CHAPTER 4 PROCESS PLANT DESIGN

4-1 Plant Design and Metallurgy

4-1-1 Introduction

The main source of process plant design data for this study is reported in two Amdel report, "Metallurgical Testing of Copper/Gold Ore, Rakah Project", Report No N108FL00,28 February 2001, and "Additional Metallurgical Testing of Copper/Gold Ore, Rakah Project", Report No N027FL01, 18 September 2001.

The testwork used samples from four ore types, Rakah Stockwork (RSW), Rakah Massive Sulphide (RMS), Hayl as Safil (HAS) and Bishara Breccia (BB).

The scope of work included ball mill work index tests, flotation circuit development testwork, gold recovery and dewatering test work. Flotation test work culminated in a single locked cycle test on a blended feed.

Additional testwork performed for this study included the following:

- Rod mill work index tests of the four Amdel testwork samples.
- Ball mill and rod mill work index tests on an additional four Rakah Stockwork and four Hayl As Safil samples to provide some variability data. (Note that these two ore types make up approximately 80% of the resource).
- Bench scale dynamic High Rate Thickening (HRT) tests for the tailings stream.
- Vender vacuum and pressure filtration testing for the tailings stream.

The source of the samples used for the variability work index testwork, was remaining half drill core from the core used for the 2000/1 Amdel testwork. For the variability work, four samles were taken at increasing depth, whereas in the Amdel 2001/1 work the whole intercept length was composited into a single ore type sample.

The tailings sample for the HRT and filtration testwork was produced in a large-scale batch flotation test. The tests included rougher/scavenger regrind to 20 μ m and the first stage of cleaning.

Cleaner and Scavenger tails were combined to form the final tailing sample

4-1-2 Process Design

(1) Comminution

A summary of comminution data from previous testwork is given in TableⅢ-4-1, with the inclusion of Rod mill indices for the year 2000 samples from this study.

The variability data performed for this study is presented in TableⅢ-4-2. From these data, the design criteria values were selected, as presented in TableⅢ-4-3.

Table III-4-1 Summary of comminution data

Parameter	Ore Type							
	1993 Rakah	2000 Rakah SW	2000 Rakah MS	1993 Hayl as Safil	2000 Hayl as Safil	1993 Little Gossan	1993 Gotcha	2000 Bishara Breccia
UCS (ave)Mpa	78			82				
ICWi kWh/t 76-51mm	13.5			10.1				
51-38mm	10.9			7.7				
38-25mm	6.9			7.0				
25-19mm	5.5			5.7				
Rwi kwh/t	20.0	23.6	12.4	15.9	20.5			19.6
Bwi kwh/t	17.6	19.1	14.2	14.5	16.2	13.2	16.8	15.5
$BwiP_{80}$ μm	69	82	85	72	84			80
A	44.08			38.49		60.89	57.12	
B	1.73			1.47		1.19	0.58	
A x b	76.2			56.6		72.4	33.1	
Ta	1.22			0.23		1.57	0.82	
Ai	0.166			0.483				

Table III-4-2 Variability Sample Work Index Results

Sample	Rwi	BWi	$BwiP_{80}$
RSW	KWh/t	KWh/t	μm
P2-1	20.5	17.6	66
P2-2	25.6	21.6	66
P2-3	25.2	20.7	65
P2-4	24.5	20.0	65
Average	24.0	20.0	66
Hayl as Safil			
P3-1	15.6	15.4	73
P3-2	15.7	15.2	70
P3-3	19.4	17.1	72
P3-4	19.5	20.1	65
Average	17.6	17.0	70

Table Ⅲ-4-3 Design Criteria

	RSW	HAS	BB	RMS	Design
Blend	40%	40%	10%	10%	
UCS(MPa)	78	82			82
ICWi(kWh/t)					
-76-51 mm	13.5	10.1			13.5
-50-38 mm	10.9	7.7			10.9
-38-25 mm	6.9	7.0			7.0
-25-19 mm	5.5	5.7			5.7
Rwi(kWh/t)	24.0	20.5	19.6	12.4	21.0
Bwi(kWh/t)	20.0	17.0	15.5	14.2	17.8
BWiP80 (μm)	65	70	80	85	70

The options considered for comminution circuit design include SAB(semi-autogeneous and ball mill) and SABC(semi- autogeneous, ball mill and pebble crusher), as well as the more traditional crushing and milling circuits. For the semi-autogeneous milling options, considerable additional sampling and the testworks such as careful preliminary and pilot testworks on the representative ores from the each deposit would be required.

Considering the size of the deposits and the variety of the ore types, the decision was taken to select a traditional crushing and grinding circuit for this study. To minimize overall capital, a crushing and single ball mill circuit was selected.

The crusher selection was made by Metso Minerals based on Yanqul ore characteristics and using Metso's in-house crusher circuit sizing and mass balance programme.

A three stage crushing circuit was selected for the 3,000t/d case, producing a product with a P_{100} of 12 mm and a P_{80} of 8.5 mm. The primary jaw crusher selected was a 30" by 40" single toggle crusher, with HP200(or equivalent) cone crushers selected for the secondary and tertiary stages. A 1.8 m by 6.1 m double deck, multi-slope screen was selected.

The ball mill sizing was conducted using Minproc's inhouse mill sizing programs, based on Bond's method. The product size used was P_{80}70μm as determined in the laboratory flotation testwork by batch milling. The valuable components of the ore are sulfide minerals in a predominantly siliceous host, they are softer and heavier than the gangue minerals. the use of a cyclone classification system in the milling circuit, which cuts finer for minerals of higher SG, results in finer grinding of the sulfides.

To produce a sulfide P_{80} of 70μm, the whole stream grind size is increased to a P_{80} of 100μm.

Minproc's inhouse mill sizing programme was used to size the ball mill using the Bond work indices shown in Table Ⅲ-4-3, and a 10% uncertainty factor. The mill is designed to operate in closed circuit with cyclones producing a cyclone overflow of 35% solids density for feed to flotation. For the base case, a 5.03 m diameter by 7.20 m long (inside shell) overflow ball mill with a 2,800 kW motor was selected, along with 510 mm diameter cyclones.

(2) Flotation

The flotation circuit developed by Amdel consisted of a combined Rougher-Scavenger, regrind of the Rougher-Scavenger concentrate to a P80 of 20 μ m, followed by three stages of cleaning.

This circuit was locked cycle tested (Test 17), with a Cleaner-Scavenger unit, however the concentrate was not recycled. For the mass balance it was assumed the cleaner 1 tailing joined the rougher-scavenger tailing as combined final tailings.

A summary of the locked cycle test is presented in TableⅢ-4-4.

TableⅢ-4-4 Locked Cycle Test Result *

Stream	Weight (%)	Cu Grade (% Cu)	Au Grade (g/t)	Cu-Rec (%)	Au-Rec (%)
Cleaner 3 Concentrate	4.33	20.5	5.5	79.2	32.2
Cleaner-Scavenger Conc.	1.22	2.37	2.6	2.6	4.2
Cleaner-Scavenger Tailing	4.13	2.40	1.9	8.8	10.4
Scavenger Tailing	90.3	0.12	0.45	9.4	53.2
Pyrite Concentrate	14.8	0.26	2.3	3.5	44.9
Pyrite Tailing	75.4	0.09	0.08	5.9	8.3
Cleaner 1 Tailing	5.36	2.39	2.1	11.4	14.6
Rougher Concentrate	9.68	10.5	3.6	90.6	46.8
Calculated Head	100.0	1.12	0.76	100.0	100.0

* Locked cycle test result is shown as the average values of cycle 4,5 and 6 results

The inclusion of flash flotation and unit cleaner cells in the grinding circuit was considered appropriate to maximise gold recovery and minimise fine copper loss. A more detailed analysis has shown that the gold head grade is insufficient to justify the increased capital. Accordingly, flash flotation has not been included in the final flowsheet design.

The basic design philosophy for plants such as this is to recover the fast floating coarse liberated mineral into the cleaner stream without regrind in a high grade rougher concentrate. The slower floating composite and fine particles recovered in the scavenger concentrate are then reground to liberate and create fresh sulfide surface prior to cleaning. The first cleaner is run to maintain grade, and a cleaner-scavenger unit is used to control cleaner circuit recovery. Cleaner-scavenger concentrate is returned to regrind, while the tail joins the scavenger tail as final tailings. This approach to circuit design has been integrated with the results from Amdel testwork. The mass balance developed for the circuit has been based around the Amdel Test 17 locked cycle test result, with appropriate changes for a full scale circuit configuration.

Double launders have been specified for the Rougher-Scavenger and Cleaner- Scavenger flotation banks to allow adustment of the Rougher and Cleaner Scavenger concentrates cut points respectively. This flotation cell arrangement allows the plant to be operated in different modes (separate rougher concentrate and cleaner-scavenger, combined rougher-scavenger and no cleaner-scavenger, or a combination of both) simply by altering the destination of the concentrates from the double laundered flotation banks (refer process flowsheets).

A regrind size P_{80} of 20 μ m for the composite particles has been maintained, as described in the Amdel testwork, and a regrind ball mill has been selected for the duty to minimise the capital cost.

The product size is at the bottom end of the range for ball mills, and fine media will be required to maintain milling efficiency. The regrind mill has been sized using Minproc's in-house mill sizing program. An industry standard Ball mill work index (BWi) of 14kWh/t at P_{80} of 70 μ m was used, giving an estimated power of 18.7kWh/t at P_{80} of 20 μ m for the regrind mill sizing calculation.

A 10% uncertainty factor is included. The design mill throughput is 16t/h, based on the designed circuit mass balance, this compares with a nominal feed rate of 14 t/h for the combined rougher-scavenger regrind, no cleaner-scavenger regrind" circuit configuration.

Flotation cell size selection is based on the circuit mass balance using average pulp flow, and a testwork scale up factor of 2.4. Selection of large flotation cells to minimise the number of cells, and associated capital cost, was made as far as possible.

(3) Concentrate Dewatering

Amdel conducted static setting and vacuum filtration tests on unground rougher concentrates.

However, the final flotation circuit design includes a regrind of up to 100% of the final concentrate to P_{80} 20 μ m. Therefore conservative design for concentrate thickener and filter sizing.

Data is presented in TableⅢ-4-5.

TableⅢ-4-5 Concentrate Dewatering Data

Thickening	Unit capacity (t/m^2.h)
Amdel2000 testwork Rgh conc P_{80} 70 μ m	0.32
1994 Rakah Copper Study Cu conc P_{80} 30 μ m	0.38
This study Cu conc P_{80} 20 μ m	0.25
Vacuum Filtration	
Amdel2000 testwork Rgh conc P_{80} 70 μ m	0.9
This study Cu conc P_{80} 20 μ m	0.2

(4) Tailing Dewatering

Tailings thickener and filtration equipment selection and sizing has been based on vendor testwork as summarized in TableⅢ-4-6.

TableⅢ-4-6 Tailings Dewatering Data

Thickening	Unit Capacity (t/㎡.h)
Outokumpu testwork	0.54
1994 Rakah Copper Study	0.42
Filtration	
Svedala Vacuum leaf filtration	0.28
Svedala Pressure filtration	0.32
Larox Pressure Filtration	0.41

4-1-3 Metallurgical Performance

The metallurgical performance was predicted based on the flotation testwork conducted by Amdel, in particular the locked cycle test.

TableⅢ-4-7 Full Scale Performance Prediction for Selected Blend*

Metal	Concentrate Grade	Recovery
Copper	20.0% Cu	82%
Gold	5.6 g/t Au	33%

* Based on the composited samples of Rakah Stockwork, Hyal as Safil Stockwork, Rakah Massive and Bishara Breccia ores in proportion to the weight of 40%,40%,10% and 10% respectively.

4-1-4 Design Criteria

The predicted mass balances of the cases of 3,000t/d and 2,000t/d are shown in TableⅢ-4-14 and TableⅢ-4-15.

(1) Material Characteristics

a) Nominal Plant Feed		%Cu	Au g/t	%S
		1.24	0.75	11.0
b) ROM Size—100% passing	mm	800		
Ore moisture content		%	3.0	
Loose Bulk Density		t/m^3	1.85	
Angle of Repose		degree	37	
Angle of Drawdown		degree	65	

Angle of surcharge	degree	20	
Ore SG		3.2	

(2) Production Criteria

		Base Case	Option
Annual throughput	t/a	1,095,000	730,000
	t/d	3,000	2,000
Copper Concentrate			
Cu Grade	%	20.0	20.0
Cu Recovery	%	82	82
Au grade	g/t	4.8	4.8
Au Recovery	%	33	33
Cu Concentrate production	t/a	55,670	37,113

(3) Estimated Metallurgical Performance of Each Ore Type

TableⅢ-4-8 Estimated Metallurgical Performance of Each Ore Type

Ore Type	Cu Concentrate		
	Grade Cu %	Recovery	
		Cu %	Au %
Rakah SW	20.0	88.5	66
Hayl As Safil SW	20.0	88.5	36
Breccia (Bishara, Al Jadeed)	20.0	60.0	11
Massive Sulfide(Rakah,HAS,Asghar)	20.0	87.5	25

This metallurgical performance is a basis for the yearly mill production schedule.

(4) ROM Stockpile and Crushing

		Base Case	Option
a) Operating Schedule			
Annual throughput	t/a	1,095,000	730,000
Nominal throughput	t/h	3,000	2,000
Operating days per year	No.	365	365
Operating hours per day	h	24	24
Effective yearly operating hours	h	7,000	7,000
Plant availability	%	80	80
Nominal feed rate	t/h	156	104

b) ROM Bin

Live capacity	h	0.5	0.5
Live volume	m^3	42	28
ROM bin grizzly bar spacing	mm	600	600
Feeder type		Vibrating	Vibrating
Primary crusher grizzly	mm	75	75

c) Primary Crusher

Type		Single toggle jaw	
Size		C100 or equal	
Feed size F100	mm	600	600
CSS nominal	mm	100	100
CSS minimum	mm	75	75

d) Secondary Crusher

Type		Cone	
Size		Hp200 or equal	Hp300 or equal
Feed size F100	mm	150	150
CSS nominal	mm	25	25
CSS minimum	mm	20	20

e) Tertiary Crusher

Type		Cone	Not required
Size		Hp200 or equal	
Feed size F100	mm	28	
CSS nominal	mm	14	
CSS minimum	mm	10	

d) Screen

Type		Double Deck	
Size	m	1.8×6.1	
Top deck aperture	mm	28	28
Bottom deck aperture	mm	12	14
Feed rate	t/h	415	330
Screen undersize P80	mm	8.5	10.3

e) Fine Ore Bin

Live capacity	h	12	12
Live volume	m^3	810	540
Feeder type		Vibrating	Vibrating

(5) Grinding and Classification

		Base Case	Option
a) Operating Schedule			
Nominal throughput	t/a	1,095,000	730,000
Nominal throughput	t/d	3,000	2,000
Operating days per year	No.	365	365
Operating hours per day	h	24	24
Effective yearly operating hours	h	8,000	8,000
Plant availability	%	91.3	91.3
Nominal feed rate	t/h	137	91
Circuit type		Single stage closed circuit ball mill	
Feed size F80	mm	8.5	10.3
Product size (whole stream) P80	μ m	100	100
Product size (sulphides) P80	μ m	70	70

b) Ball Mill			
Mill type		Overflow	
Mill size (inside shell diameter)	m	5.03	4.42
〃 (Effective Grinding Length)	m	7.20	6.24
Pinion power design	kW	2,530	1,748
Pinion power maximum	kW	2,744	1,912
Installed motor power	kW	2,800	2,000
Ball charge-design	% volume	35	37
〃 -Maximum	% volume	39	41
Operating speed	% of critical	70	70
Ball size	mm	80	90

c) Classification			
Type		Cyclones	
Size (diameter)	mm	510	510
No, Operating		4	3
No. standby		1	1
Overflow density	% solid	35	35
Feed density	% solid	58	58
Underflow density	% solid	75	75
Circulating load	%	300	300
Cyclone Pressure	k Pa	65	65

(6) Flotation and Regrind

a) Rougher/Scavenger Flotation

Feed density	% solid	35	35
Rougher Laboratory retention time	min	1	1
Scavenger laboratory retention time	min	7	7
Scale up factor		2.4	2.4
Rougher Concentrate grade	%Cu	20	20
Rougher Concentrate recovery	%	74	74
Scavenger Concentrate grade	%	3	3
Scavenger Concentrate recovery	%	77	7
Flotation Cell size and No.		16m3/cell、6 cell（3＋3）	

b) Regrind

Feedrate	t/h	16	11
F80	μm	60	60
P80	μm	20	20
Mill type		Overflow ball mill in closed circuit	
Mill Size-diameter(inside shell)	m	3.05	2.74
-Length（EGL）	m	4.05	3.55
Pinion power design	kW	464	317
Pinion power maximum	kW	539	361
Installed motor power	kW	550	400
Ball charge -design	%volume	40	40
-maximum	%volume	45	45
Ball size	mm	25	25
Operating Speed	% of critical	75	75
Classification Type		Cyclones	
Size (diameter)	mm	250	250
No, operating		2	1
No. standby		1	1
Circulating load	%	300	300
Cyclone Pressure	kPa	85	85

c) Cleaner 1

Feed density	%solid	15	15
Laboratory retention time	min	2	2
Scale up factor		2.4	2.4
Concentrate grade	%Cu	16	16

Concentrate recovery	%	75	75
Cleaner-Scavenger Feed density	%solid	11	11
Laboratory retention time	min	10	10
Scale up factor		2.4	2.4
Cleaner-Scavenger grade	%Cu	10	10
recovery	%	67	67
Flotation cell size and No.		8m3/cell、 8 cells(2 +6)	

d) Cleaner 2

Feed pulp density	%solid	15	15
Laboratory retention time	min	7	7
Scale up factor		2.4	2.4
Concentrate grade	%Cu	18	18
recovery	%	80	80
Flotation cell size and No.		8m3/cell, 4 cells	

e) Cleaner 3

Feed density	%solid	15	15
Laboratory retention time	min	7	7
Scale up factor		2.4	2.4
Concentrate grade	%Cu	20	20
recovery	%	80	80
Flotation cell size and No.		8m3/cell, 2 cells	

(7) Thickening and Filtration

		Base Case	Option
a) Concentrate			
Concentrate Thickener			
Type		Conventional	
Concentrate tonnage (nominal)	t/h	7	4.7
Solids loading	t/m2h	0.25	0.25
Underflow density	%solid	65	65
Thickener diameter	m	6	5
Concentrate Filtration			
Type		Vacuum	
Filter availability	%	75	75
Production rate	t/h	8.5	5.6
Cake moisture	%	14	14
Filtration rate	t/m2h	0.2	0.2

Filter area	m2	42	28
Concentrate storage			
Cake bulk density	t/m3	1.7	1.7
Storage type		Open stockpile	
Storage capacity	d	7	7
Storage volume	m3	630	420

b) Tailings

Tailings Thickener			
Type		Hi Rate	
Tailings tonnages (nominal)	t/h	130	87
Solids loading	t/m2h	0.54	0.54
Underflow density	%solid	65	65
Thickener diameter	m	18	15
Thickened Tailing Storage			
Type		Agitated Tank	
Storage capacity	h	1	1
Storage volume	m3	112	75
Tailing Filtration			
Type		Pressure	
Filter availability	%	75	75
Production rate	t/h	130	87
cake moisture	%	15	15
Filtration rate	t/m2h	0.4	0.4
Filter area	m2	400	270
Cake bulk density	t/m3	1.9	1.9
Storage type		Concrete bund under filter	
Storage Capacity	h	12	12
Storage volume	m3	70	46

(8) Reagents

		Base Case	Option
a) Lime			
Delivery Form		Powder	
Strength	%CaO	88	
Consumption-nominal	kg/t	5.4	5.4
	t/day	16	11
Distribution type		Direct powder dosing to mill feed	
		Lime slaking and slurry distribution elsewhere	

b) Collector

M2030 (A3894 Alternate)

Packaging		200 kg Drum	
Consumption- design	g/t	40	40
	kg/day	120	80
Distribution type		Metering pumps from head tank(100%/w/v)	

Na-IPX

Packing		200 kg Drum	
Consumption	g/t	40	40
	kg/day	120	80
Distribution type		Metering pumps from head tank (10%/w/v)	

c) Frother—MIBC

Packing Form		200 kg Drum	
Consumption	g/t	25	25
	kg/day	75	50
Distribution type		Metering pumps from head tank via on/off valves(100%v/v)	

d) Flocculant

Type		Nalco 9903 or equal	
Packing		25kg Powder bags	
Consumption —Tailings Thickener	g/t	20	
-Cu concentrate	g/t	20	
-Total	kg/d	120	80
Distribution type		Dosing pump from storage tank (0.5%/w/v)	

(9) Water

		Base Case	Option
a) Raw Water			
Consumption (including 10 m3/h of potable water)			
	m3/h	40	30
Raw storage capacity	h	12	30
	m3	350	350
Fire storage capacity	m3	150	150
Total capacity	m3	500	500

b) Process Water

Process water pumps			
- average flow	m3/h	324	243
- design flow	m3/h	405	300
Storage capacity	hr	6	8
Process water storage volume	m3	2,000	2,000

4-1-5 Process Plant

(1) Introduction

The plant design is based on a flowsheet established after a review of testwork programmes conducted by Amdel. The design and innovative layout of the operation recognises the need to conserve capital while still providing a plant that is easy to operate and maintain. There are some aspects of the metallurgical test programme that are incomplete, particularly in relation to ore variability. The process design recognises this and provides for some flexibility in operations to enable the redirection of key process streams within the flotation circuit. This flexibility will allow for differing flotation responses expected from the various ore types.

The mill plant flowsheets are shown in Fig.Ⅲ-4-1, Ⅲ-4-2 and Ⅲ-4-3.

(2) Plant location and layout

The plant layout and general arrangements of the plant showing layouts and cross sections are presented in Fig.Ⅲ-4- 4 and Appendix 2A.

The mill plant is situated in the area of co-ordinates 454300E and 2417500N. The ground is gently sloping which will require general earthworks so that the plant is founded on cut ground.

The process plant layout is typical of flotation plant layouts. The crushing and screening plant is located alongside the ore receival pad. This section of the plant is operated as a dry plant with dust collectors placed in appropriate locations. The crushing and screening plant layout is designed to minimise capital through the integration of the fine ore bin with the structual support of the screen, crushers and crusher feed bins. This arrangement also leads to efficient operations and ease of maintenance.

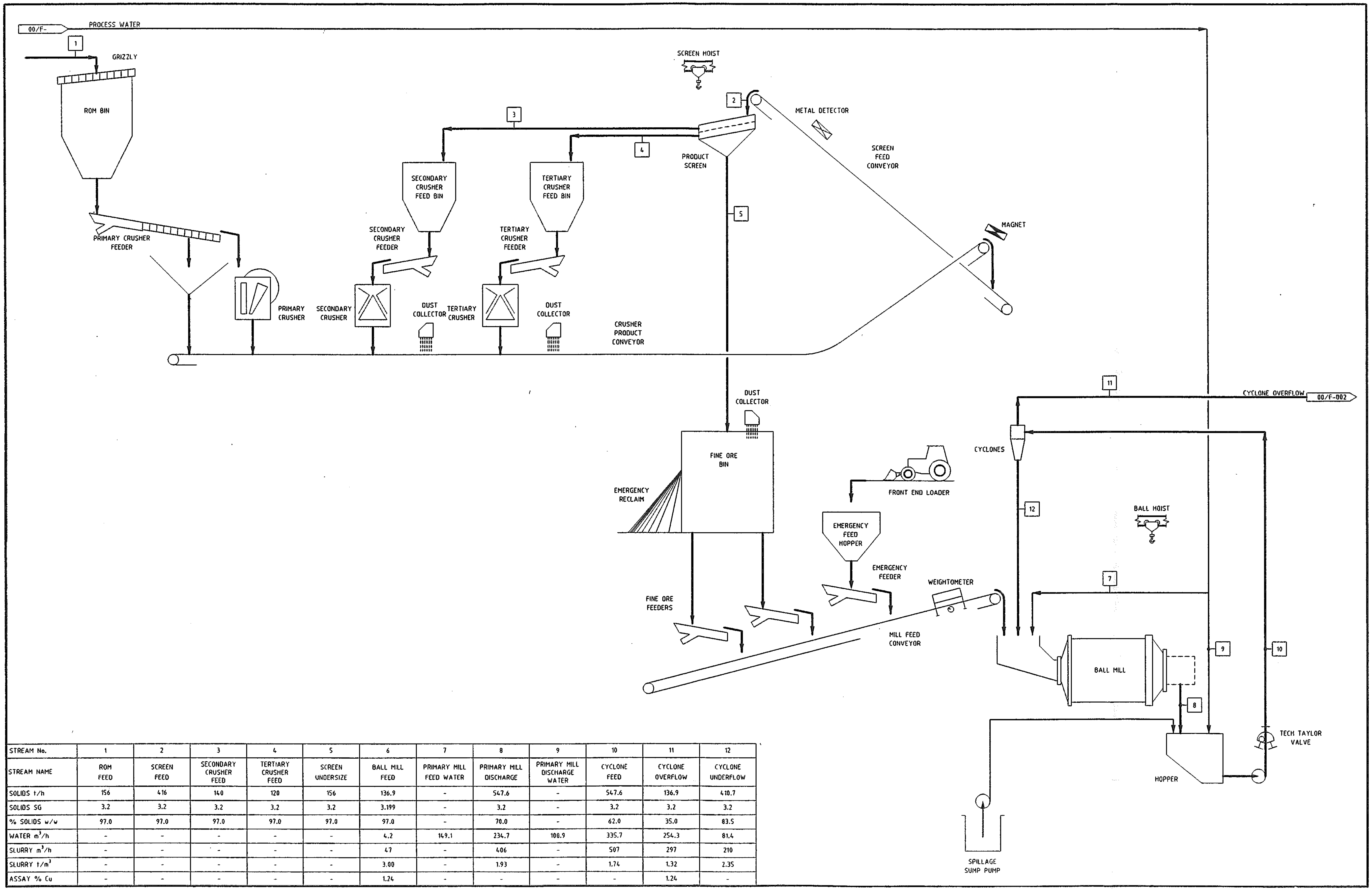
The grinding and flotation sections of the plant are arranged in a compact layout to reduce piping, support steelwork and electrical reticulation. This minimises capital expenditure and results in a plant easy to operate and maintain.

The grinding mills are arranged alongside each other, this simplifies access to the various operating levels.

The flotation plant is arranged in line with the mills and is accessible from a common walkway. This structure contains the major pipe rack running through the flotation area.

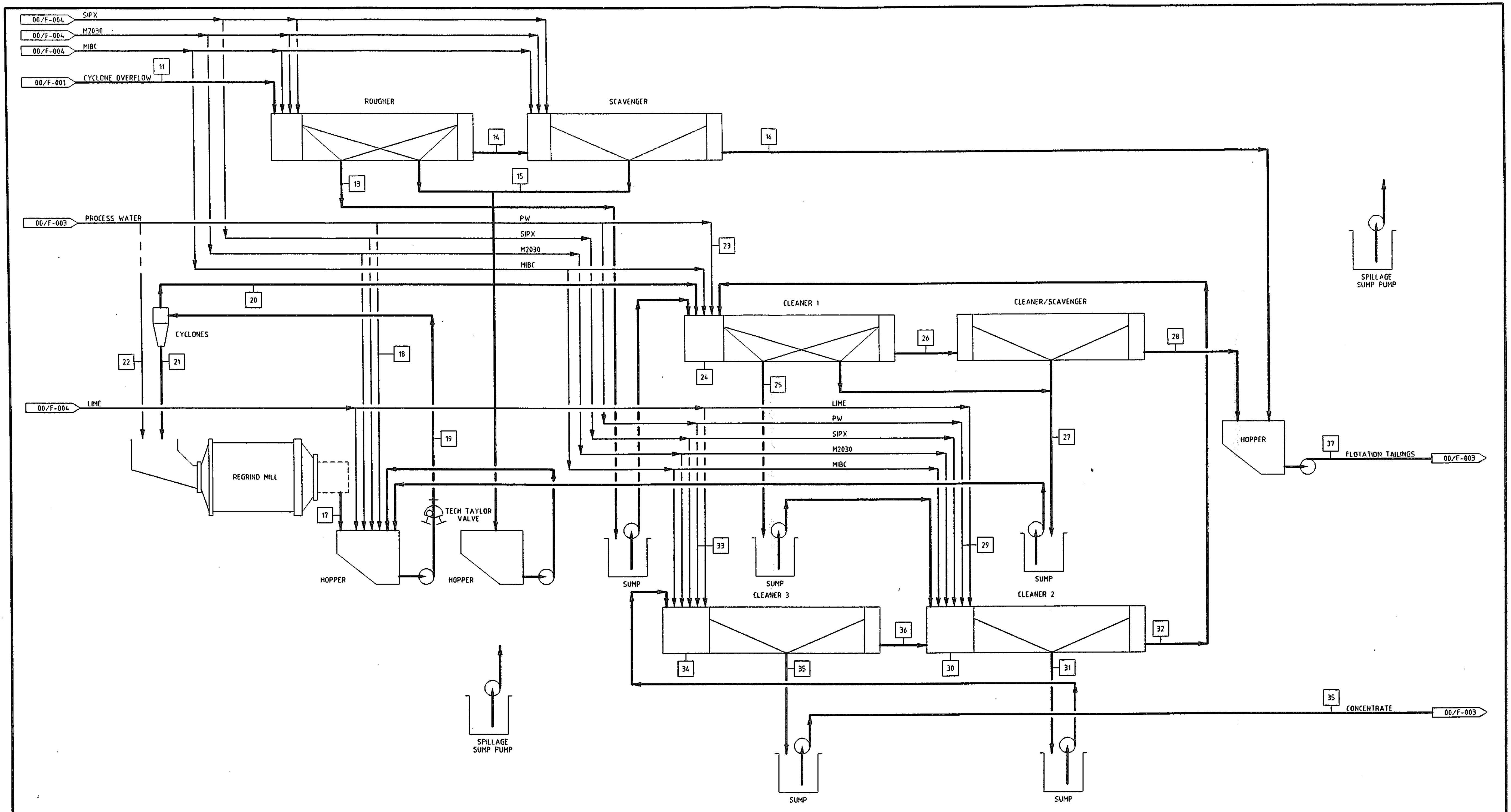
Gravity flow for product direction has been used wherever practical within the flotation circuit.

Product thickening, filtration and reagent preparation areas are grouped together and accessible via a common walkway.



STREAM No.	1	2	3	4	5	6	7	8	9	10	11	12
STREAM NAME	ROM FEED	SCREEN FEED	SECONDARY CRUSHER FEED	TERTIARY CRUSHER FEED	SCREEN UNDERSIZE	BALL MILL FEED	PRIMARY MILL FEED WATER	PRIMARY MILL DISCHARGE	PRIMARY MILL DISCHARGE WATER	CYCLONE FEED	CYCLONE OVERFLOW	CYCLONE UNDERFLOW
SOLIDS t/h	156	416	140	120	156	136.9	-	547.6	-	547.6	136.9	410.7
SOLIDS SG	3.2	3.2	3.2	3.2	3.2	3.199	-	3.2	-	3.2	3.2	3.2
% SOLIDS w/w	97.0	97.0	97.0	97.0	97.0	97.0	-	70.0	-	62.0	35.0	83.5
WATER m^3/h	-	-	-	-	-	4.2	149.1	234.7	100.9	335.7	254.3	81.4
SLURRY m^3/h	-	-	-	-	-	47	-	406	-	507	297	210
SLURRY t/m^3	-	-	-	-	-	3.00	-	1.93	-	1.74	1.32	2.35
ASSAY % Cu	-	-	-	-	-	1.24	-	-	-	-	1.24	-

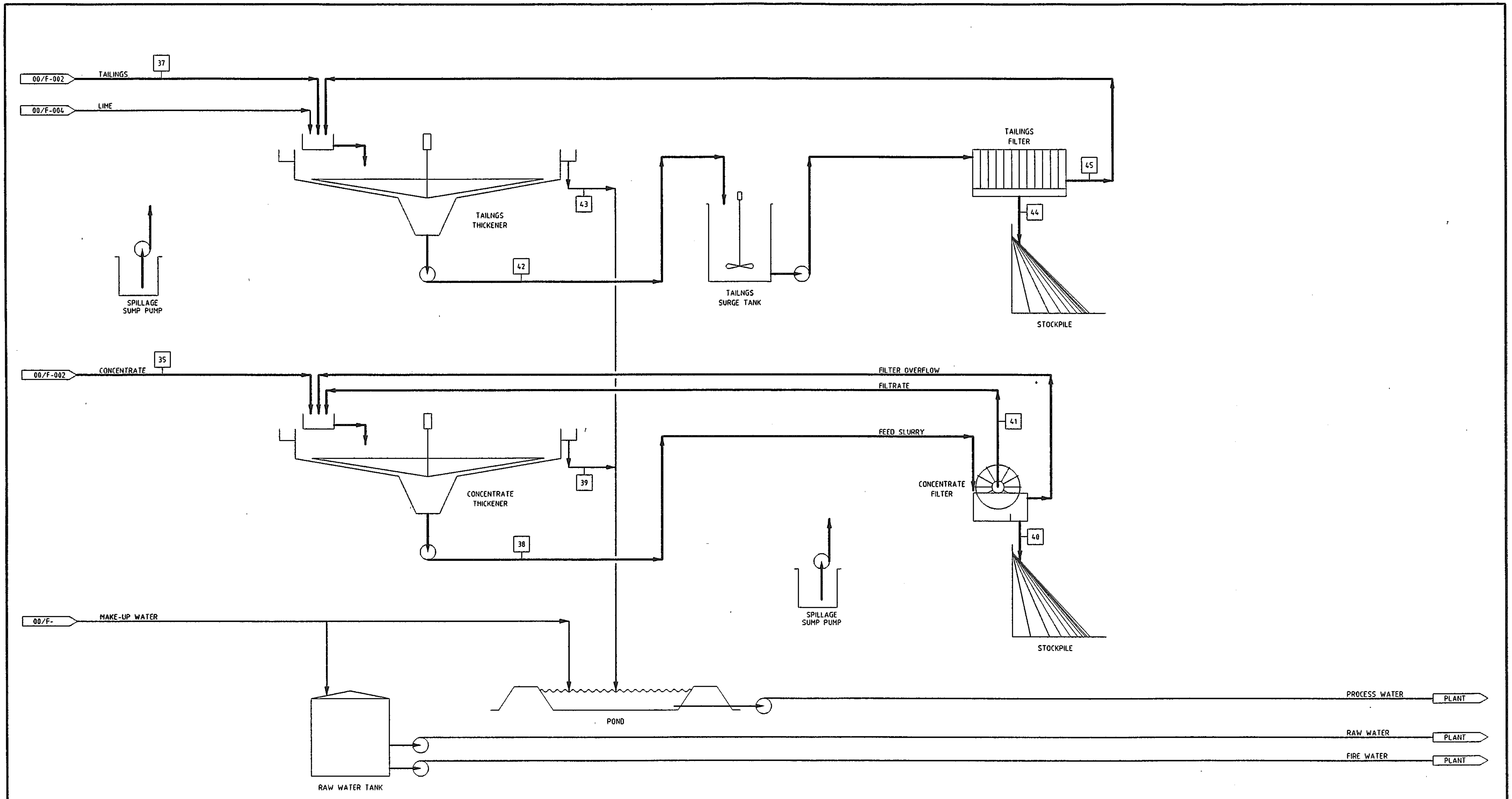
Fig.Ⅲ-4-1 Process flow diagram (crushing and grinding)



STREAM No.	11	13	14	15	16	17	18	19	20	21	22	23	24
STREAM NAME	CYCLONE OVERFLOW	ROUGHER CONCENTRATE	ROUGHER TAILINGS	SCAVENGER CONCENTRATE	SCAVENGER TAILINGS	REGRIND MILL DISCHARGE	REGRIND MILL DISCHARGE WATER	REGRIND CYCLONE FEED	REGRIND CYCLONE OVERFLOW	REGRIND CYCLONE UNDERFLOW	REGRIND MILL FEED WATER	CLEANER 1 DILUTION WATER	CLEANER 1 FEED
SOLIDS t/h	136.9	6.3	130.6	10.6	120.0	18.8	-	33.6	14.7	18.8	-	-	25.5
SOLIDS SG	3.2	4.3	3.1	3.3	3.1	4.3	-	4.3	4.3	4.3	-	-	3.7
% SOLIDS w/w	35.0	35.0	35.0	25.0	36.3	60.0	-	38.0	25.0	64.0	-	-	15.0
WATER m^3/h	254.3	11.7	242.6	31.9	210.7	12.6	30.5	54.7	44.1	10.6	1.9	16.2	144.3
SLURRY m^3/h	297	13	284	35	249	17	-	63	48	15	-	-	151.2
SLURRY t/m^3	1.32	1.37	1.31	1.21	1.33	1.85	-	1.41	1.24	1.96	-	-	1.12
ASSAY % Cu	1.24	20.00	0.34	3.20	0.08	-	-	-	5.09	-	-	-	9.59

STREAM No.	25	26	27	28	29	30	31	32	33	34	35	36	37
STREAM NAME	CLEANER 1 CONCENTRATE	CLEANER 1 TAILINGS	CLEANER/ SCAVENGER CONCENTRATE	CLEANER/ SCAVENGER TAILINGS	CLEANER 2 DILUTION WATER	CLEANER 2 FEED	CLEANER 2 CONCENTRATE	CLEANER 2 TAILINGS	CLEANER 3 DILUTION WATER	CLEANER 3 FEED	CLEANER 3 CONCENTRATE	CLEANER 3 TAILINGS	COMBINED TAILINGS
SOLIDS t/h	11.4	14.0	4.1	9.9	-	14.2	9.7	4.5	-	9.7	7.0	2.7	129.9
SOLIDS SG	4.1	3.4	3.7	3.2	-	4.0	4.2	3.7	-	4.2	4.3	3.9	3.1
% SOLIDS w/w	30.0	10.7	20.0	8.9	-	15.0	30.0	7.2	-	15.0	30.0	6.6	29.4
WATER m^3/h	26.7	117.6	16.4	101.2	14.9	80.2	22.6	57.6	32.3	54.9	16.3	38.6	312.0
SLURRY m^3/h	29.5	121.7	17.5	104.3	-	83.7	24.9	58.8	-	57.2	17.9	39.3	353.3
SLURRY t/m^3	1.29	1.08	1.17	1.07	-	1.13	1.30	1.06	-	1.13	1.30	1.05	1.25
ASSAY % Cu	16.0	4.35	10.0	2.03	-	15.4	18.0	9.76	-	18.0	20.0	12.86	0.23

Fig.Ⅲ-4-2 Process flow diagram (flotation)



STREAM No.	38	39	40	41	42	43	44	45
STREAM NAME	Cu THICKENER UNDERFLOW	Cu THICKENER OVERFLOW	Cu FILTER CAKE	Cu FILTRATE	TAILINGS THICKENER UNDERFLOW	TAILINGS THICKENER OVERFLOW	TAILINGS FILTER CAKE	TAILINGS FILTRATE
SOLIDS t/h	7.0	0.0	7.0	-	129.9	0.0	129.9	-
SOLIDS SG	4.3	4.3	4.3	-	3.1	3.1	3.1	-
% SOLIDS w/w	65.0	-	84.0	-	65.0	-	85.0	-
WATER m^3/h	3.8	14.9	1.3	2.4	70.0	242.0	22.9	52.2
SLURRY m^3/h	5.4	14.9	2.9	-	111.9	242.0	64.8	-
SLURRY t/m^3	2.00	1.00	2.81	-	1.79	1.00	2.36	-
ASSAY % Cu	-	-	-	-	-	-	-	-

Fig.Ⅲ-4-3 Process flow diagram (dewatering)

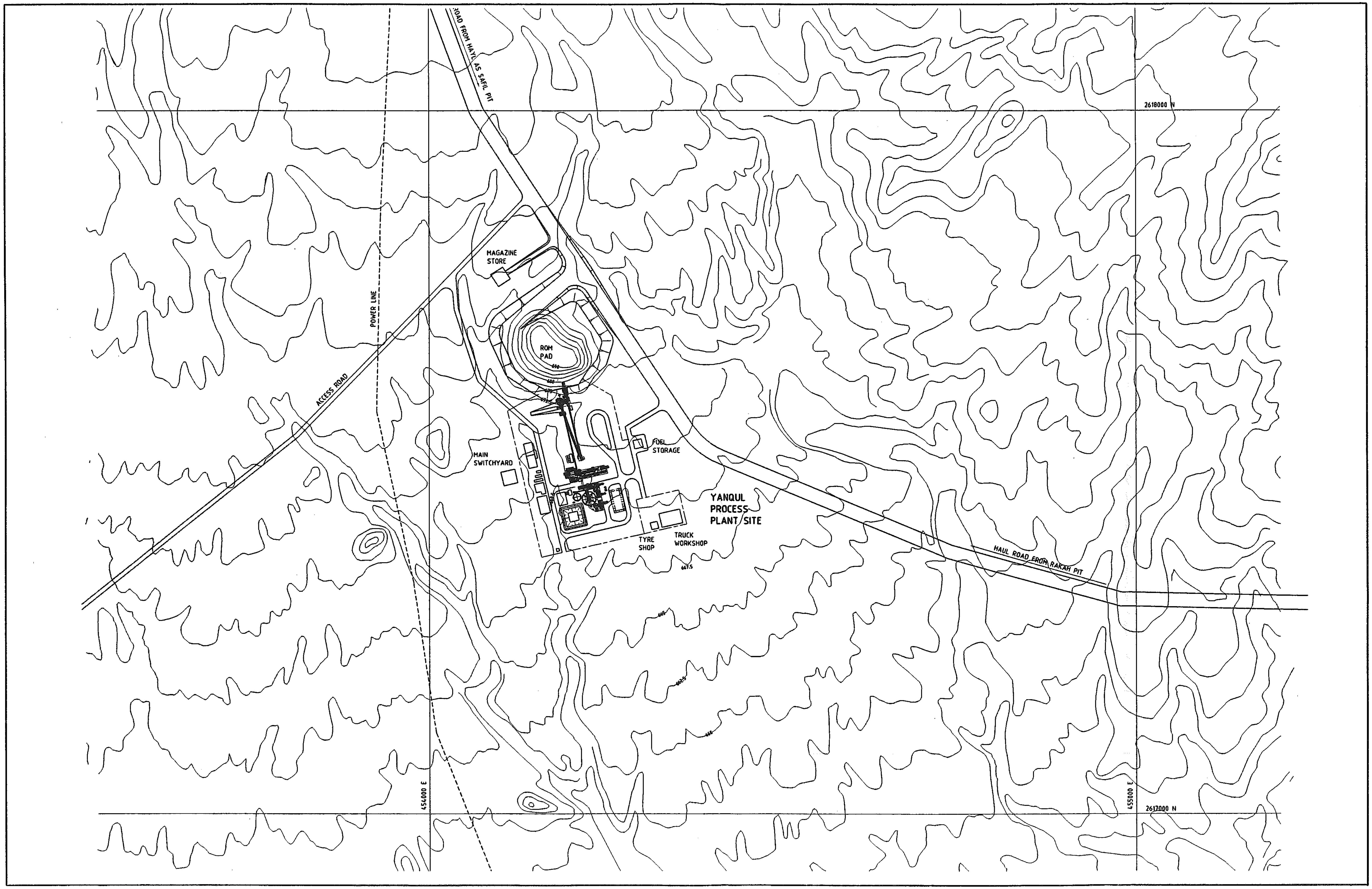


Fig.Ⅲ-4-4 Location of process plant site

A main central control room is located near the mill and flotation plant for general operating efficiency. Easy access between the control room and various areas of the plant(mainly at an elevated level)has been implemented in the layout.

(3) Crushing and Screening

The ROM bin is housed in a steel structure with the primary crusher mounted separately on concrete.

The secondary and tertiary cone crushers are located adjacent to the jaw crusher so that all three crushers discharge onto a common conveyor. The crusher discharge conveyor transfers crushed ore via the screen feed conveyor to the double deck vibrating product screen. The product screen is located over the fine ore bin (FOB) with internal screen oversize feed bins catering for the secondary and tertiary cone crushers surge requirement. The crusher feed bins are an integral part of the fine ore bin to reduce the support steelwork and chute-work.

Major maintenance activities on the crushing plant are conducted with a mobile crane.

The crushing and screening plant consists of the following key components:

- ROM bin- 80t live capacity.
- Primary crusher feeder and grizzly- vibrating feeder, 160t/h nominal capacity.
- Primary crusher- single toggle jaw, 1000 mm × 760mm.
- Crusher discharge and screen feed conveyors- 415t/h nominal capacity.
- Tramp metal magnet and metal detector- located at head end of crusher product conveyor.
- Product screen- 1.8 m wide × 6.1 m long double deck.
- Secondary crusher- Standard cone, nominal capacity 140t/h.
- Tertiary crusher- Shorthead cone, nominal capacity 120t/h

Ore is delivered from the mines by haul trucks and tipped direct into the ROM Bin through a 600 mm aperture grizzly. The bin provides approximately 30 minutes of feed to the crushing plant.

Oversize rocks from the grizzly are removed manually. The grizzly is designed for easy removal by mobile crane for maintenance purposes.

Ore is withdrawn from the ROM bin by a 1500 mm wide × 4400 mm long variable speed vibrating feeder. The feeder discharge end includes a 100 mm aperture vibrating grizzly section. Fine ore falling through the vibrating grizzly bypasses the jaw crusher while the grizzly oversize is directed into the jaw crusher. A 1000 mm × 760 mm single toggle jaw crusher has been selected for the duty.

The crusher is capable of receiving the largest lump that can pass through the ROM bin grizzly. Product from the crusher is expected to be 80% passing 100 mm.

The grizzly fines and jaw crusher discharge fall onto the 900 mm wide × 70 m long crusher product conveyor. The crusher discharge area is fitted with suitable impact idlers for further protection of the belt. A stationary manually cleaned magnet is located at the head of the crusher product conveyor to remove tramp metal.

The crusher product conveyor discharges crushed ore onto the screen feed conveyor. Ore is passed under a metal detector, which alarms when tramp metal is detected, and fed onto the 1.8 mm wide × 6.1 m long double deck screen.

Oversize ore from the 28 mm aperture screen top deck passes into the 40 t capacity secondary crusher feed bin. The bin is discharged at a controlled rate by a vibrating feeder into the secondary standard cone crusher. Ore from the screen intermediate deck passes into the 40 t capacity tertiary crusher feed bin. The bin is discharged at a controlled rate by a vibrating feeder into the tertiary shorthead cone crusher. Undersize ore from the screen , at minus 12 mm, is directed into the FOB.

The FOB has a live capacity of 1500 t – sufficient for 12 h operation of the downstream processing pant. The FOB is fitted with wear plates at outlet areas and two variable rate vibrating feeders located at the base.

The FOB also features an overflow facility to enable the establishment of an emergency fine ore stockpile on the side of the bin. Ore is withdrawn from the FOB by one or both vibrating feeders and discharged onto the mill feed conveyor. The vibrating feeders variable speed drives are linked to a weightometer fitted on the mill feed conveyor for control of the feed rate to the mill.

The mill feed conveyor is fitted with an emergency feed bin and associated vibrating feeder to enable continued mill operation in the event of upstream plant failure. Fine ore can be reclaimed by front end loader from the emergency fine stockpile and deposited into the emergency feed bin. The vibrating feeder variable speed drive is linked to the weightometer fitted on the mill feed conveyor for control of the feed rate to the mill.

(4) Grinding and Classification

The mill feed conveyor discharges crushed ore into the ball mill via s chute and feed spout lined with abrasion- resistant rubber. The feed chute assembly is rail-mounted to facilitate withdrawal and maintenance. Process water and cyclone underflow slurry is also added to the ball mill through the mill feed chute. The operation of the ball mill reduces the particle size of the ore from a feed size of 80% passing 9 mm to a nominal product size of 80% passing 100 μ m.

The ball mill is a rubber-lined 5.0 m diameter by 7.2 m effective grinding length (EGL) overflow unit fitted with a discharge end trommel and operating in closed circuit with a hydrocyclone cluster.

The mill is powered by a fixed speed 2800 kW wound rotor induction motor through a gearbox, with soft starting capability through the use of a liquid resistance starter. Pinion and trunion bearings are oil lubricated. The lubrication system for the girth gear is a fully automatic, pneumatically-operated spray system.

The ball mill discharges through its attached trommel screen with oversize scats reporting to a collection bunker. Trommel screen undersize reports to a rubber-lined pump hopper providing approximately 1.5 minutes residence time. Variable speed duty and standby pumps are installed on the discharge of the hopper and transfer mill discharge slurry to a cluster of hydrocyclones.

The cyclone cluster consists of 5×510 mm (4 duty, 1 standby) Warman cyclones. fine slurry overflowing the cyclone at 80% passing 100 μ m gravitate to the rougher flotation circuit. Coarse product from the cyclone underflow is directed back to the ball mill feed chute assembly.

The grinding area is fully bunded, with the floor sloped to direct spillage to a centrally located collection sump. The sump pump delivers any spillage to the mill discharge hopper. A drive-in concrete bunker is provided to collect trommel oversize scats for periodic removal. Grinding media for the ball mill is added via a kibble using an electric monorail hoist located above the mill feed chute. New grinding balls are added to the kibble at ground level from drums and are then hoisted into position for discharge into the ball mill feed chute.

(5) Flotation and Regrinding

The flotation circuit is designed to upgrade sulphide copper minerals into a final copper concentrate assaying 20% Cu. The flotation circuit consists of rougher, scavenging and three steps of cleaning.

A regrinding mill is included to allow for regrinding intermediate flotation products.

These intermediate components invariably consist of composite particles that require particle size reduction to allow for optimum recovery and grade. The plant is not covered and is arranged in-line and as an extension of the grinding plant. The flotation cells are positioned close to ground level, with the concentrate pumps being located in in-ground sumps. This approach to the design has been necessitated by the desire to minimise capital expenditure, as the conventional design approach of having elevated cells and conventional pump hoppers would require significant additional quantities of steel and concrete. The concentrate pumps are of the vertical spindle type and located in the sumps.

Major maintenance activities will be conducted by a mobile crane or smaller yard crane. The flotation plant consists of the following key components:

- Rougher flotation cells, three 16 cu.m cells.
- Scavenger flotation cells, three 16 cu.m cells.
- Cleaner 1 flotation cells, two 8 cu.m cells.
- Cleaner-scavenger flotation cells, six 8 cu.m cells.
- Cleaner 2 flotation cells, three 8 cu.m cells.
- Cleaner 3 flotation cells, two 8 cu.m cells.
- Regrind ball mill, 3.05 m diameter by 4.05 effective grinding length. The mill is powered by a fixed speed 550kW wound rotor induction motor through a gearbox and pinion/girth gear drive. Soft starting of the drive is accomplished with a liquid resistance starter. Pinion and trunnion bearings are oil lubricated. The lubrication system for the girth gear is a fully automatic, pneumatically-operated spray system.
- Regrind cyclones, two duty and one stand- by 250 mm diameter units.

Cyclone overflow slurry from the grinding circuit at 35% solids gravitates into the feed box ahead of the rougher flotation cells. The rougher circuit consists of three 16 cu.m trough shaped cells fitted with double sided launders arranged for variable product partitioning.

The scavenger circuit is also comprised of three 16 cu.m flotation cells fitted with similarly arranged double sided product launders. The total retention time provided in the roughing and scavenging circuit is 20 minutes. Concentrate from the rougher circuit is pumped to the cleaner one circuit or it can be pumped directly to the concentrate thickener. Concentrate from the scavenging circuit is directed into the mill discharge hopper of the regrinding mill. Tailings from the scavenging circuit is directed to the final tails hopper and transferred to the tailings thickener.

All the cleaner cells are conventional mechanical cells with 6, 24, 17 and 17 minutes residence times for the cleaner one, cleaner scavenger, cleaner two and cleaner three circuits respectively. Feed to the cleaner one circuit consists of rougher concentrate, cleaner two tails and ground product from the regrinding circuit. Concentrate from the cleaner one circuit is pumped to the feed box of the cleaner two flotation cells. Concentrate from the cleaner two circuit is pumped to the feed box of the cleaner three flotation cells.

Final cleaner concentrate gravitates to the cleaner concentrate sump and is pumped to the concentrate thickener. Cleaner one tails gravitate to the cleaner/scavenger cells. Cleaner scavenger concentrate is pumped to the discharge hopper of the regrinding mill. Cleaner scavenger tails is directed to the final tails hopper and transferred to the tailings thickener.

The rougher/scavenger and cleaner/scavenger intermediate flotation products are directed to the mill discharge hopper of the regrind ball mill. The mill discharge pumps transfer the intermediate products and mill discharge slurry to the regrind cyclones. The coarse underflow is directed to the head of the cleaner one flotation cells. The regrind circuit is designed to produce a ground product with a P_{80} sizing of 20 μ m.

(6) Concentrate Handling

Flotation concentrate is thickened and filtered prior to dispatch from the plant site.

Concentrate from the flotation circuit is pumped to the concentrate thickener feed box, where it is combined with filtrate recovered from the filter. Flocculant is also added to the feed box and the slurry then flows into the thickener feedwell. Concentrate thickener underflow at 65%solids is pumped to the filter and dewatered to less than 14% moisture. Thickener overflow is piped to the process water pond for recycling by the process water system.

A vacuum disc filter with a total filter area of 42 sq.m has been selected for the duty and is located on an elevated platform, this allows the dewatered filter cake to fall into a bunded area. Filtered concentrate is recovered from the bunded area by front end loader and loaded into delivery trucks.

The bunded area has the capacity to hold 1000 t of concentrate as well as additional space for manoeuvring the loader. Filtrate from the filter press gravitates back into the concentrate thickener where it is used as feed dilution water.

(7) Tailings Handling

Flotation tailings is thickened and filtered prior to dispatch from the plant site. The recovery of water from the tailings fraction is important in minimising the total project water usage.

Tailings from the flotation circuit is pumped to the tailings thickener feed box, where it is combined with filtrate recovered from the tailings filter. Flocculant is also added to the feed box and the slurry then flows into the thickener feedwell. Tailings thickener underflow at 65% solids w/w is pumped to the filter and dewatered to approximately 15% moisture.

Thickener overflow is piped to the process water pond for recycling by the process water system.

Two plate pressure filters with a total filter volume of 32 m^3 has been selected for the duty and are located on an elevated platform to allow the dewatered cake to discharge into a bunded area.

Tailings filter cake is recovered from the bunded area by front end loader and loaded into delivery trucks for the tailings disposal dam. The bunded area has the capacity to hold 1500 t of tailings, with additional space for manoeuvring a loader. Filtrate from the filter press gravitates back to the tailings thickener where it is used as dilution water.

(8) Reagents

The reagent mixing and storage facility is located on the main access way between the grinding plant and the thickening section. This area is favoured because it allows easy truck and forklift access and it centralises reagent distribution around the plant. The reagent facility is designed so that reagents are mixed on day shift only with provision of sufficient reagent storage to enable 24 h plant operation.

The reagent area is separately bunded with any spillage directed to a dedicated sump. Safety showers are located at strategic locations near reagent mixing and dosing points throughout the plant.

Reagents distribution diagram is shown in Appendix 2A.

The reagents used on site are as follows:

a) Promoter M2030

Promoter is delivered to site in 200 L drums and is pumped from the drums to a head tank in the flotation area using an air operated drum pump. Promoter is added to the process via metering pumps to the ball mill discharge hopper, the regrind mill discharge hopper and the cleaner flotation circuit.

b) Xanthate SIPX

Xanthate is delivered to the site in 200 L drums. The drums are lifted onto an enclosed drum tipper located on top of the combined mixing/storage tank. An overhead monorail hoist within the reagent area is used for drum movement.

From the mixing/storage tank, the xanthate is added to the process via metering pumps to the ball mill discharge hopper, the regrind mill discharge hopper, the scavenger flotation circuit and the cleaner circuit.

c) Frother MIBC

Frother is delivered to the site in 200L drums and is pumped direct from the drums to a head tank in the flotation area. The concentrated frother is added to the process via metering pumps to the head of the roughing, scavenging and cleaning flotation circuits.

d) Flocculant

Flocculant is added to the feedwells of both the concentrate and tailings thickeners to assist in improving thickener overflow clarity and general thickener performance. The flocculant plant is a packaged unit. The system is supplied complete with a storage tank, variable speed feeder, mixing agitator, mixing tank, storage tank and pump for the transfer of flocculant between the mixing and storage tanks. Dedicated variable speed positive displacement pumps deliver flocculant to the concentrate and tailings thickener.

e) Lime

Lime is delivered to the site by pneumatic road tankers and piped into the site storage silo located directly over the mill feed conveyor. The storage silo discharges the lime at a controlled rate onto the mill feed conveyor through a variable speed rotary valve. The lime silo incorporates a roof mounted dust collector and has a bin activator fitted to the bottom cone.

Lime is also discharged from the silo using a variable speed screw feeder into the agitated milk-of-lime tank. The screw feeder draws lime directly from the chute above the rotary valve. A pressurised ring main distributes the lime slurry to the flotation plant and back to the tank.

The lime slurry flow rate through the ringmain is continuous and high enough to prevent scale build-up and/or settling of solids from suspension.

Lime is dosed to the required points in the circuit at the required rate using metering pumps.

(9) Air Services

Plant air flow diagram is shown in Appendix 2A.

a) Plant Air

Duty and standby rotary screw compressors produce plant air at 750 kPa. The compressor feeds a plant air receiver which is provided for surge capacity. Plant air is taken from the air receiver and distributed by a ring main around the site.

Instrument air is taken from the plant air receiver and filtered and dried by a desiccant dryer before distribution throughout the plant. An instrument air receiver is provided.

b) Low Pressure Air

One duty and one stand by positive displacement low pressure air blower provides to the flotation circuit. Each unit delivers 30 kPa air pressure to the flotation circuit. The speed of each blower is controlled to maintain a set delivery air pressure to the flotation cells. The blowers are located in sound proof enclosures.

(10) Water Services

a) Raw water

Raw water is pumped to the site and directed into a 500 cu.m raw water tank. Raw water is piped throughout the plant and is pumped from the raw water tank by a single pump with a dedicated standby unit. Raw water is used for mixing reagents, safety showers and slurry pump gland sealing water.

b) Process Water

Process water is retained in a 2000 cu.m lined dam sufficient for 6 h of normal plant operation. Process water is sourced from tailings and concentrate thickener overflows with additional make-up from the incoming raw water system. Process water is used throughout the plant with the main consumer points being the ball mill and regrind mill. Operating and standby pumps are provided.

c) Potable Water

Potable water is taken directly from the raw water tank system. This water is used for site ablutions, drinking water and safety showers.

d) Fire Water

A fire water ring main system is provided throughout the process plant. A bleed line from the raw water pumps will maintain ring main line pressure and provide water for use as fire water. Whenever the line pressure drops below a set point, due to a fire hydrant being opened, a dedicated diesel booster pump will automatically start. The diesel fire water booster pump is connected to the raw water tank. The raw water pumps draw from tank outlet nozzles located above the fire water booster pump suction nozzle. In the event that the supply of raw water to the tank is interrupted, the fire water booster pump has 150 m^3 minimum volume to draw upon.

4-2 Operating Costs

The plant operating costs for the 3,000 t/d and 2,000 t/d cases are summarised in TableⅢ-4-9 and TableⅢ-4-10 respectively.

Table Ⅲ-4-9 3,000t/d Plant Operating Costs Summary

	US $ (Per Annum)	US $/t
Labour	807,527	0.74
Power	1,933,757	1.77
Reagents	1,010,498	0.92
Maintenance Materials	395,544	0.36
Consumables	959,836	0.88
Tailings Dam Operation	747,908	0.68
Miscellaneous	577,526	0.53
Total	6,432,596	5.88

Table Ⅲ-4-10 2,000t/d Plant Operating Costs Summary

	US $ (Per Annum)	US $/t
Labour	807,527	1.11
Power	1,435,307	1.97
Reagents	673,664	0.92
Maintenance Materials	310,105	0.42
Consumables	633,868	0.87
Tailings Dam Operation	517,971	0.71
Miscellaneous	489,054	0.67
Total	4,867,496	6.67

Details of the operating cost derivation for the 3,000 t/d case are presented in the sections that follow and the Appendix 2B. Details for the 2,000 t/d case are shown in the Appendix 2B only.

4-2-1 Labour

The mill department will compromise 53 personnel including 12 committed to maintenance and 41 dedicated to the operation of the copper concentrator. Salaries include on-costs to cover annual leave and other overheads.

4-2-2 Power

Power to the project will be distributed to the site at 11 kV from the Yanqul township substation.

The total annual power consumption of the processing plant is 37,230 MWh and the unit cost for power is estimated as 0.020 Rial/kWh. No allowance has been made for any transmission or facility charges from the town of Yanqul.

4-2-3 Reagents

Reagents consumptions have been derived from the locked cycle flotation testwork. Reagents unit costs have been obtained from Omani and Australian suppliers.

There is some uncertainty on the high usage rate of lime obtained from the testwork, this may be due to sample oxidation or excess reagent addition. The rates reported by the testwork have been used for the cost calculations though full-scale operation may achieve lower usage through the re-use of process water.

4-2-4 Consumables

Consumables usage has been derived from the experiences with other plants treating similar ores, combined with estimates based on the Yanqul ores' physical characteristics of hardness and abrasion resistance.

4-2-5 Maintenance Materials

The cost of maintenance materials has been factored from the overall direct cost. Factors applied to the individual capital costs have been derived from the experience with plants treating similar ores.

4-2-6 Tailings Dam Operation

Tailings from the flotation plant, after dewatered by the filter presses and stored in the stockyard, will be loaded into the trucks and transported to the tailings disposal dam. At the dam site, the heaped materials will be spread and compacted by a bulldozer in controlled layers, to achieve optimum in situ density. Tailings deposition will be a day time operation only. Filtered tailings produced during the night shift will be stored in the stockyard.

The operation costs of the tailings dam include the cost of the tailings loading, hauling and shaping cost (US$ 1.27/cu.m.), seepage pump operation and maintenance cost (US$ 2,750/annum) and general civil maintenance cost at the dam site (US$ 50,000/annum).

Though the dam closure cost (mainly as the dam capping civil work) at the end of the mine life is not shown in TableⅢ-4-9 and TableⅢ-4-10, it is estimated as US$ 3,000,000 (30 ha, @ US$ 10/sq.m), which will be required in the following year of the mine closure. This closure cost is classified as the operating cost in the financial analyses of the project.

4-2-7 Miscellaneous

Miscellaneous costs are as follows:

a) Laboratory Costs

An allowance has been made for laboratory consumables.

Annual cost = US$ 92,744.

b) Office and Plant Supplies

・Safety/Protective Clothing

An allowance of OR 120/annum for each production and maintenance person has been made for protective clothing and safety equipment.

Annual cost = US$ 16,122

・Office Supplies

This allowance has been based on typical costs.

Annual cost = US$ 19,945

c) Administration Support

• Photocopying and Printing

This cost has been interpolated from similar operations and includes printing, photocopying paper and leasing of a photocopier.

Annual cost = US$ 9,349

• Communications

This cost has been based on similar remote mine sites and allows for fixed line costs and telecommunication charges. Courier and mail charges have also been based on sites similar in terms of personnel structure and reporting requirements.

Annual cost = US$ 22,802

• Data Processing

The costs have been derived from previous experience. An allowance for 2 computer replacements per annum, including software and printer, has been included.

Annual cost = US$ 5,194

• Recruitment

Based on recruitment of two senior personnel per year.

Annual cost = US$ 9,349

• Relocation Expenses

Based on relocating two senior personnel per year.

Annual cost = US$ 5,194

• Bank Charge

This cost has been costed taken directly from other similar sites.

Annual cost = US$ 7,272

• Training

Training has been costed based on 1 % of the total payroll.

Annual cost = US$ 8,051

• IT Support

An allowance has been made for software updates and support functions.

Annual cost = US$ 22,334

d) Temporary Employment Cover for Annual Holidays.

Annual cost = US$ 6,752

e) Contract Cleaners

Contract cleaners will be used to clean the:

• site office;

· crib rooms;

· workshop;

· store;

· ablutions block;

· control rooms, and

· switch rooms.

Annual cost = USS 6,752

f) Mobile Vehicles

It has been assumed that the ROM loader will be operated by the mine contractor, it is not included here. Other vehicles are allowed for as leased costs and include the following:

· 6 no. 4WD light vehicles

- Metallurgy manager (4WD Land Cruiser Wagon)
- Production metallurgist (4WD ute)
- 2×Maintenance (4WD utes)
- 2×Mill Shift (4WD utes)

· 1 no. 10 t all terrain crane.

· 1 no. skid steer loader (bobcat)

· 1 no. 3 t all terrain forklift.

· 1 no. 4WD truck.

Annual cost = US$ 69,080

· Vehicle fuel

This includes fuel for the six 4WD vehicles per year and the other mobile equipment mentioned above:

Annual cost = US$ 30,382

g) Freight

· Freight contracts Annual cost = US$ 63,375

h) Raw Water

· The allowance for raw water assumes a total plant raw water requirement of 40 cu.m/h at a cost of 1 baisa/Imp.gallon.

Annual cost = US$ 182,829

4-3 Capital Costs

4-3-1 Mill Plant Direct Capital Costs

The details of the mill plant capital costs are shown in Appendix 2C and the equipment specs list is shown in Appendix 2D. Capital cost estimates for each area of the plant were developed and assembled using the general methods detailed below.

a) Transportation

This was based on freight rates quoted by shipping agencies based Australia (for international rates) and the Gulf region (for in-country rates).

b) Buildings

Buildings, such as the administration office, laboratory and workshop/stores, are based on construction practices normal to Oman. They were estimated on the basis of a floor area taken off preliminary drawings and based on rates supplied by local contractors. The substation, control room and plant crib/ablutions building are based on transportable style buildings with rates supplied by local contractors. The mine workshop is provided without internal fit-out except for the provision of two overhead travelling cranes. The estimate includes for lighting, small power, heating and air conditioning in appropriate buildings. No data links are allowed for.

c) Earthworks

Take-offs were measured from general arrangement drawings and a contour survey of the site. No specific geotechnical details are available at this stage, therefore reasonable assumptions have been made about likely ground conditions. Unit rates were determined based on quotations from reputable contractors, with local experience where possible.

d) Concrete

The quantities were measured from general arrangement drawings and preliminary designs produced Minproc's civil design department. Unit rates were based on quotations from subcontractors with local experience.

e) Structual Steel

The quantities were measured from preliminary designs and general arrangement drawings produced by Minproc's structual design department. Unit rates were based on quotations from subcontractors with local experience.

f) Platework and Tankage

The quantities were measured from general arrangement drawings in conjuction with the equipment list and vender supplied data. Unit rates were based on quotations received from a local subcontractors.

g) Mechanical Equipment

Equipment Major items (>US$ 50,000) – Duty specifications for major items of equipment were prepared and formal enquiries sent to Australian or international suppliers.

Minor items (< US$ 50,000) – Based on faxed quotations from suppliers or standard in-house price lists and in-house data from recent similar projects.

h) Pipework

In-plant process piping has been factored against cost of equipment using in-house historical data.

i) Installation Labour and Erection Equipment

Manhours are based on in-house data from previous projects, subdivided into disciplines. Unit rates were based on quotations from regional contractors with appropriate experience. Labour rates have been taken from information supplied together with interpretation of steel fabrication rates supplied by regional contractors. The labour rates have the potential to be outside our quoted accuracy of 10%. Preliminaries to cover the cost of providing sub-contractor construction facilities (excluding construction camp costs, which are allowed for as a line item in the capital cost estimate) are included in the unit rates.

j) Electrical and Instrumentation

Written quotations were sought for all major items valued in excess of US$ 50,000. Other items were based on telephone and fax quotations, inhouse data or standard price lists.

k) Capital Spares

An allowance for strategic process plant spares has been made. This has been calculated as a percentage of the process plant mechanical equipment cost.

l) First Fill

An allowance for a first fill of reagents and consumables has been allowed calculated on the basis of 2 months' normal supply.

m) Indirect Capital Costs

Temporary Works and Services – Assessment was made of requirements for temporary site services (including offices, furnishings, computers etc). Hire rates were derived from in-house historical data and recent project experience.

EPCM Services – Design engineering, project engineering, construction management and commissioning labour costs were based on in-house assessment of manhours, estimated from the project schedule, extended using the Minproc schedule of rates for 2001. Project expenses costs were based on the requirements of the project schedule and experience gained in previous projects of this nature. A project fee based on 5% of the total cost is included in the estimate.

n) Qualifications and Clarifications of the Capital Cost Calculation

The estimate is based on the installation of all new equipment. It is our experience that only relatively small capital reductions can be made through the use of second-hand equipment. This is because the actual capital cost of the equipment only represents approximately 40% of the final installed cost. So that if 30%of the capital cost can be saved through the purchase of second hand equipment, after proper equipment refurbishment, then the final saving off the total installed price is about 10%. The only areas where significant savings are possible are the major capital items of crushers, ball mills and filters. The volatility of the second-hand market for this type of equipment means that it is not possible to conduct price estimations within the required accuracy of this study. At the time of project execution a survey of the second-hand market for these key items could be conducted.

Full geotechnical information is not available at this stage and, therefore, assumptions have been made with respect to the ground conditions on the site. Rock excavation has not been allowed for. For the site cut and fill, it is assumed that excavated materials are suitable as general backfill. A soil bearing pressure of 250kPa has been assumed.

No allowance has been made for the provision of computers, printers, network servers/systems or software within any building, other than the two OIS installed in the control room for operation and monitoring of the process plant.

The cost of a tailings filtration system has added substantially to the estimated capital cost. The major cost in the system apart from the filters is the air compressors required for the air blow cycle of the filtration process. No allowance has been included in the capital cost for GST or any other taxes, import duties, statutory fees or special fees that may be levied on the Project.

4-3-2 Capital Cost for Tailings Dam Construction

Detailed of the capital cost for tailings dam construction are shown in Appendix 2E. The cost includes tailings dam construction (Bill 1), Access roads from the mill site to the dam (Bill 2) and diversion water channel for the dry dam (Bill 3). TableⅢ-4-11 summarises the capital cost for the tailings dam construction.

The capital cost of the dam construction for the 3,000t/d and 2,000t/d is estimated as the same cost. In both cases, the capital cost is allocated into phase 1 (early construction period) and phase 2 (the 4th year of the operation for the 3,000t/d case and 6th year for the 2,000t/d case) in the investment schedule. As described before, the dam closure cost (US$ 3,000,000) of the capping civilwork will be required when the mine is closed. This cost is allocated as an operating cost in the following year of the final operating year.

Table Ⅲ-4-11 Capital Cost Summary for Tailings Dam Construction

Bill No.	Description	Phase 1 Total Cost	Phase 2 Total Cost	Phase 1-2 Total Cost
Bill 1	Tailings Dam Construction	1,451,772.50	1,261,912.50	2,713,685
Bill 2	Access Roads	363,788	0	363,788
Bill 3	Diversion Channel	23,915	0	23,915
	Sub-Total	1,839,475.50	1,261,912.50	3,101,388
	Add10% Preliminary&General	183,948	126,191	310,139
	Add15% Contingencies	275,921	189,287	465,208
	Add12% Engineering	220,737	151,430	372,167
	Grand Total	2,520,081.50	1,728,820.50	4,248,902

Table Ⅲ-4-12 summarises the capital cost for the case of base case (3,000t/d) associated with the design, procurement, transportation, construction and commissioning of the process plant. All direct, indirect and accuracy provision costs have been included, no contingency has been allowed.

Table Ⅲ-4-12 Summary of Process Plant Capital Cost –Base Case (3,000t/d)

Area	Description	Total (US$'000)
10	Crushing	2,313
20	Grinding	2,624
30	Flotation	1,783
40	Concentrate Thickening and Filtration	748
50	Tailings Thickening and Filtration	2,734
60	Reagents	309
70	Water and Air Services	439
80	Buildings	1,350
	Other Costs	3,358
	Tailings Dam Construction	3,101
	Direct Cost Total	18,759
	Indirects-EPCM etc., Total	4,718
	Summary Total	23,477

Table Ⅲ -4-13 summarises the capital cost for the 2,000t/d case associated with design, procurement, transportation, construction and commissioning of the process plant. The estimate is factored from the base case with similar provision and exclusions. The nature of the estimate is such that the order of accuracy will not be to the same standard as the base case. Minproc would expect that this method of compiling the capital cost results is an order of accuracy of ±20%.

TableⅢ-4-13 Summary of Process Plant Capital Cost –2,000t/d Case

Area	Description	Total (US$'000)
10	Crushing	1,709
20	Grinding	2,046
30	Flotation	1,247
40	Concentrate Thickening and Filtration	584
50	Tailings Thickening and Filtration	1,865
60	Reagents	247
70	Water and Air Services	340
80	Buildings	1,350
	Other Costs	2,890
	Tailings Dam Construction	3,101
	Direct Cost Total	15,379
	Indirects-EPCM etc., Total	4,718
	Summary Total	20,097

4-3-3 Indirect Cost

EPCM cost (US$ 2,763,000), contingency etc. cost for the tailings dam construction (US$1,147,000) and the one year labour cost for the mill department as initial owner's cost (US$ 808,000) are allowed as the total indirect costs.

Table Ⅲ-4-14 Nominal design -case 3,000t/day

Nominal Design Case
3,000 tpd
80.0 % Crusher Availability
91.3 % Mill Availability

Stream	Stream Name	Solid t/h	Solid SG	% Solids w/w	Water m^3/h	Slurry m^3/h	Slurry t/m^3	Assay % Cu	Assay % Pb	Assay % Zn	Assay g/t Ag	Assay g/t Au
1	ROM Feed	156	3.2	97.0								
2	Screen Feed	416	3.2	97.0								
3	Secondary Crusher Feed	140	3.2	97.0								
4	Tertiary Crusher Feed	120	3.2	97.0								
5	Screen Undersize	156	3.2	97.0								
6	Ball Mill Feed	136.9	3.199	97.0	4.2	47	3.00	1.24				0.75
7	Primary Mill Feed Water				88.2							
8	Primary Mill Discharge	547.6	3.2	70.0	234.7	406	1.93					
9	Primary Mill Discharge Water				161.9							
10	Cyclone Feed	547.6	3.2	58.0	396.6	568	1.66					
11	Cyclone Overflow	136.9	3.2	35.0	254.3	297	1.32	1.24				0.75
12	Cyclone Underflow	410.7	3.2	74.3	142.3	271	2.04					
13	Rougher Concentrate	6.3	4.3	35.0	11.7	13	1.37	20.00				4.90
14	Rougher Tailings	130.6	3.1	35.0	242.6	284	1.31	0.34				0.55
15	Scavenger Concentrate	10.6	3.3	25.0	31.9	35	1.21	3.20				1.49
16	Scavenger Tailings	120.0	3.1	36.3	210.7	249	1.33	0.08				0.47
	Regrind mill new feed	14.7			48.2							
17	Regrind Mill Discharge	44.1	4.3	60.0	29.4	40	1.85					
18	Regrind Mill Discharge Water				3.6							
19	Regrind Cyclone Feed	58.8	4.3	42.0	81.3	95	1.48					
20	Regrind Cyclone Overflow	14.7	4.3	20.0	58.8	62	1.18	5.09				2.04
21	Regrind Cyclone Underflow	44.1	4.3	66.3	22.4	33	2.04					
22	Regrind Mill Feed Water				7.0							
	Cleaner 1 Feed	25.5	3.7	16.6	128.1	135.0	1.14	9.59				2.79
23	Cleaner Dilution Water				16.2							
24	Cleaner 1 Feed	25.5	3.7	15.0	144.3	151.2	1.12	9.59				2.79
25	Cleaner 1 Concentrate	11.4	4.1	30.0	26.7	29.5	1.29	16.00				3.72
26	Cleaner 1 Tailings	14.0	3.4	10.7	117.6	121.7	1.08	4.35				2.02
27	Cleaner-Scavenger Concentrate	4.1	3.7	20.0	16.4	17.5	1.17	10.00				3.47
28	Cleaner-Scavenger Tailings	9.9	3.2	8.9	101.2	104.3	1.07	2.03				1.43
	Cleaner 2 Feed	14.2	4.0	17.8	65.3	68.8	1.15	15.40				3.58
29	Cleaner 2 Dilution Water				14.9							
30	Cleaner 2 Feed	14.2	4.0	15	80.2	83.7	1.13	15.40				3.58
31	Cleaner 2 Concentrate	9.7	4.2	30.0	22.6	24.9	1.30	18.0				4.19
32	Cleaner 2 Tailings	4.5	3.7	7.2	57.6	58.8	1.06	9.76				2.27
	Cleaner 3 Feed	9.7	4.2	30.0	22.6	24.9	1.3	18.00				4.19
33	Cleaner 3 Dilution Water				32.3							
34	Cleaner 3 Feed	9.7	4.2	15	54.9	57.2	1.13	18.00				4.19
35	Cleaner 3 Concentrate	7.0	4.3	30.0	16.3	17.9	1.30	20.00				4.65
36	Cleaner 3 Tailings	2.7	3.9	6.6	38.6	39.3	1.05	12.86				2.99
37	Combined Tailings	129.9	3.1	29.4	312.0	353.3	1.25	0.23				0.65
	Cu Thickener Feed	7.0	4.3	27.2	18.7	20.3	1.26					
38	Cu Thickener Underflow	7.0	4.3	65.0	3.8	5.4	2.00					
39	Cu Thickener Overflow	0.0	4.3		14.9	14.9	1.00					
40	Cu Filter Cake	7.0	4.3	84.0	1.3	2.9	2.81					
41	Cu Filtrate				2.4							
	TailingsThickener Feed	129.9	3.1	26.3	364.2	405.6	1.22					
42	Tailings Thickener Underflow	129.9	3.1	65.0	70.0	111.3	1.80					
43	Tailings Thickener Overflow	0.0	3.1		294.2	294.2	1.00					
44	Tailings Filter Cake	129.9	3.1	88.0	17.7	59.1	2.50					
45	Tailings Filtrate				52.2							
	Reclaim water				309.2							
	Process water				324.0							
	Make-up water				14.8							

Stream	Stream Name	Overall Recovery % Cu	Overall Recovery % Pb	Overall Recovery % Zn	Overall Recovery % Ag	Overall Recovery % Au	Stage Recovery % Cu	Stage Recovery % Pb	Stage Recovery % Zn	Stage Recovery % Ag	Stage Recovery % Au	Metal Mass Cu (t)	Metal Mass Pb (t)	Metal Mass Zn (t)	Metal Mass Ag (kg)	Metal Mass Au (kg)	Comment	Factor
6	Ball Mill Feed	100.0	100.0	100.0	100.0	100						1.6977	0	0	0	0.1027	PFS Production Schedule	
8	Primary Mill Discharge																C/Load	300%
11	Cyclone Overflow	100.0				100						1.6977				0.1027		
13	Rougher Concentrate	74.0				30	74.0				30.0	1.2563	0	0	0	0.0308		
14	Rougher Tailings	26.0				70						0.4414	0	0	0	0.0719		
15	Scavenger Concentrate	20.0					77.0				22	0.3399				0.0158		
16	Scavenger Tailings	6.0										0.1015				0.0561		
20	Regrind Cyclone Overflow	44.1				29						0.7487				0.0300		
21	Regrind Cyclone Underflow																C/Load	300%
	Cleaner 1 Feed	143.8				69						2.4				0.1		
24	Cleaner 1 Feed	143.8				69						2.4				0.0709		
25	Cleaner 1 Concentrate	107.8				41	75				60	1.8307				0.0426		
26	Cleaner 1 Tailings	35.9				28						0.6				0.0284		
27	Cleaner-Scavenger Concentrate	24.1				14	67				50	0.409				0.014		
28	Cleaner-Scavenger Tailings	11.9				14						0.201				0.0		
	Cleaner 2 Feed	128.4				49						2.2				0.0507		
30	Cleaner 2 Feed	128.4				49						2.2				0.0507		
31	Cleaner 2 Concentrate	102.7				39	80				80	1.7435				0.0405		
32	Cleaner 2 Tailings	25.7				10						0.4				0.0		
	Cleaner 3 Feed	102.7				39						1.74				0.0405		
34	Cleaner 3 Feed	102.7				39						1.74				0.0405		
35	Cleaner 3 Concentrate	82.2				32	80				80	1.3948				0.0324		
36	Cleaner 3 Tailings	20.5				8						0.3				0.0		
37	Combined Tailings	17.8				82						0.3				0.1		

Table Ⅲ-4-15 Nominal design -case 2,000t/day

Nominal Design Case
2,000 tpd
80.0 % Crusher Availability
91.3 % Mill Availability

Stream	Stream Name	Solid t/h	Solid SG	% Solids w/w	Water m^3/h	Slurry m^3/h	Slurry t/m^3
1	ROM Feed	104	3.2	97.0			
2	Screen Feed	278	3.2	97.0			
3	Secondary Crusher Feed	94	3.2	97.0			
4	Tertiary Crusher Feed	80	3.2	97.0			
5	Screen Undersize	104	3.2	97.0			
6	Ball Mill Feed	91.3	3.199	97.0	2.8	31	3.00
7	Primary Mill Feed Water				58.8		
8	Primary Mill Discharge	365.1	3.2	70.0	156.5	271	1.93
9	Primary Mill Discharge Water				107.9		
10	Cyclone Feed	365.1	3.2	58.0	264.4	378	1.66
11	Cyclone Overflow	91.3	3.2	35.0	169.5	198	1.32
12	Cyclone Underflow	273.8	3.2	74.3	94.9	180	2.04
13	Rougher Concentrate	4.2	4.3	35.0	7.8	9	1.37
14	Rougher Tailings	87.1	3.1	35.0	161.7	189	1.31
15	Scavenger Concentrate	7.1	3.3	25.0	21.2	23	1.21
16	Scavenger Tailings	80.0	3.1	36.3	140.5	166	1.33
	Regrind mill new feed	9.8			32.1		
17	Regrind Mill Discharge	29.4	4.3	60.0	19.6	26	1.85
18	Regrind Mill Discharge Water				2.4		
19	Regrind Cyclone Feed	39.2	4.3	42.0	54.2	63	1.48
20	Regrind Cyclone Overflow	9.8	4.3	20.0	39.2	42	1.18
21	Regrind Cyclone Underflow	29.4	4.3	66.3	14.9	22	2.04
22	Regrind Mill Feed Water				4.7		
	Cleaner 1 Feed	17.0	3.7	16.6	85.4	90.0	1.14
23	Cleaner Dilution Water				10.8		
24	Cleaner 1 Feed	17.0	3.7	15.0	96.2	100.8	1.12
25	Cleaner 1 Concentrate	7.6	4.1	30.0	17.8	19.7	1.29
26	Cleaner 1 Tailings	9.3	3.4	10.7	78.4	81.1	1.08
27	Cleaner-Scavenger Concentrate	2.7	3.7	20.0	10.9	11.6	1.17
28	Cleaner-Scavenger Tailings	6.6	3.2	8.9	67.5	69.5	1.07
	Cleaner 2 Feed	9.4	4.0	17.8	43.5	45.9	1.15
29	Cleaner 2 Dilution Water				9.9		
30	Cleaner 2 Feed	9.4	4.0	15	53.5	55.8	1.13
31	Cleaner 2 Concentrate	6.5	4.2	30.0	15.1	16.6	1.30
32	Cleaner 2 Tailings	3.0	3.7	7.2	38.4	39.2	1.06
	Cleaner 3 Feed	6.5	4.2	30.0	15.1	16.6	1.3
33	Cleaner 3 Dilution Water				21.5		
34	Cleaner 3 Feed	6.5	4.2	15	36.6	38.1	1.13
35	Cleaner 3 Concentrate	4.6	4.3	30.0	10.8	11.9	1.30
36	Cleaner 3 Tailings	1.8	3.9	6.6	25.7	26.2	1.05
37	Combined Tailings	86.6	3.1	29.4	208.0	235.6	1.25
	Cu Thickener Feed	4.6	4.3	27.2	12.5	13.5	1.26
38	Cu Thickener Underflow	4.6	4.3	65.0	2.5	3.6	2.00
39	Cu Thickener Overflow	0.0	4.3		10.0	10.0	1.00
40	Cu Filter Cake	4.6	4.3	84.0	0.9	2.0	2.81
41	Cu Filtrate				1.6		
	TailingsThickener Feed	86.6	3.1	26.3	242.8	270.4	1.22
42	Tailings Thickener Underflow	86.6	3.1	65.0	46.6	74.2	1.80
43	Tailings Thickener Overflow	0.0	3.1		196.2	196.2	1.00
44	Tailings Filter Cake	86.6	3.1	88.0	11.8	39.4	2.50
45	Tailings Filtrate				34.8		
	Reclaim water				206.1		
	Process water				216.0		
	Make-up water				9.9		

Stream	Stream Name	Assay % Cu	Assay % Pb	Assay % Zn	Assay g/t Ag	Assay g/t Au	Overall Recovery % Cu	Overall Recovery % Pb	Overall Recovery % Zn	Overall Recovery % Ag	Overall Recovery % Au	Stage Recovery % Cu	Stage Recovery % Pb	Stage Recovery % Zn	Stage Recovery % Ag	Stage Recovery % Au	Metal Mass Cu (t)	Metal Mass Pb (t)	Metal Mass Zn (t)	Metal Mass Ag (kg)	Metal Mass Au (kg)	Comment	Factor
6	Ball Mill Feed	1.24				0.75	100.0	100.0	100.0	100.0	100						1.1318	0	0	0	0.0685	PFS Production Schedule	
8	Primary Mill Discharge																					C/Load	300%
11	Cyclone Overflow	1.24				0.75	100.0				100						1.1318				0.0685		
13	Rougher Concentrate	20.00				4.90	74.0				30	74.0				30.0	0.8375	0	0	0	0.0205		
14	Rougher Tailings	0.34				0.55	26.0				70						0.2943	0	0	0	0.0479		
15	Scavenger Concentrate	3.20				1.49	20.0					77.0				22	0.2266				0.0105		
16	Scavenger Tailings	0.08				0.47	6.0										0.0677				0.0374		
20	Regrind Cyclone Overflow	5.09				2.04	44.1				29						0.4992				0.0200		
21	Regrind Cyclone Underflow																					C/Load	300%
	Cleaner 1 Feed	9.59				2.79	143.8				69						1.6				0.0		
24	Cleaner 1 Feed	9.59				2.79	143.8				69						1.6				0.0473		
25	Cleaner 1 Concentrate	16.00				3.72	107.8				41	75				60	1.2205				0.0284		
26	Cleaner 1 Tailings	4.35				2.02	35.9				28						0.4				0.0189		
27	Cleaner-Scavenger Concentrate	10.00				3.47	24.1				14	67				50	0.273				0.009		
28	Cleaner-Scavenger Tailings	2.03				1.43	11.9				14						0.134				0.0		
	Cleaner 2 Feed	15.40				3.58	128.4				49						1.5				0.0338		
30	Cleaner 2 Feed	15.40				3.58	128.4				49						1.5				0.0338		
31	Cleaner 2 Concentrate	18.0				4.19	102.7				39	80				80	1.1623				0.0270		
32	Cleaner 2 Tailings	9.76				2.27	25.7				10						0.3				0.0		
	Cleaner 3 Feed	18.00				4.19	102.7				39						1.16				0.0270		
34	Cleaner 3 Feed	18.00				4.19	102.7				39						1.16				0.0270		
35	Cleaner 3 Concentrate	20.00				4.65	82.2				32	80				80	0.9299				0.0216		
36	Cleaner 3 Tailings	12.86				2.99	20.5				8						0.2				0.0		
37	Combined Tailings	0.23				0.65	17.8				82						0.2				0.1		
38	Cu Thickener Underflow																backfill cyc						
39	Cu Thickener Overflow																mass sp	0.6					