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.

2-4-2 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 II -2-3.

Table II -2-3 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.

2-4-3 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 8B. Table II -2-4 contains a summary of the grinding work index test results.

Table II -2-4 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.

2-4-4 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 II-2-5.

Table II-2-5 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 8C.

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 II -2-6.

Table II -2-6 Gold mineralogy of Rakah stockwork

Size Fraction	Mode of occurrence	Distribution (%)
+20 μm	Liberated native gold	15
	Locked in silicates	12
	Locked in sulphides	46
-20 μm	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 8C.

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 8D.

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 8C.

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 II-2-7.

Table II-2-7 Gold mineralogy of Rakah massive sulphide ore

Size Fraction	Mode of occurrence	Distribution (%)
+20 μm	Liberated native gold	3
	Locked in silicates	4
	Locked in sulphides	57
-20 μm	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 II-2-8.

Table II-2-8 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 Ⅱ-2-9 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 Ⅱ-2-9 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 8C.

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 8D). 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 8D.

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 II-2-10. Although pH varied for the different samples, all tests used a collector mixture comprising SIPX and M2030.

Table II-2-10 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.

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%.

(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 8E. 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 Ⅱ-2-11.

Table Ⅱ-2-11 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.

2-4-5 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 II-2-12, although the optimum dose rate varied between samples.

Table II-2-12 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 II-2-13, although the optimum dose rate varied between samples.

Table II-2-13 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 II-2-14.

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

Table II -2-14 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 Organisation, 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.

2-4-6 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 II -2-15.

Table II-2-15 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 8F, while a summary is shown in Table II-2-16.

Table II-2-16 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) Cyanidation 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 8G, while a summary is shown in Table Ⅱ-2-17.

Table Ⅱ-2-17 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.

2-5 Conclusions

2-5-1 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.

2-5-2 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 II-2-18.

Table II-2-18 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 II-2-19. The figures in ***italics*** represent actual extractions of gold from the feed samples.

Table II-2-19 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 an option to be considered might be to examine ways of maximising gold recovery to copper flotation concentrates from all ore types.

Proposed ore processing flowsheet based on the phse I study is shown in Fig. II -2-3.

2-6 Recommendations for Further Testing

2-6-1 Future flotation testwork

Future flotation testing should focus on cleaner flotation, which, for the two Rakah samples and the Hayl as Safil sample, have not been successful in the limited number of tests carried out to date.

The diagnostic mineralogy carried out on products from tests on all samples, except Rakah Massive Sulphide, indicated the following.

- A higher-grade rougher concentrate 1 should be achievable on all three samples.
- Both the rougher concentrate 1 and the rougher concentrate 2 products from all three samples will have to be reground to below 40 microns, at least. The Bishara Breccia would probably benefit from a finer regrind than this. Because of the grade disparity between these products in all samples, the temptation to combine them for a single regrind should be resisted.
- The scavenger concentrate from the Bishara sample was irretrievably locked and most likely this is the case for the other samples, in which this product carried less value. This flotation stage may have to be abandoned as it hardly warrants separate (perhaps ultrafine) regrinding.

The results of diagnostic mineralogical work are quite positive, since the level of regrind is readily achievable without the use of ultrafine grinding technology. A conventional tower mill, or even a horizontal ball mill, will probably suffice. The level of regrind should be the focus of the next stage of the investigation.

2-6-2 Future gold Recovery testwork

Based on the results of the cyanide leaching carried out in the current test program, it is recommended that a program be undertaken in the future to examine issues including the following items:

- oxidation of pyrite concentrates by roasting, pressure oxidation and bacterial oxidation

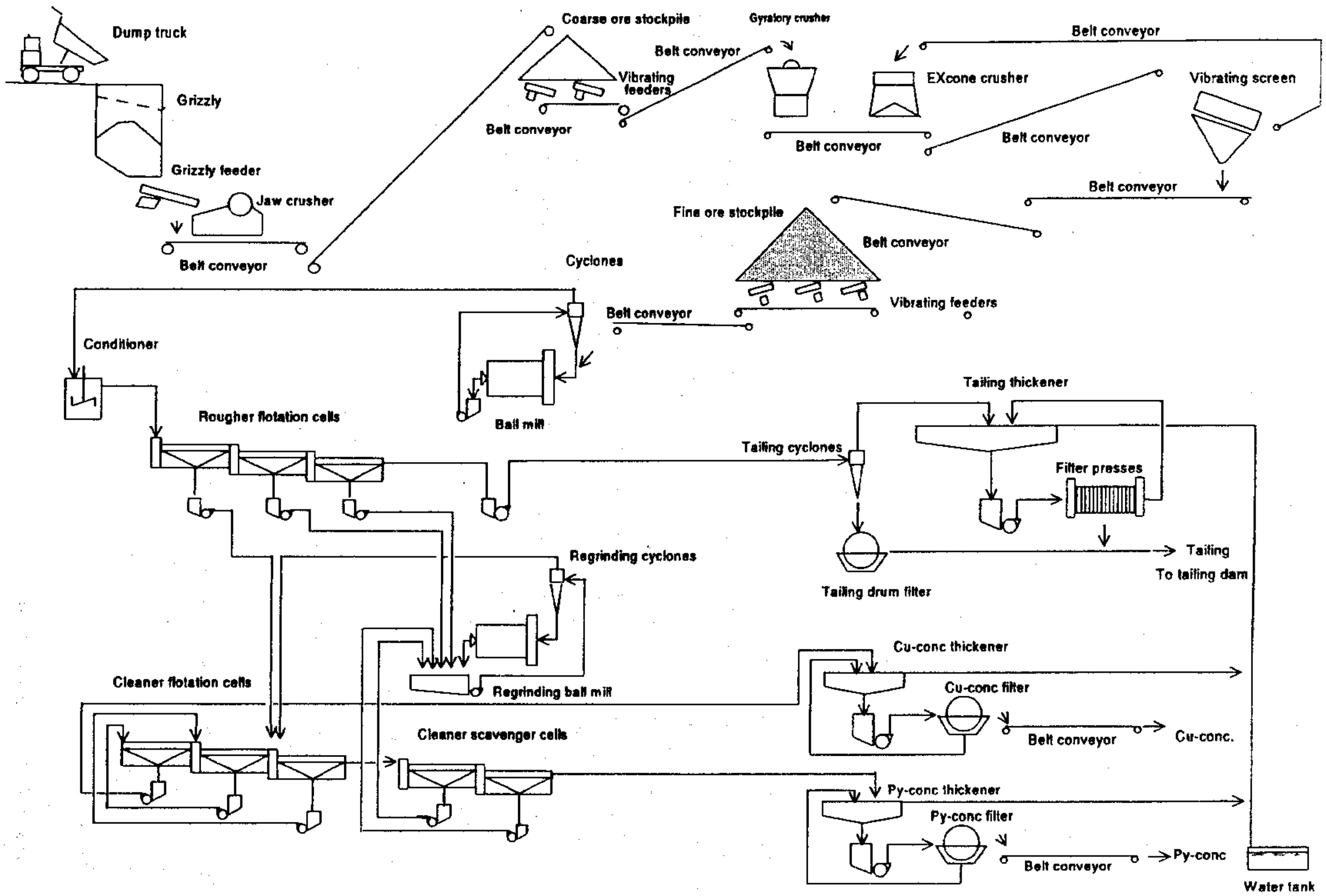


Fig. II-2-3 Proposed ore processing flowshee

- ultra-fine grinding of pyrite concentrates
- optimisation of cyanide concentrations and leach times
- mineralogical examination of flotation tailing leach residues to identify the nature of residual gold
- mineralogical examination of pyrite concentrate leach residues to obtain information on the distribution of gold between pyrite and arsenopyrite grains

Such testing should be carried out in conjunction with additional work to optimise flotation performance to maximise copper and gold recovery to copper concentrates to minimise subsequent difficulties in cyanidation.

CHAPTER 3 ENVIRONMENTAL INVESTIGATION

3-1 Objective

In order to perform groundwater investigation, concerning on behavior of the groundwater, permeability and water quality in around area of Rakah and Hayl as Safil deposits, bore holes were drilled and water level recovery tests and water quality analysis were executed.

3-2 Survey Method

3-2-1 Bore hole drilling

(1) Drilling location

For performance of the groundwater investigation, bore hole drilling works were made at 5 points presented in Fig. II -3-1.

(2) Structure of bore holes

Structure specified for bore hole is presented in Fig. II -3-2. Since each bore hole may be used as monitoring well in future, screen casing pipes were inserted. Besides, tough mouth guard was constructed at each top portion of the bore hole in order to prevent from burial due to flood water or human tamper etc.

(3) Progress and procedure of drilling

Drilling progress are presented in Table II -3-1.

Drilling procedure is as follows.

① Pilot hole mouth protection by used drum buried in hand-dug hole.

② Pilot hole drilling by 17-1/2 inch bit with sampling of cuttings.

③ Insert and fixation by mortar of 13-3/8 inch by 4.75m long casing steel pipe.

④ Main drilling by 12-1/4 inch bit with sampling of cuttings every 1 m interval.

⑤ Insert of 8 inch PVC screen casing pipes.

⑥ Pack small washed gravel around screen casing.

⑦ Insert 4 inch airlift steel pipe and water pumping for cleaning of bore hole inside.

⑧ Removal of 4 air lift pipe.

⑨ Install 13-3/8 inch top guard capping.

⑩ Assemble formwork around guard capping.

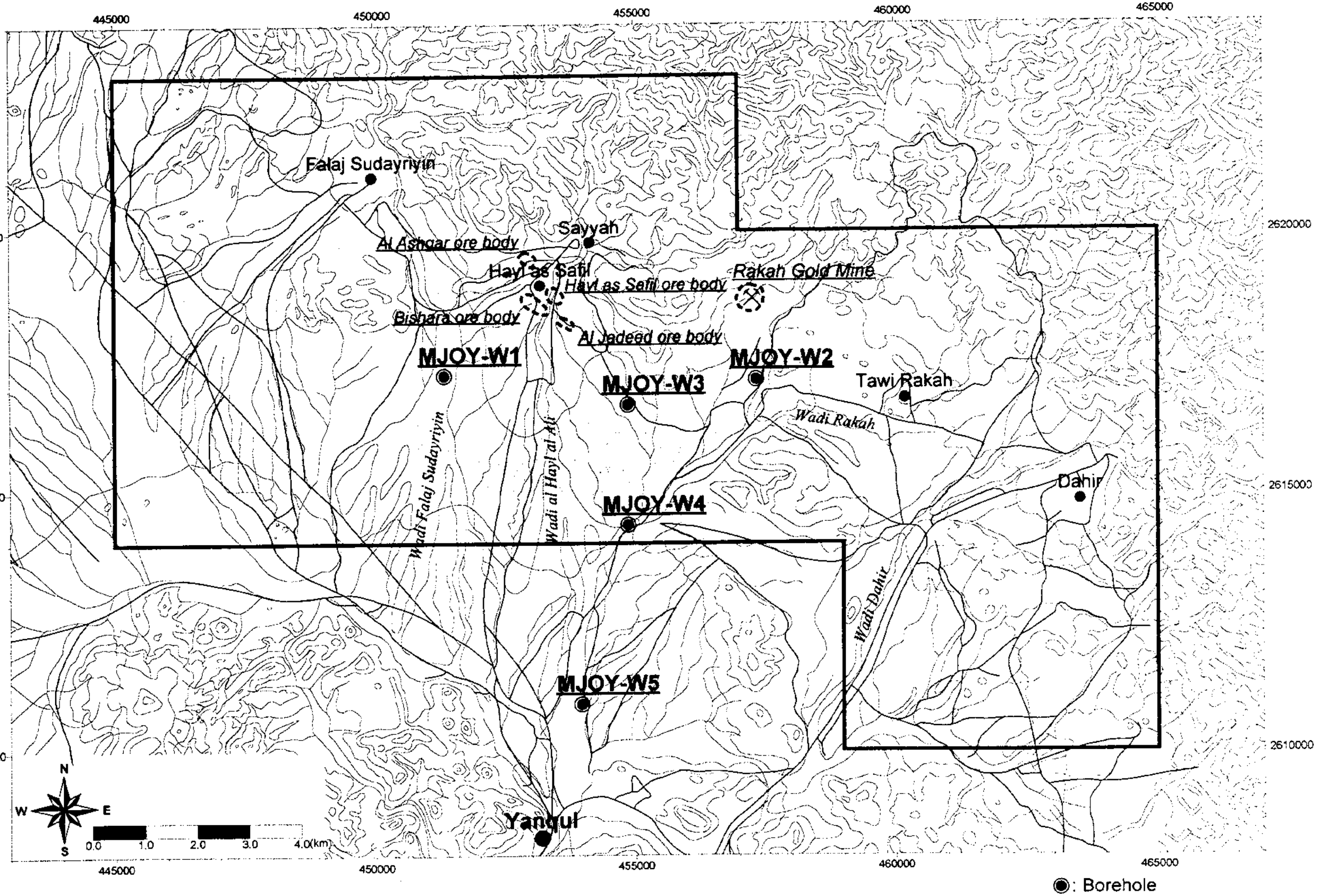


Fig. II-3-1 Location map of boreholes for groundwater survey

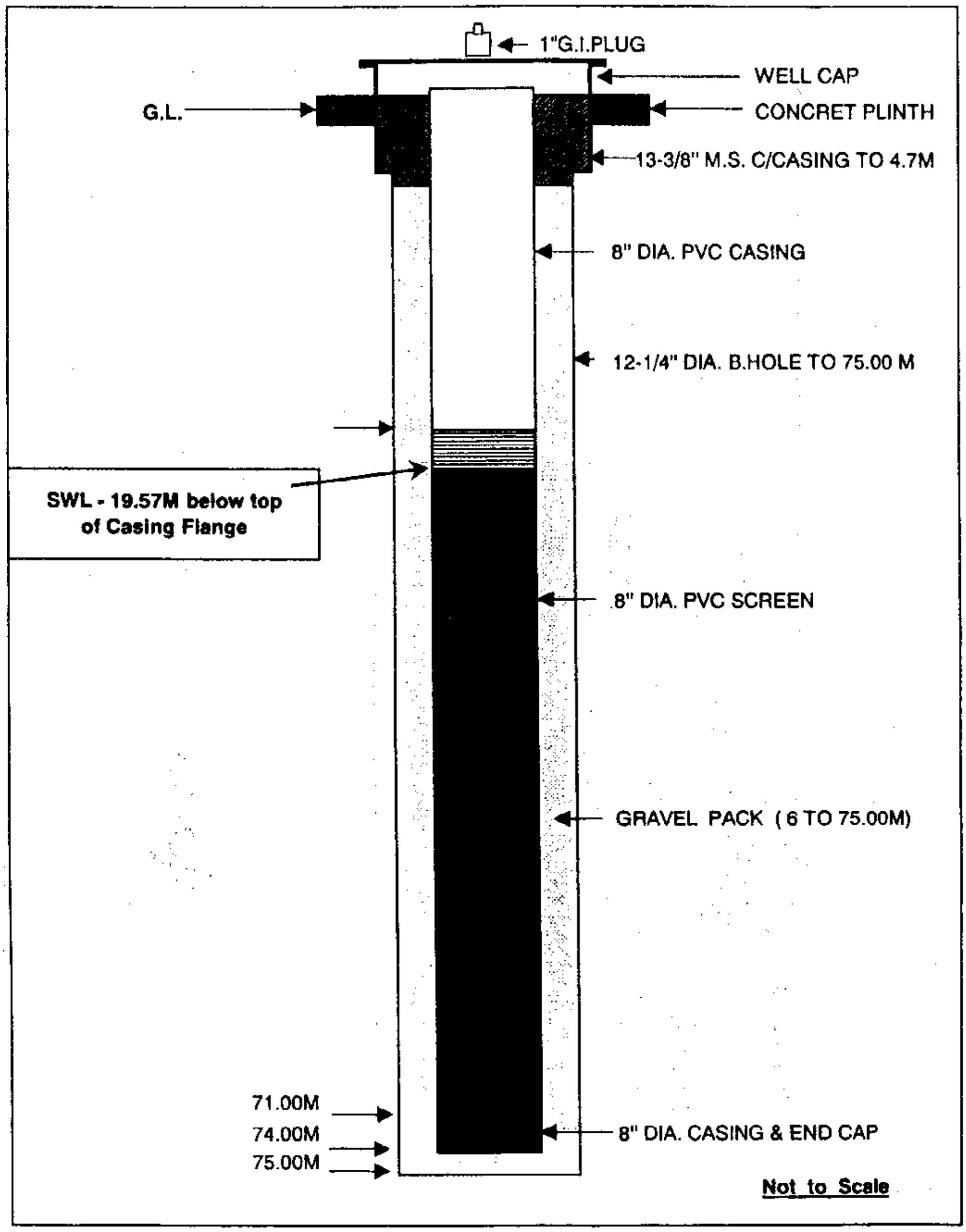


Fig. II-3-2 Bore well construction details

⑪ Concrete placing.

⑫ Removal of formwork and painting.

Table II -3-1 Drilling work for groundwater survey

Drill Hole No.	Location	UTM Coordinate		Ground Level (m)	Depth (m)	Pumping test	Water sampling
		Northing (km)	Easting (km)				
MJOY-W1	Wadi Falaj Sudayriyin	2,617.235	451.334	630	75.00	S.D.T *1 C.D.T *2 R.T *3	1
MJOY-W2	Wadi al Hayl al Ali	2,617.148	457.335	660	75.00	S.D.T *1 C.D.T *2 R.T *3	1
MJOY-W3	Wadi Rakah	2,616.673	454.867	640	75.00	S.D.T *1 C.D.T *2 R.T *3	1
MJOY-W4	Wadi Rakah & Wadi al Hayl al Ali	2,614.358	454.867	600	75.00	S.D.T *1 C.D.T *2 R.T *3	1
MJOY-W5	Wadi Rakah	2,610.903	453.964	565	75.00	S.D.T *1 C.D.T *2 R.T *3	1

*1 : S. D. T : Step Drawdown Test

*2 : C. D. T : Constant Discharge Test

*3 : R. T : Recovery Test

(4) Used equipment list, consumable items and quantity

Used equipment list, consumable items and their quantity are presented in Table II -3-2.

3-2-2 Groundwater investigation

(1) Groundwater quality

Groundwater was sampled out of each bore hole and measured pH, electric conductivity and water temperature related groundwater near water surface.

(2) Pumping test

Ground water was pumped up at each drilled bore hole and recovering situation of groundwater level was measured until natural water level to investigate recoverability of groundwater level. Measuring time interval at recovery test of water level was based on the following Table II -3-3.

Table II-3-2 List of equipments

Items	Equipments	Remarks
1. Drilling Equipment	1) Type	Model-T4W Top head drive rig
	2) Manufacturer	Ingersollrand, USA
	3) Quantity	1 set
	4) Mounting method	4WD Truck mount
	5) Drilling capacity	400m
	6) Drill rod	4-1/2" ϕ ×25'L×54m
	7) Drill	collar 6-5/8" ϕ ×7.6m
	8) Drill bit	17-1/2" and 12-1/4"TCT each 1 pc
2. Vehicles	1) Crane	4.5t UNIC hydraulic winch×1 set
	2) Water tanker	$6.5m^3$× 1 set
	3) Truck	9t Long chassis× 1 set 9t Long chassis× 1 set
	4) Pick-up	Double cab type× 1 set
3. Consumables	1) Drilling foam	Shell, UK×650Lit
	2) Drilling polymer	NL Baroid Industries Inc., USA EZ Mud×60kg Aquagel×185kg
	3) Well casing	Cosmoplast Industrial Co., Ltd 8" ϕ PVC casing pipe×104m 8" ϕ PVC screen×207m PVC End cap×5pcs 13-3/8" ϕ Mild steel conductor pipe×30m 13-3/8" ϕ Mild steel flanged caps×5pcs
	4) Pea gravel	Al-Turki Crusher, Oman×20 m^3
	5) Cement	Oman Cement Co. LLC×2000 kg
	6) Fuel	Diesel oil×11,000 Lit
	7) Lubricant	Shell Lubricant×200 Lit

Table Ⅱ-3-3 Measurement time schedule for recovery test

Setting time of measurement	Measurement interval
After 0 min. to 10 min.	Every 1 min.
After 10 min. to 20 min.	Every 5 min.
After 20 min. to 60 min.	Every 10 min.
After 60 min. to 120 min.	Every 15 min.
After 120 min. to 300 min.	Every 30 min.
After 300 min.	Every 60 min.

(3) Water quality analyses

Water samples were taken at each bore hole for water quality investigation before pumping test from zone near ground surface level February 6th to 16th in 2001. PH values and electric conductivities were measured in the field and 9 elements of heavy metallic ions including the following (Cu, Zn, Pb, Ni, Cr, Fe, Mn Hg, SO_4) were analyzed in the laboratory.

3-3 Survey Results

3-3-1 Drilling

Core log sheets of MJOY-W1~W5 are shown in Appendix 3C and summarized in Fig. Ⅱ-3-3. The profile across bore holes are shown in Fig. Ⅱ-3-4(1)~(4).

All of holes have the overburden of wadi sediments and terrace sediments with 13m to 22m in thickness. The terrace sediments reach 30m to 40m in thickness at terrace areas, but all holes located in a wadi show less thickness of the sediments because of erosion.

Wadi sediments consist of unconsolidated sand and gravels, and are only several meters in thickness. Terrace sediments also consists mainly of sand and gravels, but they are consolidated with calcarious matrix. The lowest part of terrace sediments are interbeded with sand layer in places.

(1) MJOY-W1 borehole

0.00m~-13.00m	Consisting of unconsolidated wadi sediments with sand and gravels and consolidated terrace sediments.
-13.00m~-30.00m	Reddish brown chert with interbeded grey slate.
-30.00m~-37.00m	Alternating beds of reddish brown chert and grey slate.
-37.00m~-63.00m	Reddish brown chert with small amount of interbeded grey slate.
-63.00m~-71.00m	Reddish brown chert.
-71.00m~-75.00m	Grey slate.

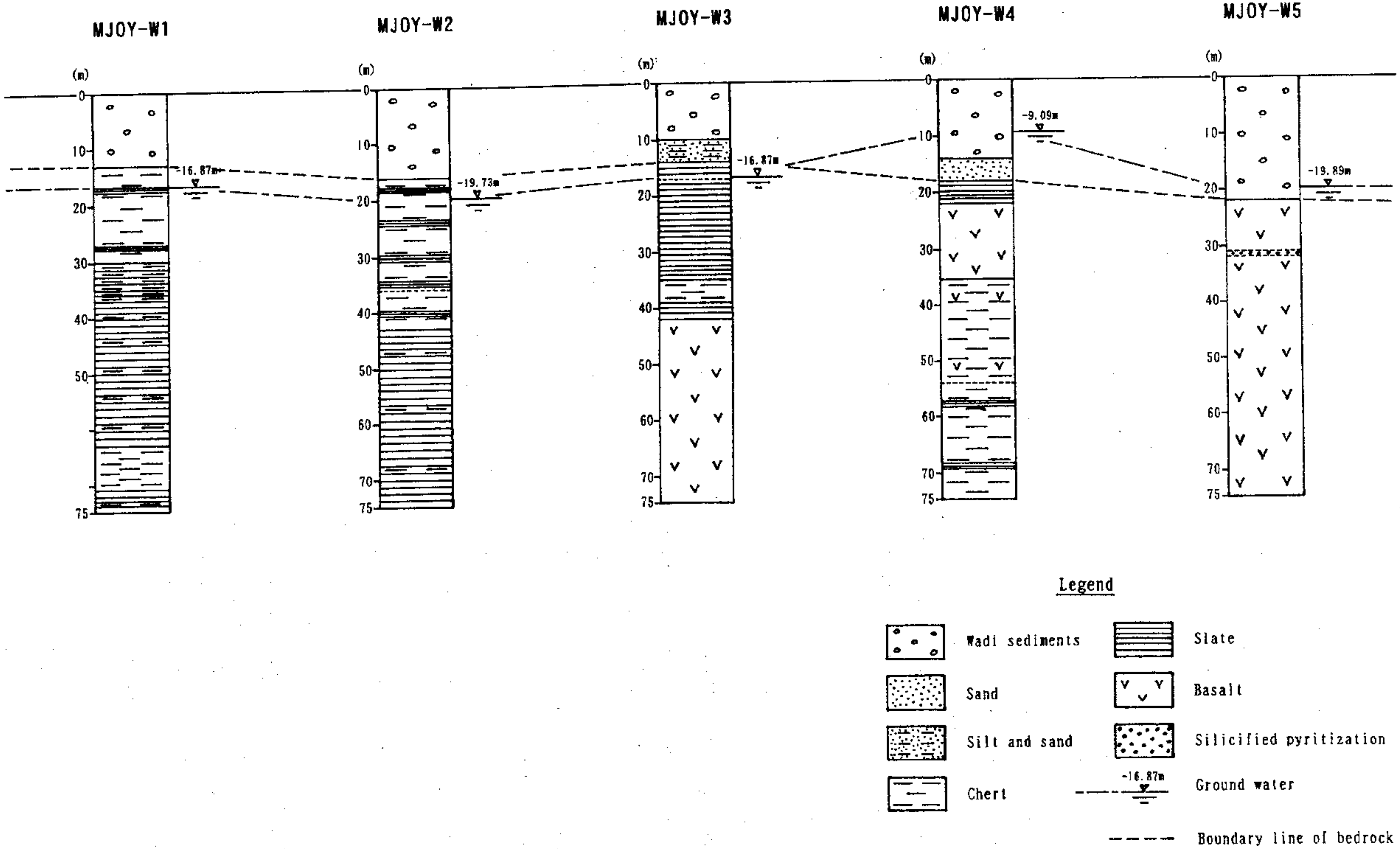
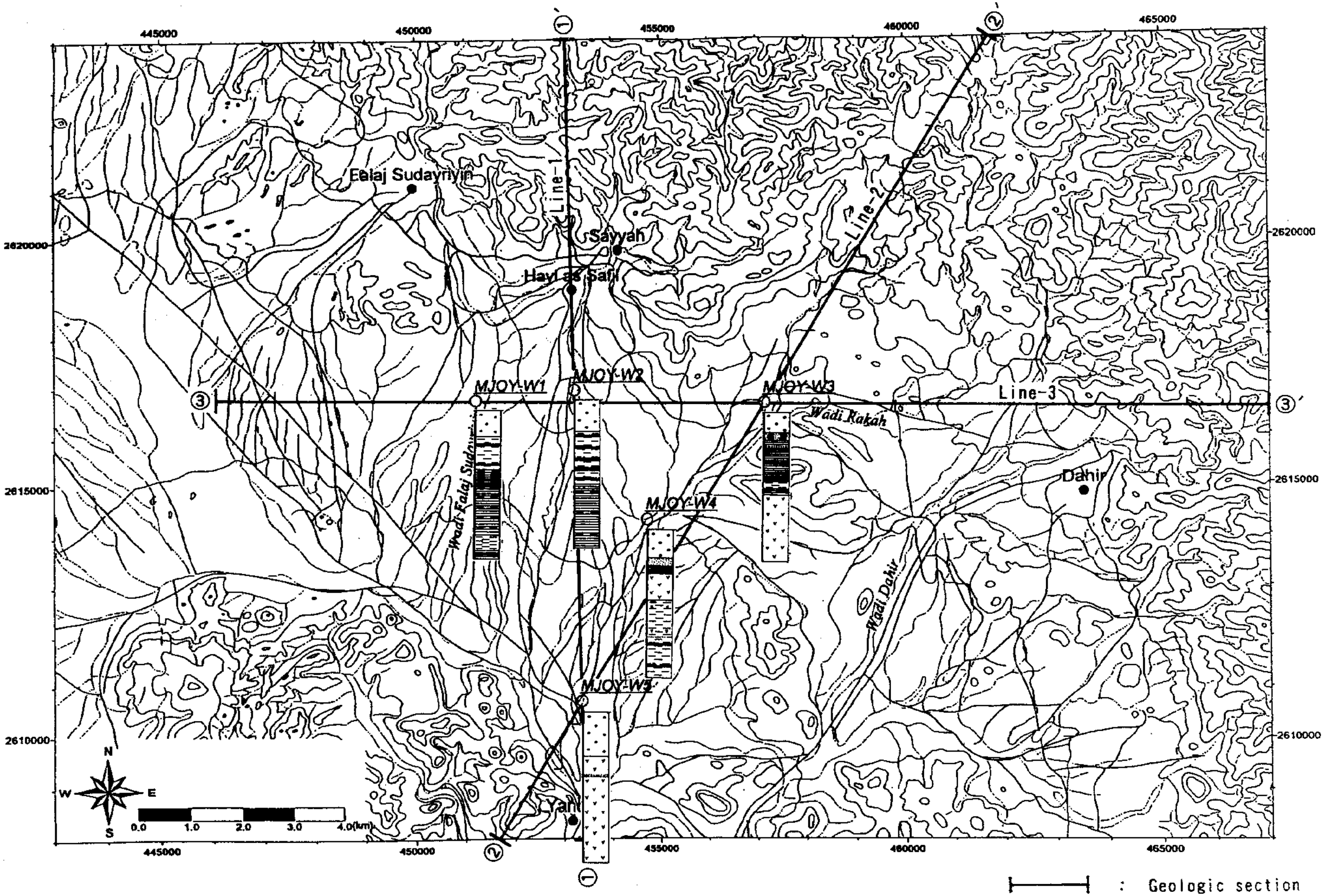


Fig. II-3-3 Geological columnar section



: Geologic section

Fig. II 3.4(1) Geological section(1)

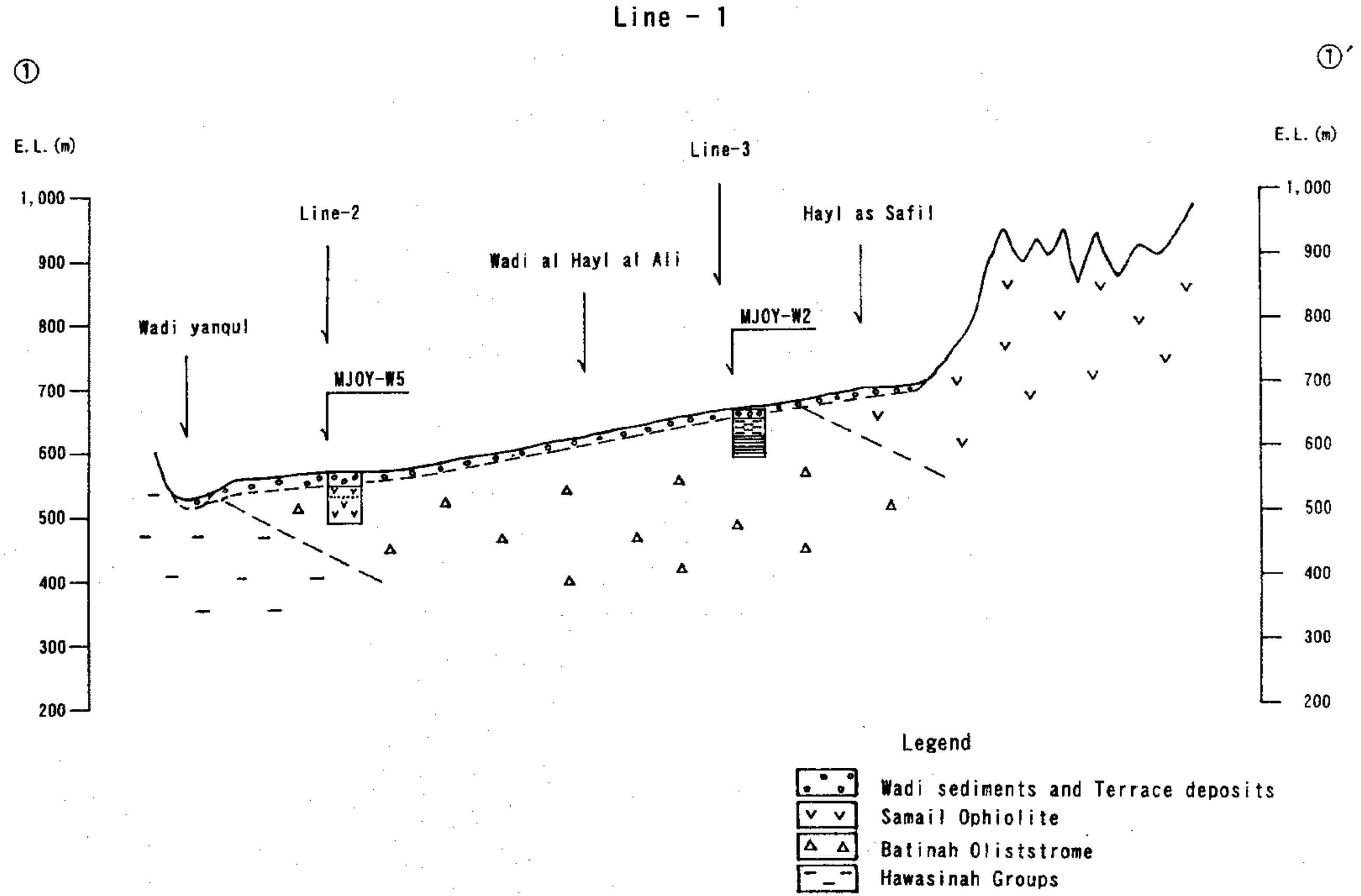


Fig. II-3-4(2) Geological section(2)

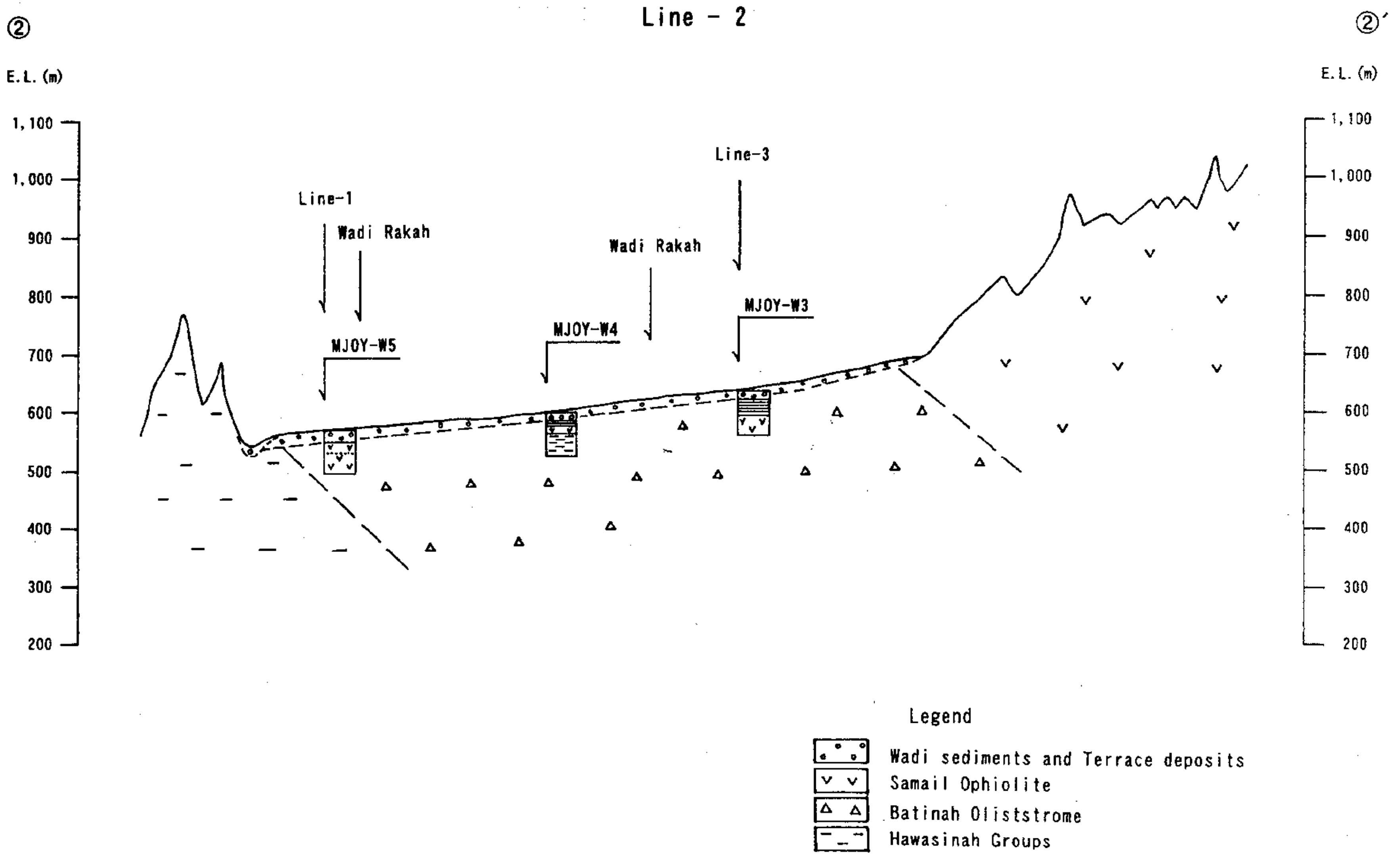


Fig. II-3-4(3) Geological section(3)

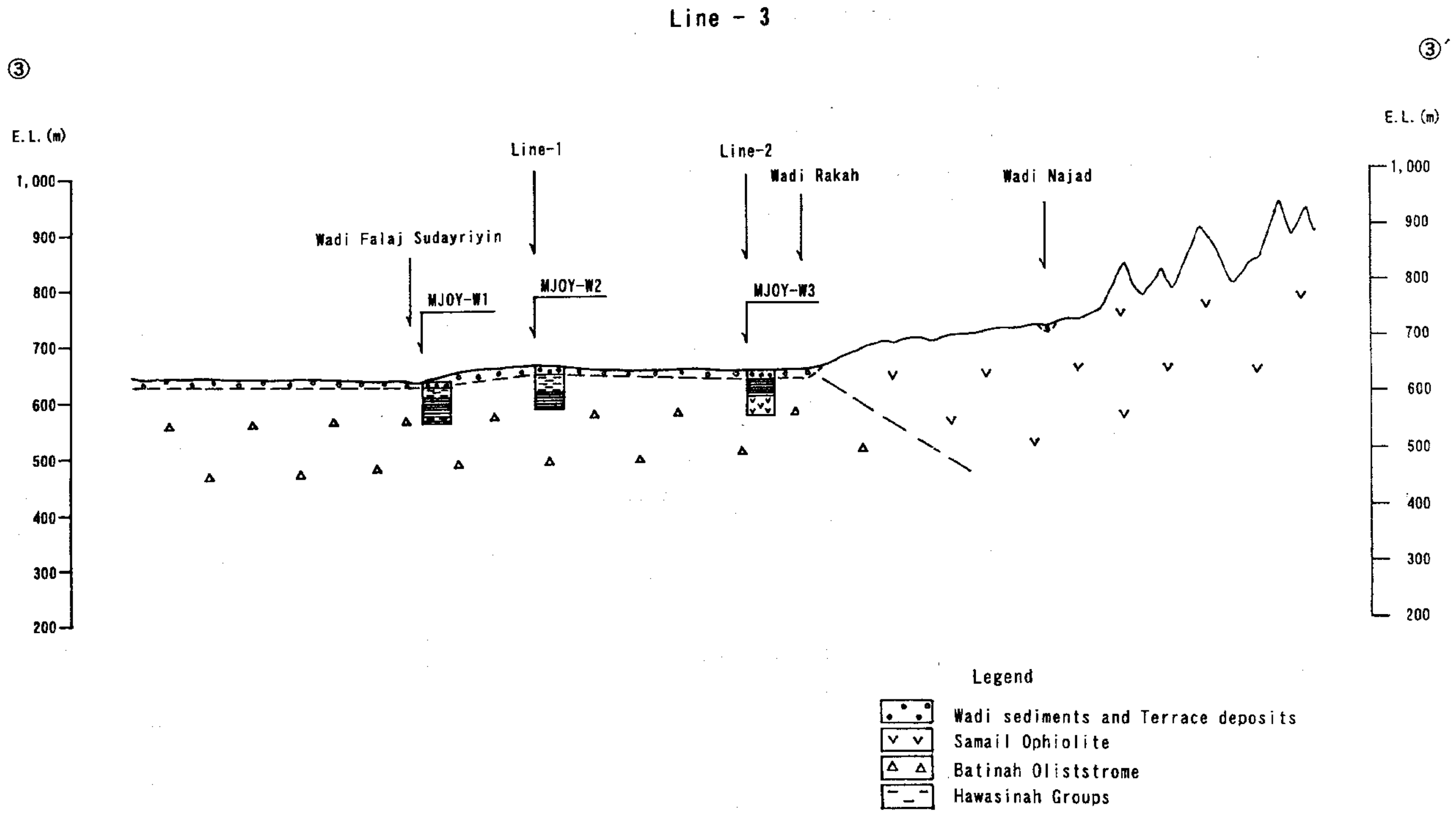


Fig. II-3-4(4) Geological section(4)

(2) MJOY-W2 borehole

0.00m~-16.00m	Consisting of unconsolidated wadi sediments with sand and gravels and consolidated terrace sediments.
-16.00m~-36.00m	Alternating beds of reddish brown chert and grey slate.
-30.00m~-43.00m	Reddish brown chert with interbeded grey slate.
-43.00m~-75.00m	Grey slate with minor amount of interbeded reddish brown chert.

(3) MJOY-W3 borehole

0.00m~-14.00m	Consisting of unconsolidated wadi sediments with sand and gravels and consolidated terrace sediments.
-14.00m~-35.00m	Grey slate.
-35.00m~-39.00m	Reddish brown.
-39.00m~-42.00m	Grey slate.
-42.00m~-75.00m	Reddish brown to grey basalt.

(4) MJOY-W4 borehole

0.00m~-18.00m	Consisting of unconsolidated wadi sediments with sand and gravels and consolidated terrace sediments.
-18.00m~-22.00m	Grey slate.
-22.00m~-35.50m	Basalt.
-35.50m~-54.00m	Reddish brown chert with minor amount of interbeded green basalt.
-54.00m~-75.00m	Reddish brown chert with interbeded grey slate.

(5) MJOY-W5 borehole

0.00m~-22.00m	Consisting of unconsolidated wadi sediments with sand and gravels and consolidated terrace sediments.
-22.00m~-75.00m	Reddish brown, dark grey and brownish grey basalt.

3-3-2 Groundwater

Groundwater was found in the time of drilling works at the following three points of MJOY-W3, MJOY-W4 and MJOY-W5. Since considerable amount of groundwater gushed out of bore hole of the MJOY-W5 at depth of 23m in drilling operation, it can be estimated that there exists confined groundwater. While, only small amount of groundwater was found at bore holes of MJOY-W3 and MJOY-W4.

Groundwater levels at each point are presented in Table II-3-4 and Figure II-3-3.

The groundwater levels distributed in a range of –9.09 ~-19.89m. These levels were identical levels of basement rocks at bore holes of MJOY-W1,W2,W3 and W5, however, only that of W4 located within the gravel bed.

Table II -3-4　Ground water level (2001/2/16)

Drill Hole No.	Depth (m)	Drill hole No.	Depth (m)
MJOY-W1	-16.87	MJOY-W4	-9.09
MJOY-W2	-19.73	MJOY-W5	-19.89
MJOY-W3	-16.87		

3-3-3 Results of pumping tests

Results of pumping tests are shown in Table II -3-5 and Appendix 7.

As result of calculation of permeability coefficient based on pumping test at each bore hole, permeability coefficient in aquifer of this area distributed in a range of 4.55E-5 ~ 1.66E-7cm.s and showed tendency of the higher value in the lower downstream.

Table II -3-5　Pumping test of groundwater

Drill Hole No.	Location	Groundwater Level (m)	Pumping Test	
			Pumping Volume (L/s)	Permeability Coefficient (cm/s)
MJOY-W1	Wadi Falaj Sudayriyin	-16.87	-	1.66 E-7
MJOY-W2	Wadi al Hayl al Ali	-19.73	-	3.23 E-6
MJOY-W3	Wadi Rakah	-16.87	0.4	1.47 E-6
MJOY-W4	Wadi Rakah	-9.09	1.5	2.92 E-5
MJOY-W5	Wadi Rakah	-19.89	5.0	4.55 E-5

3-3-4 Results of water quality analysis

(1) Characteristics of water quality

Results of water quality analysis are presented in Table II -3-6. The characteristics of water quality are described below.

Calcium (as $CaCO_3$) ranged in concentration of 17~216 mg/L, showed the highest value at MJOY-W3 and the groundwater in the western part of showed relatively lower concentration.

Sodium ranged in concentration of 34~164 mg/L, and showed the maximum value at MJOY-W3.

Bicarbonate ion distributed in range of concentration of 96~ 257 mg/L and showed the maximum value at MJOY-W4.

Sulfate ion ranged in concentration of 11~415 mg/L, and showed the highest value at MJOY-W3 which locates in downstream area of the Rakah deposit.

Nitrate ion distributed in range of concentration of 3.1~ 12.3 mg/L and showed the maximum value at MJOY-W1. Generally speaking, the ground water contains 0.1 ~ 1 mg/L of nitrate approximately, however, all bore holes of this study showed relatively higher values.

PH values at 25°C distributed in 7.6 ~ 7.76 of neutral range with almost stable state.

Table Ⅱ-3-6 Results of water quality analysis

Items	Unit	MJOY-W1	MJOY-W2	MJOY-W3	MJOY-W4	MJOY-W5
Total dissolved solids	mg/L	449	207	1200	460	504
Total alkalinity as CaCO3	mg/L	98	98	79	211	127
Calcium hardness as CaCO3	mg/L	65	43	540	105	133
Magnesium hardness as CaCO3	mg/L	49	70	41	148	95
Total hardness	mg/L	114	113	581	253	228
Calcium	mg/L	26	17	216	42	53
Magnesium	mg/L	12	17	10	36	23
Sodium	mg/L	110	34	164	70	120
Potassium	mg/L	10	3.4	4.8	6.6	6.1
Carbonate	mg/L	1	1	1	1	1
Bi-Carbonate	mg/L	120	120	96	257	165
Sulphate	mg/L	77	11	415	73	31
Chloride	mg/L	125	23	305	82	174
Nitrate	mg/L	12.3	3.1	6.6	10.5	8.4
PH Value @ 25℃		7.76	7.60	7.62	7.67	7.68
Electric conductivity @ 25℃	μ S/cm	816	378	1949	814	911

(2) Classification of the water quality

Hexagonal-diagram of the ground water quality in each bore hole is presented in Fig. Ⅱ-3-5, including bore holes (GW-1and GW-2) in Ghuzayn area for reference.

The groundwater in the study area can be classified into 3 groups as the following. (Fig. Ⅱ-3-6)

Group 1 : MJOY-W3

Group 2 : MJOY-W2, MJOY-W4 ≒GW-1, GW-2

Group 3 : MJOY-W1, MJOY-W5

In the Group 1, ground water of MJOY-W3 is classified here, indicating characteristics of high

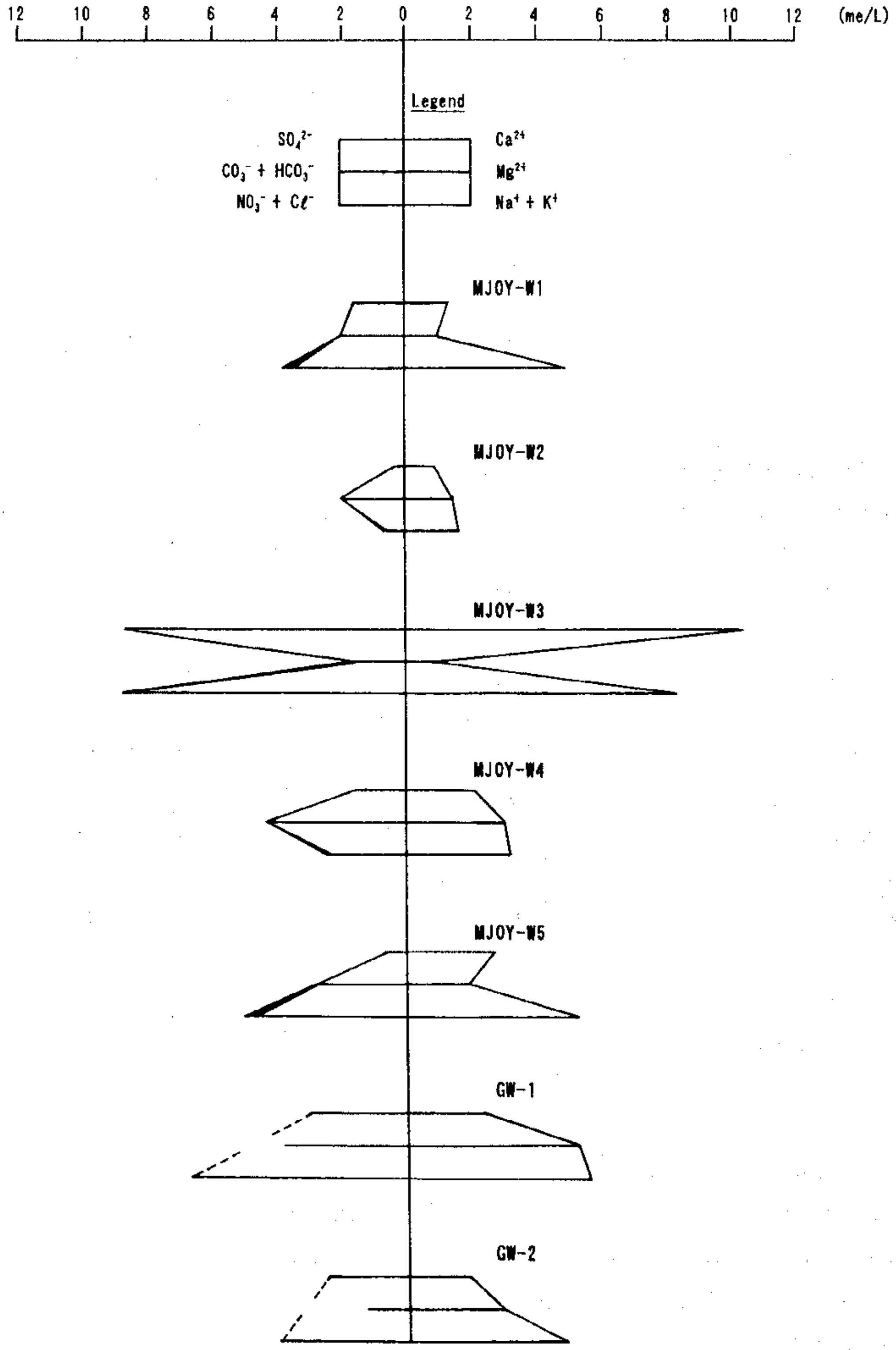


Fig. II-3-5 Hexa-diagram of groundwater

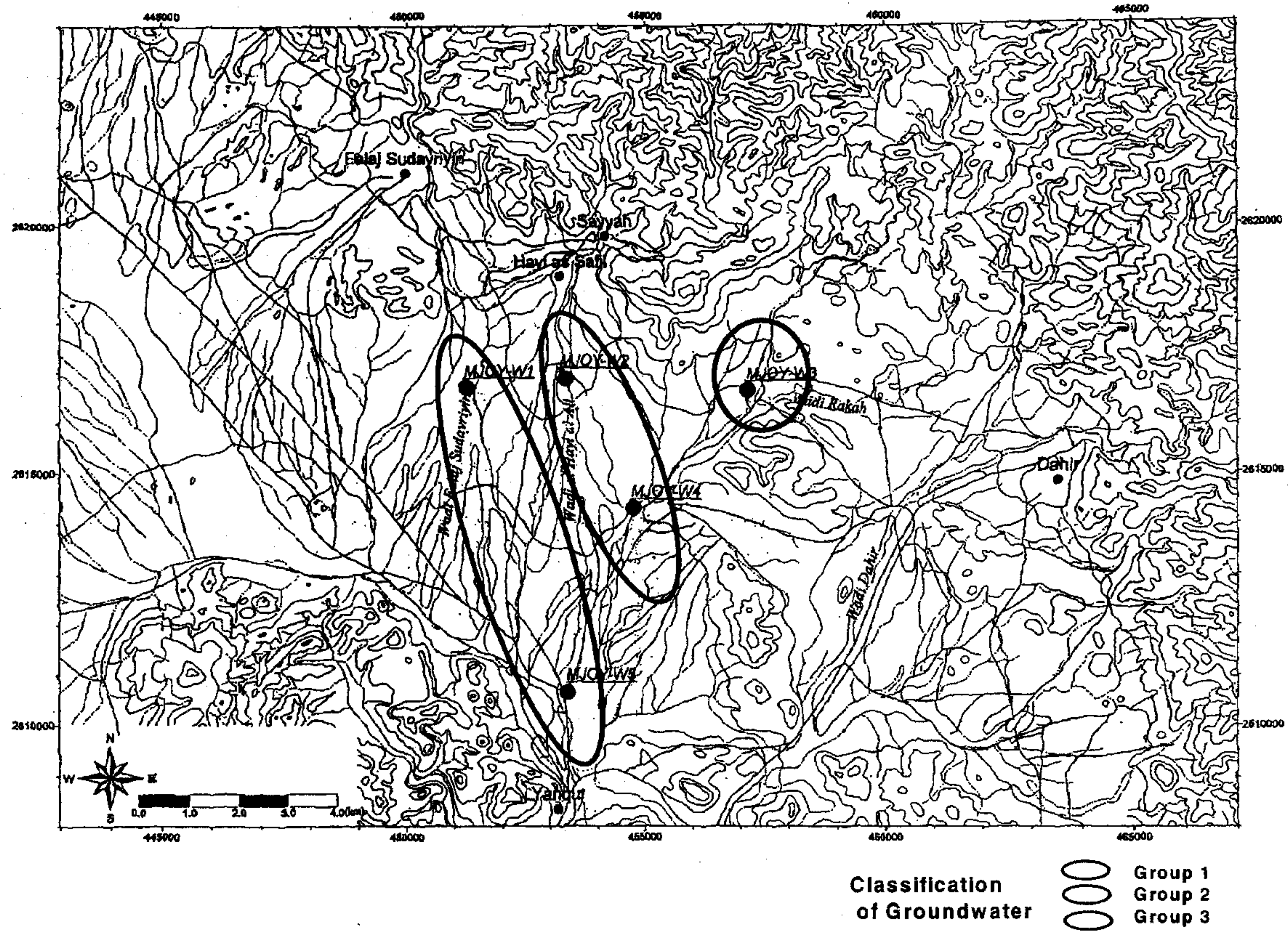


Fig. II-3-6 Classification of groundwater

contents of calcium and sulfate ions (gypsum) and also sodium and chlorine ions (salt).

Group 2 consists of the groundwater of MJOY-W2 and MJOY-W4,showing relatively higher contents of bicarbonate ion. TDS in MJOY-W4 was little bit higher than that of MJOY-W1. To belong to same group can be substantiated topographically by a fact that the Wadi al Hayl ali and the Wadi Rakah join at near points of MJOY-W4. Characteristics of water quality of the Group 1 resemble to those of well water –1 and GW-2 in Ghuzayn area.

Group 3 consists of the groundwater of MJOY-W1 and MJOY-W5,showing higher contents of sodium and chlorine ions. Coincidence of water qualities of both bore holes can be indicated topographically by a fact that the Wādi Falaj Sudayriyin the Wadi Rakah join at near points of MJOY-W5.

(3) Water quality and quantity of the groundwater

It is presumed that the groundwater of Group 1 is detained in near area of MJOY-W3 and the Rakah deposit in the upstream of the Wadi Rakah and its quantity is poor.

The groundwater in the Group 2 belongs to that of along the Wadi al Hayl ali and coincides with the groundwater in middle stream of the Wadi Rakah. Hence, quantity of the groundwater around MJOY-W4 flowed in from upstream area of the Wadi Rakah may be estimated to be very little.

The groundwater in the Group 3 belongs to that of along the Wadi Falaj Sudayriyin its water quality coincides with the groundwater in down stream of the Wadi Rakah. This fact may suggest that quantity of the Wadi Falaj Sudayriyin exceeds that along the Wadi al Hayl ali.

Based on the fact that spring water volume at MJOY-W5 was 5 L/sec (18 m^3/h) and confined, quantity of the groundwater here may be presumed to be relatively abundant.

CHAPTER 4 EXISTING DATA ANALYSIS

4-1 Objectives

Previous data acquired by Omani government projects and JICA/MMAJ projects, was collected and analyzed.

4-2 Data of Previous Projects and Studies

Data of projects and studies conducted previously are shown in Table II-4-1.

4-2-1 Review of previous feasibility study

(1) Geological and mine audit of the Hayl as Safil and Rakah copper deposits by Robertson Group plc (1991)

Robertson Group plc carried out the audit the OMCO's geological ore reserve calculation, minable ore reserve calculation, mining schedule, capital cost for mine development and operating cost, based on the available data as of June 7, 1991. Report on the mineral exploration project at the Rakah Area in 1988/9 by JICA /MMAJ is included among those available data. Robertson made clear excellent correlation between specific gravity of the ore and Fe contents, but with poor correlation between specific gravity and copper grade. Robertson proposed the geological ore reserve calculation adding the result of infill drilling. Audit did not cover the essential area of mine operation such as ore processing and environmental assessment. Preliminary financial analysis based on the wide range of assumption concluded that the recommended cut-off copper grade is 0.4%, when break even copper price resulted in U.S.$1.12/lb. Robertson recommended that systematic exploration of this area should be carried out in order to improve the viability of the mine development.

In spite of the description that Robertson carried out the extensive metallurgical test and feasibility study of the whole project, these data are not presently available.

(2) Engineering and capital cost estimate for Rakah copper project feasibility study by Minproc Engineers Limited (1994)

This is a feasibility study on process facility and related infrastructure for Rakah copper project and was completed in February 1994. The objective and approach of the study were to undertake preliminary design of the process facilities to generate a ±15% capital cost by utilizing the existing equipment from Sohar Copper Concentrator after the expected closure of the Sohar mines in late 1994. The project aimed at minimizing the capital cost without jeopardizing the metallurgical or operational performances.

Table II-4-1 List of existing data

	Title of Reports on each field	Prepared by	Date
Feasibility Study			
1	Feasibility for the Development of a Copper Mine and Recovery Plant at Hayl as Safil and Rakah (Near Yanqul) in the Sultanete of Oman	Oman Mining Company LLC	Mar-94
2	Rakah Copper Project Environmental Impact Assessment Draft	Oman Mining Company LLC	May-94
3	Rakah Copper Project Revised Feasibility Study	Minproc Engineers Limited	Jun-95
4	Geological and Mine Audit of the Hayl as Safil and Rakah Copper Deposits, Sultanate of Oman	The Robertson Group plc	Sep-91
Exploration			
1	Core log sheets of Rakah body	Oman Mining Company LLC	-
2	Core log sheets of Hayl as Safil body	Oman Mining Company LLC	-
3	Core log sheets of Bishara body	Oman Mining Company LLC	-
4	Core log sheets of Al Jadeed body	Oman Mining Company LLC	-
5	Core log sheets of Al Ashgar body	Oman Mining Company LLC	-
6	Report on The Application of mechanised mining techniques to the exploitation of Copper ore reserves in Oman	OMCO Sohar Copper Project	Apr-93
Water Resource			
1	Al Masarrat Water Supply Scheme	Ministry of Water Resources	-
2	Groundwater Resource Evaluation Upper Wadi Dank Catchment	Ministry of Water Resources	Jun-93
3	Detailed Hydrotechnical Recharge Studies in Wadi Al Aridh and Wadi Yanqul	Ministry of Water Resources	Oct-93
4	Groundwater Resource Evaluation for a Proposed Mine near Yanqul	Ministry of Water Resources	Jan-92
Environment			
1	Guidelines for Obtaining Environmental Permits	Directorate General of Environmental Affairs	-
2	Ministerial Dicision 79/94 Issuing Regulation for Noise Pollution Control in Public Environment	Ministry of Regional Municipalities and Environment	Jan-00
3	Ministerial Dicision 17/93 Regulation for the Management of Solid Non-Hazardous Waste	Ministry of Regional Municipalities and Environment	Feb-93
4	Ministerial Dicision 18/93 Regulation for the Management of Hazardous Waste	Ministry of Regional Municipalities and Environment	Feb-93
5	Ministerial Dicision 248/97 Regulation for the Registration of Hazardous Chemical Substances and the Relevant Permits	Ministry of Regional Municipalities and Environment	Jul-97
6	Interim Guideline on Environmental Impact Assessment		Jun-99
Others			
1	Daily Rainfall Record at Al Falaj Sudayriyn 1992-2000	Ministry of Water Resources	-
2	Daily Rainfall Record at Al Rudhah, Al Dhahirah-W.Yanqul Region 1993-2000	Ministry of Water Resources	-
3	Daily Rainfall Record at Yanqul 1994-2000	Ministry of Water Resources	-

This study included the implementation and review of existing metallurgical test work, carrying out additional metallurgical test work required, inspection of Sohar and Rakah project sites, and local Omani fabrication and supply facilities relevant to the construction of copper concentrator plant, establish the process design criteria, development of flow sheet, preliminary P & ID and plant layout and design of relevant electrical design.

Operation scale of this study is assumed to be 3,000tpd sourcing from four deposits not including Bishara orebody. Sub-aerial wet tailing dam method was adopted in this study.

(3) Feasibility for the development of a copper mine and recovery plant at Hayl as Safil and Rakah by Oman Mining Company LLC (1994)

This study was completed in March 1994 based on the previous studies to make preparations for the expected closure of Sohar Copper Project. Operation scale was 3,000tpd, sourcing from five orebodies. Mineable ore reserve was expected to be 11,600 thousand tones (0.93% Cu, 0.40 g/t Au). Open pit mining method was adopted followed by the underground mining method, where applicable.

Environmental Impact assessment was carried out in May 1995.

(4) Revised feasibility study on Rakah copper project by Minproc Engineers Ltd.

Due to the fact that OMCO's Feasibility Study proved to be uneconomical, an operation scale of 1,000 tpd process facilities was subsequently studied. This revision indicated that the profitability of the project could be improved and easy to implement through decreasing the capital cost by utilizing the Sohar equipment to maximum extent and through increasing the flexibility of operation. Dry dam method was employed in Tailings dam.

(5) Report on cooperative exploration at the Rakah area (follow up study) in 1997/8 by JICA/MMAJ

Subsequent to the Cooperative Exploration at the Rakah area in 1988/89, a Preliminary Feasibility Study was carried out by implementing and reviewing the existing data acquired by OMCO and Directorate General of Minerals, Ministry of Commerce and Industry. Additional studies required for further improvement were recommended.

4-3 Result of Review and Abstracted Subject

The result of review of the above mentioned studies and subject to further investigations in this study is described below.

4-3-1 Geological ore reserve calculations

Geological ore reserve calculations were carried out by JICA/MMAJ (1990, 1998), Robertson Group (1991) and OMCO (1992, 1994, 1995). The comparisons of these results are shown in Table II -4-2.

Table II-4-2 Comparison of geologic ore reserves of each ore body

Ore Body	Company and Date	Tonne	%Cu	g/tAu	ton Cu	Kg Au
Rakah	Robertson Group (Jun.1991)	3,760,090	0.90	0.40	32,995	1,384
	Minproc Engineers Ltd. (Mar.1994)	4,357,178	0.88	0.30	37,385	1,203
	Oman Mining Company (Aug.1994)	2,836,000	1.25	-	34,564	-
	Oman Mining Company (Jan.1995)	2,392,363	1.24	0.99	28,987	2,314
	JICA/MMAJ (1998)	3,679,000	1.09	2.42	39,099	8,191
Hayl As Safil	Robertson Group (Jun.1991)	7,046,720	1.03	0.30	70,767	1,945
	Minproc Engineers Ltd. (Mar.1994)	6,480,634	0.91	0.32	57,499	1,908
	Oman Mining Company (Aug.1994)	6,284,000	1.30	-	79,650	-
	Oman Mining Company (Jan.1995)	4,748,443	1.37	0.47	63,460	2,160
	JICA/MMAJ (1998)	6,100,000	1.37	0.60	81,481	3,367
Al Jadeed	Robertson Group (Jun.1991)	-	-	-	-	-
	Minproc Engineers Ltd. (Mar.1994)	445,766	1.51	0.92	6,563	377
	Oman Mining Company (Aug.1994)	809,946	1.29	-	10,187	-
	Oman Mining Company (Jan.1995)	660,442	1.38	0.85	8,912	547
	JICA/MMAJ (1998)	907,000	1.17	0.39	10,347	325
Al Asghar	Robertson Group (Jun.1991)	-	-	-	-	-
	Minproc Engineers Ltd. (Mar.1994)	379,817	2.36	1.37	8,740	479
	Oman Mining Company (Aug.1994)	946,947	2.67	-	24,651	-
	Oman Mining Company (Jan.1995)	893,679	2.48	1.22	21,531	1,063
	JICA/MMAJ (1998)	1,742,000	2.23	0.90	37,875	1,442
Al Bishara	Robertson Group (Jun.1991)	-	-	-	-	-
	Minproc Engineers Ltd. (Mar.1994)	1,133,400	1.00	0.35	11,051	365
	Oman Mining Company (Aug.1994)	2,024,500	1.19	-	23,489	-
	Oman Mining Company (Jan.1995)	2,116,259	1.32	0.74	27,227	1,518
	JICA/MMAJ (1998)	3,697,000	0.98	1.83	35,325	6,224
Total	Robertson Group (Jun.1991)	10,806,810	1.93	0.70	103,761	3,329
	Minproc Engineers Ltd. (Mar.1994)	12,796,795	0.95	0.37	121,237	4,331
	Oman Mining Company (Aug.1994)	12,901,393	1.34	-	172,541	-
	Oman Mining Company (Jan.1995)	10,811,186	1.39	0.76	150,117	7,602
	JICA/MMAJ (1998)	16,125,000	1.30	1.32	204,126	19,550

* Robertson G. The calculation was conducted only about Rakah and Hayl as Safil bodies.

Type of ore located in each ore body consists of primary mineralized sulfide ore (siliceous ore, massive ore, brecciated ore and stock work ore), gossan and secondary enriched ore. All of the existing ore reserve calculations treated the different orebodies in the same category without considering the difference in ore type. To this it can be added that the classification of ore itself seems to be inconsistent with the results of the drill core log records. Eminent difference of specific gravity and recovery rate in processing plant is expected among these types of ores. However, in the previous study the ore reserve calculation did not take into account the difference of ore type.

Moreover, the ore grade of each ore type was not considered for the interpolation analysis in each ore body and due to this reason, the previous calculations seem to have failed in applying statistical method. It seems necessary to check whether statistical method can be applied by considering the domain of each ore type.

As for specific gravity of ore and host rock, it is closely correlated with the iron contents as mentioned by Robertson Group. However, considering the distribution of Fe in the ore body, it will be practical to apply the average specific gravity for each ore body.

4-3-2 Geo-technical characteristics of the rock

Robertson Group's audit report mentioned that necessary data to design open pit is satisfactory and that through the experience of development planning and actual operation in Sohar copper project, these data are reflected properly in the open pit design. As it passed a long time since the closure of Sohar copper mining, the data are not fully available. But as far as inspection of old mine sites of Aarja (open pit) and Lasail and Bayda (underground mine), the design criteria of pits by OMCO are considered to be reliable.

4-3-3 Mineral processing

So far the metallurgical performances have been predicted on the bases of the two flotation test series. One series of tests was conducted at the Central Laboratory of Mitsubishi Metal Mining Corporation during the JICA/MMAJ pre-feasibility study (1988) and another was tested at the laboratory of Amdel Limited when Minproc Engineers Limited prepared the feasibility study (1994). Judging from these test results, though the feed sample ores for the two test series were not the same, there seemed to be two groups of copper minerals in the ores of the deposits that presented different flotation characteristics. The first group of the copper minerals presented sharp selectivity by flotation separation and the other, the second group, presented difficulties during the separation by flotation due to the fine combination with pyrite minerals.

In the past predictions, the copper recovery rates of the first group were well forecasted to be as from 70 to 80% based on the actual flotation test results. However, the test series which were conducted on the second group at the above mentioned two laboratories were considered insufficient and therefore, the predicted metallurgical performance figures inferred from these test results are judged not fully reliable.

In this study, the most careful attention should be focused on the ore group of difficult separation and it needs to conduct additional laboratory tests on these ores to get more reliable predicted performances, as the profitability of this project depends much on the flotation performances of these ore group. Gold recovery rate of the ores has been forecasted as around 40% based on the previous test results, assuming that gold is recovered as a by-product of the copper concentrates. But few investigations on gold mineralogy in the ores have been done so far. More research and test focused on the increase of gold recovery of the ores are recommended to improve the finance of this project.

4-3-4 Tailings dam

Many tailings dam methods have been investigated, namely conventional wet dam, semi-aerial dam lined with clay or synthetic resin at the bottom or side wall of the dam, dry dam or wet dam utilized mine out pit hole. But these investigations are not detailed and considered insufficient from the point of the actual dam construction. The tailings dam issue of this project is a crucial one and needs to investigate further, as the financial position of the project depends much on this issue.

4-3-5 Industrial water

In any existing feasibility studies, process water is to be recovered as much as possible and to be recycled. Deficiency in water is to be made by pumping underground water. These studies were based on an extensive investigation carried out by Ministry of Water Resources, Sultanate of Oman. (Refer to Ground water Resource Evaluation for a Proposed Mine near Yanqul, 1992). However, the Ministry of Water Resources announced that underground water from within the water shed of Dank-Fida can not be supplied to the copper mine development project, considering the increasing water demand due to the future economical development of the area and also due to some considerations presented by the local dwellers of the area.

For these reasons, supply of different water sources should be taken into consideration.

4-4 Ore Reserve Calculation

The ore reserve calculation and metallurgical tests were studied in this phase. The metallurgical tests were mentioned in the chapter 2.

The ore reserve calculation is the fundamental study for F/S and therefore it is necessary that the calculation should be conducted as accurate as possible in order to plan the mineral processing and mining schedule. To achieve the above requirements, it is necessary to confirm an accurate reserve and distribution of each ore type.

4-4-1 Preparation of data

Geologic profiles with showing the distribution of each ore type were prepared based on the core log sheets of previous drilling work. Ore types classified and those distribution are as follows;

Massive ore: Al Ashgar ore body, part of Rakah and Hayl as Safil ore bodies.

Breccia ore: Bishara and Al Jadeed ore bodies (grade into a massive ore in places).

Stockwork ore: main part of Rakah and Hayl as Safil bodies.

The spacings of geologic profiles were decided in order to get the proper accuracy accuracy regarding the density of previous boreholes as follows:

20m spacing: Rakah and Al Jadeed ore bodies.

25m spacing: Bishara and Al Ashgar ore bodies

50m spacing: Hayl as Safil ore body

4-4-2 Software used and parameter for calculation

Minex of Australian ECSI Pty Ltd was selected as the software for ore reserve calculation as well as the pit design. Several parameters for the calculation were fixed are shown in Table II -4-3.

Table II-4-3 Parameters for ore resources calculation

<table>
<tr><th>Parameter</th><th>Contents</th><th>Printed results</th><th>Contents</th></tr>
<tr><td>Specific Density</td><td>Determined by each ore type.</td><td rowspan="3">Wireframing</td><td rowspan="3">Including Cu, Au by domains,
Drill traces.</td></tr>
<tr><td>Assay below detection limit</td><td>Adopted a half of detection limit.</td></tr>
<tr><td>Top cutting</td><td>Cut and fix by thresholds at 95 % for raw Cu and Au assay data.</td></tr>
<tr><td>Composites</td><td>Rakah : 6m
Other than Rakah : 5m</td><td>Drill hole location (Plan)</td><td>Including geology and drill collars.</td></tr>
<tr><td>Block size</td><td>Rakah : 10m×10m×12m
Other than Rakah : 10m×10m×10m</td><td>Block grade plans</td><td>Cu, Au and CuEq by each bench.</td></tr>
<tr><td>Block grade interpolation</td><td>Ordinary Kriging.</td><td rowspan="2">Block grade sections</td><td rowspan="2">Cu, Au and CuEq.
Including drill traces.</td></tr>
<tr><td>Anisotropy</td><td>Vertical anisotropy should be considered.</td></tr>
<tr><td>Domains (Ore classified)</td><td>Stockwork, Massive, and Brecciated ores.</td><td>Birds eye view</td><td>Each deposit, including domains.</td></tr>
<tr><td>Cu equivalent grade factor</td><td>Determined based on metallurgical tests.</td><td>Block classified plans</td><td>Each bench.</td></tr>
<tr><td>Area for evaluations</td><td>Determined by each deposit.</td><td>Block classified sections</td><td>Each section.</td></tr>
<tr><td>Boundary</td><td>Simplified boundary.
Hard or soft boundaries determined by each domain.</td><td rowspan="3">Ore grades and tonnages</td><td rowspan="3">COG : 0, 0.2, 0.4, 0.5, 0.6, 0.7, 0.8, 0.9, 1.0%Cu
By each ore type classified and deposit.</td></tr>
<tr><td>Search radii</td><td>Determined after geostatistics evaluations.</td></tr>
<tr><td>Ore resources classification</td><td>Determined by composites number and search radii (measured, indicated, inferred)</td></tr>
</table>

CHAPTER 5 TDIP SURVEY

5-1 Objectives

The TDIP (Time Domain Induced Polarization) survey was carried out during this phase in order to extract mineralized zones related to the existence of massive sulphide deposits in the Yanqul region on the basis of the results of the geoscientifical surveys previously carried out.

5-2 Survey Locations and Specifications

During this fiscal year, the Yanqul region was selected for exploration work. This area, with its center around the Rakah Gold Mine, extends about 9km along E-W direction and 5km along N-S direction covering an area of about $20km^2$. The geophysical investigations within this area included also the Rakah Gold Mine, the Hayl as Safil deposit as well as Tawi Rakah area.

The TDIP survey carried out this year consisted of 44 lines with a total length of 95.9 km and 3220 measured points. Resistivity as well as chargeability values of rocks and core samples were measured according to the time domain IP procedures in the laboratory.

5-3 Survey Method

5-3-1 Procedure

The Induced Polarization survey was carried out by using a time-domain method (TDIP) and adopting a dipole-dipole electrode configuration with a separation factor from 1 to 4. IP data were taken along lines every 100m by keeping a potential dipole of 100m. In the survey area, the TDIP survey was carried out by injecting a current into the earth through current electrodes and the resulting voltage was measured across potential electrodes. Fig. II-5-1 shows the array utilized as well as the location of the plotting points.

For TDIP surveys, the current is turned on for a certain length of time (on-time) then turned off (off-time). The transmitted waveform is then repeated with current flow in opposite direction. The pair of positive and negative on-off waveforms constitutes a cycle, which in this survey lasted 8 seconds, as indicated in Fig. II-5-2. According to Fig. II-5-3, the polarization of the target creates a transient decay voltage and its corresponding changing response is observed in the received waveform.

5-3-2 Instrumentation

The instrumentation used for the conventional time-domain IP survey is described in the Table II-5-2.

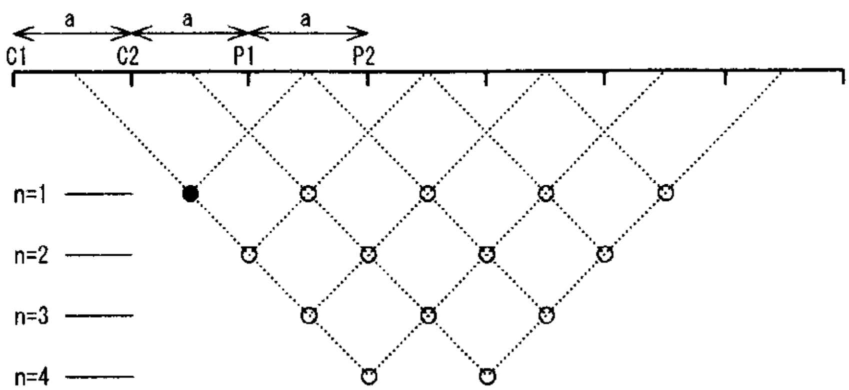


Fig. II-5-1 Dipole-dipole array and plotting procedure

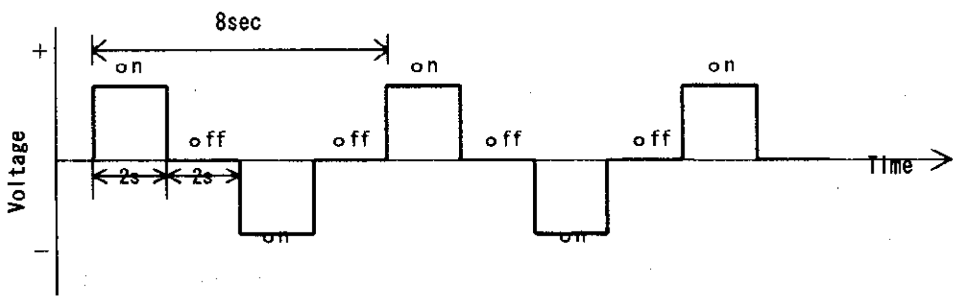


Fig. II-5-2 Waveform produced by the transmitter

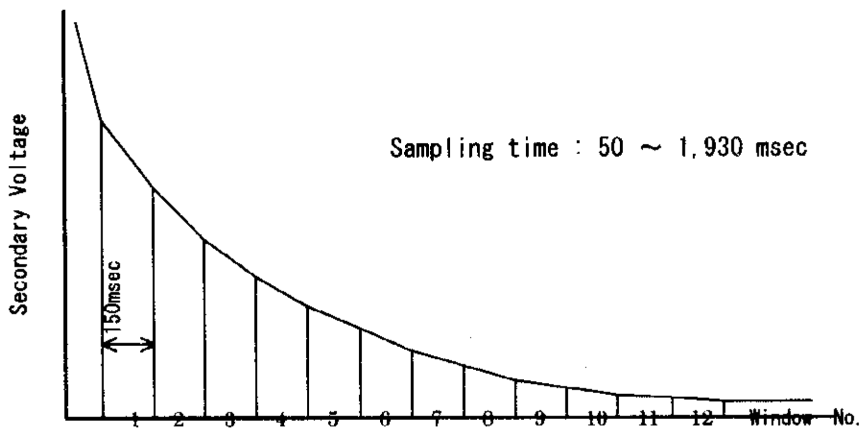


Fig. II-5-3 Sampling interval of the TDIP receiver

Table Ⅱ-5-1 Specifications of TDIP survey instruments

Receiver	**Zonge GDP-16**
Frequency range	DC to 8KHz
Number of Channels	8
Number of Stackings	8096
Detectable signal	1 μ V
A/D Conversion	16 bits
Number of Windows	13(from 50 to 1,930ms)
Transmitter	**CH-95A**
Output Power	2kw,800v,12A
Output Frequency	DC to 10KHz
Frequency control	Automatic
Generator	**Geonics GPU2000**
Maximum output	2Kw
Output Voltage	200V
Output Frequency	400Hz

5-4 Analysis Method

5-4-1 Data processing

The TDIP data processing involves the determination of 3 parameters, i.e., apparent resistivity, chargeability as well as metal factor. The first 2 parameters are calculated directly by the receiver unit during data acquisition. The third one is calculated as a simple relation between the first 2 parameters. These 3 parameters are calculated as follows:

a) Apparent resistivity (ρ)

$$\rho = K\frac{V_P}{I}$$

Where, $K = \pi an(n+1)(n+2)$, ***Vp*** is the received voltage in volts, ***a*** is the A-spacing in meters, ***n*** is the N-spacing and ***I*** is the transmitted current in Amperes.

b) Chargeability (M)

$$M = \frac{1.87}{V_P}\int_{t_1}^{t_2} V_t dt$$

Where, ***Vp*** is the primary voltage in volts and ***Vs*** is the secondary voltage in volts.
Here, the secondary voltage is calculated from 450msec. to 1,100msec.

c) Metal factor (MF)

$$MF = \frac{M}{\rho} \times 100$$

Where, M is the chargeability (mV/V) and ρ the apparent resistivity (Ωm)

5-4-2 Topographic corrections

Since the apparent resistivity is calculated here as a function of the location of the current and potential electrodes on a half-infinite plane, it is affected by topography depending on the location of the electrodes. For the case of a dipole-dipole configuration, the apparent resistivity appears to be high beneath a hill and low beneath a valley. On the other hand, the chargeability values are less affected by topography.

In order to make the appropriate corrections for the present survey, the topographic correction is calculated for each survey line by using 2D finite element method(FEM). The corrected apparent resistivity values are then used to construct the related sections and contour maps.

5-4-3 Two-dimensional analysis

The pseudo-sections do not show the real image of subsurface structure. In order to estimate the subsurface structure from TDIP data, we applied a 2D quantitative analysis method to the measured data, consisting of forward calculations using FEM and inversion calculations using non-linear least square method.

In order to make the model calculations, the subsurface structure is divided into many small blocks, each of them having initially assigned their own chargeability and resistivity value. The size of the blocks are relatively small at the shallower part, but large at the deeper part.

Theoretical values are calculated from the block models using FEM, and the parameter of each block is made to change until the difference between the theoretical value and measured value is sufficiently small.

5-5 Survey Results

The lines were all set up along N-S direction and with varied lengths from 1.1 km to 3.9 km, depending on the assumed distribution of the geological boundaries.

Fig. II-5-4 shows the location of all the IP lines surveyed, while the measured pseudo sections of apparent resistivity, chargeability and metal factor values are presented from Fig. II-5-5 to Fig. II-5-11. Plane maps of apparent resistivity, chargeability and metal factor from N=1 to 4 are presented from Fig. II-5-12 to Fig. II-5-15.

Apparent resistivity values ranged from several Ωm to about 1,300Ω-m with an average of about 160Ωm. To the east of line 200W, high resistivity values are generally seen in places where sheeted dikes are distributed with resistivity distribution about 500Ωm. To the west side of this line, low to medium resistivities are seen distributed, representing the existence of sedimentary rocks. Chargeabilities from 0 to 45mV/V are distributed with an average value of about 7.5mV/V. Five places with high chargeability distributions were confirmed as follows:

- Rakah Gold Mine and surroundings

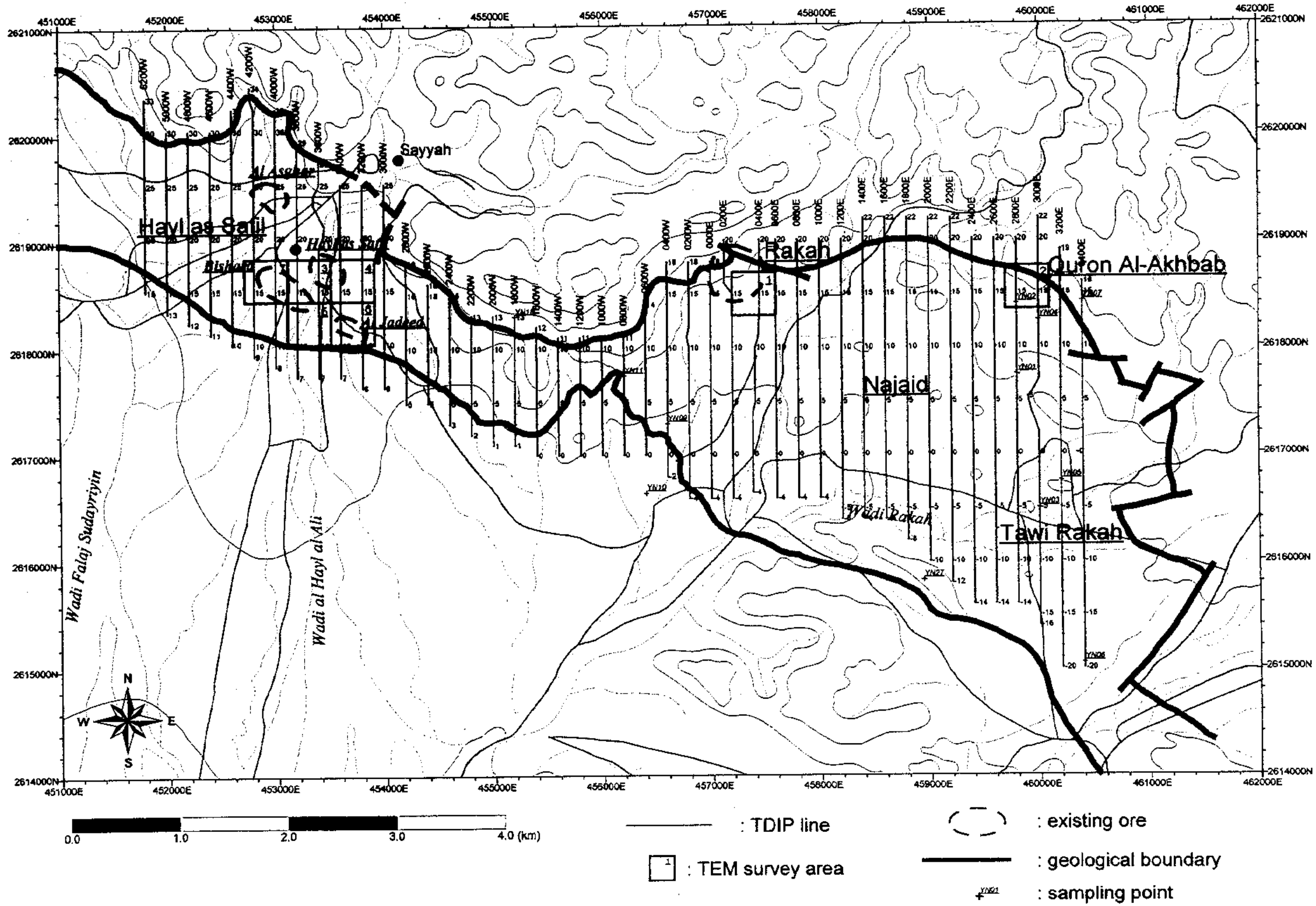


Fig. II -5-4 Geophysical survey location in Yanqul area

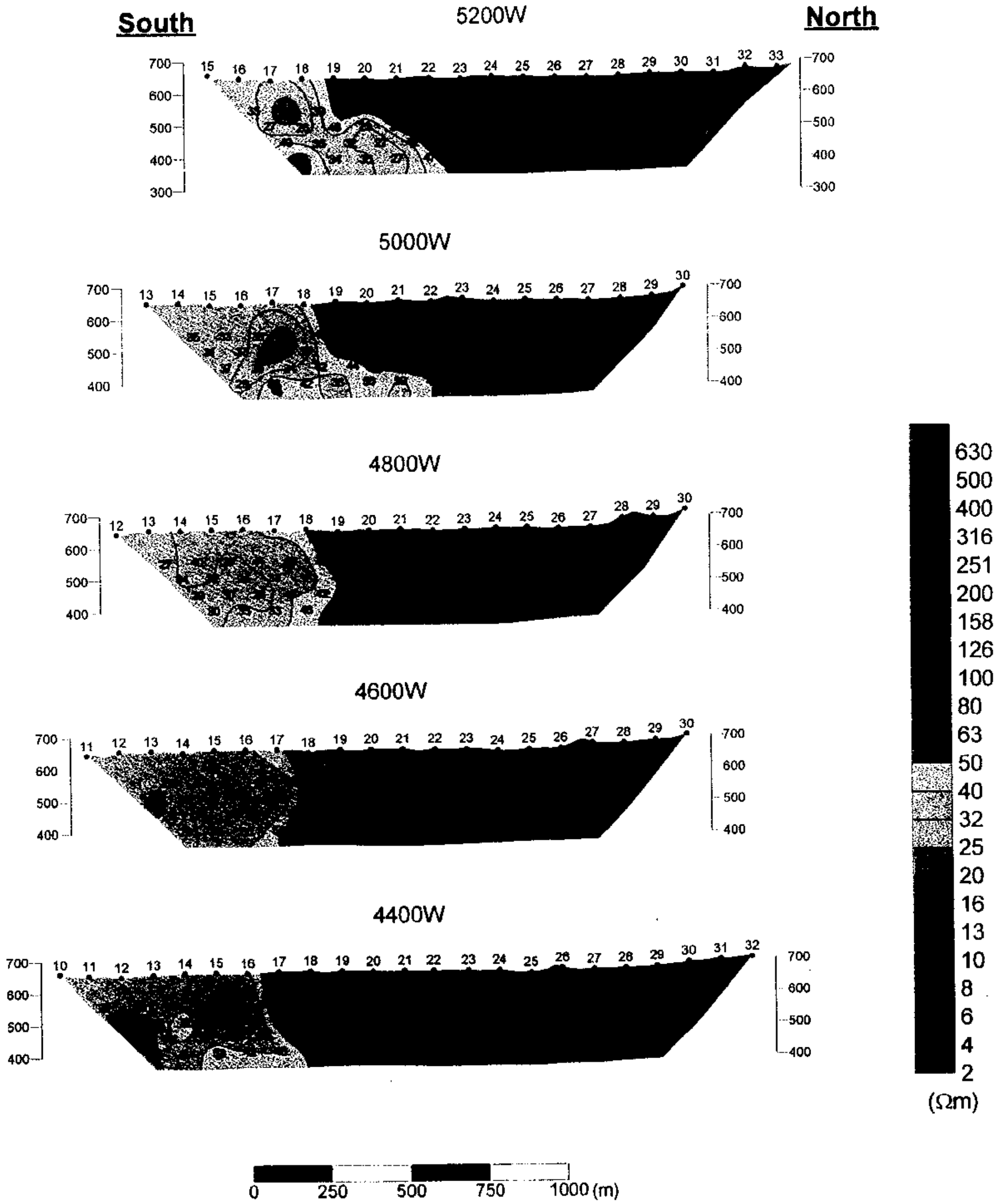


Fig. II-5-5(1) Apparent resistivity pseudo-sections(5200W-4400W)

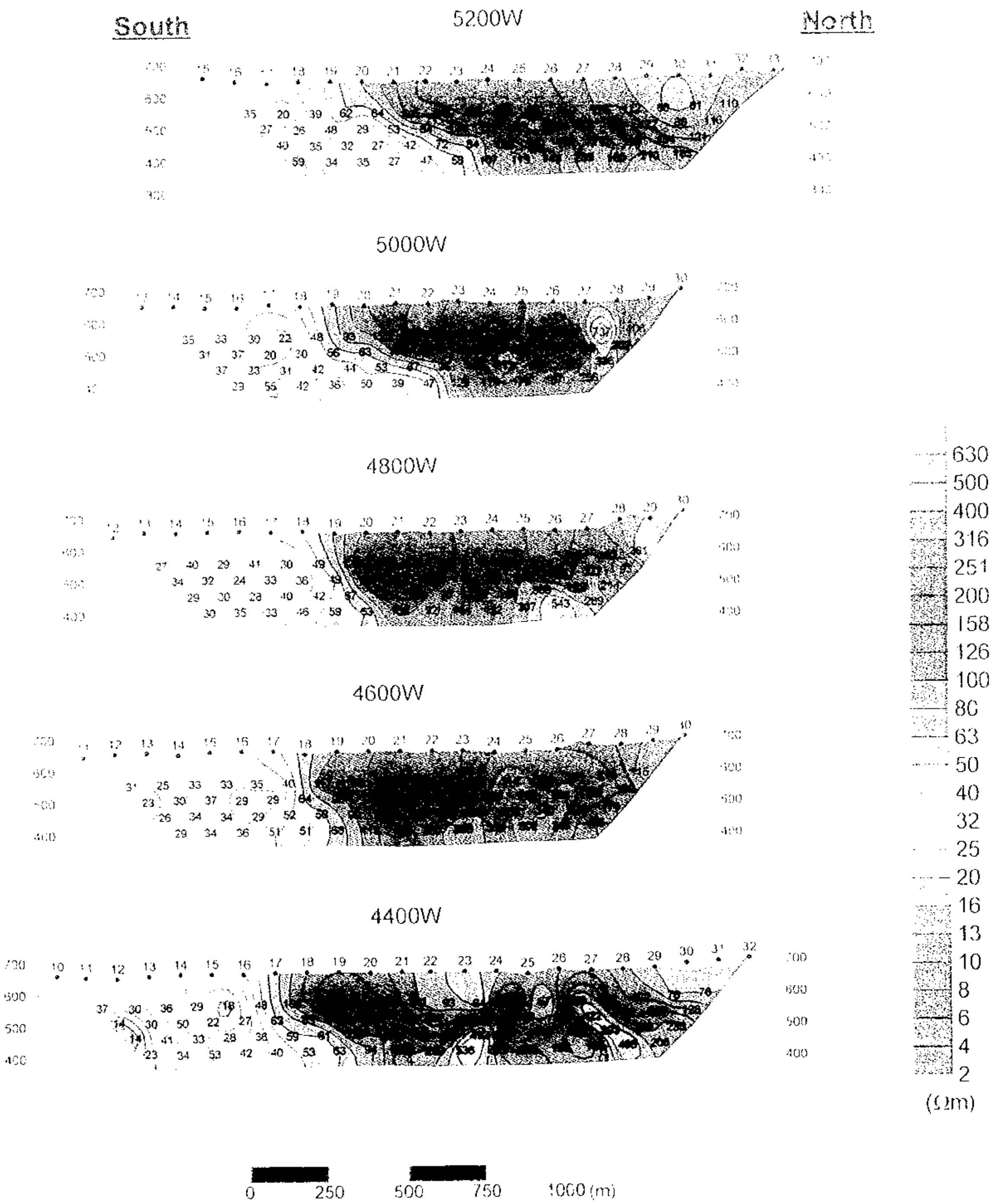


Fig. II -5-5(1) Apparent resistivity pseudo-sections(5200W-4400W)

Fig. II-5-5(1) Apparent resistivity pseudosections (5200W–4400W)

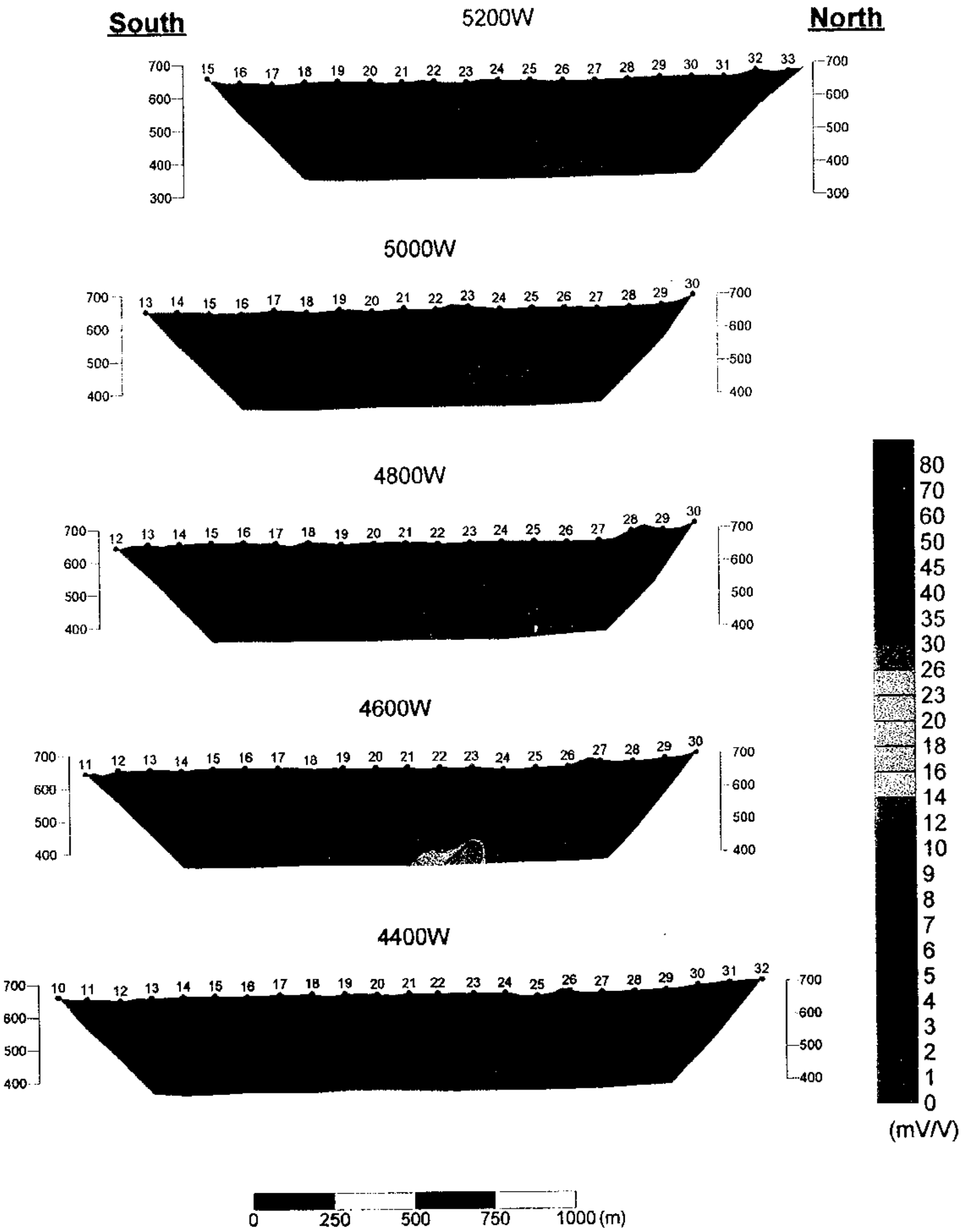


Fig. II-5-5(2) Chargeability pseudo-sections(5200W-4400W)

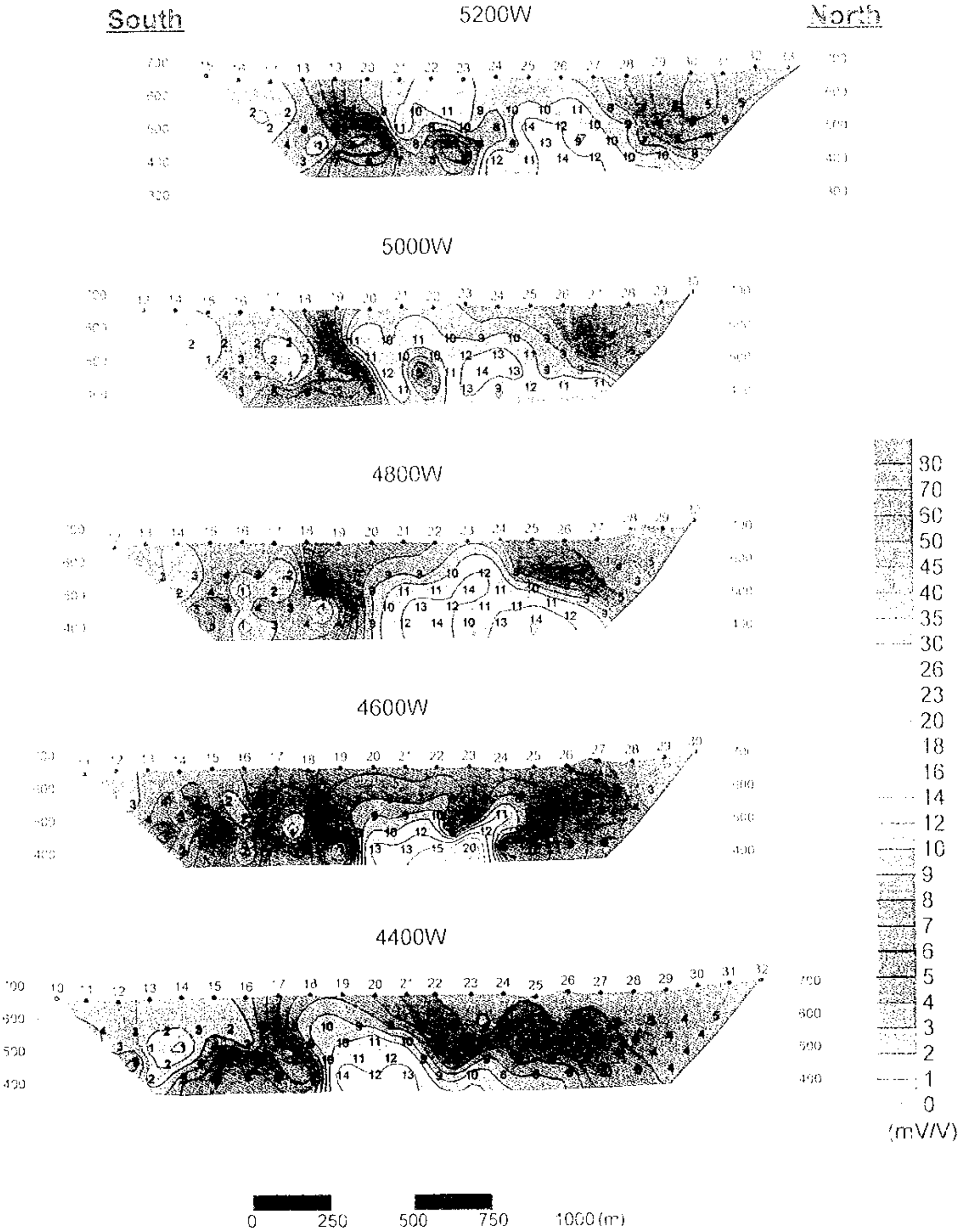


Fig. II-5-5(2) Chargeability pseudo-sections(5200W-4400W)

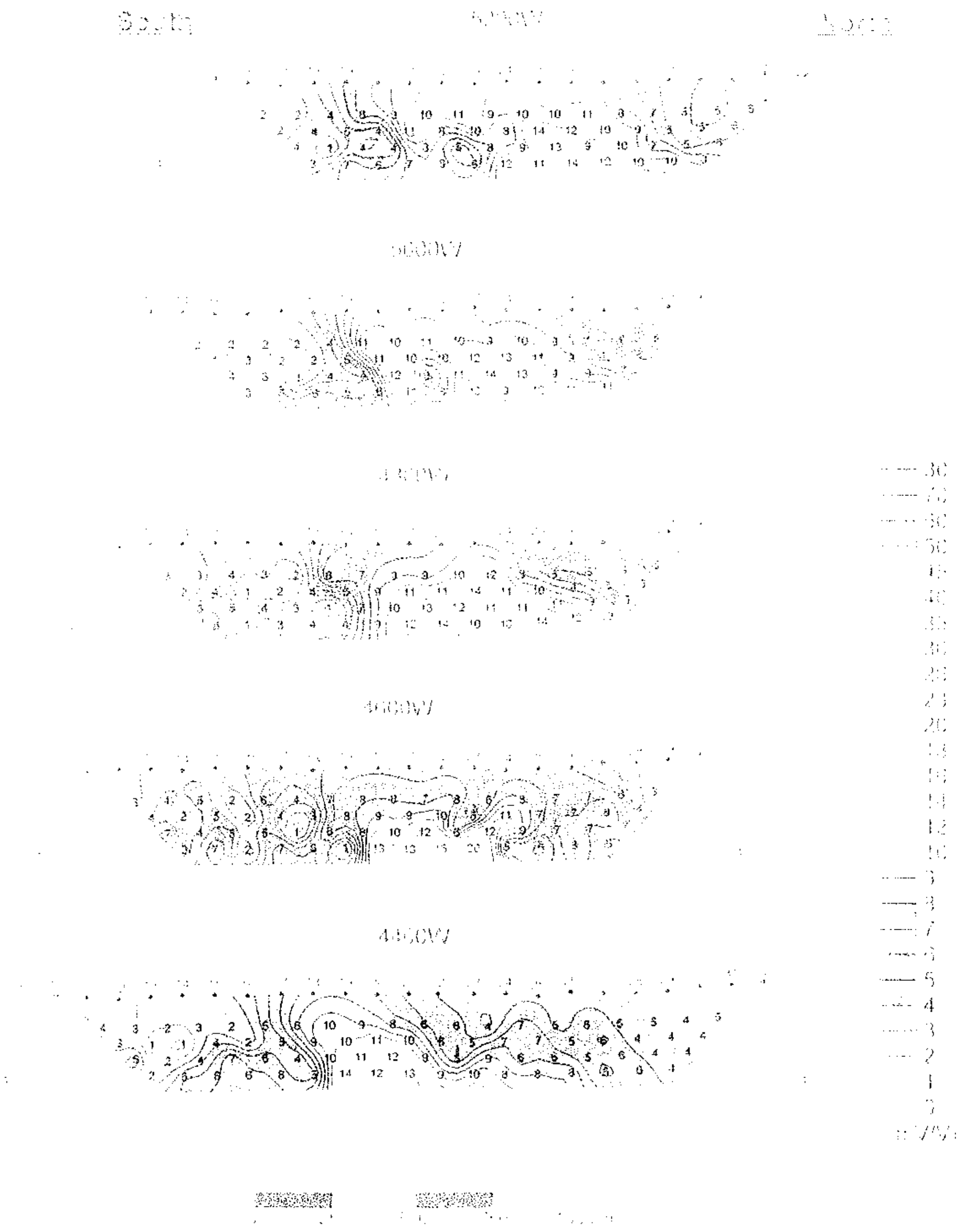


Fig. II-8-8-2 Chargeability pseudo sections (5200W - 4400W)

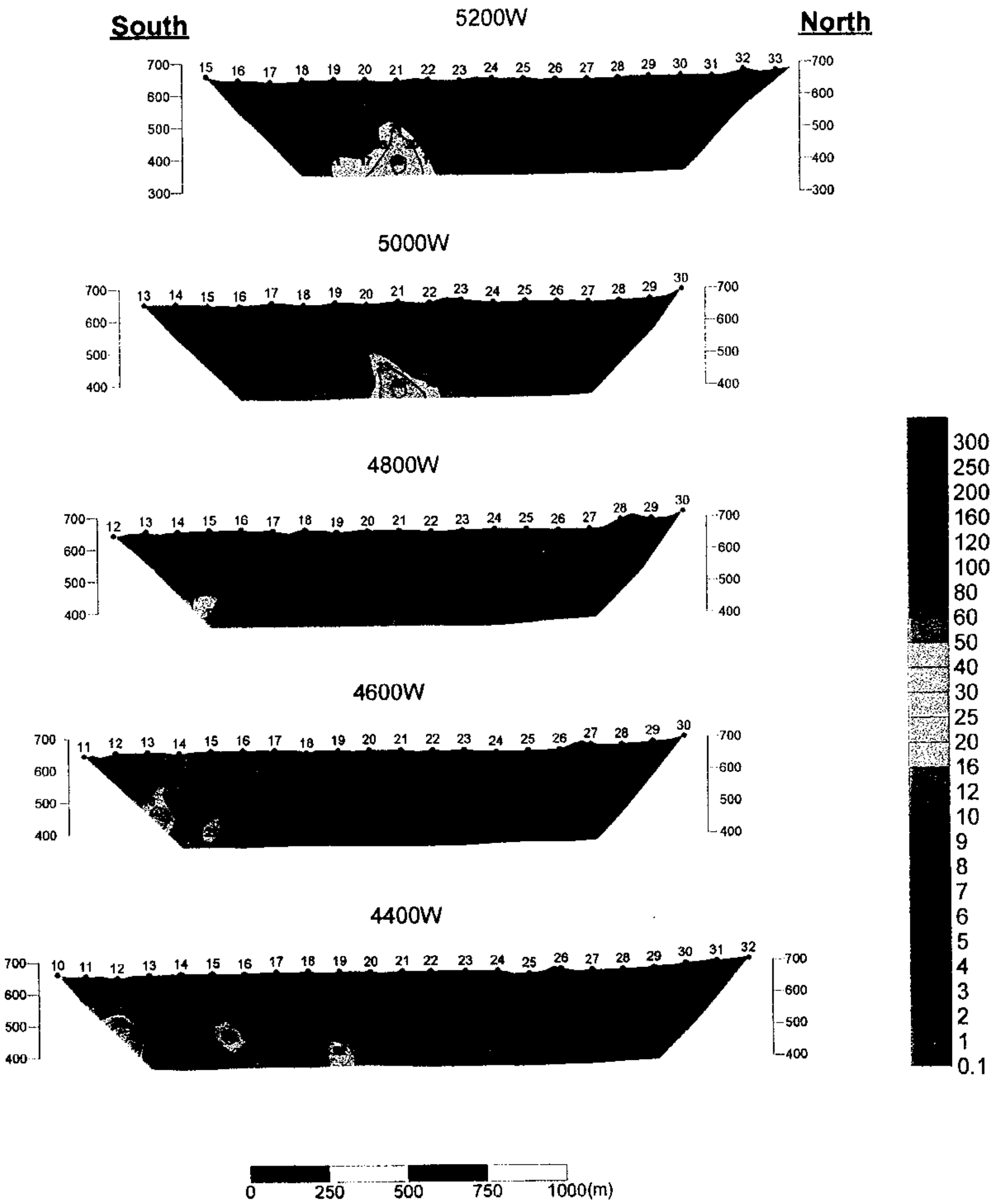


Fig. II-5-5(3) Metal factor pseudo-sections(5200W-4400W)

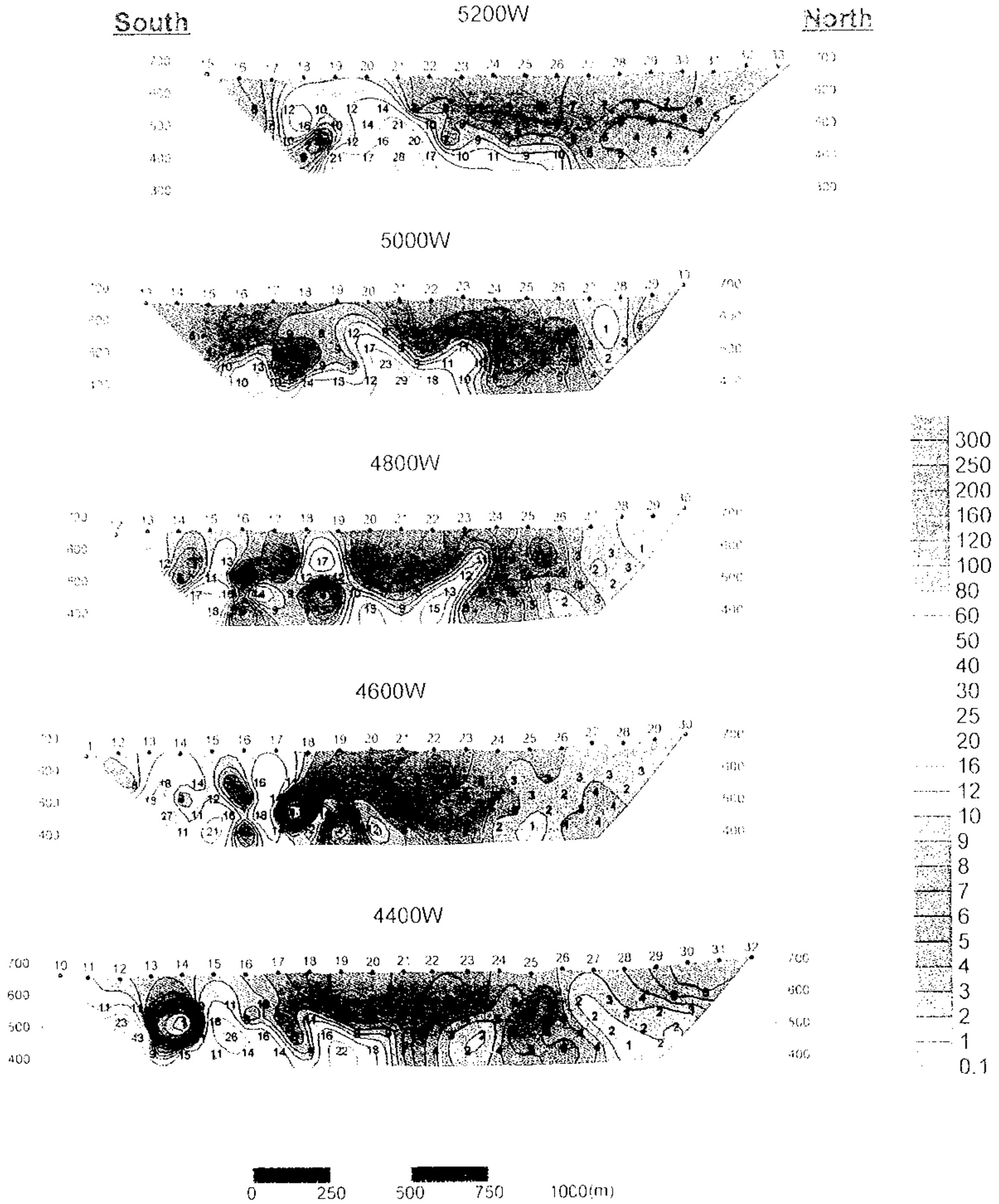


Fig. II-5-5(3) Metal factor pseudo-sections(5200W-4400W)

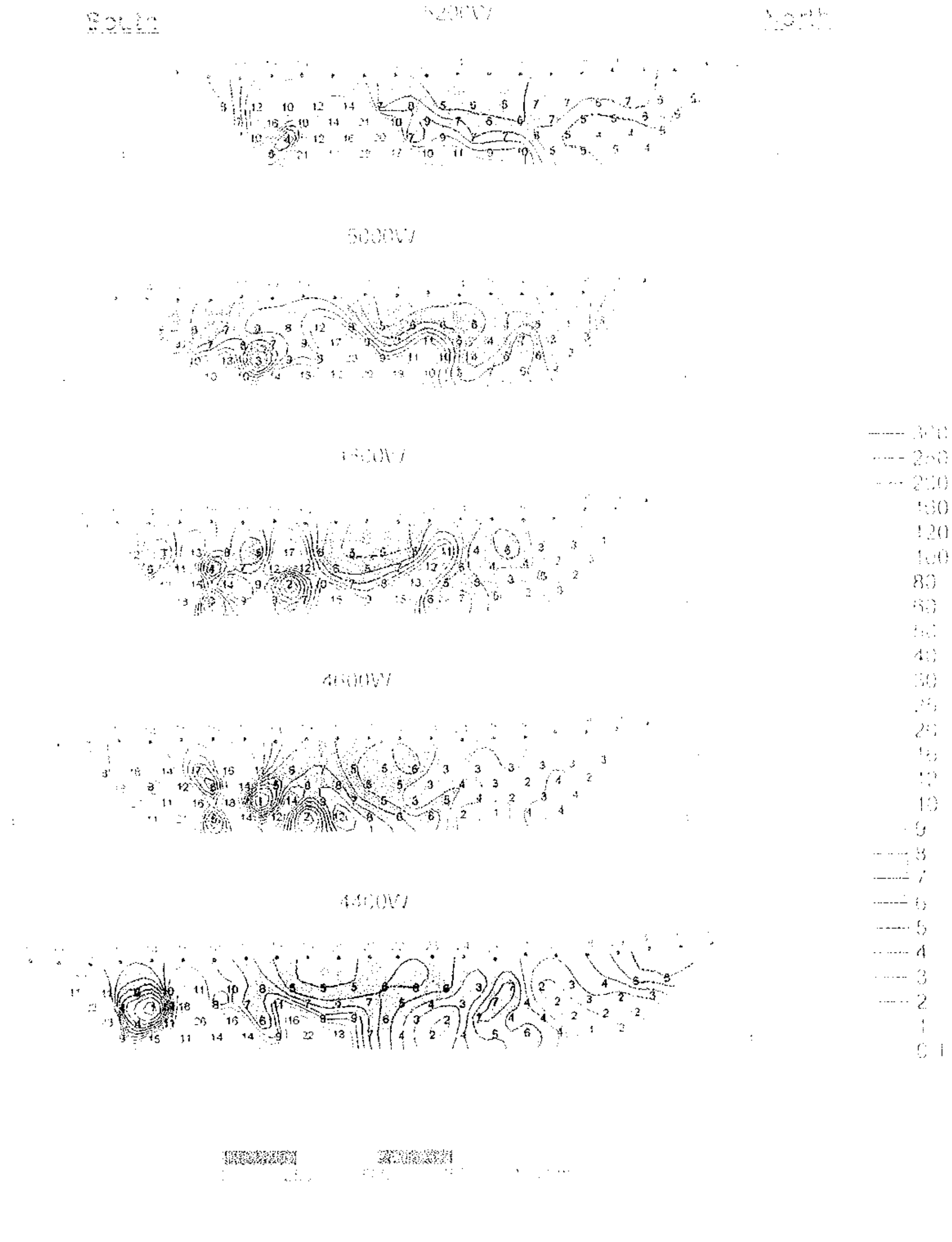


Fig. II-5-5(8) Metal factor pseudo sections (5200W-4400W)

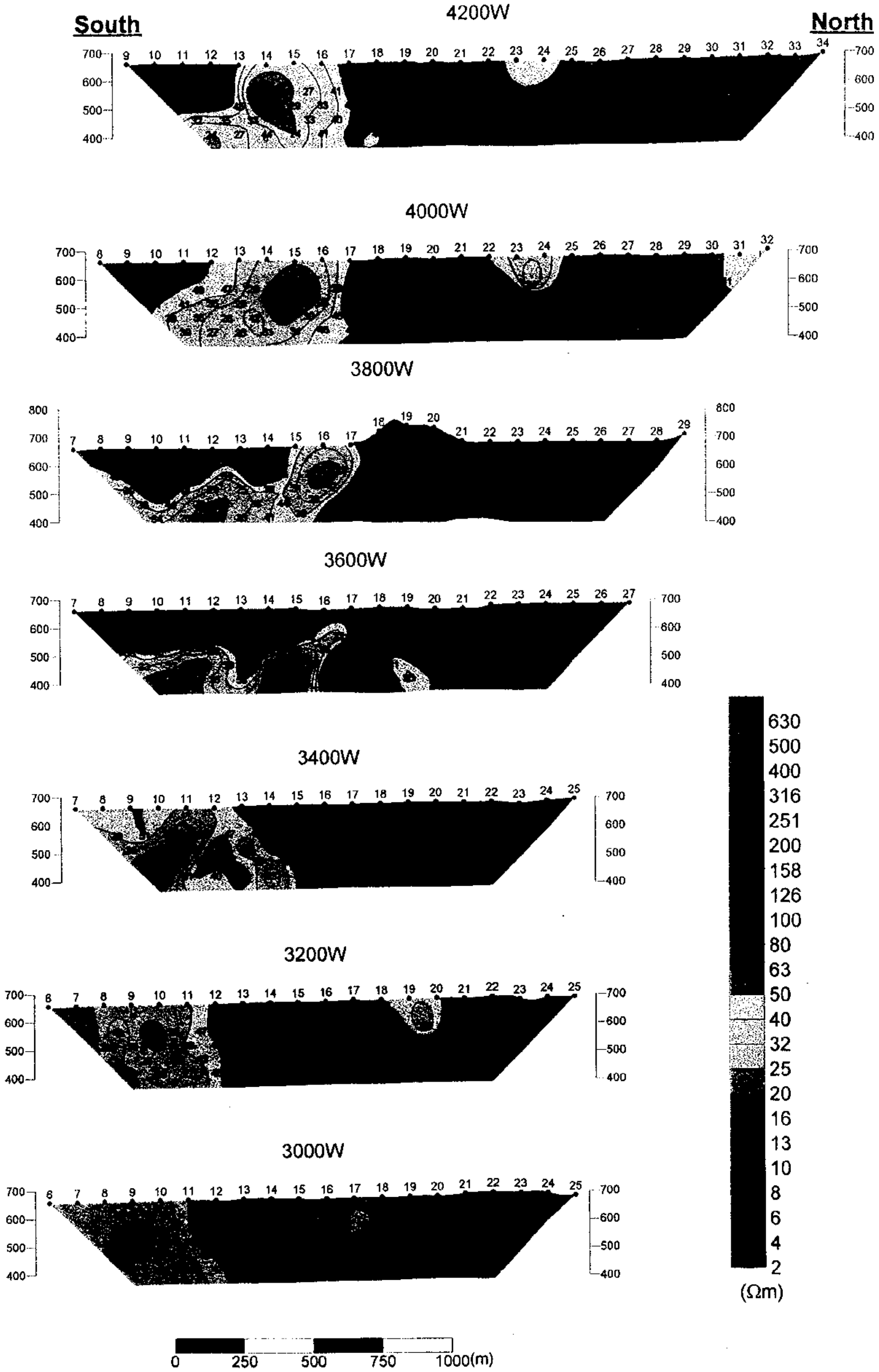


Fig. II -5-6(1) Apparent resistivity pseudo-sections(4200W-3000W)

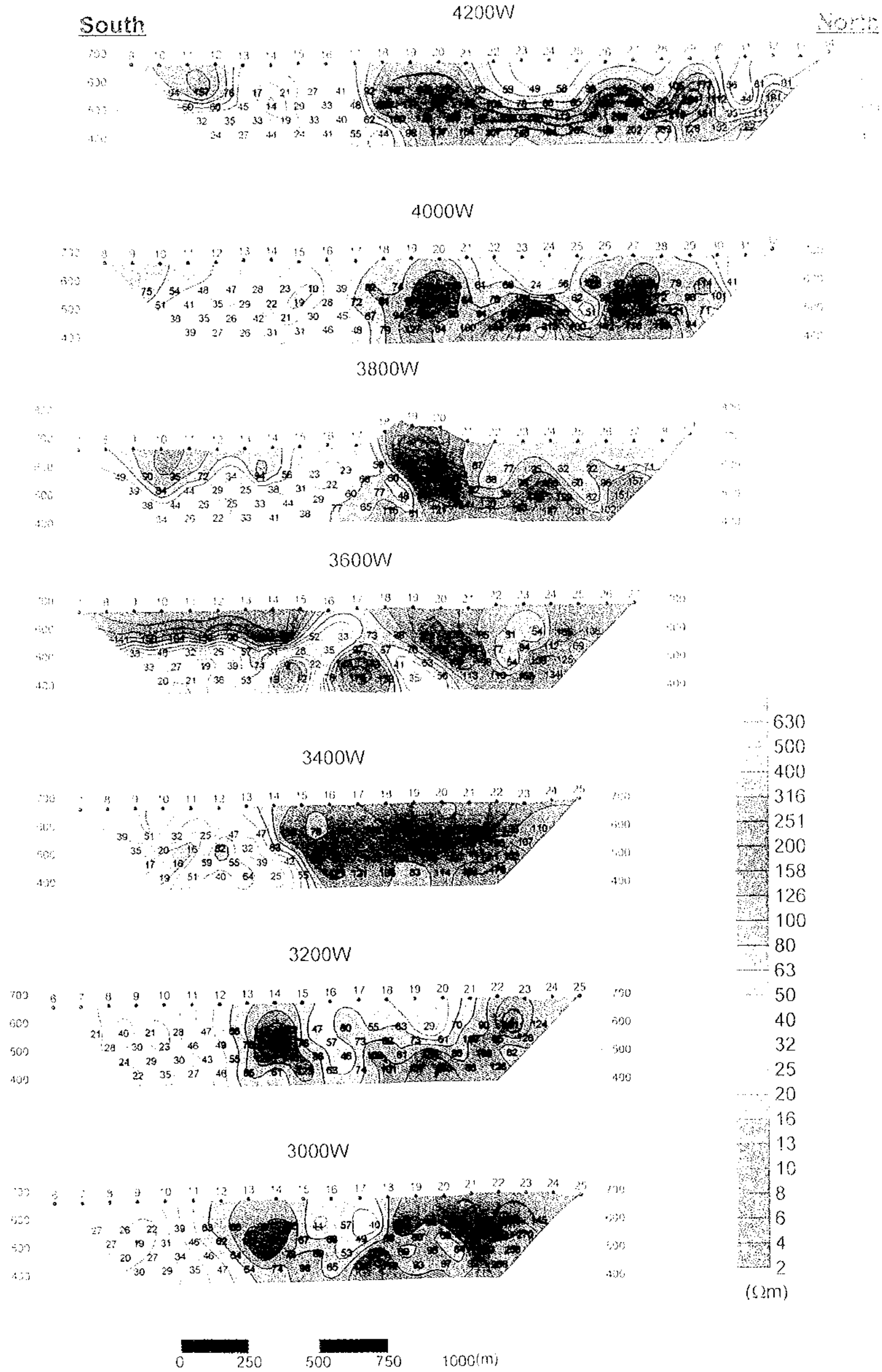


Fig. II-5-6(1) Apparent resistivity pseudo-sections(4200W-3000W)

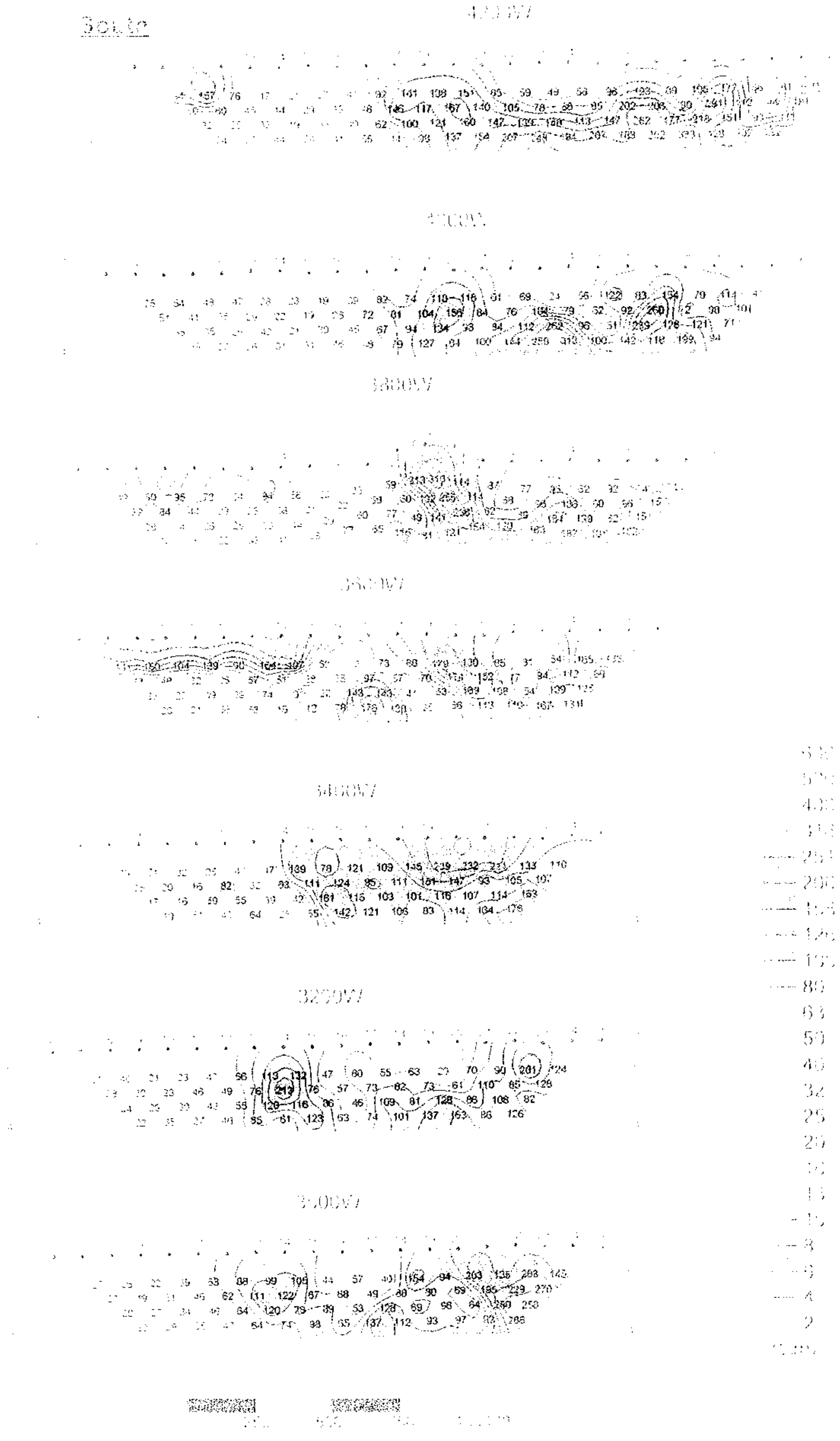


Fig. E-8 (b) Apparent resistivity pseudo sections [illegible]

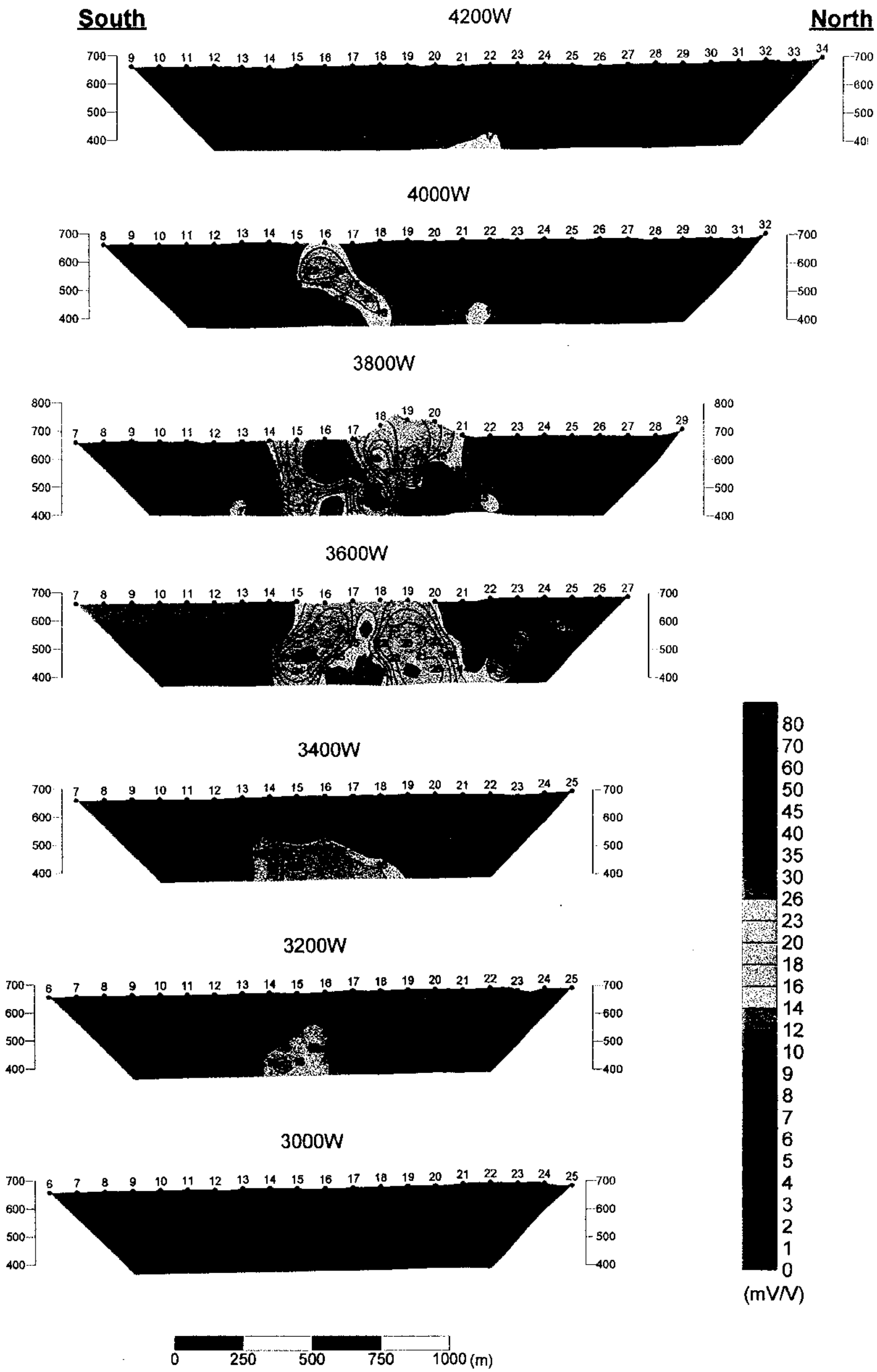


Fig. II-5-6(2)　Chargeability pseudo-sections(4200W-3000W)

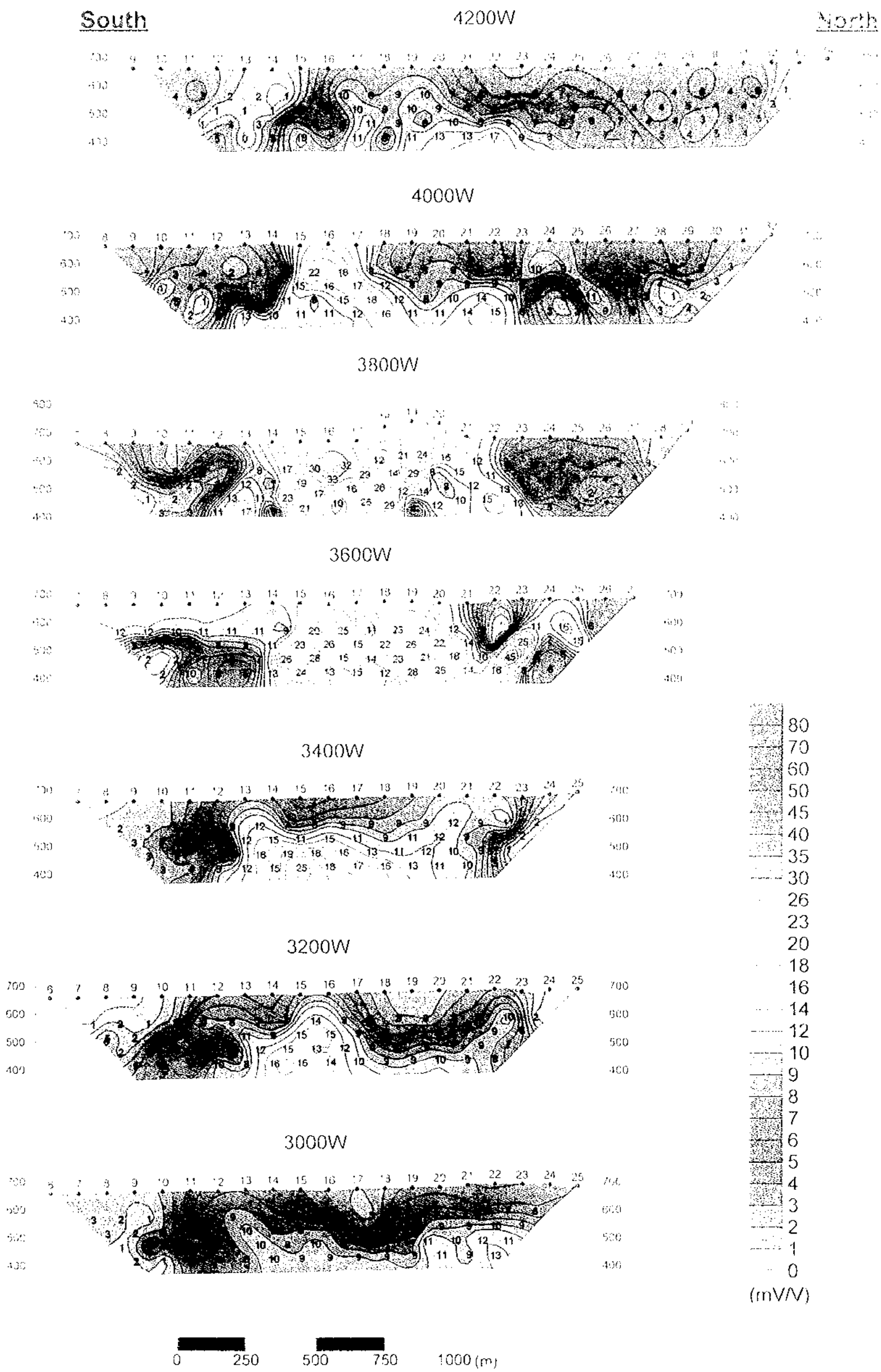


Fig. II-5-6(2) Chargeability pseudo-sections(4200W-3000W)

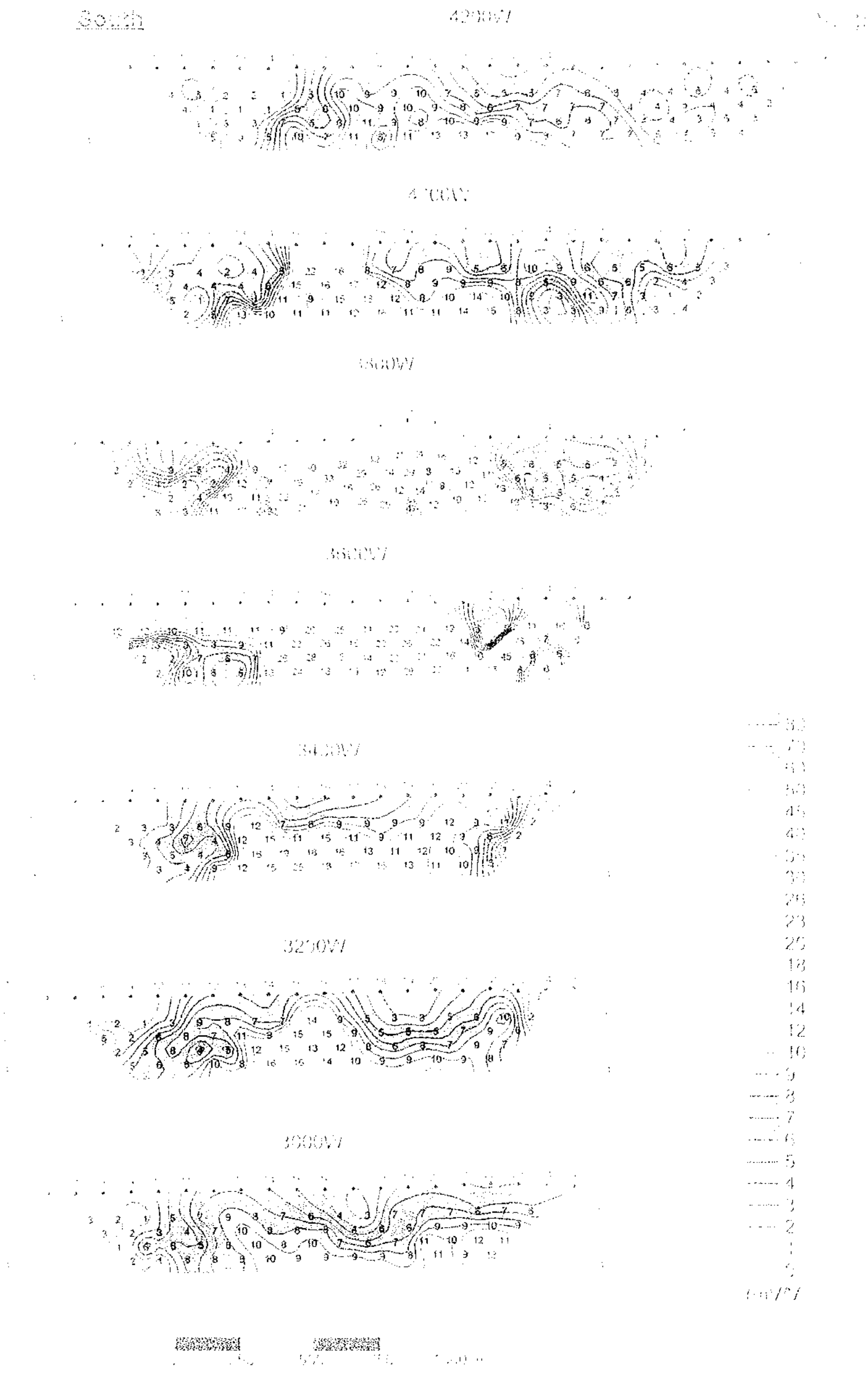


Fig. II-3-4(2) Chargeability pseudo sections (4200W-3000W)