11-1 Piping for flotation new feed

11-1-1 Design concept

Piping design is made so that overflow of the primary grinding cyclones flows by gravity to rougher flotation head tanks. Check should be doneif it is available or not.

Flow rate: 13.92m3 /min・line
Pulp density: 35%by wt, 17% by volume
Sp.Gr. of pulp: 1.28
Size of solid: 135 microns 80% passing

11-1-2 Calculation of critical settling velocity of flow after Smorzilev

$$V_c = C'\sqrt{D}\left(\frac{C_V(S-1)V''_t}{\sqrt{d_{av}}}\right)^{\frac{1}{3}}$$

where C' : coefficient 8 or 9
C_V : pulp density by volume in [%/100]
S : specific gravity of solid ; 2.7
V''_t : settling velocity of particles in pipe ; 0.0005 [m/sec]
d_{av}: average particle size (50% passing) ; 0.000074 [m]
D :diameter inside pipe 0.478 [m]

Then

$$V_c = 9 \times \sqrt{0.478}\left(\frac{0.478 \times (2.7 - 1) \times 0.0005}{\sqrt{0.000074}}\right)^{1/3} = 0.83 \text{ m/sec}$$

Flow velocity in the pipe should be bigger than this Vc

11-1-3. Calculation of friction coefficient λ

1] λw :(λ for water flow)

a) Reynolds number of water

Re N= v・D/ν

where ν: viscosity at temperature of 25℃ 0.898×10^{-6} [m^2 /sec]
v: flow velocity 13.92×4/(60×3.14× 0.4782)=1.29 [m/sec]
D: diameter inside pipe 0.478 [m]
then Re N=(1.29×0.478) ×1/0.898×10^{-6}=0.687×10^{6}

b) λw : obtained by formula after Schiller & Hermann

$\lambda w = 0.054 + 0.396/Re^{0.3} = 0.054 + 0.396/56.4 = 0.0124$

c) λs : Reynolds number of slurry

$\lambda s = \lambda w \times 1.25 = 0.0124 \times 1.25 = 0.0155$

11-1-4 Calculation of friction loss

No.2 flotation unit has longer pipe line so that its friction loss is bigger than that of No.1 unit. Then, check for friction loss should be done based on No.2 unit.

The friction loss is determined by Darcy's formula.

$$hs = f \cdot \frac{L \cdot V^2}{D \cdot 2g}$$

where h_s : friction head loss in [m]
L : pipe length in [m]
D : pipe inner diameter in [m]
V : average flow velocity in [m/sec]
g : acceleration of gravity in [m/sec^2]
f : friction loss factor

then

$$hs = 0.0155 \times \frac{101.2 \times 1.290^2}{0.478 \times 2 \times 9.8} = 0.278 \text{ m}$$

Since actual head difference is approximately 4 meters, no problem will occur concerning to the friction loss. From point of view of flow resistance, it is possible to use smaller pipe, but it is better to adopt this size of pipe recommended by KREBS ENGINEERING in order to minimize abrasion problem.

11-2 Flotation cells

11-2-1 Design concept

1] Type selection

Several brands and models of flotation cells such as Galigher Agitair#120, #120A,#60, Denver DR-300, 200, 100, Fagergren#120 and Sala 600-2L, have been discussed on cell volume, installation area, impeller speed, air consumption, service life of wearing parts and operational experience and so on.

Consequently, Agitair #120 flotation cells were decided to be used.

2] Line capacity

Since grinding system has two circuit, basic flotation capacity was designed by two rougher lines, too.

Advantages and disadvantages of these flotation cells may be summarized as the following.

3] Advantages and disadvantages

[Advantages]

a)Reliability

They have long operational experiences since 1965 and 836 cells or more have been already installed in all over the world before 1973 and operating successfully.

b) Cell volume

They have the largest nominal cell volume among competitors at present and cell volume can be increased by adding of froth bars on cell lips until maximum volume of 400 cub.ft, if necessary.

c) Impeller speed

They have the smallest tip speed of impellers which may minimize maintenance cost by high durability in relatively coarse size operation.

d) Air pressure

They require the minimum air pressure positively supplied by blowers. This may be a critical factor in case of installation at high land because mechanical efficiency of blower drops enormously in low atmospheric pressure and this fact relates with rated output of blower driving motor.

e) Head loss

They require the minimum head loss per bank to let required volume of pulp flow and this enables installation of the maximum number of cells on same floor level.

f) Versatility

They have common mechanism with #60 cells, so that it is available to minimize stock inventory of spare parts.

[Disadvantages]

a) Complex structure

They have 4 shafts per each cell and take time for maintenance.

b) Power consumption

They require relatively higher power consumption including power consumed by blowers, i.e.0.13 ～0.16Hp/cu.ft. of the cell.

c) Installation area

They require the largest installation area among same class flotation cell because of shallow structure.

On introduction of Agitair #120A (single shaft type), discussions were also made. At time of 1973, however, they have only a little experiences except several concentrators such as 6 cells at Mt. Lyell of Australia, 15 cells at Pima in Arizona, USA, 12 cells at Miami in Arizona, 7 cells at Mufulira of Zambia and the largest user 28 cells for rougher and scavenger,21 cells for cleaner and recleaner at Lynn Lake in Canada.

Besides, some troubles due to sand settling in the cell and waving of froth surface have been reported.

11-2-2 Calculations of material balance

The following relationships can be consisted.

$$F = C + T$$

$$fF = cC + tT$$

$$R = cC/fF \times 100$$

where F: tonnage of feed [mt/h]

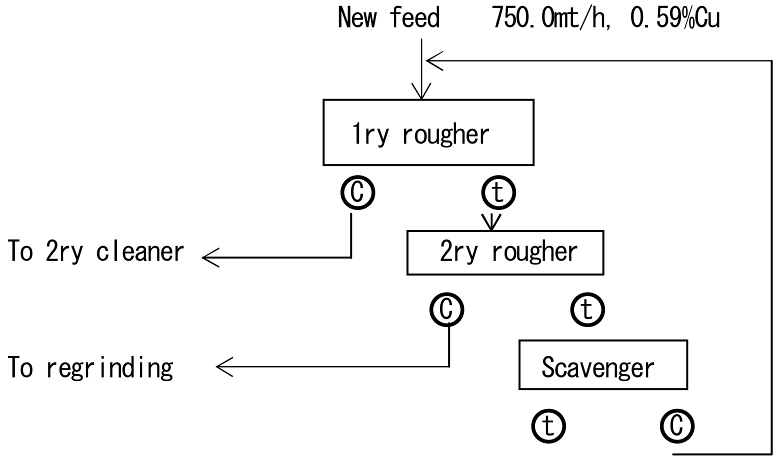
C: tonnage of concentrate [mt/h]

T: tonnage of tailing [mt/h]

f: copper grade of feed [%]

c: copper grade of concentrate [%]
t: copper grade of tailing [%]
R: copper recovery percent [%]

1]Rougher and scavenger



The following assumptions were made on tonnages, grades and recoveries、based on laboratory tests and pilot plant tests.

Tonnage of new feed:	750.0 mt/h,
Copper grade of new feed:	0.59%Cu
Copper grade of 1ry rougher concentrate:	10.00%Cu
Copper grade of 2ry rougher concentrate:	5.00%Cu
Copper grade of scavenger concentrate:	1.50%Cu
Copper distribution of 1ry rougher concentrate:	75.0% to new feed
Copper distribution of 2ry rougher concentrate:	18.0% to new feed
Copper distribution of scavenger concentrate:	3.0% to new feed

Then the next relationships can be consisted.

[Copper content of new feed]

fNF=0.59%/100×750.0mt/h=4.425 mt/h

[Copper content of 1ry rougher concentrate]

c1RC=4.425mt/h×75%/100=3.319mt/h

[Copper content of 2ry rougher concentrate]

c2RC=4.425mt/h×18%/100=0.797mt/h

[Copper content of scavenger concentrate]

cSC=4.425mt/h× 3%/100=0.133mt/h

[Copper content of 1ry rougher tailing]

t1RT=fF+cSC−c1RC=4.425+0.133−3.319=1.239mt/h

[Copper content of 2ry rougher tailing]

t2RT=t1RT−c2RC=1.239−0.797=0.442mt/h

[Copper content of scavenger tailing]

tST=t2RT−cSC=0.442−0.133=0.309mt/h

[Tonnage of scavenger concentrate]

SC =cSC/c=0.133mt/h÷1.5%/100=8.9mt/h

[Tonnage of 1ry rougher feed]

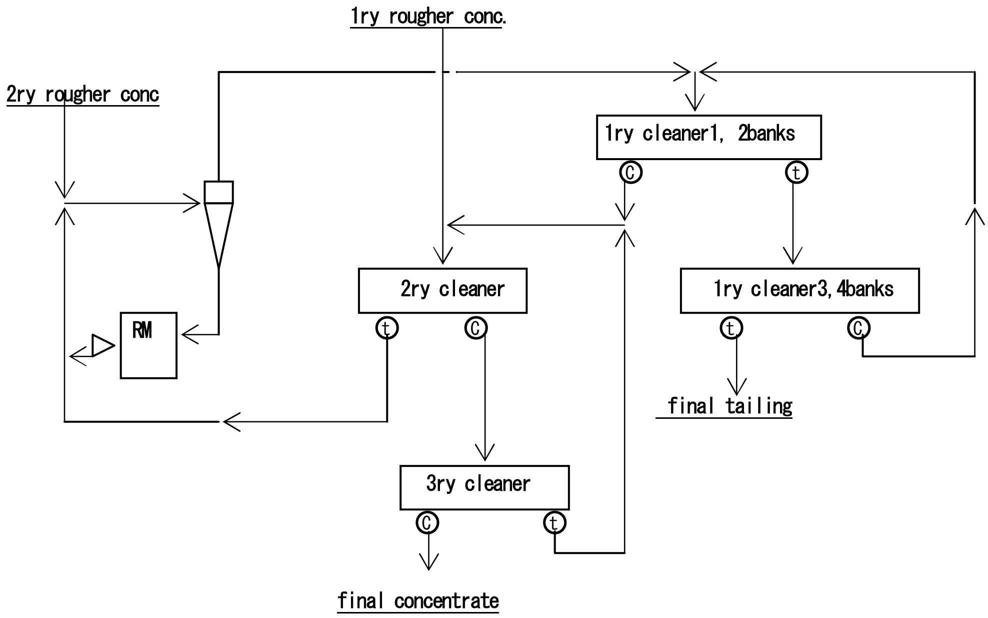
1RF＝NF＋SC＝750.0＋8.9＝758.9mt/h
[Tonnage of 1ry rougher concentrate]
1RC＝c1RC/c＝3.319mt/h÷10.0%/100＝33.2mt/h
[Tonnage of 1ry rougher tailing]
1RT＝1RF－1RC＝758.9－33.2＝725.7mt/h
[Tonnage of 2ry rougher concentrate]
2RC＝c2RC/c＝0.797mt/h÷5.0%/100＝15.9mt/h
[Tonnage of 2ry rougher tailing]
2RT＝1RT－2RC＝725.7－15.9＝709.8mt/h
[Tonnage of scavenger tailing]
ST ＝2RT－SC＝709.8－8.9＝700.9mt/h

Flow rates of products were calculated assuming the following conditions.

Materials	Sp.Gr. of solid	Pulp density	Pulp Sp.Gr.
New feed	2.7	35 wt%	1.283
1ry Rougher concentrate	4.0	30	1.290
2ry Rougher concentrate	4.0	25	1.208
Scavenger concentrates	3.0	25	1.201

[Flow rate of new feed]
750.0mt/h÷(60min/h×1.283×35%/100)＝27.833 m3 /min
[Flow rate of 1ry Rougher concentrate]
33.2mt/h÷(60min/h×1.290×30%/100)＝1.430 m3 /min
[Flow rate of 2ry Rougher concentrate]
15.9mt/h÷(60min/h×1.208×25%/100)＝0.877 m3 /min
[Flow rate of scavenger concentrate]
8.9mt/h÷(60min/h×1.201×25%/100)＝0.493 m3 /min
[Flow rate of 1ry Rougher feed]
27.833＋0.493＝28.326m3 /min
[Flow rate of scavenger tailing]
28.326－1.430－ 0.877－0.493＝25.526m3 /min

2] Cleaner and recleaner



The following assumptions and calculations were made on the base of former calculations.

Product	tonnage	copper grade	Cu distribution	Cu content
1RC	33.2mt/h	10.00%	75.0% to N.Fd	3.319mt/h
2RC	15.9	5.00	18.0	0.797
1C-1,2BC	-	12.00	30.0	-
1C-3,4BC	-	5.00	6.0	-
2CC	-	20.00	95.0	-
3CC	-	25.00	90.0	-

[Copper content of 1ry cleaner 1,2 bank concentrate]
　c1C1,2BC＝4.425mt/h×30%/100＝1.328mt/h
[Copper content of 1ry cleaner 3,4 bank concentrate]
　c1C3,4BC＝4.425mt/h× 6%/100＝0.266mt/h
[Copper content of 2ry cleaner concentrate]
　c2CC＝4.425mt/h× 95%/100＝4.204mt/h
[Copper content of 3ry cleaner concentrate]
　c3CC＝4.425mt/h× 90%/100＝3.983mt/h
[Copper content of 3ry cleaner tailing]
　t3CT＝c2CC－c3CC＝4.204－3.983＝0.221mt/h
[Copper content of 2ry cleaner feed]
　f2CF＝c1RC＋c1C1,2BC＋t3CT＝3.319＋1.328 ＋0.221＝ 4.868mt/h
[Copper content of 2ry cleaner tailing]
　t2CT＝f2CF－c2CC＝4.868－4.204＝0.664mt/h
[Copper content of 1ry cleaner 1,2 bank feed]

f1C1,2BF＝c2RC＋t2CT＋c1C3,4BC＝0.797＋0.664＋0.266＝1.727mt/h
[Copper content of 1ry cleaner 1,2 bank tailing]
t1C1,2BT＝f1C1,2BF－c1C1,2BC＝1.727－1.328＝0.399mt/h
[Copper content of 1ry cleaner 3,4 bank tailing]
t1C3,4BT＝t1C1,2BT－c1C3,4BC＝0.399－0.266＝0.133mt/h

[Tonnage of 3ry cleaner concentrate]
3CC＝c3CC/c＝3.983mt/h÷25.00%/100＝15.57mt/h
[Tonnage of 2ry cleaner concentrate]
2CC＝c2CC/c＝4.204mt/h÷20.00%/100＝21.02mt/h
[Tonnage of 3ry cleaner tailing]
3CT＝2CC－3CC＝21.02－15.57＝5.45mt/h
[Tonnage of 1ry cleaner 1,2 bank concentrate]
1C1,2BC＝c1C1,2BC/c＝1.328÷12.00%/100＝11.07mt/h
[Tonnage of 2ry cleaner feed]
2CF＝1RC＋1C1,2BC/＋ 3CT＝33.20＋11.07＋5.45＝49.72mt/h
[Tonnage of 2ry cleaner tailing]
2CT＝2CF－2CC＝49.72－21.02＝28.70mt/h
[Tonnage of 1ry cleaner 3,4 bank concentrate]
1C3,4BC＝c1C3,4BC/c＝0.266÷ 5.00%/100＝5.32mt/h
[Tonnage of 1ry cleaner 1,2 bank feed]
1C1,2BF＝2RC＋2CT.＋1C3,4BC＝15.90 ＋28.70＋5.32＝49.92mt/h
[Tonnage of 1ry cleaner 1,2 bank tailing]
1C1,2BT＝1C1,2BF－1C1,2BC＝49.92－11.07 ＝38.85mt/h
[Tonnage of 1ry cleaner 3,4 bank tailing]
1C3,4BT＝1C1,2BT－1C3,4BC＝38.85－5.32 ＝33.53mt/h
[Tonnage of final tailing]
T＝ST＋1C3,4T＝700.90＋33.53＝734.43mt/h

[Flow rate of 1ry cleaner 1,2 bank feed]
49.92mt/h÷(60min/h×1.234×25%/100)＝2.697 m3 /min
[Flow rate of 2ry cleaner feed]
49.72mt/h÷(60min/h×1.182×20%/100)＝3.358 m3 /min
[Flow rate of 3ry cleaner feed]
21.02mt/h÷(60min/h×1.201×22%/100)＝1.326 m3 /min
[Flow rate of 3ry cleaner concentrate]
15.57mt/h÷(60min/h×1.234×25%/100)＝0.841 m3 /min
[Flow rate of 1ry cleaner 3,4 bank tailing]
33.53mt/h÷(60min/h×1.180×21%/100)＝4.116 m3 /min
[Flow rate of final tailing]
25.526＋4.116＝29.6426 m3 /min

11-2-3. Calculations of number of cell requirement N

$$N = \frac{Q \cdot t}{V}$$

where Q: Flow rate of pulp [cu ft/min]

T: Flotation time required [min]

V: Efficient cell volume [cu ft]

1] Rougher and scavenger

Q: 28.326m^3 /min=1,000 cu ft/min

T: 10 min

V: 356 cu ft×0.9 =320 cu ft

then N=(1,000×10)/ 320

=31.2→32 cells=4 cells/bank×4 banks/row ×2 rows

2] Primary cleaner

Q: 3.358m3 /min=118.6 cu ft/min

T: 5 min

V: 60 cu ft×0.9 =54 cu ft

then N= (118.6× 5)/54=11.0→12 cells=6 cells/bank×2 banks/row ×1 row

3] Secondary cleaner

Q: 2.697m3 /min=95.2 cu ft/min

T: 12 min

V: 60cu ft×0.9 =54 cuft

then N= (95.20×12)/ 54=21.2 →24 cells=6 cells/bank×4 banks/row ×1 row

4] Tertiary cleaner

Q:1.326m3 /min=46.8cu ft/min

T: 7 min

V:60 cu ft×0.9 =54 cu ft

then N=(46.8× 7)/ 54=6.1 →6 ells=6 cells/bank×1 banks/row ×1 row

5] Quaternary cleaner

Taking account of trouble caused by serpentinized ore, it is recommendable to install quaternary cleaner of same size with the tertiary cleaner.

Then, total flotation time for cleaning will be as the following.

12+ 5+ 7+ 7=31.0min

11-3 Blowers for flotation

11-3-1 Design concept

Based on manufacturer' s experiences, the followings were assumed.

Type: Single suction turbo blower

Capacity: 660Nm^3 /min

Pressure: Pd :1,300 mmAq, Ps :−50 mmAq

11-3-2. Required air volume and power requirement

#120 Agitair cell: 500 cu ft/min at 21℃=14.2 Nm^3 /min・cell

60 Agitair cell: 120 cu ft/min at 21℃= 3.4 Nm^3 /min・cell

Number of flotation cells to be installed

#120: 32 cells

60: 48 cells

Total air requirement

Q_r=14.2Nm^3 /min・cell×32cells+ 3.4Nm^3 /min・cell×48 cells

=617.6Nm^3 /min

Taking account of loss ,capacity of the blower should be 660Nm^3 /min・

Estimated suction pressure due to strainer and silencer: P_s :−50 mmAq

Estimated delivery pressure: P_d :1,300 mmAq,

Pressure difference:$\Delta p = P_d - P_s$ =1,300−(−50)=1,350 mmAq

Suction temperature: Ts =17～23℃

Place of installation: 1,400m above sea level,

Atmospheric pressure: 642 mmHg=642 mmHg×13.6 Aq/Hg=8,731 mmAq

Total suction pressure: 8,731−50=8,681 mmAq

Statical power consumption is calculated by the following formula.

A Hp = (Q_s ×Δp) /6,120 [kw]

where Q_s :Air volume at suction side in [m^3/min]

660Nm^3 /min×10,632/8,681=780 S* m^3 /min

S*:substantive then A=780×1,350/6,120=172 kw

Brake horse power: RP=AHp /ηs

whereηs : statical efficiency of blower

then RP=172 kw/0.62=277kw

In the case where temperature at suction side descends to 17℃, the brake horse power will rise to

277kw×(273+23)/(273+17)=283 kw

Including surplus of 8%, installed motor power will be 283/0.92= 300 kw.

In actual installation, this motor is devided into two 150 kw motors driving in tandem system.

Revolutionary speed of the motors will be 1,460～1,470 rpm, by adopting 4 pole motors by direct coupling for money saving.

Delivery opening and valve are 600 mm in diameter but piping of down stream should be 1,000 mm dia. for saving friction loss.

11-4 Reagent addition

11-4-1 Design concept

Based on laboratory test by drilling core samples and pilot plant tests by ore derived from adit, the following reagent will be used at initial start-up stage.

Frother:	Dowfroth #250,	5～30 g/dry mt of ore milled
Collector:	Dow Z-200,	2～30 g/dry mt of ore milled
pH Regulator:	Slaked lime,	750 g/dry mt of ore milled
Activator:	Sodium sulphide,	3～100 g/dry mt of ore milled

As a reagent feeder, cup feeder and metering pump have been discussed.

From point of view of reliability for operation and maintenance, cup feeder type of CLARKSON Co. was selected except lime feeder.

Lime will be fed through automatic valves controlled by pH controllers.

Model: E-type Clarkson feeder
Capacity:10～2,000 m・/min
Material:18-8 stainless steel
No. of cups: 20

Since all reagents will be supplied in the state of liquids in drums, they will be fed as original liquid through piping.

The slaked lime will be supplied in paper bags. Bags will be opened on site and the lime will be mixed with water as 10% slurry and circulated between storage tank and flotation bay by slurry pumps and main piping. Lime milk will be fed by automatic valves through branched pipes directly into flotation cells.

11-4-2. Calculation of reagent flow rates

1] Reagents except slaked lime

Minimum flow rate: 375 mt/h・line÷60min/h×2g/mt×1.0 mℓ/g=12.5 mℓ/min・line
Maximum flow rate: 375 mt/h・line÷60min/h×100g/mt×1.0 mℓ/g=625ℓ/min・line
This type of the reagent feeder will meet above calculated demand.

2] Slaked lime

The slaked lime will be fed in the state of lime milk as pH regulator to roughing and cleaning circuits. Total consumption is estimated to be 750 g/dry mt of ore milled.

So, required tonnage T will be

0.75kg/mt×750 mt/h=562.5 kg/h

Then, assuming milk density as 10% Wt, required milk volume will be

562.5 kg/h×100/10÷1.070kg/ℓ=5,257 ℓ/h=0.088m^3/min

Since pH value is expressed by logarithm, when pH varies 0.3 lime requirement will be doubled. Then we should estimate the maximum probable consumption to be double of above-said value, and we should install a system which will be able to circulate triple volume of the maximum and consume one third of the circulating volume in order to maintain smooth and quick control of pH in the flotation circuits.

Hence capacity of the circulating pump Q will be

Q=0.088m^3/mi×2×3=0.528 m^3/min.

Specification of the actual installed pump:

0.580 m^3/min×30 mH×11 kw×2 sets (1set stand-by)

Storage tank volume of the lime milk V shall be 4 hours of circulating pump capacity.

V=0.580 m^3/min×4h×60min/h=139.2 m^3