3-1-2 Characteristics of head samples

(1) Assay of head samples

Assay results of head samples are shown in Table 3-6.

Table 3-6 Assay of head samples

Component		Hayl as Safil Ore	Rakah Stockwork Ore	Rakah Massive Ore
Au	g/t	0.5	0.5	9.1
Ag	g/t	5.1	1.4	18.8
Cu	%	1.12	1.25	1.60
Pb	%	<0.01	<0.01	0.04
Zn	%	0.22	0.23	0.11
Fe	%	16.52	20.43	35.73
S	%	14.60	10.52	39.87
SiO_2	%	57.89	40.86	19.61
CaO	%	0.04	0.20	0.05
MgO	%	2.35	6.13	0.02
Al_2O_3	%	3.17	10.13	0.35
Mo	ppm	11	<1	4
As	ppm	22	140	2,944
Cd	ppm	6	3	6
Hg	ppm	0.46	0.51	9.03

The results were almost same as the calculated grade of drill cores shown in Table 3-1.

(2) Mineralogical examination of head samples

X-ray diffraction analyses were conducted in mineral identification to estimate mineral constituents of head samples. X-ray diffraction pattern is shown in Appendix 5 and the analytical results expressed in Quartz Index are shown in Table 3-7. Quartz Index (QI) is given from a following equation.

$QI = Im/Iq \times 100$

QI : Quartz Index

Table 3-7 Mineral identification by X-ray diffraction analysis

Identified mineral	Hayl as Safil ore	Rakah stockwork ore	Rakah massive ore
chalcopyrite?	1.08	2.12	0.36
pyrite	5.64	2.36	8.76
marcasite	nil	nil	1.68
hematite	3.92	nil	nil
quartz	33.70	15.13	6.73
chlorite	0.40	4.36	nil

nil : no identification by X-ray diffraction analysis

Table 3-8 Mineral identification by microscopic observation

Identified mineral	Hayl as Safil ore	Rakah stockwork ore	Rakah massive ore
chalcopyrite	○	○	○
covelline	○		○
chalcocite	○		○
bornite	○		
pyrite	○	○	○
sphalerite	○	○	

Table 3-9 Estimate of constituent minerals

Mineral	Chemical formula	Hayl as Safil ore	Rakah stockwork ore	Rakah massive ore
		%	%	%
Chalcopyrite	$CuFeS_2$	3.2	3.6	0.9
Secondary copper minerals		very rare		2.0
Galena	PbS	0.01	0.01	0.05
Sphalerite	ZnS	0.33	0.34	0.16
Pyrite	FeS_2	25.0	17.1	72.7
Marcasite	FeS_2			
Hematite	Fe_2O_3	4.8		
Quartz	SiO_2	38.0	15.0	10.0
Chlorite	$(Mg,Fe^2,Al)_{12}\cdot(Si,Al)_8O_{20}(OH)_{16}$	9.0	39.0	

Im : Maximum X-ray intensity of objective mineral (cps)

Iq : Maximum X-ray intensity of artificial quartz (cps)

Pyrite and chalcopyrite were identified in all samples. Marcasite was identified in Rakah massive ore. As for gangue minerals, quartz was identified in all samples, hematite and chlorite were in Hayl as Safil ore and chlorite was in Rakah stockwork ore.

Mineral identification by X-ray diffraction is invalid when an objective mineral is little content in a sample; therefore microscopical examination should be applied simultaneously. Table 3-8 shows identified minerals by microscopical observation. Secondary copper minerals were found in Hayl as Safil ore and Rakah massive ore.

Mineral composition was estimated from chemical assays and identification of minerals by X-ray diffraction or microscope. Estimated results are shown in Table 3-9. These tables indicate following observation on the ore samples.

Hayl as Safil ore

This ore contained the sulfide minerals approximately 28% weight. The copper minerals identified were chalcopyrite as the major copper mineral, covelline and chalcocite as the secondary minerals in minor quantity. The gangue minerals identified were quartz and hematite.

Rakah stockwork ore

This ore contained the sulfide minerals approximately 21% weight. The copper mineral identified was chalcopyrite with no secondary minerals. The gangue minerals identified were quartz and chlorite.

Rakah massive ore

This ore contained the sulfide minerals approximately 75% weight. Amounts of iron minerals were excessively higher than that of copper minerals in sulfide minerals. Because the ore was highly oxidized and the secondary minerals, namely covelline, chalcocite and bornite were identified as the major copper minerals. A lot of marcasite was identified as much as pyrite.

(3) Work Index

Hardgrove Index (Hd) and Work Index (Wi) which were calculated from Hd are shown in Table 3-10. The table indicates that the Work Index figure of the Rakah stockwork ore was unreasonably higher than the other ore samples. This figure was due to the Hardgrove testing conducted on a sample containing a lot of clay minerals such as chlorite.

Table 3-10 Hardgrove index and work index

Item	Hayl as Safil Ore	Rakah Stockwork Ore	Rakah Massive Ore
Hardgrove Index (Hd)	54	45	56
Work Index (kwh/st)			
Wi [1)]	12.9	15.1	12.5
Wi [2)]	11.5	13.6	11.2

1): given by Ishihara equation ($Wi=400/(Hd)^{0.86}$)
2): given by Bond equation ($Wi=435/(Hd)^{0.91}$)

Table 3-11 Specific gravity of head samples

Hayl as Safil Ore	Rakah Stockwork Ore	Rakah Massive Ore
3.11	3.11	3.97

Table 3-12 Concentration of Ion dissolved from sample

Item		Hayl as Safil Ore	Rakah Stockwork Ore	Rakah Massive Ore
pH		4.8	6.1	4.3
Soluble Ions				
Cu	ppm	0.202	0.014	0.073
Zn	ppm	34.8	28.7	97.0
Fe	ppm	611	301	10,807

(4) **Specific gravity**

Measurement results of specific gravity on the head samples are shown in Table 3-11.

(5) **Assays of soluble ion**

As shown in the Paragraph of 3-1-2 (Mineral examination), the ore is oxidized in some degree and seemed to contain much soluble ions which have harmful effect on flotation. The concentration of these heavy metal ions dissolved from the ore are shown in Table 3-12.

① Rakah stockwork ore

The filtrate from filtering of ground pulp was almost neutral as pH value indicated 6.1, and concentration of soluble ions were rather low.

② Hayl as Safil ore, Rakah massive ore

The filtrate of ground pulp was rather acidic as pH value indicated 4.8 and 4.3 respectively, and concentration of soluble ions was higher than Rakah Stockwork Ore.

③ The concentration of Cu ion was extremely lower than that of Zn or Fe ions.

This result is explained as follows. Dissolved Cu ion is fixed in the place of Zn or Fe on the surface of sphalerite or pyrite and Zn or Fe ions are leached out in replacement. The sphalerite or pyrite coated with copper are easily floatable as well as fresh copper minerals; therefore they will disturb the copper mineral's selectivity in the flotation.

3-1-3 Fundamental flotation tests results

(1) **Hayl as Safil ore test results**

(i) **Copper selective flotation**

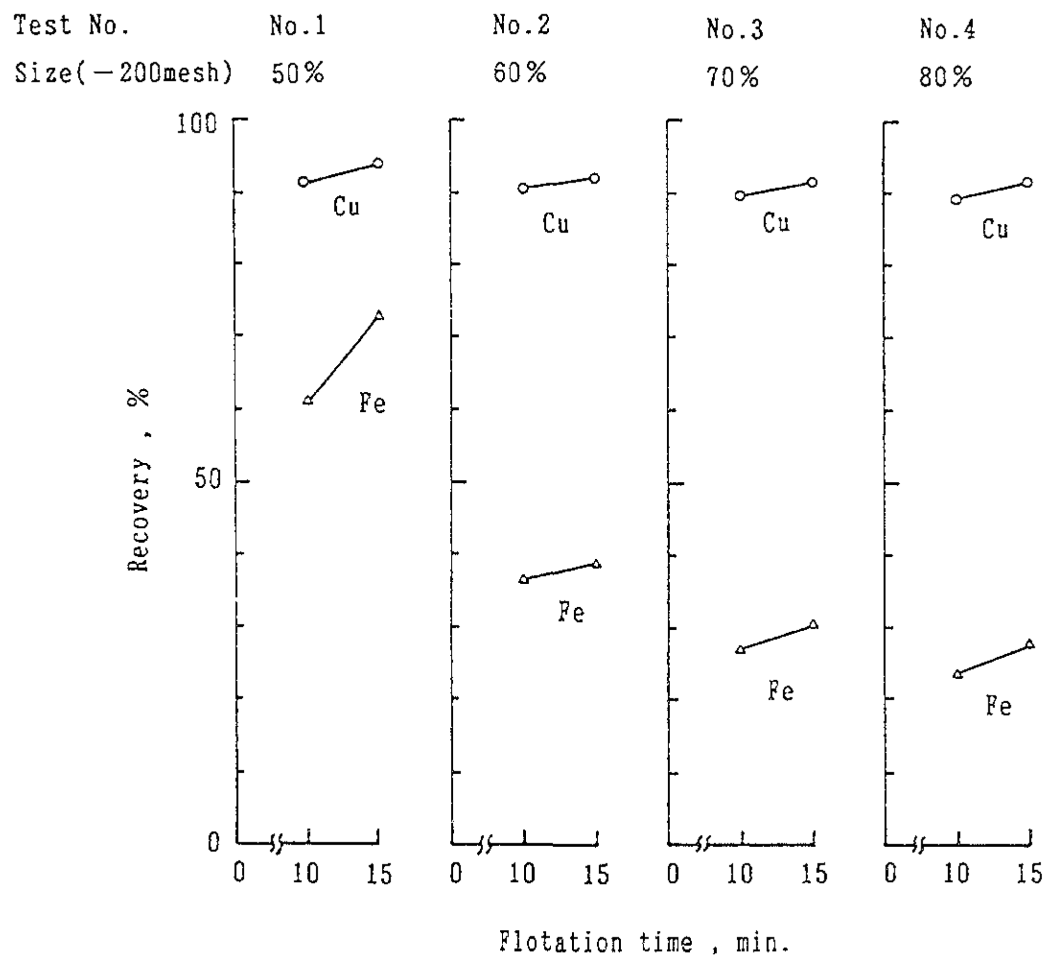
(a) **Comparison tests of grind size**

Flotation tests were performed to determine the optimum grind size under following conditions.

Grind size	minus 200 mesh	50%, 60%, 70%, 80%
KAX dosage	35 g/t	
Lime dosage	2,000 g/t	

Details of the test results are shown in Appendix 6 (Table 1) and Fig. 3-2. The results are summarized as follows.

① The test on a grind size of 50% passing 200 mesh produced rougher concentrate with a grade of 3.43% copper at a high recovery of 94%.



Flotation conditions

pulp density	35%
pH	12.1
KAX	35 g/t
lime	2000 g/t
flotation time	15 min.

Fig. 3-2 Effect of feed size on copper selective flotation of Hayl as Safil ore

② The grind size reduced to 80% passing 200 mesh a grade of rough concentrate increased to 8.17% copper, however, copper recovery slightly decreased to 91.7%.

③ The cause of reduction of copper recovery may be fine copper mineral grain locked in pyrite particles sank to tail.

④ The conclusion drawn from the test results was that optimum grind size seemed to be 80% passing 200 mesh; therefore the following copper selective flotation tests were conducted on this grind size.

(b) Comparison tests of collector

The comparison tests were conducted on four kinds of collectors varying pH value, to find a collector which would produce a high grade concentrate at high copper recovery. The collectors and their dosage were KAX (35 g/t), AP3501 (52.5 g/t), AP3418 (54.1 g/t) and AP404 (61.8 g/t). The initial pulp pH value was adjusted in the range of 8.6 ~ 12.0 by lime addition.

Details of the test results are shown in Appendix 6 (Tables 2,3, 4 and 5) and Fig. 3-3. The results are summarized as follows.

① KAX: Relationship between pH value and copper recovery showed the highest recovery of 91.7% at pH 12.0 and recovery slightly down in pH 10 ~ 11.

② AP3501: Copper recovery was constant figure of 90.3% at pH 9 or more. Pyrite was depressed in the higher pH range.

③ AP3418: The higher pH value rose, the better copper recovery achieved. The separation of copper mineral and pyrite in the higher pH range was not so good as other collector.

④ AP404: The higher pH value rose, the better copper recovery achieved. Pyrite was strongly depressed in the higher pH range.

⑤ The pH value of flotation pulp should be maintained at 12 or more to get high copper recovery.

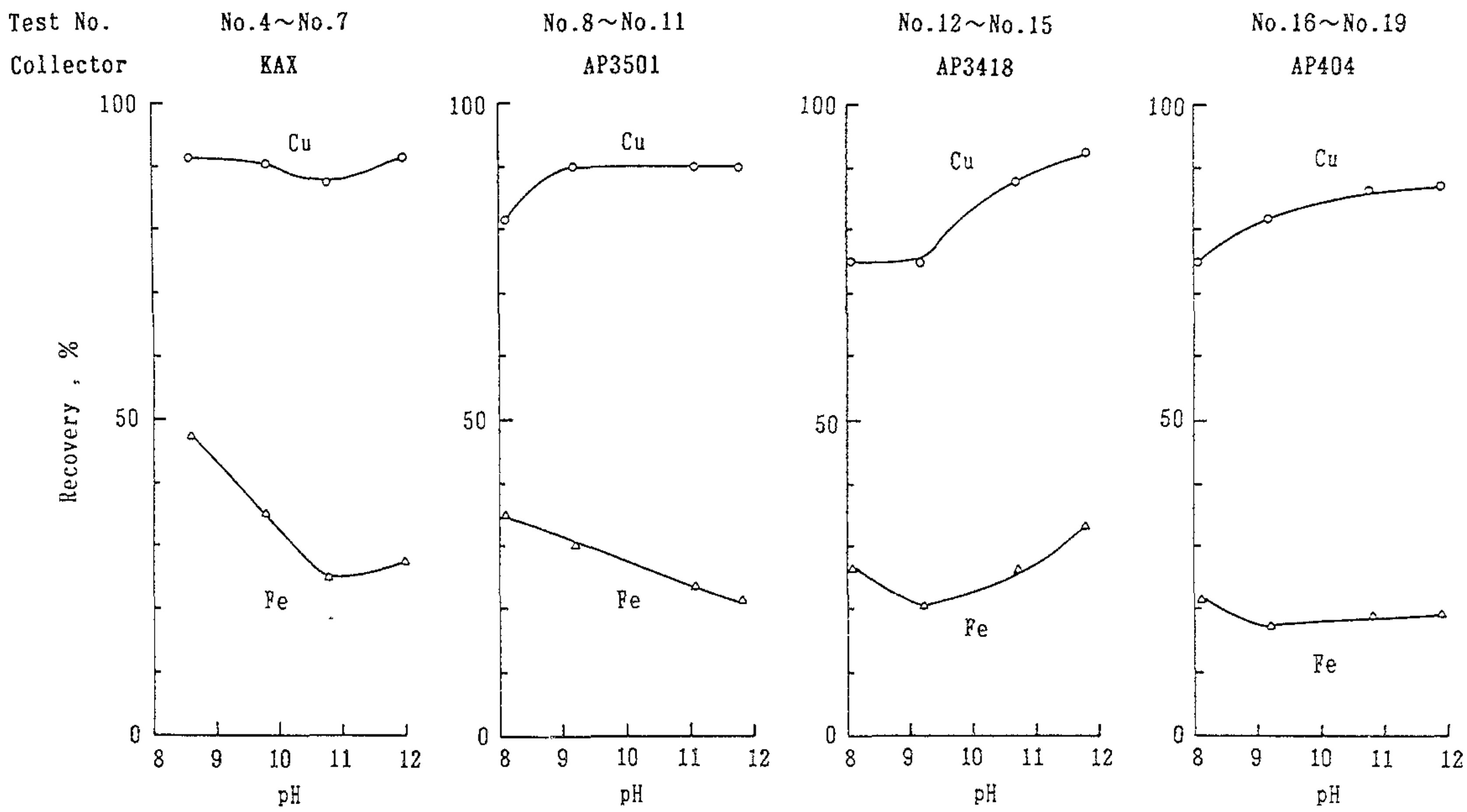


Fig. 3-3 Effect of collector on copper selective flotation of Hayl as Safil ore varying pH value

The results of comparison test between the collectors at pH 12 are shown in Table 3-13 and summarized as follows.

Table 3-13 Comparison of collectors on copper selective flotation of Hayl as Safil ore

Collector	KAX	AP 3501	AP 3418	AP 404
Dosage g/t	35.0	52.5	54.1	61.8
pH value	12.0	11.8	11.8	11.9
Recovery				
Cu %	91.7	90.3	92.7	87.5
Fe %	27.7	21.5	33.0	19.2
Grade of conc.				
Cu %	8.17	10.05	6.79	10.12
Fe %	34.25	33.20	32.80	29.80
Wt of conc. %	13.0	10.7	15.7	10.0

① KAX showed strong collecting power for copper minerals, while the other collector needed about 1.5 times of dosage as that of KAX to get same copper recovery.

② AP3501 showed superior ability in separation of copper minerals and pyrite, however, KAX which has strong collecting power should be adopted to get high copper recovery in rougher flotation.

(ii) Bulk and differential flotation

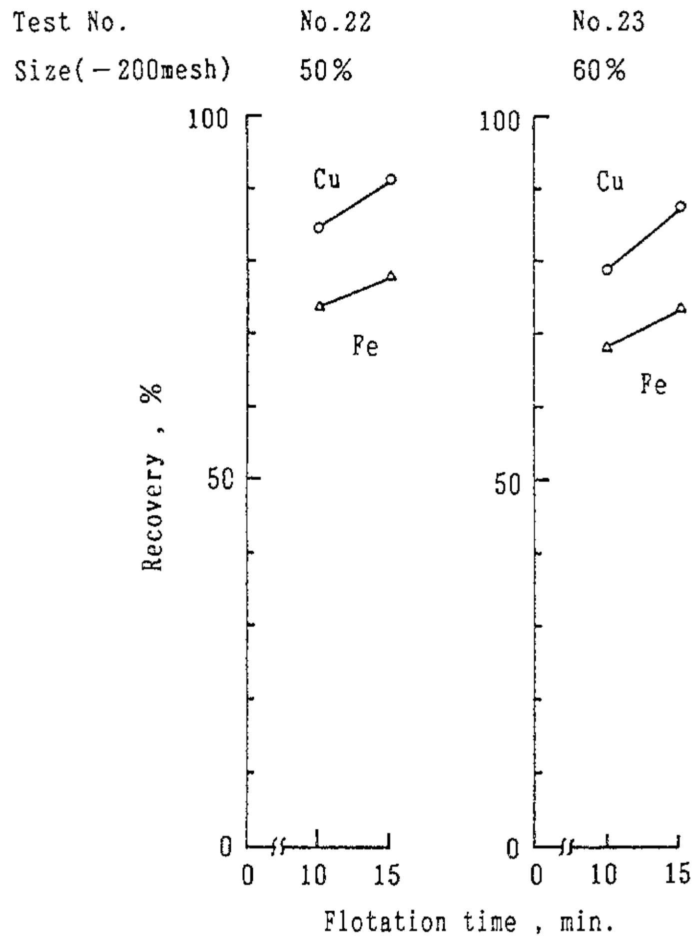
(a) Comparison tests of grind size in bulk flotation

Bulk flotation tests were performed to determine optimum grind size under following conditions.

Grind size	minus 200 mesh	50%, 60%
KAX dosage	35 g/t	
Pulp pH value	7.2	

Details of the test results are shown in Appendix 6 (Table 7) and Fig. 3-4. The results are summarized as follows.

① In the grind size of 50% passing 200 mesh, rougher concentrate with a grade of 2.94% copper was produced at recovery of 91.2%. The grind size was reduced to 60% passing 200 mesh, copper recovery was 3.5% lower than the former test.



Flotation conditions

pulp density	35 %
KAX	35 g/t
flotation time	15 min.
pH	7.2 - 7.4

Fig. 3-4 Effect of feed size on bulk flotation of Hayl as Safil ore

② From the test results the optimum grind size seemed to be 50% passing 200 mesh; therefore the following bulk flotation tests were conducted on this grind size.

(b) Comparison tests of pH value in bulk flotation

Bulk flotation tests were performed to determine optimum pH value under following conditions.

Pulp pH value	4.0, 7.2, 8.0, 10.0, 12.1	
Grind size	minus 200 mesh	50%
KAX dosage	35 g/t	

Details of the test results are shown in Appendix 6 (Table 8) and Fig. 3-5. The results are summarized as follows.

① Copper recovery of 93.7% at pH 4 and 94% at pH 12.1 was achieved in bulk flotation, however, copper recovery was rather lower in the range between pH 4 and pH 12.1.

② Pyrite was depressed in the higher pH range.

③ From the test results the optimum pH value in bulk flotation seemed to be 12.0. No. 1 test produced a bulk concentrate with a grade of 3.43% copper at 94.0% recovery in the higher pH condition.

(c) Comparison test of flotation time in bulk flotation

No. 27 flotation test was performed to study the relation between copper recovery and flotation time with KAX dosage following condition.

KAX dosage	60 g/t
Flotation time	30 minutes

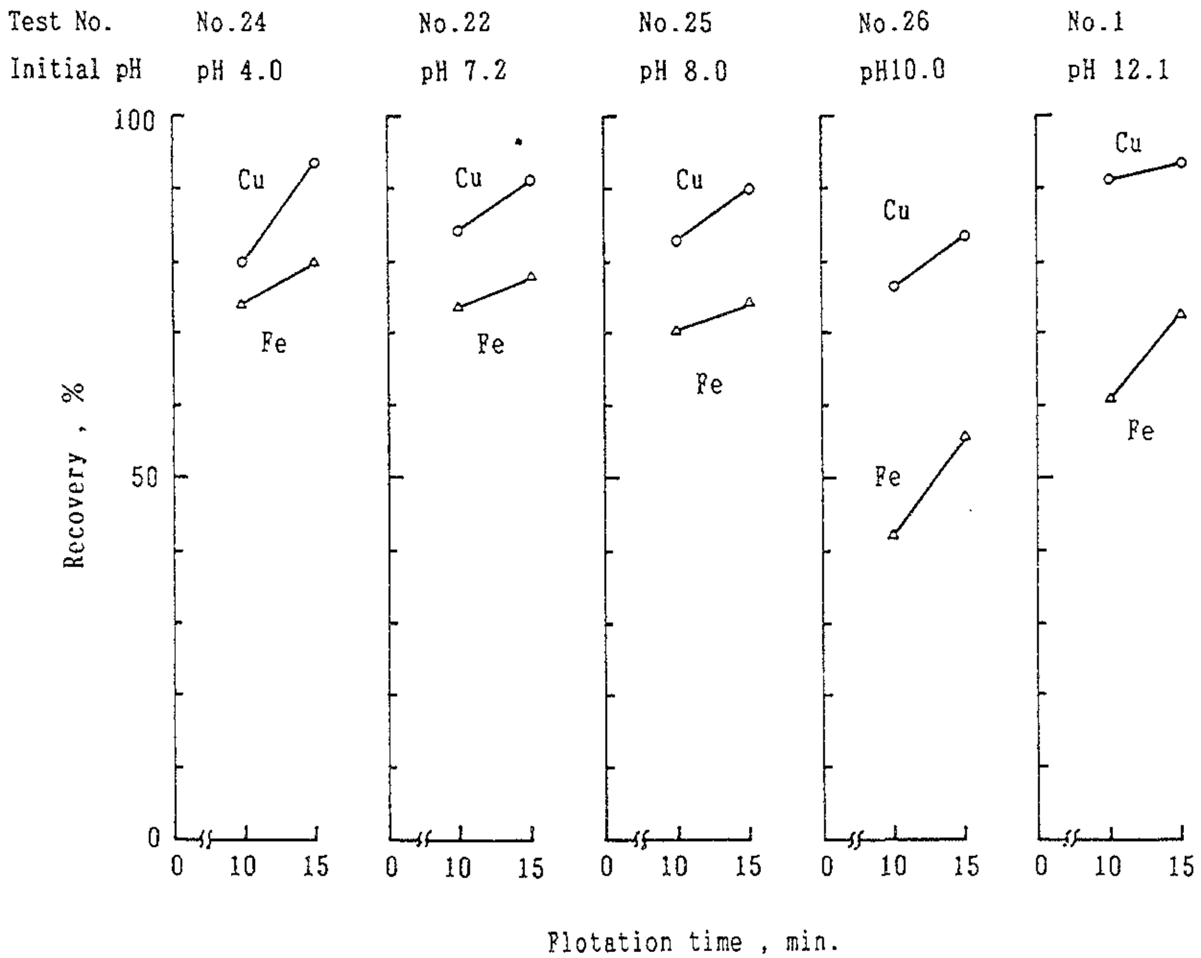
The details of the test result is shown in with No. 1 test result (flotation time 15 minutes) in Appendix 8 (Table 9) and Fig. 3-6. The results are summarized as follows.

① No. 27 test produced a bulk concentrate with a grade of 2.90% copper at 96.8% recovery versus 3.43% copper at 94.0% recovery for No. 1 test. By extending of flotation time, copper grade of concentrate was 0.53% lower and copper recovery was 2.8% higher than the former test.

② The condition of KAX dosage 60 g/t and flotation time 30 minutes was to be adequate for bulk flotation to get a higher copper recovery.

(d) Comparison tests of pH value in cleaner flotation

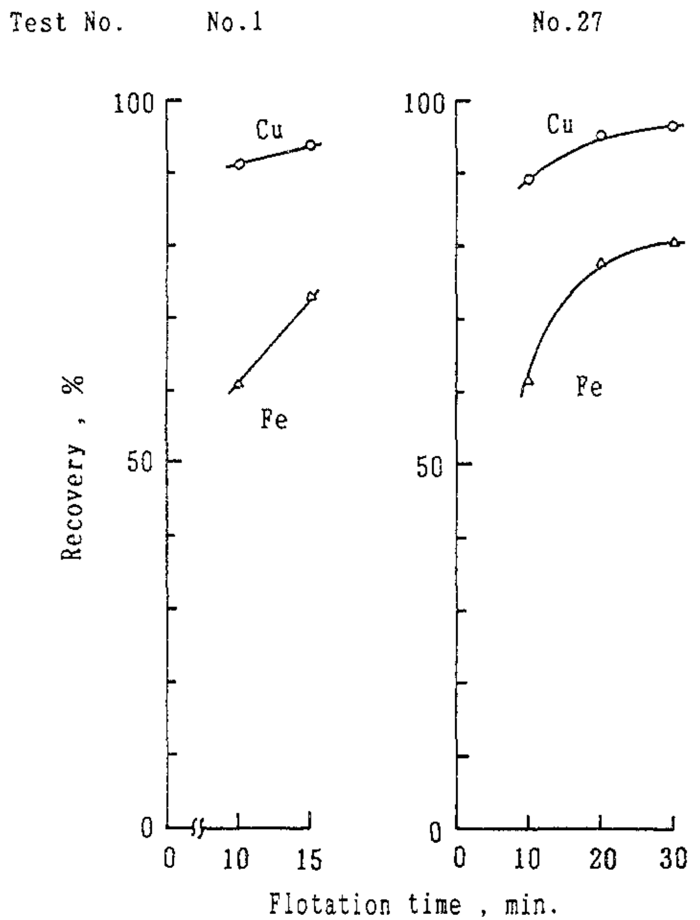
Cleaner flotation tests were performed on rougher concentrate to determine optimum pH value under following conditions.



Flotation conditions

Feed size (-200 mesh)	50 %
pulp density	35 %
KAX	35 g/t
flotation time	15 min.

Fig. 3-5 Effect of pH value on bulk flotation of Hayl as Safil ore



Flotation conditions

feed size (-200mesh)	50 %	
pulp density	35 %	
Test No.	No.1	No.27
KAX	35 g/t	60 g/t
pH	12.1	11.6
flotation time	15 min.	30 min.

Fig. 3-6 Recovery - flotation time curves on bulk flotation of Hayl as Safil ore varying KAX dosage

Pulp pH value	10.6, 11.4, 12.3	
Regrind size	minus 200 mesh	90%, 95%
Cleaning stage	2	

The details of the test results are shown in Appendix 6 (Table 10) and Fig. 3-7. The results are summarized as follows.

① No. 30 test under the condition of regrind size of 90% passing 200 mesh and high pH value of 12.3 produced copper concentrate with a grade of 12.76% copper at 73.65% recovery.

② Regrind size was reduced to 95% passing 200 mesh, however, there was no significant improvement in recovery.

③ Copper recovery of all the tests was not so high. It was caused by the shortage of cleaning flotation time.

④ The number of cleaning stage and pH value should be increased in cleaning flotation in order to get high grade copper concentrate.

(2) Rakah stockwork ore test results

(i) Copper selective flotation

(a) Comparison tests of grind size

Flotation tests were performed to determine optimum grind size under following conditions.

Grind size	minus 200 mesh	50%, 60%, 70%, 80%
KAX dosage	30 g/t	

The details of the test results are shown in Appendix 6 (Table 11) and Fig. 3-8. The results are summarized as follows.

① The test on a grind size of 50% passing 200 mesh produced rougher concentrate with a grade of 5.03% copper at a high recovery of 95.1%

② The Copper recovery was 95% in all the grind size. As the grind size reduced, the separation of copper mineral and pyrite became improved, and the grade of concentrate rose up. The rougher concentrate with a grade of 7.50% copper was produced at the grind size of 80% passing 200 mesh.

③ The conclusion drawn from the test results was that optimum grind size seemed to be 80% passing 200 mesh; therefore the following copper selective flotation tests were conducted on this grind size.

(b) Comparison tests of collector

The comparison tests were conducted on four kinds of collectors varying pH value, to find a collector which would produce a high grade concentrate at high copper recovery. The collectors

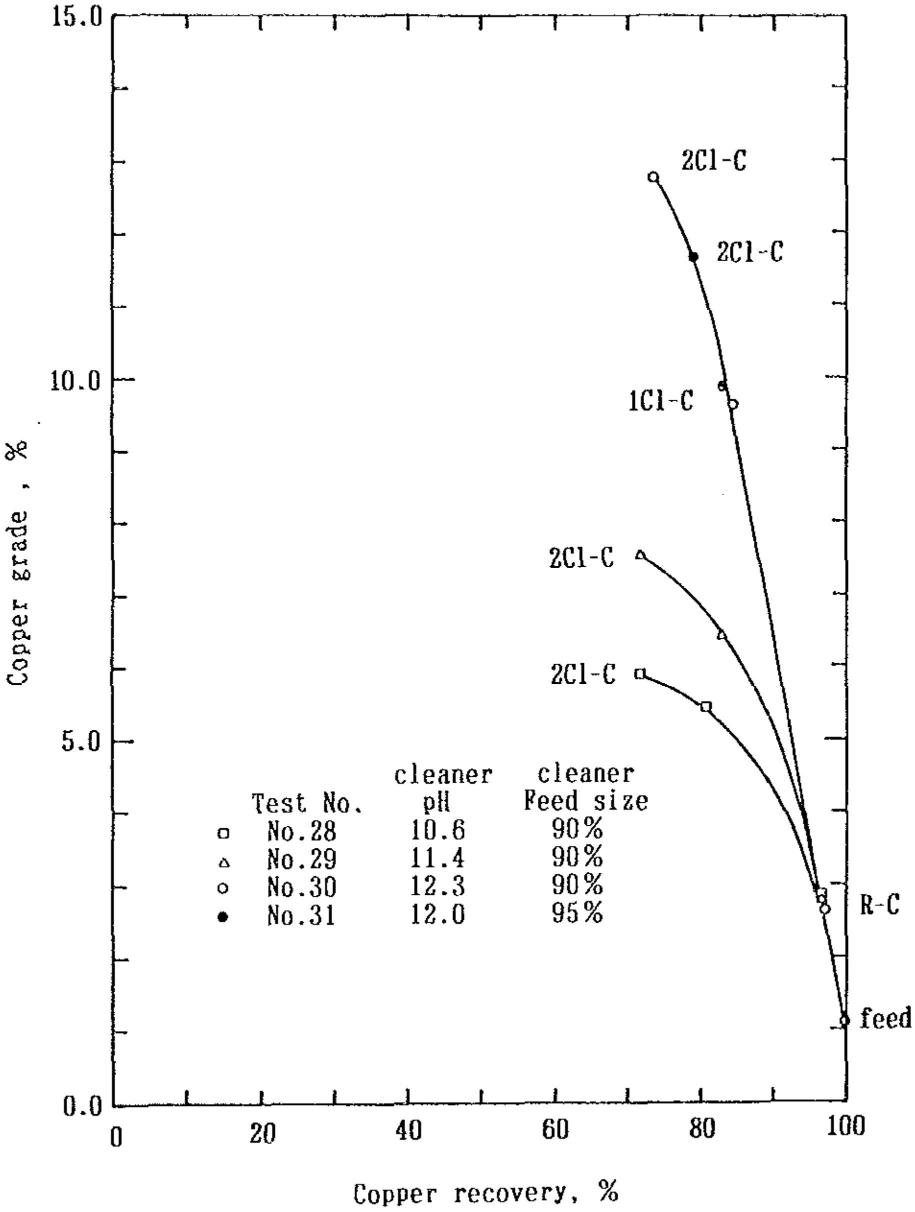
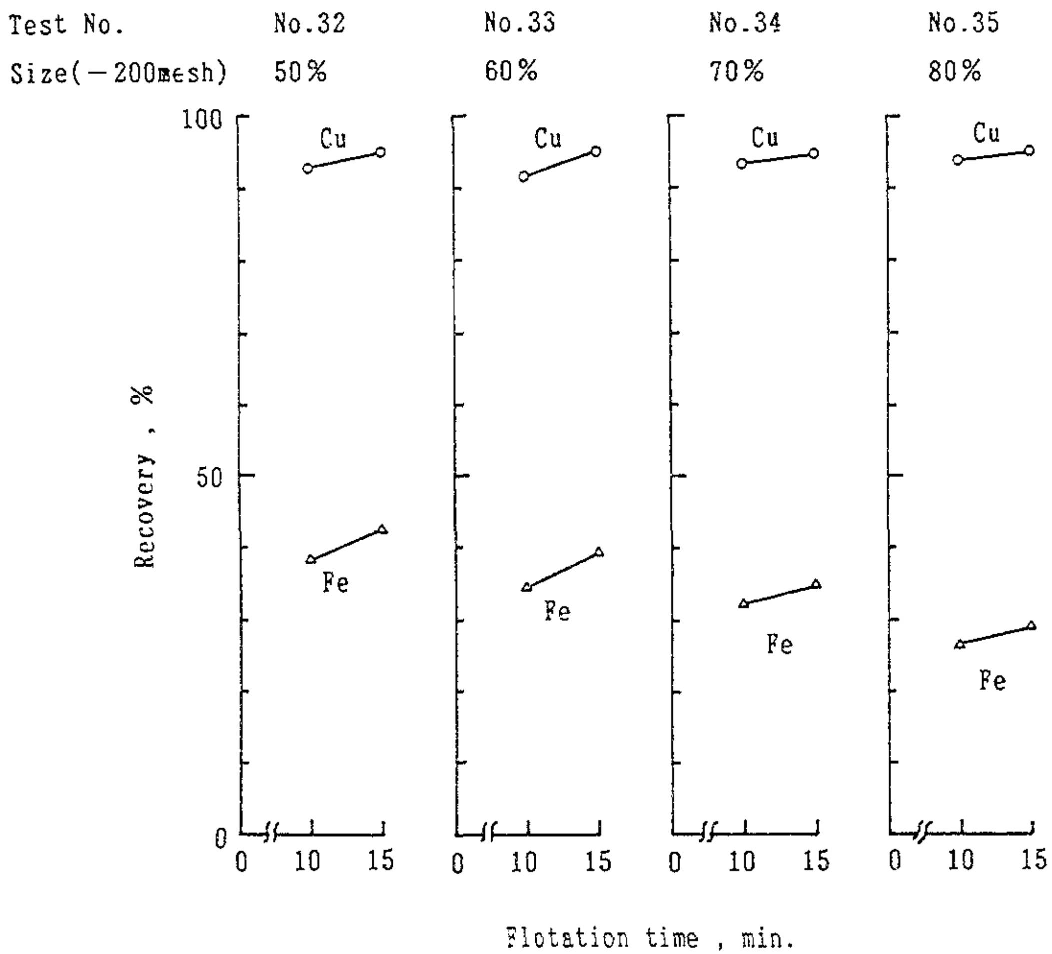


Fig. 3-7 Copper grade - copper recovery curves on bulk rougher/cleaner flotation of Hayl as Safil ore



Flotation conditions

pulp density	35%
pH	10.8
KAX	30 g/t
lime	2000 g/t
flotation time	15 min.

Fig. 3-8 Effect of feed size on copper selective flotation of Rakah stockwork ore

evaluated and their dosage were KAX (30 g/t), AP3501 (46.7 g/t), AP3418 (46.4 g/t) and AP404 (46.4 g/t). The initial pH value of pulp was adjusted in the range of 8.0 ~ 12.0 by lime addition.

The details of the test results are shown in Appendix 6 (Tables 12, 13, 14 and 15) and Fig. 3-9. The results are summarized as follows.

① The copper recovery of 95% ~ 96% was almost same on any kind of collector, however, high copper grade of rougher concentrate was produced at high pH value.

② As the pH values of pulp rose up to 12.2 or more in the flotation using AP 3501 as the collector, the froth became abnormally soften. This reagent could not be used in high pH range on the ore containing clay minerals such as chlorite.

The results of comparison test between the collectors at pH 10.6 are shown in Table 3-14 and summarized as follows.

Table 3-14　Comparison of collectors on copper selective flotation of Rakah stockwork ore

Collector	KAX	AP 3501	AP 3418	AP 404
Dosage g/t	30.0	46.7	46.4	46.4
pH value	10.6	10.6	10.3	11.0
Recovery				
Cu %	95.3	95.1	96.0	93.5
Fe %	29.2	20.9	26.1	13.7
Grade of conc.				
Cu %	7.50	9.11	7.54	13.26
Fe %	36.81	32.26	34.61	30.14
Wt of conc. %	15.9	13.0	16.0	8.9

① KAX showed strong collecting power for copper minerals while the other collector needed about 1.5 times of dosage as that of KAX to get same copper recovery.

② AP3501 showed superior ability in separation of copper minerals and pyrite, however, this reagent could not be used in high pH range on this ore.

KAX which has strong collecting power should be used to get high copper recovery in rougher flotation.

(ii) Bulk and differential flotation

Bulk flotation tests were performed to determine optimum grind size under following conditions.

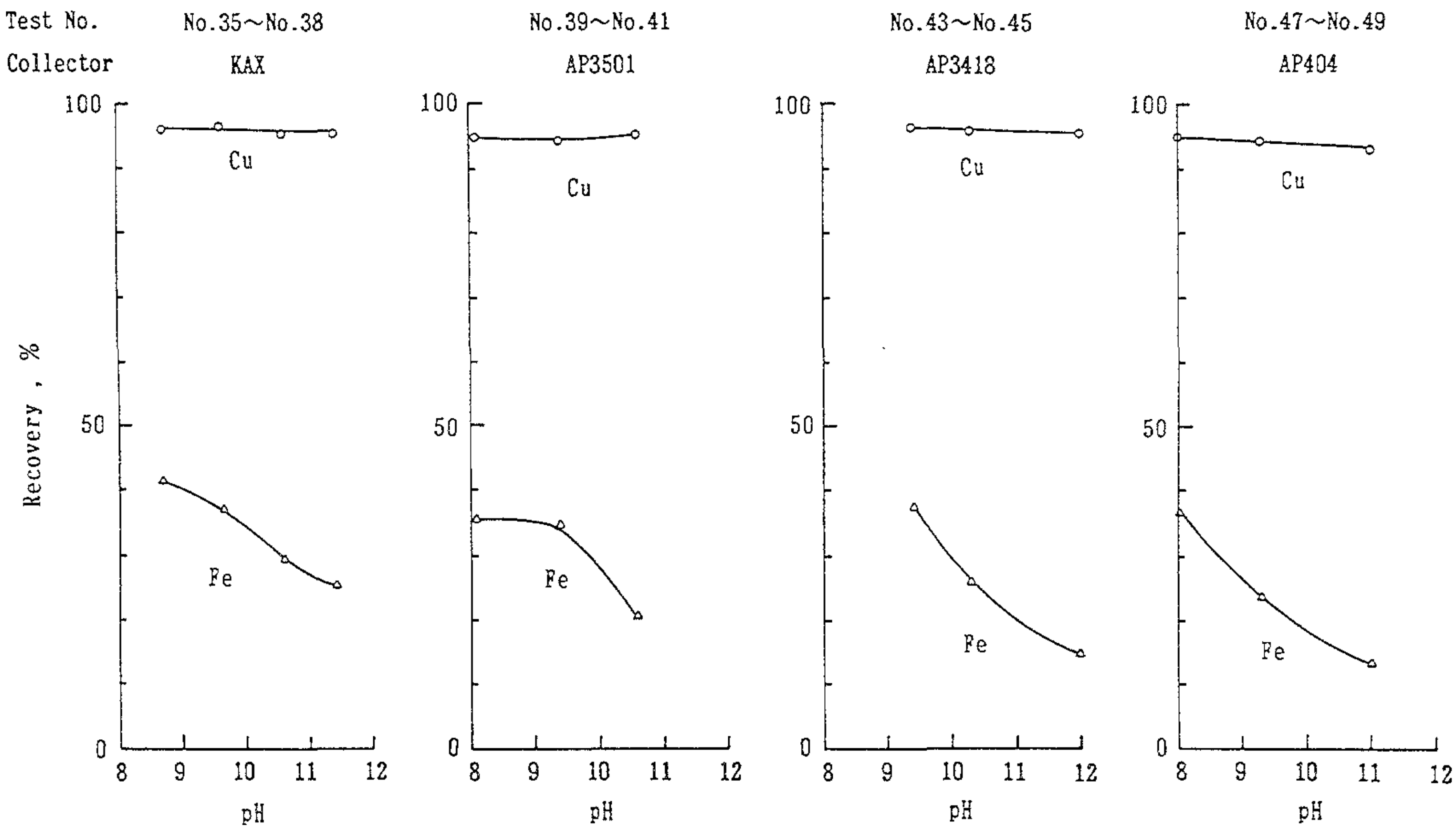


Fig. 3-9 Effect of collector on copper selective flotation of Rakah stockwork ore varying pH value

Grind size	minus 200 mesh	50%, 60%
KAX dosage	30 g/t	
Pulp pH value	7.0	

Details of the results are shown in Appendix 6 (Table 17) and Fig. 3-10. The results are summarized as follows.

① In the grind size of 50% passing 200 mesh rougher concentrate with a grade of 4.26% copper produced at recovery of 95.1%. The grind size reduced to 60% passing 200 mesh the result was almost same as that of the former test.

② From the test results the optimum grind size seemed to be 50% passing 200 mesh; therefore the following bulk flotation tests were conducted on this grind size.

(b) Comparison tests of pH value in bulk flotation

Bulk flotation tests were performed to determine the optimum pH value under following conditions.

Pulp pH value	3.9, 7.0, 8.0, 9.0	
Grind size	minus 200 mesh	50%
KAX dosage	30 g/t	

The details of the test results are shown in Appendix 6 (Table 18) and Fig. 3-11. The results are summarized as follows.

① Copper recovery of 93.5% at pH 3.9 and 95.1% at pH 10.8 was achieved in bulk flotation, however, copper recovery was slightly lower in the range between pH 3.9 and pH 10.8. Pyrite was depressed in the higher pH range.

② From the test results the optimum pH value in bulk flotation seemed to be 11.0. No. 32 test produced a bulk concentrate with a grade of 5.03% copper at 95.1% recovery.

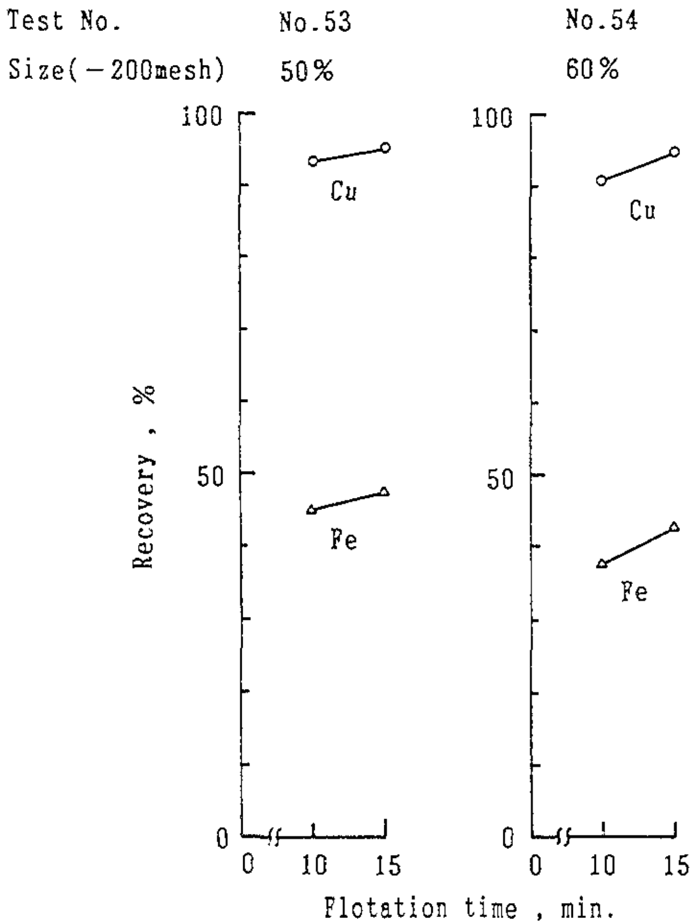
(c) Comparison test of flotation time in bulk flotation

No. 58 flotation test was performed to study the relation between copper recovery and flotation time with KAX dosage under following condition.

KAX dosage	45 g/t
Flotation time	30 minute

The details of the test results are shown with No. 32 test result (flotation time 15 minute) in Appendix 6 (Table 19) and Fig. 3-12. The results are summarized as follows.

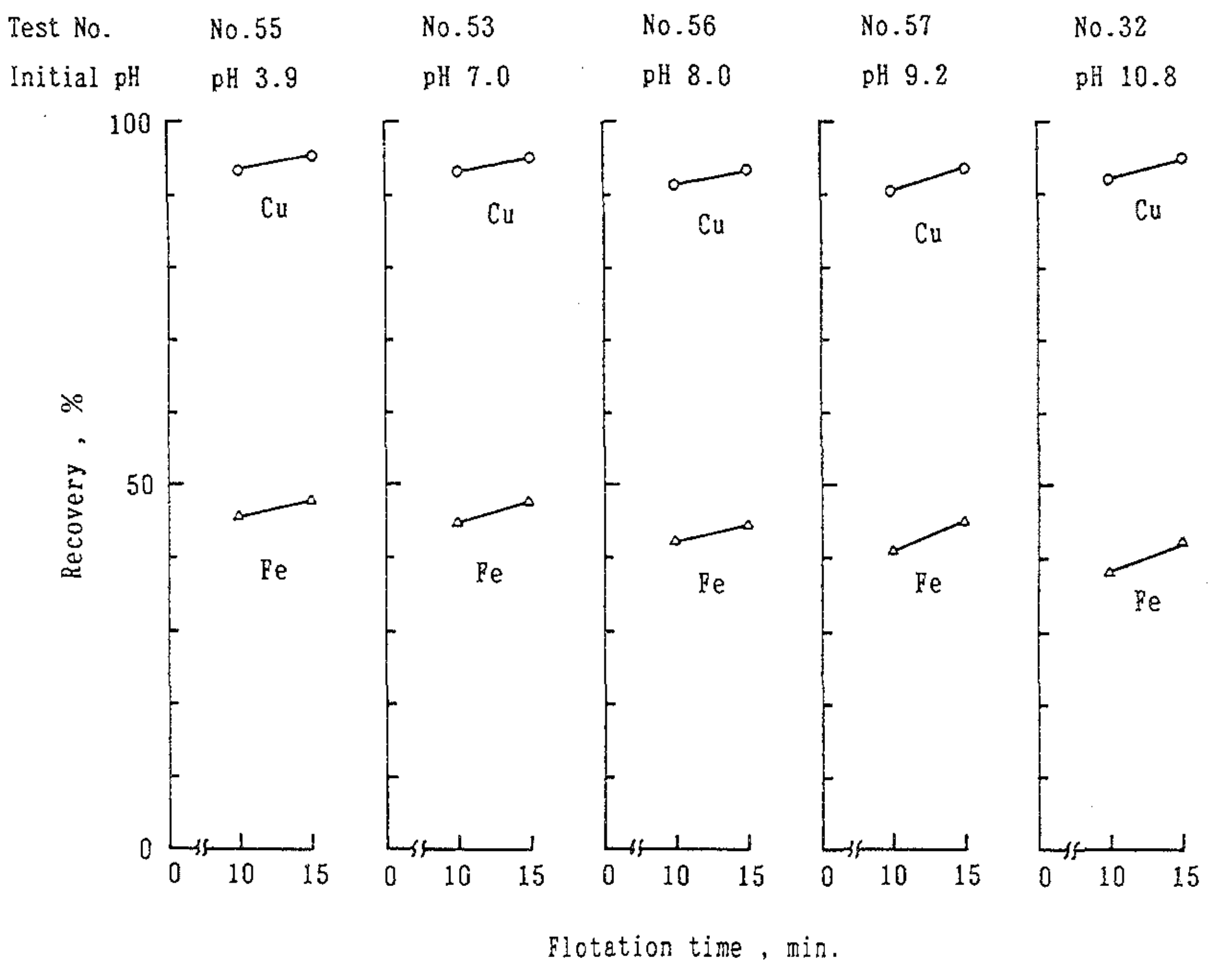
① No. 58 test produced a bulk concentrate with a grade of 4.57% copper at 96.64% recovery versus 5.03% copper at 95.1% recovery for No.32 test. By extending flotation time, the



Flotation conditions

pulp density	35 %
KAX	30 g/t
flotation time	15 min.
pH	7.0 - 7.3

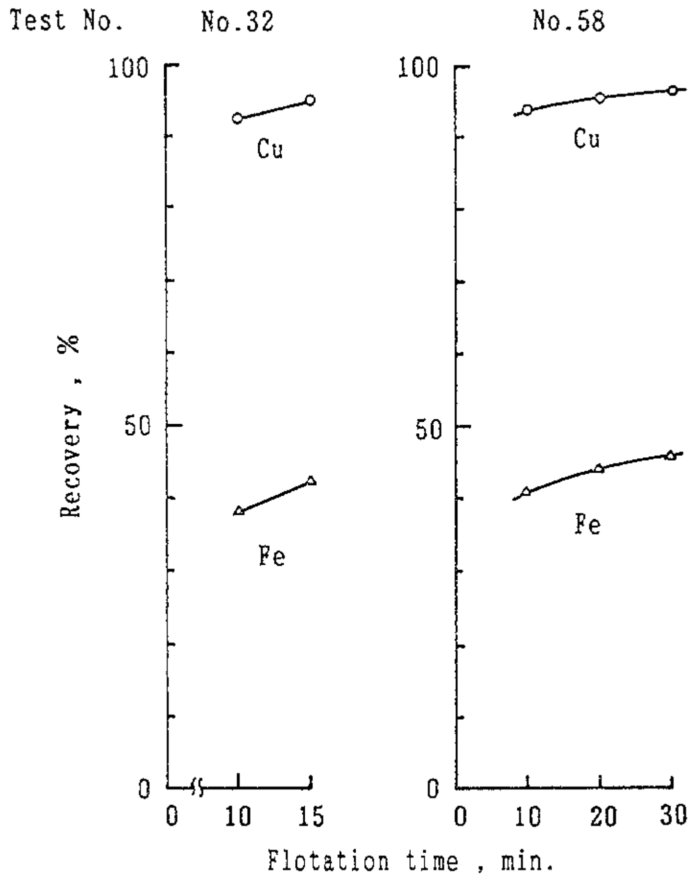
Fig. 3-10 Effect of feed size on bulk flotation of Rakah stockwork ore



Flotation conditions

Feed size (-200 mesh)	50 %
pulp density	35 %
KAX	30 g/t
flotation time	15 min.

Fig. 3-11 Effect of pH value on bulk flotation of Rakah stockwork ore



Flotation conditions

feed size (-200mesh)	50 %	
pulp density	35 %	
Test No.	No.32	No.58
KAX	30 g/t	45 g/t
pH	10.8	11.0
flotation time	15 min.	30 min.

Fig. 3-12 Recovery - flotation time curves on bulk flotation of Rakah stockwork ore varying KAX dosage

copper grade of concentrate was 0.46% lower and copper recovery was 1.54% higher than the former test.

② The condition of KAX dosage 45 g/t and flotation time 30 minutes was to be adequate for bulk flotation to get the higher copper recovery.

(d) Comparison tests of pH value in cleaner flotation

Cleaner flotation tests were performed to determine the optimum pH value under following conditions.

Pulp pH value	10.2, 11.4, 12.2	
Regrind size	minus 200 mesh	90%, 95%
Cleaning stage	2	

Details of the test results are shown in Appendix 6 (Table 20) and Fig. 3-13. The results are summarized as follows.

① No. 61 test under the condition of 90% passing 200 mesh and the high pH value of 12.2 produced copper concentrate with a grade of 13.05% copper at 76.8% recovery.

② No. 62 test under the condition of regrind size of 95% passing 200 mesh and the high pH value of 12.2 produced copper concentrate with a grade of 14.18% copper at the high recovery of 79.50%.

③ The optimum regrind size seemed to be 95% passing 200 mesh.

④ Copper recovery of all the tests was not so high. It was caused by the shortage of cleaning flotation time.

⑤ The number of cleaning stage and pH value should be increased in cleaning flotation in order to get high grade copper concentrate.

(3) Rakah massive ore test results

(i) Copper selective flotation

(a) Preliminary test

As shown in section 3-1-2, the characteristics and constituents of Rakah massive ore were extremely differed from the other ores. The following preliminary tests were conducted as this ore gave a particular performance.

The details of the test results are shown in Appendix 6 (Table 21) and Fig. 3-14.

No. 63 test: This test was conducted on the Rakah massive ore sample under the following conditions.

Grind size	minus 200 mesh	80%
KAX dosage	200 g/t	

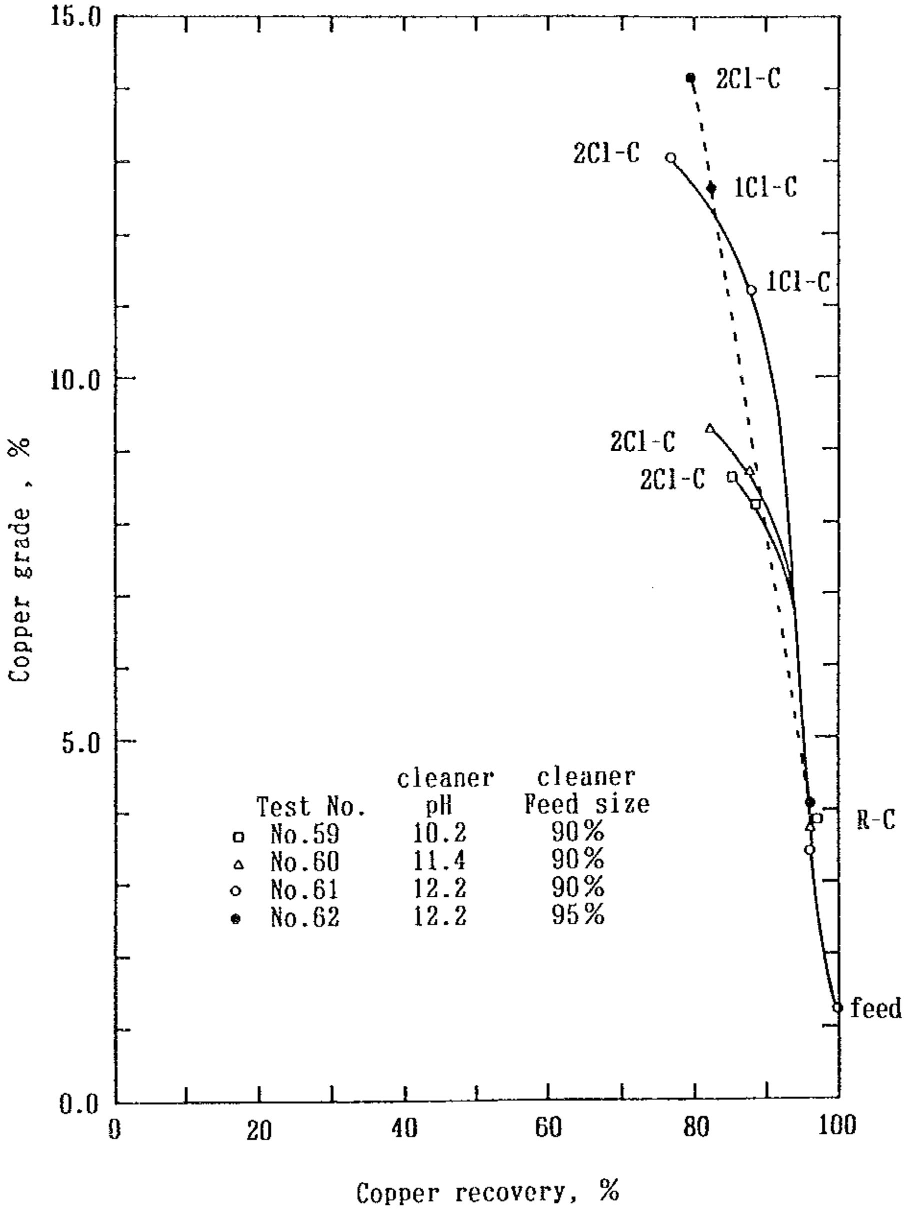


Fig. 3-13 Copper grade - copper recovery curves on bulk rougher/cleaner flotation of Rakah stockwork ore

Test No.	No.63	No.64	No.65	No.66
Size(-200mesh)	80%	80%	94%	94%
KAX , g/t	200	250	250	250
Lime , kg/t				
Grinding	0	0	0	12.5
Flotation	11.6	12.0	15.6	6.1

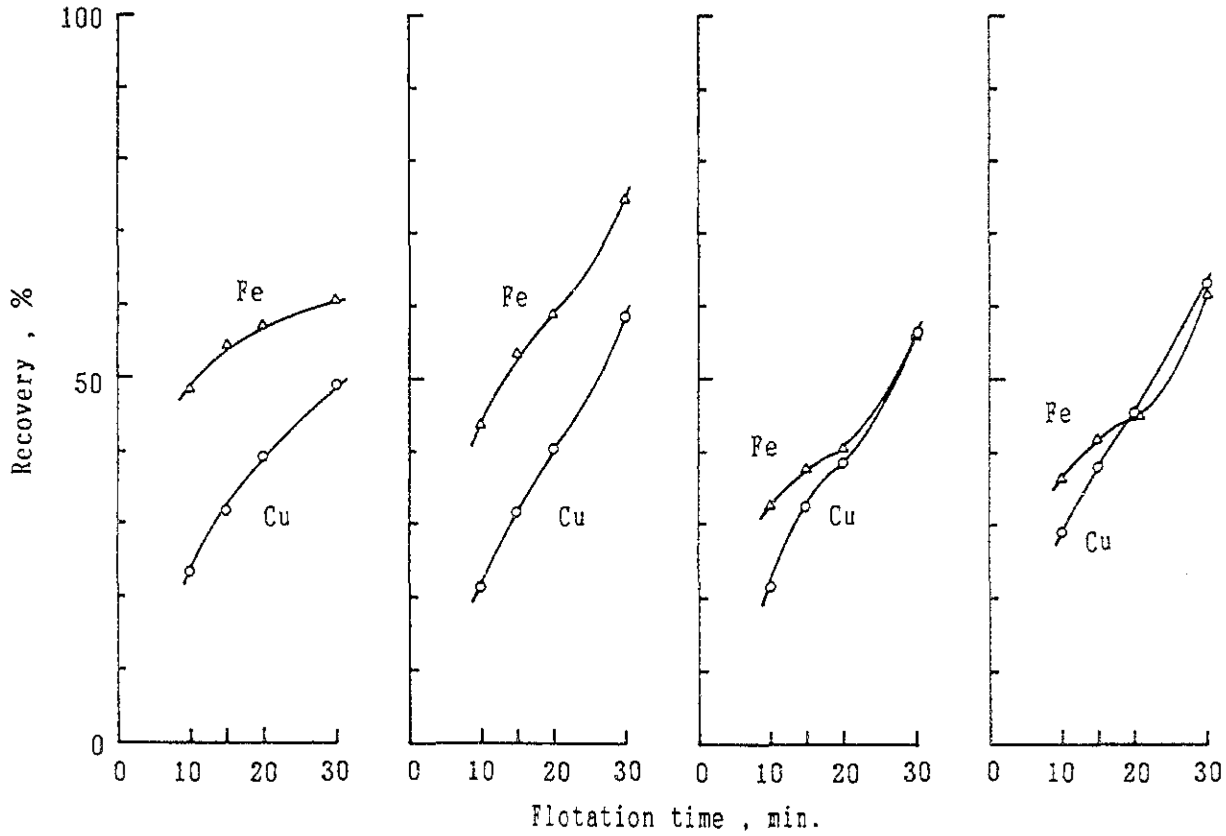


Fig. 3-14 Preliminary tests on copper selective flotation of Rakah massive ore

Lime dosage	11,600 g/t
Pulp pH value	11.1
Flotation time	30 minutes

The test results are summarized as follows.

① This test produced rougher concentrate and three scavenger concentrate. It was abnormal result from the metallurgical knowledge that copper grade of rougher concentrate was lower than that of any other scavenger concentrate.

② It was due to the floatability of copper mineral was lower than that of iron sulfide mineral. As the copper mineral was oxidized, iron sulfide mineral was activated with the copper ion delivered from the oxidization of copper minerals.

No. 64 test: This test was performed to know an effect of increasing the collector dosage from 200 g/t to 250 g/t. The test result shows that there was no significant improvement in separation.

No. 65 test: This test was performed to know an effect of reducing the grind size in percentage minus 200 mesh from 80% to 94%. The test result shows that there was no significant improvement in copper recovery, however, iron recovery was decreased markedly. This result means that the liberation between copper mineral and iron sulfide mineral was improved.

No. 66 test: 12.5 kg/t of lime was added to ball mill and adjusted to 11.0 with additional lime. The pulp was conditioned with selected dosage of collector and frother followed by flotation same as No. 65 test. The test result showed that floatability of copper was mineral improved notably and it's recovery rose to 63.2%. As the addition of lime in ball mill would prevented the activation on the surface of iron sulfide mineral by fixing of copper ion.

(b) Comparison tests of grind size

Flotation tests were performed to determine grind size under the following conditions.

Grind size	minus 200 mesh	50%, 60%, 70%, 80%, 94%
KAX dosage	200 g/t	
Pulp pH value	11.1	
Flotation time	30 minute	

The details of the test results are shown in Appendix 6 (Table 22) and Fig. 3-15. The results are summarized as follows.

The copper recovery increased as the grind size reduced 70% or more minus 200 mesh, though it reached to the limit of 72.8% at 94% minus 200 mesh.

As the cause of the unsatisfactory separation between copper minerals and iron sulfide minerals in previous tests seemed to be excess addition of KAX, the following tests were conducted on the condition of collector decreased from 200 g/t to 150 g/t. The details of test results are shown in Appendix 6 (Table 23) and Fig. 3-16.

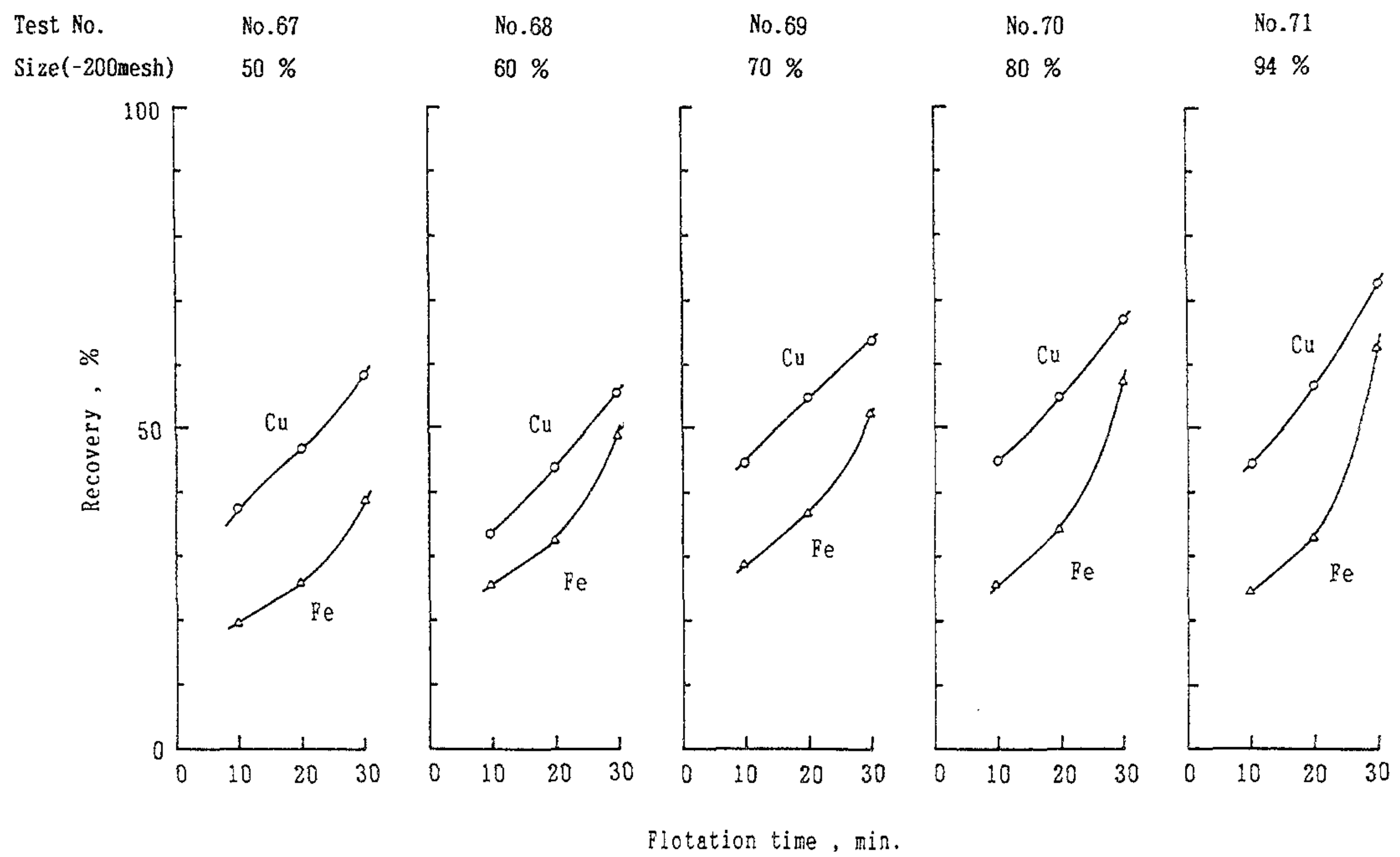


Fig. 3-15 Effect of feed size on copper selective flotation of Rakah massive ore

The separation between copper minerals and iron sulfide mineral was improved by decreasing the dosage of KAX, though copper recovery fell to 68.8%.

The tests were conducted on the ore sample ground to 99% passing 200 mesh and KAX dosage of 120 g/t. The details of test results are shown in Appendix 6 (Table 23) and Fig. 3-16 and are summarized as follows.

① The flotation rate of the ore sample at 99% passing 200 mesh was extremely lower than that of 94% passing 200 mesh. The copper recovery was only 51.5% and there was no significant improvement in separation of copper minerals and iron sulfide mineral.

② The following copper selective flotation tests on Rakah massive ore were conducted at a primary grind of 94% passing 200 mesh.

(c) **Comparison tests of collectors**

The comparison tests were conducted on four kinds of collectors varying pH value, to find a collector which would produce a high grade concentrate at high copper recovery. The collectors and their dosage were KAX (150 g/t), AP3501 (204.4 g/t), AP3418 (202.9 g/t) and AP404 (202.9 g/t). The initial pH value of pulp was adjusted in the range of 7.4 ~ 11.2 by lime addition.

Details of the test results are shown in Appendix 6 (Table 24, 25, 26, 27) and Fig. 3-17 and summarized as follows.

① The recovery vs pH value curve of all kinds of collector except AP 404 shows same tendency that all sulfide minerals floated at pH value nearly 7 and the separation between copper minerals and iron sulfide mineral was improved with rising of pH value.

② Copper recovery of AP 404 was slightly lower than the other collector, however, iron recovery was extremely low in all pH ranges.

③ The pH value should be maintained at 11.2 or more to depress iron sulfide mineral in all the collector use.

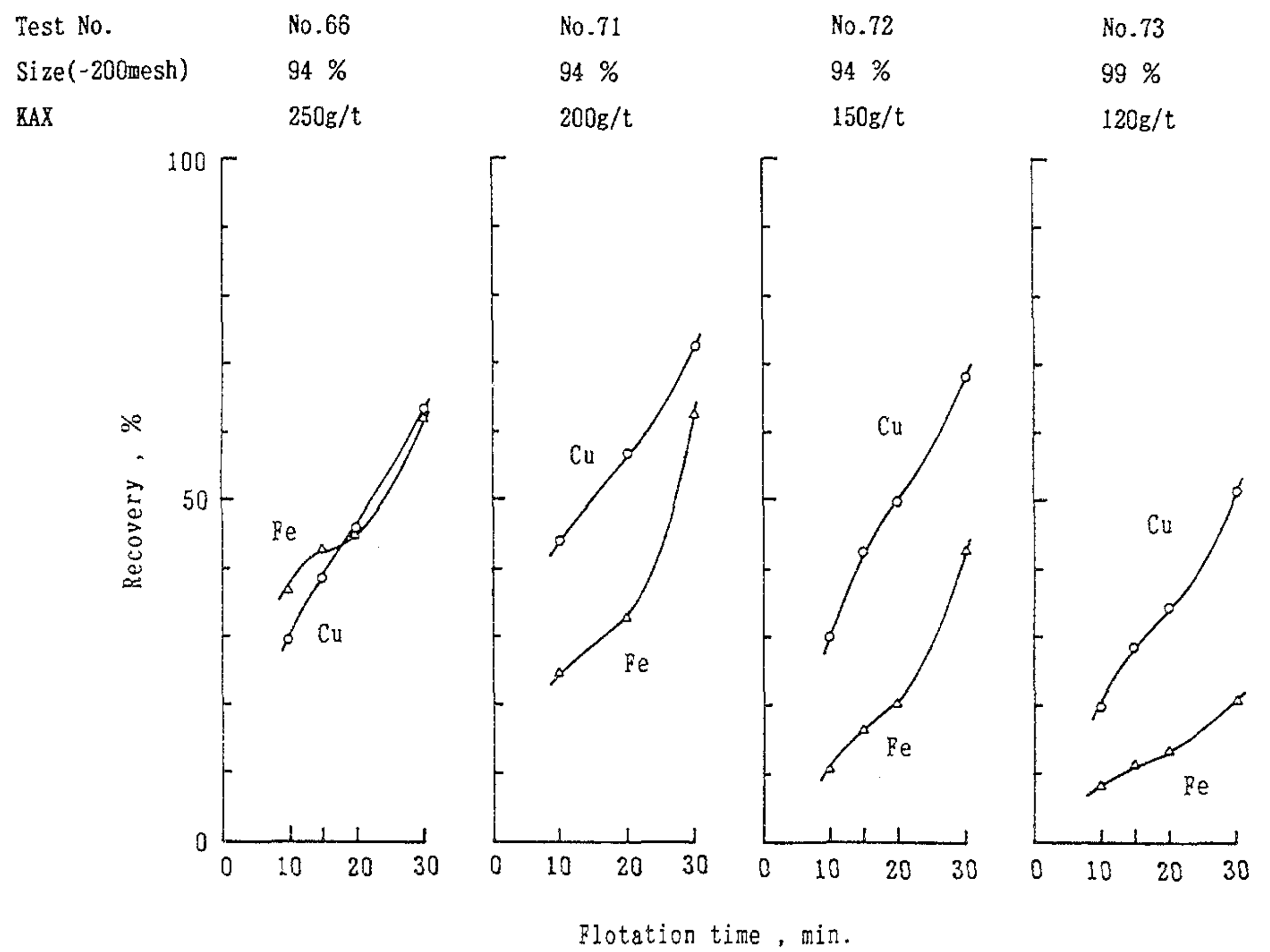


Fig. 3-16 Effect of KAX dosage on copper selective flotation of Rakah massive ore

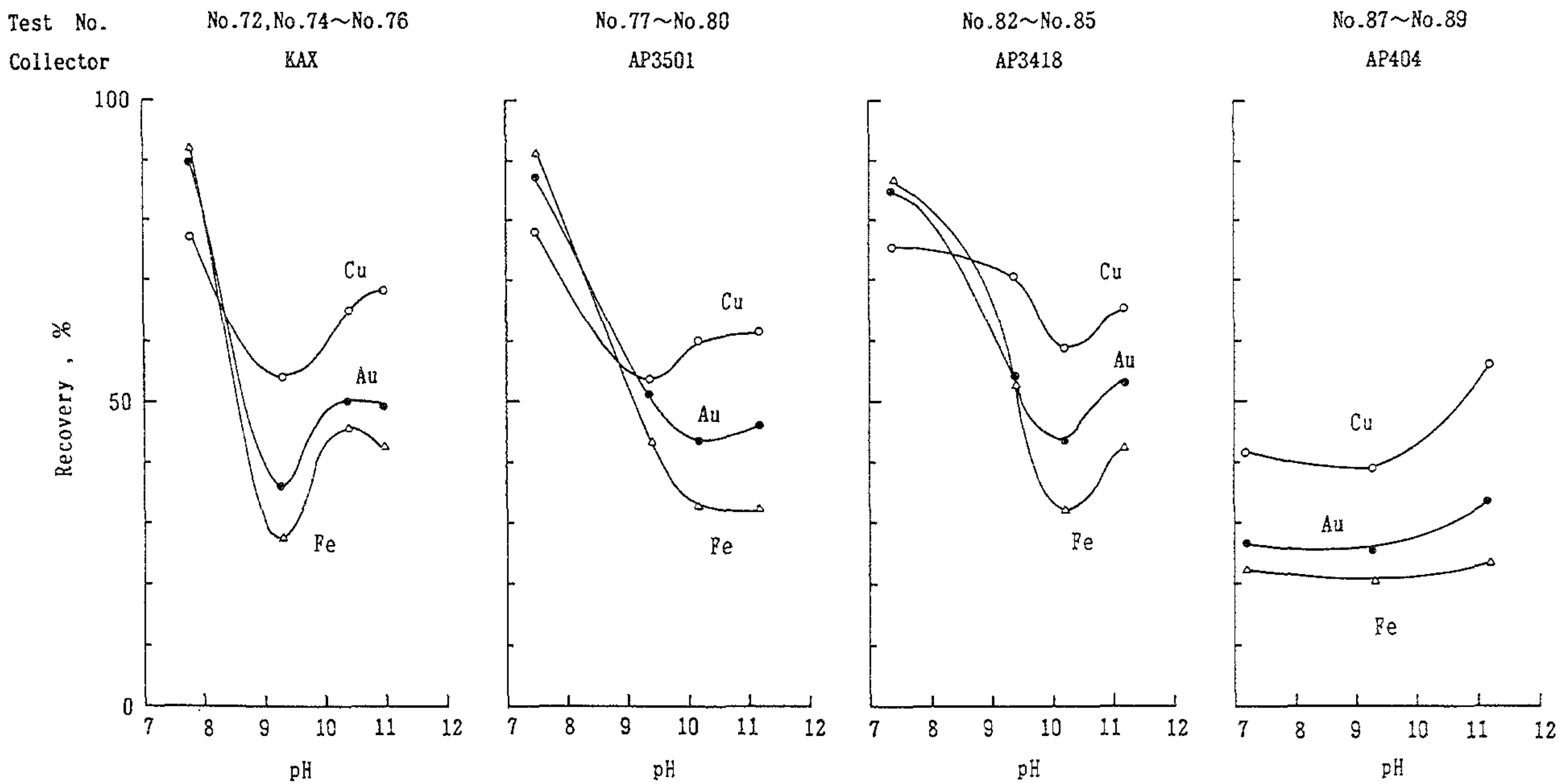


Fig. 3-17 Effect of collector on copper selective flotation of Rakah massive ore varying pH value

The results of comparison test between the collectors at pH 11.2 are shown in Table 3-15 and summarized as follows.

Table 3-15 Comparison of collectors on copper selective flotation of Rakah massive ore

Collector	KAX	AP 3501	AP 3418	AP 404
Dosage g/t	150.0	204.4	202.9	202.9
pH value	11.0	11.2	11.2	11.2
Recovery				
Cu %	68.2	61.7	65.8	56.3
Fe %	42.9	32.5	42.6	23.4
Au %	49.5	46.4	53.2	33.4
Ag %	55.7	49.0	40.6	42.5
Grade of conc.				
Cu %	2.89	3.37	2.81	4.10
Fe %	41.02	39.62	40.30	39.27
Au %	13.74	16.65	15.58	17.55
Ag %	5.05	7.15	5.33	4.72
Wt of conc. %	38.2	29.8	37.8	22.1

KAX showed higher collecting power for copper minerals than other collectors, however it's metallurgical result was poorer than that was shown in other ores.

(d) Behaviour of gold of Rakah massive ore in flotation

The possibility of gold recovery is a key point to evaluate Rakah massive ore with high grade of 9.1 g/t gold. As shown in Table 3-15 of previous section, the gold recovery was not so high. The flotation test was performed to examine the behaviour of gold. The test result was shown in Appendix 6 (test No. 91) and Fig. 3-18.

Recovery vs flotation time curve of gold was almost agreed with that of iron. It seemed most of gold was disseminated in iron sulfide mineral in fine grain, which could not be liberated from iron sulfide mineral at even a fine grind of 94% passing 200 mesh. All the gold grains locked in iron sulfide mineral particles were depressed in flotation.

(ii) Bulk and differential flotation

(a) Comparison tests of grind size and pH value

Bulk flotation tests were performed to determine the optimum grind size and pH value under following conditions.

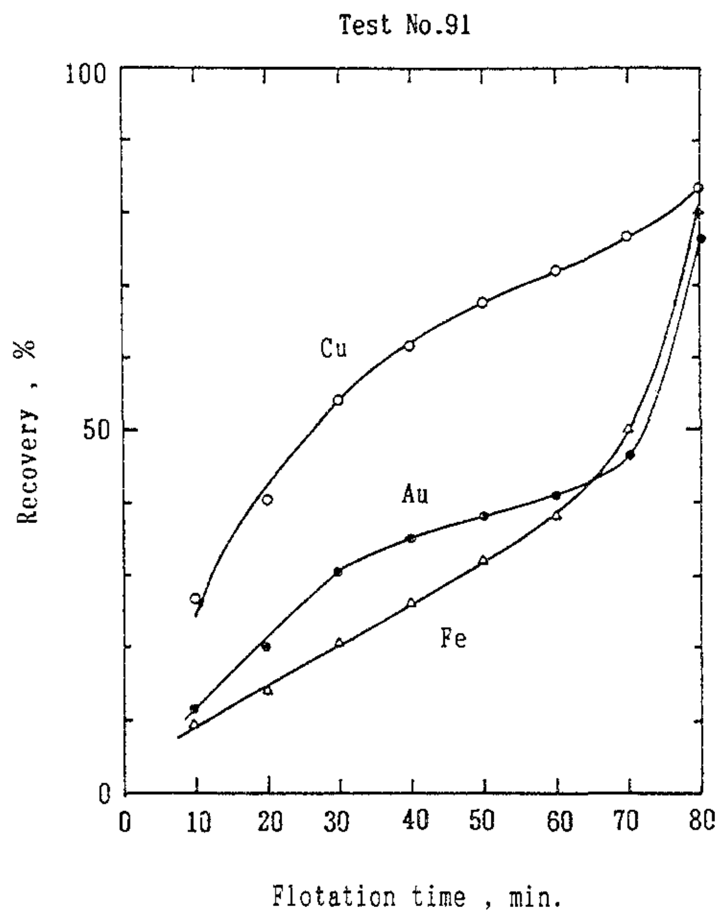


Fig. 3-18 Recovery - flotation time curves on copper selective flotation of Rakah massive ore

Grind size	minus 200 mesh	50%, 70%
KAX dosage	300 g/t	
Pulp pH value	8, 7	
Flotation time	30 minutes	

The details of the test results are shown in Appendix 6 (Table 29) and Fig. 3-19. All the tests showed poor metallurgical results and no particular difference between both conditions.

(b) Cleaner flotation

The cleaning tests on bulk concentrate were performed to study the behaviour of copper and gold in flotation under following conditions.

Regrind size	minus 200 mesh	95%
Stage of cleaning	3 stages	
Flotation time	7 min. 5 min. 4 min.	

The details of the test results are shown in Appendix 6 (Table 30) and Fig. 3-20. The results are summarized as follows.

① The copper recovery dropped exceedingly by repeating in cleaning stages with no improving of copper grade in froth products.

② As above test results were very poor, it seems impossible for Rakah massive ore to be processed by bulk and differential flotation.

(iii) Mineralogical examination of flotation products.

As the test results on Rakah massive ore showed that the processing of this ore by flotation seemed to be impossible the Microscopical Examination and Electron Probe Micro Analyzer were applied on the products of flotation test to make the cause of poor separation result clear. Microscopical photographs of flotation products and photomicrographs and X-ray images by EPMA are shown in Appendix 7 and microscopical observation results are shown in Table 3-16. The results are summarized as follows.

① It is estimated that 90 ~ 95% of copper minerals recovered in first 10 minutes were covelline and others were chalcopyrite and chalcocite, their grain sizes were passing 50 μm and 90% of them were liberated.

② The copper minerals recovered after 30 minutes were covelline and chalcopyrite locked in gangue minerals or iron sulfide minerals, their grain size were 20 ~ 30 μm.

③ The copper mineral in tails could not be identified by reflective microscopical observation. EPMA identified the grain of which Cu Kα X-ray image was in agreement with Si Kα X-ray image. It was estimated as a copper silicate mineral.

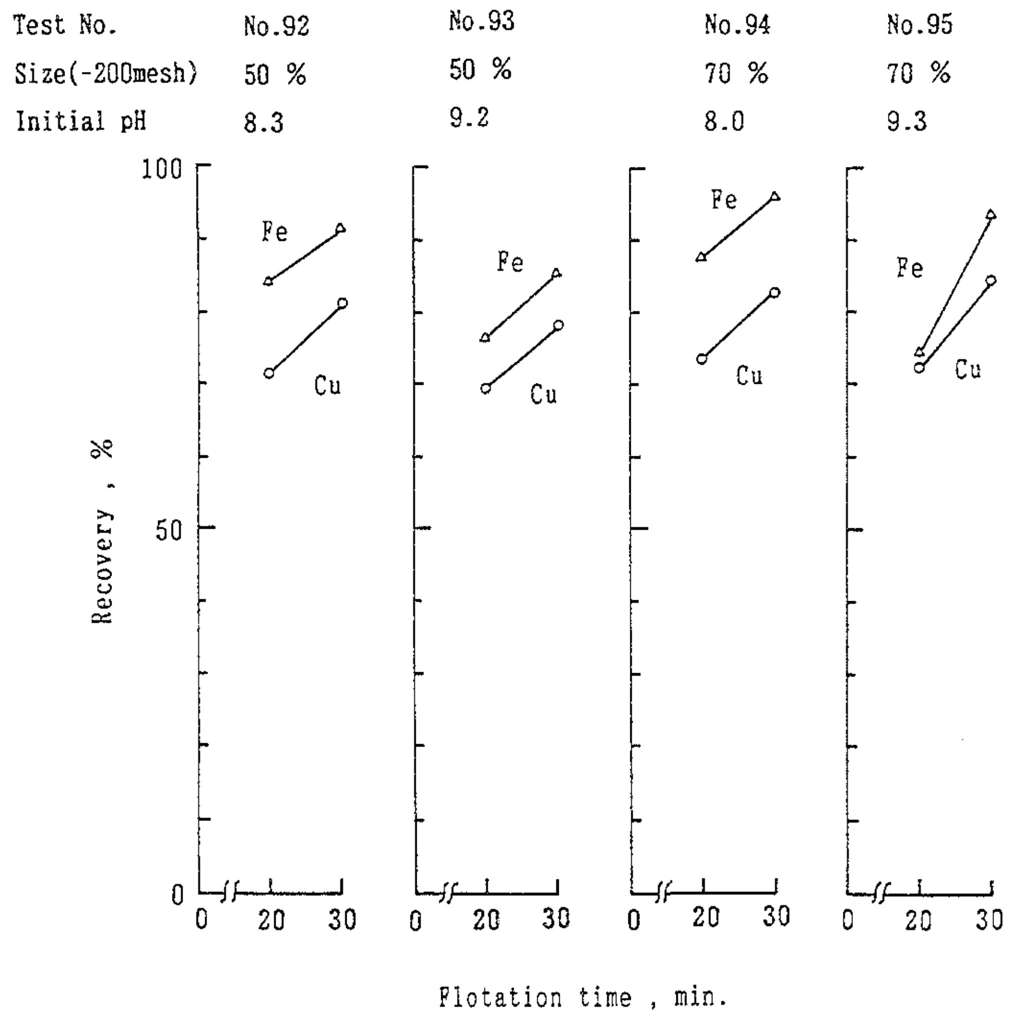


Fig. 3-19 Effect of feed size and pH value on bulk flotation of Rakah massive ore

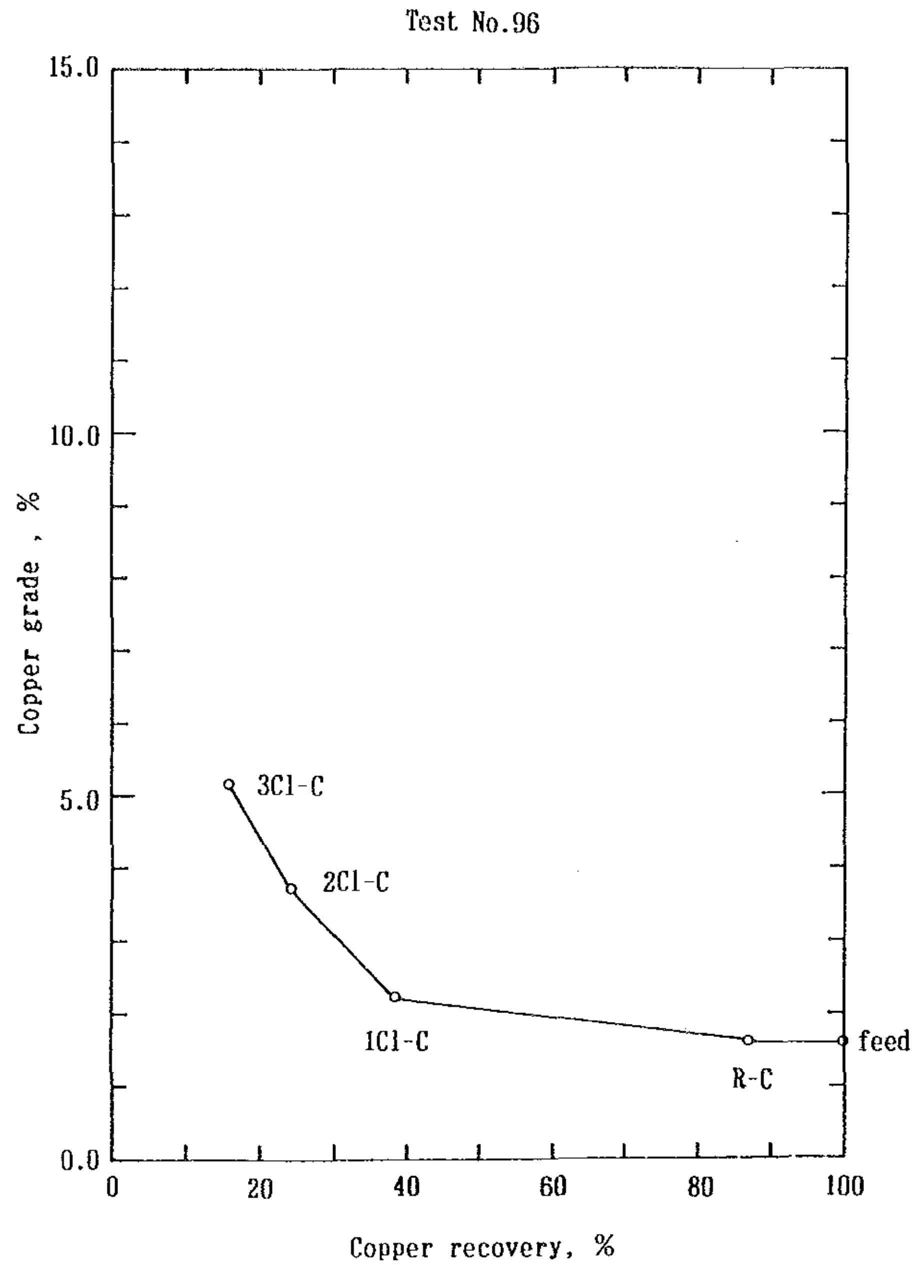


Fig. 3-20 Copper grade - copper recovery curve on bulk rougher/cleaner flotation of Rakah massive ore

Table 3-16 Microscopic examination of flotation products on Rakah massive ore

Items			10min. product	30min. product	50min. product	80min. product	tailing
Grade	Cu	, %	4.89	3.67	1.78	0.45	0.89
	Fe	, %	38.03	38.87	39.83	43.55	23.04
Distribution	Cu	, %	27.0	13.7	5.9	6.9	16.3
	Fe	, %	9.6	6.6	6.0	30.5	19.3
Grain size		, μm	< 70	< 50	<120	<120	
Percent of constituent minerals							
Cu-minerals		, %	10	1 - 2	1	very rare	nil
Fe-minerals		, %	80	80 - 90	70	90	40
gangue minerals		, %	10	10 - 20	30	10	60
Percent of Cu-minerals							
chalcopyrite		, %	5	1	99		
covelline		, %	90 - 95	99	1	100	
chalcocite		, %	5 ?				
bornite		, %			< 1		
Percent of Fe-minerals							
pyrite		, %	20				70 - 80
marcasite		, %	80				20 - 30
Liberation of Cu-minerals							
chalcopyrite		, %	95		10 - 20		
covelline		, %	90	5		0	
Size of Cu-minerals							
single grain		, μm	< 50	< 20	20 - 50		
middling		, μm	< 30 (Cv:with G,Sp Cp:with Cv,Sp)	< 30 (with G>Sp)	< 30 (with G,Sp)	< 20 (with G,Sp)	
Remarks			sphalerite : very rare		sphalerite : very rare		

Abbreviations ; Cp: chalcopyrite, Cv: covelline, Sp: iron sulfide minerals, G: gangue minerals

④ The estimate of copper mineral constituents from microscopical observation and EPMA were summarized as follows.

Copper silicate mineral	16%
Copper mineral liberated	30%
Copper mineral in middling	55%

⑤ The cause of poor metallurgical results of Rakah massive ore was that the most of copper minerals in the ore could not be liberated in spite of fine grinding. It contained the copper silicate mineral which could not be recovered by ordinary flotation process.

(4) The Characteristics of head samples in copper flotation

(i) Characteristics of Hayl as Safil ore and Rakah stockwork ore on cleaner flotation

The following test procedure was adopted for the comparison of Hayl as Safil ore and Rakah stockwork ore at a grind of 80% minus 200 mesh.

① 10 minute rougher flotation with addition of AP 3501 followed by 20 minute scavenger flotation with addition of KAX 10 g/t.

② After cleaning of rougher concentrate without regrinding, the scalp concentrate (Cu Conc 1) was produced.

③ 6 stages cleaner flotations were conducted on scavenger concentrate and tail of scalping flotation with regrind of 95% passing 200 mesh, and then the cleaner concentrate (Cu Conc 2) was produced.

The test results were shown in Appendix 6 (Table 31) and Table 3-17. The results are summarized as follows.

① Rakah stock ore showed good separation results producing Cu conc 1 with a grade of 23.36% copper, as the grain sizes of copper minerals in this ore were sufficiently large to be easily liberated.

② A copper grade of Cu conc 1 in Hayl as Safil ore was only 18.99%

③ Copper recovery and a copper grade of Cu conc 2 in Hayl as Safil ore were lower than those of Rakah stockwork ore. The grain sizes of copper minerals in this ore were very fine to be liberated with difficulty.

④ The test results gave that it would be easier to process Rakah stockwork ore than Hayl as Safil ore.

Table 3-17 Effect of scalping on copper selective flotation

Item			Hayl as Safil ore	Rakah stockwork ore
Cu conc 1				
Cu recovery		%	49.70	53.15
Grade	Au	g/t	3.10	4.62
	Cu	%	18.99	23.36
Cu conc 2				
Cu recovery		%	15.55	27.12
Grade	Au	%	2.57	3.56
	Cu	%	13.98	19.95
Cu conc total				
Cu recovery		%	65.25	80.27
Cu grade		%	17.50	20.08

(ii) Characteristics of ores in rougher flotation

The optimum conditions which produced a concentrate at the maximum copper recovery and the fundamental flotation test results are shown in Table 3-18.

The characteristics of ores in rougher flotation are summarized as follows.

① Hayl as Safil ore: The data of mineralogical examination or assays of soluble matters contained in the ore showed that this ore was slightly oxidized. A pulp pH value should be maintained at 12 or more to eliminate influence of soluble matters. The fine grinding prior to cleaning was necessary to liberate the fine grain copper minerals locked in pyrite. The copper grade and recovery of concentrate were not so high because of fine grinding.

② Rakah stockwork ore: The copper minerals of this ore were liberated from pyrite or gangue in coarser size. This ore achieved a better result in separation between copper minerals and pyrite than Hayl as Safil ore.

③ Rakah massive ore: This ore was strongly oxidized and most of the sulfide minerals were turned to secondary minerals. Iron sulfide minerals activated by copper ion were easily floatable. Fine grain of copper minerals locked in iron minerals could not be easily liberated. This ore contained copper silicate mineral which could not be recoverable by ordinary flotation. Above mentioned reason shows that this ore seemed to be impossible to be processed by any methods of flotation.

(iii) Flotation rate tests

Flotation tests were performed on three ore samples to compare the flotation characteristics of each ore. the details of the test results are shown in Appendix 6 (Tables 6, 16 and 28) and are

Table 3-18 Comparison of optimum flotation conditions and test results

Item		Hayl as Safil ore	Rakah stockwork ore	Rakah massive ore
Copper Selective Flotation				
Conditions				
Feed Size (-200 mesh)		80 %	80 %	94 %
Reagent dosage				
Collector	g/t	45	30	160
Lime	g/t	2,450	2,740	22,350
Flotation Time	min	30	30	50
PH value		11.6	10.6	11.4
Test Results				
Recovery				
Cu	%	94.9	94.2	67.8
Fe	%	38.5	23.5	32.4
Grade of Cu conc				
Cu	%	5.84	8.12	3.71
Fe	%	32.36	31.24	38.75
Wt% of Cu conc		18.2	14.6	29.8
Bulk Flotation				
Conditions				
Feed Size (-200 mesh)		50 %	50 %	70 %
Reagent dosage				
Collector	g/t	60	45	300
Lime	g/t	2,400	2,600	16,475
Flotation Time	min	30	30	30
PH value		11.6	11.0	9.3
Test Results				
Recovery				
Cu	%	96.8	96.6	84.3
Fe	%	80.4	46.2	93.8
Grade of Cu conc				
Cu	%	2.90	4.57	1.60
Fe	%	32.72	32.24	41.76
Wt% of Cu conc		38.8	27.4	83.0

plotted in the form of flotation rate curve in Fig. 3-21. The results are summarized as follows.

In generally each particle of head samples which are ground to the size of flotation test, has a different flotation rate. A flotable mineral particle has a high flotation rate if it is liberated, but in middling with non-floatable minerals the rate may be lower than the former.

The flotation rate curve of Rakah stockwork ore shown in Fig. 3-21 is the typical one. The gradient of a point on the curve shows its flotation rate. The curve gradient of 0 ~ 3 minute portion is steep, 3 ~ 7 minute portion is medium and 10 ~ 30 minute portion is gentle. It means following behaviour of mineral particles in flotation.

In the first stage, liberated chalcopyrite particle which has a high flotation rate float. In the second stage, chalcopyrite particle with some pyrite or gangue locking float. In the final stage, pyrite or gangue particle locked with same chalcopyrite float.

The flotation rate curve of Rakah massive ore shows almost straight line with gentle gradient. That means the flotation rate of any time is almost same and low. It seems all the particles float with difficulty and could not find easy floatable particles.

The flotation rate curve of Hayl as Safil ore shows similar pattern that of Rakah stockwork ore, however, the curve located at higher position and has fewer steep gradient part than Rakah stockwork ore. It means this ore has fewer liberated chalcopyrite particle and has much middling particles of chalcopyrite and pyrite.

(iv) Mixing tests with Rakah massive ore

The flotation tests were performed on samples mixed with Rakah massive ore in selected ratio to check its harmful effect and mixing ratio. Head samples were composite of 65/35% weight mixture of Hayl as Safil ore/Rakah stockwork ore. The details of test results were shown in Appendix 6 (Table 32) and Fig. 3-22.

Copper recovery and copper grade of concentrate in rougher flotation were decreased by mixing of Rakah massive ore. The decreased value of test results were come from only low result of Rakah massive ore. It did not give a severe blow to the result of rougher flotation, however, it will give harmful effect on separation of copper mineral and iron sulfide minerals in cleaner flotation. Therefore, the mixing of the Rakah massive ore should be avoided in practice.

3-1-4 Overall flotation tests results

(1) Overall flotation tests

The overall flotation tests were performed to confirm the optimum conditions which were based on the fundamental test results, and to determine the optimum flotation method. The head samples of the tests were composite of 65/35% weight mixture of Hayl as Safil ore/Rakah stockwork ore.

The tests were carried out on three flotation methods, namely, bulk and differential flotation,

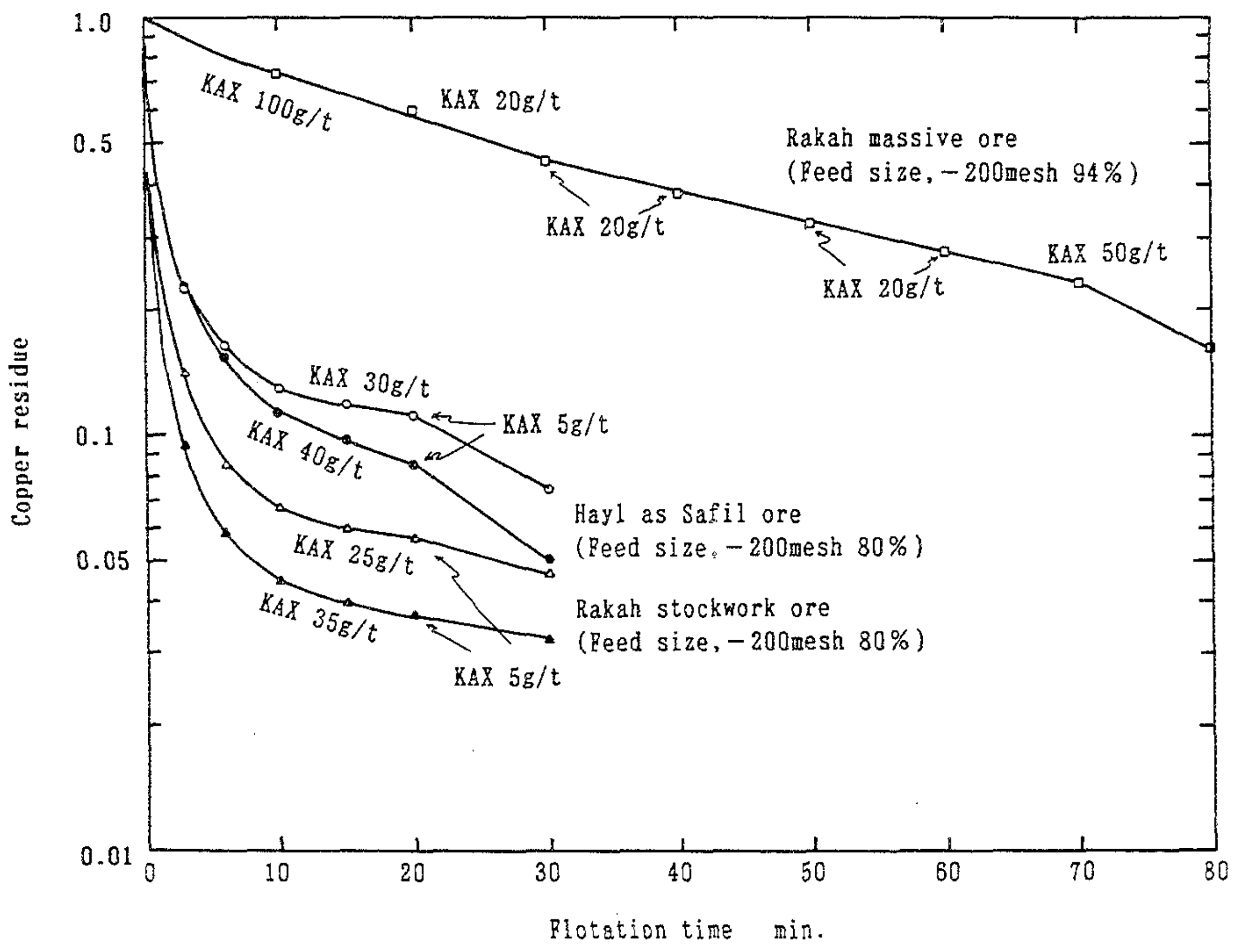


Fig. 3-21 Flotation rate curves on copper selective flotation of Hayl as Safil ore, Rakah stockwork ore and Rakah massive ore

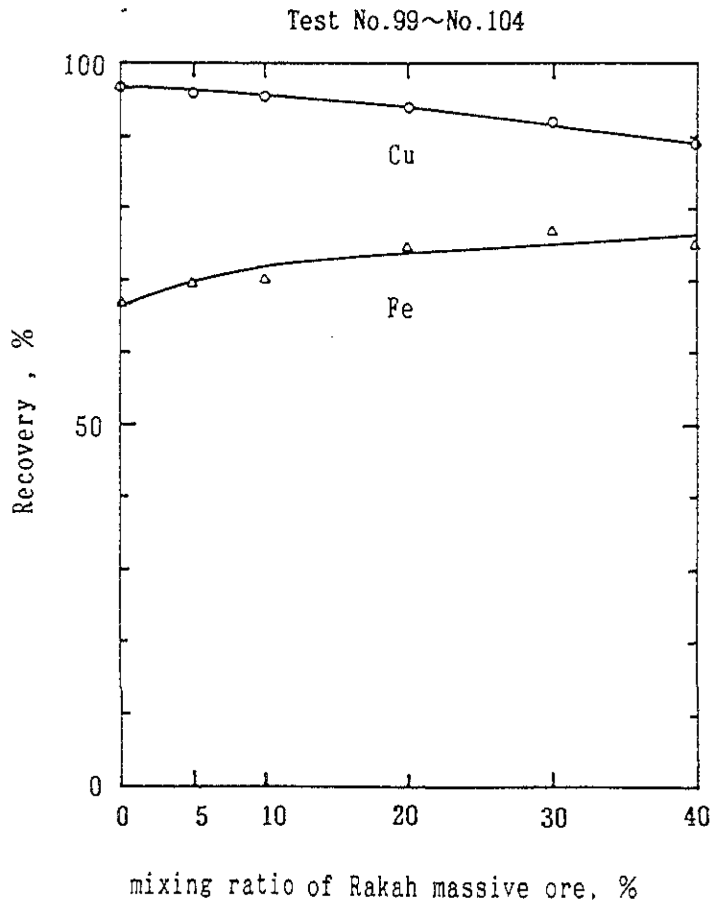


Fig. 3-22 Effect of mixing ratio of Rakah massive ore in bulk flotation of composite ore

copper selective flotation and copper selective flotation with scalping. The details of tests are shown in Appendix 6 (Tables 33 and 34) and the test results are summarized in Table 3-19.

(i) **Bulk and differential flotation**

Bulk flotation was performed under the condition of primary grind size 48% passing 200 mesh, KAX dosage 50 g/t, pulp pH value 11.4 and flotation time 30 minute. Prior to differential flotation, the bulk concentrate was reground to 95% passing 200 mesh and cleaned in six stages at pH of 12.8.

(ii) **Copper selective flotation**

Rougher flotation was performed under the condition of primary grind size 80% passing 200 mesh, KAX dosage 45 g/t, pulp pH value 11.6 and flotation time 30 minute. Prior to cleaning flotation, the rougher concentrate was reground to 95% passing 200 mesh and cleaned in six stages at pH of 12.8

(iii) **Copper selective flotation with scalping**

A froth product of initial stage on rougher flotation was scalped using selective collector of AP 3501 instead of KAX. After cleaning the froth product scalped concentrate was produced. Prior to cleaning flotation, the rougher concentrate together with the tailing of scalp cleaner were reground to 95% passing 200 mesh and cleaned in six stages at pH of 12.8.

(iv) **Behavior of gold in cleaner flotation**

Copper grade vs gold grade curves of froth products are shown in Fig. 3-23. There is a linear relationship between both grades. The grade ratio of Au/Cu of froth products of each flotation process shows almost same value as follows.

Flotation process	Average grade ration (Au/Cu)
Bulk & differential flotation	0.26
Copper selective flotation	0.20
Copper selective flotation with scalping	0.24

Gold accompanied by copper minerals is recovered in every cleaning stages. While the major part of gold disseminated in iron sulfide mineral is discarded with iron sulfide mineral in cleaning flotation.

Table 3-19 Summary of flotation test results on composite ore

Product	W %	Assay		Distribution	
		Au g/t	Cu %	Au %	Cu %
Bulk & Diferential Flotation					
Calc. Head	100.00	0.48	1.17	100.00	100.00
Concentrate	4.11	5.20	19.95	44.30	69.82
Clnr Middling	5.35	1.39	4.27	15.40	19.45
Clnr Scav Tail	31.02	0.53	0.31	34.12	8.20
Rougher Tail	59.52	0.05	0.05	6.18	2.54
Copper Selective Flotation					
Calc. Head	100.00	0.53	1.26	100.00	100.00
Concentrate	2.77	4.77	23.68	25.08	52.24
Clnr Middling	5.02	2.04	7.18	19.39	28.69
Clnr Scav Tail	11.49	1.07	0.89	23.34	8.14
Rougher Tail	80.72	0.21	0.17	32.18	10.93
Copper Selective Flotation with Scalping					
Calc. Head	100.00	0.56	1.17	100.00	100.00
Scalp Conc.	1.08	3.96	22.67	7.55	20.83
Concentrate	3.42	4.01	17.33	24.30	50.68
Clnr Middling	3.57	2.21	5.19	13.97	15.84
Clnr Scav Tail	6.59	1.01	0.95	11.81	5.35
Rougher Tail	85.34	0.28	0.10	42.37	7.29

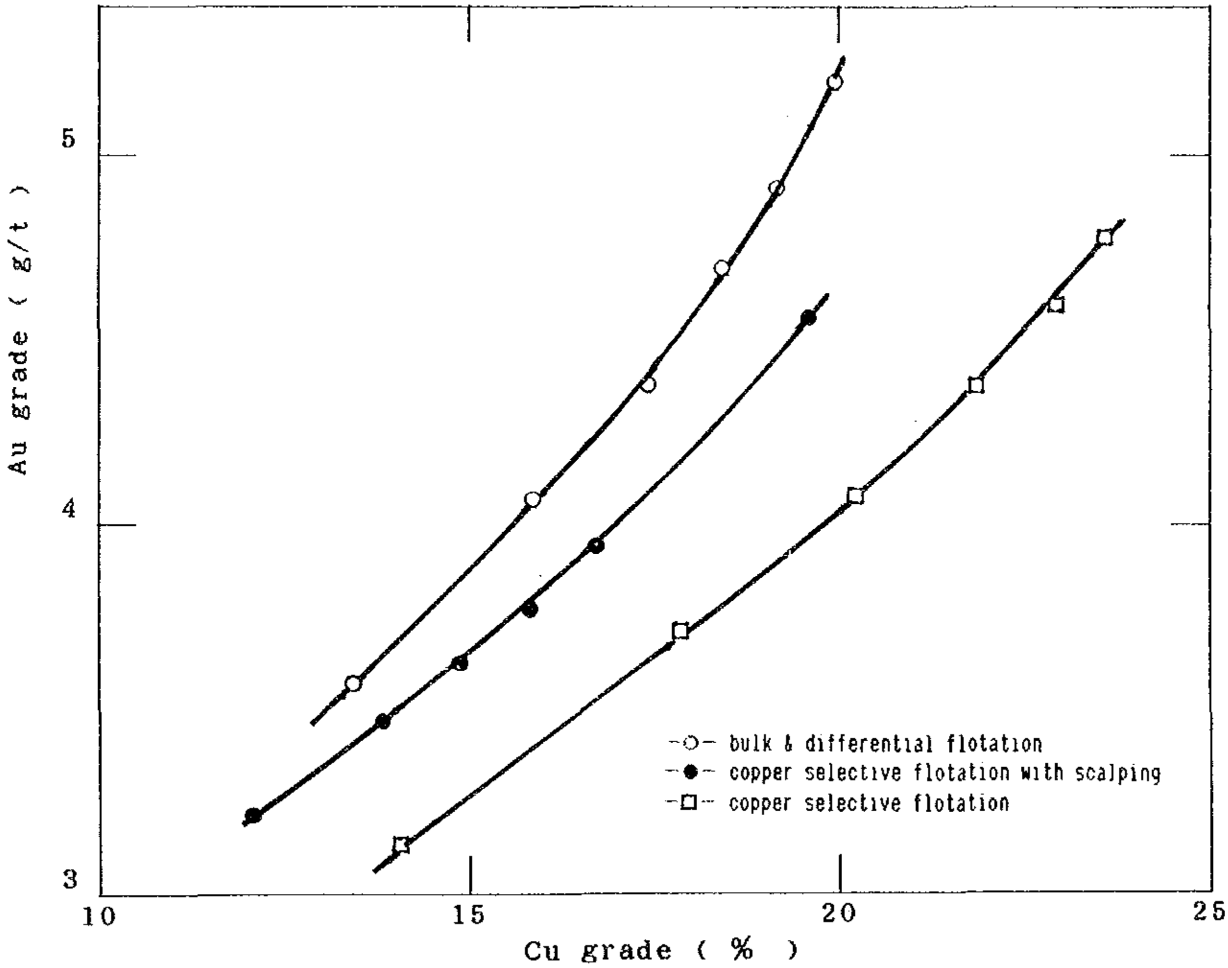


Fig. 3-23 Cu grade - Au grade of cleaner froth product

(2) **Determination of flotation process**

Table 3-20 shows metallurgical results of three flotation methods at the concentrate grade of 20% copper. Bulk and differential flotation method showed the highest result in copper and gold recoveries.

Table 3-20 Comparison of overall flotation results

Item		Bulk & differential flotation	Copper selective flotation	Copper selective flotation with scalping
No. of cleaning		6	3	6
Grade of Cu conc				
Au	g/t	5.20	4.07	4.14
Ag	g/t	25.94	18.86	16.51
Cu	%	19.95	20.24	19.81
Cu Distribution				
Conc	%	69.82	66.55	61.03
Middling	%	19.45	14.38	26.32
Tailing	%	10.73	19.07	12.65
Au Distribution				
Conc	%	44.30	25.08	26.45

The optimum flotation method of this ore seemed to be Bulk and Differential process. The flotation condition of this process is shown as follows.

Rougher	Primary grind size	50% minus 200 mesh
	KAX dosage	50 g/t
	Initial pH value	11.4
	Flotation time	30 minute
Cleaner	Regrind size	95% minus 200 mesh
	pH value	12.8
	No. of cleaning	6 stages

3-1-5 Summary of the tests

The metallurgical tests were conducted on the samples of the Hayl as Safil and Rakah deposits to obtain the fundamental data for milling design. The three type of head samples of Hayl as Safil, Rakah stockwork and Rakah massive ores were tested. All the samples were taken from the drill cores which were performed in this project in 1988. The main chemical components of samples are shown as follows.

Component		Hayl as Safil ore	Rakah stockwork ore	Rakah massive ore
Au	g/t	0.5	0.5	9.1
Cu	%	1.12	1.25	1.60
Fe	%	16.52	20.43	35.73
S	%	14.60	10.52	39.87
SiO_2	%	57.89	40.86	19.61
MgO	%	2.35	6.13	0.02
Al_2O_3	%	3.17	10.13	0.35

Mineral constituent of head samples are shown as follows.

Hayl as Safil ore: This ore contained sulfide minerals approximately 28% weight, the copper minerals identified were chalcopyrite as the major copper mineral, covelline and chalcocite as the secondary minerals in minor quantity. The fine grain size of copper minerals inclusion locked with pyrite were found. The gangue minerals identified were quartz, chlorite and hematite.

Rakah stockwork ore: This ore contained sulfide minerals approximately 21% weight, the copper mineral identified was only chalcopyrite and no secondary mineral. The gangue minerals identified were quartz and chlorite.

Rakah massive ore: This ore contained sulfide minerals approximately 75% weight. Because the ore was highly oxidized, the most of copper minerals were secondary minerals including silicate mineral. A lot of marcasite was identified as much as pyrite.

The flotation characteristics of head samples are shown as follows.

Hayl as Safil ore: The flotation selectivity of minerals of this ore was not so good because of fine mineral combination of chalcopyrite and pyrite and the oxidization of ore.

Rakah stockwork ore: The selectivity of minerals of this ore was good because the grain size of chalcopyrite and pyrite was coarse and easily liberated. A lot of chlorite was identified as gangue mineral, however, no harmful effect was found on flotation.

Rakah massive ore: This ore could not be processed by flotation, because the ore consists of highly oxidized mineral combination of very fine grain of copper minerals and iron sulfide minerals.

The overall flotation tests were conducted on composite sample of Hayl as Safil and Rakah stockwork ores to determine the optimum flotation method of bulk and differential flotation, copper selective flotation and copper selective flotation with scalping. Bulk and differential flotation process achieved the best results on copper and gold recoveries.

3-2 Mineral processing plant description

3-2-1 Site consideration

The plant processes run of mine ores from the Hayl as Safil and Rakah pits. Suitable locations for the plant could be found near both pits. Because of the ore transportation cost the plant location was selected near the Hayl as Safil pit which has twice tonnage of ore reserves as that of the Rakah. The plant was planned to be built on a flat area 500 m south of the Hayl as Safil pit perimeter.

In the selected site area, sedimentary rocks of the Olistostrome are found and these rocks have sufficient load bearing capacity to install heavy machines.

A vast area close to south side of the plant is suitable for tailing dam. This dam location near the plant is very convenient for technical control on the operation of feeding tailing or feeding back reclaimed water. It seems that no mineral deposit exists under the plant and tailing dam sites from the geological structure of this area. However, the reconfirmation by drilling should be carried out prior to the construction works.

3-2-2 Fundamental conception of plant design

The mineral processing plant for Hayl as Safil and Rakah ores was designed on the base of laboratory testwork which was carried out in this project and operating data of the Sohar plant. The following two processes are designed in different way from those of Sohar operation.

(1) Crushing and grinding

The Sohar plant is operating in autogenous grinding but this plant's comminution process is designed in conventional way of three stage crushing and one stage grinding.

Autogenous grinding system generally replace all of the crushing stage of conventional flowsheet except for primary crusher, but because the autogenous mill facilities are so expensive that total capital cost may be same as that of conventional way.

Although power consumption and liner wears in autogenous mill are generally higher than those in rod mill and ball mill, total operating cost would be reduced by the effect of saving grinding media.

Product of primary crusher is fed to autogenous mill through coarse ore stockpile. The size distribution range of the ore is so wide that the coarser fraction of the ore easy to segregate from the finer fraction when it is delivered to the stockpile.

It is difficult to draw mill feed ore from the stockpile in constant ratio of coarse and fine fractions, therefore size distribution of autogenous mill feed ore may wildly fluctuate usually.

In the operation of autogenous grinding different from that of ball mill grinding, the fluctuation of size distribution of mill feed ore directly dominate quantity of grinding media in the mill and have a serous influence upon characters of ground product. Double stage grinding circuit, closed circuit operation and automation control have some action on controlling operational variations, however, notable effects could not be expected. Moreover in this project processing two different type of ore, another trouble occurs caused by the variation on ore mixing ratio.

Although various difficulties occur in operation such large scale plants as porphyry copper mills or iron ore mills take autogenous grinding system as a comminution process, because operating costs of autogenous grinding are lower than that of conventional comminution system.

In small scale plants, however, the cost advantage of autogenous grinding is not so high as large scale plants, from a viewpoint of the operation control it may be better to avoid taking autogenous grinding as a comminution process.

As this ore has few moisture and sticky materials, conventional system of three stage crushing will be operated with no troubles, and produces fine mill feed ore which has goood effects on stable operation of grinding and flotation.

(2) Flotation

In Sohar plant the ore is ground to 80% passing 200 mesh, copper minerals selectively float in the condition of depressing pyrite and other gangues in rougher flotation circuit.

Chalcopyrite and pyrite in this ore are very finely combined and fine grinding is necessary to liberate chalcopyrite from pyrite, but combination between sulfide minerals and gangues is not so fine and easy to liberate each other. It is confirmed in the tests that copper recovery of coarse size in bulk flotation is almost same as that of fine size, because chalcopyrite locked in pyrite easily float.

Bulk and differential flotation system is taken as a most feasible flotation process of this kind of ore. Bulk flotation circuit is operated in coarse size and the bulk concentrate is reground in fine size prior to differential flotation.

A reduction in capital and running cost of grinding and high flotation rate in rougher flotation are expected by coarse size operation, and coarse size tailing will be effective on stabilization of bank body in tailing dam.

All the mine ore is processed in the plant except Rakah massive ore. Plant metallurgical balance shown in Table 3-21 is estimated from the data of No. 105 bulk and differential flotation test which showed the best separation result on the composite sample of Hayl as Safil ore and Rakah stockwork ore.

Table 3-21 Plant metallurgical balance

Product	Weight (%)	Grade		Distribution	
		Copper (%)	Gold (g/t)	Copper (%)	Gold (%)
(Rougher)					
Mild feed	100.00	1.26	0.59	100.0	100.0
Rou'r conc.	40.00	3.06	1.40	97.1	94.9
Rou'r tail	60.00	0.06	0.05	2.9	5.1
(Cleaner)					
Cl'r feed	40.00	3.06	1.40	97.1	94.9
Cl'r conc.	5.60	20.00	5.20	88.9	49.3
Cl'r tail	34.40	0.30	0.78	8.2	45.6
(Overall)					
Mill feed	100.00	1.26	0.59	100.0	100.0
Final conc.	5.60	20.00	5.20	88.9	49.3
Final tail	94.40	0.15	0.32	11.1	50.7

3-2-3 Process design criteria

The selection of plant processes and machines have been designed on the bases of fundamental conception and design criteria shown in Table 3-22.

3-2-4 Flowsheet description

This section briefly describes the process of the plant as designed on the bases of design criteria. It should be read conjunction with Fig. 3-24 Flow diagram, Fig. 3-25 Flowsheet and Table 3-23 Equipment list. The general layout for the plant is illustrated in Appendix 8 (Fig. 1).

Table 3-22 Process design criteria (1)

Process	Item	Unit	Quantity
Operating condition	Anual processing tonnage	mt/y	1,080,000
	Scheduled operating days	day/y	360
	Average throughput	mt/day	3,000
Characteristics of mill feed ore	Main component minerals of Mill feed ore (estimate)		
	Sulphide minerals	%	27
	Hematite	%	3
	Chlorite	%	20
	Quartz	%	30
	Moisture content		nearly 0
Crushing	Three stage crushing		
	Scheduled operating hrs	hr/day	24
	Availability	%	69
	Average running hrs	hr/day	16.7
	Crushing rate	mt/hr	180
	Max size of mine ore	mm	1,200
	Crushing product size (80% passing)		
	Primary (open circuit)	mm	150
	Secondary (open circuit)	mm	28
	Tertiary (closed circuit)	mm	9.4
	Coarse ore stockpile (primary crusher product)	mt	2,000
	Fine ore stockpile (tertiary crusher product)	mt	3,000

Table 3-22 Process design criteria (2)

Process	Item	Unit	Quantity
Grinding	Single stage ball mill grinding closed circuit with cyclones		
	Scheduled operating hrs	hr/day	24
	Availability	%	83
	Average running hrs	hr/day	20
	Milling rate	mt/hr	150
	Grinding feed (80% passing)	μ	9,400
	Grinding pro't (80% passing)	μ	150
	Bond's Work Index	kwh/st	Wir= 13
		kwh/st	Wib= 12
	Pulp density (W%)		
	Ball mill discharge	%	75
	Cyclone overflow	%	38
	Circulating load (Percentage of new feed)	%	350
Regrinding	Closed circuit operation with ball mill & cyclone		
	Capacity	mt/hr	63
	Grinding feed (80% passing)	μ	100
	Grinding pro't (80% passing)	μ	38
	Bond's Work Index	kwh/st	Wi = 14
	Pulp density (W%)		
	Ball mill discharge	%	75
	Cyclone overflow	%	25
	Circulating load (percentage of new feed)	%	250
Flotation	Bulk & differential flotation		

Circuit	Pulp flow rate (m^3/min)	No. of cells (300cf)	Flotation retention time (min)
Rougher	4.89	12	20.8
First clr	5.41	6	9.4
Second clr	2.25	4	15.1
Third clr	1.46	2	11.7
Fourth clr	.72	2	23.6
Clr scav'r	4.80	6	10.6

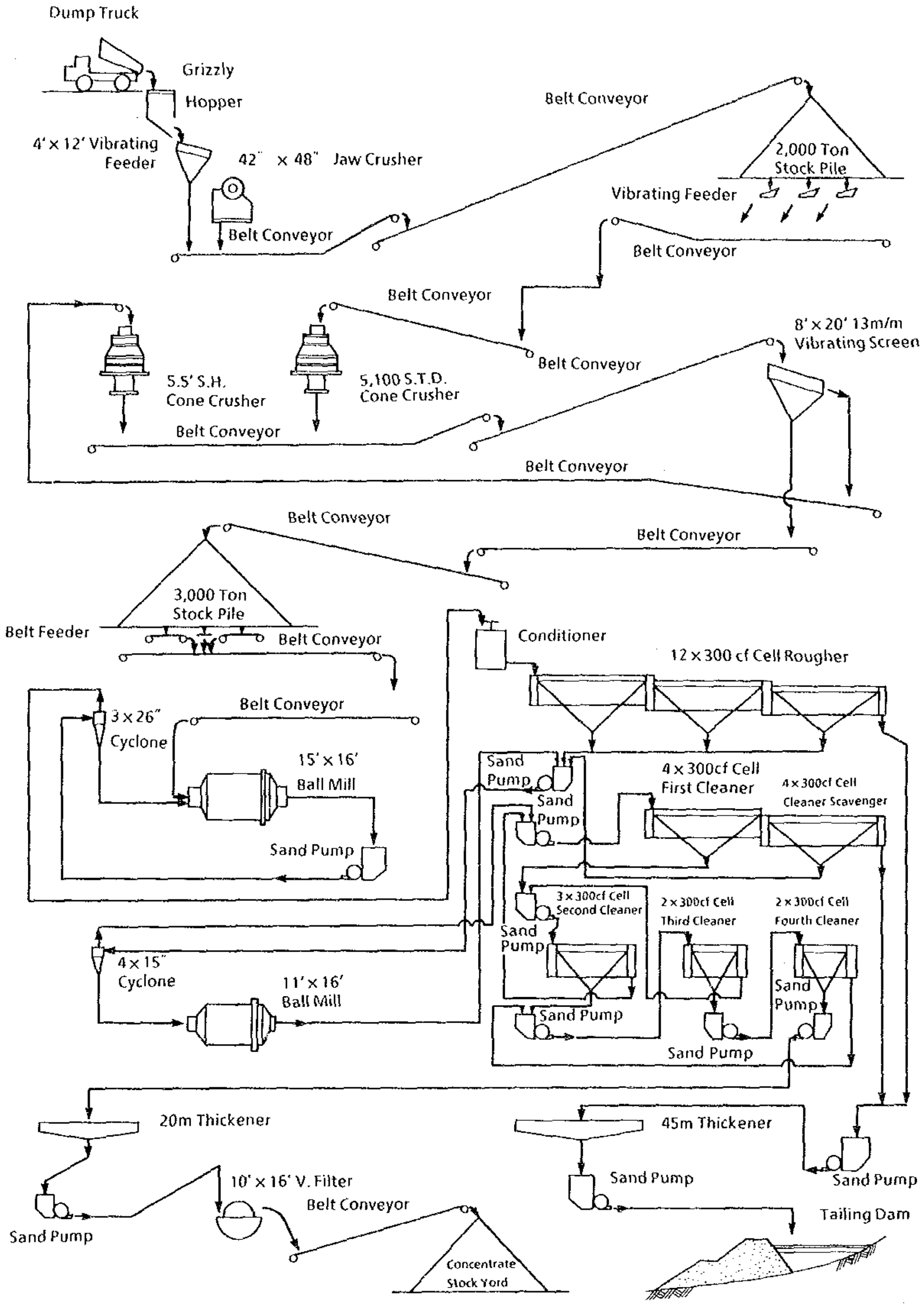


Fig. 3-24 Mineral processing plant flow diagram

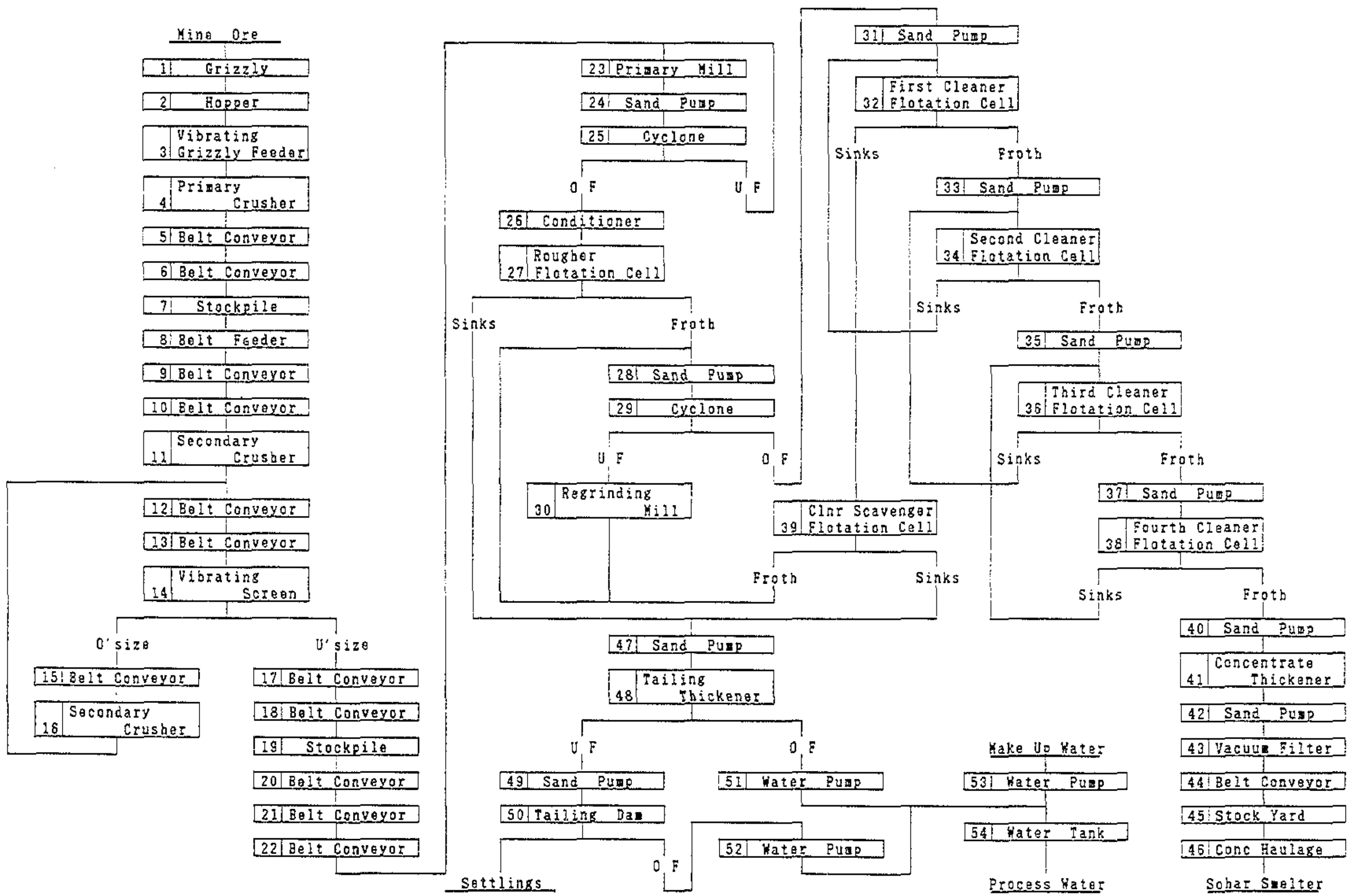


Fig. 3-25 Proposed mineral processing flowsheet

Table 3-23 Proposed plant equipment list (3,000 t/day) (1)

No	Equipment	Size	Details	Motor (kw)	No of Unit
1	Grizzly	5mx4.5m	grizzly bar spacing 800mm		1
2	Hopper		live capacity 30t		1
3	Vibrating grizzly feeder	4'x12'	grizzly bar spacing 150mm	15	1
4	Primary crusher	42"x48"	double toggle type openside setting 150mm	130	1
5	Belt conveyor	1.2mx35m	12° inclined	11	1
6	Belt conveyor	1.05mx35m	15° inclined	7.5	1
7	Coarse ore stockpile		live capacity 2,000t		1
8	Vibrating feeder	.8mx1.2m	valiable speed motor drive (3.7kwx3)	11.1	3
9	Belt conveyor	1.05mx35m	14° inclined	5.5	1
10	Belt conveyor	1.05mx40m	15° inclined	7.5	1
11	Secondary crusher	5,100	STD cone crusher coarse type setting 22mm	130	1
			oil pump & others	5.9	
12	Belt conveyor	1.2mx25m	9° inclined	7.5	1
13	Belt conveyor	1.2mx30m	18° inclined	7.5	1
14	Vibrating screen	8'x20'	single deck Ripl-Flow type screen aperture 13mm	22	1
15	Belt conveyor	1.05mx50m	13° inclined	11	1
16	Tertiary crusher	5'-6"	SH cone crusher fine type	130	1
			setting 9.5mm	5.9	
			oil pump & others	5.9	

Table 3-23 Proposed plant equipment list (2)

No	Equipment	Size	Details	Motor (kw)	No of Unit
17	Belt conveyor	1.05mx25m	15° inclined	5.5	1
18	Belt conveyor	1.05mx60m	14° inclined	11	1
19	Fine ore stockpile		live capacity 3,000t		1
20	Belt feeder	1.2mx6.5m	horizontal(7.5kwx2)	15	2(2)
21	Belt conveyor	.9mx50m	5° inclined	5.5	1
22	Belt conveyor	.9mx18m	7° inclined	5.5	1
23	Primary mill	15'x16'	overflow type ball mill compressor	1,550 3.7	1
24	Sand pump	10"x8"		100	1(1)
25	Cyclone	26"			2(1)
26	Conditioner	2mϕ x2m		7.5	1
27	Rougher flotation cells	300cf	(22.5Kwx12)	270	12
28	Sand pump	8"x6"		30	1(1)
29	Cyclone	15"			3(1)
30	Regrinding mill	11'x16'	overflow type ball mill compressor	750 3.7	1
31	Sand pump	6"x4"		15	1(1)
32	First cleaner Flotation Cells	300cf	(22.5Kwx 6)	135	6
33	Sand Pump	3"x2"		5.5	1(1)
34	Second Cleaner flotation cells	300cf	(22.5Kwx 4)	90	4
35	Sand pump	3"x2"		5.5	1(1)

No of Stand-by Unit ( )

Table 3-23 Proposed plant equipment list (3)

No	Equipment	Size	Details	Motor (kw)	No of Unit
36	Third cleaner flotation cells	300cf	(22.5Kwx 2)	45	2
37	Sand pump	3"x2"		5.5	1(1)
38	Fourth cleaner flotation cells	300cf	(22.5Kwx 2)	45	2
39	Cleaner scavenge flotation cells	300cf	(22.5Kwx 6)	135	6
40	Sand pump	3"x2"		5.5	1(1)
41	Concentrate thickener	20mφ	centre drive type	3.7	1
42	Sand pump	3"x2"		5.5	1(1)
43	Vacuum filter	10'x16'	drum type filter vacuum pump compressor filtrate pump	5.9 65 22.5 3.7	1
44	Belt conveyor	.4mx35m	horizontal	3.7	1
45	Stockyard	36mx12m	live capacity 1,500t		1
46	Cu conc haulage		20t Truck		10
47	Sand pump	10"x8"		50	1
48	Tailing thickener	45mφ	centre drive type	4.4	1
49	Sand pump	6"x4"		30	1(1)
50	Tailing dam		capacity 7 milion t		1
51	Water pump			75	1(1)
52	Water pump			22.5	1(1)
53	Water pump			75	1(1)
54	Water tank		capacity 400 cub m		1

No of Stand-by Unit ( )

(1) **Primary crushing (see Appendix 8, Fig. 2)**

Run of mine ore hauled by 30 t lorry is tipped on a scalping grizzly. The undersize product falls into a surge hopper of 30 t live capacity, large rocks on grizzly are spalled by impact breaker. Under the surge hopper a 4' × 12' Vibrating grizzly feeder is installed for withdrawing the ore to a 42" × 48" Jaw crusher. Its product discharges onto a 1.2 m wide belt conveyor. A second conveyor of 1.05 m width delivers the ore to a coarse ore stockpile with live capacity of 2,000 t.

(2) **Secondary and tertiary crushing (see Appendix 8, Figs. 3 and 4)**

The ore is reclaimed from stockpile by three .8 m × 1.2 m feeders and fed to a 5,100 STD cone crusher through two belt conveyors. The crusher products are conveyed to a 2.4 m × 6.0 m single deck vibrating screen with 13 mm × 13 mm aperture woven wire cloth, and undersize product is taken as mill feed and conveyed to a fine ore stockpile with live capacity of 3,000 t. Oversize product from the screen is conveyed to a 5'-6" SH cone crusher. For the sake of capital costs saving, fine ore stockpile is selected instead of fine ore bin. it seems that no technical troubles will occur in stockpile at few rainfall climate of mine site.

(3) **Grinding (see Appendix 8, Figs. 5 and 6)**

Mill feed is reclaimed from the stockpile by four 1.2 m × 6.5 m belt feeders controlled by constant feed weigher and fed into a 15' × 16' ball mill which is in closed circuit with three 26" cyclones. Grinding operation shall be controlled to get the cyclones overflow at 38% solid, 55% minus 200 mesh. Regrinding system is a 11' × 16' ball mill with four 15" cyclones.

(4) **Flotation (see Appendix 8, Fig. 6)**

The fine overflow from cyclones is piped to 12 cells of 300 cf (8.5 m^3) rougher circuit. The tailing from rougher circuit joining the tailing of cleaner circuit are discarded as final tailings. The bulk concentrate floated in rougher circuit is ground to the size of 90% minus 200 mesh in regrinding mill to liberate chalcopyrite from pyrite and fed to 20 cells of 300 cf cleaner circuit where the final copper concentrate is recovered by depressing pyrite through four stage cleaning. The tailside 6 cells of the cleaner circuit are arranged as scavenger, enabling the overall cleaner tailing to be discarded as final tailings.

(5) **Concentrate dewatering (see Appendix 8, Fig. 7)**

The concentrate dewatering process consists of a 20 mφ × thickener and a 10' × 16' drum type vacuum filter.

The copper concentrates are loaded into lorries and hauled to Sohar smelter. If the concentrates contain high moisture, they will be liquidized by the quake during the way and may cause flow loss.

The moisture contents in copper concentrate in this ore may be higher than other ordinary concentrate, because very fine regrinding is necessary prior to cleaner flotation. Installation of pressure filter which has high dewatering performance may be considered in this case. This machine, however, cannot operate continuously and dewatering capacity is much lower than the same size vacuum filter. If the drum type vacuum filter is selected, it will not be impossible to get filter cakes nearly 10% moisture contents in good operating condition.

The copper concentrate stock yard is designed to have sufficient capacity for the fiter cake storage before loading into lorries. The excess moisture in the filter cakes will evaporate during the storage in very dry climate at mine site.

(6) Tailing disposal (see Appendix 8, Fig. 8)

Final tailings are pumped to a 45 mϕ thickener. The underflow from thickener is pumped at 60% solids to the tailing dam located south side of the plant. Final tailing size is coarser than that of Sohar mine by coarse grinding in primary mill, and a lot of lime added in flotation assist flocculating of fine particles and clear decant water could be obtained in the thickener and tailing dam.

(7) Mill water

Mill water balance is shown in Table 3-24. Mill water requirement is grinding 5.5 m^3/min, flotation and others 2.0 m^3/min total 7.5 m^3/min, that is equivalent to 3.0 m^3/t. Water loss in tailing dam is 1.5 m^3/min and remaining 6.0 m^3/min of water will be reclaimed mainly from tailing stream.

Raw water make-up for water loss is supplied from underflow of river by well. The plant site is situated on upper stream region of the river where only few water may be supplied, the water intake facility is planned to install near Yanqul 15 km downstream from the plant site where many rivers join together. The survey on the water intake in Yanqul will be carried out by the Ministry of Petroleum and Minerals.

(8) Power supply

Electric power is supplied from power transmission network of the Ministry of Electricity and Water. The Hayl substation which is located some 20 km south of mine site have a sufficient capacity to supply power for the plant. The construction cost of power transmission line from Hayl

Table 3-24 Mill water balance

Process water		Loss of water		Reclaimed water		Make-up water	
Plants	m^3/h	Plants	m^3/h	Plants	m^3/h	Plants	m^3/h
Grinding m^3/h	327	Tailing dam	60	Tailing dam	61	Water intake	90
Primary grinding 245		Filter cake	1	Tailing thickener	260		
Regrinding 82		Evaporation	22	Concentrate thickener	20		
Flotation	97	Surface area of water pool (Tailing dam, Thickener) 22,000 m^2					
First clr. launder 30							
Second clr. launder 30							
Third clr. launder 25		Rate of evaporation					
Fourth clr. launder 12		day time) 1 mm/hr					
Others	26	Others	7	Others	19		
Total	450	Total	90	Total	360	Total	90
	(7.5m^3/min)		(1.5m^3/min)		(6.0m^3/min)		(1.5m^3/min)

substation to mine site through Yanqul substation shall be born by this project.

3-2-5 Operation

(1) Plant organization and manpower

Under the metallurgical superintendent the department is divided into three sections, plant operation, laboratory and assays and maintenance. The manpower of operation and maintenance on mineral processing plant and tailing disposal is planned to be 104 persons. The details of manpower requirement are shown in Table 3-25.

The plant is to be operated continuously on a 3 × 8 hour shift system. The plant operation consists of eight processes namely primary crushing, secondary and tertiary crushing, grinding, flotation, concentrate dewatering, tailing disposal, lime plant and Yanqul pump station. Under a shift forman 13 operators compose a shift crew. Total 51 persons are in plant operation, 42 persons in shift crews and 9 persons in day works.

Metallurgical tests and technical studies concerning processing operation are perfomed by laboratory. Samples of mining and processing operations are assayed by assay plant. Total 22 persons belong to laboratory and assay section.

Daily maintenance and repair works of machines and equipments are carried out by mechanical and electrical technicians. Two civil technicians are in charge of safty and pollution contrl of tailing dam. Total 28 persons belong to maintenance section

(2) Operating materials

The operating material consumptions are shown in Table 3-26. The figures of flotation reagents and grinding materials are estimated from the data of metallurgical test, and the other materials mainly from the operating results of Sohar plant. Because the grain size of the component minerals in this ore is so fine that fine grinding is necessary to get enough liberation of each minerals. Some part of the ore are suffered from oxidization by weathering and have no good selectivity in flotation. Therefore the consumptions of grinding material and flotation reagent are higher than other ordinary ore.

(3) Power consumption

The power demand of the plant is estimated to be 3,800 kw from designed motor powers as shown in Table 3-27. The unit power consumption will be 25 kwh/t. Power is supplied from the Ministry of Electricity and Water at an annual average rate of 0.019 R.O./kwh.

Table 3-25 Manpower requirement of Metallurgical department

Section	Salary and Wage Grade											Total
	1	2	3	4	5	6	7	8	9	10	11	
Superintendent	1											1
(PROCESSING PLANT)												
General Foreman			1									1
Sift Foreman						4						4
Primary crushing								3		3		6
Sec/Ter'y crushing								3		3		6
Grinding								3	3			6
Flotation								3				3
Conc. dewatering								3				3
Tailing disposal								3		3		6
Lime plant								3		3		6
Yanqul water pump									3			3
Day works								1			6	7
(Subtotal)												51
(LABORATORY & ASSAY)												
Chief Metallurgist			1									1
Laboratory					1		3	4		3		11
Assay						2	3	3		3		11
(Subtotal)												23
(MAINTENANCE)												
Supervisor			1									1
Mechanical					1	3	6	6		4		20
Electrical					1	1	4					6
Civil(Tailing Dam)						1	1					2
(Subtotal)												29
Total	1	0	3	0	3	11	17	35	6	22	6	104

Table 3-26 Estimate operating material consumption

Items	Unit g/t	Q'ty t/y	Unit price R.O.	Unit price US$	Amount US$/y	US$/t
Limes	5,000	5,400	12	31	168,480	
Frother	20	22	778	2,023	43,692	
Collector	50	54	557	1,448	78,203	
Reagent total					290,375	0.269
Ball 80mm	1,000	1,080	258	671	724,464	
Ball 30mm	400	432	308	801	345,946	
Ball total					1,070,410	0.991
Crusher liner					66,528	
Ball mill liner					74,080	
Liner total					140,608	0.130
Operating consumables					44,678	
Machine parts					725,253	
Vehicle fuel parts					111,049	
Lab'ry materials					28,244	
Miscellaneous					18,909	
Others total					928,133	0.859
Total					2,429,526	2.250

Table 3-27 Estimated power consumption

Equipment	Calculation	Results
	(Motor power) (Availablity)	
Primary mill	1,550 kw × .93 =	1,447 kw
Regrinding mill	750 kw × .94 =	704 kw
Mill(others)	1,295 kw × .85 =	1,100 kw
(Mill subtotal)	(3,595 kw × .904)=	(3,251 kw)
Crushing plant	527 kw × .70 =	369 kw
Lighting & others		180 kw
Total		3,800 kw
Crushing plant	527 kw ÷ 180 t/h =	2.05 kwh/t
Mill	3,251 kw ÷ 150 t/h =	21.67 kwh/t
Lighting & others	total × 5 % =	1.28 kwh/t
Total		25.00 kwh/t
Total consumption	25.00kwh/t× 1,080,000t/y =	27,000,000 kwh/y
Power cost	.019 RO × 2.6 US$/RO × 27,000,000kwh/y=	1,333,800 US$/y

(4) Total operating costs

Total operating costs shown in Table 3-28 are estimated from unit prices and the consumption data of manpower required, operating materials and powers.

Table 3-28 Estimated operating costs

1,080,000 tons/year

Item	Details	US$/y	US$/t
Wage & salaries	number of employees = 104	1,043,230	0.966
Operating materials		2,429,526	2.250
Power	4.94c/kwh×27,000,000 kwh=	1,333,800	1.235
General expenses		12,400	0.011
Total		4,819,006	4.462

Chapter 4 Waste dump and tailings dam

4-1 Waste dump

Two waste dumps are to be constructed for the mine development. They are shown in Fig. 1 and Fig. 4-2 (1). One is adjacent to the Hayl as Safil pit and the other is to Rakah pit. The sizes of the dumps are 500,000 square meters and 300,000 square meters for Hayl as Safil and Rakah respectively. Owing to sufficient area available for the dumps, the height of both dumps are only 20 meters which is considerably low and consequently the dumps are very stable.

Before the commencement of waste dump construction, it is essential to check the sites by drilling in order to confirm non existence of mineral deposits beneath the proposed sites.

4-2 Tailing dam

4-2-1 Selection of tailing dam site

After studying tailing dam site on valleys of Oman mountain range or flat areas around Hayl as Safil, the location of dam site has been selected to vast flat area close to south side of the plant, because the valleys of the mountain range are so narrow that could not acquire a sufficient capacity.

The tailing dam site shown in Fig. 4-1 is slightly elevated area between two rivers Wadi Hayl al Ali and Wadi Falaj Sudayriyin. In few rainfall region like this mine site rivers flow usually in underground, but when a heavy rain falls the water level of under stream will rise up to the surface, though, in such cases it will have no serious trouble on this dam site.

4-2-2 Topography and nature of ground

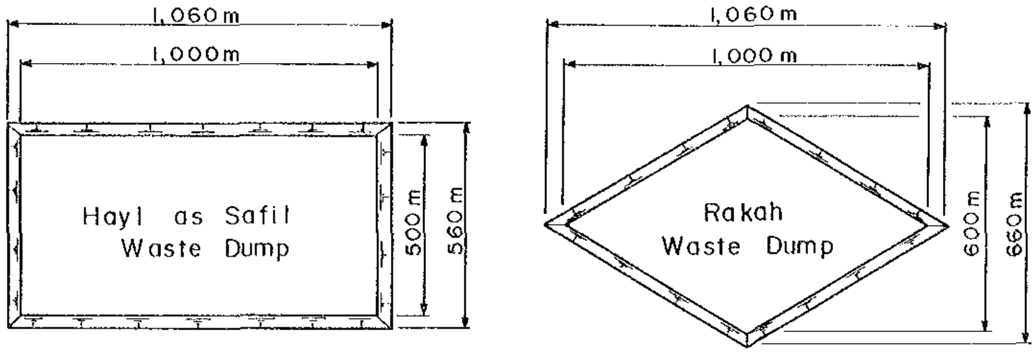
Topography of the site is slightly inclined at north-east gradient of 1/60, and the area of the site is east-west 1.2 km, north-south 1.2 km.

The geology of this site is terrace structure same as that of the processing plant. The nature of ground is gravel and sand layer cemented by calcite having unpermeable quality and sufficient load bearing capacity to the construction of banking or drain culvert. The decant water separated from settlings will not permeate into ground.

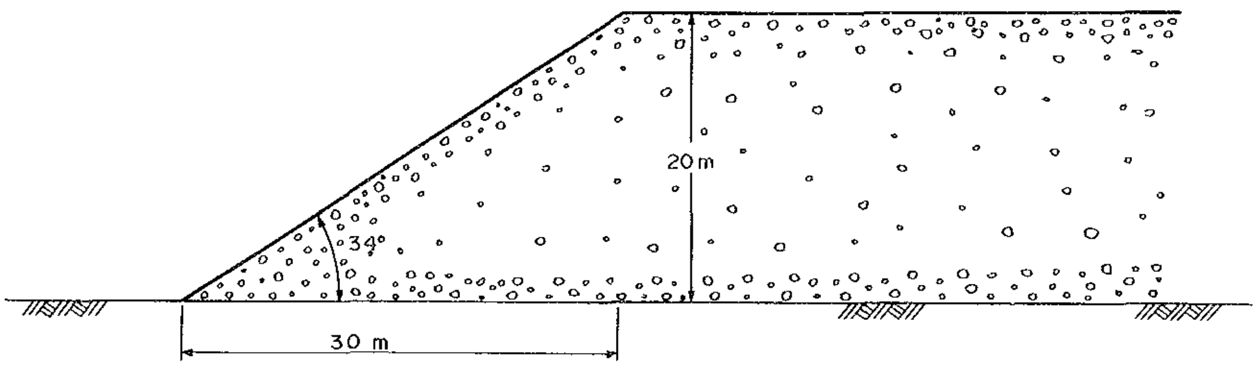
4-2-3 Dam

The 3,400 m long dam is constructed around the site area as shown in Fig. 4-1. Height of dam is 18 m in south side and 0.5 m in north end. The dam is constructed by piling up wast rocks of

Fig. 4-1 Plan of tailing dam



Plan of Waste Dump



Section of Waste Dump

Fig. 4-2 Standard section of tailing dam and waste dump (1)

Hayl as Safil pit and using soils of site area for filter section of dam as shown in Fig. 4-2. The design specifications of banking are shown in Table 4-1.

4-2-4 Drainage facility

Final tailings of the plant are discharged from the downstream side of the dam. The coarser fraction of tailings will settle near the dam and the finer fraction will flow to upstream and settle there. Inside the dam area two lines of underlying drain culverts shown in Fig. 4-1 are constructed on the bottom. The decant water separated from the settlings will flow into the drain culvert through spillways shown in Fig. 4-3.

All the rain water that falls inside the dam area flows into the drain culvert through the spillways and then discharges outside of the dam. It is not necessary to provide an emergency drain because there is no basin other than the inside of the dam (120 ha). Two lines of culverts are designed to act as a stand by unit each other. The design criteria of drainage facility are shown in Table 4-2.

The discharge capacity of drain culvert is four times of maximum rain water inflow. The spillway has sufficient capacity of collecting water in operation (0.017 m^3/sec). Many spillways should be opened to collect a lot of water in heavy rain, of course excess water could be stocked temporary in the dam. For the sake of safety the end of the drain culvert must be opened to act as an emergency drain.

4-3 Other facilities

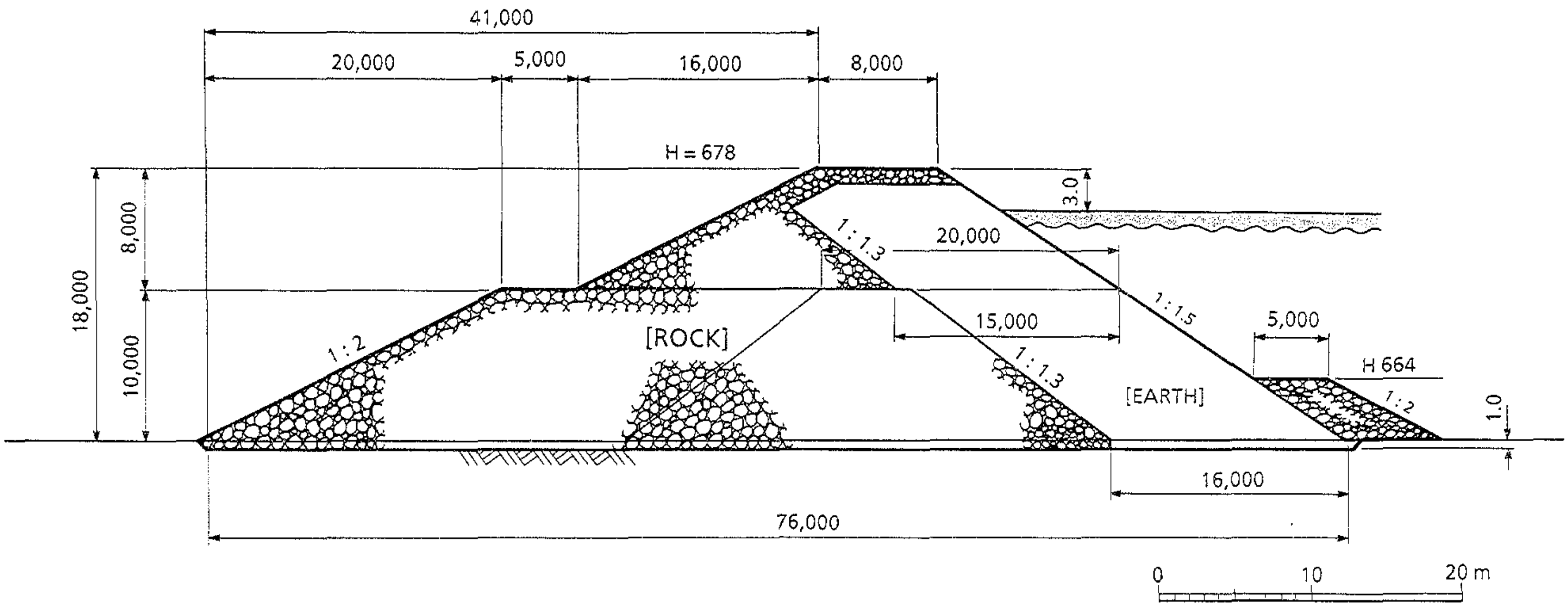
(1) Pit drainage

The seepage water in the pit is to be pumped up by submergible pumps. The water is directly delivered to the tailing thickener through pipe line. The heavy metal ion in the water is to be neutralized by thickener water which has high pH value due to concentrator operation.

(2) Dust prevention for the mining hauling road

Two units of water tank lorries are allocated for dust prevention for the mining hauling road. They are for Hayl as Safil pit and Rakah pit respectively.

Reclaimed water from the concentrator is to be supplied to these lorries in order to save the fresh water.



Section of Tailing Dam

Fig. 4-2 Standard section of tailing dam and waste dump (2)

Table 4-1 Specification of banking

	Item	Quantity
Dam	Final height of dam	18.0 m
	Final altitude of dam	678.0 m
	Final altitude of settlings	675.0 m
Earth Volume	Rock fill material	1,400,000 m^3
	Filter material	610,000 m^3
	Total	2,010,000 m^3
Storage Capacity	Alutitude	Cumulative volume
	665.0 m	600,000 m^3
	667.5 m	1,600,000 m^3
	670.0 m	3,100,000 m^3
	672.5 m	4,900,000 m^3
	675.0 m	7,200,000 m^3

Table 4-2 Design criteria of drainage facility

	Item	Quantity
Rain water	Rainfall intensity	10 mm/hr
	Catchment area	1.2 km^2
	Coefficient of discharge	1.0
	Max. rain water inflow	3.4 m^3/sec
Culvert	Discharge capacity of a drain culvert	6.8 m^3/sec
Spillway	Flow capacity of a spillway	
	Water depth 0.55 m	.16 m^3/sec
	0.75 m	.19 m^3/sec
	1.25 m	.24 m^3/sec

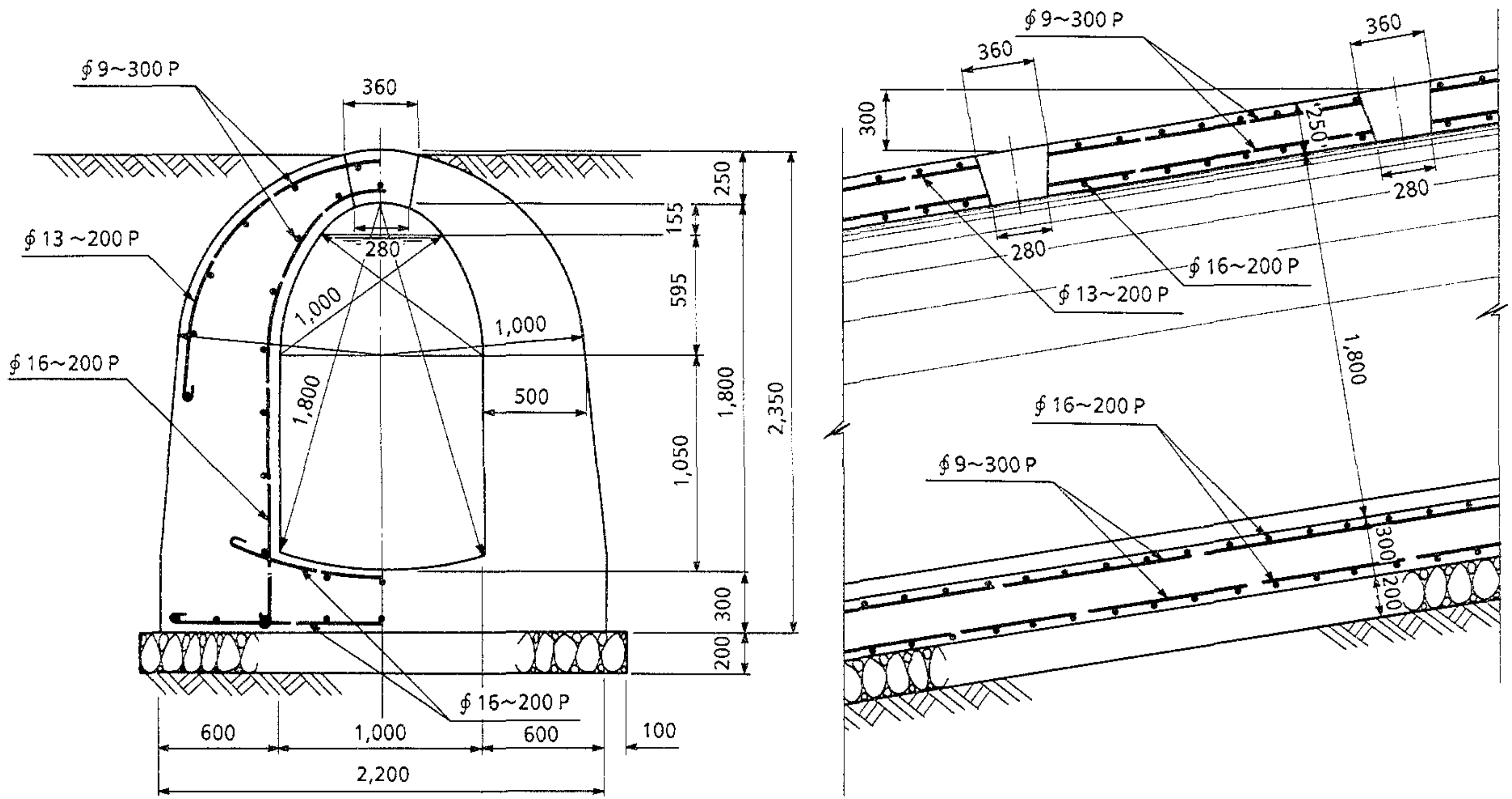


Fig. 4-3 Section of drain culvert

Chapter 5 Supporting

Finance, commerce, general affairs and personnel sections are planned as supporting department in the operation of the project. In addition to the above mentioned, engineering department is working as the independent body in supporting department at the Sohar mine which is now under operation. It is basing on the idea that overall maintenance for operations of mining, mineral processing and smelting /refining are under control of the engineering department at the Sohar mine.

In this project, engineering department was not independently established, because without smelting/refining section, a size of operation will be supposed much smaller than that of the Sohar mine.

Proposed organization chart of supporting department is shown in Fig. 6-1 (4), (5) of Chapter 6.

Construction cost of the facility such as the buildings for office of supporting department, warehouse of materials necessary for operation etc. is amounted to US$275,200 (Table 8-2).

Operation cost of supporting department is amounted to US$1,566,900/year in total, consisting of US$816,700/year for personnel and US$750,200/year for material.

Chapter 6 Organization and manning plan

Basic concept for the organization and manning plan in this development project is derived from the idea of the Sohar mine which is now under operation and having a similar operating environment, with the exception of thinking on engineering department described in Chapter 5.

Regarding to the engineering department, it is planned for maintenance of mineral processing plant by maintenance team of mineral processing department and maintenance of mine heavy equipment by heavy equipment repair team of mining department (Fig. 6-1 (2), (3)).

Every effort is made for establishing an efficient organization with reducing as possible the number of persons in manning plan of each department. However, on the other hand, some increment of personnel are estimated during 5 years from commencement of operation, to promote systematically the Omanization of employees.

In concrete terms, vocational schooling mainly through on the job training for a duration of one year, is carried out for 20 newly joined employees of the ranking from middle standing engineers to nucleus workers level, and it should be continued 5 years over. By this training, it is planned to upgrade 100 Omani employees and to replace the foreign employees with them.

Estimating from the present circumstances of the Sohar mine, ratio of Omani employees at the stage of entering into mine production, is supposed to be less than 30% of total employees, considering that the Omanization to be available from the initial stage is limited to only drivers of mining dump truck. However, it is possible to improve the ratio to be more than 50% of total employees after 5 years operation, with promoting the above mentioned Omanization programme.

Proposed organization charts are shown in Fig. 6-1 (1)~(5) and a breakdown of personnel cost is shown in Table 6-1.

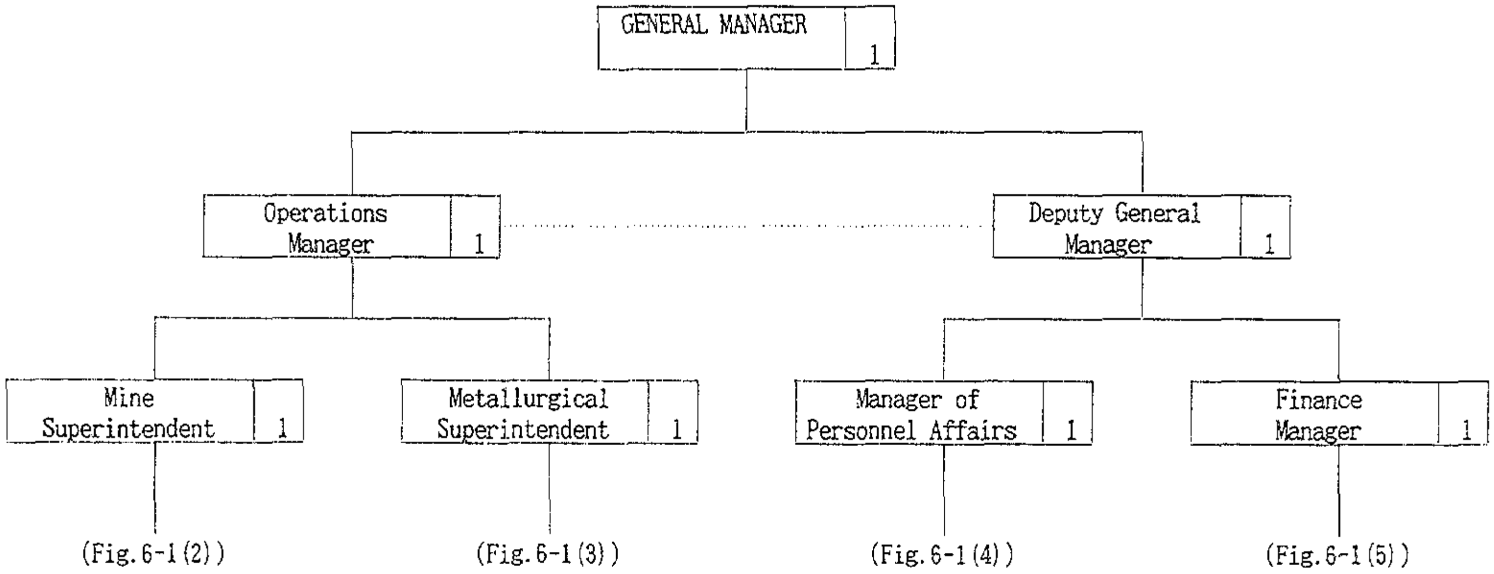


Fig. 6-1 Proposed organization for mine operation (1)

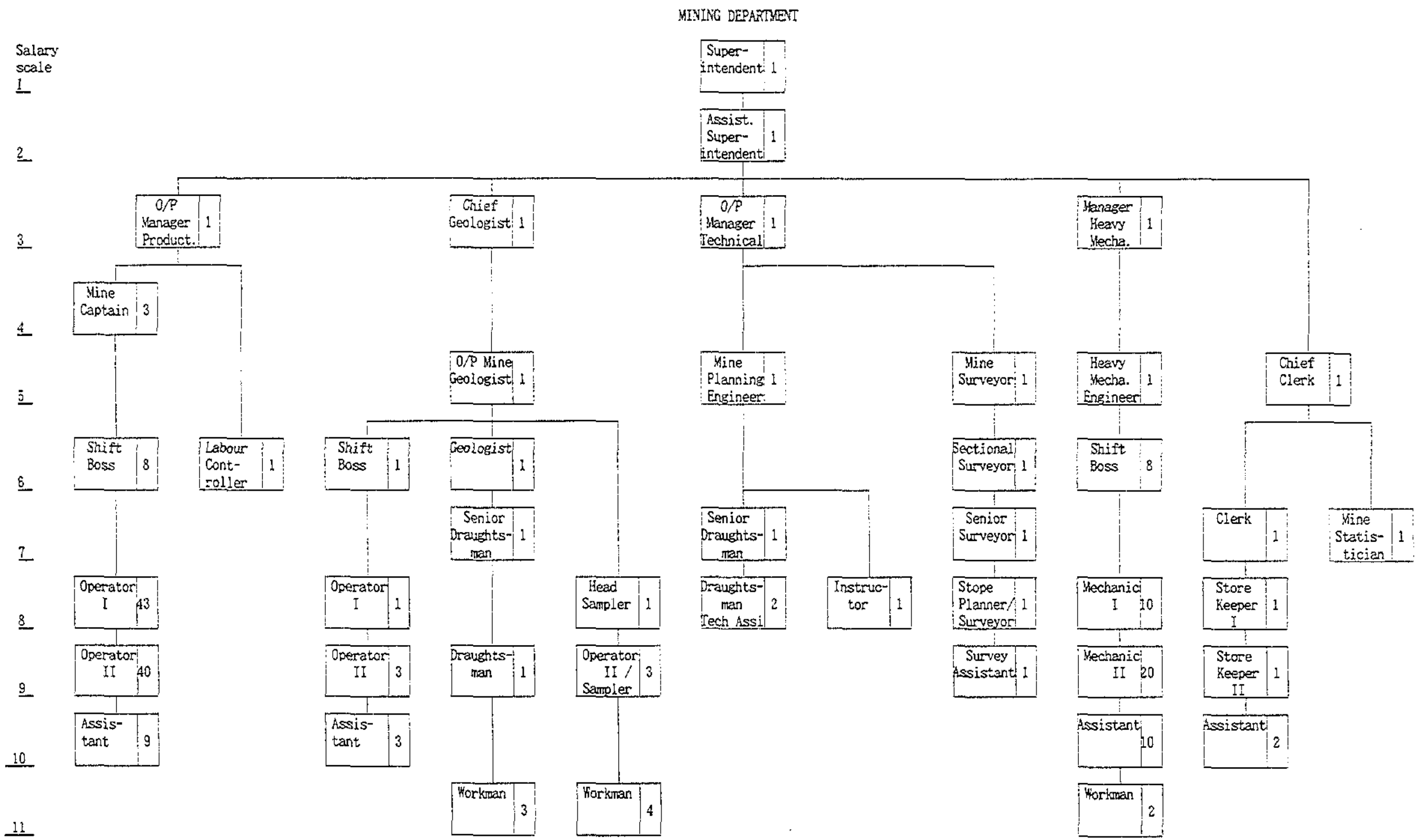


Fig. 6-1 Proposed organization for mine operation (2)

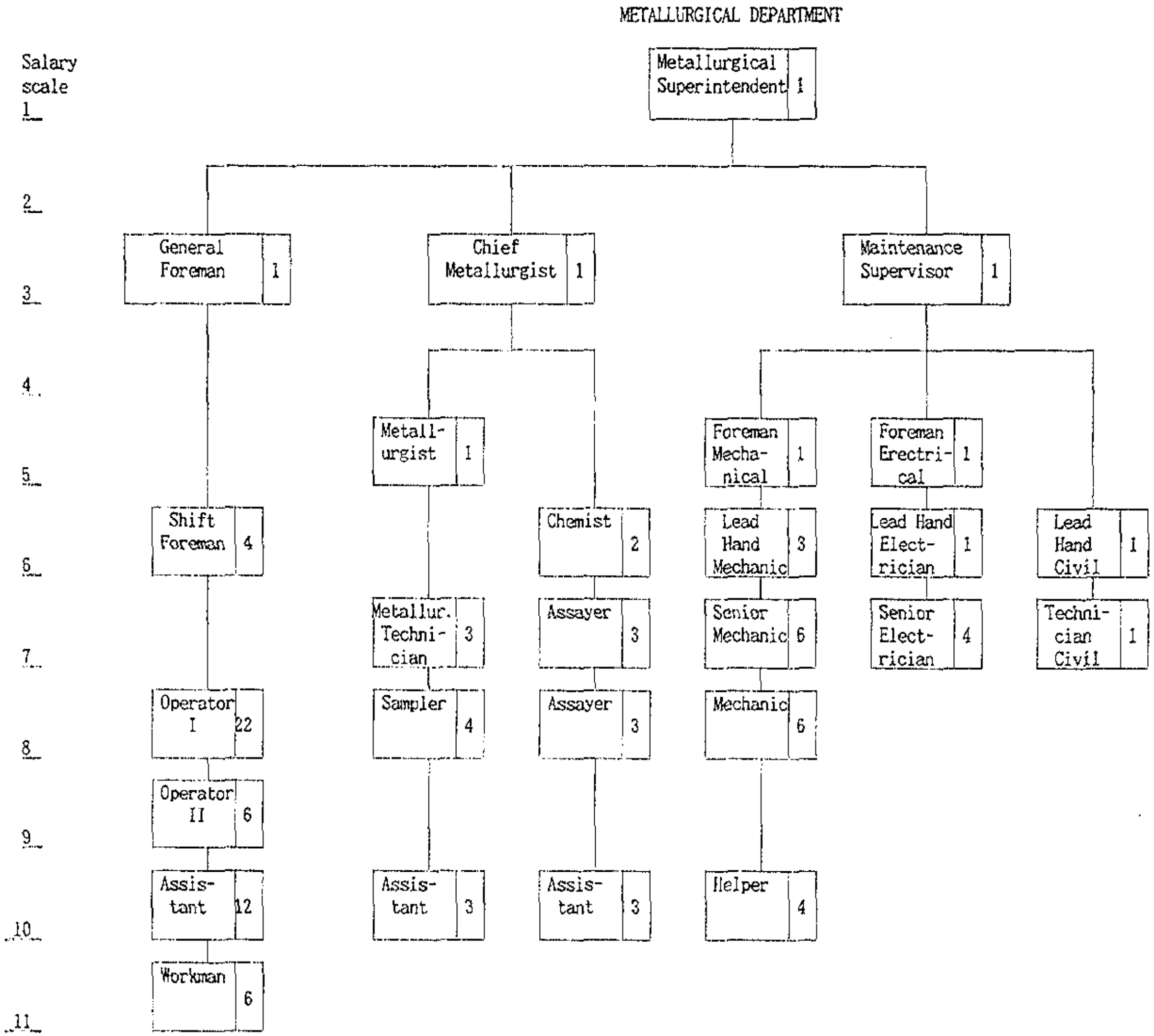


Fig. 6-1 Proposed organization for mine operation (3)

SUPPORT SERVICE - ADMINISTRATION

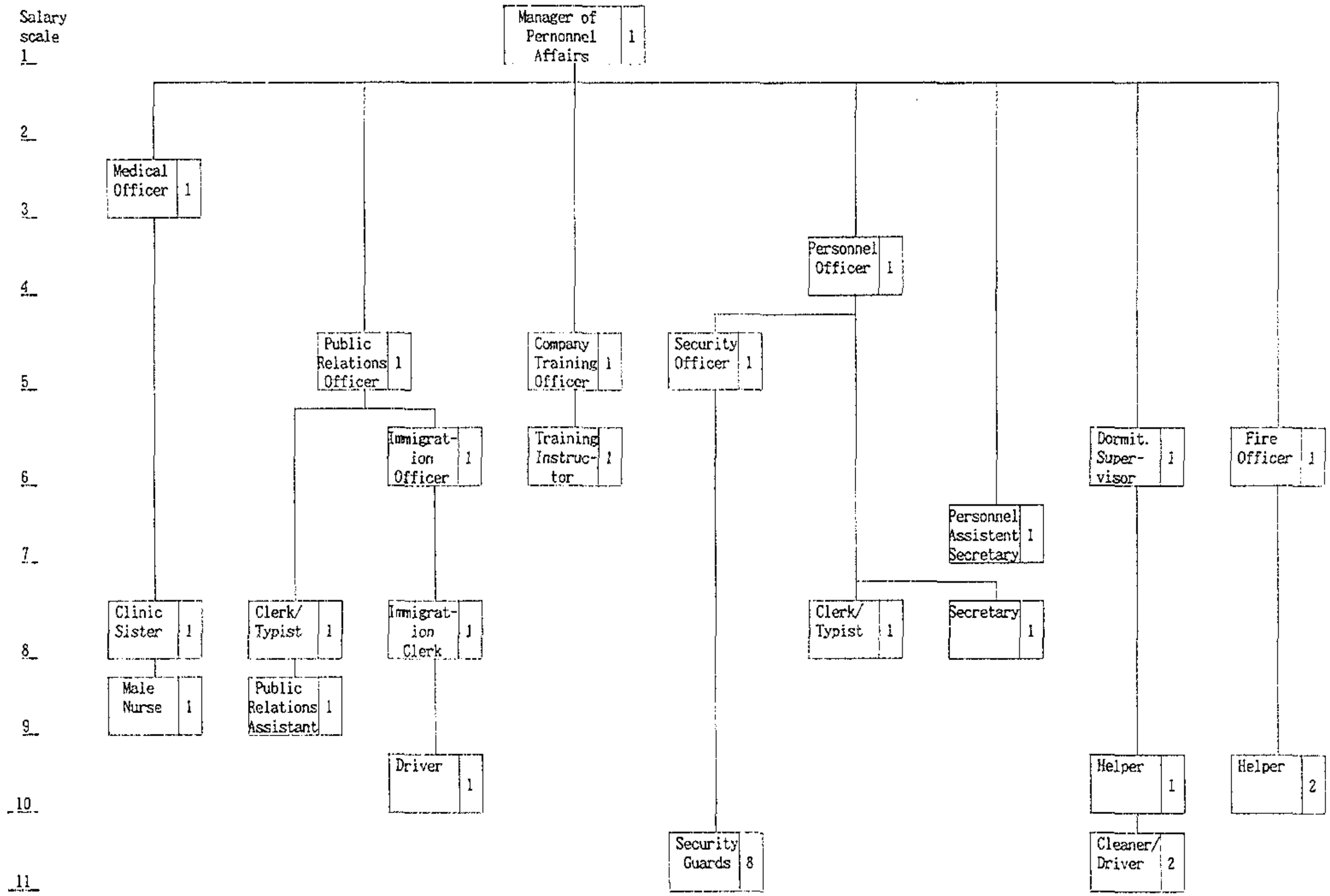


Fig. 6-1 Proposed organization for mine operation (4)

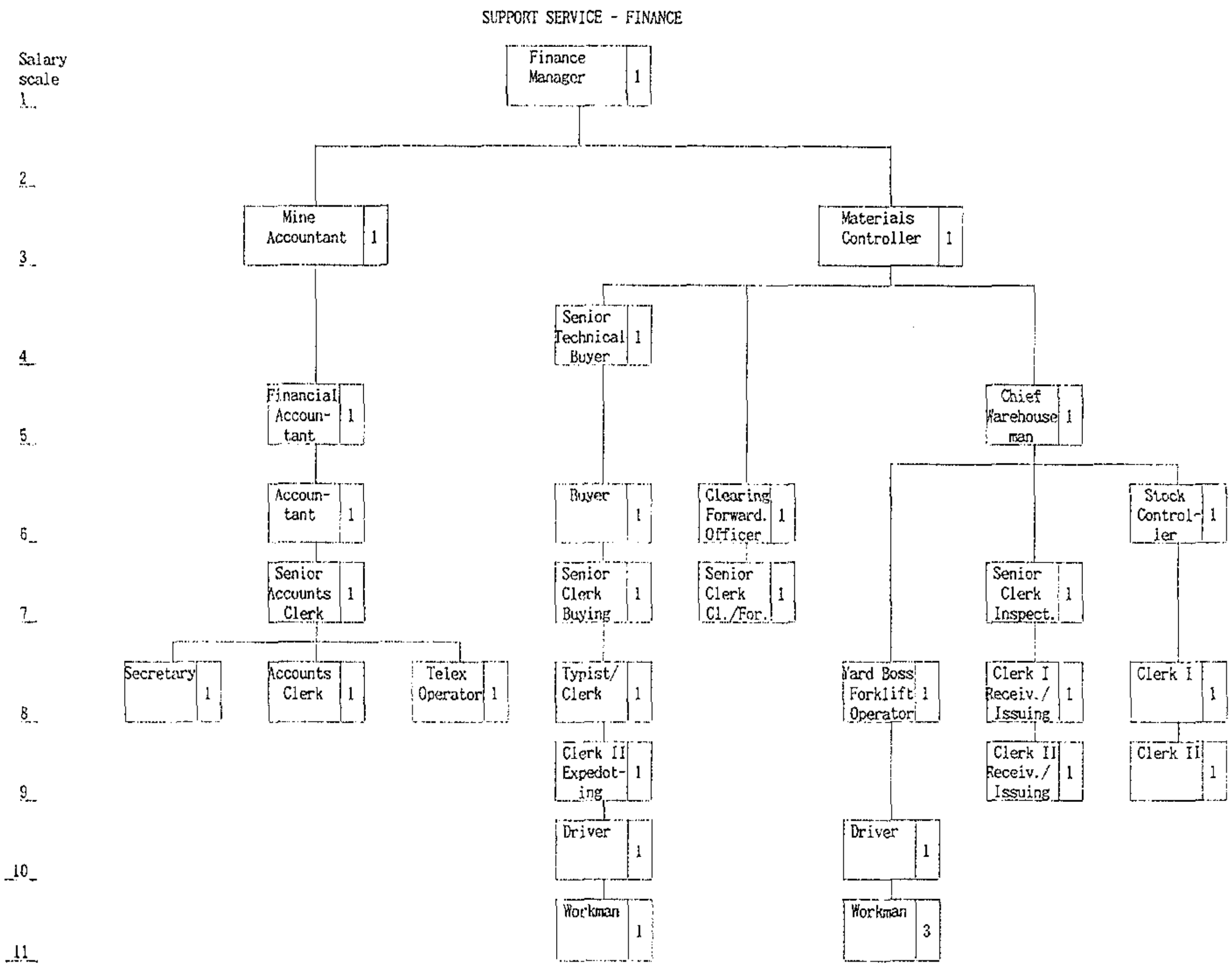


Fig. 6-1 Proposed organization for mine operation (5)

Table 6-1 Proposed manpower salary and wage

(Unit:US$)

Salary		Mining		Concentrator		Finance		Administration		Trainee		Total	
Grade	Scale	Num.	Amount	Num.	Amount	Num.	Amount	Num.	Amount	Num.	Amount	Num.	Amount
G.M.	3,300							1	3,300		0	1	3,300
M.	3,000							2	6,000		0	2	6,000
1	2,670	1	2,670	1	2,670	1	2,670	1	2,670		0	4	10,680
2	2,370	1	2,370		0		0		0		0	1	2,370
3	2,150	4	8,600	3	6,450	2	4,300	1	2,150	2	4,300	12	25,800
4	1,910	3	5,730		0	1	1,910	1	1,910		0	5	9,550
5	1,620	5	8,100	3	4,860	2	3,240	3	4,860	2	3,240	15	24,300
6	1,370	20	27,400	11	15,070	4	5,480	4	5,480	8	10,960	47	64,390
7	930	5	4,650	17	15,810	4	3,720	1	930	4	3,720	31	28,830
8	760	60	45,600	35	26,600	7	5,320	5	3,800	4	3,040	111	84,360
9	600	69	41,400	6	3,600	3	1,800	2	1,200		0	80	48,000
10	450	24	10,800	22	9,900	2	900	4	1,800		0	52	23,400
11	330	9	2,970	6	1,980	4	1,320	10	3,300		0	29	9,570
Total		201	160,290	104	86,940	30	30,660	35	37,400	20	25,260	390	340,550

Salary scale calculation

R.		O.		Ave.	x 1.40	x 2.6
min.	max.	min.	max.			
						3,300
						3,000
650	700	690	890	733	1,026	2,666
550	650	600	800	650	910	2,366
500	625	520	720	591	828	2,152
450	600	450	600	525	735	1,911
350	500	390	540	445	623	1,620
275	400	340	490	376	527	1,370
205	300	210	310	256	359	933
170	225	170	270	209	292	760
110	180	135	235	165	231	601
85	120	105	185	124	173	450
65	90	80	130	91	128	332

* 1.4 : Average factor of overtime. overhead and catering
* 2.6 : Exchange rate (US$/R.O.)

Chapter 7 Infrastructure

7-1 Transportation

7-1-1 Transportation of construction materials and operating supply

The route of transportation for mine construction machinery, equipment and materials and operating supply is appropriate from Muscat – Nizwa – Ibri – Yankul to proposed location of mine construction basing on the results of site investigation. Within the above route, 365 km of road between Muscat and Yankul is well paved and it is adequate to overland transport including large trailer truck. The local road of 13 km from Yankul to proposed mine site is not paved, but above mentioned overland transport is practicable subject to maintenance work with motor grader and others. Purchasing cost of motor grader etc. and maintenance cost of the portion of local road are included in mine planning.

7-1-2 Transportation of copper concentrate

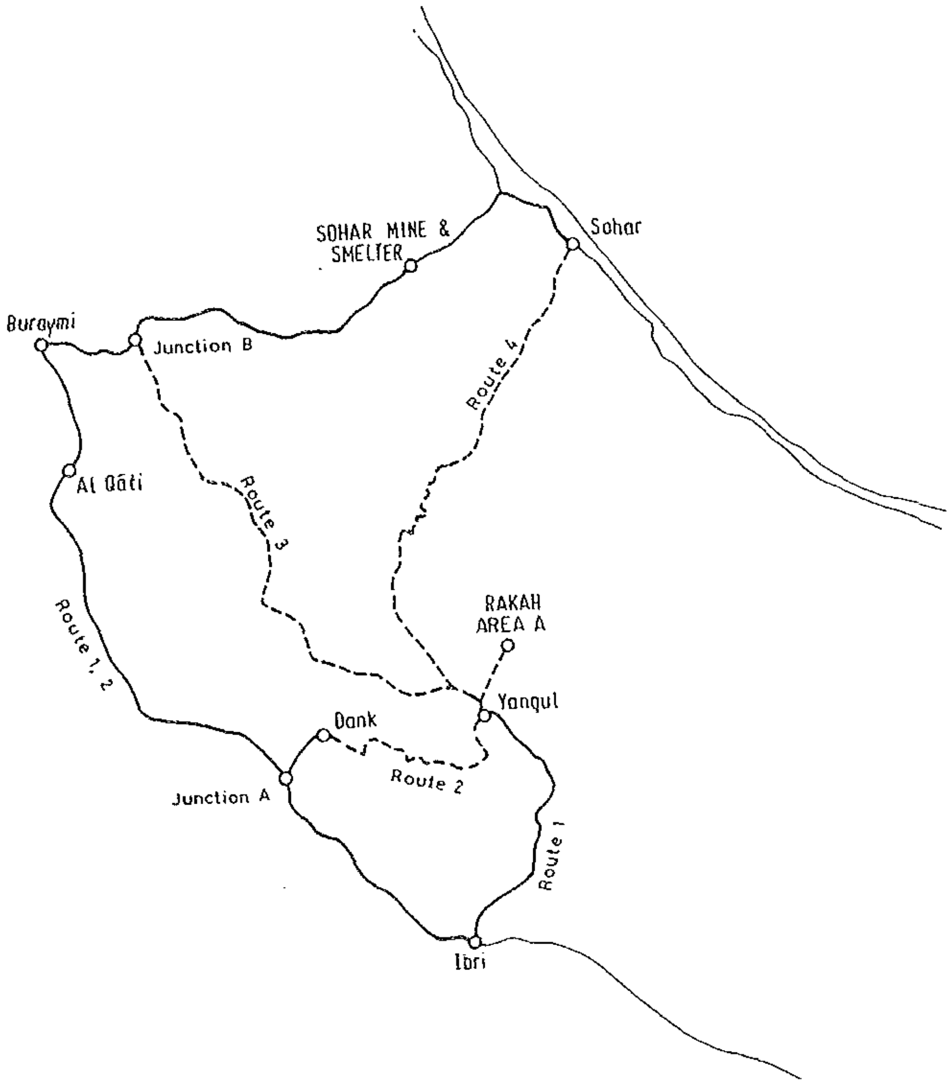
(1) Route of transportation

Copper concentrate produced at the mineral processing plant is sent to the Sohar smelter using truck transport. Site investigation of the route shown in Fig. 7-1 was carried out in the first year and the phase II (final year). Route 3 and Route 4 out of total 4 routes have shorter distance of transportation, but approximately 70% of the total extension is not paved. In addition both routes contain severe conditions in view of inclination and curvature due to traversing the Oman Mountains. As the results, these two routes are considered to be not satisfactory roads for truck transport, even though they will be paved in future. Consequently, route 1 which is completely paved from Yankul to the Sohar Smelter should be adopted for the time being, and it is planned to change with Route 2 of much shorter distance transport, if and when the road between Yankul and Dank will be paved in accordance with a road improvement work by the Government in future.

(2) Method of transportation

Volume of concentrate product (Dry)	171 t/day
Volume of concentrate product (Wet)	190 t/day
Number of trucks required	20 t dump truck × 10 unit
Loader	2.3 m^3 wheel loader × 1 unit

Copper concentrate filtered and dewatered is once piled at stock yard, and then it is loaded on trucks by wheel loader. Transportation is normally operated by one trip per day. Purchasing cost



Route	Paved road	Gravel road		Total
		Minesite -Yanqul	Yanqul-	
	(km)	(km)	(km)	(km)
1	275	13	0	288
2	186	13	42	241
3	60	11	101	172
4	40	11	110	161

Fig. 7-1 Copper concentrate haulage road

of trucks and a loader is included in capital cost of mineral processing department. Running and management of trucks are planned on the assumption to be carried out by a contracted transporter.

7-2 Water supply

The operation of mine, based on this development project requires fresh water of 1.5 cubic meter per minute, or about 2,200 cubic meter per day, mainly for using in mineral processing operation as described in 3-2-4 (7) of this report under the heading of Mill water. Whether the fresh water can be secured or not is a fatal matter for this project. Apart from a problem of water utilization right, it is considered to be able technically to secure fresh water through taking subsurface water as a result of site investigation. Construction cost of pump and delivery pipe is included in capital cost of mineral processing department. Location map of the estimated delivery pipe line is shown in Fig. 7-2.

7-3 Electricity

The estimated demand of electricity is amounted to about 4,000 kw mainly for operation of mineral processing and demand of other departments is quite small. The necessary electricity is supplied by construction of power line extension of about 23 km from the Hayl substation. Construction cost etc. are described in 3-2 of this report, and these are included in capital cost of mineral processing department. Location map of power line is also shown in Fig. 7-2.

7-4 Communicating system

Communication network in the Sultanate of Oman is composed of microwave system connecting principal cities and wire telephone within short distanced region. Communicating system of this project is planned with wire telephone and facsimile by constructing telephone wire line for a distance of about 13 km from Yanqul telephone office. Capital cost of this part is amounted to US$104,800.

7-5 Housing facilities

Housing facilities for this project are planned to set up in the urban district of Yankul. Stores, schools, clinics, mosques and other facilities necessary for lives of employees and their families are already existing in Yanqul.

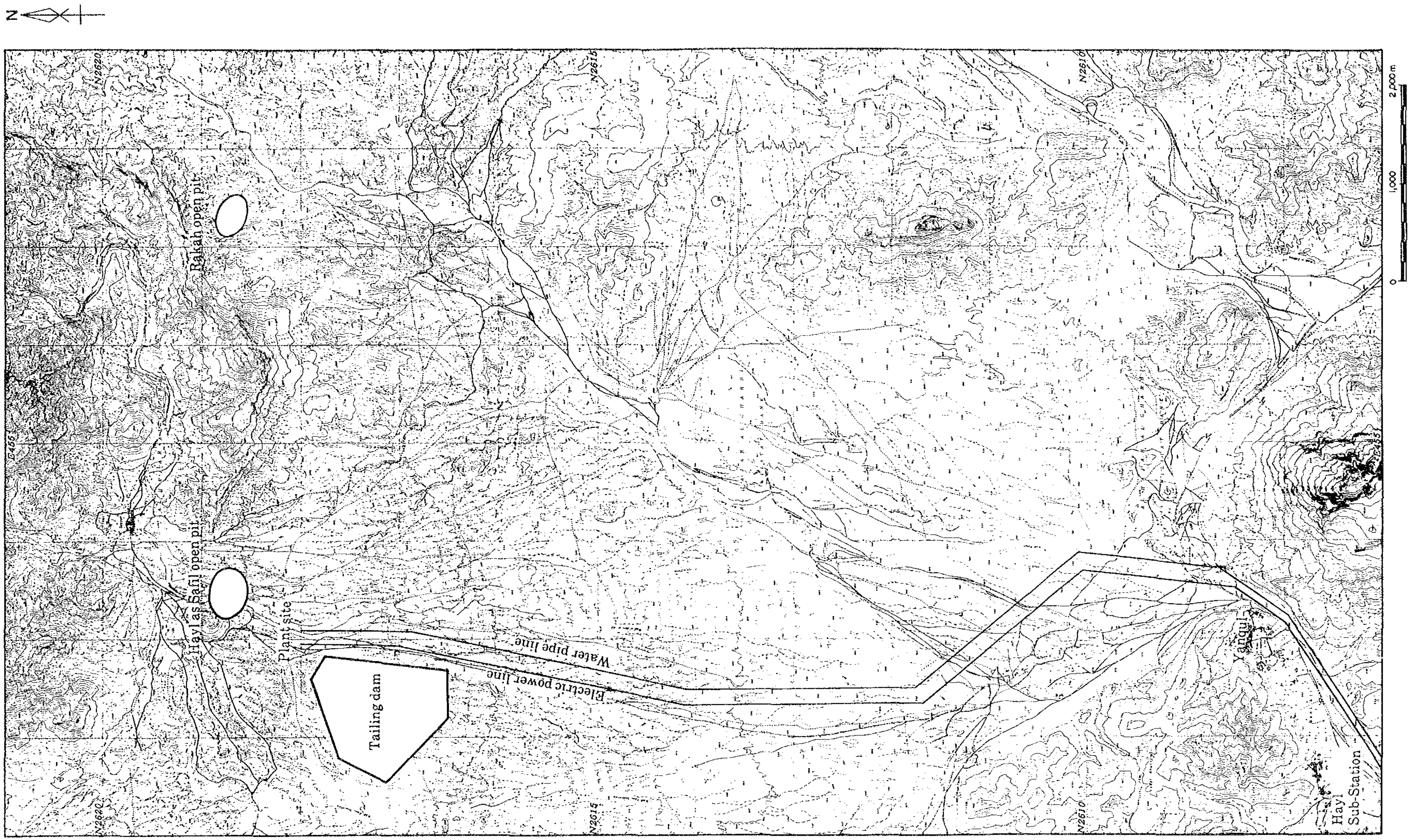


Fig. 7-2 Proposed water pipe line and electric power line

Number of company house is calculated in the manner of multiplying the number of employees of this project by a ratio between number of employees and number of company houses of the Sohar mine. Among these, high grade houses are prepared for the manager of each department, mine superintendent and metallurgical superintendent. Middle grade houses are used for married employees of over the salary grade 5. Standard grade houses are allotted for married employees of under the salary grade 6. Single employees move into a room of equivalent grade in bachelors quarter, according to their salary grade. It is expected that fairly large number of Omani employees come to work from their own house in the same way as the Sohar mine.

Company houses to be constructed are as follows and their construction cost is amounted to US$2,849,000.

For married employees:

High grade	7 houses
Middle grade	17
Standard grade	25

For single employees:

Dormitory	103 rooms

Chapter 8 Initial and additional investment, operating cost

(1) Initial investment

The initial investment necessary for this development project is amounted to US$54,815,500 in total. Breakdown of the above mentioned is shown in Table 8-1. As a direct construction cost, total of mining and mineral processing departments is amounted to US$40,735,300 and the ratio is about 75% of the total initial investment. Construction cost for supporting department and infrastructure is amounted to US$6,060,300 and the ratio is about 11% of the total initial investment.

Principle mine heavy equipment and mineral processing machinery among others are integrated basing on the quotations from respective sales agents. The other construction cost is estimated with adopting the most suitable assumption price at the present stage.

Furthermore, a 5.4% of the above mentioned direct construction cost is allocated as a contingency. And also 11.8% is estimated for engineering fee of detailed design and construction management fee.

Table 8-1 Summary of construction cost

Item	Construction cost (US$1,000)	Percentage (%)
Mining	19,172.7	35.0
Concentrator	21,562.6	39.3
Mine general items	2,935.6	5.4
Infrastructure	3,124.7	5.7
Sub-total	46,795.6	85.4
Contingency	2,506.3	4.6
Design, Engineering and Construction management fee	5,513.6	10.1
Sub-total	8,019.9	14.6
Total	54,815.5	100.0

(2) Additional investment

The capital cost estimated for the years after commencement of operation as the additional investment is used for stripping work of surface to be carried out in the 1st year of operation at the Rakah deposit and supplement or replace of mining heavy equipment. Stripping work cost of the Rakah deposit is amounted to US$1,410,300 and purchasing cost of heavy equipment is amounted to US$1,917,700.

Detailed annual programe of initial investment and additional investment is shown in Table 8-2.

(3) Operating cost

In operating cost of mining, unit cost of mining operation is calculated separately for ores from the Rakah deposit and the others (ores from the Hayl as Safil deposit and wastes from both deposits), basing on a difference of hauling distance. Unit cost of ores from the Rakah deposit is amounted to US$1.539/ton and of the others is amounted to US$1.121/ton.

Operating cost of mineral processing is amounted to US$4.462/ton. Operating cost of supporting department is amounted to US%1,566,900/year and transportation cost of copper concentrate is amounted to US$10.00/ton.

In addition to the above, US$303,100/year is allocated as a training cost of newly joined employees for a period of 5 years after commencement of operation to push forward the Omanization programe.

Annual operating cost computed with the above mentioned calculation basis is shown in Table 9-3 of the next chapter, under a heading of Direct operating cost.

Table 8-2 Initial and additional investment schedule

(Unit : US$1,000)

Items	Total	Initial Investment		Additional Investment							
		Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
(MINING)											
Production Development											
Mining Heavy Equipment Purchasing	9,310.0	7,540.0	350.0	440.0		350.0	630.0				
Clearing	22.2	22.2									
Drill & Blast Overburden & Waste											
Hayl as Safil	2,796.8	1,398.4	1,398.3								
Rakah	497.7			497.7							
Excavation, Load & Haul											
Hayl as Safil	7,918.6	3,959.3	3,959.3								
Rakah	1,410.3			1,410.3							
Wadi Diversion	545.2	545.2									
Sub-total	22,500.8	13,465.1	5,707.6	2,348.0	0.0	350.0	630.0	0.0	0.0	0.0	0.0
(CONCENTRATOR)											
Concentrator Construction											
Primary Crushing	1,615.8		1,615.8								
Secondary & Tertiary Crushing	3,166.7		3,166.7								
Grinding & Flotation	6,458.4		6,458.4								
Concentrate & Tailing	1,799.3		1,799.3								
Mill Water Supply	1,400.2		1,400.2								
General Works	3,503.6	1,500.0	2,003.6								
Concentrate Haulage	1,618.6		1,618.6								
Tailing Dam Construction	2,000.0	500.0	1,500.0								
Sub-total	21,562.6	2,000.0	19,562.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
(MINE GENERAL ITEMS)											
Land Acquisition	620.0	620.0									
Communication System	104.8	52.4	52.4								
Fueling System	75.9	75.9									
Offices for Mining & Concentrator	393.5		393.5								
Heavy Equipment Repair Shop	1,010.3	252.6	757.8								
Warehouses	116.2		116.2								
Surface Buildings	159.0		159.0								
Drainage System	156.0		156.0								
Construction Materials Transportation Cost	300.0	30.0	270.0								
Sub-total	2,935.6	1,030.8	1,904.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
(INFRASTRUCTURE)											
Access Road	275.7		275.7								
Townsite	2,849.0		2,849.0								
Sub-total	3,124.7	0.0	3,124.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total	50,123.8	16,495.9	30,299.8	2,348.0	0.0	350.0	630.0	0.0	0.0	0.0	0.0
Contingency	2,506.2	883.5	1,622.7								
Design, Engineering & Construction Management Fee	5,513.6	1,943.6	3,570.0								
Sub-total	8,019.8	2,827.1	5,192.7								
Grand Total	58,143.6	19,323.0	35,492.5	2,348.0	0.0	350.0	630.0	0.0	0.0	0.0	0.0

Chapter 9 Overall evaluation

9-1 Financial evaluation

(1) Annual production schedule

Annual production schedule has been developed based on Table 2-6, 2-7 and 3-21. It is shown in Table 9-1. The copper recovery is adjusted according to the fluctuation of crude ore copper grade. Total copper concentrate production through 8 years operation is 473,054 t (Cu 20.0%, Au 5.2g/t).

(2) Estimated annual revenue

The major premise for the financial analysis and economic analysis on this project is that the project is financially independent from Sohar mine.

The smelter terms used in this evaluation is international standard condition at present. The actual cost and recovery of Sohar smelter was not used in order to evaluate this project objectively. An objective evaluation of this project is meaningful when the project is considered as a combination with Sohar mine and smelter.

Estimated annual revenue is shown in Table 9-2. The metal price, Cu 100 USc/lb and Au 400 US$/oz, used in this calculation are the estimation from recent 30 years metal price record. The tonnage and grade of concentrate is given in Table 9-1 which is shown previously. Payable metal content is obtained by multiplying metal content in the concentrate by smelter recovery.

Gross revenue is obtained by multiplying the payable metal content by the metal price. Realization cost consists from T/C and R/C. Total net revenue is US$ 177,131,600.

(3) Annual profit (loss) and cash flow (Financial evaluation)

Annual profit (loss) and cash flow is shown in Table 9-3. Net revenue is delivered from Table 9-2. Direct operating costs are described in chapter 8.

Royalty calculation has followed the MINING PERMITS REGULATIONS, Second Schedule.

Depreciation has been obtained on the basis of "proportionate to the crude ore production". Interest rate is 10% and interest is calculated on the principal at the beginning of the year. Income tax has been calculated following the regulation.

Total profit before tax is US$ 871,800, tax is US$ 272,400 and net profit after tax is US$ 599,400. The copper break even is US¢ 99.7/lb.

Table 9-1 Annual production schedule

		Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Mining Production												
Tonnage												
Ore	(1,000t)			848.2	1,080.0	1,080.0	1,080.0	1,080.0	1,080.0	1,080.0	1,080.0	8,408.2
Waste	(1,000t)	6,000.0	6,000.0	3,850.6	3,506.8	3,151.5	2,859.7	1,869.2	1,210.4	1,091.3	854.7	30,394.2
Total	(1,000t)	6,000.0	6,000.0	4,698.8	4,586.8	4,231.5	3,939.7	2,949.2	2,290.4	2,171.3	1,934.7	38,802.4
Grade												
Copper	(%)			1.34	1.46	1.39	1.56	1.38	0.91	1.17	0.92	1.26
Gold	(g/t)			0.59	0.70	0.71	0.69	0.47	0.57	0.62	0.38	0.59
Content												
Copper	(t)			11,399.6	15,762.9	15,062.5	16,836.2	14,875.2	9,881.1	12,658.2	9,884.4	106,360.1
Gold	(Kg)			497.77	754.84	764.60	748.59	510.65	620.03	669.56	405.28	4,971.32
Concentrate												
Tonnage	(t)			50,768.1	70,405.0	67,171.2	75,367.2	66,313.6	43,432.4	56,139.1	43,456.8	473,053.5
Grade												
Copper	(%)			20.0	20.0	20.0	20.0	20.0	20.0	20.0	20.0	20.0
Gold	(g/t)			5.2	5.2	5.2	5.2	5.2	5.2	5.2	5.2	5.2
Copper recovery	(%)			89.07	89.33	89.19	89.53	89.16	87.91	88.70	87.93	88.95
Content												
Copper	(t)			10,153.6	14,081.0	13,434.2	15,073.4	13,262.7	8,686.5	11,227.8	8,691.4	94,610.7
Gold	(Kg)			263.99	366.11	349.29	391.91	344.83	225.85	291.92	225.98	2,459.88

Table 9-2 Estimated annual revenue

Metal Price			Smelter Terms					
			Copper			Gold		
Copper	(US¢/1b)	100	T/C	(us$/dmt)	65	R/C	(us$/troz)	6
Gold	(US$/troz)	400	R/C	(us¢/1b)	8.5	Recovery	(g/t-1.0)x98%	
Cu price escalation		0.0%	Recovery	(%)	96			

		Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Concentrate												
Tonnage	(t)			50,768.1	70,405.0	67,171.2	75,367.2	66,313.6	43,432.4	56,139.1	43,456.8	473,053.5
Grade												
Copper	(%)			20.0	20.0	20.0	20.0	20.0	20.0	20.0	20.0	20.0
Gold	(g/t)			5.2	5.2	5.2	5.2	5.2	5.2	5.2	5.2	5.2
Content												
Copper	(t)			10,153.6	14,081.0	13,434.2	15,073.4	13,262.7	8,686.5	11,227.8	8,691.4	94,610.7
Gold	(Kg)			263.99	366.11	349.29	391.91	344.83	225.85	291.92	225.98	2,459.88
Payable Metal Content												
Copper	(1,0001b)			21,489.3	29,801.3	28,432.4	31,901.7	28,069.5	18,384.2	23,762.7	18,394.5	200,235.6
Gold	(troz)			6,718.3	9,316.9	8,888.9	9,973.5	8,775.4	5,747.5	7,429.0	5,750.7	62,600.3
Metal Price												
Copper	(US¢/1b)			100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	
Gold	(US$/troz)			400.00	400.00	400.00	400.00	400.00	400.00	400.00	400.00	
Gross Revenue												
Copper	(US$1,000)			21,489.3	29,801.3	28,432.4	31,901.7	28,069.5	18,384.2	23,762.7	18,394.5	200,235.6
Gold	(US$1,000)			2,687.3	3,726.7	3,555.6	3,989.4	3,510.2	2,299.0	2,971.6	2,300.3	25,040.1
Total	(US$1,000)			24,176.6	33,528.0	31,988.0	35,891.1	31,579.6	20,683.2	26,734.4	20,694.8	225,275.7
Realization Costs	(US$1,000)											
Copper T/C	(US$1,000)			3,299.9	4,576.3	4,366.1	4,898.9	4,310.4	2,823.1	3,649.0	2,824.7	30,748.5
R/C	(US$1,000)			1,826.6	2,533.1	2,416.8	2,711.6	2,385.9	1,562.7	2,019.8	1,563.5	17,020.0
Gold R/C	(US$1,000)			40.3	55.9	53.3	59.8	52.7	34.5	44.6	34.5	375.6
Total	(US$1,000)			5,166.8	7,165.3	6,836.2	7,670.4	6,748.9	4,420.2	5,713.5	4,422.7	48,144.1
Net Revenue	(US$1,000)			19,009.8	26,362.7	25,151.8	28,220.7	24,830.7	16,263.0	21,020.9	16,272.1	177,131.6

Table 9-3 Annual profit (loss) and cash flow (financial evaluation)

Metal Price			IRR
Copper	(US¢/1b)	100	6.40% as R.O.I.
Gold	(US$/troz)	400	0.50% as R.O.E.

(Unit : US$1.000)

	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
(PROFIT & LOSS STATEMENT)											
1. NET REVENUE			19,009.8	26,362.7	25,151.8	28,220.7	24,830.7	16,263.0	21,020.9	16,272.1	177,131.6
2. COSTS											
Direct Operating Costs											
Mining			2,872.9	5,268.6	4,870.3	4,543.2	3,432.9	2,694.4	2,560.8	2,295.7	28,538.8
Concentrator			3,784.7	4,819.0	4,819.0	4,819.0	4,819.0	4,819.0	4,819.0	4,819.0	37,517.4
Supporting			1,566.9	1,566.9	1,566.9	1,566.9	1,566.9	1,566.9	1,566.9	1,566.9	12,535.2
Conc. Transportation			507.7	704.0	671.7	753.7	663.1	434.3	561.4	434.6	4,730.5
Training Cost			303.1	303.1	303.1	303.1	303.1	0.0	0.0	0.0	1,515.6
Sub-total			9,035.3	12,661.7	12,231.0	11,985.9	10,785.0	9,514.5	9,508.1	9,116.1	84,837.5
Royalty			925.1	1,282.9	1,224.0	1,373.4	1,208.4	791.4	1,023.0	791.9	8,620.1
Depreciation			6,001.1	7,976.5	7,976.5	8,046.5	8,204.0	8,204.0	8,204.0	8,204.0	62,816.4
Interest 10.00%			4,804.3	4,614.6	3,925.0	3,172.0	2,059.9	961.8	448.1	0.0	19,985.8
Total Costs			20,765.8	26,535.7	25,356.5	24,577.7	22,257.3	19,471.8	19,183.2	18,111.9	176,259.8
3. PROFIT BEFORE TAX			-1,756.0	-173.0	-204.7	3,643.0	2,573.4	-3,208.8	1,837.7	-1,839.9	871.8
4. INCOME TAX			0.0	0.0	0.0	96.3	176.1	0.0	0.0	0.0	272.4
5. NET PROFIT AFTER TAX			-1,756.0	-173.0	-204.7	3,546.7	2,397.3	-3,208.8	1,837.7	-1,839.9	599.4
(CASH FLOW STATEMENT)											
Net Profit After Tax			-1,756.0	-173.0	-204.7	3,643.0	2,477.1	-3,384.9	1,837.7	-1,839.9	599.4
Depreciation			6,001.1	7,976.5	7,976.5	8,046.5	8,204.0	8,204.0	8,204.0	8,204.0	62,816.4
Equity	13,703.8										13,703.8
Loan	6,181.0	41,862.4									48,043.4
Capital Expenditure	-19,322.9	-35,492.4									-54,815.4
Interest During Construction	-561.9	-4,111.2									-4,673.1
Additional Capital Expenditure			-2,348.0	0.0	-350.0	-630.0	0.0	0.0	0.0	0.0	-3,328.0
Working Capital Increase (Decrease)		-2,258.8	0.0	-906.6	107.7	61.3	300.2	317.6	1.6	2,377.0	0.0
Loan Repayment			-1,897.1	-6,896.8	-7,529.5	-11,120.8	-10,981.3	-5,136.7	-4,481.2	0.0	-48,043.4
Net Generated Cash			0.0	0.0	0.0	0.0	0.0	0.0	5,562.1	8,741.1	14,303.2
PRINCIPAL		48,043.4	46,146.4	39,249.5	31,720.0	20,599.2	9,617.9	4,481.2	0.0	0.0	
(RATE OF RETURN)											
Net Generated Cash			0.0	0.0	0.0	0.0	0.0	0.0	5,562.1	8,741.1	14,303.2
Capital Expenditure	-19,884.9	-41,862.4									-61,747.3
Repayment Flow Adjustment			1,897.1	6,896.8	7,529.5	11,120.8	10,981.3	5,136.7	4,481.2	0.0	48,043.4
Interest Flow Adjustment			4,804.3	4,614.6	3,925.0	3,172.0	2,059.9	961.8	448.1	0.0	19,985.8
Cash Flow Out and In	-19,884.9	-41,862.4	6,701.4	11,511.5	11,454.4	14,292.8	13,041.2	6,098.5	10,491.4	8,741.1	20,585.2
Discounted Cash Flow at 6.40%	-18,689.3	-36,979.9	5,563.9	8,982.9	8,400.9	9,852.4	8,449.2	3,713.5	6,004.4	4,701.9	-0.0
(RATE OF RETURN TO THE EQUITY)											
Net Generated Cash			0.0	0.0	0.0	0.0	0.0	0.0	5,562.1	8,741.1	14,303.2
Capital Expenditure	-13,703.8										-13,703.8
Cash Flow Out and In	-13,703.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	5,562.1	8,741.1	599.4
Discounted Cash Flow at 0.50%	-13,635.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	5,318.7	8,317.2	-0.0

Tax in cash flow statement is to be deducted in the next year.

Equity is 25% of the initial investment. The rest which is 75% of the initial investment, interest during construction period and working capital are to be covered by loan. Interest during construction period is depreciated.

Additional investment is given from Table 8-2. Working capital has been increased/decreased so that it is maintained at a level of 3 months operation cost.

All the net generated cash has been used for loan repayment in order to reduce the principal as early as possible and to reduce interest payment.

The IRR to the project and IRR to the equity have been calculated. They are 6.40% and 0.50% respectively.

9-2 Economic evaluation

Annual profit (loss) and cash flow (Economic evaluation) is shown in Table 9-4. "Annual production schedule" and "Estimated annual revenue" are the same as former section.

In this economic evaluation, royalty and income tax are exempted. Regarding labor cost, there is no room for adjustment because Oman is highly depending on the foreign workers. Operating supply cost also can not be adjusted because most of operating supply is imported from foreign countries.

Economic IRR to the project is 8.90% and IRR to the equity is 7.96%.

9-3 Sensitivity analysis

Sensitivity analysis has been conducted on the financial IRR to the project. Results are shown in Table 9-5 and Fig. 9-1. The items tested in this analysis are copper price, capital cost and operating cost.

The base conditions are the same as that of financial evaluation.

The analysis has proven that this project is most sensitive to copper price.

Table 9-4 Annual profit (loss) and cash flow (economic evaluation)

Metal Price			IRR
Copper	(US¢/lb)	100	8.90% as R.O.I.
Gold	(US$/troz)	400	7.96% as R.O.E.

(Unit : US$1,000)

	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
(PROFIT & LOSS STATEMENT)											
1. NET REVENUE			19,009.8	26,362.7	25,151.8	28,220.7	24,830.7	16,263.0	21,020.9	16,272.1	177,131.6
2. COSTS											
Direct Operating Costs											
Mining			2,872.9	5,268.6	4,870.3	4,543.2	3,432.9	2,694.4	2,560.8	2,295.7	28,538.8
Concentrator			3,784.7	4,819.0	4,819.0	4,819.0	4,819.0	4,819.0	4,819.0	4,819.0	37,517.4
Supporting			1,566.9	1,566.9	1,566.9	1,566.9	1,566.9	1,566.9	1,566.9	1,566.9	12,535.2
Conc. Transportation			507.7	704.0	671.7	753.7	663.1	434.3	561.4	434.6	4,730.5
Training Cost			303.1	303.1	303.1	303.1	303.1	0.0	0.0	0.0	1,515.6
Sub-total			9,035.3	12,661.7	12,231.0	11,985.9	10,785.0	9,514.5	9,508.1	9,116.1	84,837.5
Royalty			0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Depreciation			6,001.1	7,976.5	7,976.5	8,046.5	8,204.0	8,204.0	8,204.0	8,204.0	62,816.4
Interest 10.00%			4,804.3	4,522.1	3,694.9	2,796.5	1,509.6	225.9	0.0	0.0	17,553.4
Total Costs			19,840.7	25,160.3	23,902.4	22,828.9	20,498.6	17,944.5	17,712.1	17,320.1	165,207.4
3. PROFIT BEFORE TAX			-830.9	1,202.4	1,249.4	5,391.8	4,332.1	-1,681.5	3,308.8	-1,048.0	11,924.2
4. INCOME TAX			0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
5. NET PROFIT AFTER TAX			-830.9	1,202.4	1,249.4	5,391.8	4,332.1	-1,681.5	3,308.8	-1,048.0	11,924.2
(CASH FLOW STATEMENT)											
Net Profit After Tax			-830.9	1,202.4	1,249.4	5,391.8	4,332.1	-1,681.5	3,308.8	-1,048.0	11,924.2
Depreciation			6,001.1	7,976.5	7,976.5	8,046.5	8,204.0	8,204.0	8,204.0	8,204.0	62,816.4
Equity	13,703.8										13,703.8
Loan	6,181.0	41,862.4									48,043.4
Capital Expenditure	-19,322.9	-35,492.4									-54,815.4
Interest During Construction	-561.9	-4,111.2									-4,673.1
Additional Capital Expenditure			-2,348.0	0.0	-350.0	-630.0	0.0	0.0	0.0	0.0	-3,328.0
Working Capital Increase (Decrease)		-2,258.8	0.0	-906.6	107.7	61.3	300.2	317.6	1.6	2,377.0	0.0
Loan Repayment			-2,822.2	-8,272.3	-8,983.5	-12,869.6	-12,836.3	-2,259.5	0.0	0.0	-48,043.4
Net Generated Cash			0.0	0.0	0.0	0.0	0.0	4,580.6	11,514.4	9,533.0	25,628.0
PRINCIPAL		48,043.4	45,221.3	36,949.0	27,965.4	15,095.8	2,259.5	0.0	0.0	0.0	
(RATE OF RETURN)											
Net Generated Cash			0.0	0.0	0.0	0.0	0.0	4,580.6	11,514.4	9,533.0	25,628.0
Capital Expenditure	-19,884.9	-41,862.4									-61,747.3
Repayment Flow Adjustment			2,822.2	8,272.3	8,983.5	12,869.6	12,836.3	2,259.5	0.0	0.0	48,043.4
Interest Flow Adjustment			4,804.3	4,522.1	3,694.9	2,796.5	1,509.6	225.9	0.0	0.0	17,553.4
Cash Flow Out and In	-19,884.9	-41,862.4	7,626.5	12,794.4	12,678.4	15,666.2	14,345.9	7,066.0	11,514.4	9,533.0	29,477.6
Discounted Cash Flow at 8.90%	-18,260.2	-35,301.2	5,905.7	9,098.1	8,279.0	9,394.2	7,899.6	3,573.0	5,346.7	4,065.0	-0.0
(RATE OF RETURN TO THE EQUITY)											
Net Generated Cash			0.0	0.0	0.0	0.0	0.0	4,580.6	11,514.4	9,533.0	25,628.0
Capital Expenditure	-13,703.8										-13,703.8
Cash Flow Out and In	-13,703.8	0.0	0.0	0.0	0.0	0.0	0.0	4,580.6	11,514.4	9,533.0	11,924.2
Discounted Cash Flow at 7.96%	-12,693.4	0.0	0.0	0.0	0.0	0.0	0.0	2,482.1	5,779.3	4,432.0	-0.0

Table 9-5 Sensitivity analysis on the FIRR (project)

Cu price	Capital cost	Operating cost +20%	+10%	0%	-10%	-20%
-20%	+20%	-16.24	-12.59	-9.34	-6.37	-3.63
	+10%	-15.02	-11.25	-7.88	-4.81	-1.96
	0%	-13.66	-9.75	-6.25	-3.05	-0.07
	-10%	-12.12	-8.04	-4.39	-1.04	2.08
	-20%	-10.35	-6.08	-2.23	1.30	4.60
-10%	+20%	-8.49	-5.63	-2.96	-0.45	1.92
	+10%	-7.03	-4.05	-1.28	1.32	3.80
	0%	-5.38	-2.28	0.61	3.34	5.89
	-10%	-3.50	-0.26	2.78	5.61	8.19
	-20%	-1.33	2.09	5.28	8.15	10.87
0%	+20%	-2.31	0.13	2.46	4.68	6.73
	+10%	-0.63	1.92	4.34	6.58	8.67
	0%	1.27	3.93	6.40	8.68	10.88
	-10%	3.44	6.18	8.70	11.10	13.43
	-20%	5.91	8.72	11.38	13.94	16.43
+10%	+20%	2.98	5.16	7.15	9.04	10.88
	+10%	4.86	7.03	9.08	11.07	13.00
	0%	6.89	9.13	11.29	13.38	15.42
	-10%	9.19	11.55	13.84	16.06	18.24
	-20%	11.87	14.39	16.84	19.23	21.56
+20%	+20%	7.55	9.41	11.23	13.00	14.72
	+10%	9.48	11.44	13.35	15.21	17.03
	0%	11.69	13.76	15.77	17.74	19.67
	-10%	14.24	16.44	18.58	20.68	22.74
	-20%	17.24	19.59	21.89	24.15	26.37

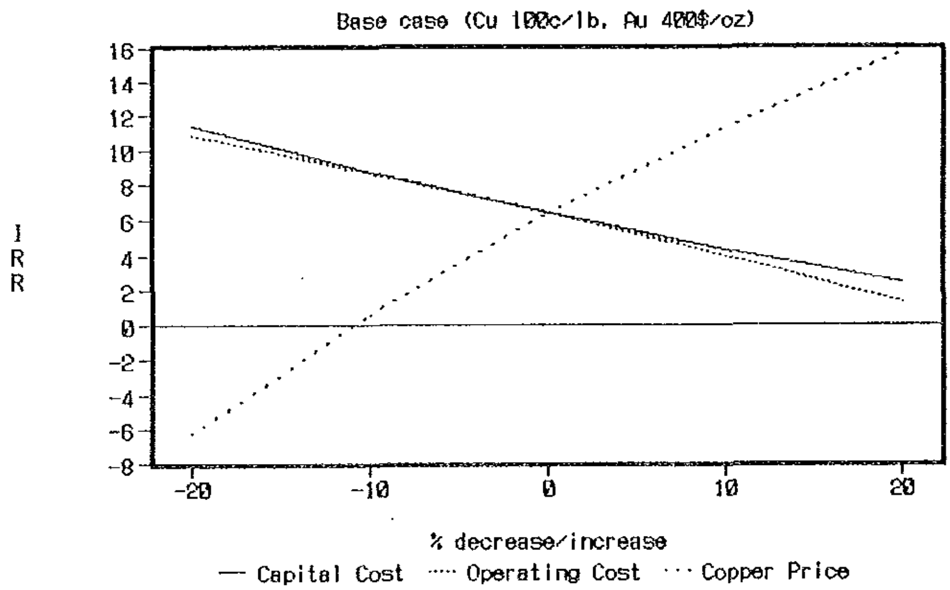


Fig. 9-1 FIRR sensitivity analysis

Chapter 10 Conclusion

(1) Open pit mining method is the most suitable mining method for these deposits. The cut-off grade is Cu 0.35%. Maximum allowable stripping ratio for Hayl as Safil is 11.3, and for Rakah it is 6.8.

(2) 3,000 t/day, 8 years operation is optimum operation size for this project.

(3) Minable ore reserves are;

	Tonnage (t)	Cu (%)	Au (g/t)
Hayl as Safil deposit	6,284,436	1.28	0.58
Rakah deposit	2,123,833	1.22	0.62
Total	8,408,269	1.26	0.59

(4) Hayl as Safil deposit requires 12,000,000 t of pre-stripping and Rakah deposit does 3,136,000t. Wadi al Hayl al Ali needs to be diverted.

(5) The flotation characteristics of head samples are;

Hayl as Safil ore; The flotation selectivity of minerals of this ore was not so good because of fine mineral combination of chalcopyrite and pyrite and the oxidization of ore.

Rakah stockwork ore; The selectivity of minerals of this ore was good because the grain size of chalcopyrite and pyrite were coarse and was easily liberated.

Rakah massive ore; This ore could not be processed by flotation because the ore consists of highly oxidized mineral combination of very fine grain of copper minerals and iron sulfide minerals.

(6) The comminution process is designed in conventional way of three stage crushing and one stage grinding.

Bulk and differential flotation system is taken as a most feasible flotation process of this kind of ore.

Estimated concentrate Cu grade is 20.0%, Au grade is 5.2g/t, copper recovery is 88.9% and gold recovery is 49.3%.

(7) Waste dumps and tailing dam are designed.

(8) Supporting department has been planned. Organization and manning plan have been developed.

(9) Transportation, water, electricity and housing facilities were designed.

(10) Total construction cost is US$ 54,815,500.

(11) Financial IRR to the project is 6.40% and IRR to the equity is 0.50%.

Sensitivity analysis on the financial IRR has shown that the project is most sensitive to copper price.

(12) Economic IRR to the project is 8.90% and IRR to the equity is 7.96%.

FIGURES

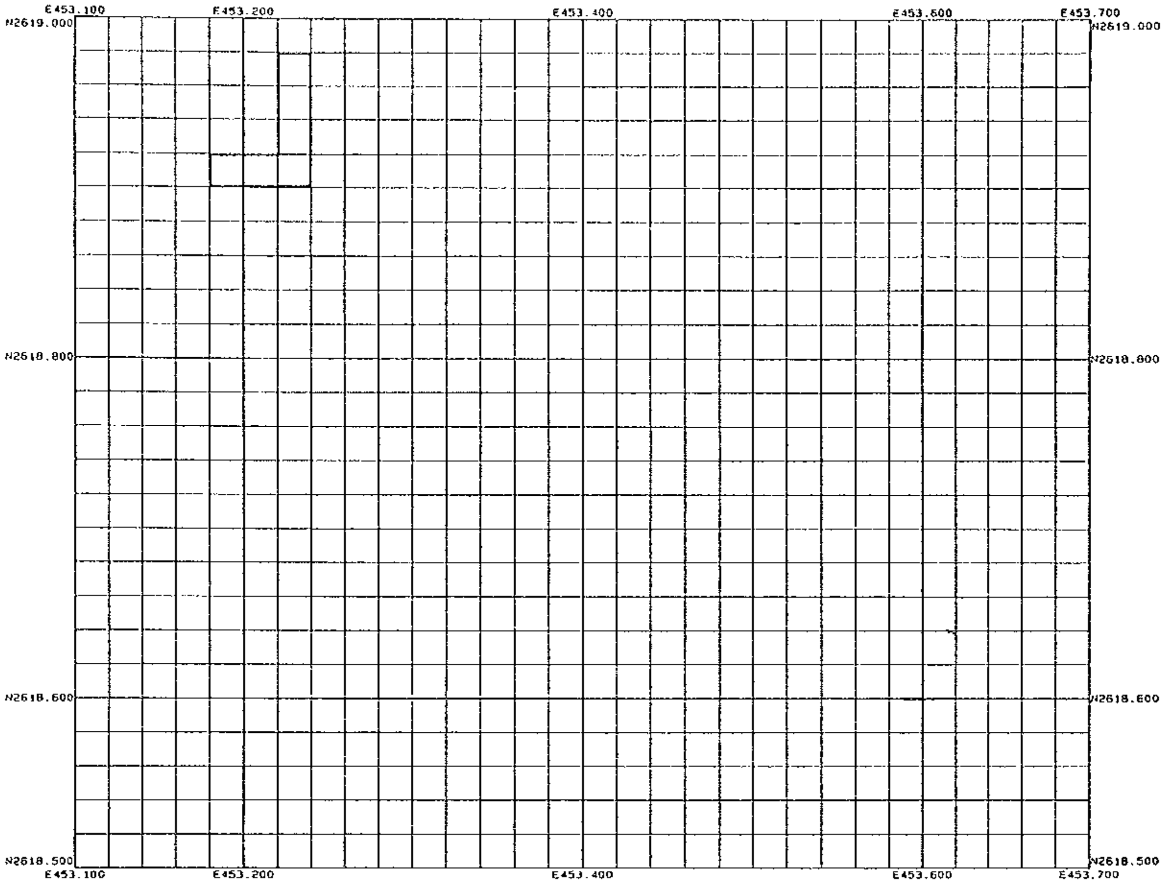
TABLES

APPENDICES

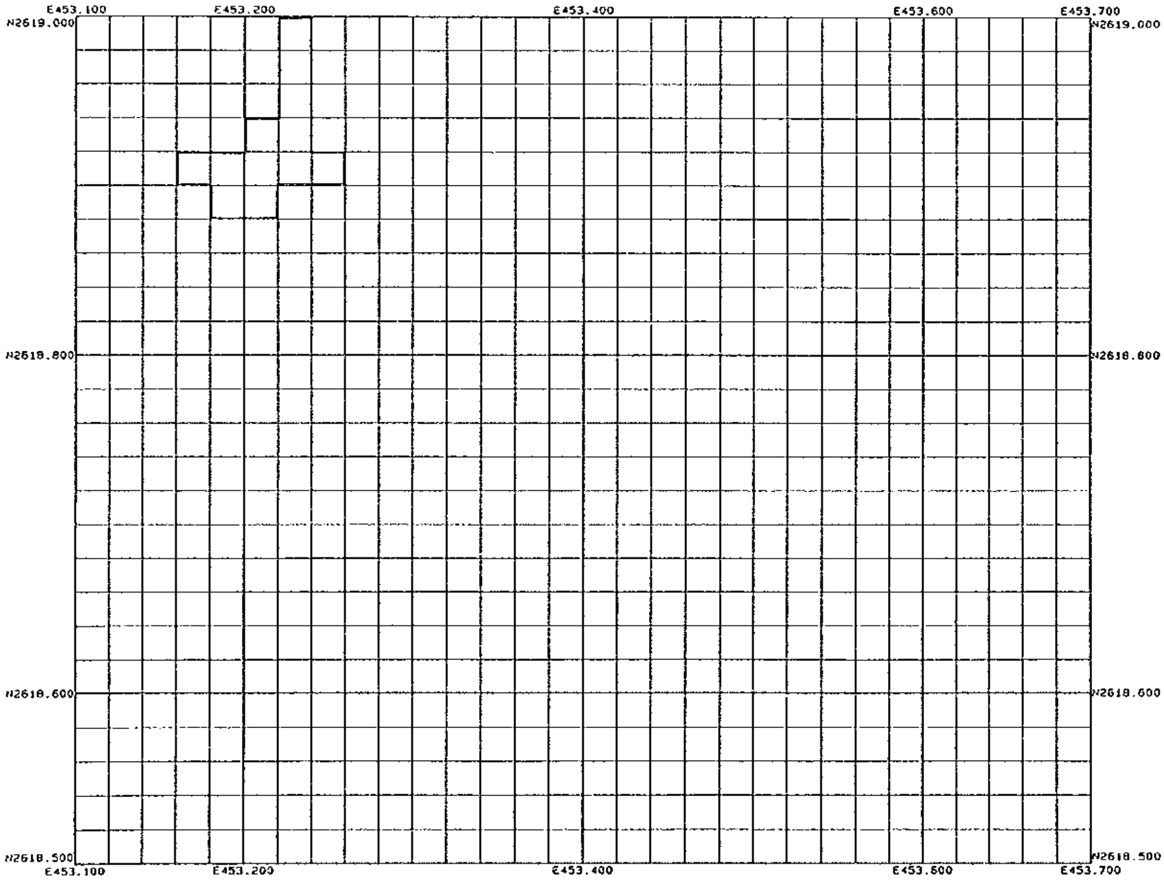
Appendix 1

Plan maps for each mining level of the Hayl as Safil deposit

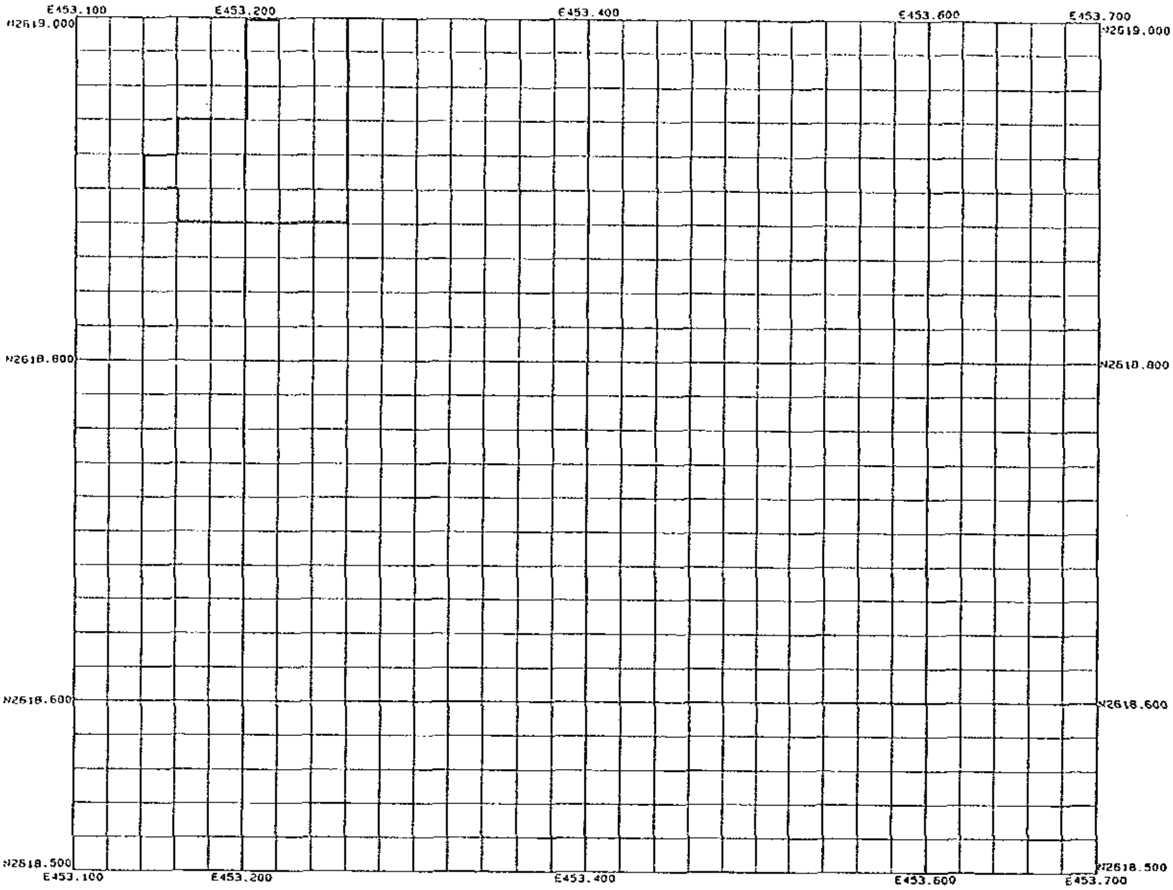
Hayl as Safil 780 mL



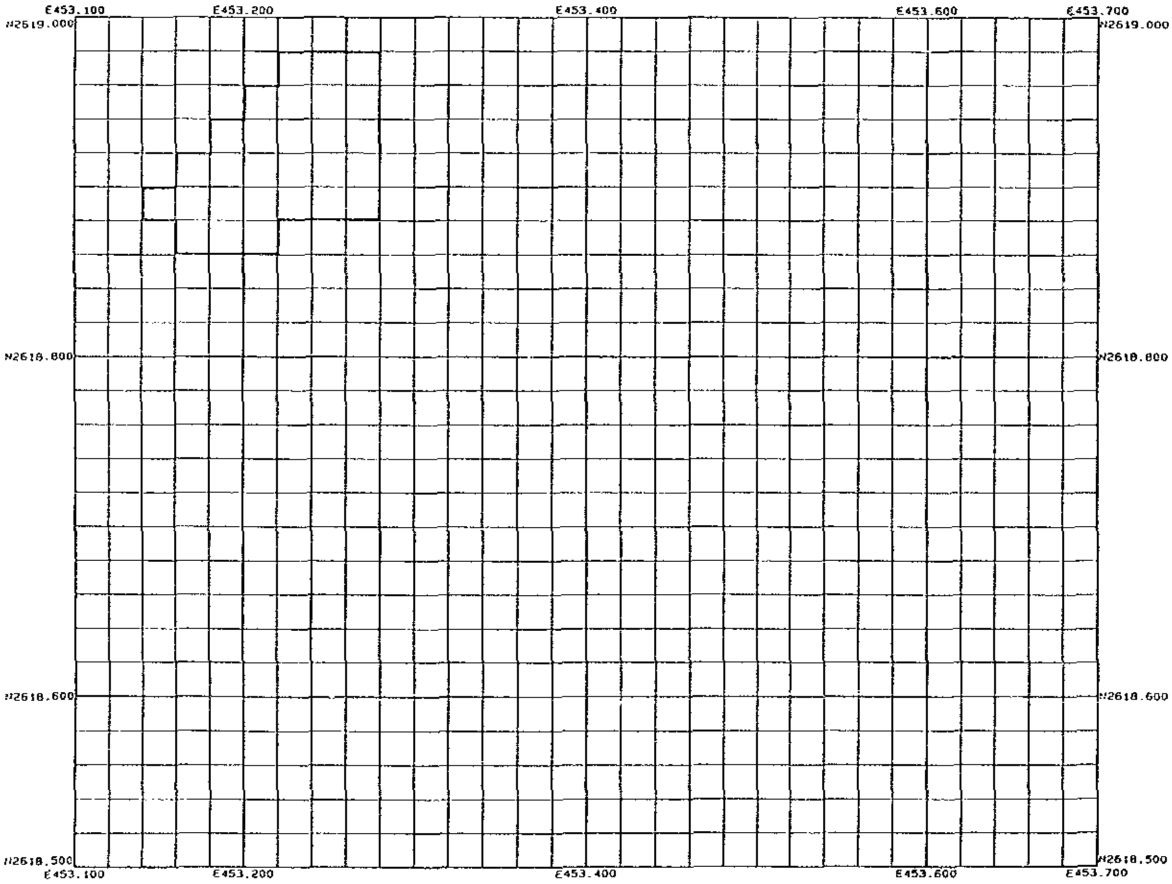
Hayl as Safil 770 mL



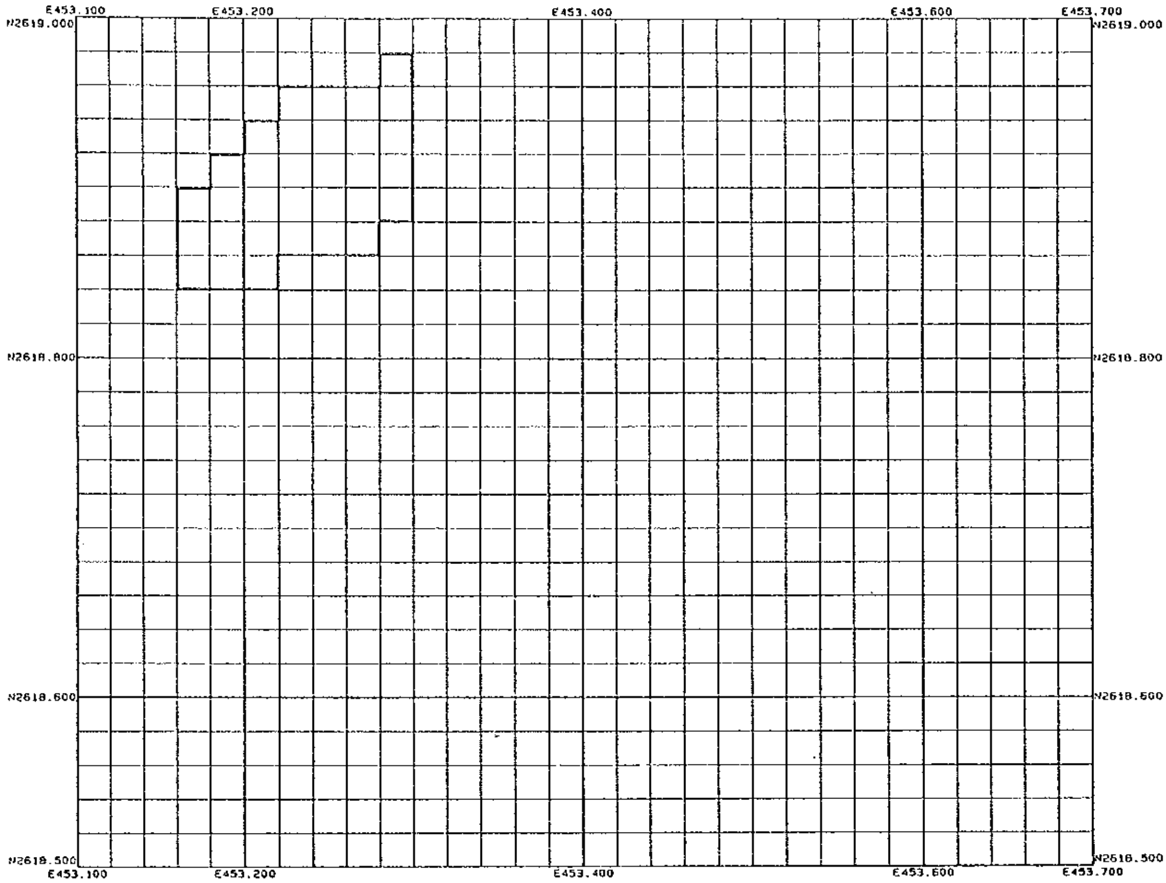
Hayl as Safil 760 mL



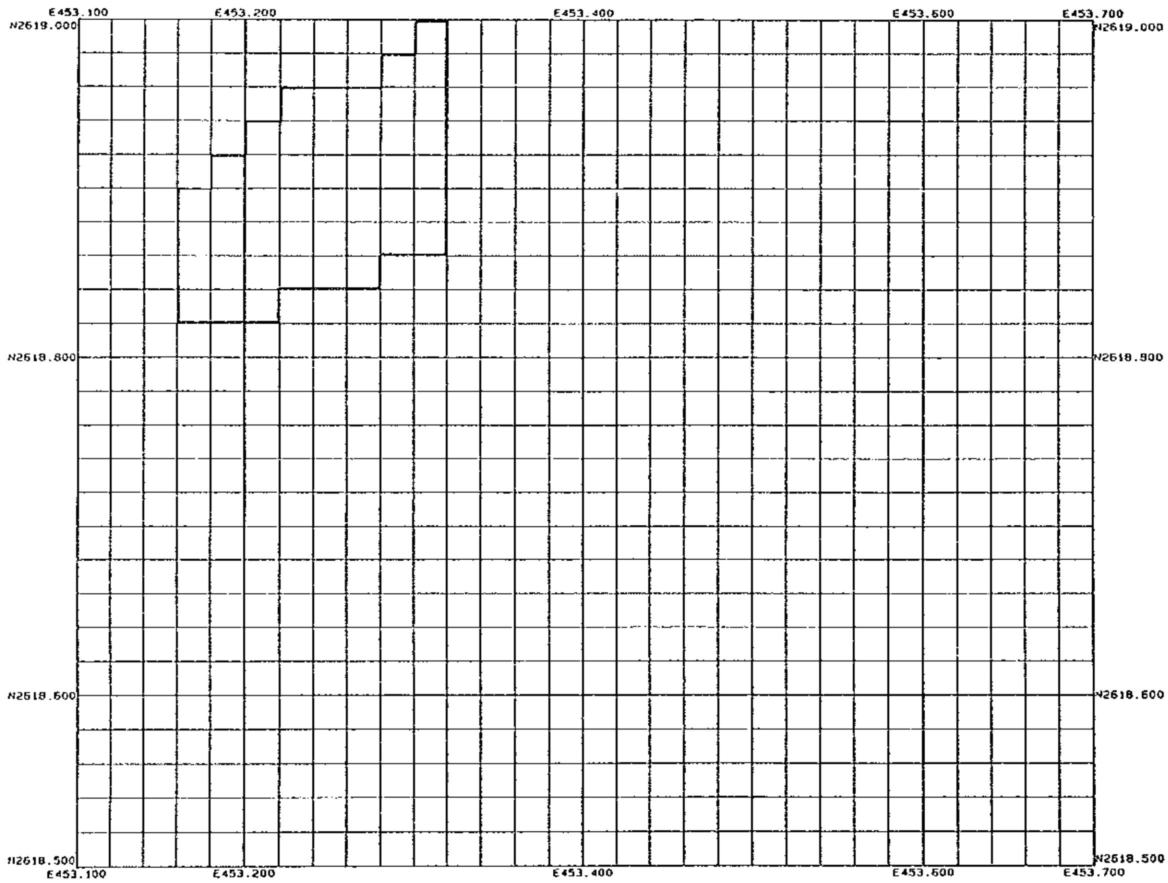
Hayl as Safil 750 mL



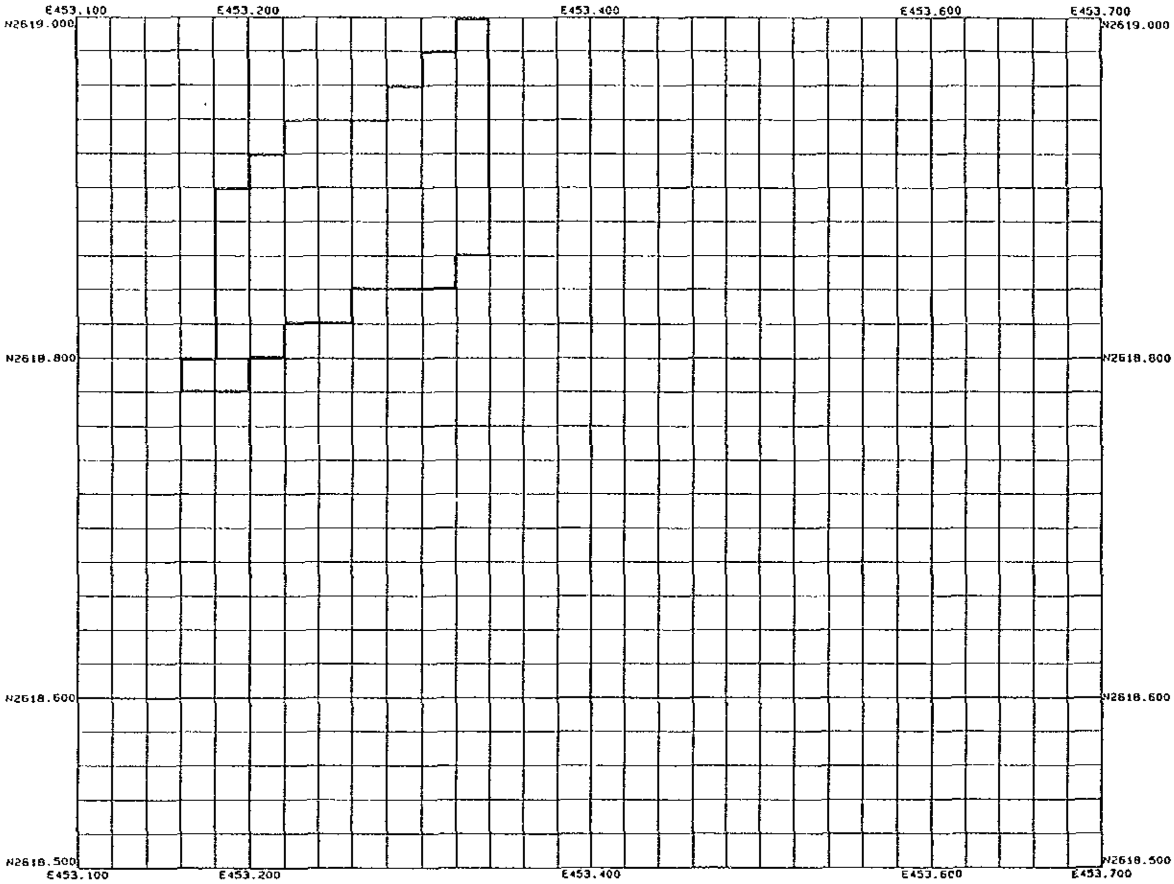
Hayl as Safil 740 mL



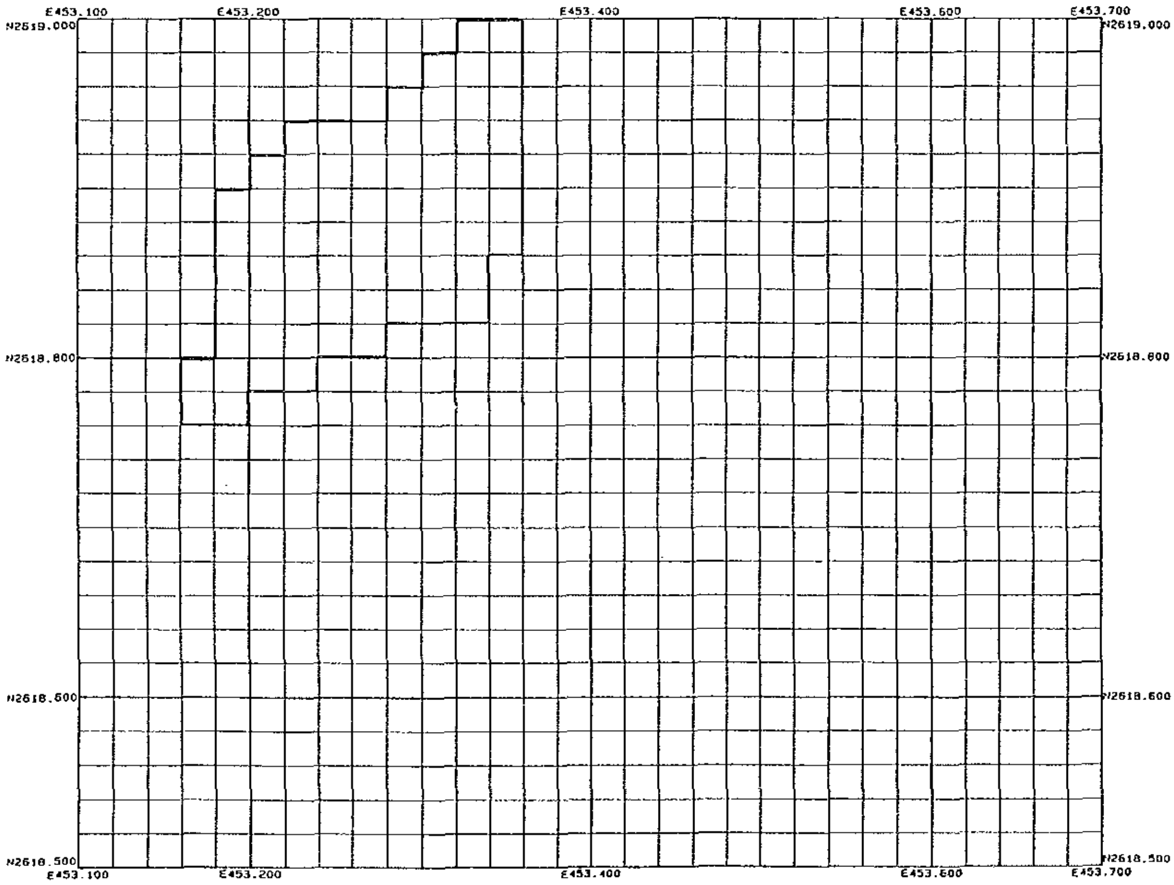
Hayl as Safil 730 mL



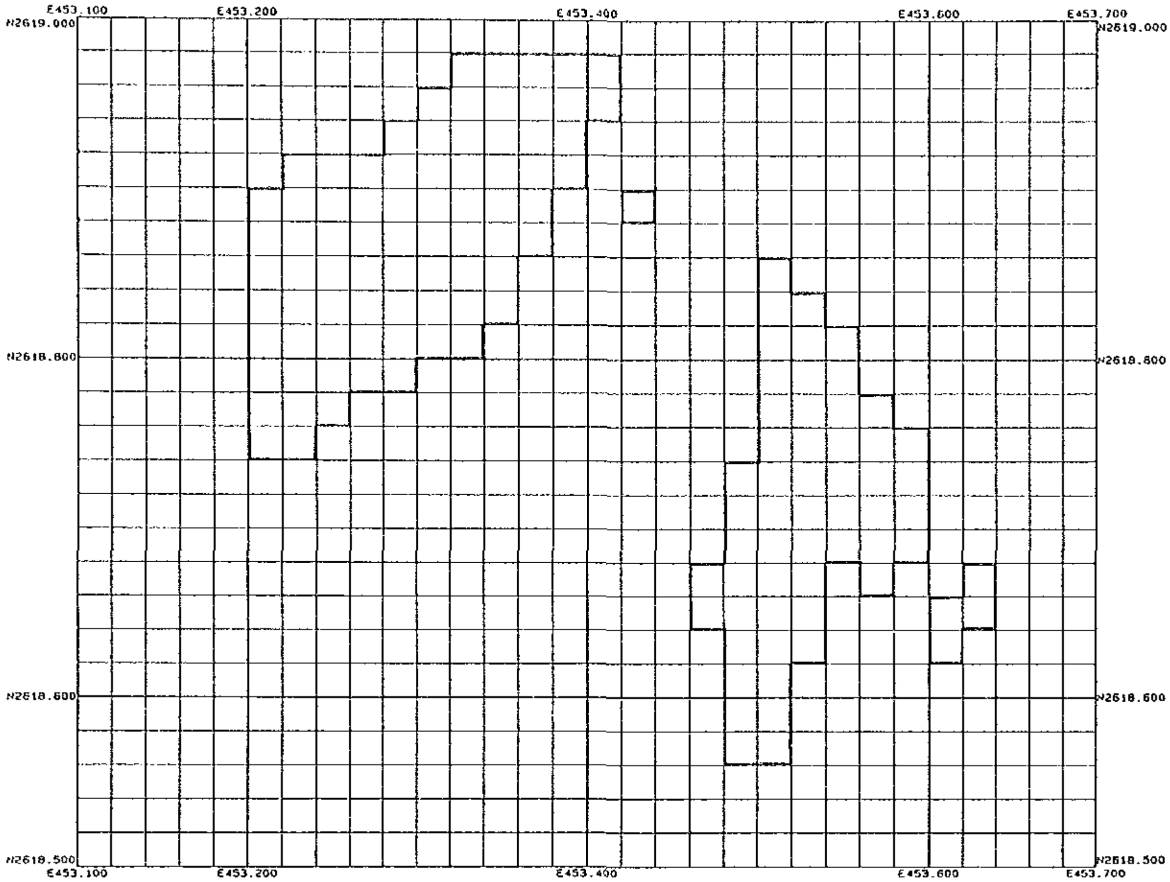
Hayl as Safil 720 mL



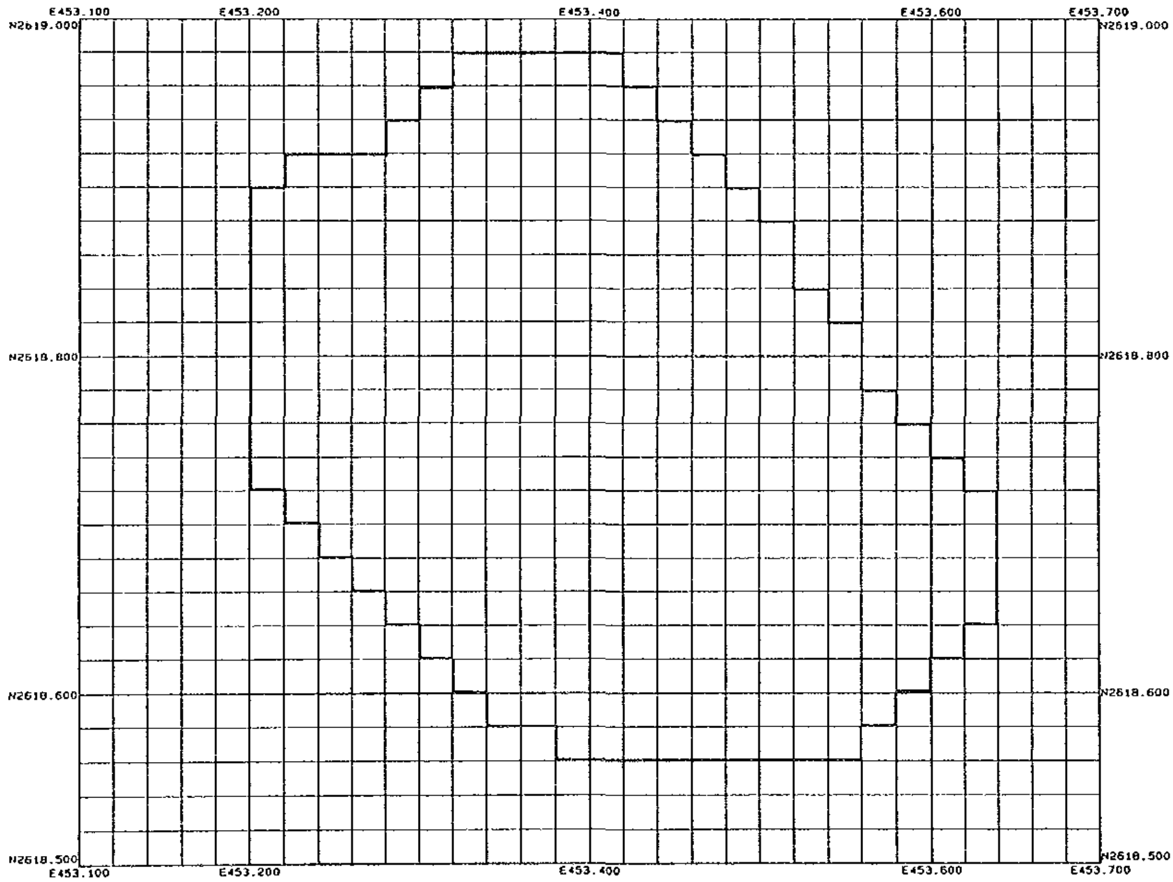
Hayl AS Safil 710 mL



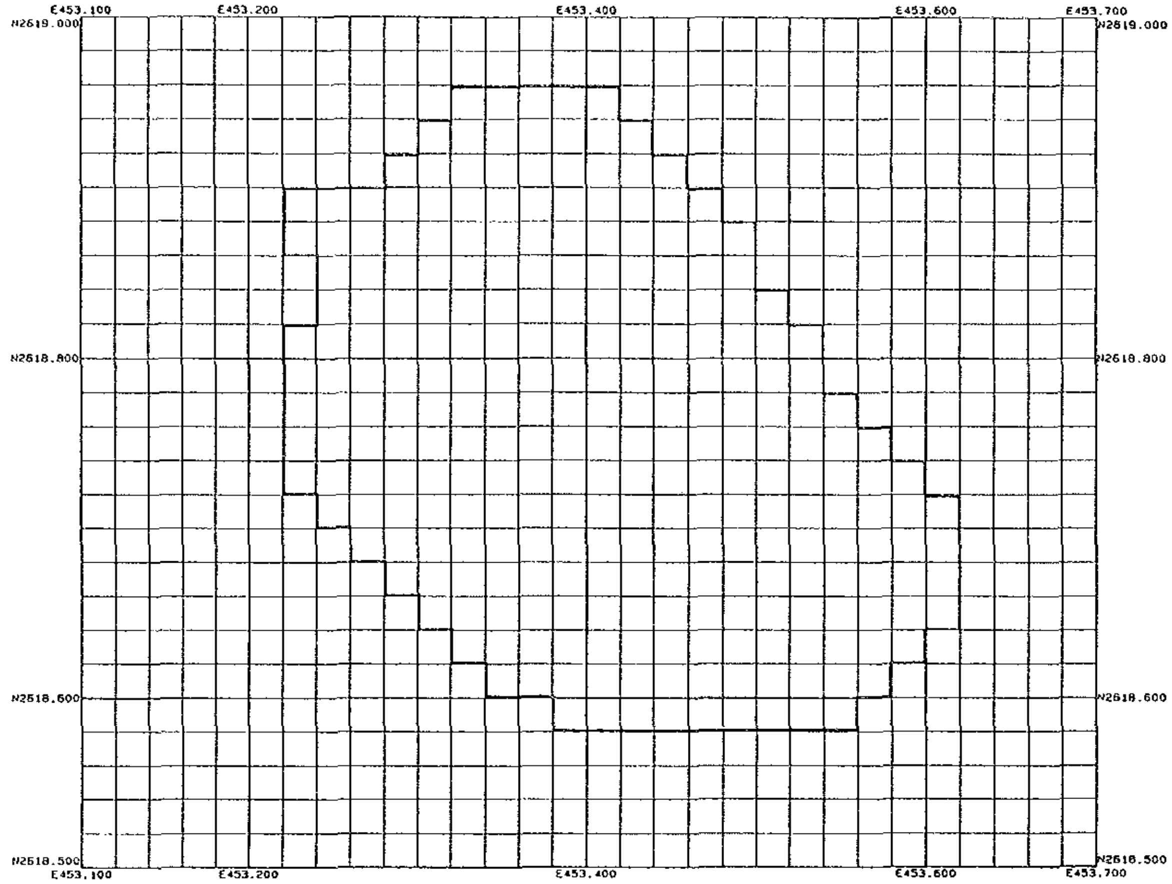
Hayl as Safil 700 mL



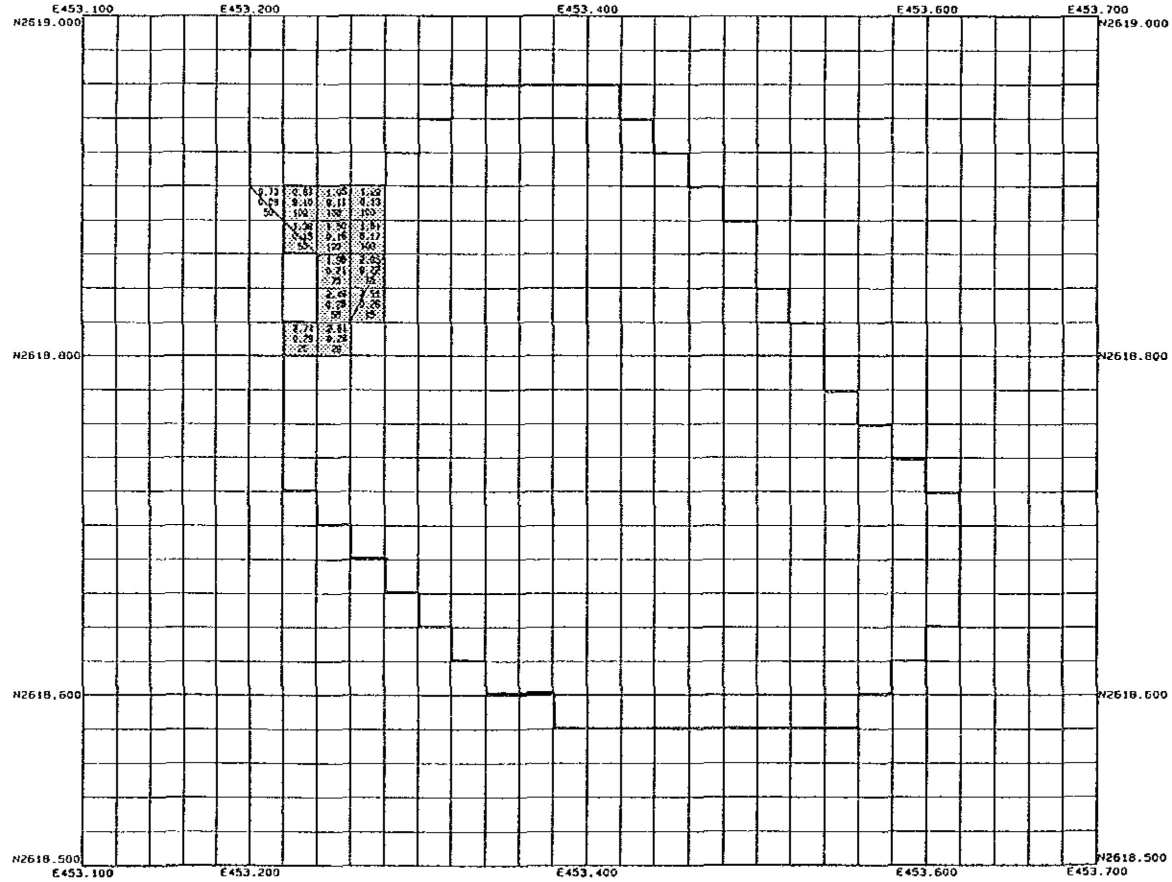
Hayl as Safil 690 mL



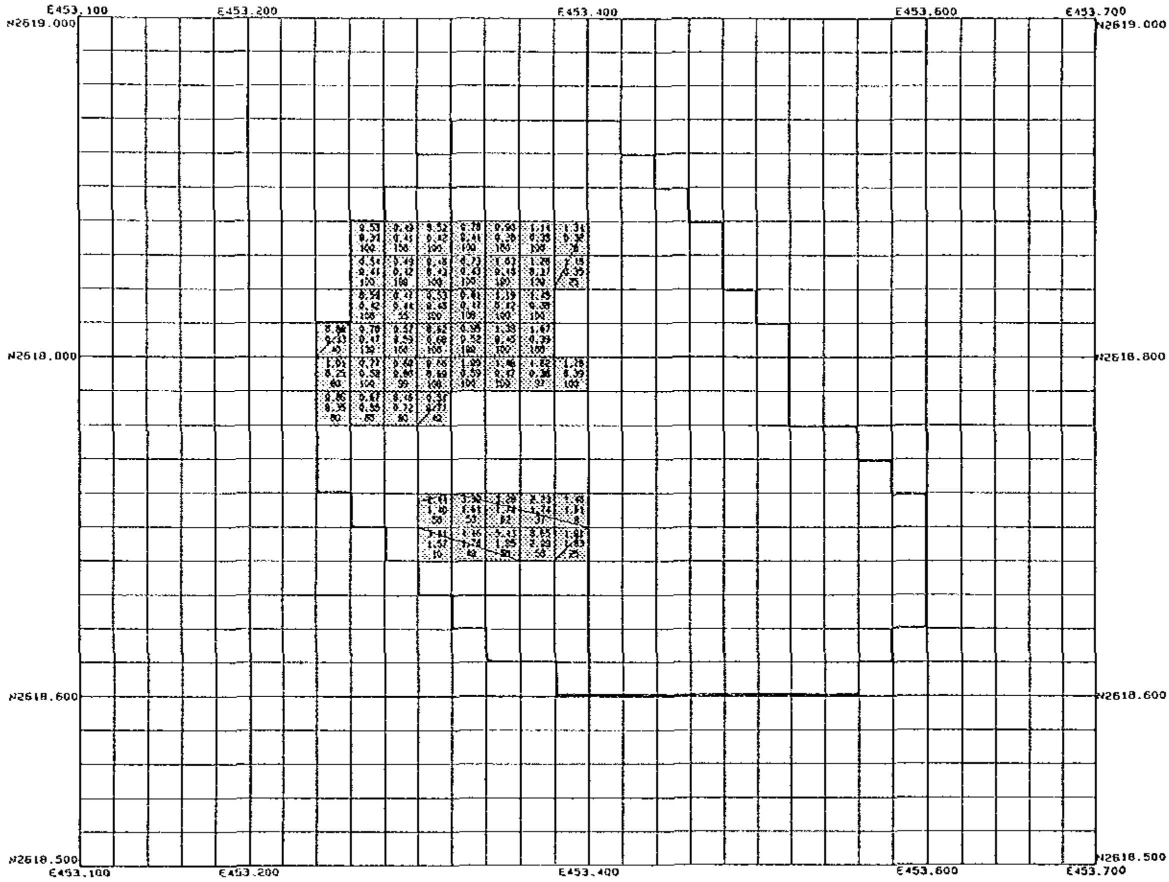
Hayl as Safil 680 mL



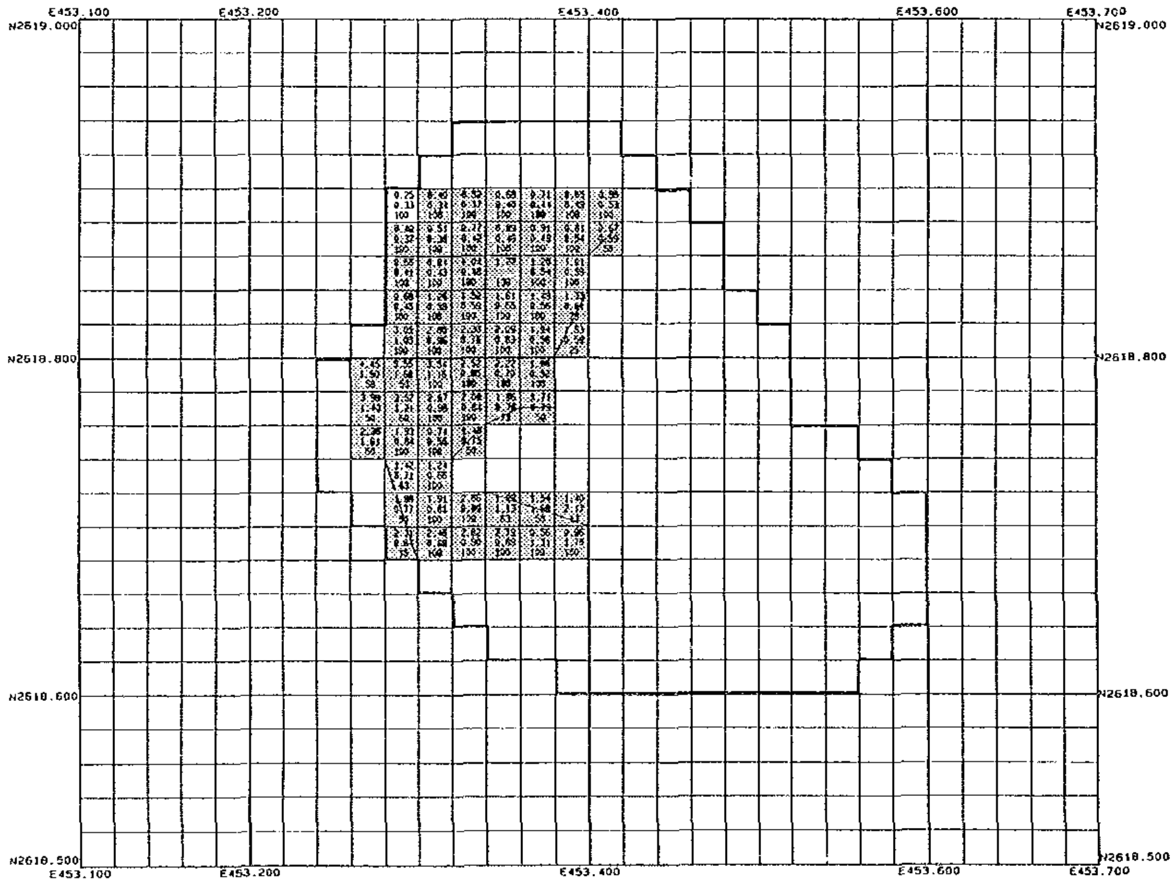
Hayl as Safil 670 mL



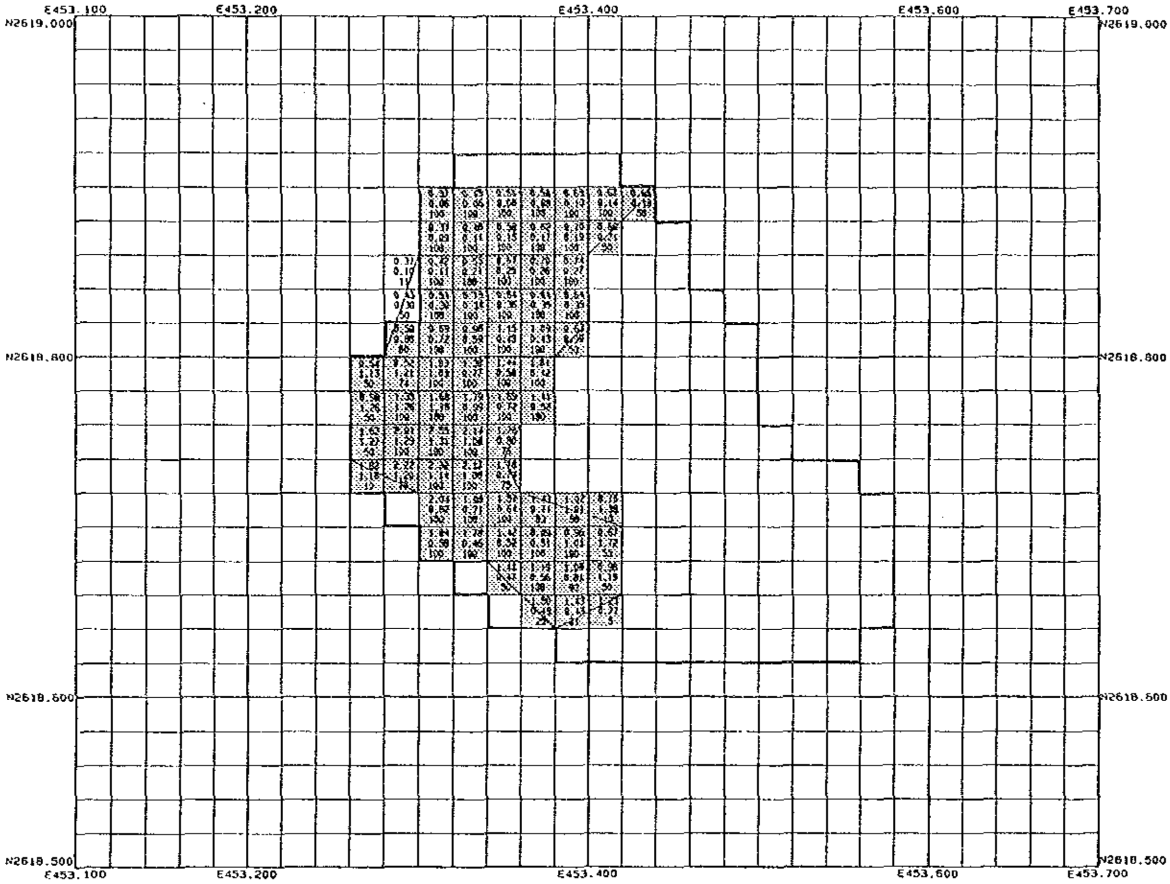
Hayl as Safil 660 mL



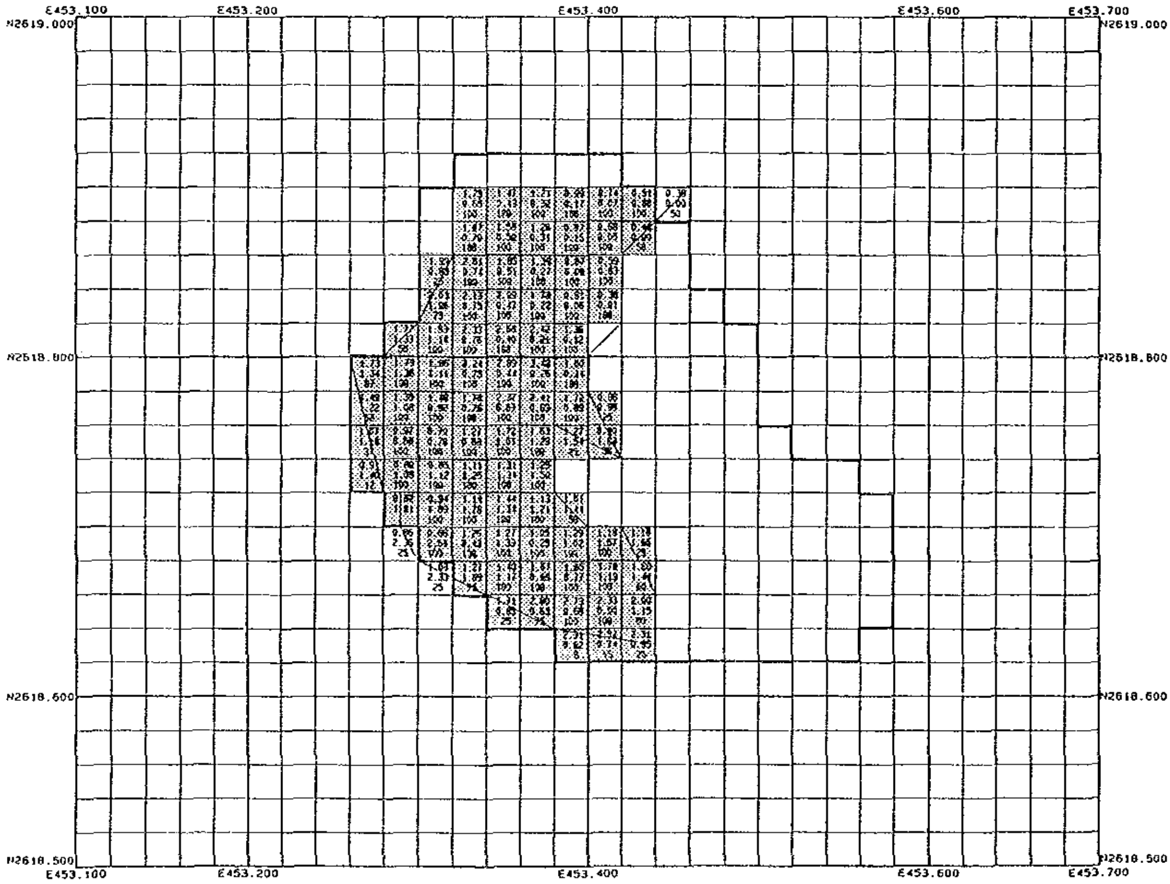
Hayl as Safil 650 mL



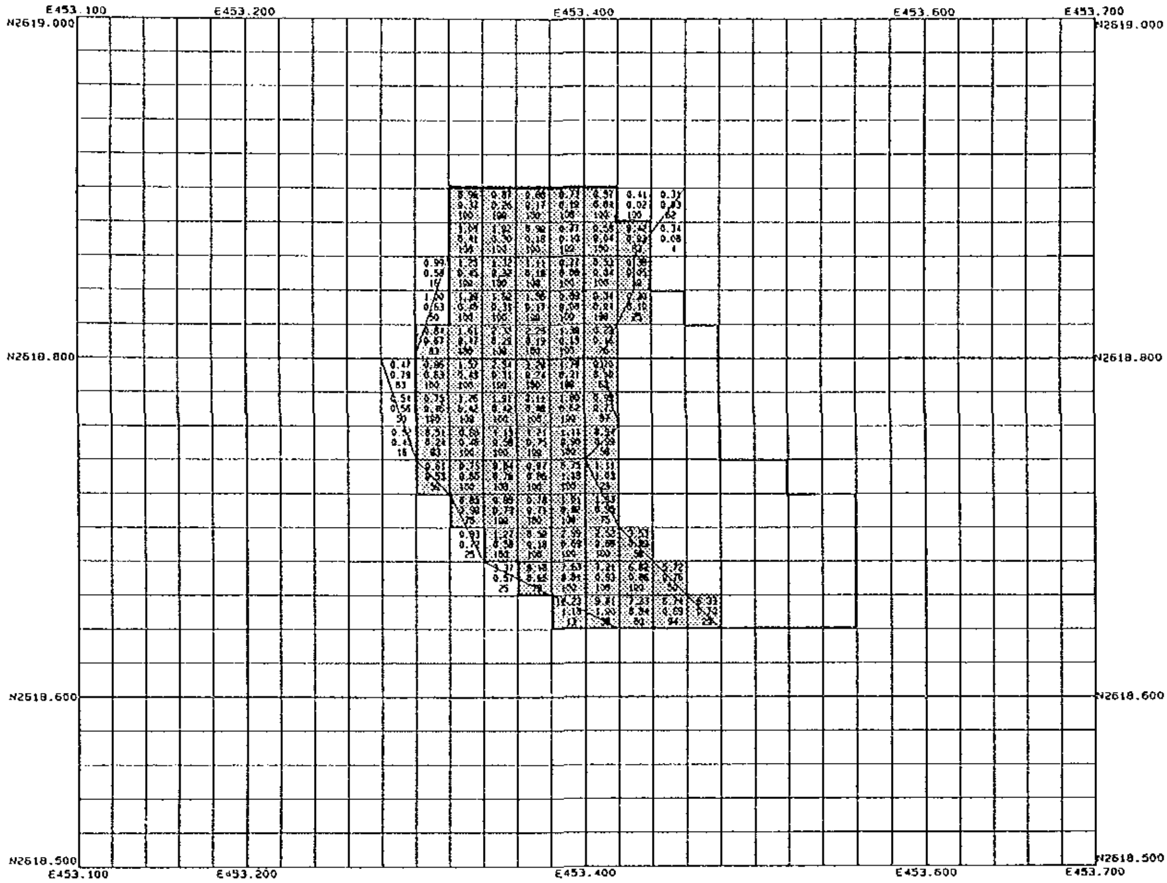
Hayl as Safil 640 mL



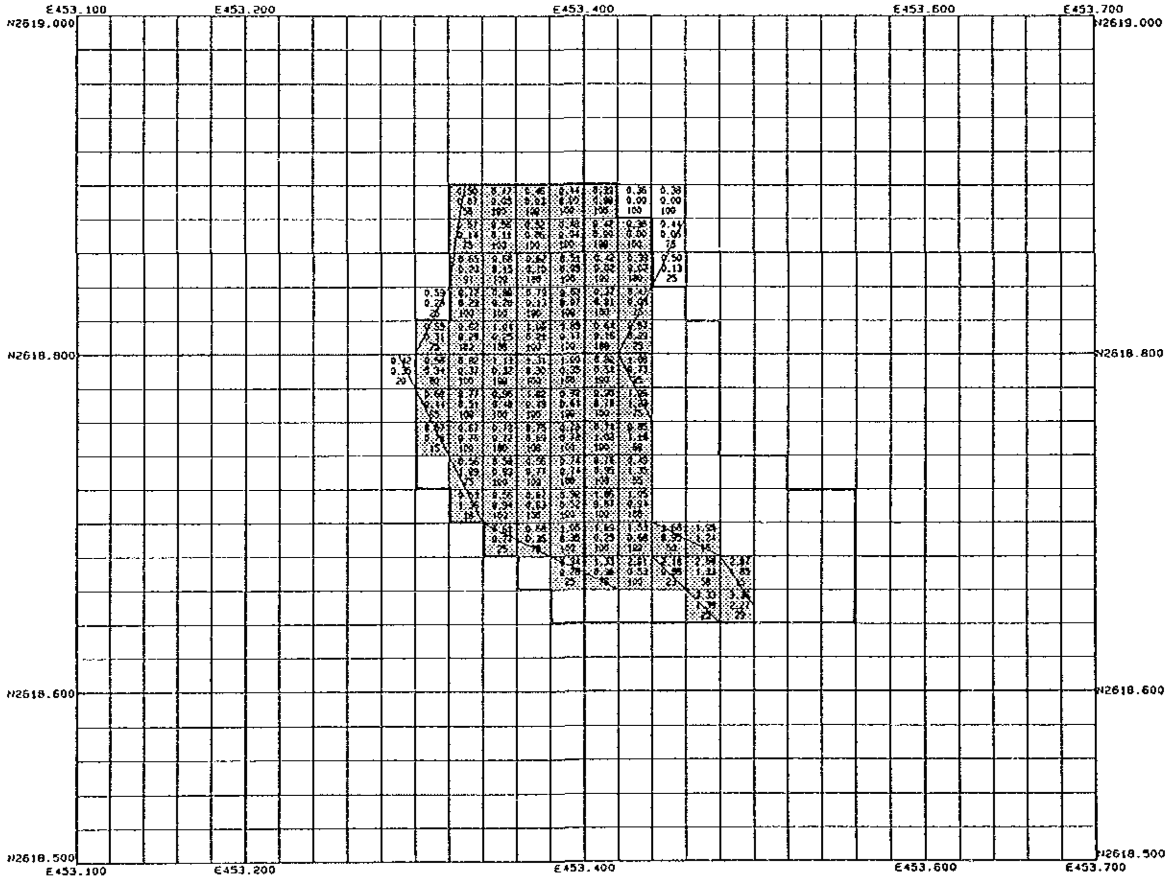
Hayl as Safil 630 mL



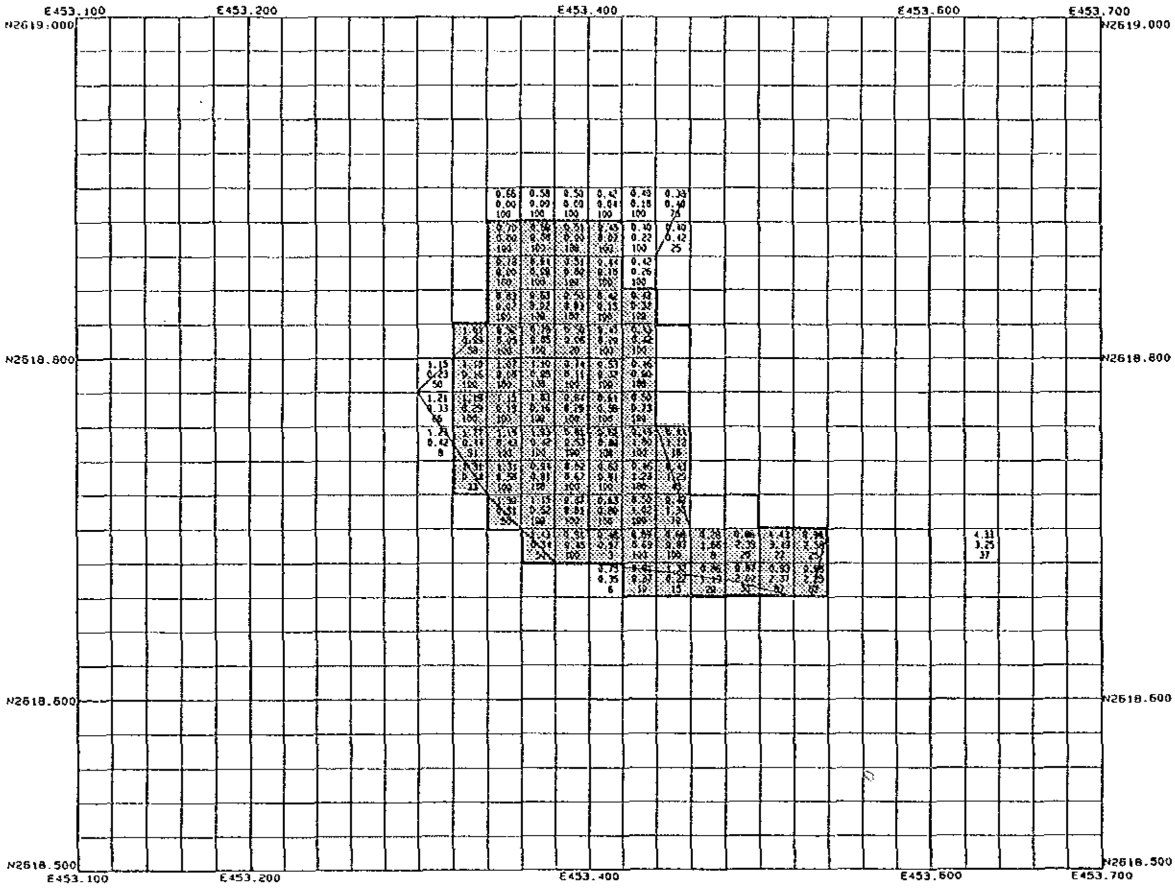
Hayl as Safil 620 mL



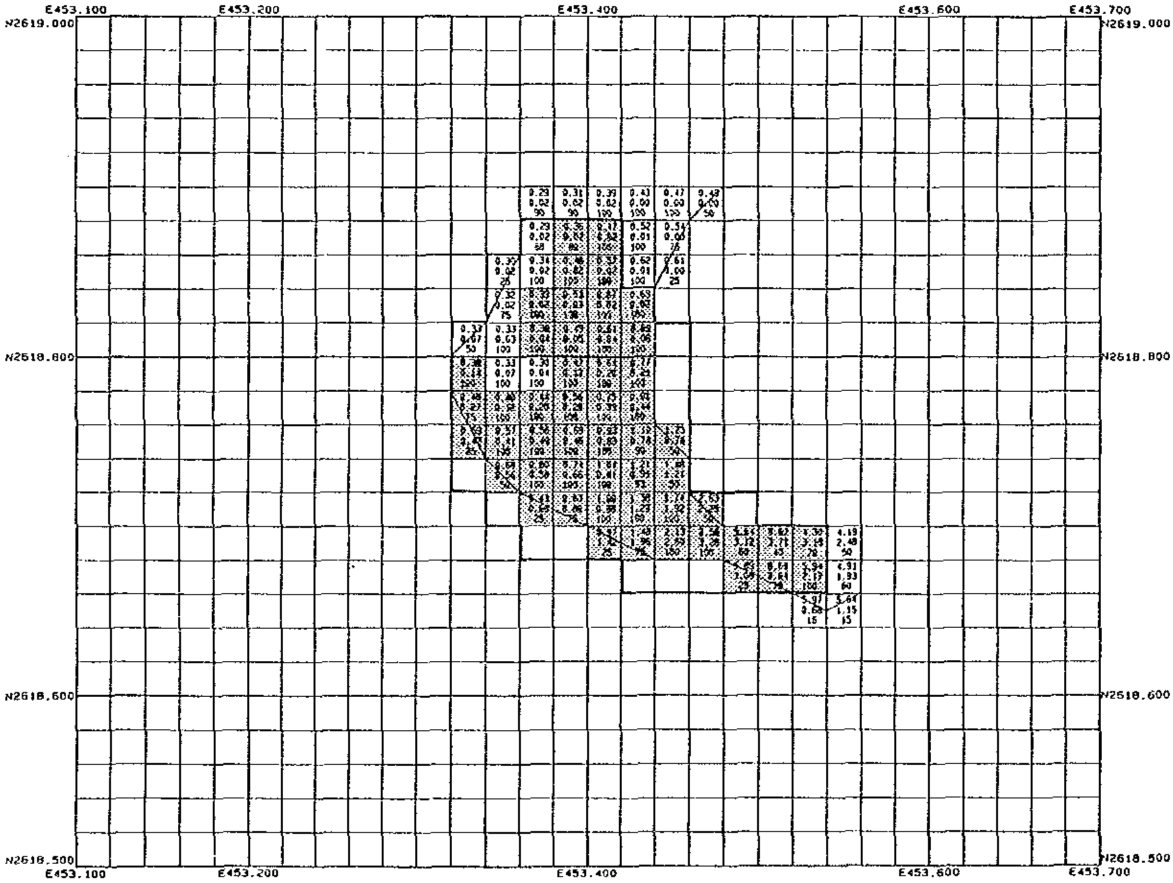
Hayl as Safil 610 mL



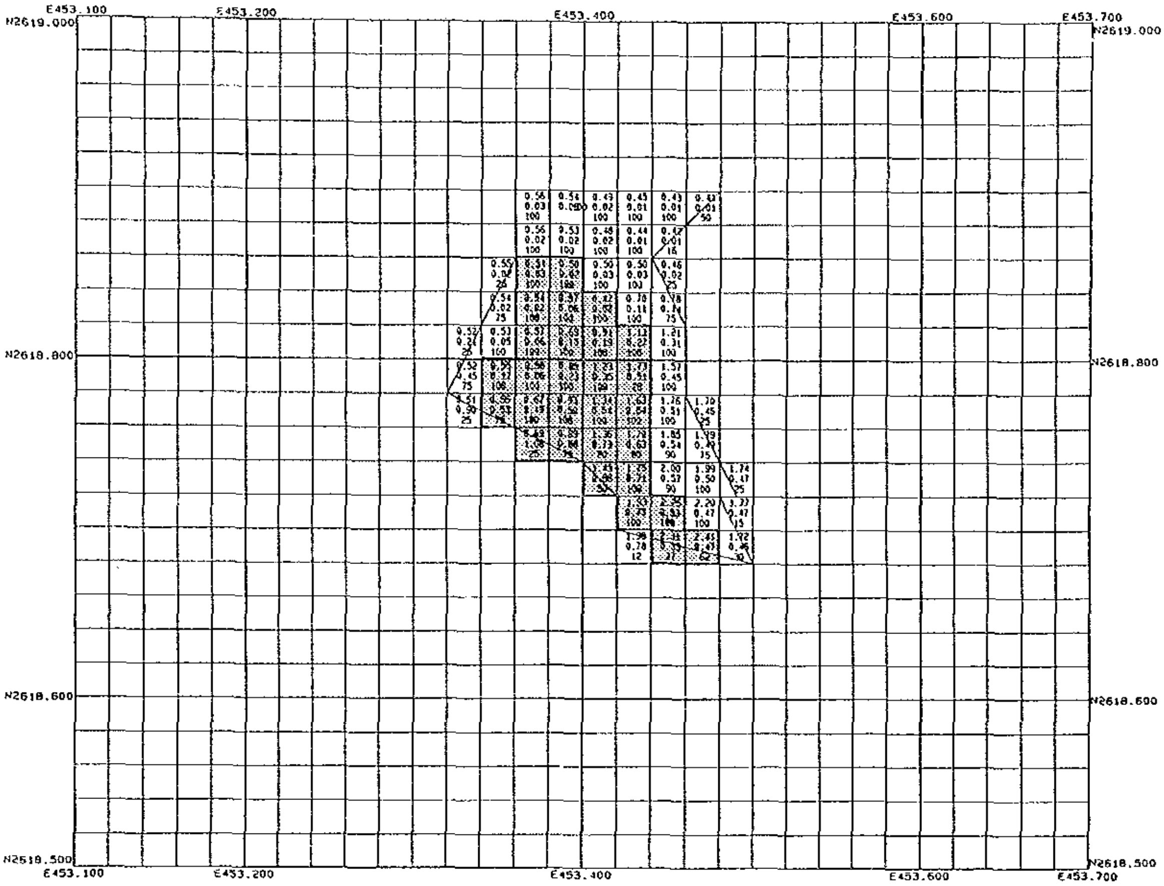
Hayl as Safil 600 mL



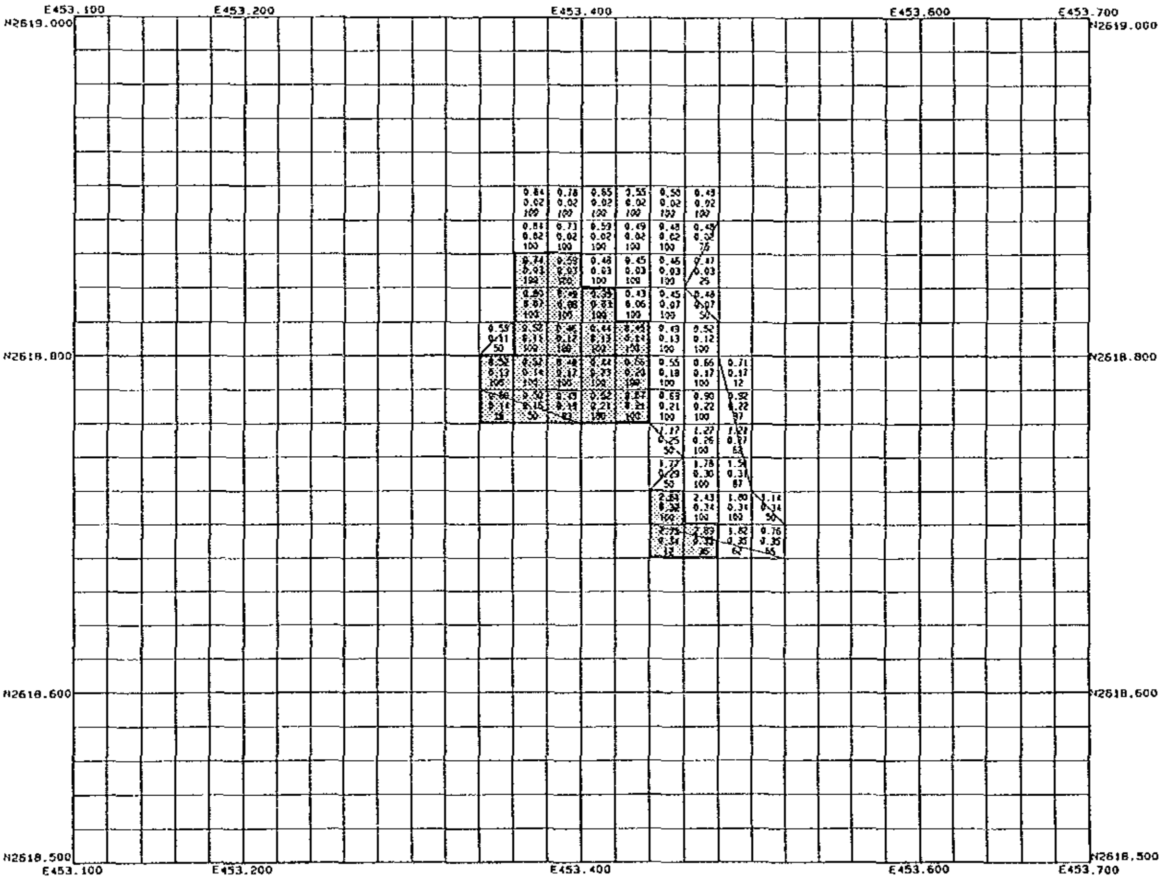
Hail as Safil 590 mL



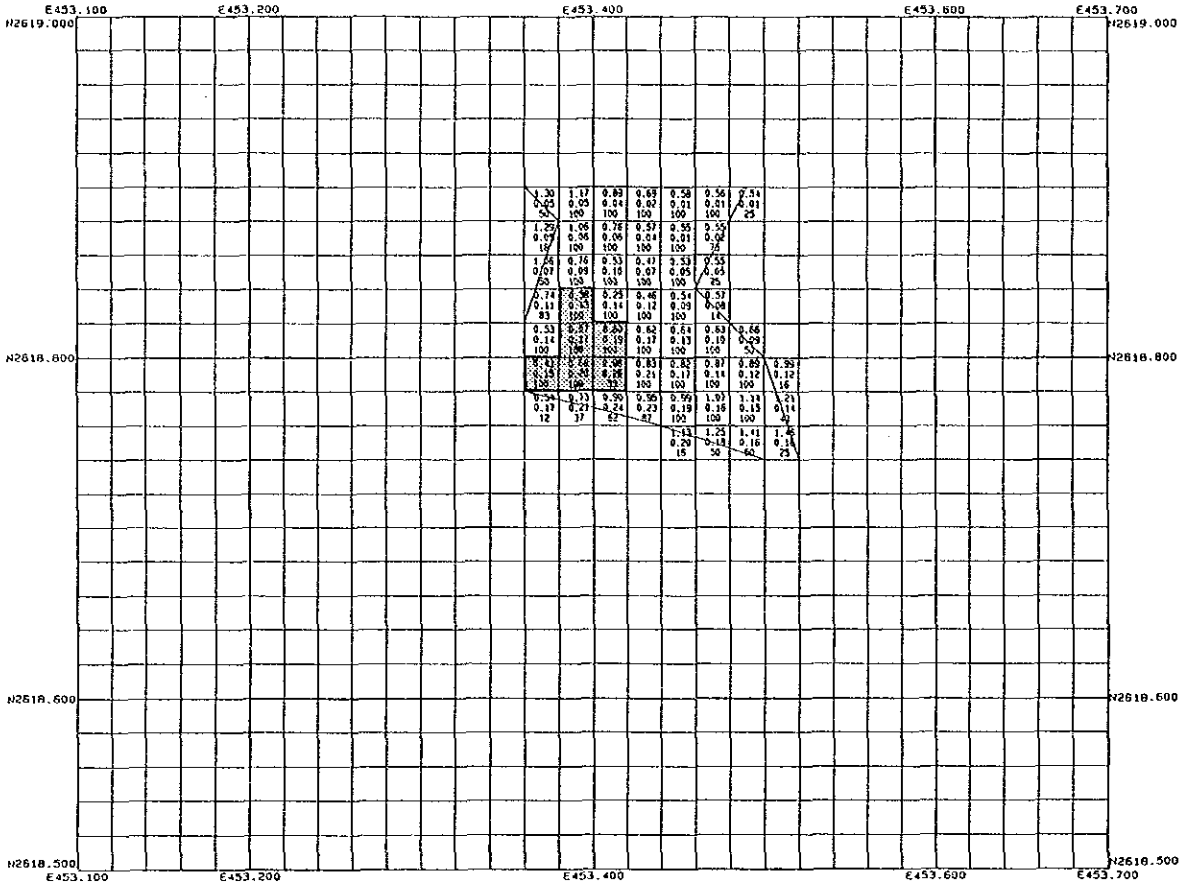
Hayl as Safil 580 mL



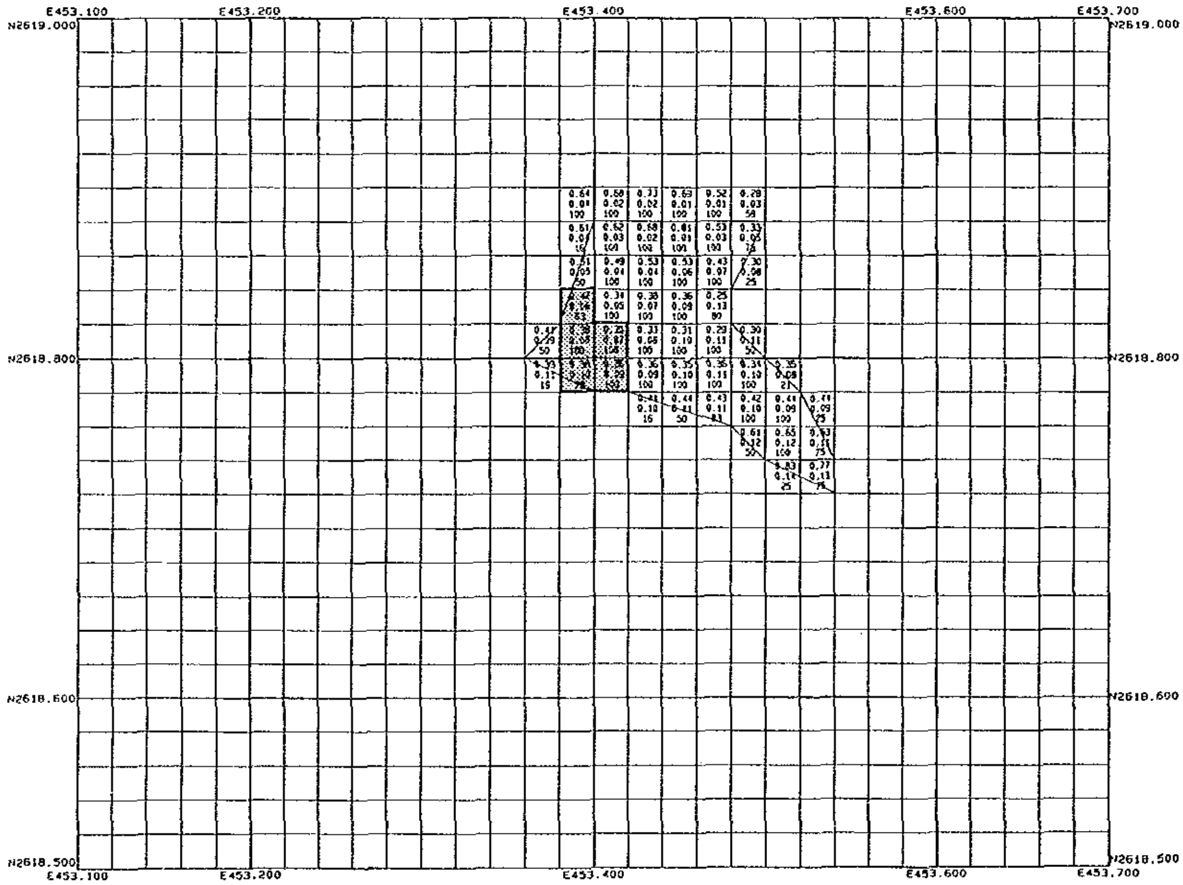
Hayl as Safil 570 mL



Hayl as Safil 560 mL



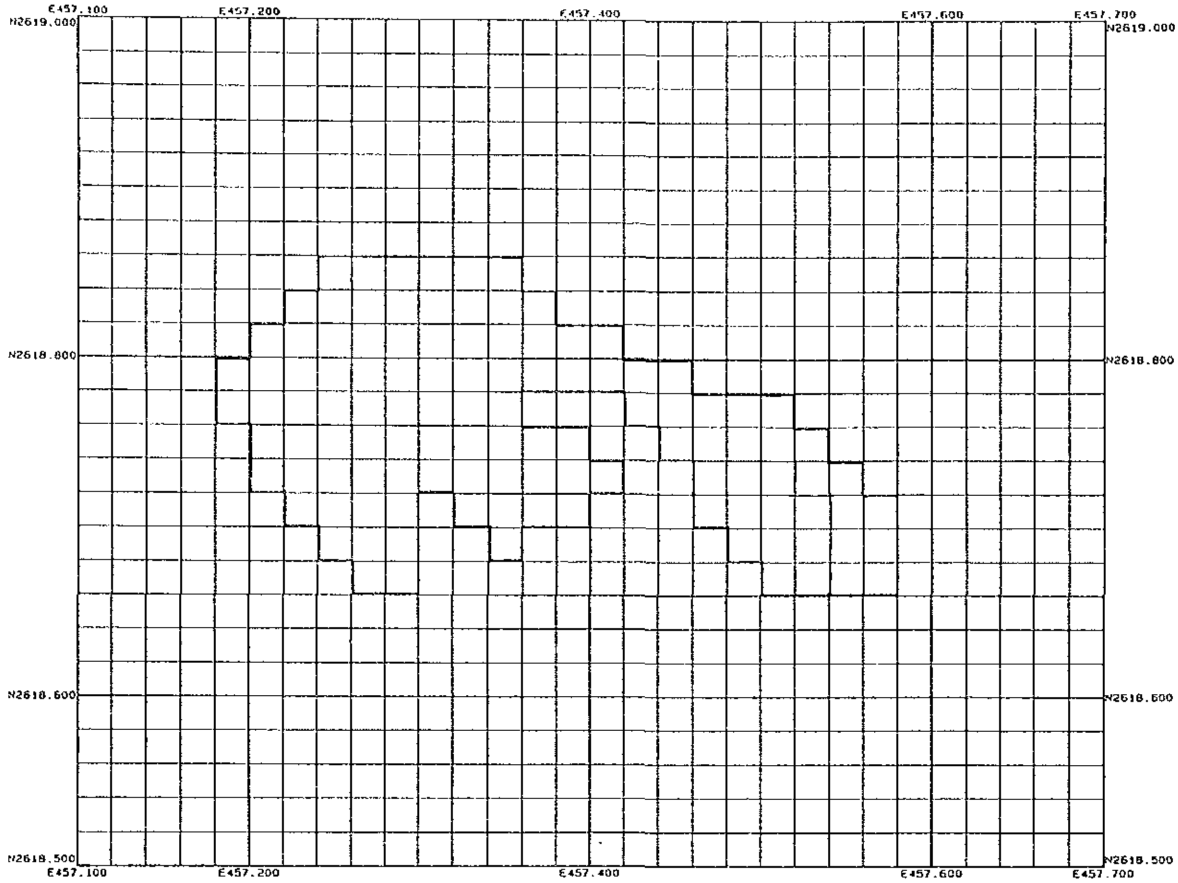
Hayl as Safil 550 mL



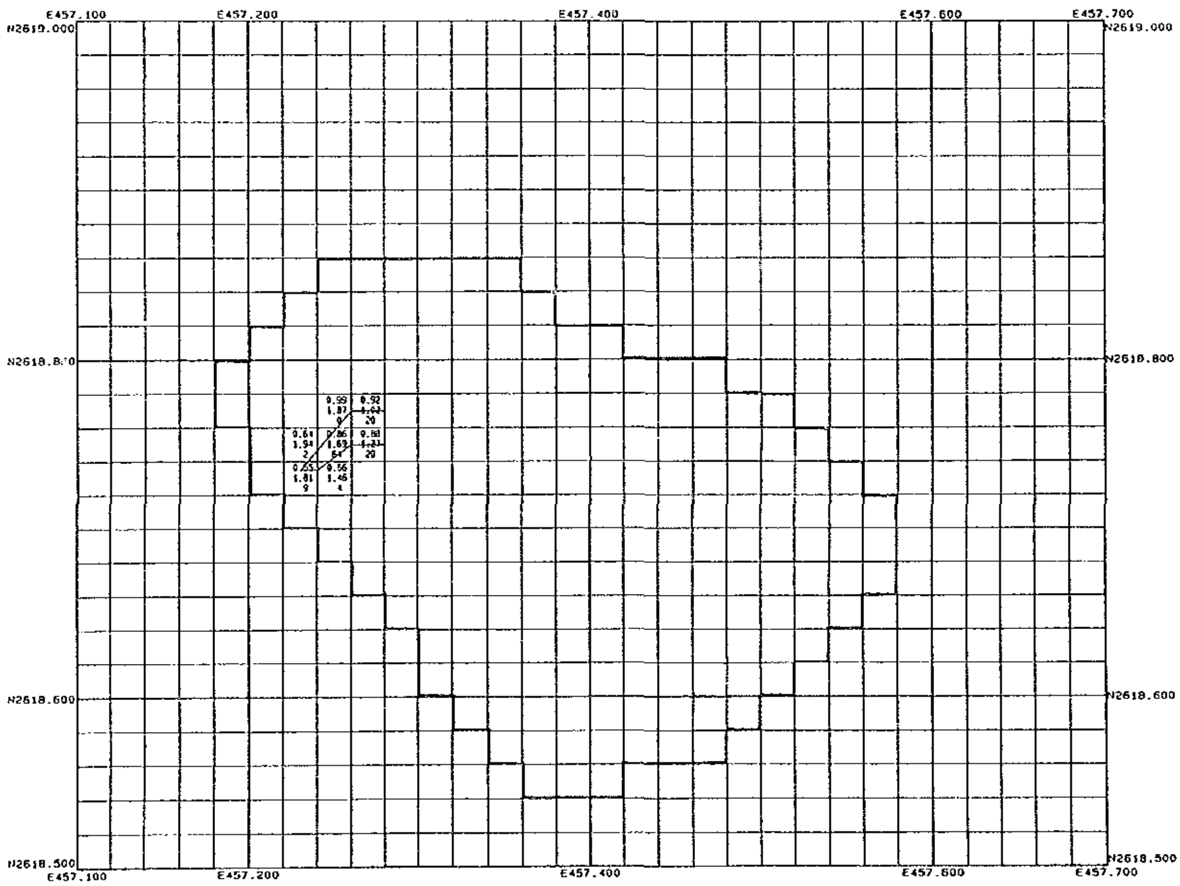
Appendix 2

Plan maps for each mining level of the Rakah deposit

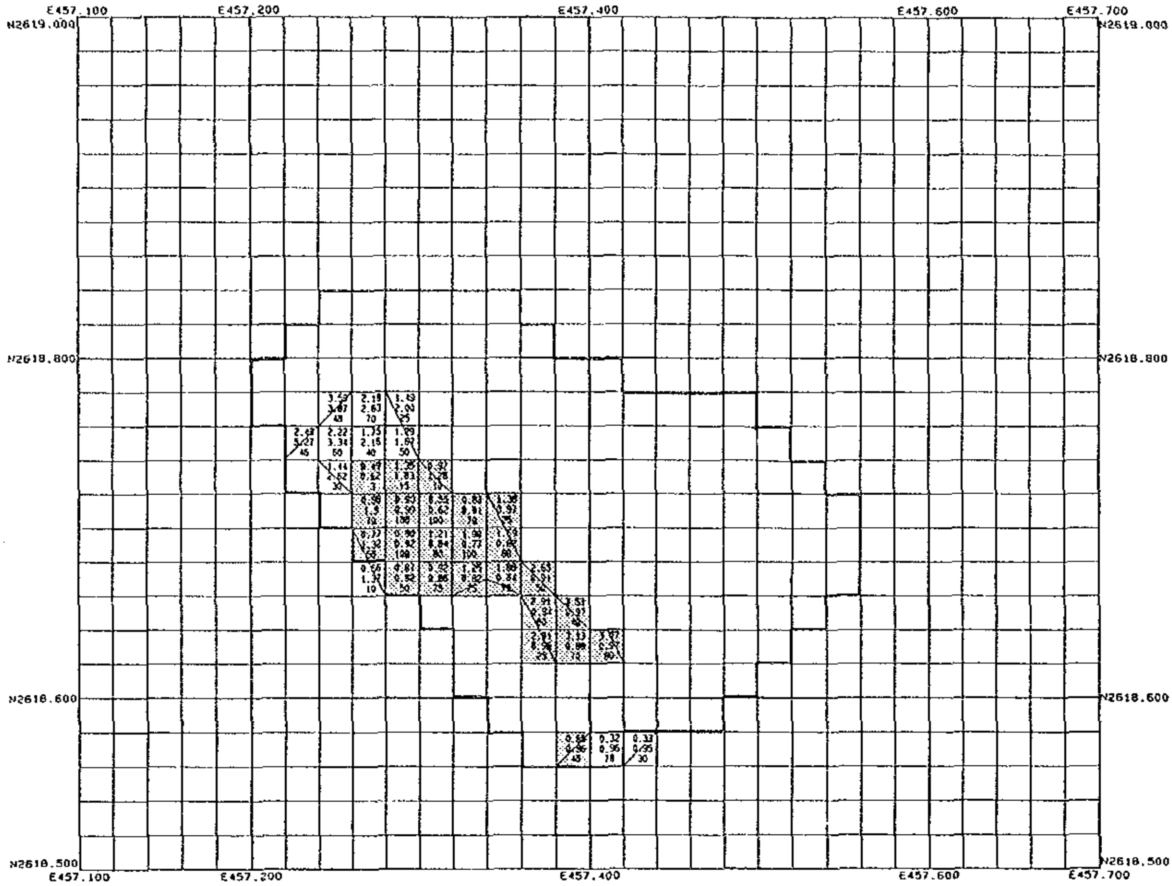
Rakah 670 mL



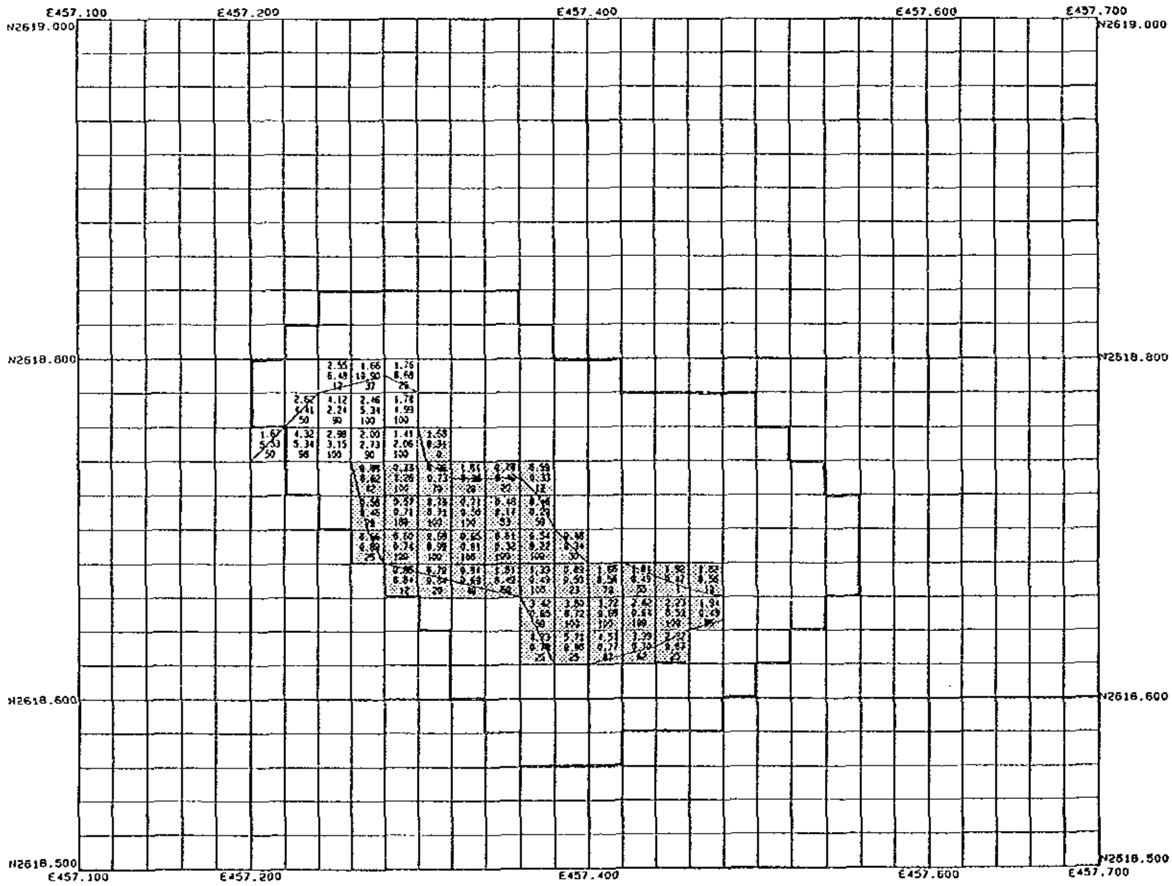
Rakah 660 mL

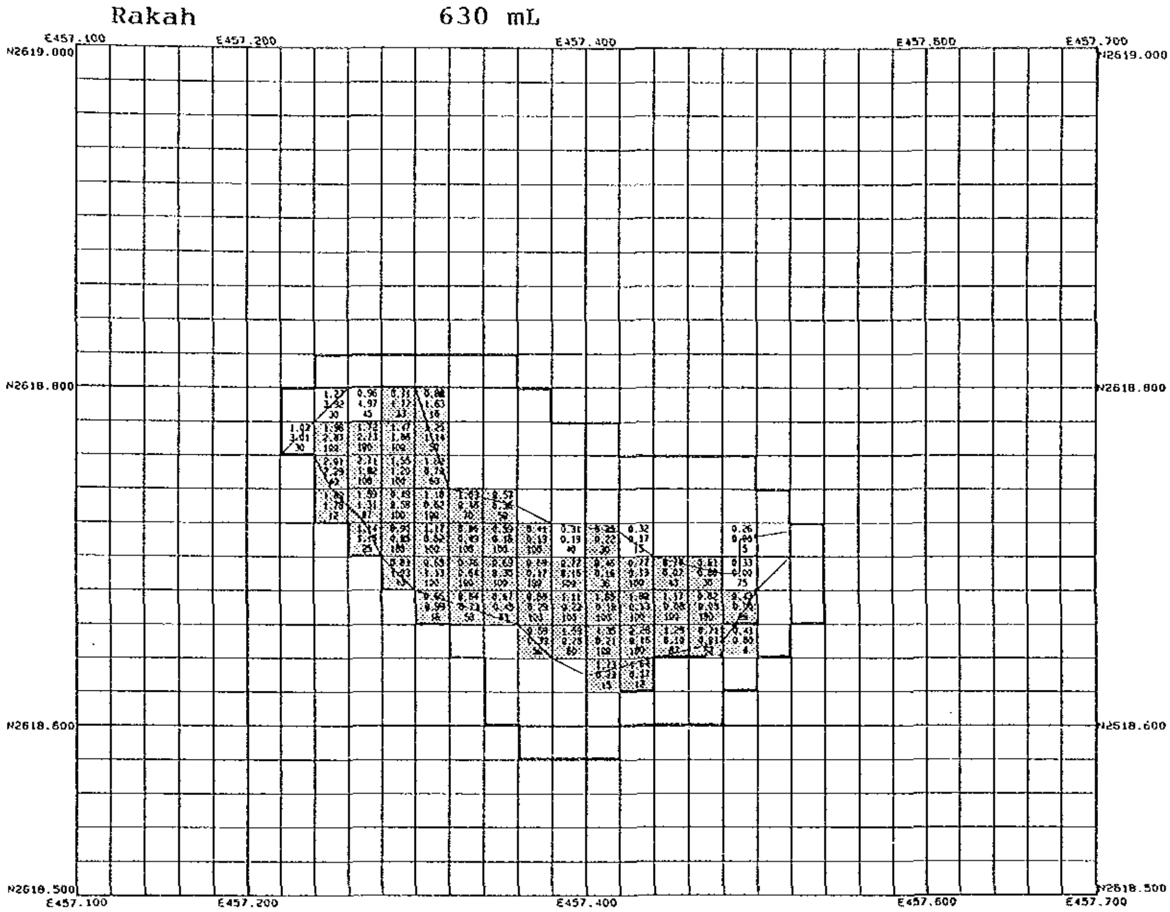


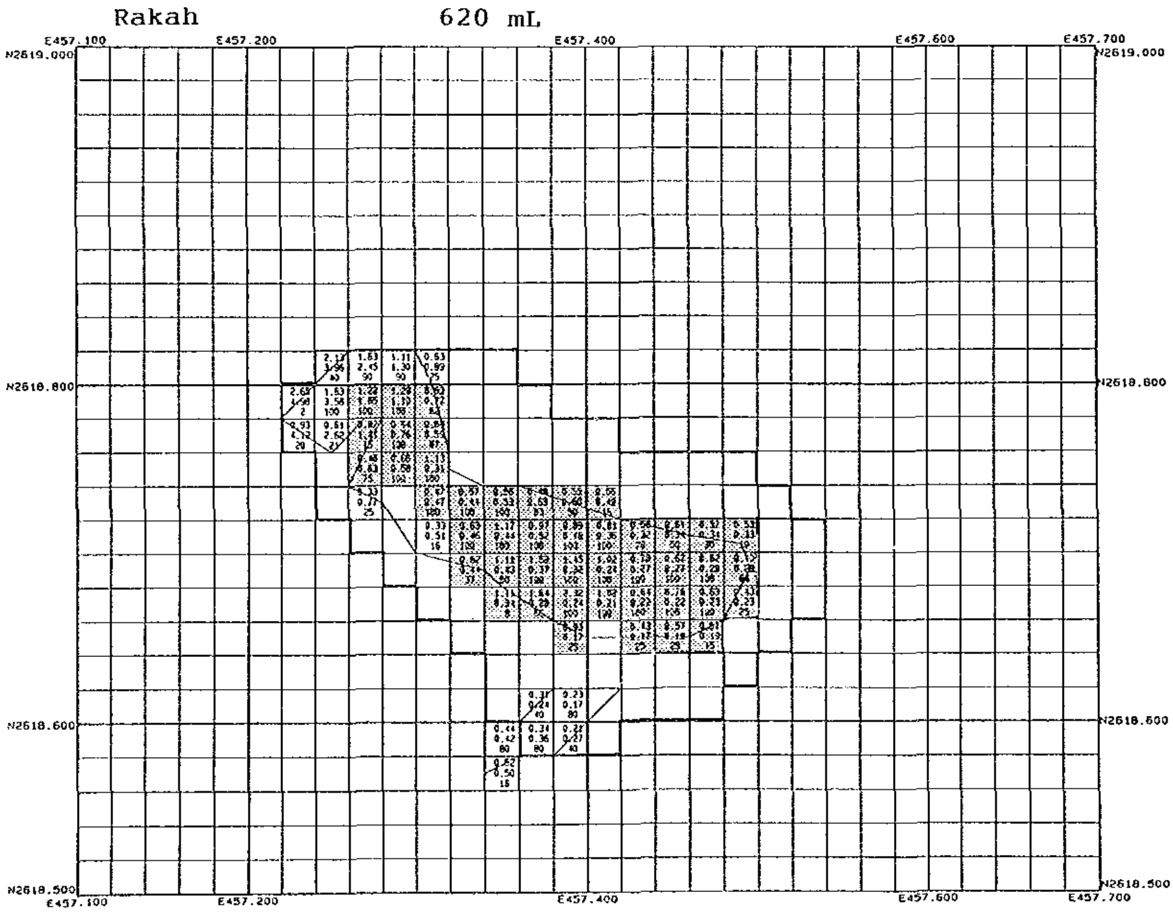
Rakah 650 mL



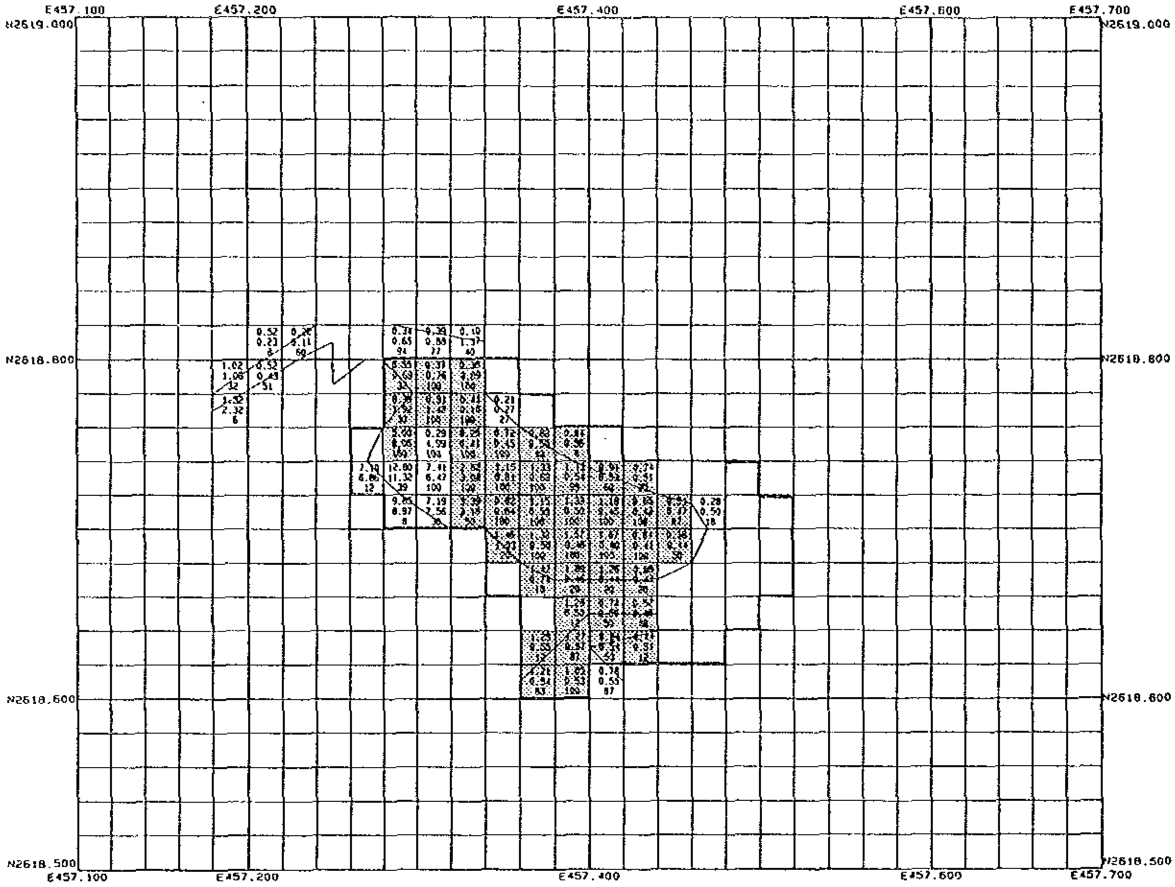
Rakah 640 mL



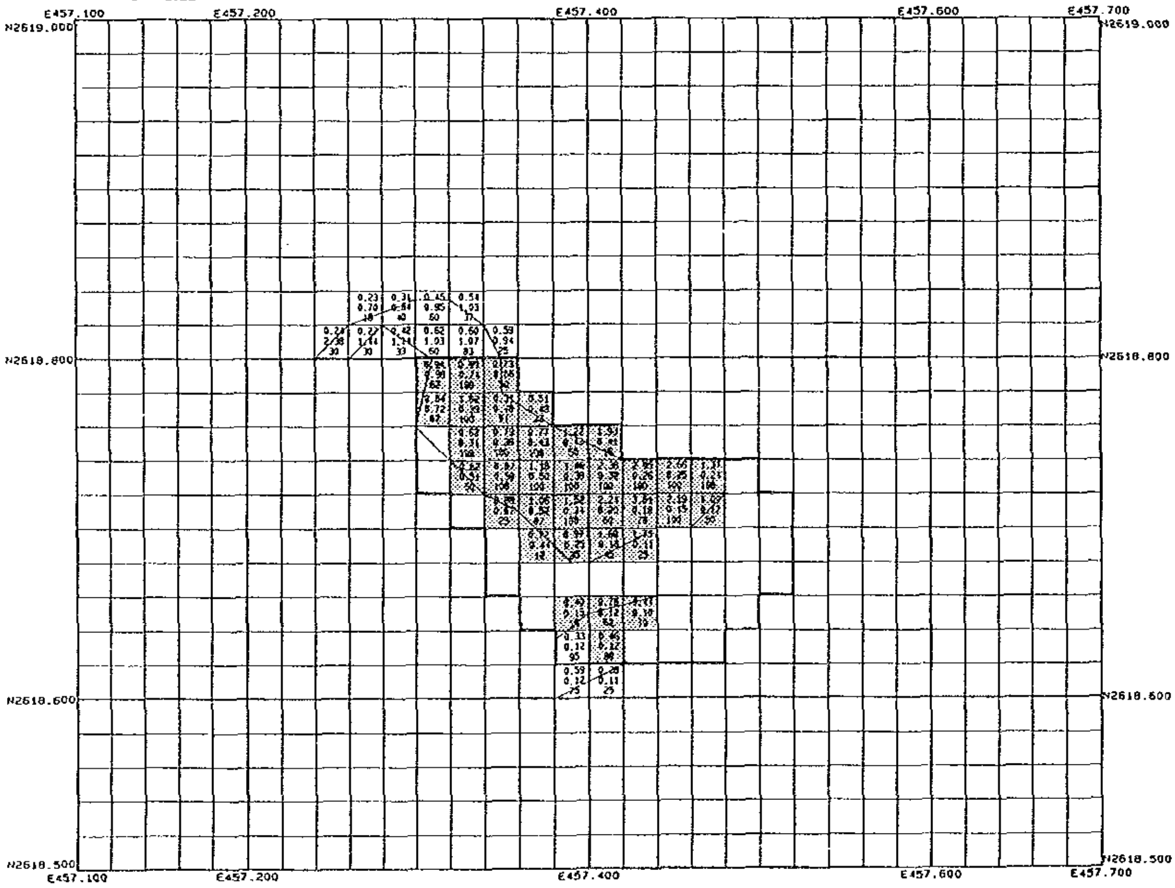

Rakah
630 mL
E457.100
E457.200
E457.400
E457.600
E457.700
N2619.000
N2618.800
N2618.600
N2618.500


Rakah
620 mL
E457.100
E457.200
E457.400
E457.600
E457.700
N2619.000
N2618.800
N2618.600
N2618.500

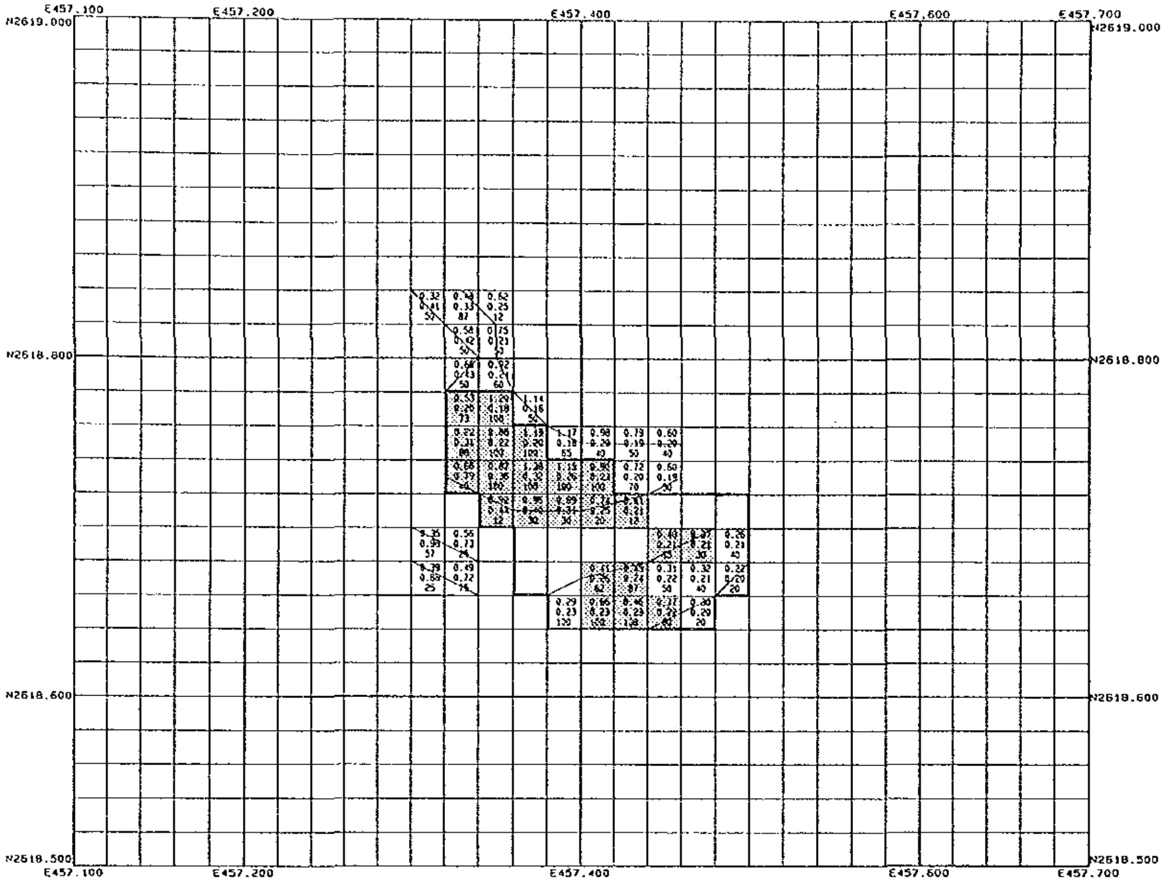
Rakah 610 mL



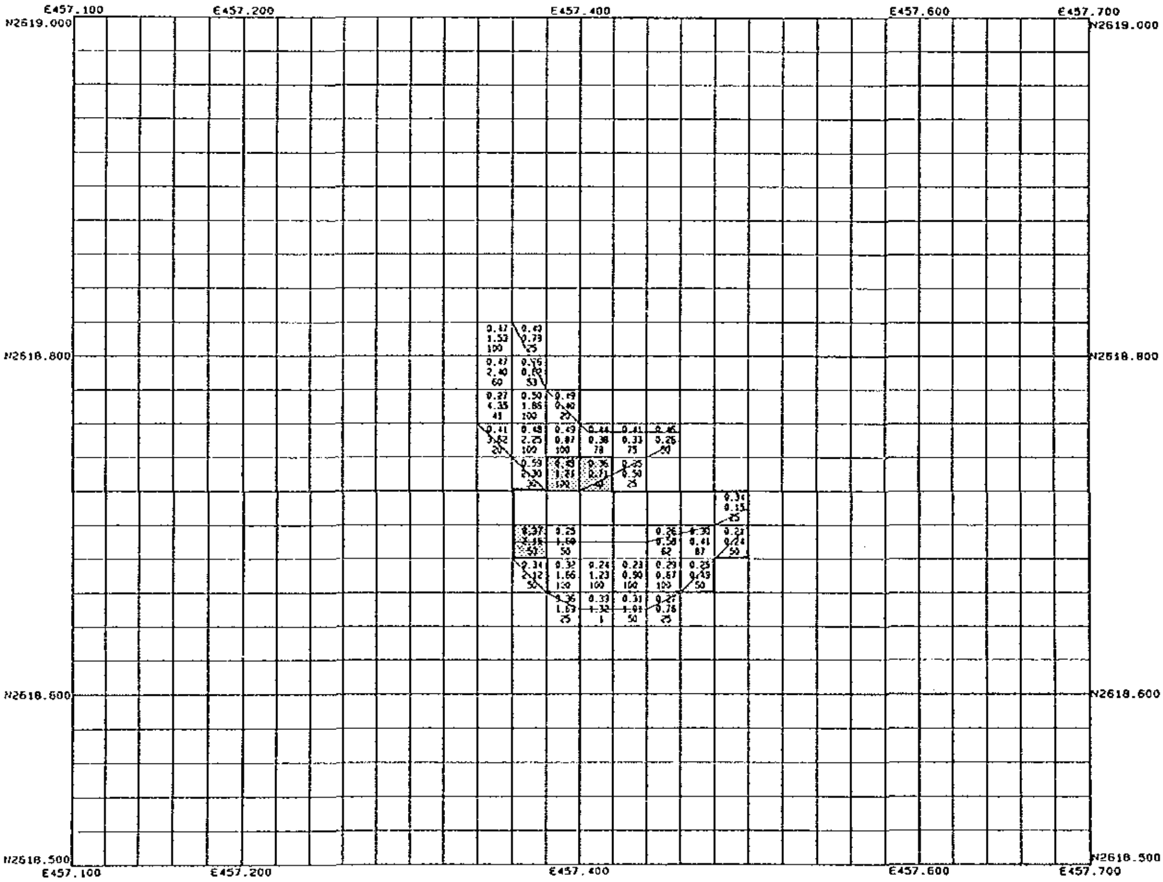
Rakah 600 mL



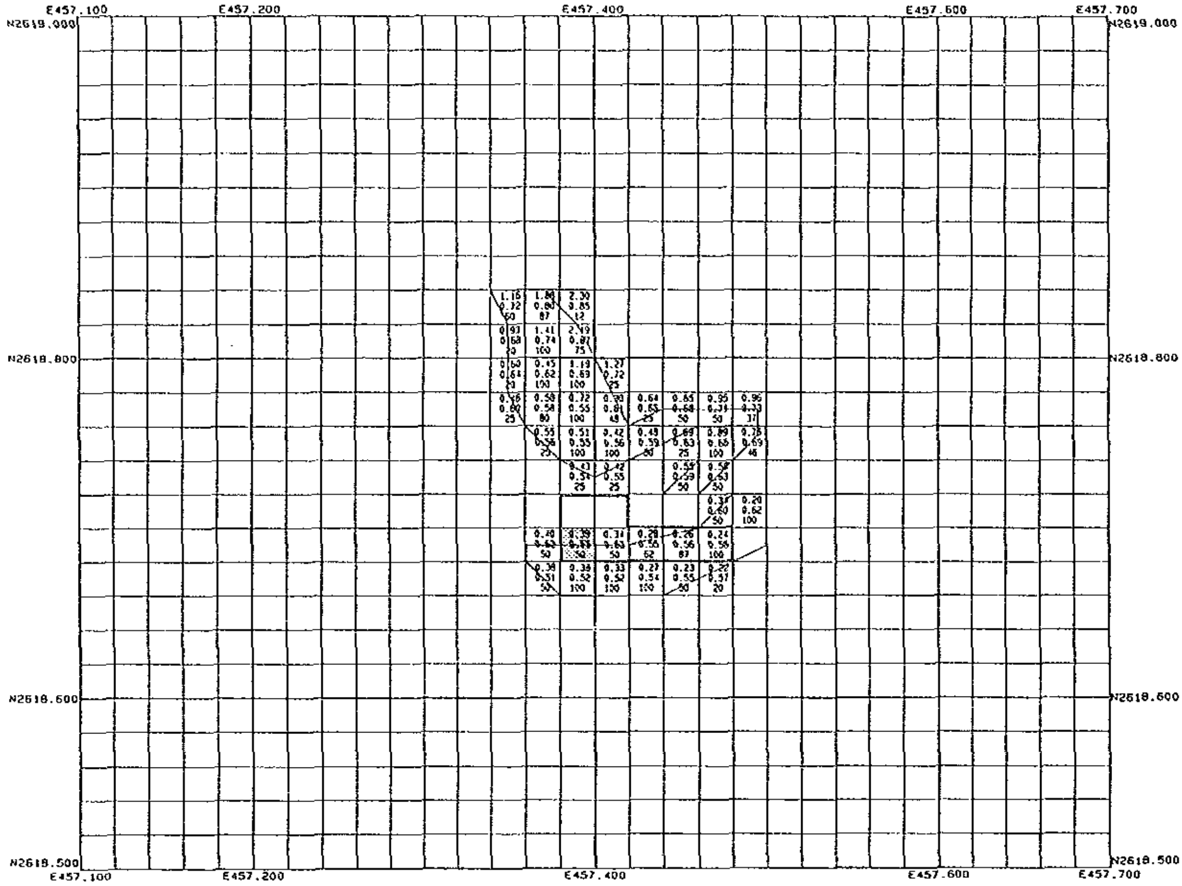
Rakah 590 mL



Rakah 580 mL



Rakah 570 mL



Rakah 560 mL