1–2–5 *Transportation, Ventilation and Drainage*

(1) Transportation

The transportation in the sublevel stoping includes the transportation of ores, waste, materials and persons. The quantities of stoping product, the capacity of supplying materials and actual working time in stopes must be maintained. In this mining plan, existing equipment will be utilized to the maximum and will not be changed greatly.

In the transportation of ores, ores dumped from sublevel stopes into ore passes are drawn out by hand from steel chutes and loaded on mine cars in the L650 main haulage drift. In the L650 main haulage drift, one train is composed by connecting 13 mine cars to a ten-ton trolley locomotive and transports ores to the entrance of the ore dressing factory.

For transporting materials, small winches will be installed in ore passes to supply materials to each level.

For transporting persons, ladders will be made in the existing shafts and ore passes.

(2) Ventilation

In the sublevel stoping, ventilation is required to remove fumes after long-hole blasting. In this mining plan, the existing ventilation system in Catavi Mine will be used as it is without being changed greatly.

The ventilation system inlets the air from the L650 main haulage level and distributes it to each level through each service chute, and foul air is exhausted from the top ventilation chute via exclusive outlet adits. To accelerate ventilation, it will be necessary to install local fans as auxiliary equipment in the L551 scram drift.

In the long-hole blasting, if large scale blasting is carried out once a week on the day before a holiday and fumes exhausted during the holiday, the fumes can be exhausted almost completely.

(3) Drainage

The quantity of mine water in Catavi Mine is small, and most of it is that accompanied, the development of deep levels, so that a special drainage plan for the sublevel stoping plan will not be required. Accordingly, the existing equipment will be used unchanged.

1–2–6 Rock Pressure Around

In the sublevel stoping, pillars play an important role, and they have to maintain stoping caverns at least until the stoping of the whole area is finished. Therefore, the earth pressure which will be exerted on the pillars when stoping is finished becomes an item to be studied.

We simulated the pressure by the "Finite Element Method" using computers.

Block A was selected as an object ore block, and to avoid the influence of boundary conditions, the computation was carried out for a range of 800 mW x 800 mH. Fig. II–1–17 shows the mesh pattern of the entire computation range. As input data, the physical properties of rocks according to the test results of rocks obtained last year were used.

Table II–1–14 In-Put Data of F.E.M.

Kinds	Value
Depth from surface	428.5 m
Specific gravity	2.75 g/cm³
Young's module	3.34x10⁵ kg/cm²
Poisson's ratio	0.25

Computation results by the FEM method are shown in Fig. II–1–18.

According to the simulation results, maximum compressive stress which will act on pillars will be about 240 kg/cm² and tensile stress will not be generated, so that there will be no problem in carrying out this sublevel stoping.

In the current simulation, computation was carried out on the supposition of uniform rock, the results therefore cannot be applied directly to sublevel stoping in old mining blocks, but an approximate estimation can be made. However, measured such as the filling of old mining blocks, the reinforcement of pillars with concrete, etc., will be required.

1–2–7 Total Production Efficiency and Mining Cost

Total production efficiency and mining cost are computed as follows.

(1) Total Production Efficiency

1) Production Efficiency

Production efficiency computed for a stope in block A based on Table II–1–12 and Table II–1–13 is as shown in Table II–1–15.

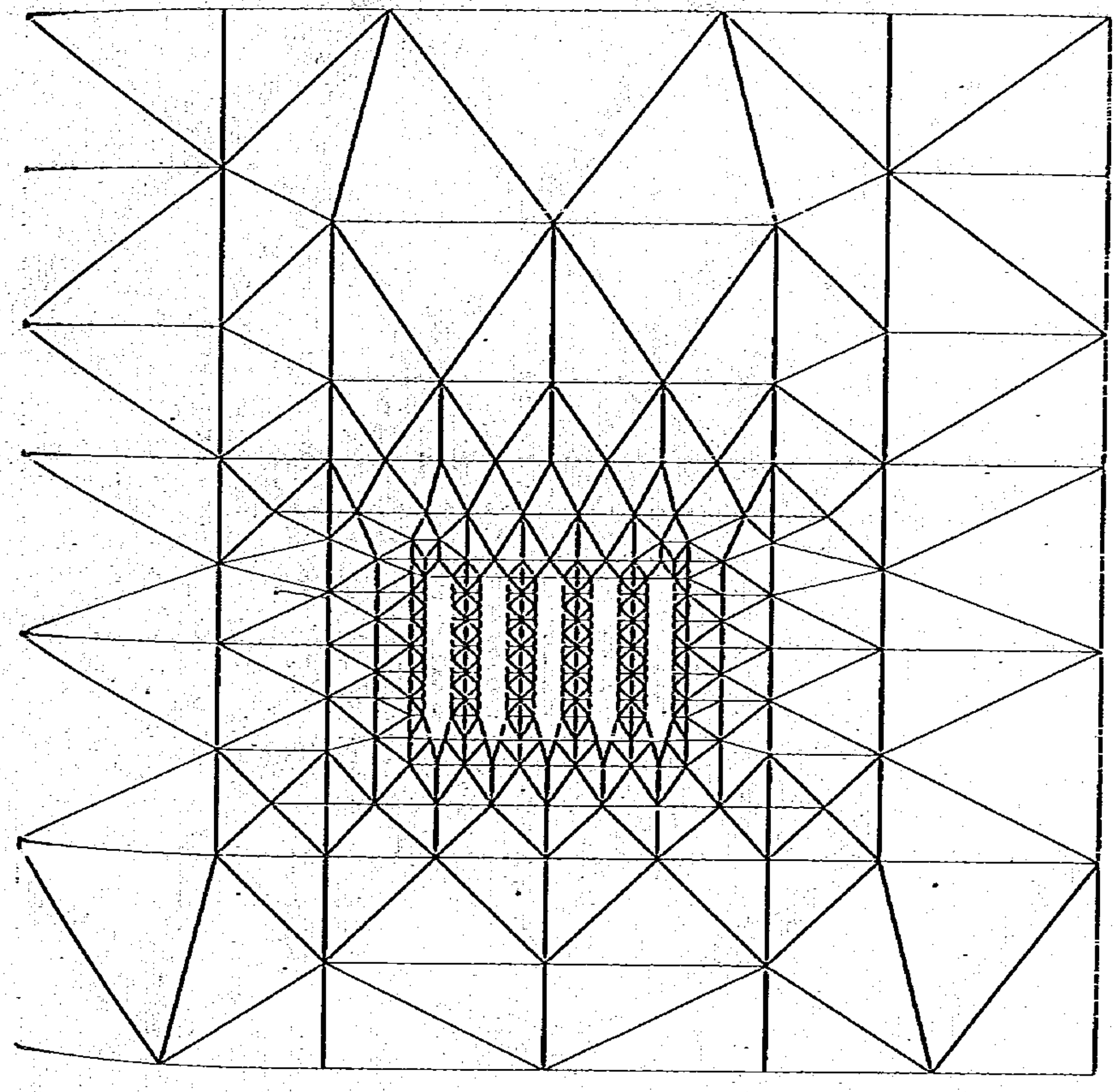


Fig. II–1–17 Mesh Pattern of F.E.M.

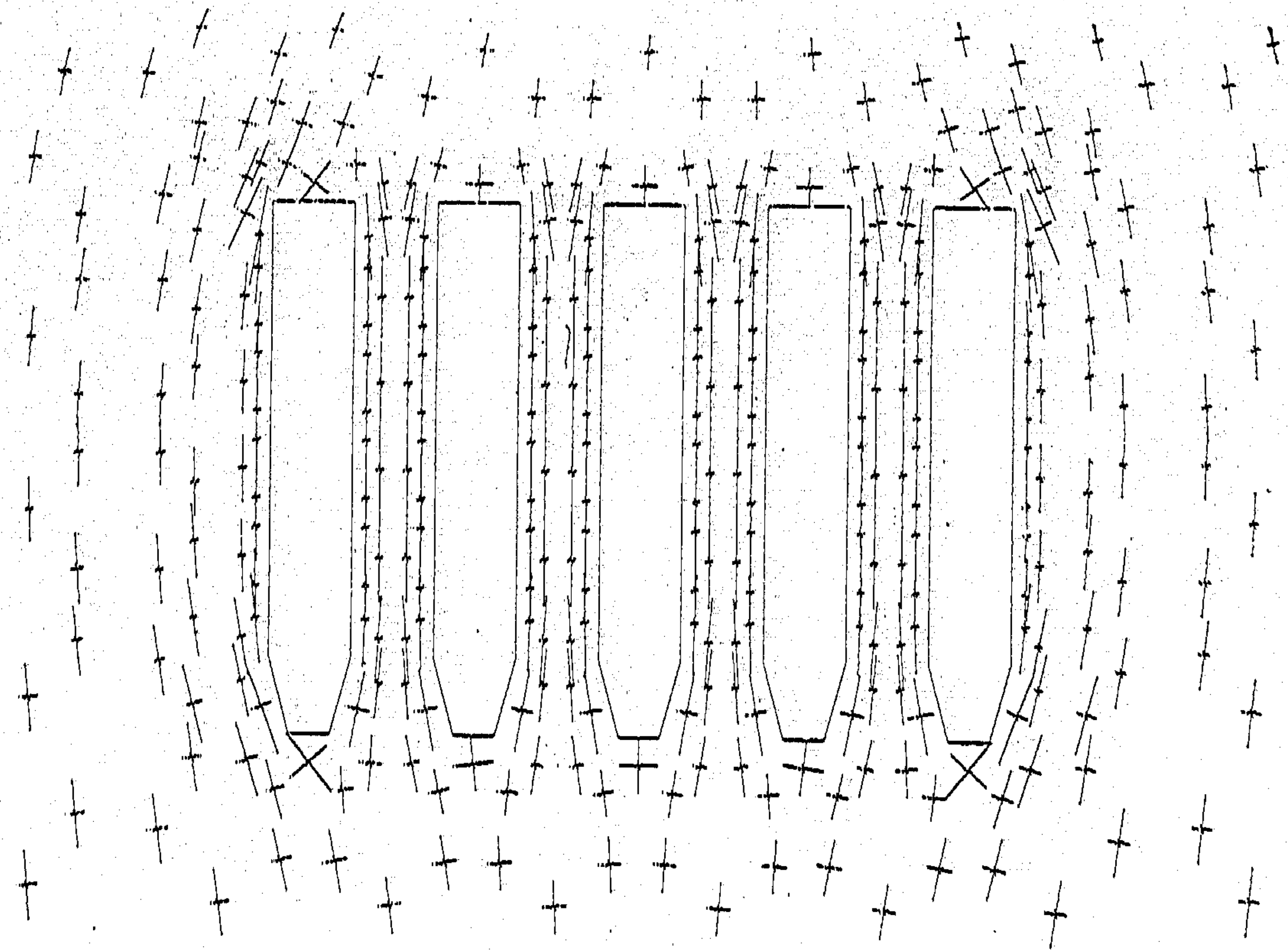


Fig. II–1–18 Stress Analysis by F.E.M.

Table II-1-15 Production Efficiency

Kinds	Figure	Unit per man-shift	Manpower requirement
Long-hole fan drilling	29,457.3 m	10 m/man-shift	2,946
Undercut drilling	1,168.3 m	15 m/man-shift	78
Slot drilling	2,014.6 m	10 m/man-shift	201
Draw cone drilling	2,340.3 m	4 m/man-shift	585
Boring	1,050.0 m	10 m/man-shift	105
Sub total			3,916
Ore handling	547,135 t	200 t/man-shift	2,736
Charge and blasting	547,135 t	300 t/man-shift	1,823
Maintenance equipment	547,135 t	300 t/man-shift	1,823
Rainforcement of rib pillar	547,135 t	300 t/man-shift	1,823
Others	547,135 t	300 t/man-shift	1,823
Sub total			10,028
Total			13,944
Tons per man-shift			39.3

2) Development Efficiency

Development efficiency in block A is computed as shown in Table II-1-16.

Table II-1-16 Development Efficiency

Items	Level	Chute
Development length (m)	10,290	2,363
Minable ore (t)	2,780,540	2,780,540
Development length per 1,000 tons (m)	3.7	0.85
Minable ore per one stope (t)	547,135	547,135
Average development length per one stope (m)	2,024.4	465.1
Manpower per meter	4.0	5.5
Manpower requirement	8,098	2,558
Total manpower	10,656	
Meter per man-shift	0.234	

From the above, the total production efficiency calculated including pure production and development shows that minable ore reserve is 547,135 tons, manpower requirement is 24,600 man-days, and the total production efficiency is 22.2 t/man-day. This is a reasonable value compared with records in Japanese mines.

(2) Mining Cost

The mining cost was calculated by dividing it into development cost, equipment cost and operating cost.

[Prerequisites]

1 The mining period shall be 10 years, crude ore production shall be 3,500 t/day until the seventh year and that from the 8th year on shall be 2,000 t/day.

2 The number of operation days per year shall be 300 days.

3 Actual working time per day shall be 15 hours (5 hrs/shift x 3 shifts) for the drilling and blasting operation and 18 hours (6 hrs/shift x 3 shifts) for the ore handling operation.

4 Equipment shall be depreciated fully in ten years, and as maintenance cost in ten years, 85% of the purchase price was counted in.

5 The equipment costs and transportation costs do not include interest.

1) Equipment costs

The equipment costs are as shown in Table II–1–17.

Table II–1–17 Mining Equipment Costs

Items	Scraper hoist	Ring drill crawler	Hopper loader	Alimak climber	Rocker shovel
Number (Unit)	4	6	7	2	1
F.O.B. costs ($US)	166,600	323,077	281,615	495,430	65,385
Transportation Costs ($US)	16,408	32,627	28,369	37,931	8,030
Sub total ($US)	183,008	355,704	309,984	533,361	73,415
Total ($US)	1,455,472				

2) Development cost

Excavation unit costs were estimated respectively as follows for drifting and raise cutting based on records in Catavi Mine in 1981.

Drifting :	L650 Main haulage drift development	US$ 217.04 /m
	Excavated volume per meter	7.91 m^3/m
	Unit cost per m^3 of excavation	US$ 23.4 /m
Raise Cutting :	Exploration raise cutting	US$ 117.44 /m
	Unit cost per m^3 of excavation	US$ 52.2 /m^3

Table II–1–18 Development Cost

Kinds		Excavation area (m^3/m)	Length (m)	Unit cost ($US/$m^3$)	Sum ($US)
Main haulage drift (L650)		14.57	230	341	78,430
Service shaft		5.0	478	261	124,758
Ore pass		9.0	495	470	232,650
Access		10.5	1,420	246	349,320
Scram drift		9.0	550	211	116,050
Blasthole drift		12.25	7,840	287	2,250,080
Slot raise		4.0	402.5	209	84,122.5
Draw cone	Cross cut	12.0	250	281	70,250
	Raise	4.0	900	209	188,100
Ventilation shaft		4.0	87.5	209	18,287.5
Total					3,512,048
Minable ore (ton)		2,780,540			
Development cost per ton ($US)		1.26			

3) Operating Cost

The operating cost is as shown in Table II–1–19.

Table II–1–19 Operating Cost

Items		Factor	Unit cost	Costs 0–7 years	($US/t) 8–10years
Labor cost		39.3 t/PMS	20.0 $US/PMS	0.509	0.509
Material costs	Explosive	0.118 kg/t	0.64 $US/kg	0.076	0.076
	Electric fuse	0.036 piece/t	1.23 $US/piece	0.044	0.044
	Drill bit	0.0013 piece/t	116.5 $US/piece	0.151	0.151
	Drill rod	0.0007 piece/t	145.4 $US/piece	0.102	0.102
	Timber	0.001 m^3/t	46.7 $US/$m^3$	0.047	0.047
	Iron material	0.1 kg/t	1.75 $US/kg	0.175	0.175
	Cement	0.58 kg/t	0.16 $US/kg	0.093	0.093
	Others			0.100	0.100
	Sub total			0.788	0.788
Maintenance cost	Drifts and shafts			0.250	0.160
	Equipment			0.470	0.280
	Sub total			0.720	0.440
Haulage costs				0.944	0.944
Machine shop cost				0.124	0.124
Indirect service cost				0.401	0.153
Administration cost				0.760	0.380
Total				4.246	3.338

1–3 Mining Plan for Desmonte

1–3–1 General Layout

Desmonte is the float of sink-and-float separation produced in Siglo XX Mill, which has been accumulated since 1947 over a long period. The ore reserve of Desmonte is about 22 million tons, and its average grade is about 0.27%, but there is a tendency that the older waste, which is nearer to the bottom of the heap, has a higher grade, while the newer on, which is near the surface has a lower grade. In addition, the grade of waste nearer to the Mill is higher and gets lower the farther it gets from the Mill.

Because of the landscape of the area, the float was not heaped up on a flat place, but has been accumulated, while crossing the small ridge groups of mountains, on gentle slopes, its foot spreading radially, over an area as wide as about 520,000 m^2, and the height of accumulation is also supposed to have seriously expanded or contracted.

Desmonte has comparatively equal grain sizes of 30~40 mm and rounded shapes, so that it can be regarded as product easily loaded and transported. The state of its accumulation is comparatively loose and not so tightly tamped, it can therefore be easily crumbled by poking, but can support the traffic of heavy machines.

Taking these conditions into account, the mining plan for Desmonte this time has been prepared. From the viewpoint of balance with the production of underground mining ores, the quantity of production per day under the plan will be 6,500 t/day druing the first seven years and will be 8,000 t/day during three remaining years.

1–3–2 Mining Plan

The mining of Desmonte is a kind of large scale open-pit mining, requiring no breaking by blasting, and is regarded similar to the gravel cutting of river beds or the barrowing pit of sand. The important point is how cheaply the Desmonte is loaded and transported. As the result of comparing various kinds of mining methods, a method combining excavation equipment and haulage equipment was judged to be most advantageous.

When mining, the method of slicing down from the top or cutting from the toes of slopes and mining in vertical sections may be used, but in consideration of the topographic conditions that ridges branch variously and the height of accumulation expands or contracts greatly, we thought it better to adopt the former method. As excavation equipment, common large and heavy machines will be used, and about haulage systems, the three following kinds of systems were compared.

1 Dump Truck System

2 Load and Carry System

3 Belt Conveyer System

As mentioned above, a slice down system is adopted in this mining plan. If the belt conveyer system 3 is adopted, the belt conveyer must be moved every time a slice down is carried out. If the belt conveyer is installed in the central part, slicing down becomes impossible. For such reasons, belt conveyers must be installed around the accumulated area, and, if portable belt conveyers are extended along valleys, the extension of conveyers will become very long and moreover, mining procedures will become complicated. For such reasons, this system is thought to be unsuitable for mining large quantities of ores.

Accordingly, the economical efficiency of the dump truck system and that of the load and carry system were investigated by comparing them as a haulage system for Desmonte.

1-3-3 Haulage Plan

To determine the haulage plan, loading operation, transporting operation and the number of required machines were investigated for the dump truck and load and carry systems.

(1) Dump Truck System

When dump trucks are used as haulage equipment, power shovels tractor shovels etc. are considered as excavation and loading equipment and each has its merits and demerits. From the easiness of excavation and operation efficiency, the power shovel was thought superior, so that a combination of power shovels and dump trucks was studied.

1) Loading

[Prerequisites]

One loading cycle	28 sec (90° swivelling)
Actual loading time per day	15 hr. (5 hr/shift x 3 shifts)

Bucket capacity of power shovel when ore production per day is 6,500 ton/day :

$6{,}500$ t/d x 28 sec $\div$ (15 hr x 3,600 x f x E x k) $\div$ ρ = 3.3 m^3

f : earth quantity conversion coefficient = 1.0

E : operation efficiency = 0.7

k : bucket coefficient = 0.9

ρ : unit volume weight = 1.63 t/m^3

Bucket capacity of power shovel when ore production is 8,000 t/day :

8,000 t/d x 28 sec $\div$ (15 hr x 3,600 x 1.0 x 0.7 x 0.9) $\div$ 1.63 = 4.0 m^3

From the above, two power shovels of a bucket volume closs 2.0 m^3 are required, and as auxiliary equipment, two bulldozers (21-ton class) will be arranged.

2) Handling

[Prerequisites]

Loading capacity of dump truck used 11.0 m^3 (18 ton dump)

Haulage distance : 700 m (to the grizzly of the plant)

One cycle time of dump truck :

$$Cmt = \frac{C}{Q \times K} \times \frac{Cms}{60 \times Es} + \left(\frac{D}{V_1} + t_1 + \frac{D}{V_2} + t_2\right) = 14.5 \text{ min}$$

Cms : one cycle time of power shovel = 28 sec

Es : operation efficiency of shovel = 0.7

D : haulage distance = 700 m

V_1, V_2 : average speed of truck = 10 km/hr = 167 m/min

t_1, t_2 : waiting time of truck for loading and unloading = 1 min

C : loading capacity = 11.0 m^3

Q : bucket capacity of power shovel = 2.0 m^3

K : bucket coefficient of power shovel = 0.9

Actual haulage operation time of truck : 18.0 hr (6 hr/shift x 3 shifts)

Haulage capacity per truck :

$$\frac{60}{14.5} \times 11.0 \text{ m}^3 \times 1.0 \times 0.9 \times 18 \text{ hr} = 737.4 \text{ m}^3/\text{day}$$

Number of trucks required for daily ore production, 6,500 t/day :

6,500 t/day ÷ 1.63 t/m^3 ÷ 737.4 m^3/day = 5.4 ÷ 6

Number of trucks required for daily ore production, 8,000 t/day :

8,000 t/day ÷ 1.63 t/m^3 ÷ 737.4 m^3/day = 6.7 ÷ 7

From the above, the required number of machines in the case of the dump truck system is shown in Table II–1–20. As reserve machines, about a 50% of the required number has been added.

Table II–1–20 Equipment Required by Dump Truck System

Ore production tons per day	Power Shovel 2.0 m^3	Dump truck 18 ton	Bulldozer 21 ton
6,500	3	9	2
8,000	3	12	2

(2) Load and Carry System

In the load and carry system, a selecting condition was set that excavation, loading and carrying can be performed by one machine, and a motor scraper was selected.

This machine can travel steadily even on an uneven ground and is thought to be a machine most suitable for mining in a vast plane place.

When excavating with this machine, if the excavation is especially difficult, a pusher (bulldozer) will be required, but it is necessary to decide whether the pusher is required or not by actually carrying out excavation tests on the spot. In the plan of this time, we judged that it is possible to excavate with the motor scraper and investigated the plan without using a pusher.

Although there are various capacities of motor scrapers, we selected that of a capacity of 11.0 m^3 class, considering economical efficiency, operation efficiency, etc.

The efficiency and the required number of motor scrapers are computed as follows.

[Prerequisites]

Capacity of motor scraper : 11.0 m^3

Haulage distance : 700 m

One cycle time of motor scraper :

$$Cm = \frac{H}{vh} + \frac{R}{vr} + td + ts + tg = 6.85 \text{ min}$$

H : haulage distance on going = 700 m

R : haulage distance on returning (including turnabout) = 800 m

vh : haulage speed on going = 16 km/hr (= 267 m/min)

vr : haulage speed on returning = 22 km/hr (= 367 m/min)

td : loading time = $\frac{D}{v_1}$ = 0.75 min

v_1 : loading speed = 40 m/min

D : loading distance = 30 m

ts : unloading time = $\frac{S}{v_2}$ = 0.3 min

v_2 : unloading speed = 100 m/min

S : unloading distance = 30 m

tg : time required for turnabout and gear change = 1.0 min

Actual operating time of motor scraper : 18 hr (6 hr/shift x 3 shifts)

Haulage capacity per motor scraper :

$$\frac{60}{6.85} \times 11.0 \text{ m}^3 \times E \times 18 \text{ hr} = 1{,}300.7 \text{ m}^3/\text{day}$$

E : operation efficiency of motor scraper = 0.75

Number of motor scrapers required for daily ore production of 6,500 t :

6,500 t/d ÷ 1.63 t/m^3 ÷ 1,300.7 m^3/d = 3.0 ≑ 3

Number of motor scrapers required for daily ore production of 8,000 t :

8,000 t/d ÷ 1.63 t/m^3 ÷ 1,300.7 m^3/d = 3.8 ≑ 4

In addition, one bulldozer (21-ton class) will be arranged as auxiliary equipment.

From the above, the required number of machines in the case of the load and carry system is as shown in Table II–1–21. As reserve machines, about a 50% of the required number has been added.

Table II–1–21 Equipment Required by Load and Carry System

Ore production tons per day	Motor scrapper 11.0 m^3	Bulldozer 21 ton
6,500	5	1
8,000	6	1

1–3–4 Mining Costs

Mining costs in the case of the dump truck system were compared with those of the load and carry system after being divided into equipment costs and operating cost in the computation.

Prerequisites in the cost computation are as follows. The equipment costs are shown in Table II–1–22, and operating costs in Table II–1–23.

[Prerequisites]

1 The mining period shall be 10 years, ore production shall be 6,500 t/day until the seventh year, and that from the eighth year on shall be 8,000 t/day.

2 Unit man-hour per unit work shall conform to the standard value of the Ministry of Construction, and efficiency drop in highlands has not been taken into account.

3 The actual operation time of heavy machines per day shall be 18 hours (3-shift system), and the machines shall be fully depreciated in 10 years.

4 The number of working days per year shall be 300 days.

5 As maintenance costs, 85% of the cost of each machine was summed up.

6 Equipment costs and haulage costs do not include interest.

Table II-1-22 Open Pit Mining Equipment

Kinds	Dump truck system			Load & carry system	
	Power Shovel	Dump truck	Bulldozer	Motor scrapper	Bulldozer
Numbers	3	12	2	6	1
F.O.B. costs ($US)	637,500	932,400	204,000	1,140,000	102,000
Transportation costs ($US)	88,771	121,290	27,499	133,365	13,750
Total ($US)	2,011,460			1,389,115	
Total operating days	3,000			3,000	
Equipment cost per day	670.49 $US			463.04 $US	
Average ore production per day	6,950 t/Day			6,950 t/Day	
Equipment costs per ton	0.097 $US/t			0.067 $US/t	

Table II-1-23 Operating Costs

(Unit : $US)

Years	Kinds	Unit	Dump truck system				Load & carry system			
			Figure	Unit cost	Sum	Costs per ton	Figure	Unit cost	Sum	Costs per ton
0 ∫ 7	Gasolines	ℓ	1,839	0.2535	466.19	($US/t)	3,045	0.2535	771.91	($US/t)
	Oils and fats		1		93.24		1		154.38	
	Labors		33.3	8.0	266.40		20.4	8.0	163.20	
	Tyre consumption	number	6	48.7	292.20		3	90.7	272.10	
	Maintenance		1		436.56		1		298.07	
	Total		6,500 t/Day		1,554.59	0.239	6,500 t/Day		1,659.66	0.255
8 ∫ 10	Gasolines	ℓ	2,128	0.2535	539.45		3,945	0.2535	1,000.06	
	Oils and fats		1		107.89		1		200.01	
	Labors		39.3	8.0	314.40		25.5	8.0	204.00	
	Tyre consumption	number	8	48.7	389.60		4	90.7	362.80	
	Maintenance		1		502.61		1		351.90	
	Total		8,000 t/Day		1,853.95	0.232	8,000 t/Day		2,118.77	0.265
Average operating cost of 10 years			0.237							0.258

According to this results, the mining costs by the load and carry system are a little lower, but this system includes an uncertain element that it cannot be judged if this system fits the conditions of the spot without carrying out excavation tests there, so the current mining plan was made based on the dump truck system which is thought more reliable.

As a result, the production efficiency of Desmonte was estimated to be about 200 t/manpower and the total production efficiency, 130 t/man-day.

Table II–1–24 Comparison with Mining Costs

Kinds	Dump truck system	Load & carry system
Equipment cost ($US/t)	0.097	0.067
Operating cost ($US/t)	0.277	0.283
Mining cost ($US/t)	0.374	0.350

1–4 Mining Equipment

1–4–1 Measures for Mining Equipment from the Viewpoint of Machines Used

(1) Underground

When introducing "sublevel stoping", rapid drifting must be realized to adopt large long-hole drilling machines, to increase ore handling, etc., for improvement in development efficiency, and for such reasons, increase in air compression equipment capacity and air compression pressure is required.

The number of main machines which will be used in the current plan and air consumption by them are as shown in Table II–1–25.

Table II–1–25 Air Consumption

Kinds	Ring drill crawler	Hopper loader	Rocker shovel	Leg drill	Stoper
Type	CJ–641	ME–803	RS–150	BBD–90–W	RB–83
Air consumption	13.5 m^3/min	13.0 m^3/min	Max. 20 m^3/min	5 m^3/min	4 m^3/min
Number	4	5	1	20	10
Sub total	54 m^3/min	65 m^3/min	20 m^3/min	100 m^3/min	40 m^3/min
Total	279 m^3/min				

From this result, air consumption is 279 m^3/min, and the capacity of compressed air required becomes, by assuming their operating efficiency to be 70%, about 400 m^3/min.

As air compression equipment, we selected oil free air compressors (VGP 120 Turbo Type). The specification of the compressor are as follows.

Item	Data
Type	VGP–120 Turbo Type
Capacity	103.3 m^3/min
Intake Condition	0.637 kg/cm^2 A
Discharge Condition	9.0 kg/cm^2 G
Motor Output	520 KW (2P)
Cooling Water	56 m^3/hr (temperature difference 8°C)

From the above data, the required number of compressors becomes 4. For other equipment, the existing equipment will be used without substantial change.

(2) Open-Pit

As a large number of large heavy machines will be used in the mining of Desmonte, maintenance of the heavy machines and storage of fuel are required, so a maintenance machine shop and fuel tanks will have to be built. In consideration of the number of heavy machines used, the maintenance machine shop requires a scale of the extent that one power shovel (bulldozer) and three dump trucks can always be serviced.

About fuel tanks, the quantity of gas oil used in a day is about 2 kℓ, and for securing storage of a quantity required for 2~3 days, a tank with a storage capacity of about 6 Kℓ is required.

1-4-2 Mining Equipment Cost

Main equipment investment for mining equipment is that for air compressors, the maintenance machine shop and the fuel tank, but the maintenance machine shop and the fuel tank are planned as equipment common with the engineering work section, so that the equipment costs of only compressors are computed here.

The equipment costs do not include interest.

Table II-1-26 Compressor Equipment

Kinds	Value
Numbers	4
F.O.B. costs ($US)	738,461
Transportation costs ($US)	65,670
Installation cost ($US)	24,123
Total ($US)	828,254

1–5 Plan of Long Term Production

In the current long term production plan under the idea of balancing underground mining ores and open-pit mining ores, it has been planned to carry out selective mining by adopting the sublevel stoping method for underground mining ores and to mine also the low-grade open-pit ores, and, by making a production plan, a personnel plan and a construction plan and computing mining operation costs for ten years from now, it was investigated what extent of mining costs and productivity can be attained.

1–5–1 Production Plan

The production plan in ten years consisting of the sublevel stoping plan and the mining plan of Desmonte can be summarized as shown in Table II–1–27.

Table II–1–27 Production Planning of 10 Years

Years		Sublevel stoping			Desmonte			Total		
		Min. ton (t)	Sn% (%)	Fino ton (TMF)	Min. ton (t)	Sn% (%)	Fino ton (TMF)	Min. ton (t)	Sn% (%)	Fino Ton (TMF)
1~7	Monthly production	87,500	0.41	361.25	162,500	0.27	438.75	250,000	0.32	800.00
	Total	7,350,000	0.41	30,345.00	13,650,000	0.27	36,855.00	21,000,000	0.32	67,200.00
8~10	Monthly production	50,000	0.22	110.00	200,000	0.27	540.00	250,000	0.26	650.00
	Total	1,800,000	0.22	3,960.00	7,200,000	0.27	19,440.00	9,000,000	0.26	23,400.00
Total		9,150,000	0.37	34,305.00	20,850,000	0.27	56,295.00	30,000,000	0.30	90,600.00

1–5–2 Personnel Plan

The personnel plan for 10 years which should be computed appropriately in accordance with the quantities of ore production by the sublevel stoping plan and the mining plan of Desmonte can be summarized as shown in Table II–1–28.

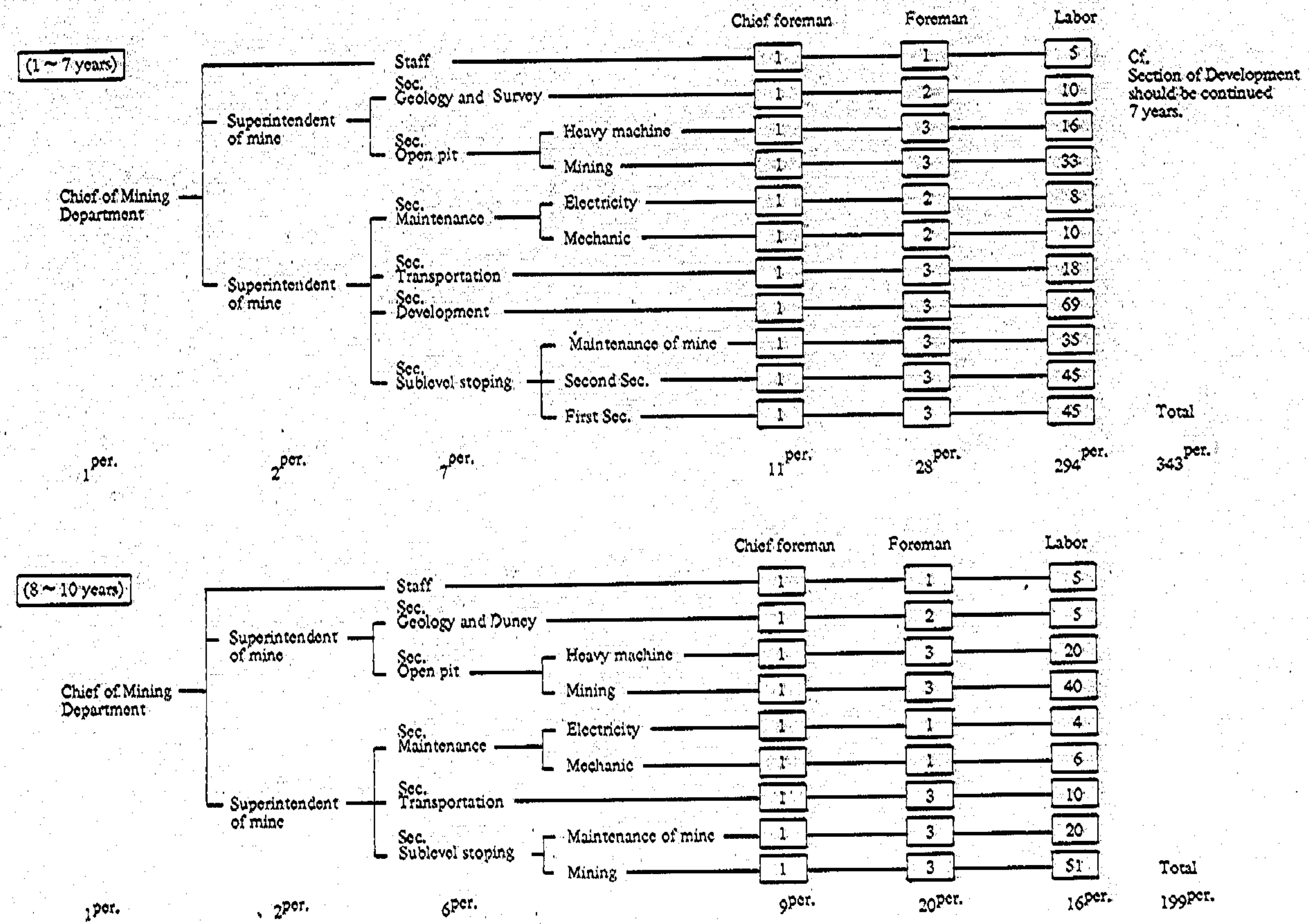


Fig. II–1-19 Organization of Mining Department

Table II–1–28 Personnel Planning of 10 Years

Kinds	Occupation		Unit per man shift	Manpower required	
				1~3 years	4~10 years
Sublevel stoping	Miners	Extraction	39.3 t/MS	90	51
		Development	0.134 m/MS	69	-
	Technicians		120 t/MS	35	20
	Repairmen		200 t/MS	18	10
	Haulage men		300 t/MS	18	10
	Sub total			230	91
Desmonte	Operators		200 t/MS	33	40
	Repairmen		400 t/MS	16	20
	Sub total			49	60
Others	Staff			65	10
Total				344	161

By studying the organization of mining department based on this personnel planning, the forms shown in Fig. II–1–19 can be organized.

1–5–3 Construction Planning

The construction in the 10-year long term plan includes the development work before beginning the sublevel stoping and equipment supply. The construction schedule and development costs are summarized as follows, the former being shown in Table II–1–29, and the latter in Table II–1–30. Mining equipment investment is shown in Table II–1–31.

Table II–1–29 Development Schedule

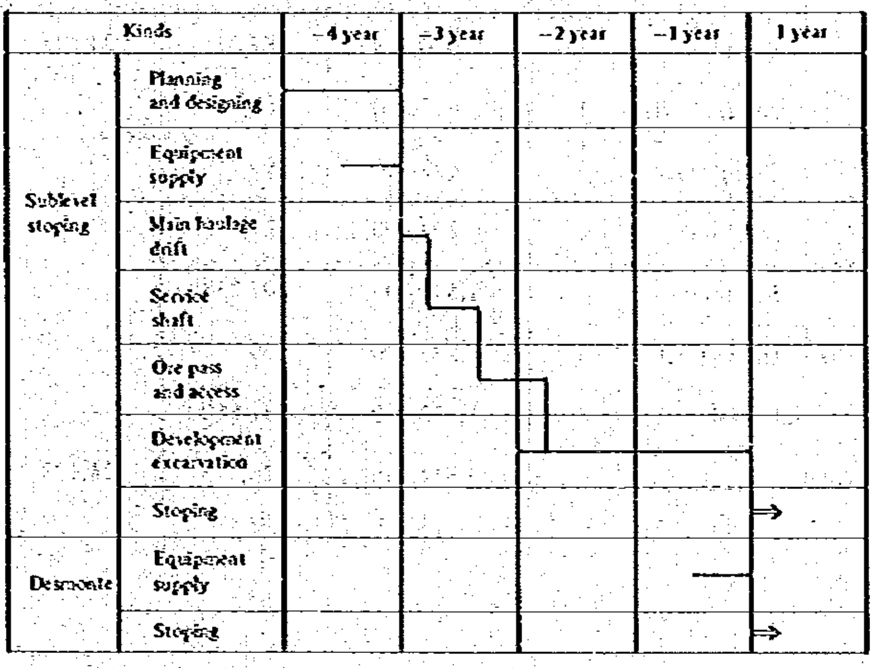


Table II–1–30 Development Costs with Sublevel Stoping

Development year	Stoping year	Yearly production (t)	Unit cost ($US/t)	Yearly costs ($US)	Development cost ($US)
–4 ~ –1	1 ~ 3	1,050,000	1.26	1,323,000	3,969,000
1 ~ 4	4 ~ 7	1,050,000	1.26	1,323,000	5,292,000
5 ~ 7	8 ~ 10	600,000	1.26	756,000	2,268,000
Total					11,529,000

Table II–1–31 Mining Equipment Costs

Kinds		Equipment	Number	F.O.B. costs ($US)	Transportation costs ($US)	Installation costs ($US)	Total cost ($US)
Sublevel Stoping		Scraper	4	166,600	16,408	–	183,008
		Ring drill crawler	6	323,077	32,627	–	355,704
		Hopper loader	7	281,615	28,369	–	309,984
		Alimak climber	2	495,430	37,931	–	533,361
		Rocker shovel	1	65,385	8,030	–	73,415
		Compressor	4	738,461	65,670	24,123	823,254
		Sub total		2,070,568	189,035	24,123	2,283,726
Desmonte	1 ∫ 7 Year	Power shovel	3	637,500	88,771	–	726,271
		Dump truck	9	699,300	90,968	–	790,268
		Bulldozer	2	204,000	27,499	–	231,499
		Sub total		1,540,800	207,238	–	1,748,038
	8~10	Dump truck	3	233,100	30,322	–	263,422
	Sub total			1,773,900	237,560	–	2,011,460
Total				3,844,468	426,595	24,123	4,295,186

I-5-4 Operating Costs and Productivity

The operating costs in ten years required for the sublevel stoping plan and the mining plan for Desmonte are shown in Table II-I-32. The productivity is shown in Table II-I-33.

Table II-I-32 Operating Costs of 10 Years

Kinds		1 ~ 7 years		8 ~ 10 years		Total
		Yearly	Sub total	Yearly	Sub total	
Sublevel stoping	Ore production (t)	1,050,000	7,350,000	600,000	1,800,000	9,150,000
	Operating cost ($US/t)	4,246	4,246	3,338	3,338	4,067
	Sum ($US)	4,458,300	31,208,100	2,002,800	6,008,400	37,216,500
Desmonte	Ore production (t)	1,950,000	13,650,000	2,400,000	7,200,000	20,850,000
	Operating cost ($US/t)	0.280	0.280	0.271	0.271	0.277
	Sum ($US)	466,050	3,262,350	556,800	1,670,400	4,932,750
Total	Ore production (t)	3,000,000	21,000,000	3,000,000	9,000,000	30,000,000
	Operating Cost ($US/t)	1.668	1.668	0.884	0.884	1.433
	Sum ($US)	5,004,300	35,030,100	2,653,200	7,959,600	42,989,700

Table II-I-33 Productivity of 10 Years

Kinds		1 ~ 7 years	8 ~ 10 years
Sublevel stoping	Monthly production	87,500	50,000
	Manpower	230	91
	Monthly productivity	380.4	549.5
Desmonte	Monthly production	162,500	200,000
	Manpower	49	60
	Monthly productivity	3,316.3	3,333.3
Total	Monthly production	250,000	250,000
	Manpower	294	161
	Monthly productivity	850.3	1,552.8

The above investigation results show that the average mining cost in 10 years will become US$ 1.433 ton, a comparatively low value, and the productivity will rise greatly to about 850 ton/man-month.

In conclusion, great improvement can be realized by putting this project into practice.

CHAPTER 2. PLAN FOR THE BENEFICIATION DEPARTMENT

2–1 Background of Planning for New Concentration Mill

2–1–1 Trends of Concentration Results at Catavi Mine

Fig. II–2–1 shows the trends of concentration results since the start of Catavi Mine. At first, in 1923, the grade of tin crude ore was as rich as several percent, and, although this grade dropped to the 2% level over the following 20 years, in Victoria Mill, where gravity concentration using jigs and tables is carried out, such good results as a grade of concentrate of more than 60% and a recovery of more than 80% have been maintained.

After that, corresponding to the drop in the grade of tin crude ore to a level of 1%, pre-concentration by means of sink-and-float separation was added to Siglo XX in 1947. Recovery in the pre-concentration stage was above 90% at first, but in the latter half of the 1960's, when the grade of tin ore became lower than 0.6%, recovery sometimes dropped below 70%, and for that reason, the addition of various items of equipment and changes in the systems seem to have been repeated.

Interesting characteristics can be seen in the fact that the recovery at Siglo XX shows three periodic changes, each covering five years (1 , 2 and 3 in Fig. II–2–1), and during the low recovery period of Siglo XX, the recovery at Victoria was on a high level. As a result, total recovery has been, since the 1960's, on a level a little lower than 50%; i.e., pre-concentration 70% x main concentration 70% = 49%, or pre-concentration 75% x main concentration 65% = 49%.

On the other hand, the grade of tailings at the commencement of the mine, when the grade of tin crude ore was high, was as high as more than 1%, but dropped to the 0.5% level after 20 years. From the time when pre-concentration was started, the grade of float in the sink-and-float separation process progressed to around 0.3%, and the grade of tailings after the main concentration process by means of gravity concentration has made progress to slightly higher levels of 0.35~0.4%. In total, tailings of grades above 0.3% had been accumulated outside the mine in vast quantities as so-called Desmonte over a period of 40 years, with the exception of the last 10 years. When these tailings are compared with the proven ore reserves and their grade at present, the quantity of exploitable ores be greatly increased if an appropriate concentration system for the tailings is provided.

2–1–2 Recent Results of Concentration Operations

As shown in Fig. II–2–2, the results of operations have dropped greatly these past several years, while on the other hand, operating costs have increased greatly. The grade of crude ore dropped from 0.46% to 0.31%, a decline of about 30%, in five years, and the grade of concentrate also dropped from 50% to 38% almost linearly.

The total recovery of tin (pre-concentration x main-concentration), which was about 50% in 1976 and 1977, has dropped every year after that, and marked a particularly serious drop in the past year, from 45.1% in 1980 to 40.8% in 1981. The main cause of this was a rapid drop in the amount of pre-concentration recovery over the past one year, from a level of 75% to 70% or less.

The operating cost per ton of tin crude ore was 4.5 US$/ton in 1976 and 1977, but has

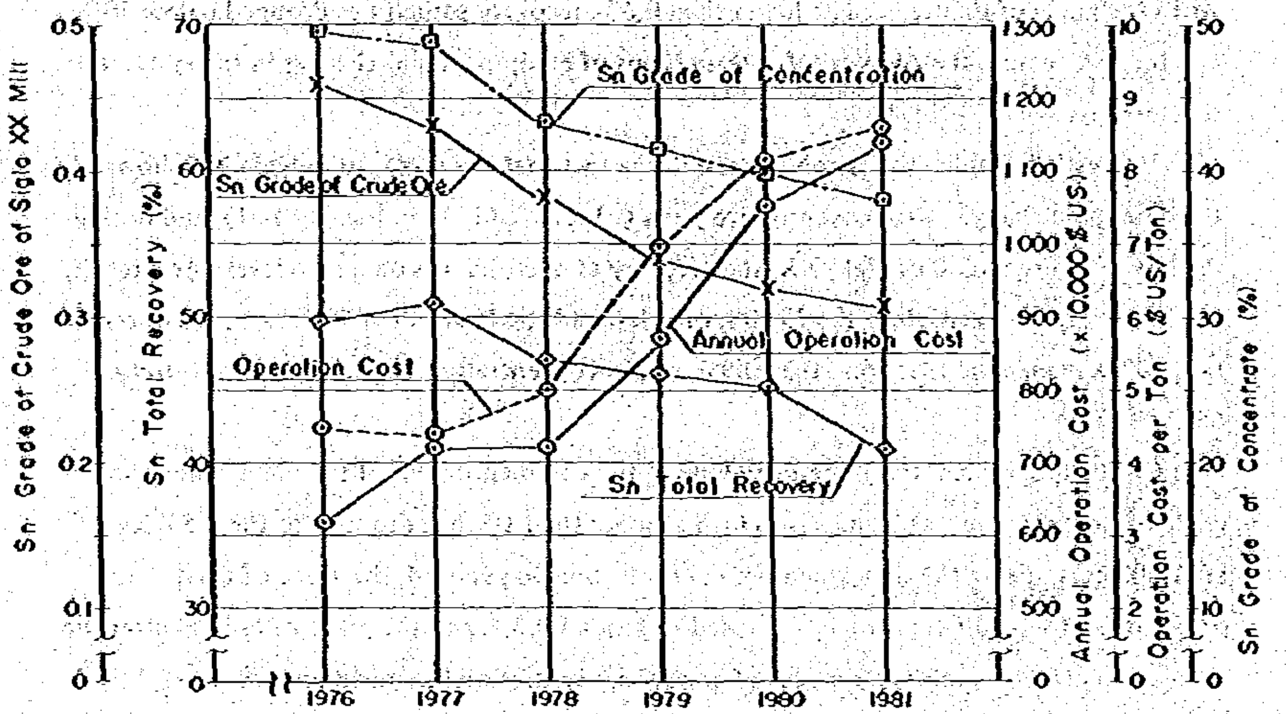


Fig. II-2-2 Result of Recent Concentration Work

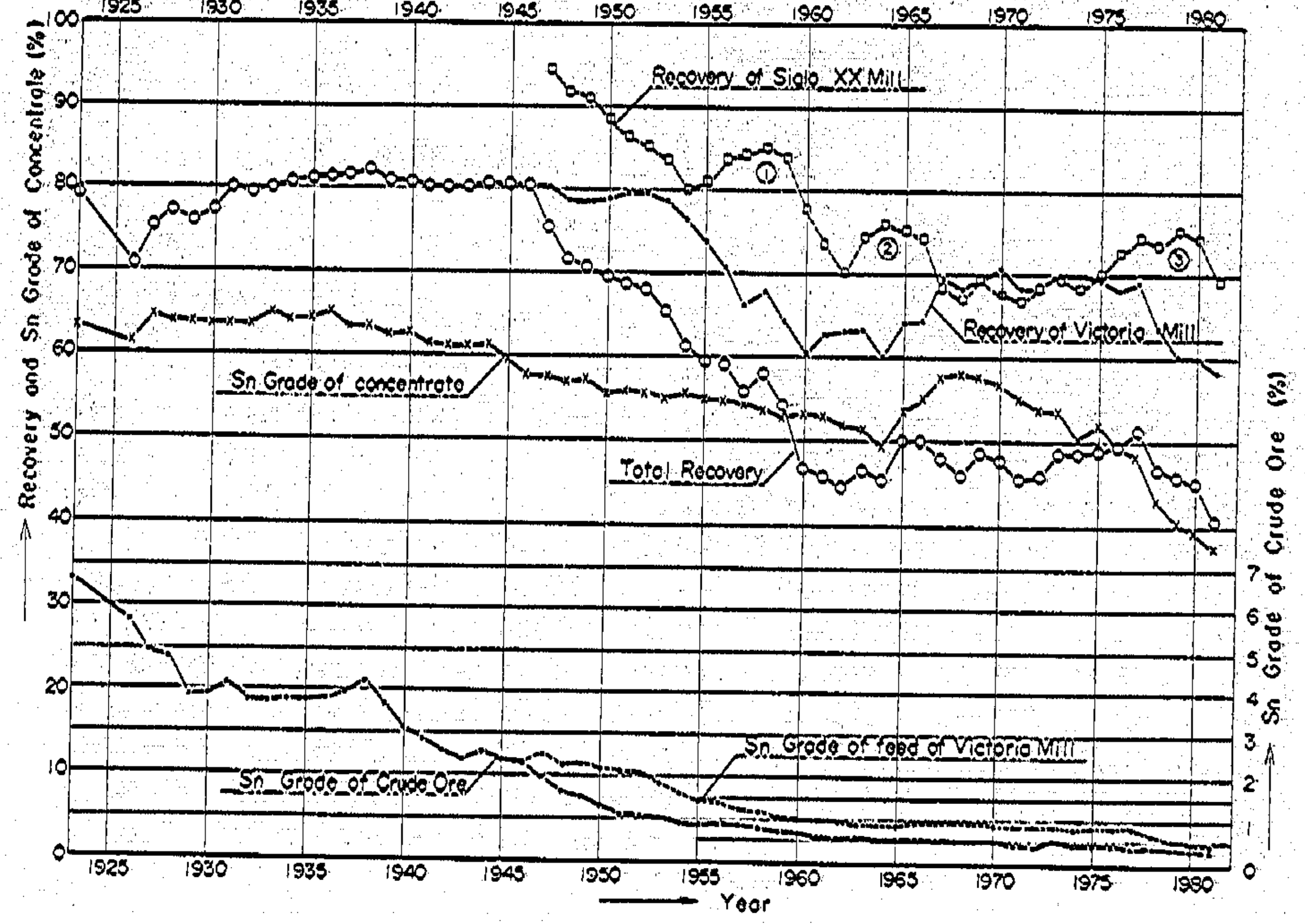


Fig. II-2-1 Transition of Concentration Results of Catavi Mine

rapidly increased after that, exceeding 8 US$/ton in 1980 and reaching 8.6 US$/ton in 1981, nearly twice the previous value.

2–1–3 Current Problems

Technical and economic problems in concentration operations, combined with the results of investigation in the first year, can be summarized as follows:

(1) Drop in Concentration Results

As mentioned in the two preceding sections, the quantity of tin production in Catavi Mine has dropped at present to the lowest level ever in the long history of the mine on account of the drop in concentration results, and this drop has come to be the largest factor in the increasing deficits.

According to the results of surveys to date, each of the problems described in the following is related directly or indirectly with the drop in concentration results, but, in short, it is most important to establish a concentration system suitable for the low-grade ores.

It can of course be supposed that processes and equipment have often been improved in the past from the trends in the concentration results. For example, it was planned to raise the grade of concentrate by adding the fine particle table process and the flotation process. However, there is a limit to the improvement of results obtainable by a partial improvement of the process. Such kinds of improvements will rather make the process more complicated, and when the quantity or quality of crude ore changes, the process must be reviewed each time, so that the control of operation becomes difficult.

(2) Problems of the Concentration System and its Capacity

Tin grades of crude ore and products in Siglo XX since the opening of the mill are shown in Fig. II–2–3. At present, compared with the grade of about 0.3% for crude ore, the grade of sink is only 0.45% and in addition, its recovery is also low as mentioned above. In terms of quantity, about a half of the crude ore, i.e., about a third of the tin content in crude ore, is disposed of as Desmonte in the pre-concentration stage.

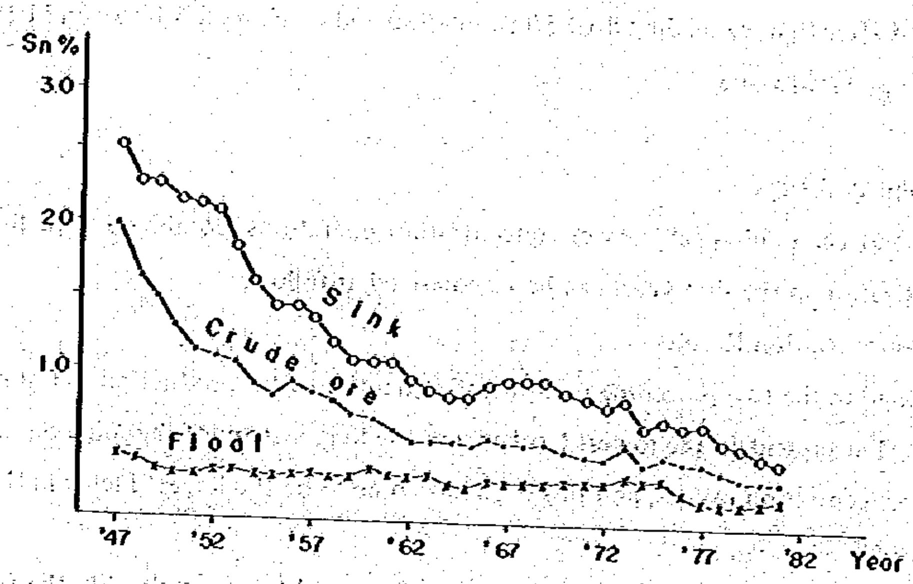


Fig. II-2-3 Transition of Sn Grade of Siglo XX Mill Plant

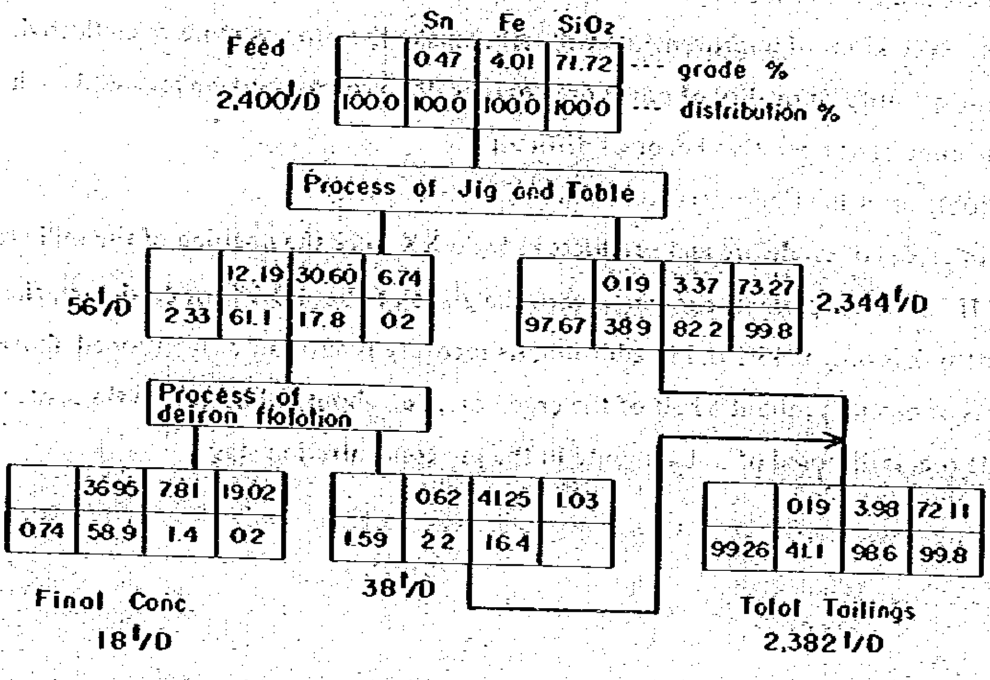


Fig. II-2-4 Result of Metal Balance (at 1981) pre-concentration stage.

This fact shows that the application of the sink-and-float concentration process itself to cassiterite distributed in the field of fine grain has become a bottleneck in improving the results of concentration as described in 2–2–4.

The quantities of ores and metal balance in Victoria Mill are outlined in Fig. II–2–4, and the quantities disposed of as Colas Arenas (final tailings) are also very large.

To improve these results, it is desirable to check each type of equipment for mineral dressing in detail, from the viewpoints of the quantity and capacities of the machines and the material balance. Fundamentally, however, these results are thought to arise from the fact that low-grade ores are at present treated with equipment used in the days of high-grade ores with their contents and capacities unchanged, and also from the fact that although the size of cassiterite is considered to be 50μ maximum, the particle size of the tailings is comparatively coarse. For example, the number of tables is 0.75 t/h table, which is thought to be too few for such low-grade crude ores as those below 0.5% obtained at present. In other words, the processing capacity of Victoria must be reduced by 2,500 t/day in the planning of operations.

(3) Instability of Operation

The facilities of the Siglo XX and Victoria Mills include considerably deteriorated equipment, and many defects and considerable damage are noticeable in driving devices, feed chutes, feed troughs, piping, valves, etc. In addition, as a preventive maintenance system for the equipment has not been employed, unexpected troubles occur and greatly lower the operating rate of the concentration mills.

The details of suspensions of operation in the two Mills in 1981 are as shown in Table II–2–1. The rate of operation is only about 70%, and more than 30% of the suspension time is accounted for equipment troubles.

Table II–2–1 Result of Suspension of Operation

(at 1981)

	Siglo XX	Victoria
Time of Operation	5,788 hrs	6,717 hrs
Suspension Time	2,972 hrs	2,043 hrs
Efficiency of Operation	66.1 %	76.7 %
Shortage of Crude Ore	6.9 %	3.8 %
Shortage of Electric Power	2.4	4.1
Shortage of Water	1.1	0.8
Maintenance	32.8	44.1
Mixing of Block	16.5	–
Labor Affairs	14.4	23.0
Shortage of Worker	15.6	14.4
Others	10.3	9.8
Total	100.0 %	100.0 %

Furthermore, there is the problem of labor management comparable with equipment trouble as a reason for suspension, and in Siglo XX, the inclusion of large blocks at the time of receiving ores is also a large factor.

In addition, it must be pointed out that there are too many factors which make operation unsteady, e.g., operators do not have sufficient knowledge of the function of the head motion of the shaking tables, which is the very principle of gravity concentration; or of the maintenance of table surfaces; or, although many kinds of data related with the computer system are stored, the data are not used efficiently in operational control, etc.

It is not a part of the purpose of this report to analyze the contents of the above-mentioned problems in labor management. However, against the background of the occurrence of so many suspensions for this reason, in spite of the existence of a large number of employees, this problem is inferred as follows:

"There is an uneasiness that unexpected equipment troubles may occur at any time, and once a problem in the main equipment occurs, the equipment must be stopped for a long period because of a lack of spare parts storage facilities, or else the equipment must be carried to a repair shop. Moreover, when the equipment is repaired and operation is to be started, the arrangements for all the required personnel have not yet been simultaneously completed."

It must also be pointed out that there is no constant feed weigher fitted to the rod mills, which is the most important device for suppressing fluctuations in the feed to make the grinding operation stable. Furthermore, feed the tables is often disturbed by unstable hydraulic pressure or unstable supply of water for hydraulic classifiers.

(4) Increase in Operating Costs

Fig. II–2–5 shows the total operating cost per ton of crude ore at Siglo XX and Victoria Mills. Although affected by the decrease of processed ore quantities, the figures show a rapid increase in cost from 1979, and it is understood that labor costs, material costs and auxiliary section costs have also increased. Above all the labor costs have increased very steeply.

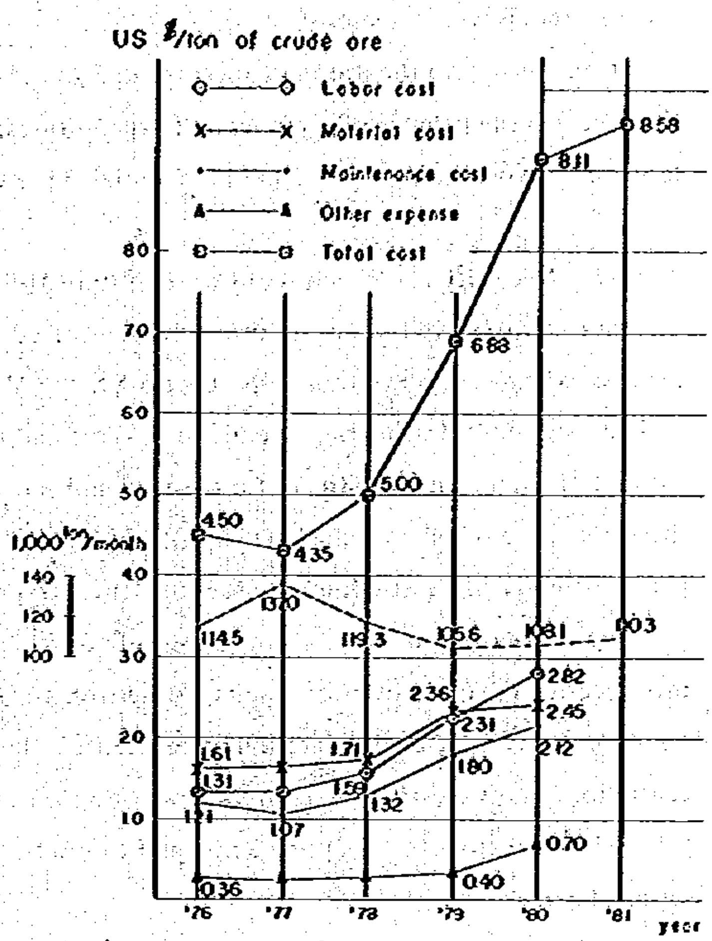


Fig. II-2-5 Transition of Operation Cost by elements in recent year

As a background to the labor cost increases, the fact that the level of automation in the concentration mills is very low is indicative of the large number of persons employed, and also the rapid increase of wages in these three years is the main factor influencing the unit labor cost. As an additional factor to the increase of material costs, ferrosilicon and sulfuric acid, which have been the main materials in the concentration system to date cannot be supplied sufficiently for the industrial level of this country and for the present conditions of Catavi Mine. Accordingly, there is another problem in that the prices of

Table II–2–2 Total Personal

(at 1981)

		Siglo XX	Victoria	Total
Salary Worker	Engineer	6	5	11
	Foreman	72	10	182
	Total	78	115	193
Labor		264	393	657
Total		342	508	850

Note: This Table contains personal of maintenance and 65 persons of Kenko is not included.

these materials will increase.

Combined with the drop in concentration results mentioned above, these adverse conditions worsen the balance of revenues and expenditures, so that it is fundamentally important to solve this cost increase problem in modernizing the Catavi Mine.

2–1–4 Necessity for Renewal of Concentration Mill

In order to solve the various problems considered above, it is not sufficient to remodel the two existing concentration mills at Siglo XX and Victoria, but there is no other way to build a new concentration mill as early as possible and renew the operations system. A large amount of equipment investment for the new mill is considered necessary. The reasons for these conclusions being reached are clarified by the following viewpoints:

(1) Possibility of Drastic Improvement in Concentration Results

Pre-concentration by means of sink-and-float separation is in fact one of the major factors lowering the concentration results. Moreover, gravity concentration with shaking tables does not suit the degree of liberation of cassiterite either. It may be possible to improve concentration results greatly if an entirely new concentration system can be built by removing these adverse factors and adopting a new and carefully-planned usage of tables, etc., as described later.

The means of removing the above-mentioned adverse factors are the abolition of the sink-and-float concentration equipment at Siglo XX and a large increase in the number of tables at Victoria. As a method for combining these two aspects into one measure, we have the following opinions:

(2) Possibility of Utilizing Victoria Mill

In order to process 5,000 t/day, the present volume of production, at least three times the existing number of tables is required. A space for increasing the number of tables can be obtained by the expansion of Victoria, but large-scale construction work similar to overall renewal will be required to also accommodate the ore crushing process and grizzly processes.

It will be better to build another plant at a position near the adits and the accumulation of Desmonte, and there will be no advantage in adhering to Victoria as a location.

If the entrance and the crushing equipment are left in Siglo XX and the processes from grinding onwards are redirected to Victoria, the amount of handling required between the two mills will remain forever as a large burden. Division between the two mills will not be advantageous if the existing equipment of the two mills cannot be used almost unchanged. However, both in Siglo XX and Victoria, the rate of operation of the existing equipment is low and there are many items of equipment which have deteriorated, and if they were to be repaired, most components of the old system would not provide a sufficient level of performance, so that it is not possible to improve the quality of the equipment in Bolivia.

In addition, to make the grain sizes of the products for crushing and grinding smaller so as to suit the liberation size of cassiterite, the capacities of the crushers and ball mills are insufficient. If the partial strengthening of these machines or an increase in their sizes were planned, it would be difficult to arrange plant layout appropriately, or if spaces were somehow able to be made to accommodate the machines, operational control would become very difficult.

(3) Possibility of Utilizing Siglo XX Mill

This mill is near adits and Desmonte, but as mentioned above, equipment capacity per machine is small and the quality of the equipment is also low, so that the mill must be almost entirely renewed rather than being diverted to other purposes near in its present condition.

Moreover, at a location this mill, there is a place suitable for constructing a mill with a capacity of up to 10,000 t/day, so that there is no need to remain with Siglo XX only.

(4) Possibility of Reducting Operating Cost

In order to improve the balance of revenues and expenditures without reducing the number of employees in Catavi as far as possible, it is first required to impove the concentration results drastically, and next to greatly increase the quantity of processing, i.e., to expand the scale of the concentration mill. Fortunately, a great improvement in the operation results may be expected, on the scale of double in quantitative terms, by utilizing Desmonte.

However, even if the scale is doubled, i.e., personnel cost is halved, deficits cannot yet be eliminated. In a practical sense, if the existence of Catavi is given priority, expenditures including personnel costs must be thorough reduced.

This investigation committee has carried out various fundamental tests on these subjects to determine the appropriate concentration system as will be described subsequently, and as a result, although in the stage of laboratory tests, a prospect for concentration results of over 50% for concentrate grade and of 55% for recovery rate has been obtained. Hence, the committee has decided to make a plan for a new concentration mill of a scale of up to 10,000 t/D. Furthermore, on the basis of eliminating deficits arising from various elements of cost, personnel costs have also been computed on the basis of the minimum required number of correctly assigned persons.

As this is a plan whose main part is table concentration simple note is appended about the number of tables.

If the area scales of the concentration mills are compared in relation to the number of tables in the table concentration process, there are about 150 tables in the existing Victoria mill, while in the newly-planned system nearly 1,100 tables will be provided about seven times the number in Victoria, a scale which is unparalleled in the world. When comparing the mills in relation to processed ore quantities, the figures are 2,500 t/D to 10,000 t/D, i.e., a little over four times. However, recently in Victoria, where the grade of crude ore has dropped, about 1,900 t/D of sink is regarded as its object, so that if this value is taken as the reference for comparison, the processed ore quantity ratio becomes a little over five to one.

If compared with other mine concentration mills which are obtaining good results, in Huanuni, 0.60 t/h/table (crude ore Sn grade 1.20%, concentrate Sn 50.0%, tailings Sn 0.39% recovery 68%), in Mikohata, 0.37 t/h/table (the above values are respectively 0.25%, 60.0%, 0.11%, 58%), and in Victoria, 0.60 t/h/table (2,040 t/D of crude ore containing Sn 0.47%; concentrate Sn 38%, recovery 59%).

Generally, to improve concentrate grade when crude ore grade drops, i.e., to increase concentration ratio, and yet to raise recovery, it is important to reduce the quantity of treatment per table simultaneously with keeping pulp concentration at a low level so that the movement of each particle may not be interferred with by other particles. In this plan, from the fact that the tin grade of crude ore is in a range below 0.3% and the grain size of a single cassiterite is around 50μ, the quantity of treatment per table in the above-mentioned new system was determined to be 0.38 t/h/table, a value close to that of Mikohata.

2–1–5 Outline of New Concentration Mill Plan

As mentioned above, it is required to greatly improve the balance between the income and expenditures of the concentration mill, reducing the percentage of personnel costs in the operating cost of the mine at least to less than a half of the present value, and, with regard to the concentration results, to renew the concentration system entirely to one suitable for processing low-grade crude ores, in order to reconstruct Catavi Mine. Based on the results of two surveys about the state of the Mine and laboratory tests in Japan for several months over the past two years, this investigation committee has planned the framework of a new concentration mill, which will be able to meet these requirements and solve various problems, as follows:

(1) A concentration mill with a processing capacity of 10,000 t/D, at the ratio of underground-mined ores 1 : open-pit-mined ores 2, will be built on the eastern slope of Cerro-Pichakani neighboring Siglo XX.

(2) The concentration system, for low-grade crude ores of about 0.3% Sn, will employ table concentration combining the crushing and classification processes and placing emphasis on the

liberating property of cassiterite as the main element of the system. Pre-concentration by means of sink-and-float separation will not be employed because there is a possibility of a drop in recovery rates, a cost hike for materials (mainly ferrosilicon) and instability of operation.

(3) Planning for the concentration results is based on the results of the above-mentioned tests (for their contents, refer to the following section). When putting this construction plan into practice, as with the usual steps taken when starting an enterprise, a pilot plant (20 t/D or more) will be built to increase the accuracy of the sampling data before entering into a detailed design. Incidentally, the expected values for the concentration results are a grade of concentrate of Sn 50% for high-grade crude ore and Sn 4% for low-grade crude ore, and a total recovery rate of Sn 60%. To fully realize these values, sufficient monitoring by means of real time processing will be employed in instrumentation and analyses.

(4) By premising the adoption of a preventive maintenance system, the number of annual working days is planned to be 300 days, equal to that in Japanese mills. The form of operation will follow a pattern consisting of two continuous operation weeks and two continuous holidays so that sufficient maintenance of equipment may be carried out.

(5) A personnel plan will be made based on the proper arrangement of workers in contrast with the present conditions. As too many workers are employed at present, computations based on the present personnel costs may not be correct, but even when the quality of equipment is improved and operational control becomes easier, an increase in the quantity of work performed per man cannot be expected, so that personnel costs will be computed on the basis of attaining an overall improvement in the balance of payments.

(6) This new plan is based on the assumption of 10 years of operation. Regarding ore production, two cases, 9,000 t/D and 8,000 t/D, reduced from the initial value of 10,000 t/D when there is no reserve of underground-mined ores that can maintain the expected grade for ten years, will also be examined.

(7) Simultaneously with scale expansion, disposal of tailings and recovery of water used will become more important than in the previous mill, corresponding to the finer crushed particle sizes.

At first, the water balance will be planned so as to keep the pH value of the water supply at the neutral value and to secure a sufficient quantity of water.

Concerning the disposal of tailings, it may be better, insofar as the construction cost is concerned, to continue the so-called "free discharge" system, based on the tailings flowing into Lupi-Lupi, similar to the present system. However, to give priority to the recovery of used water mentioned above, a plan for tailings dams to be built at certain positions for recovering the supernatant water to secure an adequate water supply for concentration will be made.

Such a plan for dams is thought to be the first of its kind in Bolivia, so that the plan will be examined in detail as a model case.

2–2 Various Tests for Planning New Concentration System

2–2–1 Investigation of Liberation Property Cassiterite

As the tin grade of Desmonte (old tailings of sink-and-float separation) and Block Central (underground-mined ores), which are the object ores of the new mill, is as low as about 0.3%, it is difficult to calculate liberation accurately.

According to results from observing ores by EPMA (Electron Probe-microanalyzer), the crystalline grain sizes of cassiterite are, both in Desmonte and Block Central ores, about 50μ maximum, fine ones being of the order of submicron size, and some ores in which cassiterite exists in the state of dissemination were also found. Accordingly, grinding close to sink-and-float separation particle sizes is thought to be required, and crushing to at least 200~300μ will be required to attain a sufficient degree of liberation.

On the other hand, in cassiterite recovered by tables from ores entirely crushed to 150μ, many particles of over 50μ in size were found.

From these results, the above-mentioned cassiterite existing in submicron particle size range or excessively crushed cassiterite are thought to be difficult to recover. In conclusion, it can be said that the optimum particle size range for the object ores shall be an appropriate grinding particle size of around several tens of microns, taking into account both the selectivity and the degree of liberation of the minerals.

2–2–2 Crushing Test

To test the crushability of an ore mixture comprising Desmonte and Block Central ores at the ratio of 65:35, the ores were crushed with a jaw-crusher (testing machine). Grades by grain sizes and size distribution curves are shown in Table II–2–3 and Fig. II–1–6.

Table II-2-3 Crushing Test (Mixed Ore -20mm)

Product	Wt %	Grade %		Dist. %	
		Sn	Fe	Sn	Fe
Total	100.0	0.39	2.10	100.0	100.0
20mm -11.1	18.6	0.31	2.24	16.2	19.8
- 9.52	10.4	0.34	2.12	9.0	10.5
- 6.73	16.6	0.34	2.02	14.6	16.0
- 4.76	7.0	0.29	2.02	5.2	6.7
- 3.36	11.7	0.40	2.24	12.0	12.5
- 1.68	10.5	0.36	2.20	9.8	11.0
- 0.84	7.2	0.38	1.82	7.1	6.2
- 0.30	7.0	0.59	1.60	10.7	6.0
minus - 0.30	11.0	0.54	2.14	15.3	11.2

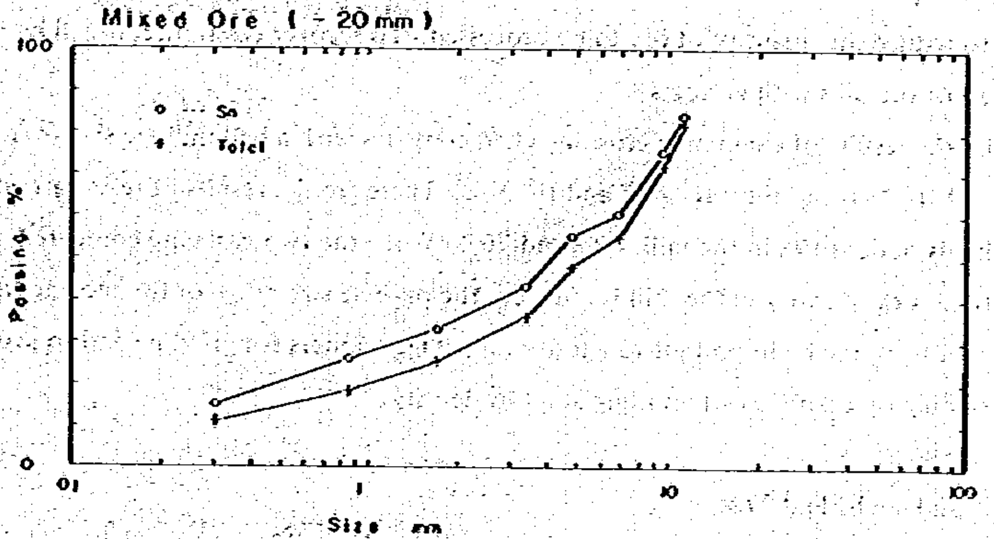


Fig.II-2-6 Size Distribution of Crushing Test (Table I)

These show that although tin is concentrated to some extent in the finer grain size range, on the whole it seems to be uniformly distributed, proving the results described in the preceding section that cassiterite is finely scattered.

2-2-3 Grinding Test

The work indexes of the object ores are 15~16 KWh/t, showing that they are comparatively hard. Accordingly, a high grinding cost is supposed, so a test was carried out to crush mixed ores, which had been crushed beforehand to -12 mesh, by a crushing roller into fine particles for the purpose of studying the possibility of energy conservation in grinding. The results are shown in Table II-2-4.

The results show that the tin content of crushed products by grain sizes is distributed almost uniformly. It will be difficult to conserve energy by an extra grinding process in the stage of grinding the coarse grain of about −12 mesh (1.397 mm) in size into fine particles and removing the low-grade portion as Desmonte in order to reduce the quantity of ore to be processed. To recover cassiterite, it will be required to grind all of the ores with grinders.

Table II-2-4 Grinding Test (Mixed Ore −12 mm)

Product	Wt %	Grade %		Dist. %	
		Sn	Fe	Sn	Fe
Total	100.0	0.38	1.50	100.0	100.0
35 meshes	31.1	0.35	1.66	28.6	28.8
48 "	11.8	0.38	1.89	11.7	12.4
65 "	11.1	0.38	1.64	10.9	10.1
100 "	6.8	0.41	1.80	7.3	6.8
200 "	9.9	0.48	1.81	12.3	10.0
400 "	7.6	0.54	2.00	10.7	8.4
−400 "	21.7	0.33	1.95	18.5	23.5

Next, the results of examining crushing characteristics with a ball mill are shown in Table II−2−5 and II−2−6 and Figs. II−2−7 and II−2−8. These are the results of tests carried out for two kinds of densities in the mill, 60% and 70%. When the two cases are compared, it appears that as the density in the mill is reduced, the particle size range of tin becomes narrower. In other words, it will be important to adopt multi-stage grinders for grinding and to avoid excessive grinding by carrying out grinding at a low density.

2−2−4 Sink-and-Float Test

The sink-and-float separation may have an important relationship with the conservation of energy in grinding. In the Mine, a pre-concentration mill is in operation.

However, as the grain sizes of cassiterite in the object ores are fine, good results were unable to be obtained by means of sink-and-float analyses carried out by this investigation committee in the preceding year. Fig. II−2−9 shows the results of sink-and-float tests of Desmonte, indicating that the specific gravity of heavy medium which gives

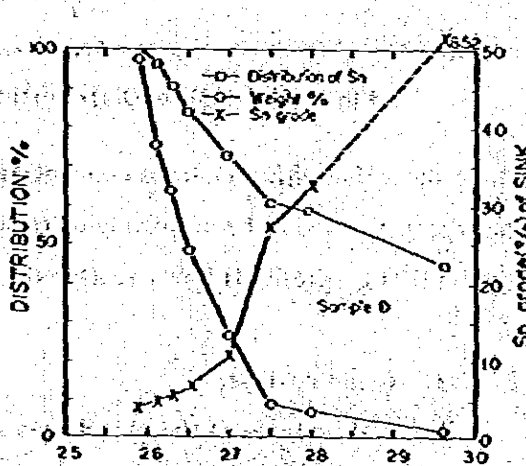


Fig II-2-9 Result of Sink and Float Test (Desmonte 168/021mm)

Table II-2-5 Grinding Test Wet Grinding P.D. 60%

Product	Wt %	Grade %			Distribution %		
		Sn	Fe	S	Sn	Fe	S
Total	100.0	0.37	1.79	0.52	100.0	100.0	100.0
+ 65 Mesh	12.0	0.17	1.72	0.29	5.4	11.5	6.7
+ 100 "	11.2	0.28	1.65	0.50	8.6	10.3	10.9
+ 150 "	15.1	0.45	1.70	0.54	18.3	14.3	15.8
+ 200 "	10.5	0.51	1.62	0.52	14.7	9.5	10.6
+ 400 "	13.8	0.53	1.67	0.55	20.0	12.9	14.7
− 400 "	37.4	0.32	1.99	0.57	33.0	41.5	41.3
− 200 Mesh	51.2	0.38	1.90	0.56	53.0	54.4	56.0
− 150 "	61.7	0.40	1.86	0.56	67.7	63.9	66.6
− 100 "	76.8	0.41	1.82	0.55	86.0	78.2	82.4
− 65 "	88.0	0.40	1.80	0.55	94.6	88.5	93.3

Table II-2-6 Grinding Test Wet Grinding P.D. 70%

Product	Wt %	Grade %			Distribution %		
		Sn	Fe	S	Sn	Fe	S
Total	100.0	0.38	2.02	0.50	100.0	100.0	100.0
+ 65 Mesh	10.7	0.28	1.83	0.52	8.1	9.7	11.0
+ 100 "	16.1	0.34	1.87	0.50	14.4	14.9	16.8
+ 150 "	10.9	0.41	1.98	0.51	11.9	10.7	11.0
+ 200 "	9.9	0.45	2.09	0.48	11.9	10.2	9.4
+ 400 "	12.8	0.50	2.04	0.46	17.0	12.9	11.7
− 400 "	39.6	0.35	2.12	0.52	36.7	41.6	40.9
− 200 Mesh	52.4	0.38	2.10	0.51	53.7	54.5	52.5
− 150 "	62.3	0.40	2.10	0.50	65.6	64.7	62.0
− 100 "	73.2	0.40	2.08	0.50	77.5	75.4	73.0
− 65 "	89.3	0.39	2.04	0.50	91.9	90.3	89.0

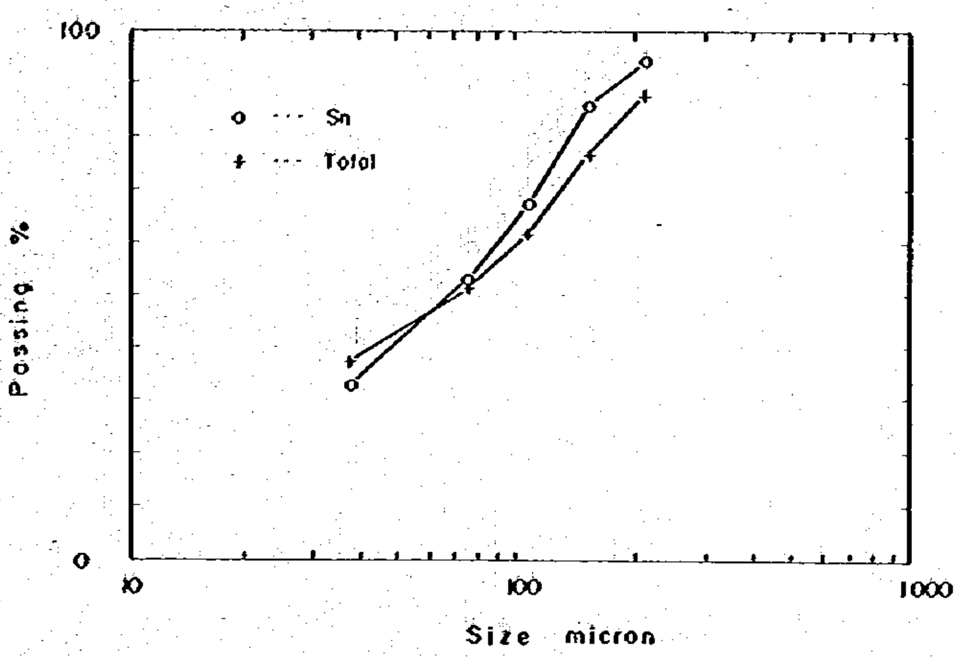


Fig.II-2-7 Size Distribution of Wet Grinding Test -PD60%
(Table 5)

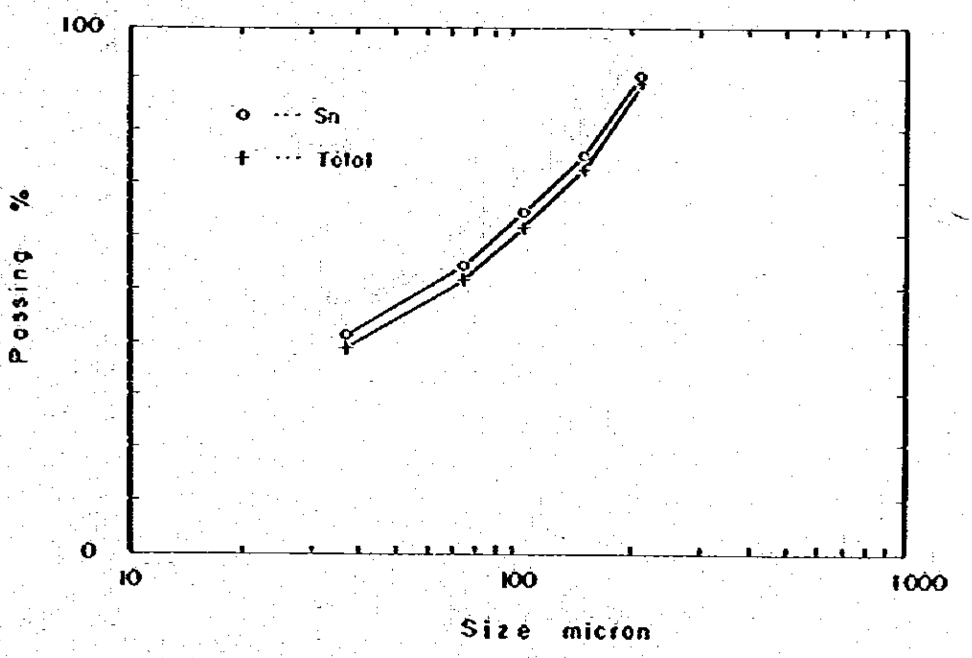


Fig.II-2-8 Size Distribution of Wet Grinding Test -PD70%
(Table 6)

comparatively good grade of separation is over 2.75 and the recovery rate of tin is about 60%.

In the range of grain sizes coarser than this, the grade of separation is extremely poor and moreover, the theoretical recovery rate is low even at a grain size of 1.68/0.21 mm in addition, an effective sink-and-float separation method at this grain size cannot be found. For these reasons, sink-and-float separation was excluded from the object of this investigation.

2–2–5 Gravity Concentration Test

From basic tests conducted last year, it was found that cassiterite showed a good grade of separation in a fine particle size range (–100 mesh). Furthermore as described in 2–2–1, grinding grain sizes should center around several tens of microns.

In this test, however, with the intention of reducing costs as far as possible, separation in a coarser grain size range was also taken into account, and separation tests with a table were carried out for different grinding grain sizes, and furthermore for several grain size classes after sieving.

1) Table Test No. 1 (Fig. II–2–10)

In this test, mixed ore (Desmonte: Block Central = 65:35) were first grinded to 80% grain size, about 180μ by a ball mill (p.d 60%). These were then classified into products of +100 mesh, 100 ~ 400 mesh, –400 mesh and slime, and separation test were carried out for all the products except slime.

The recovery rate in the preliminary flotation stage was 70.5% and the grade of preliminary concentrate was 13.4% Sn. When shown by grain sizes, the grade of separation was best in the 100 ~ 400 mesh range and the recovery rate here was 87%.

The preliminary concentrate was concentrated collectively by sulfide floatation (preliminary flotation pH4.5 Etx 100 g/t, NIKKO #125 – Cleaner flotation) and table.

The grade of final concentrate was 48.93% Sn and the recovery was 59.8%. The cleaner tailings may be subsequently treated as middlings or may be handled as low-grade concentrate for volatilization.

2) Table Test No. 2 (Fig. II–2–11)

Ores grinded to 80% grain size 560μ by a crushing roller were classified into products of +35 mesh, 35 ~ 100 mesh, 100 ~ 400 mesh and slime, and separation tests for all the product except slime were carried out.

In this test, in the range of over 400 mesh preliminary concentrate was concentrated by cleaner flotation and the tailing were then handled as middlings, while in the range of –400 mesh, the preliminary concentrate was handled directly as middlings.

Table test NO. I

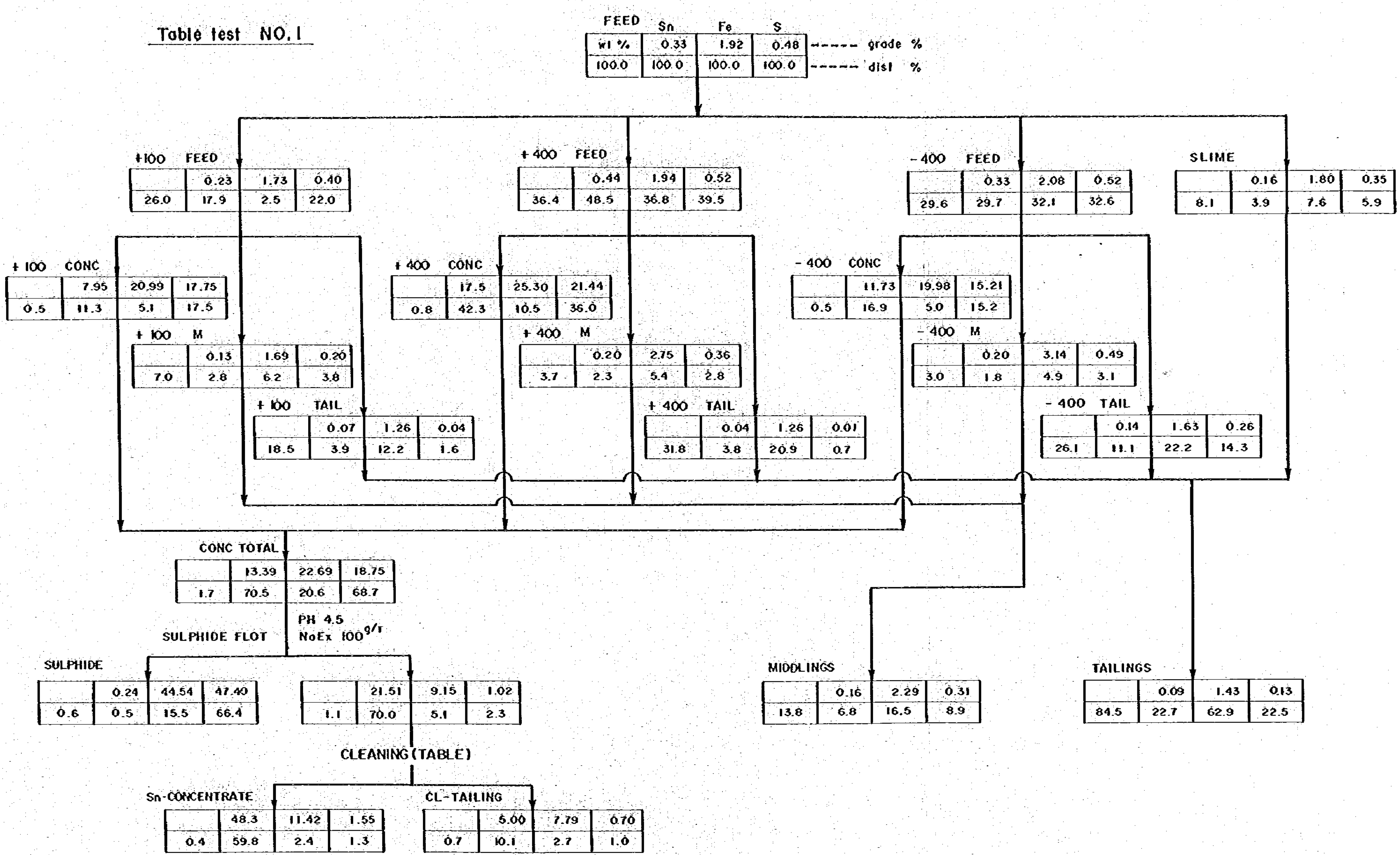


Fig. II-2-10 Table Test - Mixed Ore (D80 = 180 micron Wet Grinding)

Table test NO.2

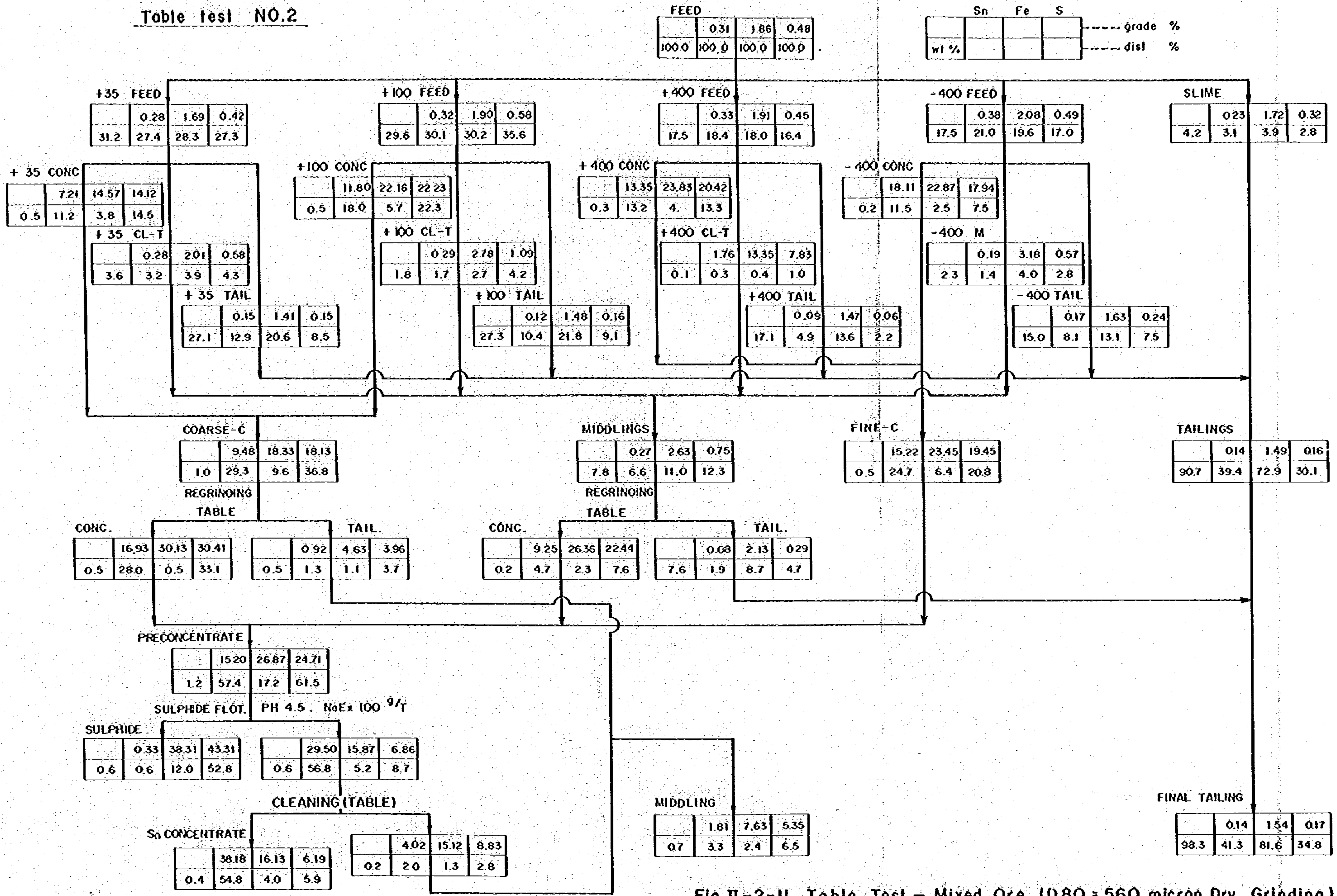


Fig.II-2-11 Table Test – Mixed Ore (D80 = 560 micron Dry Grinding)

Table test No.3

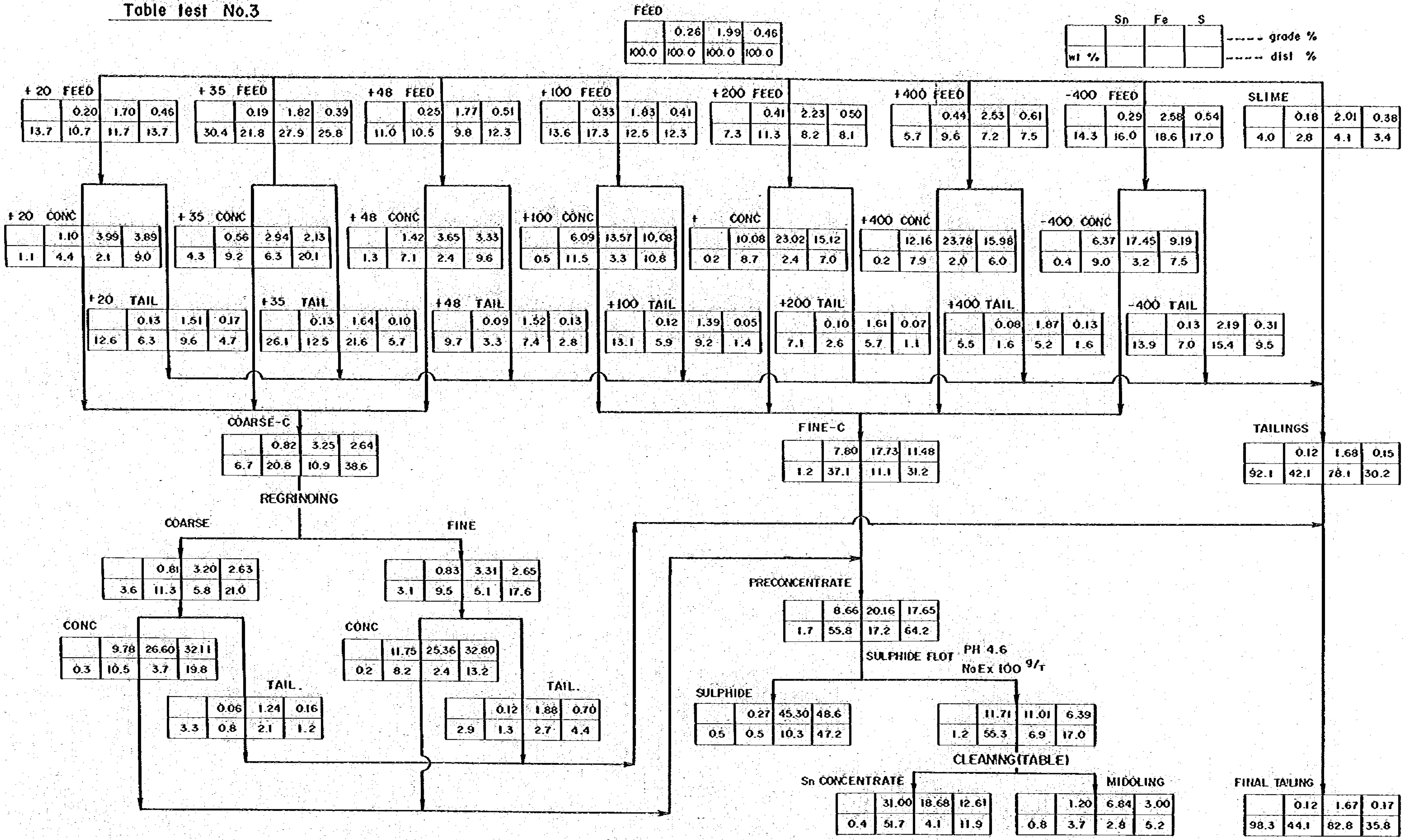
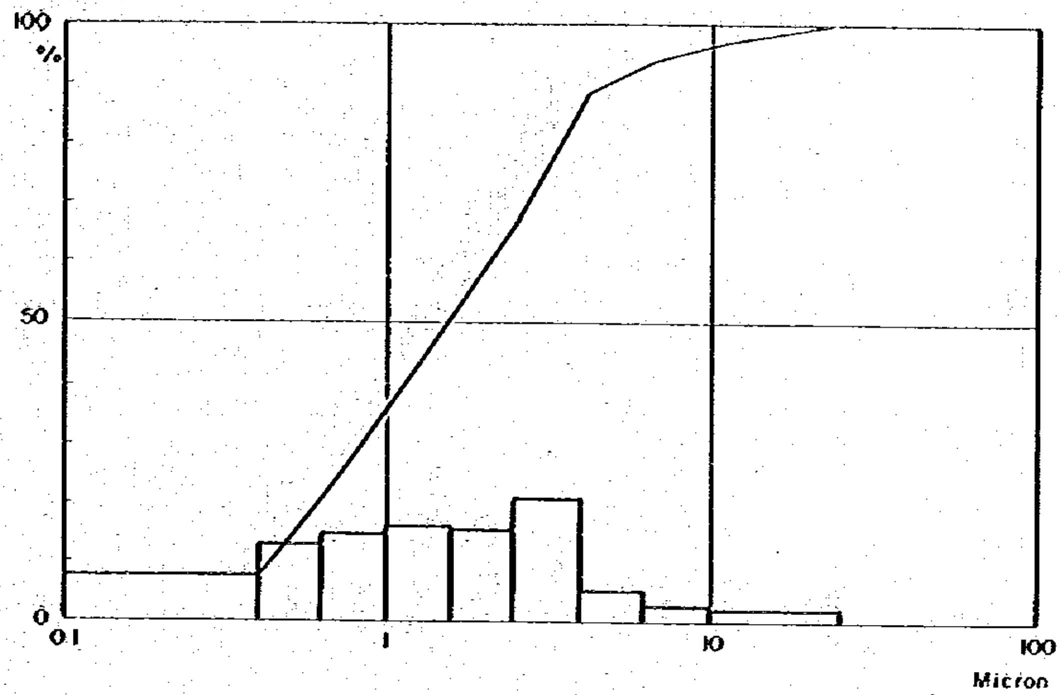


Fig. II-2-12 Table Test – Mixed Ore (D80 = 730 micron Dry Grinding)



MEMO

Size distribution of slime ;

deslimed by multiclone

Pres. 2.0 kg/cm2

P.d 7.5 %

Slime ratio 4.2% - Head

SIZE	Wt %	CUMULATIVE
25.00	0.0	100.0
16.00	1.9	98.1
10.00	1.9	96.2
6.30	2.7	93.5
4.00	5.2	88.2
2.50	20.8	67.4
1.60	15.5	51.8
1.00	16.1	35.7
0.63	14.9	20.8
0.40	13.3	7.5
0.00	7.5	0.0

Fig. II-2-13 Size Distribution of Slime of Table Test No.2

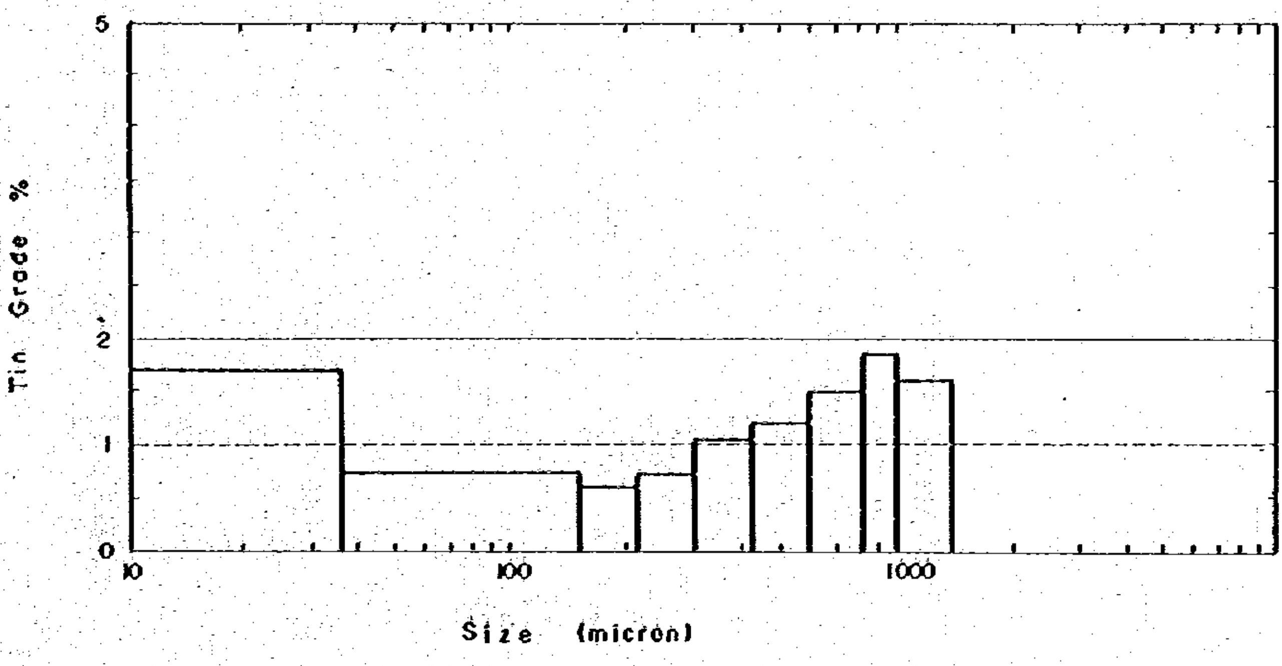


Fig. II-2-14 Particle Size & Tin Grade of Tailing - Table Test No. 2

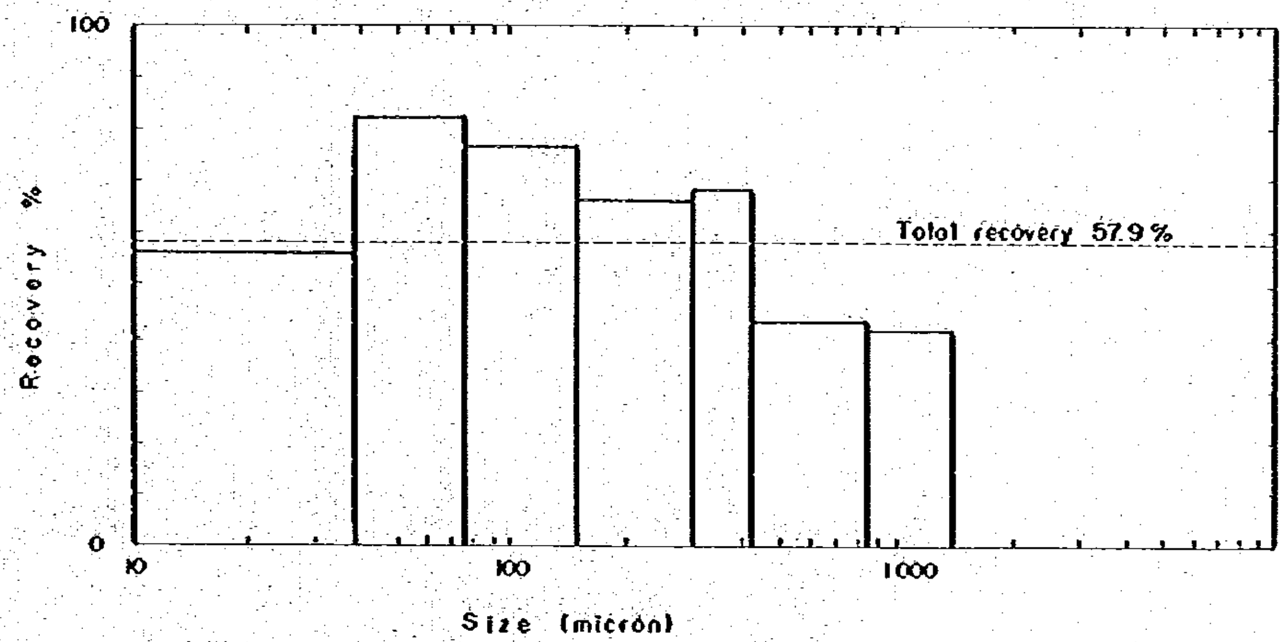


Fig. II-2-15 Particle Size & Tin Recovery - Table Test No. 3