10) Grinding

10-1. Design concept

a) Grinding capacity of one circuit system

Generally speaking, design tendency of grinding capacity of one circuit system has been changed to select larger system where it can treat 10,000 mt a day.

Since design capacity of the Mamut Mill is 18,000 mt/day, number of the grinding system should be two. Spare mill was omitted because of the reason why the recent grinding mill has higher reliability and takes expensive construction costs.

b) Grinding stage number

It is the newest tendency to adopt one stage grinding of ball mill. Two stage grinding system, however, consisting of a rod mill and also a ball mill was considered as the best selection, because they can accept relatively bigger mill feed derived by troublesome condition due to serpentine. About ten years ago, there was tendency to install two ball mills after one rod mill but in recent mill designs it has changed to install one ball mill after one rod mill, because of difficulty to distribute the feed to ball mills properly and technical progress to manufacture large mills. Hence we decided to adopt this system of one rod mill and one ball mill.

10-2. Vibrating feeders for feeding conveyors for rod mills

Two units of vibrating feeders shall feed each rod mill system as a rule.

750mt/hr÷(2 circuits×2 units)=187.5t round up to→190 mt/h・unit averaging.

The maximum capacity should be 130% of the averaging one.

Then190 mt/h×1.3=247mt/h round up to→250 mt/h・unit max.

b) Selection of type.

In order to minimize number of manufacturers, we selected

YASUKAWA & Company・KEB-32-4 950 mm-Wide×1,500 mm Long、with 6° of slope.

Based on catalogue data, we determined motor power of 5 kw×2 motors per each unit.

10-3.Rod mill feeding conveyors

a) Belt speed

$Q_m = Q_t / \gamma = 60 \cdot k_1 \cdot k_2 \cdot (0.9b - 0.05)^2 \cdot v$

Where $Q_t = 190$mt/h, $\gamma = 1.7$mt/ m^3 , $b = 0.9$ m,

[No,16,17,18 &19BC]

Conveyor slope 17° , $k_1 = 0.87$, 20° of both trough angle and surge sngle, $k_2 = 0.1245$

Then the minimum belt speed v will be

$v = 190 \div \{60 \times 1.7 \times 0.87 \times 0.1245 \times (0.9 \times 0.9 - 0.05)^2\}$

$= 29.6$m/min

Taking into account of surplus 10% , $29.6 \times 1.1 = 32.5$m/min.

Expecting of operation by one unit system, the capacity should be double.

Consequently、 $v = 32.5$m/min$\times 2 = 65$m/min.

[No,20 & 21BC]
Conveyor slope 8°, k_1=0.97 , 20° of both trough angle and surge sngle, k_2=0.1245
Then the minimum belt speed v will be
$v=380\div\{60\times1.7\times0.97\times0.1245\times(0.9\times0.9-0.05)^2\}$
$=53.4$m/min
Taking into account of surplus 50% in the case of extraordinary operation、
the belt speed should be 53.4m/min×1.5=80m/min.

[No,22 & 23BC]
Conveyor slope13°, k_1=0.92, 20° of both trough angle and surge angle, k_2=0.1245
Then the minimum belt speed v will be
$v=380\div\{60\times1.7\times0.92\times0.1245\times(0.9\times0.9-0.05)^2\}$
$=56.3$m/min
By the same reason with above-said 21 & 22 BC, v should be 80m/min.

b) Calculations of required powers

Conveyor power without load P_1 :	$P_1=0.06fWv(l+l_0)/367$	[kw]
Power by horizontal load P_2 :	$P_2=f\cdot Qt\ (l+l_0)/367b$	[kw]
Power by vertical load P_3 :	$P_3=\pm h\cdot Q_t/367$ downward −	[kw]
Required power P:	$P=P_1+P_2+P_3$	[kw]

[No,16,17,18 &19 BC]
Q_t=190mt/h, h=1.5m, l=6.5m, v=65m/min, b=900mm, W=63kg/m
$P_1=0.06\times0.03\times63\times80\times(6.5+49)/367=1.15$ kw
$P_2=0.03\times190\times(6.5+49)\times1/367=0.86$ kw
$P_3=1.50\times190\times1/367=0.78$ kw
$P=1.15+0.86+0.78=2.79$ kw
We will use recommended power of 2.79 kw/0.8=3.49→5.5kw .

[No,20 & 21BC]
Q_t=380mt/h, h=1.5m, l=48.5m, v=80m/min, b=900mm, W=63kg/m
$P_1=0.06\times0.03\times63\times80\times(6.5+49)/367=2.49$ kw
$P_2=0.03\times380\times(48.5+49)\times1/367=3.03$ kw
$P_3=1.50\times380\times1/367=1.55$ kw
$P=2.49+3.03+1.55=7.07$ kw
We will use recommended power of 7.07 kw/0.8=8.84→11 kw .

[No,22 & 23BC]
Q_t=380mt/h, h=4.4m, l=18.2m, v=80m/min, b=900mm, W=63kg/m
$P_1=0.06\times0.03\times63\times80\times(18.2+49)/367=1.71$ kw
$P_2=0.03\times380\times(18.2+49)\times1/367=2.09$ kw
$P_3=4.40\times380\times1/367=4.56$ kw
$P=1.71+2.09+4.56=8.36$ kw
We will use recommended power of 8.36 kw/0.8=10.45→15 kw.

c) Calculations of effectve tensions
F_p ＝6,120P/v

[No,16,17,18 &19 BC]
F_p ＝6,120×2.79/65＝263 kg

[No,20 & 21BC]
F_p ＝6,120×7.07/80＝541 kg

[No,22 & 23BC]
F_p ＝6,120×8.36/80＝640 kg

d) Calculation of effective tension
$F_2 = F_p / e^{\mu\theta-1}$

[No,16, 17, 18 & 19 BC]
F_2 ＝263×0.93＝ 245 kg (wrap angle 190°, take-up with screw type

[No,20 & 21BC]
F_2 ＝541×0.59＝ 319 kg (wrap angle 190°, take-up with heavy weight)

[No,22 & 23BC]
F_2 ＝640×0.59＝ 378 kg (wrap angle 190°, take-up with heavy weight)

e) 傾斜張力の計算
$F_3 = W_1 \cdot L_1 (\sin A - f \cdot \cos A)$
Where　W_1 : belt weight　[kg/m]
　L_1 : length of slope　[m]
　A : slope angle　[°]
　f : revolutionary friction coefficient of roll

Belt widths and weights

Belt width mm	400	500	750	900	1,500	1,200	1,800
W_1 kg	4.5	6.5	12	14	18	21	58

[No,16,17,18 & 19 BC]
F_3 ＝14× 6.67× 0.200＝ 19kg

[No, 20 & 21 BC]
F_3 ＝14×17.21× 0.117＝ 28kg

[No, 22 & 23 BC]
F_3 ＝14×18.72× 0.210＝ 55kg

f) Maximum tension

$F_{max} = F_p + F_2$

[No,16,17,18 & 19 BC]

$F_{max} = 263 + 245 = 508$ kg

10-4. Rod mills

10-4-1. Design concept

Rod mills were designed based on the following design concept.

Grinding method :	wet open circuit
Feeder :	spout
Feed size:	13,000 micron 80% passing
Product size :	900 micron 80% passing
Work index :	10.8 kwh/st-grinding feed
New feed moisture :	4%
Pulp density in mill :	65～75%wt
Discharge pulp density :	58%
Grinding capacity :	375 dry mt/h・set
Discharge method :	overflow through discharge trommel
Driving method :	side drive by spur gears with air clutch
Grinding media tonnage :	max. ; 177 mt、averaging ; 155 mt
Revolutionary speed :	13.15 rpm, 62.6% of critical speed
Supporting method :	trunnions
Liners :	single wave chrome・molybdenum steel
Motor:	open drip proof synchronous motor
Lubricating method :	gears; oil spray
	trunnions ; oil pump & oil cups
Trunnion cooling method :	reinforced water circulation

10-4-2. Required power

1] Required power per ton of ore milled W_r

$W_r = W_i\,(10/\sqrt{P} - 10/\sqrt{F}) \cdot f_r \cdot f_d \times 1.1$ [kwh/mt]

where W_i : work index [kwh/st]

F : feed size 13,000 micron 80% passing $\sqrt{F}$; 114

P : product size 900 micron 80% passing $\sqrt{P}$; 30

f_r : reduction factor, after F.C.Bond.at reduction ratio RR=F/P=16, $f_r = 1.0$, at RR<14 $f_r < 1$,

RR=14.5→ $f_r = 0.98$,

f_d : diameter factor, generally speaking, it is said that the mill capacity refers to2.5～3 power of mill daiameter. The Japanese cement association, however, denies this theory.

At D>8', $f_d < 1.0$

In our case, we determined $f_d = 1.0$ for safety/

Hence

$W_r = 10.8 \times (10/30 - 10/144) \times 0.98 \times 1.0 \times 1.1 = 2.859$ kwh/mt

2] Theoretical required power per one set of mill Wt

$W_t = W_r \times$ (tonnage milled)

$= 2.859$ kwh/mt × 375 mt/h・set = 1,072 kw/set

3] calibration by conditioning factor

Since actual power is influenced by such many conditions as mechanical efficiency, shock load, broken rods etc, W_t should be calibrated as the following.

$W_a = W_t \cdot f_m \cdot f_s \cdot f_b$

Where	W_a	:installed motor power	[kw]
	fm	: mechanical efficiency	0.96
	f_s	: shock load factor	0.90
	f_b	: broken rod factor	0.95

$W_a = 1{,}072$ kw/ set × 1/0.96 × 1/0.90 × 1/0.95 = 1,306 kw/ set

Taking account of the difficulties in crushing due to the serpentin, feed size may be bigger than that of we expected here, so it is advisable to hold larger surplus for the motor power. Hence we determined to install the motor with 1,400 kw for each rod mill.

4] Check calculations by power per rod ton KW_r

$KW_r = 1.07 \cdot D^{1/3} \times (6.3 - 5.4V_p) \cdot Cs \times 1.1$

where	D	:mill diameter on inside liner (unit:ft)	13.5ft $D^{1/3} = 2.38$
	V_p	:ratio of grinding media to mill volume	35%/100
	C_s	:critical speed ratio	62.6%/100

$KW_r = 1.07 \times 2.38 \times (6.3 - 5.4 \times 0.35) \times 0.626 \times 1.1 = 7.74$ kwh/mt-rod

Tonnage of rod charged

$W = \pi/4 \cdot D^2 \cdot L \cdot Vp \cdot \rho$

where	π	: the circular constant	3.14
	D	: mill diameter on inside liner (unit:ft)	13.5ft
	L	:rod length (unit:ft)	19.0ft
	V_p	: ratio of grinding media to mill volume	35%/100
	ρ	: rod bulk density (unit: mt/ft^3)	0.172 mt/ft^3

$W = 3.14/4 \times 13.5^2 \times 19.0 \times 0.35 \times 0.172 = 163.6$ mt/set

Drawable power i.e. $W_d = KW_r \times W \times f_b$ should be bigger than the theoretical required power per one set of mill W_t. If not, power would be consumed in vain and no use for grinding.

$W_d = 7.74$ kwh/mt-rod × 163.6mt/set × 0.95 = 1,202 kw > Wt = 1,072kw

Then, above result meets the requirement.

10-4-3.Rod size selection

The maximum rod size can be determined after empirical equation of F.C. Bond.

$$B = \sqrt{\frac{F \cdot Wi \sqrt{\frac{S}{\sqrt{D}}}}{K \cdot Cs}}$$

Where B : the maximum size of rod or ball in inch.

F : feed size 13,000 micron 80% passing
W_i : work index 10.8 kwh/st
K : empirical factors rod ; 300, ball ; 200
C_s : critical speed ratio 62.6%
S : actual specific gravity of the ore 2.7
D : mill diameter on inside liner (unit:ft) 13.5ft

$$\text{Then}\quad B = \sqrt{\frac{13,000 \times 10.8}{300 \times 62.6}\sqrt{\frac{2.7}{\sqrt{13.5}}}} = 2.54\ \text{in}$$

In the actual operation, especially in the case where feed size may drift into coarser size, it should take 20〜30% of safety factor.

So 2.54×1.3=3.3in rounded up to→3.5 in.

Hence, recommended rod size distribution can be estimated after table of Allis-Chalmers as the following.

Rod dia.	%	estimation	actual charge*
3-1/2 in	20	31mt	62mt
3	33	51	62
2-1/2	21	33	53
2	26	40	-
Tot	100	155	177

* Tonnage of initial charge will be the maximum volume in order to check the mill capacity.

Cf. Maximum rod size and initial size distribution to be charged(%)

Rod dia. in	5	4-1/2	4	3-1/2	3	3-1/2
5	18	-	-	-	-	-
4-1/2	22	20	-	-	-	-
4	19	23	20	-	-	-
3-1/2	14	20	27	20	-	-
3	11	15	21	33	31	-
2-1/2	7	10	15	21	39	34
2	9	12	17	26	30	66
Tot	100	100	100	100	100	100

10-4-5. Estimated grinding media consumption

1] Grinding rods

Rod consumption may be estimated based on laboratory tests and experiences of other porphyry copper mines.

rod : 400 g/mt, 189 mt/ month

Examples of other porphyry copper mines.

Gibraltar (Canada, B.C.) 13'-1/2×20'×3: 239 g/t

Marcopper (Philippines) 13'-1/2×20'×2: 425 g/t
Butte (USA, Montana) 9'×12'×6: 149 g/t
Palabora(Transvaal) 12'×16'×5: 160 g/t

2] Liners

Liner consumptions will vary depending on liner materials, mill speed, mill diameter, grinding medium size, loading conditions, ore hardness, pH of grinding water and shape of liners etc. After BECHTEL & Company , averaging rates of liner consumptions for common rod mill/ ball mill grinding operations range in the extent of 15～30 lb/h/1,000 sq ft･inner mill surface area. This means that we can estimate liner consumptions of 0.015～0.3 lb/kwh or 17～340 g/. In the case of white cast iron, however, much higher consumptions were reported and high manganese steel liners range between10～35 g/mt only. Data obtained in other copper mills are shown in the following.

Ray : 10'×13' 25 g/t
San Manuel : 10'×13' 30 g/t

On the liner material, chromium-molybdenum anti-abrasion steel is the most reliable. Since Ni-hard is brittle, it is not suitable for rod mill liner. At present, rubber rod mill liners are not available for large diameter rod mills.

Consequently, we estimate about 10 months of the liner life, this value corresponds to 34g/t-ore milled.

10-5. Ball mills

10-5-1. Design concept

Primary ball mill design was done on the bases of the following conditions.

Grinding method :	wet closed circuit with cyclones
Feeder :	spout
Feed size:	Grinding method : wet open circuit
Feeder :	spout
Feed size:	900 micron 80% passing
Product size :	135 micron 80% passing
Work index :	10.8 kwh/st-grinding feed
New feed pulp density	75%wt
Pulp density in mill :	65～72%wt
Grinding capacity :	375 dry mt/h・set
Discharge method :	overflow through discharge trommel
Driving method :	side drive by helical gears with air clutch
Grinding media tonnage :	max. ; 226 mt
Revolutionary speed :	13.58 rpm, 70.5% of critical speed
Supporting method :	both trunnions
Liners :	Skega F-type rubber liners
Motor:	open drip proof synchronous motor
Lubricating method :	gears; oil spray
	trunnions ; oil pump & oil cups
Trunnion cooling method :	reinforced water circulation

Circulating load ratio: 350%

10-5-2. Required power

1] Required power per ton of ore milled W_r

$W_r = W_i\,(10/\sqrt{P} - 10/\sqrt{F}) \cdot f_r \cdot f_d \times 1.1$ [kwh/mt]

where W_i : work index [kwh/st]

F : feed size 900 micron 80% passing $\sqrt{F}$; 30.0

P : product size 135 micron 80% passing; $\sqrt{P}$; 11.6

f_r : reduction factor, $f_r = 1.0$,

f_d : $f_d = 1.0$ for safety.

Then $W_r = 10.8 \times (10/11.6 - 10/30) \times 1.0 \times 1.0 \times 1.1 = 6.28$ kwh/mt

2] Theoretical required power per one set of mill Wt

$W_t = W_r \times$ (tonnage milled)

$= 6.28$ kwh/mt × 375 mt/h・set = 2,355 kw/set

3] calibration by conditioning factor

Since actual power is influenced by mechanical efficiency, W_t should be calibrated as the following.

$W_a = W_t \cdot f_m$

Where W_a : installed motor power [kw]

fm : mechanical efficiency 0.96

$W_a = 2{,}355$ kw/ set × 1/0.96 = 2,453 kw/ set round up to 2,500kw.

4] Check calculations by power per ball ton KWr

$KW_r = 3.1 \cdot D^{0.3} \times (3.2 - 3V_p) \cdot C_s \times (1 - 0.1/2^{9-10C_s})$

where D : mill diameter on inside liner (unit:ft) 16ft $D^{1/3} = 3.68$

V_p : ratio of grinding media to mill volume 38%/100

C_s : critical speed ratio 70.5%/100

$KW_r = 3.1 \times 3.68 \times (3.2 - 3 \times 0.38) \times 0.705 \times (1 - 0.1/3.86)$

$= 16.14$ kw/t- ball charged

Tonnage of ball charged

$W = \pi/4 \cdot D^2 \cdot L \cdot V_p \cdot \rho$

where π : the circular constant 3.14

D : mill diameter on inside liner (unit:ft) 16.0 ft

L : mill liner inside length (unit:ft) 23.0 ft

V_p : ratio of grinding media to mill volume 38%/100

ρ : rod bulk density (unit: mt/ft^3) 0.172 mt/ft^3

$W = 3.14/4 \times 16.0^2 \times 23.0 \times 0.38 \times 0.126 = 221.4$ mt/set

Drawable power i.e. $W_d = KW_r \times W \times f_b$ should be bigger than the theoretical required power per one set of mill W_t. If not, power would be consumed in vain and no use for grinding.

Wd = 16.14kwh/mt-ball × 221.4mt/set = 3,567 kw > Wt = 2,355kw

Then, above result meets the requirement.

10-5-3. Selection of ball size

The maximum ball size can be determined after empirical equation of F.C. Bond.

$$B = \sqrt{\frac{F \cdot W_i \sqrt{\frac{S}{\sqrt{D}}}}{K \cdot C_s}}$$

Where　B : the maximum size of rod or ball in inch.

F : feed size 900 micron 80% passing
W_i : work index　10.8 kwh/st
K : empirical factors　rod ; 300, ball ; 200
C_s : critical speed ratio　70.5%
S : actual specific gravity of the ore　2.7
D : mill diameter on inside liner　(unit:ft) 16.0 ft

Then　$$B = \sqrt{\frac{900 \times 10.8 \sqrt{\frac{2.7}{\sqrt{16.0}}}}{200 \times 62.6}} = 0.75 \text{ in}$$

In the actual operation, especially in the case where feed size may drift into coarser size, it should take 2.0～3.0 of safety factor.

So 0.75×2.5=1.87 2　rounded up to→2.0 in.

Hence, recommended rod size distribution can be estimated after table of Allis-Chalmers as the following.

2 in	40%	@90 mt
1-1/2	45	@100
1	15	@30
Tot	100	@220

10-5-5. Ball consumption

Ball consumption can be estimated actual performances in the pilot plant tests As the following.

Ball : 450 g/t,　213 mt/month

Operating data of other concentrators show 266～645 g/t in the case of two stage grinding by rod mill and ball mill for porphyry copper ores.

10-5-6. Estimation of liner consumption

The rate of liner consumption is highly related to the mill revolutionary speed. If the mill speed exceeds beyond 80～82% of the critical speed, liner consumption proceeds to such extent where the rubber liners are not economical at present. In the case of the Mamut Mill, however, the mill speed remains in 70.5% of the critical one, no problem will be expected.

Based on experiences of the liner manufacturer, liner lives will be expected as 12,000～18,000 hours of operation in the case where ball size ranges 2～3 inches and the mill speed is 70.5% of the critical one.　The lives of lifter bars are shorter than those of shell plates in general and expected to be about one year. In the case the lifter bars are symmetry and can be reversible, the lives will be nearly double by reversion.

Liner weights

1] Shell plates

Weights of shell plates are estimated by the following equation.

$W_s = N \cdot A \cdot L \cdot \rho / 1,000$

where W_s : weight of shell plates [kg]

N : number of shell plates in a set of mill

A : cross sectional area of a shell plate [cm²]

L : effective length of inside shell [cm]

ρ : specific gravity of rubber (1.0 approximately)

Then $W_s = 40 \times (20.2 \times 5.0) \times 701 \times 1/1,000 = 2,832$ kg/set

2] Lifter bars

$W_{sl} = N \cdot A \cdot L \cdot \rho / 1,000$

where W_{sl} :weight of lifter bars [kg]

N : number of the lifter bars in a set of mill

A : cross sectional area of a lifter bar [cm²]

L : effective length of inside shell [cm]

ρ : specific gravity of rubber (1.0 approximately)

then $W_{sl} = 40 \times 174 \times 701 \times 1/1,000 = 4,963$ kg/set

3] Head plate liners

$W_h = 2\{\pi/4 \cdot (D^2 - d^2) - 1/2 \cdot N \cdot w \cdot (D-d)\} h_s / 1,000$

where W_h : weight of head plate liners [kg]

π :circular constant

D : diameter in side liner [cm]

d : diameter of inside trunnion [cm]

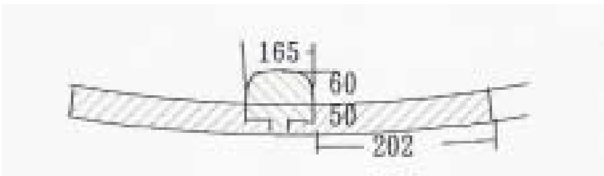
w : width of lifter bar [cm]

h : thickness of head plate [cm]

N : number of the head plate liners in a set of mill

Then $W_h = 2 \times \{3.14/4 \times (488^2 - 144^2) - 1/2 \times 24 \times (488-144)\} 5/1,000$

$= 5,953$ kg/set



4] Head liter bars

$W_{hl} = w(D-d) N \cdot h_l = 12.5(488-144) \times 24 \times 12 = 1,238$ kg/set

where w, D, d, N are dittou with 3].

h_l : height of the head lifter bar [cm]

5] Total weight of the plate liners and lifter bars

$W = W_s + W_{sl} + W_h + W_{hl}$

$= 2,832 + 4,963 + 5,953 + 1,238 = 14,986$ kg/set

6] Liner consumption per ton milled

Assuming 15,000 hours of operation as averaging life, apparent rubber liner consumption per ton milled will be estimated as the following.

14,986 kg/set×1,000g/kg×1/(15,000h×375t/h・set)＝2.6g/t

Since scrap ration is averaging 30～35%, actual liner consumption per ton milled will be about 2.0 g/ton.

10-6. Regrinding ball mill

10-6-1. Design concept

Regrinding ball mill design was done on the bases of the following conditions.

Grinding method：	wet closed circuit with cyclones
Feeder：	spout
Feed size:	130micron passing
Product size：	50 micron 80% passing
Work index：	11.0 kwh/st-grinding feed
New feed pulp density	75%wt
Pulp density in mill：	75% wt
Grinding capacity：	40 dry mt/h
Discharge method：	overflow through discharge trommel
Driving method：	side drive by helical gears with air clutch
Grinding media tonnage：	max.；41.7 mt averaging 36.5 mt
Revolutionary speed：	18.10 rpm, 74.0 % of critical speed
Supporting method：	both trunnions
Liners：	Skega K-type rubber liners
Motor:	open drip proof 3 phase induction motor
Lubricating method：	gears; oil spray
	trunnions；oil pump & oil cups
Trunnion cooling method：	reinforced water circulation
Circulating load ratio:	150%

10-6-2. Power requirement

1] Required power per ton of ore milled W_r

$W_r = W_i\,(10/\sqrt{P} - 10/\sqrt{F}) \cdot f_r \cdot f_d \times 1.1$ [kwh/mt]

where W_i : work index 11.0 [kwh/st]

F : feed size 130 micron 80% passing $\sqrt{F}$; 11.4

P : product size 50 micron 80% passing; $\sqrt{P}$; 7.1

K_1 : reduction factor, $K_1 = 1.11$

K_2 : fine grinding factor $K_2 = (P + 10.3)/1.145P = 1.05$

Then $W_r = W_r = 11.0 \times (10/7.1 - 10/11.4) \times 1.11 \times 1.05 \times 1.1 = 7.57$kwh/mt

2] Theoretical total power requirement for a set of mill W_t

Tonnage to be ground by the regrinding mill

Assuming 150% of copper content in the new feed ore is ground by the regrinding mill and this grade of the ore to be ground is 15%Cu, processing rate of the grinding mill [F] Is determined as follows.

$F = 750\ mt/h \times 0.59\%/100 \times 150\%/100 \times 1/15\%/100 = 44.25\ mt/h$

$W_t = W_r \cdot F$

$= 7.57\ kwh/mt \times 44.25\ mt/h = 335\ kw$

3] Calibration by conditioning factor

Since actual power is influenced by mechanical efficiency, W_t should be calibrated as the following.

$W_a = W_t \cdot f_m$

Where W_a :installed motor power [kw]

f_m : mechanical efficiency 0.96

$W_a = 335\ kw \times 1/0.96 \times 1.20 = 418\ kw/$ set round up to 420 kw.

4] Check calculations by power per ball ton KW_r

$KW_r = 3.1 \cdot D^{0.3} \times (3.2 - 3V_p) \cdot Cs \times (1 - 0.1/2^{9-10Cs})$

where D :mill diameter on inside liner (unit:ft) 10ft $D^{1/3} = 2.16$

V_p :ratio of grinding media to mill volume 38%/100

C_s :critical speed ratio 74.0%/100

$KW_r = 3.1 \times 3.68 \times (3.2 - 3 \times 0.38) \times 0.74 \times (1 - 0.1/3.03)$

$= 9.87$ kw/t- ball charged

Tonnage of ball charged

$W = \pi/4 \cdot D^2 \cdot L \cdot Vp \cdot \rho$

where π : the circular constant 3.14

D : mill diameter on inside liner (unit:ft) 10.0 ft

L :mill liner inside length (unit:ft) 11.0 ft

V_p : ratio of grinding media to mill volume 38%/100

ρ : rod bulk density (unit: mt/ft^3) 0.126 mt/ft^3

$W = 3.14/4 \times 10.0^2 \times 11.0 \times 0.38 \times 0.126 = 41.3\ mt$

Drawable power i.e. $W_d = KW_r \times W \times f_b$ should be bigger than the theoretical required power per one set of mill W_t. If not, power would be consumed in vain and no use for grinding.

$Wd = 9.87kwh\ /mt\text{-}ball \times 41.3mt = 407.6\ kw > Wt = 355kw$

Then, above result meets the requirement.

10-6-3. Selection of ball size

The empirical equation after F.C. Bond is not applicable for the regrinding mill, so we select 1-1/4 in bal based on ouw own experiences.

After T.E. Norman, ball consumption for copper ores can be given as averaging 0.178 lb/kwh.

Then 0.178 kwh/kwh × 335kwh × 450 g/lb ÷ 44.24 mt/h = 606 g/mt

On the other hand, generally speaking, the ball consumption is said to be 1.26 lb/st for non-ferrous ores. i.e.1.26 lb/st×450 g/lb ×1.1 mt/st = 623 g/mt. Any way the ball consumption of the regrinding mill can be estimated as 600～650 g/mt.

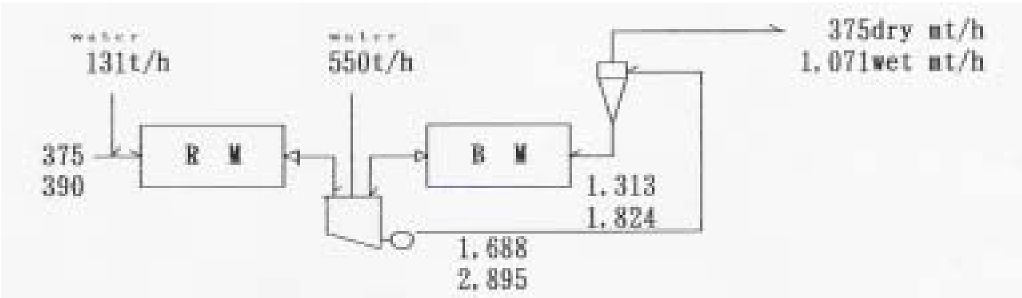
10-7. Primary grindinf cyclones

10-7-1.Material balance

The following assumptions were made.

Number of grinding circuit systems: 2

Tonnage of new feed: 375mt/h・system
Circulating load ratio: 350%
Pulp densities: underflow; 72%wt., 49% vol., Sp.Gr.1.829
overflow; 35%wt., 17% vol., Sp.Gr.1.283
feed; 58%wt., 34% vol., Sp.Gr.1.580



10-7-2. Selection of cyclone size

To minimize installation area, the maximum available type, namely KREBS 26Bmodelwas selected. Standard diameters are feed inlet 10″, overflow finder12″,and apex 4″1/2, respectively.

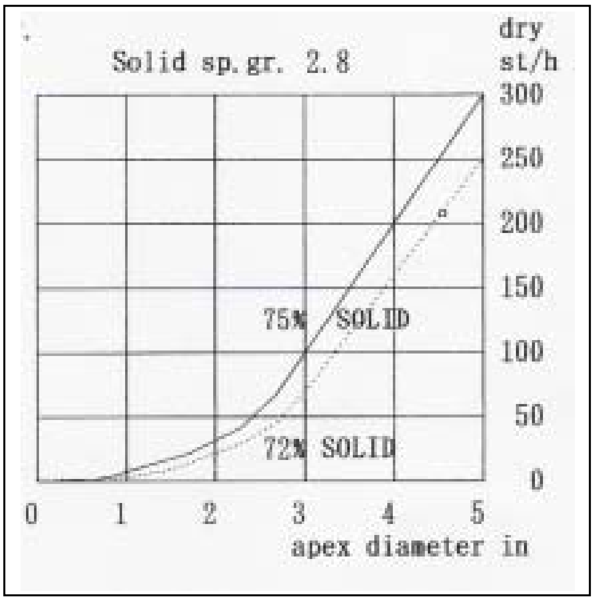
Capacity per one set Q and required numbers of cyclones N can be estimated after chart of the Krebs under the above said conditions as the following.

Q =210 st/h×1/1.1mt/st×2.7/2.8
=185mt/h・set
N=1,313mt/h÷185mt/h・set
=7.0 → 7 sets

Then, eight sets of 26B model including stand-by , will meet the requirement treating dry tonnage of 313mt/h.

Inlet pressure was selected to be 0.42 kg/cm^2 in order to minimize abrasion problems.

Cyclones should be installed radially around manifolds to keep uniform distribution of the feed pulp.



APEX CAPACITY CHART FOR 26B CYCLONE

10-7-3. Design of underflow launders

Slope of underflow launders can be calculated the following empirical equation after Caldecott.

G=(W+12)/W
P=100W/(W+1)
Where G: slope of launder

W: weight ratio of water to solid

P: weight percent of water in pulp

Since weight pulp density of the underflow is 72% , so P=100−72=28%

Then W=28/72=0.388、

Hence G=(0.388+12)/0.388=31.9%

Recommended slope by KREBS ENGINEERING is 4-3/4 in per 1 ft, namely 39%.

Inner surface of the launder should be LINED by rubber.

10-7-4. Estimated particle size distributions

Assuming circulating load ratio 350% and solid specific gravity 2.7, estimated particle size distributions of primary cyclone products are shown in the following.

Tyler mesh	Feed		Underflow		Overflow	
	Disr' tion	ΣDisr' tion	Disr' tion	ΣDisr' tion	Disr' tion	ΣDisr' tion
48	45.0%	45.0%	56.2%	56.2%	2.0%	2.0%
+ 65	12.8	57.8	14.5	70.7	6.0	8.0
+ 100	10.6	68.4	9.4	80.1	14.0	22.0
+ 150	7.1	75.5	5.2	85.3	13.0	35.0
+ 200	5.6	81.1	3.7	89.0	12.0	47.0
+ 325	2.1	83.2	1.3	90.3	5.0	52.0
− 325	16.8	100.0	9.7	100.0	48.0	100.0

10-8. Regrinding cyclones

10-8-1. Material balance

Feed of the regrinding cyclone will be concentrate of secondary roughers and tailing of secondary cleaner. The next conditions were assumed based on operational conditions of the regrinding mill and material balance in flotation circuits.

Feed characteristics	2RC	2CT	underflow	overflow	Total feed
Dry tonnage mt/h	15.9	28.7	66.9	44.6	111.5
Pulp density %Wt	25.0	18.0	75.0	19.8	34.8
Wet tonnage mt/h	63.6	159.4	89.2	225.3	267.6
Water contained mt/h	47.7	130.3	22.3	180.7	156.1
Water supplied mt/h			2.7		

Assuming specific gravity of the feed pulp as 1.46, flow rate of the feed Q_f will be

267.6 mt/h÷1.46mt/m³ = 183.3 m³/h

10-8-2. Required number of regrinding cyclones

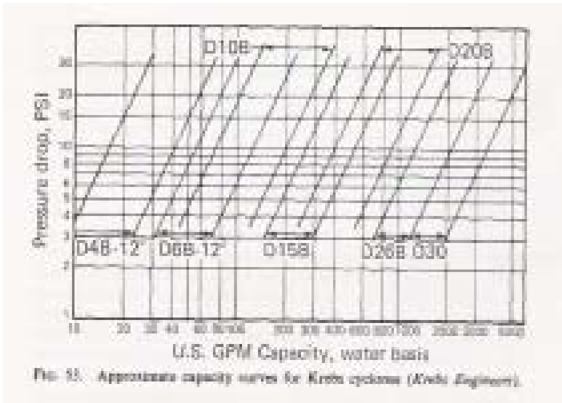
Changing the feed flow rate by ratio of 1 US gal= 3.8 ℓ and 1 h=60 min,

183.3 m³/h÷(60 min/h×3.8 ℓ/US gal×1000 ℓ/m³)=803.9 US gal/min

Assuming to adopt D15Bcyclones and their inlet pressure as 10 psi=0.7 kg/cm², capacity of one set is estimated to be 400 US gal/min/set after below chart of Krebs Engineers.

Then required number will be 803.9 US gal/min÷400 US gal/min≒2 sets.

Installed number should be 3 sets taking account of variation of feed grade and stand-by for repairing.



10-9. Primary cyclone feed pumps

10-9-1 Design concept

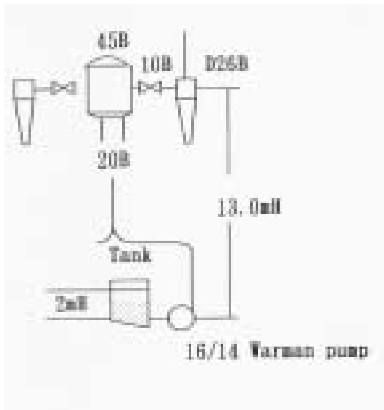
In order to minimize construction costs, we selected one pump for one circuit and to save maintenance costs one stage larger pump than its normal capacity. At the start-up operation we will use high chromium steel as lining material and rubber lining at later operation, to avoid troubles due to extraordinary hard materials intermingled in feed pulp such as bolts, nuts and tools and so on. Number of installation

Should be three, including common stand-by.

Flow rate: 30.53m^3 /min・set

Pulp density: 58%wt

Specific gravity of pulp: 1.58



Piping in this circuit is shown in right figure.

Inlet pressure:6psi=0.42 kg/cm^2

Actual head:(13−2)+4.2/1.58=13.66mH

Friction loss equivalent lengths

20B-14B reducer 1 set: 9m×1= 9.0m

45B-20B reducer 1set:10m×1=10.0m

20B elbows 4sets:13m×4=52.0m

10B valve 1set:1.2m×1= 1.2m

45B-20B sudden contraction 1set: 7.0m ×1 = 7.0m

10B right elbow 1set:10.0m×1=10.0m

Sub-total 89.2m

Total pipe length=actual pipe length+total friction loss equivalent lengths

=45m+89.2m=134.2m

Flow velocity=flow rate/sectional area of pipe

v = (30.6× 4)/(60×3.14× 0.4782)=2.845m/sec

specification of pipe

material: X-52

outer diameter: 508mm

thickness: 15mm

inner diameter: 478mm

10-9-2. Calculations of friction loss after Darcy's equation

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

Where hs :friction head loss [m]
L :pipe length [m]
D : pipe inner diameter [m]
V : averaging velocity [m/sec]
g :gravitational acceleration [m/sec^2]
f : friction loss coefficient

then

$$hs = 0.02 \times \frac{134.2 \times 2.845^2}{0.478 \times 2 \times 9.8} = 2.31\ m$$

Total head=actual head + head loss
=13.66m+2.31m=15.97m

This value of the total head should be calibrated by application of calibration factor in the case of slurry pumping.

Final total head=15.97m/0.815=19.60m

10-9-3 . Calculations of pump revolutionary speed and required power

1] pump revolutionary speed N_2

$$N_2 = N_1 \sqrt{\frac{H_2}{H_1}}$$

Where N_1 :standard speed at standard head [rpm]
H_1 : standard delivery head [m]
H_2 : required delivery head

In the case of Warman 16/14 pump, N_1 =335 rpm at H_1 =22.6mH

Then $N_2 = 335 \times \sqrt{19.60/22.60} = 312$ rpm

2] Required shaft power P_2

$P_2 = P_1 (N_2 / N_1)^3 = 237 \times (312/335)^3 = 191.4$kw

Where P_1 : standard shaft power at standard delivery head [kw]
N_1 : standard revolutionary speed [rpm]
N_2 : required revolutionary speed [rpm]

3] Installed power P

P=(required shaft power)/(pump efficiency×mechanical efficiency of V-belt×safety factor)
=191.4×1/0.72×1/0.95×1.2=335.7kw→round up to 350kw.

10-10. Regrinding cyclone feed pumps

10-10-1. Design concept

Same design concept will be applied with the primary grinding. Piping material, however, should be used stainless steel pipe STPG38 to keep anti-abrasion and anti-corrosion.

Flow rate: $3.826m^3$ /min・set
Pulp density: 34.8%Wt
Pulp specific gravity: 1.383
Cyclone inlet pressure: 10 psi = 0.7 kg/cm^2
Total pipe length = actual pipe length + total friction loss equivalent lengths
= 45m + 89.2m = 134.2m
Flow velocity $v = (3.826\times 4)/(60\times 3.14\times 0.191) = 2.845$ m/sec
Pipe specification: material: STPG38、inner diameter: 190.9mm

10-10-2. Calculations of friction loss and others

Calculations were made by same methods with 10-9 and their results are shown in the table SPECIFICATION OF SLURRY PUMPS including the results for other pumps.

10-11. Utilities

10-11-1. Grinding water

1] Water for rod mills

New feed tonnage: 375 mt/h·set ×2set = 750 mt/h
Moisture of new feed: 4%
Pulp density in mill: 70%Wt aver.、75%Wt max.
Pulp density cyclone feed: 58% Wt

Water quantity in new feed: Q_f
The following equation is given by definition of the pulp density and moisture.

$$Q_f/(375+Q_f)\times 100 = 4.0$$

Then $Q_f = (375\times 4.0)/(100-4.0) = 15.6$ mt/h・set

By the same way, water quantity of pulp in mill: Q_r is given by the following equation.

$$Q_r = 375\times(100-70)/70 = 160.7 \text{ t/h・set}$$

Then $Q = Q_r - Q_f = 160.7 - 15.6 = 145.1$ mt/h・set should be added at each rod Mill spout.

At each discharge box of rod mill, water content of pulp to be classified can be estimated as follows.

$$Q_d = 375\times(100-58)/58 = 271.5 \text{ mt/h・set}$$

Hence $Q_d - Q_r = 110.8$ mt/h・set will be further needed at each discharge box.

2] Water for primary ball mills

New feed tonnage: 375 mt/h·set ×2set = 750 mt/h
Circulating load ratio: 350%
Ball mill feed tonnage: 375 mt/h×350/100 = 1,313 mt/h
Pulp density of new feed: 74.0%
Pulp density in mill: 72% averaging

Pulp density cyclone feed : 58% Wt

Water quantity in new feed : Q_f

The following equation is given by definition of the pulp density and moisture.

Q_r =1,313 t/h×(100−74)/ 70 = 461.3 t/h・set

Similarly water quantity of pulp in mill : Q_m

Q_m =1,313 t/h×(100−72)/ 72 = 495.5 t/h・set

Then $Q = Q_d - Q_m$ =495.5−461.3=34.2mt/h・set should be added at each ball mill spout.

At each discharge box of ball mill, water content of pulp to be classified can be estimated as follows.

Q_d =1,313×(100−58)/ 58=1,222.3 mt/h・set

Hence $Q_d - Q_m$ =1,222.3−495.5=726.8mt/h・set will be further needed at each discharge box.

3] Water for regrinding ball mill

After table in section 10-8-4, 2.7 t/h of water should be added at spout of regrinding mill. Besides that water addition through pipe lines of 6B or 150A for discharge box and intake tank of cyclone O/F pump will be necessary for every start-up and shut down time of the regrinding cyclone feed and O/F pumps.

4] Cooling water for each trunnion

50 ℓ/min ·set is recommended to add at each mill in order to cool trunnion bearingsby the mill manufacturer. So, 50 ℓ/min ·set×5 sets =250 ℓ/min/5 sets=15 t/h of fresh water will be needed independently for cooling.

10-11-2. Compressed air

1] Air clutches

Assuming A equals air consumption at each time of ON-OFF,

$A = V_1 + V_2$

where V_1 : Volume of clutch element in [ℓ]

V_2 : Inner volume of piping from electromagnetic valve to the clutch in [ℓ]

F/R: Flow regulator, P/R: Pressure regulator

Assuming pipe size is 1 inch hereafter the electromagnetic valve, its volume V_2 is 6 litres.

Then each consumption of the air clutch is shown as follows.

Type of air clutch	V1 (ℓ)*	V2 (ℓ)*	A (ℓ)*
D42VC1200	27	6	33
D51VC1600	66	6	72
S42VC650	6.5	6	12.5

*The air volume is at gauge pressure of 7.0 kg/cm²

As capacity of the air compressor, it is requested to select the maximum consumption rate in the case of inching of the 16' ×23' ball mill.

From above table. the air consumption at free state of the D51VC1600 model in each time of inching, will be

$72ℓ \times (7+1.03)/1.03 = 561.3ℓ$

If inching of the mill is carried at the rate of 4 times per a minute and consumed air should be supplied in 2 minutes, the capacity of compressor will be

$0.5613\ m^3 \times 4 \times 1/2min \times 1/0.6 = 1.87\ m^3/min$ (Piston displacement)

where 0.6 is volumetric efficiency of the air compressor at sea level.

After manufacturer's catalogue, for example, HITACHI VHC WS-W type of 15 kw, its piston displacement 3.08 m³/min is selected, taking account of altitude of mill location to be installed.

Since lower limit of the air pressure for the clutches is 5.5kg/cm² (G), it is recommended to keep 6.0 kg/ cm² of minimum pressure at air tank of compressor side.

Then, $P_2\ (B+C) = P_1 \cdot C$

$C = V + 200N$

where P_1 :Pressure before start up of the mill in [kg/ cm²]

P_2 :Pressure after start up of the mill in [kg/ cm²]

B :Volume of clutch element in [ℓ]

C :Total volume of all air tanks in [ℓ]

V :Volume of compressor side air tank in [ℓ]

N :Number of clutch side air tanks

So $(6+1.03)(B+C) = (7+1.03)C$

In the case of inching for the 16' ×23' ball mill, air consumption becomes maximum, so that

$B = 72 \times 4 = 288ℓ,\ C = V + 200 \times 1$

whence $7.03 \times (288+V+200) = 8.03 \times (V+200)$

Then, $V = 488 \times 7.03 - 200 \times 8.03 = 1,824$ ・

In the case of start-up of one unit of grinding system of a 13' -1/2×16' rod mill and a 16' ×23' ball mill at same time, its air requirement will be $B = 72 + 33 = 103ℓ$

$c = V + 200 \times 2 = V + 400$

$7.03 \times (105+V+400) = 8.03 \times (V+400)$

$V = 5.03 \times 7.03 - 400 \times 8.03 = 338\ ℓ$

Consequently, as volume of compressor side air tank, effective volume of 2 m³ is recommended, assuming some surplus.

2] Selection of air compressor for oil spray system

Each oil spray unite requires air pressure of 4.5～5.5kg/cm²(G) and consumes air at rate of the following for each spray time.

2-6S2C: 720ℓ×2 of free air=1,420ℓ
2-7S2C: 960ℓ×2 of free air=1,920ℓ
1-3S2C: 185ℓ×1 of free air= 185ℓ
Total =3,525ℓ

Air tank should be installed independently at each mill side . Assuming inner pressure of air tank drops from 7 to 6kg/ cm^2(G) by each spray,

$$V_1 \times (7+1.03) - V_1 \times (6+1.03) = 1.03\ V_2$$

where V_1 :Tank volume in [ℓ]
V_2 :Air consumption of each spray time in [ℓ]
then V_1 is approximately same to V_2

Therefore, the following volumes of air tanks should be installed near side of each mill.

13' -1/2×16' rod mill : 750 ℓ
16' ×23' ball mill :1,000 ℓ
10' ×11' ball mill : 200 ℓ

Selecting spray interval as minimum 15 minutes (4 times per hour)and consumed air is supplied in 5 minutes by the compressor, its piston displacement will be

3,525 ℓ/(5min×0.6)=1,175 ℓ/min

In actual installation, it will be more economical to select same model with air compressor for air clutch.

3] Air requirement for air motor of rod charger

When an air motor is driven for rod charging, one unit of the grinding system is naturally stopped and another unit is operating normally. In this case, efficient delivery volume of the compressor is

$2.18 \times 0.6 = 1.31\ m^3$ /min

Since air consumption of the air motor is 1.1 m^3 /min, then supplying volume of spray air is 1.31 m^3 /min− 1.1 m^3 /min=0.21 m^3 /min.

After 3 operating units of the grinding mills have finished spray at a same time, the compressor requires y minutes of time to recover inner pressure from 6 to 7kg/cm^2, then y is given as follows.

$y=1.95\ m^3/0.21\ m^3/min=9.75$ min

where 1.95 m^3 is total volume of air tanks for the 3 units.

This time of y is shorter than time of interval of spray i.e. 15 min., so that it is available to supply compressed air to the air motor by the compressor for oil spray and independent compressor for the air motor is unnecessary.

It is recommendable to install a check valve at each inlet of the air tank in order to prevent from back blow .

End