CHAPTER 3   METALLURGICAL TESTS

3-1 Survey Amounts and Method

In order to establish the efficient recovery method for gold and copper, including processing technology, metallurgical testing works were carried out for drilling core samples obtained from each deposit of Rakah, Hayl as Safil and Bishara area in Yanqul region as objective ores during two years. Studies were also conducted on matters related to design of mineral processing facilities, including pollution control facilities. The flow sheet of test program is shown in Fig.Ⅲ-3-1

3-1-1 Survey Amounts

The drilling of five holes was carried out in order to collect samples for metallurgical tests as shown in Table Ⅲ-3-1. The amount of metallurgical tests is shown in Table Ⅲ-3-2.

3-1-2 Survey method

In the first year, the following basic tests were made to improve metal recoveries, based on the optimum conditions that were obtained by tests in the past.

(1) Flotation Tests for sulphide minerals

Flotation tests were made on each core sample of four kinds of ores. The tests were carried out by laboratory scale. Referring to concentration flow, based on the optimum recovery conditions in scalping bulk differential tests which were obtained as results of straight tests conducted by organization of the counter part in 1994 and follow-up study conducted by MMAJ in 1997, chalcopyrite concentrates, pyrite concentrate and tailing are recovered.

Chemical analyses, preparation and mineral identification on polished sections were also carried out on each feed, middling and product to identify behavior of copper and gold in concentrating processes. Studies on the following items were executed.

1) The optimum condition in each processing item
   - a. Grind size
   - b. Flotation reagent
   - c. Flotation time
   - d. Number of cleaning stage

2) Proper ore processing flow sheet

3) Matters related to the design of ore processing facilities

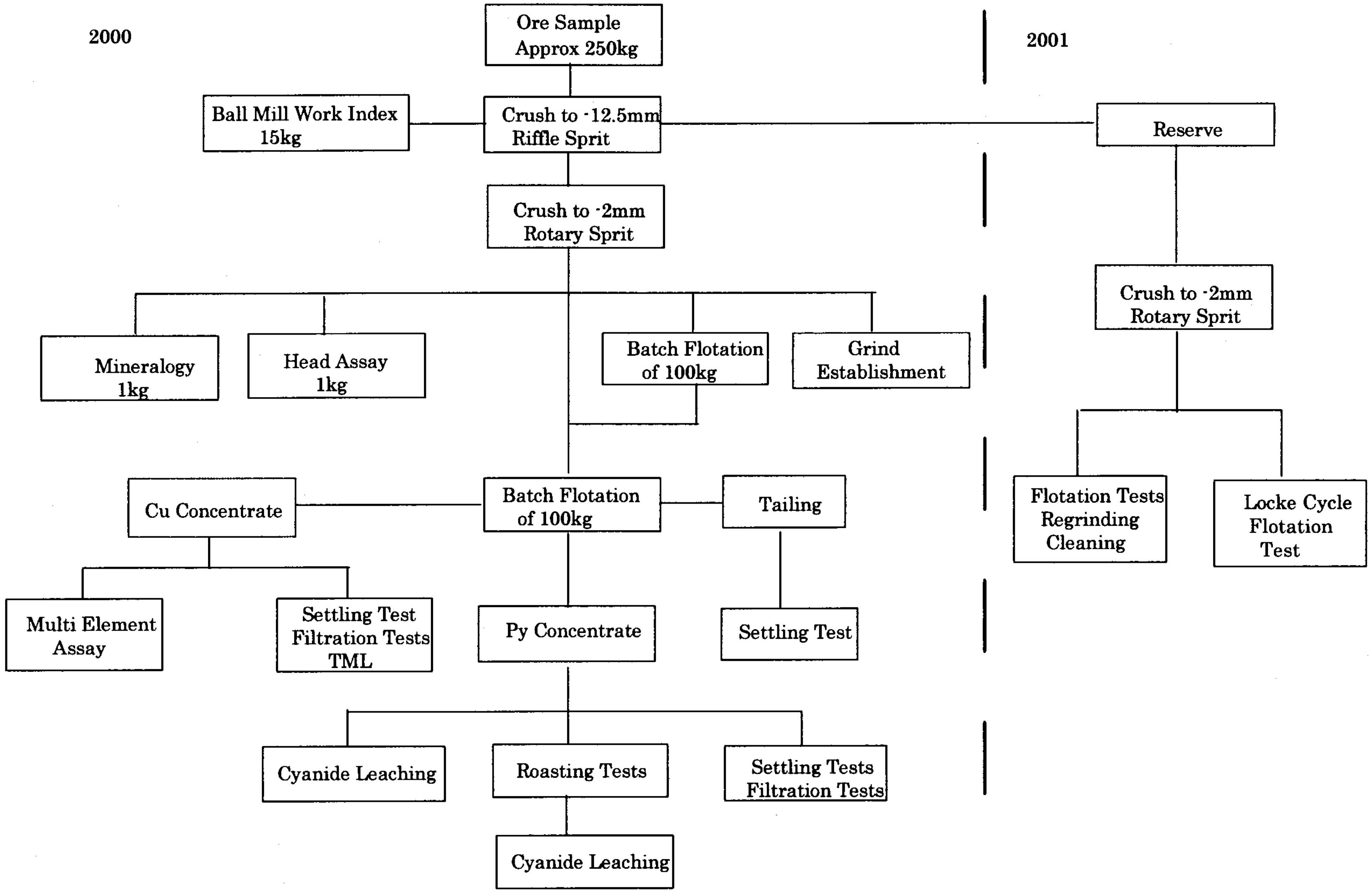

Fig.Ⅲ-3-1 Metallurgical test program flowsheet

Table Ⅲ-3-1 Content and amount of the survey

Area	No.	Length	Inclination	Direcion
Yanqul Area	MJOY-P1	125.65m	-90°	
	MJOY-P2	125.80m	-90°	
	MJOY-P3	125.65m	-75°	W
	MJOY-P4	137.55m	-90°	
	MJOY-P5	126.00m	-90°	
Grand total of length		640.65m		

Table Ⅲ-3-2 Content and amount of metallurgical test

Test Item	Details	Amount
1. Sampling	Drilling, core logging and core sampling	Number of Samples: 4 (massive ore of Rakah, stockwork ore of Rakah, stockwork ore of Hayl as Safil, breccia ore of Bishara) Total weight of samples: 1,000kg
2. Sample preparation	Weighting, crushing and blending	4 sets of ore samples, 1,000kg in total
3. Characreristics of feed ore	Chemical analysis (Cu, Au, Ag, Pb, Zn, Fe, As, Sb, S, S^{2-}, Bi, Cd, Co, Cs, Ga, In, Mo, Ni, Rb, Se, Te, Th, Tl, U, Y,)	4 samples × 25 elements
	Mineralogical test for gold	4 samples
	Measurement of work index for ball mill	4 samples
4. Flotation	To produce concentrates, middlings, and tailing by establishment of process flow through each batch test of roughing and cleaning.	23 tests (17 batch rougher flotation tests and 6 batch cleaner flotation tests)
	Chemical analysis of rugher, cleaner and tailing (Cu, Au, Ag, Fe, S,)	Rakah stockwork ore: 57 products, Hayl as Safil stockwork ore: 22 products, Rakah massive ore: 24 products: Bishara breccia ore: 27 products Total: 130 products × 5 elements
	Chemical analysis of flotation concentrates (As, Ba, Ce, Cd, Co, La, Mo, Nb, Sn, Sr, Ta, V, Y, Zr, Al_2O_3, CaO, Fe_2O_3, K_2O, MgO, MnO, Ma_2O, P_2O_5, SiO_2, TiO_2, Hg, F)	3 products × 26 elements
	Mineralogical test for flotation products	12 products
	Settling and filltering tests	each 1 set for 4 samples
5. Leaching	Sample preparation (pyrite concentrate)	Pyrite concentrate obtained from 2 samples (Rakah massive ore and Bishara breccia ore)
	Leaching tests	2 kinds (original , regrinding) × 2 samples
	Leaching tests after roasting	3 temperature conditions × 2 samples
	Chemical analysis (Au, Ag)	2 elements × 7 kinds (1 feed, 1 residue, 4 activated carbon, 1 leached liquid) × 2 times

(2) Leaching tests

By conducting agitation cyanide leaching tests in laboratory scale and agitation cyanide leaching tests combined with roasting referring to the pyrite concentrate produced in above said flotation tests, the following studies were made.

1) The optimum condition for gold leaching
   a. Grind size
   b. Cyanide concentration
   c. Additional reagents
   d. Oxygen addition
2) Proper ore processing flow sheet
3) Matters related to the design of ore processing facility

(3) Improving Tests for Copper Concentrate Grade

Since copper cleaning conditions could not have been well established in the first year, a series of cleaning test was performed in batch style, focusing the optimization of regrinding size.

(4) Locked Cycle Test

A locked cycle test was done in the second year. In the test, the rougher and scavenger concentrates were mixed and ground to be minus 20 micron using composite ore sample as feed and cleaning was executed three times to obtain final copper concentrate. In this case, the first cleaner tailing was joined with scavenger tailing as final tailing, however, the second and third cleaner tailings and water were recycled to next stage, repeating totally six stages. Test flow diagram of the locked cycle test is presented in Fig.Ⅲ-3-2.

3-2 Collecting Samples

Core samples were taken by core drilling from the three deposits of Bishara, Rakah and Hayl as Safil. Drilling locations were selected based on the existing data as shown in Fig.Ⅱ-2-2.

Samples used for metallurgical testing were summarized in Table Ⅲ-3-3.

Table Ⅲ-3-3 Samples used for metallurgical tests

Body Name	Type of Ore	Sampling Source	
		Drill Hole No.	Sampling Depth
Rakah Body	Massive sulphide ore	MJOY-P1	0m to -7.30m
Rakah Body	Stockwork ore	MJOY-P2	-13.40m to -31.35m
do.	do.	do.	-32.90m to -43.60m
do.	do.	do.	-63.15m to -122.10m
Hayl as Safil	Stockwork ore	MJOY-P3	-49.65m to -118.25m
Bishara Body	Breccia ore	MJOY-P5	-45.70m to -68.65m
do.	do.	do.	-69.75m to -93.60m

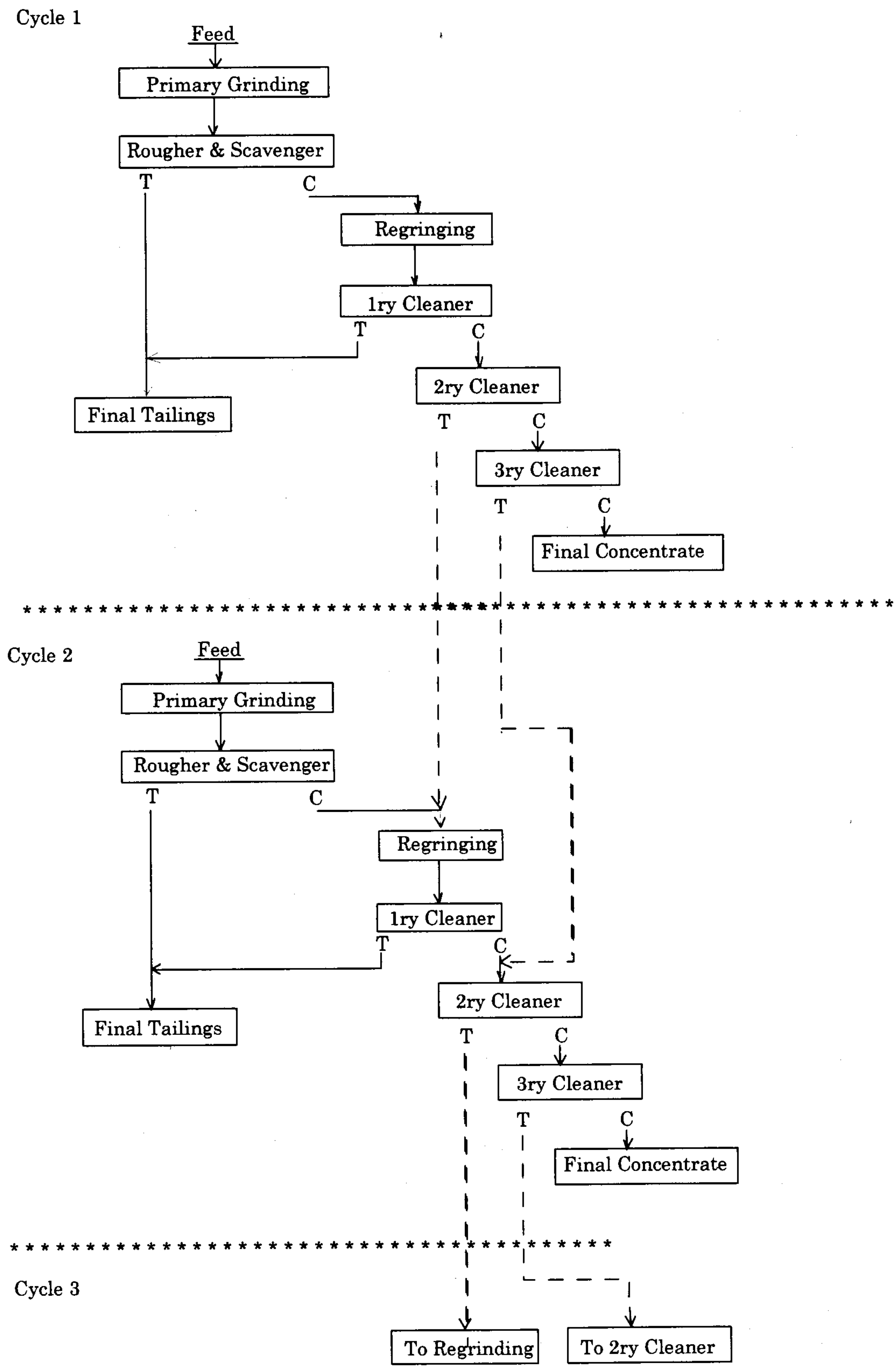

Fig.Ⅲ-3-2 Locked cycle test diagram

3-3 Survey Results

3-3-1 Head assays

One charge (1kg) of each sample was used for head assay for a range of elements. The complete head assay results are contained in Appendix 1A, while assays for the major elements are summarized in Table Ⅲ-3-4.

Table Ⅲ-3-4 Head assays of major elements

Body name and Type of ore	Rakah Body Stockwork	Hayl as Safil B. Stockwork	Rakah Body Massive	Bishara B. Breccia
Cu (%)	1.15	0.915	1.82	1.45
Au (g/t)	0.45	0.16	3.78	1.06
S in total (%)	3.35	7.40	39.0	28.3
S in sulphide (%)	3.30	7.40	38.8	28.1

Both stockwork samples were found to be low in gold content, while the Rakah Massive Sulphide and Bishara Breccia samples were of higher grade. The latter samples were also higher in copper grade than the stock samples.

The similarity between total S and sulphide S indicates that the sulphur minerals contained in all samples were primary in nature.

3-3-2 Gold mineralogy

Samples of each ore were examined mineralogically to determine the location of gold and to provide general information regarding the mineralogy of the samples. The samples were ground to 80% - 75μm prior to examination, so that the samples represented material typical of flotation feed.

A summary of the gold distribution for each ore sample is shown in Table Ⅲ-3-5.

Table Ⅲ-3-5 Occurrence of gold minerals

Body name and Type of ore	Rakah Body Stockwork	Hayl as Safil B. Stockwork	Rakah Body Massive	Bishara B. Breccia
Liberated native gold	15	20	3	4
Locked in sulphides	46	37	57	60
Locked in silicates	12	17	4	4
Undifferentiated -20μm	27	27	36	33

The predominant sulphide minerals in all samples were pyrite and chalcopyrite. The Rakah Massive Sulphide sample also contained a significant quantity of chalcocite which generally occurred as individual grains whereas the chalcopyrite grains frequently occurred as intergrowths with pyrite. There was also an unidentified mineral intergrown with pyrite in the Rakah Massive Sulphide sample which could be another copper mineral.

3-3-3 Ball mill work index

The ball mill work index for each ore sample was determined using the standard Bond procedures with closing screen of 106μm to achieve a product of approximately 80% - 75μm. The detailed results and procedure, plus some reference data, are contained in Appendix 1B. Table Ⅲ-3-6 contains a summary of the grinding work index test results.

Table Ⅲ-3-6 Ball mill work index

	Rakah Body Stockwork	Hayl as Safil B. Stockwork	Rakah Body Massive	Bishara Body Breccia
Feed 80% Passing, μm	2,415	2,167	2,214	2,405
Product 80% Passing, μm	82	84	85	80
Work Index, kWh/tonne	19.1	16.2	14.2	15.5

The Rakah Stockwork sample recorded the highest grinding work index of 19kWh/tonne, while the remaining samples recorded values of 14 to 16kWh/tonne.

3-3-4 Flotation tests

(1) Grind establishment

Several 1kg charges of each ore type were ground at 60% solids in a batch rod mill for different periods and the products screened to determine their size distribution. The 80% passing values for each products were then graphed against mill revolutions to produce grind establishment curves, from which could be interpolated the mill revolutions required to achieve the target P_{80} value.

A P_{80} of 70μm was selected as the primary grind for the batch flotation test, although P_{80} 90μm was also used in some tests. Mill revolutions required for each sample to achieve these grinds are summarized in Table Ⅲ-3-7.

Table Ⅲ-3-7 Grind requirements

	Rakah Body Stockwork	Hayl as Safil B. Stockwork	Rakah Body Massive	Bishara Body Breccia
Mill revolutions for P80 70μm	1,240	1,270	980	1,200
Mill revolutions for P80 90μm	1,060	980	760	1,010

(2) Batch flotation tests

It was expected that the stockwork ore would compromise about 70% of the mill feed, of which the Rakah Stockwork would be predominant. Hence the approach to the testwork was to focus on this sample initially. For the other three samples the aim was to fit these to the Rakah Stockwork flowsheet with the minimum of variation, in order to simplify subsequent preliminary process design by an engineering organization.

A previous program of testwork on samples from the same resource had been carried out (Minproc, 1994). The results of this program were referred to in order to establish initial flotation test conditions. The collectors used in the previous work were SIPX and Minerec 2030, so use of these same collectors was continued in the current program.

(3) Rakah stockwork

(3)-1 Rougher tests

The conditions, results and grade versus recovery curves are shown in Appendix 1C.

Tests FL01-3 were the initial tests carried out on Rakah Stockwork ore.

Test FL01 used the best conditions from the 1994 testwork, with a pyrite flotation atage added. Subsequently head assays became available which showed that there was little pyrite in the sample. Tests FL02 and FL03 were identical except the pH was in test FL02 and 11 in Test FL03.

The best of the tests was Test FL02 in which 95.6% of the copper was recovered in roughers 1-3 at a grade of 11.0% Cu. Flotation time was only three minutes, which would translate of 6-8 minutes in a commercial operation. The scavenger concentrate contained an additional 1.7% of total copper but at a grade of only 0.5% Cu. The pyrite stage proved to be of no benefit, carrying only 1% of the sulphur, 96% of which had already floated.

This sample was significantly different to the Rakah Stockwork sample tested in 1994, which had a sulphur content of 14.4% compared to 3.35% in the current sample.

The result obtained in test FL02 was good enough to warrant examining ways of reducing the flowsheet cost. Hence the following changes were made:

- In Test FL04 the grind was coarsened to a P_{80} of 90 microns
- In Test FL05 the collector addition was halved
- In Test FL06 potassium amyl xanthate (a strong and somewhat unselective collector), was replaced by sodium isopropyl xanthate. Also the addition rate in the scavenger float was halved.

The pH in the above tests was 9.5.

The coaser grind gave an inferior result with a rougher recovery of only 88% at a grade of 11.4% Cu. The test with reduced collector gave good rougher concentrate grades but copper recovery was low (86%) at a grade of 16.9% Cu. The best result was obtained in Test FL06 in rougher concentrate assayed 24.2% with a recovery of 54%.

The next test on Rakah Stockwork was test FL16 in which the SIPX addition was doubled and the collector M2030 was dropped altogether. The results obtained in the previous tests indicated that the copper mineralogy is straight forward, with little pyrite, so a simple reagent regime should be able to achieve good results. It would also be cheaper to install and operate. A second scavenger was added, as an information gathering stage, in the event that copper recovery was low in the preceding stages.

In Test FL16 while the overall copper recovery was maintained the concentrate grades were lower. The recovery into rougher concentrates 1-2 was similar but the grade was 3.1% higher with the mixed collector in Test FL16. It was decided to retain the mixed collector in future test.

(3)-2 Cleaner tests

Three cleaning tests (FL17, FL21 and FL22) were carried out in which the roughing and scavenging conditions were basically as per Test FL06.

In the first of these tests (Test FL17) rougher concentrates 1-2 were combined and cleaned/recleaned at pH 9.5 without collector addition. Rougher concentrate 3 and the scavenger were kept separate for the time being. It was envisaged that rougher concentrate 3 would probably be combined with the cleaner tailing in a later test and retreated, possibly after regrinding.

The result obtained in Test FL 17 was disappointing. The recleaner concentrate assayed 24% Cu at a recovery of 68.6%. This compares to Test FL06 in which rougher concentrate 1, alone, assayed 24.8% Cu at a recovery of 53.9%. The overall rougher concentrate 1-3 results were similar, 13.5% Cu at 93.4% recovery in FL06 and 12.2% Cu at 90.8% recovery in FL17, so the problem did not lie there.

In Test FL21 it was decided to leave rougher concentrate 1 uncleaned, similar to the high-grade flash flotation concentrate recovered from cyclone underflow in some plants. However to ensure a grade above 25% Cu the flotation tome was reduced from 30 secs to 20 secs. The rougher 2 flotation time was extended to 70 secs to maintain copper recovery. Rougher concentrate 2 was cleaned at pH 9.5. The cleaner tailing was combined with rougher concentrate 3, reground and refloated at pH 9.5 to produce a scavenger cleaner concentrate and a scavenger cleaner tailing. In a commercial operation the former would recycle to cleaner floatation and the latter would either recycle to the rougher section or directly to final tailing. A small amount of collector was added in the cleaning and scavenger cleaning stages to promote copper recovery.

The results were somewhat disappointing. As a consequence of a higher than expected weight recovery, rougher concentrate 1 assayed only 21.1% Cu. This undermined the objective of the test. In previous rougher tests rougher concentrate 2 had assayed between 9 and 13% Cu (with the exception of Test FL05). However in Test FL22 rougher concentrate 2, after cleaning, assayed only 11.2 % Cu, an inexplicable result. The rougher tailing from this cleaning stage when floated with rougher concentrate 3 gave a concentrate of 9.3% Cu, not much lower than the cleaning of rougher concentrate 2. Considering that rougher concentrate 3, in earlier tests, was low grade (2.8 %Cu in FL06) it is clear that the grade of the cleaner tail from the rougher 2 cleaner was high grade. In fact it appears that rougher concentrate 3 has simply not floated in this cleaning stage, reporting in entirety to the rougher 3 cleaner tailing. The net result was that rougher concentrate 1 together with the two other concentrates assayed only 16% Cu at a recovery of 84%.

In Test FL06, roughing alone gave a concentrate grade of 13.5 %Cu with a recovery of 93.4%

In Test FL22, which was carried out straight after FL21, conditions were similar with the exception of the cleaner and scavenger cleaner pH, which was higher at 11 to determine the effect on the grade-recovery relationship.

The result for Test FL22 was similar to FL21. Rougher concentrate 1 assayed only 20.2% and the two cleaner concentrates were again lower in copper grade than expected.

In both the latter cleaner tests gold recovery was in line with expectations from the rougher tests.

(3)-3 Diagnostic mineralogy of flotation products

In order to gain some understanding of the limitations imposed on the copper metallurgy by the mineralogy the following products were examined from Test FL22:

- Rougher concentrate 1
- Rougher 2 cleaner concentrate
- Rougher 3 cleaner tailing

In rougher concentrate 1, which assayed 20.2% Cu, chalcopyrite was the dominant copper mineral. Its liberation was approximately 50%. The pyrite and silicates in this product were poorly liberated at approximately 30% and 20% respectively. There was sufficient free silicates and pyrite especially (about 10% by weight) and locked silicates and pyrite (27% by weight) in this product to expect a higher grade to be obtained with slightly less aggressive conditions (reduced collector and flotation time). This would push the composites into the second rougher, which provides a route to regrind.

In the rougher 2 cleaner concentrate, which assayed only 10.8% Cu, only a third of the chalcopyrite was liberated. The dominant diluent was pyrite, which accounted for over 40% of the product. Of this 25% was liberated. A regrind of the rougher concentrate 2 is required. There are pyrite and silicate intergrowths of 1-30 μm in diameter in the chalcopyrite so a quite fine regrind is required.

In the tailing from the flotation of the combined rougher 2 cleaner tail and rougher concentrate 3 (2% Cu) each of the major minerals was approximately 90% liberated. It is notable that this product was mainly below 30 μm in size.

The mineralogy indicates that the rougher concentrates 2 and 3 both need to be reground to below 40 μm.

(3)-4 Gold recovery

The gold mineralogy of the Rakah Stockwork sample was examined and found to be as shown in Table Ⅲ-3-8.

Table Ⅲ-3-8 Gold mineralogy of Rakah stockwork

Size Fraction	Mode of occurrence	Distribution (%)
+20 μ m	Liberated native gold	15
	Locked in silicates	12
	Locked in sulphides	46
-20 μ m	Undifferentiated	27

In Test FL06 74% of the gold was recovered in rougher concentrates 1-3 with a further 12%

recovered in the scavenger concentrate. The latter is probably associated mainly with pyrite. The gold recovery is in line with the mineralogy since some of the liberated gold may be too coarse to float and some of the gold associated with pyrite and most of the gold associated with silicates will be lost to tailings. Gold in the rougher concentrates was above payable levels (normally 1-2 g/t).

(4) Hayl as Safil Stockwork

(4)-1 Rougher tests

The conditions, results and grade vs recovery curves are shown in Appendix 1C.

Tests FL07 and FL08 were carried out on the Hayl as Safil Stockwork sample. Test FL07 used the same conditions as Test FL02 on Rakah Stockwork. Test FL08 used a higher pH level of 12.5. Except for the pyrite flotation stage in FL07 conditions were otherwise identical.

The test at pH12.5 gave a much better result with a copper recovery of 94.8% at a grade of 12.2% Cu, in rougher concentrates 1-3. Gold was above payable levels but at a recovery of only 50%. However the gold head grade is only 0.16 ppm.

When the results of the second set of Rakah Stockwork tests came to hand a third test was carried out on Hayl as Safil. In Test FL13 the conditions were as per FL06 but at the higher pH of 12.5. The grade-recovery lines for these tests cross over at 92% recovery and 16% rougher concentrate grade. However Test FL13 used less reagent so the FL06/pH12.5 conditions were taken as standard. Additionally the calculated head in Test FL08 was a little low suggesting that the tailings assay may be low. This would have favoured Test FL13.

The overall aim was to develop a flowsheet using the same collector conditions as for Rakah Stockwork with changes to pH only required to get optimum metallurgy. This will make the plant easier to operate and control and reduce capital and possibly operating costs. At this stage of the testwork this had been achieved for Rakah Stockwork and Hayl as Safil in terms of rougher-scavenger conditions.

(4)-2 Cleaner tests

The next test carried out was FL18, which was a cleaning tests using the same conditions (pH excepted) as FL17 on Rakah Stockwork. The result was similar to the Rakah Stockwork cleaning test in that in Test FL18 the overall result was barely comparable in grade-recovery terms with roughing only. In Test FL13 rougher concentrate 1, alone, contained 73% of the copper at a grade of 22% Cu whereas in FL18 the cleaner concentrate contained 75% of the copper at a grade of 24% Cu. The recleaner concentrate grade was 28% Cu but the recovery was only 47.5%.

In line with the general approach of fitting the minor ore types as much as possible to the Rakah Stockwork flowsheet it was decided to await the outcome of the cleaning tests FL21 and FL22 on the latter before carrying out further cleaning tests on Hayl as Safil. Unfortunately the testwork programme had to be terminated before this could be carried out.

(4)-3 Diagnostic mineralogy

The following products from Test FL13 were examined mineralogically:

- Copper rougher concentrate 1
- Copper rougher concentrate 2

The mineralogical report is shown in Appendix 1D.

These two products contained 90% of the recovered copper and although a cleaner test was not carried out on this ore it was decided to determine the liberation of the copper for comparison mainly with the Rakah Stockwork testwork products.

Copper rougher concentrate 1 (22% Cu) had about the same copper content as rougher concentrate 1 from Test FL22 on Rakah Stockwork. However the liberation was 90% compared to only 50% with the latter. The silicate and pyrite mineralogies were similar. As with Rakah Stockwork a less aggressive float here should produce a higher grade and push more gangue towards regrind.

Copper concentrate 2 (8.8% Cu) was poorly liberated with similar characteristics to the rougher 2 cleaner concentrate from Test FL22 on Rakah Stockwork. The flowsheet that was recommended above, from the diagnostic mineralogy on Rakah Stockwork, should be suitable for Hayl as Safil .

(5) Rakah Massive Sulphide

(5)-1 Rougher tests

The conditions, results and grade vs recovery curves are shown in Appendix 1C.

Test FL09 used the same conditions as Test FL02, which had given high recoveries on Rakah Stockwork. It was expected that there would be far more weight in the pyrite concentrate in the massive sulphide sample. This sample contains 3.8 ppm gold so the gold distribution in the pyrite concentrate was of special interest. Test FL10 used the same conditions as Test FL09 except that the pH was 11 in copper roughing due to the need for better pyrite depression with this sample. Pyrite flotation was carried out at pH 9.

The test at the higher pH did give a better result. The rougher concentrates 1-3 assayed 26.5% Cu at a recovery of 84.9%. The pyrite concentrate assayed 32% S with a sulphur recovery of 57.9%. The sulphur recovery was only 42.5% in Test FL09 where more pyrite floated in the copper roughers. In terms of overall pyrite recovery Test FL09 was more successful with only 6% of sulphur remaining in the final tail compared to 22.3% in Test FL10.

Obviously to increase copper grade, with Test FL09 rougher conditions, pyrite would have to be rejected in the cleaning stage. However pyrite recovery may overall end up better under these conditions than with the conditions used in FL10. It was considered that an intermediate pH level could achieve good copper rougher grades yet reduce the depression of pyrite, allowing more to be recovered in the subsequent pyrite rougher.

Hence in Test FL14 the conditions were as per the best Rakah Stockwork result (Test FL06) but at a pH of 11. The result was similar to FL10 (the previous best for copper) with one exception. Although the pyrite stage in both tests was carried out under identical conditions the recovery of sulphur was much lower in FL14. The stage recovery was 54% compared to 72% in FL10.

(5)-2 Gold Recovery

When the gold assays became available they indicated a strong grade association between copper and gold in the copper roughers with 13-35 ppm gold in the rougher concentrates in test FL09. However the recovery was only 37% in the three rougher concentrates. Of the remainder, half reported to the scavenger concentrate and half to the pyrite concentrate. The gold grade of the pyrite concentrate was only 4 g/t, which may be too low to warrant recovery by roast/leach. The final tail in this test contained 10.5% of the gold and 5.5% of the sulphur indicating that there is either loss of free gold or some gold is tied up in silicates.

The gold associations in the Rakah Massive Sulphide were subsequently determined to be as Table Ⅲ-3-9.

Table Ⅲ-3-9 Gold mineralogy of Rakah massive sulphide ore

Size Fraction	Mode of occurrence	Distribution (%)
+20 μ m	Liberated native gold	3
	Locked in silicates	4
	Locked in sulphides	57
-20 μ m	Undifferentiated	36

There is a higher proportion (57%) of gold tied up in sulphides in this sample than in the Rakah Stockwork (46%). Much less of it is locked in silicates (3% versus 12%).

It was apparent that the higher sulphur recovery in Test FL09 (copper float at pH 9.5) compared to FL10 (copper float at pH11) has resulted in a higher gold recovery in both the copper concentrates (more pyrite in them) and the pyrite roughers (higher sulphur recovery).

(6) Tailings mineralogy

The reason for the lower rougher copper recovery for this sample was determined by mineralogical examination of final tailing. At 85%, copper recovery in copper roughers 1-3 was about 10% below the equivalent Rakah Stockwork and Hayl as Safil figures. The final tailing from Test FL20, which contained 10% of the copper, was mineralogically examined. The results are summarised in Table Ⅲ-3-10 .

Table Ⅲ-3-10 Tailing mineralogy of Test FL20

	Chalcopyrite	Covellite
Mineral proportion in Tailings, %	30	70
Mineralogy		
% liberated		10
% locked with pyrite	100	20
% locked with non-sulphide		50
% locked in complex particles		20

It is apparent that to achieve a higher recovery of copper from this material a finer primary grind will be required. As a minority ore this may not be warranted.

(6)-1 Cleaner tests

A single cleaning test was carried out (FL20). Copper roughing and scavenging were as per FL14 but with the same lime in the grind as FL10 (it had been observed that in Test FL14 an additional 1500 g/t of lime was consumed in the grind to no apparent benefit). The three rougher concentrates were combined and cleaned/recleaned. The copper scavenger concentrate was left separate. The pH for pyrite flotation was reduced to 8.5 in an effort to maximise sulphur recovery.

The cleaner concentrate assayed 43.4% Cu at a recovery of 78.3%. This compares with the rougher concentrates 1-3 (the cleaner feed) of 31% Cu at a recovery of 84.6%. It is very likely that the 12% higher concentrate grade is worth more in net return (after freight and smelting) than the 6.3% recovery loss.

There seems no reason why the Rakah Massive Sulphide would not respond well to the cleaner flowsheet tested on Rakah Stockwork in Tests FL21 and 22. In Test FL14 the rougher 1 concentrate assayed 41.5% Cu. If a flowsheet were developed in which a high-grade rougher concentrate 1 was produced as final concentrate then Rakah Massive Sulphide would serve as a "sweetener" in the ore blend.

In terms of the comparison between tests it is worth pointing out the copper results for Tests FL14 and FL20 in which the copper rougher conditions were identical.

Table Ⅲ-3-11 Comparative results of Test FL14 and 20

Test No.	Rougher 1-3 Grade, % Cu	Rougher 1-3 Recover, % Cu
FL14	30.9	85.5
FL20	31.0	84.6

This results as shown in Table Ⅲ-3-11 is excellent reproducibility without which the evaluation of different cleaner/recleaner flowsheets is impossible.

(7) Bishara Breccia

(7)-1 Rougher tests

The conditions, results and grade vs recovery curves are shown in Appendix 1C.

Initially two tests were carried out on the Bishara Breccia. In Test FL11 the conditions were as per Test FL06, which had been adopted as standard. In Test FL12 the conditions were as per Test FL06 but at an exploratory pH 8.5.

The results for both tests were similar with copper recovery in rougher concentrates 1-3 around only 66%. Unlike the other samples a high proportion (10%) of copper was recovered in the scavenger stage. To improve copper recovery, in Test FL15 the conditions were as per test FL11 but collector addition was increased. In the copper roughers the addition of each collector was increased by additions of 7 g/t and 3 g/t in roughers 2 and 3 respectively. The addition to the scavenger was doubled. Flotation time was left the same.

The result of Test FL15 showed a copper recovery only slightly higher than Test FL12. The scavenger tailing carried 19% of copper. Nearly 80% of the gold reported to the scavenger tailing.

In Test FL19 the Test FL15 conditions were repeated but at a finer flotation feed P_{80} of 60 microns. This had no effect on copper recovery. The scavenger tailing carried 19% of the copper.

The rougher concentrate 1 for this sample assayed 20-23% Cu in the four tests carried out to this point.

(7)-2 Tailings mineralogy

The final tail from Test FL15 was mineralogically examined to determine the nature of the copper losses (refer Appendix 1D). All of the copper was in chalcopyrite of which only 1% was liberated. Most (74%) of the chalcopyrite was locked in ternary (complex) particles.

(7)-3 Cleaner test

A cleaner test was carried out based on Test FL21, which was carried out on Rakah Stockwork. In this test a pyrite float was carried out as per Rakah Massive Sulphide and all products were assayed for gold. The Bishara head sample assayed 28% S, less than the Rakah Massive Sulphide but much higher than Hayl as Safil and Rakah Stockwork.

The result of this test was somewhat disappointing. Firstly although the flotation time had been reduced, the first rougher concentrate assayed only 19.2% Cu. This may have been a consequence of experimental technique, as much as anything, because the recovery was higher than expected. Secondly the cleaner concentrate produced from cleaning the second rougher concentrate (which had assayed about 13% Cu in the earlier tests) assayed only 13.8% Cu. The combined rougher one and cleaner concentrates assayed 17.3% Cu at a recovery of 63%, no better than the combined first two rougher concentrates in Test FL19.

Of the 84% if sulphur in the copper scavenger tail, less than half reported to the pyrite concentrate. Gold followed with 43% reporting to the final tailing. Of the 20% of copper in the copper scavenger tailing half reported to the pyrite concentrate. The pyrite concentrate itself assayed only 1.4 g/t gold.

(7)-4 Diagnostic mineralogy

The following products were examined mineralogically .

- Test FL23 copper rougher concentrate 1
- Test FL15 copper rougher concentrate 2
- Test FL23 scavenger concentrate

The mineralogical report is shown in Appendix 1D.

Copper rougher concentrate 1 from FL23 (19.2% Cu) was similar to the same product from Test 22 on Rakah Stockwork, but with more pyrite and less silicate. The liberation profiles are similar however. As with the Rakah Stockwork and Hayl as Safil samples, a higher rougher concentrate 1 should be able to be obtained.

Copper rougher concentrate 2 from Test Fl15 (13.2% Cu) was very poorly liberated and will certainly require regrinding, to below 30 microns.

Scavenger concentrate from Test FL23 (3.2% Cu and carrying 7% of the copper) was very poorly liberated. Over 90% of the chalcopyrite was locked with pyrite. This product contains more copper than the respective products from the other samples. The higher distribution of copper is indicative of the apparently poorer overall liberation of this sample at the standard primary grind size used. It is also consistent with the tailings mineralogy referred to above.

(8) Summary of batch flotation test results

The best flotation results obtained for each sample in the rougher series are shown in Table Ⅲ-3-12. Although pH varied for the different samples, all tests used a collector mixture comprising SIPX and M2030.

The proportions of the main ore types in the expected mill feed would be approximately as follows: Stockwork ore; 70%, Breccia; 15% and Massive sulphide; 15%.

The relative proportions of the two stockwork ore types (Rakah and Hayl as Safil) are not known. However taking them as 50:50 the overall weighted average rougher 1 concentrate performance for mill feed would be 25.6% Cu grade with a copper recovery of 56.2%.

Table Ⅲ-3-12 Summary of best rougher/scavenger flotation tests

	Rakah B. Stockwork	Rakah B. Massive	Hayl as Safil Stockwork	Bishara B. Breccia
Test No	FL06	FL14	FL13	FL15
Grind P_{80}, μm	70	70	70	70
pH	9.5	11	12.5	9.5
Cu Recovery, %				
Cu Ro Con 1	53.9	38.7	72.9	38.2
Cu Ro/Sc Con	95.7	88.0	94.4	80.8
Cu Grade, %				
Cu Ro Con 1	24.8	41.5	22.0	22.6
Cu Ro/Sc Con	8.6	18.1	10.7	10.3
Au Recovery, %				
Cu Ro/Sc Con	86	27.6	40.9	23.9
Py Con	--	35.6	--	28.5(1)
Au Grade, %				
Cu Ro/Sc Con	3.2	10.8	1.3	2.4
Py Con	--	3.5	--	1.4(1)

Ro: Rougher, Con: Concentrate.

(1): Results for pyrite concentrate produced in test FL23.

(9) Analyses of concentrates

High-grade rougher concentrates produced during batch flotation tests of three of the ore types were assayed for a range of elements to provide an indication of likely levels in final concentrates.

The assays for the respective concentrates are contained in Appendix 1E. The Rakah Stockwork and Hayl as Safil Stockwork concentrates were similar in content, although the Rakah sample was of higher As content (600ppm compared with 50ppm).

The Rakah Massive Sulphide concentrate was different in content to the stockwork concentrates. It contained a markedly higher As concentration of 6600ppm and a Hg concentration of 11ppm (compared to <1ppm in the stockwork concentrates). The massive sulphide concentrate was also lower in Al_2O_3, Fe_2O_3, MgO and SiO_2 concentrations.

(10) Flotation of bulk samples

In order to provide sufficient concentrate and tailing products for further testing, samples of 21kg of each ore type were processed.

Batches of 7kg ground to P_{80} 70μm were floated in a double batch cell of approximately 15L capacity using conditions established in the previous batch flotation tests. A summary of the tests and their respective conditions is provided in Table Ⅲ-3-13.

Table Ⅲ-3-13 Flotation of bulk samples

Sample	Test Conditions	Products
Rakah Stockwork	as per test FL05	Cu ro con 1+2, Cu tails
Hayl as Safil Stockwork	as per test FL13	Cu ro con 1, Cu tails
Rakah Massive Sulphide	as per test FL10	Cu ro con 1+2, Cu tails
Bishara Breccia	as per test FL23	Cu ro con 1, Cu ro con 2+3, Py con, Py tails

ro: rougher, con: concentrate.

The various products from the test were not weighed or assayed, but were retained as wet filter cakes pending settling and filtration tests.

3-3-5 Settling and filtration tests

Settling and filtration tests were conducted on copper flotation concentrates of each ore type, while settling tests only were conducted on flotation tailings of each ore type.

(1) Flocculant selection tests

Preliminary tests for flocculant selection were carried out on Rakah Stockwork concentrate and tailings.

Three flocculants were tested, namely:

- Nalco 9903 (non-ionic)
- Magnafloc M455 (cationic)

• Nalco 9602 (anionic)

Each flocculant was added at different dose rates to cylinders containing either concentrate or tailings slurry. The approximate settling rate and supernatant clarity was recorded and flocc characteristics were observed.

Two flocculants, Nalco 9903 and Magnafloc M455 were found to be of similar performance in terms of initial settling rates and supernatant clarity, and were effective at dose rates of the order of 10 to 20g/t.

Nalco 9903 was selected as the flocculant for the controlled tests.

(2) Settling test procedure

Portions of wet filter cake were added to 600mL cylinders to result in approximately 25% solids density slurry. After re-suspension of the solids using a plunger, the required dose of flocculant was then added in four stages as 0.05% solution (diluted to 50mL with water). Mixing of the solids with the flocculant continued for four more down strokes of the plunger, after which the plunger was removed and the initial slurry height recorded.

The interface level was then recorded periodically during the settling test. After the solids had settled through the compression zone, a picket rake operating at 6 rev/hour was inserted into the settled solids. The final settled height was recorded after a further 16 hours.

At the conclusion of the test the settled solids were filtered, dried and weighed, and the solids specific gravity measured. The interface height was plotted against time, and approximate unit settling area requirements calculated using the Kynch technique as modified by Talmage and Fitch ('Determining Thickener Areas', Talmage and Fitch, Ind. and Eng. Chem., Vol 47 No 1).

(3) Settling tests, copper flotation concentrates

Copper flotation concentrates of each ore type were produced for settling and filtration tests by flotation of bulk samples. The concentrates produced were not dried, but were retained as wet filter cakes prior to the settling tests.

The settling tests were conducted using 9903 flocculant at addition rates of approximately 0, 5, 10 and 20g/t. Some variations in actual addition rates occurred due to variations in sample weights which arose from estimations of moisture content in the wet filter cakes used as settling test feed.

The results obtained using flocculant dose rates of 10g/t are summarised in Table Ⅲ-3-14, although the optimum dose rate varied between samples.

Table Ⅲ-3-14 Settling tests of copper concentrates (10g/t flocculant)

	Rakah B. Stockwork	Hayl as Safil Stockwork	Rakah B. Massive	Bishara B. Breccia
Final U/F % Solids	67	62	64	64
Estimated Thickener Area m^2/tpd				
55 % Solids U/F	0.07	0.13	0.05	0.11
60 % Solids U/F	0.09	0.16	0.06	0.13
65 % Solids U/F	0.11	0.17	0.06	0.14

U/F: using 9903 flocculant.

(4) Settling tests, copper flotation tailings

Copper flotation tailings of each ore type were produced for settling tests by flotation of bulk samples. The tailings produced were not dried, but were retained as wet filter cakes prior to the settling tests. Note that for samples from which a pyrite concentrate was also produced (for gold extraction tests), the samples used for settling test represented the tailings from copper flotation (ie. prior to pyrite flotation. Thus the sample tested would represent pyrite flotation tailings recombined with pyrite concentrates after the gold extraction process, which would be final plant tailings.

The settling tests were conducted using 9903 flocculant (see Section 5.1) at addition rates of approximately 0, 10, 20 and 40g/t. Some variations in actual addition rates occurred due to variations in sample weights which arose from estimations of moisture content in the wet filter cakes used as settling test feed.

The results obtained using flocculant dose rates of 20g/t are summarised in Table Ⅲ-3-15, although the optimum dose rate varied between samples.

Table Ⅲ-3-15 Settling tests of copper tailings (20g/t flocculant)

	Rakah B. Stockwork	Hayl as Safil Stockwork	Rakah B. Massive	Bishara B. Breccia
Final U/F % Solids	63	67	70	68
Estimated Thickener Area m^2/tpd				
55 % Solids U/F	0.37	0.12	0.03	0.14
60 % Solids U/F	0.41	0.15	0.04	0.17
65 % Solids U/F	0.44	0.17	0.06	0.19

U/F: using flocculant

(5) Filtration tests, copper flotation concentrates

(5)-1 Procedure

Concentrate samples were added to water to 60% solids density and the solids flocculated using Nalco 9903 flocculant added at a rate of 10g/t. The slurry sample was then poured onto a top-feed vacuum filter fitted with Neotex 6044 filter cloth.

Vacuum was applied and the time required to form the filter cake (when the last free water disappeared from the cake surface) was recorded. The cake was kept under vacuum to dewater for a prescribed time. The final cake thickness was measured, the cake was discharged from the filter cloth and the wet weight recorded. After drying, the dry cake weight was determined.

Three tests were carried out on each sample using portions of different weight to result in different cake thicknesses. Drying times equivalent to 5, 1 and 0.2 times the form time were used to provide a range of dry time/cake moisture data.

(5)-2 Results

A summary is shown in Table Ⅲ-3-16.

The Hayl as Safil Stockwork concentrate demonstrated significantly poorer filtration characteristics than the other samples.

Table Ⅲ-3-16 Filtration tests of copper flotation concentrates

Sample	Dry: Form Time Ratio	Cake Moisture	Cake Thickness mm	Formation Rate $kg/h.m^2$
Rakah Stockwork	5 : 1	15.4	5	1,908
	1 : 1	16.3	10	1,141
	0.2 : 1	16.9	15	823
Hayl as Safil Stockwork	5 : 1	17.6	5	735
	1 : 1	17.5	10	449
	0.2 : 1	17.7	15	282
Rakah Massive Sulphide	5 : 1	16.2	5	1,968
	1 : 1	16.5	10	1,745
	0.2 : 1	16.6	15	975
Bishara Breccia	5 : 1	16.9	5	1,750
	1 : 1	17.0	10	875
	0.2 : 1	17.1	15	655

(6) Transportable moisture limit

The transportable moisture limit (TML) was determined for a combined copper concentrate comprising all copper concentrate products from the flotation of bulk samples of all ore types.

The procedure used was as described in the Code of Safe Practice for Solid Bulk Cargoes published by the International Maritime Organization, London, 1989. The determination involves sequential testing of a sample of concentrate at increasing moisture content on a slump table until a flow state is observed. The TML is taken as 90% of the flow moisture point, the latter variable being the average of moisture contents immediately before and after the flow state being achieved.

The TML for the combined copper flotation concentrate was measured as 11.25% moisture.

3-3-6 Gold recovery by cyanide leaching

Only the Rakah Massive Sulphide and Bishara Breccia samples contained sufficient quantities of gold in the copper flotation tailings to warrant investigation of recovery of gold. Furthermore, production of a pyrite concentrate from both samples resulted in significant gold in the tailings, so that extraction tests on both pyrite concentrate and tailings were undertaken.

The pyrite concentrate and tailings for the Rakah Massive Sulphide sample were obtained by combining products from batch flotation tests FL9m 10 and 14, while the Bishara Breccia samples were obtained by flotation of a bulk sample.

(1) Roasting of pyrite concentrates

Portions of each pyrite concentrate were blended with 2 parts sand (to reduce the chance of autogenous roasting at excessive temperatures) and roasted at 700° C for 4 hours in a muffle furnace. Air was injected into the furnace and the concentrates periodically rabbled to ensure oxidation of the total sample. After correcting for the effect of the dilution sand, the sample assays and weight losses during roasting are summarized in Table Ⅲ-3-17.

Table Ⅲ-3-17 Roasting of pyrite concentrates

	Rakah Massive Sulphide	Bishara Breccia
Initial sample weight, g	313	348
Final calcine weight, g	208	285
Weight loss in roast, %	34	18
Assays, Au g/t		
Before roast	3.5	1.8
After roast (1)	5.3	2.3

(1)Calculated from assay of un-roasted concentrate and weight loss during roast

(2) Cyanidation of pyrite flotation concentrates

Samples of pyrite concentrate were leached at 0.15% NaCN (nominal) and pH 11 for 48 hours. Solution samples were taken at prescribed times during the leach for Au assay and free cyanide titration. Reagents (NaCN and lime) were added as necessary to maintain the required leach conditions. Final residues were assayed for Au and Ag.

The concentrates leached were processed as follows prior to cyanide leaching:

- as floated
- re-ground
- calcined at 700° C

Detailed results for the cyanide leaches are contained in Appendix 1F, while a summary is shown in Table Ⅲ-3-18.

Table Ⅲ-3-18 Cyanide leaching of pyrite concentrates

	Rakah Massive Sulphide			Bishara Breccia		
	As Floated	Reground	Calcined	As Floated	Reground	Calcined
Grind P_{80}, μm	70	34	70	70	18	70
Head Assay, Au g/t	3.53	3.53	5.30	1.84	1.84	2.26
Calc. Head Assay, Au g/t	3.37	3.61	8.12	1.97	1.91	2.16
Residue Assay, Au g/t	2.42	1.93	1.18	1.40	1.13	0.75
Au Extraction, %						
8 hours	28	40	82	28	28	62
48 hours	28	44	85	30	30	66
NaCN Cons, kg/t	5.9	7.8	13.2	17.6	17.6	46.1

Gold recoveries from the 'as floated' concentrate of both samples were low at approximately 30%. Re-grinding increased the extraction to 40 to 45%, while roasting resulted in the highest extractions of 85% for Rakah Massive Sulphide and 66% from Bishara Breccia.

The low extractions and the relatively minor increase in extraction after re-grinding suggests that the contained gold is extremely finely disseminated in the sulphides (pyrite and, possible, arsenopyrite).

On the other hand, the higher gold extractions achieved after roasting indicates that oxidation of the concentrate by some means, such as roasting, pressure oxidation or bacterial oxidation offers the best chance of maximizing gold extraction.

However, cyanide consumptions recorded during the leaches were high, particularly after roasting. On the basis of the test results, cyanide costs would exceed the value of gold recovered from the Bishara Breccia sample and a major proportion of the gold recovered from the Rakah Massive Sulphide concentrate.

It is possible that the high cyanide consumptions are the result of dissolution of copper minerals remaining in the pyrite concentrates, or by direct reaction with the pyrite minerals.

Although consumptions would be reduced by shorter leach times (without lower gold extractions), cyanide consumption rates represent a major issue to be addressed in optimizing overall gold recovery from these samples.

(3) Cyanization of pyrite flotation tailings

Samples of pyrite flotation tailing were leached at 0.05% NaCN (nominal) and pH 11 for 48 hours. Solution samples were taken at prescribed times during the leach for Au assay and free cyanide titration. Reagents (NaCN and lime) were added as necessary to maintain the required leach conditions. Final residues were assayed for Au and Ag.

Detailed results for the cyanide leaches are contained in Appendix 1G, while a summary is shown in Table Ⅲ-3-19.

Table Ⅲ-3-19 Cyanide leaching of pyrite tailings

	Rakah Massive Sulphide	Bishara Breccia
Head Assay, Au g/t	1.93	0.92
Calc. Head Assay, Au g/t	1.66	0.87
Residue Assay, Au g/t	1.09	0.72
Au Extraction, %		
8 hours	31	16
48 hours	35	19
NaCN Cons, kg/t	6.7	4.3

Gold extractions from both samples were low at 35% from Rakah Massive Sulphide and 19% from Bishara Breccia.

As was the case for the pyrite concentrates, cyanide consumptions recorded for both samples were high, particularly in comparison with the low quantity of gold recovered.

3-4 Additional Cleaning Tests

3-4-1 Flotation Tests on Reground Cleaner Feed

A series of cleaning tests was conducted on each ore sample changing reground size as the second year test program. Their conditions are presented in Table Ⅲ-3-20. Their metallurgical results including overall comparison are shown in Table Ⅲ-3-21.

Table Ⅲ-3-20 Flotation tests on reground cleaner feed from Yanqul ore type

	Ore Type	Conditions	Cleaner Feed P80
FL01	RS	Rougher as for Test 6 N108, ball mill regrind of Ro/Sc conc. Cleaner float with mixed xanthate,	20
FL02	HASS	Rougher as for Test 13 N108, ball mill regrind of Ro/Sc conc. Cleaner float with mixed xanthate, thionocarbamate collector	20
FL03	BB	Rougher as for Test 6 N108, ball mill regrind of Ro/Sc conc. Cleaner float with mixed xanthate, thionocarbamate collector	21
FL04	RMS	Rougher as for Test 6 N108, ball mill regrind of Ro/Sc conc. Cleaner float with mixed xanthate, thionocarbamate collector plus pyrite float on Cu Sc	21
FL05	RS	Rougher as for Test FLo1, UFG mill regrind of Ro/Sc conc. Cleaner float with mixed xanthate, thionocarbamate collector	8.5
FL06	HASS	Rougher as for Test FLo2, UFG mill regrind of Ro/Sc conc. Cleaner float with mixed xanthate, thionocarbamate collector	10.3
FL07	BB	Rougher as for Test FLo3, UFG mill regrind of Ro/Sc conc. Cleaner float with mixed xanthate, thionocarbamate collector	17.8
FL08	RMS	Rougher as for Test FL04, UFGl mill regrind of Ro/Sc conc. Cleaner float with mixed xanthate, thionocarbamate collector plus pyrite float on Cu Sc	14.8
FL09	RS	As for Test FL05 ball mill regrind, low density rougher and cleaner with silicate dispersant	18.2
FL10	HASS	As for Test FL06 ball mill regrind, low density rougher and cleaner with silicate dispersant	21
FL11	BB	As for Test FL07 ball mill regrind, low density rougher and cleaner with silicate dispersant	16.5
FL12	RMS	As for Test FL08 ball mill regrind, low density rougher and cleaner with silicate dispersant	15.9
FL13	RS	As for Test FL09, then 3 cleaner stages plus CPS cleaner scavenger on 1st cleaner tail	~20
FL14	HASS	As for Test FL10, then 3 cleaner stages plus CPS cleaner scavenger on 1st cleaner tail	~20
FL15	BB	As for Test FL11, then 3 cleaner stages plus CPS cleaner scavenger on 1st cleaner tail	~20
FL16	RMS	As for Test FL12, then 3 cleaner stages plus CPS cleaner scavenger on 1st cleaner tail plus pyrite floatonCuscavengertail.	~20

Two types of regrind mill are used in practice, either a ball mill or a stirred mill for ultrafine product sizes. Conventional regrind ball mill is probably limited to the area above 20micron, where-ad the UFG mills will handle the area below 30 micron down to 5 or less.

Tests FL01-FL04 were conducted on each ore type using the ball mill regrind with a regrind target of about P_{80} 20 μ m.

Tests FL01-FL04 were conducted using the UFG mill regrind with a regrind target of about P_{80} 10-15 μ m.

The BB ore did not quite grind to the target grind in the UFG mill but all other ores were within the target ranges.

Table Ⅲ-3-21 An overall comparison of the metallurgical results from Yanqul ore type

Ore	Test No	Stage	Time	Con Cu%	CL Cu Rec%	CL Au Rec%
RS	FL01	1stCleaner	8.5	21.4	76.5	64.6
RS	FLo5	1stCleaner	8.5	18.0	94.6	69.9
RS	FL09	1stCleaner	8.5	22.8	79.4	57.2
RS	FL13	1stCleaner	11.0	17.8	90.6	70.2
RS	FL13	3rdCleaner	6.5	22.4	84.6	60.3
HASS	FL02	1stCleaner	5.5	19.4	90.1	38.3
HASS	FL06	1stCleaner	5.5	20.8	90.6	42.2
HASS	FL10	1stCleaner	5.5	21.6	72.7	27.5
HASS	FL14	1stCleaner	11.0	18.5	87.1	40.8
HASS	FL14	3rdCleaner	6.5	24.6	68.1	23.2
BB	FL03	1stCleaner	6.0	13.3	71.8	17.8
BB	FL07	1stCleaner	6.0	15.1	74.6	15.9
BB	FL11	1stCleaner	6.0	16.6	66.1	11.9
BB	FL15	1stCleaner	11	16.8	71.1	14.1
BB	FL15	3rdCleaner	6.5	20.7	57.2	9.1
RMS	FL04	1stCleaner	5.5	25.6	70.8	20.2
RMS	FL08	1stCleaner	5.5	21.5	69.6	23.5
RMS	FL12	1stCleaner	5.5	26.6	60.2	16.3
RMS	FL16	1stCleaner	11.0	12.9	70.0	26.3
RMS	FL16	3rdCleaner	6.5	23.2	65.5	19.2

The flotation tests were scoped to follow the best rougher/scavenger flowsheets of the previous work as noted on the conditions table above. After regrinding the total rougher/scavenger concentrates, a single cleaner stage was conducted with re-reagentising to compensate for the new surface produced. The pyrite float was retained for the RMS ore as nearly 50% of the Au had previously reported here. There was some very marginal improvements in UFG in the BB and HASS ores for Cu. It was possible that the RS and RMS were deficient in collector, though extra collector would have compromised the Cu grade further. For Au, the UFG gave a small improvement in Au recovery for the BB and RMS ores.

3-4-2 Locked Cycle Test

In order to check the effect of recycle, 6 stage locked cycle test was conducted on composite sample from Yanqul four types of ore, using flow sheet Fig. Ⅲ-3-2. Test conditions are presented in Table Ⅲ

-3-22. Test results are summarized in Table Ⅲ-3-23.

Table Ⅲ-3-22 Locked cycle test condition

on composite of 40%RS, 40%HASS, 10% BB & 10%RMS

Stage	Time	SIPX	M2030	MIBC	A60	A412	KAX	pH
Conditioner	2.0	15	15	25	200	--	--	7.83
Rougher	8.0	5	5	10	--	--	--	11.54
Pyrite Rougher	6.0	--	--	10	--	--	30	11.21
1st Cleaner	11.0	17	17	15	100	--	--	11.50
Cleaner-Scavenger	5.0	2	2	--	--	7.5	7.5	11.52
2nd Cleaner	7.0	2	2	5	--	--	--	11.66
3rd Cleaner	6.5	2	2	5	--	--	--	11.54

Table Ⅲ-3-23 Locked cycle test result on composite sample

PRODUCT	Weight	Assay		Recovery %	
	%	Cu %	Au g/t	Cu %	Au %
Caluculated Head	100.00	1.12	0.76	100.0	100.0
Cu 3rd Cleaner Concentrate	4.33	20.5	5.5	79.2	32.2
Cu 1st Cleaner Tail	5.36	2.39	2.17	11.4	14.6
Cu Scavenger Tail	90.31	0.12	0.45	9.4	53.2

Above results show that Cu concentrate of 20% and Cu recovery of approximately 80% will be able to be obtained in practical plant operation.

3-5 Conclusions

3-5-1 Copper flotation

Copper recoveries from rougher/scavenger flotation varied from 94% to 96% for the stockwork samples to 80 to 90 % for the breccia and massive sulphide samples. Concentrate grades also varied, ranging from 42% in the rougher 1 concentrate for Rakah Massive sulphide to 22 to 25% for the remaining samples.

With the exception of the Bishara Breccia the Oman Copper samples responded very well to rougher-scavenger flotation at a moderately fine grind P_{80} of 70 microns. Finer grinding failed to improve the metallurgical response of the Bishara sample in which the copper losses in the scavenger tailings were entirely locked. The intermediate level of rougher-scavenger recovery for the Rakah massive Sulphide sample was also due to liberation limitations.

A significant result of the test program is that all samples responded well to the same collectors, with pH being the only main variable between the conditions suitable for the respective ore types.

3-5-2 Gold recovery

Gold grades in the various samples and recovery of gold to the copper flotation concentrates (for the best rougher tests, see Section 4.8) are summarised in Table Ⅲ-3-24.

Table Ⅲ-3-24 Gold recovery to copper concentrates

	Rakah B. Stockwork	Hayl as Safil Stockwork	Rakah B. Massive	Bishara Breccia
Test No	FL06	FL13	FL14	FL15
Head assay, Au g/t	0.45	0.16	3.78	1.06
Au recovery to Cu con	86.1	40.9	27.6	23.9
Au grade in Cu tail	0.08	0.16	2.8	0.94

The stockwork samples were low in gold grade and, for the Rakah Stockwork sample, gold recovery to the copper concentrate was high.

Only the Rakah Massive Sulphide and Bishara Breccia samples contained sufficient gold in the copper flotation tailings to justify production of a pyrite concentrate to improve gold recovery.

The overall gold recoveries obtained by the various stages of processing, as a percentage of total gold, are shown in Table Ⅲ-3-25. The figures in ***italics*** represent actual extractions of gold from the feed samples.

Table Ⅲ-3-25 Overall gold recoveries

	Rakah B. Stockwork	Hayl as Safil Stockwork	Rakah B. Massive	Bishara Breccia
Test No	FL06	FL13	FL14	FL15, FL23
Head assay, Au g/t	0.45	0.16	3.78	1.06
Au recovery to Cu con, %	***86.1***	***40.9***	***27.6***	***23.9***
Au recovery to Py con, %			35.6	28.5
Au extraction by cyanidation of Py con (calcine), %			***31.3***	***18.8***
Au recovery to Py tail, %			36.8	43.4
Au extraction by cyanidation of Py tail, %			***12.9***	***8.2***
Total Au Recovery, %	***86.1***	***40.9***	***71.8***	***50.9***

Thus gold grades and recoveries varied considerably between the samples.

It should be remembered that cyanide leaching of the Rakah Massive Sulphide and Bishara Breccia pyrite concentrates and tailings resulted in excessive cyanide consumptions, to the point that the cyanide cost would exceed the value of gold recovered. In addition, oxidation of the pyrite concentrates was required to allow reasonable gold extractions to be achieved.

Based on the nature of the samples tested, the benefit of a specific circuit for gold recovery will depend on the quantity of ore containing significant gold grades.

Quantities of the various ore types aside, the relatively refractory nature of the gold will create difficulties in processing, so that ultrafine regrinding and additon of sulfurizer tests were conductede to examine ways of maximizing gold recovery to copper flotation concentrates from all ore types. Results of above tests, however, could not be found to be significantly successful.

In addition, we conducted pressure oxidation and bacterial oxidation tests in order to promote Au extraction. Small improvement could be found but it is estimated that these processes are not practical due to their expensive operating costs.