

CHAPTER 6 CONTAMINATION SOURCE INVESTIGATION

CHAPTER 6 CONTAMINATION SOURCE INVESTIGATION

Contamination sources related to the Sohar mine area consist of the tailing dam, the area around PS-2 pumping station, waste dump areas, abandoned mines, abandoned open pits, the copper smelter, and evaporation ponds.

6.1 Tailing Dam

6.1.1 Summary of Contamination Source

The characteristics of the tailing dam and tailings are presented in Table 6.1 and Table 6.2, respectively.

- The tailing dam is located in the uppermost reach of Wadi Suq (Figure 1.1). The embankment was constructed by the inside piling method.
- The dam embankment consists of two banks, one on the east side and one on the south side. The area of the tailings impoundment is 505,000 m². The total tonnage reached approximately 11 million tons.
- The tailings consist of silt to fine sized sand containing 20 to 35% pyrite.
- The top layer of the tailings impoundment is solidified by desiccation and numerous polygonal shrinkage cracks can be found. These cracks developed perpendicularly approximately 10 to 50cm in depth and 20 cm in length. The interior surfaces of the cracks are oxidized presenting a yellowish brown color.
- Tailings sampled from the surface layer showed strong acidity with a pH 2 to 3.
- Lower layers deeper than 2 to 3 m presented grayish color and softness due to high moisture content. The color of these materials indicates they are under reducing conditions.
- The top layer consists of fine particles, having thickness of 2 to 10 cm. The fine tailings from the surface are scattered by wind in every direction out of tailing impoundment (Figure 6.1).
- The whole tonnage of industrial water for ore dressing was supplied by seawater between 1982 and 1989. In 1991 all industrial water was changed to fresh water.
- The total amount of seawater discharged at tailing dam is estimated to be about 5 million m³ containing salt of 17,500 t.
- Contaminated groundwater collected in Trench-2 is returned to the tailing dam via Trench-1. The purpose of pumping the seepage back to the tailing dam is to try to contain the seepage and to lose as much as possible by evaporation.

6.1.2 Result of Leaching Test for Tailing

Tailings were sampled during the drilling investigation to execute a leaching test. The leaching tests consisted of solubility tests (15 components: pH, EC, Hg, Cd, Cr, As, Pb, Cu, Mn, Fe, Ni, Sn, Zn, SO₄, and Cl), total sulfur test (TS), Net Neutralization Potential (NNP) test, and Maximum Potential Acidity

Table 6.1 On the Tailing Dam

1. Tailing dam	Location	Uppermost part of Wadi Suq, between 0.93 km and 1.95 km from beginning of the river
	Area (m ² , ha)	567,000 m ² (56.7 ha) Piled area : 505,000 m ² (50.5 ha) Embankment area: 62,000 m ² (6.2 ha)
	Elevation (m)	265m – 235m
	Length (m)	1.02 km
	Width (m)	Max. : 850 m Length of embankment: 650 m
	Gradient (°)	Gradient of basement of dam : 1.7° (1:34) Gradient of present piled surface: -0.52° (1:110)
2. Groundwater (Polluted water)	2 drill holes surveyed for the Study project	(1) D-1A: 50 m west from embankment of the dam (2) D-1B: 110 m west from embankment of the dam
	Groundwater level (m)	Depth from surface of tailings: DH No. D-1A: -32.64 m DH No. D-1B: -18.53 m Elevation of water table : DH No. D-1A: 230.08 m DH No. D-1B: 242.98 m
	Present condition of Countermeasures	(1) Trench -1 (2) Trench -2 (3) Pumping and return polluted water from Trench-1 and Trench-2 to Tailing dam : Total volume of polluted water returned to the T/D per month and year between 1997 and 2000 : Year Monthly Yearly : (m ³) 1995 : Ave. 18,950 227,399 1996 : Ave. 17,356 208,272 1997 : Ave. 9,956 119,466 1998 : Ave. 8,589 103,073 1999 : Ave. 8,274 49,645 (6 months)

T/D: Tailing dam

Table 6.2 On the Tailings

1. Tailings	Total tonnage (t)	11,000,000 t
	Density (g/cm ³)	Dry base: Ave. 2.95 g/cm ³ Wet base: Ave. 3.35 g/cm ³
	Total volume (m ³)	3,280,000 m ³
	Total sulfur: Content of sulfur (%) Total tonnage (t)	Ave. 19.4 % 2,135,000 t as Sulfur
	Total pyrite: Content of initial pyrite (%) Total volume of pyrite (m ³) Total tonnage of pyrite (t)	(Inferred as pyrite under the initial condition) 36.4 % 834,000 m ³ (Pyrite SG: 4.8) 4,003, 00 t
	Average neutralization potential	10.31 CaCO ₃ equivalent: t/1000t dried base
	Maximum potential acidity	646.05 CaCO ₃ equivalent: t/1000t dried base
	2. Sea water for mineral dressing	Used period
Total volume (m ³)		5,000,000 m ³
Total tonnage of salt (t)		175,000 t (NaCl: 3.46%, Cl: 34,600 ppm)
Residual salt in tailing dam (t)		Approx. 150,000 t
Flowed salt from tailing dam (t)		Approx. 25,000 t
3. Groundwater (Polluted water)	Pumping volume in T/D (m ³)	Approx. 72,000 m ³ /year (6,000 m ³ /month)
	Reserving volume in T/ D (m ³)	Volume: 39,600 m ³
	Water quality	pH: 4.8, EC: 8.40 S/m, Hg: 0.0047 mg/L, Cd: 0.027 mg/L, As: 0.012 mg/L, Pb: 0.59 mg/L, Cu: 1.21 mg/L, Zn: 0.30 mg/L, SO ₄ : 1,434 mg/L, Cl: 31,600 mg/L
4. Surface water	Volume of precipitation (m ³)	198,000 m ³ /year
	Volume of evaporation (m ³)	158,400 m ³ (Rate: 80%)
	Surface water in-flowed to T/D	Volume: 39,600 m ³ (Rate: 20%)
	Outflow by groundwater	Volume: 39,600 m ³
	Recycling volume by pumping	Volume: 72,000 m ³

T/D: Tailing dam

(MPA) test.

Data on tailings samples and leaching test results are presented in Tables 6.3 and 6.4 (1) and (2). Since tailings contain a great amount of sulfur, they have a very strong acid potential. The NNP of the tailings ranged between -360 to -840 t (tons as CaCO₃/1,000 t of tailings) indicating that there is a potential to create between 360 to 840 t of acidity from each 1,000 t of tailings, if all the sulfur is oxidized. Effluent obtained from the oxidized surface layer of the tailings showed a pH 2 to 3. However, the actual seepage out of tailing dam ranges between pH 5 to 7. The following reasons are presumed to explain such phenomena.

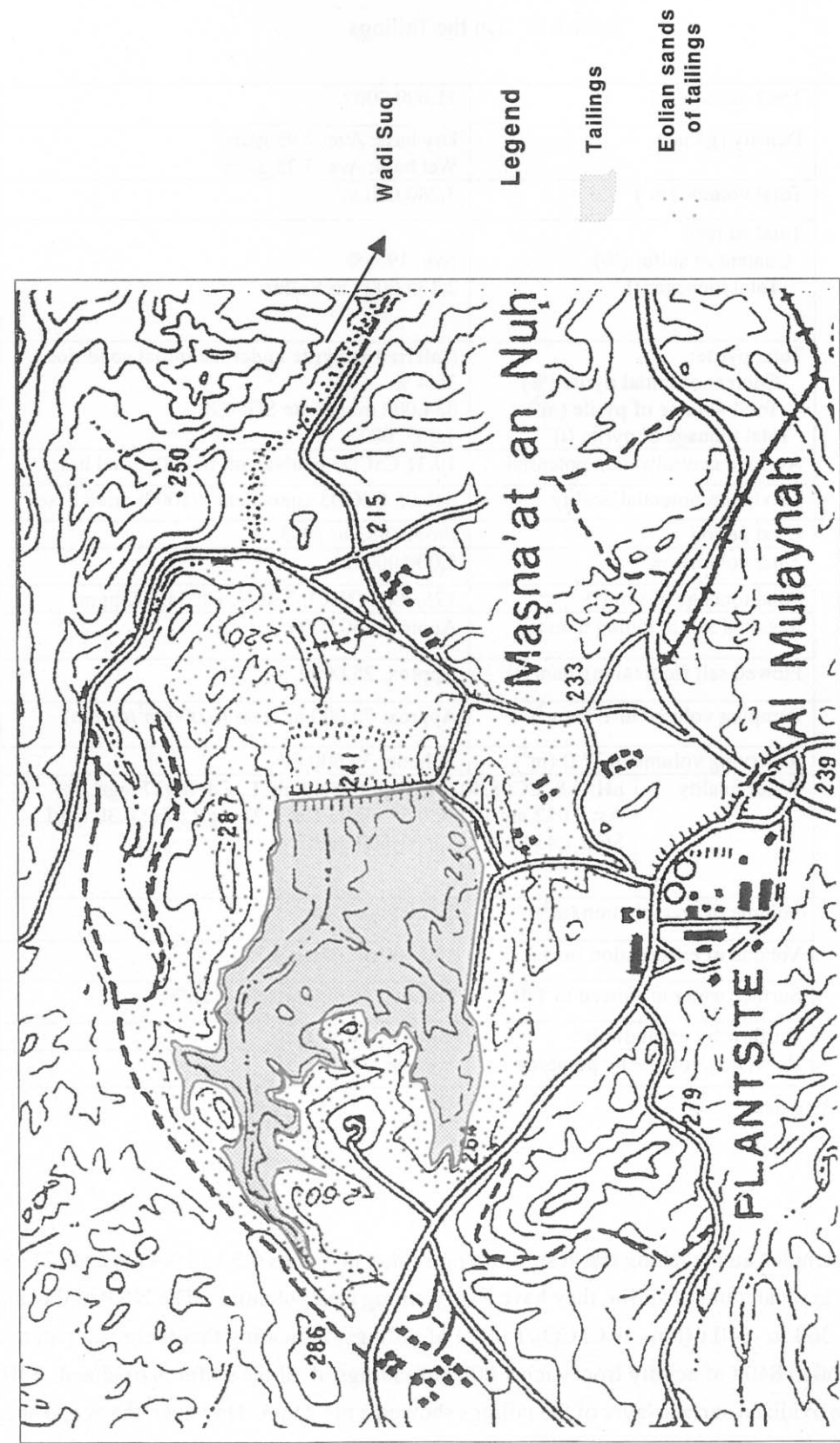


Figure 6.1 Scattered Distribution map of Tailings

Table 6.3 On the Tailings and Mine Wastes

Sample No.	Area name	Materials	Size	Color	Oxidation	Sulphide	Remarks
LT-DIA1	Tailing dam area	Tailings	Fine sand	Brownish-yellowish grey	Weakly oxidized	Rich pyrite	Dry in top surface
LT-DIA2	Tailing dam area	Tailings	Fine sand	Dark greenish grey		Rich pyrite	Wet
LT-DIA3	Tailing dam area	Tailings	Fine sand	Dark greenish grey		Rich pyrite	Wet
LT-T/D	Tailing dam area	Eolian Tailings	V. fine - fine sand	Yellowish grey	Partly oxidized	Rich pyrite	Dry
LT-DIB1	Tailing dam area	Tailings	Fine sand	Brownish-yellowish grey		Rich pyrite	Dry in top surface
LT-DIB2	Tailing dam area	Tailings	Fine sand	Dark greenish grey		Rich pyrite	Wet
LT-DIB3	Tailing dam area	Tailings	Fine sand	Dark greenish grey		Rich pyrite	Wet
LT-DIB4	Tailing dam area	Tailings	Fine sand	Dark greenish grey		Rich pyrite	Wet
LT-L1	Lasail area	Waste	Fines to boulder	Grey to dark grey	Oxidized		Dry
LT-L2	Lasail area	Waste	Fines to cobble	Grey to grey/green			Dry
LT-LW1	Lasail West area	Waste	Fines to boulder	Brownish grey	Oxidized		Dry
LT-LW2	Lasail West area	Waste	Fines to boulder	Dark greenish grey		Py disseminated	Oberburden, basalt; dry
LT-LW3	Lasail West area	Waste	Fines to boulder	Brownish-reddish grey	Gossanized		Dry
LT-LW4	Lasail West area	Waste	Fines to boulder	Dark greenish grey		Py disseminated	Dry
LT-A1	Aarja area	Waste	Fines to pebble	Light grey to tan	Oxidized		Gausen stockpile, dry
LT-A2	Aarja area	Waste	Fines to cobble	Red to reddish brown			Oberburden, dry
LT-A3	Aarja area	Waste	Fines to boulder	Yellowish to dark grey		Py disseminated	Dry
LT-A4	Aarja area	Waste	Fines to cobble	Brown to poly-color	Oxidized		Top soil, dry
LT-A5	Aarja area	Waste	Fines to boulder	Dark grey, to grey/green		Py disseminated	Dry
LT-B1	Bayda area	Waste	Fines to boulder	Dark greenish grey		Py disseminated	Oberburden, basalt; dry
LT-B2	Bayda area	Waste	Fines to boulder	Reddish brown	Gossanized		Dry

Py: pyrite

Table 6.4 Analysis Result of Leaching Test (1)

Sample No.	Area name	Depth (m)	Material	Neutralization potential	Total sulphur content	Maximum potential acidity
				CaCO ₃ Equivalent (Tons/1000t of dried material)	%	CaCO ₃ Equivalent (Tons/1000t of dried material)
LT-D1A1	Tailing dam area	0-10m	Tailings	2.59	20.43	638.44
LT-D1A2	Tailing dam area	10-20m	Tailings	14.61	26.46	826.88
LT-D1A3	Tailing dam area	20-30m	Tailings	25.71	20.98	655.63
LT-T/D	Tailing dam area	Surface	Eolian tailings	7.70	19.25	601.56
LT-D1B1	Tailing dam area	0-3.60m	Tailings	17.87	21.22	663.13
LT-D1B2	Tailing dam area	3.60-10m	Tailings	11.10	11.86	370.63
LT-D1B3	Tailing dam area	10-20m	Tailings	2.49	13.58	424.38
LT-D1B4	Tailing dam area	20-30m	Tailings	14.91	27.36	855.00
LT-L1	Lasail area	Surface	Wastes	7.30	13.47	420.94
LT-L2	Lasail area	Surface	Wastes	11.40	10.26	320.63
LT-LW1	Lasail West area	Surface	Wastes	9.56	20.09	627.81
LT-LW2	Lasail West area	Surface	Wastes	14.45	1.15	35.94
LT-LW3	Lasail West area	Surface	Wastes	14.43	0.38	11.88
LT-LW4	Lasail West area	Surface	Wastes	22.15	0.16	5.00
LT-A1	Aarja area	Surface	Wastes	9.11	4.61	144.06
LT-A2	Aarja area	Surface	Wastes	9.11	4.56	142.50
LT-A3	Aarja area	Surface	Wastes	11.26	29.16	911.25
LT-A4	Aarja area	Surface	Wastes	75.41	0.84	26.25
LT-A5	Aarja area	Surface	Wastes	43.48	3.10	96.88
LT-B1	Bayda area	Surface	Wastes	57.04	0.13	4.06
LT-B2	Bayda area	Surface	Wastes	28.66	0.67	20.94

Table 6.4 Analysis Result of Leaching Test (2)

No.	Sample No.	Area	Material	pH	E.C. S/m	Hg mg/L	Cd mg/L	Cr mg/L	As mg/L	Pb mg/L	Cu mg/L	Mn mg/L	Fe mg/L	Ni mg/L	Sn mg/L	Zn mg/L	SO ₄ mg/L	Cl mg/L
1	LT-D1A1	Tailing dam area	Tailings	4.52	0.49	0.0011	0.124	0.178	0.072	0.531	62.70	2.81	1.40	0.07	0.01	100.15	2994	369
2	LT-D1A2	Tailing dam area	Tailings	6.70	0.19	0.0007	<0.001	0.088	0.085	0.327	0.03	2.24	0.03	0.06	0.01	0.86	1062	35
3	LT-D1A3	Tailing dam area	Tailings	6.97	0.22	0.0013	0.022	0.097	0.077	0.413	0.19	4.14	0.03	0.07	0.01	28.39	983	149
4	LT-TD	Tailing dam area	Eolian tailings	3.97	0.93	0.0017	0.243	0.255	0.068	0.267	165.50	120.70	26.24	2.54	0.01	10.10	4991	865
5	LT-D1B1	Tailing dam area	Tailings	3.99	0.87	0.0024	0.118	0.246	0.028	0.285	92.90	74.70	4.86	1.39	0.01	10.08	3787	1050
6	LT-D1B2	Tailing dam area	Tailings	5.01	0.36	0.0018	0.039	0.161	0.031	0.242	4.72	4.69	0.07	0.14	0.01	8.99	1855	241
7	LT-D1B3	Tailing dam area	Tailings	5.57	0.34	0.0019	0.025	0.146	0.054	0.321	1.21	6.75	0.15	0.16	0.01	7.35	1713	255
8	LT-D1B4	Tailing dam area	Tailings	5.98	0.33	0.0013	0.045	0.146	0.048	0.372	1.50	8.07	0.14	0.21	0.01	7.14	1694	227
9	LT-L1	Lasail Mine area	Waste	3.72	0.51	0.0003	0.048	0.276	0.042	0.385	78.40	23.07	90.09	0.57	0.01	9.45	5152	14
10	LT-L2	Lasail Mine area	Waste	3.47	0.88	0.0004	0.037	0.434	0.050	0.541	841.40	34.11	103.4	1.24	0.01	8.62	10472	11
11	LT-LW1	Lasail West Mine area	Waste	3.21	0.65	0.0004	0.104	0.370	0.062	0.457	840.00	13.95	104.5	2.40	0.01	9.98	5962	11
12	LT-LW2	Lasail West Mine area	Waste	7.00	0.06	0.0010	<0.001	0.119	0.071	0.138	0.06	0.12	0.09	0.06	0.01	0.13	199	14
13	LT-LW3	Lasail West Mine area	Waste	7.10	1.47	0.0009	<0.001	0.142	0.073	0.242	0.04	0.08	0.11	0.07	0.01	0.05	746	14
14	LT-LW4	Lasail West Mine area	Waste	7.44	0.04	0.0009	<0.001	0.118	0.065	0.168	0.05	0.02	0.12	0.06	0.02	<0.01	82	14
15	LT-A1	Aarja Mine area	Waste	4.17	0.39	0.0007	1.075	0.188	0.058	0.237	53.80	44.71	31.84	1.57	0.02	9.87	2800	11
16	LT-A2	Aarja Mine area	Waste	3.73	0.25	0.0008	0.048	0.159	0.042	0.213	67.00	8.82	18.59	0.61	0.01	7.90	1393	35
17	LT-A3	Aarja Mine area	Waste	2.26	1.51	0.0005	1.115	0.949	0.112	0.627	810.10	25.05	9747	1.41	0.02	10.67	10388	7
18	LT-A4	Aarja Mine area	Waste	7.50	0.05	0.0005	<0.001	0.135	0.085	0.138	0.11	0.39	3.77	0.07	0.02	2.11	115	14
19	LT-A5	Aarja Mine area	Waste	7.21	0.08	0.0004	<0.001	0.141	0.074	0.227	0.04	0.12	0.12	0.05	0.01	<0.01	273	11
20	LT-B1	Bayda Mine area	Waste	7.51	0.04	0.0004	<0.001	0.138	0.068	0.285	0.04	<0.01	0.15	0.05	0.01	0.04	58	18
21	LT-B2	Bayda Mine area	Waste	7.21	0.15	0.0004	<0.001	0.166	0.070	0.310	0.05	1.35	0.11	0.07	0.02	0.56	770	11

Slaked lime was used to regulate the pH at 9.3 during the milling process. Also, the interior of tailing impoundment remains under reducing conditions and sulfur oxidation is not occurring or is occurring very slowly.

There is sufficient buffering capacity in the rock-water reaction due to the calcreted beds in the wadi sediments and basement rocks. In the case where oxidation might be promoted from the top layer to the deeper zones, it can be presumed that a great amount of acid might be produced and heavy metals might be dissolved. If this were the case, each component should exhibit concentrations similar to those obtained in the leaching tests presented in Table 6.3. However, the actual leachate being produced in the tailings dam is probably mitigated by buffering due to the initial surplus of lime from the milling process, as well as by the buffering capacity of the calcreted sediments and basement rocks. Nevertheless, the seepage from the tailings dam exhibits elevated metal concentrations exceeding the Omani standards, especially for cadmium and lead. For this reason, the seepage needs to be collected and treated.

6.2 PS-2 Pumping Station

A soil investigation around the PS-2 pumping station was executed during the Study. Soil was sampled from the ground surface and the soluble Cl component was analyzed. Analysis results are designated as CIS-49 to 68 in Table 3.2 and the distribution map of Cl is shown in Figure 6.2.

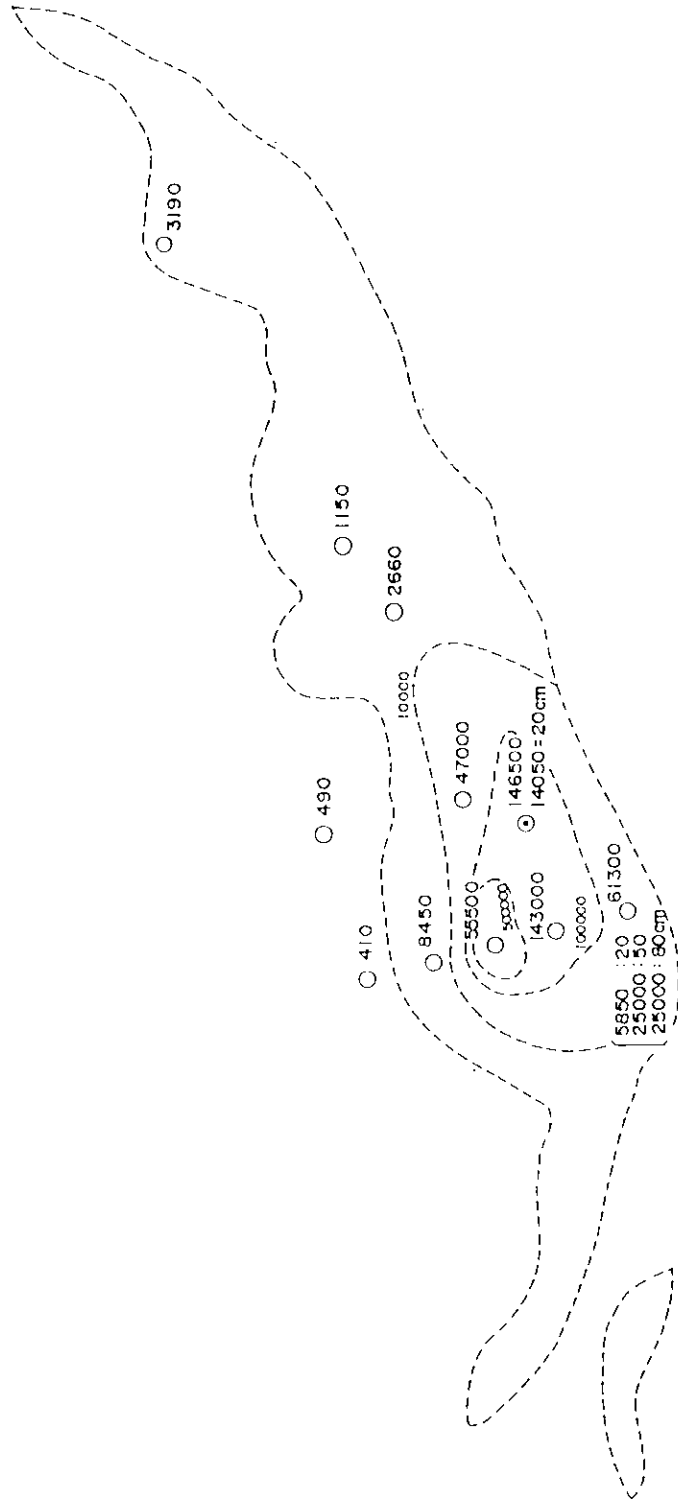
Seawater for the dressing plant was pumped up from Majis Jetty and a considerable amount of seawater leaked at booster pumping station PS-2. The JICA team confirmed the fact that contaminated soil with high salt concentration still exists in the area upstream of PS-2, and is distributed for about 200 to 350 m along the wadi. The existence of crystallized salt (halite) was observed as far as 0.2 to 0.8 m in depth below ground surface. However, Cl concentrations in groundwater in the hand dug at PS-2 and bore hole DH-3 ranged from 6,000 to 6,500 mg/L. Hence, it appears that a high Cl concentration zone was formed for about 300m in length around the PS-2 area and that this zone is limited to the top thin layer only, without direct contact with groundwater.

6.3 Waste Dump Areas

Mine wastes were deposited in dumping areas around each abandoned mine, including Lasail, Lasail West, Aarja and Bayda (Figure 6.3) .

The conditions of the waste dumps around each mine are presented in Figures 6.4 to 6.7. The feature of the wastes in each mine is summarized in Table 6.5.

- Salt contaminated zone
- > 410 mg / kg : c2
 - > 10000 mg / kg : c2
 - > 100000 mg / kg : c2
 - > 500000 mg / kg : c2



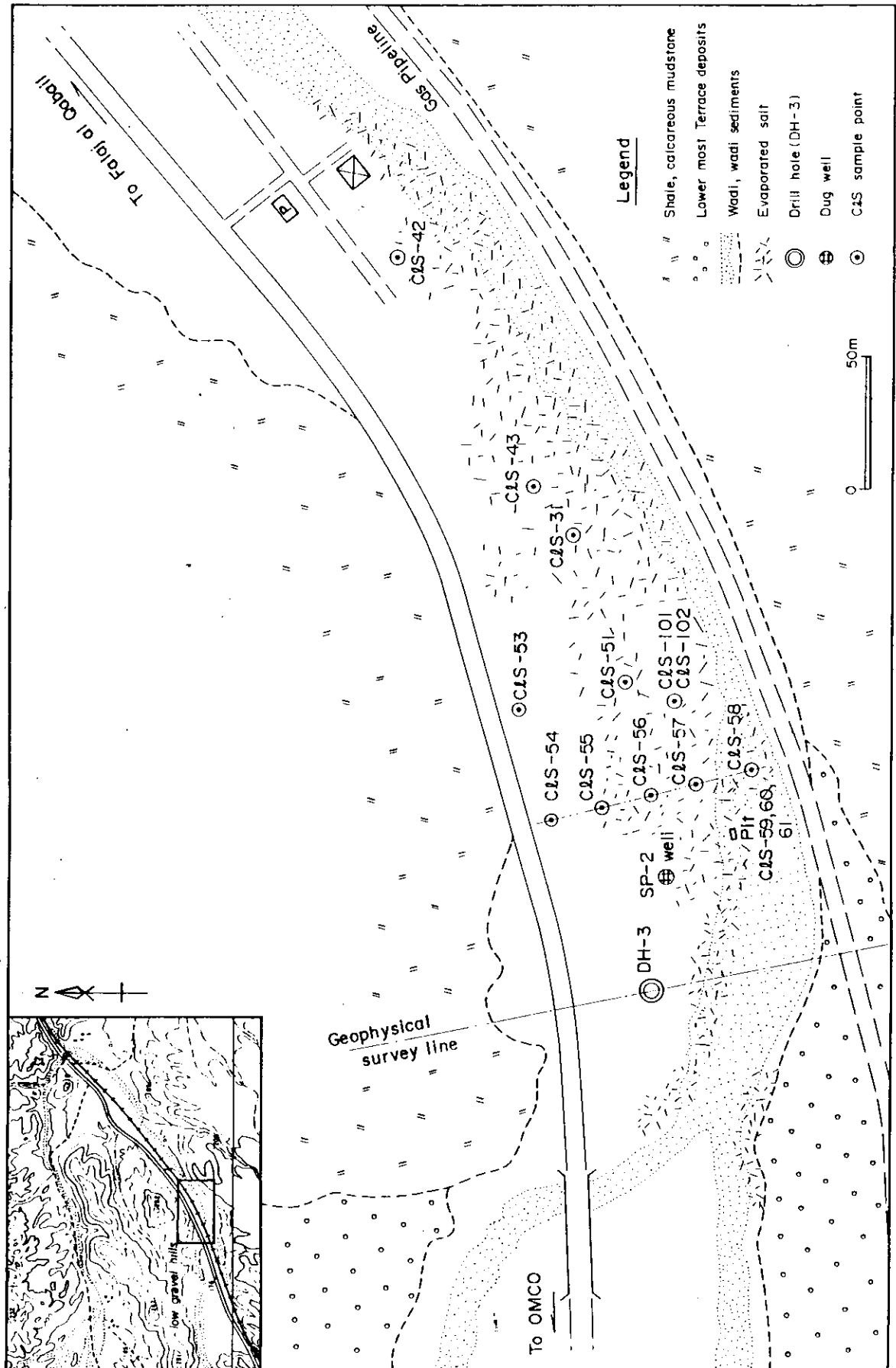


Figure 6.2 Cl Concentration Contour Map in the Soil at SP-2 Point

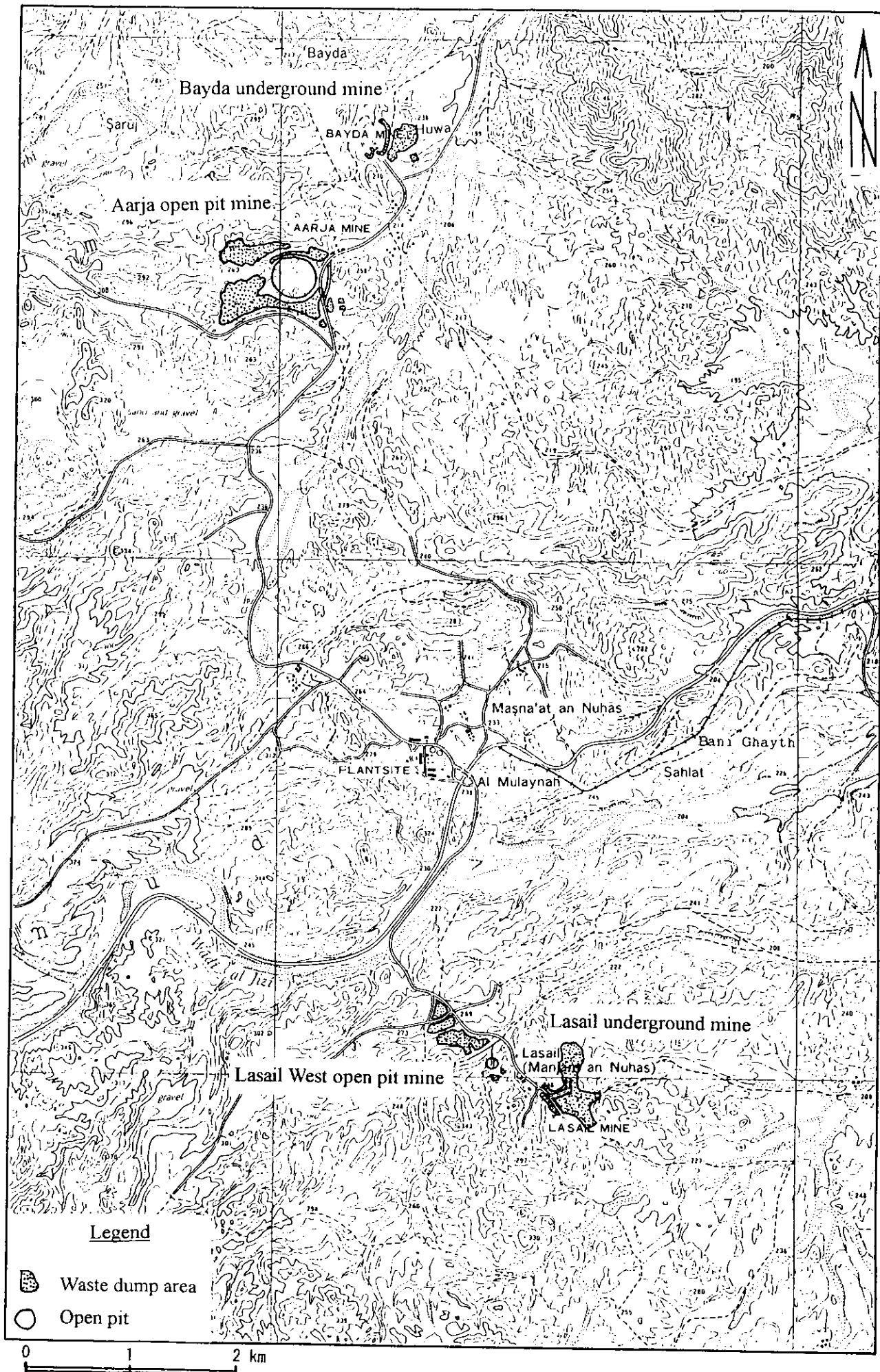


Figure 6.3 Distribution of Mine Wastes
6-10

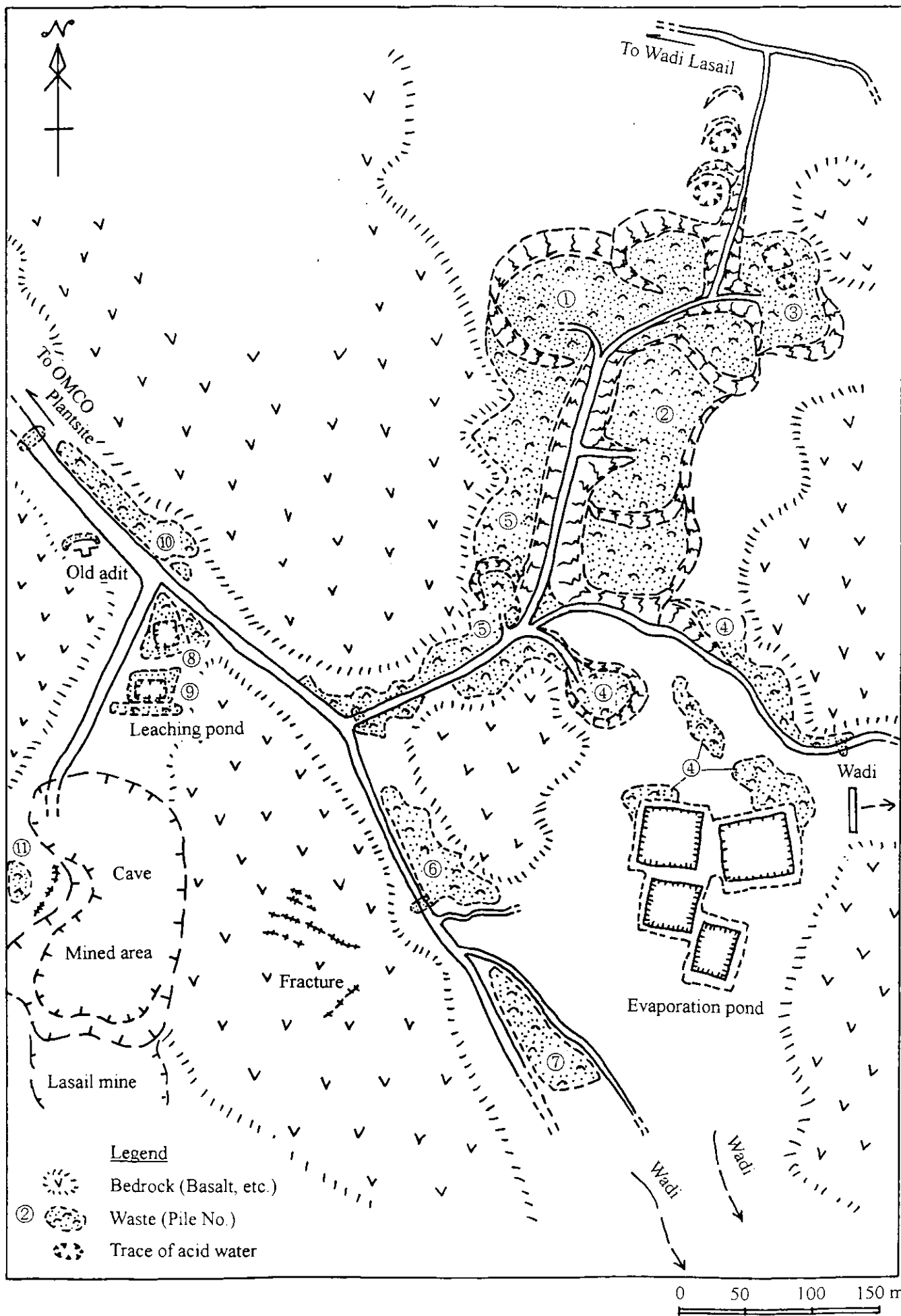


Figure 6.4 Distribution of Wastes of Lasail Mine

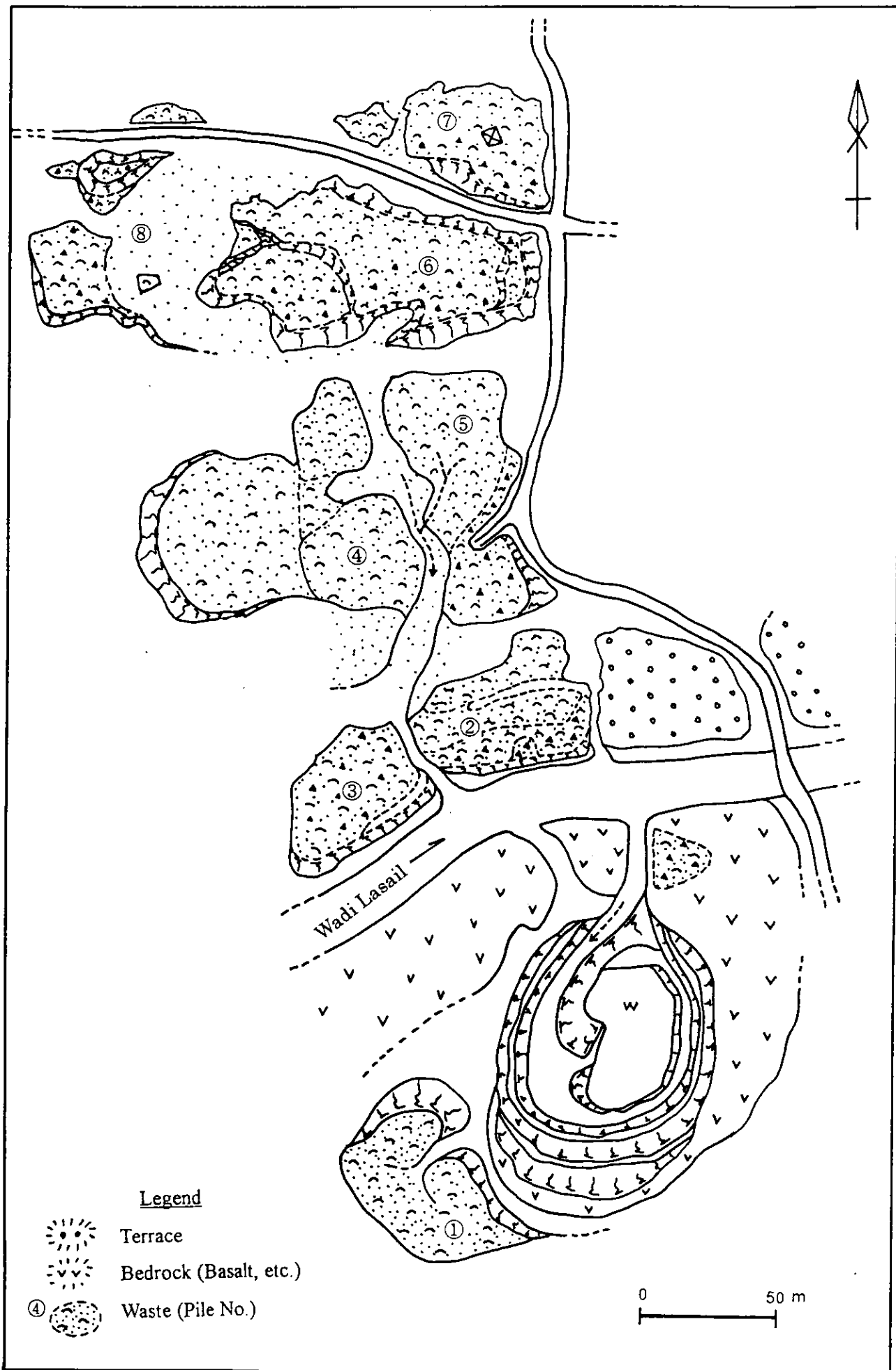


Figure 6.5 Distribution of Wastes of Lasail West Mine
6-12

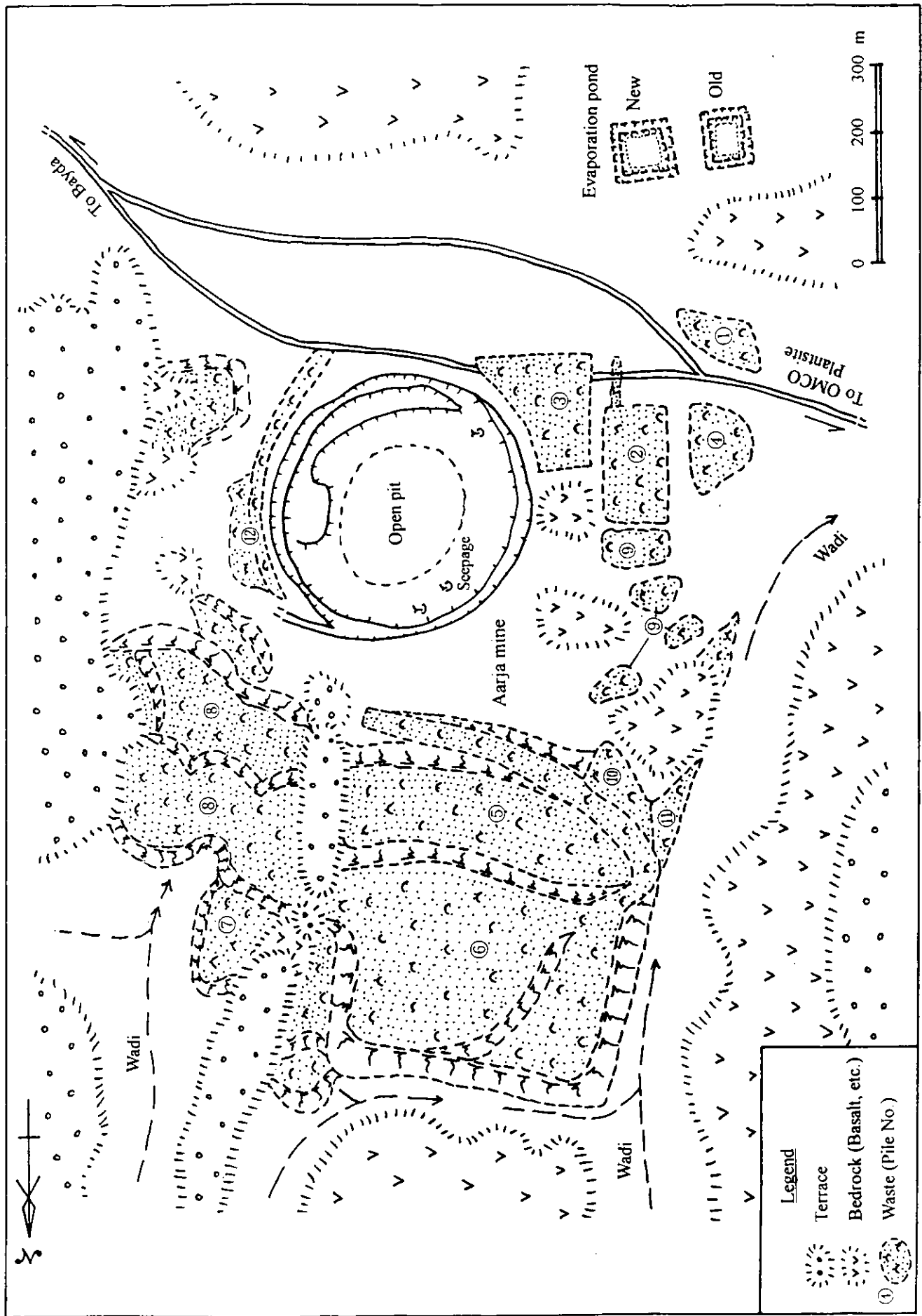


Figure 6.6 Distribution of Wastes of Aarja Mine

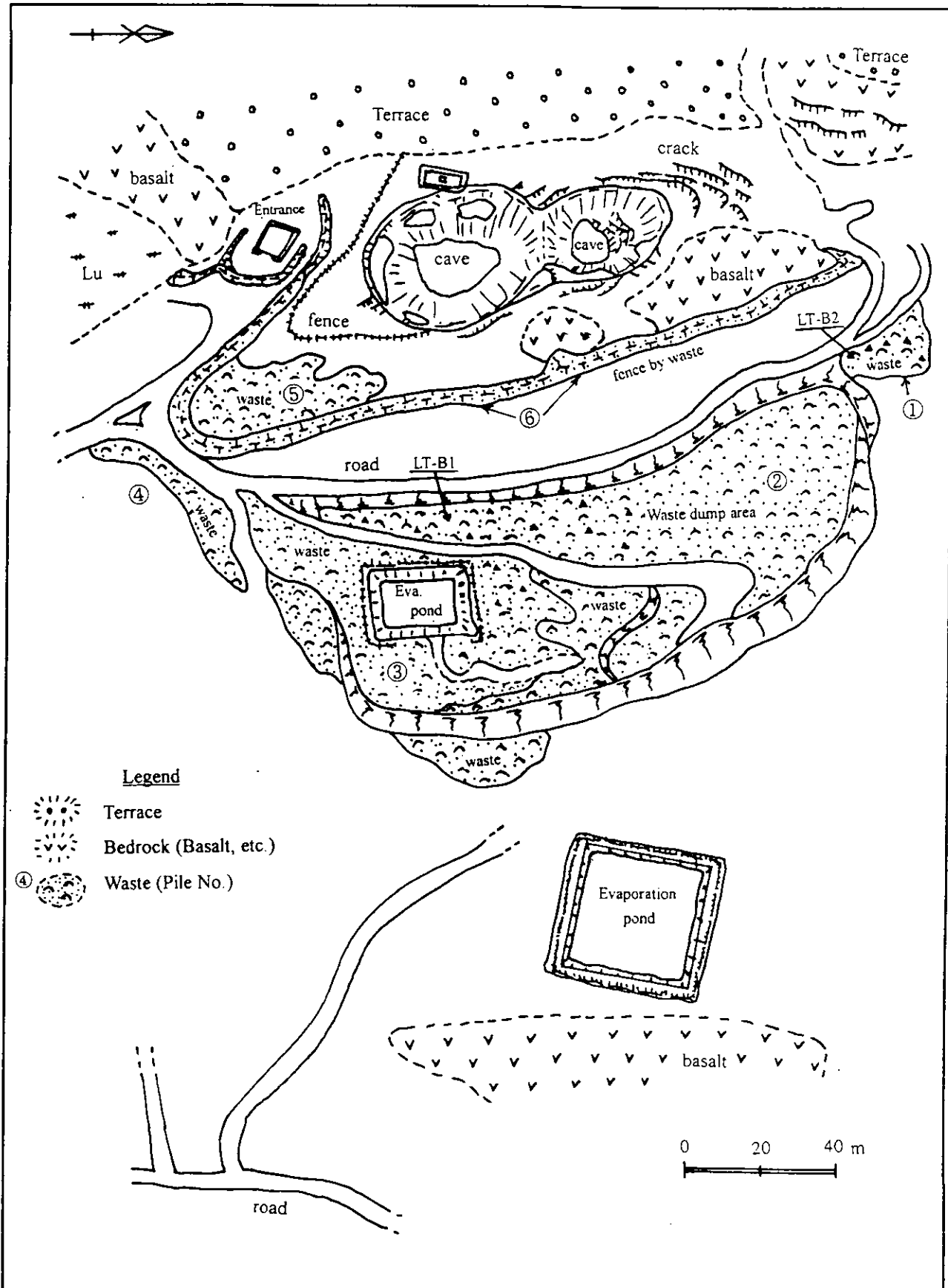


Figure 6.7 Distribution of Wastes of Bayda Mine

Table 6.5 Mine Wastes

Location #1	Description	Approximate Dimensions			Approximate volume (m ³)	Particle size	Color / Physical condition	Sample No.
		Length (m)	Width (m)	Height (m)				
Lasail underground mine								
1	Waste rock pile, weak py disseminated	90	120	14	151,200	Fines to boulder	Grey to dark gray outside, yellowish orange under surface	LT-L1
2	Waste rock pile, weak py disseminated	160	60	12	115,200	Fines to boulder	Grey to dark gray outside, yellowish orange under surface	None
3	Waste rock pile, weak py disseminated	70	70	8	39,200	Fines to boulder	Grey to dark gray outside, yellowish orange under surface	None
4	Waste rock pile, weak py disseminated	320	60	4	76,800	Fines to boulder	Grey to dark gray	None
5	Waste rock pile, weak py disseminated	280	70	6	117,600	Fines to boulder	Grey to dark gray	None
6	Waste rock pile	65	50	4	13,000	Fines to cobble	Grey to gray/green	None
7	Small leach pads	120	40	4	19,200	Gravel to small cobbles	Grey to dark gray, reddish grey	LT-L2
8	Settling pond fines, reddish overburden	70	50	3	10,500	Fine grained materials	Light tan, some piled up, others left in pond bottom	None
9	Interburden foundation area	70	70	3	14,700	Fines to cobble	Grey to gray/green	None
10	Waste rock pile, weak py disseminated	60	20	3	3,600	Fines to cobble	Reddish, rough graded	None
11	Boulder pile	60	50	3	9,000	Large boulders	Greenish grey	None
	Total				570,000 m ³			
Bayda underground mine								
1	Waste rock pile, pyrite disseminated	40	30	2	2,400	Fines to boulder	Reddish to brownish/greenish grey, oxidized	LT-B2
2	Waste rock pile, strongly pyritic dissem.	70	70	6	29,400	Fines to boulder	Greenish to yellowish grey, oxidized in surface, weathered	None
3	Waste rock pile, strongly pyritic dissem.	60	100	5	30,000	Fines to boulder	Greenish to yellowish grey, oxidized in surface, evaporation pits	LT-B1
4	Overburden	20	30	3	1,800	Boulders	Brown to reddish/greenish grey, partly oxidized	None
5	Waste of gossalized area, oxidized soil	30	40	2	2,400	Fines to cobble	Reddish to brownish grey, oxidized	None
6	Waste, used for fence of the mined area	500	3	2	3,000	Fines to cobble	Brownish to greenish grey, oxidized, including evaporation pits	None
	Total				69,000 m ³			
Aarja open pit mine								
1	Gausen stockpile	120	40	4	19,200	Fines to small gravel	Light gray to light tan, end-dumped in small piles	LT-A1
2	Reddish overburden, some sulfide	150	90	3	40,500	Fines to cobble	Red to reddish brown, end-dumped in piles	LT-A2
3	Low grade ore stockpile, w/sulfide	100	70	4	28,000	Fine to large cobbles	Yellowish to gray or dark gray, dumped in random piles	LT-A3
4	Top soil mixed with rock	70	90	6	37,800	Fine to large cobbles	Brown soil mixed with rocks of various colors, uniform pile	LT-A4
5	Waste rock pile, No. 1, w/pyrite	340	270	15	1,377,000	Cobble to boulder	Dark gray to gray/green rock, huge stock pile	LT-A5
6	Waste rock pile, No. 2, w/pyrite	460	350	12	1,932,000	Fines to boulder	Gray with streaks of yellow, end-dumped graded flat on top	None
7	Waste rock pile, No. 3, w/pyrite	160	140	6	134,400	Cobble to boulder	Dark gray to gray/green rock, huge stock pile	None
8	Waste rock pile, No. 4, w/pyrite	360	260	12	1,123,200	Cobble to boulder	Dark gray to gray/green rock, huge stock pile	None
9	Reddish overburden	-	-	-	4,000	Fines to cobble	Red to reddish brown, multiple small piles	None
10	Brown overburden	120	45	4	21,600	Fine soil to large cobbles	Brown soil mixed with rocks of various colors, uniform pile	None
11	Fine gray sulfide overburden	220	75	7	115,500	Fines with small gravel	Uniformly gray, dumped at edge of Wadi and graded flat.	None
12	Light tan pit berm	610	15	6	54,900	Fines with small gravel	Light gray to light tan, fashioned into a berm on N side of pit	None
	Total				4,888,100 m ³			
Lasail West open pit mine								
1	Waste rock pile, overburden, basalt	20	40	2	1,600	Pebble to boulder	Greenish grey, partly brown oxidized	None
2	Overburden with oxidized rocks	40	60	1	2,400	Fines to boulder	Greenish grey to reddish brown, partly py dissem.	LT-LW3
3	Waste rock pile, overburden, basalt	80	60	2	9,600	Fines to boulder	Greenish grey to dark grey, partly py dissem.	LT-LW4
4	Waste rock pile, pyrite disseminated	70	80	2.5	14,000	Fines to boulder	Greenish grey to dark grey, partly py dissem.	None
5	Waste rock pile, overburden, basalt	300	120	8	288,000	Pebble to boulder	Greenish grey, partly oxidized	LT-LW2
6	Brown to yellow overburden, py dissem.	130	70	4	36,400	Fines to cobble	Brownish to yellowish grey, strongly oxidized	LT-LW1
7	Brown to yellow overburden, py dissem.	50	60	3	9,000	Fines to cobble	Brownish to yellowish grey, strongly oxidized	None
8	Brown to yellow overburden, py dissem.	100	60	3	18,000	Fines to cobble	Brownish to yellowish grey, strongly oxidized	None
	Total				379,000 m ³			
	Grand total				5,906,100 m ³			

6.3.1 Lasail Mine

The distribution of waste around the Lasail Mine is presented in Figure 6.4. Waste was mainly dumped in the valley between the western to northeastern hilly zone. The valley belongs to three tributaries of Wadi al Owaina water system. Hence, seepage out of the waste dumps drains into these three tributaries toward the north, east and south. An embankment is constructed in the eastern tributary in order to control heavy rainwater. In the case of other tributaries, there are no mine drainage control facilities, such as ditches, etc., but the mine mouth was closed by an earthen embankment to prevent human access into the mine.

The total tonnage of waste was estimated to be 600,000 m³ and is piled 3 to 14 m in height. Waste consists of mainly greenish gray colored basalt in sizes ranging from large cobbles to gravel. Almost all basaltic rocks are influenced by pyrite mineralization and some of them were strongly oxidized presenting a yellowish brown color (LT-L1). On the south side of the waste dump there are four dried up square evaporation ponds ranging in size from 30 to 50 m and 1.5 m in depth. These dry evaporation ponds are filled by sediment like gypsum. They were already decommissioned.

6.3.2 Lasail West Mine

The distribution of waste around Lasail West Mine is presented in Figure 6.5. Waste was dumped along Wadi Lasail from the terrace on the opposite side of the mine. A part of the waste was also dumped on the west side of the open pit. Seepage out of the waste dump flows into Wadi Lasail. There are no mine drainage control facilities, but the mine entrance was closed by an earthen embankment to prevent human access into the mine.

The total tonnage of waste was estimated to be 400,000 m³ and is piled from 2 to 6 m in height. Waste consists of mainly greenish gray to yellowish brown colored basalt with sizes ranging from large cobbles, to fine gravel, to clay (LT-LW1). In the waste impoundment area there are many ruins of seepage ditches but all of them are dry.

6.3.3 Aarja Mine

The distribution of waste around Aarja Mine is presented in Figure 6.6. Waste was dumped along valleys on the north to southwest side of the mine. Part of waste was also dumped on the east side of the open pit. This waste impoundment is located in the catchment areas of two tributaries of Wadi Bani Umar al Gharbi, and part of the seepage from the waste piles flows into these tributaries.

In addition, some of the seepage out of the waste dump flows out toward the south and flows into the open pit. There are no mine drainage control facilities, but the open pit was closed by an earthen embankment and wire net fence to prevent human access into the mine.

The total tonnage of waste was estimated to be 5,000,000 m³ and consists of many kinds of colored rocks, such as greenish gray, brownish gray, yellow, yellowish brown and reddish brown, containing low grade ore with sizes ranging from large cobbles to fine gravel, as well as some clay. The wastes on the west to southwest side are especially fine-grained material. A part of the overburden was piled up separately.

Almost all waste rocks are more or less influenced by pyrite mineralization and some of them were strongly oxidized presenting yellowish brown or reddish brown color and weathered to be soft. On the southwest side of the waste dump, there are two dried up square evaporation ponds ranging in size from 60 to 80 m and having a depth of approximately 2 m. They were used for mine water disposal but are still being used for wastewater out of the smelter.

6.3.4 Bayda Mine

The distribution of waste around Bayda Mine is presented in Figure 6.7. Waste was dumped along a flat zone to the east of some hilly land. This waste impoundment is located in a tributary of Wadi Bani Umar al Gharbi. All seepage out of waste dump flows into the wadi.

There are no mine drainage control facilities, but the mine area was closed by an earthen embankment and wire net fence to prevent human access into the mine.

The total tonnage of waste was estimated to be 70,000 m³ and consists of greenish gray, brownish gray, yellowish brown and reddish brown colored basaltic rocks. Almost all waste rocks are more or less influenced by pyrite mineralization and some of them were strongly oxidized presenting yellowish gray or yellowish brown color and weathered to be soft.

On the east side of waste dump there are two dried up square evaporation ponds ranging in size from 25 to 40 m, having depths between 1 to 2 m and filled by sediments like gypsum. They were used for mine water disposal but already decommissioned.

6.3.5 Result of Leaching Tests

(1) Lasail Mine

The sulfur content of Lasail Mine waste is relatively high, as much as 10 to 13 % due to pyrite mineralization (Table 6.3 and Figure 6.8 (1), (2)). The Maximum Potential Acidity (MPA) is also high, ranging from 320 to 420 t per 1,000 t. However, the net neutralization potential is slightly lower.

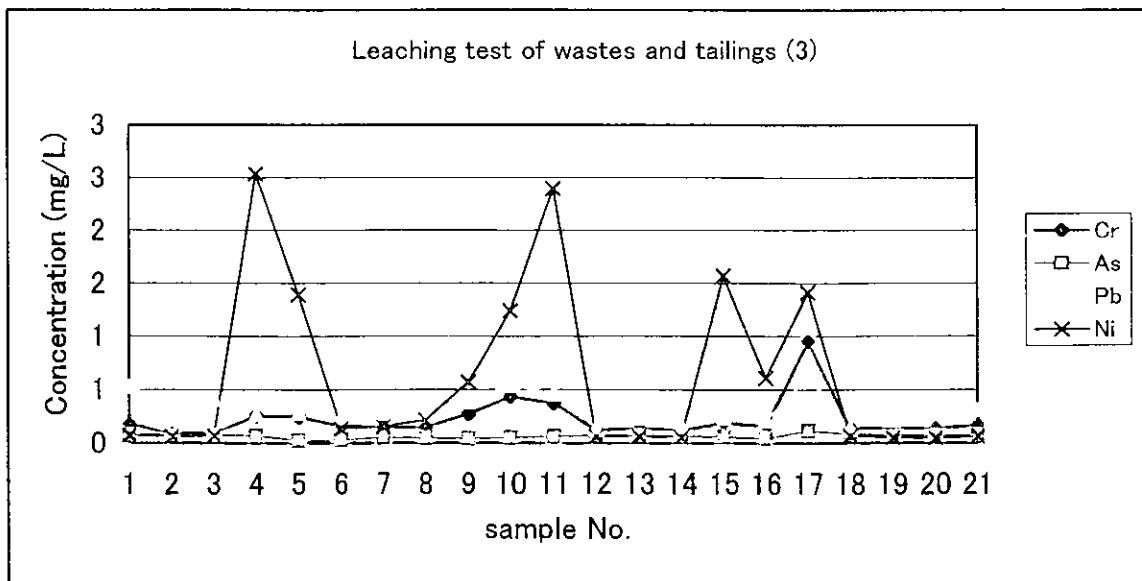
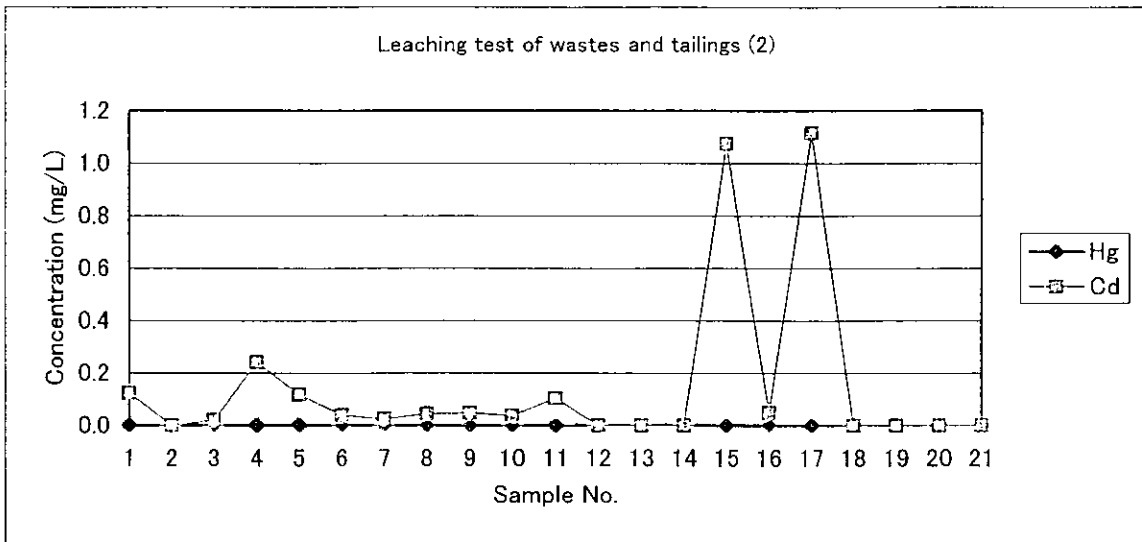
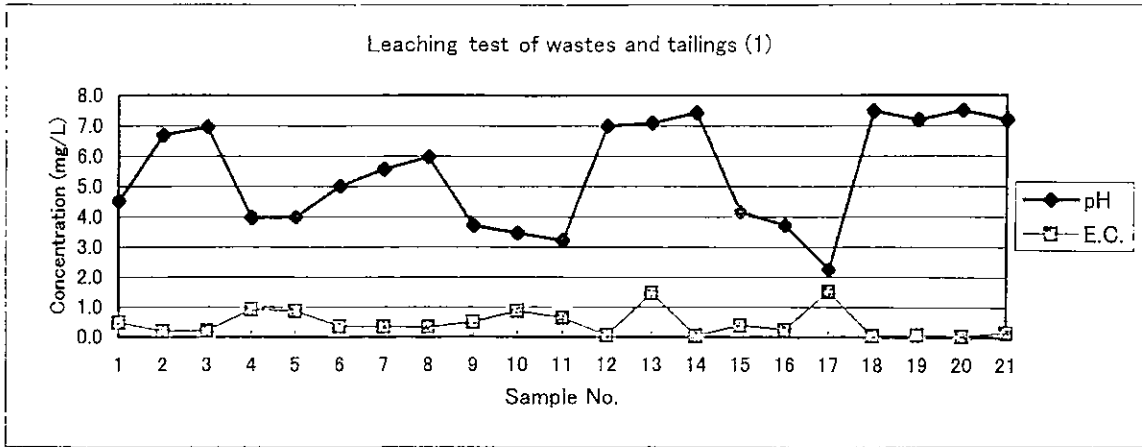
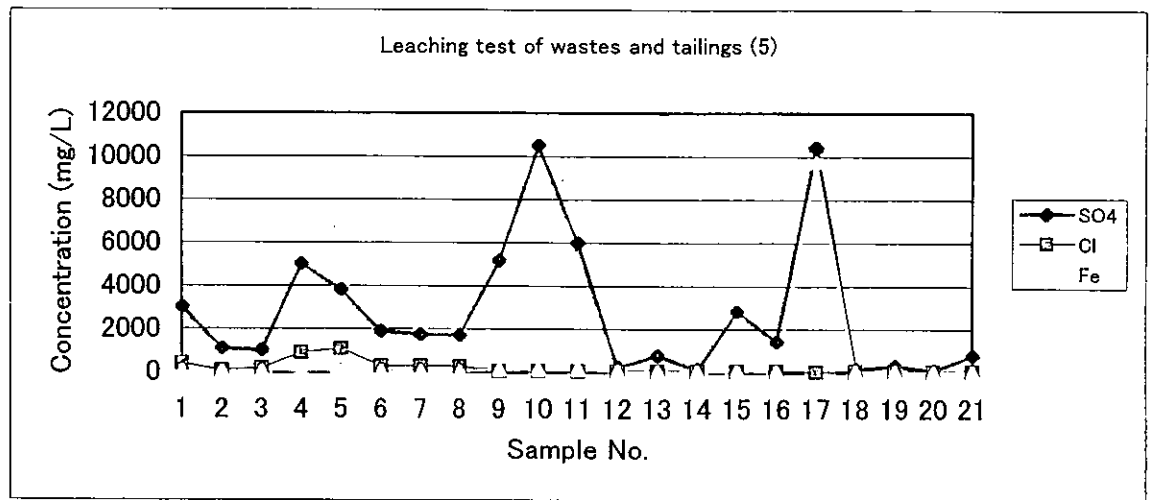
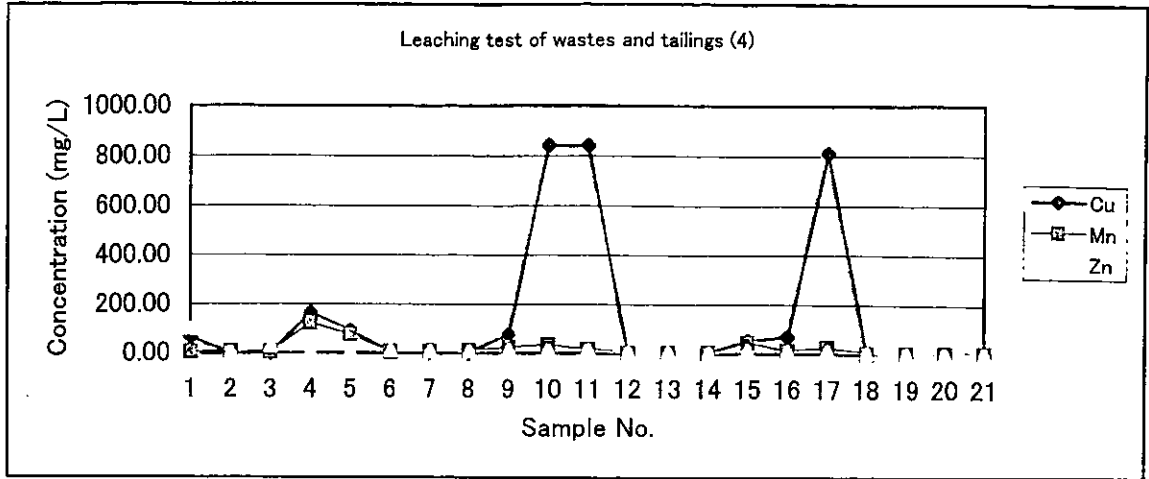


Figure 6.8 Analysis Results of Leaching Test of Tailings and Wastes (1)



No.	Sample No.
1	LT-D1A1
2	LT-D1A2
3	LT-D1A3
4	LT-D1B1
5	LT-D1B2
6	LT-D1B3
7	LT-A1
8	LT-A2
9	LT-L1
10	LT-L2
11	LT-LW1

No.	Sample No.
12	LT-LW2
13	LT-LW3
14	LT-LW4
15	LT-A1
16	LT-A2
17	LT-A3
18	LT-A4
19	LT-A5
20	LT-B1
21	LT-B2

Figure 6.8 Analysis Results of Leaching Test of Tailings and Wastes (2)

The pH value of the effluent exhibits strong acidity of 3.4 to 3.7 with high dissolved metals concentrations from the low grade ore. Greenish gray colored basaltic country rock with less pyrite mineralization is believed to have neutral effluent with low dissolved heavy metal ions, based on other experience.

It is presumed that a part of waste might produce acid drainage containing leached heavy metals in the future as a result of continued oxidation.

(2) Lasail West Mine

The sulfur content in waste from the Lasail West Mine ranges from 0.16 to 20.09% for barren country rock and rocks containing low-grade ore, respectively. The MPA of the low-grade ore is very high ranging up to 627 tons of acidity per 1,000 tons of waste. However, the barren waste presented a low MPA of 5 to 36 tons per 1,000 tons. The differences in the MPA of the different rock types can be partly explained by the degree of oxidation that has already taken place.

In the materials exhibiting low MPA, much of the sulfur may have been previously oxidized. .

The pH value of effluent presents strong acidity of 3.2 in case of low grade ore but neutral in case of other simple waste. Low grade ore showed also high solubility of heavy metals, including Cu, SO₄, Ni, Fe and Zn. Other wastes present low metals concentrations.

It is also presumed that a part of the waste might produce acid drainage containing leached heavy metals in the future if pyrite oxidation continues to occur. However, due to the arid climate and the buffering capacity of surficial sediments, acidic drainage will probably not migrate very far beyond the waste piles. Nevertheless, some of the leached metals will be transported downstream or into local groundwater by storm water runoff.

(3) Aarja Mine

The sulfur content in Aarja Mine waste ranges from 0.84 in low grade country rock to 29.16 % in rocks containing oxidized low-grade ore. The MPA of the low-grade ore is very high at 911 t per 1,000 t. However, the barren waste exhibited a low MPA of 26 to 144 t per 1,000 t. The reason for the lower MPA in the barren materials is that the initial sulfur content is low and much of the initial sulfur content is already oxidized.

The pH value of effluent is strongly acidic at 2.2 to 4.2 in the case of the low-grade ore and LT-A1 and LT-A2, but the pH is neutral in cases of LT-A4 and LT-A5. Low-grade ore also exhibited high solubility of heavy metals, including Cu, SO₄, Ni, Fe and Zn, while other waste exhibit low metals concentrations.

It is also presumed that a part of the waste might produce acid drainage containing leached heavy metals in the future if pyrite oxidation continues to occur.

(4) Bayda Mine

The sulfur content in Bayda Mine waste was rather low, ranging from 0.1 to 0.67 %, indicating the wastes might be in the final stages of sulfur oxidation. This situation coincides with field observations. The MPA of the low-grade ore is as low as 4 to 21 t per 1,000 t.

Based on leaching test results, seepage from the Bayda waste piles exhibits neutral pH and low metals concentrations. Hence, the potential for this material to leach metals in to storm runoff and local groundwater appears to be relatively low.

6.4 Underground and Open Pit Mined-out Areas

6.4.1 Mine Water of Abandoned Underground Mines

Lasail and Bayda Mines were mined by underground mining methods and so the quality of the mine water cannot be readily evaluated. The entrances to the mines have been blocked by fences and earthen embankments to prevent human access. Hence, it is impossible to collect mine water samples.

6.4.2 Mine water of Abandoned Open Pits

Lasail West and Aarja Mines were mined by open pit mining methods. At present, both these pits are filled with mine water. Although the mine entrances have been blocked by fences and earthen embankments, it is still possible to obtain mine water samples through controlled access points.

6.4.3 Water Quality of Mine Water

Open pit mine water samples were taken from Lasail West and Aarja three times during the Study, i.e. June, September and November 2000 to check water quality. Analytical results of the mine water samples are presented in Table 6.6.

- Water temperature ranged in 26.9 to 32.8 °C. The November temperature was 6 °C lower than the June temperature.
- The Arja pH ranged from weakly acidic to neutral at 5.8 to 7.45 and pH of Lasail West mine water was somewhat higher.
- Electric Conductivity presented a stable range of 0.23 to 0.57 S/m. Values at Aarja was higher.
- Hg concentrations ranged from 0.0004 to 0.0028 mg/L, with the higher values observed at Aarja. Values of Aarja exceeded that of Omani drinking water standards (0.001 mg/L).

Table 6.6 Water Quality of Mine Water

(1) First Monitoring in June, 2000

Location	Sample Type	Temp.	pH	ORP	E.C.	Hg	Cd	Cr	As	Pb	Cu	Mn	Fe	Ni	Sn	Zn	SO ₄	Na	K	Mg	Ca	Cl	HCO ₃
		(C.)		(mV)	S/m	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
Aarja	Surface Water	32.8	7.41	126	0.57	0.0028	0.09	0.03	0.006	0.01	0.07	27.5	0.04	1.01	<0.01	3.55	2046	572	1.40	88.0	663	752	58
Lasail West	Surface Water	30.2	5.80	241	0.26	0.0004	0.08	<0.01	0.003	<0.01	2.37	25.3	<0.01	0.40	<0.01	1.85	1102	177	0.70	70.0	277	100	10

(2) Second Monitoring in September, 2000

Location	Sample Type	Temp.	pH	ORP	E.C.	Hg	Cd	Cr	As	Pb	Cu	Mn	Fe	Ni	Sn	Zn	SO ₄	Na	K	Mg	Ca	Cl	HCO ₃
		(C.)		(mV)	S/m	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
Aarja	Surface Water	31.4	7.45	158	0.55	0.0025	0.01	0.05	0.006	0.01	0.10	24.9	0.05	1.36	<0.01	3.89	2026	560	2.00	93.0	665	784	55
Lasail West	Surface Water	31.5	6.42	268	0.23	0.0005	0.01	0.01	0.004	<0.01	2.33	22.3	0.01	0.38	<0.01	1.73	1138	160	1.00	74.0	274	108	11

(3) Third Monitoring in November, 2000

Location	Sample Type	Temp.	pH	ORP	E.C.	Hg	Cd	Cr	As	Pb	Cu	Mn	Fe	Ni	Sn	Zn	SO ₄	Na	K	Mg	Ca	Cl	HCO ₃
		(C.)		(mV)	S/m	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
Aarja	Surface Water	26.9	7.32	89	0.54	0.0027	0.01	0.05	0.005	0.01	0.12	24.7	0.06	1.42	<0.01	3.93	2018	610	2.00	87.0	654	842	54
Lasail West	Surface Water	26.9	6.49	220	0.24	0.0006	0.01	0.01	0.005	0.01	2.14	22.4	0.02	0.35	<0.01	1.63	1200	170	1.00	83.0	285	124	12

(4) Average among three monitoring results

Location	Sample Type	Temp.	pH	ORP	E.C.	Hg	Cd	Cr	As	Pb	Cu	Mn	Fe	Ni	Sn	Zn	SO ₄	Na	K	Mg	Ca	Cl	HCO ₃
		(C.)		(mV)	S/m	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
Aarja	Surface Water	30.4	7.39	124	0.55	0.0027	0.034	0.043	0.006	0.01	0.10	25.69	0.05	1.26	<0.01	3.79	2030	581	1.80	89.3	661	793	56
Lasail West	Surface Water	29.5	6.24	243	0.24	0.0005	0.036	0.008	0.004	0.01	2.28	23.33	0.01	0.38	<0.01	1.74	1147	169	0.90	75.7	279	111	11

- Cd concentrations ranged from 0.01 to 0.09 mg/L with the higher values observed at Aarja.
- As concentrations ranged from 0.003 to 0.006 mg/L and did not exceed the Omani drinking water standard (0.05 mg/L) .
- Pb concentrations ranged of <0.01 to 0.01 mg/L and did not exceed the Omani drinking water standard (0.1 mg/L).
- Cu concentrations ranged from 0.07 to 2.37 mg/L with the higher values observed at Lasail West.
- Mn concentrations ranged from 22.3 to 27.5 mg/L.
- Fe concentrations ranged from <0.01 to 0.06 mg/L with the higher values observed at Aarja.
- Ni concentrations ranged from 0.35 to 1.42 mg/L with the higher values observed at Aarja.
- Sn concentrations ranged from <0.01 mg/L
- SO₄ concentrations ranged from 1,102 to 2,046 mg/L with higher values observed at Aarja.
- Na concentrations ranged from 160 to 610 mg/L with the higher values observed at Aarja.
- K concentrations ranged from 0.70 to 2.00 mg/L with the higher values observed at Aarja.
- Mg concentrations ranged from 70 to 93 mg/L and somewhat higher values were observed at Aarja.
- Ca concentrations ranged 274 to 665 mg/L with the higher values observed at Aarja.
- Cl concentrations ranged from 100 to 842 mg/L with much higher values observed at Aarja. It is presumed that brine from the tailings dam may be mixed with the Arja mine water.
- HCO₃ concentrations ranged from 10 to 58 mg/L with the higher values observed at Aarja.
- As a whole, Aarja mine water contained higher dissolved components, especially higher values for Hg, Na, Ca, and Cl. On the other hand, Lasail West water exhibited lower pH values and higher copper content. There was little indication of seasonal fluctuations in any of the analytical parameters observed at either of the mine pools.
- Comparisons of the water quality between both mines and other groundwater and surface water sources will be described in section of Chapter 7.

6.5 Copper Smelter

6.5.1 General Description

(1) History

The copper smelter and electrolytic refinery plant (the so-called tank house) were constructed with the purpose of treating concentrates obtained from Sohar area mines adjacent to the ore dressing plant. The smelter and refinery plant started operation in 1983. The plants have annual nominal capacities to treat 80,000 t of copper concentrate and produce 20,000 tons of electrolytic (cathode) copper.

Beginning in 1992, imported copper concentrates began to be supplied to supplement local production due to a decrease in ore reserves in the Sohar area. In 1994, the smelter and refinery began to be supplied 100 % by imported concentrates due to the closure of all the local mines. Since then, the

smelter has become a toll smelter. Afterward, accompanied with decrease of ore reserves in the Sohar area, imported copper concentrate began to be partially supplied in 1992 and since 1994 whole concentrate has been imported by reason of closure of mines and become 100 percent toll smelter. As of the end of 1999, domestically produced concentrates of about 723,000 t and toll concentrates of 487,000 t had been treated and produced 289,000 tons of electrolytic copper. Annual tonnage of treated concentrates and electric copper production are shown in Figure 6.9.

The smelter has been operated without long term shut down since start-up in 1983. However, for the first time in 16 years, the smelter was ceased its operation in order to execute overall maintenance work for two months beginning in February of 1999 for the first time in 16 years (Figure 6.9). The main purpose of this shut down was scheduled maintenance of the electric arc furnace. During this period, the tank house continued its operation using accumulated anode stock. However, production was reduced by about 30 % compared with the previous year.

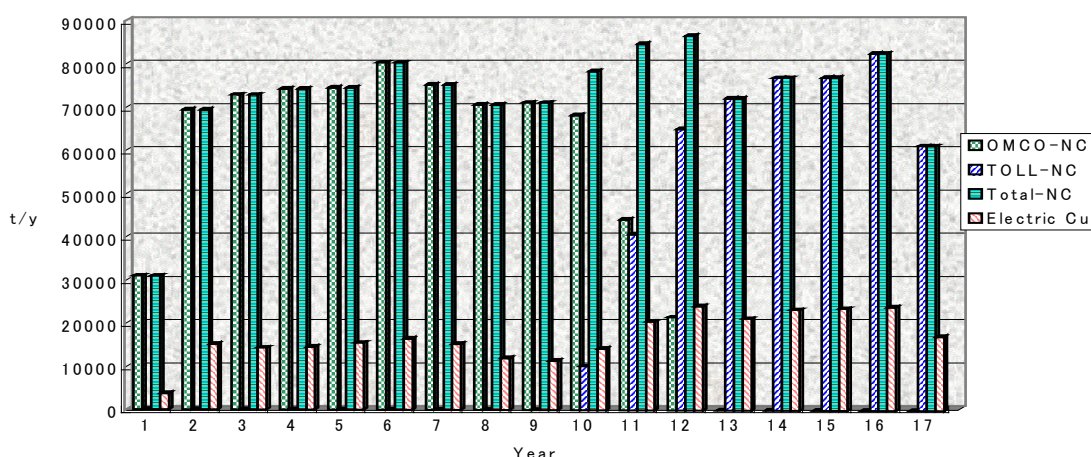


Figure 6.9 OMCO-Sohar Smelter Production

(2) Operational Performance in Recent Years

The operational performance of 1998 and 1999 were as shown in Table 6.7.

Table 6.7 Production Summary

Year	1998			1999		
	Tonnage	Grade		Tonnage	Grade	
	t	Cu%	S%	t	Cu%	S%
Copper concentrate treated	82,789	30.8	30.5	61,423	26.89	31.9
Electric copper produced	23,962	--	--	17171	--	--
Electric furnace slag	62,824	0.57	--	50,310	0.43	--

(3) Transportation of Raw Materials and Products

Currently, copper concentrate is imported from Chile, Australia, Myanmar, Peru or Iran and is unloaded at Al Fujairah port in the adjacent country of United Arab Emirates. The concentrates are then hauled by trucks to the smelter by way of 120 km of inland roads. The entire tonnage of the electrolytic copper produced is hauled to Muscat, a distance of 275 km by road. The refined copper is exported to European countries, India and Japan from the Mina Qaboos Port in Muscat.

6.5.2 Outline of Smelting Processes

The smelting processes are described in the following sections. A process flowchart and plant layout is shown in Figures 6.10 and 6.11, respectively. The principal process is copper smelting by electric furnace using pelletized concentrates charged to produce matte. The matte is treated by one of the two converters, an anode furnace and casting wheel to produce blister copper in the form of anodes. The blister anode copper is refined by conventional electrolysis in the tank house. Off-gases from the furnaces are discharged directly into the atmosphere through stacks without any desulfurization, such as acid plants etc.

Summarized specifications of the main equipment are listed in Table 6.8.

(1) Reception, Storage and Blending Process

The copper concentrate arrives at the smelter by trucks, and is hauled to an out-door stockpile for storage. From the stockpile, and appropriate amount required for blending is carried to blending bins by front-end loaders and belt conveyors. For cost purposes, the concentrates received by the plant must meet a stipulated standard grade of 30 to 35 % copper.

However, sometimes concentrates not meeting this condition may be received. In such cases, blending is a very important process to stabilize operating conditions.

(2) Pelletizing and Drying Process

The copper concentrate drawn out of the blending bins is blended with recycled flue dust and pelletizing binder. The blended concentrates are treated by a pelletizer of the tilted pan type to make pellets. The pellets are then dried by a moving grate type dryer equipped with natural gas burners. Ordinary belt conveyors and a special sloped conveyor are used to fill the feed bins installed just above the electric furnace.

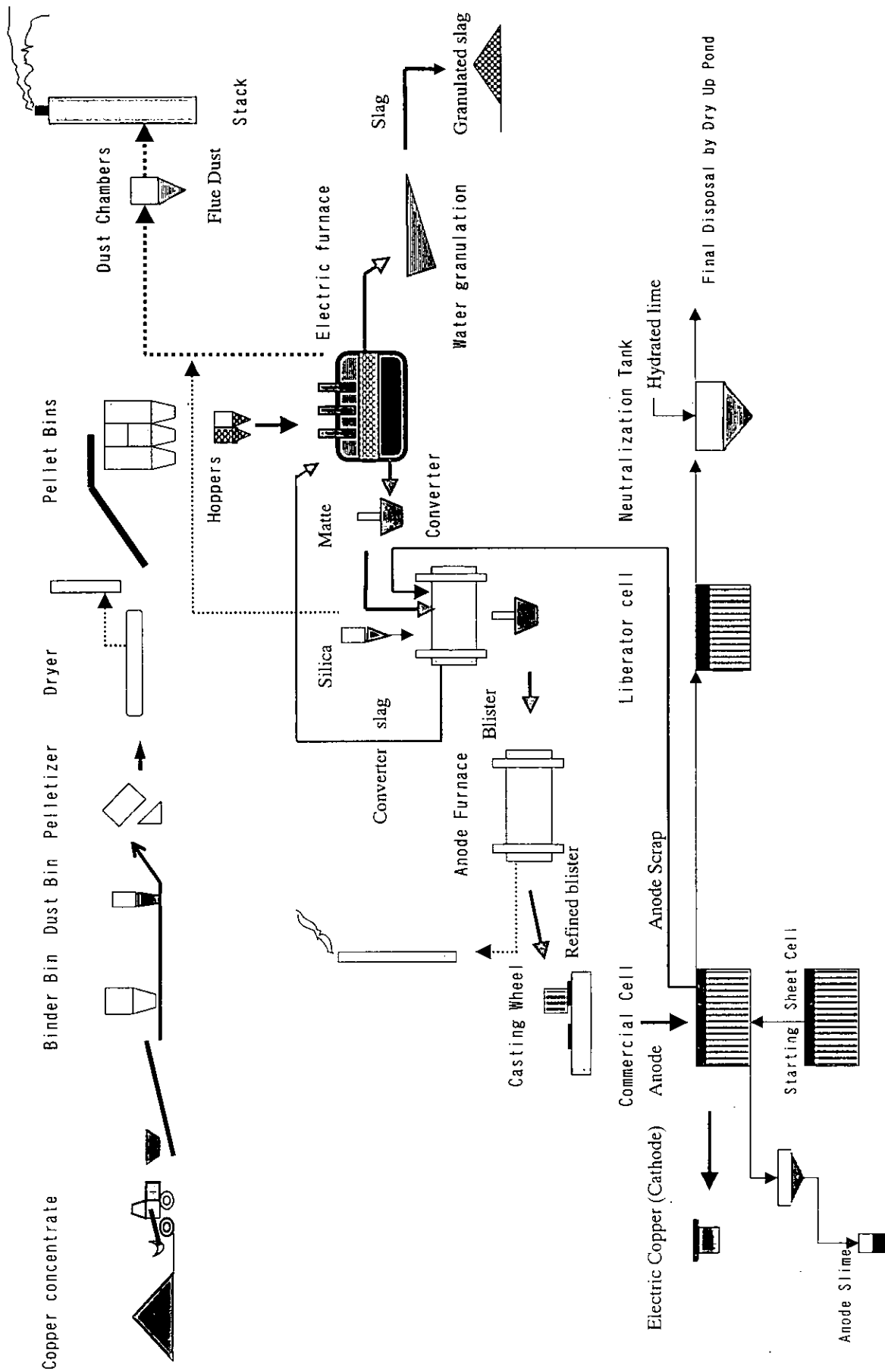


Figure 6.10 OMCO-Sohar Smelter Plant Layout

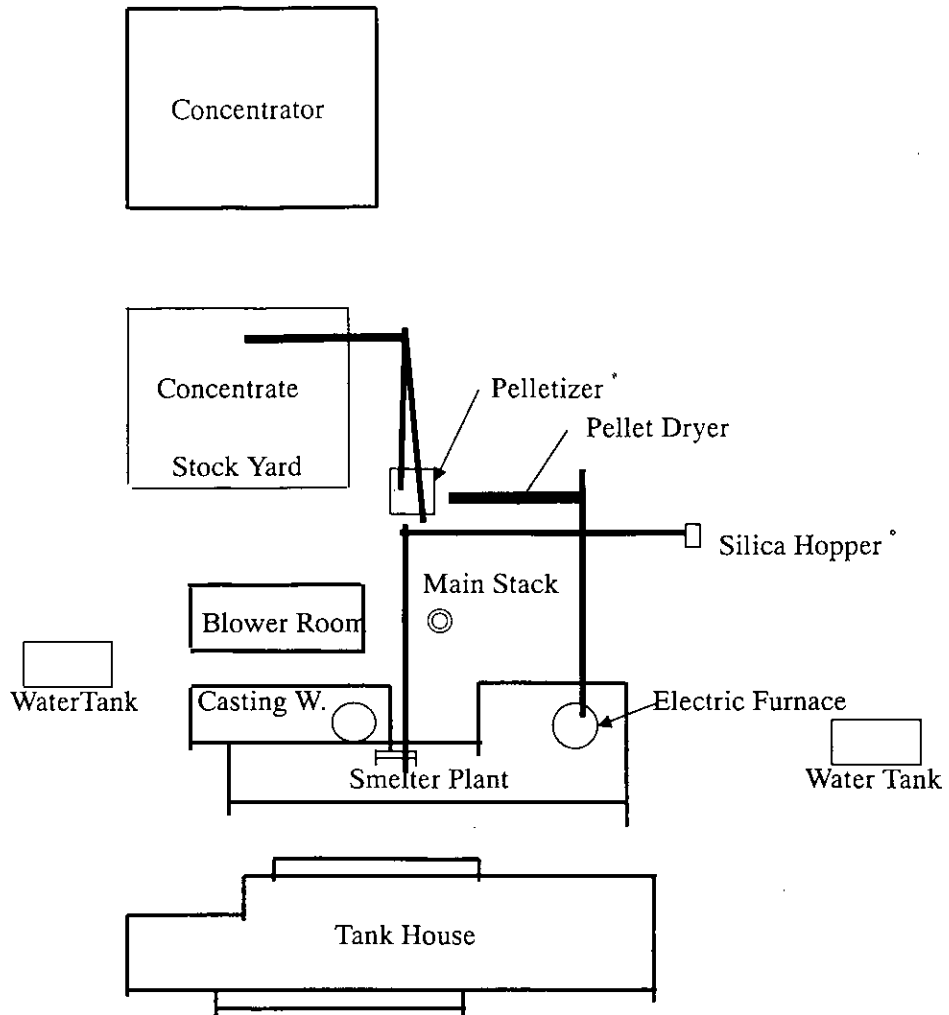


Figure 6.11 Smelter plant layout of OMCO

Table 6.8 Equipment Specifications of OMCO Smelter

Equipment	Item	Specifications	Quantity
Pellet dryer	Type Capacity	Moving grate type About 16 t/h	1 set
Electric furnace	Type Diameter Height Capacity	Heroult type electric arc furnace I.D. about 8.1m About 4.5 m About 12 t/h	1 set
Converters	Type Dimensions	Pierce-Smith type 2.8 m ϕ 6.0mL	2 sets (1 stand-by)
Anode furnace	Type Dimensions	Horizontal cylindrical type 2.8 m ϕ 6.0mL	1 set
Casting wheel	Type	Single molded Walker type	1set

(3) Electric Furnace Process

The dried pellets are charged together with limestone and silica as fluxes into an electric Heroult type furnace. The electric furnace is heated by three carbon electrodes, which are installed vertically on top of the furnace ceiling and connected with three-phase alternating current transformers. Heat is generated by arcing between the electrodes and raw materials.

Products in the furnace are melted matte, slag, and off-gas containing fume and sulfur dioxide gas. The melted matte, which is a crude mixture of sulfides formed in smelting sulfide metal ores, is pored into a ladle and carried by 40t overhead cranes to one of two converters. The melted slag overflows through launders and is quickly cooled by water flow. The slag is a waste material that is granulated and settled in a rake classifier and a water tank. The granulated slag is raked up in order to be dewatered for selling as an abrasive for sand blasting or road filler.

The off-gas is sucked out of the furnace and discharged to the atmosphere through a main stack. The off-gas is passed through dust settling chambers in the stack where the dust is collected.

(4) Converter process

The molten matte from the electric furnace and silica as flux are charged into tilted converters of the Pierce-Smith type. Then the converter is turned up to an upright position and compressed air is blasted through a number of nozzles, so-called tuyeres. An oxidizing reaction of sulfur and chemical reaction between iron and silica forms crude copper, so-called blister and slag. Since these reactions are heat generating, fuel is unnecessary under normal conditions.

After the slag forming process, air blasting is interrupted and the slag is pored into the ladle by tilting the converter. As the converter slag contains 2 to 4 % copper, the slag is carried by overhead cranes to the

electric furnace and charged again into the electric furnace through a recycle launder. After discharge of the slag and if enough blister copper is formed, the melted blister is pored into a ladle by tilting of the converter further. The ladle is used to carry the blister to an anode furnace. If not enough blister copper is formed, matte, coolant and flux are charged into the converter and the compressed air blasting is continued again. Identification of blister or slag is done by visual observation of a sample adhered to an iron rod and also by the flame color. The temperature of the converter is controlled by the amount of matte, flow rate of blast air, and addition of anode scrap as coolant. Converter off gas is discharged through a hood and duct to the main stack.

(5) Anode Furnace and Casting Wheel Process

The melted blister is refined in the anode furnace of the horizontal cylinder type to remove retained impurities by natural gas and compressed air. Refined blister is cast into anode plates by a casting wheel. The casting wheel is a Walker open horizontal single mold type. Anode plates with lugs are carried to the tank house. Off-gas from the anode furnace is discharged through a hood and duct to a local stack for discharge into the atmosphere.

(6) Electrolysis Process

There are three processes of electrolysis in the tank house namely electrolysis for electrolytic copper production, electrolysis for starting sheet production, and electrolysis for purification of electrolytic liquid. The starting sheets are produced on surfaces of mother copper boards by electrolytic reduction. Then, they are peeled off the motherboards and their strain or warp is corrected by a correcting machine. After correction, each sheet is hung by two riveted ribbons and a cross bar.

The Anodes and starting sheets are arranged alternately at regular intervals and soaked in electrolytic liquid mainly consisting of cupric sulfate and sulfuric acid solution. The sheets are electrolyzed by direct current. The copper from the anodes is dissolved by electrolytic reaction and educed on starting sheets as cathodes. Therefore, the anodes are reduced in their thickness, while the starting sheets become thicker and thicker. After about ten days, the starting sheets weigh about 120 kg and are pulled up and washed to remove slime on their surfaces, then shipped as final product after being bundled about 16 sheets to a pallet. Anode residues are returned to the converter.

The life of the anode, generally speaking, is about three weeks. Gold, silver and other impurities, such as lead in the anodes do not dissolve but settle in the electrolytic tanks as anode slime. The slime is removed periodically by washing after pulling up both anodes and cathodes. The washing wastewater is thickened and dewatered. Then, the dried slime is sent to Japan to recover gold and silver.

Since the electrolytic liquid becomes dirty by nickel, arsenic and iron, its constant volume is always separated and purified then the purified liquid is returned to electrolytic baths. From waste liquid

occurred by purification process, copper is recovered by electrolysis. After this electrolysis, the waste liquid is neutralized by lime milk and the treated waste liquid is discarded to drying ponds for final disposal. Liquid effluent from the smelter and tank house is limited to only this waste liquid and evaporation losses amounting to daily total of 120 to 140 t.

(7) Maintenance Management

There is a mechanical workshop adjacent to the smelter and maintenance work, such as mechanical fabrication, metal sheet processing, welding, overhaul or replacing of refractory is executed by OMCO itself.

6.5.3 Air Pollution Source, Situation of Off-gases

(1) General Description

There is neither an acid plant nor a desulfurization plant at the smelter. The off-gases of the electric furnace and converter are directly discharged from the main stack after simple dust collection by dust chambers consisting of balloon type ducts. Since the dust chambers are simple gravity settling system, their dust collecting efficiency is not high. Therefore, smoke and soot are always discharged out of the main stack. An especially large amount of dense smoke and soot hang over and around the smelter site matte or slag are discharged and the converter is charged. Consequently, the top of the smelter buildings becomes invisible by smoke for a while. Since it very seldom rains, only a part of the sidewall of the building is covered, so the smoke and soot often disappear in a few minutes. However, sometimes a considerably strong odor of sulfur dioxide gas can be felt in and out of the smelter.

Concentration of sulfur dioxide:	1 to 4 % fluctuating widely by cycles of Slag forming and blister forming stages
Amount of sulfur dioxide gas: discharged:	38,682 t/year of 1999, = 5.417 t/h=1,895 m ³ /h
Dust concentration :	0.3 to 0.45 g/m ³
Amount of dust discharged :	1 to 2 t/day
Dimensions of main stack:	2.16 m ϕ (I.D)x 100 mH
Designed velocity of off-gas for the main stack:	14.1 m/sec=51.6 m ³ /sec=185,908 m ³ /h

The average calculated SO₂ concentration can be given as 1.02 v/v% based on above data.

Fugitive gas around electric furnace and off-gas of the anode furnace are discharged through locally installed stacks for exclusive use. The annual sulfur balance in 1999 is shown in Table 6.9.

Table 6.9 Annual sulfur balance in 1999

Item	Sulfur t/y	Distribution %	Remarks
Copper concentrate treated	19,594	100.0	Ore tonnage 61,423 t/y, 31.1 %S
Total charge to electric fnc.	19,990	102.02	Pellet+converter slag
Electric furnace matte	14,279	72.87	5 small stacks
Granulated slag	735	2.9	Tonnage 50,310 t/y S: 1.17 %, Cu 0.57 %
Off-gas of electric furnace	4,861	24.81	
Converter matte(Blister)	13,883	70.85	Remarks
Converter slag	396	2.02	Tonnage 44,487 t/y, 0.89 %S
Anode blister	5	0.03	Tonnage 24,592 t/y
Converter off-gas.	13,480	68.79	
Off-gas of main stack	18,341	93.61	Electric furnace+converter
Fugitive off-gases	594	3.03	Small local stacks, fugitive gases

Off-gas of the pellet dryer is discharged out of a small stack after dust collection by a cyclone. A gas scrubber located in a small stack discharges dust collected from the dry pellet conveyors and flux conveyors after washing. Waste water of the gas scrubber is thickened by a old concentrate thickener and dewatered by filter, then dried by rotary dryer and returned to the pelletizing process.

(2) Sulfur oxide gas emission

a. K Value

Application of K value emission regulation for Japanese standard of sulfur dioxide gas emission

$$q = K \cdot 10^{-3} H_e^2$$

$$H_e = H_o + 0.65(H_m + H_t)$$

$$H_t = 2.01 \times 10^{-3} \cdot Q \cdot (T - 288) \cdot \left(2.30 \cdot \log J + \frac{1}{J} - 1 \right)$$

$$H_m = \frac{0.795 \sqrt{Q \cdot v}}{1 + \frac{2.58}{v}}$$

$$J = \frac{1}{\sqrt{Q \cdot v}} \left(1460 - 296 \times \frac{v}{T - 288} \right) + 1$$

where

q : Gas volume of sulfur dioxide in Nm³/h=1,895 Nm³/h

Q : Gas volume of off-gas at temperature of 15° C in m³/sec=51.6 m³/sec

v : Velocity of off-gas in m/sec=14.1 m/sec

T : Absolute temperature of off-gas in °K=250+273=523°K

r : rate of operation in smelter=0.772* * It shows about 0.9 in ordinary year.

H_o : Actual height of stack in m=100 m

H_e : Effective height of stack in m

H_m : Ascending height of off-gas due to upward momentum at top of stack in m.

H_t : Ascending height of off gas due to temperature difference between gas and air in m.

$$J = \frac{1}{\sqrt{51.6 \times 14.1}} \left(1460 - 296 \times \frac{14.1}{250} \right) + 1 = 54.5$$

$$H_m = \frac{0.795 \times \sqrt{51.6 \times 14.1}}{1 + \frac{2.58}{14.1}} = 18.1$$

$$H_t = 2.01 \times 10^{-3} \times 51.6 \times (523 - 288) \times \left(2.30 \times \log 54.5 + \frac{1}{54.5} - 1 \right) = 73.4$$

$$H_e = 100 + 0.65 \times (18.1 + 73.4) = 159.5$$

$$K = \frac{1895}{10^{-3} \times 159.5^2} = 74.5$$

This value corresponds to about 4.3 times of the most lenient allowable limit of 17.5 in Japan.

b. Maximum Ground Level Concentration and its Occurring Distance

Based on method devised by Sutton and Pasquill, the maximum ground level concentration C_m in ppm and its occurring distance x_m in km were estimated.

$$C_m = \frac{2q}{e\pi u H_e^2} \left(\frac{C_x}{C_y} \right)$$

$$x_m = \left(\frac{H_e}{C_z} \right)^{2/(2-n)}$$

where

q : Amount of sulfur dioxide gas emission in m³/h

e : Base of natural logarithm

π : The circular constant

u : mean wind velocity in m/sec

H_e : Effective height of stack in m

C_z, C_y, n : Sutton's diffusion parameters

Then, the following assumption was made.

$$q = 1895 \text{ m}^3/\text{h}$$

$$e = 2.72$$

$$\pi = 3.14$$

$$u = 6 \text{ m/sec}$$

$$H_e = 159.5 \text{ m}$$

$$C_z = 0.07$$

$$C_y = 0.12$$

$$n = 0.25$$

Therefore,

$$C_m = \frac{2 \times 1895 \times (1/3600) \times 10^6}{2.72 \times 3.14 \times 6 \times 159.5^2} \times \left(\frac{0.07}{0.12} \right) = 0.47 \text{ ppm}$$
$$x_m = \left(\frac{159.5}{0.07} \right)^{2/1.75} = 6883 \text{ m} \rightarrow 6.9 \text{ km}$$

c. Measurement of Sulfur Dioxide gas Concentration

Concentrations of sulfur dioxide gas have been measured at the following ten points in an area of radii less than 10 km from the main stack as the center. Because there is only one set of measuring instruments and removing the instrument by vehicle is required for the measurement, a 24-hour value has not yet been obtained. In addition, the instrument is set directly on the ground surface and the accuracy of measurement may have some problems due to the effect of sand storms, etc.

Hence, items and method of measurement are improved in the second site investigation together with an increase in the number of measuring points (Table 6.10).

6.5.4 Water and Waste Water Treatment

(1) Water Treatment

Because an adequate fresh water source could not be obtained at the start-up time of the plant, a great amount of the seawater was used beginning in 1983 and discarded into the tailings dump with concentrator tailings. Consequently, salt pollution has occurred in ground water down stream of the tailing dam.

Table 6.10 Measuring points of sulfur dioxide gas

Measuring point	Direction	Distance	Name of area
No.1	South west	10 km	SUHAYLAH
No.2	South west	8 km	SUHAYLAH
No.3	North east	5 km	ASAGA
No.4	North east	8 km	ASAGA
No.5	East south east	4 km	OWAINAH
No.6	East south east	8 km	OWAINAH
No.7	North west	4 km	RAHAB
No.8	North west	8 km	RAHAB
No.9	West	4 km	ALHAIL
No.10	West	8 km	ALHAIL

In 1993, OMCO began using groundwater from Wadi al Jizi and the importation of the seawater was terminated. Currently, all water supplied to the plant site consists of fresh water. Concerning the water balance, no new data were given. However, annual water consumption in a normal year ranges between 76,000 to 80,000 m³ for the smelter and 70,000 m³ for maintenance and utilities sections. About half of the smelter consumption is used for water granulation of the electric furnace slag and cooling in the casting wheel operation. The rest is consumed in the tank house.

Since the new supply water is ground water, no particular water treatment like filtration is performed.

(2) Waste water Treatment

Cooling water used in the smelter is itself cooled by a water spray in a cooling tank and reused in the smelter. Waste liquid from the electrolysis is treated in liberator cells to recover copper, neutralized by hydrated lime and transported by tank truck to evaporation ponds in the Bayda area for final disposal by natural evaporation. Hence, all water consumed in the smelter is lost by evaporation, including natural evaporation in circulation. Insolubilization or recycle of dried residue may be a problem in future.

There are no ditches in the smelter site for rainwater discharge and rainwater is disposed by natural gravity flow and infiltration into ground. Therefore, it is feared that heavy metal contamination of ground water may be occurring through the scattering of concentrates from the stockyard and dust fall.

However, it can be concluded that the frequency of rain is very rare, contaminant transport processes will be very slow, even though pollution may be occurring.