Similarly large mine tubs are put in use for main haulage levels (L650), while small mine tubs for sublevel haulages.

The classification by purchase year shows that small mine tubs are relatively new but a lot of large mine tubs were made 25 years ago which indicates few renewals.

7) Excavation equipment

Rock drills are repaired by exchanging the demaged part with a new one. However, all the parts are not always assorted because all rock drills are imported machines with a variety of types, thus a substantial number of rock drills are lost as being disabled. The parts are made in-house at the machine plant in Catavi mine and Oruro, and these parts have problems in durability from material quality and maching precision, but this in-house preparation of parts seems to be unavoidable because imported parts are not available instantly.

The classification by purchase year in five-year division is shown in Table 2–29, and some old drills are kept in storage but they should be disposed.

There are few disabled loading machines and this fact may be due to these loading machines being relatively new. The classification by purchase year for each section is listed in Table 2–30.

2–5 Underground Development

In Catavi mine, drifts and cross-cuts have been excavated like the meshes of a net, and the total length reaching about 900 km.

Most of these drifts and cross-cuts, being located in preferable rock conditions, are not equipped with timbering, while those in fault zones and likes are reinforced with wooden timbering or concrete linings, resulting in the timbering ratio of some 10 to 15 percent.

2–5–1 Underground Constructure

The basic underground structure is composed of combinations of adits and shafts (partly inclines).

There are three main levels for adits. For Patiño Level at L383, adits are developed roughly perpendicularly against the vein. For Cancañiri level at L411 and Siglo XX Level at L650, adits are excavated in parallel with the vein entering into the orebody where cross-cuts are provided.

Seven shafts are in operation at present, and located at sites with suitable rock conditions and distant from the main vein.

Vertical distance between levels are 15 m to 20 m in the upper portion (higher than

L411), and 35 m or 50 m in the lower portion (L411 to L650).

1) Principal level

The principal level is Siglo XX Level at L650, and some 1,700 m portion from mine mouth is almost straight. The sectional dimension is 3 m x 3 m having a arch shape, and a double track extending for 200 m in the middle with a dimension of 6 m x 3 m. About 100 m portion of mine mouth and about 50 m portion of the fault zone are reinforced by concrete lining, but the remaining portion is not provided with any timbering. The cross section of the level is shown in Fig. 2–6.

2) Shaft

All the cross sections of shafts are rectangular and the size differs slightly with every shafts.

For example, the New BEZA Shaft has a 3.79 m x 2.60 m section of excavation, in which timber sets are arranged in parallel cross, and the cage space is 1.57 m x 1.55 m. The distance between timber sets, constructed with 20 cm square lumbers (imported from the U.S.), is 1.8 m.

The size of the machine room is 6.0 m long x 6.40 m wide x 3.50 m high, and the distance between the winch and the head sheave is 20 m.

Shaft-related drawings are shown in Fig. 2–7 to Fig. 2–9.

3) Ore passes and ore shutes

Ore passes are about 1.5 m x 1.5 m in size, and most of them are steeply inclined about 50° to 60°. The lower portion of several ore passes are gathered into one group to reduce the number of shute gates at L650. (see Fig. 2–10)

2–5–2 Excavation System and Efficiency of Drifts and Shafts

1) Drifts

All drifts are excavated with the rail way system. Every group, consisting of two labors, has its own faces. Two rail men for extending rail, and two locomotive operators of a haulage squad for mucking cooperate with members of the first shift for excavation.

(1) Excavation system

The burncut method is used for drifting to 80 up excavation efficiency by making the length of round longer.

Each operation of drifting is explained on a typical drift of large dimension at L650 which is to augment the haulage capacity.

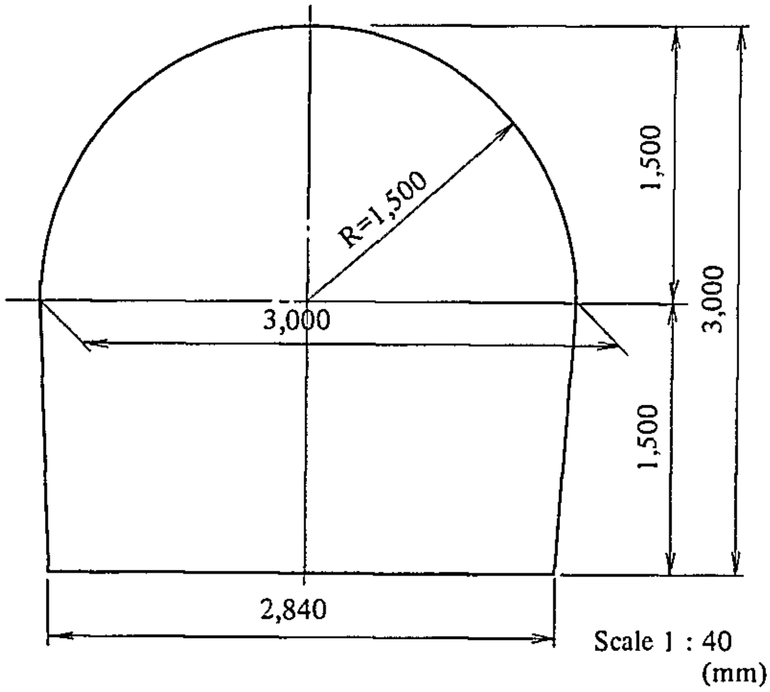


Fig. 2–6 Section of Main Level (L650)

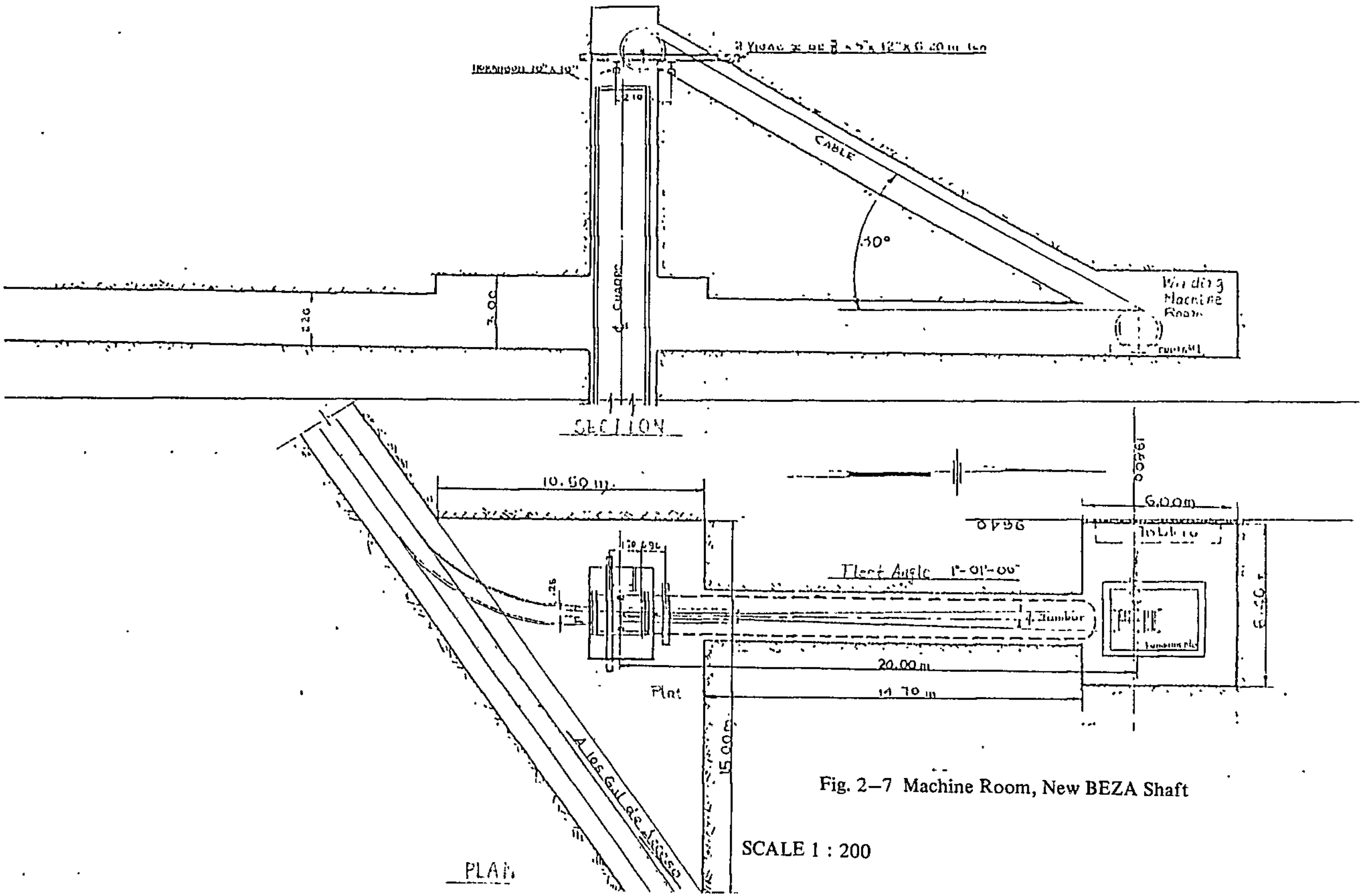


Fig. 2–7 Machine Room, New BEZA Shaft

SCALE 1 : 200

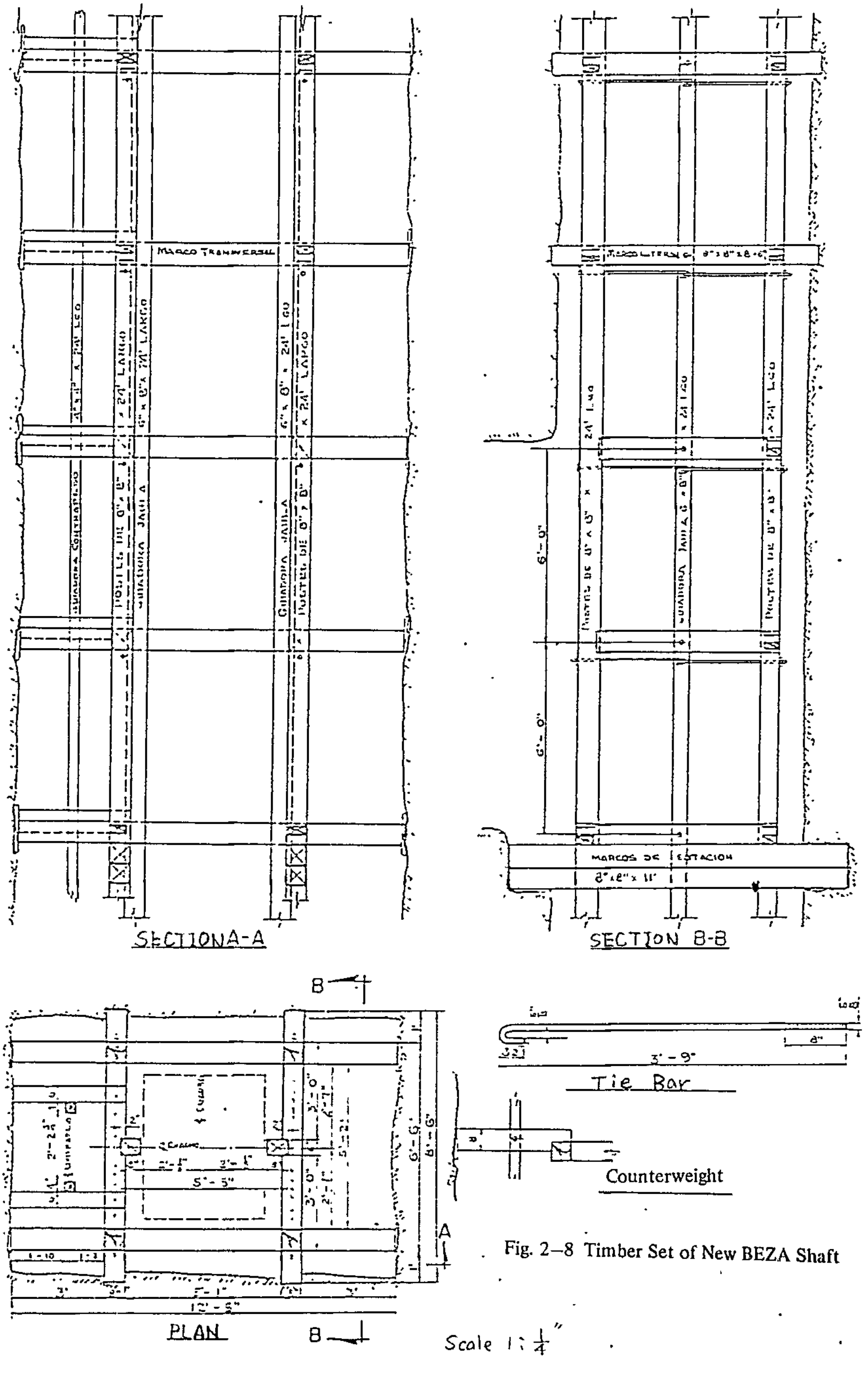


Fig. 2–8 Timber Set of New BEZA Shaft

(i) Drilling and blasting

The drilling and blasting is extended over two days: the first day for the center cut and the second day for the round blasting. The drilling pattern was the burncut method with 2.3 m per length of borehole and 2.0 m to 2.2 m per length of round.

The cut hole consisted of four holes of 2.3 m long which were drilled with the first and the second rods and all holes were charged with explosives. The drill round is shown in Fig. 2–11, and the state after the center cut in Fig. 2–12 respectively. In this example, 1 and 2 failed to explode, and only 3 and 4 were blasted.

Broken rocks by center cut spattered slightly backward forming a space on which drilling was performed with.

The drilled diameters were 40 mmϕ with a 1.2 m insert rod, and 38 mmϕ with a 2.4 m insert rod.

Drilling for the round blasting were carried out referring to the blasted state caused by the center cut. The general arrangement of drillings is given in Fig. 2–13. Two rail men prolonged the rail during the drill arrangement by the driller, then drilling was started.

The cycle time of drilling and blasting is shown in Table 2–31. A period of 2 hours and 17 minutes was required to drill 20 holes per 2.3 m long, some 5 hours were consumed for completion of blasting after the charge of explosives. Hole cleaning was carried out with a hose for charging AN-FO, but jaming was found in holes near by the cut hole, resulting in drilling again. Charge was carried out in such a way that one primer attached with a fuse was charged in each hole, and several remainders of the charge were put in the holes of the toe. This may be a proper action because of imperfect center cut and heavy toe.

AN-FO was charged with a simple loader, which only suction mouth was inserted into a 50 kg pack of AN-FO.

The consumption of explosives was about 3.0 kg/m^3. During the operation of drill, the air pressure fell down to 2.5 kg/cm^2, which made drilling impossible temporarily. The reasons for this trouble were judged that the face was too far from the compressor and drilling operations concentrated in this time.

(ii) Extraction of waste

The resultant excavation by blasting was good. Extraction of waste was performed in such a way that broken rock were loaded into mine tubs loading weight 1.7 ton with a track-type rocker shovel (Pneumatic operation), and the mine tubs were transferred by labor to the place where an locomotive was able to approach, then by the locomotive to the switching track.

The quantity of broken rock was about 74 tonnes per one round, while two days were required for mucking because of its inferior efficiency.

(2) Advancing efficiency

The advancing efficiency can be estimated from the required days per one round as follows:

Length of round	2.0 m to 2.2 m
Center cut	1 day
Found blasting	1 day
Mucking	2 days
Required days	4 days
Working days per month	28 days
Attendance ratio	78%
Cycle per day	5.5 cycles
Monthly advance	11 m to 12 m

Table 2–32 shows the drifting at L650.

(3) Equipment

Rock drill	Leg drill (Atlas Copco BBD-90W)	1
Loader	Track-type rocker	
	Shovel (made in U.S.S.R)	1
Hauling equipment	Electric locomotive	
	(trolley-type 4 t)	1
	Mine tub (1.7 t, rigid-car type)	10
Bit-rod	Made by Coromant Co.	
	1.2 m (40 mmϕ bit-dia., insert-type)	
	Made by Coromant Co.	
	2.4 m (38 mmϕ bit-dia., insert-type)	

Table 2–32

Note: As of June, 1981, eight drifting faces are being worked on by excavating cross cuts from each spot and opening in both sides.

2) Shafts

The way for shaft sinking is that a small raise is excavated upward at first, and the raise is used as a pilot through which broken ore falls down to expand the width, thus the shaft sinking is advanced downward.

The main method for the excavation of a pilot is the raise with stull timbering, but the

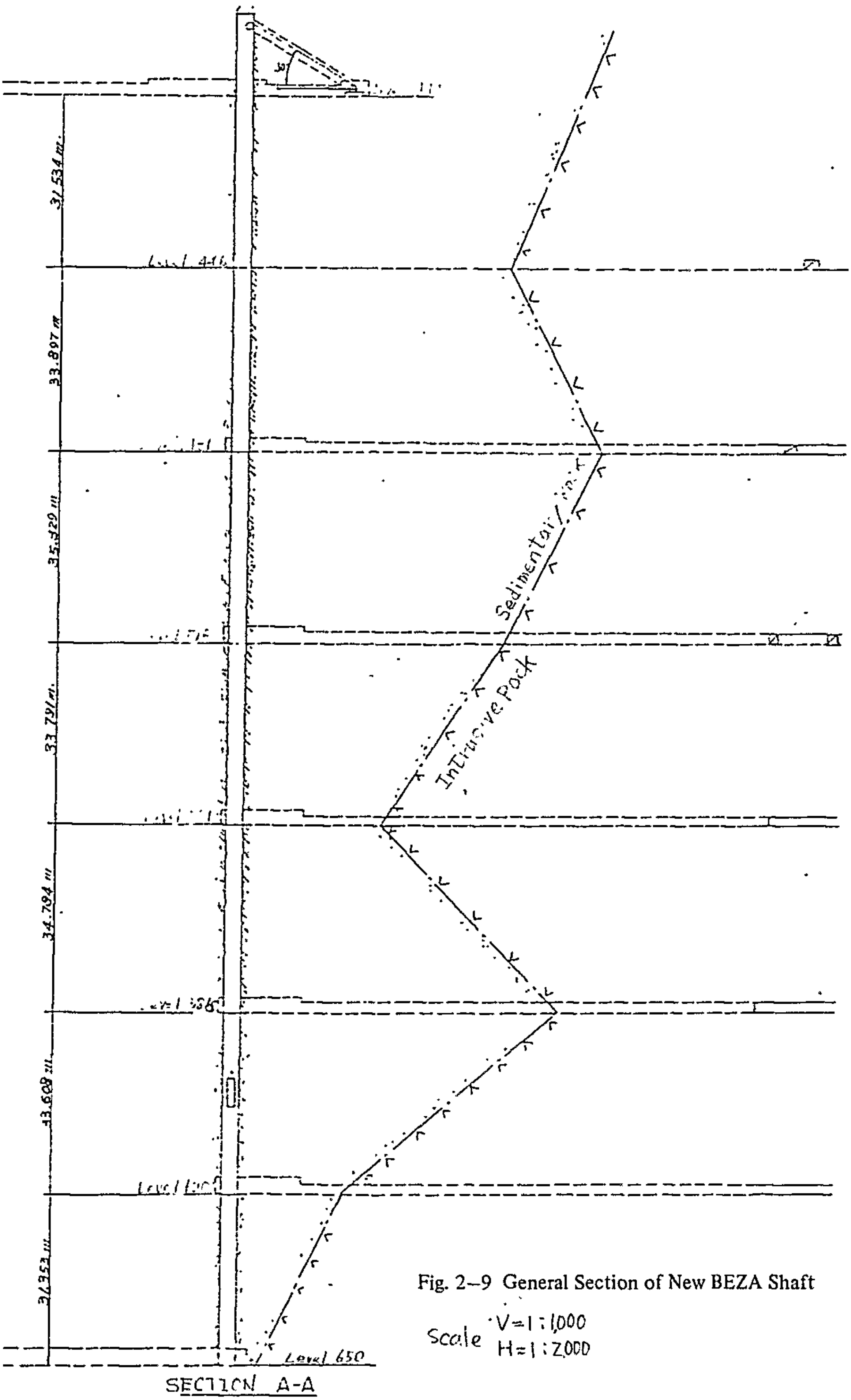


Fig. 2–9 General Section of New BEZA Shaft

Scale V=1:1,000 H=1:2,000

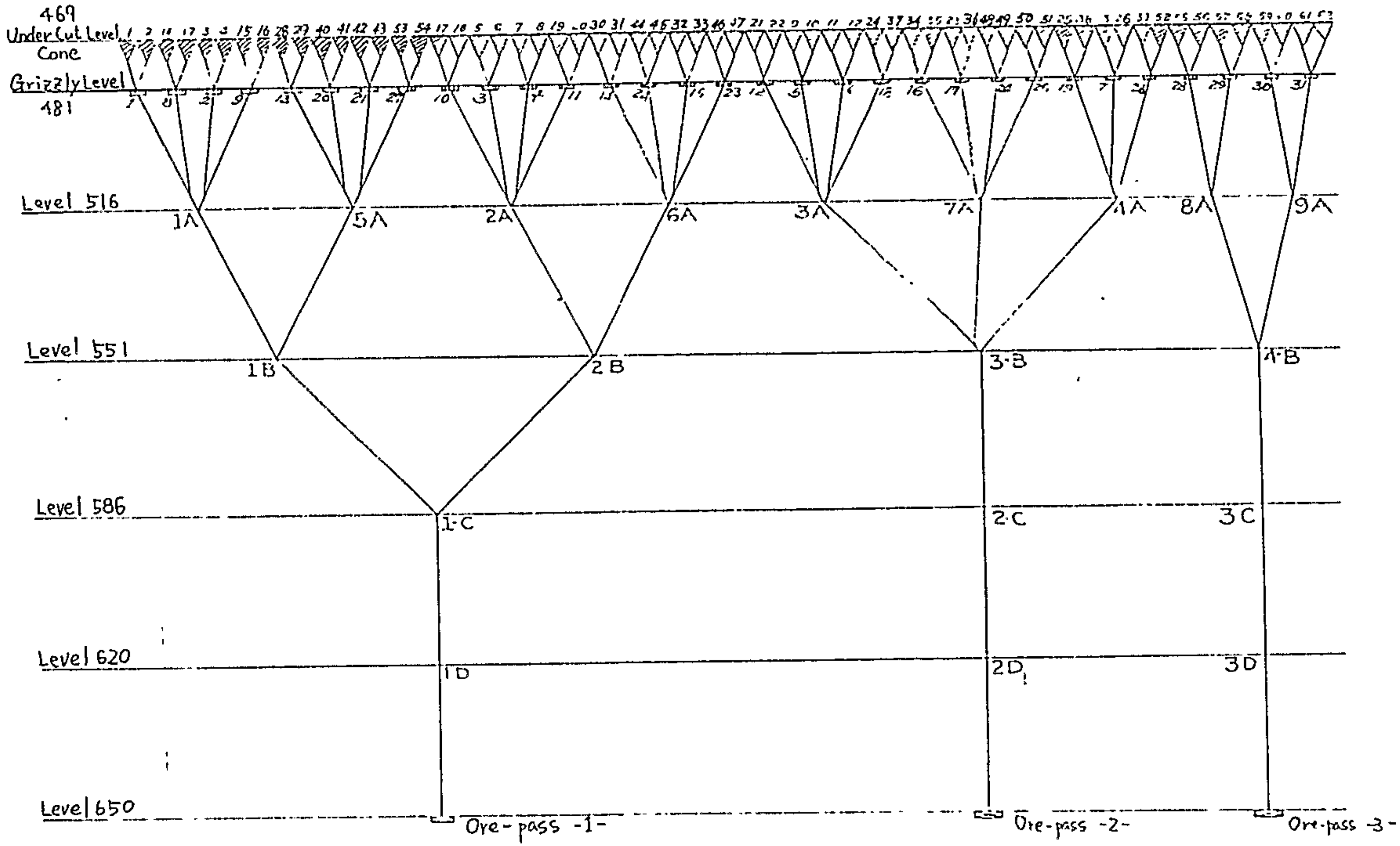


Fig. 2–10 Typical Arrangement of Ore-Pass, Block 5–D

raise-climber and the full face cutting machine (made by Turmag Co., West Germany) are also used.

In widening excavation, blasting is done by making the pilot as a free face and broken rock are forced to fall down through the pilot by labor.

After the widening has advanced to a certain extent, wooden timber sets are installed from the top.

(1) Excavation method

The shaft sinking is explained on the New BEZA Shaft which is excavated to run from L411 to L650.

(i) Excavation of pilot

The pilot from L411 to L551, the size of which is 1.5 m x 1.5 m, is excavated by the blasting method with a raise-climber. The type of blasting is the burn cut, the length of borehole is 2.4 m, and the number of boreholes is 5 for the center cut and 14 for the square-up holes; 19 in total.

At first, the center cut blasting is put in practice, followed by round blastings. Drilling is performed with a stoper, where a insert rod (22 mmϕ, hexagonal and hollow) with 1st rod (0.6 m), 2nd rod (1.2 m), 3rd rod (1.8 m) or 4th rod (2.4 m) is used.

The length of one round is about 2.0 m to 2.2 m.

Table 2-33 Advancing of New BEZA Shaft Sinking (unit : m)

Year / Month	1977		1978		1979	1980	1981	Total	
	Pilot Raise	Ripping	Pilot Raise	Ripping	Ripping	Ripping	Ripping	Pilot Raise	Ripping
January	-	-	10.6	-	-	10.1	15.0		
February	-	-	-	-	-	9.1	11.2		
March	-	-	-	-	-	13 0	10 0		
April	-	-	-	-	-	13 5	9.2		
May	5.8	-	2.4	-	-	13.0	7.0		
June	8.0	-	10.6		-	13.0	12.0		
July	10.6	-	5.7		-	-			
August	2.4	-	6.0		10.0	-			
September	11.0	-	8.1	6.5	-	13.6			
October	7.9	-	-	7.5	-	10.0			
November	8.0	-	-	7.7	-	14.1			
December	-	-	-	11.3	-	12.2			
Total	53.7	-	43.4	33.0	10 0	121.6	64.4	97.1 m	229 m

For explosives, dynamite and AN-FO are used with a fuse for ignition.
Fig. 2—14 illustrates the drilling arrangement.

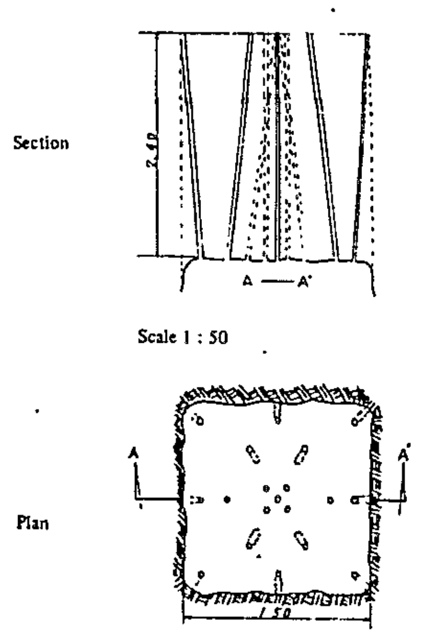


Fig 2-14 Drilling Pattern of Raise

The drilling is done only in the first shift with three labors.

The raise-climber is an air type product of Alimuk Co., Sweden. This type of climber is used also in Japan, but the air type is old and the electric type is dominant now.

The excavation of the pilot from L411 to L551 is not supported with any timbering because of adequate rock conditions.

The pilot from L551 to L650 is excavated with a raise-borer (hydraulic) of 1.5 m diameter. The borer is made in West Germany. The excavation is performed in such a way that a pilot of 250 mm diameter is bored downward from L551, and after reaching to L650 the pilot is reamed upward from below with a 1.5 m diameter cutter. This machine is reportedly able to excavate a 250 m hole, but the weak point is that the machine chamber is required considerably large space because machine height is about 5 m.

The operation of this machine is so free and easy that a German engineer staying at La Paz visits this mine once a week to operate the machine.

(ii) Widening

Windening is advanced downward from L411. Square-up holes are drilled and blasted by using the pilot as a free face and are widened to 3.8 m x 2.6 m.

The length of borehole is 2.4 m and rods 1st, 2nd, 3rd and 4th have been used.

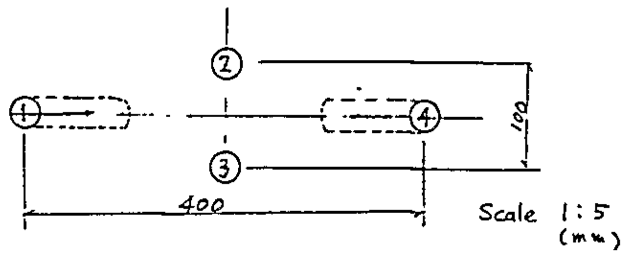
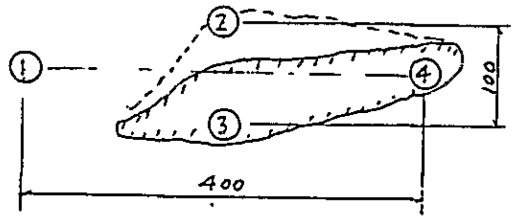


Fig. 2–11 Cut Hole Patter



Blasted Part

Fig. 2–12 State of Cut Hole after Cut Blasting

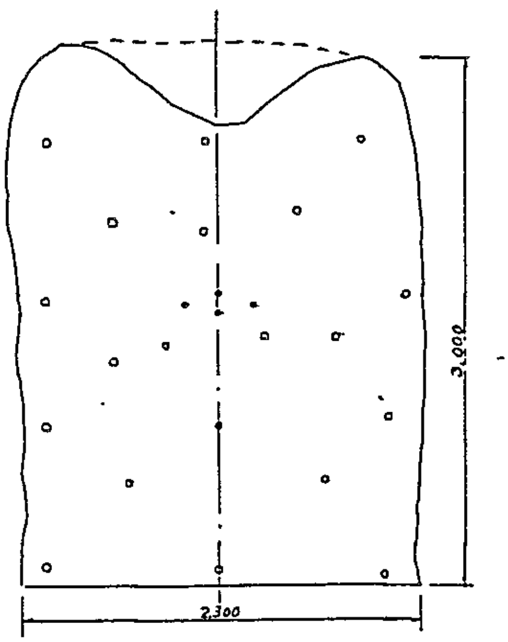


Fig. 2–13 Drilling Pattern of Drifting <650

Table 2-31 Cycletime of Drilling and Blasting

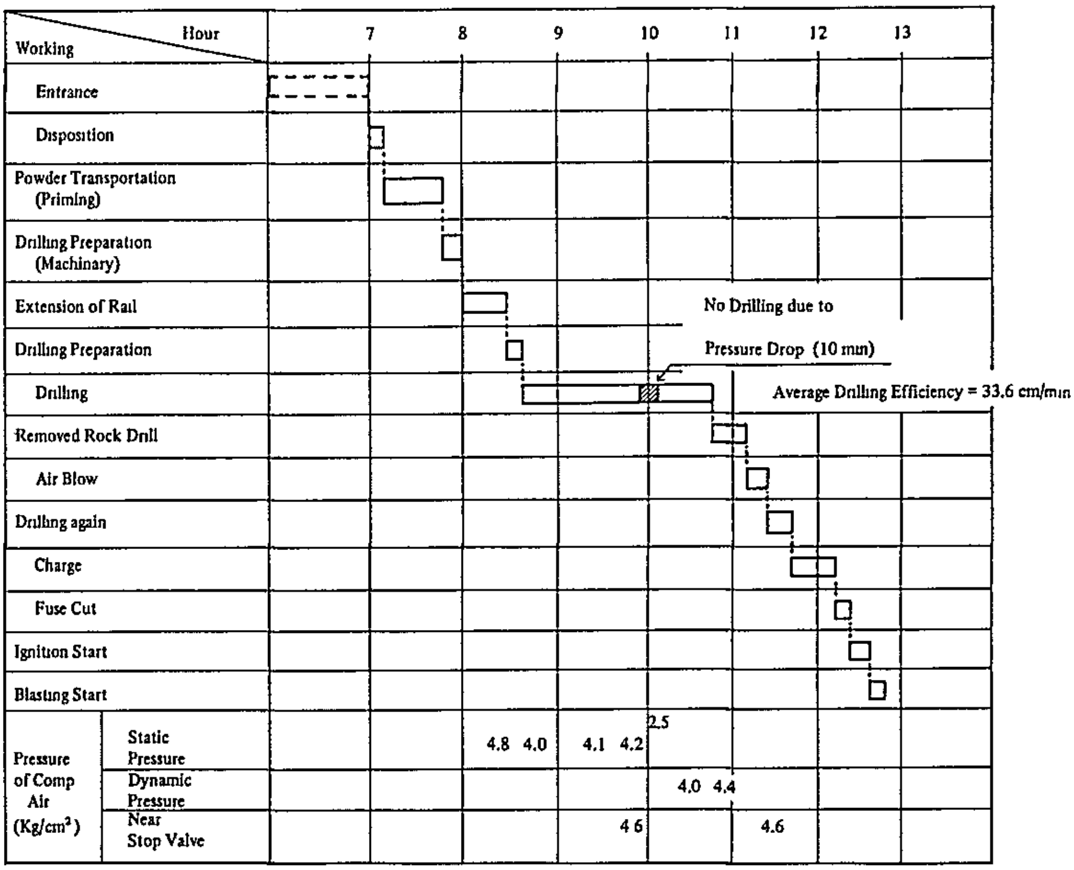


Table 2-32 Drifting in L650

(unit: m)

Month \ Year	1978	1979	1980	1981	Total
January	–	16 4	30.0	71.0	
February	–	–	22 9	91.6	
March	–	9.8	47.2	58.8	
April	–	21 3	60 6	82.7	
May	–	9 8	45.2	73 2	
June	2.0	2.0	67.2	74 5	
July	6.9	12 1	18 2		
August	7.9	21.8	44.1		
September	18.1	23 1	72 6		
October	19.8	28 3	98 5		
November	19 7	19.0	93.5		
December	3.9	6.2	69.9		
Total	78 3	169.8	669.9	451.8	1369 6

The rod for working is 22 mm diameter, hexagonal and hollow, and insert type with a bit of 36 mm diameter.

The length of one round is about 2.0 m to 2.2 m, and dynamite and AN-FO are used as explosives. The cap and fuse firing is dangerous as it gives little chance to the labors to take a safe position so an electric cap firing with a specially designed electric detonator is used. Fig. 2–15 illustrates the drill round arrangement.

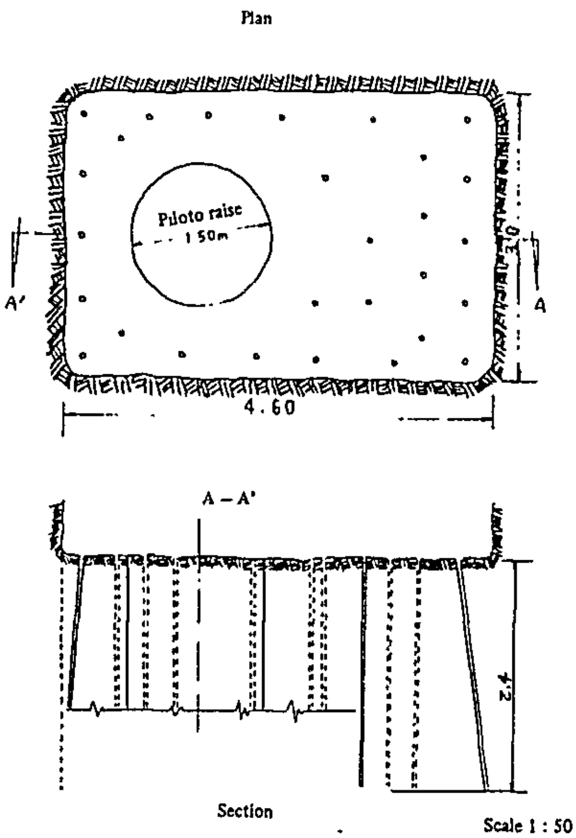


Fig. 2–15 Drilling Pattern of Shaft Sinking

After blasting, broken rock are forced to fall down through the pilot with labor. This widening is also non-timbering type because of the excellent rock conditions. The widening is also worked only in the first shift with three laborers.

(iii) Arrangement of timber sets

The arrangement of timber sets between L411 and L481 is being done now. The timber sets are built up with 20 cm square lumbers (Canadian pine 7.4 m long) by at-site processing at a 1.8 m interval. Every timber set is fixed with wedges into a carved portion of the rock.

Timbering lumbers and other material are brought in with a wire rope and an air hoist (5 HP) installed nearby the shaft mouth.

This work is conducted in the first shift by four laborers.

(2) Shaft sinking efficiency

Since there are no data of shaft sinking efficiency, the advancing of the New Beza Shaft is shown in Table 2–33.

The pilot excavation between L650 and L551 in 1977 and 1978 was actually conducted with raise borer, while no real data have recorded for the span between L411 and L551.

2–6 Transportation, Ventilation, Drainage of Mine Water

2–6–1 Transportation

Underground transportation work includes the transportation of ore, the transportation of waste rocks,the transportation of materials and the transportation of persons. Transportation occupies an important part in a mine in securing production, the capacity of supplying materials, securing real working time in stopes through the reduction of the time needed for entering and departing the mine.

(1) Transportation of ore

Ore stopped at a shrinkage stope are dumped into a shute below through sublevel haulage. Ore stopped at a block caving stope pass through an ore pass directly, and are dumped into a shute below. All the lower shutes are connected to the L650 main haulage level, and ore collected there are drawn out manually from iron shute gates and loaded on mine tubs.

The sublevel haulage is carried out by combining 4-ton or 1.5-ton class trolley locomotives and battery locomotives with 1.7-ton or 0.75-ton tubs.

In the L650 main haulage level, a train is formed by connecting thirteen 5-ton tubs to a 10-ton trolly locomotive.

Table 2–34 shows the state of round trips 10-ton trolley locomotives in the main haulage level July, 1981 data, showing that the number of total round trip times per day vary largely and the average number of times is about 64 times per day. The state of operation in a week shows that the number of round trip times reaches its peak on Wednesday, Thursday and Friday, but drops in Saturday, Sunday and Monday. The fact is also related with the rate of attendance. The fact that the average number of working locomotives is seven per day represents that four locomotives are always out of order or under repair.

When the number of times which is required two hours or more in loading and hauling in the main L650 are picked out, it becomes as large as 6.5 times per day in average, that is one among ten of the locomotives takes a long time to draw out the ore. Moreover, when the number of times requiring an hour or more in dumping on the surface is picked out, its average value becomes as large as 10.1 times per day, showing that waiting at the concentration acceptance occurs at a rate of once in six times.

Two men, a driver and an assistant driver, get on one train. In the ore loading, four shute-men per shift are engaged in loading, a group of two shute-men at each shute.

On the surface, there is a measuring point at a place about 200 m from mine mouth Siglo XX, where each tub is measured on a trackscale and the value after subtracting the empty weight of the tub from its gross weight is recorded. The sampling of ore on each tub is carried out close to this point, scooping small-sized ones alone with a shovel and keeping the two-fourths of them as one sample after dividing them into four. These works are carried out by one measuring man per shift and two sampling men per shift in a three shift system. Fig. 2–16 shows the system of main haulage.

(2) Transportation of waste rock

The transportation of waste rock is to muck waste rock produced in exploration drifts and underground development.

The waste rock above L650 is dumped into nearby shutes, drawn out manually from iron shute gates and loaded on tubs.

In the drifting of L650, one train is allocated near the face, and after being loaded, waste rock is transported to the waste rock disposal on the surface by an 8-ton or a 6-ton trolley locomotive. Sometimes, waste rock is disposed in goafs underground.

(3) Transportation of materials

Materials consumed in the mine are transported into the mine from the Cancañiri mine mouth of L411 where the principal material bay and a mechanical shop are situated and from the Siglo XX mine mouth of L650. They are transported through each vertical shaft and inclined shaft and stored in a material bay near the plat of the vertical shaft and the inclined shaft.

Among principal materials, dynamite is stored once in the underground magazine of L650 and transported to a reserve station near each stope, but timbers, cement, etc., are directly transported to shaft plats near specified stopes. AN-FO and fuses (with caps) are transported directly from the mixing plant of AN-FO on the surface to the reserve stations for explosives.

The transportation circuit of materials is shown in Fig. 2–17.

(4) Transportation of persons

The entrances for workers are at three positions: the Cancañiri mine mouth of L411, the Patiño mine mouth of L683 and the Siglo XX mine mouth of L650, and locomotives and man cars are arranged and operated regularly.

(i) L650 Siglo XX mine: 100 passenger train, 4-ton trolley locomotives.

(ii) L411 Cancañiri mine: 100 passenger train.

(iii) L383 Patiño mine: 50 passenger train.

Workers are transported to each stope mainly via the vertical shafts and the inclined shafts of each area.

2–6–2 Ventilation

The principal system of underground ventilation used in Catavi mine draws in fresh air from adits by natural ventilation and distributes it to each stope on the way of shafts.

In block caving stopes, a system which extracts air soiled with blasting fume, dust, etc., by mechanical ventilation using fans has been used, but in shrinkage stopes, natural ventilation has been used or further, air blow is added.

(1) Total ventilation system

As principal inlet levels, there are four adits, namely, Azul level (L295). Patiño level (L383), Cancañiri level (L411) and Siglo XX level (L650), and in addition, Beza level (L355) and Bismark shute (surface-L383) have been provided.

As exhaust levels, there are Salvadora shaft (LO - not clear), Animas shaft (L50–L383) and Transformador shaft (L50), and in addition, exhausted air is discharged from the surface subsidence of block caving goafs. Salvadora shaft is beginning to be broken by the effect of stopping and its inside conditions are not clear. Fig. 2–18 indicates the positions of inlet and outlet adits. The result of investigating the quantity of air flow about the principal inlet and outlet adits of the whole mine is shown in Table 2–35. The result of measuring the quantity of air flow carried out by Catavi mine itself is also shown.

The average underground temperature is approximately 10°C – 15°C.

As shown by the measured results in Table 2–35, the total quality of inlet is 3,667.2 m^3/min and the total quantity of outlet is 933.3 m^3/min, the difference between the inlet and outlet quantities is thought to be escaping from somewhere.

Especially in block caving goafs, steam is seen to rise from surface cracks.

Accordingly, it seems that outlet air is escaping little by little over a wide range from old principal goafs, etc. The quantity of inlet according to the measured results of this time

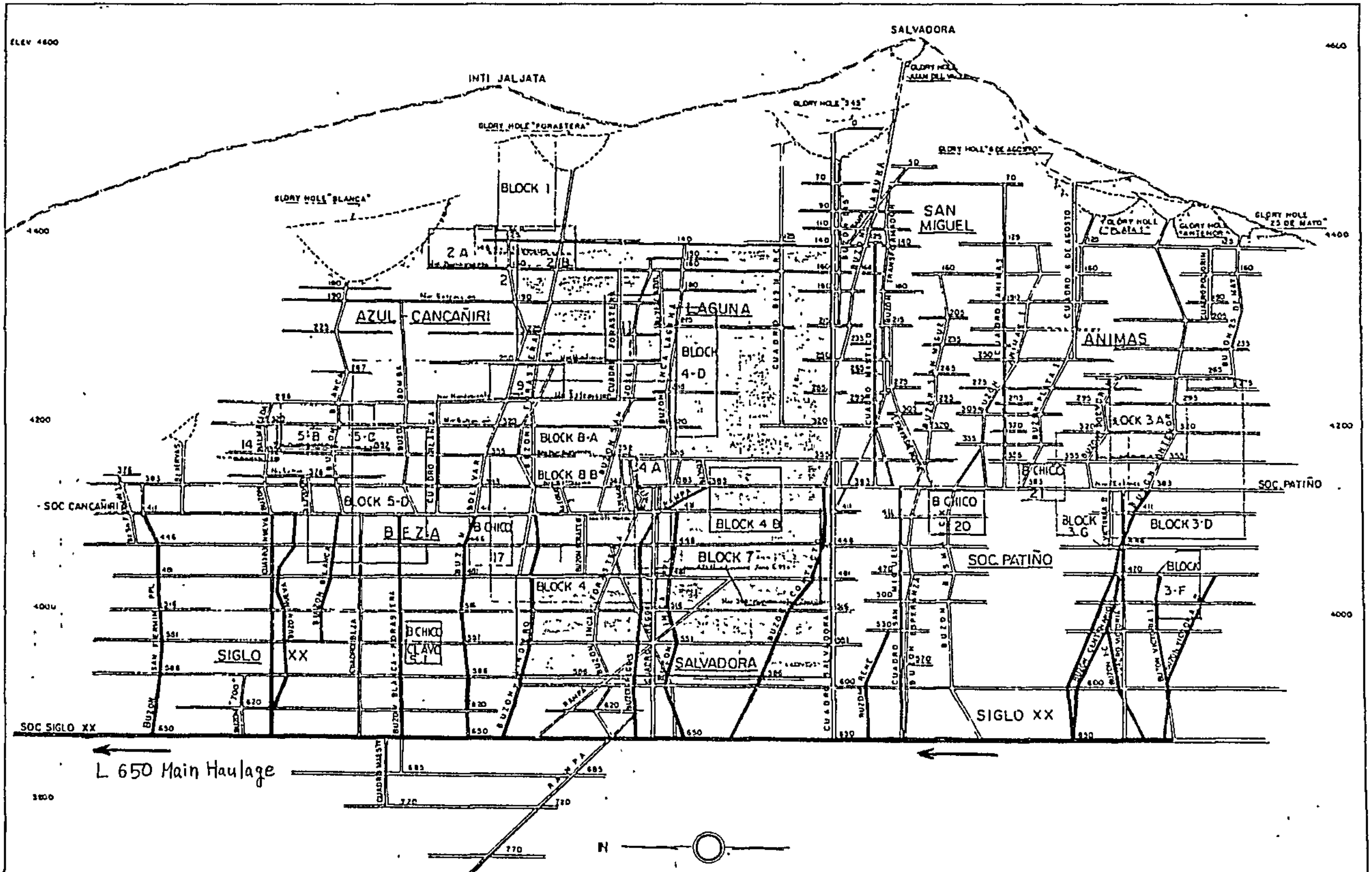


Fig. 2–16 System of Main Haulage

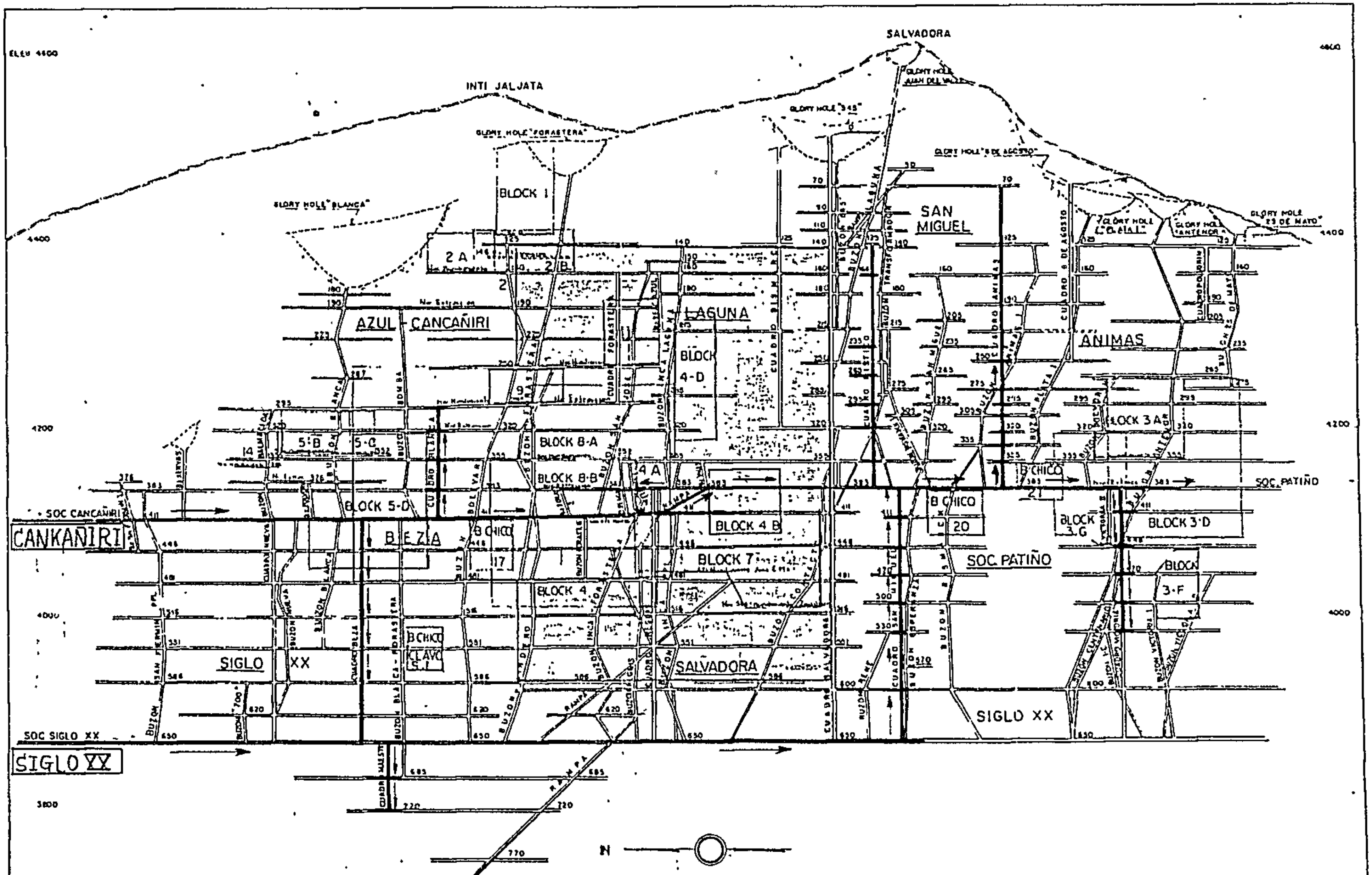


Fig. 2–17 System of Material Transportation

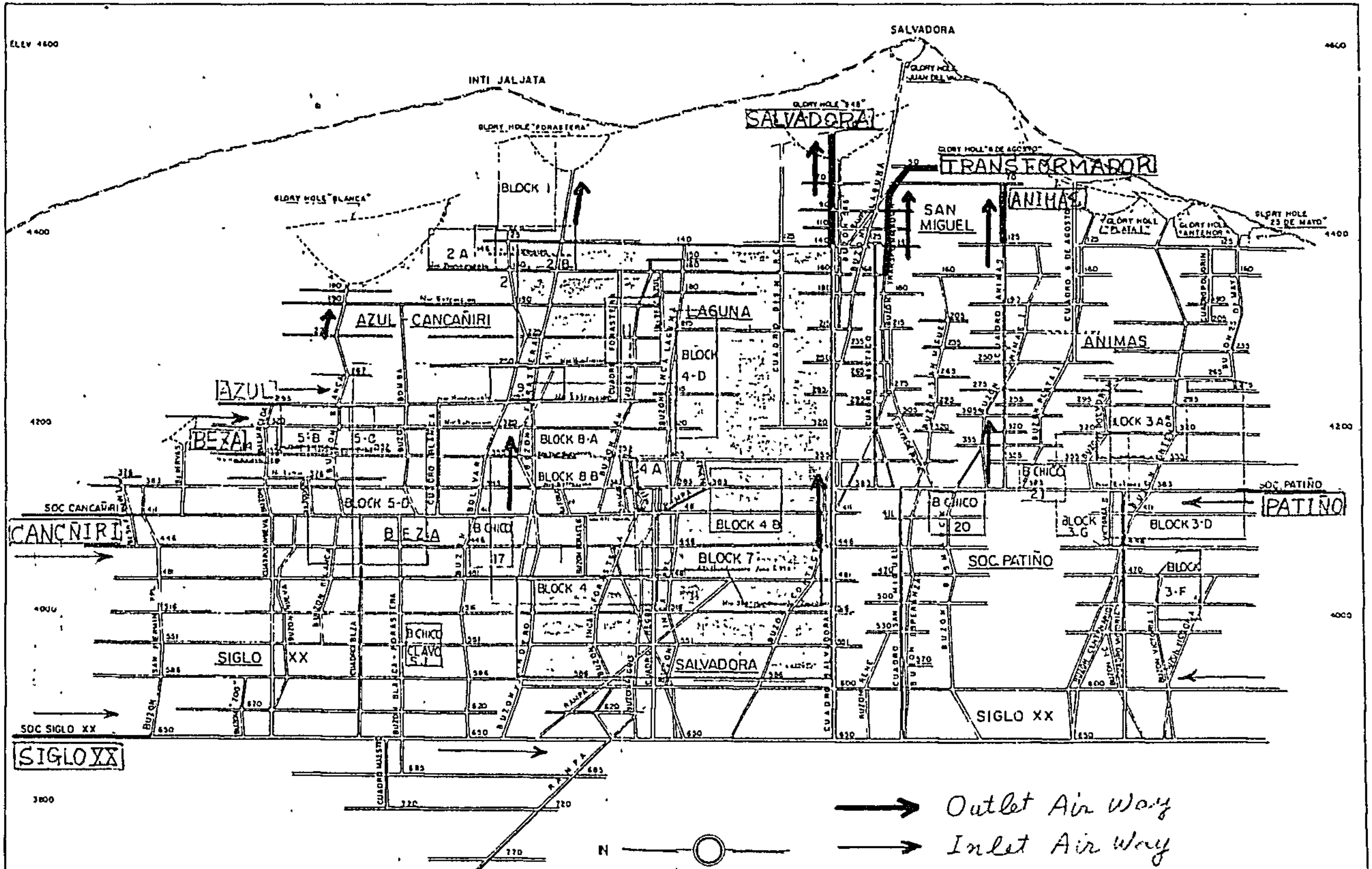
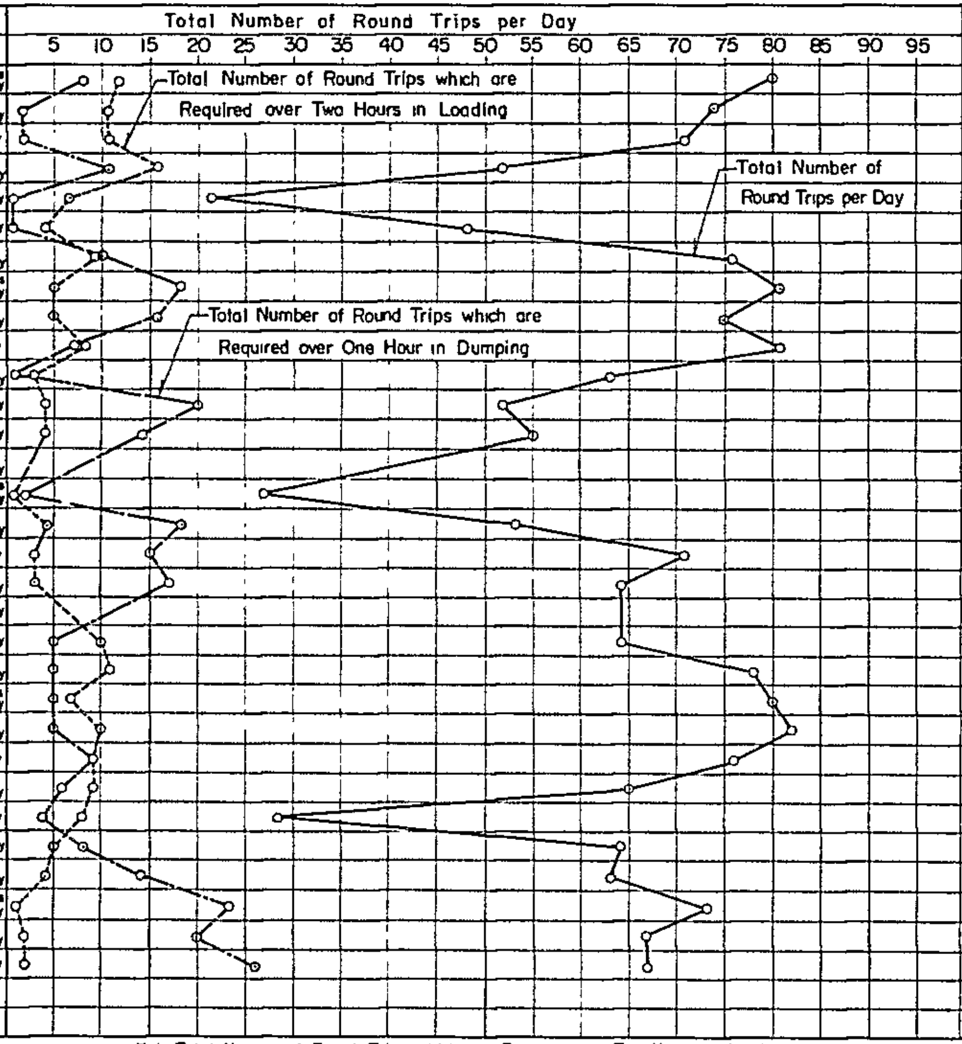


Fig. 2–18 Location of Main Air Way

Table 2-34 Number of Round Trip of Main Haulage (∠650) (July, 1981)



		Item	Total Number of Round Trips per Day	X 1	X 2	Number of Round Trips
1	Wednesday		80 time	12 time	8 time	8
2	Thursday		74	11	2	7
3	Friday		71	11	2	8
4	Saturday		52	16	11	7
5	Sunday		22	7	1	5
6	Monday		48	4	1	6
7	Tuesday		76	9	10	8
8	Wednesday		81	5	18	8
9	Thursday		75	5	16	8
10	Friday		81	8	7	8
11	Saturday		63	3	1	7
12	Sunday		52	4	20	5
13	Monday		55	4	14	6
14	Tuesday	Strike	—	—	—	—
15	Wednesday		27	1	2	3
16	Thursday		53	4	18	7
17	Friday		71	3	15	8
18	Saturday		64	3	17	7
19	Sunday	Holiday	—	—	—	—
20	Monday		64	10	5	6
21	Tuesday		78	11	5	8
22	Wednesday		80	7	5	8
23	Thursday		82	10	5	8
24	Friday		76	9	9	8
25	Saturday		65	9	6	8
26	Sunday		28	8	4	3
27	Monday		64	5	8	7
28	Tuesday		63	4	14	7
29	Wednesday		73	1	23	8
30	Thursday		67	2	20	8
31	Friday		67	2	26	8
Average			63.9 time/day	6.5 time/day	10.1 time/day	7.0

X 1 Total Number of Round Trips which are Required over Two Hours in Loading
X 2 Total Number of Round Trips which are Required over One Hour in Dumping

Table 2–35 Actual Air-quantity of Main Ventilation

Location	Level	Temperature (°C)		Average Velocity of the Air Flow (m/sec)	Sectional Area (m^2)	Air-quantity (m^3/min)	Arrangement of Air Flow	Measurement Air-quantity in Catavi (m^3/min)
		Dry	Wet					
Sigro XX	L650	4.5	3.0	1.66	7.91	785.7	Intake	1,362
Cancañiri	L411	7.3	2.5	2.16	7.51	973.3	Intake	1,650
Azul	L295	3.8	0.4	1.88	4.14	467.0	Intake	630
Beza	L355	4.5	0.9	1.58	4.77	452.2	Intake	1,020
Salvadora	(L0) ground	9.9	7.2	0.49	6.63	194.9	Exhaust	-
Arumas	(L50) ground	-	-	2.07	3.10	385.0	Exhaust	-
	L384	5.7	3.0	1.16	4.05	281.9	Intake	1,500
Transformador	L50	16.5	15.1	1.97	2.99	353.4	Exhaust	-
Patiño	L383	-	-	0.91	12.95	707.1	Intake	870

is only about a half of the quantity of air measured by Catavi mine itself, 7,032 m^3/min, there must therefore be some reason for this difference, but it is not yet clear.

Accordingly, the entire quantities of inlet and outlet cannot be grasped, and as the distribution of levels and cross cuts is very complicated, the total ventilation system can hardly be grasped and calculation of the ventilation network is impossible.

(2) Ventilation in block caving

In block caving stopes, air is largely soiled with blasting fume, dust, etc., in addition there cannot be enough performed by natural ventilation, therefore, mechanical ventilation by large fans is carried out.

Fresh air is blown into the grizzly levels of each block, and blasting fume and dust are drawn out from each grizzly via a ore-pass by a fan in the exhaust drift of the lower level and exhausted into a goaf via a horizontal drift and a vertical shaft exclusively used for exhausting.

An example of the ventilation system for a block caving is shown by that of a block 5–D in Fig. 2–19. The result of measuring the quantity of air flow in this block 5–D is shown in Table 2–36.

As shown by this table, while the quantity of inlet from L411 is 1,431 m^3/min (②+③) and the quantity of inlet from L516 is 1,318 m^3/min. (④), the quantity of outlet from L551 is only 1,428 m^3/min. (①), about a half of the quantity of inlet is therefore thought to be escaping to the goafs of block caving or to the old principal goaf.

The rated quantity of the fan in L551 is 3,396 m^3/min, but it only exhibits a capacity of 1,428 m^3/min. Hence, air pressure, the load current, the forms of vanes, etc., were examined.

The result of the examination is shown in Table 2–37.

Table 2–36 Actual Air-quantity in BLOCK 5-D

No.	Location	Level	Average Velocity of Air Flow (m/sec)	Sectional Area (m^2)	Air-quantity (m^3/min)	Arrangement of Air Flow	Note
①	Flont of Fan for Exhaust	L551	7 0	3.40	1,428	Exhaust	Exhaust from L481
②	Flont of Fan for Blowing	L411	3.6	3.91	845	Intake	L295 → L411 → L481
③	Flont of Fan for Blowing	L411	3 09	3 16	586	Intake	L295 → L411 → L481
④	Flont of Fan for Blowing	L516	6.78	3.24	1,318	Intake	L650 → L516 → L481
⑤	Shaft-mouth for Blowing	L481	5 6	4 32	1,451	Intake	L516 → L481
⑥	Shaft-mouth for Blowing	L481	0.71	2.73	116	Intake	L411 → L481
⑦	Shaft-mouth for Blowing	L481	2 31	3.0	416	Intake	"
⑧	Shaft-mouth for Blowing	L481	4.24	3 1	789	Intake	"
⑨	Rear of Fan for Exhaust	L551	419 m/min	3.41	1,433	Exhaust	Checked by Low-speed Anemometers

Table 2–37 Actual Performance of Exhaust Fan No. 1

Item	Air-quantity (m^3/min)	Air-pressure (mmAg)	Voltage (V)	Loading Electric Current (A)	Note
Theoretical	3396	127	440	150	
Actual	1428	60.3	440	100	

The result shows that the quantity of air flow, air pressure and the load current are lower than their rated values. The causes of this can be attributed to the following but are not very clear.

1) Aluminum cast vanes made in Catavi mine have been worn by dust.

2) The burnt motor was repaired by rewinding in Catavi mine, but the number of revolutions may have been dropped by a connection mistake. (The number of poles may have been increased.)

. . . (In this point, it was impossible to measure the number of revolutions so the real condition of the motor could not be found to my regret.) However, if the number of revolutions is halved, power consumption must be dropped to $1/2^3 = 1/8$, hence, the current value, 100A, is too large from this viewpoint, showing a contradiction.

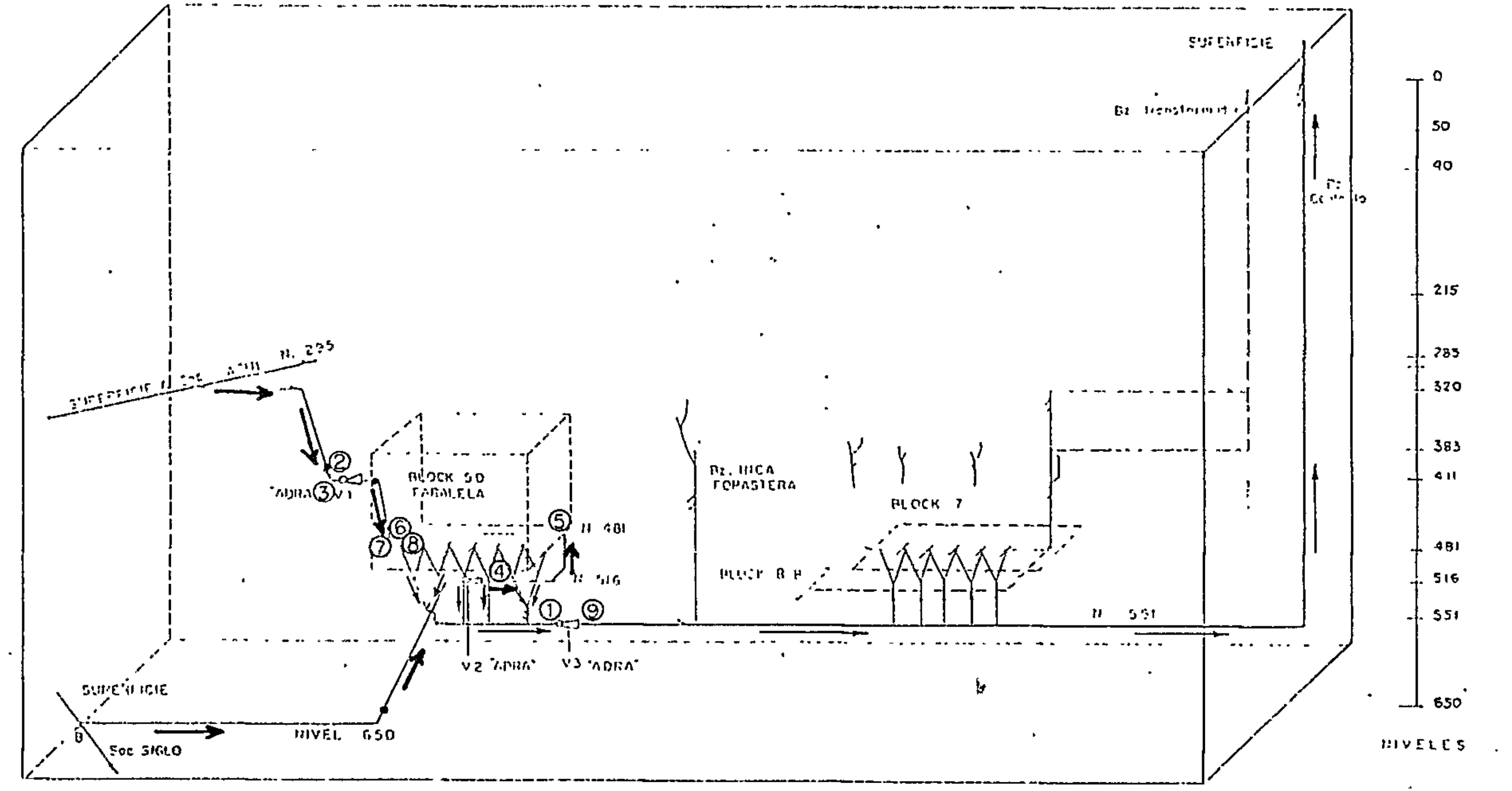


Fig. 2–19 Ventilation System of Block 5–D

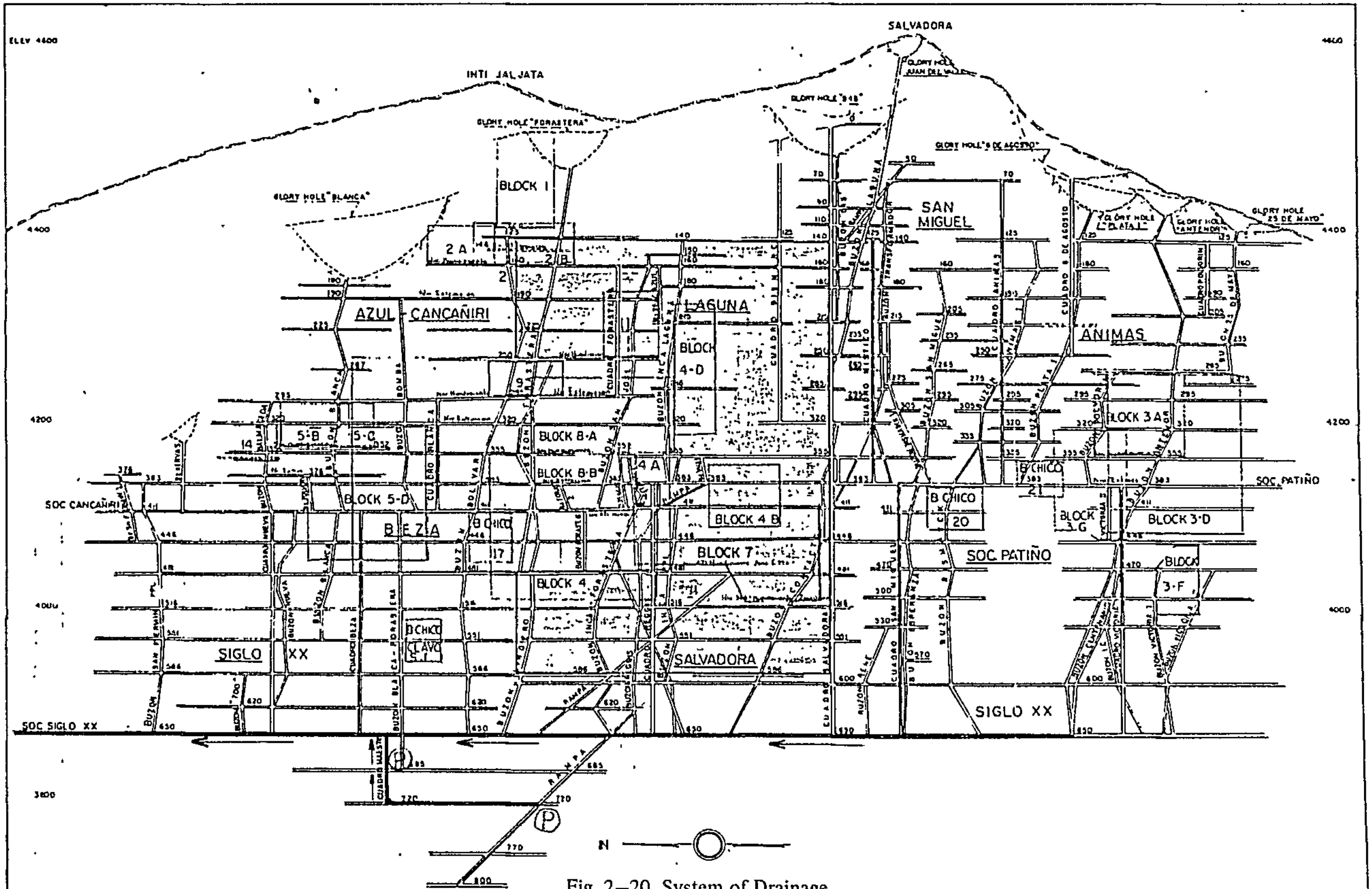


Fig. 2—20 System of Drainage

2–6–3 Drainage of Mine Water

The total amount of mine water of the Catavi mine is small and mostly is mine water from deep levels (below L650). Since there are many disseminated zones of pyrite in the deep levels, the quality of water shows strong acidity of PH. 1.5 to 2.0

The amount of drainage is 0.87 m^3/min (52.15 m^3/hr), and mine water is relayed from the pumping room of L720 to that of L685, through the drain of L650 and discharged by gravity flow. Fig. 2–20 shows the circuit of drainage. This strong acid mine water is used as flotation water in the Siglo XX preconcentration plant.

In addition to reopening of deep level, a plan has been made to drain it by installing a large pump in the lower inclined shaft that was flooded in February, 1980 on account of pump trouble.

2–7 Labor Control and System of Payment

2–7–1 Labor Control

1) Disposition of daily work

Laborers include those hired on 4 piece-work basis and those on hourly basis, the former form groups each consisting of four members, and one group is in charge of a set of faces in underground. (Here, there are no cases in which another piece-work group works in the same set of faces.)

The groups are formed as follows.

(a)	Head (ex-driller)	44.25 $b/man
	Timberer	41.35 "
	Assistant Driller	39.85 "
	Assistant	35.65 "
(b)	Head (ex-timberer)	44.25 $b/man
	Driller	41.35 "
	Assistant Driller	39.85 "
	Assistant	35.65 "

In Catavi mine, rocks are hard and drilling and blasting works are mainly part at the underground. Accordingly, the wages for drillers and timberers are the highest. The amount of money represents the basic wage (hourly payment) for each kind of labor.

A set of faces include shrinkage stopes, drifts, raises, etc., and four piece-work laborers always try to work at some face on the other.

These piece-work groups attend the first shift and carry out drilling and blasting. In the second or the third shift, other hourly laborers enter and carry out the mucking of ore and waste. When piece-work groups carry out mucking in the first shift, piece-work payment on the basis of unit price multiplied by amount of work is payed to them.

Four group members carry out drilling, timbering or mucking in faces regardless of their occupation. Similarly, hourly laborers often carry out the work other than designated by their occupation.

2) Attendance check

(1) State of attendance

Entrance time is referred to 2–1–2.

Each person has two cards of different colors, one for odd number days and other for even number days. After entering underground in the morning, he receives his card of the day at the underground office (in exchange for his round register card), and submits it to the foreman under whom he is working. The foreman signs the card and writes the laborer's name of each face in the work disposition note. When departing, each laborer receives his card, submits it to the Attendance Card Keeper and receives his round register card, and go home. The Attendance Card Keeper of the next shift returns to the office with the cards of the preceding shift, checks attendance and keeps the results as the basic data for wage calculation. The Attendance Card Keepers also must report on duty in the three shifts.

Working Time Schedule

5:30 hrs	Entrance (Locomotive)
6:00 hrs	Arrives at underground office, submits complaints about the tools, explosive of the day before.
7:00 hrs	Receives disposition of work and also receives the tools, explosives, etc., for the day.
11:00 hrs	Ends work in the morning. Lunch
11:30 hrs	Begins work in the afternoon.
14:15 hrs	Departure (Locomotive)

(2) Rate of attendance

The state of attendance of the mining section is shown in following table. As seen in the table, the rate of attendance of underground laborers is below 80%.

For underground laborers, the rate of absence without notice is highest, and that owing to illness is next to it. For the surface laborers, annual vacation, absence without notice, absence owing to illness show almost the same rate.

2–7–2 System of Payment

1) Standard wages for various occupation

Standard wages for various occupation are shown in Table 2–38.

Table 2–38 Classification of Basic Wages

(1 $US per 24.51$b)

Staff	$b/Month	Labor	$b/Man-shift	Labor	$b/Man-shift
Superintendent	22,100	Crew-Foreman	44.25	Signalman	35.65
Chief of Section	11,880	Driller	41.35	Boringman	44.25
Sub-Chief of Section	9,720	Carpenter	41.35	Support-Boringman	41.35
Chief of Foreman	2,463	Support-Driller	39.85	Samplingman	31.60
Foreman	1,911	Lamero	38.25	Trackman	39.85
Secretary	1,335	Locomotive-Operator	39.85	Guarder	32.90

2) Treatment of piece-work labors and hourly labors

Piece-work groups consisting of a group of four members carry out all kinds of works such as drilling, timbering, mucking, etc., as piece-works, but when amount of working per month was little, the work is regarded as "Falla" and hourly wages (standard wages for various kinds of occupations) are paid. In other words, when the state of the work is:

Amount of working x unit price < hourly wage x monthly operation man shift. The working per month is regarded as "Falla" this change is made automatically at the office.

Even when hourly laborers carried out piece-work jobs, only the hourly wages are paid.

3) Piece-work laborers and unit prices

Piece-work laborers (piece-work payment) are widely employed underground and on the surface. The following underground jobs are carried out by piece-work laborers and the norm is set.

(i) Development of drifts (level, incline, raise, shaft sinking)

(ii) Shrinkage stopping, block caving

(iii) Independent work (timbering, shute arrangement, grizzly setting, rehabilitation of old drifts, mucking, transportation by locomotive filling, widening, etc.)

The unit wage is not revised periodically. It seems that the wage is determined by the power struggle between the union and the company. Nearly all of the present unit wages are those of the agreement in 1978, but only the unit wages of block caving were revised in

October, 1979 and increased about 20% over the previous wages. At the same time, it was agreed to add extra wage of 20% universally to all the piece-work payments.

This is the reason for the sudden increase of mining costs in 1980 compared with those of other sections.

The unit wages for main piece-work labors are shown in Table 2–39.

In the above tables, the unit wages of block caving included A wages alone in the past, but the payment system was revised in October, 1979, to use B wages when the monthly ore production of total blocks passed 69,000 tons. Actually, the production is usually 4,000 t/day x 25 days = 100,000 ton/month, accordingly B wages are always used.

4) System of payment

(i) Hourly laborers – standard wage for job x man shift + allowances (overtime, nightshift, attendance bonus, family, annual vacation, etc.)

(ii) Piece-work laborers – sum of all piece-work x unit wages + allowances (overtime, nightshift, attendance bonus, family, annual vacation, etc.)

In the case of piece-work labors, total piece-work payment is employed i.e., piece-work payment for a particular group is distributed to the members of the group corresponding to the ratio of the standard wages for the job of the member. In the allowances, those for overtime and holiday attendance have been increased by 25%. If a person attends six days in a week, he can get the wage for seven days included Sunday. As the attendance bonus, special addition of 8 – 12 Sb/man shift is paid.

(iii) Average monthly earnings

Piece-work Labor	(1) Block caving section : 390 men x 20.2 man-shift/month		2,034 Sb/mouth
	(2) Animas section: 72 men x 19.2 man-shift/month		2,413 Sb/month
Allowances	Annual vacation, Sunday allowance	260 Sb/month	
	Attendance bonus	190	1,750 Sb/month
	Supplement bonus	1300	
Total			3,784 – 4,163 Sb/month

In addition to the above, a food allowance of about 2,300 Sb/month is paid, accordingly, a labor gets approximately 6,000 Sb/month – 6,500 Sb/month real pay with tax, becoming a very high salaried person in the Catavi area.

2–8 Various Projects in the Mine

Planning reports have been annually prepared at the mine from investigations on

Table 2–39 Price of Piece-work

	Horizontal Development (2.13^{m} x 2.44^{m})			Hard Rock	Medium Rock	Soft Rock
1	Advance Length	0.00 ~ 10.00^{m}	\$b/m	146.65	131.96	117.32
		10.01 ~ 15.00^{m}	"	161.93	142.06	129.84
		15.01 ~ 30.00^{m}	"	183.33	164.97	146.65
		over 30.01^{m}	"	219.97	197.96	175.99

	Raise (1.5^{m} x 1.5^{m})			Hard Rock	Medium Rock	Soft Rock
2	Advance Length	0.00 ~ 10.00^{m}	\$b/m	146.65	131.96	117.32
		10.01 ~ 15.00^{m}	"	183.33	164.97	146.65
		over 15.01^{m}	"	219.97	197.96	175.99

	Shaft Sinking (4.0^{m} ~ 2.0^{m})			Drilling and Blasting	Mucking
3	Advance Length	0.00 ~ 10.00^{m}	\$b/m	687.43	687.43
		⋮		⋮	⋮
		30.01 ~ 40.00^{m}		916.60	916.60

Bonus Wages		
	10.01 ~ 20.00^{m}	10 %
	20.01 ~ 30.00^{m}	15 %
	⋮	⋮
	over 40.01^{m}	25 %

4 Shrinkage Stopping (\$b/m^{3})

Width of Stop (m)	Development Area 0 ~ 80 m^{2} Hard Rock	Medium Rock	Soft Rock	80.01 m^{2} ~ 120 m^{2} Hard Rock	Medium Rock	Soft Rock
0.6	80.34	71.87	63.92	110.43	99.40	88.33
0.7	61.51	54.99	48.92	84.59	76.11	67.66
⋮	⋮	⋮	⋮	⋮	⋮	⋮
1.20	24.66	22.07	19.62	33.90	30.44	27.11

5 Block Caving (\$b/Ton)

Tonnage per Man-shift	Tonnage of Total Block Caving A. 0 ~ 69,000 Ton	B. over 69,001 Ton
14 ~ 15	4.14	5.38
19 ~ 20	7.67	9.97
24 ~ 25	9.44	12.27
29 ~ 30	10.10	13.13

economies and processes of new plans such as exploitation plans of new mining blocks, exchange plans of shafts, augmentation plans of transport capacities and so forth. This section deals with development, exploitation and operation plans which were recently planned at Catavi mine as well as those put into practice now.

2–8–1 Underground Development

1) New Beza shaft and exploitation plan of leavings ore of Blanca vein (prepared in December, 1977).

(1) Outline

The present Beza Shaft, located 2,450 meters from Siglo XX mine mouth has been used since a long time ago. Its vertical height is 250 meters, ranging from L650 to L411. The shaft is a passage to Siglo XX area and Beza area. The function of this shaft is very important because the shaft is linked with Nos. 4, 8–B and 5–D faces of block caving, and each level in Paralela and Salvadora areas.

Some parts of the shaft are now under fairly great lateral pressures caused by former exploitations of Blanca vein and San Fermin vein, so that an immediate reinforcement is required to maintain the shaft. Furthermore, there is a considerable displacement in the axis of the shaft, and some deformed side walls can be seen in the midle level (L446 to L551) of the shaft. When exploitations of leavings ore in San José and San Fermin veins are carried out, the effect of caving will reach the upper part of the shaft, resulting in the damage of the shaft itself.

Because of the above-mentioned dangerous conditions, a new shaft has to be constructed for immediate exploitation of leavings ore.

(Outline of the New Shaft)

a) Length : 250 meters (L650 to L411). (A lot of work associated with shaft construction will be required).

b) Size : 3.8 meters x 2.6 meters

c) One two-storied cage

d) Level : Four
Total length of drifts : 803 meters

e) Rope incline to the head sheave : 20 meters

f) Plat at each level and widening of hoisting room : 2,540 cubic meters in total

(Facility)

All arrangements of the new shaft resemble those of San Miguel Shaft, including compressed air pipings, high voltage wire and telephone lines. Main materials necessary and their respective amounts are as follows:

60 lbs. (27 kg.) rails	400 m
40 lbs. (18 kg.) rails	1,070 m
30 lbs. (14 kg.) rails	2,650 m
Trolley wire	2,057 m
Piping 8B	250 m
Piping 6B	300 m
Piping 4B	1,200 m
Piping 2B	2,050 m
High voltage wire	300 m
Telephone line	500 m

(2) Purpose

The purpose of the new shaft is to exploit the leavings ore in the existing area of the shaft and to supply it to the concentrator. The shaft also serves as an access to exploitation in the Northern Deposit (Siglo XX and Beza areas).

(3) Application

The new shaft, after completion of construction, will realize an exploitation of so much ore as 139,013 tons of ore reserve, the grade of which is 0.8%. The amount of Sn metal will be 1,110.14 tons , which in turn can be converted into 532.87 tons of tin with an assumption of 48% recovery as follows:

1,110.14 x 0.48 = 532.87 (Sn tonnage)

The figure can be changed into pound scale as follows:

532.87 (ton) x 2,204.62 (lbs./ton) = 1,174,769.68 (Sn lbs.)

(4) Value of ore

When the exploitation in Beza Shaft Area and 4.75 dollars metal price per pound weight are taken into consideration, the whole value will be:

1,174,769.68 (Sn lbs.) x 4.75 ($/lbs.) = 5,580,156 ($)

(5) Cost

Work	Quantity	Unit Price ($)	Amount ($)
Shaft	250 m	1,500.00	375,000
Main level (L650, L411)	732 m	145.17	106,264
Drift (each level)	803 m	76.65	61,550
Rope incline	20 m	157.80	3,156

Widening	2,540 m^3	7.95	20,193
Subtotal			566,163
Reserve	20 %		113,233
Total			679,396

(6) Depreciation and interest

Total investment	679,396 $
Annual interest (6%)	121,707 $/year
Depreciation (7 years)	851,949 $

(7) Comparison between cost and profit

Economies of shrinkage . stoping and exploitation of old filling:

Total quantity of Sn metal (calculated)	1,110.14 ton
Total recovery	48 %
Metal price	4.75 $/Sn lbs.
Production income	5,580,156 $

Running cost

Shrinkage stoping

Exploitation	28,410 (t) x 3.94 ($/t) = 111,935 ($)
Preconcentration	28,410 (t) x 1.77 ($/t) = 50,286 ($)
Concentration	11,364 (t) x 5.55 ($/t) = 63,070 ($)
Regalias (Mining Tax)	486,582 (lbs) x 1.1020 ($/lbs) = 536,213 ($)
T/C & R/C	486,582 (lbs) x 0.8923 ($/lbs) = 434,117 ($)
Administrative expenses	486,582 (lbs) x 0.6706 ($/lbs) = 236,302 ($)
Tax	486,582 (lbs) x 0.3528 ($/lbs) = 171,666 ($)
Subtotal	1,603,589 ($)

Exploitation of old filling

Exploitation	114,902 (t) x 2.23 ($/t) = 256,231 ($)
Preconcentration	114,902 (t) x 1.77 ($/t) = 203,376 ($)
Concentration	45,961 (t) x 5.55 ($/t) = 255,083 ($)
Regalias	737,137 (lbs) x 1.1020 ($/lbs) = 812,325 ($)
T/C & R/C	737,137 (lbs) x 0.8923 ($/lbs) = 657,747 ($)
Administrative expenses	737,137 (lbs) x 0.6707 ($/lbs) = 494,324 ($)
Tax	737,137 (lbs) x 0.3528 ($/lbs) = 260,062 ($)
Subtotal	2,939,148 ($)
Subtotal of running cost	4,542,737 ($)
Shaft sinking cost	851,949 ($)
Total cost	5,394,686 ($)
Production	5,580,156 ($)

Profit 185,470 ($)

Profitability of the plan : $\frac{5,580,156}{5,394,686} = 1.03 > 1$

Construction progress : The construction progress as of December 30, 1977, has covered 71.11 percent of the whole plan.

Machinery – None

Other facilities – None

Investment :	Up to November 30, 1977	128,448.70 ($)
	December, 1977 to December, 1978	575,000 ($)
	Total	703,448.70 ($)

(8) Notice

Unless the plan is put into practice in a hurry, there is an anxiety that the exploitation in the northern deposit areas may fail for continuation.

In addition, care must be taken to the fact that the cost does not include expenses for machines, and the operational cost of a full face cutting machine for pilot raise.

2) Plan for main haulage level – L650 (Prepared in December, 1980)

(1) Outline

The size of the present main haulage level at L650 is the smallest for a drift and an access to carry 7,000 to 7,500 tons are per day with mine tubs. Consequently, in prolongation of 1,728 meters, a drift has to be provided in parallel with the only drift at present having a 3.0 m x 3.0 m cross-section.

The actual work is now under way with four workers per two faces and a length of 918 meters has been excavated.

(2) Purpose

The new drift, when completed, with make it possible to transport a large quantity of ore, along with the plans for two big block cavings Central and Paralela.

(3) Application

The completion of this new haulage level will make it possible to carry ore of the large exploitation from block caving Central (the southern side) where the ore reserve is 25,772,361 tonnes with 0.33% grade and 85,719.61 Sn metal.

When the recovery during preconcentration and concentration processes is 50 percent, then 85,719.61 (t) x 0.5 = 42,859.80 (Sn, t) which can be converted into pound scale as follows:

42,859.80 (t) x 2,204.62 (lbs/t) = 94,489,583 (Sn, lbs)

(4) Value of ore

When the metal price of tin in 1981 is assumed to be 5.65 $/lbs, then the value of ore obtained from this block caving will be:

94,489,583 (Sn, lbs) x 5.65 ($/lbs) = 533,866,144 ($)

(5) Preparation for exploitation

The following is a trial calculation on the base provided by the Accounts Section of Catavi mine.

Excavation of main haulage level	810 (m) x 145.80 ($/m) =	118,098 ($)
Horizontal and vertical development	11,208 (m) x 125.58 ($/m) =	1,405,501 ($)
Widening of cone	42,224 (m³) x 10.73 ($/m³) =	453,063 ($)
Widening of pillar	28,286 (m³) x 10.73 ($/m³) =	303,509 ($)
Isolation stope	34,200 (m³) x 14.38 ($/m³) =	491,796 ($)
Subtotal		2,773,967 ($)
Reserve	20 %	554,793 ($)
Reinforcement	12,000 (m³) x 51.50 ($/m³) =	618,000 ($)
Machinery and equipment		250,000 ($)
Total for preparation		4,196,760 ($)

(6) Depreciation and cost

Total investment	4,196,760 ($)
Annual interest (6%)	996,311 ($)
Depreciation (5 years)	4,981,554 ($)

Running cost

Exploitation	25,772,361 (t) x 5.31 ($/t) =	136,851,237 ($)
Preconcentration	25,772,361 (t) x 4.02 ($/t) =	103,604,891 ($)
Concentration	10,308,994 (t) x 8.74 ($/t) =	90,100,170 ($)
Regalias	94,489,538 (lbs) x 0.6235 ($/lbs) =	58,914,255 ($)
T/C & R/C	94,489,583 (lbs) x 2.0832 ($/lbs) =	196,840,699 ($)
Administrative expenses	94,489,583 (lbs) x 0.4233 ($/lbs) =	39,902,950 ($)
Total		626,214,202 ($)

(7) Profitability

Running cost	626,214,202 ($)
Development cost	4,981,554 ($)
Total cost	631,195,756 ($)

Value of ore	533,866,144 ($)
Loss	97,329,612 ($)

Profitability of the project :

$$\frac{\text{Recovery cost of ore}}{\text{Cost for development and exploitation}} = \frac{533{,}866{,}144}{631{,}195{,}756}$$

$$= 0.846 < 1$$

Work progress: By December 31, 1980, 64% of the whole work has been completed. The work will be finished by 1982.

1982	1980	918 m
	1981	810 m
	Total	1728 m

Investment :	Up to December 31, 1980	230,967.22 ($)
	January to December 1981	184,000 ($)
	Total	414,967.22 ($)

2–8–2 Exploitation

1) Development Plan for mini-block, "San José" (Prepared in December 1977)

In this planning report, the economic efficiency was examined on both "sub-level caving" and "block caving", which has resulted in a conclusion saying the block caving is superior to the sub-level caving.

(With Sub-Level Stoping)

(1) Outline

The mini-block, "San José", is situated in the west of Salvadora vein, and is a deposit existing in the contact portion of greywacks with breccia. The block belongs to Siglo XX, and its scale is:

Length – 51 m, Number of cones	–	16
Width – 43 m, Number of grizzly	–	8
Height – 100 m, Raise, etc.	–	12
Area – 2,193 m^2, Volume	–	219,300 m^3

Preparations for exploitation common to block caving and sub-level caving are executed.

(2) Purpose

The development of this mini-block ensures a sufficient ore supply for concentration and increases the production of the mine.

(3) Application

The calculated ore reserve provided by the Geology Department indicates the following data :

Ore reserve	524,503 tonnes
Grade	0.46 % Sn
Sn metal	2,412.71 tonnes

Should the mean recovery during concentration be assumed as 44 percent, then

2,412.71 (t) x 0.44 = 1,061.59 (Sn, t)

which converts into 2,340,402 pounds.

(4) Value of ore

When a trial estimate is derived from the metal price of 4.75 $/lbs which was effective in 1978, the production of 2,340,402 pounds of the metal indicates the value of ore as follows:

2,340,402 (lbs) x 4.75 ($/lbs) = 11,116,909 ($)

(5) Preparation for exploitation

The following expenses are calculated from the prime cost data provided by the Accounts Section and are necessary for preparation of San José block :

Horizontal development	902 (m) x 76.65 ($/m) =	69,138 ($)
Vertical development	507 (m) x 52.60 ($/m) =	26,668 ($)
Widening of cone	3,280 (m) x 7.95 ($/m^3) =	26,078 ($)
Widening of pillar	4,110 (m^3) x 7.95 ($/m^3) =	32,674 ($)
Isolation stope	4,200 (m^3) x 10.65 ($/m^3) =	44,730 ($)
Subtotal		199,286 ($)
Reserve	20 %	39,857 ($)
Total		239,143 ($)

(6) Machinery and equipment

Rock drills and their accessories, fans and electric materials necessary for electric apparatuses are included.

Rock drill, GD–DH99	2	7,942 ($)
Guide shell	2	5,088 ($)
Remote control, 2MS89Y	2	506 ($)
Shank rod, 1–1/2" RD	2	62 ($)
Coupling, CL–51600 VA	90	1,110 ($)

Rod, 1–1/4" HEX, 4FTCL	50	1,610 ($)
Bit, 2–1/2"	25	932 ($)
Bit, 2–3/4"	25	1,059 ($)
Bit, 3–1/4"	25	1,425 ($)
Subtotal		19,734 ($)
Ventilation equipments		7,500 ($)
Materials and accessories		3,000 ($)
Labor cost		2,500 ($)
Subtotal		32,734 ($)
Reserve	20%	6,547 ($)
Subtotal for machinery		39,281 ($)
Subtotal for preparation		239,143 ($)
Total		278,424 ($)

(7) Depreciation and interest

Investment	278,424 ($)
Interest (6%)	66,098 ($)
Depreciation (5 years)	330,489 ($)

(8) Comparison of profit to cost

Sn metal (calculated)	2,412.71 tonnes
Recovery	44 %
Metal price	4.75 $/lbs
Production	11,116,909 dollars

Running cost

Exploitation	524,503 (t) x 4.01 ($/t) =	2,103,257 ($)
Preconcentration	524,503 (t) x 1.77 ($/t) =	928,370 ($)
Concentration	264,516 (t) x 5.55 ($/t) =	1,368,164 ($)
Regalias	2,340,402 (lbs) x 1.102 ($/lbs) =	2,579,123 ($)
T/C & R/C	2,340,402 (lbs) x 0.8923 ($/lbs) =	2,088,341 ($)
Administration expenses	2,340,402 (lbs) x 0.6706 ($/lbs) =	1,569,473 ($)
Tax	2,340,402 (lbs) x 0.3528 ($/lbs) =	825,694 ($)
Total		11,462,422 ($)

(9) Profit and loss account

Running cost	11,462,422 ($)
Development cost	330,489 ($)

Total cost	11,792,911 ($)
Production	11,116,909 ($)
Loss	676,002 ($)

Profitability of the project :

$$\frac{\text{Recovery value of ore}}{\text{Total cost required for exploitation and development}} = \frac{11,116,909}{11,792,911} = 0.94 < 1$$

Conclusion : Reject

(With Block Caving)

A block caving system is used for preparation of the mini-block, where items (1), (2), (3) and (4) are the same as in the sub-level caving.

(5) Preparation for exploitation

The following expenses are calculated from the prime cost data provided by the Accounts Section and are necessary for preparation of San José Block:

Horizontal development	586 (m) x 76.65 ($/m)	=	44,917 ($)
Vertical development	399 (m) x 52.60 ($/m)	=	20,987 ($)
Widening of cone	3,280 (m^3) x 7.95 ($/m^3$)	=	26,076 ($)
Widening of pillar	4,110 (m^3) x 7.95 ($/m^3$)	=	32,674 ($)
Isolation stope	13,090 (m^3) x 10.65 ($/m^3$)	=	139,408 ($)
Coyote blasting	380 (m) x 52.37 ($/m)	=	19,900 ($)
Subtotal			283,962 ($)
Reserve	20%		56,792 ($)
Total			340,754 ($)

(6) Machinery and equipment

Fans, machines with motor facilities and electric accessories are included.

Cost :

Ventilation equipments		7,500 ($)
Materials and accessories		2,000 ($)
Labor cost		1,500 ($)
Subtotal		11,000 ($)
Reserve	20%	2,200 ($)
Subtotal		13,200 ($)
Subtotal for machinery		13,200 ($)
Subtotal for preparation		340,754 ($)
Total		353,954 ($)

(7) Depreciation and interest

Investment	353,954 ($)
Interest (65, 5 years)	84,029 ($)
Depreciation (5 years)	420,143 ($)

(8) Comparison of profit to cost

San metal (calculated	2,412.71 tonnes
Recovery	44 %
Metal price	4.75 $/lbs.
Production	11,116,909 dollars

Running cost

Explcitation	524,503 (t) x 2.12 ($/t) =	1,111,946 ($)
Preconcentration	524,503 (t) x 1.77 ($/t) =	928,370 ($)
Concentration	246,516 (t) x 5.55 ($/t) =	1,368,164 ($)
Regalias	2,340,402 (lbs) x 1.102 ($/lbs) =	2,579,123 ($)
T/C & R/C	2,340,402 (lbs) x 0.8923 ($/lbs) =	2,088,341 ($)
Administrative expenses	2,340,402 (lbs) x 0.6706 ($/lbs) =	1,569,473 ($)
Tax	2,340,402 (lbs) x 0.3528 ($/lbs) =	825,694 ($)
Total		10,471,111 ($)

(9) Profit and loss account

Running cost	10,471,111 ($)
Development cost	420,143 ($)
Total cost	10,891,254 ($)
Production	11,116,909 ($)
Profit	225,655 ($)

Profitability of the project :

$$\frac{\text{Recovery value of ore}}{\text{Total cost required for exploitation and development}} = \frac{11,116,909}{10,891,254} = 1.02 > 1$$

Conclusion : Acceptable

2) Development plan for block caving "5–D" (Remaining third) (Prepared in December, 1980)

(1) Outline

Block caving "5–D" is in the southwest of completely excavated blocks 5–B, 5–C and 14, and lies in the area covering north 19,500 to 19,700 and south 9,800 to 10,000. The vertical effect for caving is restricted between L355 to L481, and the average mining block is as follows:

Length	–	122 m
Width	–	53 m
Height	–	116 m
Area	–	6,460 m^2
Number of cones	–	60
Number of grizzlies	–	30

The development of block caving "5–D" was started in March, 1975, and the progress up to December 30, 1981 is as follows:

Horizontal development	90 %
Vertical development	92 %
Widening of cone and pillar	78 %
Block isolation stope	85 %
Reinforcement	80 %

In the northern area, two thirds of the whole mining block was expected easily to caving while the remaining one third is assigned to No. 2 main ore bin and grizzlies are provided at nine plots; No. 6, 7, 18, 19, 24, 25, 26, 30 and 31. Because the exploitation in this area will affect Blanca Shaft, consideration was to increase the height of isolation stope and to form a degraded zone in parallel with the goaf and old stoping, thus eliminating the effect on Beza area. According to the preparation schedule, blasting work will be completed in December, 1981.

(2) Purpose

The exploitation of block caving "5–D" will hold the production from Catavi mine in its normal state, and will become a face which takes the place of 8–B Block. Then the stoppage of ore supply to pre-concentration can be avoided.

(3) Application

The fundamental data on this block have been prepared by the Geology Department, which shows that the minable ore reserve is 2,161,482 tons with a 0.44% Sn grade and 9,519 tonnes of Sn metal. Should the recovery during concentration be 50 percent, the figure will be:

$$9{,}519 \text{ (t)} \times 0.5 = 4{,}759.50 \text{ (t)}$$

This is equal to 10,492,889 pounds of tin.

(4) Value of ore

With an assumption of 5.65 dollars of Sn metal price in 1981, the exploitation of

10,492,889 pounds will be valued as follows:

10,492,889 (lbs) x 5.65 ($/lbs) = 59,284,823 ($)

(5) Preparation for exploitation

The cost necessary for preparation of the remaining one third of Block "5–D" in 1981 has been estimated on the basis of actual cost records stored by the Accounts Section. According to the estimation, the following investment will be required:

Horizontal development	150 (m) x 145.80 ($/m)	=	21,870 ($)
Vertical development	100 (m) x 68.21 ($/m)	=	6,820 ($)
Widening of cone	200 (m^3) x 10.73 ($/$m^3$)	=	2,146 ($)
Widening of pillar	700 (m^3) x 10.73 ($/$m^3$)	=	7,511 ($)
Isolation stope	800 (m^3) x 16.51 ($/$m^3$)	=	13,208 ($)
Coyote blasting	300 (m) x 125.58 ($/m)	=	37,664 ($)
Subtotal			39,220 ($)
Reserve	20%		17,844 ($)
Reinforcement	600 (m^3) x 51.50 ($/$m^3$)	=	30,900 ($)
Total			137,964 ($)

(6) Machinery and equipment

Fan (1,698 m^3/min capacity) – 1

Transformer, switch board and wiring cable are included.

Cost:

Machinery	15,000 ($)
Equipment	20,000 ($)
Material	30,000 ($)
Labor	15,000 ($)
Reserve (20%)	19,000 ($)
Total for facility	99,000 ($)
Total for preparation	137,964 ($)
Total	236,964 ($)

(7) Depreciation and interest

Investment	236,964 ($)
Interest (6%, 5 years)	56,255 ($/years)
Depreciation (5 years)	281,276 ($)

(8) Comparison of profit to cost

Sn metal (calculated)	9,519 tonnes

Recovery	50 %
Metal price	5.65 $/lbs.
Production	59,284,823 dollars

Running cost:

Exploitation	2,161,482 (t) x 5.31 ($/t)	= 11,477,469 ($)
Preconcentration	2,161,482 (t) x 4.02 ($/t)	= 8,689,158 ($)
Concentration	864,593 (t) x 8.74 ($/t)	= 7,556,543 ($)
Regalias	10,492,889 (lbs) x 0.6235 ($/lbs)	= 6,542,316 ($)
T/C & R/C	10,492,889 (lbs) x 2.0832 ($/lbs)	=21,858,786 ($)
Administrative expenses	10,492,889 (lbs) x 0.4223 ($/lbs)	= 4,431,147 ($)
Total		60,555,419 ($)

(9) Profit and loss account

Running cost	60,555,419 ($)
Development cost	281,276 ($)
Total cost	60,836,695 ($)
Production income	59,284,823 ($)
Loss	1,551,872 ($)

(10) Profitability of project:

$$\frac{\text{Recovery value of ore}}{\text{Total cost required for exploitation and development}} = \frac{59{,}284{,}823}{60{,}836{,}695} = 0.974 < 1$$

Development schedule and cost of Block "5–D"

Jan. to May, 1981

Month	Advance (m)	Widening of cone (m^3)	Reinforcement (m^3)	Widening of pillar (m^3)	Isolation stope (m^3)	Cost
Jan.	2.2	969	106	54	116	8,279.57
Feb.	7.4	698	186	60	151	11,277.25
Mar.	20.1	973	141	16	263	13,856.89
Apr.	16.8	965	171	91	155	13,749.68
May	51.9	719	61	36	24	12,184.38

2–8–3 Large Block Caving

In Catavi mine, a report on the possibility of a plan about large block caving exploitation in future was completed in October, 1981. Its abstract is presented as follows.

1) Conclusion

(1) The costs of preparation for exploitation and exploitation of Block Central by Plan B (trackless mining) are lower than those by other plans in the report submitted this time. The exploitation cost (including depreciation cost but not including interest) to the entrance of preconcentration is 5.4 dollars per ton (wet weight).

(2) The ore grade by geological estimation is 0.20% in tin.

(3) According to the result of economic analysis for the operation of processing 12,000 ton (dry weight) ores supplied per day to the preconcentration, the operation is said to be wholly unprofitable.

(4) A fact found from the sensitivity analysis of this plan is that profit can be gained when the recovery of tin is above 82.8% or processing cost per ton (dry weight) is below 4.46 dollars.

(5) When the ore grade is 0.20% and the recovery is 25%, profit cannot be gained if the metal price of tin is not 24.1 dollars per pound or more.

(6) The lowest economic ore grade for tin is 0.414% when there is no payment of interest.

(7) After all, the mining plan of block caving in Catavi mine will not be able to gain profit because of the lower grade of the veins and the lower recovery.

2) Outline

To maintain the mining operation of Catavi mine, a mining plan in the main blocks of Siglo XX was proposed. As a result of independent research since 1974, Block Paralela and Block Central were selected as object blocks and have been given particular attention to the F.S. since 1979 through detailed geological estimation and for preparing the mining plan.

This report is focused on the mining plan, describing the development of blocks selected beforehand through geological estimation, preparation for exploitation and exploitation plan drawings, selection of machines and the judgement of investment values. Moreover, economical analysis of this plan and various sensitivity analysis which varied each factor were made.

3) Investigation of mining method in block caving

The block caving system consists of various levels, the main ones being undercut levels, grizzly levels (stoping levels) and haulage levels.

The outline of main blocks are as follows:

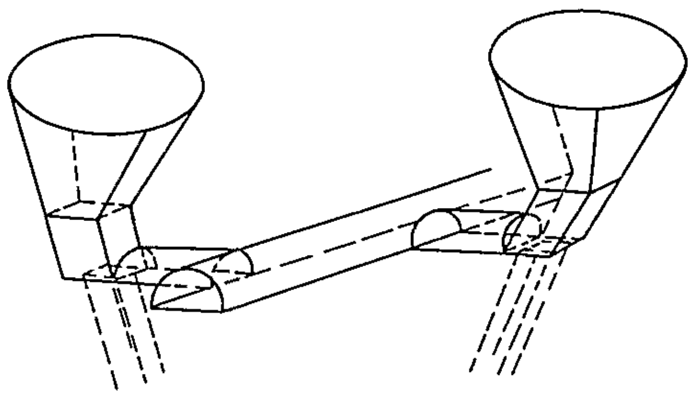
	Block Central	Block Paralela
Undercut level	L551	L481
Grizzly level	L567	L497
Haulage level	L650	L650

Exploitation area	75,848 m^2	55,313 m^2
Ore reserve	40,000,000 ton	17,000,000 ton

Various plans about a method to drop broken ores to a haulage level were examined. The plans are explained in the following.

(Plan A)

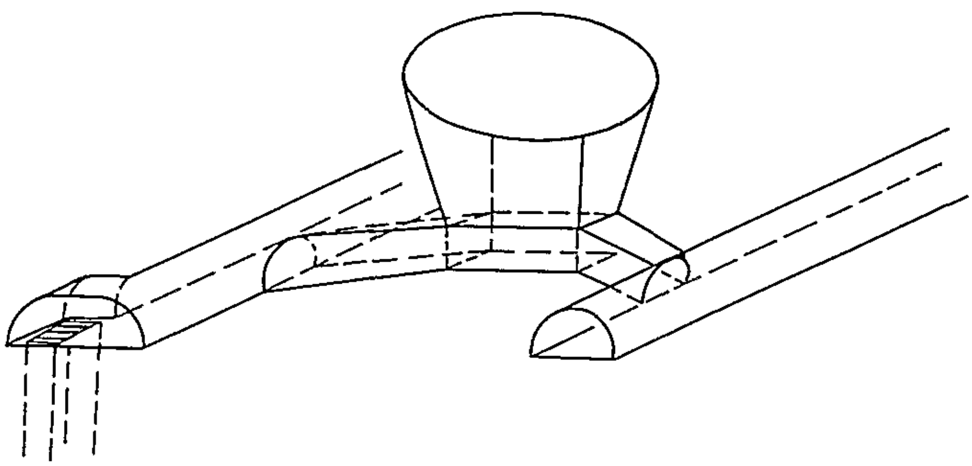
This plan uses the conventional method which has been applied for block caving. A raise of 6 m long is excavated upwards at an inclination of 45° from a grizzly level, and then excavated vertically until it reaches an undercut level. Next, this vertical part is widened into a cone shape whose top diameter is 10 m. The plan A is shown schematically in the following figure.



(Plan B)

In this plan, most of the finger raises and conventional ore passes are omitted. Cones of a diameter over 12 m and machines for hauling from the cones to ore passes are required. In addition, rock breakers are used.

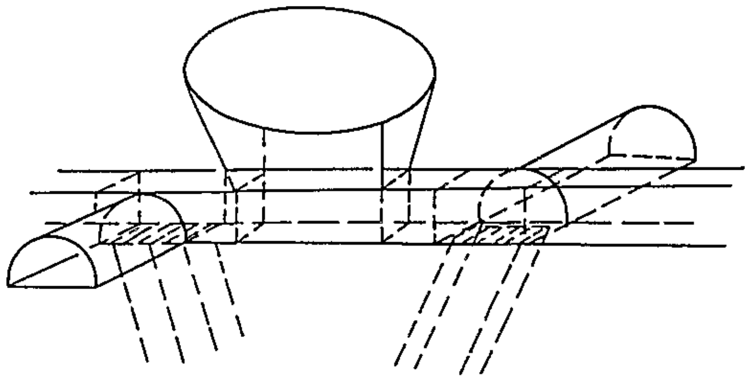
The Plan B is shown schematically in the following figure.



(Plan C)

In this plan, grizzlies are arranged between cones. It is the combination of plan A and plan B. Ore is dropped directly into ore passes by gravity, and also impact type rock breakers are used.

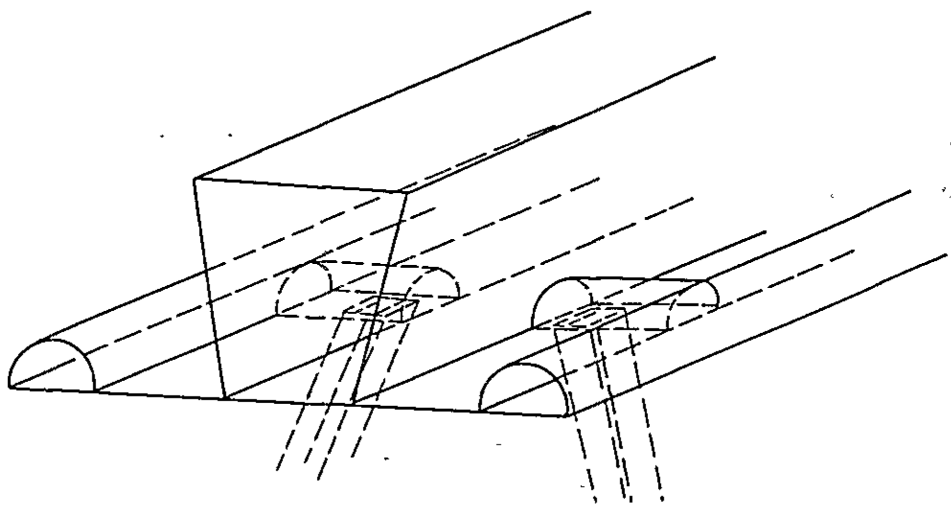
The plan C is shown schematically in the following figure.



(Plan D)

In this plan, the cone of each column ends in a groove form on the spot. In other words, cones are forming the continuous shape with an inverted trapezoidal cross section.

The plan D is shown schematically in the following figure.



From the above-mentioned results of investigation, plans A and D were excluded and plans B and C alone were investigated in detail.

4) Plan B (Trackless Mining)

In the plan B, a "Scoop Tram" with a front end loader of bucket capacity 3.8 m^3 is used, and an impact type rock breaker to break rocks into sizes which can be easily handled is used.

The upper part of a cone is 12 m in diameter and is connected with a cross cut of W 3.6 m x L 10 m, and the bottom of cone is the same level to stoping one. The cross cut is 14 m in length and intersects with a haulage level at an angle of 45°. LHD passes through this level and enters the bottom of cone. Shutes are provided for the haulage level at an interval of about 100 m. These shutes are concentrated to L620 in Block Central and to L583 in Block Paralela and connected to L650 or the main haulage level.

The quantities of excavation are as follows.

			Block Central	Block Paralela	Total
Development	Horizontal development		15,442 m	14,520 m	29,962 m
	Vertical development		4,727 m	4,606 m	9,333 m
Preparation for	Number of cones		140	112	252
Exploitation	Widening of cones		84,000 m^3	67,200 m^3	151,200 m^3
	Isolation shrinkage	Number of stopes	16	8	24
	Quantity of excavation		37,100 m^3	18,320 m^3	55,420 m^3
Widening of pillars			726,000 m^3	525,000 m^3	1251,000 m^3
Shute arrangement			21	16	37
Reinforcement work			3,484 m^3	2,786 m^3	6,270 m^3

5) Plan C (Combined Method)

A corss cut is excavated between cones from a haulage level to the bottom position of the cones, while a drift of the same size as the bottom width of the cones is excavated in parallel with the haulage level towards the bottom of the cones. A grizzly is made between cones. By introducing impact type rock breakers, the number of shoot blasting man is reduced and the use of hammers is eliminated.

The quantities of excavation are as follows:

			Block Central	Block Paralela	Total
Development	Horizontal development		19,014 m	15,081 m	34,095 m
	Vertical development		13,775 m	15,335 m	29,110 m
Preparation for	Number of cones		204	154	358
Exploitation	Widening of cones		119,544 m^3	90,244 m^3	209,788 m^3
	Isolation shrinkage	Number of stopes	16	8	24
	Quantity of excavation		37,100 m^3	18,320 m^3	55,420 m^3

Widening of pillars	715,901 m^3	518,363 m^3	1234,264 m^3
Shute arrangement	37	23	60
Reinforcement work	3,583 m^3	2,222 m^3	5,760 m^3

6) Period of preparation for exploitation and number of persons required (Plan B)

Block Central	Period (Month)	Number of persons	Number of rock drills
L650 main haulage level	14	18	3
L620 connecting level	28	24	4
L586 connecting level	15	4	1
L567 exploitation level	39	50	10
L551 undercut level	38	40	9
Subtotal	39	136	27
Spare (20 %)	8	28	5
Total	47	164	32

About four years will be required for preparing exploitation at Block Central.

7) Operating costs for plans

The costs of plan B and plan C not including the payment of interest for loan are compared as follows.

	Plan B ($1000)		Plan C ($1000)	
	Block Paralela	Block Central	Block Paralela	Block Central
Exploration	2,278	1,400	2,278	1,400
Development	5,324	5,964	6,235	7,632
Preparation for exploitation	12,133	16,439	12,388	18,135
Machine equipment	10,545	10,545	8,745	8,745
Ventilation equipment	4,965	6,301	5,567	6,695
Exploitation	47,260	112,094	88,910	206,851
Ventilation (Operation)	1,510	9,076	1,376	7,269
Haulage	15,300	36,289	15,300	36,289
Reserve fund (10%)	9,931	19,811	14,080	29,301
Cost per ton totaled up to haulage	$6.43	$5.40	$9.11	$7.99

From the above, plan B is more economical. At Block Central, preparation for exploitation will be required four years and exploitation period will be ten years. The estimation of Block Paralela will be carried out while stopping Block Central.

Table 2–40 Production Planning of 6 Years

(Ton)

Year / No. of Stop	1981	1982	1983	1984	1985	1986
Block Caving						
4	325,107	325,107	324,009	156,004	124,802	124,802
5-D	296,722	354,457	456,009	504,010	456,009	456,009
8-B	103,793	89,336	0	0	0	0
3-D	174,460	144,820	51,272	0	0	0
17-A	80,336	80,336	74,402	60,001	0	0
3-F	67,400	71,804	78,005	57,604	57,604	0
20	41,625	45,193	45,601	45,601	45,601	0
4-D	Low Grade	Low Grade	Low Grade	0	0	0
Mini Block Caving				B.C 3-A, 3-G 53,860	B.C 3-A, 3-G 53,860	0
Bayona	Under Preparation	49,500	100,800	100,800	126,000	150,004
Laguna 23	Under Preparation	41,625	80,640	80,645	80,640	80,640
Plata 24	Under Preparation	Under Preparation	100,800	100,800	151,205	223,209
San José	Under Planning	Under Preparation	(198,000)	(100,800)	(100,800)	0
"	Under Preparation	31,224	61,388	69,600	74,404	127,204
8-C	Under Planning	Under Preparation	(198,800)	(100,800)	(100,800)	0
7	–	–	–	77,636	76,801	76,801
Beza	Under Planning	Under Planning	Under Preparation	62,401	126,000	134,257
Central	Under Planning	Under Preparation	Under Preparation	Under Preparation	Under Preparation	(145,600)
Paralela	Under Planning	Under Preparation	Under Preparation	Under Preparation	Under Preparation	
Production beyond Planning	(683,693)	(639,599)	(573,289)	(450,702)	(814,677)	(657,612)
Sub-Total	1,089,443	1,233,852	1,372,926	1,372,926	1,372,926	1,372,926
Open-cut	0	0	139,074	643,074	643,074	643,074
Shrinkage Stopping	398,561	278,148	0	0	0	0
Desmonte	336,000	504,000	504,000	0	0	0
Total	1,824,004	2,016,000	2,016,000	2,016,000	2,016,000	2,016,000
Ore Grade (%)	0.32	0.32	0.32	0.32	0.32	0.32
Sn Metal (Ton)	6,052.34	6,522.52	6,411.31	6,515.52	6,515.52	6,515.52

2–8–4 Production Plan

1) Middle term production program

In Catavi mine the engineers of project section of mining department prepare the plan of middle term production which make one unit three years under the guidance of sub-manager of the mine. The table 2–40 shows an example of it, however, the plan for after four or six years seems to be only idea of one step and beside includes the block Central excavation plan which is not reality. They generally effort to maintain the actual production of tin.

2) Annual production program

As the fiscal year of operation in Catavi mine is January to December, the superintendent of mining department prepares operation program for next year at the end of the year.

3) Monthly production program

Referred to the production program of every month each chief of the sections present at the last ten days of the previous day the results of study on the annual production program arranging and studying with the chiefs of groups.

The superintendent of mining department presents it to the submanager and manager of the mine examinating totally and modifying the production program presented by chiefs of the sections and obtains manager's approval.

The office work section of the mining department prepares a table of integrate production program by each section.

2–9 Tests on Rock Mechanics

2–9–1 Outline

In investigation on mining methods for a mine, mechanical properties of the rock must be made clear. In this connection, well cores brought back from Catavi mine were subjected, as samples, to test on rock mechanics. Tests performed were specific gravity, velocity of elastic wave young's modulus, Poisson's ratio, uni-axial compressive strength and indirect tensile strength.

The tests were carried out in the test laboratory of the Resource Engineering Department of the Engineering Faculty of Yamaguchi University. All the tests were performed under dry atmosphere.

2–9–2 Test Methods

1) Preparation of test pieces

Cores were arranged and fixed vertically in a wooden frame so that individual cores were parallel with one another, and bound into a mass with cement. Core pieces were picked up from the mass by boring. Each test piece was planned with a cutter so that the top and the bottom surfaces became parallel to each other within 5/100 mm in parallelism.

2) Specific gravity

The specific gravity was measured in accordance with JIS (Japanese Industrial Standards), while the significant figure of the results was two digits or so because the test pieces were too small.

3) Uni-axial compressive strength

The dimension and shape of test piece were based on the Standard of ISRM (International Symposium of Rock Mechanics), and the ratio of L (length) to D (diameter), L/D, was set as 2.5 or a little greater.

4) Indirect tensile strength

The dimension and shape of test piece accorded with the Standard of ISRM, and the L/D fell in the range from 0.5 to 1.0

5) Velocity of elastic wave

The measurement was carried out in the following way: A 200 kHz wave from a pulse generator of Wavetek was fed to one PZT–7 element, and the waveform received at another PZT–7 element was stored in DM703 Transient Recorder. The data, as they were digital, were transferred to a computer, and the arrival time was read out from the result recorded by the printer. The minimum readable time was 50 ms (50 x 10^{-9} sec.)

6) Young's modulus and Poisson's ratio

Two 30 mm foil gauges for the axial strain and two 10 mm foil gauges for the radial strain were stuck on the side of the test piece. The data were automatically measured and recorded with a strain meter during the uni-axial compression test.

The result was picturized with a computer and a plotter, and Young's modulus and Poisson's ratio were obtained from the slope of the linear portion of the resultant chart.

2–9–3 Test Results

The test results on rock mechanics are shown in Table 2–41, where the figures on No. 2 test piece are much greater than others. With the exception of the figures of No. 2 test piece, the compressive strength is in a range 780 to 1,120 kg/cm^2, the tensile strength in a range 72 to 130 kg/cm^2 and the velocity of elastic wave in a range 4,000 to 4,500 m/sec, which indicates that the rock is relatively hard.

Table 2–41 Results of Tests in Rock Mechanics

Sample	Kinds of Rock	No. of Boring by Catavi Mine	Test Piece Size (mm)		Specific Gravity	Velocity of Elastic Wave (m/sec)	Tensile Strength (kg/cm^2)	Compressive Strength (kg/cm^2)	Young's Module EX10^5 (kg/cm^2)	Poissons's Ratio
			In Compression	In Tension						
1	Quartz Porphyry	DDH 845-366 L650	D=29 70 L=62 90	D=29.70 L=36 45	2 64	4,500	77 6	780	3 26	0 25
2	Siliceous Sandstone	DDH 782	D=29.30 L=59.80	D=29.70 L=11.15	2.76	5,710	272 4	4,510	8.82	0 19
3	Quartz -Porphyry	DDH 798 L551 A205'		D=14.10 L=10 95	2.8	–	129 6	–	–	–
4	Quartz -Porphyry	DDH 802 A185 L551	D=13.90 L=32 20	D=14 00 L=10 70	2 6	4,100	94 9	1,120	3 58	0 22
5	Sandstone	DDH 782 A285 L551		D=13 95 L= 8.85	3 0	–	72.4	–	–	–
6	Sandstone	DDH 782 A205' L551	D=13 95 L=30 25	D=14.10 L=10 25	2 9	4,400	82.7	790	3 18	0 29

2–10 Ore Sellers

In Catavi mine, there are groups of people about 2,500 in number, who, by permission of the mine, are getting tim in the mining area in a primitive manner and selling it to the mine as high quality tin, thus earning their bread (most of the tin mines in Bolivia have such people).

Although they are working in and out of the mine, their products are generally called other ores (otros fuentes), and account for 50 to 70% of the total production of the mine. They are playing an important role in the production of the Catavi mine.

2–10–1 Organization and Personnel

The types of these ore sellers are mentioned below, and they are each classified into groups of several people to several tens of people.

(1) Locatarios (mining high quality small veins in the mine)

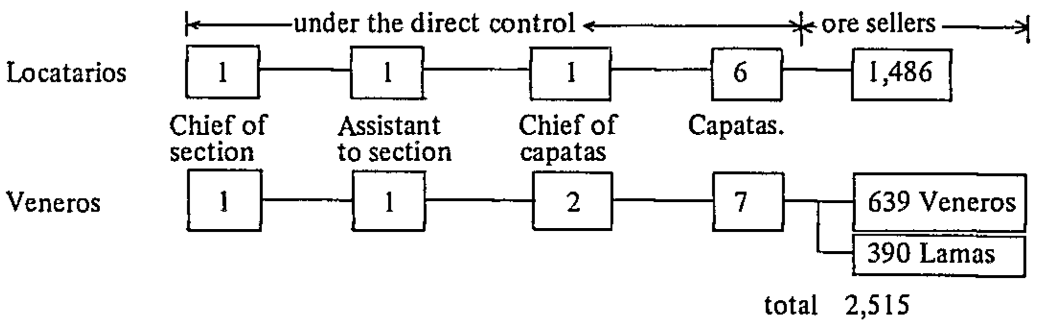
(2) Lamas (recovering tin from the tailing for heavy fluid)

(3) Veneros ⓐ mining high quality veins outside the mine, just under the surface of the earth

ⓑ mining at the placer deposits

(4) Cooperativa (other small groups)

The mining division has an organization consisting of 20 peoples including the chief, who are supervising and controlling these groups. In the Victoria mill plant, there is an organization for purchasing the products (weighing, quality analysis, etc.).



The Catavi Mine, with this supervising organization, is trying to prohibit the labors without permission (controlling the personnel), prevent the theft from the face under the direct control of the mine, check the attendance of individual workers, and also calculate the distribution of ore purchasing bills for individual labor by using a computer. The mine also gives appropriate assistance to the ore sellers, e.g., lending portable compressors.

The face inside the mine which Locatarios are mining is located in upper part of deposit, higher than the level 300, and they are mining veins with poor conditions such as leavings after mining high quality veins or offset veins.

Therefore, their faces are located in the same area as the face of the groups under the direct control of the mine such as Laguma, Animas, so that Locatarios set up a steel door and lock it in the night to prevent theft by others.

2–10–2 Production Change

The ore sellers' production by year is shown in the following tables. The numbers indicate the tin content in selected ores which can be sold.

Table 2–42 Production of Locatarios by Years

(Sn Metal ton)

Year	Locatarios(A)	Catavi Mine(B)	Total	(A)/(B) %
1980	1,662	2,288	3,950	72.6
1979	1,409	2,525	3,934	55.8
1978	1,431	2,959	4,390	48.4
1977	1,616	4,198	5,814	38.5
1976	1,876	3,542	5,418	52.9
1975	2,137	3,968	6,105	53.8

Table 2–43 Production of Locatarios

Year	High Grade Ore (Concentrate)			Low Grade Ore (Crude Ore)		
	Amount of concentrate	Ore Grade (%)	Sn Metal (ton)	Amount of crude ore	Ore Grade (%)	Sn Metal (ton)
1980	2,606.8	37.67	981.9	5,233	1.33	69.9
1979	2,132.9	33.74	719.2	8,075	1.21	97.7
1978	2,207.8	32.74	722.8	13,423	1.91	159.2
1977	2,669.2	34.19	912.5	10,792	1.26	136.0

Table 2–44 Production of Other Locatarios by Year

(Tonnage Sn Metal)

Year	Veneros	Lamas	Cooperativa
1980	448.6	89.2	38.8
1979	580.5	–	–
1978	280.8	–	–
1977	319.3	–	–
1976	309.3	–	–

The Table 2–44 indicates that the production of the Catavi mine has decreased with lowering in ore grade, while the ratio of the tin quantity produced by ore sellers has increased year to year, playing an important role for Catavi mine.

2–10–3 Mining Method

1) Mining in the pit (Actual examples at the level 295 are described)

(1) Group No.15 : 35 personnels, 1 face

The face is located at the lower part of the small vein, and one rock drilling personnel and one assistant were drilling. First the stock part is blasted and then the vein is blasted. The ores are selected, packed in a bag and carried out.

High quality ores : Sn 3–4% to the manual operation jig plant
Sn 0.7–1.4% to Victoria mill plant

(2) Group No.13 : 28 personnels, 1 face

This is located in the vicinity of No.15. Drilling water is carried in with 4 to 5ℓ plastic vessels and poured into a tank over the face to obtain pressure.

(3) Group Dolores : At the Dolores pit away from the main pit, a compressor room is

provided to the approach to the pit. The compressors are made by Ingersol, USA, and the newest one among three compressors was made in March 1905, and has a 60 Hz, 125 HP motor produced by GE. The other two compressors are even older and may be called the prototype of compressors (All of them are driven by a big leather belt.)

In this pit the workers go up and down in the small shaft of 70m depth by hanging on a rope which is pulled by a manually operated hoist, and dig the lower veins.

2) Mining in surface

(1) Vein mining

Among the workers mining on the surface of the earth, a group is called Venerista, who are mining a high grade vein at the mountainside down to 15 to 20m just like as cutting a ditch. They, without wearing a helmet, are working at great risk at the face which is equivalent to ones inside the pit.

San Pedrita : 30 personnels, 1 face, width of the vein 50 to 60 cm, contains zinc and pyrites.

(2) Placer deposit mining

At the broad placer deposit in Uncia district 6 Km distant from Catavi, nearly 80 workers are divided into groups of several persons, and mining tin. The placer deposit is of low grade and generally contains 0.01% tin, however, they are tunnelling as moles through relatively high grade sand layer 8 to 10m under the ground in the vicinity of Uncia settlement.

Ores are lifted up by a manually operated hoist (for lifting workers and ores) through a small shaft with a diameter of ca. 60 cm, and dressed by specfic gravity in the manual operation jig installed in the water channel to give high quality refined ores of 30 to 50% purity, which are then sold to the mine.

(3) Obtaining the tailing for heavy liquid separation

The tailing in the heavy liquid separation plant in Siglo XX still contains about 0.17% tin, and Lameros are recovering tin from the tailing by using a manually operated jig. They draw the tailing into a hand-made dressing plant, in which manually operated jigs are arranged to produce refined ores.

3) Concentration Method

High grade ore (4 to 25%) packed in bags in the mine is dressed into high grade refined ore and low quality in the Locatarios's dressing plant located above Siglo XX office.

Here refined ore is produced in a primitive way using all manually operated machines, where nothing is consumed and no energy cost is required so that the production cost should be markedly low.

Supplied Ore (5 to 6 cm diameter)
↓
Manual Operation Crasher (to –5 mm)
↓
Manual Operation Jig
→ Manual Operation Crusher
  ↓
  Flume
  ↓
  Round Specific Gravity Separator
  → Low grade Ore 1.5%
  → Medium grade Ore 12%
  → High grade Ore Sn 25%
→ High Grade Concentrated (Sn 45 to 50%)

Even low grade tailing contains tin several times as high as the ore worked by the mine so that it is sold to the mine as raw ore for dressing.

Lamas, who are recovering tin from the tailing at the heavy liquid separation plant, originally handles only fine particles, so that they do not have a manually operated crusher, but have a manual operation jig and other equipment used in the latter stages.

Also Veneros, working at the placer deposit use only a manually operated jig to obtain refined ore.

2–10–4 Terms of Payment

Low quality ores and high quality ores both produced by the ore sellers were measured, the former at the dressing plant and the latter at the purchasing place, their weight, water content, and quality to calculate the price corresponding to the market price and quality. The rough estimate of the purchase price is shown in Table 2–45.

Table 2–45 General Remarks of the Purchase Price

High grade Ore (1981) Sn Price $US 6.5 = 14.32 $US/kg			Low Grade Ore (1980) Sn Price $US 5.30 = 11.68 $US/kg		
10%	8.40$b/kg	0.34$US/kg	1.0%	17.68$US/t	*21.68$US/t
20	41.21	1.68	1.5	27.90	34.22
30	72.36	2.95	2.0	39.14	48.00
40	107.03	4.37	2.5	49.49	60.69
50	140.86	5.74	3.0	59.84	73.38

(Note 1) The figures in the * column are calculated by multiplying $\frac{6.50}{5.30}$ by the purchase prices, where $ 6.5/lb is taken as the price of tin.

(Note 2) Ore price per kg for high grade ore and per ton for low grade ore are shown.

The purchase prices are shown in the table, but the comparison with the value of tin content is as follows:

Ore Grade	Ore Value($US/t)	Purchase Price$US/t	Ratio %
40%	5,728	4,370	76.3
2%	286.4	48.0	16.8

Since the low grade ore (Sn 2%) is handled as crude ore for dressing, it can be found that it is purchased at very low price even by taking into consideration the expenses and the actual dressing yield (approx. 50%)

2–10–5 Analysis on the Present Situation

The cost and revenue and expenditure of the division under the direct control of the mine concerning the high grade ore purchased from ore sellers are shown in Table 2–49 in the next page. It shows that the budget for the 1981 business year shows a deficit, but the sum of actual account from January to June is in the black.

Table 2–46 Average Prices on Concentrate of Locatarios

Average Metal Price $US 6.500

Changed by $b. 24.51 per $US. 1.00 Date of Aug. 1981

Ore Grade (%)	Price per Kg $b.	$b.
10	8.40	3.280
15	24.80	3.282
20	41.21	3.116
25	56.79	3.114
30	72.36	3.714
35	90.93	3.220
40	107.03	3.452
45	124.29	3.314
50	140.86	3.316
55	157.44	3.314
60	174.01	3.314
65	190.58	3.314

Table 2–47 Prices for Crude Ore Tonnage of Locatarios

Average Metal Price $US 5.30

Ore Grade	0.00	0.01	0.02	0.03	0.04	0.05	0.06	0.07	0.08	0.09
0.70	9.92	10.29	10.66	11.04	11.41	11.78	12.15	12.52	12.90	13.27
0.80	13.64	13.84	14.04	14.25	14.45	14.65	14.85	15.05	15.26	15.46
0.90	15.66	15.86	16.06	16.27	16.47	16.67	16.87	17.07	17.28	17.48
1.00	17.68	17.95	18.21	18.48	18.74	19.01	19.27	19.54	19.80	20.07
1.10	20.33	20.48	20.63	20.77	20.92	21.07	21.22	21.37	21.51	21.66
1.20	21.81	22.08	22.34	22.61	22.87	23.14	23.41	23.67	23.94	24.20
1.30	24.47	24.61	24.75	24.89	25.03	25.18	25.32	25.46	25.60	25.74
1.40	25.88	26.08	26.28	26.49	26.69	26.89	27.09	27.29	27.50	27.70
1.50	27.90	28.10	28.31	28.51	28.71	28.92	29.12	29.32	29.52	29.73
1.60	29.93	30.14	30.35	30.56	30.77	30.98	31.19	31.40	31.61	31.82
1.70	32.03	32.32	32.61	32.90	33.19	33.48	33.77	34.06	34.35	34.64
1.80	34.93	35.14	35.34	35.55	35.76	35.97	36.17	36.38	36.59	36.79
1.90	37.00	37.21	37.43	37.64	37.86	38.07	38.28	38.50	38.71	38.93
2.00	39.14	39.35	39.55	39.76	39.97	40.18	40.38	40.59	40.80	41.00
2.10	41.21	41.42	41.62	41.83	42.04	42.25	42.45	42.66	42.87	43.07
2.20	43.28	43.49	43.69	43.90	44.11	44.32	44.52	44.73	44.94	45.14
2.30	45.35	45.56	45.76	45.97	46.18	46.39	46.59	46.80	47.01	47.21
2.40	47.42	47.63	47.83	48.04	48.25	48.46	48.66	48.87	49.08	49.28
2.50	49.49	49.70	49.90	50.11	50.32	50.53	50.73	50.94	51.15	51.35
2.60	51.56	51.77	51.97	52.18	52.39	52.60	52.80	53.01	53.24	53.42
2.70	53.63	53.04	54.04	54.25	54.46	54.67	54.87	55.08	55.28	55.49
2.80	55.70	55.91	56.11	56.32	56.53	56.74	56.94	57.15	57.35	57.56
2.90	57.77	57.98	58.18	58.39	58.60	58.81	59.01	59.22	59.42	59.63
3.00	59.84	60.05	60.25	60.46	60.67	60.88	61.08	61.29	61.50	61.70

Table 2–48 Cost Balance on Concentrate of Locatarios

from Jan. to Jun. 1981

($US)

	Estimate	Results	Cost per lb
Accepted Concentrate (Ton)	1,799.8	1,606.5 *	
Average Ore Grade (Sn %)	39.44	43.27	
Sn Metal (Ton)	710.0	695.2	
Sn Metal (lb)	1,564.4	1,532.6	
Sn Metal Price ($US/lb)	6.0	5.46	
Proceeds	9,386,388	8,372,086	5.460
Direct Labor Cost	121,941	101,610	0.066
Indirect Labor Cost	96,274	87,797	0.057
Material Cost	53,977	39,278	0.026
Cartage Cost	19,542	30,104	0.020
Purchase Cost *	5,948,395	4,872,199	3.177
Finance Cost	382,508	219,248	0.143
Common Cost	47,795	30,383	0.020
Administration Cost	117,567	110,955	0.072
Others	141,190	104,724	
Production Costs	6,929,189	5,596,298	3.650
Production Profit		2,775,788	
Production Charge	1,249,482	808,145	0.527
Total Freightage	411,811	367,827	0.240
Smelting Cost	1,242,649	1,087,885	0.709
Loss and Others	489,441	507,512	0.331
Sales Costs	3,393,383	2,771,369	1.807
Total Costs	10,322,572	8,367,668	5.457
Final Profits	Δ 936,184	4,419	0.003

* There is special income $US194,978 as purchase cost of low grade ore about 1.5%

Table 2–49 Analysis of Final Profits

		Locatarios		Catavi Mine	Note
		High Grade Ore	Low Grade Ore		
Sn Metal (Ton)		695.2	( )	1,139.9	The tonnage of low grade ore is included with Catavi Mine
Proceeds (\$US)		8,372,086		12,464,682	
Direct Production Cost (\$US)		724,099		20,839,302	
Purchase Cost (\$US)		4,872,199	194,978	included	(Direct Labor Cost 4,206,432 Living Compensation 2,778,358) Total 6,984,790
Production Profit (\$US)		2,775,298		Δ 8,374,620	
Sales Costs (\$US)		2,771,512		4,523,178	
Total Costs (\$US)		8,367,668		25,362,480	
Final Profits (\$US)		4,419		Δ12,897,798	
Number of Labor		2,500		4,500	
Wage per Man-Month	\$US/Month	*1 338		*2 259	*1 $\frac{(4,872,199 + 194,978) \div 6}{2,500}$
"	\$b/Month	8,400		6,400	*2 $\frac{6,984,790 \div 6}{4,500}$
Profit per Sn Metal Ton	\$US/Ton	6.4		Δ 11,315	

2–11 Consideration and Proposal

Mining costs account for a very large part of the production cost, especially the proportion of the labor cost is large. Accordingly, the most effective countermeasure for cost reduction of labor cost based on improvement of the productivity. The improvement of productivity is to establish a rational production system suitable for the ore deposit conditions.

The biggest problem in mining division, Catavi mine is how to product low grade ore in low cost according to decreasing of ore grade and the present problem is that a large scale block caving method was introduced without giving due consideration to the existing equipments and ore deposit conditions.

A serious neck has occurred from the condition that only mining stopes are those of large scale and mass production system, and processes from grizzlies and afterwards are those of the small scale production system of the conventional stoping of the vein type deposit.

2–11–1 Mining Method

The block caving method is mainly used for stoping, but a problem exists that ore is very hard and there are few cracks, accordingly, many large size blocks over 1 m – 2 m are occurred and much labor and explosives are required for secondary blasting. That is:

(1) The strength of ore is very high, the ore does not break down themselves. (See Table 2–41)

(2) There are few cracks, and it is difficult to cave. From these conditions, it can be said that the block caving method is not suitable for stoping in Catavi mine.

The block caving method is essentially a high efficiency and low cost stoping method suitable for large scale ore deposits, where many cracks have developed both in hanging walls and ore making it easy to cave the deposit, accordingly, little drilling is required and the main part of stoping is the drawing and hauling on lower levels. Then, this method is little required labors and explosives.

Although Catavi mine is a large scale ore deposit, the natural conditions are as mentioned above in (1) and (2) and yet the stoping method not applicable to them has been adopted. That is:

(1) Since the ore deposit which cannot be caved easily is forcibly caved, shrinkage stoping must be carried out to isolate around mining block, and further a drift must be developed in the ore and coyote blasting must be carried out to break and cave the ore. As a whole, much drilling and blasting are required.

(2) Above the draw-points and grizzlies many large blocks are seen, which require much labor and explosives for processing them, and the consumption of explosives is far more than that of other stoping methods.

(3) Although the scale of stoping is large, grizzly meshes are too small (250 mm – 300 mm), and if the meshes are made larger, the following problems occur hence some countermeasures have to be taken.

(i) Since the ore passes are small and may be stopped up due to bite with each other of ore.

(ii) Large size blocks can hardly be drawn out from shute gates.

(iii) The small-sized crushers of preconcentration cannot be received large size blocks.

In Catavi mine, large-sized mining system which used diesel engine type loading machines in block caving are planned at present, but it seems that this cannot solve the above-mentioned problems and will not lead to improvement of production and productivity. That method will only serve to reduce the number of shutes.

After all, an important point is to reduce the amount of large size blocks on grizzlies, countermeasures for improvement will be summarized to the following two points:

(1) To enlarge the size of grizzly meshes – an enlargement of shute cross section in underground and crusher in preconcentration is required accompanied with the grizzly modification, and

(2) Not to occur large size blocks in stopes.

As a method to reduce the occurrence of large size blocks, an increase in the range and amount of drilling and blasting in mining blocks and breaking into small pieces at the stopes is envisaged. In other words, it is to provide sublevels and carry out long hole blasting in the widest range possible to break caving ore into small pieces.

When the above-mentioned deposit conditions are considered, the sub level stoping method should be introduced. By this method,

(1) broken ore can be crushed into small pieces,

(2) large size blocks may also be produced, but their quantity is supposed to be small, and

(3) various problems mentioned before in (i), (ii) and (iii) can be solved.

The man-shift required for drilling and the number of rock drills and bit rods may be increased, but when the present state consuming much labors and explosives in isolation stoping and coyote blasting is considered, it is expected that the extent of the increase will not be very large and the substantial effect of improvement in efficiency and cost reduction will be obtained.

2–11–2 Transportation System

1) L650 trolley locomotive transportation

(1) Present state

At present, a 10-ton trolley locomotive pulls thirteen 5-ton tubs in main haulage L650, but the transportation capacity of locomotive is not fully utilized because of the following reasons.

(i) Feed is often stopped by the trouble of preconcentration.

(ii) There are many large size block ore and can hardly be drawn out from the shute gates.

(iii) Ore is not supplied sufficiently because of hanging in the ore passes.

(iv) The slow speed of locomotives to weigh all the cars.

The state of round trips of locomotives, as stated in 2–6–1, is: seven trains are operated 64 times per day in average and transport.

106,000 ton/month of ore (Actual figures for July, 1981)

106,000 ton/month ÷ 29 day/month = 3,655 ton/day

3,655 ton/day ÷ 64 times/day = 57 ton/time (57 ton/train, rate of loading 88%)

Since the maximum number of round trips is 82 times/day, the maximum quantity per day is, 57 ton/time x 82 times/day = 4,674 ton/day. If this value is examined from round trip records (the mine mouth is taken as the reference point of round trip time), minimum required time in underground (entrance, departure, loading) = 30 minutes (measured value) minimum required time on surface (weighting, dumping) = 15 minutes (measured value) allowance for standby, etc. of entering and departing trains = 10 minutes (estimated value)

Total 55 min/time

Real working time per shift 6 hr = 360 min.

Per day (three shifts) 360 min x 3 = 1,080 min

Total round trip times of seven trains per day is,

1,080 min ÷ 55 min x 7 = 137 times

Haulage quantity per day = 137 times/day x 57 ton/time = 7,809 ≒ 7,800 ton/day

It is found that theoretical haulage capacity per day is 7,800 tons. As the average actual haulage quantity is about 64 times/day, 3,600 ton/day, the number of required trains can be halved. If the seven trains are reduced to four to five trains, underground and surface waiting time will be reduced, the real working time per train = the real haulage quantity will be increased and haulage costs can be reduced.

Also, the trouble of stoping trains caused voltage drop by the shortage of rectifier capacity can be eliminated.

(2) Estimation of haulage capacity after the completion of double track

The haulage capacity when double track in progress at the mine now is completed was investigated using an operation diagram based on the assumption of the following conditions.

Assumptions

(i) Time required for making a round trip from the mine mouth through the track-scale and the grizzly of preconcentration is assumed to be 15 minutes.

(ii) Underground loading time is assumed to be 15 minutes.

(iii) Travelling speed is assumed to be 12 kg/hr.

(iv) The diagram is prepared considering the time taken for (i) and (ii) as stoppage-time at both the terminals.

(v) The departure interval of trains at the mine mouth is assumed to be two minutes.

(vi) As the underground loading cannot be carried out for the seven trains at the same time because of the disposition of the shute gates, it was assumed to operate the trains in the two groups of four and three trains.

The operation diagram thus organized is shown in Fig. 2–22 and 2–23, and the point which needs attention is that it is most important to carry out the underground loading smoothly. For that reason, if double tracks are not provided to allow the trains to pass each other freely, trains must wait there in sequence and the operation as planned cannot be realized. Therefore, when double-tracking the main line, it is very important to double-track at the parts of the shute gate without fail.

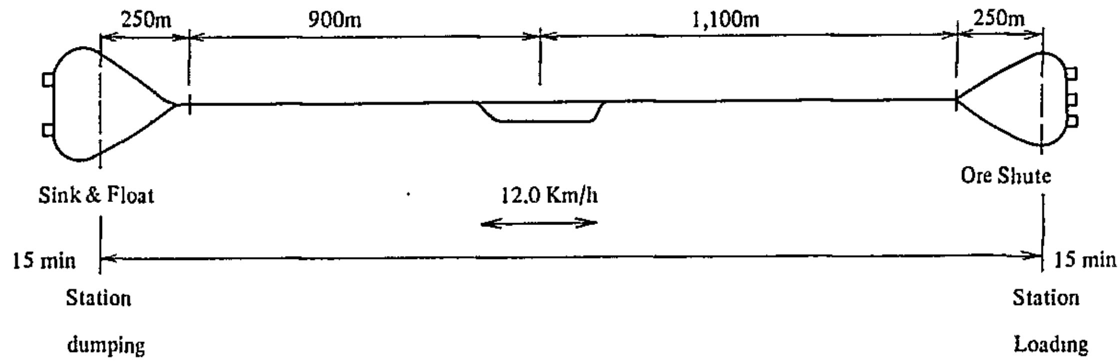
According to the operation diagram, the operation time for one cycle of a train is 55 minutes. Then the haulage capacity of seven trains is 7,809 tons/day.

360 min x 3 shift/day = 1,080 min/day

(1,080 min ÷ 55 min) x 7 trains = 137 times/day

137 times/day x 57 tons/time = 7,809 tons/day

1) Present



2) After double tracking

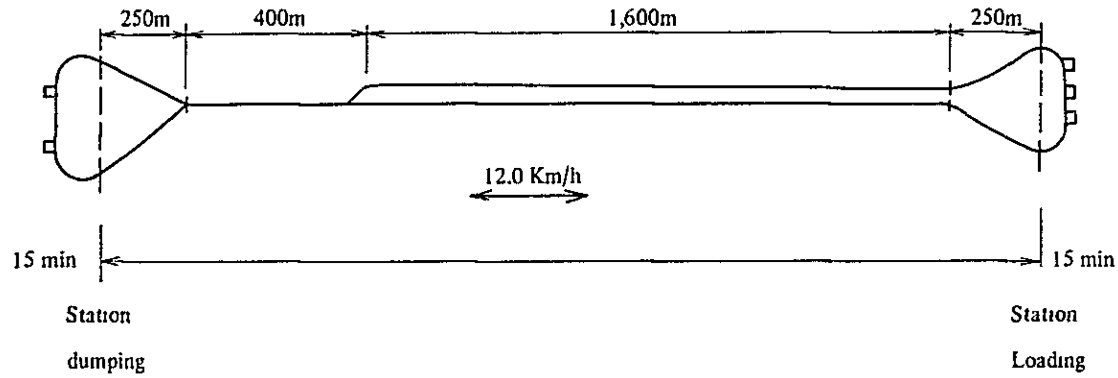


Fig. 2–22 Track of L650, Main Hanlage Level

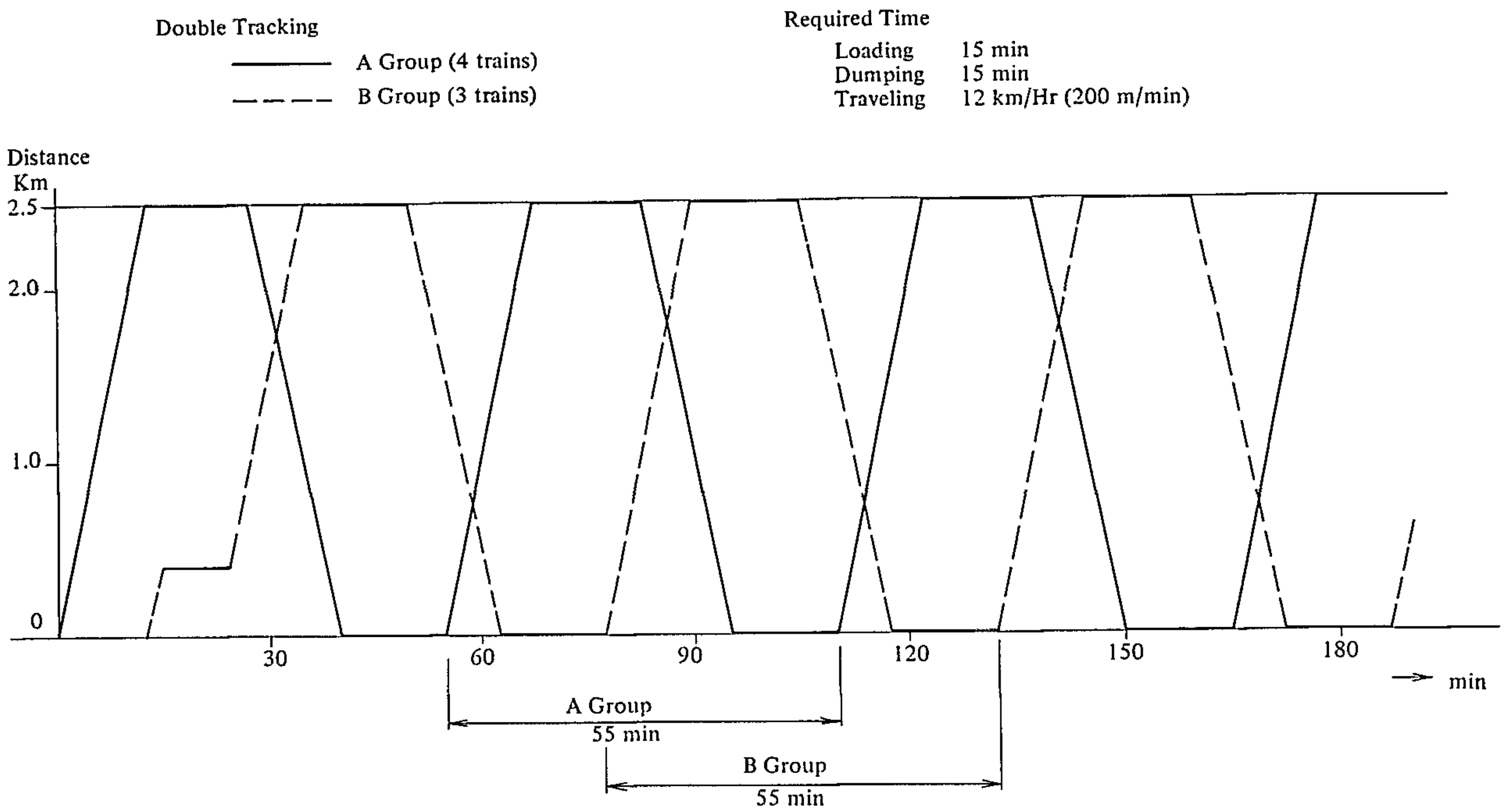


Fig. 2–23 Diagram of Round Trips

The above-mentioned theoretical haulage capacity is a value obtained when loading at underground and receiving at the preconcentration are always carried out smoothly, and the capacity during actual operation will drop to about 70% of the value. Accordingly, the following countermeasures will be required after the completion of the double-tracking.

(a) If the production remains the same as now: the number of locomotives should be reduced.

(b) If the production is doubled: the sizes of tubs and locomotives should be enlarged.

Even in the case of (b), the planned haulage capacity cannot be attained if the loading time in the underground is increased on account of various troubles, it is therefore quite important to adopt the mining method so as to eliminate large size blocks and improve the efficiency of loading.

(3) Haulage by belt conveyor

The idea of changing the locomotive haulage into a belt conveyor system is L650 was examined. In this case, the following important problems are faced:

(i) An primary crusher in underground will be required.

(ii) It is necessary to make a main ore-bin above the crusher, and moreover, sublevel haulage from each ore-pass to main ore-bin will be required.

(iii) A large sum of constructing costs (1.8 – 2.2 million dollars) will be required for installing a new conveyor.

(iv) The maintenance of a very long belt conveyor (about 3,000 m) is very difficult.

As the number of labors must be increased from (ii), it will be better to consider improvement in productivity by the track system.

(4) Shaft transportation system

Shaft for transporting materials and persons in Catavi mine have the following features.

(i) All the shafts are of the single cage system with counter-weight.

(ii) The dimensions of the cage are as small as about 1.5 m x 1.5 m.

Concerning (i), the level intervals of horizontal levels are irregular (15 – 50 m), and there are about ten levels per shaft, accordingly, if the usual double cage system is used, the stop place will become too complicated so that the operation may become impossible, the use of the present system is therefore inevitable. Concerning (ii), it may be thought to have come from a fact that large-sized machines were not used in the past, but it cannot be understood that Beza shafts now excavated have the same dimensions as those of conventional ones. In this case, there must have been planned the increase of transportation capacity by enlarging the size of cages.

2–11–3 Ventilation Problem

1) Complicated Underground Constructure

As the underground constructure of Catavi mine is very complicated because it was developed following the veins, it is impossible to establish a systematic underground ventilation system. However, a auxiliary ventilation system for a certain mining block (for example, like the present block caving) can be established.

The most urgently required things at present are large-sized exhaust levels. Although large-sized local fans have been installed in block caving stopes, sufficient air quantity does not flow through because exhaust levels are not directly connected with surface.

When considered from the total flow system of air, it will be appropriate to use the lowest L650 as a main intake level and use Azul level (L295) 350 m above L650 or the uppermost (Transformador) level (L50) as a main exhaust level. When installing fans in large-sized exhaust levels, it will be necessary to blockage many branch levels so as not to intake air from many minor drifts.

2) Ventilation of block caving stopes

As referred to the section of ventilation, each stope has independently inlet and outlet fans, trying to extract the blasting fume and dust in the grizzly level (or the working level), but is not so very effective. The following two factors are thought to be the reason for it.

(i) Grizzly meshes are almost always clogged with large size blocks and air cannot flow through them.

(ii) On account of the insufficient capacity or the performance drop of suction fans on lower levels, negative pressure is too low and suction effect is too small. (This relates also to the defect of exhaust levels directly). To improve these conditions, the Safety and Hygiene Center of Oruro is planning to develop raises exclusively used for suction between grizzlies so that dust and blasting fume can always be extracted, and the effect of the plan can be expected. Even in this case, maintenance of fans and improvement in exhaust levels are of course required.

2–11–4 Operation Control

1) Concentration of drilling and blasting to the first shift

In Catavi mine, drilling is concentrated to the first shift, and the reason for it can be considered as follows.

(i) Stopes are fixed to each piece-work group.

(ii) All the supervisors including superintendants enter to underground together with the first-shift group, so that work can be controlled sufficiently.

(iii) The operation system at the time of start of mining operation when the quantity of production was small still, remains.

The disadvantage owing to the concentration of drilling to the first shift is that the consumption of compreseed air is concentrated and pressure drops to such an extent that drilling efficiency is dropped and laborers are complaining about it.

In ordinary mines, the quantity of work is dispersed evenly to each shift to reduce the number of machines and increase the rate of operation. Also in Catavi mine, if drilling is dispersed to two shifts (or to the first, second and third shifts), the present number of compressors will be sufficient for the requirements, and as the quantity of required compressed air per shift is reduced, the drop of air pressure will be reduced and the efficiency of work will be increased. (In Catavi mine, enlarge in the size of compressed air piping and the number of compressors is being planned as a countermeasure for the drop of air pressure).

However, the dispersion of blasting to each shift will require the workers of the reserve stations of explosives for each shift, dispersing the responsibility of controlling the quantities of explosives, and may cause problems, therefore, some countermeasures must be taken beforehand.

2) Kind of occupations and work

Though the laborers have work allotted according to the work group to which the belong, various activities (transportation of AN-FO, cleaning of levels, etc.) are allotted to hourly laborers in turn regardless of the group to which they belong, and this is a desirable form from the view point of making laborers versatile and improving efficiency.

3) Accumulation of various data

Among the data of ore reserve, ore grade of the exploration section and the data of production, crude ore grade of the mining section, there are many similar data and we are at a loss as to which we should choose. The more the kinds of data, the larger the number of persons required to prepare the data, it is therefore necessary to investigate which data are really required, omit unnecessary data and integrate similar data.

Next, there are many yearly or long term plans, but it seems that a little more detailed analysis is required about the elements which form the bases of the plans. For example, in the comparison of the block caving method and the sublevel stoping method, the analysis and the detailed examination of works about the mining efficiency and the mining costs of each method, which should be main subjects, are not mentioned but only conclusions are referred (the stoping costs of the block caving method are far lower), but correct comparison cannot be made in such a way.

Also about the increase plan of haulage capacity for the main haulage level of L650, it is necessary to understand the real state of the shute gate drawing in detail and try the round trip simulation of many trains utilizing a computer.

2–11–5 Ore Sellers (Locatarios, Lamas, Veneros)

In Catavi mine, Locatarios have been regarded as a kind of encumbrance, but actually they help to increase the profit of the mine very much. There are many future problems especially about Locatarios among them, who work underground. The following problems are pointed out by the mine authority.

(i) Ore reserves have been exhausted to below minable limits.

(ii) They have no more than elementary technique.

(iii) Their machines and accessories are old-fashioned.

(iv) Their productivity is low.

(v) The cost of controlling them is increasing.

(vi) They have too large social burden for their limited organization.

(vii) Sales conditions are getting worse.

So much as we surveyed, they may not in fact have modern technique, but they are mining and concentrating at low cost selecting high grade veins to be product, which produce coarse grain tin minerals which can easily be gravity-concentration, they therefore cannot be regarded altogether worthless. The biggest problem at present is (i), ore reserve, and the mine is not exploring for them, but it is urgently necessary to discover high grade veins even if they may be veinlets.

2–11–6 Underground Development

The efficiency of development is too low. This problem must be considered dividing it into two parts. One is the efficiency of elemental works such as drilling, blasting, hauling, etc., and the other is the problem of the working system how to increase the length of excavation per day or per month.

The former is determined automatically by rock drills, loading machines, locomotives, tubs, etc., used. One problem here is the drop of efficiency caused by a fact that blasting is carried out in two parts, that of center-cut holes and that of round holes, so that the time of preparation work for drilling, blasting, etc., and the time of waiting for fume exhausting must be doubled. The reason for adopting such method is thought to be as follows.

Since center-cut is carried out by the burn-cut method, center-cut holes may not sometimes be broken completely on account of the misfire of fuses, etc., but even in such a case, auxiliary center-cut can be added at the time of blasting round holes.

However, center-cut blasting can be performed more surely by making the center-cut holes into a V-cut or Pyramid-cut, so that the center-cut holes can be blasted simultaneously with round and the drifting efficiency can be improved.

An item to be studied and carried out in future in Catavi mine is the latter, improvement in the excavation system. Usually, when sinking shafts or excavating long distance in mines, great effort is made to complete the total work in the shortest period possibly, employing the so-called "quick excavation" system, but this way of thinking is lacking in Catavi mine.

In the case of the "quick excavation" system the following countermeasures are required.

1) The twenty-four hours in day should be used fully.

2) By dividing a day into three or four shifts, the laborers of three or four teams should carry out the work continuously.

3) The rock drills, loading machines, tubs, etc., of modern large-sized type should be used.

When this system is employed, many laborers must cooperate in the same work and share the total piece-work payment for the work, but this is completely different from the custom in the past, so that the education of laborers is required in addition to the training of skilled workers.

CHAPTER 3 METALLURGY DIVISION

3–1 Outline

The production of crude ore in Catavi mine at present is about 5,000 t/day with about 0.3% tin grade ore. There are two concentrating plant called Siglo XX Sink & Float Plant and Victoria Mill to treat these crude ore.

The circuit of Siglo XX Sink & Float Plant consists mainly of a washing and crushing circuit and a heavy medium operation circuit and recovers pre-concentrate by treating the crude ore carried out from underground. The tin grade of the pre-concentrate is about 0.45% and the tin recovery is about 75%. In addition to the above, this plant has a slime treatment circuit in which a part of tin is recovered by table concentration and flotation.

Pre-concentrate produced in Siglo XX Sink & Float Plant is transported to Victoria Mill about 5 km away by train.

In Victoria Mill, the pre-concentrate is crushed, and after a part of tin is recovered by jigs, tin is concentrated and recovered with tables from the pre-concentrate classified to each sizes. The ore concentrated by these gravity separation methods are crushed again by rod mill, and pass through sulphide mineral flotation and produced as final concentrate.

Capacity of this plant is about 2,500 ton/day, recovering the concentrate of about 40% tin grade at a tin recovery of 60%.

On the other hand, the work of Siglo XX Sink & Float Plant and the tailings from the coarse table of Victoria mill are dumped on the surface and the fine tailings of the two plants are sent to neighboring Lake Kenko.

In the flotation plant called Kenko Plant, tin has been recovered since 1970 from the fine tailings which have been accumulated in Lake Kenko for many years.

In Kenko Plant, old fine tailings are dredged by a dredger, and after the optimum size for flotation is arranged in a classification circuit, tin concentrate is recovered by flotation.

However, the quantity of tin production in this mine is decreasing year by year as the grade of crude ore drops rapidly (Fig. 3–1). Moreover, no improvement in the process of operation has been achieved for many years, so that there are many surplus personnels, and the operation costs are rising sharply, thus the economical situation of the mine is steadily getting worse.

On the other hand, existing equipment has been deteriorated, because the renewal of circuits and machines has rarely been carried out over these many years. In addition, the

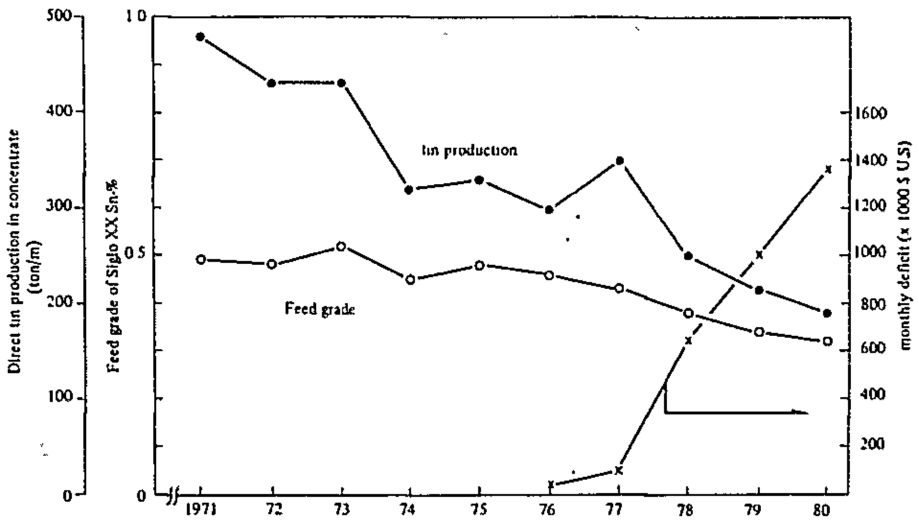


Fig. 3–1 Feed Grade and Tin Production

production first policy up to now has left the maintenance system of the plant unsatisfactory, resulting in frequent machine trouble and making operation unstable.

The necessity of renewal of machines and instruments has been recognized recently at the mine site too and a plan to renew a part of equipments is being put into practice.

However, the above-mentioned aggravation of economical situation can no longer be covered up with only renewing a few machines, and further, a serious problem is coming up, it is that ore stoped at present will soon be exhausted.

Accordingly, the most important subject at this time is not the improvement of the existing equipments and circuits but the selection of new kinds of ore to be treated and the establishment of techniques to treat and concentrate the ore.

Under such an urgent and difficult state, the present survey commission chose Desmonte (tailings of heavy medium flotation) of Siglo XX Sink & Float Plant, Colas Arenas (coarse table tailings) of Victoria Mill and Block Central (a name of undeveloped underground low grade ore) as test object and carried out basic tests about concentration techniques with them.

Table 3–1 shows ore reserves in this mine which have been reported up to now. A part of Colas lamas in the table is the object of treatment of Kenko Plant.

Table 3–1 Reserves of Catavi Mine

	Min. ton	Sn %	Fino ton
Vetas	443,472	1.52	6,757.71
Vetas en Blocks	115,399	2.08	2,398.34
Puentes	44,338	2.88	1,275.16
Taqueos	–	–	–
Block Caving	3,255,329	0.39	12,797.36
Block Chicos	89,698	0.40	363.14
Existencias	103,478	0.92	948.04
Total Mina	4,051,714	0.61	24,539.75
Desmontes	21,961,820	0.27	59,845.16
Veneros	297,249,015	0.01	30,558.49
Relaves	32,262,227	0.37	118,686.20
Total Superficie	351,473,062	0.06	209,089.85
Gran Total	355,524.776	0.07	233,629.60

The outlook of the three concentration plants of this mine and a series of flotation test results carried out this time are mentioned in the following.

3–2 Siglo XX Sink & Float Plant

Main ore treated in this plant is low grade ore mined by the block caving method and high grade ore mined by the shrinkage method, and about 5,000tonsore is treated in a day. Main circuits are those of washing, crushing and heavy medium flotation. To reduce crushing and grinding costs, ore is pre-concentrated in the state of comparatively coarse size. From the fine grains produced in the washing and crushing process, tin is recovered by mineral jigs and classifiers, and also from slime including finer grains, tin is recovered by shaking tables and flotation.

The tin grade of crude ore is about 0.3%and the tin grade of the recovered pre-concentrate is about 0.45%.

On the other hand, the tin recovery is about 75%, and the quantity of ore transported to Victoria Mill is reduced to half by the pre-concentration in this plant.

3–2–1 Organization, Personnel

In this plant, 78 salaried personnel including six engineers, and 264 day laborers are engaged in the operation and the repair works of the plant. The outline of the organization is shown in Fig. 3–2.

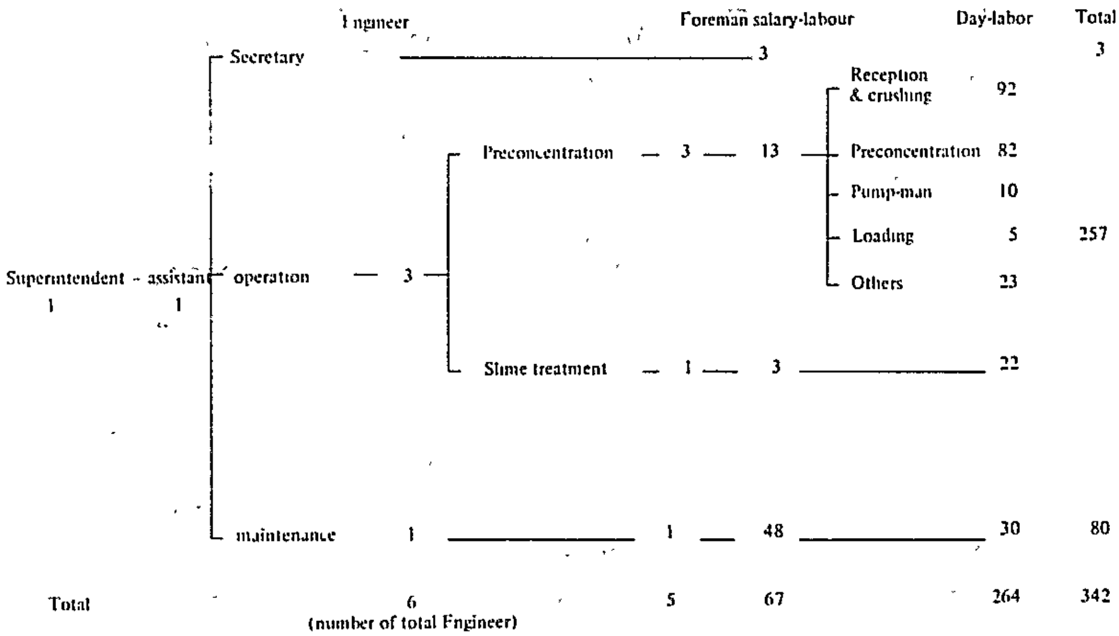


Fig 3–2 Organization of the Sink & Float Plant

3–2–2 Metallurgical Results

The metallurgical balance of this plant in 1980 is shown in Table 3–2.

Table 3–2 Metallurgical Balance of Siglo XX Sink & Float Plant (1980)

	Weight		Grade	Sn.	Distribution
	Ton / Y	%	Sn-%	Ton / Y	%
Feed	1,283,515	100 0	0.32	4,151.1	100.0
Fines	347,141	27.0	–	–	–
Feed of S & F	936,374	73.0	–	–	–
Sink	290,141	22.7	–	–	–
Desmonte	646,233	50 3	0.16	1,045.8	25.2
Preconcentrate	637,282	49 7	0.49	3,105.3	74.8

Preconcentrate = Fines + Sink

3–2–3 Process and Equipment

1) Receiving and crushing circuit

Receiving equipment includes three circuits. At first, ore mined by block caving method is received into a crude ore bin passing through a grizzly of 10 inch opening, then fed to a 5′ x 12′ low head screen of 3 inch opening. The oversize is crushed by two 13 inch gyratory crushers and stored once in an ore storage bin together with the undersize. Next, the ore is drawn out by a belt conveyor and washed in a 5′ϕ x 10′ drum washer. The washed ore is screened by a 5′ x 12′ low head screen of 3/8 inch opening and the oversize is conveyed to a secondary crushing circuit, while the undersize is sent to a classification circuit.

The secondary crushing circuit consists of two Symons 5 1/2′ type cone crushers, four 1 1/2′ inch opening and four 3/8 inch opening low head screens, and while crushing coarser ore in the closed circuit, classifies the ore finally into 1 1/2 – 3/8 inch product and product under 3/8 inches. The 1 1/2 – 3/8 inch product is stored in bins to be used as feed for heavy medium flotation, while the product under 3/8 inches is fed to jigs, and the sink is sent to the classification circuit, while the float is disposed as tailings.

The classification circuit has two 9′ x 28′ x 6′ drag classifiers as primary classifiers and one 6′ x 18′ x 4′ drag classifier as a secondary classifier, and it recovers sand as pre-concentrate in two steps and sends the overflow of the second classifier to a slime treatment circuit.

The above is the outline of main circuits, and ore stopped by the block caving method accounts for about 75% of the total quantity of ore treated. Most of the other ore is mined by the shrinkage method.

Ore mined by the shrinkage method is received into another crude ore bin with a grizzly of 10-inch opening. Ore drawn out from the crude ore bin is fed to a 5′ x 12′ low head screen of 3-inch opening and the undersize is fed to the drum washer of the main circuit. On the other hand, the oversize is crushed by two 13-inch gyratory crushers, and then sent to the secondary crushing circuit of the main circuit.

In this circuit, a 3′ x 3′ jaw crusher made in Soviet Union was installed recently as a auxiliary.

In addition to these circuits, there is a small scale receiving crushing circuit for treating other various minerals of a little higher grade. These minerals crushed here are sent to Victoria Mill directly as preconcentrate.

2) Heavy medium separation circuit

The ore of 1 1/2 – 3/8 inch sizes obtained by screening in the crushing circuit is fed to two 10' x 13' cone heavy media separators. Ferrosilicon is used as heavy media. The apparent specific gravity of the heavy liquid is about 2.4.

The sink is washed on two 4' x 14' low head screens of 1 mm x 10 mm opening and stored in a bin as preconcentrate.

The float is washed on four similar low head screens and then transported by a ropeway and accumulated on the surface as tailings.

Pulp produced by washing heavy medium separation products is sent to a heavy medium recovery circuit. In this circuit, a cyclone and a spiral separator are used for thickening and classification, and as ferrosilicon recovery equipment, four wet type belt magnetic separators are used for coarse grains, and eight wet type drum magnetic separators are used for fine grains. Recovered heavy media are recycled to the heavy medium separators after being deslimed by a spiral classifier. On the other hand, final non-magnetic product is sent to a water recovery circuit.

The tin recovery in this circuit is about 60%, and the tin distribution to the sink is about 30% of the crude ore.

The recovery of ferrosilicon in the heavy medium recovery circuit is about 60%.

3) Slime treatment circuit

In the washing-and-crushing circuit, slime is produced at a ratio of about 6.5% to feed ore. To treat this slime, there is a circuit for recovering tin by flotation and table concentration.

In this circuit, received slime is first divided into the coarse grain part and fine grain part by two spiral classifiers. The coarse grain part is concentrated by 14 4-inch cyclones and the overflow from the cyclones is pumped to a water recovery circuit. The underflow is fed to pyrite flotation with xanthate Z–11, and the tailing is fed to tin flotation. The tin flotation is carried out in an acid circuit using aeropromoter 860 as a collector and eight Denver type flotation machines are used for rougher flotation, while six Denver type flotation machines are used for cleaner flotation. The cleaner concentrate is collected as tin-concentrate, and from the cleaner tailing and rougher tailing, tin is again recovered by table concentration.

For the treatment of the fine grain part from the classifiers, thirteen Deister tables are provided to recover tin.

Table tailing and pyrite flotation float are drained into Kenko Lake through an open ditch as final tailing.

At present, the operation of the tin flotation circuit is stopped because of the shortage of sulfuric acid, and the shaking tables are utilized temporarily to recover tin.

4) Other equipments

In this sink-and-float plant, water of about 3.6 m^3 per ton of treated ore is used and the recycling ratio of water is about 70% of the total water consumption.

Fresh water is pumped up from underground and nearby rivers and there are four water supply lines. Pumped up fresh water is stored in a concrete water tank at the higher part of this plant and supplied to each circuit by a natural head.

As water recovery equipments, there are four thickeners whose diameters are 60 feet, 50 feet, 50 feet and 30 feet, and recovered water is stored in the above-mentioned water tank or supplied to various circuits directly according to the each characteristics of recovered water.

In addition, there are two ropeways of 2,700 m long and 800 m long as equipments for transporting tailings on the surface.

For reference, the balance sheet (1974) of this sink-and-float plant is shown in Fig. 3–3.

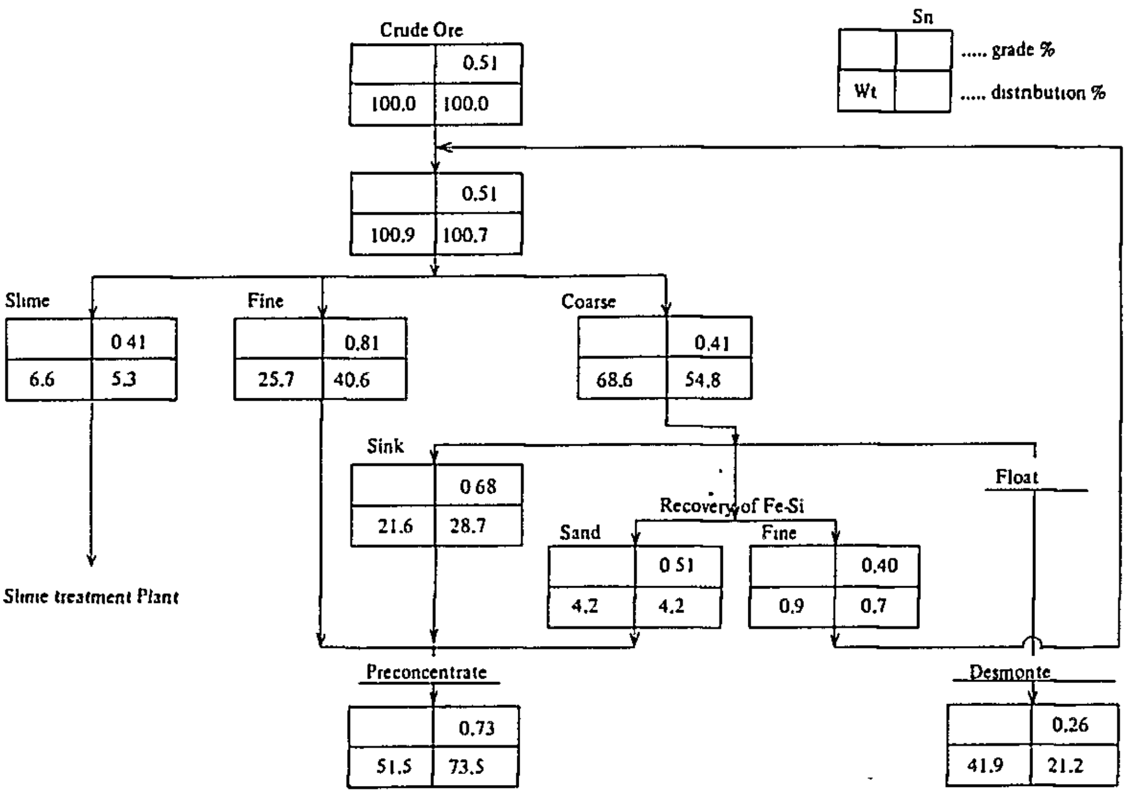


Fig. 3–3 Balance Sheet of Sink and Float Plant (1974)

3–2–4 Operating Cost

The outline of operating cost in the past five years and the detail of the cost are shown in Table 3–3 and Table 3–4.

Electric power consumption in 1980 was about 12.8 million KWh, unit consumption was 9.8 KWh/ton, unit price was 0.29 US$/ton, and annual cost of electric power was about US$ 375,000.

Table 3–3 Operating Cost of Siglo XX Sink and Float Plant

	1976		1977		1978		1979		1980	
	$ U.S/Y	%	$ U.S/Y	%	$ U.S/Y	%	$ U.S/Y	%	$ U.S/Y	%
Personnel Expenses										
Direct	344,339	13.2	417,656	14.2	468,223	14.0	583,627	15.9	753,907	17.0
Indirect	384,549	14.7	412,794	14.1	466,065	14.0	588,931	16.0	740,889	16.8
Total	728,888	27.9	830,450	28.3	934,288	28.0	1,172,558	31.9	1,494,796	33.8
Materials	885,218	33.9	1,074,449	36.7	1,196,227	35.9	1,220,146	33.2	1,407,782	31.8
Transportation	324,913	12.4	343,000	11.7	324,796	9.7	298,425	8.1	323,259	7.3
Auxiliary Service	622,659	23.8	643,597	22.0	807,864	24.2	921,443	25.1	1,130,275	25.5
Depreciation	36,894	1.4	27,015	0.9	30,293	0.9	33,026	0.9	43,011	1.0
Others	15,458	0.6	12,059	0.4	43,000	1.3	29,430	0.8	26,662	0.6
Total	2,614,030	100.0	2,930,570	100.0	3,336,468	100.0	3,675,028	100.0	4,425,785	100.0
Unit Cost $ U.S/T	1.90		1.78		2.33		2.90		3.41	
Treatment Ton/M	114,539		137,074		119,339		105,552		108,065	

Table 3–4 Cost of Materials (Sink & Float)

1980.7 – 1981.6

	$ U.S/Y	$ U.S/Ton	%
Parts of Crusher	64,795	0.05	3.6
Parts of Screen	122,602	0.09	6.8
Conveyor Belt	17,100	0.01	0.9
Parts of Pump	174,836	0.13	9.6
V-belt	47,718	0.04	2.6
Ferro Silicon	492,115	0.38	27.2
Flotation Reagent	12,290	0.01	0.7
Steel	249,744	0.19	13.8
Others	629,335	0.48	34.8
Total	1,810,535	1.38	100.0

3–2–5 Actual Problems

1) General problems

(1) This plant is suffering from the shortage of heavy medium, the most important material for this plant; and because of the shortage of sulfuric acid, the operation of the tin flotation circuit has been stopped. The supply of these principal materials is a problem which must be solved before the operation, and to realize smooth procurement, not only Catavi Mine but all Comibol must make its best effort.

(2) There are frequent breakdowns of machines during operation, which obstructs smooth operation greatly.

To improve such a state, it is necessary to regard maintenance work as a part of the operations and carry out at least the required minimum maintenance work periodically.

2) Receiving and crushing circuit

(1) Ore carried out from underground includes many large blocks, while primary crushers are too small to crush such blocks, therefore, many large blocks have to be disposed as refuse, increasing loss of resources. To deal with such conditions, it is necessary to employ large-sized crushers or to provide rock breakers for the receiving circuits.

(2) As the capacity of the drum washer is too small and the capability of the lifter to lift ore is insufficient, the effect of washing seems insufficient.

(3) The quantity of feed ore is not controled appropriately and sometimes too much ore is fed compared with the capacity of washers. It is necessary to introduce an automatic ore feed system to stabilize ore feed.

3) Sink and float circuit

(1) Also to feed ore to the cone heavy media separators, it is desirable to provide an automatic ore feed system.

(2) To remove factors obstructing the recovery of heavy media, detailed analysis of the recovery circuit itself, investigation of the conditions of classification and concentration, etc., will be required in addition to the above-mentioned improvement in washing.

3–3 Victoria Mill Plant

In this mill, mainly preconcentrate from Siglo XX Plant is treated, and in addition, purchased concentrate, crude ore, etc., is also received.

The capacity of this mill is about 2,500 ton/day, and main circuits are a receiving and crushing circuit, a jig separation circuit, a sand table circuit, a slime table circuit, a fine grain table circuit, a pyrite flotation circuit, etc.

As tin concentrate, concentrate of about 40% grade is produced at a recovery of about 60%.

3–3–1 Organization and Personnel

In this mill, 115 salaried personnel including five engineers and 393 day-laborers are engaged in the operation and the repair of facilities. The outline of the organization is shown in Fig. 3–4.

		Engineer	Foreman	Salary-labour	Day-labour	Total
superintendent 1 — assistant 1	administration	1	1	2		6
	concentration		4	33	235	272
	flotation plant		1	3	31	35
	sampling		1	1	16	18
	maintenance		2	36	25	63
	carpenter-shop		1	6	8	15
	tailing treatment		1	8	41	50
	concentrate control		3	5	26	34
	others			1	11	12
superintendent	cheef of laboratory	1	1			
Total		5 (number of Engineers)	15	95	393	508

Fig 3–4 Organization of Victoria Mill

3–3–2 Metallurgical Results

The metallurgical balance of this mill in 1980 is shown in Table 3–5.

Table 3–5 Metallurgical Balance of Victoria Mill Plant (1980)

		Weight		Grade	Sn	Distribution
		Ton/y	%	Sn-%	Ton / y	%
Feed	Siglo XX mine	636,831	92.2	0.49	3,102.4	88.1
	Others	53,876	7.8	0.88	420.6	11.9
	Total	690,707	100.0	0.51	3,523.0	100.0
Concentrate of Jig		778	0.1	21.34	166.0	4.7
Concentrate of Table		14,036	2.0	14.29	2,005.5	56.9
Tailing of Table		675,893	97.9	0.20	1,351.5	38.4
Feed of Pyrite flotation		14,814	2.1	14.66	2,171.5	61.6
Froth of Pyrite flotation		9,477	1.3	0.50	47.0	1.3
Final Concentrate		5,337	0.8	39.81	2,124.5	60.3
Total Tailing		685,370	99.2	0.20	1,398.5	39.7

3–3–3 Process and Equipment

1) Receiving and crushing circuit

The feed ore is dumped from train into a 2,000-ton bin. Then ore is drawn out from the 2,000-ton bin by a belt conveyor and crushed into all minus 1/2 inch in parallel two primary crushing circuits, each of them is closed circuit consisting of low head screens of 1/2-inch mesh and a 57"D x 20"L roll crusher.

After the primary crushing, the ore is stored once in a 2,200 ton bin and then sent to parallel two secondary crushing circuit.

Each secondary crushing circuit is a closed circuit consisting of a 42"D x 16"L roll crusher and each two low head screens of 6 mm opening, 3 mm opening and 1.5 mm opening, and ore is finally classified into 6 mm – 3 mm, 3 mm – 1.5 mm and under 1.5 mm size and sent to the next separation circuit.

In the secondary crushing circuit, the feed is weighed with a merrick scale, and sampled intermittently by an automatic sampler.

2) Jig separation circuit

From the 6 mm – 3 mm ore and the 3 mm – 1.5 mm ore classifed in the crushing circuits, coarse cassiterite is recovered with jigs.

Four Bunker Hill two-compartment jigs are used for coarse grains, and six Bunker Hill four-compartment jigs for fine grains. The tailings from the jigs are sent to a grinding circuit.

3) Sand table circuit

In this circuit, ground jig tailings and ore under 1.5 mm size from the crushing circuits are the new feed.

The feed is classified by two Bawl-Rake classifiers and the overflow is sent to a slime table circuit, while the sand is sent to 6-compartment Dorrco Sizers and classified into five kinds of coarse products and two kinds of fine products.

The coarse products are fed to coarse grain shaking tables and separated into concentrate, middlings and tailings. A part of the middlings is ground together with jig tailings and cycled as the feed of this sand table circuit.

Other middlings are again fed to shaking tables in three stages and separated in sequence into concentrate, middlings and tailings.

The fine products are fed to fine grain shaking tables, and after separation into concentrate and middlings, the middlings are fed to a classifying and shaking table circuit including regrinding mills and undergo tin recovery again.

47 Deister Plat-0 tables are used as coarse grain shaking tables and 14 Deister Super Duty No. 6 tables as fine grain shaking tables.

The regrinding circuit includes four 5 ft D x 10 ft L rod mills and two 4 ft D x 10 ft L rod mills set in and drag classifiers coupled respectively with the sets. As regrinding mills for fine grains 4 ft D x 10 ft L rod mills are used.

Tailings from this circuit, after being dewatered by a drag classifier, are transported by ropeways and accumulated on the surface.

4) Slime table circuit

The overflow from the Bawl Rake classifiers is thickened by sixteen 10 ft D x 2 m cone type thickeners, and next, tin is recovered from it by shaking tables in two steps.

The tailings are fed to a Bawl Rake classifier together with the overflow from the drug classifier for dewatering Colas Arenas, and the coarse part classified there is disposed as Colas Arenas. On the other hand, the overflow from the Bawl Rake classifiers is fed to a fine grain table circuit.

The middlings from the tables and the overflow from the cone type thickeners are thickened by thickeners or cyclones, and tin is recovered again from it by shaking tables.

The overflow from the thickeners and the cyclones is pumped into the water recovery circuit.

5) Finer grain table circuit

In the finer grain circuit, pulp is thickened first by cyclones and thickeners, and next, tin is recovered in three different parallel circuits

The circuit are as follows.

(1) A circuit which recovers tin at each grain size with Deister tables.

(2) A circuit which consists of rougher separation with Denver Buckman tables and cleaner separation with Deister tables.

(3) A circuit which consists of rougher separation with Bartles Mozley tables and cleaner separation with a Crossbelt Concentrator.

In these circuits, 19 Deister Super Duty No. 6 tables, eight Denver Buckman tables, two Bartles Mozley tables and one Crossbelt Concentrator are used.

Tailings and the overflow from the thickeners are transferred to the water recovery circuit.

6) Pyrite flotation circuit

The concentrate recovered by the jigs and the tables is stored once in a bit together with purchased concentrate.

After being ground in a closed circuit consisting of 5 ft D x 10 ft L rod mill and drag classifier, the concentrate is fed to pyrite floatation with xanthate. The pyrite flotation consists of rougher flotation and cleaner flotation. For roughing six cells of 48-inch are used and eight cells of 28-inch Denver flotation machines are used for cleaning.

The float is disposed as tailings and the non-float is shipped as final concentrate after being dewatered by disk filters and being dried by driers.

7) Other equipment

(1) Circuit for treating other various minerals

In addition to the main circuits mentioned above, there is a circuit for treating other various minerals in this mill.

After being crushed by ball mills, those minerals are fed to three rougher flotation with three Panamerican jigs, then fed cleaner with four Deister Plat-0 tables, and tin concentrate is recovered.

Tailings are pumped to the sand table circuit after regrinding by rod mill.

(2) Water recovery facilities

In this mill, about 10 m^3 water is used per ton of treated ore, and about 70% of the water is recycled water. One 120 ft D thickener and two 75 ft D thickeners are used as the water recovery facilities.

Fig. 3–5 shows the brief balance sheet of this mill.

3–3–4 Operating Cost

The outline of operating cost in the past five years and the contents of material cost are shown in Table 3–6 and Table 3–7.

Electric power consumption in 1980 was about 19.0 million KWh, unit consumption was 14.6 KWh/ton, unit price was US$/ton 0.43 and annual cost of electric power was US$ 558,000.

Table 3–6 Operating Cost of Victoria Mill Plant

	1976		1977		1978		1979		1980	
	$ U.S.	%	$ U.S.	%	$ U.S.	%	$ U.S.	%	$ U.S.	%
Personnel Expenses										
Direct	499,464	14.0	634,820	15.0	652,802	17.1	863,967	17.1	1,103,402	18.1
Indirect	573,242	16.1	694,227	16.5	688,306	18.0	895,340	17.8	1,066,475	17.5
Total	1,072,706	30.1	1,329,047	31.5	1,341,108	35.1	1,759,307	34.9	2,169,877	35.6
Materials	1,332,093	37.3	1,627,282	38.6	1,250,857	32.7	1,771,895	35.2	2,133,220	35.0
Transportation	4,537	0.1	4,242	0.1	2,941	0.1	15,739	0.3	13,707	0.2
Auxiliary Service	1,045,638	29.3	1,116,102	26.4	1,087,288	28.4	1,358,496	27.0	1,614,024	26.5
Depreciation	105,689	2.9	132,041	3.1	103,050	2.7	93,558	1.8	99,677	1.7
Others	10,142	0.3	10,941	0.3	38,782	1.0	38,891	0.8	59,562	1.0
Total	3,570,805	100.0	4,219,655	100.0	3,824,026	100.0	5,037,886	100.0	6,090,067	100.0
Unit Cost $U S/T	5.30		5.48		5.58		7.36		8.82	
Treatment Ton/M	56,137		64,122		57,130		57,014		57,559	

Table 3–7 Cost of Material (Victoria Mill Plant)

(1980.7 ~ 1981.6)

	$ U.S/Y	$ U.S/Ton	%
Parts of Crusher	212,766	0.31	10.7
Parts of Screen	40,506	0.06	2.0
Conveyor Belt	38,688	0.06	1.9
V-belt	15,087	0.02	0.7
Parts of Mill	79,413	0.11	4.0
Parts of Pump	203,936	0.29	10.2
Rod	228,716	0.33	11.5
Flotation Reagent	23,549	0.03	1.2
Steel	158,089	0.23	7.9
Wood	54,974	0.08	2.8
Others	939,256	1.35	47.1
Total	1,994,980	2.87	100.0

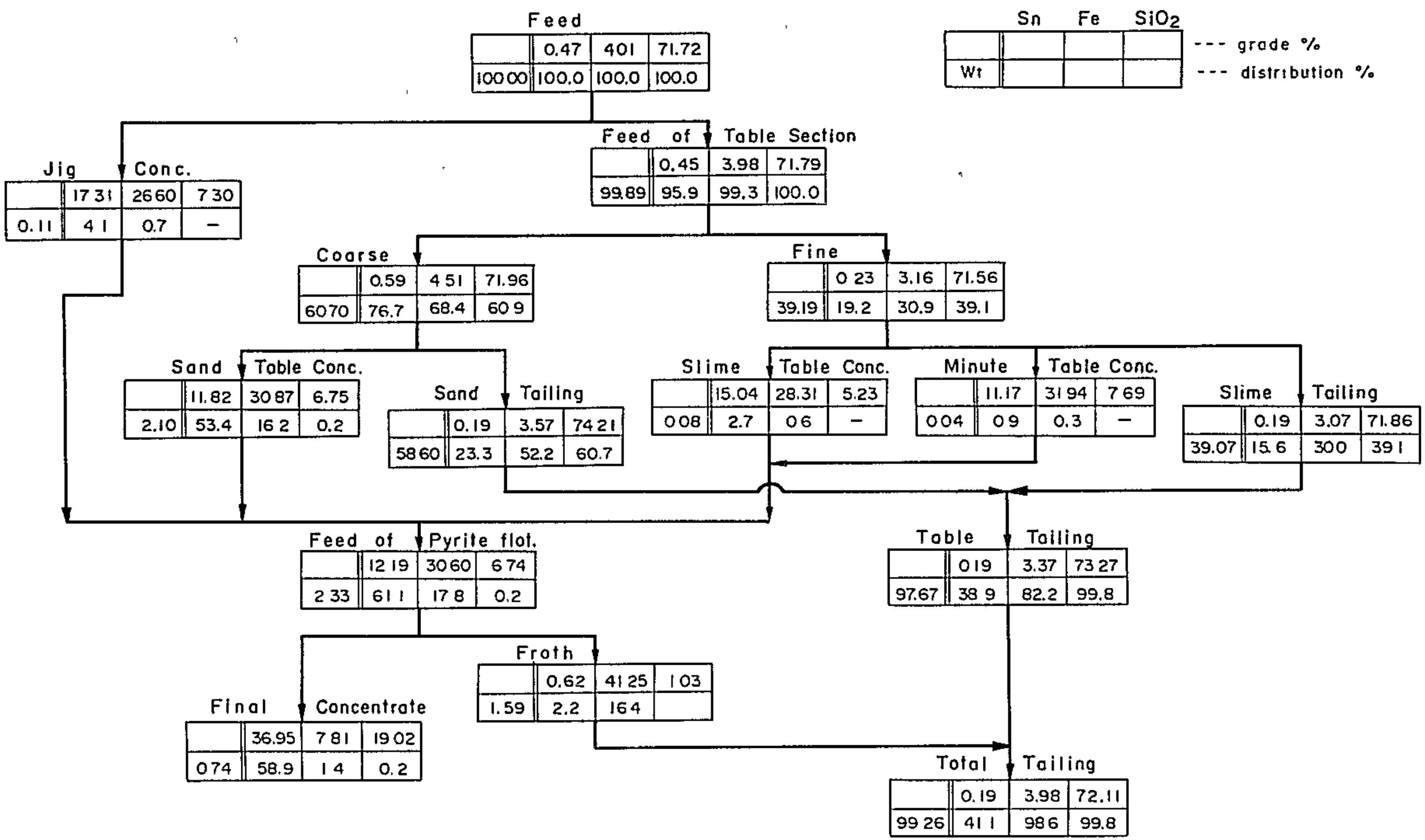


Fig. 3-5 Balance Sheet of Victoria Mill (1981. 6)

3–3–5 Actual Problems

1) General problems

Also in this mill, procurement of rods, one of the most important material is not smooth and for lack of rods grinding circuit cannot work effectively.

2) Receiving and crushing circuit

(1) In this circuit, two stage roll crushers have deteriorated seriously and must be frequently repaired, which disturbs the operation of the separation circuit. Although countermeasures are taken at the mill, it is necessary to install cone crushers, etc., as soon as possible.

(2) To make the operation of the separation circuit stable an automatic feed system must be introduced.

3) Sand table circuit

(1) The quantity of tin disposed as Colas Arenas is large. It will be more effective to intend to recover tin from coaser tailing by strengthening the regrinding circuit than the present policy of improving the recovery of tin from the finer table circuit.

(2) The classification of feed ore is quite important to operate the table efficiently. When the operation is normal and stable, the classification works efficiently, but the problem in the crushing circuit mentioned above greatly fluctuates the quantities and the particle sizes of the feed to the tables, making the operation of the tables unstable.

(3) The circuit is considerably complicated, and it is difficult to analyze the state of operation to plan the improvement of the circuit.

4) Slime table circuit

Cone type thickeners or cyclones are used for the thickening of feed to tables, but the operation control of the former is difficult and the control of the latter is not also performed satisfactorily.

3–4 Kinko Mill Plant

This is a concentration plant in which tin is recovered by the flotation process from old slime tailings accumulated in Lake Kenko.

However, from the condition that Lake Kenko has a function to recover water for the Victoria mill from slime tailings and it is difficult to control the level of the lake freely because the climate of the district has the wet season and the dry season, the plant has an essential problem that dredging cannot be carried out efficiently.

Moreover, the proper operation of the plant cannot be carried out at present because of trouble with the dredger and the plant is in an anomalous state that a part of old tailings accumulated on the surface is transported by tracks and treated temporarily.

Fig. 3–6 shows the change in quantity of recently treated ore.

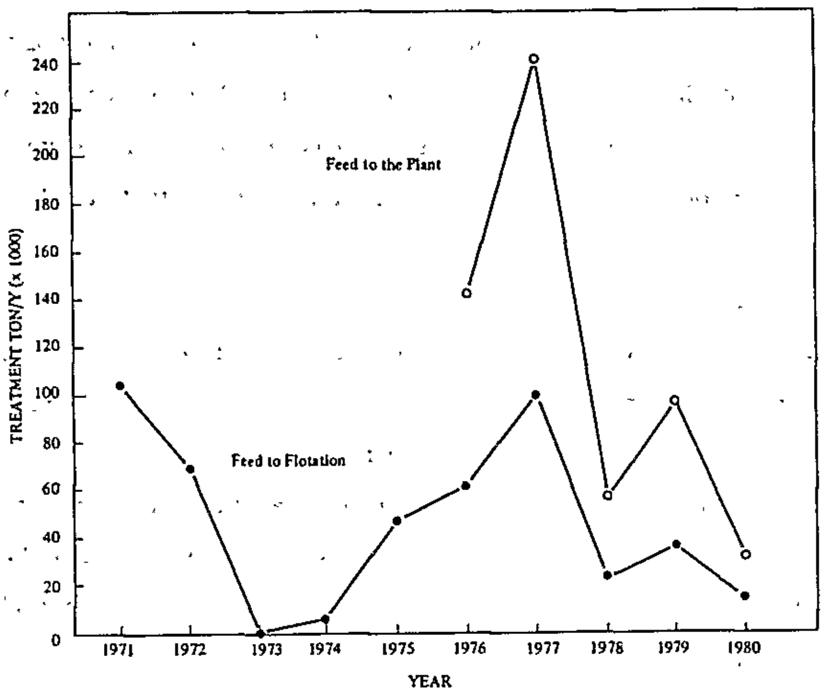


Fig. 3–6 Anual Treatment of Kenko Plant

3–4–1 Organization and Personnel

In this plant, 23 salaried personnel including three engineers and 42 day-laborers are engaged in the operation of the plant.

Fig. 3–7 shows the outline of the organization.

		Foreman	Salary-labour	Day-labour	Total
Superintendent — assistant 1 2	Secretary		1		1
	Dredger	1	2		3
	Operation	3	3	28	34
	Concentrate Treatment	1		9	10
	Maintenance	1	7		8
	Sampling			3	3
	Others		1	2	3
Total 3 (number of Engineers)		6	14	42	65

Fig 3–7 Organization of the Kenko Plant

3–4–2 Metallurgical Results

The metallurgical result of this plant in 1980 is shown in Table 3–8. As mentioned above, the form of operation is a temporary one.

	Weight		Grade	Sn	Distribution
	Ton / y	%	Sn-%	Ton / y	%
Feed	31,160	100.0	0.70	218.6	100.0
Classification Tailing	17,508	56.2	0.67	117.4	53.7
Flotation Feed	13,652	43.8	0.74	101.2	46.3
Sn Concentrate	324	1.0	14.18	45.9	21.0
Flotation Tailing	13,328	42.8	0.41	55.3	25.3
Total Tailing	30,836	99.0	0.56	172.7	79.0

Table 3–8 Metallurgical Balance of Kenko Mill Plant (1980)

3–4–3 Process and Equipment

The proper process of this plant is explained in the following.

1) Dredging

In this circuit, an Ellicot 4005 type dredger is used for dredging and it pumps old slime tailings from Lake Kenko to the plant.

The nominal capacity of the dredger is 150 – 200 ton/h at a pulp density of 20%.

After scalping of misplaced materials by low head screens, dredged pulp is stored in a wooden tank of 30 ft D x 20 ft.

2) Classification circuit

Pulp drawn out from the wooden tank is pumped into a circuit, where coarse particle are removed at first. There are two circuits for coarse particle removal, one of whcih consists of one Krebs 20-inch cyclone and an Akins classifier.

The another circuit consists of four Patterson 14-inch cyclones, four DSM screen and six Abe-Mathews screens.

Next, the pulp from which the coarse particles have been removed is classified by thirty Krebs 4-inch cyclones and twenty-eight 4-inch Dorrclone's, and is declimed.

The coarse particles and the slime are disposed into Lake Golden City nearby and the particles classified into 150 – 100 μ sizes are stored in two wooden agitators of 32 ft D x 20 ft for flotation feed.

3) Flotation circuit

Flotation feed is thickened in twenty-eight 4-inch Dorrclone's and conditioned in two Wemco No. 20 attrition tanks with Xanthate Z-11, which is used as a collector for pyrite. In the pyrite flotation, six cells of Fagergren No. 66 flotation machines are used, and the float from them is disposed into Lake Golden City.

Tin flotation after the pyrite flotation is carried out at two-stages, which is rougher flotation and cleaner flotation, it is an acid circuit flotation using AP 860 as a collector for cassisterite.

As step prior to rougher floatation, thickening the feed is treated by cyclone classification attrition-dilution-thickening method, and then pulp density and pH of pulp are adjusted in agitators. Twenty-eight 4-inch Dorrclones are used for thickening and eight Wemco No. 20 attrition tanks for attrition, while flotation fourteen cells of Fagergren No. 66 flotation machines arranged in two parallel lines are used.

After excess reagent is removed from rougher float in twelve Krebs 4-inch cyclones and twenty-five Patterson 3-inch cyclones, the rougher float is conditioned for tourmaline depression in four Wemco No. 20 attrition tanks with waterglass and caustic soda. The pH value of the pulp in these tanks is 9 – 11. The rougher float is again diluted, and after being subjected to the removal of excess reagent by twelve Patterson 3-inch cyclones, its concentration and pH value are adjusted, and the float is fed to the cleaner flotation stage. The pH value in the cleaner flotation stage is about 1.5.

In the cleaner flotation stage, twelve cells of Denver Sub-A No. 15 flotation machines are used as the main circuit in two parallel lines and four cells are used as an auxiliary circuit and tin concentrate of about 15 – 20% grade is recovered in the stage.

The concentrate is concentrated in cyclones and settling tanks, dewatered by a disc filter of 4-ft diameter and four leaves, dried by a drier and shipped.

Table 3–9 shows reagent consumption, and Fig. 3–8 shows the balance sheet of ore quantities surveyed in 1971.

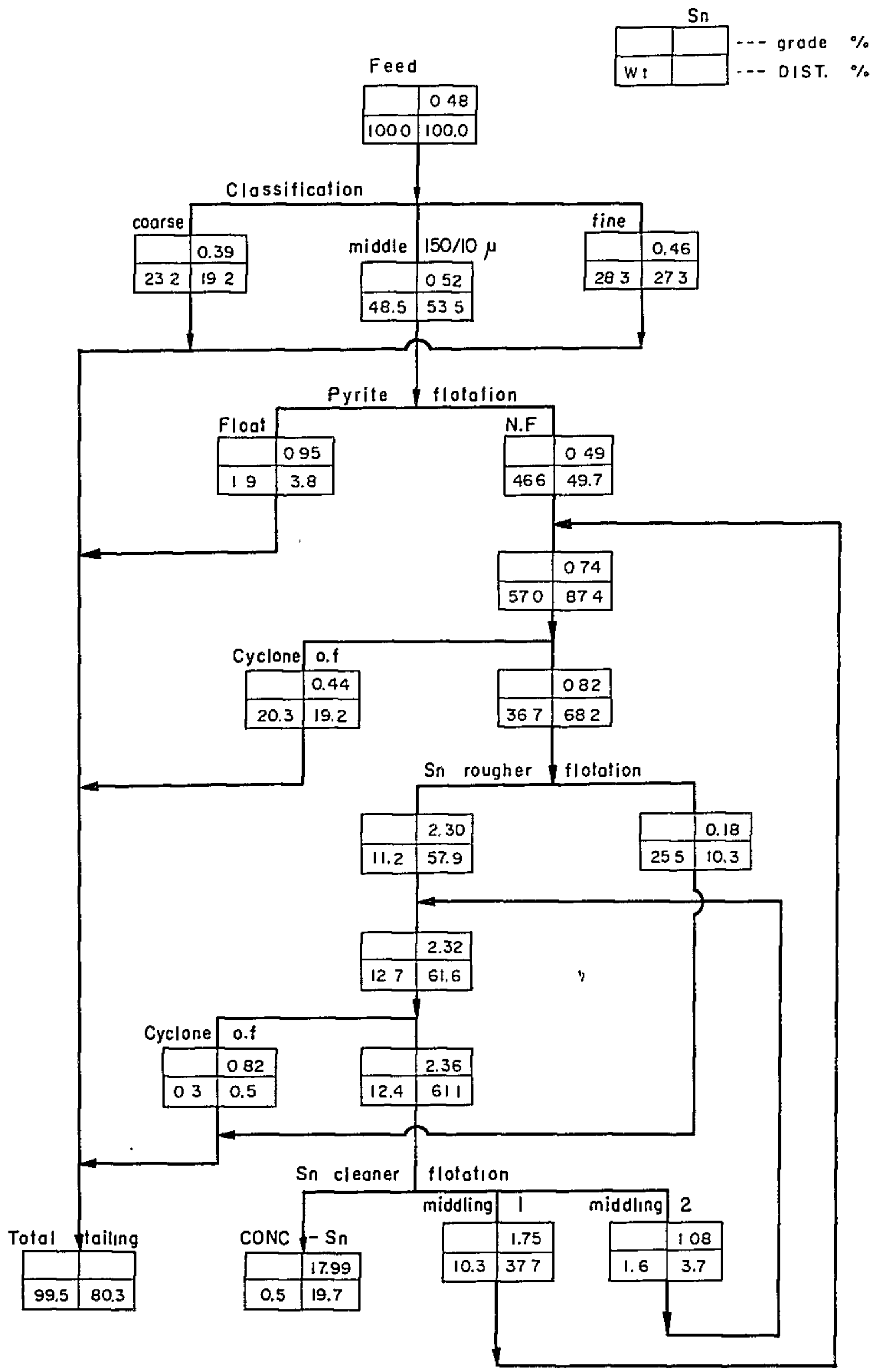


Fig. 3-8 Balance Sheet of Kenko Plant (1971)

Table 3–9 Reagent Consumption in Flotation Section

	g/ton
Sulphuric acid	5,347
A.P. #860	853
Sodium silicate 'E	1,262
Sodium hydroxide	341
Xanthate z–11	27
A.F #65	15
Terpene	13
Methanol	5

3–4–4 Operating Cost

The outline of operating cost in the past five years and the contents of material cost are shown in Tables 3–10 and 3–11.

Table 3–10 Operating Cost of Kenko Mill Plant

	1976		1977		1978		1979		1980	
	$ U.S/Y	%	$ U.S/Y	%	$ U S/Y	%	$ U.S/Y	%	$ U S/Y	%
Personnel Expenses										
Direct	37,857	8.9	75,680	10.1	53,292	12.6	109,856	15.5	118,404	15.5
Indirect	44,708	10.5	90,712	12.2	59,828	14.2	118,290	16.7	136,156	17.8
Total	82,565	19.4	166,392	22.3	113,120	26.8	228,146	32.2	254,560	33.3
Materials	173,880	40 9	313,160	42.0	158,188	37.5	261,754	36.9	262,153	34.3
Transportation	763	0.2	1,144	0.2	1,045	0.2	11,477	1.6	8,638	1.2
Auxiliary Service	127,038	29.9	168,502	22.6	104,656	24.8	165,054	23.3	198,075	25.9
Depreciation	39,749	9.3	94,628	12.7	39,659	9.4	38,060	5.4	39,868	5.2
Others	1,069	0 3	1,051	0.2	5,469	1.3	4,210	0.6	1,020	0 1
Total	425,064	100.0	744,877	100 0	422,137	100.0	708,701	100.0	764,314	100.0
Unit Cost $U S/T	3.04		3.11		7.48		7.46		24.53	
Treatment Ton/M	11,658		19,990		4,701		7,920		2,597	

Table 3–11 Cost of Materials (Kenko Mill Plant)

(1980.7 ~ 1981.6)

	$ U.S/y	%
Parts of Dredger	11,997	5.3
Parts of Cyclone	2,755	1.2
Parts of Pump	4,891	2.1
Sulphuric Acid	93,219	41.0
Flotation Reagent	48,409	21.3
Fuel Oil	6,169	2.7
Steel	6,455	2.8
Wood	7,202	3.2
Others	46,448	20.4
Total	227,545	100.0

3–4–5 Actual Problems

1) At present, the plant cannot be operated properly because of the dredger trouble, it is therefore necessary to replace the dredger immediately.

2) For the cleaner flotation of tin, a costly method to increase the pH value once to about 10 and then decrease it to 1.5 is used to depress gangue minerals. To reduce costs, it is necessary to discover through research same new gangue depressing technology that can be substituted for the present method or look for some new separating method which can be substituted for flotation.

3) In the classification circuit, much tin is lost by being disposed as coarse particles and slime. To reduce the loss of tin into the slime, a more efficient separator, for example, a centrifugal separator, must be investigated and about the coarse particles, the appropriateness of the regrinding circuit must be investigated.

4) During the survey this time, it was impossible to investigate the state of proper operation, but in so far as the state of operation at the time of test-running of the dredger is concerned, it seemed that the rate of distribution of underflow and overflow of each cyclone was inappropriate and this circuit combining various kinds of classifiers was not balanced well.

3–5 Metallurgical Test

3–5–1 Outline

Since it is a very important subject in this mine to select some new ore as the object of treatment and to establish new treatment techniques for the ore, we chose Colas Arenas, Desmonte and Block Central ore for the test treatment and carried out a series of metallurgical tests.

Prior to separation tests, chemical analyses, observation under microscopes and by EPMA and grindability tests were carried out first to grasp the properties of each ore, and then sink-and-float analyses, flotation tests and gravity concentration tests were carried out. In the cases of these ores, cassiterite exists distributed finely and it seems difficult to separate it by the present method in coarse grain size ranges, accordingly, concentration tests in finer grain size ranges were carried out.

Since the tests conducted this time were merely basic tests, a final concentration technique cannot be determined from them, but they will give an index for the future study of treatment techniques.

3–5–2 Property and Constituent of Ore

1) Chemical composition

Table 3–12 shows the results of the chemical analyses of three samples, Colas Arenas, Desmonte and Block Central, used for the tests.

Table 3–12 Chemical Analysis of Crude Ore

	Cu	Zn	Fe	S	Sn	Ti	WO_3	Mn
Colas Arenas	0.004	0.01	2.06	0.24	0.28	0.23	0.015	0.013
Desmonte	0.005	0.02	1.96	0.34	0.25	0.27	0.014	0.010
Block Central	0.017	0.13	2.45	0.96	0.22	0.30	0.045	0.017

	CaO	K_2O	Na_2O	Ni	Al_2O_3	SiO_2	MgO
Colas Arenas	0.05	1.60	0.24	0.55	12.82	76.02	0.51
Desmonte	0.05	2.14	0.25	0.53	15.01	71.66	0.54
Block Central	0.20	1.44	0.25	0.71	15.51	68.22	0.70

2) X-ray analyses

In the X-ray fluorescence analyses of the three samples, the elements, Fe, Sn, Zr, Zn, As, Al, K, Ti, Si, S and Ca, were detected, and by X-ray diffraction analysis, tourmaline, sericite, phlogopite and quartz, were identified.

3) Microscopic observation and analysis by EPMA

No big difference was found among Colas Arenas, Desmonte and Block Central. Cassiterite, tourmaline, quartz, pyrite and pyrrhotite were discovered in these samples under the optical microscope.

Tin exists, as cassiterite whose larger grain size is about 50 μ, while smaller grain size is submicron, and sometimes the cassiterite exists in nearly a disseminated state; such cassiterites are often found in the state of paragenesis with a titanium mineral which seems to be rutile (TiO_2), and also such tin exists forming solid solution with titanium minerals. On the other hand, cassiterite exists in a state that it fills the gaps of tourmaline or it is sandwiched between tourmaline and quartz.

From the conditions above mentioned considerable difficulty is anticipated in the concentration of these three kinds of ore, and the key point in the recovery of tin concentrate is thought to be to recover also titanium minerals together with tin concentrate and to separate it from tourmaline.

3–5–3 Grindability Test of Ore Milling Test of the Ore

To examine the grindability of these ore, work index measurement by the usual method was carried out. The result gave values 15 – 16 KWh/T, which show that these ores are considerably hard, since the work indices of usual ores are 7 – 11 KWh/T.

Also in the case when the ore was ground to flotation sizes using a standard pot mill, considerably longer grinding time was required compared with usual ore, and the test result showed that Colas Arenas was especially hard to grind.

3–5–4 Sink-and-Float Analysis

1) Desmonte

A Desmonte sample ground entirely to under 36 mm sizes and screened as shown in Table 3–13 was subjected to sink-and-float analysis. As heavy liquid, terabromoethane + ethyl-alcohol was used. No test was carried out with sample E.

The results of the sink-and-float analysis are shown in Figs. 3–11 – 14. These results show that as the grain size becomes smaller, selectivity according to specific gravity is improved. About samples A, B and C, tin distribution curves and weight distribution curves are close, which lower selectivity, and moreover, the gradients of the tin distribution curves are steep. The fact means that a small fluctuation of separating specific gravity has a big influence on the recovery of tin and it is very difficult to carry out stable operation.

As high selectivity is seen with sample D, and as this is a hard ore, heavy media separation will be effective.

When the existence of cassiterite in the state of very fine grain sizes found by microscopic observation is taken into account, it is necessary to carry out heavy medium separation at grain sizes finer than the heavy medium separation grain sizes employed in the plant at present, and heavy liquid cyclones can be used.

Table 3–13 Size Distribution of Sample

Sample	Size (mm)	Dist (%)
A	36 ~ 9.25	20.5
B	9.25 ~ 4.76	31.9
C	4.76 ~ 1.68	21.3
D	1.68 ~ 0.21	17.0
E	−0.21	9.3

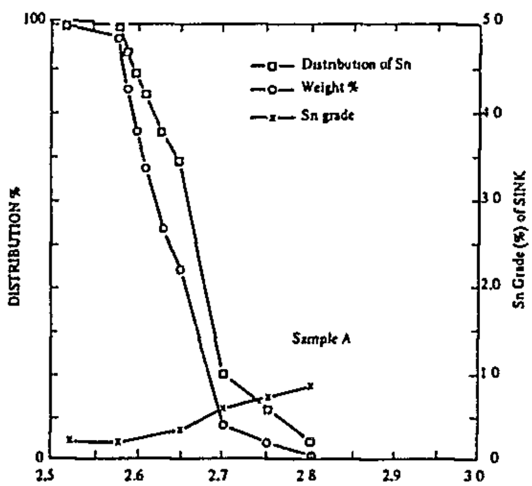


Fig. 3–9 Result of Sink and Float Test (Desmonte 36/9.25 mm)

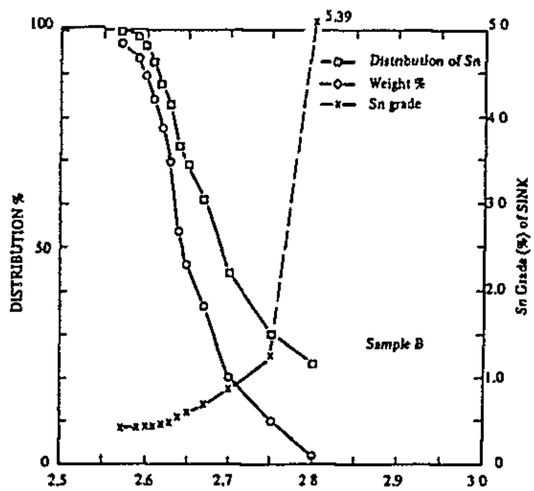


Fig. 3–10 Result of Sink and Float Test (Desmonte 9.25/4.76 mm)

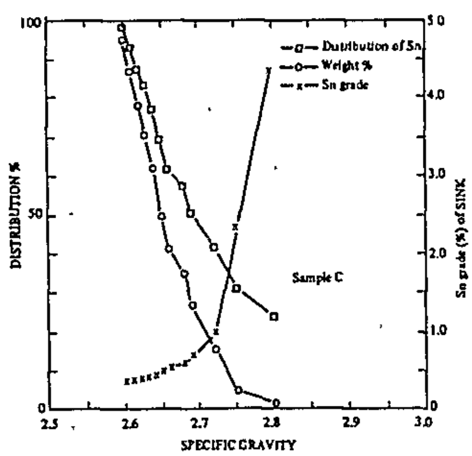


Fig. 3–11 Result of Sink and Float Test (Desmonte 4.76/1.68 mm)

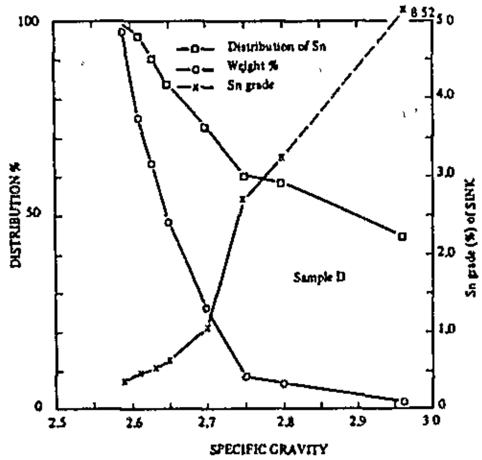


Fig. 3–12 Result of Sink and Float Test (Desmonte 1.68/0.21 mm)

2) Block Central

Similar sink-and-float analyses were carried out with Block Central ore. Figs. 3–13 ~ 15 show the results.

Also here, a tendency is clearly seen that selectivity improves as grain sizes become smaller, but selectivity is not high enough even in the case of the 5–1 mm sample, and it can be concluded that it is necessary to grind to still finer grain sizes and improve selectivity before carrying out heavy medium separation.

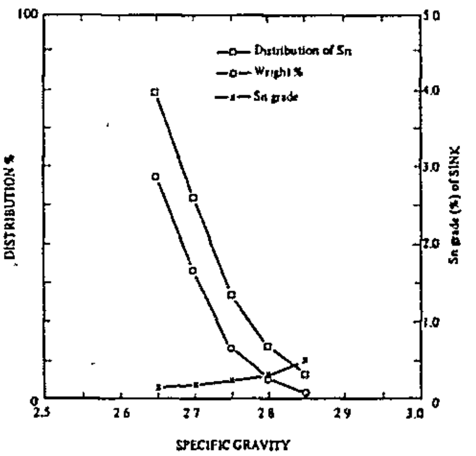


Fig. 3–13 Result of Sink and Float Test (Block Central 20/10 mm)

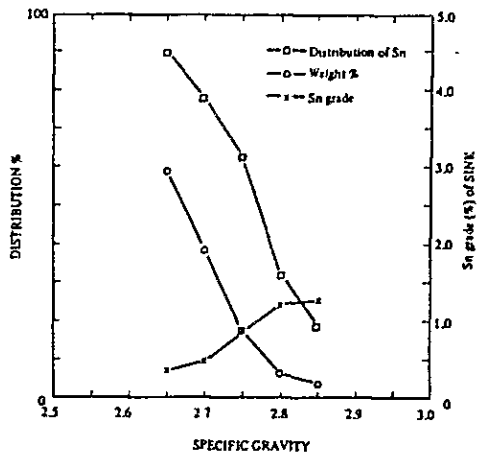


Fig. 3–14 Result of Sink and Float Test (Block Central 10/5 mm)

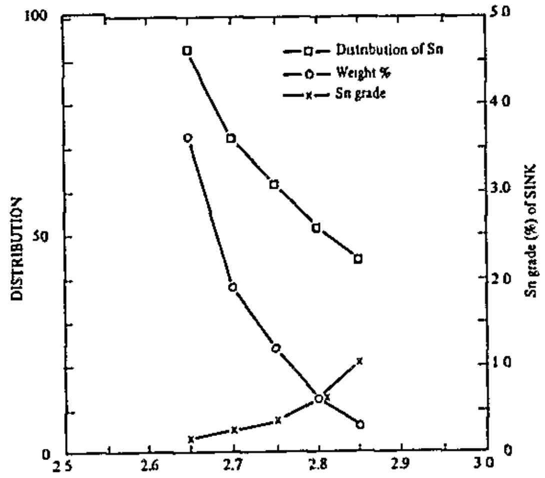


Fig. 3-15 Result of Sink and Float Test

(Block Central 5/1 mm)

3-5-5 Flotation Test

When carrying out flotation tests, a series, grinding-desliming-pyrite-flotation-rougher tin flotation-cleaner tin flotation of test were employed as a basic flow chart and various conditions were investigated.

1) Selection of collector

As a result of microscopic observation, it was found that tin grains were very fine, accordingly, sample ore was ground to 75% −400 mesh and fed to desliming and pyrite flotation with amilxanthate, then the floatability tests of tin with various collectors were carried out. Fig. 3-16 shows the example of the recovery and the grade of tin in the rougher flotation about Colas Arenas. As seen in the figure, ACC 830 and KL 2,700 showed good results as collectors, it was therefore decided to carry out subsequent tests using these two collectors.

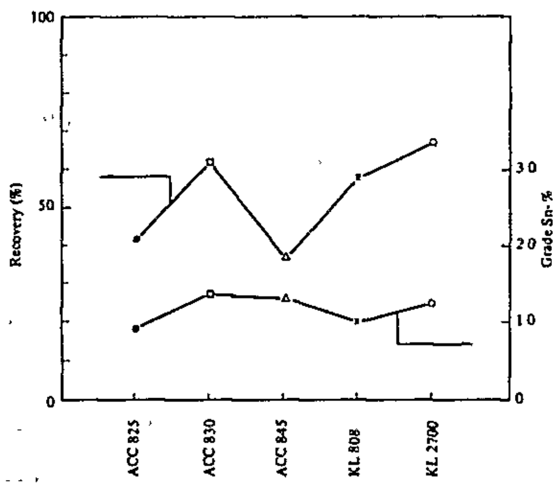


Fig. 3-16 Floatability with various collectors

2) Investigation on flotation grain size

Generally, the flotation selectivity of fine grains is low, and as cassiterite is especially easily over ground, and moreover, as there is a problem that tin is lost at the desliming stage, overgrinding must be avoided as much as possible.

On the other hand, to secure the adequate degree of liberation in the case of these ore, considerable degree of grinding is supposed to be required, accordingly, it is quite important to find the optimum grain size.

When compared the variation of the flotation characteristics of the three ores with grain sizes in the rougher flotation of tin, a little difference was found among the three ores. For example, in the case of Colas Arenas, the recovery percentage in rougher flotation is almost constant regardless of its grain size, but the tin grade of the froth drops as the grain size becomes smaller. While in the case of Desmonte, the tin grade of the froth lowers as its grain size becomes smaller similarly to Colas Arenas, but a tendency appeared that the recovery increases. However, when a total recovery including the loss of tin at the stage of desliming or grinding cost is considered, it is better to carry out flotation in comparatively a coarse grain size range, and finally a flotation grain size of about 45% – 400 mesh was adopted for all of the three kinds of ore.

3) Cleaner flotation test

For the depression of gangues, several depression conditions were set and their comparison tests were carried out. As a result of the tests, it was found that a method to use sodium silicate as main and sodium silicofluoride with it for depresant can get good results, accordingly, this method was adopted for the cleaner flotation test. Up to now, ACC 830 and KL 2,700 have been used as collector, but it was found that, in the cleaner flotation process, KL 2,700 was more effective in the improvement of concentrate grade. An example is shown in Fig. 3–19. The value of pH was 2.3 in rougher floation and 1.7 in cleaner flotation.

Using the various conditions selected as mentioned above as optimum conditions, the flotation tests of Colas Arenas, Desmonte and Block Central ores were carried out and the results as shown in Tables 3–14 ~ 3–16 were obtained.

The grades of tin concentrate obtained by flotation were 13.64%, 14.65% and 18.41% respectively, and it was fairly difficult to improve grades. Perhaps this is because it is difficult to depress tourmaline, and multi stage of cleaner flotation will be required to improve grades.

As the cleaner flotation included many stages, tin recovery percentage was low, but since cleaner flotation tailings are returned as flotation feed ore, the actual recovery percen-

Table 3–14 Flotation Test of Colas Arenas (No C25)

Product	W %	Grade %		Distribution %	
		Sn	Fe	Sn	Fe
Feed	100 0	0 26	1 78	100.0	100 0
Cy Of	12 0	0.16	2.74	7 3	18 5
Py C	1.9	0 30	5 60	2.2	6.1
Sn Feed	86 1	0.28	1.56	90 5	75.4
Sn RC	18.1	1.09	3.43	75 0	35 0
Sn RT	68.0	0 06	1 06	15.5	40 4
Sn 1C1C	5 4	3.18	5.28	65.3	16 1
Sn 1C1T	12.7	0 02	2.65	9.7	18 9
Sn 2C1C	2.0	7.54	6.11	57.0	6 9
Sn 2C1T	3 4	0 64	4.79	8 3	9.2
Sn 3C1C	1.4	10.32	3 72	52 9	2.9
Sn 3C1C	0 6	1 69	11.14	4 1	4 0
Sn 4C1C	0.9	13 63	3.18	45 3	1 6
Sn 4C1T	0.5	4 20	4.73	7 6	1.3
Total C1T	17 2	0 45	3 44	29.7	33 4

Table 3–15 Flotation Test of Desmonte (No D7)

Product	W %	Grade %		Distribution %	
		Sn	Fe	Sn	Fe
Feed	100 0	0 31	1 68	100 0	100 0
Cy Of	15 2	0 17	1 87	8 3	16 8
Py C	1 6	0.37	20 23	1.9	18.7
Sn Feed	83.2	0.33	1.30	89 8	64 5
Sn RC	16.3	1.49	2 91	79 0	28 4
Sn RT	66 9	0 05	0 91	10 8	36.1
Sn 1C1C	4 3	5 16	4 16	72 8	10 9
Sn 1C1T	12 0	0 16	2.46	6 2	17 5
Sn 2C1C	2 4	8 45	4 02	67 3	6 0
Sn 2C1T	1 9	0 89	4 34	5 5	4.9
Sn 3C1C	1 7	11 39	3.64	63.1	3 8
Sn 3C1T	0.7	1 72	4 90	4 2	2 2
Sn 4C1C	1.2	14 64	3 29	56 5	2 4
Sn 4C1T	0 5	3 92	4 43	6 6	1.4
Total C1T	15.1	0 45	2 87	22.5	26 0

Table 3–16 Flotation Test of Block Central (No 3)

Product	W %	Grade %		Distribution %	
		Sn	Fe	Sn	Fe
FEED	100 0	0 23	2.78	100 0	100 0
Cy Of	16 7	0 18	4.23	13 2	25 4
Py C	4.1	0 31	19 41	5 7	28 9
Sn Feed	79 2	0 23	1 61	81.1	45.7
Sn RC	21.1	0.71	2 95	65 8	22.3
Sn RT	58 1	0 06	1 12	15.3	23 4
Sn 1C1C	2.5	4 54	4 05	51.1	3 6
Sn 1C1T	18 6	0 18	2 80	14 7	18 7
Sn 2C1C	1.0	9 63	3.76	46 4	1 4
Sn 2C1T	1.5	0.73	4.26	4.7	2.2
Sn 3C1C	0 6	14 04	3.34	41.8	0.7
Sn 3C1T	0 4	2.50	4 45	4 6	0.7
Sn 4C1C	0 3	18 41	2.83	33 4	0 4
Sn 4C1T	0 3	7.21	4.13	8 4	0.3
Total C1T	20.8	0 37	2 96	32.4	21 9

tage will be higher than these recovery rates.

The main flotation conditions are as follows.

Grain size :	45% ~ 400 mesh	
Pyrite Flotation :	pH 4.0, KAX 150 g/T	
	DOW # 250 20 g/T	
Tin Flotation :	Rougher flotation pH 2.3, Cleaner flotation pH 1.7	
	KL 2700	700 g/T
	Na_2SiO_3	2 kg/T
	Na_2SiF_2	500 g/T

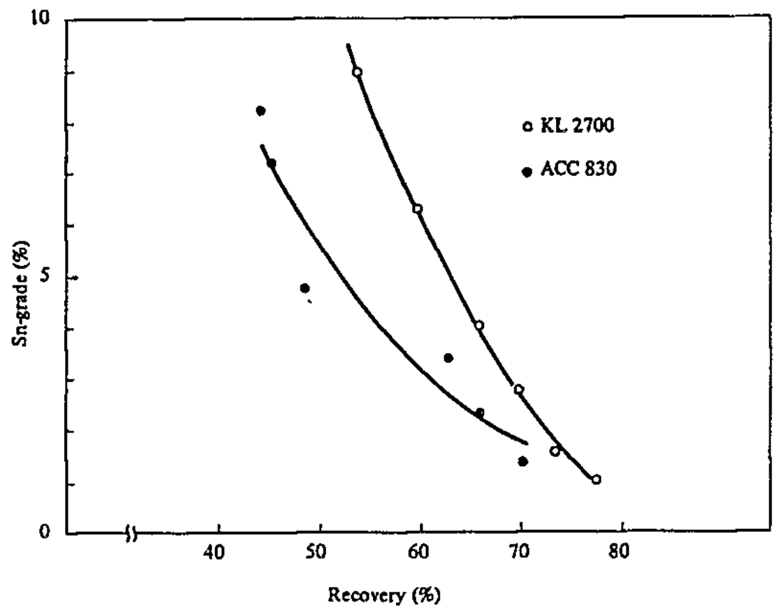


Fig. 3-17 Reration between grade and Recovery
(Cleaning Test of Colas Arenas)

3–5–6 Gravity Concentration Test

From the fact that cassiterite exists in finely dispersed states, we got an idea that flotation will be better suited than the table concentration method which has been the key method and repeated flotation tests, but the cost of grinding ore to the optimum flotation grain size is high because the ores are hard and the results of flotation experiment were not so good. Because of these results and also because it was found from the results of microscopic observation that the degree of liberation of cassiterite was comparatively high, the method of concentration with shaking tables was investigated.

However, for the reason that cassiterite exists in these ores in the state of fine grains, these tests were carried out in the range of grain sizes finer than those used in the table concentration in the mine particularly paying attention to the grain sizes.

In the table concentration tests carried out at first in a comparatively coarse grain size range, desirable results were not obtained, and it was found that it was difficult to improve the grade of concentrate because the degree of liberation was particularly low. Since it was anticipated that it would be difficult to recover concentrate in a range of several millimeter grain sizes from the result of sink-and-float analyses, each kind of ore was all ground to under 34 mesh for use as separation feed ore in later tests.

In the tests, each kind of ore was again screened with 65 mesh or 100 mesh and the coarser grain part and the finer grain part were fed to tables separately. In the case of coarser grains the grade of tailings could be lowered easily (0.02 ~ 0.05%), but the grade of concentrate could be improved only by a few percent at maximum. On the other hand, in the case of finer grain part, the grades of concentrate were obtained, 12.92% for Colas Arenas, 24.73% for Desmonte and 14.24% for Block Central showing results that the grade of concentrate can be improved more easily in the case of fine grains.

Based on the above results, each kind of ore was ground to under 100 mesh and subjected to pyrite flotation and table concentration. The results are shown in Tables 3–17 ~ 3–19.

The recovery percentage was about 45% for all three kinds of ores, while the grades of concentrate were, 33% for Colas Arenas, 44% for Desmonte and 21% for Block Central.

In each case, the grade of tailing was about 0.12%, a comparatively high value, but it seems that the loss of finely ground tin grains is the cause of the high value, and improvement in the recovery rate can be expected by developing a grinding circuit which generates less fine grains.

Table 3–17 Metallurgical Balance – Colas Arenas

	Wt %	Grade %		Distribution %	
		Sn	Fe	Sn	Fe
Feed	100.0	0.29	0.93	100.0	100.0
Pyrite-C	0.4	0.31	21.60	0.4	9.3
Pyrite-T	99.6	0.29	0.85	99.6	90.7
Rougher Table-C	7.6	2.18	3.51	58.1	28.6
Rougher Table-T	92.0	0.13	0.63	41.5	62.1
Cleaner Table-C	0.4	33.06	8.70	46.3	3.7
Cleaner Table-T	7.2	0.47	3.22	11.8	24.9

Table 3–18 Metallurgical Balance - Desmonte

	Wt %	Grade %		Distribution %	
		Sn	Fe	Sn	Fe
Feed	100.0	0.31	0.77	100.0	100.0
Pyrite-C	1.2	0.43	19.20	1.7	29.8
Pyrite-T	98.8	0.31	0.55	98.3	70.2
Rougher Table-C	4.9	3.91	1.51	61.9	9.5
Rougher Table-T	93.9	0.12	0.50	36.4	60.7
Cleaner Table-C	0.3	44.48	6.50	43.2	2.5
Cleaner Table-T	4.6	1.26	1.18	18.7	7.0

Table 3–19 Metallurgical Balance - Block Central

	Wt %	Grade %		Distribution %	
		Sn	Fe	Sn	Fe
Feed	100.0	0.24	1.14	100.0	100.0
Pyrite-C	2.1	0.40	27.70	3.6	51.1
Pyrite-T	97.9	0.24	0.57	96.4	48.9
Rougher Table-C	8.3	1.50	2.07	52.2	15.1
Rougher Table-T	89.6	0.12	0.43	44.2	33.8
Cleaner Table-C	0.5	20.53	17.20	43.0	7.5
Cleaner Table-T	7.8	0.28	1.10	9.2	7.6

About table feed ore, it seems that it will be more effective to employ classification and carry out separation by grain sizes than to carry out the separation of all the grain sizes at one time as in this test.

3–5–7 Property and Constituent of Products

1) Chemical composition

The chemical compositions of the flotation concentrate and the table concentrate obtained from the three kinds of ores, test objects, are shown in Tables 3–20 and 3–21.

Table 3–20 Chemical Analysis of Flotation Concentrates

	Cu	Zn	Fe	S	Sn	Ti	WO_3	Mn
Colas Arenas	0 005	0 019	3 18	0 03	13 63	4 88	0 011	0 012
Desmonte	0 007	0 016	3 29	0 19	14 64	6 06	0.024	0 013
Block Central	0 016	0 069	2 83	0 90	18 41	6 40	0 033	0.010

	CaO	MgO	K_2O	Na_2O	Al_2O_3	SiO_2	Ni
Colas Arenas	0 48	2.82	0 15	0 45	21 27	28 18	0 001
Desmonte	0 47	2 48	0 15	0.38	21 04	29 22	0 002
Block Central	0 27	2 28	0 05	0.37	17 85	21 74	0 003

Table 3–21 Chemical Analysis of Table Concentrates

	Cu	Zn	Fe	S	Sn	Ti	WO_3	Mn
Colas Arenas	0 061	0 135	8 70	3.23	33 06	7 04	1 12	0 195
Desmonte	0 030	0.068	6 50	1.85	44 48	4 52	0 916	0 172
Block Central	0 088	0 111	17 20	6 01	20 53	4.36	3.76	0.692

	CaO	MgO	K_2O	Na_2O	Al_2O_3	SiO_2	Ni
Colas Arenas	0 22	1.18	0 117	0 116	10 76	21 61	0 005
Desmonte	0.37	–	0.165	0 135	3 65	9 70	0 011
Block Central	0.34	0 61	0 100	0.107	6.42	16.98	0 099

2) X-ray analysis

In addition to cassiterite, tourmaline, rutile and quartz were identified by X-ray diffraction. The low peak values of tourmaline in the table concentrate compared with those in the flotation concentrate can be regarded as characteristics.

3) Analysis by means of microscopic observation and EPMA

(1) No substantial difference was found in the results of microscopic observation of the three kinds of ores, and much cassiterite and rutile were observed. Most of the free particles in them are about 10 ~ 50 μ in size, but coarse particles of 50 ~ 100 μ order also exist. The two kinds of ores often existed being middling of each other, and the particles of solid solution of them were also observed. Most of other minerals are tourmaline, and alumino-silicate minerals were partly observed.

(2) Also in the case of table concentrate, the same thing can be said about the state of existence of tin and titanium as flotation concentrate, but particles in this case are usually coarse and most of them are those over 50 μ. As other heavy element minerals, those which seem to be wolframite, zircon and rare earth minerals (Ce, La, P) exist, and these minerals were observed as comparatively coarse particles (50 ~ 150 μ) and yet as free particles. Particularly, woframite exists in comparatively large quantities.

3–5–8 Consideration

1) Three kinds of ores selected as test objects are all hard and it is anticipated that their grinding cost will be high, it is therefore important to investigate the adoption of preconcentration.

2) However, the form of existence of cassiterite is very fine, and so even preconcentration must be carried out in a fine grain size range under several millimeters. For such preconcentration, it will be advantageous to adopt heavy liquid cyclones.

3) As a kind of ore to be selected, the selectivity of Desmonte is best. Perhaps this is because the ore was subjected to separation processes not so much and cassiterite is more easily liberated as compared with other ores.

4) To recover high grade concentrate, table concentration will be superior to flotation. Because depression of tourmaline is difficult and the grade of concentrate can hardly be improved, and more stages are required in cleaner flotation in the case of flotation, accordingly, flotation has such disadvantages that higher cost of flotation reagents including sulfuric acid and electric power will be required.

5) Accordingly, it is difficult to adopt flotation independently as means for concentration, and it will be necessary to develop a method to apply flotation partly to the treatment of fine particles or to apply it only to rougher flotation in the stage of comparatively coarse particles since flotation is suitable for mass-treatment.

6) Also in the case of adopting shaking tables, grinding to the same particle size range as that of flotation is required finally.

7) As it is easy to lower the tin grade of table tailings in a coarse particle range, it is necessary to investigate a method to carry out rougher separation with shaking tables before fine grinding to meke preconcentration effective. By so doing, the cost of fine grinding required to recover good concentrate can be reduced.

8) As a result of analysis by EPMA, much rutile was observed as an interesting mineral and also in table concentrate were observed wolframite, zircon, rare earth minerals, etc., in the form of coarse free particles.

9) To analyze the present state of operation of three concentrating plants in Catavi Mine, considerable amounts of samples were brought back and various analyses were carried out. However, it was very difficult to analyze the present state of the plants from the data of the analyses of the samples obtained by asking the people there. Perhaps there may have been some problems in the method of sampling or the treatment there may have fluctuated too much, but that is not certain. Accordingly, that data are omitted in this report.

3–6 Conclusion and Suggestion

From the result of pointing out problems based on the field survey of the present state of the metallurgical section in Catavi Mine and the result of basic metallurgical tests carried out in Japan about the ores which will be treated in future, the following conclusion can be drawn.

All the problems are resulting from a fact that presently used ores are of low grade and low profit type, nevertheless the operation form of high grade and high profit type is still continued. Moreover, the crude ore of low tin grade itself is going to be exhausted at present and yet the easy-going way of operation is still continued. Such conditions must be reconsidered seriously, and it must be urgent that the present form of operation be changed for that matching the treatment of low tin grade ores. Hence, the following proposals are made.

1) Proposals for present operation

(1) Study on technical improvement to overcome cost increase is required

The result of analyzing the actual state clearly shows that the biggest factor in increasing cost is the increase of personnel cost. It is necessary to wrestle with the reasonable disposition of personnel use. It is necessary to realize a system which can operate the plants with fewer personnel by simplifying circuits, increasing the size of equipment, etc.

The second factor of cost increase is the increase of material costs. The reduction of repairing costs by the thorough execution of periodic maintenance, the reduction of main material costs by the simplification of circuits, etc., are required.

(2) It is necessary to improve the system of supplying main materials and secure balls, rods, sulfuric acid, etc., regularly.

(3) It is necessary to strengthen the system of maintenance and reduce the breakdown of equipments.

(4) As the libration size of cassiterite has become smaller, it is necessary to strengthen the grinding circuit and make the particle size smaller than that at present. In this case it is necessary to investigate whether the preconcentration with the cone heavy medium separator in Siglo XX Sink & Float Plant is appropriate or not.

(5) It is necessary to simplify circuits so that the state of operation can be grasped and analyzed more easily.

Particularly in the case of Victoria Mill, it is an urgent work to change the method of using tables from the scavenging first type to cleaner main type, simplify circuits by strengthening of regrinding for middlings, and substituting the fine grain circuit for that of flotation to improve the present state of disposing much tin as Colas Arenas.

(6) It is necessary to strengthen classification and carry out the collection of tin concentrate by each grain sizes very carefully. It should also be planned to stabilize the quantity of feed ore to classifiers by means of constant ore feeding and to employ a centrifugal separator for the classification of fine grains.

2) Proposals about ores to be treated in future

(1) Among the three kinds of ores tested this time, Desmonte is most easy to treat and Colas Arenas and Block Central follow in order. Desmonte and Colas Arenas can be treated on appropriate scales, but Block Central ore will not be profitable when its development cost is taken into account.

(2) The sizes of cassiterite grains are about 50 μ at the largest and many particles of submicron order exist, therefore, cassiterite cannot be liberated unless it is generally ground into fine particles.

This is clear also when it was observed from the result of table tests that tin was recovered effectively when table concentration was carried out by grinding all the ores to under 100 mesh.

(3) Compared with the past, considerably higher degree of fine grinding is required, it is therefore quite natural that the method of treatment has to be changed and it is necessary to select the methods of concentration suitable for various grain sizes in the fine grain range and sufficiently investigate the combination of the methods in future.

(4) As a result of analyses in Japan this time, the existence of rutile, wolframite, zircon, rare earth minerals, etc., was confirmed in addition to tin. Study on the recovery techniques for these minerals will also be a subject in future.