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

3-2 Mineral processing plant description

3-2-1 Site consideration

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

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

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

3-2-2 Fundamental conception of plant design

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

(1) Crushing and grinding

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

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

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

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

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

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

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

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

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

(2) Flotation

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

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

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

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

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

Table 3-21 Plant metallurgical balance

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

3-2-3 Process design criteria

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

3-2-4 Flowsheet description

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

Table 3-22 Process design criteria (1)

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

Table 3-22 Process design criteria (2)

<table>
<tr><th>Process</th><th colspan="4">Item</th><th>Unit</th><th>Quantity</th></tr>
<tr><td rowspan="2">Grinding</td><td colspan="6">Single stage ball mill grinding
closed circuit with cyclones</td></tr>
<tr><td colspan="4">Scheduled operating hrs
Availability
Average running hrs
Milling rate

Grinding feed (80% passing)
Grinding pro't (80% passing)

Bond's Work Index

Pulp density (W%)
Ball mill discharge
Cyclone overflow

Circulating load
(Percentage of new feed)</td><td>hr/day
%
hr/day
mt/hr

μ
μ

kwh/st
kwh/st

%
%

%</td><td>24
83
20
150

9,400
150

Wir= 13
Wib= 12

75
38

350</td></tr>
<tr><td rowspan="2">Regrinding</td><td colspan="4">Closed circuit operation with</td><td>ball mill</td><td>& cyclone</td></tr>
<tr><td colspan="4">Capacity

Grinding feed (80% passing)
Grinding pro't (80% passing)

Bond's Work Index

Pulp density (W%)
Ball mill discharge
Cyclone overflow

Circulating load
(percentage of new feed)</td><td>mt/hr

μ
μ

kwh/st

%
%

%</td><td>63

100
38

Wi = 14

75
25

250</td></tr>
<tr><td rowspan="3">Flotation</td><td colspan="6">Bulk & differential flotation</td></tr>
<tr><td>Circuit</td><td>Pulp flow rate (m^3/min)</td><td colspan="2">No. of cells (300cf)</td><td colspan="3">Flotation retention time (min)</td></tr>
<tr><td>Rougher
First clr
Second clr
Third clr
Fourth clr
Clr scav'r</td><td>4.89
5.41
2.25
1.46
.72
4.80</td><td colspan="2">12
6
4
2
2
6</td><td colspan="3">20.8
9.4
15.1
11.7
23.6
10.6</td></tr>
</table>

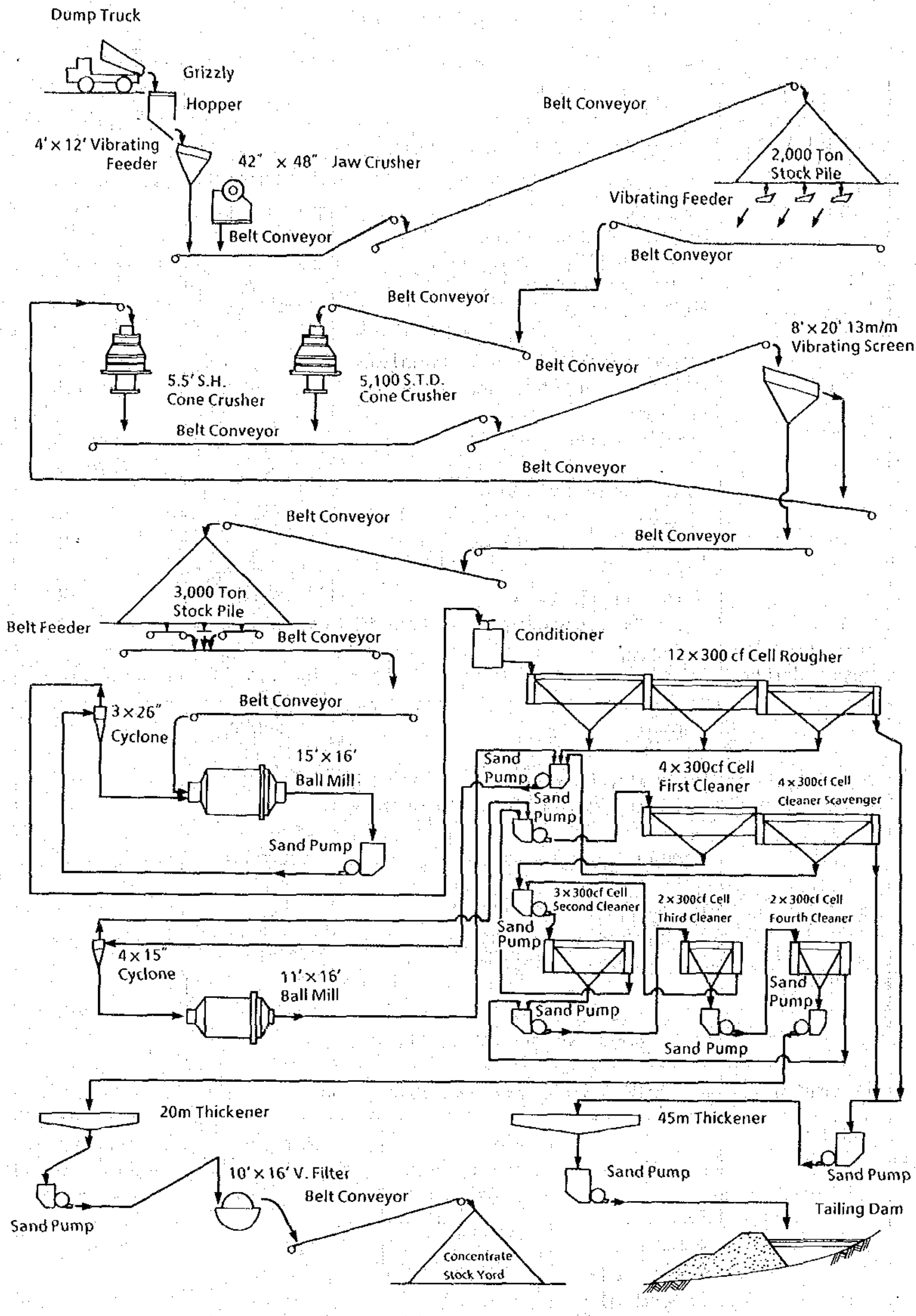


Fig. 3-24 Mineral processing plant flow diagram

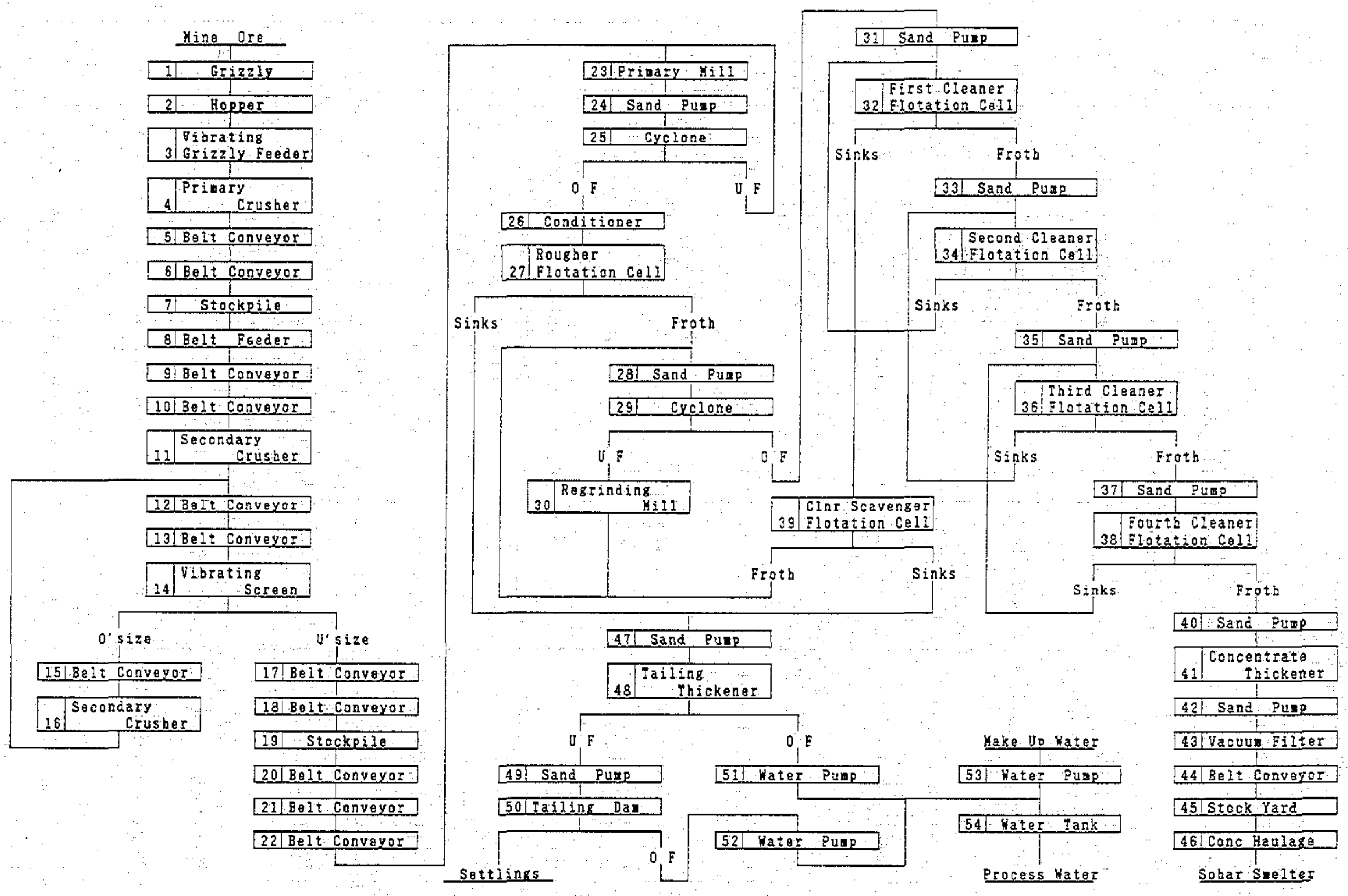


Fig. 3-25 Proposed mineral processing flowsheet

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

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

Table 3-23 Proposed plant equipment list (2)

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

No of Stand-by Unit ( )

Table 3-23 Proposed plant equipment list (3)

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

No of Stand-by Unit ( )

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

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

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

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

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

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

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

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

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

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

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

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

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

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

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

(7) Mill water

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

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

(8) Power supply

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

Table 3-24 Mill water balance

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

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

3-2-5 Operation

(1) Plant organization and manpower

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

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

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

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

(2) Operating materials

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

(3) Power consumption

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

Table 3-25 Manpower requirement of Metallurgical department

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

Table 3-26 Estimate operating material consumption

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

Table 3-27 Estimated power consumption

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

(4) **Total operating costs**

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

Table 3-28 Estimated operating costs

1,080,000 tons/year

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

Chapter 4 Waste dump and tailings dam

4-1 Waste dump

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

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

4-2 Tailing dam

4-2-1 Selection of tailing dam site

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

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

4-2-2 Topography and nature of ground

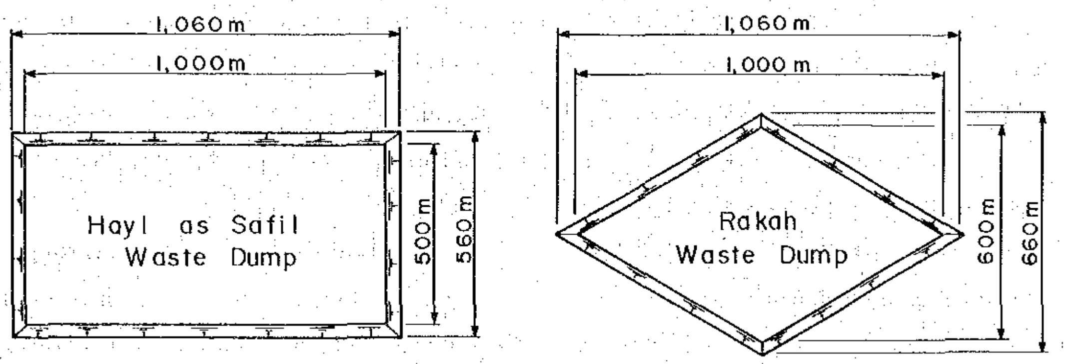
Topography of the site is slightly inclined at north-east gradient of 1/60, and the area of the site is east-west 1.2 km, north-south 1.2 km.

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

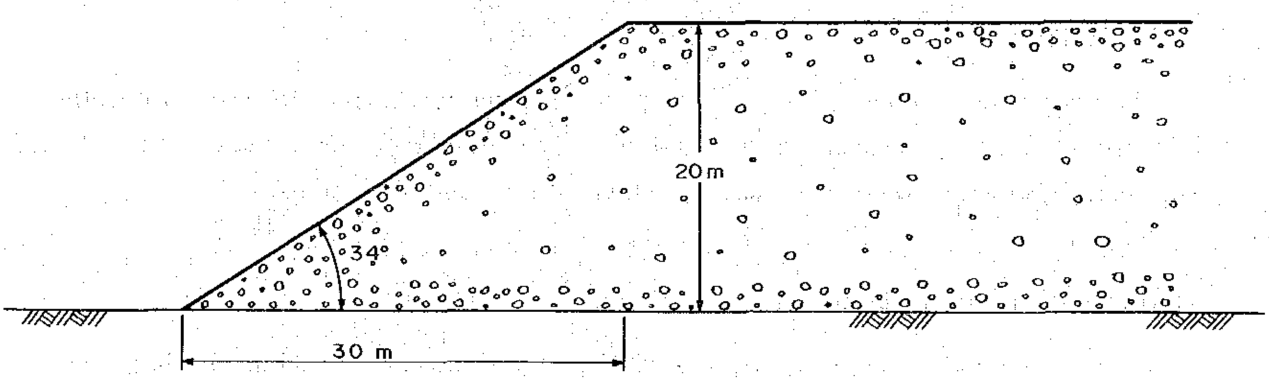
4-2-3 Dam

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

Fig. 4-1 Plan of tailing dam



Plan of Waste Dump



Section of Waste Dump

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

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

4-2-4 Drainage facility

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

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

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

4-3 Other facilities

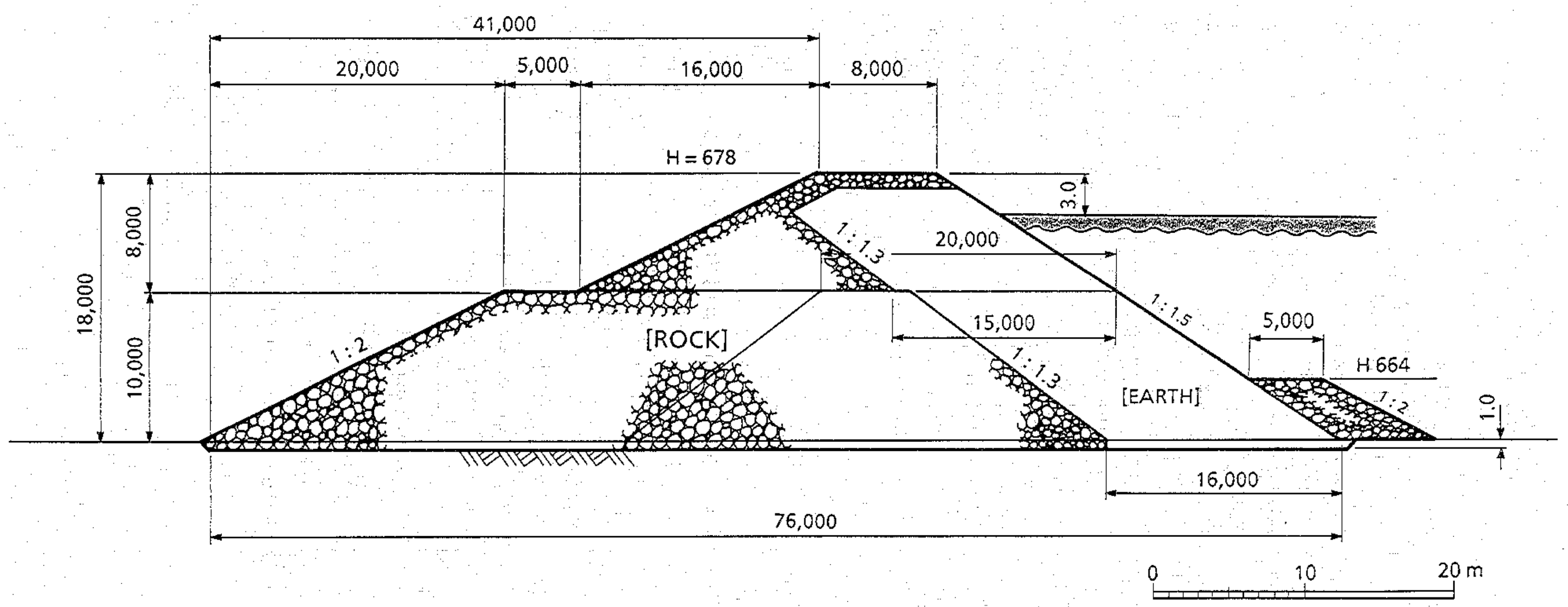
(1) Pit drainage

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

(2) Dust prevention for the mining hauling road

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

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



Section of Tailing Dam

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

Table 4-1 Specification of banking

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

Table 4-2 Design criteria of drainage facility

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

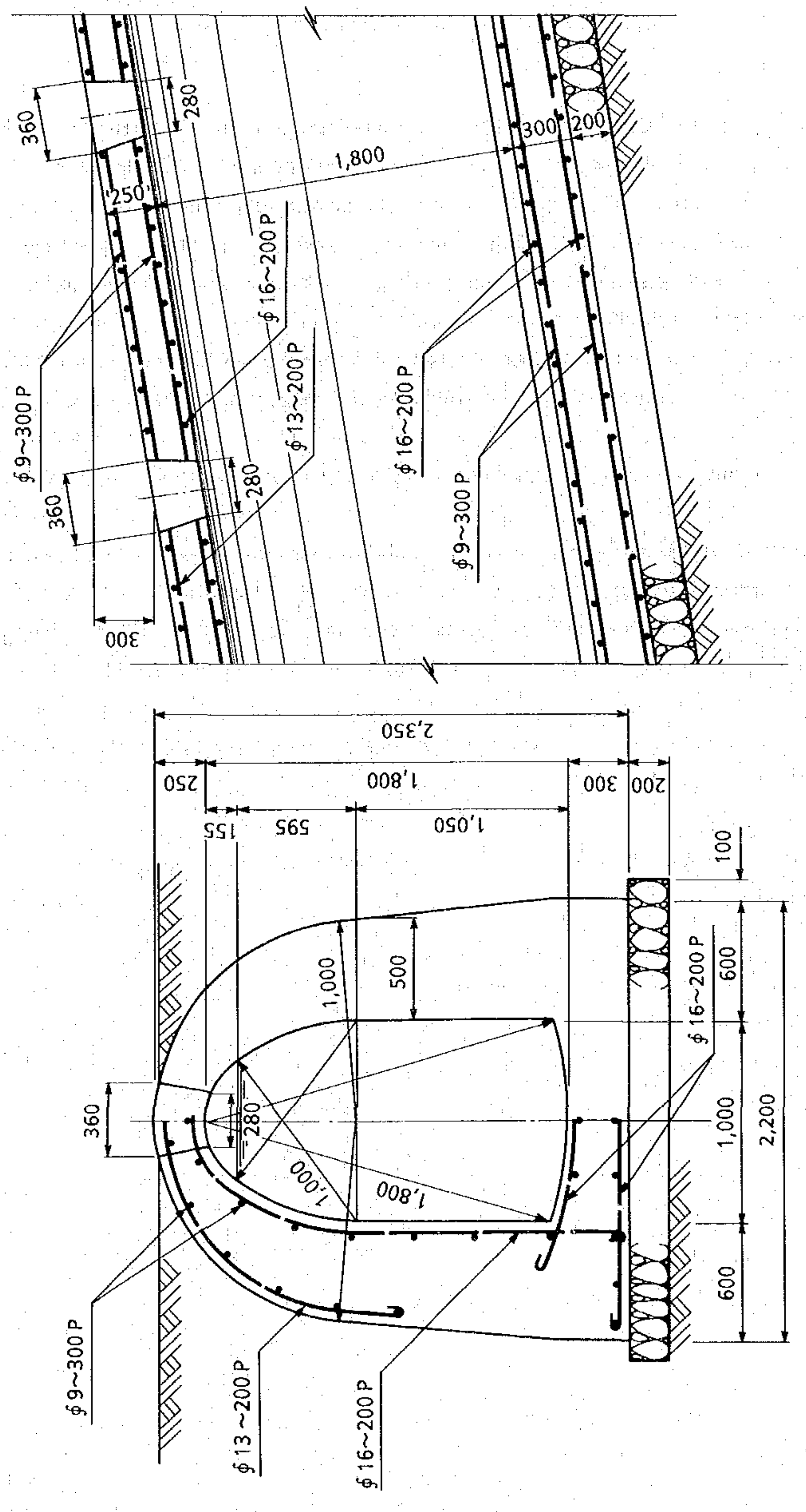


Fig. 4-3 Section of drain culvert

Chapter 5 Supporting

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

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

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

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

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

Chapter 6 Organization and manning plan

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

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

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

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

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

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

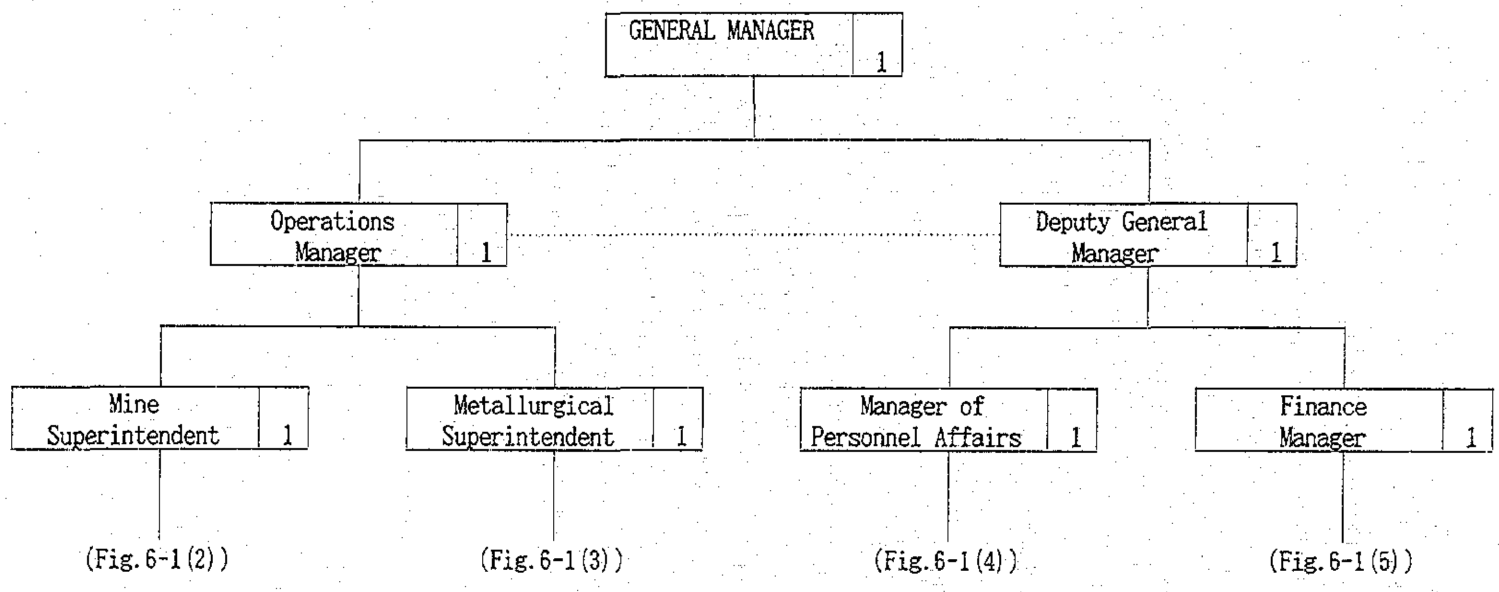


Fig. 6-1 Proposed organization for mine operation (1)

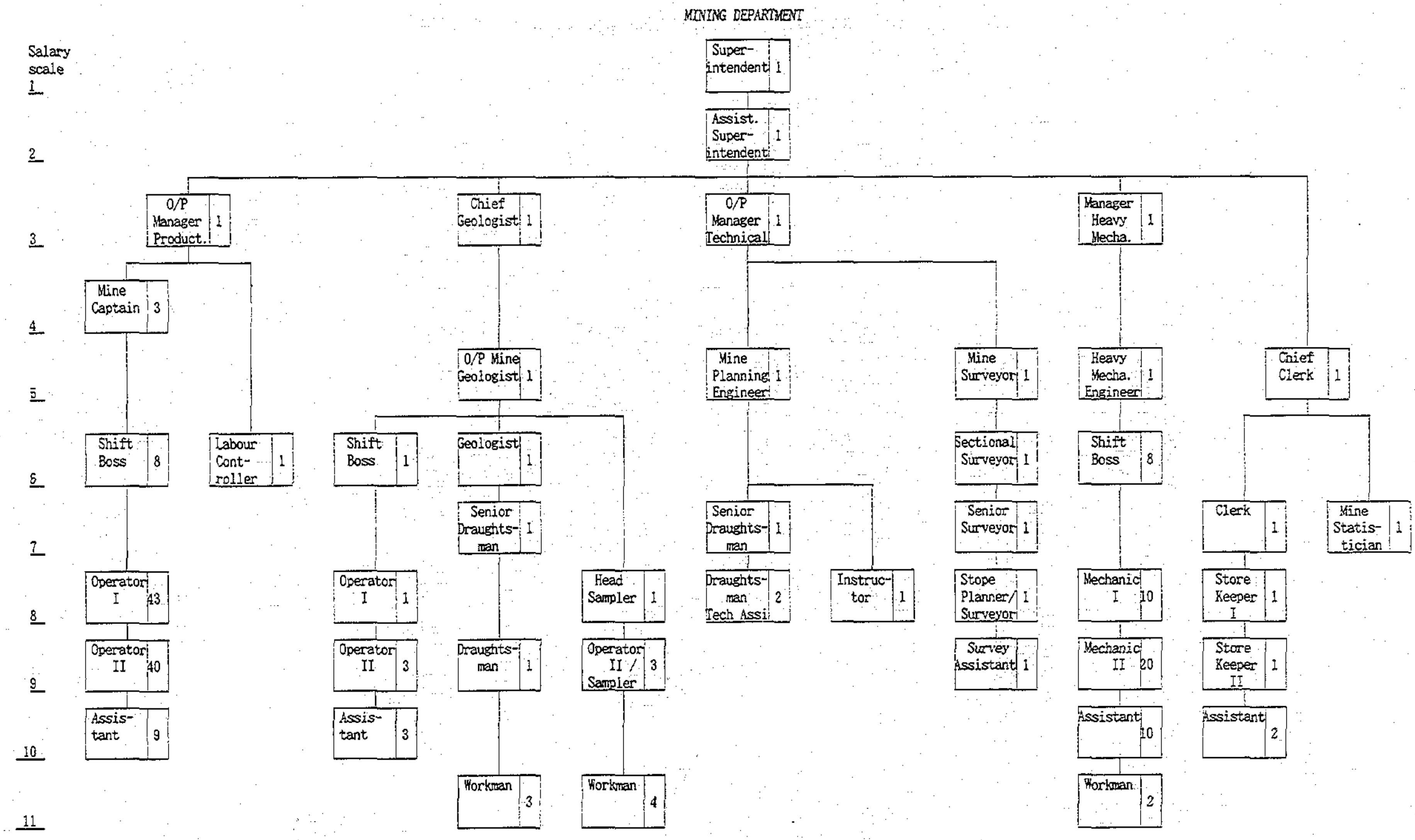


Fig. 6-1 Proposed organization for mine operation (2)

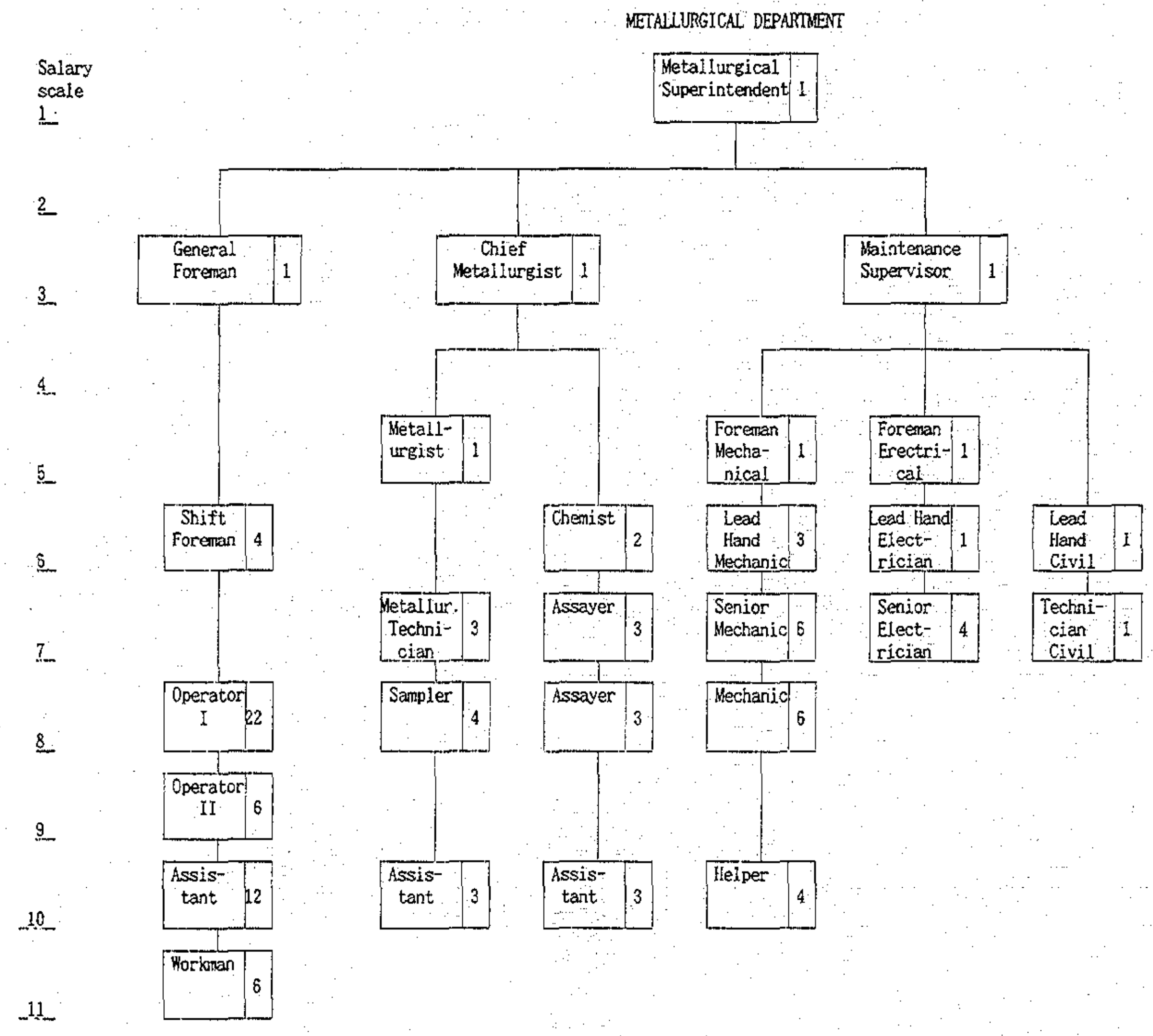


Fig. 6-1 Proposed organization for mine operation (3)

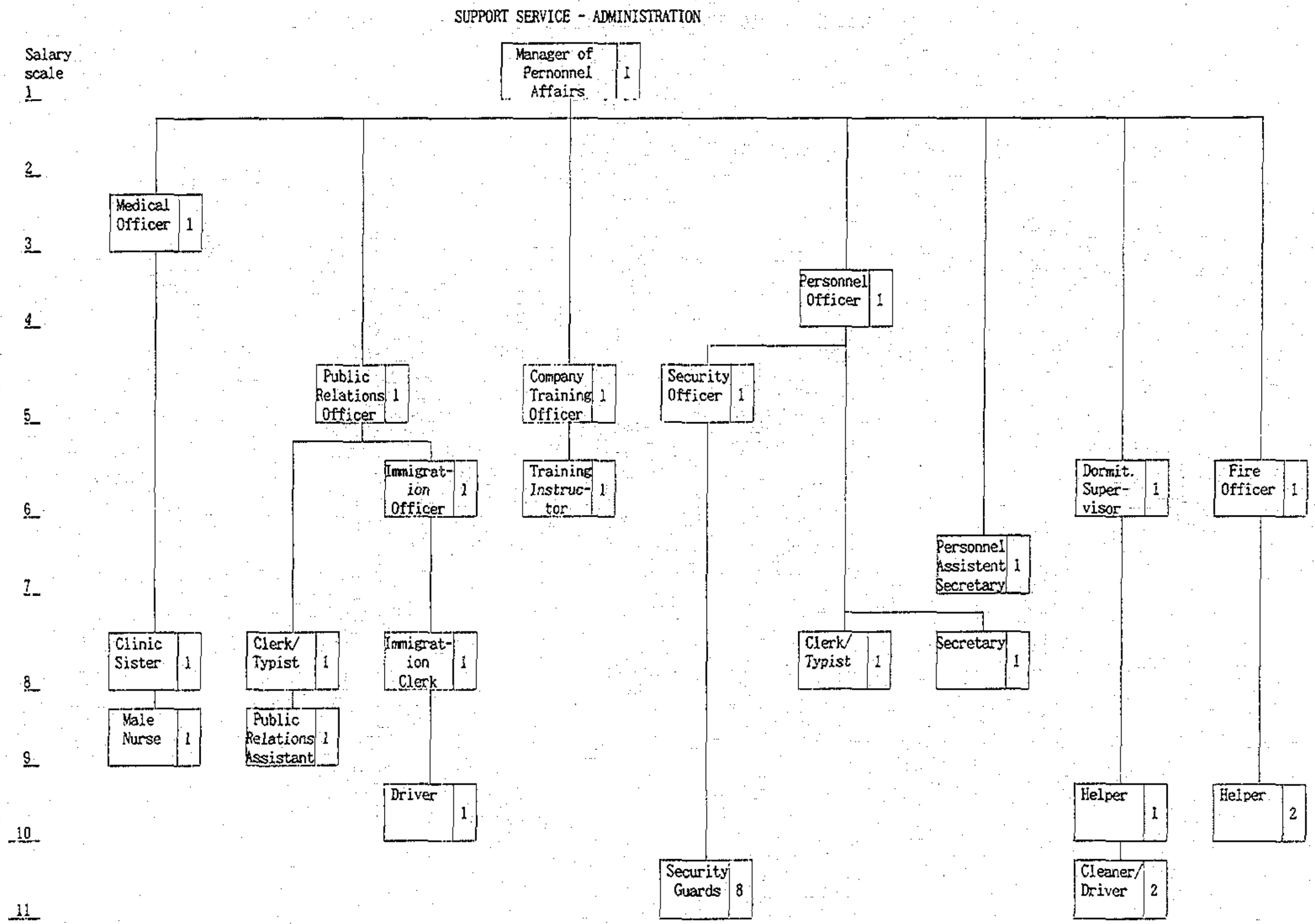


Fig. 6-1 Proposed organization for mine operation (4)

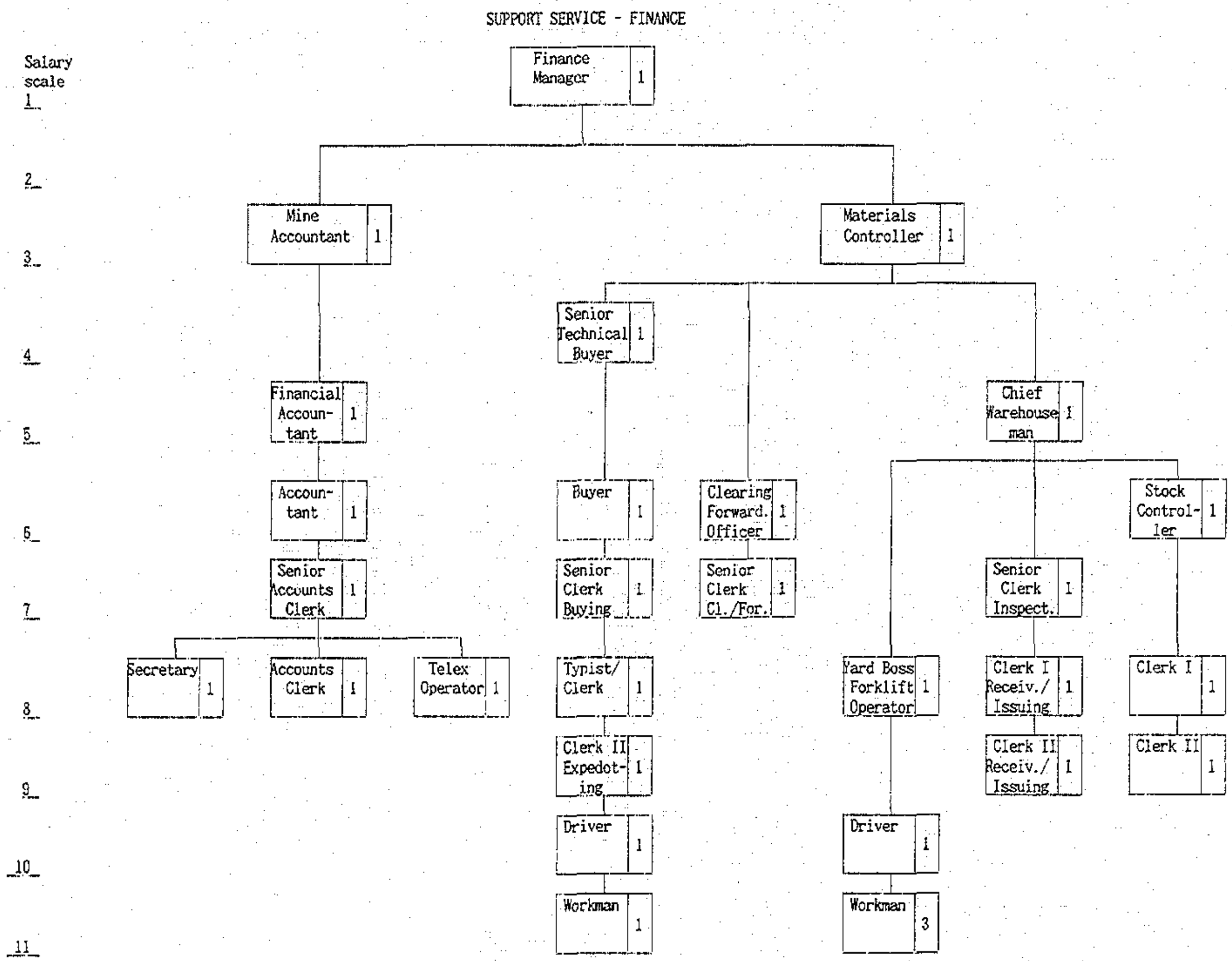


Fig. 6-1 Proposed organization for mine operation (5)

Table 6-1 Proposed manpower salary and wage

(Unit:US$)

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

Salary scale calculation

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

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

Chapter 7 Infrastructure

7-1 Transportation

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

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

7-1-2 Transportation of copper concentrate

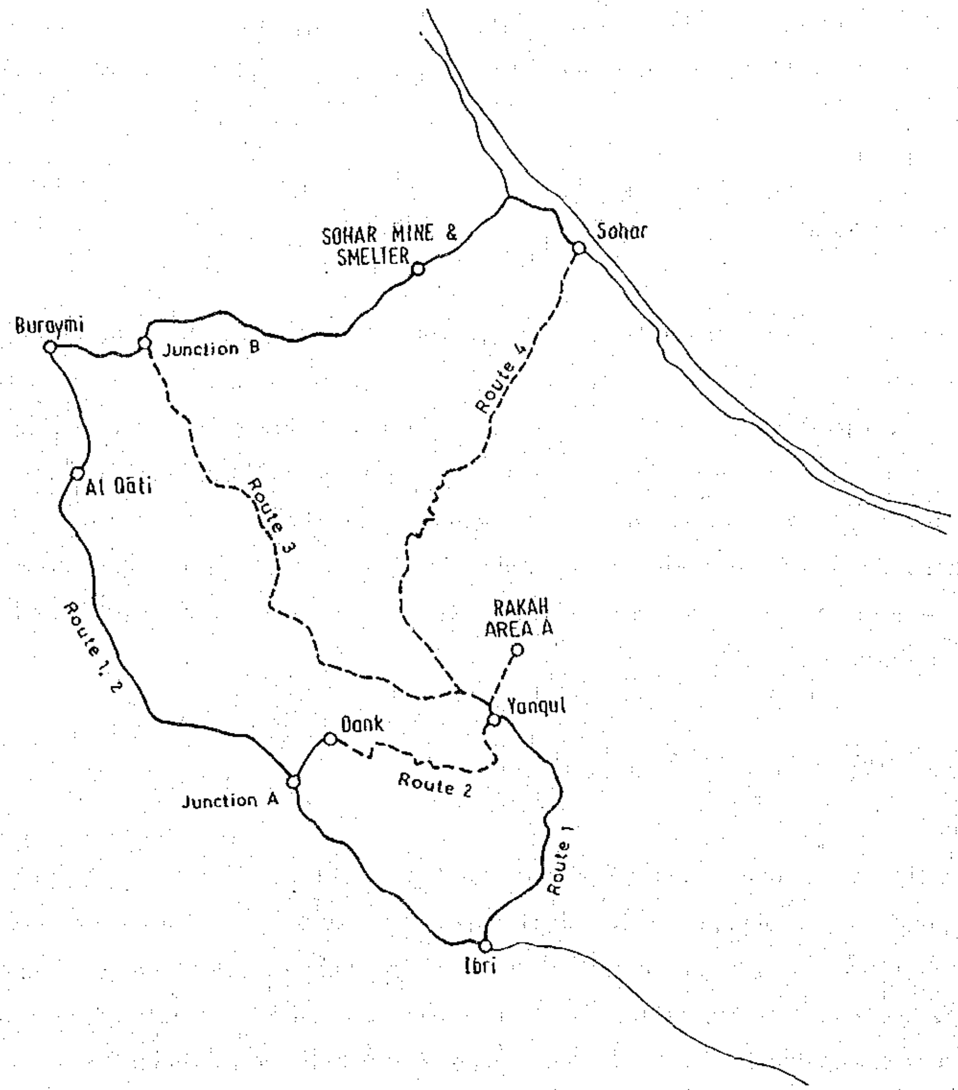
(1) Route of transportation

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

(2) Method of transportation

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

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



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

Fig. 7-1 Copper concentrate haulage road

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

7-2 Water supply

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

7-3 Electricity

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

7-4 Communicating system

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

7-5 Housing facilities

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

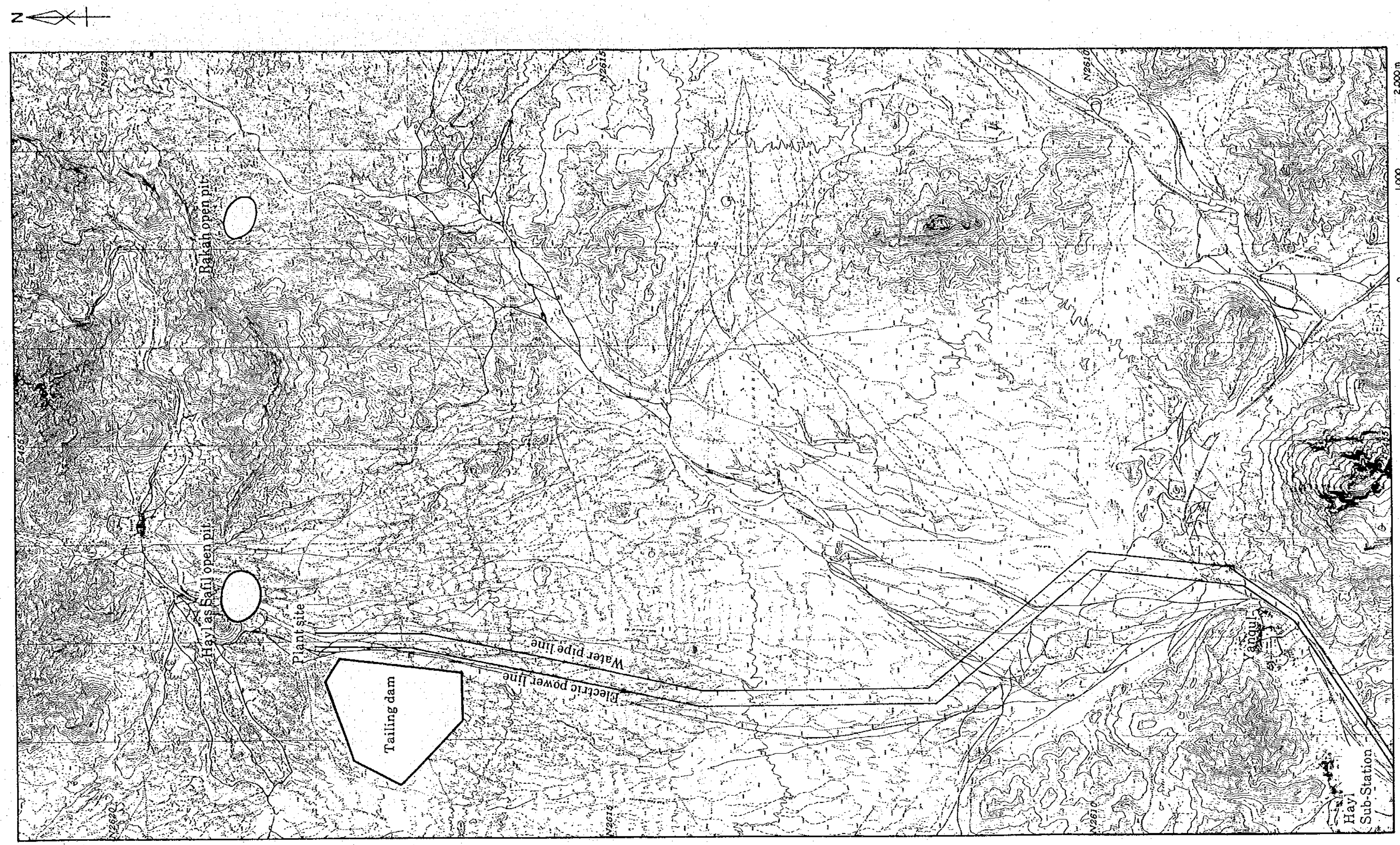


Fig. 7-2 Proposed water pipe line and electric power line

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

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

For married employees:

High grade	7 houses
Middle grade	17
Standard grade	25

For single employees:

Dormitory	103 rooms

Chapter 8 Initial and additional investment, operating cost

(1) Initial investment

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

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

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

Table 8-1 Summary of construction cost

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

(2) Additional investment

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

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

(3) Operating cost

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

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

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

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

Table 8-2 Initial and additional investment schedule

(Unit : US$1,000)

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

Chapter 9 Overall evaluation

9-1 Financial evaluation

(1) Annual production schedule

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

(2) Estimated annual revenue

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

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

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

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

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

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

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

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

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

Table 9-1 Annual production schedule

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

Table 9-2 Estimated annual revenue

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

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

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

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

(Unit : US$1,000)

		Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
(PROFIT & LOSS STATEMENT)												
1. NET REVENUE				19,009.8	26,362.7	25,151.8	28,220.7	24,830.7	16,263.0	21,020.9	16,272.1	177,131.6
2. COSTS												
Direct Operating Costs												
Mining				2,872.9	5,268.6	4,870.3	4,543.2	3,432.9	2,694.4	2,560.8	2,295.7	28,538.8
Concentrator				3,784.7	4,819.0	4,819.0	4,819.0	4,819.0	4,819.0	4,819.0	4,819.0	37,517.4
Supporting				1,566.9	1,566.9	1,566.9	1,566.9	1,566.9	1,566.9	1,566.9	1,566.9	12,535.2
Conc. Transportation				507.7	704.0	671.7	753.7	663.1	434.3	561.4	434.6	4,730.5
Training Cost				303.1	303.1	303.1	303.1	303.1	0.0	0.0	0.0	1,515.6
Sub-total				9,035.3	12,661.7	12,231.0	11,985.9	10,785.0	9,514.5	9,508.1	9,116.1	84,837.5
Royalty				925.1	1,282.9	1,224.0	1,373.4	1,208.4	791.4	1,023.0	791.9	8,620.1
Depreciation				6,001.1	7,976.5	7,976.5	8,046.5	8,204.0	8,204.0	8,204.0	8,204.0	62,816.4
Interest	10.00%			4,804.3	4,614.6	3,925.0	3,172.0	2,059.9	961.8	448.1	0.0	19,985.8
Total Costs				20,765.8	26,535.7	25,356.5	24,577.7	22,257.3	19,471.8	19,183.2	18,111.9	176,259.8
3. PROFIT BEFORE TAX				-1,756.0	-173.0	-204.7	3,643.0	2,573.4	-3,208.8	1,837.7	-1,839.9	871.8
4. INCOME TAX				0.0	0.0	0.0	96.3	176.1	0.0	0.0	0.0	272.4
5. NET PROFIT AFTER TAX				-1,756.0	-173.0	-204.7	3,546.7	2,397.3	-3,208.8	1,837.7	-1,839.9	599.4
(CASH FLOW STATEMENT)												
Net Profit After Tax				-1,756.0	-173.0	-204.7	3,643.0	2,477.1	-3,384.9	1,837.7	-1,839.9	599.4
Depreciation				6,001.1	7,976.5	7,976.5	8,046.5	8,204.0	8,204.0	8,204.0	8,204.0	62,816.4
Equity		13,703.8										13,703.8
Loan		6,181.0	41,862.4									48,043.4
Capital Expenditure		-19,322.9	-35,492.4									-54,815.4
Interest During Construction		-561.9	-4,111.2									-4,673.1
Additional Capital Expenditure				-2,348.0	0.0	-350.0	-630.0	0.0	0.0	0.0	0.0	-3,328.0
Working Capital Increase (Decrease)			-2,258.8	0.0	-906.6	107.7	61.3	300.2	317.6	1.5	2,377.0	0.0
Loan Repayment				-1,897.1	-6,896.8	-7,529.5	-11,120.8	-10,981.3	-5,136.7	-4,481.2	0.0	-48,043.4
Net Generated Cash				0.0	0.0	0.0	0.0	0.0	0.0	5,562.1	8,741.1	14,303.2
PRINCIPAL			48,043.4	46,146.4	39,249.5	31,720.0	20,599.2	9,617.9	4,481.2	0.0	0.0	
(RATE OF RETURN)												
Net Generated Cash				0.0	0.0	0.0	0.0	0.0	0.0	5,562.1	8,741.1	14,303.2
Capital Expenditure		-19,884.9	-41,862.4									-61,747.3
Repayment Flow Adjustment				1,897.1	6,896.8	7,529.5	11,120.8	10,981.3	5,136.7	4,481.2	0.0	48,043.4
Interest Flow Adjustment				4,804.3	4,614.6	3,925.0	3,172.0	2,059.9	961.8	448.1	0.0	19,985.8
Cash Flow Out and In		-19,884.9	-41,862.4	6,701.4	11,511.5	11,454.4	14,292.8	13,041.2	6,098.5	10,491.4	8,741.1	20,585.2
Discounted Cash Flow at	6.40%	-18,689.3	-36,979.9	5,563.9	8,982.9	8,400.9	9,852.4	8,449.2	3,713.5	6,004.4	4,701.9	-0.0
(RATE OF RETURN TO THE EQUITY)												
Net Generated Cash				0.0	0.0	0.0	0.0	0.0	0.0	5,562.1	8,741.1	14,303.2
Capital Expenditure		-13,703.8										-13,703.8
Cash Flow Out and In		-13,703.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	5,562.1	8,741.1	599.4
Discounted Cash Flow at	0.50%	-13,635.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	5,318.7	8,317.2	-0.0

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

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

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

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

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

9-2 Economic evaluation

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

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

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

9-3 Sensitivity analysis

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

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

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

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

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

(Unit : US$1,000)

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

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

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

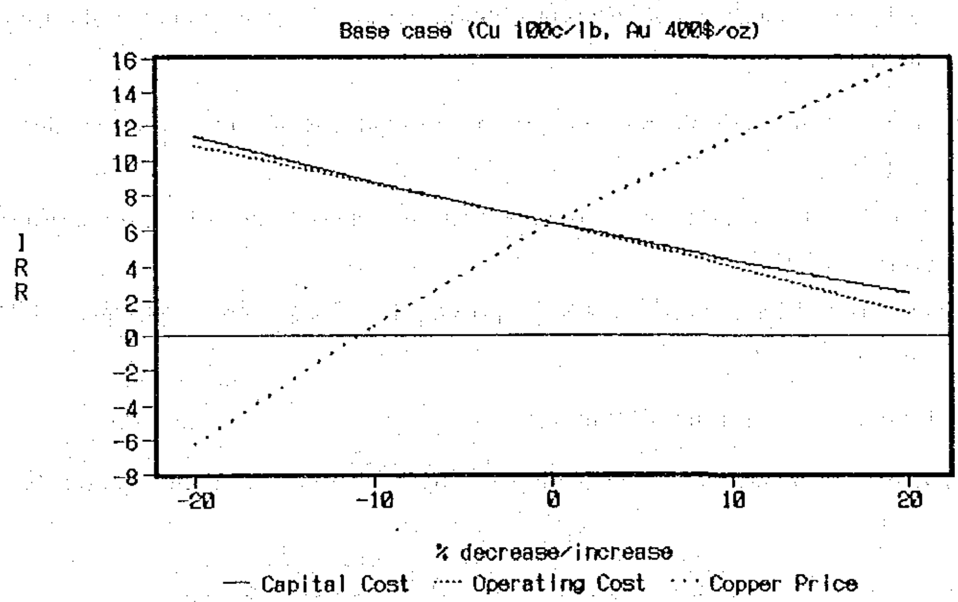


Fig. 9-1　FIRR sensitivity analysis

Chapter 10 Conclusion

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

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

(3) Minable ore reserves are;

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

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

(5) The flotation characteristics of head samples are;

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

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

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

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

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

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

(7) Waste dumps and tailing dam are designed.

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

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

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

(11) Financial IRR to the project is 6.40% and IRR to the equity is 0.50%.
Sensitivity analysis on the financial IRR has shown that the project is most sensitive to copper price.

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

FIGURES

TABLES