CHAPTER 2. CIRCUMSTANCES AND PURPOSE OF MODERNIZATION PROGRAM FOR CATAVI MINE

In April 1980, the Japanese Government accepted a request to provide renewal and modernization plans for mines under the control of the Public Mining Corporation of Bolivia (COMIBOL). According to this acceptance, the Japan International Corporation Agency (JICA) dispatched a preliminary investigation mission to the site to consult with COMIBOL. As a result, the mission selected Catavi Mine as the target for modernization based on the agreement.

In the first-year term from July 1981 through March 1982, the team, in order to prepare the model plan requested by COMIBOL, studied the status quo and gathered samples and data of the Mine at the site, while investigating the data and testing the samples in various ways to extract and analyze problems to be worked on in Japan, and key points were obtained to prepare for the second-year term investigation.

The following are the essential problems among those grasped by the first-year term investigation.

(1) Ore reserve higher in grade than the break-even point is reaching its end; i.e. the reserves of such high-grade ores were only at a three-year level at the end of 1981.

(2) The large-scale block caving method which is now being planned is not suited to the existing facilities and the ore deposit conditions.

(3) The existing concentration facilities and their operating systems, which were provided to treat high-grade ores, are inadequate for low-grade ores with a low yield, which occupy the major part of present ore production.

(4) Concentration tests were carried out on low-grade underground ores, Siglo XX's Desmonte and Victoria's tailings. The results have shown that the grain size of these ores is as fine as less than 50 mesh, and that Siglo XX's Desmonte are the easiest to concentrate.

(5) Striking problems in the administrative aspect are the deterioration of machines and facilities, a lack of maintenance, deficiencies in administration and complicated organization.

(6) If the present operating system continues in the future, it is apparent that a pre-tax loss of over US$ 10 million/year continue year after year.

(7) Because Catavi Mine is one of the important sources of national revenue, whether the Mine can survive or not is of great concern to both the local community and the nation.

The targets for the second-year term investigation were determined as follows, from the overall examination of the above problems and repeated discussions with COMIBOL.

(1) Basic designs for a new operating system for modernization, including (a) the design of sub-level stoping methods to selectively exploit high-grade ore deposits within the Block Central area which is now under planning; (b) conceptual designs for a new mill plant with double the previous treatment capacity and a new system which is matched to underground ores and Siglo XX's Desmonte; and (c) a proposal for a new administrative system.

(2) Economic appraisals of the new operating system.

(3) The establishment of mid-term exploitation plans in the vicinity.

In the second-year term investigation, based on the above fundamental policies, the team surveyed the site again, and after returning to Japan carried out additional concentration tests on samples, and examined and analyzed the data. Thus the stage has now been reached for the presentation of proposals on a new operating system for modernizing the Mine, the expected target of the team, described hereinafter.

CHAPTER 3. OUTLINE OF THE INVESTIGATION

The investigation of Catavi Mine was carried out to prepare the conceptional design for a new operating system for modernizing the Mine, on the basis of investigation results mainly obtained from analyses of the present situation. These basic designs are: (1) the preparation of an exploitation program using the sub-level stoping method for the high-grade ore underground portions, (2) examination of basic designs and new management systems for a new concentration plant, and (3) planned exploration to detect new ore deposits in the vicinity. Because the investigation covered every field of the mining industry, the team consisted of experts on exploitation, mining, concentration, facilities engineering and economic analysis, and after its homecoming the team analyzed the data with the support of specialists in the respective fields.

During the on-site investigation, specialized counterparts of COMIBOL participated in the survey, which greatly helped the smooth progress of the investigation. The on-site survey mainly aimed at collecting the necessary data for the various designs and for planning methods and sites of exploitation.

The team was organized with nine persons including one who joined at the site. The eight persons left Japan on July 2, 1982, and three returned home on August 4, while the remaining five returned on August 5. Thus the term covered 35 days, from July 2, through August 5.

Prior to this second-term investigation, various concentration tests such as crushing, grinding and gravity separation were performed, and after homecoming these test results were reconfirmed on samples which had been brought back. These examinations were (1) chemical analyses, observations through microscopy, EPMA appraisals and the physical characteristics of ores for exploitation, (2) the planning of exploitation using the sub-level stoping method for mining, (3) various concentration tests and successive designs for a new concentration plant and related incidental facilities, and estimations of the corresponding construction and operating costs for facilities engineering, and (4) revenue and expenditure calculations and an investigation of the profitability of the project for administration. All of these are described and explained in the report for the second-term investigation. The work schedule is shown in Fig. 1–1.

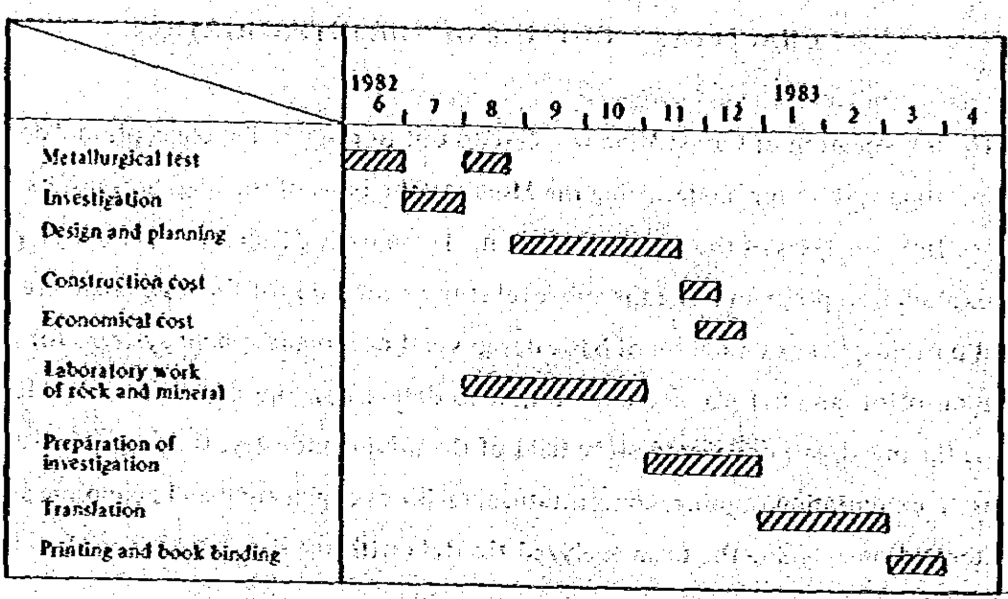


Fig. I–1 Schedule of Work

The Japanese members and their Bolivian counterparts participating in the investigation are listed below.

Japanese Team

Leader	Minoru Sumita (General, Geology)*	Dowa Koei Co., Ltd.
Member	Hedio Janome (Geology)*	"
"	Hiroji Kuronuma (Geology)	"
"	Masanori Ochiisi (Mining)*	"
"	Mamoru Takeda (Mining)	"
"	Kazuo Shuto (Metallurgy)*	"
"	Noboru Sato (Metallurgy)*	"
"	Kazuo Shojaku (Metallurgy)*	"
"	Hiroo Washimi (Metallurgy Facilities Eng.)	"
"	Akio Hiroshima (Metallurgy Facilities Eng.)	"
"	Hironori Akasaka (Facilities Eng.)*	"
"	Yuji Takemoto (Facilities Eng.)*	"
"	Takao Matsumoto (Facilities Eng.)	"
"	Kenji Sawaguchi (Facilities Eng.)	"
"	Minoru Shinozaki (Economic Analysis)*	"
"	Soichi Ito (Economic Analysis)	"

* : Participated at the site.

Bolivian Counterparts

Leader	César Mercado (Mining)	COMIBOL
Member	Aurelio Bustos (Geology)	"
"	Juán Maita (Metallurgy)	"
"	Edmundo Contreras (Civil)	"
"	Jorge Collazos (Mechanical Eng.)	"

CHAPTER 4. BASIC CONCEPTS OF THE PROJECT

(1) A new operating system must be planned, which is suitable for the mining and treating of crude ores on a scale of 10,000 t/day, with the aim of greatly increasing production of metallic tin and improving the balance of revenues and expenditures in relation to the ore reserves, with the recovery of concentrates expected from the new operating system.

Table I–3 Revenues and Expenditures

(Sum of 10 years)

Article				10,000 t/day treatment					
				First year ~ 7th year total		8th year ~ 10th year total		Sub total	
Production	Mining	(Per day)	Mine	0.41%	3,500t	0.22%	2,000t	0.38%	3,050t
			Desmonte	0.27	6,500	0.27	8,000	0.27	6,950
			Total	0.32	10,000	0.26	10,000	0.302	10,000
			Mine		1,000t 7,350		1,000t 1,800		1,000t 9,150
			Desmonte		13,650		7,200		20,850
			Total	0.32	21,000	0.26	9,000	0.302	30,000
	Concentration	High Grade Conc.	Dry Conc.	50.0	73,900t	45.05	26,900t	48.76	98,830t
			Tin Metal		36,960		11,232		48,192
			Recor.		55%		48%		53%
		Low Grade Conc.	Dry Conc.	4.10	81,900t	4.40	41,850t	4.2	123,850t
			Tin Metal		3,358		1,841		5,199
			Recor.		5%		7.9%		5.7%

(2) The new concentration plant must treat mixed ores consisting mainly of Desmonte from Siglo XX with high-grade ores which are selectively mined from block caving areas, in order to maintain the operation of the Mine for at least 10 years with a possible mining ore reserve against the production scale shown above.

(3) The productivity must be improved by adopting a new system in each of the Mining, Concentration and Administration Departments, to improve as much as possible the worsening balance of revenues and expenditures which has mainly been caused by yearly rises in costs.

(4) The personnel allocation and administrative systems must be so efficient that the above-mentioned policies can be smoothly put into practice.

(5) The new operating system must be evaluated in terms of its economic aspects. Some case

studies must also be estimated with variations in ore grades and stoped quantities of crude ores, in order to find the case in which the best balance of revenues and expenditures can be obtained within the ten-year period.

(6) A mid-term integrated exploration program is required which is well suited to the existing conditions of the ore deposits, since the discovery of new ore deposits in the vicinity of the Mine is the most desirable, in order to develop COMIBOL itself and the local community along with the modernization of Catavi Mine.

PART II CONCEPTUAL DESIGN OF THE PROJECT

CHAPTER 1. PLAN FOR THE MINING DEPARTMENT

1–1 Circumstances and Design of Mining Plan

Among the production costs of Catavi Mine, mining costs account for a major portion, Personnel costs rate above all. Accordingly, the most effective measure for cost reduction is the reduction of personnel costs based on improvement in productivity. To improve productivity, it is necessary to establish a rational production system adapted to the ore conditions.

To cope with the decrease of income resulting from minable ore grade degradation, the mining department has implemented low cost exploitation by means of mass production. Under this policy, the block caving method, a representative large-scale low-cost mining method, has been introduced extensively.

As a result, a mass production system has completely introduced, but on the other hand, the grade of ores had dropped substantially, and in addition, such difficulties as rapid increase in problems in large block processing at mining stopes.

If the mining method is not selected appropriately based on the comprehensive and thorough consideration of ore deposit conditions (country rock properties, ore grade distribution, etc.), transportation equipment, concentration equipment, etc., the method will not be helpful in improving overall productivity and profitability.

In Catavi Mine, high grade ores have been exhausted and the grade of underground mining ores has fallen to levels lower than those of Desmonte (waste rocks after sink-and-float separation) and relaves (tailings) accumulated on the surface. (See Table 1.)

As urgent improvement measures under these circumstances, the following measures are thought to be required.

1 Adoption of an underground mining method suitable for ore deposit conditions.

2 Use of open-pit mining ores.

(1) Adoption of an Underground Mining Method Suitable for Ore Deposit Conditions

As the object area for underground mining in future, the largest Block Central has been selected. The ore reserve of this area is 38 million tons and its average ore grade is 0.20%.

The ore grade distribution in the Block Central area (according to the ore reserve estimation carried out in Catavi Mine) shows grade difference locally, so that improvement in the grade of underground mining ores can be expected by setting ore blocks appropriately.

In addition, the geological features of this deposit are hard and have few cracks and is judged to be a deposit which is hard to cave. The block caving method adopted at present is suitable for large scale mining and deposits under geological conditions easily caved. To adopt block caving in such a mine as Catavi whose geological condition is hard and has few cracks, it is required to widen the object area so that caving may become easier and to make its underground structure larger so that large blocks can be processed easily.

In addition to the increase in the size of the mining department, it becomes necessary to increase the size of the crushing system and requires a large amount of investment, putting up another big problem. Moreover, the expansion of the mining area means that mining is expanded to low-grade ores, so that the degradation of crude ore grade is inevitable. Under the deposit conditions of this mine, improvement in crude ore grade can be expected by limiting ore blocks.

When the following are taken into consideration in the selection of the method.

1 A method which can mine selectively to a certain extent (such a method can lead to improvement in ore grade),

2 A method suitable for geology which is hard and has few cracks (reduction of large block processing),

3 A method capable of low-cost mass production,

4 A method which does not require to change the existing equipment greatly,

The "sublevel, stoping method" is judged to be most appropriate.

If the sublevel stoping method is applied, the following advantages can be obtained:

1 Imporvement in crude ore grade can be expected by arranging ore blocks appropriately,

2 Ores can be broken into small pieces, so that there are few problems which are incurred in processing large blocks,

3 Stopes can be standardized and orderly circumstances can easily be mainteined.

On the other hand, the method has the following disadvantages:

1 Drilling man-hours will increase,

2 The extent of development will increase.

However, compared with the present conditions in which a large number of man-hours and a large amount of explosives are consumed for isolation shrinkage stoping and coyote blasting, it is expected that man-hours will not increase so much, while the effect of efficiency improvement and cost reduction will come through substantially.

(2) Use of Open-pit Mining Ores

An example of open-pit mining ores is Desmonte in Siglo XX. The ore reserve of it is 22 million tons and its average grade is 0.27%, so that it is comparatively worthy. Although about 100 thousand tons per year are mined, the ores can be mined on a large scale immediately. As these ores are the float of sink-and-float separation, their grain sizes are compratively

Table II–1–1 Summary of Reserves of the Catavi Mine (Jun. 30 1981)

Tipo de reserva	Tons. min.	%Sn	Tons. fino
Vetas	443,472	1.52	6,757.71
Vetas en blocks	115,399	2.08	2,398.34
Ruentes	44,338	2.88	1,275.16
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	209,000.00
Gran total	355,524,776	0.07	233,629.60
*Block central	38,000,000	0.20	76,000.00

uniform and the loading and haulage of the ores is easy and stable operations can be expected, therefore, costs can greatly be reduced by the mining of Desmonite.

From the above, in planning the mining project this time, a production plan covering 10 years in future will be made including the transition period from the present state to a new mining system to reduce mining costs and improve productivity by balancing the underground mining ores and open-pit mining ores, realizing selective mining by means of sublevel stoping and using low-grade open-pit ores together.

Under this mining plan, a production of 10,000 tons/day will be secured, and, by considering from ore reserves, the production plan will be based on a ratio of 3,500 tons/day of underground production and 6,500 tons/day of open-pit production.

1–2 Sublevel Stoping Plan

1–2–1 Integral Layout

In the sublevel stoping method, an ore blocks between an upper drift and a lower drift is divided with horizontal drifts (sublevels) to make several horizontal pillars. Next, in a horizontal pillar just on the lower scram drift are made several short raises, and, by spreading their upper parts into cone shapes, the raises are made into draw cones (chutes) which receive falling ores when mining horizontal pillars above. The mining of horizontal pillars is begun with the lower one, and the upper horizontal pillar is mined a little later in a retreating system. Accordingly, all the crushed ores fall directly down to the lower chutes, the ores that fall down the chutes flow out to the scram drift, are scraped and poured into a chute.

A diagram of the sublevel stoping is shown in Fig. II–1–1.

In the case of the Block Central area in the mining plan this time, when the part between L411 and L551 is considered, ores flow down in the following order: stope → L551 → L650, while air flows up in the order. L650 → L551 → stope → L411. For the flow of men and materials, a service shaft will be made outside the mined area which will be connected with each level.

As the extent of development drifts will become a considerable length, it is desirable to increase the quantity of development ore product by arranging as many drifts of various kinds as possible in the ore body. Accordingly, the scram drift will be developed on the level of L551.

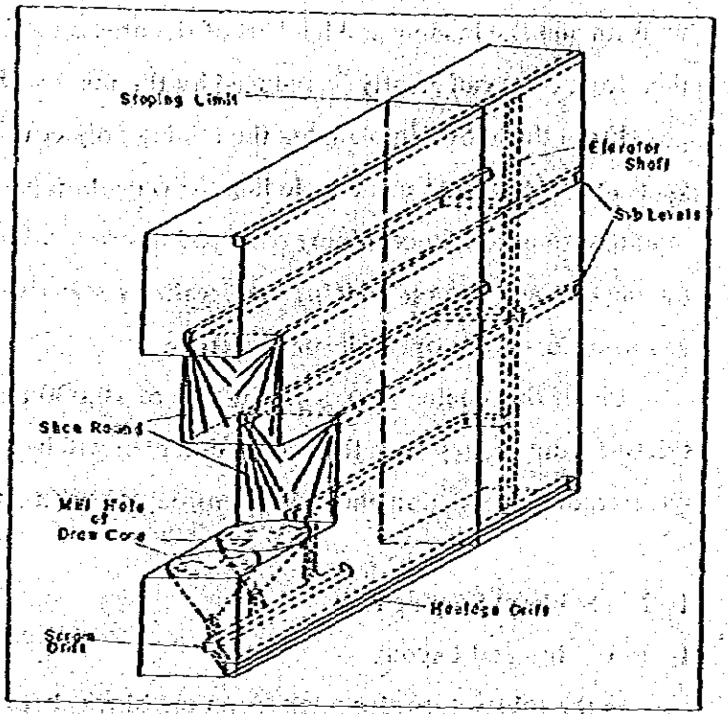


Fig. Ⅱ-1-1 Isometric View of Sublevel -Stoping Activities

(1) Pillar Width

Pillars in the sublevel stoping are required for maintaining stoping caverns and mining safely, and the size of a pillar must be determined taking ground conditions, safety and mining recovery into account. The size of a pillar is determined taking its supportable strength and stress generated in the pillar and in its neighborhood, and as design standard for the pillars, "The Second Report on the Design of Pillars" states the requirements prevalent in Japan. We therefore investigate the required width of pillars from the standpoint of safety in accordance with the report.

(a) Supportable Strength of Pillar

The supportable strength of a pillar Sp is estimated, referring to the uni-axial compressive strength Sc of a test piece taken from the pillar, to be a little lower than Sc, usually about 70% of Sc.

In the case of Catavi Mine, the average value of the uni-axial compressive strength Sc obtained from the results of the tests in rock mechanics carried out last year is 897 kg/cm^2, so that the supportable strength of pillar Sp becomes,

$$Sp = 897 \times 0.7 = 628 \text{ Kg/cm}^2$$

(a) Relation between Stress in Pillar and Pillar Thickness

Stress in a pillar is determined by the condition of rock stress, the geometrical conditions of stoping caverns and the pillar, but here, we assume that the stress is given by the average stress caused by earth pressure acting vertically. The value of the average stress can be calculated from the following equation.

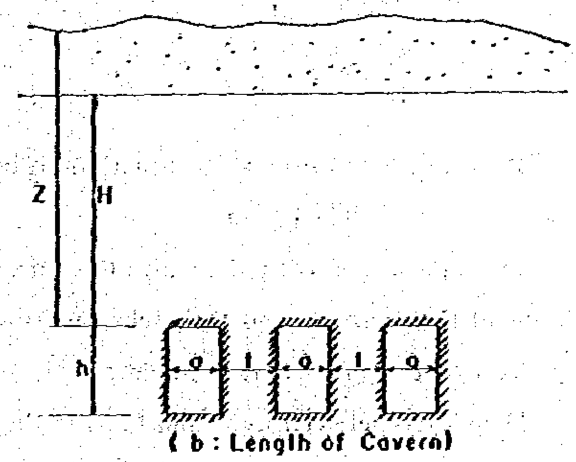


Fig. II-1-2 Geometrical Condition of Stoping Cavern

$\bar{\sigma} \doteqdot Kr \cdot Kb \cdot \gamma Z \frac{a+t}{t}$

$\bar{\sigma}$: average stress (t/m^2)

γ : average specific gravity of rock (t/m^3)

z : vertical height from cavern roof to the surface (m)

t : pillar thickness (m)

a : cavern width (m) b : cavern depth (m)

Kr: distribution factor of earth pressure when cavern length is assumed to be infinite

Kb: distribution factor of earth pressure when cavern length is assumed to be finite

By using values $\gamma = 2.70$ t/m^3 and $z = 400$ m, Kb, the distribution factor when the cavern length is assumed to be finite is determined to be 0.9, on the safer side. If the cavern width "a" is determined to be 20 m based on past records in Japanese mines, the relation between the average stress $\bar{\sigma}$ and the pillar width t becomes as shown by the following equation

$$\bar{\sigma} \doteqdot 0.9 \times 2.70 \times 400 \times \frac{20+t}{t} = 972 \times \frac{20+t}{t}$$

(b) Pillar Width Seen from the Safety Factor

The safety factor is represented as the ratio of the supportable strength of a pillar to the average stress, and although the value may vary with the method used for processing stoping caverns and the skill of stress control, a value over 3 is usually used.

If a pillar width which makes possible to secure safety factor f above 3 is determined, it can be expressed as follows.

$Sp = 628\ kg/cm^2 = 6{,}280\ t/m^2$

$$\bar{\sigma} = 972 \times \frac{20 + t}{t}$$

$$f = \frac{Sp}{\bar{\sigma}} > 3$$

If t is calculated from these equations, it becomes,

$$t > 17.4\ m$$

From the above, the pillar width this time is determined to be 20 m considering the conditions of the ore body and the rocks and loosening near the surface caused by blasting.

(2) Vertical Distance between Sublevels

Main levels between L411 and L551 in the Block Central area, the area under consideration this time, have the interval of 35 m (L446, L481 and L516). Sublevels (intermediate drifts) can be developed more easily by utilizing these existing drifts.

If the main levels are adopted as vertical distances between sublevels, the sublevel distance becomes 35 m, which is a little too large.

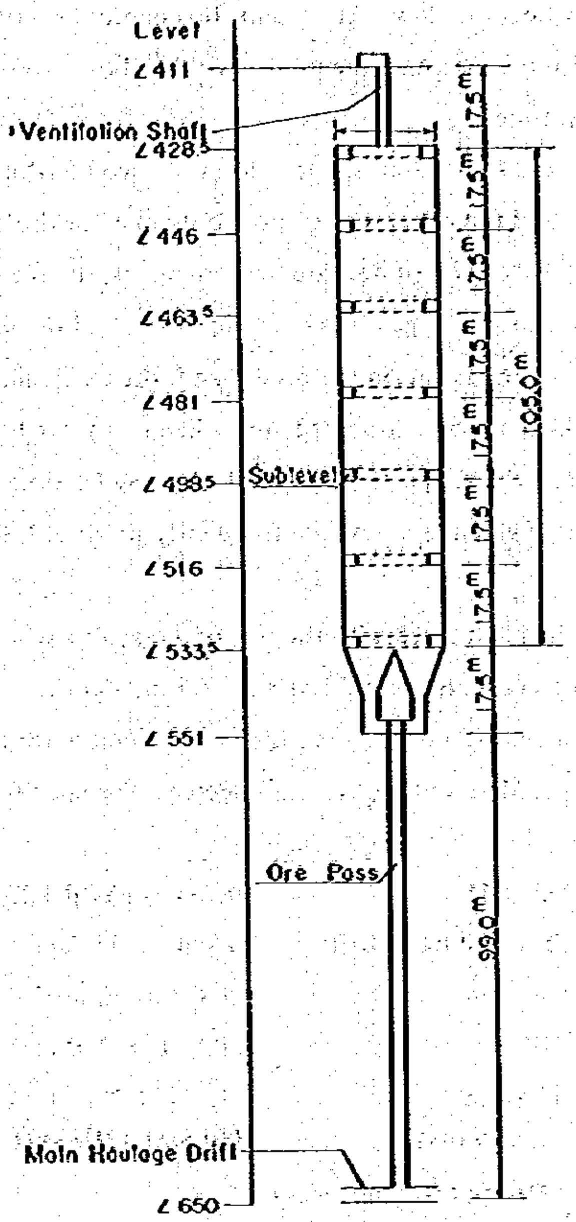


Fig. II-1-3 Arrangement of Sublevel

(3) Disposition and Size of Blasthole Drifts

The disposition of blasthole drifts is very important, because it is closely related with the requirements of long-hole drilling and improvement in blasting efficiency.

The system of long-hole drilling includes parallel cut method and fan cut method. In the case of the parallel cut method, blasting efficiency is high, but floor mucking operation is required prior to widening in sublevels and long-hole drilling, so that operation becomes complicated and its efficiency becomes low. At present, this method is not so widely applied except when specially large drilling machines are used. For the above reason, it was decided to use the fan cut method this time.

The fan cut method includes two systems, one is to make two blasthole drifts one each on both sides of an ore block, and the other is to dispose only one blasthole drift in the center of an ore block. Although the extension of development increases, the former has advantages that blasting efficiency is high and few large blocks are produced, and the loosening of ore blocks is minimal. In the case of the latter on the other hand, the extension of development may be reduced, but ore blocks tend to be loosened, and when many ore blocks are prepared on such a large scale as this time, the supporting of ore blocks may become risky. In addition, there are many other problems, e.g., large blocks are frequently produced, the former system therefore will be adopted.

The size of a blasthole drift must agree with the size of a machine which carries out long-hole fan drilling. This time, interval between sublevels is 17.5 m, and the length of a blast-hole becomes 10 – 20 m. As a compact drilling machine most suitable for these conditions, a ring drill crawler (Toyo Kogyo, Model CS–641) was selected. The specifications of this machine are as follows.

Type	: TYCJ-641	Drifter : TYPR 120 Baby Drifter
Mounting Machine	: TYPR 120 Baby Drifter	Weight : 175 kg
Total Length	: 4,570 mm	Air Consumption : 13.5 m^3/min.
Total Width	: 2,214 mm	R.P.M. 0 ~ 220 rpm
Total Height	: 3,400 mm	Bit ϕ65 ~ 75 mm
Weight	: approx. 7,000 kg	Red Size 32H, 38H
Propulsion	: Air Driven (10.5 m^3/min)	

From the above conditions, the size of a blast-hole drift was determined to be a cross section of 3.5 mW x 3.5 mH.

(4) Interval between Draw Cones and their Dimensions

When determining the interval between draw cones, the width of a mining slope, the flowing ability of ores and the strength of rock are taken into account, and the center distance of draw cones is determined to be 20 m. In sublevel stoping, the hang up of draw cones is an important factor in relation to the safety of operation, and the most effective measure against the hang up is to increase the cross-sectional area of the draw cone. In addition, the increase of the draw cone section area has an advantage of reduce settling quantities remaining on slope bottoms. On the other hand, if the draw cones are made too big, draw points and scram drifts will be loosened and the maintenance of drifts will become difficult. The standard diagram of a draw cone based on these conditions and records in the past is shown in Fig. II-1-4.

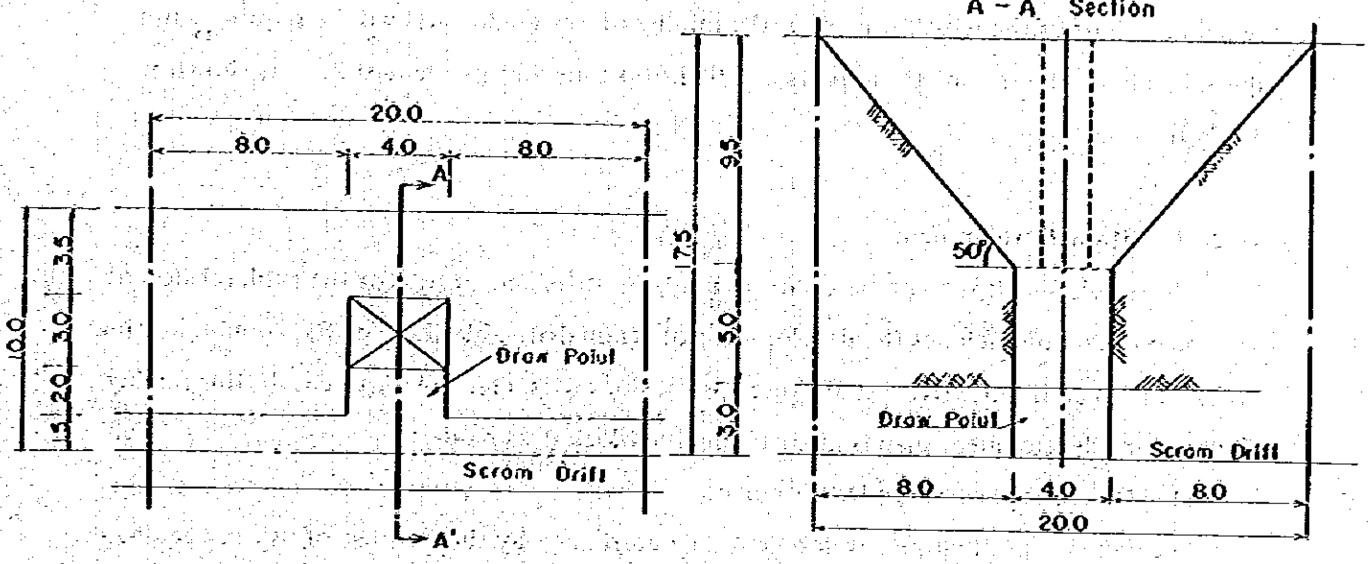


Fig. II-1-4 Plan and Section of Draw Cone (unit : m)

From the above-mentioned conditions, the integral layout of the sublevel stoping can be arranged as illustrated in Fig. II-1-5.

1-2-2 Ore Blocks and Ore Reserve

The Block Central area, which has been selected as a mining object this time, is an ore body where veins concentrate most densely, and block caving is comparatively developed. The geological evaluation of this area has made over these several years and the grade distribution of deposits has been calculated with computers. Based on these data and by considering the most advantageous disposition of sublevel stoping ore blocks, four blocks A, B, C and D have been selected as illustrated in Fig. II-1-6. The results of computing ore reserve in each block by dividing each block into 20 m wide zones are shown in Table II-1-2. (The height in this ore reserve computation is 122.5 m between L551 and L428.5 m.) From these results, the object ore reserve is 18,724,090 tons and the average ore grade is 0.33%.

In this area to be mined, based on the results of investigating the integral layout, a mining width of 20 m and a pillar width of 20 m have been adopted, and mining stopes were arranged to make the grade of crude ores as high as possible. By so arranging stopes, it was found that the minable ore reserve of high grade blocks would be 7,378,070 tons and its grade would be 0.41%, i.e., the target grade in this plan, 0.4% or more, could be secured. However, if the operation at the rate of 3,500 tons/day is continued for this minable reserve, the reserve will be exhausted in seven years, and, if mining is carried out in the Block Central area in the three remaining years of the ten-year project, the mining of low-grade parts will be required, and the minable ore reserve of these parts is 1,981,560 tons and its grade 0.22% as shown in Table II–1–3.

1–2–3 Stope Development

The development is to prepare various kinds of drifts and chutes for the sublevel stoping. The main operation includes the development of scram drifts, blasthole drifts, communicating drifts, etc. In addition, service shafts, ore passes, slot raises, etc., are required. In this mining plan, a stope development plan is made taking the block A as model.

1) Dimensions of Stope Development

The dimensions of stope development are determined by the purpose of use and geological conditions. As Catavi Mine has good geological conditions, the dimensions of sections may be determined appropriate for the purpose. In this mining plan, main haulage drifts are made into those with double-line sections to make possible to load at ore pass draw points, and the ore passes have been planned to be as large as possible to prevent hang-ups in the chutes. The dimensions of connection drifts were determined to secure dimensions which can pass fan-cut drilling machines (ring drill crawlers), and for scram drifts, dimensions were determined to secure passage of loading and transporting equipment (hopper loaders). As the scram drifts tend to be loosened by the disposition of draw cones, etc., they were planned to be reinforced with rock bolts. About the timbering of drifts, as the rock is hard, drifts were planned to be those without timbering. If required

Table II–1–4 Dimensions of Stope Development

Kinds		Dimensions width x height (m)	Areas (m^2)	Level or chute	Note
Main haulage drift		5.0 x 3.0	14.75	Level	
Service shaft		2.5 x 2.0	5.0	Chute	
Ore pass		3.0 x 3.0	9.0	Chute	
Access		3.5 x 3.0	10.5	Level	
Scram drift		3.0 x 3.0	9.0	Level	Supported rock bolts
Blasthole drift		3.5 x 3.5	12.25	Level	
Slot raise		2.0 x 2.0	4.0	Chute	
Draw cone	Cross cut	4.0 x 3.0	12.0	Level	
	Raise	2.0 x 2.0	4.0	Chute	For widening of draw cone.
Ventilation shaft		2.0 x 2.0	4.0	Chute	

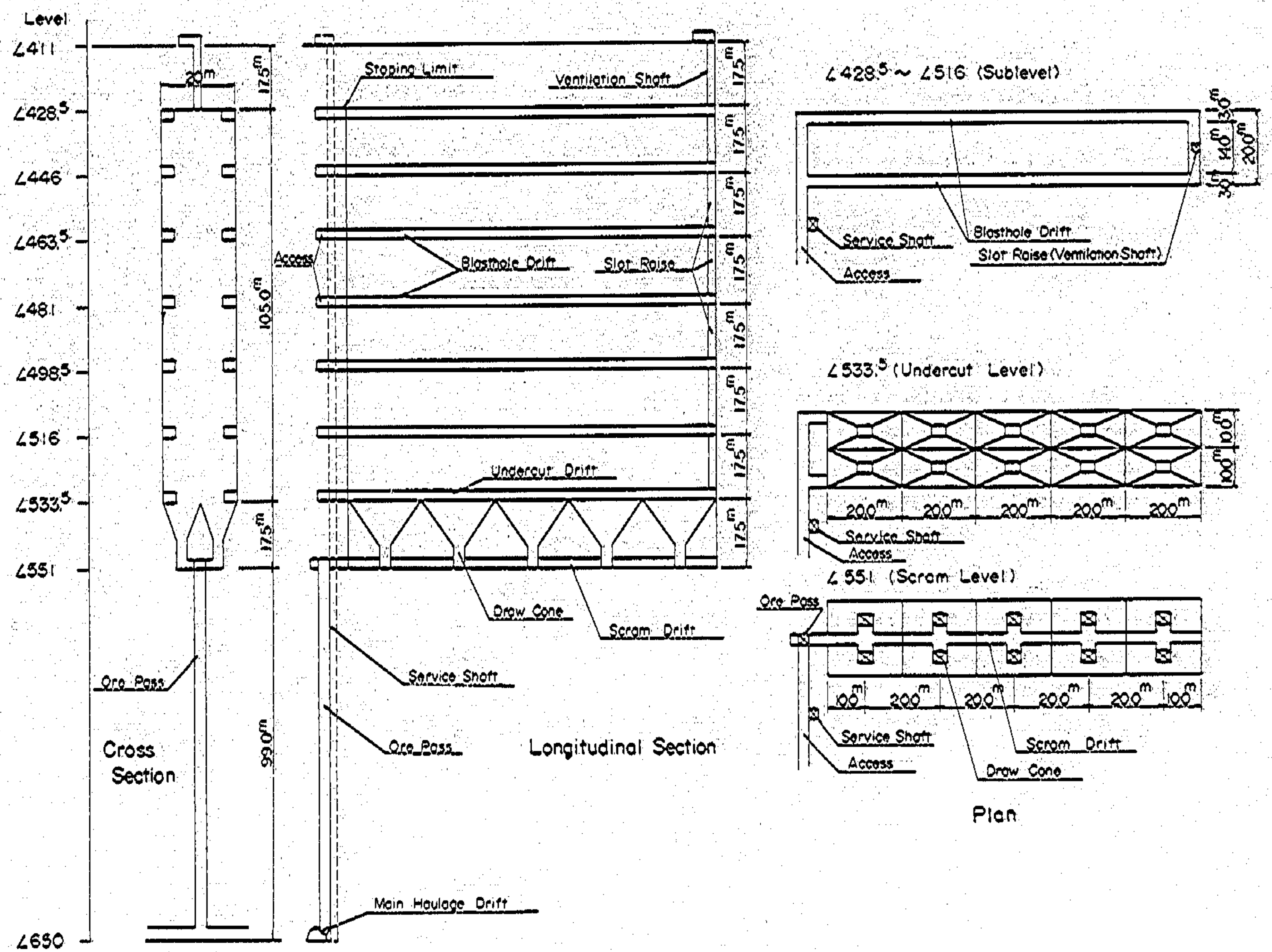


Fig. II-1-5 Plan and Section of Typical Sublevel Stopes

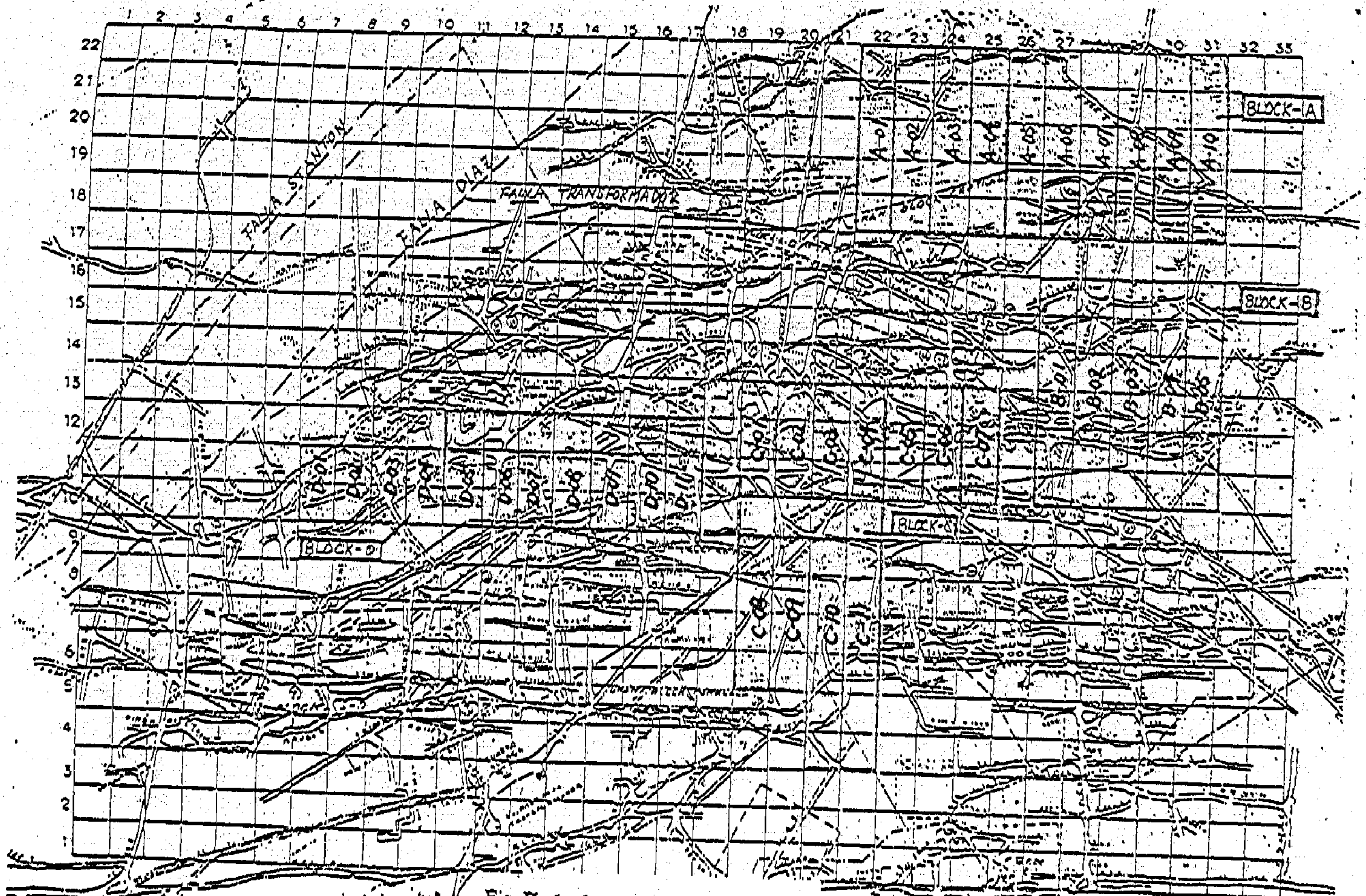


Fig. II–1–6 Arrangement of Ore Block

Table II-1-2 Ore Reserves and Ore Grade

Block-A	Min. Ton	Sn%	Fino Ton
A-01	497,700	0.34	1,692.180
A-02	557,270	0.46	2,563.442
A-03	593,390	0.33	1,958.187
A-04	592,340	0.38	2,250.892
A-05	611,240	0.28	1,711,472
A-06	631,680	0.35	2,210.880
A-07	612,080	0.33	2,019,864
A-08	614,460	0.45	2,765.070
A-09	596,120	0.48	2,861,376
A-10	384,790	0.59	2,270.261
Total	5,691,070	0.39	22,303.624

Block-B	Min. Ton	Sn%	Fino Ton
B-01	804,090	0.35	2,814.315
B-02	804,230	0.35	2,814.805
B-03	805,420	0.40	3,221,680
B-04	806,540	0.43	3,468.122
B-05	710,220	0.42	2,982.924
Total	3,930,500	0.39	15,301.846

Block-C	Min. Ton	Sn%	Fino Ton
C-01	402,920	0.42	1,692.264
C-02	400,750	0.27	1,082.025
C-03	399,210	0.17	678.657
C-04	402,080	0.36	1,447.488
C-05	402,430	0.37	1,488,991
C-06	404,320	0.35	1,415,120
C-07	403,970	0.49	1,979.453
C-08	342,230	0.16	547.568
C-09	533,820	0.24	1,281.168
C-10	800,590	0.23	1,841.357
C-11	800,870	0.24	1,922,088
Total	5,293,190	0.29	15,376.179

Block-D	Min. Ton	Sn%	Fino Ton
D-01	247,660	0.23	569.618
D-02	266,280	0.19	505.932
D-03	268,240	0.39	1,046.136
D-04	269,360	0.48	1,292.928
D-05	401,170	0.40	1,604,680
D-06	400,330	0.24	960.792
D-07	398,510	0.12	478.212
D-08	361,130	0.16	577.808
D-09	398,790	0.14	558.306
D-10	398,790	0.14	558.306
D-11	399,070	0.36	1,436.652
Total	3,809,330	0.25	9,589.370

Block No.	Min. Ton	Sn%	Fino Ton
Block-A	5,691,070	0.39	22,303.624
Block-B	3,930,500	0.39	15,301.846
Block-C	5,293,190	0.29	15,376.179
Block-D	3,809,330	0.25	9,589.370
Total	18,724,090	0.33	62,571.019

Table II–1–3 Minable Ore and Ore Grade

Block–A	Min. ton	Sn%	Fino ton
A–02	557,270	0.46	2,563.442
A–04	592,340	0.38	2,250.892
A–06	631,680	0.35	2,210.880
A–08	614,460	0.45	2,765.070
A–10	384,790	0.59	2,270,261
Total	2,780,540	0.43	12,060.545

Block–B	Min. ton	Sn%	Fino ton
B–01	804,090	0.35	2,814,315
B–03	805,420	0.40	3,221.680
B–05	710,220	0.42	2,982.924
Total	2,319,730	0.39	9,018.919

Block–C	Min. ton	Sn%	Fino ton
C–01	402,920	0.42	1,692.264
C–03	(399,210)	(0.17)	(678.651)
C–05	402,430	0.37	1,488.991
C–07	403,970	0.49	1,979.453
C–09	(533,820)	(0.24)	(1,281,168)
C–11	(800,870)	(0.24)	(1,922.088)
Total	1,209,320	0.43	5,160.708

Block–D	Min. ton	Sn%	Fino ton
D–01	(247,660)	(0.23)	(569,618)
D–03	268,240	0.39	1,046.136
D–05	401,170	0.40	1,604.680
D–07	–	–	–
D–09	–	–	–
D–11	399,070	0.36	1,436.652
Total	1,068,480	0.38	4,087,468

Note : ( ) shows the parts of low grade.

High grade

Total block	Min. ton	Sn%	Fino ton
Block–A	2,780,540	0.43	12,060.545
Block–B	2,319,730	0.39	9,018,919
Block–C	1,209,320	0.43	5,160.708
Block–D	1,068,480	0.38	4,087.468
Total	7,378,070	0.41	30,327,640

Low grade

Total block	Min. Ton	Sn%	Fino Ton
Block–C	1,733,900	0.22	3,881.913
Block–D	247,660	0.23	569.618
Total	1,981,560	0.22	4,451.531

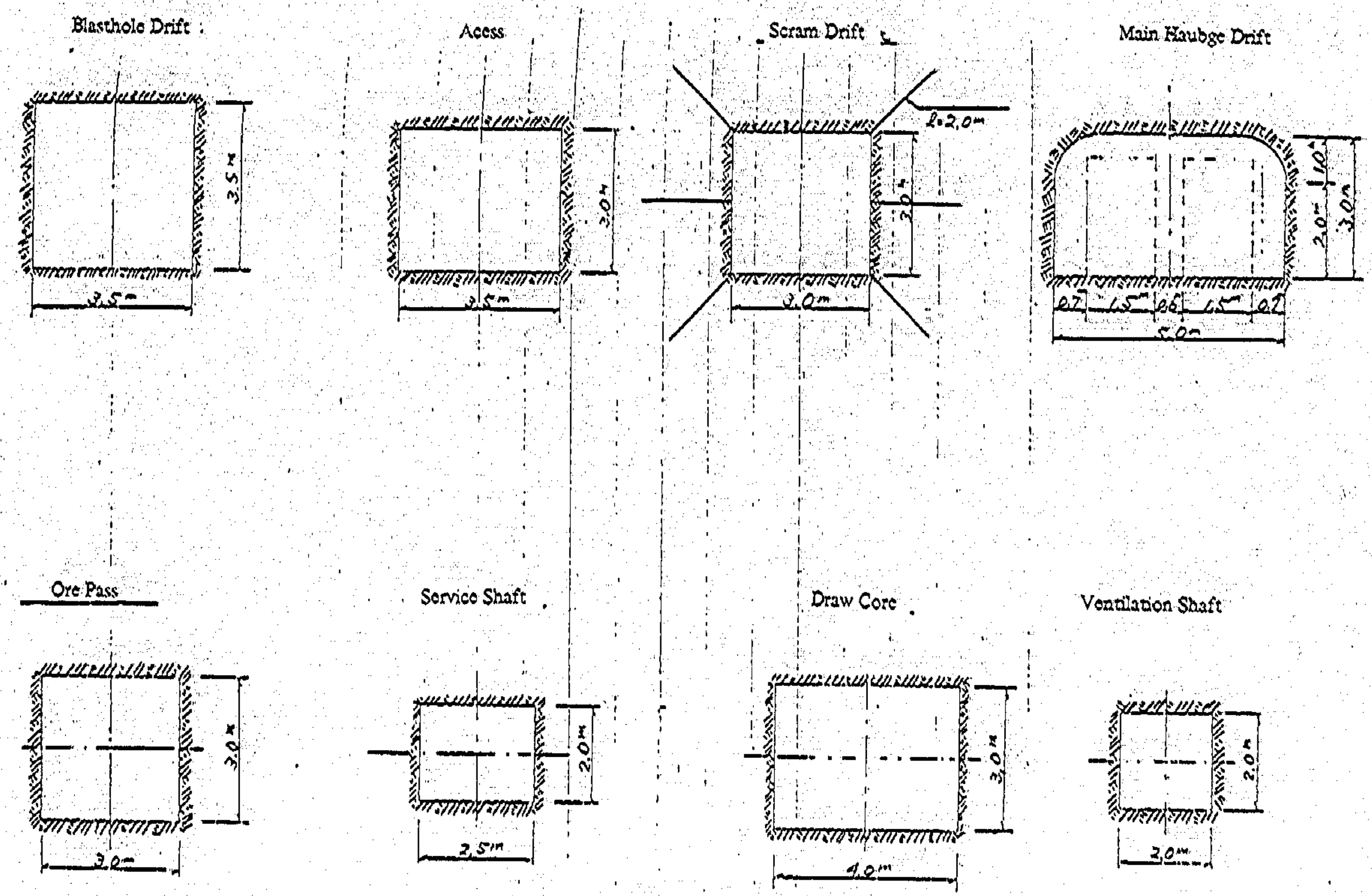


Fig. II–1–7 Sections of Drifts and Shafts

when actually developing, it is recommended to reinforce drifts with rock bolts, concrete, etc. Various drifts, shafts, etc., are as shown in Table II–1–4, and the standard sections of drifts and shafts are shown in Fig. II–1–7.

2) Development Work

Development in block A is as shown in Fig. II–1–8 ~ Fig. II–1–11. First, a haulage drift is provided on the main haulage level, L650, and from one side of this drift to the scraper level are cut chutes which will be used as ore passes, and they are provided with draw points at L650. In parallel to them, from the L650 haulage drift to the L411 level, chutes are made as service shafts for carrying in men and materials. From the service shafts horizontally are cut connection drifts so as to penetrate the ore passes, and grizzlies are made.

From here, scram drifts are dug horizontally, and on both sides of this drift crosscuts are made at 20 m intervals to be formed into the draw points of draw cones. In the center of this draw cone a raise is cut to the undercut level (L533.5) to be used as the free face of widening the draw cone.

On each sublevel, connection drifts are cut horizontally from the service shafts, and connected with them are drilled blasthole drifts. After cuts this drift, at a position where long-hole blasting is started, a slot (raise) is cut to the next level to form a free face required for the long-hole blasting. This slot is used also as a ventilation chute during development operation and mining.

The length of stope development for block A computed in the above plan is shown in Table II–1–5. The total length of development becomes 12,653 m. From Table II–1–3, the minable ore reserve in block A is 2,780,540 tons, the stope production per m of development resulted in 216 t/m. Compared with the records of Japanese mines, this value is a considerably better value, so that the plan this time can be said to be on a reasonable scale. The stope production per m of drift development becomes 266 t/m.

3) Equipment and Excavation Efficiency

Drifts in sublevel stoping are disposed in three dimensions except the main haulage drift, and for improving excavation efficiency, excavating equipment of high mobility is suitable.

As excavating and haulage equipment of high mobility, wheel-drive-type overshot loaders and LHD's (load-haul-dump's) were considered, and from the actual condition of Catavi Mine, the wheel-drive-type overshot loader was judged to be better and the hopper loader (ME 803D) was adopted.

In the main haulage drift, a rail system is planned for the haulage of mined ores, so a rail type overshot loader (RS 150, with a conveyor), a loading machine suitable for an excavation width of 5 m, will be used. The specifications of these machines are as follows.

Hopper Loader		Rail Type Overshot Loader	
Type	MB 803D	Type	RS 150
Bucket Volume	$0.28\ m^3$	Bucket Volume	$0.68\ m^3$
Hopper Capacity	$1.4\ m^3$	Total Weight	12,750 kg
Total Weight	5,200 kg	Working Air Pressure	$4.5 \sim 7.0\ kg/cm^2$
Working Air Pressure	$4.5 \sim 6.3\ kg/cm^2$	Air Consumption	$15 \sim 20\ m^3/min$
Air Consumption	$13\ m^3/min$	Total Height	2,700 mm
Total Height	2,225 mm	Maximum Scraping Width	5,500 mm
Total Length	2,910 mm	Rail Gauge	914 mm
Total Width	1,840 mm	Conveyer Belt Width	835 mm

In ore passes and service shafts, as they are long and people have experience in Catavi Mine, raise climbers will be used, and in this mining plan, Alimak Climber (STH–5E Type) was adopted. The specifications of this machine are as follows.

Alimak Climber	
Type	STH–5E
Maximum Platform Area	$7 m^2$
Lifting Speed	18 m/min
Lowering Speed	19 m/min
Motor	Electric Motor 10 PS
Aricab	5B (3 passengers)
Cable Reel	MKV–4 Type

As rock drills, existing machines will be used, and for the excavation of horizontal drifts, leg drills and stopers will be used. The results of computing the excavation efficiency of drifts and chutes on the above conditions are shown in Table II–1–6.

Assuming that three-shift system will be used here, the operation time per day was determined to be 900 minutes (15 hours).

The required number of main machines computed from development process becomes as follows.

(Hopper Loader)

Excavation efficiency per machine : 2.8 m/day (= 70 m/month)

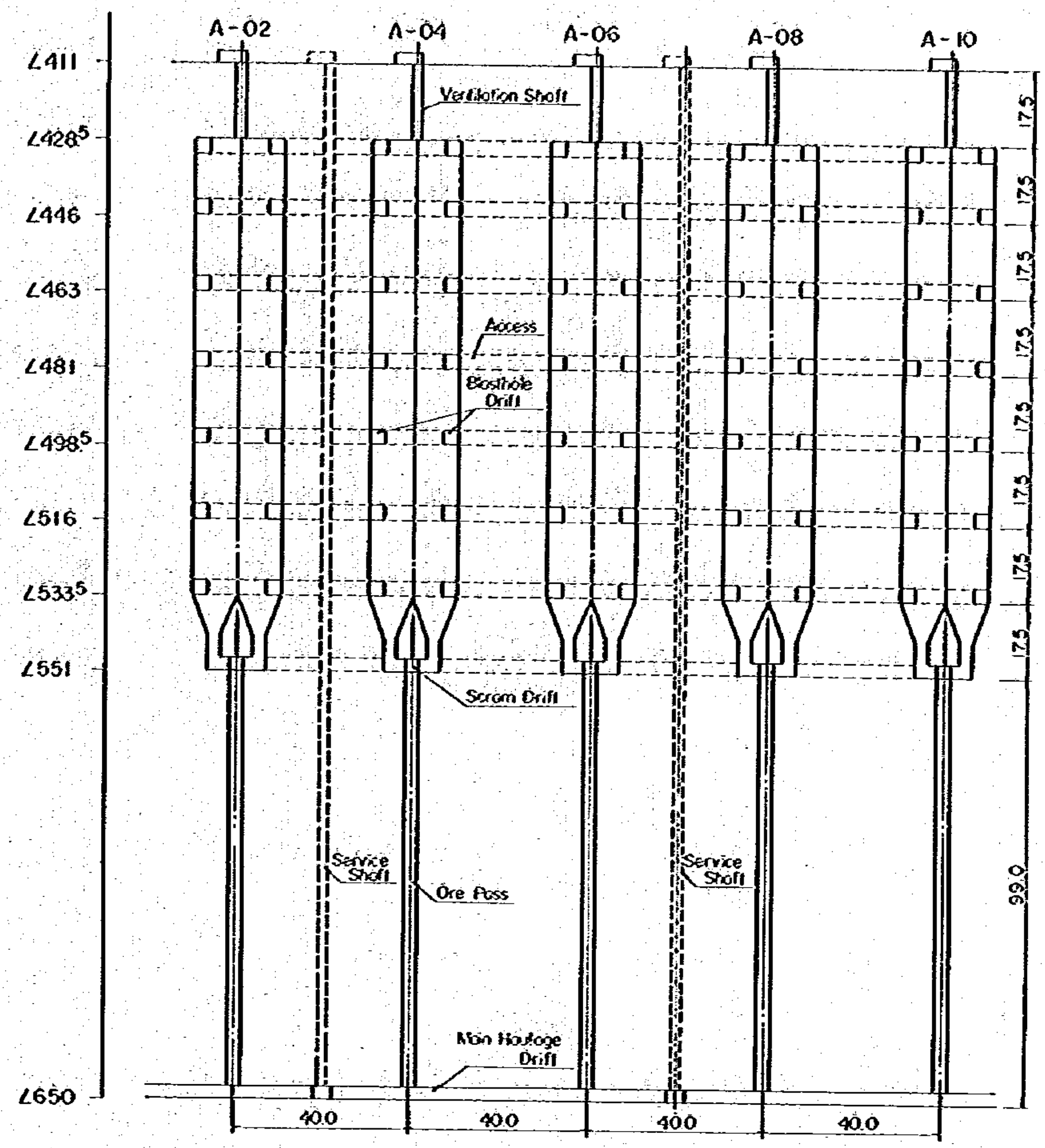


Fig. II-1-8 Section of Block-A
(Unit ; m)

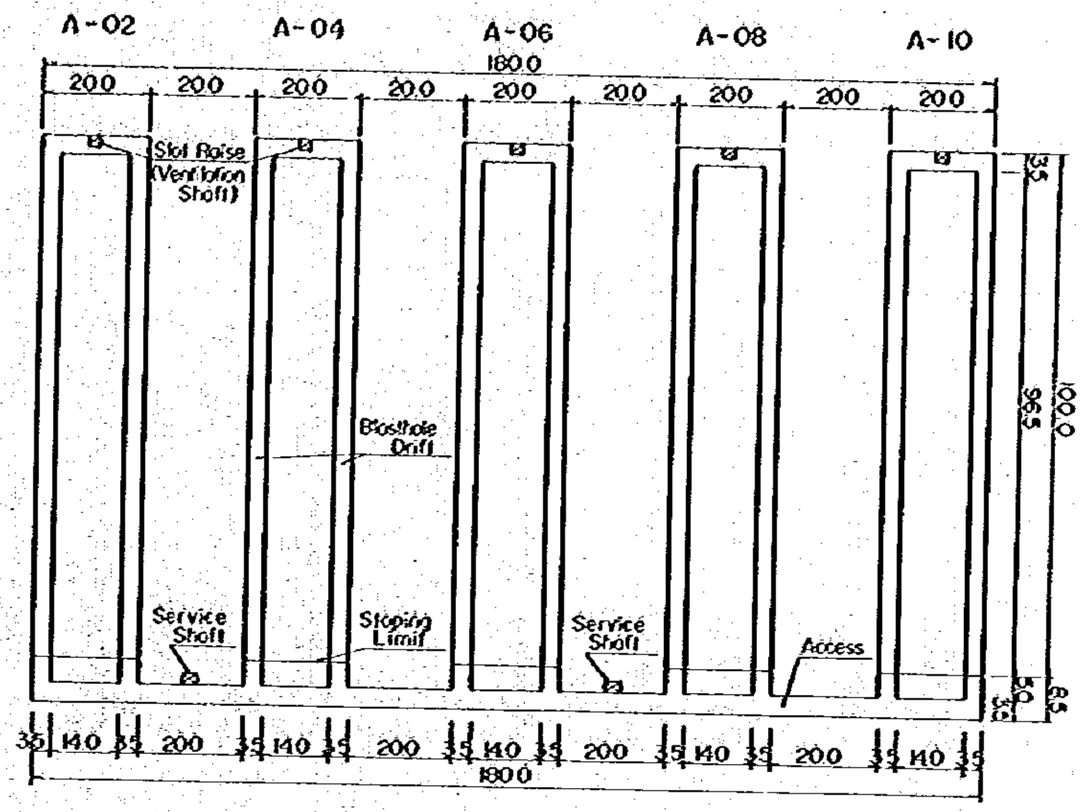


Fig. II-1-9 Plan of Sublevel for Block-A

(Unit ; m)

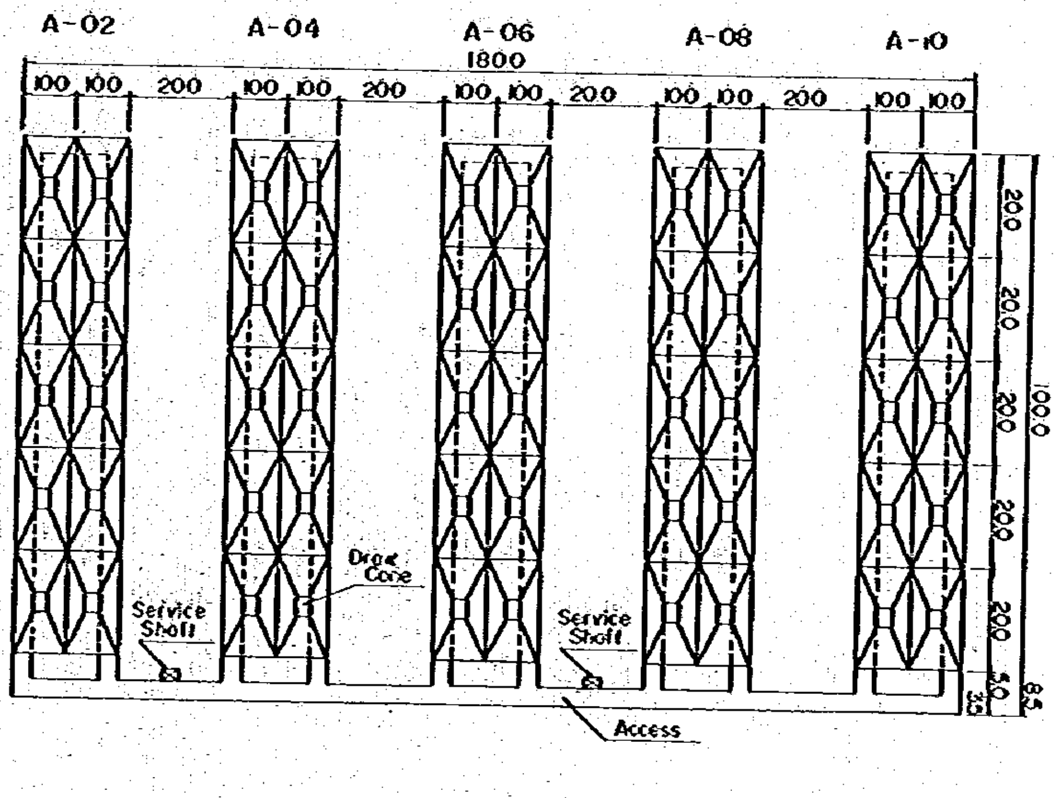


Fig. II-1-10 Plan of Undercut Level for Block-A

(Unit ; m)

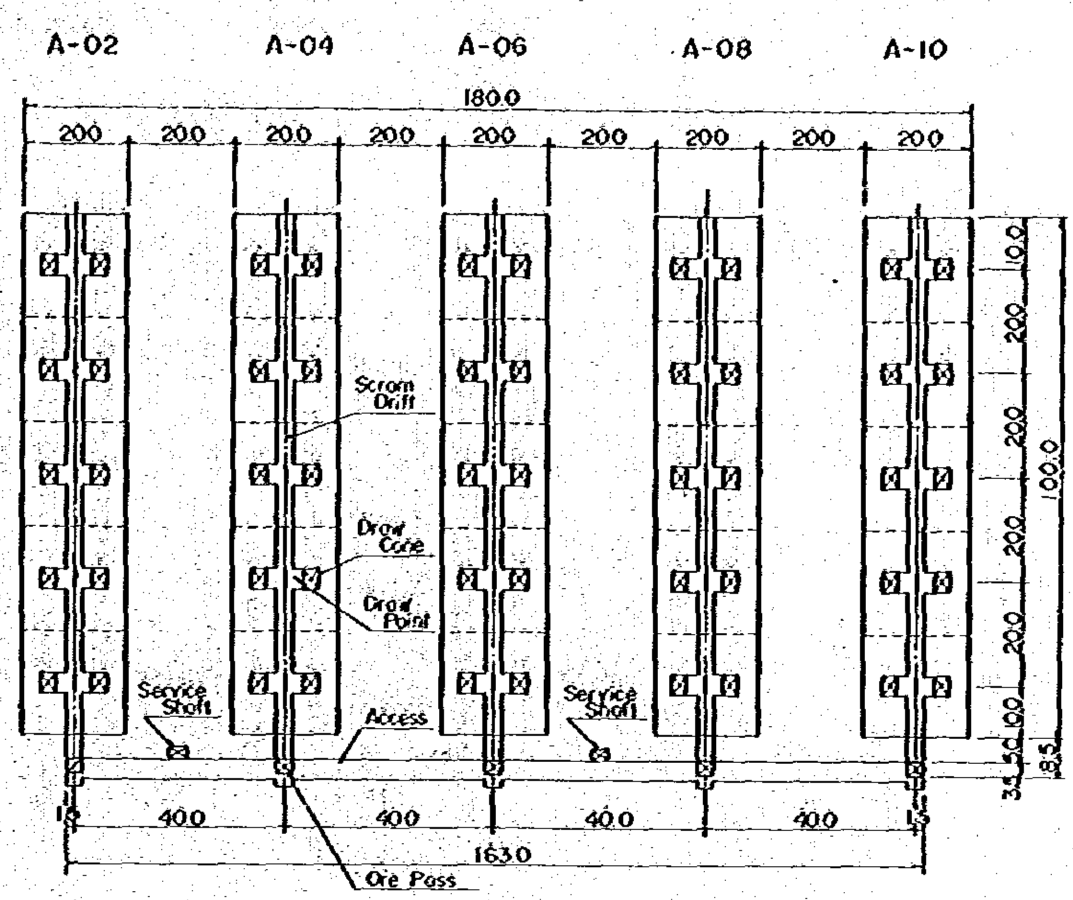


Fig. II-1-11 Plan of Scram Level for Block-A

(Unit; m)

Table II–1–5 Length of Stope Development for Block–A

Kinds	Location	Level or chute		Ore or waste		Total (m)
		Level (m)	Chute (m)	Ore (m)	Waste (m)	
Main haulage drift	L650	230			230	230
Service shaft	No.1		239		239	239
	No.2		239		239	239
L650~L411	Total		478		478	478
Ore Pass	A–02		99		99	99
	A–04		99		99	99
	A–06		99		99	99
	A–08		99		99	99
	A–10		99		99	99
	Total		495		495	495
Access	L551	160			160	160
	L533.5	180			180	180
	L516	180			180	180
	L498.5	180			180	180
	L481	180			180	180
	L463.5	180			180	180
	L446	180			180	180
	L428.5	180			180	180
	Total	1,420			1,420	1,420
Scram drift	A–02	110		100	10	110
	A–04	110		100	10	110
	A–06	110		100	10	110
	A–08	110		100	10	110
	A–10	110		100	10	110
	Total	550		500	50	550
Blasthole drift	A–02	1,568		1,498	70	1,568
	A–04	1,568		1,498	70	1,568
	A–06	1,568		1,498	70	1,568
	A–08	1,568		1,498	70	1,568
	A–10	1,568		1,498	70	1,568
	Total	7,840		7,490	350	7,840
Slot raise	A–02		80.5	80.5		80.5
	A–04		80.5	80.5		80.5
	A–06		80.5	80.5		80.5
	A–08		80.5	80.5		80.5
	A–10		80.5	80.5		80.5
	Total		402.5	402.5		402.5
Draw cone	A–02	50	180	230		230
	A–04	50	180	230		230
	A–06	50	180	230		230
	A–08	50	180	230		230
	A–10	50	180	230		230
	Total	250	900	1,150		1,150
Ventilation shaft L411~L428.5	A–02		17.5	17.5		17.5
	A–04		17.5	17.5		17.5
	A–06		17.5	17.5		17.5
	A–08		17.5	17.5		17.5
	A–10		17.5	17.5		17.5
	Total		87.5	87.5		87.5
Total		10,290	2,363	9,630	3,023	12,653

Table II–1–6 Excavation Efficiency of Drifts and Chutes

Kinds		Blasthole drift	Access	Scram drift	Ore pass	Service shaft	Ventilation shaft	Main haulage drift
Excavating areas (m^2)		12.25	10.5	9.0	9.0	5.0	4.0	14.57
Length of round (m)		1.5	1.5	1.5	1.2	1.2	1.2	1.5
Excavating volumes (m^3)		18.4	15.75	13.5	10.8	6.0	4.8	21.86
Drilling efficiency (m/min)		0.3	0.3	0.3	0.2	0.2	0.2	0.3
Mucking efficiency (m^3/min)		0.09	0.09	0.09	–	–	–	0.16
Drilling time	Preparation (min)	20	20	20	50	50	30	20
	Drilling (″)	98	84	72	175	195	156	116
	Charge and blasting (″)	35	35	35	60	60	50	35
	Ventilating (″)	30	30	30	30	30	30	30
	Taking off fragmented rock (″)	20	20	20	30	30	30	20
	Sub total (″)	203	189	177	345	365	296	221
Mucking time	Preparation (″)	10	10	10	–	–	–	10
	Mucking (″)	204	175	150	–	–	–	136
	Removing (″)	10	10	10	–	–	–	10
	Extending air and water (″)	20	20	20	–	–	–	20
	Sub total (″)	244	215	190	–	–	–	176
Timbering time	Preparation (″)	–	–	10	60	180	60	–
	Timbering (″)	–	–	80	–	–	–	–
	Sub total (″)	–	–	90	60	180	60	–
Loss time (″)		30	30	30	60	60	60	30
Total time (″)		477	434	487	465	605	416	427
Working time per day (min)		900	900	900	900	900	900	900
Number of cycle		1.89	2.07	1.85	1.94	1.49	2.16	2.11
Advancing meter per day		2.84	3.11	2.78	2.33	1.79	2.59	3.17
Advancing meter per month		71.0	77.8	69.5	58.3	44.8	64.8	79.3

Table II-1-7 Development Schedule for Block – A

Kinds of Working	Length	Efficiency (Metre per month)	Required month (month)
∠650 Main Haulage Drift	230 m	79.3	2.9
∠650~∠411			
Service Shaft No.1	239 m	58.3	4.6
No.2	239 m	58.3	4.6 (Included preparation)
∠650~∠551			(Included preparation)
Ore Pass A – 02	99 m	58.3	2.2
A – 04	99 m	58.3	2.2
A – 06	99 m	58.3	2.2
A – 08	99 m	58.3	2.2
A – 10	99 m	58.3	2.2
Access ∠551	160 m	77.8	2.1
∠533^5	180 m	77.8	2.3
∠516	180 m	77.8	2.3
∠498^5	180 m	77.8	2.3
∠481	180 m	77.8	2.3
∠463^5	180 m	77.8	2.3
∠446	180 m	77.8	2.3
∠428^5	180 m	77.8	2.3
Advancing			
A – 02			7.0
A – 04			7.0
A – 06			7.0
A – 08			7.0
A – 10			7.0

1st Year 5 10 2nd Year 15 20 25 3rd Year 30 35 4th Year 40 45 48

Schedule of Advancing per One Stope

Kinds of Working		Length	Efficiency	Required month (month)
Scram Drift		11.0 m	69.5	1.6
Blasthole Drift		156.8 m	71 m/month×4 ≒284 m/month	5.5
Slot Raise		80.5 m	64.8 m/month	1.2
Draw Cone	Crosscut	5.0 m	69.5 m/month	0.7
	Raise	18.0 m	64.8 m/month	2.8
Ventilation Shaft		17.5 m	64.8 m/month	0.3

1 2 3 4 5 6 7 8 9 (month)

Stope production per m of horizontal drift : 266 t/m

Stope production per day : 3,500 t/day

The required number of machines : 3,500 t/day ÷ 266 t/m ÷ 2.8 m/day

= 4.7 ÷ 5

By adding two spare machines, Total 7 machines

(Rail Type Overshot Loader)

The required number of machine : 1

(Alimak Climber)

The required number of machines : 2

4) Stope Development Schedule

The stope development schedule was planned as shown in Table II–1–7.

Prerequisites of this schedule are :

(a) Number of operation days per month is 25 days, i.e., the number of operation days per year is 300 days.

(b) Three-shift system will be used, so that operation time per day is 900 minutes.

If this schedule is observed, it takes about four years to complete the entire development of block A, but if mining is begun with stopes whose development are finished, stope production can be started in the latter half of the third year or in the fourth year.

1–2–4 Stoping

After the stope development work is completed, the sublevel stoping is begun with the widening of draw cones into cone shapes by the undercutting of the undercut drift.

Next, for the purpose of forming free faces for long-hole fan drilling and blasting, widening blasting around slots is carried out. After that, long-hole fan drilling is carried out and ores are broken by long-hole blasting and mined.

(1) Drilling and Blasting

1) Long-hole Fan Drilling and Blasting

The long-hole fan drilling and blasting is most important in sublevel stoping and also has a great influence on other operations. The extent of ore breakage is a matter which is thought to be especially important in this drilling-and-blasting. This broken size not only directly influences ore handling but also gives greatly affects mining costs. When designing the long-hole fan drilling and blasting, it must be designed so that the breakage of ores may produce appropriate size ores for ore handling.

(a) Borehole diameter

The borehole diameter is closely related with ring burden and hole spacing, and in addition, is also related with the blasting efficiency of explosives, blasting scales and drilling rates, so that drill bit diameters must be selected taking the scale of production, minable ore reserve, drilling costs, etc., into account.

The borehole diameter (bit diameter) is determined to be 65 mm after considering of the size of the heavy drifter and drilling lengths.

(b) Ring burden

In the case of drilling and blasting using crawler drills, the ring burden is usually 2.5 ~ 3.0 m for the borehole diameter of 65 mm. By a method which determines a ring burden based on a borehole diameter, the standard burden is determined to be 40 ~ 45 times the borehole diameter.

In this sublevel stoping plan, the ring burden is determined to be 3.0 m in consideration of avoiding overcharging, reduction of powder factor, etc.

(c) Borehole spacing

The borehole spacing is related with the ring burden, and the dimension of the spacing is related with broken sizes. Normally, the standard interval is said to be 1.25 times the ring burden, but usually, intervals of 0.8 ~ 1.4 times the ring burden are used.

In this sublevel stoping plan, the hole spacing is determined to be 3.5 m.

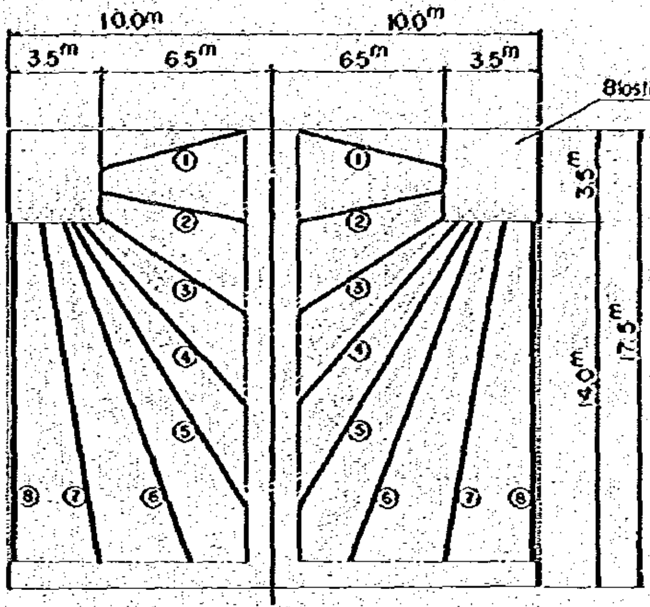
The pattern of long-hole fan drilling is shown in Fig. II−1−12.

As explosives, low-priced AN-FO will be used and dynamite will be used as primer cartridge. Powerder factor is computed as shown in Table II−1−8.

Table II−1−8 Powder Factor of Long-Hole Blasting

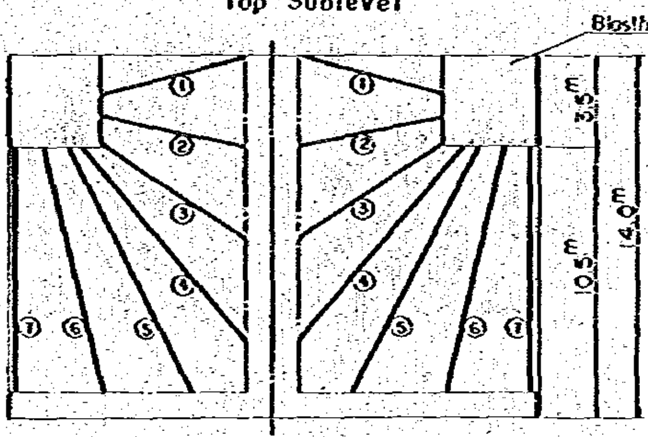
Items	Middle sublevel	Top sublevel
Total drilling length (m)	160.4	114.6
Borehole diameter (D) (m)	0.065	0.065
Loading rate (α)	0.6	0.6
Loading density (ρ) kg/m^3	850	850
Amount of charge $L = \frac{\pi D^2}{4} \times \ell \times \alpha \times \rho$ kg	271.4	193.9
Ore brocken per slice round (ton)	2,675.4	2,100.0
Powder factor (kg/ton)	0.101	0.092
Ore brocken per meter of drilling (ton/m)	16.7	18.3

Middle Sublevel



Borehole No.	Hole Depth(m)	Hole	Total(m)
①	5.7	2	11.4
②	5.6	2	11.2
③	6.6	2	13.2
④	9.4	2	18.8
⑤	12.8	2	25.6
⑥	14.0	2	28.0
⑦	13.1	2	26.2
⑧	13.0	2	26.0
Total			160.4m

Top Sublevel



Borehole No.	Hole Depth(m)	Hole	Total(m)
①	5.7	2	11.4
②	5.6	2	11.2
③	6.6	2	13.2
④	9.7	2	19.4
⑤	10.6	2	21.2
⑥	9.6	2	19.2
⑦	9.5	2	19.0
Total			114.6m

Fig. II-1-12 Long-Hole Fan Drilling Pattern

(Unit ; m)

The powder factor is about 0.1 kg/t, a comparatively high efficiency value compared with the results of Japanese mines.

2) Slot Blasting

The slot-blasting is carried out by drilling with a ring drill crawler used for fan drilling to form a slot into a free face and slot blasting with a width of 3.5 m. The disposition of drill holes for this slot blasting is shown in Fig. II–1–13.

The powder factor is as shown in Table II–1–9.

Table II–1–9 Powder Factor of Slot Blasting

Items	Middle sublevel	Top sublevel
Total drilling length (ℓ) (m)	351	256.5
Borehole diameter (D) (m)	0.065	0.065
Loading rate (α)	0.6	0.6
Loading density (ρ) (kg/m³)	850	850
Amount of charge $L=\frac{\pi D^2}{4} \times \ell \times \alpha \times \rho$ kg	594.0	434.1
Ore brocken per slice round (ton)	2,531.6	1,698.8
Powder factor (kg/ton)	0.235	0.229
Ore broken per meter of drilling (ton/m)	7.2	7.4

3) Undercutting Round in Undercut Drift

The undercutting round in the undercut drift is carried out at every 20 m intervals in accordance with the development of mining stopes. By drilling with the crawler drill, undercutting round is carried out at every 20 m intervals. The disposition of drill holes for this undercutting round is as shown in Fig. II–1–14. As explosives, ANFO and dynamite will be used and the powder factor will become as shown in Table II–1–10.

Table II–1–10 Powder Factor of Undercutting Round

Items	Value
Total drilling length (ℓ) (m)	241.5
Borehole diameter (D) (m)	0.065
Loading rate (α)	0.6
Loading density (ρ) (kg/m³)	1,000
Amount of charge $L=\frac{\pi D^2}{4} \times \ell \times \alpha \times \ell$(kg)	450.8
Ore brocken per slice round (ton)	2,494
Powder factor (kg/ton)	0.193
Ore brocken per meter of drilling (ton/m)	10.3

4) Widening Blasting of Draw Cone

The widening blasting of a draw cones is carried out after the undercutting round of the undercut drift, and using a raise as a free face, an area 10.0 mW x 20.0 mL is widened by blasting into a cone shape. The disposition of drill holes for this widening blasting is as shown in Fig. II–1–15, and the powder factor is as shown in Table II–1–11.

Table II–1–11 Powder Factor of Widening Draw Cone

Items	the Value
Total drilling length (ℓ) (m)	234.5
Borehole diameter (D) (m)	0.065
Loading rate (α)	0.6
Loading density (ρ) (kg/m³)	850
Amount of charge $L=\frac{\pi D^2}{4}\times\ell\times\alpha\times\rho$ (kg)	396.9
Ore brocken per slice round (ton)	2,270.1
Powder factor (kg/ton)	0.175
Ore brocken per meter of drilling (ton/m)	9.7

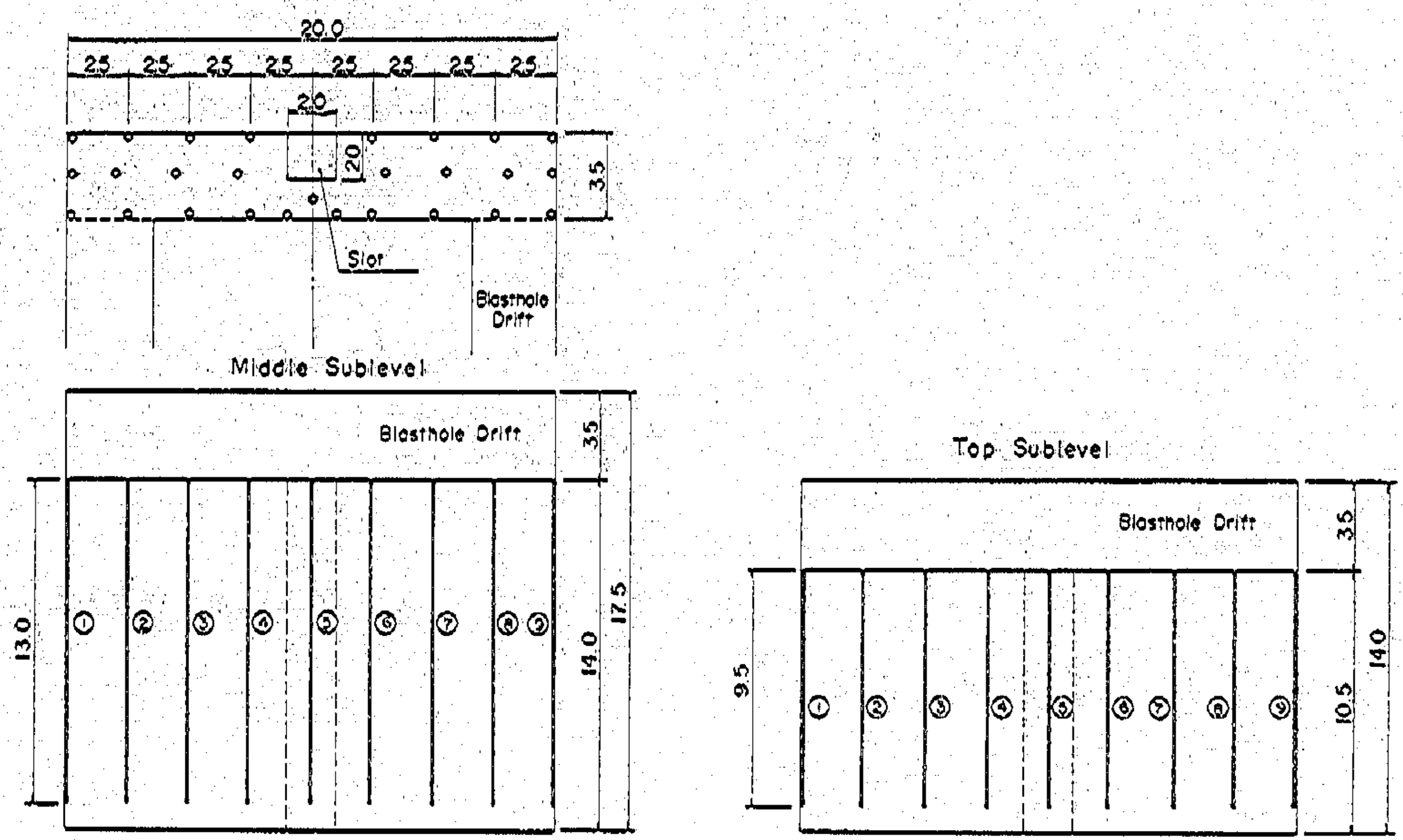
The drilling and blasting plan in the sublevel stoping plan has been described in the above. If total drilling length per one stope is determined from ore production per one stope in block A (Table II–1–12) is determined, it is as shown in Table II–1–13.

As this mining stope is in an old stoping block, boring for checking old working is required, and including such boring length, the average ore production per meter of drilling length was determined to be 15.2 t/m.

(2) Ore Handling

Ores broken by blasting flow out from draw cones, are scraped by the scraper hoist and dumped into ore chutes. The entrance of this chute is provided with a grizzly, and large blocks remaining on the grizzly are broken by secondary blasting. The size of the grizzly will be determined to be that which will not hang up blocks in ore passes and will be arranged into paralkel crosses of 0.6 m x 0.6 m.

Draw points will considerably be loosened by the falling down and flowing out of ores, so that they must be reinforced previously. The section of a draw point reinforced by driving in rock bolts allover and lining with concrete is as shown in Fig. II–1–16.

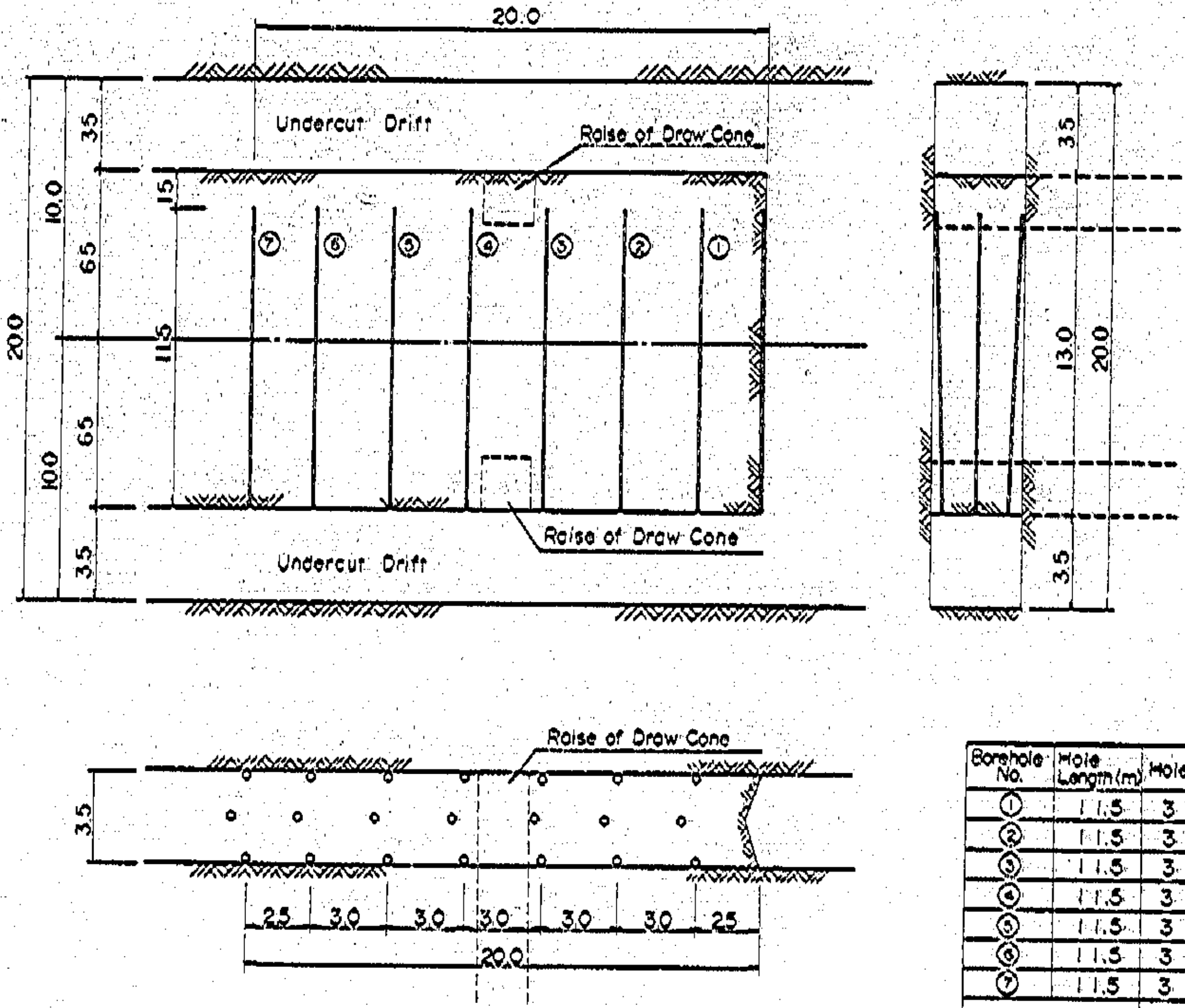


Borehole No.	Hole Length (m)	Hole	Total (m)	Borehole No.	Hole Length (m)	Hole	Total (m)
①	13.0	3	39.0	6	13.0	3	39.0
②	13.0	3	39.0	7	13.0	3	39.0
③	13.0	3	39.0	8	13.0	3	39.0
④	13.0	3	39.0	9	13.0	3	39.0
⑤	13.0	3	39.0	Total			351 m

Borehole No.	Hole Length (m)	Hole	Total (m)	Borehole No.	Hole Length (m)	Hole	Total (m)
①	9.5	3	28.5	6	9.5	3	28.5
②	9.5	3	28.5	7	9.5	3	28.5
③	9.5	3	28.5	8	9.5	3	28.5
④	9.5	3	28.5	9	9.5	3	28.5
⑤	9.5	3	28.5	Total			256.5 m

Fig. II-1-13 Drilling Pattern of Slot Blasting

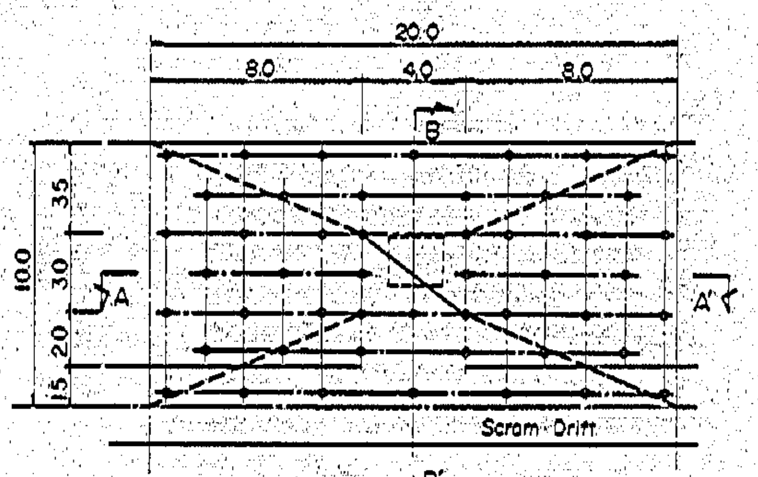
(Unit ; m)



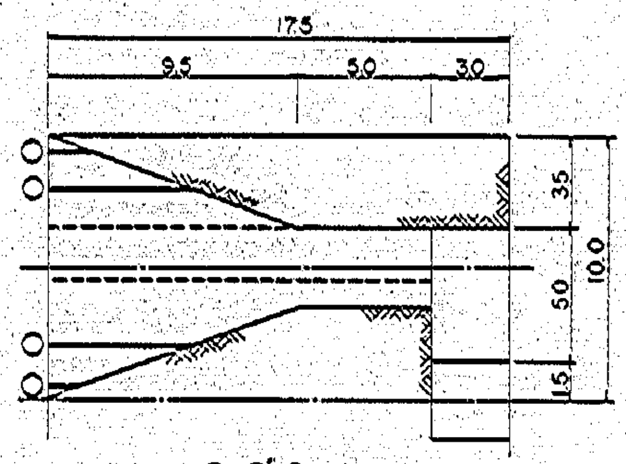
Borehole No.	Hole Length (m)	Hole	Total (m)
①	11.5	3	34.5
②	11.5	3	34.5
③	11.5	3	34.5
④	11.5	3	34.5
⑤	11.5	3	34.5
⑥	11.5	3	34.5
⑦	11.5	3	34.5
Total			241.5

Fig. II-1-14 Drilling Pattern of Undercutting Round

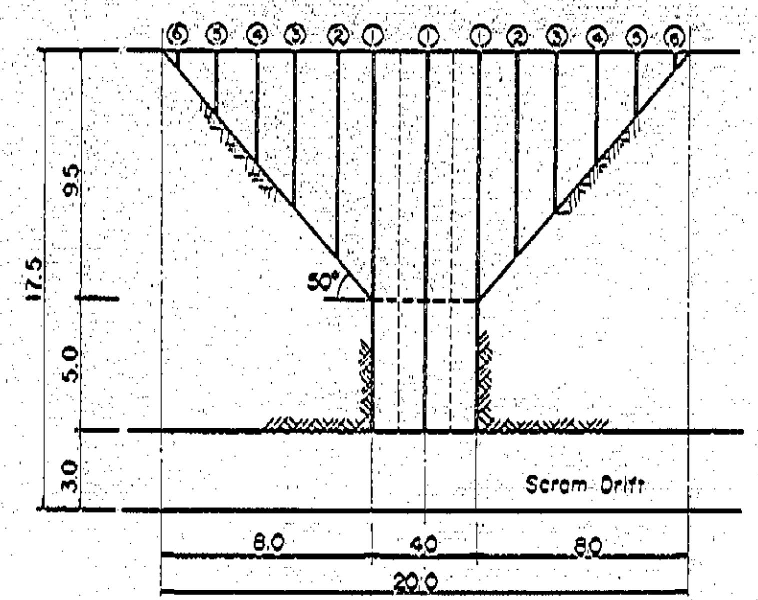
(Unit ; m)



Plan



B-B' Section



A-A' Section

Borehole NO.	Hole Depth (m)	Hole	Total (m)
1	14.5	7	101.5
2	7.7	4	30.8
3	6.0	2	12.0
4	4.2	4	16.8
5	2.3	6	13.8
6	0.7	8	5.6
7	5.5	8	44.0
8	1.0	10	10.0
	Total	49	234.5m

Fig.II-1-15 Drilling Pattern of Draw Cone

(Unit ; m)

Table II-1-12 Ore Production per One Stope

Kinds	Location	Ore production per unit	Value	Ore production (ton)
Long-hole Fan drilling	L533.5 ~ L516	891.8 t/m	96.5 m	86,058.7
	L516 ~ L498.5	891.8 "	96.5 "	86,058.7
	L498.5 ~ L481	891.8 "	96.5 "	86,058.7
	L481 ~ L463.5	891.8 "	96.5 "	86,058.7
	L463.5 ~ L446	891.8 "	96.5 "	86,058.7
	L446 ~ L428.5	700.0 "	96.5 "	67,550
	Sub Total			497,843.5
Undercut drilling	L533.5	124.7 t/m	96.5 m	12,033.6
Slot drilling	L533.5 ~ L516	2,531.6 t/unit	1 unit	2,531.6
	L516 ~ L498.5	2,531.6 "	1 "	2,531.6
	L498.5 ~ L481	2,531.6 "	1 "	2,531.6
	L481 ~ L463.5	2,531.6 "	1 "	2,531.6
	L463.5 ~ L446	2,531.6 "	1 "	2,531.6
	L446 ~ L428.5	1,898.8 "	1 "	1,898.8
	Sub Total			14,556.8
Draw cone drilling	L551 ~ L533.5	2,2701.1 Ton/unit	10 units	22,701.0
Total				547,134.9

Table II-1-13 Drilling Length per One Stope

Kinds	Location	Ore production per meter drilling (t/m)	Ore production (t)	Drilling length (m)
Long-hole Fan drilling	L533.5 ~ L516	16.7	86,058.7	5,153.2
	L516 ~ L498.5	16.7	86,058.7	5,153.2
	L498.5 ~ L481	16.7	86,058.7	5,153.2
	L481 ~ L463.5	16.7	86,058.7	5,153.2
	L463.5 ~ L446	16.7	86,058.7	5,153.2
	L446 ~ L428.5	18.3	67,550	3,691.3
				29,457.3
Undercut drilling	L533.5	10.3	12,033.6	1,168.3
Slot drilling	L533.5 ~ L516	7.2	2,531.6	351.6
	L516 ~ L498.5	7.2	2,531.6	351.6
	L498.5 ~ L481	7.2	2,531.6	351.6
	L481 ~ L463.5	7.2	2,531.6	351.6
	L463.5 ~ L446	7.2	2,531.6	351.6
	L446 ~ L428.5	7.4	1,898.8	256.6
				2,014.6
Draw cone drilling	L551 ~ L533.5	9.7	22,701.0	2,340.3
Boring	everywhere	15 m hole	70 holes	1,050.0
Total		Average value 15.2 t/m	547,134.9	36,030.5

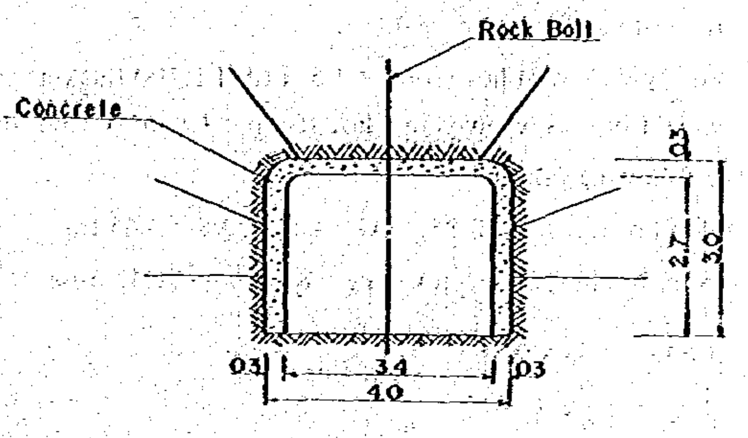


Fig. II-1-16 Section of Draw Point (Unit : m)

(3) Required Number of Stoping Equipment

1) Long-Hole Drilling Machine

As the long-hole drilling machines crawler drills (Toyo Kogyo, CJ-641), will be used as special purpose machines for the operation. If the drilling operation is carried out in a three-shift system, actual drilling hours per day becomes 15 hours (five hours per shift), so that a drilling rate secured by one crawler drill is estimated to be 60 m/unit/day. Accordingly, the number of crawler drills required for securing a daily crude ore production of 3,500 t/day is,

3,500 t/day ÷ 15.2 t/m ÷ 60 m/unit day = 3.84 ≑ 4 units

In this mining plan, by adding two reserve machines, the required number of the crawler drills is determined to be six.

2) Ore Handling Machine

For the ore handling machine, a 75 kw electric scraper hoist (Akimoku Steel Works, 2 DS-100M Type) was selected. Its handling capacity is calculated as follows:

[Handling capacity per one scraper]

$$Q = \frac{3{,}600 \times G \times \varphi}{\frac{L}{v_1} + \frac{L}{v_2} + t}$$

where : Q : handling capacity per one scraping (reciprocation) (t/hr)

G : scraping weight per time (t) $(G = G_0 + G_1)$

L : handling distance (m)

φ : scraping factor = 0.8 ~ 0.9

v_1, v_2 = pulling and returning speeds of scraper (m/sec)

t = time of pause per time = 40 sec

G_0 = empty weight of hoe scoop = 1.5 t (AM 1,400 Bucket)

G_1 = weight of ores received in a hoe scoop = 1.8 m^3 x 1.64 ton/m^3 = 2.95 tons

In the case of the following values :

L = 80 m (average), φ = 0.85, G = 1.5 + 2.95 = 4.45 ton

v_1 = 75 m/min (= 1.25 m/sec), v_2 = 85 m/min (1.42 m/sec),

$$Q = \frac{3{,}600 \times 4.45 \times 0.85}{\frac{80}{1.25} + \frac{80}{1.42} + 40} = 84.9 \ \text{t/hr}$$

If the ore handling operation is carried out in a three-shift system, actual operation hours per day becomes 18 hours (6 hours/shift). As a certain extent of time is required for secondary blasting, we estimate the operation time to be 3 hour/day, then actual ore handling time per day becomes 15 hours. Accordingly, the number of scraper hoists required for securing a stoping production of 3,500 t/day is.

3,500 t/day ÷ 15 hr/day ÷ 84.9 t/hr. unit = 2.75 ≒ 3

In this mining plan, the number of machines is determined to be 4 considering the rate of operation. The specifications of the electric scraper hoist are as follows.

Item		Value
Type		2DS = 100M
Drum Dimensions		500 $^\phi$ x 850 x 260 mm
Rope Load		6,100 kg
Rope Speeds	Pull	75 m/min
	Return	85 m/min
Drum Capacity		22 $^\phi$ x 170 m
Output		75 kw (100 IP)
Voltage Used		220 / 440 V
Hoe Scoop	Volume	1.8 m^3
	Weight	1,500 kg