.5 minutes and the bulk cleaning was practiced only once. As the result, the sulphur content in the pyrite concentrate was as low as 40%.

In the test 15, the regrinding time was prolonged to 5 minutes and the bulk cleaning was practiced twice. As the result, the pyrite concentrate of 47%S was obtained.

Table 35 Metallurgical Result (Test 13)

Product		Wt %	Assay %		Distribution %	
			Cu	S	Cu	S
Feed		200.00	1.84	7.58	200.00	200.00
1 st Run	f(1)	8.17	20.25	30.20	89.96	32.55
"	Tailing	82.72	0.110	2.02	4.95	22.05
2 nd Run	f(1)	8.67	20.31	28.94	95.75	33.10
"	Tailing	93.09	0.116	6.00	5.87	73.70
"	f(2)	7.35	0.87	39.80	3.47	38.60

Fig 10 Test Flowsheet (Test 14)

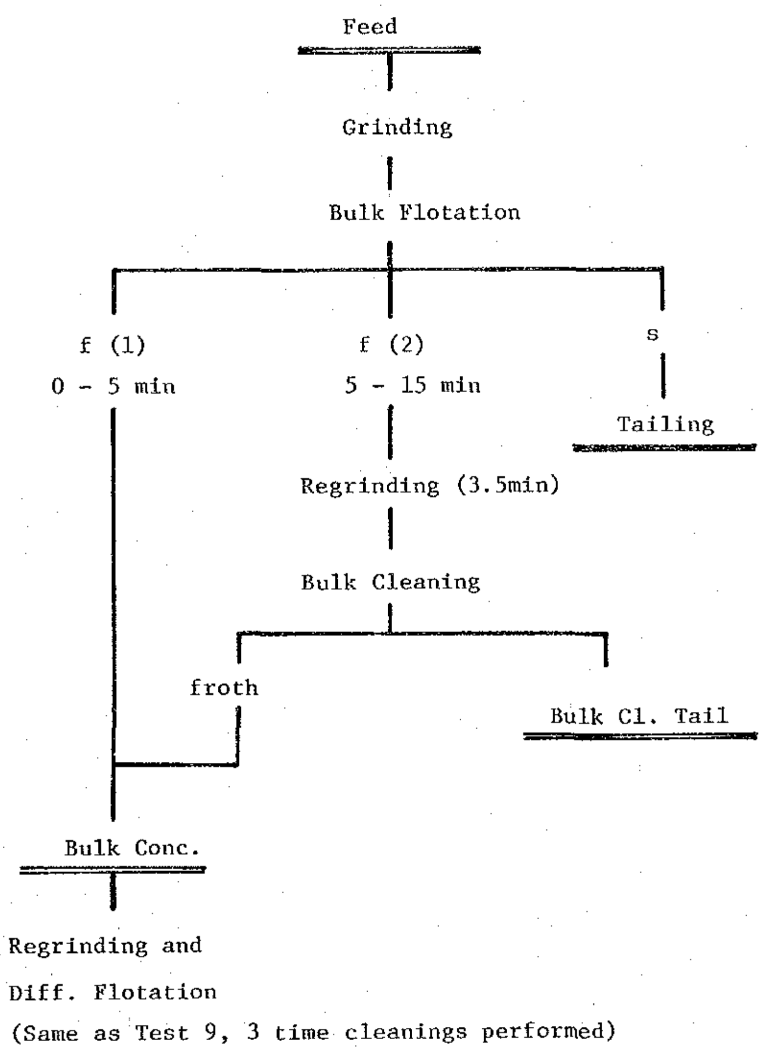


Table 36 Test Condition (Test 14)

	Bulk f(1)	Bulk f(2)	Bulk Cleaning	Total (Bulk)
Test Feed	Mill Feed (800g)			
Grinding mill	8"x9" RM		6"x7" RM	
Grinding Time(min)	10		3.5	
Size -200 mesh(%)	50			
Flotation Time (min)	5	10	5	
Pulp Density (%)	40			
Pulp pH	7.7			
Reagent (g/T)				
Lime	-	-	-	-
Mixed Xanthate	10	20	-	30
Z-200	5	-	2	7
DF-250	10	20	2	22
Kerosene	-	200	-	200

Table 37 Metallurgical Result (Test 14)

Product		Wt %	Assay %		Distribution %	
			Cu	S	Cu	S
Feed		100.00	1.85	7.37	100.00	100.00
(1)	Cu-Conc.	5.90	29.46	34.35	93.86	27.49
(2)	Cu-Mid (3)	1.00	2.31	42.23	1.25	5.73
(3)	Cu-Mid (2)	0.94	0.92	41.05	0.46	5.23
(4)	Cu-Mid (1)	1.13	0.39	33.20	0.24	5.09
(5)	Py-Conc.	9.66	0.082	40.42	0.43	52.97
(6)	Bulk Cl-Tail	4.52	0.28	2.12	0.69	1.30
(7)	Tailing	76.85	0.074	0.21	3.07	2.19
(1)+(2)		6.90	25.53	35.49	95.11	33.22
(1)+(2)+(3)		7.84	22.58	36.16	95.57	38.45
(1)+(2)+(3)+(4)		8.97	19.78	35.79	95.81	43.54
(1)+(2)+(3)+(4)+(5)		18.63	9.57	38.19	96.24	96.51
(6)+(7)		81.37	0.085	0.32	3.76	3.49

Fig 11 Test Flowsheet (Test 15)

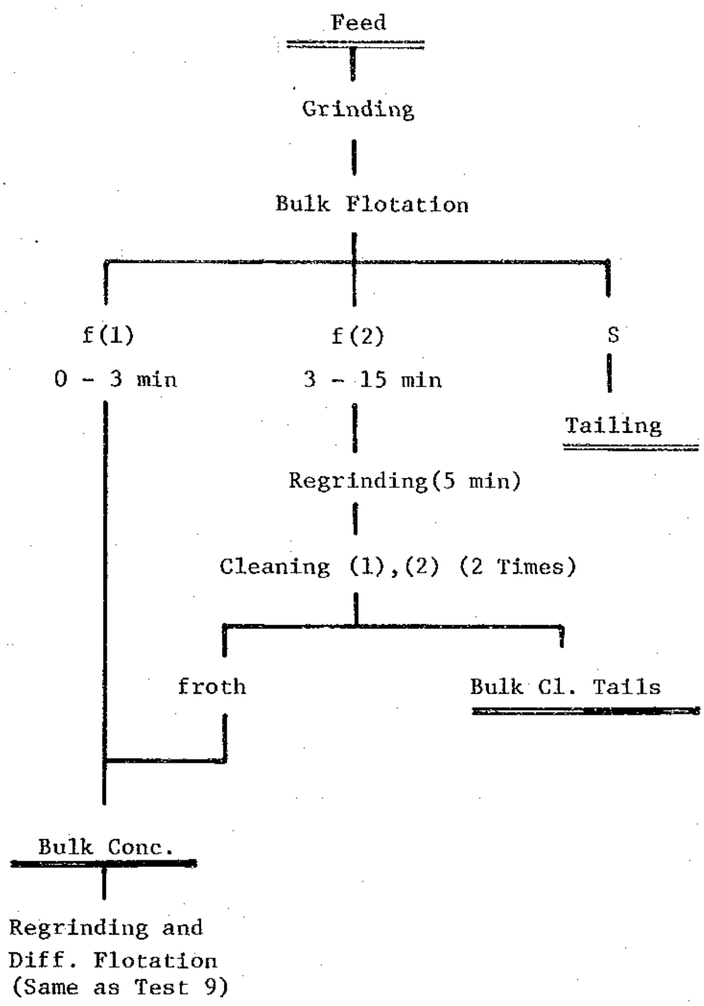


Table 38 Test Condition (Test 15)

	Bulk f(1)	Bulk f(2)	Cleaning (1)	Cleaning (2)	Total (Bulk)
Test Feed	Mill Feed (800g)				
Grinding Mill	8"x9" RM		6"x7" RM		
Grinding Time (min)	10		5		
Size -200 mesh (%)	50				
Flotation Time (min)	3	12	5	5	
Pulp Density (%)	40				
Pulp pH	7.7				
Reagent (g/T)					
Lime	-	-	-	-	-
Mixed Xanthate	10	20	2	-	32
Z-200	5	-	2	-	7
DF-250	10	20	2	-	32
Kerosene	-	240	-	-	240

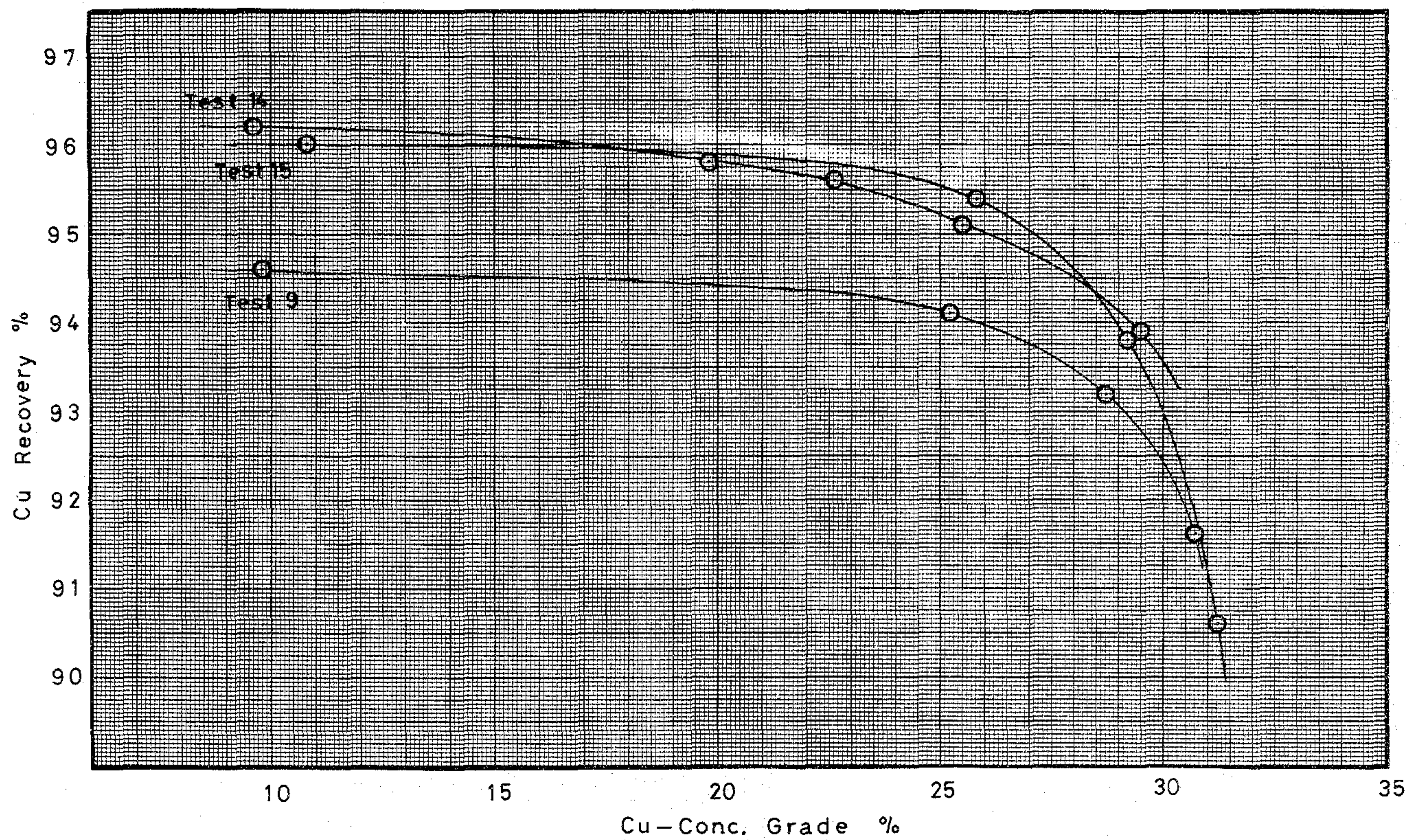
Table 39 Metallurgical Result (Test 15)

Product		Wt %	Assay %		Distribution %	
			Cu	S	Cu	S
Feed		100.00	1.86	7.35	100.00	100.00
(1)	Cu-Conc.	5.40	31.23	33.92	90.62	24.90
(2)	Cu-Mid (2)	0.57	10.30	34.25	3.15	2.65
(3)	Cu-Mid (1)	0.92	3.30	36.90	1.63	4.59
(4)	Py-Conc.	9.80	0.12	47.82	0.63	63.72
(5)	Bulk Cl-Tail	6.11	0.26	2.30	0.86	1.93
(6)	Tailing	77.20	0.075	0.21	3.11	2.21
(1)+(2)		5.97	29.23	33.95	93.77	27.55
(1)+(2)+(3)		6.89	25.76	34.32	95.40	32.14
(1)+(2)+(3)+(4)		16.63	10.75	42.39	96.03	95.86
(5)+(6)		83.37	0.089	0.36	3.97	4.14

Table 40 Screen Analysis of Tailing (Test 15)

Size (micron)	Wt %	Assay %		Distribution %	
		Cu	S	Cu	S
210	6.1	0.100	0.35	8.8	9.7
149	9.7	0.097	0.23	12.5	10.2
105	22.0	0.084	0.22	24.6	22.0
74	14.6	0.080	0.19	15.5	12.6
-74	47.6	0.061	0.21	38.6	45.5
Total	100.0	0.075	0.22	100.0	100.0

Fig. 12 Cu Recovery as a function of Cu-Conc. Grade —(2)



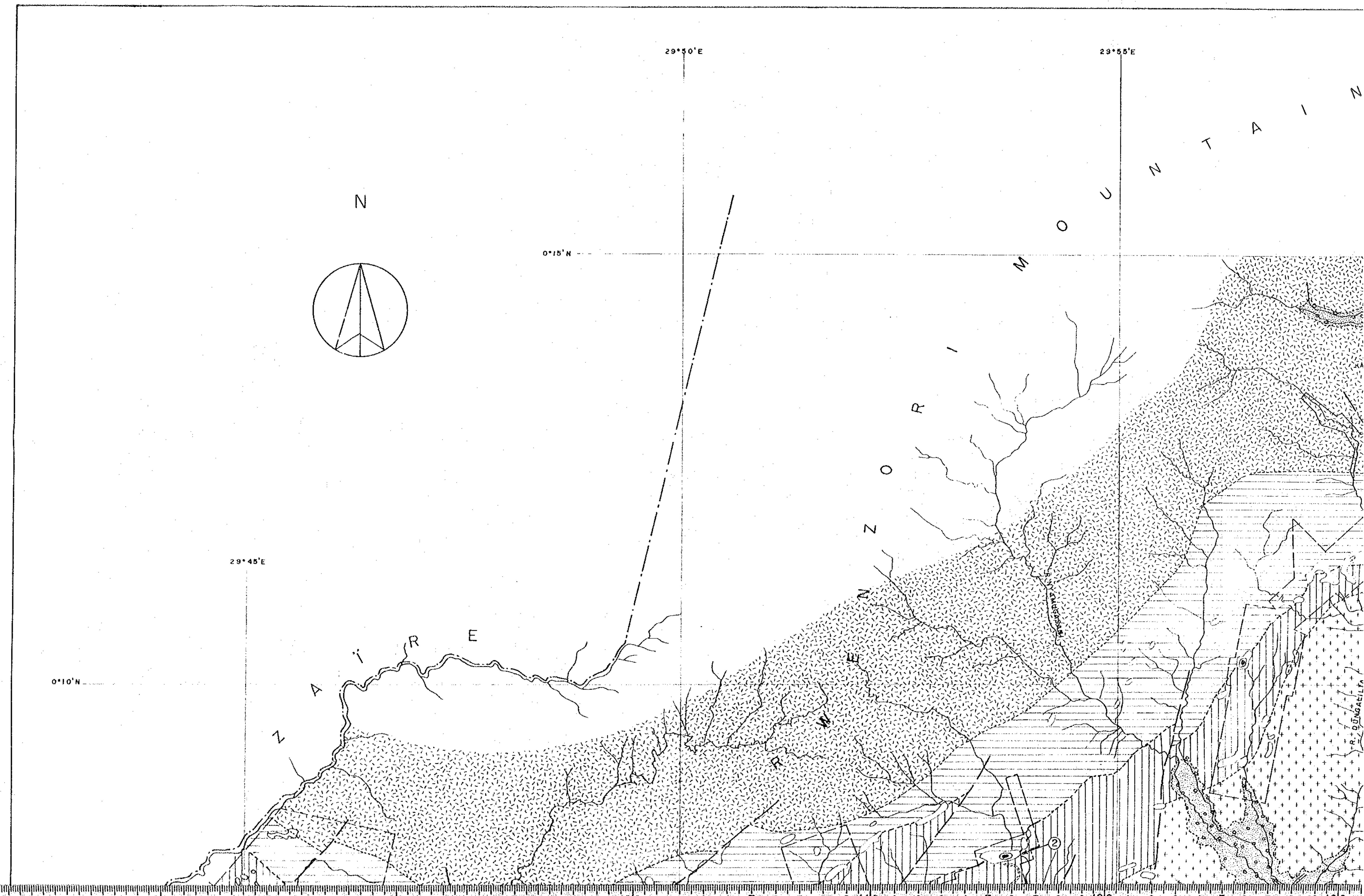
The sulfur content in pyrite concentrates is not controlled in the actual milling operation at the present time. However, the higher sulfur content in pyrite concentrates is desirable to increase the cobalt recovery.

Effects of regrinding the bulk middlings and cleaning should be tested more in detail before applying the procedure of these test in the actual milling operation.

In the test 14, copper concentrate of about 30% Cu was obtained by three times Cu-cleanings. However, twice cleanings will be practically adequate as practiced in the test 15.

It is the writer's observation that the copper content in tailings seems to decrease by increasing amount of collectors and the flotation time.

The results of the screening test on the products of the test 15 are given in Table 40. The copper contents in the coarser fractions are apparently lower than those of the screened tailings of the test 9 (Table 27).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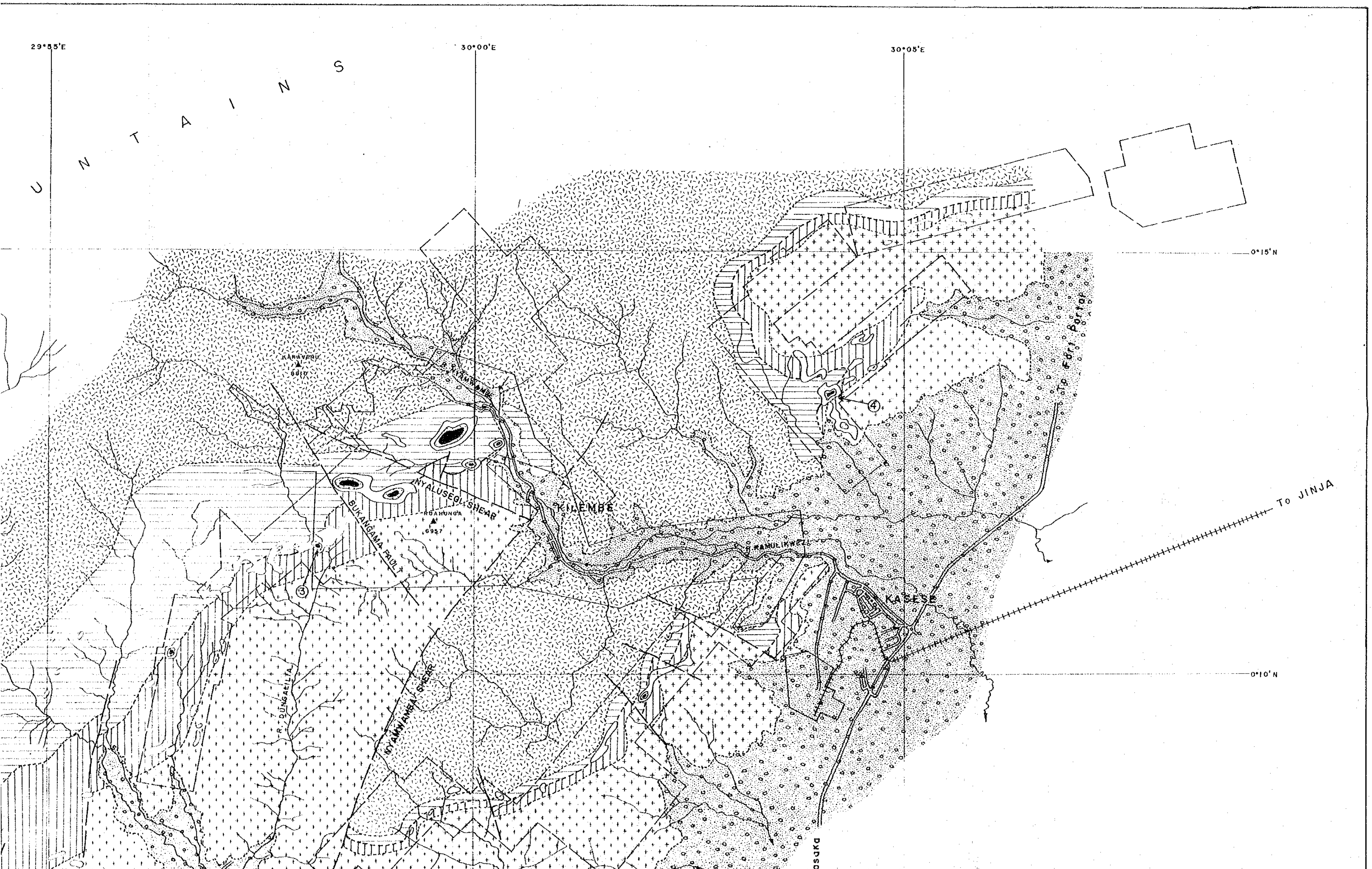
RWENZORI MOUNTAIN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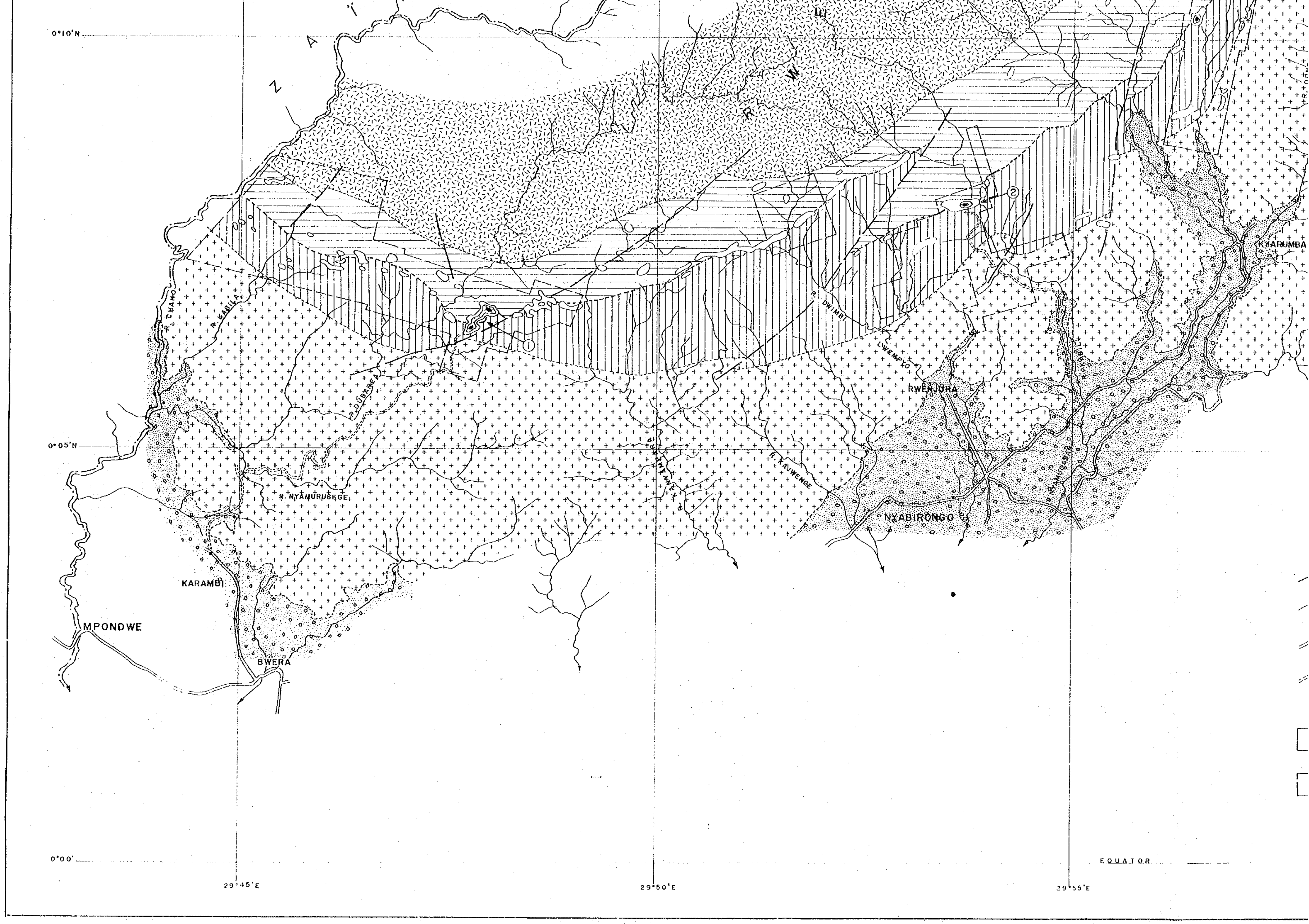
To JINJA
To Fort Portal
NYALUSEGI SHEAR
BUKANGAMA FAULT
KYAMWANGA SHEAR
③
④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R. DWIMBI
RWENJURA
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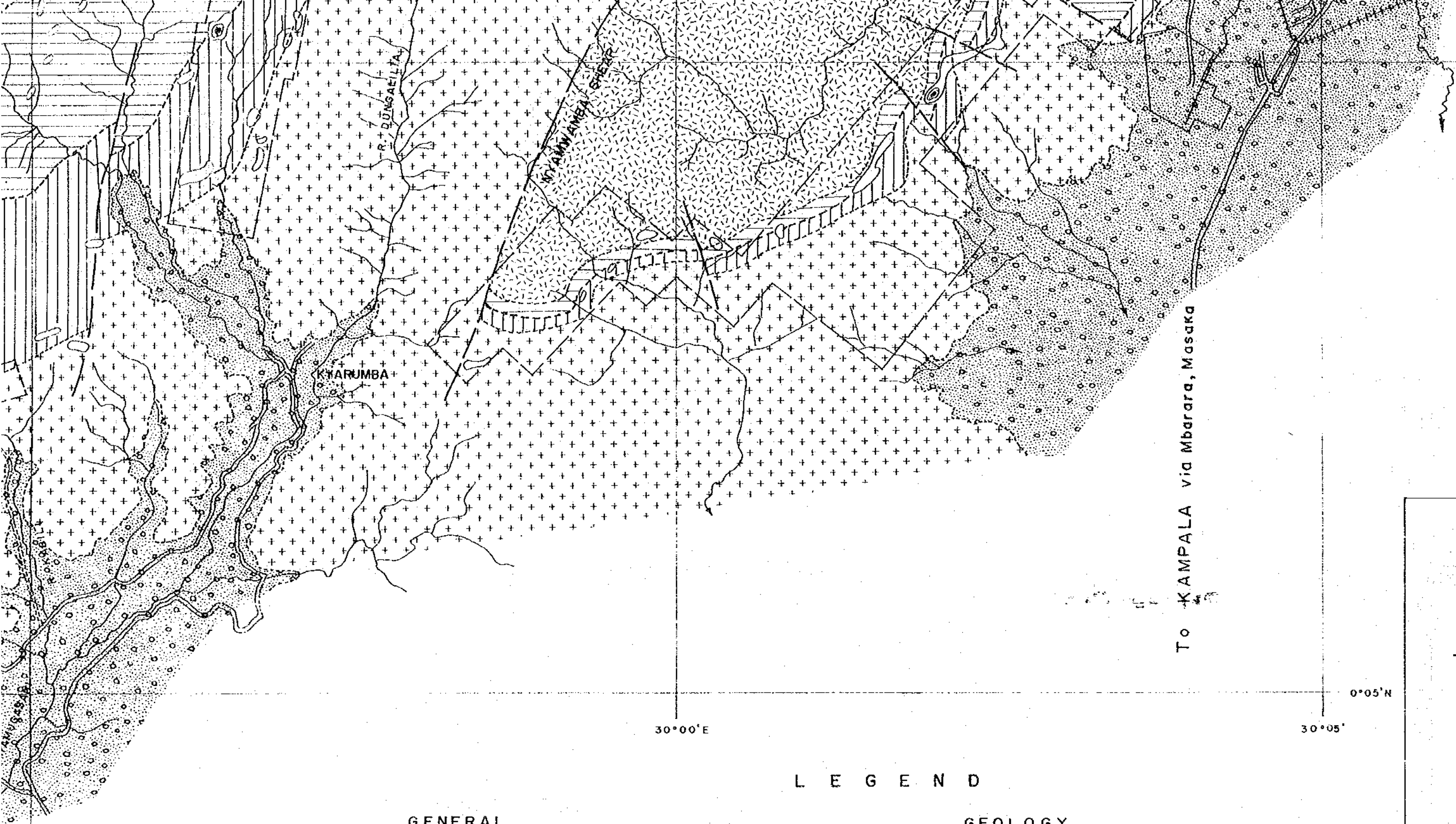


L E G E N D

GENERAL

- INTERNATIONAL BOUNDARY
- RAILWAY
- MAIN ROAD (Tarred Surface)
- ACCESS ROAD (Dirt)
- 8910 PEAK (Elevation in Foot)
- MINING PROPERTY
- AREA OF DETAILED SOIL SURVEY

AREA OF FOLLOW-UP EXPLORATION

1. DUBAREA
2. KABILI
3. BUKANGAMA WEST
4. NKENDA

GEOLOGY

- CENOZOIC
- PRE-CAMBRIAN
  - RUAHUNGA GNEISS
  - KILEMBE SERIES
    - UPPER KILEMBE
    - MIDDLE KILEMBE
    - LOWER KILEMBE
  - KATIRI GNEISS & LK-2

GEOCHEMISTRY (Cu In Soil)

- > 1,000 PPM
- 500 – 1,000 PPM
- 250 – 500 PPM

DWG. No. 1

ENGINEERING STUDY
FOR
THE DEVELOPMENT OF THE KILEMBE MINE
UGANDA

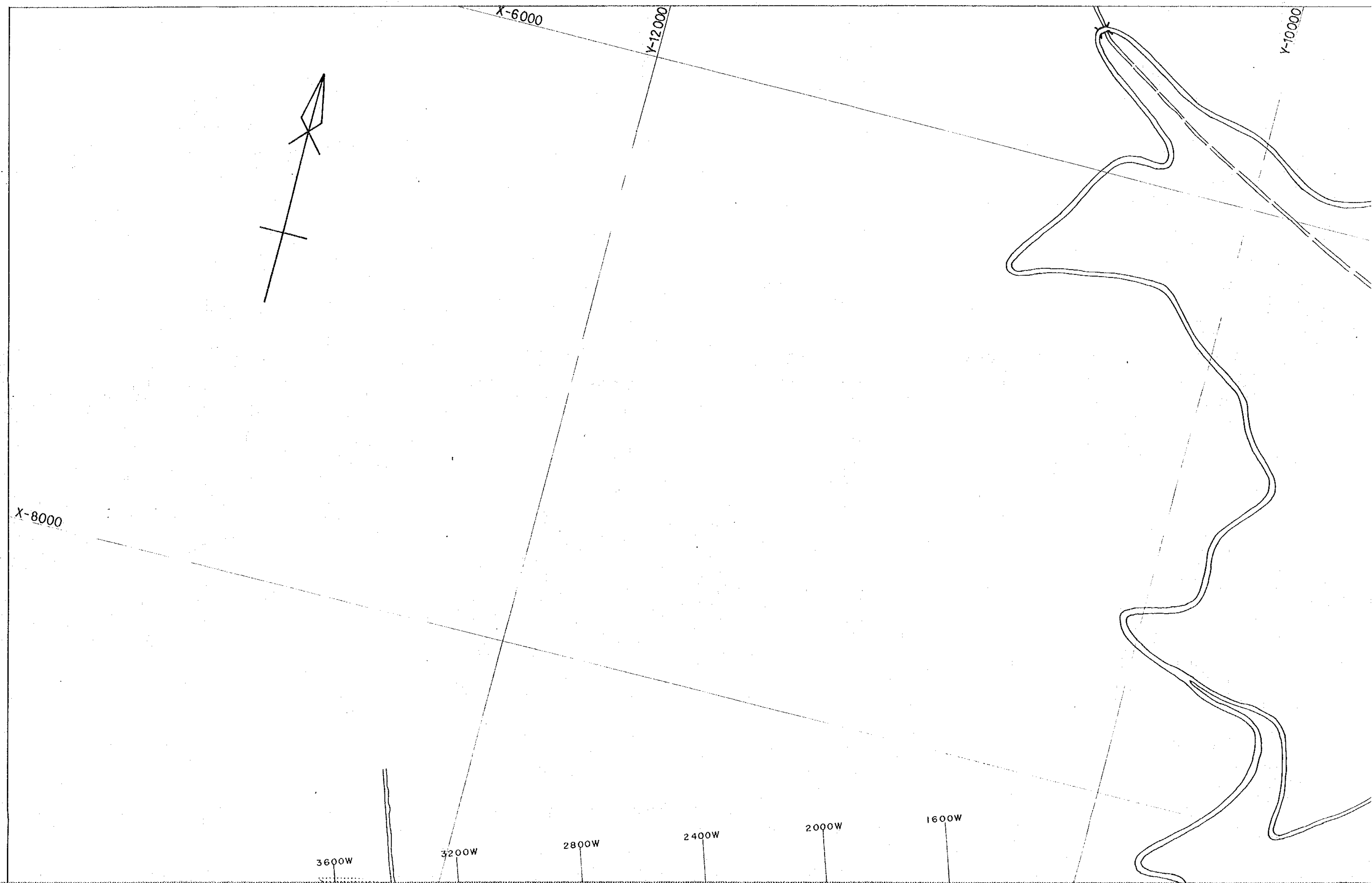
GENERALIZED GEOLOGIC MAP

SCALE 1 : 50,000

1 0 1 2 3 4 5 KM

JAPAN INTERNATIONAL CO-OPERATION AGENCY
GOVERNMENT OF JAPAN

AUG. 1978


X-6000
Y-12000
Y-10000
X-8000
3600W
3200W
2800W
2400W
2000W
1600W

DWG. No. 2

ENGINEERING STUDY
FOR
THE DEVELOPMENT OF THE KILEMBE MINE
UGANDA

UNDERGROUND PLAN
SHOWING COPPER ACCUMULATION
BUKANGAMA DEPOSITS

SCALE 1"=200'

0 200 400 600 800 1000 Feet

JAPAN INTERNATIONAL CO-OPERATION AGENCY
GOVERNMENT OF JAPAN

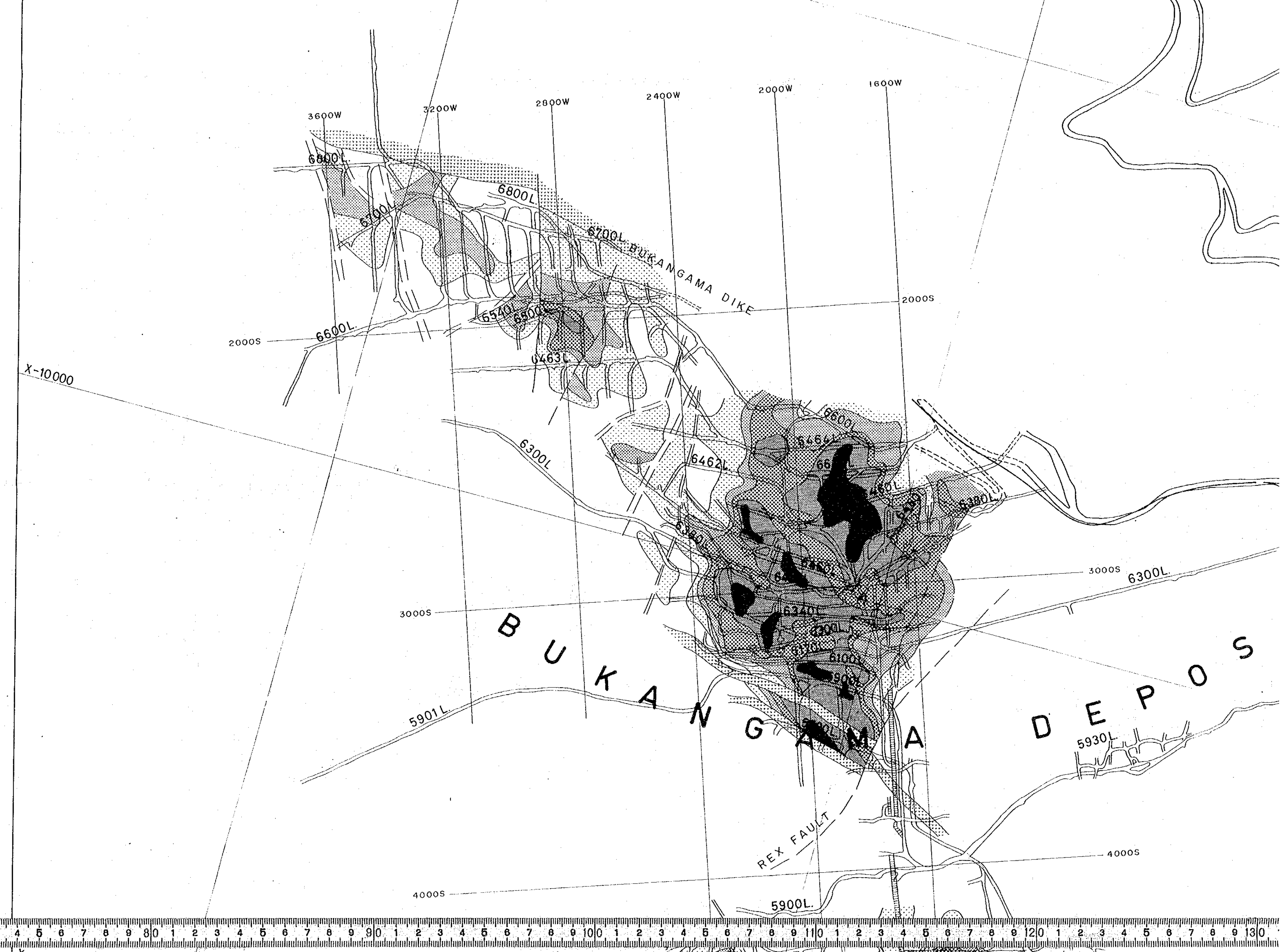
AUG. 1978

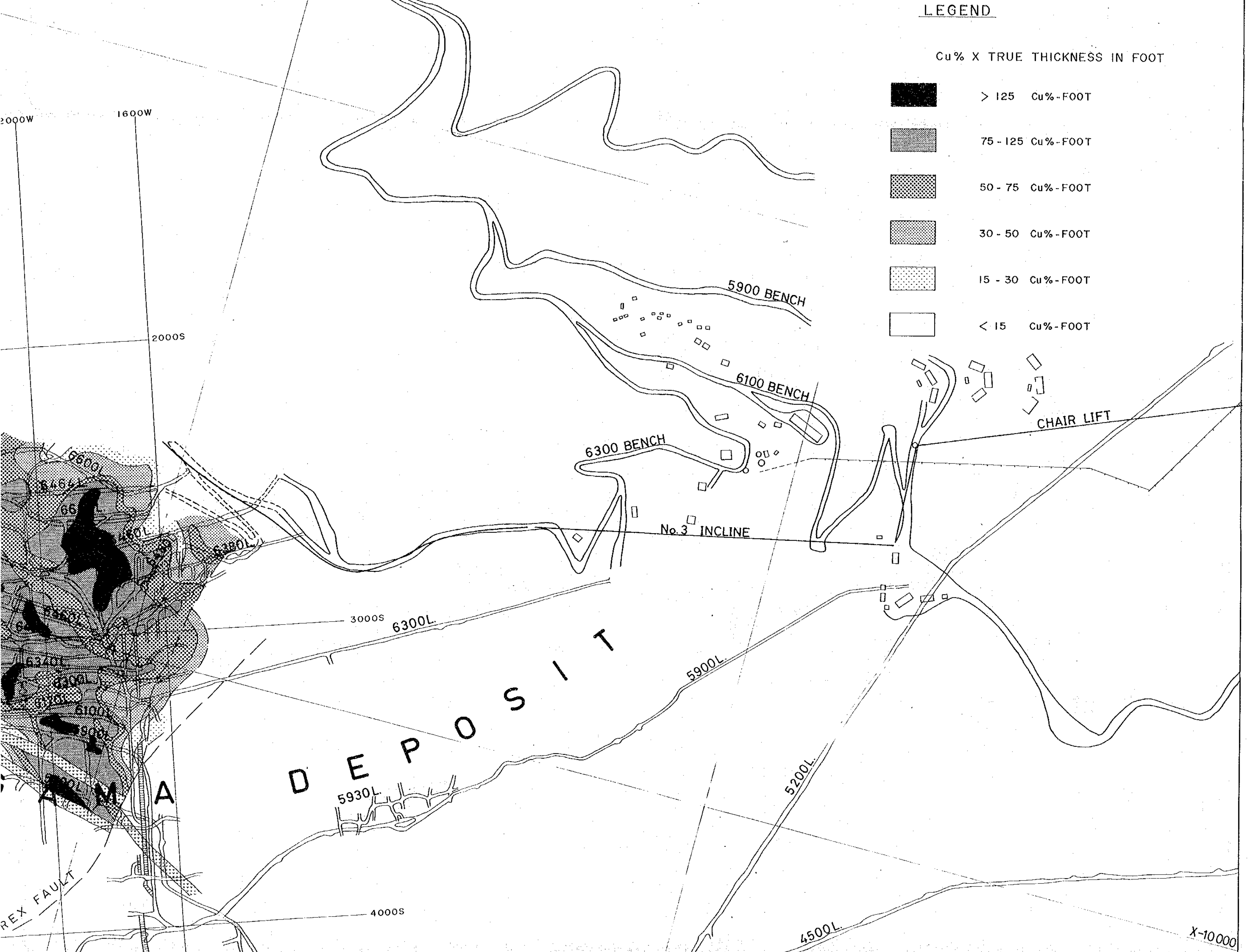
LEGEND

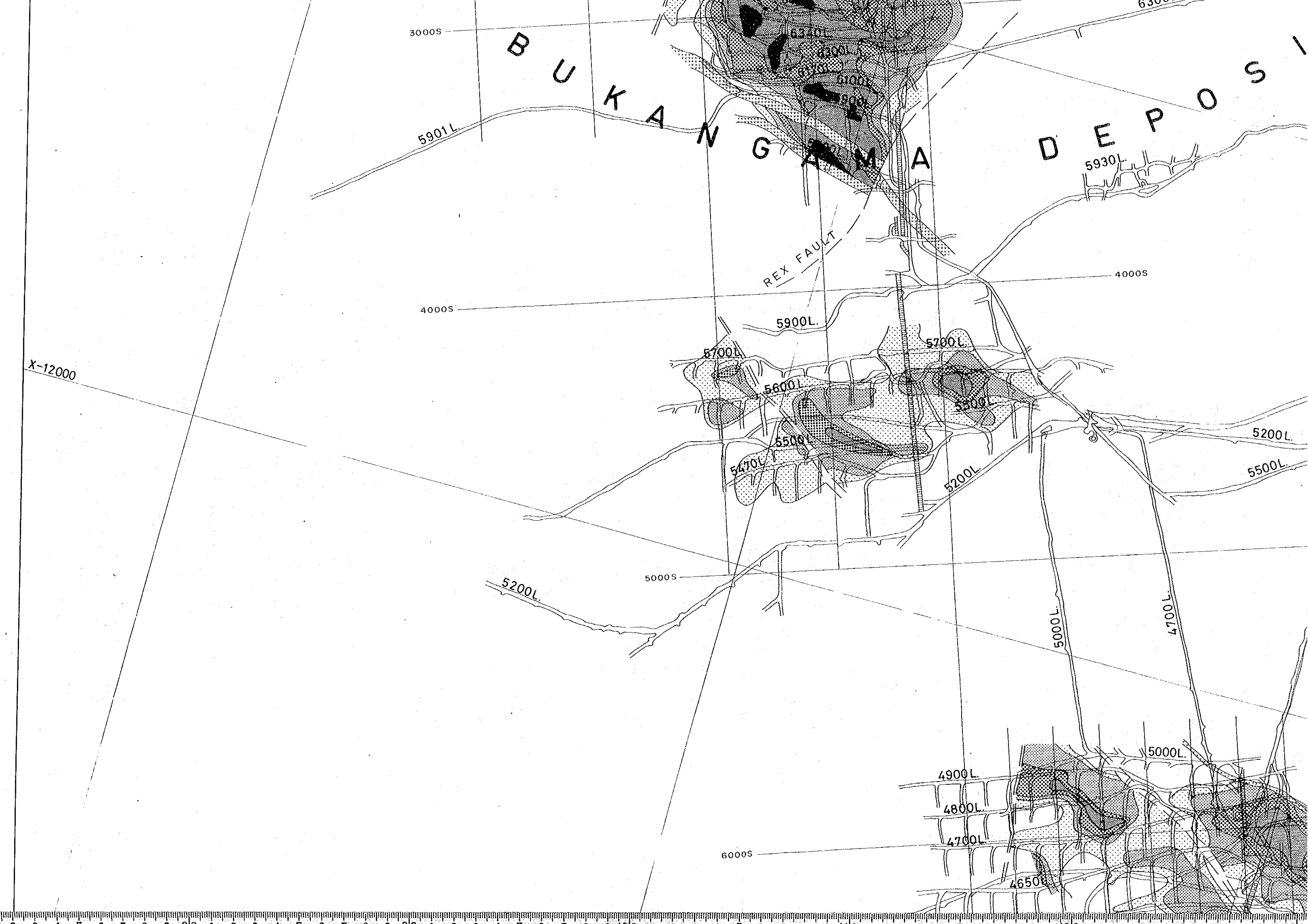
Cu% X TRUE THICKNESS IN FOOT

> 125 Cu%-FOOT

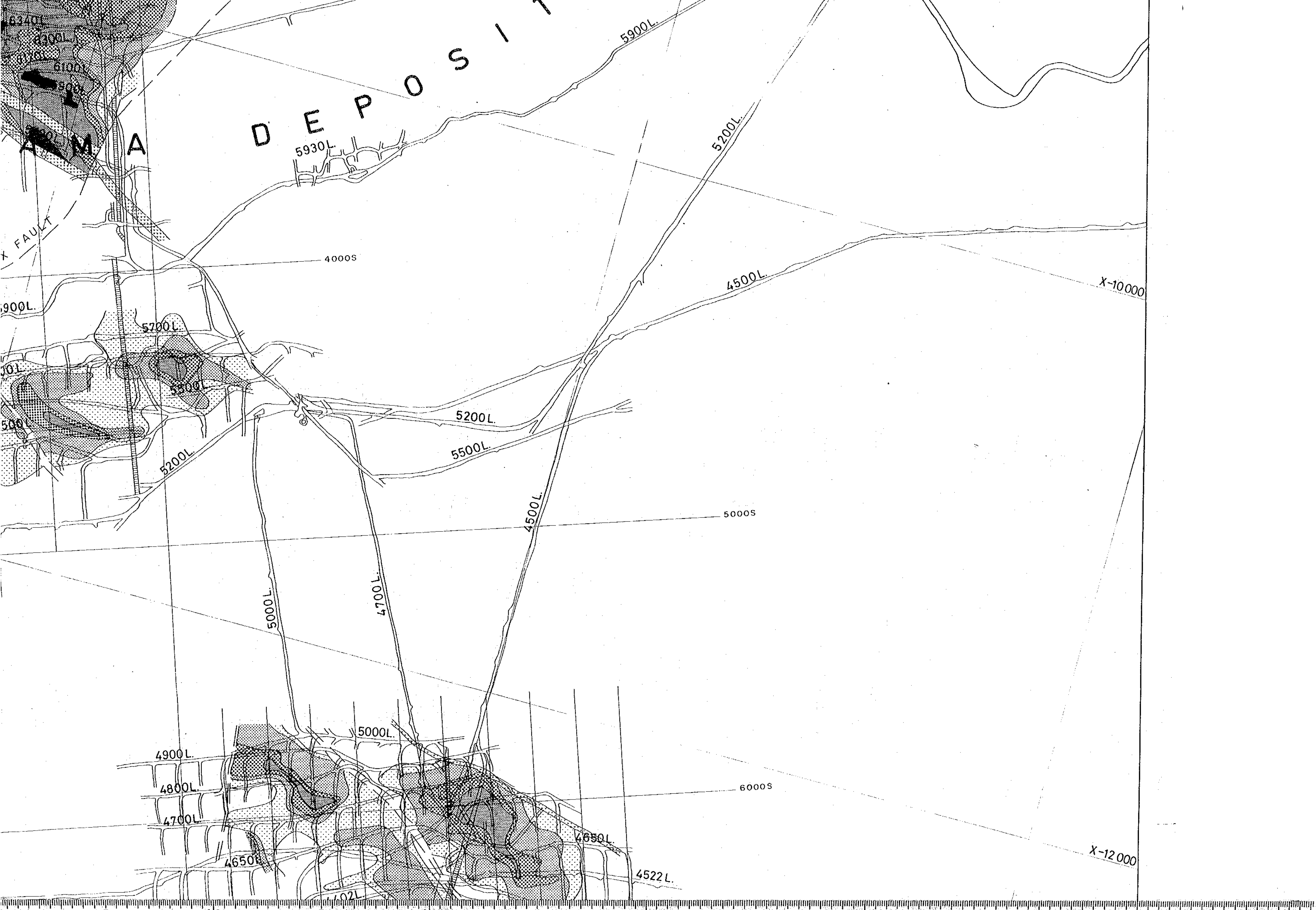
75 - 125 Cu%-FOOT


BUKANGAMA DEPOS
BUKANGAMA DIKE
REX FAULT
X-10000
3600W
3200W
2800W
2400W
2000W
1600W
2000S
3000S
4000S
6800L
6700L
6600L
6300L
5901L
5900L
5930L
6540L
6463L
6462L
6340L

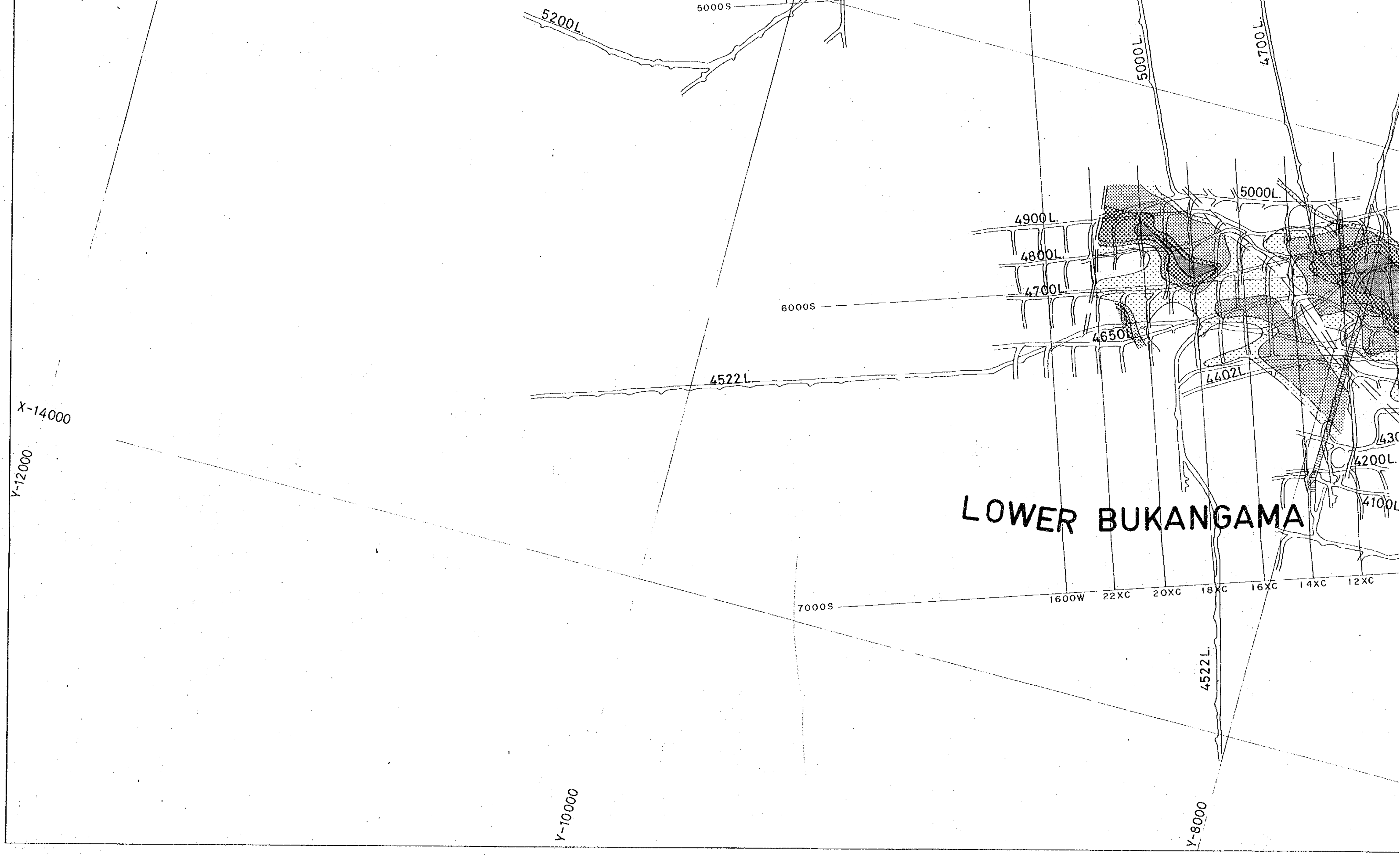

LEGEND
Cu% X TRUE THICKNESS IN FOOT
> 125 Cu%-FOOT
75 - 125 Cu%-FOOT
50 - 75 Cu%-FOOT
30 - 50 Cu%-FOOT
15 - 30 Cu%-FOOT
< 15 Cu%-FOOT
2000W
1600W
2000S
3000S
4000S
5900 BENCH
6100 BENCH
6300 BENCH
CHAIR LIFT
No.3 INCLINE
6600L
6300L
6380L
6340L
6100L
5900L
5930L
5200L
4500L
DEPOSIT
A M A
REX FAULT
X-10000



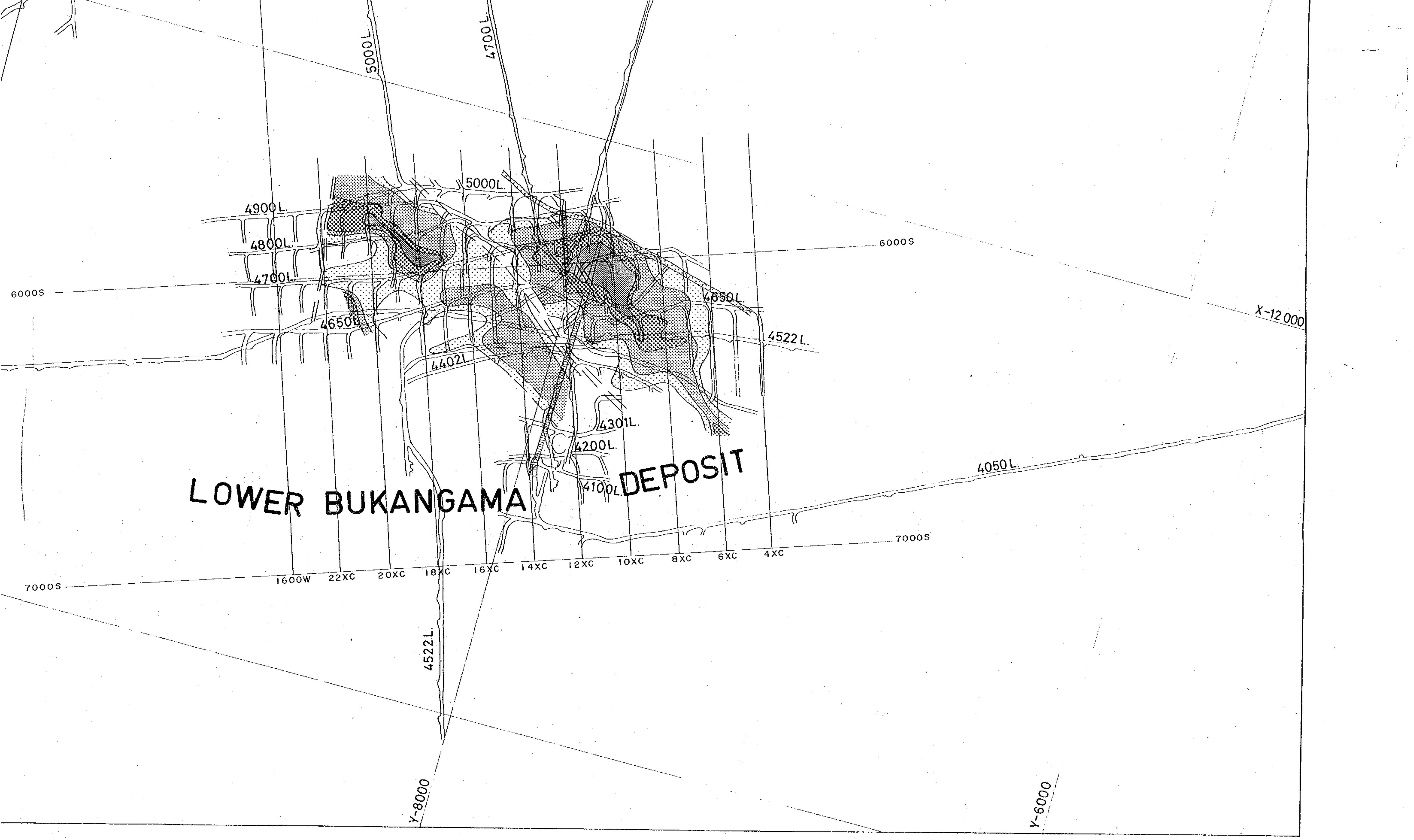
BUKANGAMA DEPOSI
3000S
6340L.
6300L.
6100L.
5900L.
5901L.
5930L.
REX FAULT
4000S
4000S
5900L.
5700L.
5700L.
5600L.
5500L.
5500L.
5470L.
5200L.
5200L.
5500L.
X-12000
5000S
5200L.
5000L.
4700L.
5000L.
4900L.
4800L.
4700L.
4650L.
6000S



DEPOSIT
AMA
X FAULT
6340L.
6300L.
6170L.
6100L.
5900L.
5930L.
5900L.
5200L.
4000S
4500L.
X-10,000
5900L.
5700L.
5500L.
5500L.
5200L.
5500L.
5200L.
4500L.
5000S
5000L.
4700L.
5000L.
4900L.
4800L.
4700L.
4650L.
4650L.
6000S
4522L.
4402L.
X-12,000



LOWER BUKANGAMA
5000S
5200L.
5000L.
4700L.
5000L.
4900L.
4800L.
4700L.
6000S
4650L
4522L.
4402L.
4200L.
4100L
X-14000
Y-12000
7000S
1600W
22XC
20XC
18XC
16XC
14XC
12XC
4522L.
Y-10000
Y-8000



5000L.
4700L.
5000L.
4900L.
4800L.
4700L.
6000S
6000S
4650L.
4650L.
4522L.
4402L.
X-12 000
4301L.
4200L.
4100L.
4050L.
LOWER BUKANGAMA DEPOSIT
7000S
7000S
1600W
22XC
20XC
18XC
16XC
14XC
12XC
10XC
8XC
6XC
4XC
4522L.
Y-8000
Y-6000