CHAPTER 3. PLAN FOR TAILING DAM

3–1 Position and Construction of Tailing Dam

At first, it was planned that an earth rock dam will be used for the construction of the tailing dam and a search was made for a place suitable for the dam. A ravine at Estancia Chajlluma was thought to be the most suitable place, because there is a large pocket for water and a throat formation suitable for constructing a dam.

On the other hand, the construction cost would be very large, and moreover there is a problem that it will be difficult to transport tailings hydraulically or recover water as there is a valley and a mountain between that ravine and the new concentration plant. Accordingly, this plan was set aside as Plan 2 and a more economical and rational tailing dam plan was studied.

As a result, a dam to deposit earth from outside corresponding to the increase of the quantity of sediments was adopted as the construction of the tailing dam and it was also decided to use a part of tailings transported hydraulically as dam material. The planned site for the dam is a flat place situated on the west of Cerro Sacamarca and south of Colas Arenas. As this plan is superior to Plan 2 in that its construction cost will be lower and the hydraulic transportation of tailings and water recirculation will be easier, etc., this plan was set as Plan 1 and further investigation was carried out concerning it.

The design conditions in the basic design of the tailing dam are as follows.

Quantity of processed ore :	10,000 t/day
Quantity of tailings (dry weight) :	9,865.8 t/day
Water in tailings :	34,859.0 t/day
Density of tailings transported :	P.D. ≑ 22%
Specific gravity of solids in tailings :	Gs = 2.75
Required quantity of recovered water :	approx. 17,000 m^3/day
Annual number of working days of concentration mill :	300 days
Number of operating years of concentration mill :	10 years

The state of the proposed dam site and the idea of selecting the site are described as follows.

From the western side to the north western side of Cerro Sacamarca, a tailing dam will be constructed. About the materials and method of constructing the dam, as the dam material for the first year sediments of conglomeratic soil around the dam site will be used to build an earth dam, and from the following year on, coarse grains (above 100μ) in tailings separated by classifying at the time of discharging the tailings to the tailing dam will be piled up from outside to increase the height.

The height of crest required will be 3,800 m with a free board of 2 m added to the sedimentation height. The capacity at the time of completing sedimentation will be 36,256,000 m^3. The slope of the dam shall be 1:2.0 both on the inside and outside. Before carrying forward practical design, it is required to test earth quality and based on the test data, to examine the plan of slump and then determine the slope of the dam and the crest width. In the case where water intercepting material is required, the surface (inner slope) of the dam will be covered with viscous clay at a place as near as possible from the tailing dam.

Dams will be constructed at the two positions, north and south. The crest heights and shapes of the dams when the sedimentation of tailings will have been finished are shown in the

Table II–3–1 Shape on Height of Dam When Sedimentation of Tailing Mill Plant Finished

Location	Height (m) when sedimentation of tailing mill have finished				Average size of dam
	Level of sedimentation	Crest of dam	Toe of shape of inside	Toe of shape of outside	Width of crest x height x length
North	3,798	3,800	3,765	3,760	8.0 x 17.5 x 2,400
South			3,745	3,735	10.0 x 30.0 x 970

The required dam size and dam material determined by finding out the quantity of sedimentation and the height of sediments from the sedimentation quantity curve (Fig. II–3–2) are as shown in Table II–3–2.

3–2 Quantity of Tailing Sedimentation and Recovered Water

(1) Quantity of Tailing Sedimentation

By estimating tailing density within the tailing sedimentation area, the quantity of tailing sedimentation and the quantity of supernatant water are determined. Quantity of tailing V, transported hydraulically from the concentration mill is,

$$V = \frac{9,865.8}{2.75} + 34,859.0 = 38,466.6 \text{ m}^3/\text{day}$$

The specific gravity (Gr) of tailings when hydraulically transported is,

$$Gr = \frac{W}{\gamma_W V} = \frac{1}{\gamma_W} \times \frac{W_S + W_W}{V}$$

$$= \frac{1}{1.0} \times \frac{9,865.8 + 34,859.0}{38,466.6} = 1.163$$

Fig. II-3-1 Tailing Dam Layout

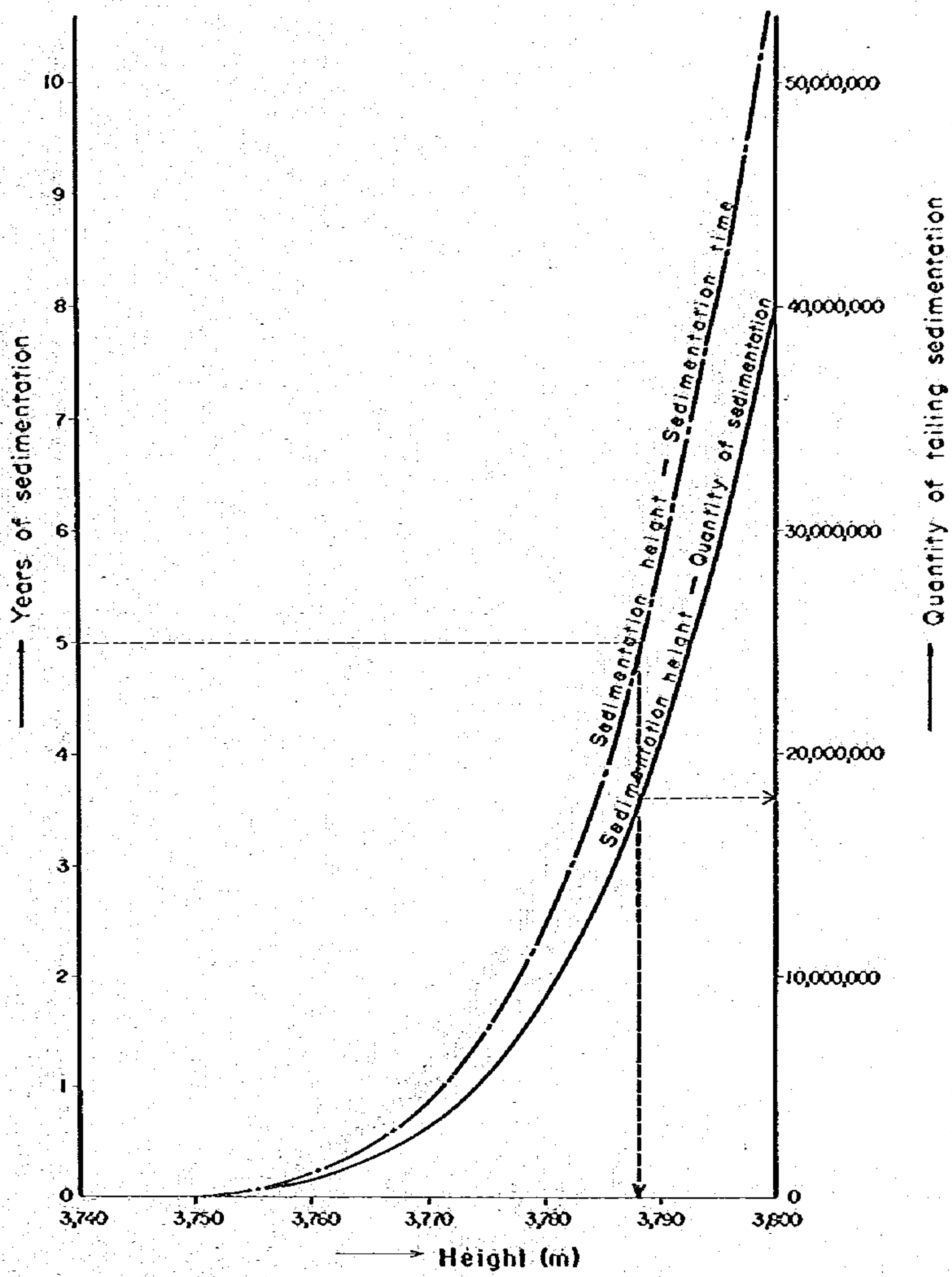

Fig. II-3-2 Sedimentation Quantity Curve

Table II-3-2 Annual Quantity of Tailing Sedimentation and Material of Dam

Year of Sedimentation	Quantity of Sedimentation	Height of Sediments	Required size of dam and material		Construction (m^3)		Note
			Size of dam (m) Width of crest x Height x Length	Material to Construct dam	For One Year	Total	
-2.5 ∫ 0					498,000	498,000	Conglomeratic earth
1	3,625,600	3,771.0	N 8.0 x 4.0 x 900 S 8.0 x 16.5 x 650	58,000 440,000 (498,000)	372,000	870,000	Tailing
2	7,251,200	3,777.5	8.0 x 7.25 x 1,330 8.5 x 19.75 x 730	217,000 692,000 (909,000)	372,000	1,242,000	
3	10,876,800	3,782.0	8.0 x 9.5 x 1,900 9.0 x 22.0 x 790	487,000 921,000 (1,408,000)	372,000	1,614,000	
4	14,502,400	3,785.5	8.0 x 11.25 x 1,950 9.0 x 23.75 x 830	669,000 1,114,000 (1,783,000)	372,000	1,986,000	
5	18,128,000	3,788.0	8.0 x 12.5 x 1,970 9.5 x 25.0 x 860	813,000 1,279,000 (2,092,000)	372,000	2,358,000	
6	21,753,600	3,790.5	8.0 x 13.75 x 2,080 9.5 x 25.0 x 890	1,015,000 1,448,000 (2,463,000)	372,000	2,730,000	
7	25,379,200	3,792.5	8.0 x 14.75 x 2,160 10.0 x 27.25 x 910	1,195,000 1,599,000 (2,794,000)	372,000	3,102,000	
8	29,004,800	3,794.5	8.0 x 15.75 x 2,240 10.0 x 28.25 x 930	1,394,000 1,747,000 (3,141,000)	372,000	3,474,000	
9	32,630,400	3,796.5	8.0 x 16.75 x 2,320 10.0 x 29.25 x 950	1,613,000 1,903,000 (3,516,000)	369,000	3,843,000	Conglomeratic earth 498,000 Tailing 3,345,000
10	36,256,000	3,798.0	8.0 x 17.5 x 2,400 10.0 x 30.0 x 970	1,806,000 2,037,000 (3,843,000)			

The average of dry bulk density (γd) in the tailing dam will be at the usual level of 0.8 t/m^3, hence the average density of tailings in the tailing dam is estimated from the above value.

$$\gamma d = \frac{Ws}{V} = \frac{Ws}{Vs + Ww}$$

where Vs : quantity of tailings (volume),

Vw : water contents in tailings

hence,

$$Vw = \frac{Ws}{0.8} - Vs$$

$$= \frac{9,865.8}{0.8} - \frac{9,865.8}{2.75} = 8,744.7\ m^3 = 8,744.7^t = Ww$$

$$P.D. = \frac{Ws}{Ws + Ww} \times 100$$

$$= \frac{9,865.8}{9,865.8 + 8,744.7} \times 100 \div 50\%$$

The quantity of supernatant water (Q) is,

$$Q = 34,859.0 - \frac{9,865.8 \times (100 - 50)}{50} = 24,993.2\ m^3/day$$

Therefore, the quantity of tailing sedimentation (Q_T) is,

$$Q_T = 38,446.6 - 24,993.2 = 13,453.4\ m^3/day$$

The quantity of tailing sedimentation in a year becomes,

$$13,453.4 \times 300\ (D) = 4,036,000\ m^3/year$$

Wet density (γt) and dry density (γd) of sedimented tailings at this time are,

$$\gamma t = \frac{9,865.8 + 9,865.8\ (water\ 50\%)}{13,453.4} = 1.467\ t/m^3$$

$$\gamma d = \frac{9,865.8}{13,453.4} = 0.733\ t/m^3$$

If the required quantity of tailings (in the wet state) is determined when using the tailings within the sedimentation area as dam material by assuming that the density of tailings after being compacted equals to 1.8 t/m^3, the required quantity of dam material is 3,843,000 m^3 (See Table II–3–2) among which 498,000 m^3 is the quantity for the first year. For dam material for the first year conglomeratic earth around the tailing dams will be used, the quantity of tailings required for the dams therefore becomes:

$$3,843,000 - 498,000 = 3,345,000\ m^3$$

If the density after being compacted is assumed to be 1.8 t/m^3, the quantity of tailings in the wet state is :

$$3,345,000 \times \frac{1.8}{1.467} \div 4,104,000 \text{ m}^3$$

Hence, the quantity of tailings which will be sedimented inside the dams in ten years will become:

$$4,036,000 \times 10 \text{ (years)} = 36,256,000 \text{ m}^3$$

(2) Loss Resulting from Evaporation, etc.

In the computation of evaporation quantity the average value of annual evaporation quantity in 1964 ~ 1976, 1,567 mm/year, is used.

For the inflow of daily rainfall, the average value of annual rainfall in 1934 ~ 1982, 542.63 mm/year, is used for the catchment of the tailing dams, 3,448,000 m². The coefficient of run-off in this case is 1.0 inside the overflowing level, and 0.3 outside the overflowing level.

According to the above-mentioned computation of the quantity of tailing sedimentation, the quantity of supernatant water is 24,993.2 m³/day, the annual quantity of supernatant water therefore is:

$$24,993.2 \times 300 = 7,497,960 \text{ m}^3/\text{year}$$

If the water filled area corresponding to the quantity of tailing sedimentation and the height of sediments each year (See Table II–3–8) is determined, and, by determining the difference between evaporation quantity and the inflow of rainfall next, the annual loss is computed, the following table is obtained.

The annual loss will become maximum in the 10th year, and the ratio of loss to the quantity of supernatant water is :

$$1,959,600 \div 7,497,960 \times 100 = 26.1\%$$

By estimating that other loss is 3.9%, total loss is estimated to be 30%.

(3) Recovered Water

If the ratio of loss is assumed to be 30% from the result of computation in (2), the quantity of water that can be recovered is :

$$24,993.2 \times (1-0.3) = 17,495 \text{ m}^3/\text{day} > \text{required quantity} \div 17,000 \text{ m}^3/\text{day}$$

Ratio of recovered water to total water in tailings becomes,

$$17,495 \div 34,859.0 \times 100 = 50.2\%$$

Table II-3-3 Annual Loss of Rainfall

Years of Sedimentation	Height of Sedimentation (m)	Area of water surface A (m²)	Annual evapolation quantity A x 1.567 (m³)	Annual rainfall water to inflow (m³)		Annual loss (m³)
				Inside of overflowing level A x 0.54263	Outside of over-flowing-level (3,448,000-A) x 0.54263 x 0.3	
1	3,771.0	400,600	627,700	217,400	459,100	– 48,800
2	3,777.5	690,400	1,081,900	374,600	448,900	258,400
3	3,782.0	967,500	1,516,100	525,000	403,800	587,300
4	3,785.5	1,257,500	1,970,500	682,400	356,600	931,500
5	3,788.0	1,464,700	2,295,200	794,800	322,900	1,177,500
6	3,790.5	1,661,200	2,603,100	901,400	290,900	1,410,800
7	3,792.5	1,784,500	2,796,300	968,300	270,800	1,557,200
8	3,794.5	1,907,700	2,989,400	1,035,200	250,700	1,703,500
9	3,796.5	2,031,000	3,182,600	1,102,100	230,700	1,849,800
10	3,798.0	2,123,400	3,327,400	1,152,200	215,600	1,959,600

3–3 Drainage Safety Facilities

(1) Estimation of Rainfall Intensity

Rainfall data obtained in the survey this time are the monthly quantities of rainfall and the annual rainfall quantities in the 48 years from 1934 to 1981 in Catavi, and the daily quantities of rainfall in 13 years from 1970 to 1982 in Catavi, Siglo XX, Miraflores and Lupi-Lupi.

On the other hand, the catchment area of the tailing dams is about 7.6 km² and comparatively small, and when considering the outflow of rainwater, daily quantity of rainfall and hourly quantity of rainfall begin to matter, but in the above-mentioned data of the daily quantity of rainfall, there are no records of the duration of continuous rainfall, so it seems impossible to estimate the maximum daily rainfall quantity and the maximum hourly rainfall quantity. Accordingly, we estimated these rainfall quantities concerning the three following cases.

1 By arranging the records of rainfall quantities in 40 years in Catavi in the descending order from the largest annual rainfall quantity, obtaining the arithmetic averages of the first 10 years and the first 20 years, and assuming that the number of rainy days per month is five,

and the duration of continuous rainfall is four hours, the daily quantity of rainfall and the hourly quantity in 100 year probability and 200 year probability were obtained from the maximum value of the average monthly rainfall quantity.

2 By arranging the maximum values of the monthly rainfall quantities in each year in the descending order, and from the monthly rainfall quantities of 100 year probability and 200 year probability obtained on a logarithmic probability paper, the daily rainfall quantity and the hourly rainfall quantity in 100 year probability and 200 year probability were obtained by assuming that the number of rainy days in a month is five and the duration of continuous rainfall is four hours.

3 By arranging the monthly rainfall quantities of each year in the descending order and obtaining the arithmetic averages of the first 10 years and the first 20 years, the average maximum daily rainfall quantity and the average maximum hourly rainfall quantity were obtained from the maximum value of the average monthly rainfall quantity by assuming that the number of rainy days in a month is five and the duration of continuous rainfall is four hours.

As a result of trial computation with regard to the above three cases, Case 2 gave the maximum value, so we decided to adopt this value, and computation about case 2 is described in the following.

If the maximum values of the monthly rainfall quantities of each year are taken out and arranged in the descending order, the following table is obtained.

Table II-3-4 Order of Maximum Monthly Rainfall

Order (i)	Year, month of the rainfall	Maximum monthly Rainfall y_i (mm)	$\frac{2i-1}{2N} \times 100$	Order (i)	Year, month of the rainfall	Maximum monthly Rainfall y_i (mm)	$\frac{2i-1}{2N} \times 100$
1	1941.1	268.1	1.25	21	1956.1	164.0	51.25
2	1934.2	265.2	3.75	22	1942.1	157.5	53.75
3	1960.1	241.5	6.25	23	1950.1	155.0	56.25
4	1959.3	240.0	8.75	24	1930.1	153.3	58.75
5	1939.1	236.9	11.25	25	1964.11	153.0	61.25
6	1947.2	230.1	13.75	26	1962.12	152.0	63.75
7	1968.2	203.2	16.25	27	1951.1	150.0	66.25
8	1945.8	200.8	18.75	28	1969.1	149.2	68.75
9	1941.8	200.4	21.25	29	1965.1	145.5	71.25
10	1949.12	198.5	23.75	30	1963.1	144.0	73.75
11	1945.1	196.2	26.25	31	1947.8	143.5	76.25
12	1972.1	195.2	28.75	32	1953.2	141.0	78.75
13	1957.8	191.0	31.25	33	1943.3	134.7	81.25
14	1933.8	190.0	33.75	34	1954.1	131.5	83.75
15	1952.9	183.0	36.25	35	1961.2	125.3	86.25
16	1955.8	181.0	38.75	36	1936.1	125.2	88.75
17	1973.12	172.0	41.25	37	1971.12	123.0	91.25
18	1935.1	171.5	43.75	38	1943.1	121.2	93.75
19	1958.1	170.0	46.25	39	1938.2	117.0	96.25
20	1966.12	168.0	48.75	40	1967.2	110.0	98.75

Number of data = 40

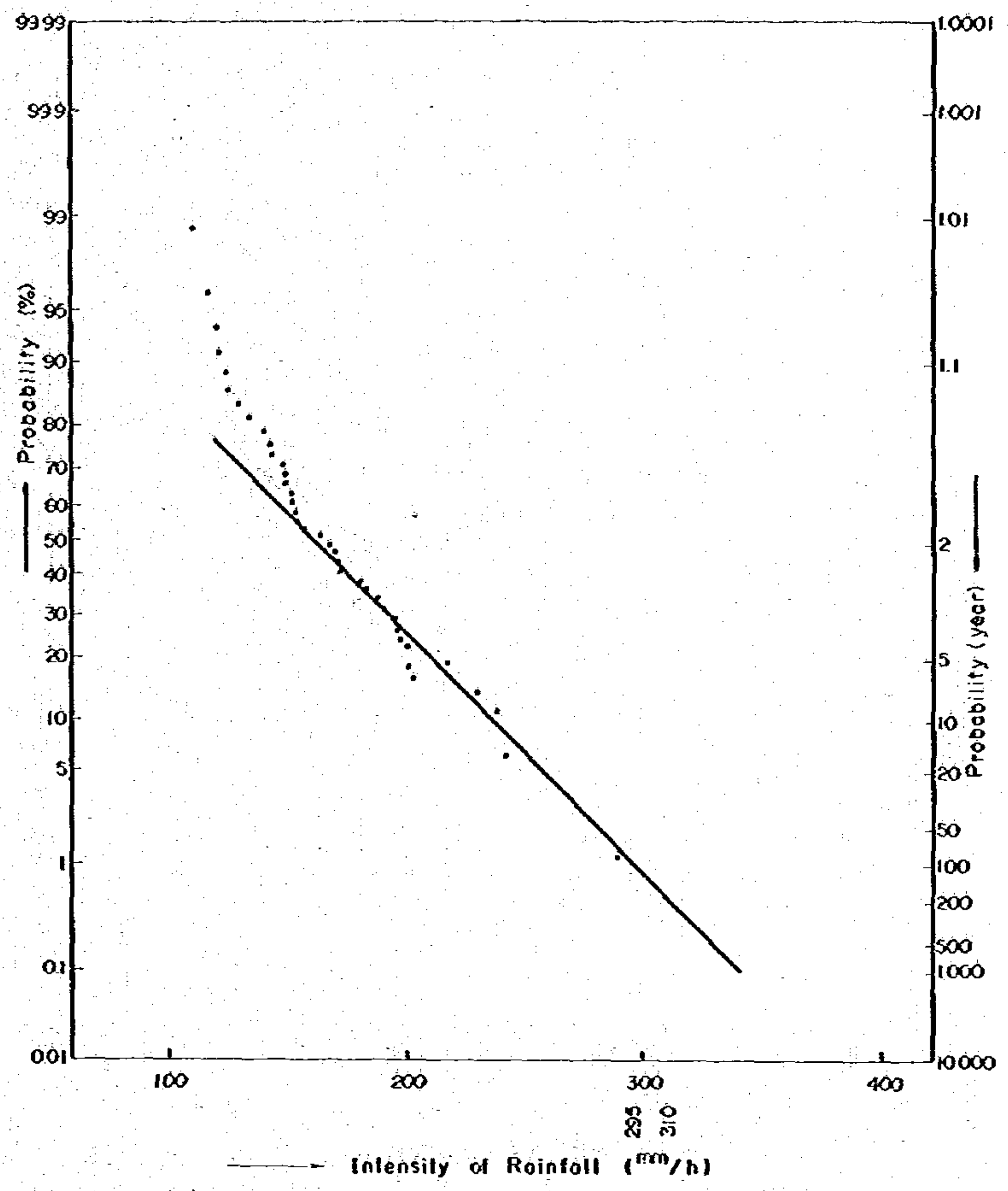

Fig. II-3-3 Probability of Rainfall

If γi and $(2i-1)/2N$ are plotted on a normal probability paper (Fig. II–3–3) to obtain 100 year and 200 year probability rainfall quantities ;

100 year probability rainfall quantity : 295 mm/month

200 year probability rainfall quantity : 310 mm/month

If the number of rainy days in a month and the duration of continuous rainfall are respectively assumed to be five days and four hours, the daily rainfall quantities and the monthly rainfall quantities in 100 year probability and 200 year probability become ;

100 year probability daily rainfall quantity : $295 \div 5 = 59.0$ mm/day

100 year probability hourly rainfall quantity : $59.0 \div 4 = 14.8$ mm/hr

200 year probability daily rainfall quantity : $310 \div 5 = 62.0$ mm/day

200 year probability hourly rainfall quantity : $62.0 \div 4 = 15.5$ mm/hr

If rainfall intensity equations are determined from the above data, the 100 year probability rainfall intensity equation is :

$$14.8 = \frac{a}{b + 60} \quad \text{--------} \quad (1)$$

$$\frac{59.0}{24} = 2.46 = \frac{a}{b + 1,440} \quad (2)$$

from equations 1 and 2 ,

$$14.8\,b + 888 = a$$

$$2.46\,b + 3,540 = a$$

from these equations,

$$b = \frac{3,540 - 888}{14.8 - 2.46} = 214.9$$

$$a = 4,068.5$$

Therefore, the 100 year probability rainfall intensity equation is determined as follows.

$$\gamma_{100} = \frac{4,068.5}{t + 214.9}$$

Next, the 200 year probability rainfall intensity equation is :

$$15.5 = \frac{a}{b + 60} \qquad 3$$

$$\frac{62.0}{24} = 2.58 = \frac{a}{b + 1,440} \qquad 4$$

from equation 3 and 4 ,

$$15.5\,b + 930 = a$$

$$2.58\,b + 3,720 = a$$

from these equations,

$$b = \frac{3,720 - 930}{15.5 - 2.58} = 215.9$$

$$a = 4,277.1$$

Therefore, the 200 year possibility rainfall intensity equation is determined as follows.

$$\gamma_{200} = \frac{4,277.1}{t + 215.9}$$

(2) Drainage Equipment Design

i) Prerequisites

About the drainage equipment, design calculations are carried out based on the following conditions.

a. In the area of valley including inside area 2,126 km^2, outside area 5,480 km^2, rainfall over 4,158 km^2 of the outside area shall be drained by providing waterways on the sides of mountains, and the rainfall quantity over the remaining 1,322 km^2 shall flow into the area.

b. The mountainside waterways shall have a sufficient sectional area which can drain water inflow according to 100 year probability rainfall into the mountainside waterway safely.

c. The drainage equipment within the site (underdrain) shall have a sufficient sectional area which can safely drain inflowing water according to the 100 year probability rainfall into the area.

d. Emergency drainage equipment shall have a sufficient sectional area which can safely drain inflowing water according to the 200 year probability rainfall into the area.

e. In the above calculation, the quantity of evaporation is omitted.

f. The ratio of outflow shall be 30% for the outside area, and 100% for the inside area.

g. The quantity of water to be drained is the sum of supernatant water and inflow water according to rainfall, and spring water and valley water shall be neglected.

ii) Calculation of outflow quantity

According to the experimental equation by Shirasaka Water Measurement Laboratory, taken for water to reach (t) is,

$$t = 0.33 L^{0.63} \text{ (min)}$$

L : flow passage length from the farthest point of the area of valley to the position where flow quantity is calculated.

Rainfall intensity equations from the above-mentioned are as follows:

100 year probability rainfall Intensity (γ_{100})

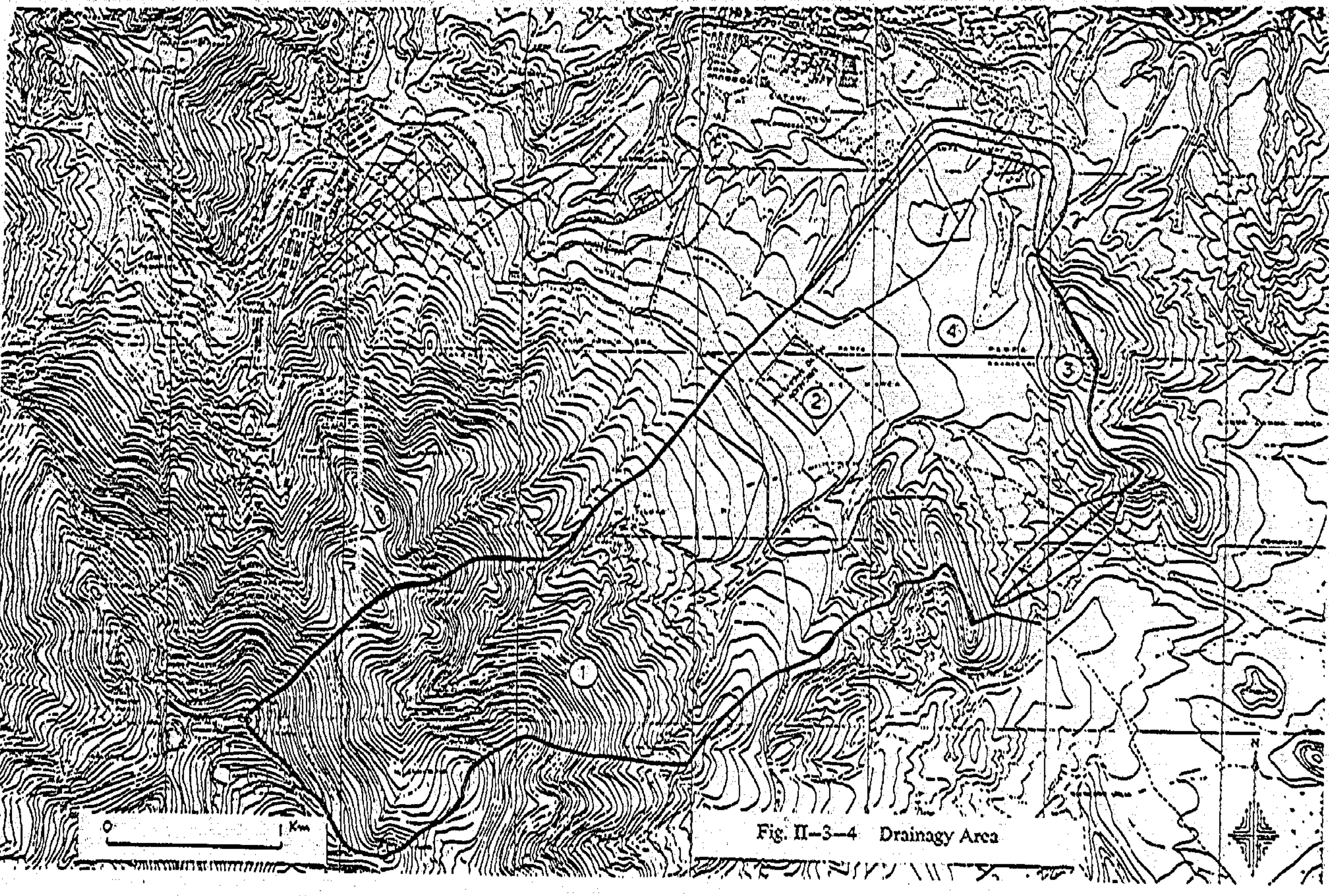

Fig. II–3–4 Drainagy Area

$$\gamma_{100} = \frac{4{,}068.5}{t + 214.9}$$

200 year probability rainfall intensity (γ_{200})

$$\gamma_{200} = \frac{4{,}277.1}{t + 215.9}$$

The calculation of planned floodover water quantity (Q) is carried out by Rational's formula:

$$Q = \frac{1}{3.6} \, f \cdot \gamma \cdot A \quad (m^3/sec)$$

f : coefficient of outflow, within mill plant = 1.0, outside mill plant = 0.3

γ : rainfall intensity (mm/hr)

A : area of valley (km^2)

The results of carrying out the above calculations by determining the area of valley and the length of current from Fig. II–3–4, the figures for the valley, are shown in the following table.

Table II–3–5 Water Way and Drainage Equipment

Class of construction	Number of the basin	Length of current L (m)	Area of the basin A (km^2)	Time to arrive t (min)	Coefficient of out flow f	Intensity of rainfall γ_{100}	Intensity of rainfall γ_{200}	Quantity of out flow γ_{100}	Quantity of out flow γ_{200}	Note
Water way in hill side	1	4,500	4.158	66.1	0.3	14.48	--	5.017		
Drainage equipment in mill plant	2	2,000	1.134	39.6	0.3	15.99		1.511		
	3	1,500	0.158	33.1	0.3	16.41		0.227		
	4	2,500	2.126	45.6	1.0	15.62		9.224		
							Total	10.992		
Emergency drainage equipment	2	2,000	1.134	39.6	0.3		16.74		1.582	
	3	1,500	0.158	33.1	0.3		17.18		0.269	
	4	2,500	2.126	45.6	1.0		16.36		9.661	
								Total	11.512	

iii) Hillside waterway design

The hillside waterway shall be an open waterway of reinforced concrete construction.

Design flow velocity (v) and flow quantity (Q) are calculated according to Manning's formula.

$$v = \frac{1}{n} R^{2/3} I^{1/2} \quad (m/sec)$$

$$Q = A \cdot v$$

n : coefficient of roughness of the inner surface of the waterway = 0.015 (concrete)

R : hydraulic radius = $\frac{A}{P}$ (m)

A : water flow sectional area (m^2)

I : waterway gradient ≒ 0.03 (from the figures for the valley)

The section of the hillside waterway shall be as shown in the figure on the right.

$A = 1.00 \times 1.00 = 1.00\ m^2$

$P = 1.00 \times 3 = 3.00\ m$

$R = 1.00 \div 3.00 = 0.333\ m$

$$v = \frac{1}{0.015} \times 0.333^{2/3} \times 0.03^{1/2} = 5.551\ m/sec$$

$$\therefore Q = 1.00 \times 5.551 = 5.551\ m^3/sec > Q_{100}$$

$$= 5.017\ m^3/sec$$

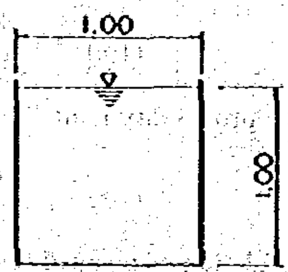

Fig. II-3-5 Profile of Water Way in Hill Side

iv) Design of drainage equipment for water in tailing dam

The drainage equipment for water in the tailing dam shall use underdrains (constructed by fitting cover plates to open drains of reinforced concrete) which will be extended corresponding to the increase of the quantity of sediments. As the waterway gradient is as small as 1% on the north side and 3% on the south side, inlet parts in uncovered state can be regarded as dams with very large widths. Accordingly, the flowing capacity of the waterways as underdrains after water has entered can be calculated as follows.

The flow quantity is calculated by Manning's formula:

$A = 1.20 \times 1.30 = 1.56\ m^2$

$P = 1.20 + 1.30 \times 2 = 3.80\ m$

$R = 1.56 \div 3.80 = 0.411\ m$

$I = 0.01$ (north side)

$$v = \frac{1}{0.015} \times 0.411^{2/3} \times 0.01^{1/2}$$

$$= 3.685\ m/sec$$

$$Q = 1.56 \times 3.685 = 5.749\ m^3/sec$$

$$> \frac{Q_{100}}{2} = \frac{10.992}{2} = 5.496\ m^3/sec$$

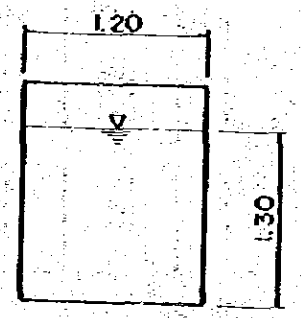

Fig. II-3-6 Blind Ditch in the Bottom

v) Design of Emergency Drainage Equipment

The emergency drainage equipment (spill way) shall be a canal without timbering made by ditching the base mountain ground at dam and part along the dam, and the entrance of the canal will be required to be raised corresponding to levee raising in the dry season every year.

The quantity of flow is calculated by Francis' formula by assuming that contracted flow does not occur at both ends of the dam and there is no approaching velocity. By assuming that the same quantity of water is discharged through the southern and northern emergency drainage canals, we will use a crosssection shown in the figure on the right.

$Q = 1.84\, bH^{3/2}$

b : bottom width = 2.0 m

H : overflow depth = 1.5 m

$Q = 1.84 \times 2.00 \times 1.50^{3/2} = 6.71\ m^3/sec$

$= \frac{Q_{200}}{2} = \frac{11.512}{2} = 5.756\ m^3/sec$

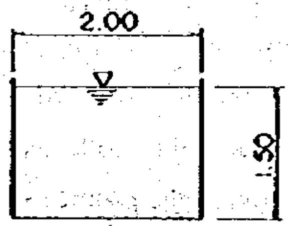

Fig. II−3−7 Emergency Drainage Canal Section

3−4 Hydraulic Transportation of Tailings

The waterway for hydraulic transportation of tailings will use an open waterway of plain concrete construction for its 4 km long part from the concentration mill to the head tank and use a plastic pipe for its 500 m long part from the head tank to the cyclone. The quantity of flow can be calculated by Manning's formula. The section of the open waterway is shown by the drawing on the right.

$A = (0.20 + 0.45) \times 1/2 \times 0.125 + 0.45 \times 0.475$

$= 0.254\ m^2$

$P = (0.475 + 0.177) \times 2 + 0.20 = 1.504\ m$

$R = 0.254 \div 1.504 = 0.169\ m$

$n = 0.015$

$I = 0.01$

$v = \frac{1}{0.015} \times 0.169^{2/3} \times 0.01^{1/2} = 2.038\ m/sec$

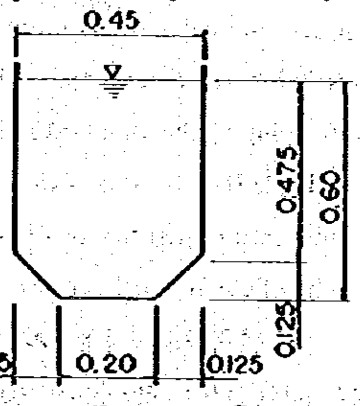

Fig. II−3−8 Tailing Conveyance Channel line Section

$\therefore Q = 0.254 \times 2.038 = 0.518\ m^3/sec >$ tailing quantity $= 38{,}446.6\ m^3/day$

$= 0.455\ m^3/sec$

Blind waterway section is shown in figure on the right.

$A = 0.24^2 \times \pi = 0.181\ m^2$

$P = 0.48 \times \pi = 1.508\ m$

$R = 0.181 \div 1.508 = 0.120\ m$

$n = 0.013$

$I = 0.03$

$v = \frac{1}{0.013} \times 0.120^{2/3} \times 0.03^{1/2} = 3.241\ m/sec$

Fig. II−3−9 Blind Waterway Section

$Q = 0.181 \times 3.241 = 0.587\ m^3/sec >$ tailing quantity $= 0.445\ m^3/sec$

3–5 Computation of Approximate Construction Costs

(1) Prerequisites

Conglomeratic earth used as dam material to secure volume for receiving tailing sediments in the first year will be collected within the a radius of 500 m average transporting distance (one way) around the planned site for the dam, and the dam shall be completed in 2.5 years.

For the dam material from the second year on, tailing above 100μ classified and drained in the sedimenting area will be carried to the dam by dump trucks, spread and compacted by bulldozers to increase the dam height. To simplify calculation, it is assumed that water is removed from the tailings to make the density of the dam material tailings to be 1.8 ton/m³.

Actual working time of heavy machines in a day shall be 6 hours in the case of dump trucks and 5 hours for other heavy machines. The heavy machines are regarded to be fully depreciated leaving no remaining value. The number of heavy machines to be purchased is required number + reserve number (÷30% x required number), and repair cost to be summed up is purchase price x 85% / 10 years for all the machines. Operation capacity is computed conforming to the standard rate of the Ministry of Construction.

Table II–3–2 shows that the material for dam construction required for the first year sedimentation quantity is 498,000 m³, and the construction period is 2.5 years, so that the volume of earth work per day is :

$$498{,}000 \div (2.5 \times 300) = 664 \text{ m}^3/\text{day}$$

The material for dam construction from the second year on is,

$$3{,}843{,}000 - 498{,}000 = 3{,}345{,}000 \text{ m}^3$$

If this amount of work is performed uniformly in 9 years, annual required amount of dam construction is performed (if the amount is divided by 10 (years), totalled construction amount will become insufficient on the way). If the dam construction from the second year on is completed in 9 years, the volume of earth work per day becomes,

$$372{,}000 \div 300 = 1{,}240 \text{ m}^3/\text{day}$$

(2) Computation of Operation Capacity and the Required Number of Heavy Machines

i) Bulldozer 32t

a) Operation capacity

Volume of excavating and moving earth (V_B) :

$$V_B = \frac{60}{Cm} \times q \times E$$

q : volume of excavating and moving earth per cycle = 4.55 m³

E : operation efficiency = 0.55 (conglomeratic earth)

Cm : cycle time = $\frac{\ell}{V_1} + \frac{\ell}{V_2} + 5$ (min)

V_1 : forward movement speed = 41.6 m/min

V_2 : backward movement speed = 78.6 m/min

t : gear change time = 0.25 min

ℓ : average distance of excavating and moving earth (including stripping-off of surface soil) = 50 m

hence, $Cm = \frac{50}{41.6} + \frac{50}{78.6} + 0.25 = 2.09$ m

thus, $V_B = \frac{60}{2.09} \times 4.55 \times 0.55 = 71.8\ m^3/hr = 359.0\ m^3/day$

Volume of spreading earth (Q_1) :

$$Q_1 = \frac{W \times V \times H \times E}{P}$$

W : effective operation width = 3.8 m (Komatsu D155A)

V : operation speed = 1,700 m/hr

H : thickness of spreading earth = 0.30 m

E : operation efficiency = 0.5

P : times of spreading = 3 times

heney, $Q_1 = \frac{3.8 \times 1700 \times 0.30 \times 0.5}{3} = 323.0\ m^3/hr$

Volume of compacting earth (Q_2):

$$Q_2 = \frac{W \times V \times D \times E}{P}$$

W : width of effective compacting = 0.8 m

V : operation speed = 4,000 m/hr

D : finished thickness = 0.30m

E : operation efficiency = 0.7

P : times of compacting = 5 times

hence, $Q_2 = \frac{0.8 \times 4000 \times 0.3 \times 0.7}{5} = 134.4\ m^3/hr$

Volume of spreading and compacting earth :

$$Q = \frac{Q_1 \times Q_2}{Q_1 + Q_2} = \frac{323.0 \times 134.4}{323.0 + 134.4} = 94.9\ m^3/hr = 474.5\ m^3/day$$

b. Required number of bulldozers

Number of bulldozers required for the first year :

Earth excavation and moving : $664 \div 359.0 = 2$ bulldozers

Earth spreading and compacting : $664 \div 474.5 = 2$ bulldozers

Total 4 bulldozers

Number of bulldozers required from the second year onwards :

Spreading and compacting : $1,240 \div 474.5 = 3$ bulldozers

ii) Tractor shovel 3.2 m^3

a. Operation capacity

First year :

Volume of excavation and loading (Vt)

$$Vt = \frac{3,600}{Cm} \times q \times E \quad (m^3/hr)$$

q : volume of excavation and loading per cycle

$= 0.84 \times q_0 - 0.03 = 0.84 \times 3.2 - 0.03 = 2.66\ m^3$

E : operation efficiency = 0.5

Cm : cycle time = 46 sec

hence,

$$Vt = \frac{3,600}{46} \times 2.66 \times 0.5 = 104.1\ m^3/hr = 520.5\ m/day$$

Second year onwards

q = 2.66 m^3

E = 0.6

Cm = 46 sec

hence,

$$Vt = \frac{3,600}{46} \times 2.66 \times 0.6 = 124.9\ m^3/hr = 624.5\ m^3/day$$

b. Required number of tractor shovels,

First year : $664 \div 520.5 = 2$

Second year onwards : $1,240 \div 624.5 \doteqdot 2$

iii) Dump truck 18t

a. Operation capacity

First year :

Volume of earth carried (Vt) :

$$Vt = \frac{60}{Cm_T} \times q_T \times E_T \ (m^3/hr)$$

q_T : capacity of truck (weight of unit volume earth 1.8 t/m³, variation ratio = 1.0 are used) = 18 ÷ 1.8 = 10.0 m³

E : operation efficiency = 0.9

Cm_T : cycle time = $\frac{Cms \cdot n}{60\ Es} + T_1 + T_2 + t_1 + t_2$ (min.)

Cms : cycle time of loading machine = 46 sec (tractor shovel)

n : loading times of loading machine = $\frac{q_T}{qs}$ (times)

qs : loading volume per cycle of loading machine = 2.66 m³

hence,

$$n = \frac{10.0}{2.66} \doteqdot 4 \text{ times}$$

Es : operation efficiency of loading machine = 0.5

T_1 : cycle time of earth haulage = $\frac{\ell}{v} \times 60$ (min)

T_2 : return traveling time (min) = T_1

ℓ : one way traveling distance = 1.0 km

v : traveling speed = 15 km/hr

$$T_1 = \frac{1.0}{15} \times 60 = 4.0 \text{ min} = T_2$$

t_1 : unloading time = 1.0 min

t_2 : waiting time, etc. = 0.5 min

thus,

$$Cmt = \frac{46 \times 4}{60 \times 0.5} \times 4.0 + 4.0 + 1.0 + 0.5 = 15.63 \text{ min}$$

therefore,

$$V_T = \frac{60}{15.63} \times 10.0 \times 0.9 = 34.5 \text{ m}^3/\text{hr} = 207.0 \text{ m}^3/\text{day}$$

From second year onwards :

q_T = 10.0 m³

E_T = 0.9

Cms = 46 sec

n = 4 times

Es = 0.6

ℓ = 1.5 km

v = 15 km/hr

$$\therefore T_1 = \frac{1.5}{15} \times 60 = 6.0 \text{ min} = T_2$$

(3) Persons Necessary to Operate Heavy Machines (per day)

Table II–3–6 Persons necessary to operate heavy machines for a day (–2.5 ∿ 0 year)

Machine	Specification	Number	Kerosin (ℓ)	Worker (person)					Note
				special operator	operator	assistant	extra	total	
Buldozer	32 ^{t}	4	680	4.0		2.0	0.8	6.8	Excavation 2 Movement of earth 2
Track showvel	3.2 m^3	2	300	2.0		1.0	0.4	3.4	
Dump track	18 ^{t}	4	192		4.0			4.0	
Total			1,172	6.0	4.0	3.0	1.2	14.2	

Table II–3–7 Persons necessary to operate heavy machines for a day (0 ∿ 9 year)

Machine	Specification	Number	Kerosin (ℓ)	Worker (person)					Note
				special operator	operator	assistant	extra	total	
Buldozer	32 ^{t}	3	510	3.0		1.5	0.6	5.1	
Track showvel	3.2 m^2	2	300	2.0		1.0	0.4	3.4	
Dump track	18 ^{t}	8	384		8.0			8.0	
Total			1,194	5.0	8.0	2.5	1.0	16.5	

(4) Cost of Operating Construction Equipment (per day)

Table II–3–8 Cost of Operating Construction Equipment (–2.5 ∿ 0 year)

Machine	Number	Class of construction	Unit	Quantity	Unit cost ($US)	Amount ($US)	Note
Buldozer	32^{t} x 4	Kerosin	ℓ	1,172	0.2535	297.1	
		Grease	set	1		59.4	Kerosin x 20%
Track showvel	3.2^{m^3} x 2	Worker	person	14.2	8.0	113.6	
		consumption of tire		4	16.2	64.8	$4,600 \div 1,700^{hr}$ x 6 hr/day=16.2
Dump track	18^{t} x 4	Repair	set	1		415.3	$1,465,700 \times \frac{0.85}{10 \times 300} = 415.3$
		Total				950.2	

Table II-3-9 Cost of Operating Construction Equipment (0 ∿ 9 year)

Machine	Number	Class of construction	Unit	Quantity	Unit cost ($US)	Amount ($US)	Note
Buldozer	32^t x 3	Kerosin	ℓ	1,194	0.2535	302.7	
		Grease	set	1		60.5	
Track Showvel	3.2^{m^3} x 2	Worker	person	16.5	8.0	132.0	
		consumption of tire		8	16.2	129.6	
Dump track	18^t x 8	Repair	set	1		635.4	$(1,465,000 + 777,000) \times \frac{0.85}{10 \times 300} = 635.4$
		Total				1,260.2	

Table II-3-10 Heavy Machinery and its Transportation Cost (–2.5 year)

In $US

Item			Buldozer	Track showvel	Dumptrack	Note
Mark			Komatsu D155A–1	Komatsu D95S–2	Komatsu HD180–4	
Number of Machinery			5	3	5	
Weight (t/machine)			38.4	29.8	16.8	
Total weight (t)			192.0	89.4	84.0	
Price			142,300	121,900	77,700	F.O.B
Purchase price			711,500	365,700	388,500	F.O.B Purchase cost Total 1,465,700
Transportation Cost	Marine	Freight	36,900	17,200	16,200	Total weight x 192.44
		Insurance	5,800	3,000	3,100	(Freight + Purchase cost) x 0.7775%
		Total	42,700	20,200	19,300	
	Inland	Freight	24,800	11,500	10,800	Total weight x 129
		Insurance	37,700	19,300	20,400	Marine transportation cost + Purchase cost x 5%
		Total	62,500	30,800	31,200	
	Transportation Cost Total		105,200	51,000	50,500	Transportation cost total 206,700
Total by classes of machine			816,700	416,700	439,000	
Sum Total			1,672,400			

Table II-3-11 Heavy Machinery and its Transportation Cost (0 year)

In $US

Item			Buldozer	Track showel	Dumptrack	Note
Mark					Komatsu HD180-4	
Number of Machinery					5	
Weight (t/machine)					16.8	
Total weight					84.0	
Price					77,700	
Purchase price					388,500	F.O.B
Transportation Cost	Marine	Freight			16,200	
		Insurance			3,100	
		Total			19,300	
	Inland	Freight			10,800	
		Import tax etc.			20,400	
		Total			31,200	
	Transportation Cost Total				50,500	
Total by classes of machine					439,000	
Sum Total				439,000		

(5) Dranage Equipment Construction Cost

Table II-3-12 Drainage Equipment Construction Cost (−2.5 ~ 0 year)

Class of Construction	Specification	Unit	Quantity	Unit price ($US)	Amount ($US)	Note
Water way in Mill side	Reinforced concrete open canal $1.0^{W} x 1.0^{H}$	m	3,000	170	510,000	
Dramage Equipment in Mill Plant	Reinforced concrete open canal $1.2^{W} x 1.5^{H}$	m	2,400	360	864,000	South 1,000 m North 1,400 m
	Reinforced concrete box culvat $1.2^{W} x 1.5^{H}$	m	600	470	282,000	South 300 m North 300 m
Emergency Dramage Equipment	Tunnel water way in rock $20^{W} x 1.5^{H}$	m	900	50	45,000	South 450 m North 450 m
Total					1,701,000	

Table II-3-13 Drainage Equipment Construction Cost (0 ~ 9 year)

Class of Construction	Specification	Unit	Quantity	Unit price ($US)	Amount ($US)	Note
Drainage Equipment in Hill Plant	Putting on lid reinforced concrete 1.7^{W}	m	2,400	130	312,000	
Emergency Drainage Equipment	Tunnel water way $2.0^{W} x 1.5^{H}$	m	1,800	50	90,000	South 100 x 9 = 900 North 100 x 9 = 900
Total					402,000	

(6) Tailing Transportation Equipment Construction Cost

The construction cost occurres only in construction stage (−2.5 year ~ 0 year). The dranage canal frome mill plant to head tank, and cycron is included in mineral concentration equipment cost.

Table II-3-14 Tailing Transportation Equipment Construction Cost

Class of Construction	Specification	Unit	Quantity	Unit price ($US)	Amount ($US)	Note
Tailing Transportation open water way	Concrete open water way $0.45^{W} x 0.6^{H}$	m	4,000	90	360,000	
Tailing Transportation Equipment	Head Tank		22		226,000	
	Piping $530^{\phi} x 200^{M} x 2^{Une}$					
	Cyclone underflow pump		3			
	Cyclone $20''^{\phi}$ x 2 pieces		3			
Total					586,000	

t_1 = 1.0 min

t_2 = 0.5 min

thus, $Cm_T = \frac{46 \times 4}{60 \times 0.6} + 6.0 + 6.0 + 1.0 + 1.5 = 18.61$ min

therefore, $V_T = \frac{60}{18.61} \times 10.0 \times 0.9 = 29.0\ m^3/hr = 174.0\ m^3/day$

b. Required number of dump trucks

First year : 664 ÷ 207.0 = 4 trucks

Second year onwards : 1,240 ÷ 174.0 = 8 trucks

(7) Tailing Dam Construction Cost

Construction period is 11.5 years (dam construction period for first year sediments 2.5 years + dam construction period for 2nd ~10th year sediments 9 years), and dam construction cost is summed up as dam raising cost in operation costs.

The construction costs are shown in the following table, but the construction cost of water recovery equipment is not included in this table.

Table II-3-15 Construction Cost of Tailing Dam

in US$

Processing ore (t/day)	10,000 t	9,000 t	8,000 t
Years	2.5	2.5	2.5
Heavy Machine Equip.	1,672,000	1,283,000	1,585,000
Construction of Dam	713,000	518,000	667,000
Drainage Equip. Construction	1,701,000	1,629,000	1,593,000
Tailing Transportation Equipment	586,000	533,000	574,000
Road Construction	60,000	60,000	60,000
Total	4,732	4,023	4,479,000

CHAPTER 4. Utility Equipment Plan

4-1 Water Supply System Plan

In planning the new concentration plant, security of water for concentration must be recokoned as one of the most important subjects, but as the site for the plant is in a dry land area with the average rainfall of the level of 500 ~ 600 mm, from where rivers rise, serious difficulty is anticipated in securing the planned quantity of water supply of 32,000 m^3/day.

In the current state of operation, this mine is suffering from water shortage to some extent, but as there is possibility of securing supplementary water supply from bored wells under way with effort in Centenario area, the quantity of water collection and the quantity of water used will be kept in the present state until the new plant starts its operation. For securing the planned quantity of water supply after starting the operation of the new plant, the drinking water system and the mining water supply system may be kept in the present state, but water supply for concentration, which may be insufficient, has been investigated as follows.

4-1-1 Water Collecting Equipment Reviewing

There are mine water sources at present, Sauta, Tomas-Ventilla, dry season, we will estimate it possible to collect 70% of the quantity in average, then the increase of collected water quantity is :

$5\ m^3/min \times 60\ min/hr \times 24\ hr/day \times 0.7 \fallingdotseq 5{,}000\ m^3/day$

Since the present water-conveyance equipment will be maintained for supplying drinking water for Catavi area, the following new equipment will be required to send water of 5,000 m^3/day:

① Debris barrier to be built somewhere upstream of Sauta dam

② Water-conveyance pipe between Sauta dam and pump station

③ Pump 380 kw x 1, Reserve pump 1

④ Piping between the pump station and the new plant 250 mmϕ x 11,000 m(L)

2) Ventilla

We heard that a concrete ditch had been buried here to take water in Patiño days, but as it has been clogged naturally and become impossible to use, a reservoir of water has been made by excavating the river bed at a position about 100 m upstream of the ditch position to collect surface water and river bed water. Rio Ventilla has comparatively a large basin (87.21 km^2) in this district, so that a large quantity of underground water is expected to exist apart from the surface water.

According to the first survey report, the measured quantity of flow was 2,721.6 m^3/day while the quantity of intake water was 1,637.9 m^3/day, therefore, if the apparent values alone are simply compared, the difference between the two quantities, i.e., 1,083.7 m^3/day is thought to be a possible increase in the quantity of collected water, but since the above-mentioned reservoir occupies only a small part of the river bed, the quantity of water collected now may be negligibly small compared with the quantity of underground water flowing away at present.

The quality of water at the present intake point is acid water of the level of pH=4 and yet contaminated water including much suspended iron compounds. This fact is thought to be one of the factors which clogged the water collecting ditch in Patiño days, so the point of collecting water should be transferred upstream.

At a point about 2 km upstream of the present water intake point, spring water is gushing out, which is clear water and its quality is pH ≧ 7. Accordingly, we want to set a new water intake point at the junction of Rio Ventilla and Rio Catiri.

According to the records of rainfall observation, average annual rainfall is about 540 mm, but 83% of the rainfall is concentrated to the rainy season of only four months from December to March. The topographic feature of the area is steep, bold and rocky surface with a large co-efficient of outflow, so that most of the rain water is supposed to flow away immediately after rainfall, and, when the quantity of evaporation in the dry season that begins in April is taken into account, it is anticipated that underground water can hardly be reserved and surface water flow will not exist, but actually, surface water existed in the river even at the time of survey (July).

To explain this phenomenon, we suggest the following hypotheses :

① Whether or not the rock formation that seems to scarcely reserve underground water includes much sedimentary rocks such as slate, sandstone, etc., a certain amount of clay is sandwitched between layers and yet fissures have developed, so that the formation may reserve underground water.

② Whether or not the Llallagua formation has features to reserve underground water, e.g. a certain degree of fissures have developed in the formation, when the fact that much mine water exists is taken into account.

③ As underground water is reserved at a substantial depth, and apart from the water, underground water which stays near the ground surface are both frozen when oozing out, therefore, whether or not such underground water is hardly affected by evaporation. In fact we observed ice sticking to rock in many places in the shade of the mountainuous area. As the dry season is winter, relation between oozing out quantity and evaporated quantity is not

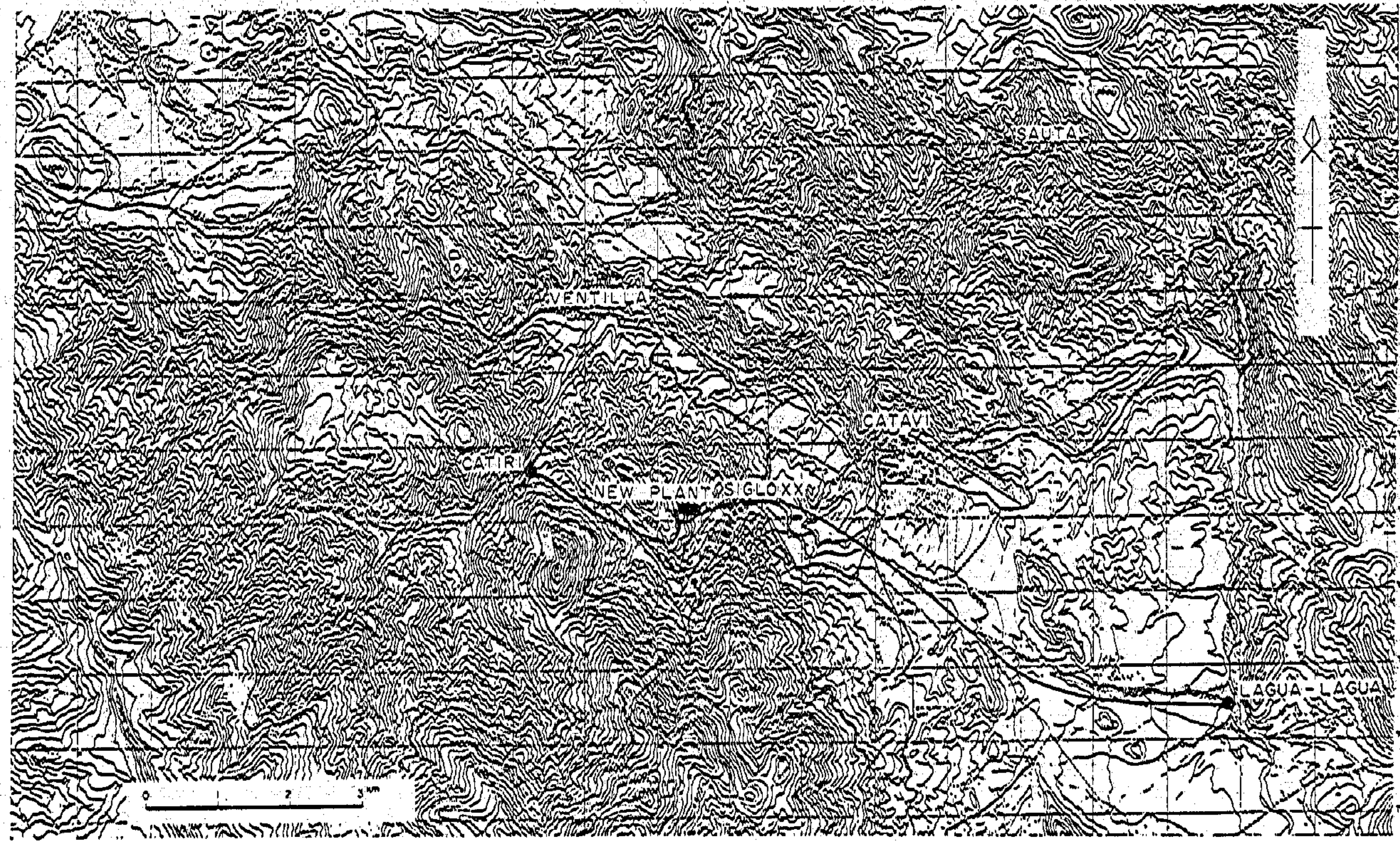

Fig. II-4-1 PIPING ROUTE OF WATER SUPPLY

simple and they are not always linearly proportionate.

By supposing so, the fact that a certain amount of surface water exists in the river even in the dry season, in which rainfall can hardly be expected, can be explained to some extent.

According to Journal of Hydrology, Vol. 53 No. 3/4 (pp. 213 ~ 227, October, 1981), the mean ground water rechargeability in the drainage area in a semi-dry area with an annual rainfall of about 250 mm in the south-western part of Idaho State, USA is estimated to be about 4.6×10^{-4} cm/min (coefficient of permeability, 0.023 cm/min, average hydraulic gradient 0.02 cm/cm). Although it may be unreasonable to apply this value directly to this area where geological conditions, rainfall conditions, etc., are quite different, if it is assumed that the water rechargeability of that degree exists, as the catchment area of Rio Ventilla is 87.2 km^2 of which 44.51 km^2 is occupied by Catiri water system described in article 3), the catchment area of the junction of Rio Ventilla and Rio Catiri becomes 42.7 km^2, and if it is further assumed that rain water is recharged continuously during the rainy season of four months from December to March, the quantity of underground water reservation is determined as follows :

4.6×10^{-4} cm/min x 1,440 min/day x 30 days/month x 4 months x $42.7 \times 10^{10} cm^2$
$= 3.394 \times 10^{13} cm^3$
$= 33{,}941{,}000 m^3$

However, it is impossible to collect the entire quantity of this reserved water, but the effective porosity of underground fissures through which water can actually pass matters, and if the porosity is assumed to be about 5%, the quantity of collectable water becomes,

$33{,}941{,}000 \times 0.05 = 1{,}697{,}000 m^3$

If the quantity of collected water is assumed to be 12,000 m^3/day,

$1{,}697{,}000 \div 12{,}000 = 140$ days = 4.6 months

accordingly, underground water reserve corresponding to the quantity of collected water for 4.6 months can be secured.

During four months in the rainy season, there is abundant surface water and water can be collected through a water collecting pipe installed on the river bed, and during four months in the first half of the dry season, underground water of the river bed and near the bed and a certain amount of surface water will be able to meet the required quantity, so it is only for the four months of the latter half of the dry season during which the above-mentioned underground water reserve must be collected. Accordingly, continuous water collection will be possible.

Equipment required for taking in water of 12,000 m^3/day are as follows :

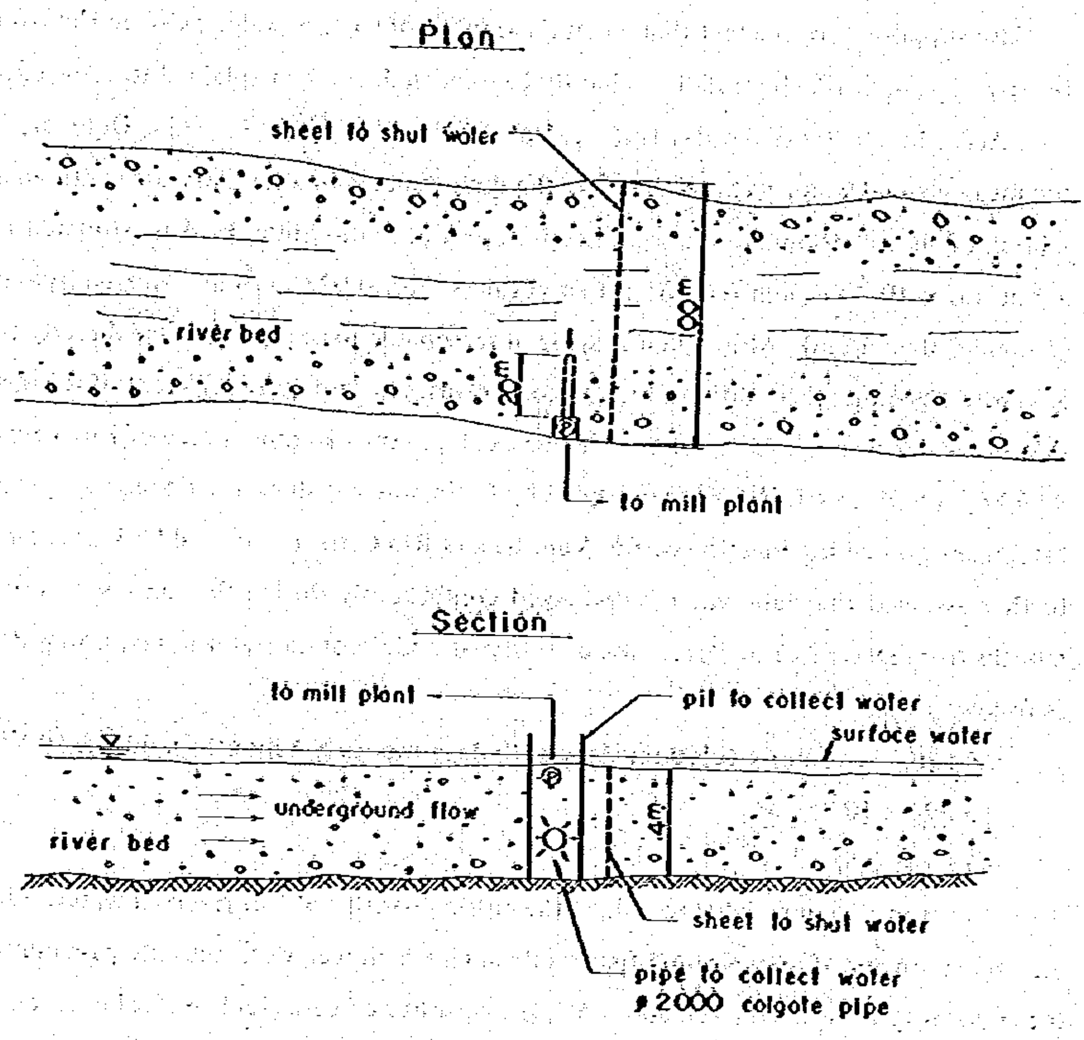

Fig. II - 4 - 2 Idealized Irustration of Taking Water

As the sheet for shutting off water does not require strength and its main purpose is to reduce the coefficient of permeability of the river bed, water leakage may be allowed to some extent.

1 Equipment to take water : Collecting pipe, 2,000 mmϕ x 20 m (L)
corrugated pipe with holes.

Sheet to shut water : h = 4 m, ℓ = 100 m

Catch pit : 200 m^3 x 1 unit

2 Water conveyance equipment :

Pump : 350 kW x 2, Reserve pump 1

Piping : 350 mmϕ x 4,500 m (L)

An underground dam type will be adopted as the method of collecting water, which can be illustrated typically by Fig. II–4–1.

3) Catiri

Here, water is collected directly from the concrete arch dam of a capacity of 2,500,000 m^3 through a 6" pipe, and in addition, a water collecting dam has been built at a point about 1.5 km upstream of the arch dam to conduct water through an 8" pipe from there. At both intake points, water is reserved once in the intake tanks of pump stations before being pumped up to be supplied as industrial water and drinking water. A noteworthy point about Catiri is the fact that in spite of collecting the above-mentioned quantity, the dam was in a fully filled state (at the time of survey on July 5, 1982), which suggests that this place can also be reckoned as an object of water collection.

The quantity of water intake at present is about 1 m^3/min including water for concentration, domestic use water, etc., and as the dam is equilibrated in the fully filled state while the above quantity of water is taken, we will assume that 85% of the dam capacity can be taken, then the increase of collected water quantity is :

$2,500,000\ m^3 \times 0.85 = 2,125,000\ m^3$

If the number of days in which water can be taken is calculated from the above quantity.

The number of days water can be taken = 2,125,000 ÷ 20,000 = 110.75 days

As the number of working days of the new mill plant is 25 days/month, the above quantity of water can meet the requirement of water for approximately 4.4 months.

On the other hand, about eight months from April to November belong to the dry season here, and especially in four months from April to August, almost no rainfall can be expected. It is supposed that the peak time of the dry season of the river upstream of the dam naturally lags a little from the peak time of the dry season of the district because of the outflow time of rainfall, and as surface water exists at the time of July, a period in which inflow to the dam cannot be expected is thought to be about four months from August to November. Accordingly, these four months can be got through sufficiently with the reserve water of the dam.

To collect the quantity of water of 20,000 m^3/day at Catiri, the following equipments will be required :

① Water pipe from Catiri dam to pump station.

② Pump : 5 m^3/min x 270 m(H) x 370 kW x 3, Reserve 1

③ Piping from pump station to new mill plant : 450 mmϕ x 3,000 m(L)

The existing water conveyance equipments cannot be diverted for this purpose in relation to their specifications, but as they are used for supplying drinking water, they will be left unchanged.

4) Centenario and Lagua-Lagua

The alluvial fan developed in the east of Uncia town at the foot of Juan del Valle mountain makes us expect intuitively the existence of underground water, but now, water is pumped up from only one deep well, so that it can be said without exaggeration that this place can be regarded a so-called undeveloped virgin soil as water source.

This alluvial fan is said to be an alluvium bed formed by eroding and sedimenting Juan del Valle peak, etc., in the glacial period and is distributed over a range of 7 km from east to south, 5 km from north to south and the maximum depth of 90 m. The geology mainly consists of coarse gravel layers, and when observed through the deep well at Centenario, the layers have been compacted hard, and the well itself is as bored having no timbering.

The depth of the well is 45 m and the water level in the well is 5 m below the ground surface. On the other hand, the erosion of glaciers also carried and sedimented the mineralized zones of Juan del Valle peak, this alluvial fan forms an alluvial deposit of tin and the upper layer has become the mining place of Veneros.

COMIBOL has also noticed this land form and is working on a plan of boring three wells each having a section of 2 m x 2 m and a depth of 30 m. At the time of the last survey, one of them was being bored and hit the underground water level at 13 m below the ground surface and the quantity of water springing out was confirmed to be 2 ℓ/sec. According to the plan, the wells are said to be bored on both sides of Rio Temporal, and apart from the plan, a trench with a section of 5 m x 5 m and a length of about 200 m has been ditched at a position downstream of the well and the water exists about 3 m below the ground surface level, so the water can surely be found by boring a well anywhere around the place. From the wells under boring now, the quantity of water equal to that of the well in operation is supposed to be pumped up at the stage when they are bored to the planned depth of 30 m.

On the other hand, we have heard that COMIBOL authorities have particular concern for Rio Lagua-Lagua and are planning to supply water from the river to Catavi area.

In the flow quantity survey in August, 1981, the quantity of surface water of 11.5 ℓ/sec was confirmed and a pH value of 7.8 was measured. Although the quantity of surface water is not so large, its wide basin area of 150 km² is worthy of notice.

The rainfall in this basin entirely flows into Tranque except loss due to evaporation, etc., and is utilized for hydraulic power generation, but as the water is acid water (pH=3), the water is not used in any other way. Tranque is an artificial lake with a capacity of 38,000,000 m³, but as acid water of concentration tailings, No. 2 Kenko Lake effluent, etc., has flowed into the Lake for many years, the lake has become reservoir of chronic acid water, and it is regrettable that the water of the lake has not been utilized efficiently in spite of the fact that this lake is a sole abundant source which can be confirmed visually in this area. However, the river through which flow the above-mentioned concentration tailings, etc., is Rio Andavilque alone, and the water quality of Rio Lagua-Lagua and Rio Santa is almost neutral, so that everyone will agree that is would be far more advantageous to take the neutral water before flowing into the lake than to neutralize acid water before collection.

Moreover, Centenario area alone, where the quantity of water reserve has been confirmed to a certain extent, has a basin area of 60 km², and the land form of the basin forms a state that the whole quantity of underground water flows into Rio Lagua-Lagua, so that Lagua-Lagua seems to have a considerably large scale of underground water in existence. When compared Rio Lagua-Lagua with other rivers in the neighborhood, i.e., Rio Ventilla and Rio Sauta, Rio Lagua-Lagua has a sand river bed formed of materials including much sand, while other rivers have gravel river beds mainly of boulders. Perhaps, this is because Rio Lagua-Lagua has considerably a large basin area and a long river length, and materials flowed by the river stream have been sized down to a larger degree as they flow down. Although these rivers flow into Lago Tranque independently at present, Tranque is essentially a lake made by damming Rio Lague-Lagua and Rio Ventilla and Rio Sauta were originally tributaries of Rio Lagua-Lagua, so that the above-mentioned fact is quite natural. Other rivers therefore have steeper gradients than that of Rio Lagua-Lagua and also have larger quantities of surface water. Accordingly, although it may sound paradoxical, as Rio Lagua-Lagua has a wide basin area, it is anticipated that the river has a considerably high underground water recharging ability and the seasonal fluctuation of its underground water quantity is far smaller than those of other neighboring rivers. Also for this reason, the collection of underground water here is sufficiently worthy of investigation.

About the possible quantity of water collection, assumptions are made similar to the case of Rio Ventilla.

4.6×10^{-4} cm/min x 1,440 min/day x 30 days/month x 4 months

x 150 x 10 cm^2 x 0.05 = 5.9616×10^{12} cm^3 = 5,961,600 m^3

If the quantity of water collection is assumed to be 12,000 m^3/day,

5,961,600 m^3 ÷ 12,000 m^3/day = 496.8 days

Accordingly, there will hardly be any problem even if water is collected all through the year.

To take in water, a catch pit will be built on the river bed of Rio Lagua-Lagua, water collected on the left bank and pumped up from there to the new mill plant. Accordingly, the following equipment will be required :

Catch pit : 200 mmφ x 20 mL corrugated pipe

Water collecting pit : 200 m^3 x 1

Water pump : 4.5 m^3/min x 340 mH x 420 kW x 2, Reserve pump 1

Water conveyance piping : 350 mmφ x 8,500 mL

4-1-2 Determination of Water-Intake Point

As described in 4-1-1, Sauta requires comparatively a large amount of equipment investment compared with the small increase of water collection of only 5,000 m^3/day. As Ventilla is comparatively near to the new mill plant, construction cost may not be so large, but it has a weak point in securing its ability to secure the sufficient quantity of water. Accordingly, Sauta and Ventilla are omitted from investigation objects, and two points, Catiri and Lagua-Lagua are determined as water-intake points.

As Catiri has a reserve dam and the quantity of water can be confirmed, and it is also at the nearest place to the new plant mill, construction cost will be comparatively low, so that water should be collected here as much as possible. Thus, the quantity of water collection is determined to be 20,000 m^3/day.

Lagua-Lagua has somewhat unsteady factors in securing water quantity and is also far from the new mill plant, the quantity of water collection is limited as small as possible, i.e., 12,000 m^3/day. Although there may be such problem, underground water actually exists in Centenario area, so that the remaining problem is the quantity of existing water.

If it is assumed that the water reserving gravel layer here has an area of about 30 km^2, average depth of 10 m and effective porosity of 15%, the quantity of water that can be collected is estimated to be:

30×10^6 m^2 x 10 m x 0.15 = 45,000,000 m^3

then,

$45,000,000 \, m^3 \div 12,000 \, m^3/day = 3,750$ days

This means that the existence of the quantity of water reservation corresponding to the quantity of collection for ten years is expected. This expectation can be several times surer than the uncertain computation based on the assumption of underground water rechargeability of Lagua-Lagua described in 4–1–1. As Centenario is an area where such enormous possibility is hidden, we think it indispensable to confirm the quantity of existing underground water. However, the purpose of this survey was not the survey of underground water, we only describe possibility in this report and want to suggest another survey concerning underground water at the earliest possible occasion.

4–1–3 Measures to Get through Water-Shortage Period.

It was described in 4–1–1, 2) that about 83% of the average annual rainfall of 540 mm here is concentrated to the rainy season of four months. In the eight-month dry season, there may be a certain amount of rainfall in four months including April, September, October and November (16% annual rainfall), but in four months from May to August, almost no rainfall can be expected (1% of annual rainfall). In addition to the existence of no rainfall period, it happens about once in three years that annual rainfall falls short of 500 mm, and sometimes there is a year with annual rainfall below 400 mm. In such a year, the dry season tends to last longer, and in Catiri, it may be impossible to collect the required quantity of water. The occurrence of such a state at a place where 55% of required fresh water is to be taken will have a serious impact on production plans, so the best way is to prepare a reserve system of collecting water. From this viewpoint, we suggest to prepare the water collecting equipment described in 4–1–1, 2) in the Ventilla water system as early as possible during the operation period.

4–1–4 Operating System for Water Collecting Equipment

The operating system described so far can be summarized as follows :

The drinking water systems of Sauta and Catiri will be left unchanged.

For supplying concentration water, new equipments will be constructed in Catiri and Lagua-Lagua.

Accordingly, the following pump operators will be required (Refer to Primary Report).

Sauta system 2 x 3 shifts = 6 operators

Catiri system 3 x 3 shifts = 9 operators

Lagua-Lagua system 1 x 3 shifts = 3 operators

After the operating system has been prepared, it is desirable to change the drinking water system into a distribution system utilizing natural head by preparing a head tank at a high place or anyhow to avoid keeping pump operators in places other than the water-intake places.

4–1–5 Computation of Approximate Construction Costs

Construction costs of water collecting equipment at each water-intake point can be summarized as follows.

Table II–4–1 Cost of Construction of Water Collecting Equipment (in 1,000 US$)

	Pump	Piping	Electric Construction	Civil engineering construction	Total
Catiri			164	98	
Lagua-lagua			558	57	
Total	743	1,914	722	155	3,534

4–2 Plan for Electric Power Source

4–2–1 Basic Idea

Catavi Mine has an existing substation (132 KV / 66 KV) with a capacity of 50 MVA, among which 18 MVA is supplied to the surrounding mines, pumps, towns, houses, etc. Accordingly, there is a surplus capacity of 32 MVA at present, so that this capacity will be allotted to the power sources for the increased equipments including the new mill plant and water supply equipments, etc., accompanied by the plant.

In addition, the existing hydraulic power plant should be abolished, and the emergency power source is required only for ventilation in the mining department and for thickness in the concentration department, and the capacity of the existing diesel power generation equipment will be sufficient for the emergency power source, so that the equipment will be diverted for the emergency purposes.

4–2–2 Power Transmission to New Concentration Mill

To the afore-mentioned incoming power substation (capacity 25 MVA) to be built within the site of the new concentration mill, overhead wires of a length of transmission of 2.5 km will be newly provided from the secondary busses (three phase three wire system 66 KV) of the above-mentioned main substation.

Power transmission system & transmission voltage : 3-phase 3-wire system, 66 KV 50 Hz 1 circuit.

Transmission capacity : 30 MVA

Length of transmission : 2.5 km

Kind of supporters : Steel towers of angle steel

Number of steel towers : 11 (average span 230 m)

Transmission wire material : ACSR 160 mm^2

Aerial ground wire : Twisted wire of galvanized steel

4–2–3 Power Transmission to Pump Stations

To Catiri and Lagua-Lagua where pumps will be increased greatly as the result of the afore-mentioned water collecting equipment reviewing, to the former will be supplied power from the Siglo XX substation by changing the thickness of the existing transmission wires (length 3 km) from 45 □ to 100 □ from the Siglo XX substation to the Catiri pump station, and to the latter power will be supplied from the Victoria power substation by newly drawing 6 km long 11 KV transmission wires.

4–2–4 Rough Estimation of Construction Costs

66 KV transmission wires	total amount	105	million yen
Substation equipments	"	379	"
Substation outer structures	"	33	"

Notes: Estimating conditions are the same as those of the concentration mill.

The cost of water supply pump is included in 4–1–5.

CHAPTER 5. PLAN FOR MAINTENANCE SHOP

5–1 Policy of Managing Machine Shop

In Catavi Mine there are large organizations of the department of maintenance and the electrical section historically. The number of total workers in them is, including 370 mm in the department of maintenance and 180 mm in the electrical section, 550. Jobs in which they are engaged cover all the operations and repair work of surface and underground machines and electrical equipment, but for the operation and minor repair work of the equipment of concentration shops, there are in total 138 workers who belong to the departments of concentration of three mills, Siglo XX, Victoria and Kenko. In consequence, 688 workers in total are directly engaged in jobs concerning equipment.

The capacities of equipment, the operation of which is directly administrated by the maintenance section, are roughly shown in the following table.

Table II–5–1 Equipment that Maintenance Section Administrates

Division	Equipment	Number	Capacity	Note
Mining	Compressor		3,400 KW	
	Winch	13	850	
	Drainage pump	19	2,100	
	Fan	21	1,050	
	Water pump	5	80	
	Trolley (Direct current)	1	75	
	Total		7,555 KW	Sum total ≒ 11,000 KW
Metallurgical	In Incline, others	13	1,440	
	Water pump	23	2,020	
	Total		3,460 KW	
Electric power	Hydraulic generator	8	2,300	
	Diesel generator	5	2,000	
	Siglo XX substation		8,800	66 KV/10 KV

Functions performed by the machine shop cover a casting shop which includes pattern making and mold making, a lathe shop, a welding and plate working shop, automobile repair work, tire repair work, motor rewinding work, maintenance work of bits and drilling machines, etc., in short, all of the maintenance and repair work of various equipment in the mine.

Such a system of the machine shop covering various functions as the above is regarded to be just the same system once possessed also in Japan by most of the large scale mines. The reason for adopting such a system is that the mining industry itself started its business activities prior to other general industries, and moreover, different from so-called urban industries, the mining industry usually constructed mining towns in remote and secluded places in the mountains.

To cope with these recent basic problems, exploration for new ore deposits is continued in mines, and if a new ore deposit of considerable scale is found fortunately, the maintenance section will tentatively be very busily occupied by various developmental and starting jobs.

After the construction work of the concentration mill is finished, the maintenance section will be engaged in the maintenance of the equipment. However, compared with the old deteriorated equipment, as various measures for wear resisting, corrosion resisting, etc., are incorporated into new equipments, the quantity and frequency of repair work will normally be reduced to a large extent.

Consequently, the number of workers of the maintenance section naturally will become too large, so some measures will be required to balance the income and expenditure of the maintenance section itself. In short it is suggested as countermeasures to get rid of entrusting the mine only with everything of the section and to develop some outside income sources, or to reduce expenditures by reducing the number of workers of the maintenance section or by transferring them to some other posts, etc.

5–2 Maintenance of Equipment after Modernization

We have suggested construction of a large concentration mill with a processing capacity of 10,000 t/day in this modernization plan.

In this article, we want to describe the way of developing PM (production maintenance) for reference of developing the maintenance of equipment by supposing Catavi Mine after completing the new concentration mill.

(1) Adoption of PM system

The purpose of PM is to carry out the maintenance of equipment in the most appropriate way.

The purpose of production is to increase earnings and maintenance prevents unexpected suspension of production caused by equipment trouble.

Of course production cannot be performed only by maintenance but can be performed satisfactorily by close cooperation of various sections including engineering, maintenance, operation, management, etc.

PM can at least serve to harmonize maintenance and operation. Accordingly, maintenance work after modernization of the plant will be performed most efficiently by PM system. PM aims at the increase of earnings by maintenance, but a very careful and rational method is required to attain the aim. However, such a perfect method has not yet been accomplished, although the efforts to make PM approach completeness are always continued.

The way PM is progressing in Japan will be described as follows.

(2) Preparation for PM

a. The current status of equipment maintenance is checked carefully. The reference point for measuring future progress is determined.

b. Important equipments are selected

Equipment which can increase earnings most effectively by apply PM are selected.

c. Important points are determined

Highly effective points of each important equipment in carrying out PM are determined.

d. The most economical repair limit is determined

The most economical repair limit of each machine is determined by comparing the effect of recovering the functions of the machine when it is deteriorated with the cost required for the repair.

e. Consent is obtained from related persons

The current status of maintenance and improvement expectation are explained to them to obtain their consent.

f. Organization is prepared

The most appropriate organization for obtaining the effect of PM is prepared and persons are arranged.

g. Rules and procedures are established

The most appropriate rules and procedures for obtaining the effect of PM are established.

(3) Carrying out PM

a. Equipment which has reached its repair limit is found

Equipments are inspected periodically not to miss the equipment which have reached their repair limit.

b. Proper repair is carried out

Repair which meets requirements most economically is carried out.

c. Appropriate improvement is undertaken

Appropriate improvement is undertaken to increase profit by saving maintenance cost and preventing the deterioration of equipments.

d. Proper renewal is carried out

Proper renewal is carried out by examining the change of conditions of repair limits resulting from the deterioration of the equipment.

e. Effects are checked

The results of carrying out the PM are analyzed and investigated to check whether the expected profit has been obtained or not and undesirable points are corrected if there are any.

Profit which can be expected by carrying out PM comes from earnings obtained by production. The profit cannot be gained without production. Even if PM is carried out very well so that earnings may be increased, the valuable PM will give no fruit if there are defects in operation which cancel the effect of PM. Accordingly, proper control of operation must be carried out concurrently with PM.

The concept of PM, requires preparation before carrying out PM things to be done under the PM system have been listed above followed by brief explanation.

In Japan, PM system has been adopted in the procedures briefly explained above and great effect has been obtained, so that it is desirable to adopt this system referring to the Japanese example.

(4) Proposal

Essentially, the mining equipment has been developed by the combination of various industries. In other words, they can be looked upon as a department store of technology. Accordingly, mining facilities cannot be improved without the improvement of the level of all the engineering fields – which cover mechanical, electrical, civil engineering, building, applied chemistry, metallurgical and industrial material industries, etc. – however hard the maintenance workers may make effort.

Accordingly, excessive demands cannot be placed upon the maintenance workers of Catavi Mine, but we believe it important to adopt the advanced maintenance techniques of developed countries positively from those which can be adopted and put them into practice surely.

Especially for the maintenance of equipment of the new large concentration plant which we suggest and for operation control and maintenance techniques, it is desirable to prepare a factory of wear resisting rubber lining near the mine and to equip sufficient instrumentation devices and further to make workers master the handling and maintenance techniques of various devices as early as possible.

According to the impression of our survey committee members, most of the native workers seem to be very dexterous and have obedient character, so that they will be able to master techniques early, if appropriate training and labor control are carried out.

CHAPTER 6. ADMINISTRATION DEPARTMENT PLAN

6–1 Organization

Based on the results of research carried out last year, conceptual designs for a new mining method and new concentration mill plant together with a production program have been drawn up as a case study. The following organizational setup has been planned so as to provide an appropriate administration department for the execution of the above plan.

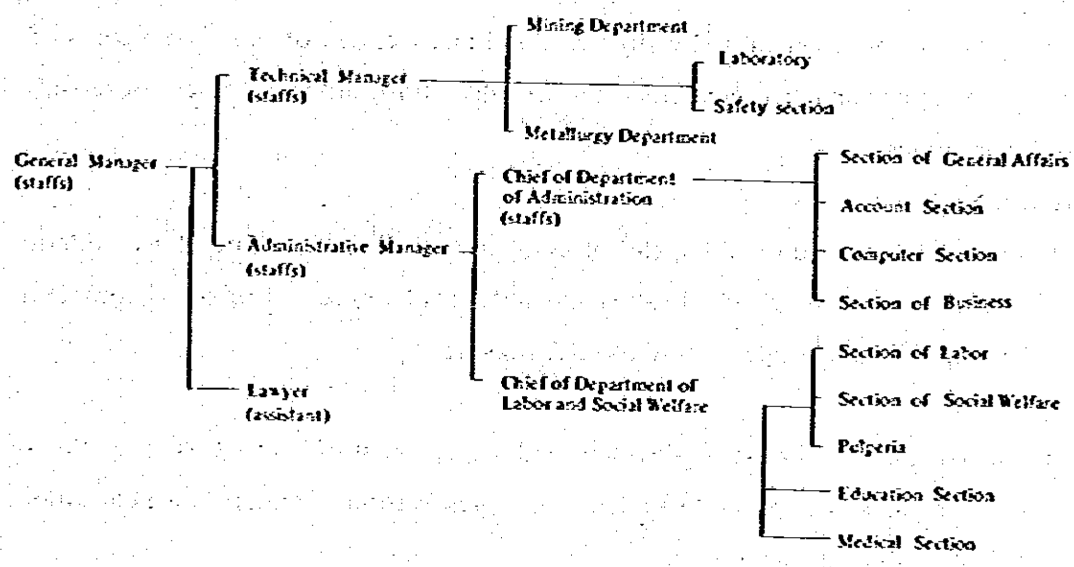

Fig. II–6-1 Organization

In the above plan, the Education and Medical Sections have been prepared taking the circumstances at the site into consideration. An Administrative Manager controls two subordinate departments and their respective sections to reinforce the administrative activities.

The tasks allotted to each section are as follows.

General Affairs Section :

Management and control of the receiving and sending of documents, wireless telecommunications, telephone exchange room, data room, club for visitors; management of mining rights, land, land surface rights, water rights, and corporeal fixed assets; cleaning and security of premises; and public relations with the community.

Accounts Section :

Financing, accounting, cost accounting, budgeting and settling accounts.

Computer Section :

Calculation of wages, receipts and payments of material supplies, costs, mined quantities and others.

Business Sections :

Management and control of materials warehouses (purchasing, receipts and payments, and storage) and printing shop; management of ore deliveries, transportation of ores, materials and supplies, and vehicle operation; and the loading of concentrates.

Labor Section :

Formalities for employment and retirement, and arrangement of personal records of employees; preparations and arrangements for wage calculation; and arrangement of records related to labor unions.

Social Welfare Section :

Management, preservation and control of company residences, welfare clubs, athletic facilities, movie theaters, and lighting, heating and water for living; uplifting social consciousness; and planning and executing hygienic countermeasures.

Education Section :

Setting up education policies; employment and management of instructors; operation of special classes; purchases, receipts and payments, and preservation of materials for education; and integrated controls over each school.

Medical Section :

Setting up health policies, and planning and executing countermeasures for health and prevention; management of purchases, receipts and payments for medical instruments and drugs; management of employment etc. of doctors and nurses; management and operation of nurse-training schools.

6–2 Personnel Planning

The personnel have been assigned as follows to ensure that the above organization fulfills its functions effectively.

Table II-6-1 List of Personel of Administration department

	Salaried Worker	Day Labour	Total
Leading members	6 per	– per	6 per
General manager, Technical and Administrative manager and staffs	18	–	18
General affair section	21	38	59
Account section	11	1	12
Computer section	8	1	9
Section of Business	26	46	72
Section of Labour	16	6	22
Section of Social Welfare	12	20	32
Pulperia	29	12	41
Education section	80	17	97
Medical section	50	18	68
Total	277 per	159 per	436 per

Note : The 28 personnel for loading concentrates in the Business Section and the 68 personnel in the Medical Section are not included in the operating costs.

6-3 Construction Costs

Because the project has been based on the fundamental concept of a case study which can be applied to operational mining, differing from conventional projects which aim to newly develop ore deposits which have not yet been producing, no construction program has been provided for the department since it is thought that existing facilities should be utilized as much as possible. However, the changes in organization described in the previous section require some improvements and repairs to the offices, so that the necessary expenses for this have been appropriated. In addition, according to the plans described above, all of the production sites will come together in Siglo XX area while the major part of the administration department will remain in Catavi area. This will require good communication between the production sites and the administration department, so that an increase of vehicles has been projected to reinforce the communication system. The outlined construction costs for the administration department are as follows. (For a breakdown of the construction costs, refer to "Details of Construction Costs of Administration Department" in the Appendix.)

(1)	Vehicles 15 x US$ 12,000 (average)	US$ 180,000
(2)	Business machines Communication instruments	US$ 175,000
(3)	Improvements to offices	US$ 100,000
	Total	US$ 455,000

CHAPTER 7. ECONOMIC FEASIBILITY OF THE PROJECT

7–1 Feasibility Study

Pre-tax incomes were in the black until 1979, but turned into the red after 1980, and the deficit for each of 1981 and 1982 reached about 15 millions dollars.

In this plan, the revenues and expenditures have been calculated on the premise that revenue should be raised by activating the merits of the plan as much as possible, obtainable from the collection of production sites into one area and the enlargement of the production scale with the modernization of various facilities. To put the premise concretely, the treatment capacity for crude ores has been vastly increased from the present 5,000 t/day up to 10,000 t/day, 9,000 t/day or 8,000 t/day.

The calculated results, as seen in Table II–7–4, show that the annual pre-tax revenue can be expected to be 6.4 million dollars with 10,000 t/day capacity, 7.6 million dollars with 9,000 t/day capacity and 5.6 millions dollar with 8,000 t/day capacity, which means some 21 million dollars increase in annual revenue. The incomes per ton of crude ore are 2.123 dollars, 2.799 dollars and 2.335 dollars for 10,000 t/day, 9,000 t/day and 8,000 t/day respectively, provided that the quotation is $6/lb. The calculation result shows that an income improvement of some 12 dollars per ton of crude ores can be realized with the fundamental concept; i.e. collection of facilities, increase of production, utilizing of tailing and modernization of production management. Details are as follows.

(1) Cut-Off Grade

As shown in Table II–7–1, the operating cost calculated in the plan indicates that the cut-off grade is 0.28% tin content in the case of 10,000 ton production when the quotation is $6/lb. An increment of the operating cost by one dollar per ton of crude ores requires a 0.01% lifting of the cut-off grade.

(2) Cut-Off Quotation

Based on the ore grade and the operating cost in the plan, the cut-off quotation is estimated to be $5.29/lb in the case of 10,000 ton production. An increment of the operating cost by one dollar per ton of crude ores equals a rise of the quotation by 0.25 to 0.37 dollars per pound.

(3) Construction Costs and Depreciation Expenses

A 10% increase of the construction cost results in a 0.437 dollars rise in the case of 10,000 ton production in depreciation expenses per ton of crude ore. This estimate is based on all funds being raised by loans and the capital being fully repaid within 10 years. When the operating cost for six months is borrowed as working capital, or one percent increase of the interest necessitates a 0.287 dollars rise in the depreciation cost per ton of crude ore.

7–2 Discount Cash Flow (DCF) Calculation

Based on the Calculation of Revenues and Expenditures shown in Table II–7–4, the DCF has been calculated at $6.0/lb quotation, 6% interest rate, and 35% total tax rate including 30% corporate tax and 5% other taxes which were not estimated in the Calculation of Revenues and Expenditures. The calculated results are listed in Table II–7–1, 2, 3, and the Internal Rate of Return are 10.50% for 10,000 t/day treatment, 10.79% for 9,000 t/day treatment and 9.95% for 8,000 t/day treatment; each of these is around 10%. As a result of a sensitivity analysis, a 10% income increase at 10,000 t/day treatment elevates the Internal Rate of Return from 10.50% to 13.25%. Increases of the operating and construction costs both lower Internal Ratio of Return; for 10,000 t/day treatment, a 10% increase in the operating cost reduces the Internal Ratio of Return by 1.6%, down to 9.31%. As shown above, such factors as a rise of quotation and an upturn in grade improve the Internal Ratio of Return remarkably. This represents the general characteristic of the mining industry, like the proverbial saying: "First if the ore situation, second is the market conditions, and fifth is technology, with nothing in third and fourth place."

7–3 Calculation of Revenues and Expenditures

7–3–1 Calculation of Revenues

The calculation of revenues has been based on the whole amount of high-grade concentrates produced under the project being sold to the tin refinery at Vinto, Oruro City, which belongs to the Empresa Nacional de Fundiciones under the present terms of sale. The entire amount of low-grade concentrates is sold to the volatilization plant that is now under construction at Machacamarca by the Corporación Minera de Bolivia, under the intracompany trade conditions which are being applied to a similar volatilization plant at Palca. The selling expenses are estimated also under conditions corresponding to the above, and the quotation is regarded as $6/lb. Key results of the calculation of the amount of revenue are shown as follows, and details of the calculation are explained later.

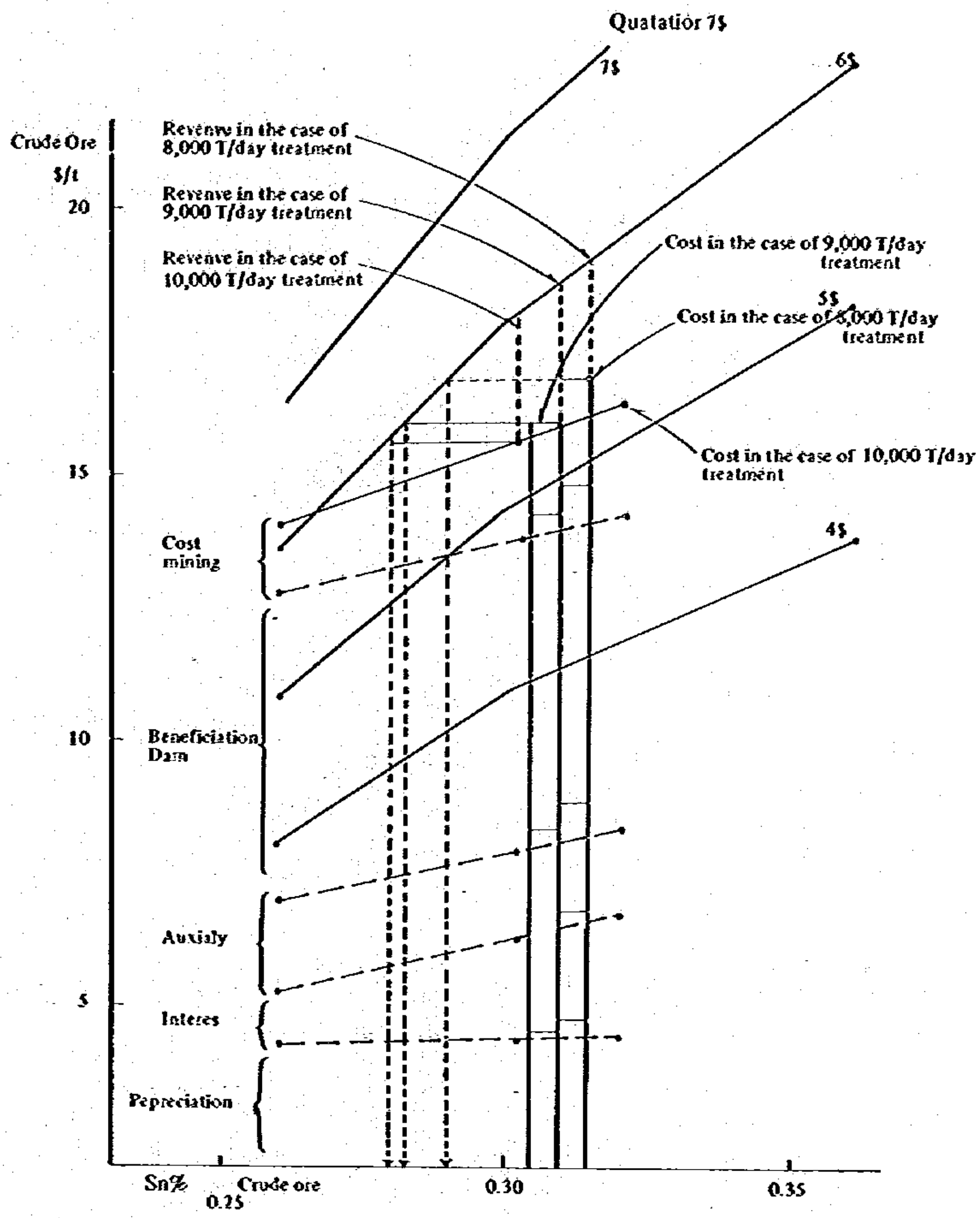

Fig. II–7–1 Net Income for Ton of Crude Ore by Grades Quotation and Cost

Table II–7–1 Profit/Loss Forecast and DCF Calculation
(10,000 T/Day)

1,000 US$

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	Accum.
Processed Ore (1000T)	0	0	0	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	30,000
Sn-Conc H. (T)	0	0	0	10,557	10,557	10,557	10,557	10,557	10,557	10,557	8,310	8,310	8,310	98,830
Sn-Conc L. (T)	0	0	0	11,700	11,700	11,700	11,700	11,700	11,700	11,700	13,950	13,950	13,950	123,750
Net Revenue	0	0	0	58,431	58,431	58,431	58,431	58,431	58,431	58,431	41,007	41,007	41,007	532,040
Operating Cost	0	0	0	28,804	28,804	28,804	28,804	28,804	28,804	28,804	26,098	26,098	26,098	279,922
Operating Profit	0	0	0	29,627	29,627	29,627	29,627	29,627	29,627	29,627	14,909	14,909	14,909	252,119
(–) Depreciation	0	0	0	12,478	12,478	12,478	12,478	12,478	12,478	12,478	12,478	12,478	12,478	124,775
(–) Interest (6.00%)	0	3,075	5,375	7,487	7,225	6,089	4,909	3,683	2,410	1,086	0	0	0	32,889
Profit Before Tax	0	0	0	9,663	9,925	11,060	12,240	13,466	14,740	16,064	2,432	2,432	2,432	94,454
(–) Income Tax (35.00%)	0	0	0	3,382	3,474	3,871	4,284	4,713	5,159	5,622	851	851	851	33,059
Profit After Tax	0	0	0	6,281	6,451	7,187	7,956	8,753	9,581	10,441	1,581	1,581	1,581	61,395
(–) Initial Investment	51,258	35,251	29,816	0	0	0	0	0	0	0	0	0	0	116,325
(–) Working Capital Change	0	0	0	14,402	0	0	0	0	0	0	–1,353	0	–13,049	0
(+) Primary Bank Loans	51,258	38,326	35,191	0	0	0	0	0	0	0	0	0	0	124,776
(–) Loan Repayment	0	0	0	4,357	18,929	19,667	20,434	21,231	22,059	18,100	0	0	0	124,776
Surplus	0	0	0							4,819	15,411	14,058	27,107	61,395
Debt Outstanding	51,258	89,584	124,776	120,419	101,490	81,823	61,390	40,159	18,100	0	0	0	0	0
Before Interest														
Rate of Return 10.50%														
Net Cash Flow	–51,258	–35,251	–29,816	11,843	26,154	25,756	25,343	24,914	24,468	24,005	15,411	14,058	27,107	102,735
Discounted Cash Flow	–51,258	–31,901	–24,418	8,777	17,541	15,633	13,920	12,384	11,007	9,772	5,677	4,687	8,178	0

Table II-7-2 Profit/Loss Forecast and DCF Calculation
(9,000 T/Day)

Unit: 1,000 U.S.$

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	Accum.
Processed Ore (1000 T)	0	0	0	2,700	2,700	2,700	2,700	2,700	2,700	2,700	2,700	2,700	2,700	27,000
Sn-Conc H. (T)	0	0	0	9,174	9,174	9,174	9,174	9,174	9,174	9,174	9,174	9,174	9,174	91,740
Sn-Conc L. (T)	0	0	0	10,170	10,170	10,170	10,170	10,170	10,170	10,170	10,170	10,170	10,170	101,700
Net Revenue	0	0	0	50,773	50,773	50,773	50,773	50,773	50,773	50,773	50,773	50,773	50,773	507,728
(−) Operating Cost	0	0	0	25,670	25,670	25,670	25,670	25,670	25,670	25,670	25,670	25,670	25,670	256,699
Operating Profit	0	0	0	25,103	25,103	25,103	25,103	25,103	25,103	25,103	25,103	25,103	25,103	251,028
(−) Depreciation	0	0	0	11,595	11,595	11,595	11,595	11,595	11,595	11,595	11,595	11,595	11,595	115,953
(−) Interest (6.00%)	0	2,862	4,998	6,957	6,776	5,818	4,822	3,788	2,713	1,596	436	0	0	32,906
Profit Before Tax	0	0	0	6,550	6,732	7,690	8,685	9,720	10,795	11,911	13,072	13,508	13,508	102,170
(−) Income Tax (35.00%)	0	0	0	2,293	2,356	2,691	3,040	3,042	3,778	4,169	4,575	4,728	4,728	35,759
Profit After Tax	0	0	0	4,258	4,375	4,998	5,646	6,318	7,017	7,742	8,497	8,780	8,780	66,410
(−) Initial Investment	47,706	32,739	27,647	0	0	0	0	0	0	0	0	0	0	108,092
(−) Working Capital Change	0	0	0	12,835	0	0	0	0	0	0	0	0	−12,835	0
(+) Primary Bank Loans	47,706	35,601	32,645	0	0	0	0	0	0	0	0	0	0	115,953
(−) Loan Repayment	0	0	0	3,018	15,971	16,594	17,241	17,913	18,612	19,338	77,267	0	0	115,953
Surplus	0	0	0	0	0	0	0	0	0	0	12,825	20,375	33,210	66,410
Debt Outstanding	47,706	83,307	115,953	112,935	96,964	80,370	63,129	45,216	26,604	7,267	0	0	0	0
Before Interest														
Rate of Return 10.79%														
Net Cash Flow	−47,706	−32,739	−27,647	9,975	22,747	22,411	22,063	21,701	21,325	20,934	20,528	20,375	33,210	107,177
Discounted Cash Flow	−47,706	−29,551	−22,524	7,335	15,098	13,427	11,931	10,592	9,395	8,324	7,368	6,601	9,711	1

Table II-7-3 Profit/Loss Forecast and DCF Calculation
(8,000 T/Day)

1,000 U.S.$

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	Accum.
Processed Ore (1000 T)	0	0	0	2,400	2,400	2,400	2,400	2,400	2,400	2,400	2,400	2,400	2,400	24,000
Sn-Conc H. (T)	0	0	0	8,283	8,283	8,283	8,283	8,283	8,283	8,283	8,283	8,283	8,283	82,830
Sn-Conc L. (T)	0	0	0	9,183	9,183	9,183	9,162	9,183	9,183	9,183	9,183	9,183	9,183	91,830
Net Revenue	0	0	0	45,842	45,842	45,842	45,842	45,842	45,842	45,842	45,842	45,842	45,842	458,423
(–) Operation Cost	0	0	0	23,744	23,744	23,744	23,744	23,744	23,744	23,744	23,744	23,744	23,744	237,440
Operating Profit	0	0	0	22,098	22,098	22,098	22,098	22,098	22,098	22,098	22,098	22,098	22,098	220,983
(–) Depreciation	0	0	0	10,914	10,914	10,914	10,914	10,914	10,914	10,914	10,914	10,914	10,914	109,140
(–) Interest (6.00%)	0	2,692	4,704	6,548	6,425	5,585	4,711	3,804	2,861	1,882	864	0	0	32,682
Profit Before Tax	0	0	0	4,636	4,759	5,600	6,473	7,380	8,323	9,302	10,320	11,184	11,184	79,162
(–) Income Tax (35.00%)	0	0	0	1,623	1,666	1,960	2,266	2,583	2,913	3,256	3,612	3,915	3,915	27,707
Profit After Tax	0	0	0	3,013	3,093	3,640	4,207	4,797	5,410	6,046	6,708	7,270	7,270	51,455
(–) Initial Investment	44,870	30,841	26,033	0	0	0	0	0	0	0	0	0	0	
(–) Working Capital Change	0	0	0	11,872	0	0	0	0	0	0	0	0	–11,872	0
(+) Primary Bank Loan	44,870	33,533	30,737	0	0	0	0	0	0	0	0	0	0	109,140
(–) Loan Repayment	0	0	0	2,055	14,008	14,554	15,121	15,711	16,324	16,961	14,407	0	0	109,140
Surplus	0	0	0	0	0	0	0	0	0	0	3,215	18,184	30,056	51,455
Debt Outstanding	44,870	78,403	109,140	107,085	93,077	78,524	63,402	47,691	31,367	14,407	0	0	0	0
Before Interest														
Rated Return 9.95%														
Net Cash Flow	–44,870	–30,841	–26,033	8,604	20,433	20,138	19,833	19,515	19,185	18,843	18,486	18,184	30,056	91,533
Discounted Cash Flow	–44,870	–28,050	–21,534	6,473	13,981	12,532	11,225	10,006	8,982	8,023	7,159	6,405	9,628	0

		(\$1,000/day)	(\$1,000/year)	(\$/t of crude ore)
(A)	For 10,000 t/day (1st–7th Year)	194.8	58,443	19.48
	" (8th–10th Year)	136.7	41,007	13.67
(B)	For 9,000 t/day (10th Year)	169.2	50,779	18.81
(C)	For 8,000 t/day (10th Year)	152.8	45,842	19.10

As a reference taking 10,000 t/day treatment as an example, changes in revenues per ton of crude ore by grades, caused by a \$1/lb change in the quotation, are as follows.

(Grade of crude ore)	0.34	0.32	0.30	0.28	0.26
(Change in dollars)	4.13	3.82	3.52	3.13	2.77

Calculation of Revenue (A)

Example A : 10,000 t/day treatment under new operating system

First Term (7 years after operation start) : Income per day

1. Lot : Catavi tin concentrates (high-grade)

(1) Weight : 1,034 bags, wet wt. 36.2 t, moisture 2.8%, dry wt. 35.2 t (Sn 17.6 t)

(2) Grade : Sn 50%, Fe 6%, S 1%, As, Sb $<$0.15%, Bi $<$0.40%

(3) Quotation : US\$ 6.00/lb

(4) Value : US\$ 232,808 (17.6 x 2,204.62 x 6.00)

(5) Refining cost :

Recovery subtraction	US\$ 6,984 (35.2 x 0.015 x 2,204.62 x 6.00)
Smelting and refining cost	US\$28,356 (35.2 x 805.58)
Total	US\$35,340

(No penalty condition applicable)

(6) Selling expenses :

(Mining site – Refinery)

Freight rate	US\$ 243 (36.2 x 6.70)
Loading change	US\$ 109 (36.2 x 3.00)
Bag cost	US\$ 827 (1,034 x 0.80)
Insurance bill	US\$ 70 (198.000 x 1.10 x 0.00032)
(Subtotal)	US\$ 1,249

(Refining – Destination)

Marine transport charges	US\$ 2,312 $(17.6 \times \frac{0.485}{0.5} \times 135.42)$
Carriage	US\$ 922 (" x 54.00)
Loading charges	US\$ 102 (" x 6.00)

Table II-7-4 Calculation of Revenue and Expenditure (Sum of 10 years)

Article				10,000 t/day treatment: First year~7th year Total	10,000 t/day treatment: 8th year~10th year Total	10,000 t/day treatment: Sum Total	9,000 t/day Treatment	8,000 t/day Treatment
Production	Mining	(per day)	Mine	0.41% 3,500t	0.22% 2,000t	0.38% 3,050t	0.41% 2,500t	0.41% 2,500t
Production	Mining	(per day)	Desmonte	0.27 6,500	0.27 8,000	0.27 6,950	0.27 6,500	0.27 5,500
Production	Mining	(per day)	Total	0.32 10,000	0.26 10,000	0.302 10,000	0.309 9,000	0.314 8,000
Production	Mining		Mine	1000t 7,350	1000t 1,800	1000t 9,150	1000t 7,500	1000t 7,500
Production	Mining		Desmonte	13,650	7,200	20,850	19,500	16,500
Production	Mining		Total	0.32 21,000	0.26 9,000	0.302 30,000	0.309 27,000	0.314 24,000
Production	Concentration	High Grade Conc.	Dry Conc.	50.0 73,900t	45.05 24,930t	48.76 98,830t	50.0 91,740t	50.0 82,830t
Production	Concentration	High Grade Conc.	Tin Metal	36,960	11,232	48,192	45,870	41,415
Production	Concentration	High Grade Conc.	Recov.	55%	48%	53%	55%	55%
Production	Concentration	Low Grade Conc.	Dry Conc.	4.10 81,900t	4.40 41,850t	4.2 123,750t	4.1 101,700t	4.1 91,830t
Production	Concentration	Low Grade Conc.	Tin Metal	3,358	1,841	5,199	4,170	3,765
Production	Concentration	Low Grade Conc.	Recov.	5%	7.9%	5.7%	5%	5%

(In 1000 U.S.$)

Article	First year~7th year Total	8th year~10th year Total	Sum Total	9,000 t/day Treatment	8,000 t/day Treatment
Quatation	6 $/lb	"	"	"	"
Sum of Value	533,314.–	172,931.–	706,245.–	661,917.–	597,630.–
Net Revenue	409.104.–	123,022.–	532,126.–	507,729.–	458,424.–

Article			First year~7th year Total	8th year~10th year Total	Sum Total	9,000 t/day Treatment	8,000 t/day Treatment
Operation cost	Mining	Development	8,204	633	8,837	7,500	7,300
Operation cost	Mining	Mine	24,535	6,009	30,544	25,430	25,430
Operation cost	Mining	Desmonte	3,822	1,953	5,775	5,460	4,830
Operation cost	Mining	Others	6,671	2,445	9,116	8,750	8,750
Operation cost	Mining	Total	43,232	11,040	54,272	46,940	46,310
Operation cost		Concentration	119,721	51,309	171,030	154,650	138,640
Operation cost		Dam	3,346	957	4,303	4,110	3,430
Operation cost		Maintenance	12,012	5,097	17,109	18,000	16,180
Operation cost		Laboratory Safety	2,625	1,089	3,714	3,660	3,540
Operation cost		Administration	10,843	4,647	15,490	15,490	15,490
Operation cost		Labour, Social	9,849	4,155	14,004	13,850	13,850
Operation cost		Total	201,628	78,294	279,922	256,700	237,440

Article	First year~7th year Total	8th year~10th year Total	Sum Total	9,000 t/day Treatment	8,000 t/day Treatment
Depreciation	91,848	39,363	131,211	122,267	115,084
Payment of Interests	48,376	8,924	57,300	53,183	49,850
Net Profit	67,252	△ 3,559	63,693	75,579	56,050

Unloading charges	US$ 102	$(17.6 \times \frac{0.485}{0.5} \times 6.00)$
Consular fees	US$ 1	(" x 0.05)
Commission	US$ 222	(" x 13.00)
Marine insurance	US$ 264	(198.000 x 1.1 x 0.00121)
(Subtotal)	US$3,925	
Total	US$5,174	

(7) Net income US$ 192,294

2. Lot : Catavi tin concentrates (low-grade, for volatilization)

(1) Weight : 800 bsgs, wet wt. 40 t, moisture 2.5%, dry wt. 39 t (Sn 1.599 t)

(2) Grade : Sn 4.1 %

(3) Quotation : US$ 6.00/lb

(4) Value : US$ 21,151 (1.599 x 2,204.62 x 6.00)

(5) Smelting and volatilization cost :

US$ 17,310 [21,151 – (39 x 98.50)]

Intracompany invoiced prices for

US$ 6.00 quotation,

Sn 4% : US$ 94.70/t,

Sn 4.2% : US$ 102.30/t

(6) Selling expenses :

Carriage	US$ 268	(40.0 x 6.70)
Loading charges	US$ 120	(40.0 x 3.00)
Bag costs	US$ 936	(800 x 1.17)
Total	US$1,324	

(7) Net income US$2,517

3. Revenue amount US$ 194,811 / day

US$ 58,443,300 / year

US$ 409,103,700 / 7 years

US$ 19.48 / ton of crude ore

Second Term (8th to 10th year after operation start for three years)

1. Lot : Catavi tin concentrates (high-grade)

(1) Weight : 827 bags, wet wt. 28.6 t, moisture 3%, dry wt. 27.7 t (Sn 12.48 t)

(2) Grade : Sn 45.05%, Fe 6%, S 1%, As, Sb $< 0.15\%$, Bi $< 0.40\%$

(3) Quotation : US$ 6.00/lb

(4) Value : US$ 165,082 (12.48 x 2,204.62 x 6.00)

(5) Refining cost :

Recovery subtraction	US$ 5,859	(27.7 x 0.01599 x 2,204.62 x 6.00)

Note : For Sn 50% (–) 1.5% base, add 0.02% (–) for each –1%,

∴ –4.95 x 0.02% = –0.099%

Smelting and refining cost	US$ 22,315	(27.7 x 805.58)
Total	US$ 28,174	

(No penalty condition applicable)

(6) Selling cost :

(Mining site – Refinery)

Freight rate	US$ 192	(28.6 x 6.70)
Loading charges	US$ 86	(28.6 x 3.00)
Bag costs	US$ 662	(827 x 0.80)
Insurance bill	US$ 48	(137.000 x 1.10 x 0.00032)
(Subtotal)	US$ 988	
(Refinery – Destination)		
Marine transport charges	US$1,630	$(12.48 \times \frac{0.43451}{0.4505} \times 135.42)$
Carriage	US$ 650	(" x 54.00)
Loading charges	US$ 72	(" x 6.00)
Unloading charges	US$ 72	(" x 6.00)
Consular fees	US$ 1	(" x 0.05)
Commission	US$ 156	(" x 13.00)
Marine insurance	US$ 182	(137.000 x 1.1 x 0.00121)
(Subtotal)	US$2,763	
Total	US$3,751	

(7) Net income US$133,157

2. Lot : Catavi tin concentrates (low-grade, for volatilization)

(1) Weight : 958 bags, wet wt. 47.9 t, moisture 3%, dry wt. 46.5 t (Sn 2.046 t)

(2) Grade : Sn 4.4%

(3) Quotation : US$ 6.00/lb

(4) Value : US$ 27,064 (2.046 x 2,204.62 x 6.00)

(5) Smelting and volatilization cost : US$ 21,944 [27,064 – (46.5 x 110.10)]

(6) Selling expenses :

Carriage	US$ 321	(47.9 x 6.70)
Loading charges	US$ 144	(47.9 x 3.00)
Bag costs	US$ 1,121	(958 x 1.17)
Total	US$ 1,586	

(7) Net income US$ 3,534

3. Revenue amount US$ 136,691 / day
US$ 41,007,300 / year
US$ 123,021,900 / 3 years
US$ 13.67 / ton of crude ore

(The entire amount for 10 years in Example A reaches US$ 532,125,600)

Calculation of Revenue (B)

Example B : 9,000 t/day treatment under new operating system.

For whole term (10 years after operation start) : Income per day

1. Lot : Catavi tin concentrate (high-grade)

(1) Weight : 899 bags, wet wt. 31.46 t, moisture 2.8%, dry wt. 30.58 t (Sn 15.29 t)

(2) Grade : Sn 50%, Fe 6%, S 1%, As Sb $<$ 0.15%, Bi $<$ 0.40%

(3) Quotation : US$ 6.00/lb

(4) Value : US$ 202,252 (15.29 x 2,204.62 x 6.00)

(5) Refinery cost :

Recovery subtraction	US$ 6.068	(30.58 x 0.015 x 2,204.62 x 6.00)
Smelting and refining cost	US$ 24,635	(30.58 x 805.58)
Total	US$ 30,703	

(No penalty condition applicable)

(6) Selling expenses :

(Mining site – Refinery)

Freight rate	US$ 211	(31.46 x 6.70)
Loading charges	US$ 94	(31.46 x 3.00)
Bag costs	US$ 719	(899 x 0.80)
Insurance bill	US$ 61	(172.000 x 1.1 x 0.00032)
(Subtotal)	US$ 1,085	

(Refinery – Destination)

Marine transport charges	US$	2,008	$(15.29 \times \frac{0.485}{0.5} \times 135.42)$
Carriage	US$	801	(" x 54.00)
Loading charges	US$	89	(" x 6.00)
Unloading charges	US$	89	(" x 6.00)
Consular fees	US$	1	(" x 0.05)
Commission	US$	193	(" x 13.00)
Marine insurance	US$	229	(172.000 x 1.1 x 0.00121)
(Subtotal)	US$	3,410	
Total	US$	4,495	

(7) Net income US$ 167,054

2. Lot : Catavi tin concentrate (low-grade, for volatilization)

(1) Weight : 695 bags, wet wt. 34.77 t, moisture 2.5%, dry wt. 33.9 t (Sn 1.39 t)

(2) Grade : Sn 4.1%

(3) Quotation : US$ 6.00/lb

(4) Value : US$ 18,387 (1.39 x 2,204.62 x 6.00)

(5) Smelting and volatilization cost : US$ 15,048 [18,387 – (33.9 x 98.50)]

(6) Selling expenses :

Carriage	US$	233	(34.77 x 6.70)
Loading charges	US$	104	(34.77 x 3.00)
Bag costs	US$	813	(695 x 1.17)
Total	US$	1,150	

(7) Net income US$ 2,189

3. Revenue amount US$ 169,243 / day
US$ 50,772,900 / year
US$ 507,729,000 / 10 years
US$ 18.805 / ton of crude ore

Calculation of Revenue (C)

Example C : 8,000 t/day treatment under new operating system.

For whole term (10 years after operation start) : Income per day

1. Lot : Catavi tin concentrate (high-grade)

(1) Weight : 812 bags, wet wt. 28.41 t, moisture 2.8%, dry wt. 27.61 t (Sn 13.805 t)

(2) Grade : Sn 50%, Fe 6%, S 1%, As, Sb < 0.15%, Bi < 0.40%

(3) Quotation : US$ 6.00/lb

(4) Value : US$ 182,609 (13.805 x 2,204.62 x 6.00)

(5) Refinery cost :

Net yield subtraction	US$	5,478	(27.61 x 0.015 x 2,204.62 x 6.00)
Smelting and refining cost	US$	22,242	(27.61 x 805.58)
Total	US$	27,720	

(No penalty condition applicable)

(6) Selling expenses :

(Mining site – Refinery)			
Freight rate	US$	190	(28.41 x 6.70)
Loading charges	US$	85	(28.41 x 3.00)
Bag costs	US$	650	(812 x 0.80)
Insurance bill	US$	55	(155.000 x 1.1 x 0.00032)
(Subtotal)	US$	980	
(Refinery – Destination)			
Marine transport charges	US$	1,813	$(13.805 \times \frac{0.485}{0.5} \times 135.42)$
Carriage	US$	723	(" x 54.00)
Loading charges	US$	80	(" x 6.00)
Unloading charges	US$	80	(" x 6.00)
Consular fees	US$	1	(" x 0.05)
Commission	US$	174	(" x 13.00)
Marine insurance	US$	206	(155.000 x 1.1 x 0.00121)
(Subtotal)	US$	3,077	
Total	US$	4,057	

(7) Net income US$ 150,832

2. Lot : Catavi tin concentrate (low-grade, for volatilization)

(1) Weight : 628 bags, wet wt. 31.39 t, moisture 2.5%, dry wt. 30.61 t (Sn 1.255 t)

(2) Grade : Sn 4.1%

(3) Quotation : US$ 6.00/lb

(4) Value : US$ 16,601 (1.255 x 2,204.62 x 6.00)

(5) Smelting and volatilization cost : US$ 13,586 [16,601 – (30.61 x 98.50)]