Design Calculations of Mamut MIll

[1]Design criteria

Annual production: 5,670,000 tons (metric tons as a rule)

Operating days・Running time: 350 Days/year, 3 shifts/day, 8 hours/shift

Availability: Crushing; 60%
Concentrator; 90%

Capacities: Crushing ; 1,100 dry tons/hour
Concentrator; 750 dry tons/hour

Method of concentration: All slime flotation

Concentarate recovered: Copper concentrate

Characteristics of crude ore:

Grade of ore;	0.59%Cu
Specific gravity;	2.7
Apparent Sp. Gr.;	1.7
Moisture;	5.0%
Maximum size of lump;	700mm×900mm×1,400mm
Angle of repose;	40°
Angle of drawoff;	70°

Characteristics of concentrate:

Grade;	25% Cu
Specific gravity;	4.1
Apparent Sp. Gr.;	2.0
Moisture;	8.0%
Particle size;	−200 mesh 98%
Angle of repose;	50°

Grinding feed:

Particle size;	13mm 80% passing
Work index ;	10.8 kwh/dry short ton

Conditions of flotation:

	Pulp densities	Flotation times
Roughing;	35%Wt	10 min
Cleaning;	22%Wt	12 min
Recleaning;	20%Wt	19 min
Overall recovery;	90%	

[2] Design Calculations

1)Primary crusher

Capacity: $Q = \frac{5,670,000\ t/y}{350\ d/y \times 24\ h/d \times 0.6} = 1,125\ t/h$

As a primary crusher, it is suitable to select jaw crusher or gyratory crusher. In the case of the jaw crusher, however, its maximum capacity is limited to less to 1,000 mt/h, so we rejected it from object of selection. Referring to catalogue of manufacturers, the gyratory crusher of 42-65-type was selected, taking account of maximum feed size and capacity. Number of 42 means that feed openingis 42 in i.e. 1,070 mm and 65 shows that maximum daiameter is 65 in (1,650mm).

This value of capacity expresses tonnage of feed including fine ore, namely, there is no grizzly in upstream of the crusher.

1,150 t/h of the capacity is estimated in the case of 38mm of eccentric throw at 165mm(O.S.S. 6″1/2) of open side setting.

Particle size distribution of crushed products was estimated as follows.

Particle sizes		Distributions	Cumulative Distr' ns
+6-1/2in	(165mm)	10.0%	10.0%
5	(125mm)	15.0%	25.0
4	(100mm)	12.0	37.0
3	(75mm)	15.0	52.0
2	(50mm)	14.0	66.0
1	(25mm)	14.0	80.0
− 1		20.0	100.0

Maximum product size will be estimated to be 165mm× 2.2= 365 mm.

Power requirement per ton of feed is calculated as the following.

$$Kw/mt = \frac{Wi \times 11.06 \times (\sqrt{F}-\sqrt{P})}{\sqrt{F} \times \sqrt{P}}$$

Hence,

$$Kw/mt = \frac{10.8 \times 11.06\ (925\ 356)}{925 \times 356}$$
$$= 0.206\ kwh/mt$$

Total power requirement can be obtained by the following equation.

Kt=(Rating capacity of crusher mt/h) × Kw/mt

=1,125 mt/h×0.20 6kwh/mt=231.8kw

Recommended motor shuld be sized from 20% to 25% larger than the above-mentioned kw in order to overcome shock load.

Then, 231.8×1.25=290 kw..be rounded to 300 kw consequently.

2) Apron feeder

Required capacity: Min. 400 mt/h, Max. 1,600 mt/h

Apparent specific gravity of the crude ore:1.7 mt/ m^3

Volumes of the ore: 400 mt/h÷ 1.7 mt/ m^3 = 235 m^3/h Min.
1,600mt/h÷ 1.7 mt/ m^3= 941 m^3/h Max.
1,125mt/h÷ 1.7 mt/ m^3=662 m^3/h Averagingly

Sectional area of discharge opening: 1,800 mmW×1,600 mmH= 2.88 m^2

Voluminal efficiency: 0.9

Apron speed: V 235 m^3/h÷(60min/h×2.88 m^2×0.9)=1.5 m/min Min.
941 m^3/h÷(60min/h×2.88 m^2×0.9)=6.0 m/min Max.
662 m^3/h÷(60min/h×2.88 m^2×0.9)=4.3m/min Av

Design allowable tension: T 30,000 kg

Power requirement: K= VT/6,120 = 6m/min×30,000kg/6,120 = 29.4→30 kw

Model selected: Kobe Steel Ltd.; Ultra-heavy duty type 18-66AFH

Generally speaking, capacity of the Apron fedder is influenced by type of feed. shape and size of particles, moisture, clay content and so on. To meet such factors successfully, it is advisable to adop variable speed motor.

Maximum feed lump size is estimated to be 165 mm×230 mm×330mm approximately, based on the product size distribution of the primary crusher.

3) Primary screen

It is desirable to use double decked screes, using upper deck as gaurd screen for lower deck. Opening size of each deck aperture will be 70 mm for the upper deck and 28 mm for the lower deck, respectively.

Material balance is estimated as follows.

Upper screen undersize=1,125mt/h×45%×90%=456 mt/h
(Assuming screening efficiency as 90%.)

Lower screen undersize=456 mt/h×20/45%×80%=162 mt/h

In the case of lower deck, efficiencytends to be lower than upper deck, based on smaller feed size etc.

Total screen oversizes =1,125 mt/h−162 mt/h= 963mt/h

Required screen area is given by Gluck' s equation.

$$A = \frac{Q}{B \cdot I \cdot D \cdot So \cdot Sh \cdot F \cdot O \cdot W \cdot Y \cdot M \cdot Z}$$

Where A: Required screen area in (sq ft)
Q: Capacity in (st/h)
B:Basic capacity based on opening size 0.757 (st/h/ft^2)
I:Factor due to slope of the screen 0.95 at 20 degrees
D:Deck factor; Top deck 1.00, second deck 0.90, bottom deck 0.80
S_o : Oversize factor; Factor due to bigger particles than opening. 1.0
S_h : Halfsize factor; % of material in the feed less than one half the size of opening; 20%; 0.7, 40% 1.0, 70%; 1.8
F :Factor due to shape of the opening Square;1.0, circle; 0.8, rectangular:L/S=2～３ ;1.6, L/S=3～6;1.4, L/S>6; 1.1

O :Open area facto;=1−0,02(50−opening%)

W :Factor depending on bulk density=1.7/1.6=1.06

Y :Shape factor due to shape of particles which varies by percentage of particles in total feed where major axis/minor axis ratio is bigger than 3, besides minor axis ranges from 1/2 to 2/3.
(5%; 1.00, 10%; 0.95, 20%; 0.85)

M: Wet screening factor
Dry;1.0、
Wet (In the case where water is sprayed 25〜50ℓ per m^3/h of feed)

Opening	25.4mm	19.1	12.7	9.5	7.9	4.8	3.2	1.6	0.8
M	2.9	2.71	2.5	2.25	2.1	1.9	1.75	1.5	1.25

Z: Moisture and cohesion factor
Wet muddy or sticky gravel, gypsum, apatite etc. ; 0.75
Ore with wet surfaces, Moisture>6% and not water absorpivet materials ; 0.85
Dry limp materials, Moistur<4%;1.0

Assuming I=0.95, F=1, W=1.06, Y=1,M=1,Z=0.75, abovesaid equation can be expressed as the following.

$$A = \frac{Q}{0.757 \times B \cdot I \cdot D \cdot So \cdot Sh \cdot O \cdot}$$

Then

$$\text{Upper deck}\quad A = \frac{1{,}125 \times 1.102}{0.757 \times 9.0 \times 1 \times 1.19 \times 0.82 \times 1.28} = 145\ ft^2$$

$$\text{Lower deck}\quad A = \frac{456 \times 1.102}{0.757 \times 5.9 \times 0.9 \times 1.\ 5 \times 0.90 \times 1.16} = 104\ ft^2$$

8'×20'Ripl-FloScreen(160 ft^2) has capacity to meet the abovementioned requirement.

Check after Allis Chalmars' s method.

Ta= Tb×V×H×K×W

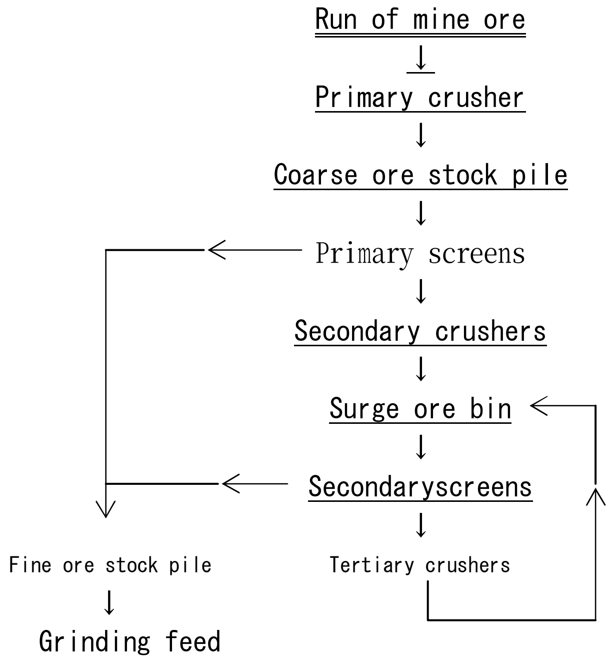
where Ta: Actual capacity (st/ft^2 /h)
Tb: Basic capacity (In the case of opening 2″3/4; 8.8st/ft^2/h)
V: Oversize factorof upper and lowr decks;1.42
H: Halfsize factor, upperdeck; 0.8,lowerdeck; 0.70
K: Condition factor; dry uncrushed materials with 6% moisture 1.25
W: Weight factor; Bulk density 105 lb/ft^3 → 1.05

Then, Ta=8.8×1.42×0.8×1.25×1.05÷1.102=11.91 mt/ft^2・h

Hence required screen area A
A=1,125mt/h/11.91 mt/ft^2・h=94ft^2<160 ft^2

This will meet the above said requirement.

4) Flow sheet of the secondary crushing



7 ft Symons Standaerd type cone crushers were recommended as the secondary crushers. This model has 750 st/h ofcapacity in standard conditions. The bulk density in the standard conditions is 1.6.

Changing into metric tons,

Capacity/Unit＝750 st/h×0.907mt/st×1.7/1.6＝722 mt/h·Unit

Then, required unit number is obtained as the following.

Required units of 2ry crushers＝963mt/h÷722 mt/h·Unit
＝1.33→ 2 Units

After Taggart:Hand book of Mineral Dressing 4-54.,particle size distribution of the crushed product is estimated as the following.

Particle size distribution of the crushed product

Size (mm)	Oversize%	Cumulative oversize%
＋ 35	28	28
25	16	44
17.5	15	59
12.5	12	71
－ 12.5	29	100

The maximum size of the crushed product is estimated to be, 32 mm×2.2＝70 mm from open side setting.

Required power of the standard cone crushers

Required power per ton of ore

$$\text{Kw/mt} = \frac{\text{Wi} \times 11.06 \times (\sqrt{F}-\sqrt{P})}{\sqrt{F} \times \sqrt{P}}$$

Where Wi: Work index 10.8 kwh/st
F: Feed size 80% passing 140,000 microns $\sqrt{F}=374$
P: Product size 80% passing 40,000microns $\sqrt{P}=200$

Hence,

$$\text{Kw/mt} = \frac{10.8 \times 11.06 \times (374 - 200)}{374 \times 200}$$
$$= 0.278 \text{ kwh/mt}$$

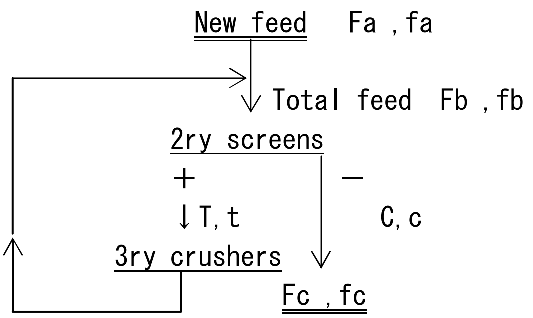
Total power requirement is obtained by multiplication of standard capacityto this.

Kw = 0.278 kwh/mt×722 mt/h=200.7 kw。

Then recommended motor power perunit should be

200.7 kw/0.9=222kw → 225 kw.

5) Material balance sheet of 2ry screens and 3ry crushers



After Taggart Handbook 19-201, the following equations are given.

$K = F_b / C = (c - t)/(f_b - t)$

$C = F_a$

$F_b = F_a (c - t)/(f_b - t)$

where
F_a : Tonnage of new feed=Feed of 2ry crushers [mt/h]
F_b : Total feed of 2ry screens [mt/h]
T : Oversize of 2ry screens [mt/h]
C : Undersize of 2ry screens [mt/h]
F_c : Tonnage of 3ry cusher product [mt/h]
f_a : Percentage of smaller than a certain siz in new feed [%]
f_b : Percentage of smaller than a certain size in total feed [%]
t : Percentage of smaller than a certain size in oversize [%]
c : Percentage of smaller than a certain size in undersize [%]

Content of minus 18 mm size in each material is estimated after given catalogue and Taggart Handbook, as the following.

$Fa = 963$ mt/h $= C$
$fb = 53\%$
$c = 85\%$
$t = 11\%$

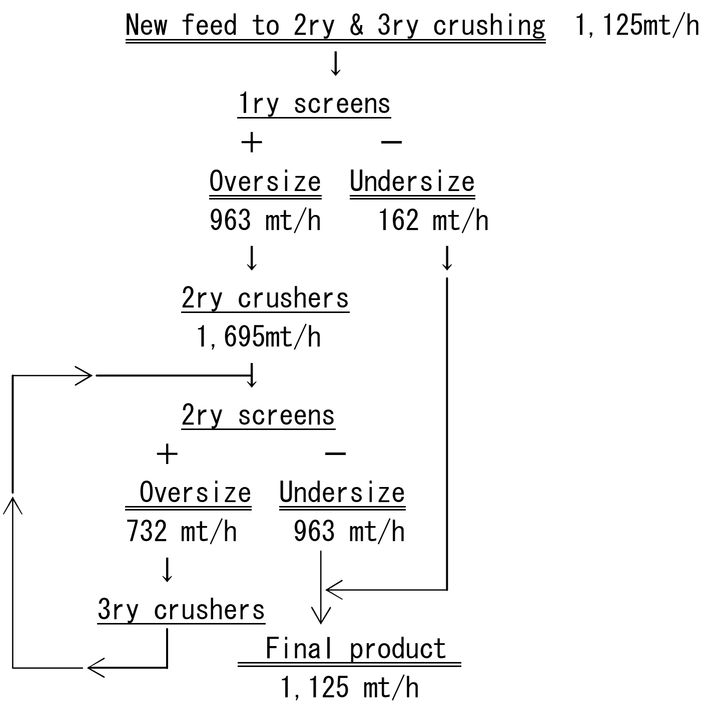
Then

$$K = \frac{c - t}{fb - t} = \frac{85 - 11}{53 - 11} = 1.76$$

$F_b = F_a \times K = 963\text{mt/h} \times 1.76 = 1{,}695$ mt/h
$T = F_b - C = 1{,}695$ mt/h $- 963$ mt/h $= 732$ mt/h

Hence the material balance is obtained as the follows.

Material balance sheet of 2ry screens and 3ry crushers



Required area of the 2ry screens is calculated by same mthod with 1ry screens.

$$A = \frac{1{,}695 \times 1{,}102}{0.757 \times 5.0 \times 1 \times 1.05 \times 0.95 \times 1.16} = 426 \text{ ft}^2$$

Hence unit number of the 2ry screens of 8ft by 20 ft will be estimated to be

$N = 426 \div (8 \times 20) = 2.6 \rightarrow 3$ units

As 3ry crushers, 7' Symons short head cone crushers are recommended assuming closedset setting (C.S.S) 13 mm.

Recommended capacity of the 7' short head cone crushers is estimated after manufacturer's catalogue.

Q＝300 st/h×0.907 mt/st×1.7/1.6＝289 mt/h

Then unit number of the 3ry crushers will be

N＝732mt/h÷289mt/h・台＝2.5→ 3 units.

Required power per ton of ore

$$Kw/mt = \frac{Wi \times 11.06 \times (\sqrt{F}-\sqrt{P})}{\sqrt{F} \times \sqrt{P}}$$

Where Wi: Work index 10.8 kwh/st

F: Feed size 80% passing 55,000 microns √F＝235

P: Product size 80% passing 26,000microns √P＝161

Hence,

$$Kw/mt = \frac{10.8 \times 11.06 \times (235 - 161)}{235 \times 161}$$

$$= 0.198 \ kwh/mt$$

Total power requirement is obtained by multiplication of standard capacityto this.

Kw = 0.198 kwh/mt×450 mt/h×1.25＝ 112 kw。

This vakue is less than actual installed power of 225 kw and will meet the above said requirement. It is recommendable to install same motor with 2ry crushers in order to minimize spare motors.

6) Vibrating feeders

Taking account of easy controllability, noise and costs, it is advantageous to select vibrating feeders, i.e. YASUKAWA Uras feeders. To drawoff the ore out of the coarse ore stock pile, running of 2 units of vibrating feeders will be expected normally.So averaging throughput per unit will be 1,125 mt/h/2＝563 mt/h. Selection of the feeder, however, it is necessary to take account of temporary increased production. After catalogue data of the manufacturer, model of 1,400 mmW× 1,500 mmL (capacity:1,000 mt/h·unit) will be suitable for this mission.

In order to increase live capacity of the coarse ore stock pil, we decided to install 4 units offeeders. (2 units operating, 2 units stand-by).Theoritical throughput out of surge bin will be estimated as 1,695 mt/h/3 units＝565 mt/h・unit. It is also advisable to select the same model for this service with coarse ore stockpile drawoff, based on above-said reasons.

Installed motor will be 3.7 kw×2/unit after manufacturer’s catalogue.

Each trough of the vibrating feeders should have 12 degrees of down slope for down stream. For these services, both 2 units of variable controllers will be installed to meet such conditions as valiation of moisture and clay content in the ore and ore sizes etc. Speed control will be able to realize by electrical remote controllers in central control room automatically or manually.

7) Coarse ore stock pile

The following assumptions were made.

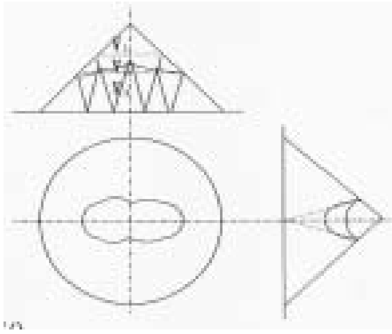
Live inventory: Ore volume should should be for 6 hours of crushing operation.be for 6 hours of crushing operation.

Total tonnage including dead should be for 24 hours of crushing operation.

Angle of repose: 40°

Angle of drawoff: 70°

Draw opening : 4



Strictly speaking, calculations of movable inventory are very complicated, but in our case it is unnecessary to require such accuracy as small ore bin.

To simplify calculation of the movable tonnage, its volume was divided into three parts.

V_1 : Perfect conical part at top of the pile.

V_2 : Circular truncated parts at intermediate of pile.

V_3 : Four inverted cones in the bottom

$V_1 = \pi/3 \cdot r^2 h = 3.14 \times 82 \times 6.4 \div 3 = 429 m^3$

$V_2 = 1/3 \cdot h(B + b + Bb)$

Where h: Height=7.5m

b: Bottome base area= $3.14 \times 52 \times 4 = 314 m^2$

B: Top base area = $420 m^2$ Obtained graphic calculation

Then $V_2 = 1/3 \cdot 7.5 \times (314 + 420 + 314 \times 420) = 2,742 m^3$

$V_3 = \pi/3 \cdot r^2 h \times 4 = 3.14 \times 52 \times 11 \times 4 \div 3 = 1,151 m^3$

$V = V_1 + V_2 + V_3 = 429 m^3 + 2,742 m^3 + 1,151 m^3 = 4,322 m^3$

Hence, the movable tonnage of ore inventory T should be

$T = 4,322 m^3 \times 1.7 t/m^3 = 7,347$ mt.

In the case where lack of the movable inventory occurred by reasons of shortage of the run of mine, unscheduled shutdown of the primary crushing plant and so on, a part of dead stock shall be raked. A bulldozer of D6 class will be used for this purpose.

Roofing of the coarse ore stock pile shall not be installed. The reason is we expect that rain water will not flow into drawoff opening in the bottom and occur severe trouble, because ore sizes are coarse and very permeable.

No,3 belt conveyor and floor of drawoff tunnel shall be sloped down with about 1/50 of gentle slope.

In the case of extraordinal heavy rains, feeding to the No,3 belt conveyor shall be stopped in order to prevent the ore in crater from sliding down. We expect that probability will be two or three times in every rainiy season.

8) Beltconveyors in the crushing plant

8-1. Design of belt conveyors

Because of ore charactristics, the maximum slope of belt conveyr will be limited to below 16° on the primary crushing product and below 18° on the 2ry and 3ry cushing circuits respectively.

Taking into consideration that supply of special type belt will be defficult in local market we decided to use standard plain belts as a rule

8-2 Minimum belt widths

Minimum belt widths should be determined based on maximum size of lump ore.

Relationship between maximum lump ore size and belt widths in the following table.

Table 1. Relationship between maximum lump ore size and belt widths

Min. belt widths (mm)	400	500	600	750	900	1,050	1,200
Max. lump size (mm)	100	150	200	250	300	400	500

The width of No.1 belt conveyor(No.1BC) should be naturally wider than that of the Apron feeder in order to function as ore spill catcher.

Other conveyor widths were determined by tonnages of ore to be conveyed.

Standard belt speeds were selected based on the next table.

Table 2. Standard belt speeds for ore:v

belt widths (mm)	400	600	900	1,500	≧ 1,600
belt speeds (m/min)	75	90	120	150	180

8-3. Transporting capacity of belt conveyors Qm

$Q_m = Q_t / \gamma = 60 \cdot k_1 \cdot K_2 \cdot (0.9b - 0.05)^2 \cdot v$

where Q_m : Belt conveyor capacity [m³/h]

Q_t : Tonnage to be carried [mt/h]

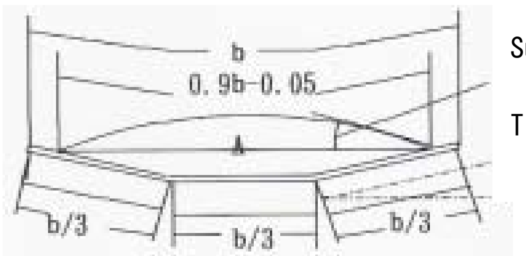
γ : Bulk density [t/m³]

k_1 : Factor based on conveyor slope

K_2 : Factor based on trough angle and surcharge angle

v : belt speed [m/min]

b : Belt width [m]



Surcharge angle

Trough angle

Table 3 : .k_1: Factor based on conveyor slope

Slope angle (°)	2	4	6	8	10	12	14	16	18	20
K_1	1.00	0.09	0.98	0.97	0.95	0.93	0.91	0.89	0.85	0.81
Slope angle (°)	21	22	23	24	25	26	27	28	29	30
K_1	0.78	0.76	0.73	0.71	0.68	0.66	0.64	0.61	0.59	0.56

Table 4： k_2 Factor based on trough angle and surcharge angle

α \ β	10°	20°	30°
0°	0.0292	0.0591	0.0906
20°	0.0963	0.1245	0.1538
30°	0.1248	0.1488	0.1757
40°	0.1406	0.1628	0.1862
45°	0.1485	0.1698	0.1915

Table 5:Relationship between belt width and trough angle

Belt width	Standard trough angle	Max. trough angle
400 mm	20°	20°
400～500	20°	30°
600～750	30°	45°
900～	30°	60°

8-3-1. Minimum belt speed of No.1 BC

$Q_m = Q_t / \gamma = 60 \cdot k_1 \cdot K_2 \cdot (0.9b - 0.05)^2 \cdot v$

Q_m :1,340 mt/h max., γ:1.7, k_1:1.0, K_2:0.1245, b:1.80m

$v = 1{,}340 \div \{60 \times 1.7 \times 1.0 \times 0.1245 \times (0.9 \times 1.8 - 0.05)^2\}$

$= 42.8$m/min

Taking account of future capacity expansion and compensation of shut down, belt speed of the No.1 BCwas determined to be 70 m/min.

8-3-2. Minimum belt speed of No.2 BC

$Qm = Qt / \gamma = 60 \cdot k_1 \cdot K2 \cdot (0.9b - 0.05)^2 \cdot v$

Qm:1,340 mt/h max., γ:1.7, k1:0.91 for 14° 30',K2:0.1488, b:1.05m

$v = 1{,}340 \div \{60 \times 1.7 \times 0.91 \times 0.1488 \times (0.9 \times 1.05 - 0.05)^2\} = 121.1$m/min

Taking account of 10% of surplus, belt speed was determined to be 121.1m/min×1.1＝133.2→135 m/min consequently.

8-3-3. Minimum belt speed of No.3 BC

$Qm = Qt / \gamma = 60 \cdot k_1 \cdot K_2 \cdot (0.9b - 0.05)^2 \cdot v$

Qm:1,500 mt/h max., γ:1.7,k1:0.93for 12° ,K2:0.1488, b:1.05m

$v = 1{,}500 \div \{60 \times 1.7 \times 0.93 \times 0.1488 \times (0.9 \times 1.05 - 0.05)^2\}$

$= 132.7$m/min

Taking account of 15% of surplus, belt speed was determined to be 132.7m/min×1.15＝152.6→155 m/min consequently.

8-3-4. Minimum belt speed of No.4 & 5BC

$Qm = Qt / \gamma = 60 \cdot k_1 \cdot K_2 \cdot (0.9b - 0.05)^2 \cdot v$

Qm :750 mt/h max., γ:1.7,k1: 0.93 for 13° 16' K2:0.1488, b:0.90m

$v = 750 \div \{60 \times 1.7 \times 0.93 \times 0.1488 \times (0.9 \times 0.90 - 0.05)^2\}$

$= 60.9$m/min

Taking account of 180% of surplus due to compensation for shutdown of one side system, belt speed was determined to be

60.9m/min×1.80＝109.6→110m/min consequently.

8-3-5. Minimum belt speed of No.6 BC

$Qm = Qt / \gamma = 60 \cdot k_1 \cdot K_2 \cdot (0.9b - 0.05)^2 \cdot v$

Qm:1,870 mt/h max., γ:1.7,k1:0.95 for 10° , K2:0.1488, b:1.20m

$v = 1,870 \div \{60 \times 1.7 \times 0.95 \times 0.1488 \times (0.9 \times 1.20 - 0.05)^2\}$

$= 122.2$m/min

Taking account of 15% of surplus, belt speed was determined to be

122.2m/min×1.15＝140.5→1501m/min consequently.

8-3-6. Minimum belt speed of No.7 BC

$Qm = Qt / \gamma = 60 \cdot k_1 \cdot K_2 \cdot (0.9b - 0.05)^2 \cdot v$

Qm :1,870 mt/h max., γ:1.7, k1:0.90for 15° , K2:0.1488, b:1.20m

$v = 1,870 \div \{60 \times 1.7 \times 0.90 \times 0.1488 \times (0.9 \times 1.20 - 0.05)^2\}$

$= 129.0$m/min

Taking account of 15% of surplus, belt speed was determined to be

129.0m/min×1.15＝148.4→1501m/min consequently.

8-3-7. Minimum belt speed of No.8,9 & 10BC

$Qm = Qt / \gamma = 60 \cdot k_1 \cdot K_2 \cdot (0.9b - 0.05)^2 \cdot v$

Qm :625 mt/h max., γ:1.7,k1:0.91for 13° 53'K2: 0.1488, b:0.90m

$v = 625 \div \{60 \times 1.7 \times 0.91 \times 0.1488 \times (0.9 \times 0.90 - 0.05)^2\}$

$= 78.3$m/min

Taking account of 40% of surplus due to compensation for shutdown of one system, belt speed was determined to be

78.3m/min×1.40＝109.6→110m/min consequently.

8-3-8. Minimum belt speed of No.11 BC

$Qm = Qt / \gamma = 60 \cdot k_1 \cdot K_2 \cdot (0.9b - 0.05)^2 \cdot v$

Qm :250 mt/h max., γ:1.7,k1:1.00for 0° ,K2: 0.1488, b:1.20m

$v = 250 \div \{60 \times 1.7 \times 1.00 \times 0.1488 \times (0.9 \times 1.20 - 0.05)^2\}$

$= 15.5$m/min

Taking account of shock load when ore bridges slide down in the surge bin, belt speed was determined to be 135 m/min consequently.

8-3-9　Minimum belt speed of No.12 BC

$Qm = Qt / \gamma = 60 \cdot k_1 \cdot K_2 \cdot (0.9b - 0.05)^2 \cdot v$

Qm　:1,500 mt/h max., γ:1.7, k1:0.97 for 6° 57’, K2:0.1488, b:1.20m

$v = 1{,}500 \div \{60 \times 1.7 \times 0.97 \times 0.1488 \times (0.9 \times 1.20 - 0.05)^2\}$

$= 96.0$m/min

Taking account of 40% of surplus belt speed was determined to be
96.0m/min×1.40＝134.4→1351m/min consequently.

8-3-10 Minimum belt speed of No.13 BC

$Qm = Qt / \gamma = 60 \cdot k_1 \cdot K_2 \cdot (0.9b - 0.05)^2 \cdot v$

Qm　:1,500 mt/h max., γ:1.7, k1:0.90for 14° 39’, K2:0.1488, b:1.05m

$v = 1{,}500 \div \{60 \times 1.7 \times 0.90 \times 0.1488 \times (0.9 \times 1.05 - 0.05)^2\}$

$= 137.0$m/min

Taking account of 10% of surplus belt speed was determined to be
137.0m/min×1.10＝150.7→1551m/min consequently.

8-3-11 Minimum belt speed of No.14 & 15BC

$Qm = Qt / \gamma = 60 \cdot k_1 \cdot K_2 \cdot (0.9b - 0.05)2 \cdot v$

Qm　:1,500 mt/h max., γ:1.7,k1: 0.88for 16° 20'K2: 0.1488, b:1.20m

$v = 1{,}500 \div \{60 \times 1.7 \times 0.88 \times 0.1488 \times (0.9 \times 1.20 - 0.05)^2\}$

$= 105.9$m/min

Taking account of 20% of surplus belt speed was determined to be
105.9m/min×1.20 ＝127.1→135m/min consequently.

8-4.所要動力の計算

Required power P:　$P = P_1 + P_2 \pm P_3 + P_t$　[kw]

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]

Power by tripper P_t :　$P_t = 0$　[kw]

where l : horizontal length of conveyor (distance between axis and axis of pulleys)　[m]

l_0 : calibrated horizontal conveyor length cf. Table 6　[m]

h : lift　[m]

Q_t : tonnage to be carried　[mt/h]

v : belt speed　[m/min]

W : weight of revolving parts without load　[kg/m]

W_1 : weight of belt　[kg/m]

μ : friction coefficient between belt and pulley

θ : wrap angle of belt　[rad]

l_c : carrier roller spacing　[m]

l_r : return roller spacing　[m]

f : revolutionary friction coefficient of roller

Table 6. Revolutionary friction coefficient of roller f & calibrated horizontal conveyor length l_0

Structure of roller	f	l_0
Common roller	0.03	49
Special low resistant, precision finished roller	0.022	66
Down sloped conveyor	0.012	156

Table.7 Belt width, belt weight W, weight of revolving parts W1 & tripper power Pt

Belt width mm	in	W kg/m	W_1 kg/m	P_t kw
400	16	22.4	4.5	1.5
450	18	28	7.0	
500	20	30	7.5	
600	24	35.5	9.0	
750	30	53	13.0	2.65
900	36	63	15.5	
1,050	42	80	23.0	3.55
1,200	48	90	26.0	
1,400	56	112	33.0	5.0
1,600	64	125	38.0	
1,800	72	150	46.0	6.0
2,000	80	160	51.0	

Table 8. Friction coefficient between belt and pulley μ

pulley	Operating conditions	μ
Naked steel	Wet by muddy water Moistened dry	0.1 0.1～0.2 0.25
Corrugated rubber lagging	Wet by muddy water Moistened dry	0.2 0.2～0.3 0.35

a) Calculations of belt tensions

Effective tension: $F_p = 6{,}120P/v$ [kg]

Tension on slack side : $F_2 = F_p / e^{\mu\theta-1}$ [kg]

Slope tension : $F_3 = W1\ (h \pm f \cdot l)$ down slope ＋ [kg]

Minimum tension : Adopt bigger value from the following

Carrier side ; $F_4 = 50/8 \cdot l_c\ (Qt/0.06v + W_1)$

Return side ; $F_4 = 50/8 \cdot l_r \cdot W_1$

b)Maximum tension： Adopt the maximum value in the following.

$F_{max} = F_p + F_2$ or $F_p + F_4$ (horizontal)

$F_{max} = F_p + F_2$ or $F_p + F_3 + F_4$ (sloped)

c)Safety factor S:

Canvas belt　$S = \sigma_c \times b \times n / F_{max}$

Steel code belt　$S = \sigma_s \times n / F_{max}$

Where　σ_c : strength per one ply　[kg/cm・ply]

σ_s : strength per one code　[kg/cord]

8-4-1.Required power of No.1 BC

In this case, B=1,800mm, Q_t =1,340mt/h, h=0m, l=11.7m, v=70m/min, rubber lagged pulley, θ:200° =3.49rad, μ=0.3, W=150kg/m, f=0.03

Then　$P_1 = 0.06 \times 0.03 \times 150 \times 70 \times (11.7+49)/367 = 3.13$kw

$P_2 = 0.03 \times 1,340 \times (11.7+49)/367 = 6.65$kw

$P_3 = +(0 \times 1,340)/367 = 0$

$P_t = 0$

Hence　$P = 3.13 + 6.65 = 9.78$kw

Recommended motor power P_m is 9.78/0.8=12.23→15kw.

8-4-2. Required power of No.2 BC

In this case, B=1,050mm, Q_t =1,340mt/h, h=51.5m, l=197.5m, v=135m/min, rubber lagged pulley, θ:200° =3.49rad, μ=0.3, W=150kg/m, f=0.03

Then　$P_1 = 0.06 \times 0.03 \times 80 \times 135 \times (197.5+49)/367 = 13.06$kw

$P_2 = 0.03 \times 1,340 \times (197.5+49)/367 = 27.00$kw

$P_3 = +(51.5 \times 1,340)/367 = 188.04$kw

$P_t = 0$

Hence　$P = 13.06 + 27.00 + 188.04 = 228.1$kw

Recommended motor power P_m is 228.1/0.8=285.12→150kw×2=300kw

8-4-3. Required power of No.3 BC

In this case, B=1,050mm, Q_t =1,500mt/h, h=5.5m, l=68.5m, v=155m/min, rubber lagged pulley, θ:200° =3.49rad, μ=0.3, W=150kg/m, f=0.03

Then　$P_1 = 0.06 \times 0.03 \times 80 \times 155 \times (68.5+49)/367 = 7.15$kw

$P_2 = 0.03 \times 1,500 \times (68.5+49)/367 = 14.41$kw

$P_3 = +(5.5 \times 1,500)/367 = 22.48$kw

$P_t = 0$

Hence　$P = 7.15 + 14.41 + 22.48 = 44.04$kw

Recommended motor power P_m is 44.04/0.8=55.05→55kw

8-4-4. Required power of No.4 & 5BC

In this case, B= 900mm, Q_t = 750mt/h, h=10.8m, l= 45.8m, v=110m/min, rubber lagged pulley, θ:200° =3.49rad, μ=0.3, W=150kg/m, f=0.03

Then　$P_1 = 0.06 \times 0.03 \times 65 \times 110 \times (45.8+49)/367 = 3.32$kw

$P_2 = 0.03 \times 750 \times (45.8+49)/367 = 5.81$kw

$P_3 = +(10.8 \times 750)/367 = 22.07$kw

P_t =0
Hence　P =3.32＋5.81＋22.07＝31.2kw
Recommended motor power P_m is 31.2/0.8＝39→45kw

8-4-5. Required power of No.6 BC

In this case,B=1,200mm, Q_t ＝1,870mt/h,h＝3.2m,l＝69.2m,v＝150m/min, rubber lagged pulley,θ:200° =3.49rad,μ=0.3, W=150kg/m, f=0.03

Then　P_1 =0.06×0.03×90×150×(69.2＋49)/367＝7.83kw
P_2 =0.03×1,870×(69.2＋49)/367＝18.07kw
P_3 =＋(3.2×1,870)/367＝16.31kw
Pt =0
Hence　P =7.83＋18.07＋16.31＝42.21kw
Recommended motor power P_m is 42.21/0.8＝52.76→55kw

8-4-6. Required power of No.7 BC

In this case, B=1,200mm, Q_t =1,870mt/h,h=19.4m,l=68.31m,v=150m/min, rubber lagged pulley,θ:200° =3.49rad,μ=0.3,W=150kg/m,f=0.03

Then　P_1 =0.06×0.03×90×150×(68.31＋49)/367＝7.77kw
P_2 =0.03×1,870×(68.31＋49)/367＝17.93kw
P_3 =＋(19.4×1,870)/367＝98.85kw
P_t =0
Hence　P =7.77＋17.73＋98.85＝124.55
Recommended motor power　Pm is 125.55/0.8＝155.69→150kw

8-4-7. Required power of No.8,9 & 10 BC

In this case, B= 900mm, Q_t = 625mt/h,h=10.4m,l=44.68m,v=105m/min, rubber lagged pulley,θ:200° =3.49rad,μ=0.3,W=150kg/m,f=0.03

Then　P_1 =0.06×0.03×65×105×(44.68＋49)/367＝ 3.14kw
P_2 =0.03× 625×(44.68＋49)/367＝4.79kw
P_3 =＋(10.4× 625)/367＝17.71kw
Pt =0
Hence　P =3.14＋4.79＋17.71＝25.64kw
Recommended motor power P_m is 25.64/0.8＝32.05→ 37kw

8-4-8. Required power of No.11 BC

In this case, B =1,200mm, Q_t =1,500mt/h, h=0.0m,l=14.0m,v=135m/min, rubber lagged pulley,θ:200° =3.49rad,μ=0.3,W=150kg/m,f=0.03

Then　P_1 =0.06×0.03×90×135×(14.0＋49)/367＝3.75kw
P_2 =0.03×1,500×(14.0＋49)/367＝ 7.72kw
P_3 =＋(0.0×1,500)/367＝ 0.00kw
P_t =0
Hence　P =3.75＋ 7.72＝11.47kw
Recommended motor power P_m is 11.47/0.8＝14.34→15kw

8-4-9. Required power of No.12 BC

In this case, B =1,200mm, Q_t =1,500mt/h, h= 4.6m, l= 46.7m, v=135m/min, rubber lagged pulley, θ:200° =3.49rad, μ=0.3, W=150kg/m, f=0.03

Then　P_1 =0.06×0.03×90×135×(46.7+49)/367=5.70kw

　　　P_2 =0.03×1,500×(46.7+49)/367=11.73kw

　　　P_3 =+(4.6×1,500)/367=18.80kw

　　　P_t =0

Hence　P =3.70+11.73+18.80=36.23kw

Recommended motor power Pm is 36.23/0.8=45.28 →45kw

8-4-9. Required power of No.13 BC

In this case, B=1,050mm, Q_t =1,500mt/h, h=20.1m, l=77.1m, v=155m/min, rubber lagged

pulley, θ:200° =3.49rad, μ=0.3, W=150kg/m, f=0.03

Then　P_1 =0.06×0.03×80×155×(77.1+49)/367=6.67kw

　　　P_2 =0.03×1,500×(77.1+49)/367=15.46kw

　　　P_3 =+(20.1×1,500)/367=82.15kw

　　　P_t =0

Hence　P =6.67+15.46+82.15=104.28kw

Recommended motor power P_m is 104.28/0.8=130.35→150kw

8-4-10. Required power of No.14 & 15

In this case, B=1,200mm, Q_t =1,500mt/h, h= 4.5m, l= 15.5m, v=135m/min, rubber lagged pulley, θ:200° =3.49rad, μ=0.3, W=150kg/m, f=0.03

Then　P_1 =0.06×0.03×90×135×(15.5+49)/367=3.84kw

　　　P_2 =0.03×1,500×(15.5+49)/367= 7.91kw

　　　P_3 =+(4.5×1,500)/367=18.39kw

　　　P_t =0

Hence　P =3.84+7.91+18.39=30.14kw

Recommended motor power P_m is 30.14/0.8=37.68 →37kw

Table 9. Driving coefficient 1 / $e^{\mu\theta-1}$

	μ			
θ	0.35	0.30	0.25	0.20
180°	0.499	0.638	0.838	1.140
200	0.418	0.541	0.718	0.990
210	0.384	0.499	0.667	0.925
220	0.353	0.462	0.621	0.866
230	0.325	0.428	0.579	0.812
240	0.300	0.399	0.541	0.763
360	0.125	0.179	0.262	0.399
420	0.083	0.125	0.190	0.300

8-5. Calculation of effective tension of No.1 BC

F_p =6,120×9.78kw/70m/min=855 kg

F_2 = F_p /$e^{\mu\theta-1}$ =855kg×0.541=462kg (wrap angle 200°, μ=0.3)

Minimum tension :F_4 =50/8・lc (Qt /0.06v+ W_1)

Here l_c :1.0m, l_r :2.0m, v:70m/min, Q_t :1,340mt/h, W_1 :58kg/m

T_{4c}=50/8×1.0m(1,340mt/h/0.06×70m/min+58kg/m)=2,357kg

T_{4r}=50/8×2.0m×58kg/m

Maximun tension : F_{max} = F_p + T_{4c}=855kg+2,357kg=3,212kg

Calculations on other belt conveyors are omitted here, but their results are listed Annexed table[SPECIFICATIONS OF BELT CONVEYORS].

8-6. Revolutionary speeds of head pulleys (n:rpm)

[No.1BC] Here v=70m/min, pulley daiameter D=900 mmφ=0.9mφ
Then n= v/(πD)=70 m/min/(3.14×0.9m)=24.76rpm

[No.2BC] Here v=135m/min, D=900mmφ=1.0mφ
Then n= v/(πD)=135m/min/(3.14×1.0m)=42.78rpm

[No.3BC] Here v=155m/min, D=800 mmφ=0.8mφ
Then n= v/(πD)=155m/min/(3.14×0.8m)=61.67rpm

[No.4 & 5BC] Here v=110m/min, D=800mmφ=0.8mφ
Then n= v/(πD)=110m/min/(3.14×0.8m)=44.77rpm

[No.6BC] Here v=150m/min, D=800mmφ=0.8mφ
Then n= v/(πD)=150m/min/(3.14×0.8m)=59.68rpm

[No.7BC] Here v=150m/min, D=980 mmφ=0.98mφ
Then n= v/(πD)=150m/min/(3.14×0.98m)=48.72rpm

[No.8,9,10BC] Here v=105m/min, D=800 mmφ=0.8mφ
Then n= v/(πD)=105m/min/(3.14×0.8m)=41.78rpm

[No.11BC] Herev=135m/min, D=800mmφ=0.8mφ
Then n= v/(πD)=135m/min/(3.14×0.8m)=53.71rpm

[No.12BC] Here v=135m/min, D=800 mmφ=0.8mφ
Then n= v/(πD)=135m/min/(3.14×0.8m)=53.71rpm

[No.13BC] Here v=155m/min, D=1,20mm, φ=1.2mφ
Then n= v/(πD)=155m/min/(3.14×1.2m)=41.11rpm

[No.14&15BC] Here v=135m/min, D=800mmφ=0.8mφ
Then n= v/(πD)=135m/min/(3.14×0.8m)=53.71rpm

8-7. Revolutionary speeds of motors N、andreductionratio ofsprocket gears i

[No.1BC]　Here GM-15-DNP, 4p, 50Hz, 400V,1/30
N=1,500(1−0.03)×1/30=48.5rpm
i_t =48.5/24.76=1.956　theoretical gear ratio
i_a = N_{ts}/N_{tb}=35/19=1.842　actual gear ratio

[No.2BC]　Here TKB-300, 6p, 1/22.5
N=1,000(1−0.03)×1/22.5=43.1rpm direct drive

[No.3BC]　Here GM-55-DSP, 4p, 50Hz, 400V, 1/15
N=1,500(1−0.03)×1/15=97rpm,　i_t =97.0/61.7=1.573
i_a =30/19=1.579

[No.4 & 5BC]　Here GM-45-DSP, 4p, 50Hz, 400V, 1/15
N=1,500(1−0.03)×1/15=97rpm,　i_t =97.0/44. 8=2.165
i_a =40/18=2.220

[No.6BC]　Here GM-55-DSP, 4p, 50Hz, 400V, 1/15
N=1,500(1−0.03)×1/15=97rpm, i_t =97.0/59.7=1.624
i_a =30/18=1.667

[No.7BC]　Here SB-E150, 6p, 1/20
N=1,000(1−0.03)×1/20=48.5rpm,　direct drive

[No.8,9,10BC]　Here GM-37-DRP,4p,50Hz,400V,1/15
N=1,500(1−0.03)×1/15=97rpm,　i_t =97.0/41.78=2.322
i_a =35/15=2.333

[No.11BC]　Here GM-11-DL, 6p, 1/20
N=1,500(1−0.03)×1/15=97rpm,　i_t =97.0/53.71=1.805
i_a =30/17=1.765

[No.12BC]　Here GM-55-DSP, 4p, 50Hz, 400V, 1/15
N=1,500(1−0.03)×1/15=97rpm,　i_t =97.0/53.71=1.805
i_a =30/17=1.765

[No.13BC]　Here SB-E150, 6p, 1/20
N=1,000(1−0.03)×1/20=48.5rpm, direct drive

[No.14&15BC]　Here GM-37-DRP, 4p, 50Hz, 400V, 1/15
N=1,500(1−0.03)×1/15=97rpm,　i_t =97.0/53.71=1.805
i_a =30/17=1.765

8-8. Conveyor belt selection

In order to hold capabilities of anti abrasion and anti shock against rugged lump ores, it is necessary to have a certain rubber thickness to each ore size.

8-8-1. Maximum ore sizes for each conveyor

Maximum ore sizes for each conveyorare shown in Table.10.

Table10. Maximum ore sizes for each conveyor

Maximum ore size	ConveyorNo.
350mm (14″)	No,1,2,3,4,5BC
70mm (2″1/2)	No,6,7,8,9,10 BC
25mm (1″)	No,11,12,13,14,15,BC

8-8-2.Specifications of conveyor belts (BANDO CHEMICAL INDUSTRIES, LTD.)

Referring to above mentioned data and manufacturer' s catalogueswe selected conveyor belts as the following.

[Steel code ST-1250]: No.2BC

Belt width: 1,250 mm, top cover：8 mm, ：bottom cover 6 ㎜
Tensile strength: more than 1,250 kg/cm
Code diameter: 4.6 mmϕ, code structure:7×7, code pitch：14.5 mm,
Code number:70, galvanization for element:zinc

[Banlon#900] : No.1 BC

Belt width: 1,800 mm, top cover：15 mm, bottom cover： 6 mm
Cushion rubber:anti-abrasion & anti-shock load,
Tensile strength: more than: 900 kg/cm
Elongation of cover rubber: new priduct 450%, after aging 400%

[Banlon #800]: No,3, 4, 5, 7, 14, 15 BC

Belt width: 1,200 mm, 1,050 mm, 900 mm, 3 plies
top cover： 6 mm, bottom cover： 2 mm,
Tensile strength: more than 800 kg/cm
Elongation of cover rubber: new priduct 450%, after aging 380%

[Banlon #500]: No,6,11,12,13 BC

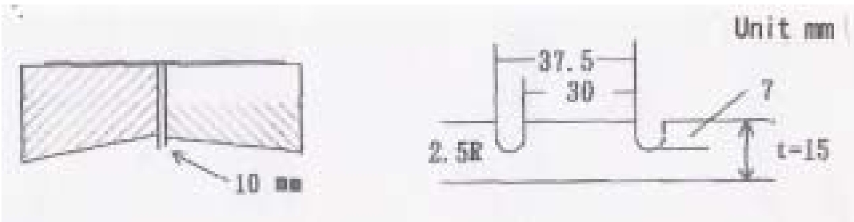
Belt width: 1,200 mm, 750 mm, 3 or 2 plies
top cover： 6 mm or 4 mm, bottom cover：2 mm,
Tensile strength: more than 500 kg/cm
Elongation of cover rubber: new priduct: 450%, after aging 380%

[Banlon#400]: No,8,9,10 BC

Belt width: 900 mm, 2 plies
top cover： 6 mm, bottom cover: 2 mm,
Tensile strength: more than: 400 kg/cm
Elongation of cover rubber: new priduct: 450%, after aging 380%

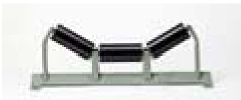
8-9. Specifications of rubber lagging for head pulleys

Type:Model 5, special double helical
Pulley width:belt width＋150 mm

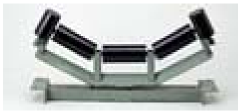


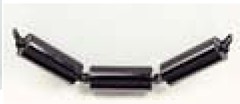
8-10. Specifications of rollers

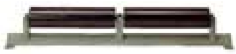
Standard carrier :	[Sanki] BKM 20° 3 roller type
	BKM 30° 3 roller type
Self-aligning carrier :	2° inclind forward 3 roller type
Standard idler :	flat 1 roller type
Self-aligning idler :	2° inclind forward 2 roller type


JIS 3 roller trough carrier roller


Impact roller

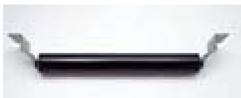

Self-aligning carrier

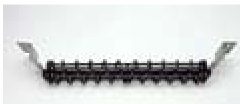

Garland Carrier roller/GCR


Flat Carier roller/KFC


Flat impact Carrier roller


Self-aligning idler roller/KAR


idler roller/KR


Spiral idler roller/KSR


Rubber sleeved idler roller


Guide roller for carrier


Guide roller for flat belt

8-11. Accessories

Belt cleaner : scraper with carbide tips for outside head pulley type
V-type scraper for belt inside
Emergency switch : pull switch for conveyor side
Snaking detector : tumbler switch type
Choking detector : hanging switch type

Belt cleaner
scraper with carbide tips
for outside head pulley type

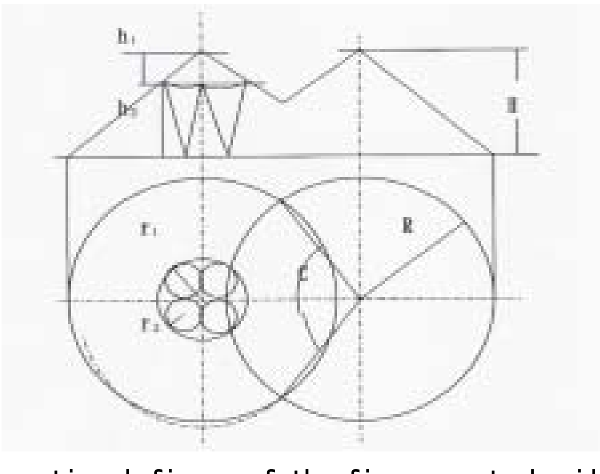
Emergency switch
tumbler switch type

Choking detector
hanging switch type

9) Fine ore stock pile

9-1. Design concept

Structure of fine ore stock pile was designed to get necessary inventory capacity and minimize construction costs. The stock pile will consist of two cones which a part of them penetrate each other. We estimated dimensiones of each cone to be 55.6mϕ ×24mH with angle of repose 40° .



Conceptional figure of the fine ore stock pile

9-2. Calculation of total volume including dead stock V

$V = (V_1 - V_2) \times 2$

Where volume of a perfect cone: V_1

$V_1 = \pi/3 \cdot R^2 H = 3.14 \times 27.8^2 \times 24/3 = 19{,}413\ m^3$

Volume of penetrated part : V_2

Area of bow: S

$S = 1/2 \cdot R2\ (C - \sin C)$

Where unit of central angle C is radian.

$= 1/2 \cdot 27.82\ \ (1.918 - 0.940) = 378\ m^2$

Then $V_2 = 378\ m^2 \times 6.7/3 = 844\ m^3$

$V = (19,413 - 844) \times 2 = 37,138\ m^3$

Hence maximum total tonnage T

$T = 37,138\ m^3 \times 1.7\ mt/m^3 = 63,135\ mt.$

9-3. Calculations of live inventory V_a

To simplifycaluculation, the live inventory was devided into volume of two perfect cones V'_1 and eight inverted cones V'_2 .

$V'_1 = \pi/3 \cdot r_1{}^2\ h_1 \times 2 = 3.14 \times 122 \times 9 \times 2/3 = 2,713\ m^3$

$V'_2 = \pi/3 \cdot r_2{}^2\ h_2 \times 8 = 3.14 \times 62 \times 15 \times 8/3 = 4,521\ m^3$

Hence volume of the live inventory is givenby the following equation.

$V' = V'_1 + V'_2 = 2,713\ m^3 + 4,521\ m^3 = 7,234\ m^3$

Then tonnage of the live inventory will be

$7,234\ m^3 \times 1.7\ mt/m^3 = 12,298\ mt.$

It was decided that central parts of the each con of the fine ore stock pile will be covered by octagonal pyramid shaped roofs in order to prevent from sliding due to the heavy rains.

End