FEASIBILITY STUDY

FOR

THE OMBILIN COAL MINE DEVELOPMENT PROJECT

IN THE REPUBLIC OF INDONESIA

JUNE 1981

Japan International Cooperation Agency

P R E F A C E

In response to a request of the Government of the Republic of Indonesia, the Japanese Government decided to conduct a feasibility study on the Sawah Lunto Coal Development Project and entrusted the work to the Japan International Cooperation Agency (J.I.C.A.).

The J.I.C.A. sent to Indonesia a survey team headed by Mr. E. Kawai of the Sumitomo Coal Mining Company from 22nd July to 10th August, 1980.

The team had discussions with the officials concerned of the Government of Indonesia and conducted a field survey. After the team returned to Japan, further studies were made and the present report has been prepared.

I hope that this report will serve for the development of the Project and contribute to the promotion of friendly relations between our two countries.

I wish to express my deep appreciation to the officials concerned of the Government of the Republic of Indonesia and of the related organizations for their close cooperation extended to the team.

July, 1981

Keisuke Arita
President
Japan International Cooperation Agency

ACKNOWLEDGMENTS

The Government of Japan, in respons to the request extended by the Government of Republic of Indonesia has decided to make a survey for the rehabilitation of Ombilin Coal Mine in West Sumatra, Indonesia. The implementation is commissioned to the Japan International Cooperation Agency.

The survey was commenced in January 1978 and the geological exploration part of the survey corresponding to CTA-79, continued to CTA-115 of BAPPENAS's blue book code number was completed in June 1980.

The work was carried out jointly by the Japanese team and Indonesian conterparts : The Directorate of Mineral Resources and PN Tambang Batubara, with the cooperation of the Ministry of Communication and Tourism.

At the conclusion of the project, a feasibility study on the development of the exploration area including railway transportation, coal storage and shiploading and the economic evaluation was made and hereby submitted.

We wish to take this opportunity to express our gratitute to all members concerned with the preparation of the study.

May 1981

Prof. Dr. J.A. Katili
Director General
Ministry of Mines and Energy
Republic of Indonesia

LETTER OF TRANSMITTAL

Mr. Keisuke Arita, President
Japan International Cooperation Agency

Dear Sir:

I have a great pleasure to submit herewith the report in consequence of our feasibility study following the previous report on geological exploration, both of which aim at a rehabilitation of the Ombilin Coal Mine, West Sumatra, Republic of Indonesia.

The survey team, headed by Eiichi Kawai, Sumitomo Coal Mining Co. Ltd., consisted of ten specialists in mining, railways, harbor facilities and economics. The field study was commenced by them in July, 1980, and has been completed after its results were analyzed in detail in Japan and were explained to the Indonesian personnel concerned.

The report describes the plan for development of coal mine and for expansion of infrastructure equipments accompaning the increased coal production, i.e. railway transportation, coal storage and shiploading. The economic evaluation as a whole based on the above plans is also presented in the report.

In consideration of the energy situation in Indonesia at present, the coal development appears to be urgently required. Thus, the expectation on Indonesian side toward the rehabilitation of the Ombilin Coal Mine seems to be hightened. The survey team eagerly expects that the report would be of good service to them for promoting future coal exploration.

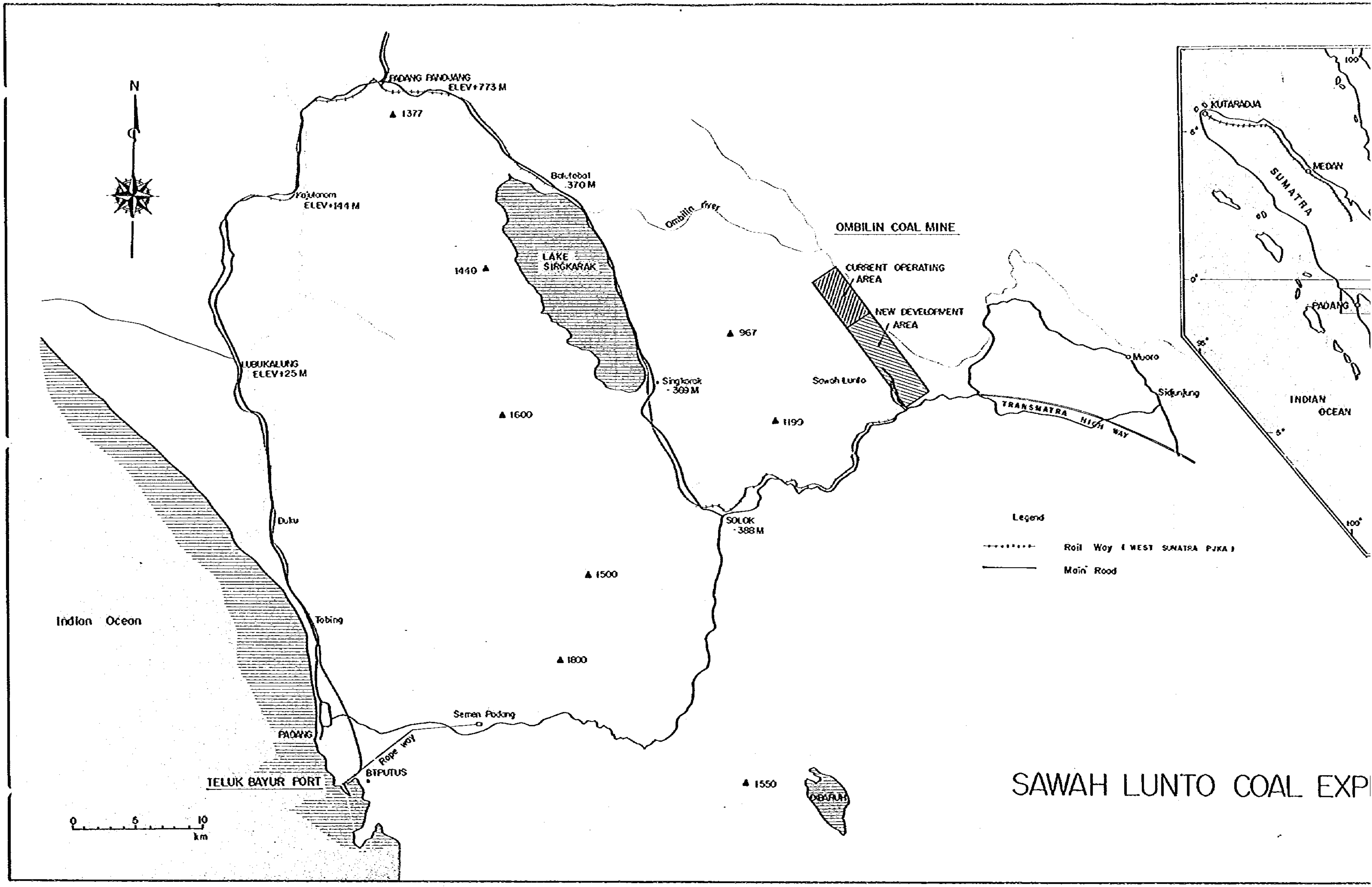
On submitting the report, I would like to express our sincere gratitude to all those personnel in the Government of Indonesia, Embassy of JAPAN in Indonesia, the Ministry of Foreign Affairs, the Ministry of International Trade and Industry and Japan International Cooperation Agency, who gave the survey team generous cooperations for the execution of survey.

Yours truly,

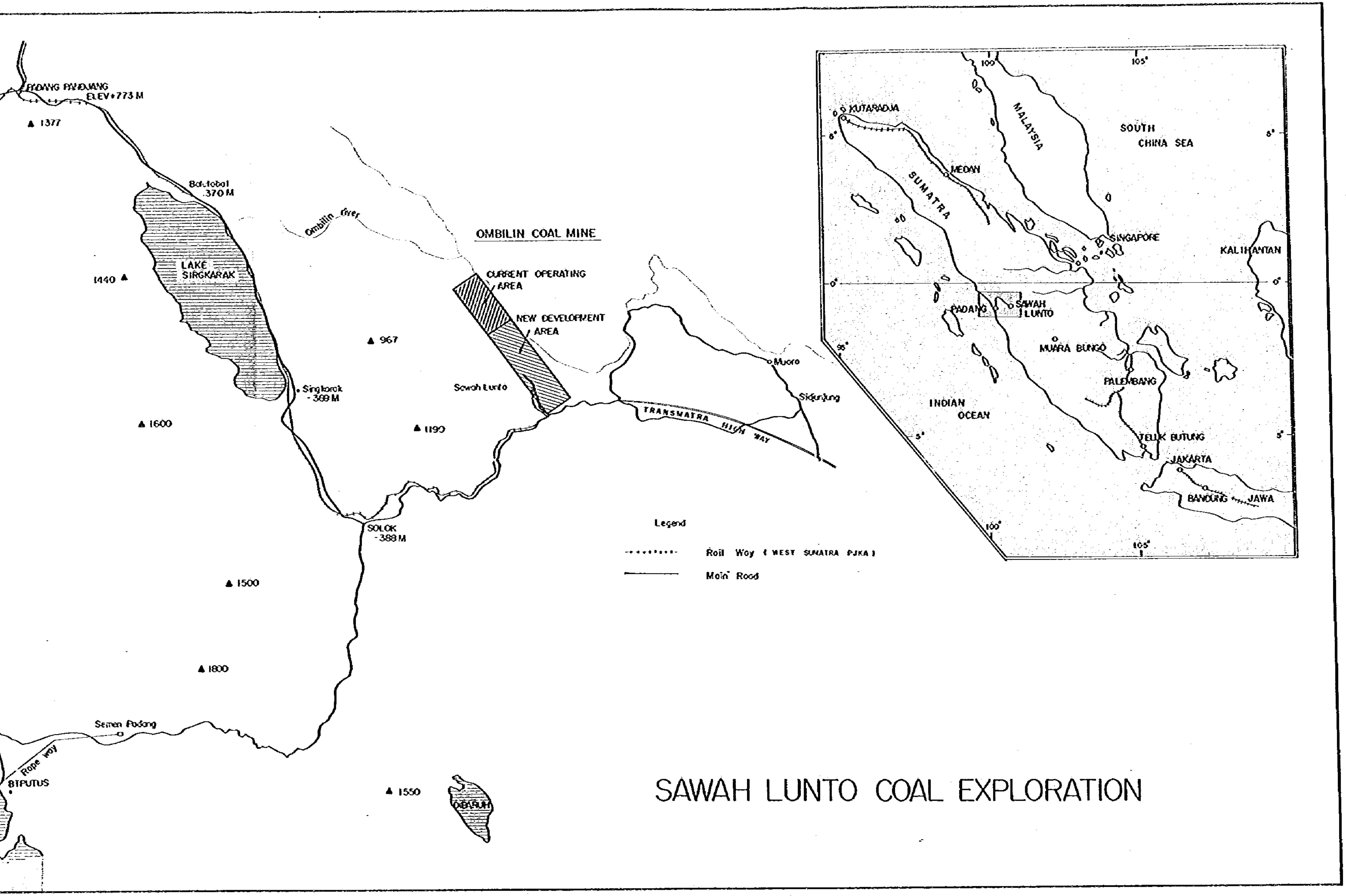
E. Kawai

Eiichi Kawai
Team Leader

C O N T E N T S

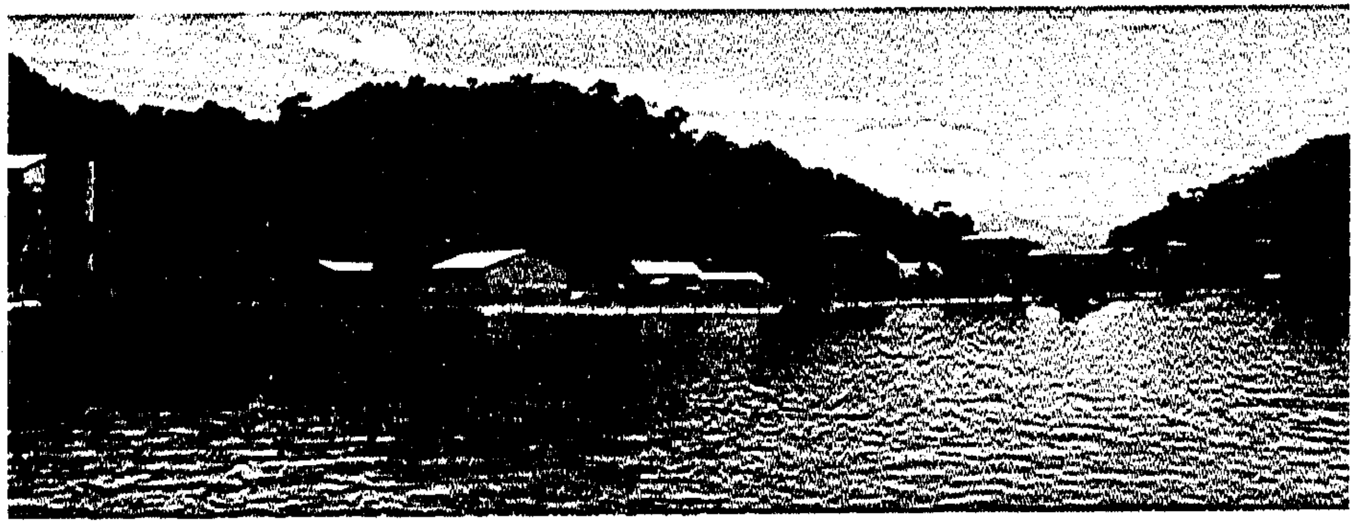


N
PADANG PANDJANG
ELEV+773 M
▲ 1377
Kajutanam
ELEV+144 M
Batutabal
370 M
Ombilin river
OMBILIN COAL MINE
CURRENT OPERATING
AREA
NEW DEVELOPMENT
AREA
1440 ▲
LAKE
SINGKARAK
▲ 967
LUBUKALUNG
ELEV+25 M
Singkarak
369 M
Sawah Lunto
Muaro
Sidjunjung
TRANSMATRA HIGH WAY
▲ 1600
▲ 1190
SOLOK
388 M
Legend
Rail Way (WEST SUMATRA PJKA)
Main Road
Duku
▲ 1500
Indian Ocean
Tobing
▲ 1800
Semen Padang
Rope way
PADANG
BTPUTUS
TELUK BAYUR PORT
▲ 1550
DIBARUH
0 5 10
km
KUTARADJA
MEDAN
SUMATRA
PADANG
INDIAN
OCEAN
SAWAH LUNTO COAL EXP



SAWAH LUNTO COAL EXPLORATION

Sawah Lunto



Teluk Bayur Port

STUDY MEMBER

Name	Charge
Eiichi KAWAI	Leader
Hiroaki TATSUNO	Mining
Toshihiko YOSHIMURA	Mining equipment
Soichiro TAKAGI	Economics
Yoshio UEDA	Civil engineering
Yutaka ISHIMODA	Civil engineering
Akio NAKAMURA	Port facility
Susumu KAMIYA	Railway
Shinichi OHASHI	Railway
Hiroshi NAMBO	Mining

B R I E F

(1) Project schedule

. Decision	1981
. Engineering design	1982
. Procurement, construction and installation	1983 - 85
. Commencement of production from the new developing area	1986

Production from the current operating area is continued in the future with increased output.

(2) Coal output

Aiming at 1.0 million tons a year (clean coal basis)

comprising,

400,000 tons from the current operating area

600,000 tons from the new developing area

(3) Initial investment

US$ 107 million (basis of the end of 1980)

comprising,

US$ 49 million for mine development

US$ 22 million for storage and shiploading

US$ 36 million for railway transportation

(4) Coal sale (correspond to 1.0 million tons output)

60,000 tons : Own use

480,000 tons : Domestic sale (330,000 for Padang Cement and 150,000 for Andaras Cement)

460,000 tons : Export (Malaysia etc.)

(5) Economics (As most likely case)

. Selling price

Domestic	US$ 22.0/ton	FOR basis
Export	US$ 30.0/ton	FOB basis

. Escalation

10%/yr for both of the price and all expenses

. DCF rate of return

About 17 % (Evaluation period: 1981 - 2005, Present value of 1980)

Table 1 PRODUCTION AND SALES SCHEDULE

[1,000t]

		1981	1982	1983	1984	1985	1986	1987	1988	1989	1990~2005
Production	Current operating area	200	300	400	400	400	400	400	400	400	Equal to the amount of 1989.
	New developing area						150	300	450	600	
	Total	200	300	400	400	400	550	700	850	1,000	
Sale	Own consumption	15	15	20	20	20	40	40	60	60	Equal to the amount of 1989.
	Domestic sale	140	235	235	330	330	480	480	480	480	
	(Padang Cement)	(140)	(235)	(235)	(330)	(330)	(330)	(330)	(330)	(330)	
	(Andaras Cement)						(150)	(150)	(150)	(150)	
	Export	45	50	45	50	50	30	180	310	460	
	Total	200	300	300	400	400	550	700	850	1,000	
Remarks	Amount for railway transport	185	285	280	380	380	510	660	810	940	Equal to the amount of 1989.
	Amount for shiploading	45	50	45	50	50	180	330	460	610	

Table 2 INVESTMENT SCHEDULE

[10^6US$]

( ): Additional

	Item	Investment Initial	Investment Additional	Investment Total	1980	1981	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991~2005
Mine	Current operating area	13.6	36.2	49.8	0.7	1.8	8.2	0.2	1.8	0.9	(1.8)	(0.8)	(0.2)	(3.0)	(5.8)	(24.6)
	New developing area	29.2	41.4	70.6				4.0	6.2	10.4	2.9	5.7 (0.9)	(0.6)	(0.6)	(1.8)	(37.5)
	Contingency	2.9		2.9				0.4	0.6	1.0	0.3	0.6				
	Engineering Design	3.6		3.6			0.4	0.1	0.9	2.1	0.0	0.1				
	Sub total	49.3	77.6	126.9	0.7	1.8	8.6	4.7	9.5	14.4	3.2 (1.8)	6.4 (1.7)	(0.8)	(3.6)	(7.6)	(62.1)
Port	Wharf	6.4		6.4					4.5	1.9						
	Shiploading	10.4		10.4					7.3	2.8	0.3					
	Coal storage	2.4		2.4					1.8	0.6						
	Contingency	0.9		0.9					0.6	0.3						
	Engineering Design	1.6		1.6			0.5		0.9	0.2						
	Sub total	21.7		21.7			0.5		15.1	5.8	0.3					
	Total	71.0	77.6	148.6	0.7	1.8	9.1	4.7	24.6	20.2	3.5 (1.8)	6.4 (1.7)	(0.8)	(3.6)	(7.6)	(62.1)
Railway	Rolling stock	21.9		21.9			4.6	0.2	1.0	3.3	10.0	1.6	1.2			
	Rail track	8.0		8.0				5.0	3.0							
	Others	3.9	2.6	6.5			0.4	2.1	1.4				(0.3)	(0.3)		(2.0)
	Contingency Engineering Design	2.0		2.0			0.6	0.9	0.5							
	Total	35.8	2.6	38.4			5.6	8.2	5.9	3.3	10.0	1.6	1.2 (0.3)	(0.3)		(2.0)
	Grand total	106.8	80.2	187.0	0.7	1.8	14.7	12.9	30.5	23.5	13.5 (1.8)	8.0 (1.7)	1.2 (1.1)	(3.9)	(7.6)	(64.1)

Table 3 Ombilin Coal Mine profit and loss, and cash flow

	Item	Unit		1980	1981	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992
(1)	Capital investment (loan)	US$ 10^3														
(2)	" (cash)	"		717	1,839	9,133	4,758	24,573	20,174	5,777	8,090	874	3,618	7,839	4,117	3,779
(3)	Dep. asset	"		717	1,839	9,133	4,758	24,523	20,174	5,777	8,090	874	3,618	7,839	4,117	3,779
(4)	Clean coal production	T 10^3		190	200	300	300	400	400	560	700	890	1,000	1,000	1,000	1,000
(5)	Saleable amount (inland use)	"		125	140	235	234	330	330	480	480	480	480	480	480	480
(6)	" (export)	"		10	45	50	46	50	50	30	160	310	460	460	460	460
(7)	" (total)	"		135	185	285	280	380	380	510	640	790	940	940	940	940
(8)	Sales	US$ 10^3		3,040	4,430	6,670	6,620	8,760	8,760	11,440	15,960	19,800	24,360	24,360	24,360	24,360
(9)	Operating cost (current operating area)	"		2,925	3,033	3,744	4,063	4,150	4,171	4,121	4,121	4,121	4,121	4,121	4,121	4,121
(10)	" (planning area)	"								4,171	4,417	4,384	4,457	4,457	4,447	4,447
(11)	Common expenses	"		1,400	1,513	1,527	1,492	1,493	1,468	1,443	1,498	1,514	1,530	1,530	1,530	1,530
(12)	Selling expense (railroad fare)	"		41	164	209	194	205	205	123	736	1,268	1,881	1,881	1,881	1,881
(13)	(port charge)	"		30	139	150	135	150	150	Δ491	Δ340	Δ214	Δ100	Δ100	Δ9	Δ9
(14)	Overhead of head office	"		843	843	843	843	843	843	843	843	843	843	843	843	843
(15)	Operating profit	"	8 - 9 - 10 - 11 - 12 - 13 - 14	Δ2,288	Δ1,278	700	Δ197	1,959	1,973	1,607	4,729	7,942	11,628	11,628	11,537	11,537
(16)	Interest on loan	"														
(17)	Depreciation	"		960	960	1,004	2,013	2,013	2,013	5,606	6,034	6,417	6,851	6,890	5,834	5,858
(18)	Fixed cost	"														
(19)	Profit B.F. tax	"	15 - 16 - 17 - 18	Δ3,256	Δ2,240	Δ304	Δ2,210	Δ54	Δ40	Δ4,079	Δ1,313	1,125	4,777	4,738	5,703	5,679
(20)	Income tax	"														
(21)	Net profit	"	19 - 20	Δ3,256	Δ2,244	Δ304	Δ2,210	Δ54	Δ40	Δ4,079	Δ1,313	1,125	4,777	4,738	5,703	5,679
(22)	Cash flow	"	21 + 17	0	0	0	0	1,959	1,973	1,607	4,725	7,942	11,628	11,628	11,537	11,537
(23)	Repayment	"														
(24)	Net cash flow	"	22 - 23	0	0	0	0	1,959	1,973	1,607	4,729	7,942	11,628	11,628	11,537	11,537
(25)	Cumulative N.C.F.	"		0	0	0	0	1,959	3,932	5,539	10,264	18,206	29,834	41,462	52,999	64,536
(26)	Balance of debt	"														

	Item	Unit		1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005
(1)	Capital investment (loan)	US$ 10^3														
(2)	" (cash)	"		3,097	6,875	8,793	4,040	7,404	8,161	3,887	4,071	2,793	2,296	3,117	3,314	2,885
(3)	Dep. asset	"		3,097	6,875	8,793	4,040	7,404	8,161	3,887	4,071	2,793	2,296	3,117	3,314	2,885
(4)	Clean coal production	T 10^3		1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000
(5)	Saleable amount (inland use)	"		480	480	480	480	480	480	480	480	480	480	480	480	480
(6)	" (export)	"		460	460	460	460	460	460	460	460	460	460	460	460	460
(7)	" (total)	"		940	940	940	940	940	940	940	940	940	940	940	940	940
(8)	Sales	US$ 10^3		24,360	24,360	24,360	24,360	24,360	24,360	24,360	24,360	24,360	24,360	24,360	24,360	24,360
(9)	Operating cost (current operating area)	"		4,121	4,121	4,121	4,121	4,121	4,121	4,121	4,121	4,121	4,121	4,121	4,121	4,121
(10)	" (planning area)	"		4,457	4,457	4,457	4,447	4,447	4,447	4,447	4,447	4,447	4,447	4,447	4,447	4,447
(11)	Common expenses	"		1,530	1,530	1,530	1,530	1,530	1,530	1,530	1,530	1,530	1,530	1,530	1,530	1,530
(12)	Selling expense (railroad fare)	"		1,881	1,881	1,881	1,881	1,881	1,881	1,881	1,881	1,881	1,881	1,881	1,881	1,881
(13)	(port charge)	"		Δ9	Δ9	Δ9	Δ9	Δ9	Δ9	Δ9	Δ9	Δ9	Δ9	Δ9	Δ9	Δ9
(14)	Overhead of head office	"		843	843	843	843	843	843	843	843	843	843	843	843	843
(15)	Operating profit	"	8 - 9 - 10 - 11 - 12 - 13 - 14	11,537	11,537	11,537	11,547	11,547	11,547	11,547	11,547	11,547	11,547	11,547	11,547	11,547
(16)	Interest on loan	"														
(17)	Depreciation	"		5,817	5,737	5,771	5,674	5,738	5,833	5,861	5,868	5,928	5,949	5,967	5,335	5,364
(18)	Fixed cost	"														
(19)	Profit B.F. tax	"	15 - 16 - 17 - 18	5,720	5,800	5,766	5,873	5,809	5,714	5,686	5,679	5,619	5,598	5,580	6,212	6,183
(20)	Income tax	"														
(21)	Net profit	"	19 - 20	5,720	5,800	5,766	5,873	5,809	5,714	5,686	5,679	5,619	5,598	5,580	6,212	6,183
(22)	Cash flow	"	21 + 17	11,537	11,537	11,537	11,547	11,547	11,547	11,547	11,547	11,547	11,547	11,547	11,547	11,547
(23)	Repayment	"														
(24)	Net cash flow	"	22 - 23	11,537	11,537	11,537	11,547	11,547	11,547	11,547	11,547	11,547	11,547	11,547	11,547	11,547
(25)	Cumulative N.C.F.	"		76,073	87,610	99,147	110,694	122,241	133,788	145,335	156,882	168,429	179,976	191,523	203,070	214,617
(26)	Balance of debt	"														

Fig. 1 SENSITIVITY ANALYSIS .... (1)

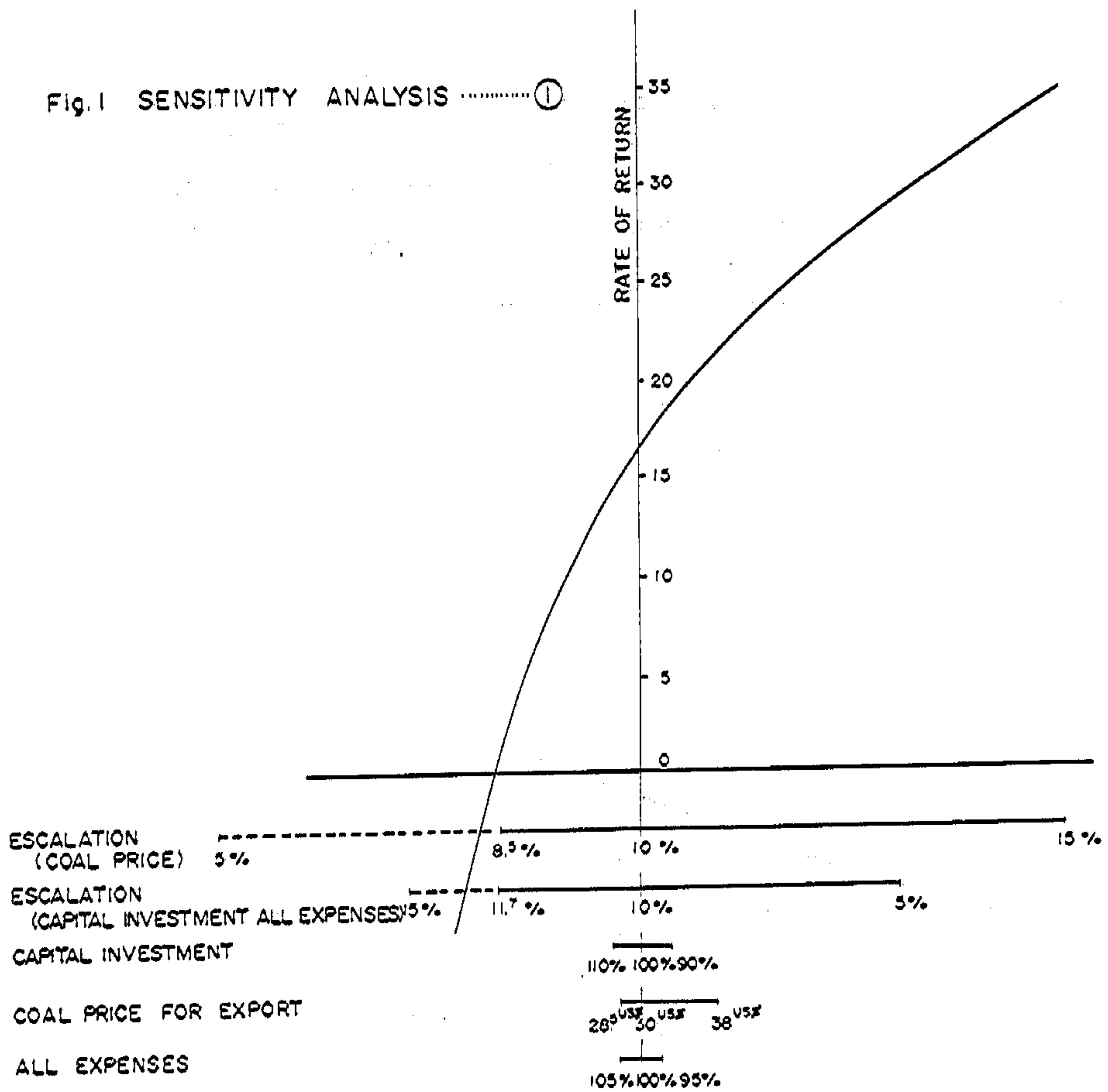


Fig 2 SENSITIVITY ANALYSIS .... ②

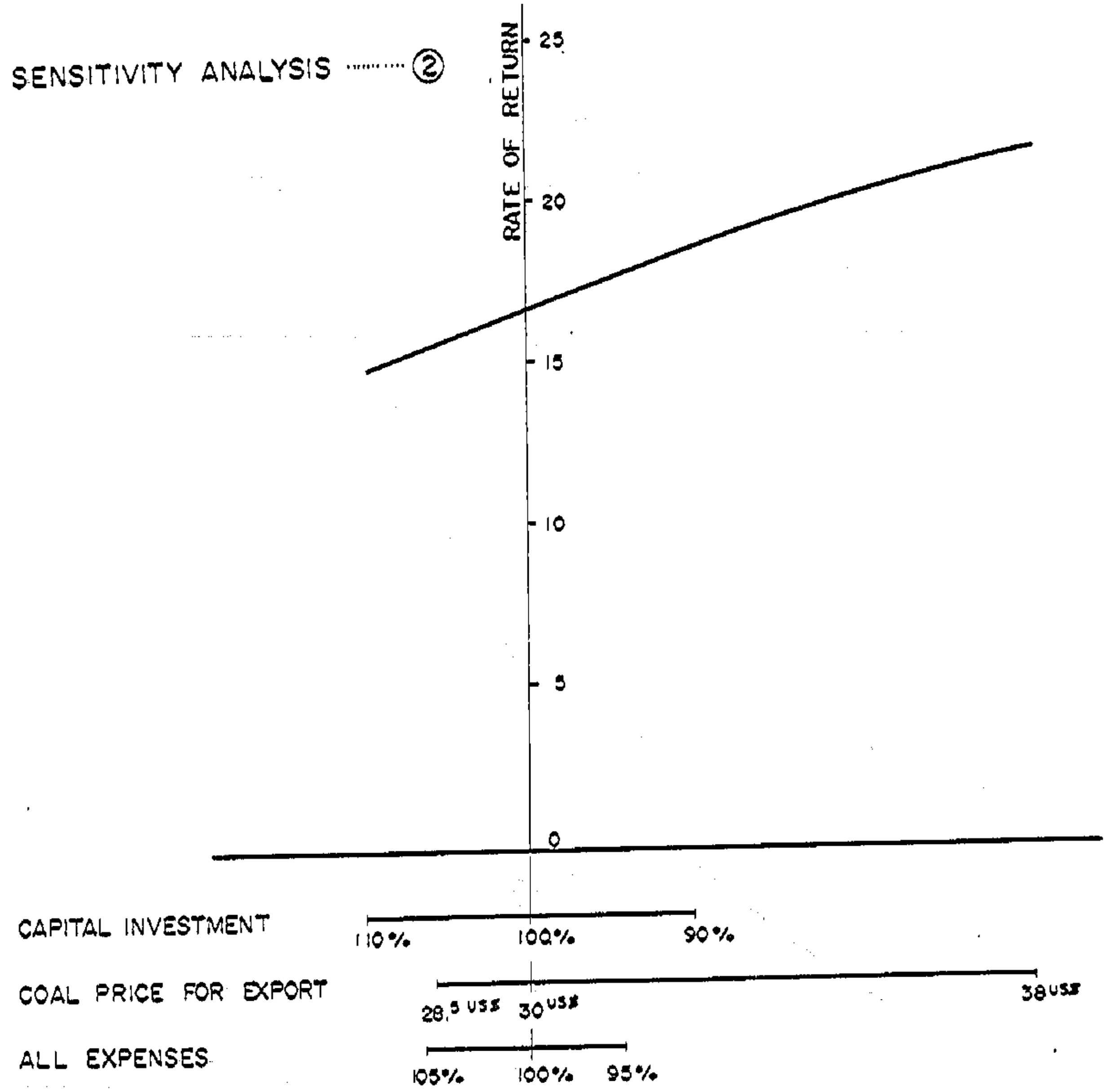
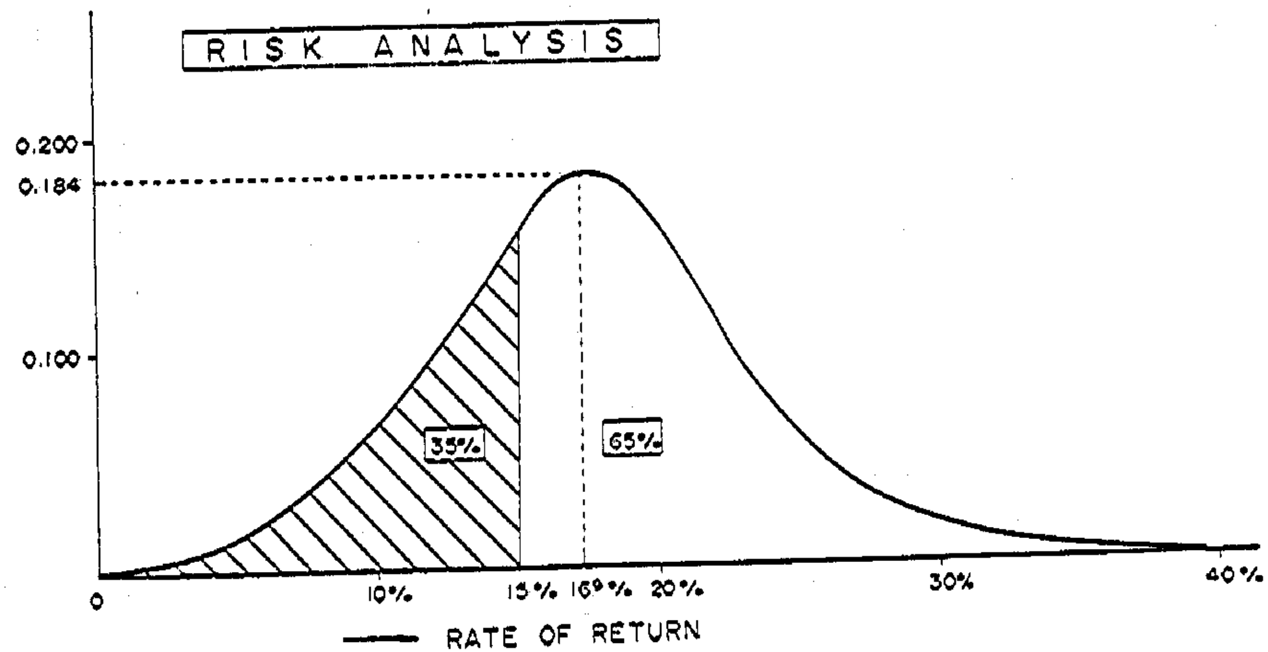


Fig. 3 RISK ANALYSIS



CONCLUSION AND RECOMMENDATION

The feasibility study for the rehabilitation of the Ombilin Coal Mine which has been carried out since the beginning of 1978, thus, comes to a conclusion.

As far as an economic view is concerned, fairly high possibility is expected to carry out the project successfully on the basis of the coal production of 1 million tons a year with the corresponding improvement of the existing infrastructure.

Furthermore, circumstances in Indonesia is strongly positive toward the project promotion even if the certain amount of the social costs could be paid, since the demand for coal as a petroleum substitute is indisputable in consideration of the needs coping with the latest energy situation, reformation of the industrial structures, regional development and the like.

However, it should be pointed out that an attainment of the project depends on not only the economic view but also the following terms and conditions which will remarkably affect each other.

(1) Sophisticated management organization for the project promotion.

(2) Securing the skillful workers in each field.

(3) Quantitative and qualitative balances in developing the fields of coal production, its market and the transportation system.

(4) Adequate political support to the project and the related matters.

(5) Timely fund supply

Last but not least, it is strongly hoped that the project will successfully contribute to the economic development in Indonesia in the near future with the powerful support, eagerness and effort of the Authorities concerned.

CHAPTER I

DEVELOPMENT OF COAL MINE

CHAPTER I DEVELOPMENT OF COAL MINE

Following the energy policy by the Government of Indonesia, the Ombilin Coal is planned to be materialized for the cement factories, power plants, etc., which will be newly and/or extensively constructed around the area besides the existing ones.

In order to cope with the above plan, P.N. Tambang Batubara is providing a new production plan such as an increase of coal output in the current mining areas, i.e. Sawah Rassaw Pit-V, Tanah Hitam and the surrounding areas. Furthermore, it intends to improve such systems on the mining operation both in the underground and on the surface by introducing the advanced coal getting method, a construction of new transportation route, an installation of coal washing and sizing plant and so on.

Meanwhile, the Ministry of Mines and Energy in Indonesia and the Authorities concerned have studied a feasibility of the development of whole Ombilin Coal field in the long term view, which is presupposing the technical cooperation between Japan and Indonesia on Governmental basis.

In this chapter, the new coal mine development in the areas where the latest geological exploration has been carried out is mainly examined. However, the project feasibility has to be judged in consideration of the current operating areas. Thus, an assessment on the investment and operation costs is made for whole Ombilin Coal Mine, and the technical terms and conditions are studied only for the new developing areas.

This study is carried out in accordance with the "Guide line for the feasibility study of the Sawah Lunto Coal Exploration dated August 7, 1980" which is agreed with the Authorities in Indonesia.

1. GENERAL DEVELOPMENT IDEA

1.1 Structural Concept

In consideration of prospective coal seam distribution, topography, current mining area, and existing surface facilities etc., "Planning area", Waringin and Sugar, are devided into three Blocks, i.e. Block-1, Block-2 and Block-3.

Block-1 is the main object area in the current feasibility study, and the basic underground structure in Block-2 and Block-3 are briefly studied as a referrence to the future development. (Fig. 1-1)

(1) Block-1

South Waringin and North Sugar

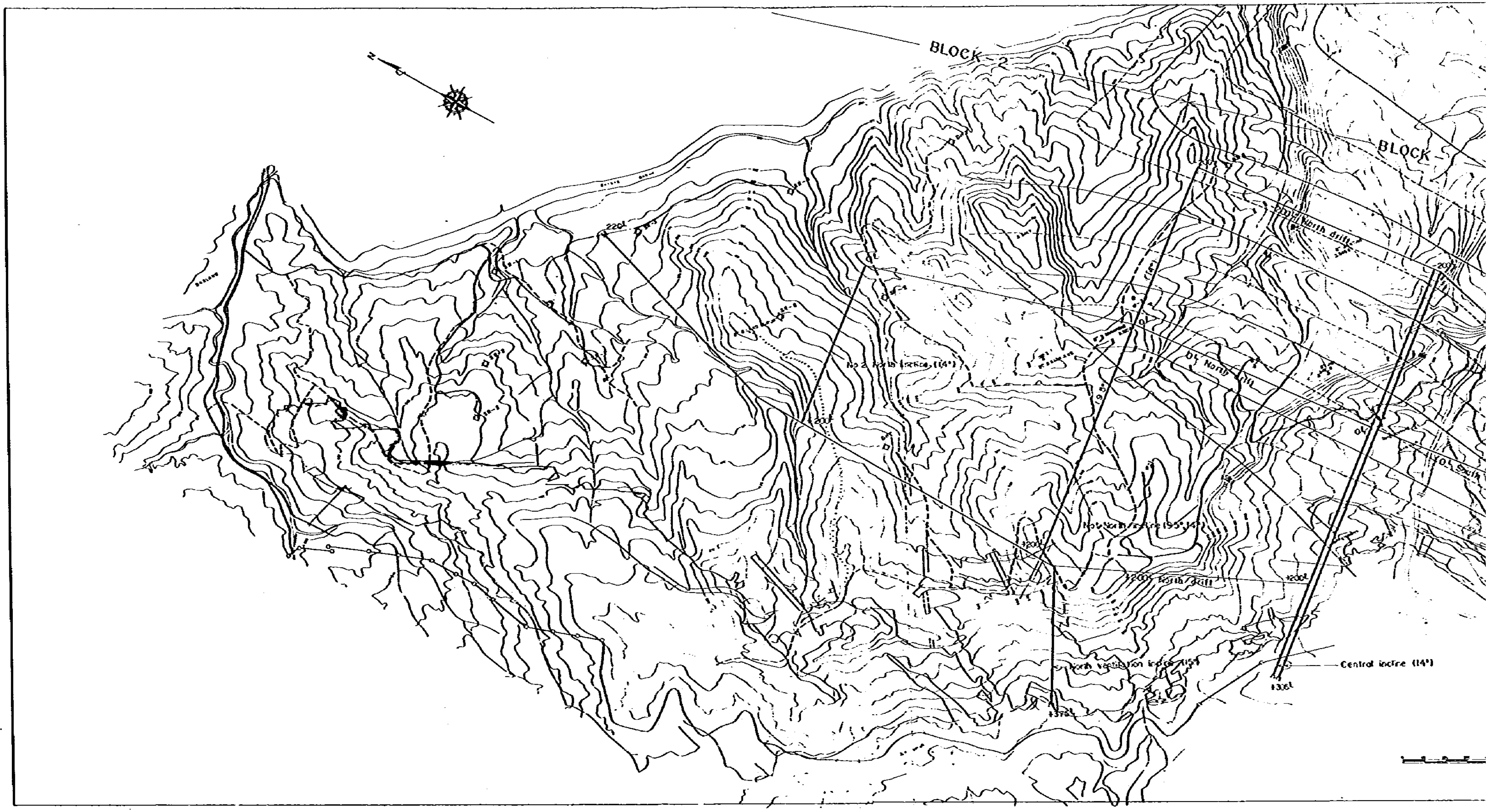
The current geological exploration has been carried out in these portions. In this study, the scope of the underground development plan is tentatively proposed down to 200 m below sea level, but its frame structure is arranged to be able to develop the lower portion below -200 L.

Twin inclined shafts, hereinafter "Central inclines", are planned for the mine opening in the center of Block-1. Frame rock drifts are arranged in such levels as +200, 0 and -200 L. Furthermore, additional inclines for ventilation are provided.

(2) Block-2

Central Waringin - Sawah Luhung

+200, 0 and -200 L frame rock drifts are extended from Block-1 to Sawah Luhung of Block-2. All of the production is transported to the surface through the central incline in Block-1.



BLOCK - 2
BLOCK -
North drift
Central incline (14°)

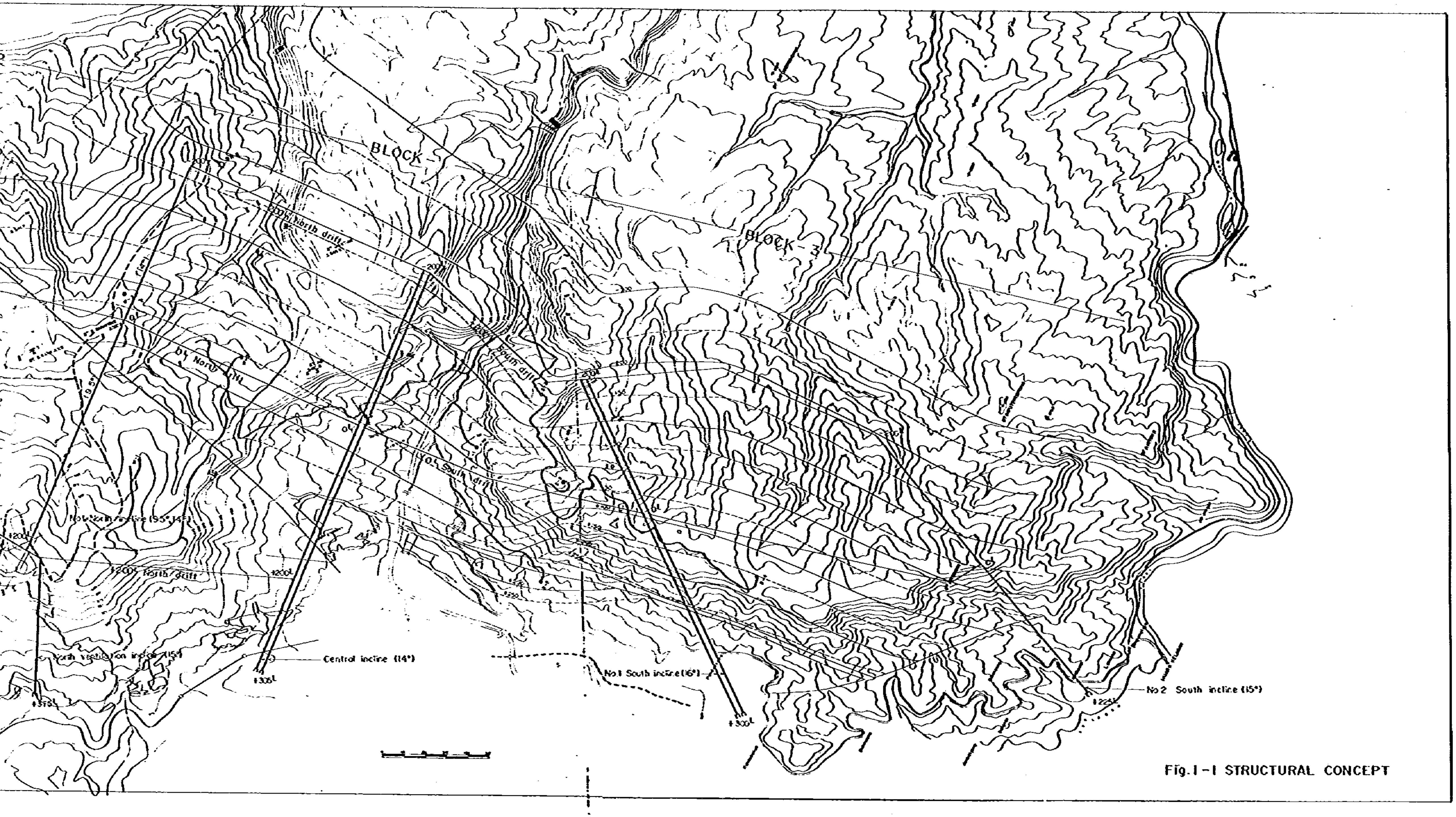


Fig.1-1 STRUCTURAL CONCEPT

(3) Block-3

Central and South Sugar

Frame rock drifts are extended from Block-1 and connected to the inclines which are newly provided from the surface of Block-3, one is in the central part of the area and another in the southern end. Coal transportation is carried out through the central incline in Block-1. If, however, enough coal reserves in Block-3 is confirmed by additional geological exploration, the incline in the southern end of Block-3 is to be used for coal transportation. In the above case, major parts of the surface facilities will be transferred to the new pit mouth area in South Sugar.

1.2 Underground Structure in Block-1

1.2.1 Opening method and the location of new pit mouth:

In relation to the existing facilities, topography and waste rock disposal place, Block-1 is proposed to be opened by twin inclines, named the central inclines, from "Kapara Rantai". Pit mouth is located at around 305 m above sea level. The inclines are generally developed below C Seam with the inclination of about 14 degrees and the length of about 2,000 meters. (Fig. 1-2)

1.2.2 Level distance

In consideration of the inclination of coal seams, coal getting methods and drifting capacity, 40 meters of the vertical level distance is adopted.

1.2.3 Frame rock drifts

Frame rock drifts are arranged in +200 m, 0 m and -200 m L which are branched from the central incline respectively toward both the north and south wings. These drifts will be also used for main level transportation when Block-2 and Block-3 are subsequently developed in the future.

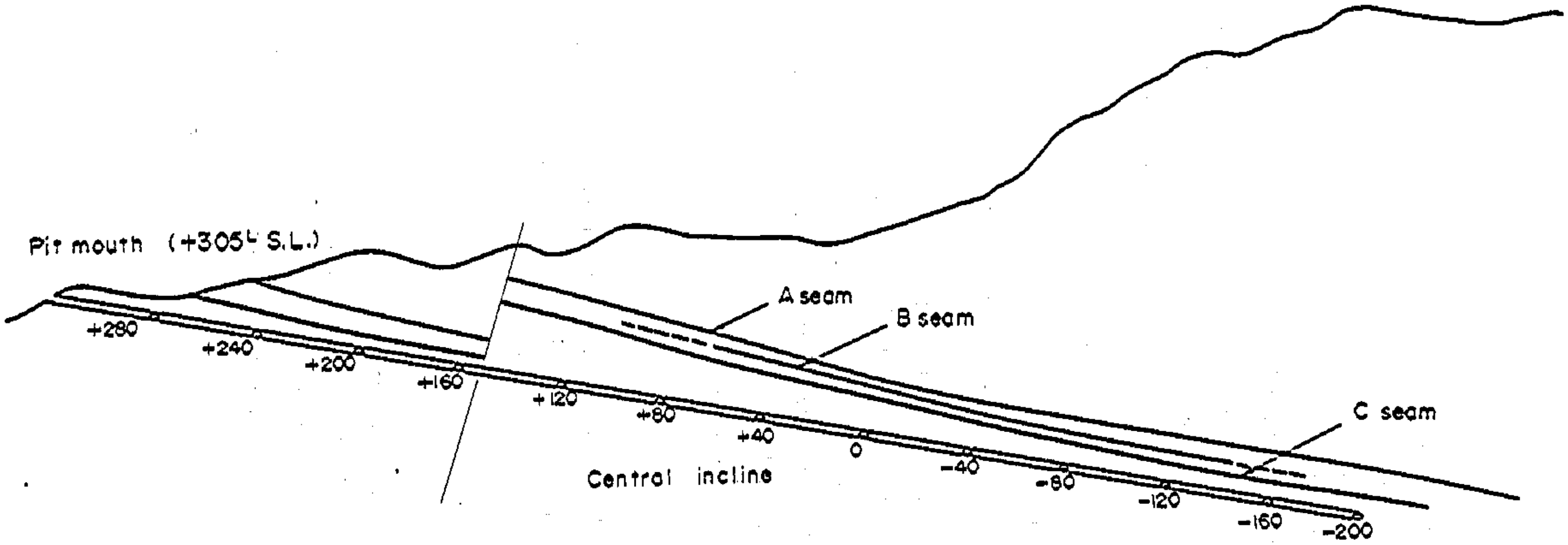


Fig. 1-2 SECTION OF CENTRAL INCLINE

1.2.4 Inclines for ventilation

Though the central ventilation system is adopted in the initial stage, the ventilation capacity will reach to the limit in a few years. Therefore, the new inclines for ventilation are required to be developed, when necessary, in the north and south end of the Block.

1.2.5 Layout of drifts and roads

(1) Cross cut

Cross cut drifts are provided towards A, B and C seams from the central incline in each level.

(2) Seam road

Seam roads are provided in and along coal seam from the junction with cross cut as straight as possible. This road is used for the coal transportation when upper coal getting face is operated, and also it is used as top gate road when lower coal getting face is operated.

(3) Gate road

Gate roads are made in pallarel to the seam roads with an interval of 15 to 20 meters of barrier pillar.
Connection roads between seam roads and gate roads are provided with a distance of 30 to 50 meters.

(4) Other miscellaneous roads

Other roads used for exclusive ventilation, pump station, power station, hoisting machine room, underground warehouse, coal bin and the like are made when necessary.

1.2.6 Transportation and ventilation

(1) Transportation

Coal is transported by belt-conveyor and waste rock, materials and personnel are sent by hoisting machine to the central inclines. Meanwhile, coal haulage is carried out by conveyor system and the others by battery locomotive - wagon haulage system in the underground level transportation.

(2) Ventilation

Central ventilation system, in which one side of the central incline is used for intake and another is used for upcast, is introduced for around the first decade of production period. Later, new inclines exclusively used as upcast are to be provided by degrees in north and south wings. Ventilation is thus performed by both of the central inclines for intake and north and south inclines for upcast, that is to say diagonal ventilation system.

2. COAL GETTING

2.1 Minable Reserves (R.O.M.)

2.1.1 Proposed area

Following area is proposed for mining:

Strikeward : 1.2 km north and 0.7 km south of the central incline.

Dipward : down to 200 m below sea level.

2.1.2 Mining panel (Fig. 1-3, 1-4, 1-5)

The proposed area mentioned above is devided into the mining panels of A, B and C seams respectively at each level. However, barrier pillars are to be left in order to avoid an influence of coal getting upon the central inclines and the Lunto River.

As to one stage mining of each seam, the seam thickness of mineable coal is taken as 1 meter or more. As to two stage mining which might be applied in only C seam, it is considered as 3 meters or more.

2.1.3 Minerable reserves

Mineable coal reserves are calculated in each mining panel by the following formula;

$$V = S \times \sec\theta \times t \times 1.3 \times 0.9$$

V : Mineable reserves (t)

S : Area of panel (m^2)
(actual mesurement on Fig. 1-3, 1-4, 1-5)

t : Seam thickness to be mined (m)

θ : Average inclination of coal seam

1.3 : Average specific gravity of coal

0.9 : Safety ratio

As to seam thickness to be mined (t), following figures are applied;

One stage mining : 2.0 meters

Two stage mining : 4.0 meters

Mineable coal reserves in each level (R.O.M.) and of each mining panel (R.O.M.) are shown in Table 1-1 and Table 1-2 respectively.

Table 1-1 Mineable Coal Reserves in Each Level

Level \ Seam	A Seam	B Seam	C Seam	Total
+240	214			214
+200	222		292	514
+160	207		532	739
+120	405		456	861
+80	431	158	458	1,047
+40	450	218	500	1,168
0	482	232	552	1,266
-40	473	203	767	1,443
-80	514	192	939	1,645
-120	544	155	1,007	1,706
-160	604	149	993	1,746
-200	788	95	652	1,535
Total	5,334	1,402	7,148	13,884

Remark : Panel name

CN (L) +120 L

Level +120 L

Lower slice, while (U) means Upper slice

North wing, while "S" means South wing

"C" means Coal seam

Mining panels of B seam are not divided into north and south wing.

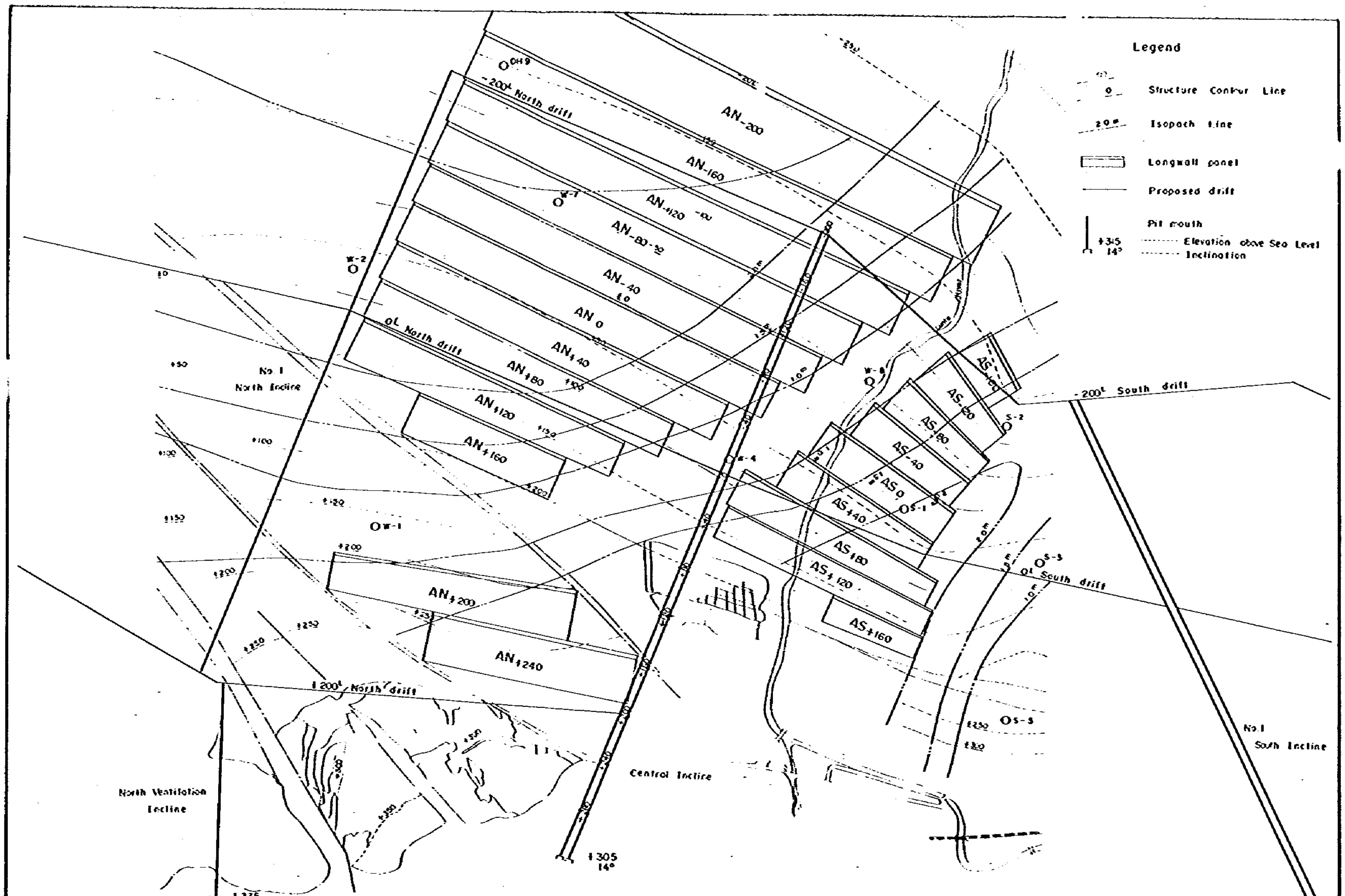


Fig. 1-3 MINING PANEL (A seam)

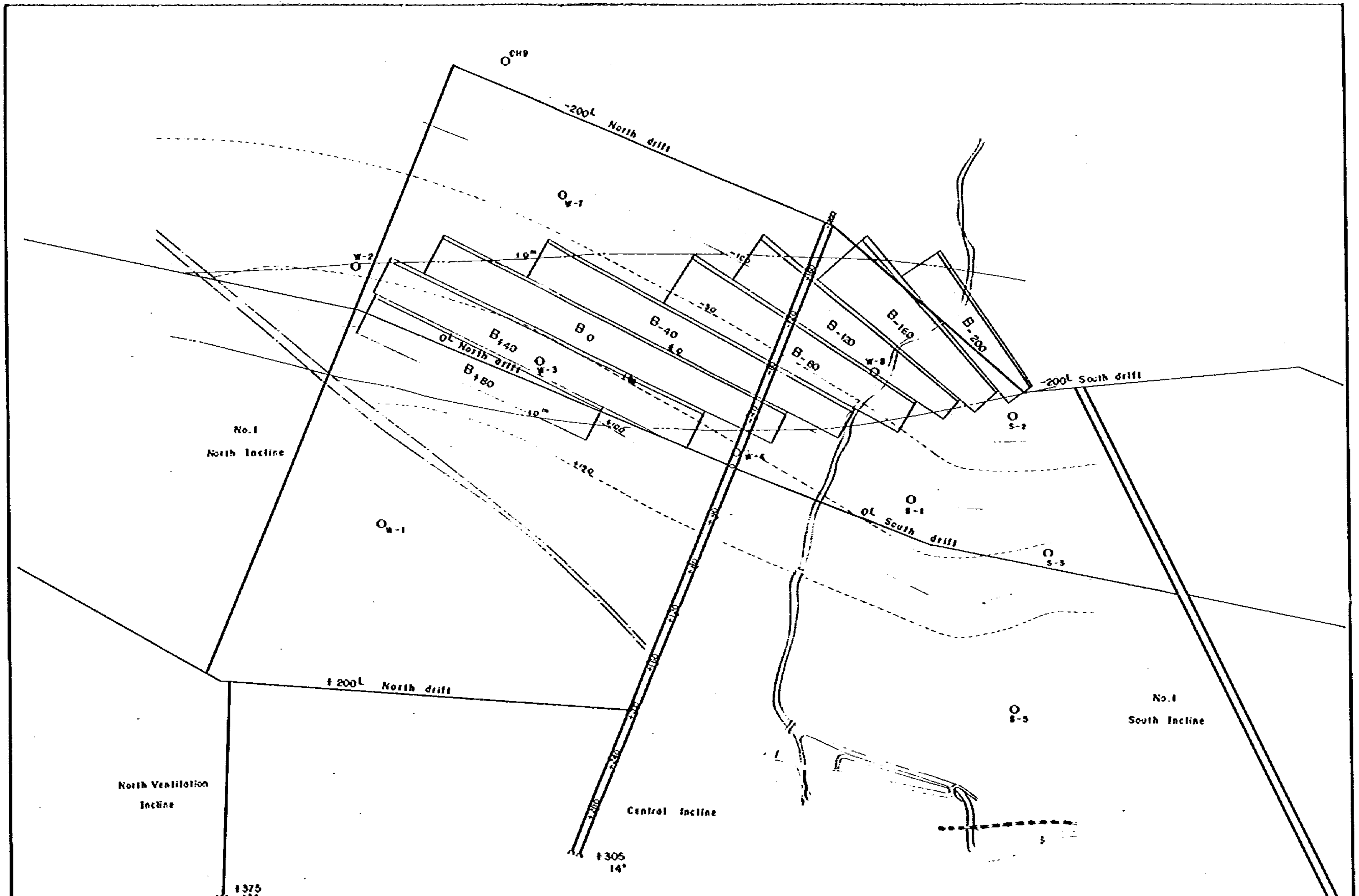


Fig.1-4 MINING PANEL (B seam)

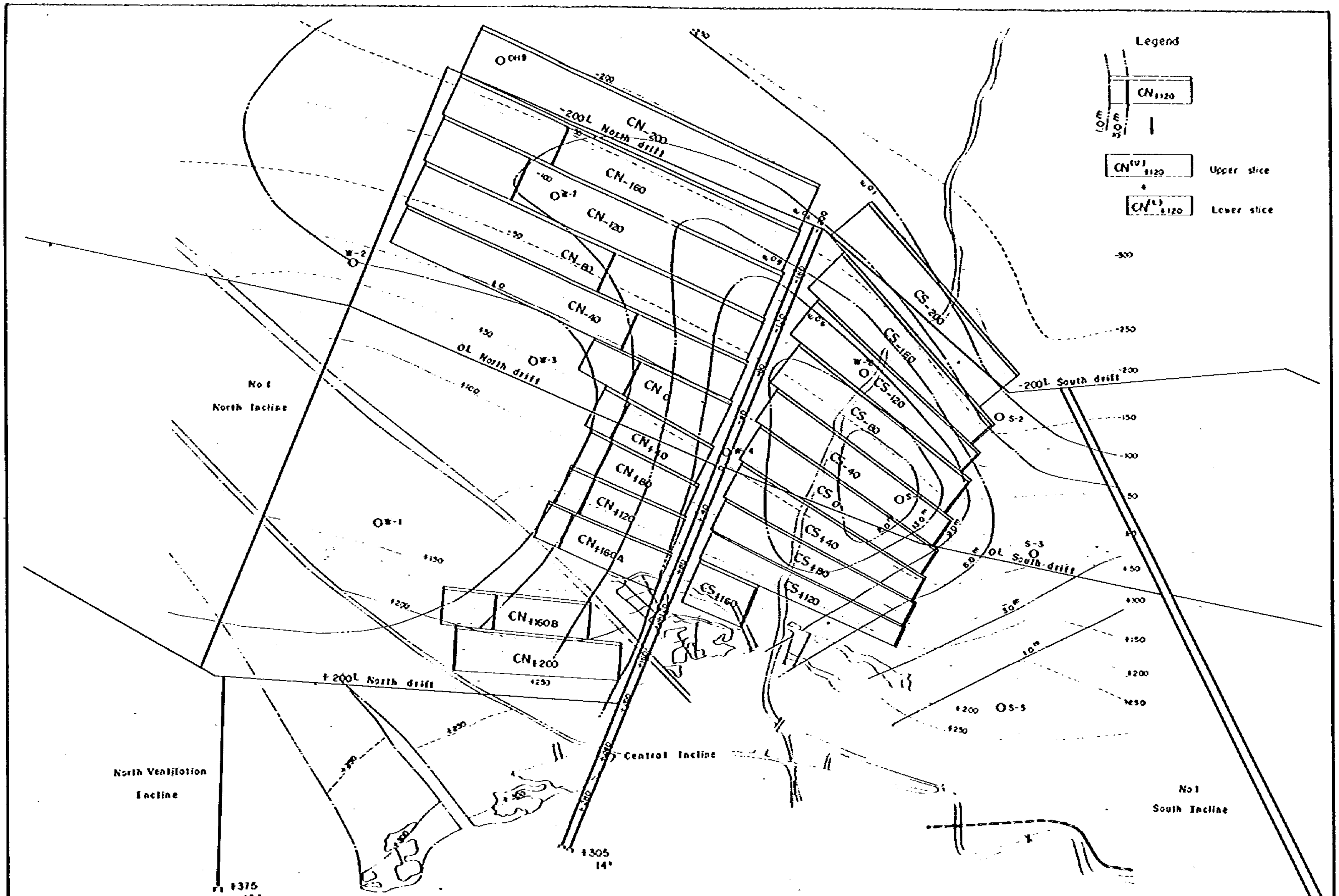


Fig. I-5 MINING PANEL (C seam)

Table 1 - 2 MINEABLE COAL RESERVES OF EACH MINING PANEL (R. O. M.)

(Unit: 1.000t)

A seam				B seam		C seam				Level
Name of panel	Reserves	Name of panel	Reserves	Name of panel	Reserves	Name of panel	Reserves	Name of panel	Reserves	
AN +240	214									+240
AN +200	222					CN(U+L) +200	292			+200
AN +160	151	AS +160	56			CN(U) +160A CN(L) +160A CN(U) +160B CN(L) +160B	115 92 113 81	CS(U+L) +160	131	+160
AN +120	251	AS +120	154			CN(U) +120 CN(L) +120	107 84	CS(U+L) +120	265	+120
AN +80	272	AS +80	159	B +80	158	CN(U) +80 CN(L) +80	96 77	CS(U+L) +80	285	+80
AN +40	320	AS +40	130	B +40	218	CN(U) +40 CN(L) +40	110 84	CS(U+L) +40	306	+40
AN 0	356	AS 0	126	B 0	232	CN(U) 0 CN(L) 0	136 90	CS(U+L) 0	326	0
AN -40	380	AS -40	93	B -40	203	CN(U) -40 CN(L) -40	330 107	CS(U+L) -40	330	-40
AN -80	431	AS -80	83	B -80	192	CN(U) -80 CN(L) -80	356 156	CS(U+L) -80	427	-80
AN -120	462	AS -120	82	B -120	155	CN(U) -120 CN(L) -120	382 285	CS(U+L) -120	340	-120
AN -160	550	AS -160	54	B -160	149	CN(U) -160 CN(L) -160	354 242	CS(U+L) -160	397	-160
AN -200	788			B -200	95	CN -200	401	CS -200	251	-200

2.2 Coal Getting Method

2.2.1 Sequence of face transition

Coal getting operation is carried out from the upper most seam to the lower seams when the seams are to be mined in the same level and locality. Consequently, A seam face is operated prior to B and C seam faces in the above case.

2.2.2 Coal getting system

Longwall retreating coal getting system is introduced in principle, as shown in Fig. 1-6. A self advancing shield support face and a single steel prop face are generally provided for the operation. Besides them, a single steel prop face is provided for a spare. Thick portion of C seam is to be mined with the upper preceding slicing method, of which two stage slicing is only introduced. (Fig. 1-7)

Coal getting equipment applied for each seam are selected in consideration of the sequence of coal getting face transition, depth from the surface, working height and natural rock conditions of roof and floor. Single steel props are introduced in all of B seam faces because of their relatively thin working height.

Coal getting equipment at A and C seam faces will be selected in accordance with the several conditions mentioned above. In the slicing method, both self-advancing shield support and single steel prop system may be applied for the upper stage. However, the former system is only introduced at the lower stage considering supporting effect in regard to the artificial roof and safety operation. Coal getting at the lower stage slicing will be commenced, giving enough time after the upper stage face is completed.

2.2.3 Face productivity

Face productivity in the actual operation is estimated 2,000 t/d at self-advancing shield support face and 600 t/d at single steel prop face on R.O.M. basis in consideration of the practice in Japan, Western Europe, the other countries and local factors in Ombilin Coal Mine. In the self-advancing shield support face, coal getting operation is to be suspended in the period of removal and installation of the face equipment after mining out one coal getting panel.

Fig. 1-6 TYPICAL ARRANGEMENT OF LONGWALL PANEL

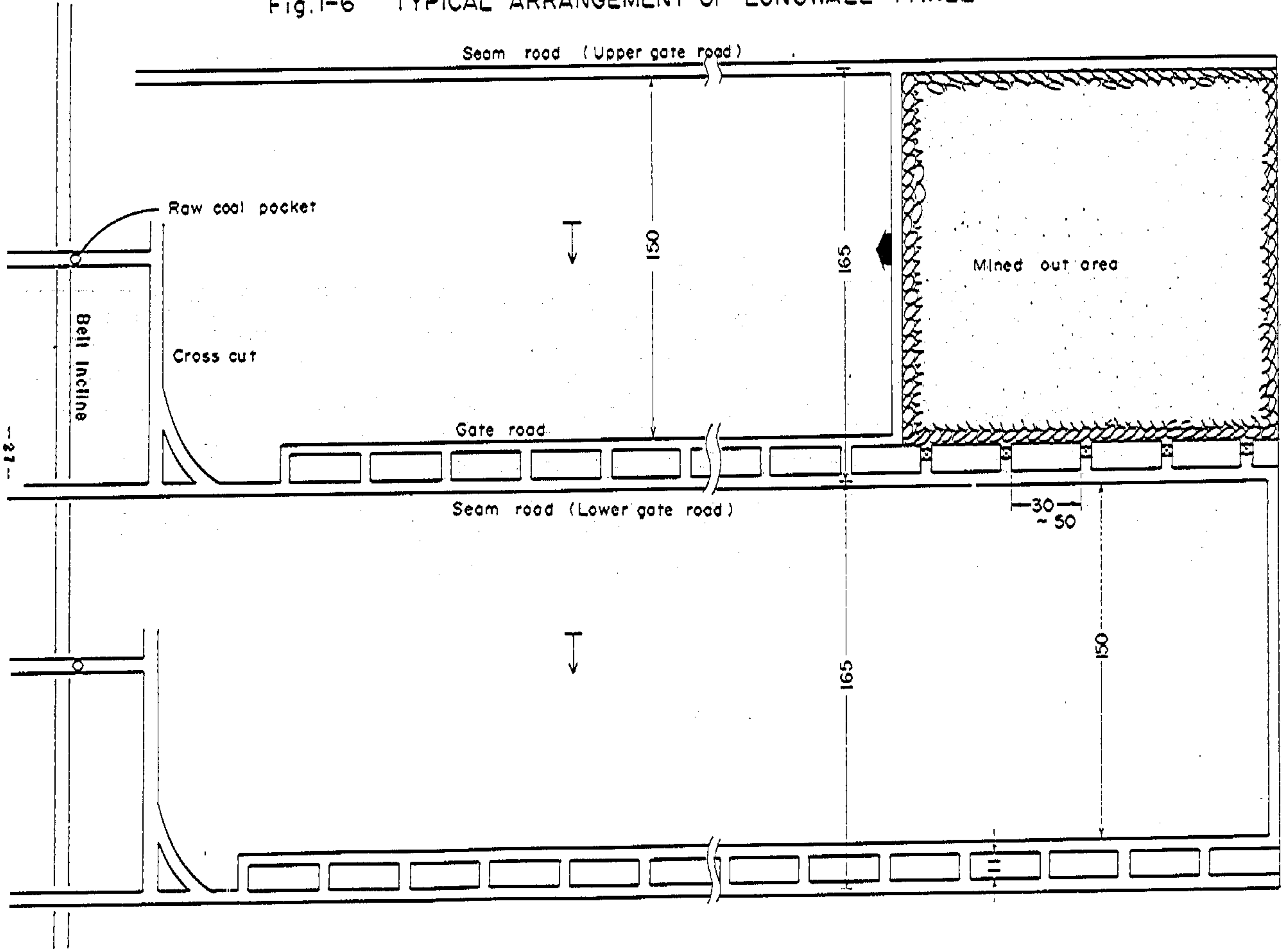
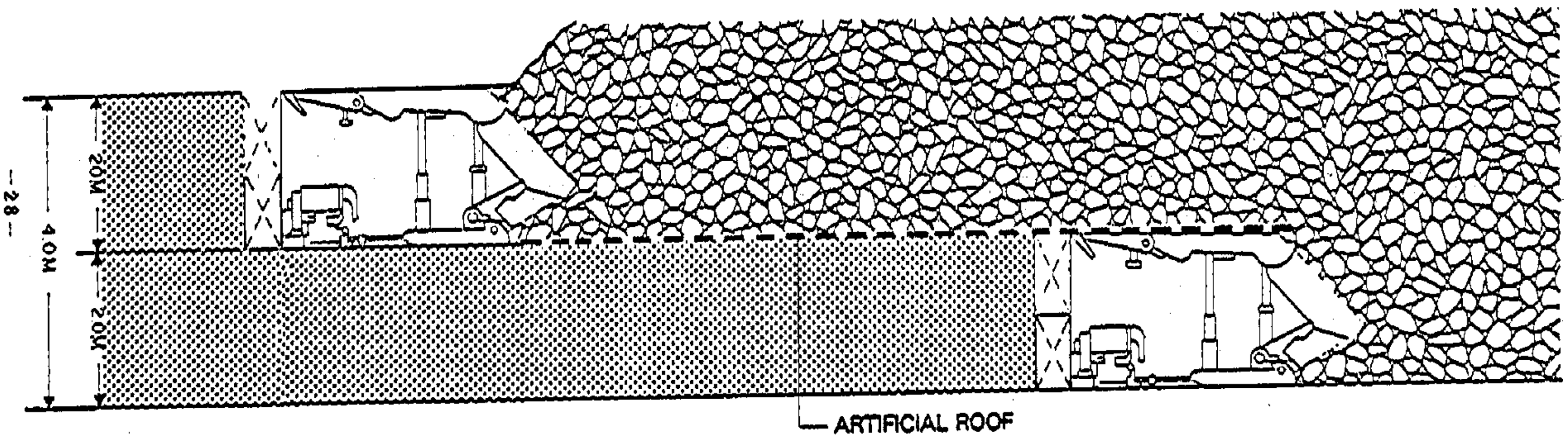


Fig. 1-7 SLICING MINNG METHOD



Consequently, an annual average face productivity in planning is to be revised as mentioned later.

In the single steel prop face, coal getting is carried out continuously by moving the workers to a new stand-by face when one coal getting panel is mined out. Equipment in the single steel prop face is planned to be removed to the new stand-by face by indirect workers who engage themselves not directly in coal getting operation. Accordingly, an annual average face productivity is considered to be equivalent to the above mentioned amount. An average productivity at the self-advancing shield support face in consideration of removal and installation is estimated as follows; At the standard panel with 2.0 m of thickness, 150 m of face length and 900 m of panel length, the amount of mineable coal reserves is obtained by the following calculation.

$$2.0 \times 150 \times 900 \times 1.3 = 351{,}000 \text{ t}$$

Supposing that an actual face productivity is 2,000 t/d, it requires about 176 days to mine out a panel. Since th period of the removal and installation is estimated about 40* days, the ratio to one cycle of face activity is obtained as $\frac{40}{176 + 40}$.

Average period required for the removal and installation per year is 56 days $(300 \times \frac{40}{176 + 40})$.

Thus, 244 days (300 - 56) in average are available for an actual coal getting operation per year. Consequently, average productivity is considered as follows;

$$2{,}000 \times \frac{244}{300} = 1{,}630 \text{ t/d}$$

* Average period for the removal and installation practice in Japan: 32 days

32 days x 125% (Allowance) = 40 days

2.2.4 Personnel allocation and productivity

(1) Personnel allocation

The production capacity of coal getting faces with self-advancing support and single steel prop are to be 2,000 t/day and 600 t/day respectively. The number of workers necessary and their roles in each longwall face are shown in Table 1-3 below;

Table 1-3 Number of Workers in Coal Getting Face

Role \ Longwall	Self advancing support L.W.	Steel prop support L.W.
Deputy	3	2
Foreman	3	2
Shearer	6	4
Support	12	30
Stable	6	4
Fitter	3	2
Electrician	3	2
Packing	6	4
Material	-	4
Total	42	54

There exist totalled 4 shifts for the self advancing support face and for the single prop face respectively. Three shifts out of those for the former and two for the latter are actually to be operated a day. The workers required for maintenance are not included in the above direct workers who engage themselves directly in coal getting.

(2) Productivity

Face productivity on this study is assumed as follows;
(R.O.M. basis)

Self advancing support longwall face

2,000 t/42 men = 47.6 t/man day

Steel prop longwall face

600 t/54 men = 11.1 t/man day

For referrence, an average face productivity in Japan is as follows;

Self advancing support longwall face

approx. 70 t/man day

Steel prop longwall face

approx. 20 t/man day

2.2.5 Major equipments in the self advancing support longwall face

Layout of major equipments is shown in Fig. 1-8, and the specification of each equipment is as follows:

(1) Drum shearer

Following functions are required to be equipped.

* Simultaneous cutting from top to floor of coal seam
  ----- Double ranging drum
* Simultaneous cutting and loading
  ----- Screw drum with helix
* Non stable type
  ----- Drum with tip bits for sumping-in
* Stable and safe driving mechanism for face undulation
  ----- Chainless drive by rack wheel system
* Prevention device for coal dust depression
  ----- Sprinkler in drum

A model below meets the above requirements, and its outline is drawn in Fig. 1-9.

EICKHOFF EDW - 200 L

Motor power	200 kw, 1,000 V
Diameter of drum	1,300 mm
Cutting depth	750 mm
Operating range	1.6 - 2.4 m
Cutting speed	5 m/min
Travelling speed	0 - 10 m/min
Gradient	0 - 23 degree

Double ranging drum shearer

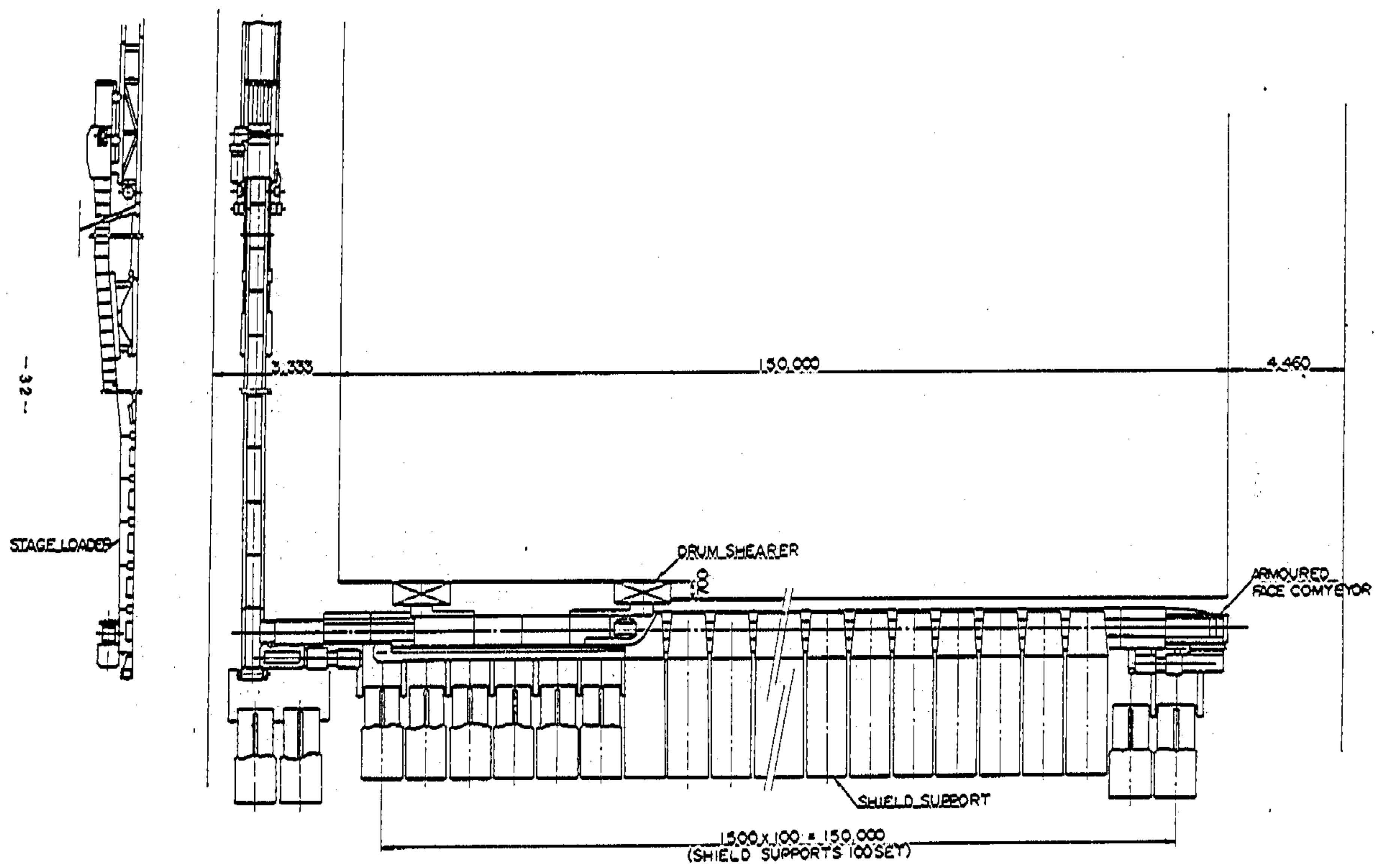
Rack wheel driving system (chainless)

Helical drums with spray nozzle

(2) Self advancing support

With the increased cutting capacity by the largely powered and speedily worked shearer, the role required for face support becomes more important in the working face activity. Following points shall be taken into account for an introduction of the self advancing supports.

Fig 1 - 8 LONGWALL UNIT LAYOUT



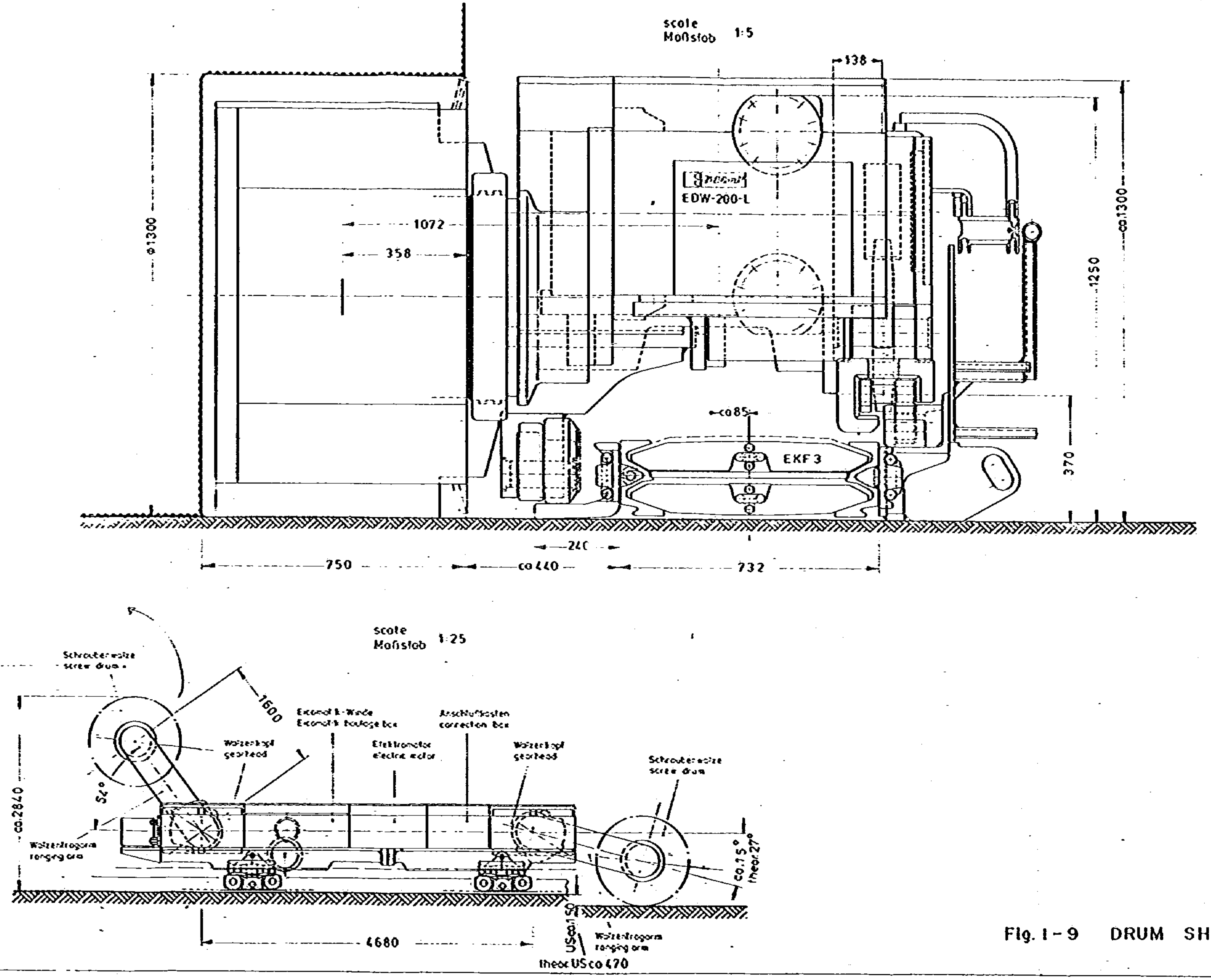


Fig. I-9 DRUM SHEARER

* to support the heavy load of the roof
* to overcome difficulties in soft roof and disturbed zone
* to support the bare roof quickly
* to be easy to install and remove
* to prevent falling of stones into the face from roof and goaf

Based on the above considerations, the supporting system below is considered to be appropriate.
Its outline is shown in Fig. 1-10.

HEMSCHEIDT G 320 - 12/27.5

Maximum extended height	2,750 m/m
Minimum closed height	1,250 m/m
Width	1,500 m/m
Yielding load per support	320 t
Setting load per support	240 t
No. of props	2
Power Pack (HAU - HINCO)	
Pressure	350 kg/cm^2
Capacity	80 l/min
Electromotor	75 kw

(3) Face conveyor

The following formula for transportation capacity by face conveyor is well known.

$$Q = m \times f \times v \times 1.30 \times 60 \times 0.70$$

m :	Cutting height	Av. 2.0 m
f :	Cutting depth	0.70 m
v :	Cutting speed	5 m/min
1.30:	Specific gravity of coal	
60 :	min./hr.	
0.70:	Loading ratio	
Q :	Transportation capacity	

The transportation capacity is thus calculated as 380 t/h. Given the above figure, the following formula is used to obtain the power requirement.

$$N = \frac{(W_1Lf_1 + WLf_1 + WLf_2)\cos.\alpha + W_1L\sin\alpha\ V\ S}{6{,}120 \times \eta}$$

N	: Power to be required	
W	: Weight of transported materials	240 kg/m
W	: Weight of chains and scrapers	40 kg/m
L	: Length of conveyor	150 m
v	: Chain speed	55 m/min
f_1	: Coefficient of friction between coal and pan	0.3
f_2	: Coefficient of friction between returning chain and pan	0.2
S	: Safety ratio	1.75
η	: Efficiency	0.75
α	: Inclination of face	14°

About 160 kw of power requirement is obtained from the calculation. As a result of the above examination, a model specification of an appropriate face conveyor is shown as follows:

* Tonnage	max. 600 t/h
* Chain speed	55 m/min.
* Power installed	90 kw x 2
* Width of pan	730 m/m
* Face length	150 m
* Single chain type	

(4) Stage loader

Movable stage loader is used for transportation between face conveyor and gate belt conveyor. Model specification of stage loader is shown below and its outline is shown in Fig. 1-11.

* Tonnage	600 t/h
* Chain speed	55 m/min.
* Power installed	75 km
* Moter power of advancing unit	3.7 kw x 2

2.2.6 Major equipments in the single steel prop face

The specification of each equipment is as follows;

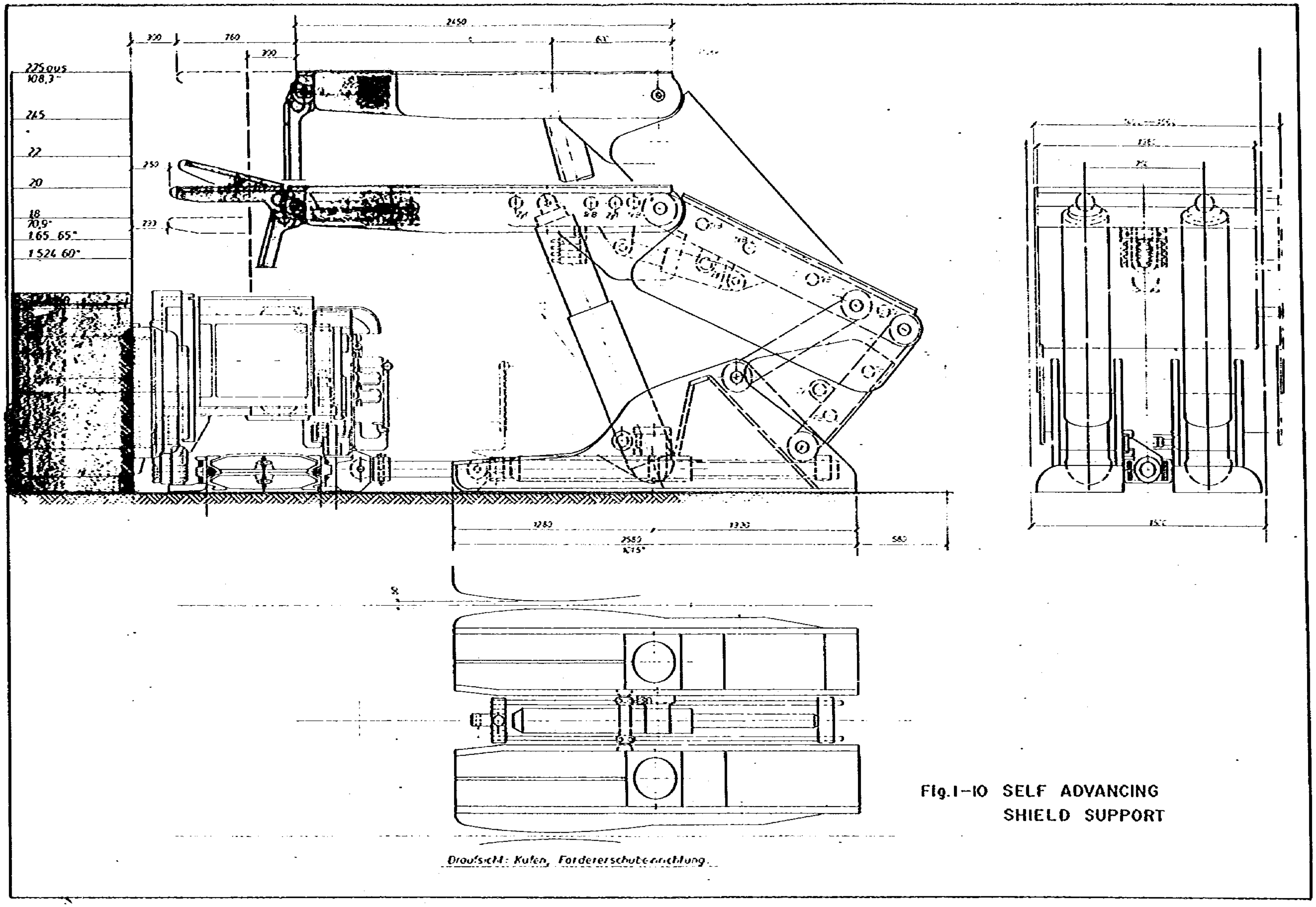


Fig.1-10 SELF ADVANCING SHIELD SUPPORT

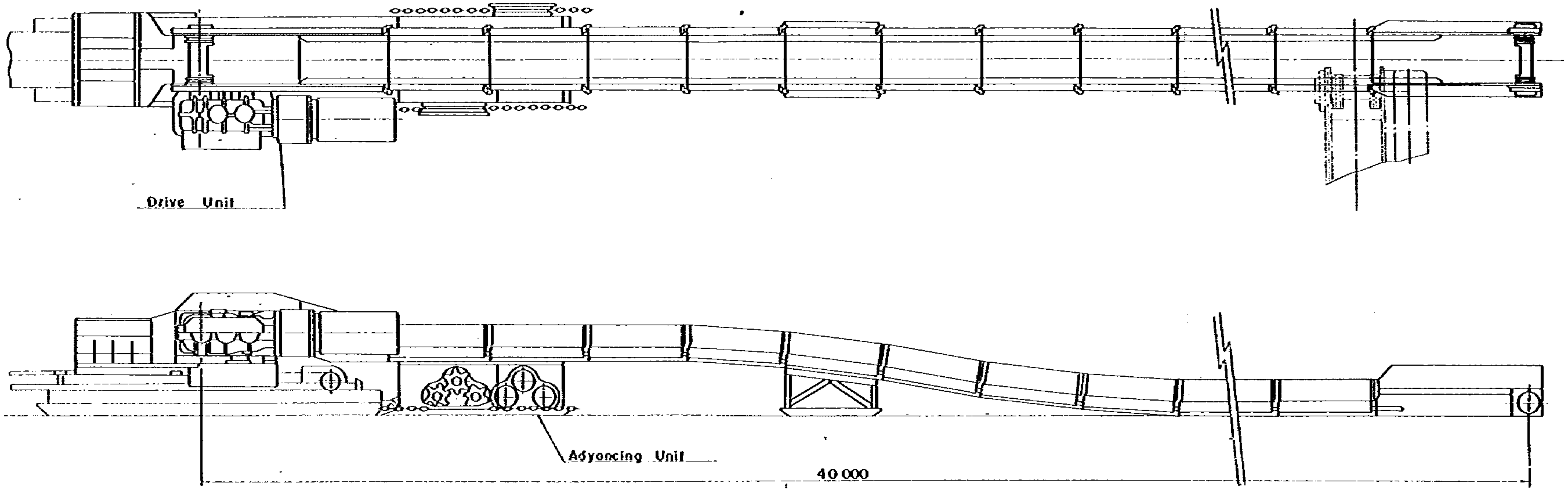


Fig.1-11 STAGE LOADER

(1) Hydraulic prop and joint bar "KAPPE" (Fig. 1-12)

The following functions are necessary to be equipped for hydraulic steel prop and joint bar in the selection of a type of model.

- * to fit the thickness of coal seam 1.6 - 2.4 m
- * to be easy to handle (Light weight)
- * to be strong enough against bend
- * to obtain sufficient supporting power

Based on the above requirements, the following steel prop and joint bar are considered to be appropriate.

* Light metal hydraulic steel prop

NIHON KOKI 40 AL 2400

Max. height	2,400 m/m
Min. height	1,500 m/m
Yielding load	40 t
Setting load	30 t
Weight	51 kg

* Joint bar

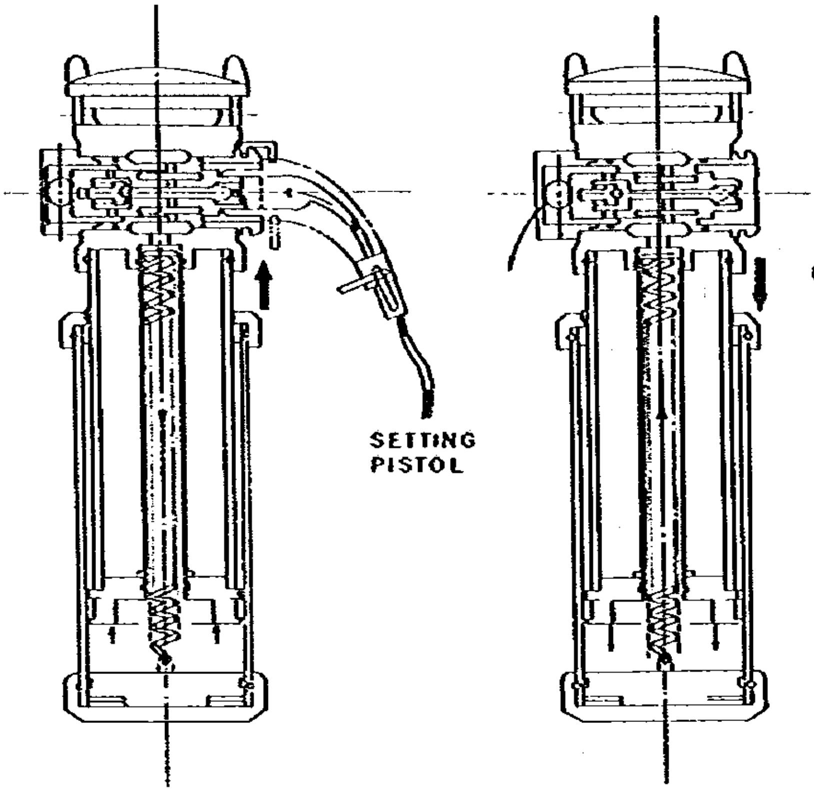
NITTO SKA - 110 1.4

Effective length	1,400 mm
Overall length	1,526 mm
Weight	47.7 kg
Beam strength (700 span)	63^{+7}_{-5} tons
Joint strength (1,200 span)	25^{+3}_{-5} tons
Vertical adjusting range	up 13°
	down 9°
Horizontal adjusting range	left 3°
	right 3°

* Hydraulic pump

Working pressure	150 kg/cm^2
Capacity	80 l/min.
Electro motor	25 kw
2 pump 1 tunk	

Light metal hydraulic prop



Maximum extension 2.400mm
Minimum extension 1.500mm
Weight 51 kg

Joint bar "KAPPE"

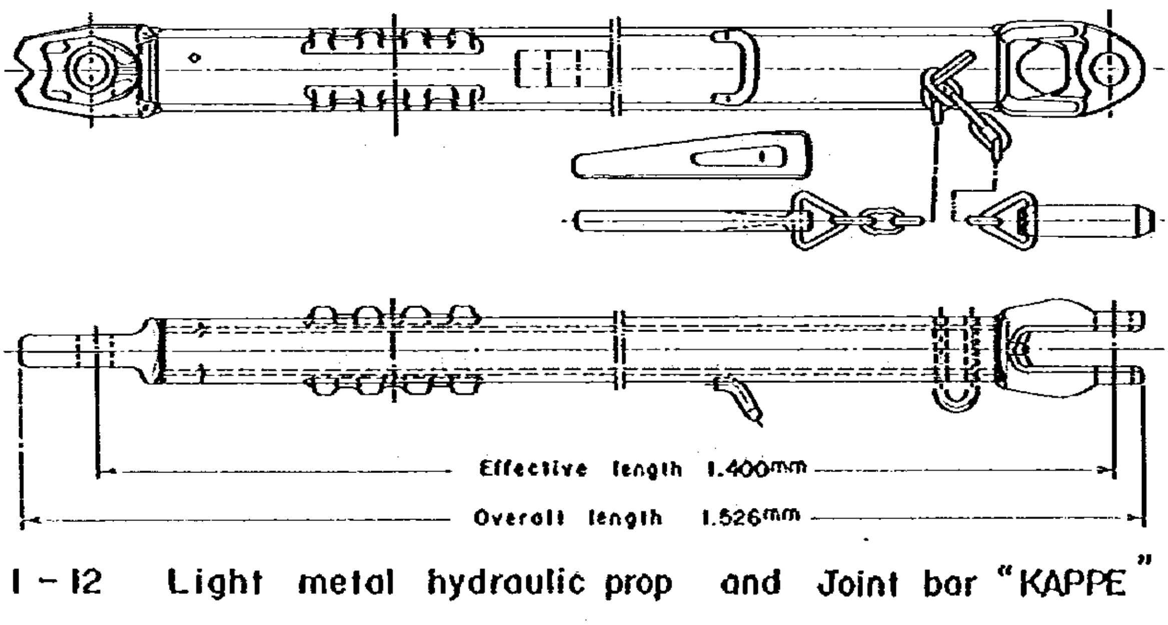
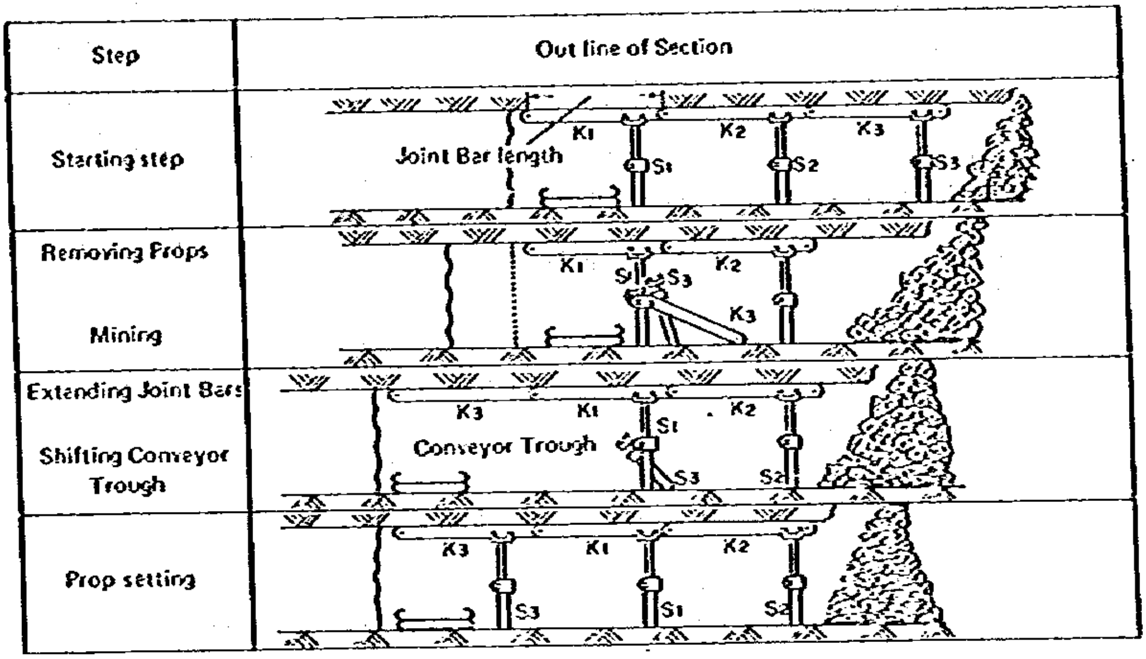


Fig. 1 - 12 Light metal hydraulic prop and Joint bar "KAPPE"

(2) Drum shearer, face conveyor, stage loader

The types of drum shearer and face conveyor are the same with those installed in self advancing mining face. This selection is adopted in consideration of their exchangeability. The figure below shows the sequence to set props and joint bars.

Fig. 1-13 Longwall mining method by props and joint bars



2.3 Coal Output

2.3.1 Annual production

An annual production from coal getting faces and seam road headings are shown in Table 1-4.

Table 1-4 Annual production

(Thousand ton)

Item / Year	Clean coal	R. O. M. Coal getting	R. O. M. Road heading	R. O. M. Total
1986	150	159	18	177
1987	300	318	35	353
1988	450	477	53	530
1989-2005	600	635	71	706
1986-2005 Total	11,100	11,749	1,313	13,062

In consideration of the changes of coal thickness and/or effective mining height, clean coal recovery is to be 85 percent.

2.3.2 Coal output

The longwall faces, one is equipped with the self advancing support and another with the single steel prop, are operated simultaneously in principle. However, as an output from the former would be commenced in 1988, a production in 1986 and 1987 is only brought from the latter.

In addition, a production in 1988 is estimated lower than the normal condition since it may take time to be accustomed to the self advancing support face in the first year of its use.

A coal output from each face is planned as in Table 1-5.

Table 1-5 Coal output from coal getting faces

	Per year	Per day	Self advancing Support L. W.	Steel prop Support L. W.
1986	159,000	530		530
1987	318,000	1,060		530 x 2
1988	477,000	1,590	1,060	530
1989-2005	636,000	2,120	1,590	530

An annual coal output from each seam and level is shown in Table 1-6. The table indicates an output in each year until 1990 and in every five years after 1991.

In this project, average speed for downward development is estimated about 20 m a year in level distance. That is relatively high in comparison with the Japanese one which is about 10 - 15 m a year.

Therefore, an organizations are required to maintain their interests to establish the adequate development system with a proper future scope.

In other words, it is necessary for them to recognize that the appropriate management system related to the high drifting ability only enables them to achieve the plan in this study.

Table 1 - 6 ANNUAL COAL OUTPUT PLAN FROM EACH SEAM AND LEVEL

(Unit: 1,000t)

	Coal getting	Seam road	A seam		B seam		C seam	
			Output	Level	Output	Level	Output	Level
1986	159	18	177	+240L				
1987	318	35	37 222	+240L +200L			94	+200L
1988	477	53	207	+160L			198 125	+200L +160L
1989	635	71	405	+120L			301	+160L
1990	635	71	431	+80L			106 169	+160L +120L
1991 ~ 1995	3,177	353	450 482 473	+40L 0L -40L	158 218 232	+80L +40L 0L	287 458 500 272	+120L +80L +40L 0L
1996 ~ 2000	3,177	353	514 544	-80L -120L	203 192 91	-40L -80L -120L	280 767 939	0L -40L -80L
2001 ~ 2005	3,177	353	604 713	-160L -200L	64 149	-120L -160L	1,007 993	-120L -160L
Balance			75	-200L	95	-200L	652	-200L

3. DRIFT AND ROAD EXCAVATION

3.1 Excavation Amount

3.1.1 Proposed cross section

In consideration of the excavation capacity, the necessity of road maintenance, the transportation and ventilation, proposed cross sections are shown in Table 1-7 and Fig. 1-14.

Table 1-7 Proposed cross section

Name of frame	Excavating section area (m^2)	Effective section area (m^2)	Application
5.0 m Arch	18.02	14.82	Incline
4.5 m Arch	14.46	12.05	Frame rock drift and incline
4.2 m Arch	12.83	10.58	Crosscut, seam road
2.4m x 2.4m	8.28	5.72	Gate road, face preparation for steel prop L.W.
3.6m x 2.4m	11.61	8.28	Face preparation for self advancing support L.W.

3.1.2 Terms for planning

An excavation amount is estimated under the following conditions.

(1) Incline and frame rock drift

20 percent of length mesured on the planning map is added as the allowance between actual and planning amount.

(2) Crosscut, seam road and gate road

Allowance of 50 percent of length measured on the planning map is added as difference between the actual and planning amount which is observed in the most cases in Japan. It is because many necessities take place in the mining practices such as re-excavation caused by faults and/or folds, provision of pump station, electric power station and so on.

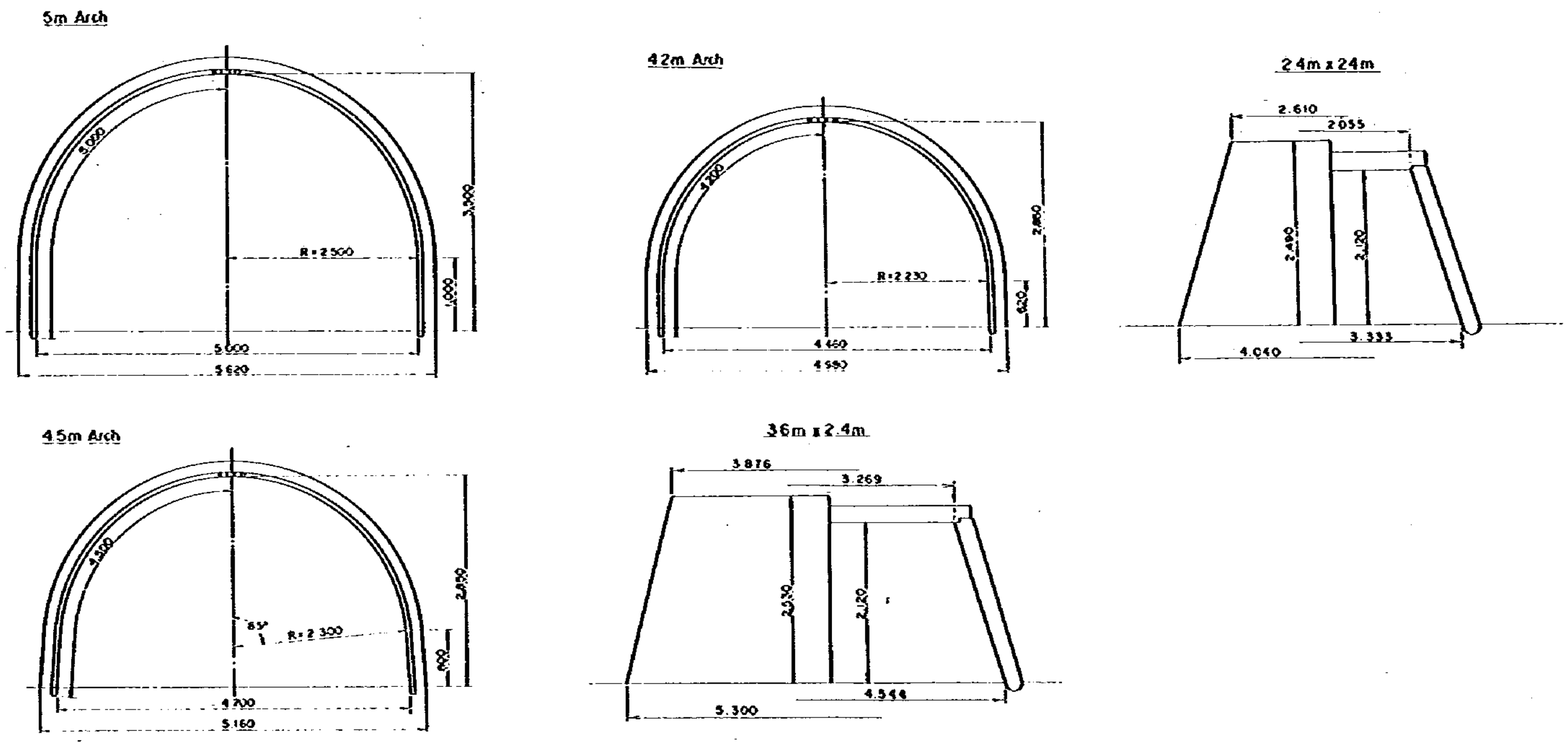


Fig. I-14 STANDARD OF CROSS SECTION

3.1.3 Excavation amount

Proposed excavation amount in this study is indicated below.

Table 1-8 Excavation amount

Name of frame	Length (m)	Quantity (m^3)
5.0 m Arch (Incline)	9,460	170,420
4.5 m Arch (Incline)	3,260	47,130
4.5 m Arch (Rock drift)	11,990	173,380
4.2 m Arch (Rock drift)	16,170	207,370
4.2 m Arch (Seam road)	85,560	1,097,670
2.4m x 2.4m (Gate road)	86,130	713,120

Total length and length per 1,000 ton of coal output are as follows;

Rock	40,880 m	3.1 m/1,000 t
Coal	171,690 m	13.1 m/1,000 t
Total	212,570 m	16.2 m/1,000 t

3.2 Excavation Method

3.2.1 The scope of works by direct workers

The scope of works by direct workers are drilling, blasting, loading, supporting and extending of temporary track, compressed air hose and air duct. Laying contemporary track, steel pipe and air hoist operation are done by indirect workers.

3.2.2 Major equipment at headings

Specifications of major equipment used at headings are shown in Table 1-9.

Table 1-9 Major equipment and specification

(per heading)

		Rock		Coal seam	
		Incline	Level drift	Arch frame	Others
Drilling	Air hummer	4	4	1	-
	Air auger	-	-	3	3
Loading	Side dump loader	$0.7m^3$ bucket; with tugger hoist, x 1	$0.7m^3$ bucket, x 1	$0.7m^3$ bucket, x 1	-
	Scraper loader	-	-	-	1
Haulage	Wagon	$2m^3$ mine car	$2m^3$ mine car	$2m^3$ dump car	-
	Belt conveyor	-	-	-	1 set
	chain conveyor	-	-	-	1 set
	Hoist	(75kw)	(10kw)	(10kw)	-
Ventilation	Local fan	$1,000m^3/min$	$1,000m^3/min$	$260m^3/min$	$260m^3/min$

3.2.3 Excavating capacity

Current record of an excavating capacity in Japan is shown in Table 1-10.

Table 1-10 Excavating capacity

	Average section area (m^2)	Excavating capacity	
		Length (m/m.s)	Quantity (m^3/m.s)
Rock drift	15.6 *	0.233	3.364
Seam road	13.3 **	0.501	6.640

* The figure corresponds approximately to the area of 4.5m arch.

** The figure corresponds approximately to the area of 4.2m arch.

Excavating capacity in this study is estimated by refering to the above mentioned Japanese standard with the adequate consideration for the local factors.

* As for the actual working period,

  Japan : 6 hours (8 hours/shift)

  Indonesia : 4 hours (6 hours/shift)

  Therefore, Japanese capacity is reduced to be 4/6 of them in Indonesia.

* It is also suggested to reduce 30% of the capacity considering the local factors such as natural conditions, working customs and the like.

* The Japanese standard of excavating capacity in the length is adjusted as well.

5.0 m Arch	: 20 percent less than 4.5 m Arch
4.5 m Arch	: 20 percent less than 4.2 m Arch
4.2 m Arch	: 20 percent less than 2.4 m x 2.4 m
Incline	: 20 percnet less than rock drift (of the same section area)
Rock drift	: 20 percents less than seam road (of the same section area)

Thus average excavating capacities are calculated as follows;

Rock drift	4.5 m Arch	0.233 x 4/6 x 0.7 = 0.11 (m/man shift)
	4.2 m Arch	0.11 ÷ 0.8 = 0.14
Seam road	4.2 m Arch	0.501 x 4/6 x 0.7 = 0.23
	2.4 m x 2.4 m	0.23 ÷ 0.8 = 0.29
Incline	4.5 m Arch	0.11 x 0.8 = 0.09
	5.0 m Arch	0.09 x 0.8 = 0.07

The number of direct workers at every kind of headings is proposed as follows in order to carry out the effective operation.

5.0 m Arch, 4.5 m Arch and 4.2 m Arch rock drift

4 - 5 workers/shift

(5 workers are allocated in the planning.)

4.2 m Arch, 2.4 m x 2.4 m, 2.4 m x 3.6 m seam road

3 - 4 workers/shift

(4 workers are allocated in the planning.)

3.2.4 Performance of excavation

The performance of excavation at each heading are estimated as the following table taking the consideration of excavation capacity, personnel allocation mentioned before and continuous operation in four shifts a day.

Table 1-11 Yearly and monthly excavation performance

	Per year	Per month
5.0m Arch incline	360 6,490	30 541
4.5m Arch incline	480 6,940	40 578
4.5m Arch rock drift	600 8,680	50 723
4.2m Arch rock drift	720 9,240	60 770
4.2m Arch seam road	960 12,320	80 1,030
2.4m × 2.4m seam road	1,200 9,940	100 828

Upper: Length (m)
Lower: Quantity (m^3)

3.3 Execution Plan

Drifts and roads required for coal getting have to be completed at least one year prior to the commencement of coal getting operation. Yearly execution plans are shown in the Table 1-12 and Table 1-13.

Table 1-12 YEARLY EXECUTION PLAN (1)

(Unit : m)

	Name of drift								Total					
	Central incline	No.1 South incline	No.1 North incline & North ventilation incline	-200L South drift	0L South drift	+200L North drift	0L North drift	-200L North drift	5.0m Arch incline		4.5m Arch incline		4.5m Arch horizontal drift	
Standard	5.0m Arch	5.0m Arch	4.5m Arch	4.5m Arch	4.5m Arch	4.5m Arch	4.5m Arch	4.5m Arch	Number of faces	Yearly performance	Number of faces	Yearly performance	Number of faces	Yearly performance
1983	180x2								2	360				
1984	360x2								2	720				
1985	360x2								2	720				
1986		360x2							2	720				
1987		360x2	480						2	720	1	480		
1988	360x2		480						2	720	1	480		
1989		360x2	480						2	720	1	480		
1990			430		600	600					1	480	2	1,200
1991		360x2	480						2	720	1	480		
1992	360x2				600				2	720			1	600
1993		360x2					600		2	720			1	600
1994					420	600	600	180					3	1,800
1995				600	30	330	240	600					3	1,800
1996	360x2								2	720				
1997		360x2							2	720				
1998	360x2								2	720				
1999	210x2	20x2							2	460				
2000			480					600			1	480	1	600
2001			380	530				60			1	380	1	590
2002~2005	—	—	—	—	—	—	—	—					2	1,200

Table 1-13 YEARLY EXECUTION PLAN (2)

	5.0m Arch incline			4.5m Arch incline			4.5m Arch horizontal rock drift			4.2m Arch horizontal rock drift			4.2m Arch horizontal seam road			2.4mx2.4m horizontal seam road			Rock drifts Total		Seam roads Total	
	Number of faces	Yearly performance m	m3	Number of faces	Yearly performance m	m3	Number of faces	Yearly performance m	m3	Number of faces	Yearly performance m	m3	Number of faces	Yearly performance m	m3	Number of faces	Yearly performance m	m3	m	m3	m	m3
1983	2	360	6,490							1	120	1,540							480	8,030		
1984	2	720	12,970							3	1,650	21,170	3	720	9,240				2,370	34,140	720	9,240
1985	2	720	12,970							3	2,000	25,660	6	4,040	51,830	2	1,730	14,320	2,720	38,630	5,770	66,150
1986	2	720	12,970							1	620	7,950	6	4,040	51,830	4	4,220	34,940	1,340	20,920	8,260	86,770
1987	2	720	12,970	1	480	6,940				1	620	7,950	6	4,040	51,830	4	4,220	34,940	1,820	27,860	8,260	86,770
1988	2	720	12,970	1	480	6,940				1	620	7,950	6	4,040	51,830	4	4,220	34,940	1,820	27,860	8,260	86,770
1989	2	720	12,970	1	480	6,940				1	620	7,950	6	4,040	51,830	4	4,220	34,940	1,820	27,860	8,260	86,770
1990				1	480	6,940	2	1,200	17,350	1	620	7,950	6	4,040	51,830	4	4,220	34,940	2,300	32,240	8,260	86,770
1991	2	720	12,970	1	480	6,940				1	620	7,950	6	4,040	51,830	4	4,220	34,940	1,820	27,860	8,260	86,770
1992	2	720	12,970				1	600	8,680	1	620	7,950	6	4,040	51,830	4	4,220	34,940	1,940	29,600	8,260	86,770
1993	2	720	12,970				1	600	8,650	1	620	7,950	6	4,040	51,830	4	4,220	34,940	1,940	29,600	8,260	86,770
1994							3	1,800	26,030	1	620	7,950	6	4,040	51,830	4	4,220	34,940	2,420	33,980	8,260	86,770
1995							3	1,800	26,030	1	620	7,950	6	4,040	51,830	4	4,220	34,940	2,420	33,980	8,260	86,770
1996	2	720	12,970							1	620	7,950	6	4,040	51,830	4	4,220	34,940	1,340	20,920	8,260	86,770
1997	2	720	12,970							1	620	7,950	6	4,040	51,830	4	4,220	34,940	1,340	20,920	8,260	86,770
1998	2	720	12,970							1	620	7,950	6	4,040	51,830	4	4,220	34,940	1,340	20,920	8,260	86,770
1999	2	460	8,290							1	620	7,950	6	4,040	51,830	4	4,220	34,940	1,080	16,240	8,260	86,770
2000				1	480	6,940	1	600	8,680	1	620	7,950	6	4,040	51,830	4	4,220	34,940	1,700	23,570	8,260	86,770
2001				1	380	5,490	1	590	8,530	1	620	7,950	6	4,040	51,830	4	4,220	34,940	1,590	21,970	8,260	86,770
2002~2005							2	x4 1,200	x4 17,350	1	x4 620	x4 7,950	6	x4 4,040	x4 51,830	4	x4 4,220	x4 34,940	7,260	101,200	33,040	347,080
Total	—	9,460	170,420	—	3,260	47,130	—	11,990	173,380	—	16,170	207,370	—	85,560	1,097,670	—	86,130	713,120	40,880	598,300	171,690	1,810,790

4. MINE TRANSPORTATION

Transportation is one of the most determinant factors in actual mining operation. Following accounts are taken for the transportation in this study.

* Adoption of continuous flow system
* Constant load for each equipment
* Flexibility for transportation capacity
* Securing safety
* Exchangeability of equipment unit

4.1 Personnel Transportation

Since the time required for personnel transportation gives a direct influence on the actual operation, the minimized time for it is to be examined.

Personnel cars are provided for the incline. However, as the distance to the working place in each level is about 500 m in average and max. about 1,200 m from the incline, no particular transportation system is provided for personnel in the level transportation.

4.1.1 Number of personnel

The plan is made based on the following maximum personnel allocation.

Table 1-14 Personnel distribution in each shift and required travelling cycles of the car

Shift	In	Out	Travelling cycles of car**
1	183	110	4
2	183	183	4
3	156	183	4
4	110	156	3
Total	632*	632	15

* Refer to "Personnel plan"

** Capacity of a unit of personnel car is as follows;

Composition of a unit	Number of cars	Number of personnel for car unit	Capacity
Ambulance	1	6	6
Normal	6	15	90
Tail end	1	12	12
Total	8	33	108

4.1.2 Personnel car and hoisting equipment

Two hoists equipped at the pit mouth of the central incline are used for both of wagon and personnel car hoisting in exchange whenever needed.

(1) Personnel car

A unit of personnel car consists of the followings;

* 1 ambulance car : Net weight 2,000 kg, Light metal, equipped with wireless signal device, 7 men including 1 operator.
* 6 normal car : Net weight 1,350 kg/each, Light metal, 15 men/each.
* 1 tail end car : Net weight 1,350 kg, Light metal, equipped with emergency operation device, 13 men including 1 operator.

(2) Hoist

Capacity and specifications of hoist is planned to be equipped as following table in relation to the wagon transportation discribed later. (See: 4.2, 4.3)

Table 1-15 Capacity and specifications of hoist

Item \ Level	120	0	-120	-200
Rope speed (m/min)	160	200	240	240
Rope pull (kg)	9,000	9,500	12,500	13,500
Motor power (kw)	400	400	650	650
Diameter of drum (mm)	2,800			
Width of drum (mm)	1,600			
Brake	650 kw V-S brake			
Emergency brake	Thruster brake			
Wire rope	28 mm B	30 mm B	31.5 mm B	33.5 mm B
Rope length (m)	1,170	1,670	2,160	2,630

4.2 Transportation of Raw Coal

Continuous transportation system from underground to the surface with various conveyors is adopted for raw coal in connection with the mechanized coal getting system.

Raw coal from seam road headings is hauled by the dump typed wagons and battery loco. to the coal pocket located above the central belt incline and discharged out into it. (See: Fig. 1-15)

4.2.1 Transportation equipment for raw coal from coal getting face

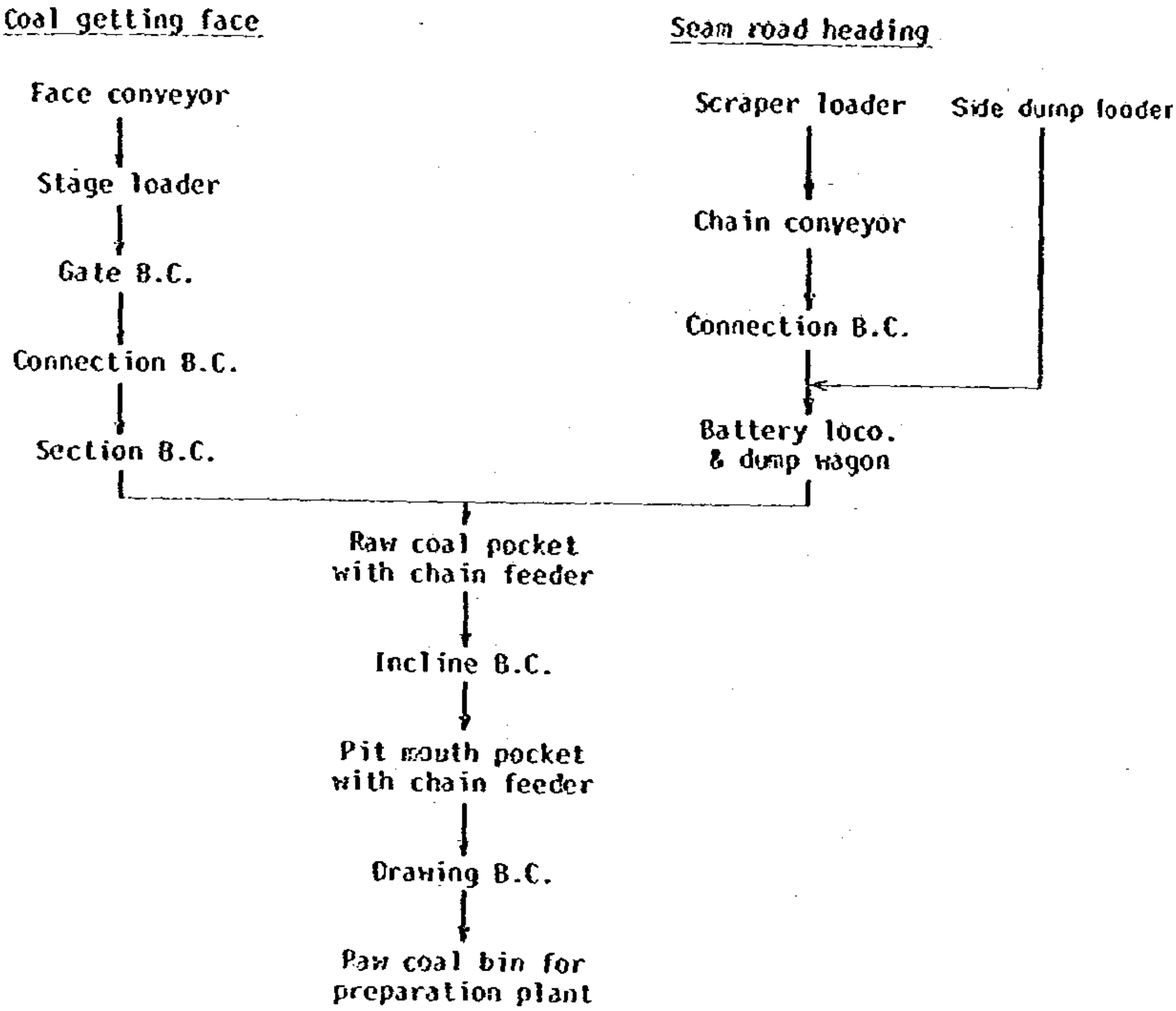
(1) Gate belt conveyor

Gate belt conveyor which has the same specifications with the section conveyor mentioned later is equipped in continuation from the stage loader.

Specifications,

Motor	15 kw
Belt speed	120 m/min
Belt width	750 mm
Trough angle	30°
Length	30 m
Capacity	400 t/h

Fig. 1-15 Raw coal transportation flow



(2) Connection belt conveyor

Connection belt conveyor is used for transporting coal between gate conveyor and section conveyor. Motor pulley is equipped for driving the conveyor because of its short distance of transport route and its convenience for frequent shifts.

Specifications,

Motor pulley	15 kw
Belt speed	120 m/min
Belt width	750 mm
Trough angle	30°
Length	30 m
Capacity	400 t/h

(3) Section belt conveyor

A Type of section belt conveyor including driving motor, installation length etc. is selected in consideration of the following terms.

* To be easy for removal and installation.
* To be the same specifications of the type for the convenience of exchangeability.

13 units of the conveyor shall be provided, in which 10 units are for the operation and 3 units for the spare.

Specifications,

Motor with Reducer	27 kw
Belt speed	120 m/min
Belt width	750 mm
Trough angle	30°
Length	250 m
Capacity	400 t/h

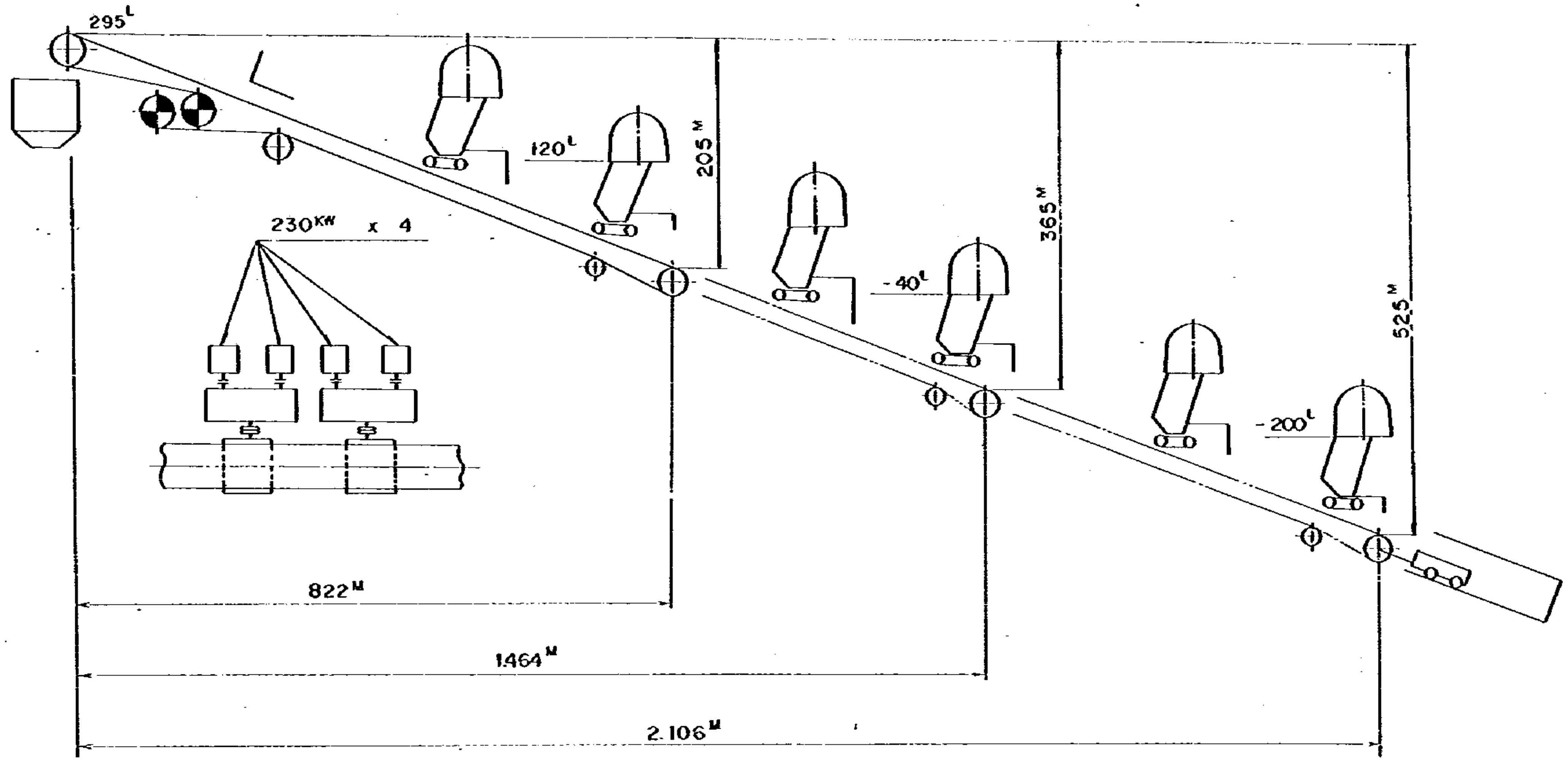
(4) Incline belt conveyor

Incline belt conveyor is installed in separation of 3 stages subsequent to the progress of development toward the deep area. Raw coal is constantly fed to the belt conveyor by menas of the raw coal pocket with the capacity of about 50 m^3 which is provided at each level. Chain feeder is equipped at the bottom of pocket for the constant feed. (Fig. 1-16)

Table 1-16 Specifications of incline belt conveyor in each stage

Item \ Stage	1	2	3
Lowest level	+120 L	-40 L	-200 L
Horizontal length (m)	822	1,464	2,106
Vertical distance (m)	205	365	525
Driving system	Tandem	Tandem	Tandem
Transport capacity (t/h)	400	400	400
Belt speed (m/min)	140	140	140
Belt width (mm)	750	750	750
Max. tension (kg)	16,490	27,500	38,580
Applied belt	ST-2,600	ST-2,600	ST-4,000
Covering rubber	5×5 Non-flamable	5×5 Non-flamable	5×5 Non-flamable

Fig. 1-16 LAYOUT OF INCLINE BELT CONVEYOR



	1ST	2ND	3RD
CAPACITY T/H	400	400	400
KILOWATT OF ELECTRIC MOTOR	230 x 2	230 x 3	230 x 4
APPLICATED BELT	ST-2600	ST-2600	ST-4000

4.2.2 Raw coal from road heading

Transportation of raw coal from road heading is carried out by dump typed wagons and battery locomotives.

(1) Transporting capacity

Output of 130,000 tons per year including waste rock is planned from seam road head excavation. Daily required transportation capacity is estimated as follows taking 30% of the fluctuation factor into account.

$$\frac{130{,}000\ t \times 1.3}{300} = 563\ (t/day)$$

Converting into the number of wagons

$$563 \div 2 = 282\ (wagons) \quad (regard\ as\ 1\ t/m^3)$$

(2) Required number of wagons

Average possible cycles of actual usage of wagons are supposed to be twice per day.

$$282 \div 2 = 141$$

Adding about 30% of the above as the spare, 180 wagons are to be provided.

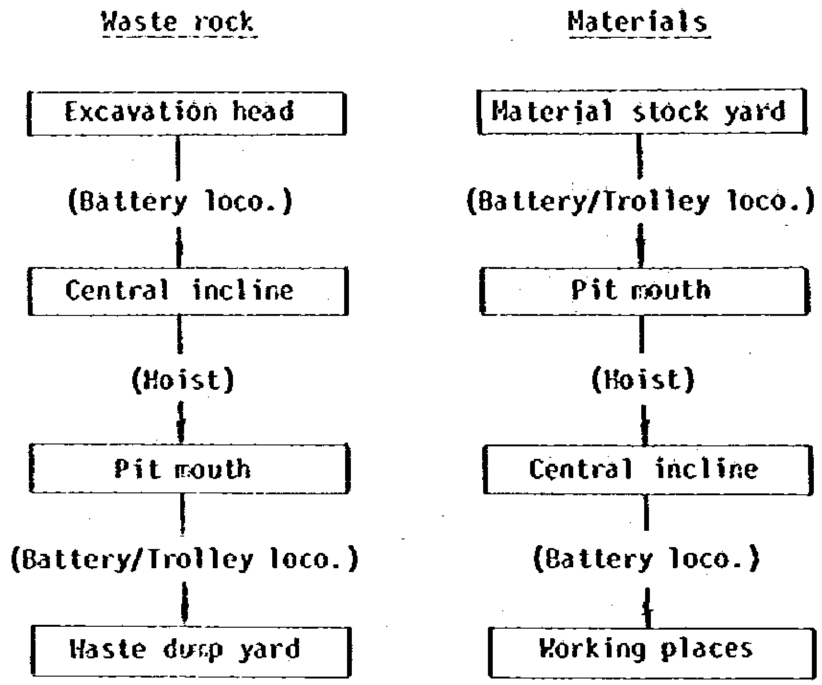
4.3 Transportation of Waste Rock and Materials

Waste rock loaded into wagons from rock drift headings and road maintenance is hauled out to the surface by hoisting machine of the central incline through level transportation by battery locomotives.

They are, then, transported to the waste dumping yard by surface locomotives.

Materials such as steel arch frames, wooden materials and the like are hauled into underground by hoist of the central incline as just opposed to waste rock flow.

Fig. 1-17 Transportation flow of waste rock and materials



4.3.1 Transporting amount of waste rock

(1) Daily output

An average annual output of waste rock including road maintenance* is supposed to be 80,000 m^3 in loose. Daily required transportation capacity is estimated as follows taking 30% of the fluctuation factor into consideration.

$$\frac{80,000 \times 1.3}{300} \fallingdotseq 350 \ (m^3)$$

It corresponds to about 180 wagons with a capacity of 2 m^3.

* Performance of road maintenance is expected 1 m^3 per man-shift in loose rock base.

(2) Required number of wagons

Average possible cycles of actual usage of wagons are supposed to be 1.2 times per day,

$$180 \div 1.2 = 150$$

Considering the spare of 20% of the above number, 180 wagons are to be provided.

4.3.2 Material transportation

(1) Daily consumption

An average annual and daily consumption of main materials is estimated as follows.

	Annual	Daily*
Wooden material	15,500 m^3	67 m^3
Steel arch frame	6,500 sets	28 sets

* Daily consumption is assumed with 30% of an allowance factor.
They correspond to about 76 wagons based on the following terms.

- o 1.2 m^3 of wooden materials are available to load into a wagon.
- o 3 sets of steel arch frame is available to load into a pair of wagons.

(2) Required number of wagons

An average daily cycle is estimated as 0.7.

$$76 \div 0.7 = 109$$

Considering 20% of the wagons for miscellaneous materials and the spare, 130 wagons are to be required.

4.3.3 Incline hoisting

(1) Operating time

Transportation for personnel, waste rock and other materials is carried out by the same hoists equipped at the pit mouth of the central incline. Therefore, 3 hours per shift is available for the actual hoisting operation since 2 hours of personnel car operation and 1 hour of interim spare are taken into account.

(2) Required operating number of wagons

Required operating number of wagons per day is proposed to be as follows in consideration of 30 wagons for miscellaneous uses.

Hoisting up,

Waste rock	180
Miscellaneous	30

Total	210
Hoisting down,	
Steel arch frame	20
Wooden material	56
Miscellaneous	30
Vacant	104
Total	210

(3) Capacity of the incline hoists

Wagon numbers of a hoisting unit train and hoisting rope speed shall be increased so as not to reduce the transportation capacity in accordance with advancing down to the deep level.

Table 1-17 Capacity and required number of hoists

Level	200	0	-120	-200
Rope speed (m/min)	160	200	240	240
Wagon number per unit train	7	7	9	9
Cycle time (sec)	951	1,128	1,281	1,419
Actual working time * (sec)	43,200	43,200	43,200	43,200
Actual operating ratio	0.60	0.60	0.60	0.60
Hoisting number of wagons per day	190	160	141	127
Required number of hoists **	(1.1) 2	(1.3) 2	(1.5) 2	(1.7) 2

* 3 hours x 3,600 x 4 shifts

** 210 ÷ hoisting number of wagons per day

4.4 Locomotive

Trolley, battery and diesel locomotive are generally used in coal mines.

In this project, battery locomotives are desirably introduced on account of relatively short hauling distance, easiness of maintenance, high security for safety, etc.

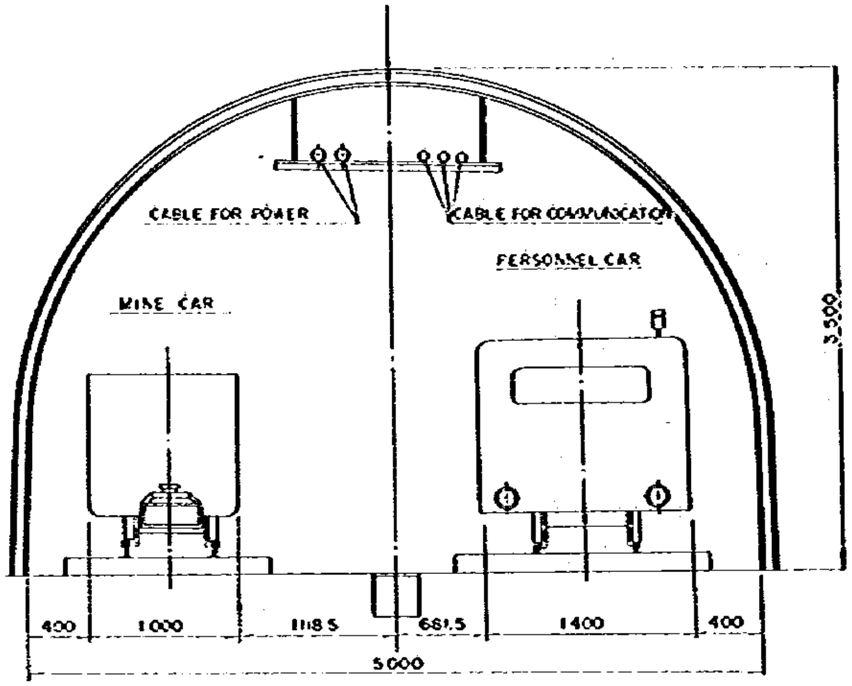
(1) Specifications of battery locomotive

Self weight	6,000 kg
Tractive force	1,000 kg
Travelling speed	7 km/h
Power	2 x 10 kw
Voltage of battery	96 V
Capacity of battery	496 AH

(2) Required number

12 locomotives comprising 2 for waste rock, 2 for coal, 1 for materials, 3 for the surface and 4 of the spare are to be introduced on the basis of the following wagon number per unit train and travelling cycle per shift.

	Number of wagons per unit	Travelling cycle per shift
Coal	20	1.5
Waste rock	15	2.0
Other materials and vacant	25	1.5



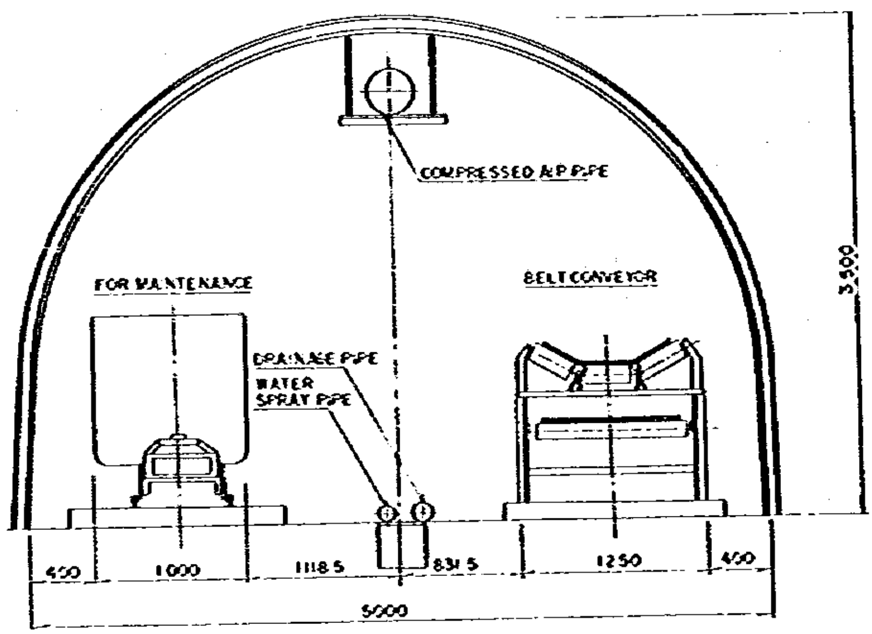


Fig. I-18

CROSS SECTION OF CENTRAL INCLINE

5. MINE SAFETY

5.1 Ventilation

5.1.1 Ventilation system

At the beginning of mining operations in Block - 1, the centralized ventilation system is applied, i.e. one side of the central inclines is used for air intake, and another for upcast.

However, since it becomes difficult to secure the required air sufficiently as the mining area is deepened, exclusive ventilation inclines are to be opened in the both wings of north and south of the mining block, and the central inclines are utilized for only intake, that is to say "diagonal ventilation system".

5.1.2 Typical ventilation network

In accordance with the progress of the underground development, a ventilation network is widely changed. Following three cases are taken into consideration as typical ventilation network in this study.

Case 1 (in 1990)	5th year from the beginning of production. Mining operations are executed in C seam at +120 L and A sean at +80 L. (Fig. 1-19)
Case 2 (in 1995)	10th year. Operations are in C seam at -40 L and A seam at -80 L. In the case 2, it is defined as Case 2A that centralized ventilation system is adopted, and defined as Case 2B that diagonal one is adopted. (Fig. 1-20, 1-21)
Case 3 (in 2005)	20th year. Operations are in C seam at -160 L and A seam at -200 L. (Fig. 1-22)

5.1.3 Required air quantity for ventilation

In the said four cases, required air quantity for coal getting faces is to be as follows;

Coal getting face under operation:

av. 800 m^3/min x 2 = 1,600 m^3/min

Spare coal getting face	: av.	500 m^3/min
Total	:	2,100 m^3/min

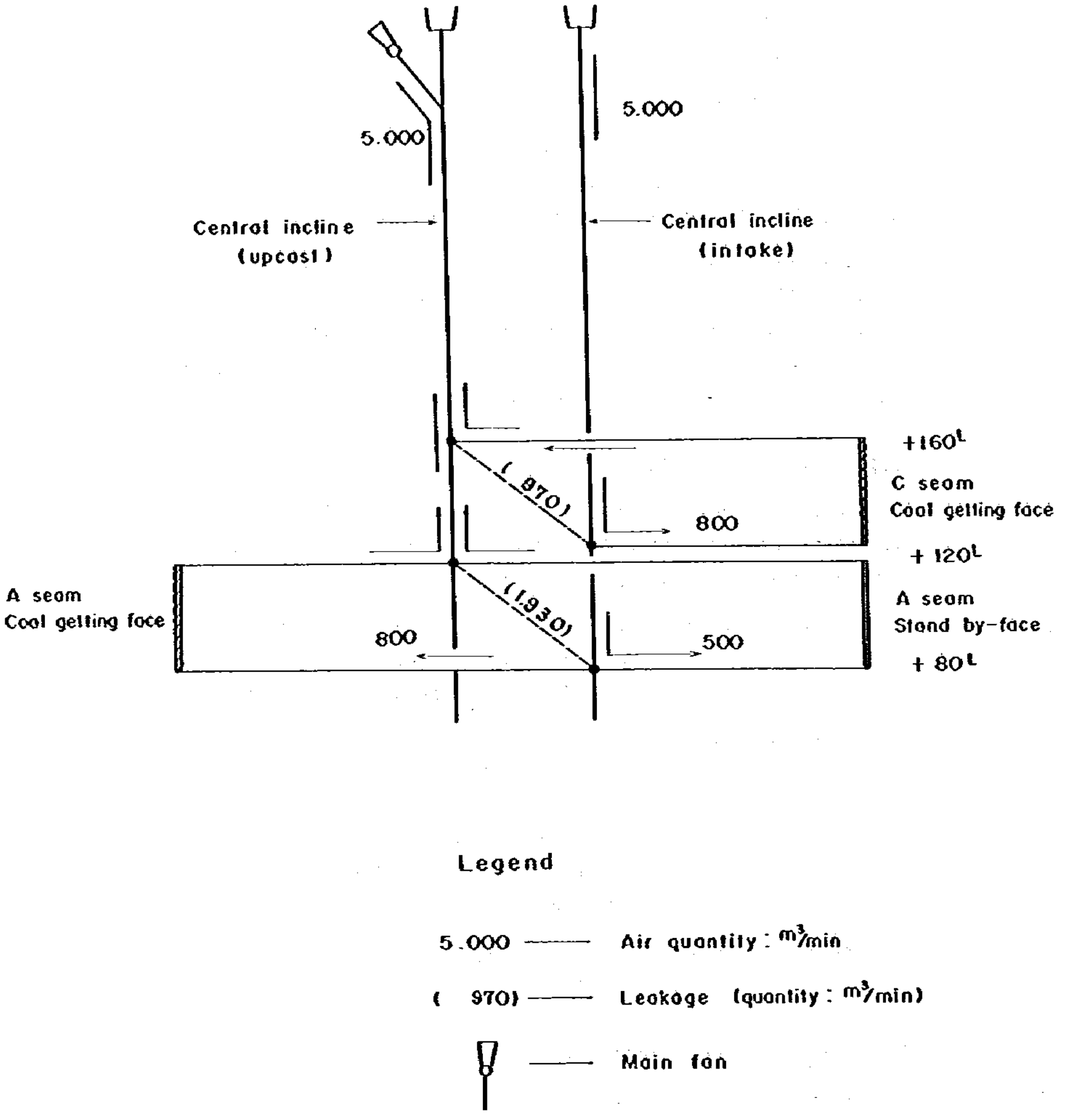


Fig. I-19 VENTILATION NETWORK (CASE-I)

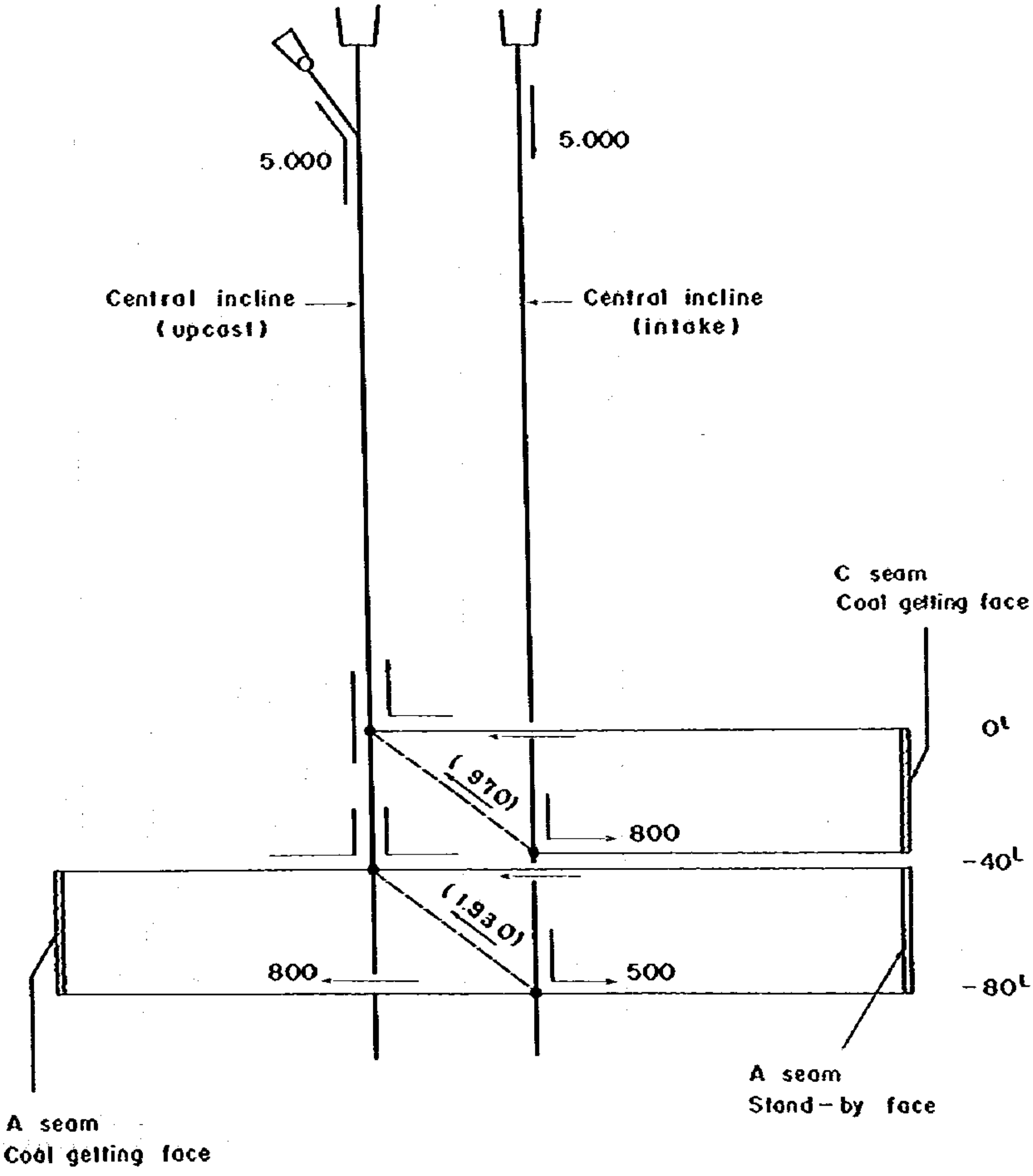


Fig.I-20 VENTILATION NETWORK (CASE-2A)

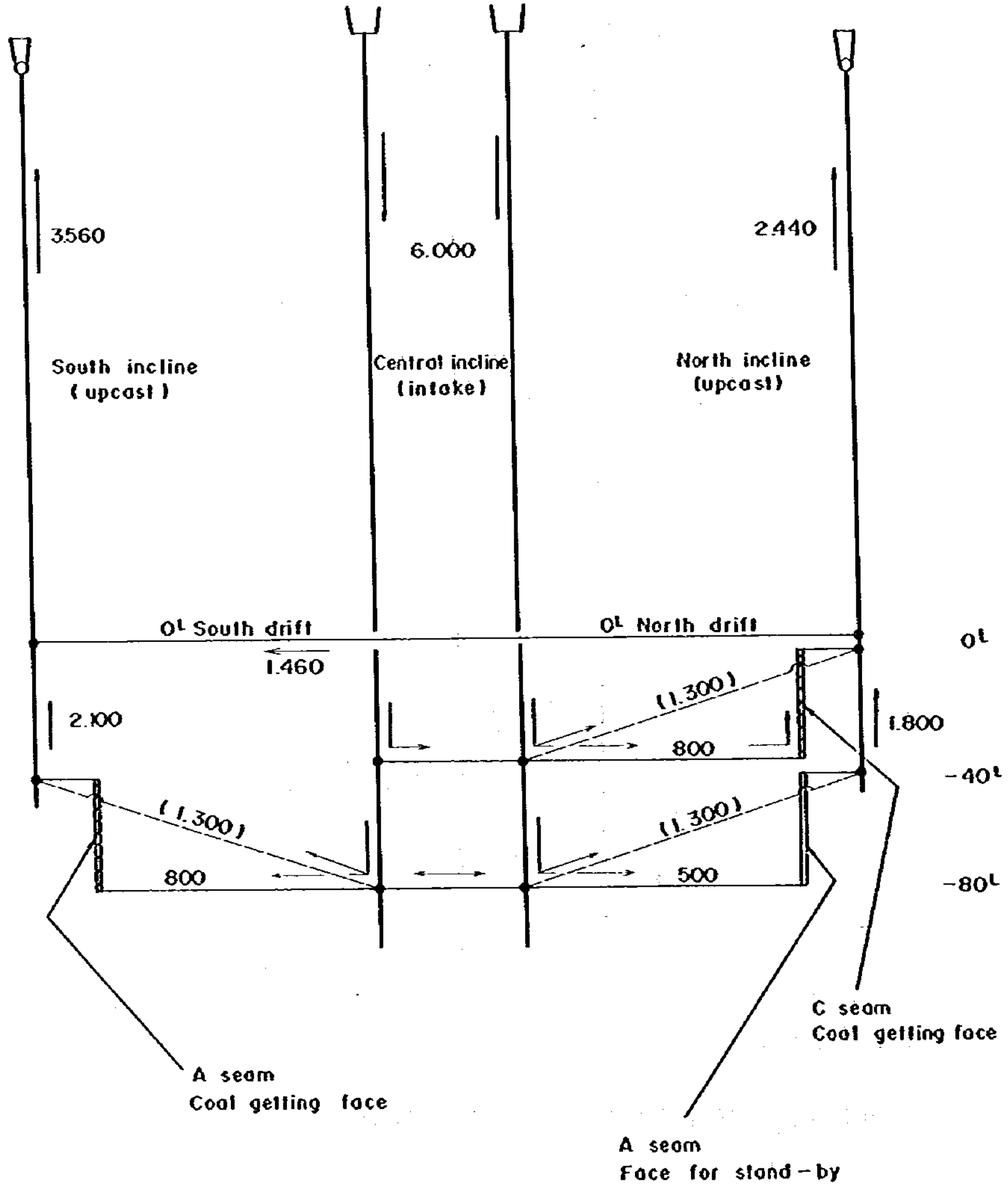


Fig. I-21 VENTILATION NETWORK (CASE-2B)

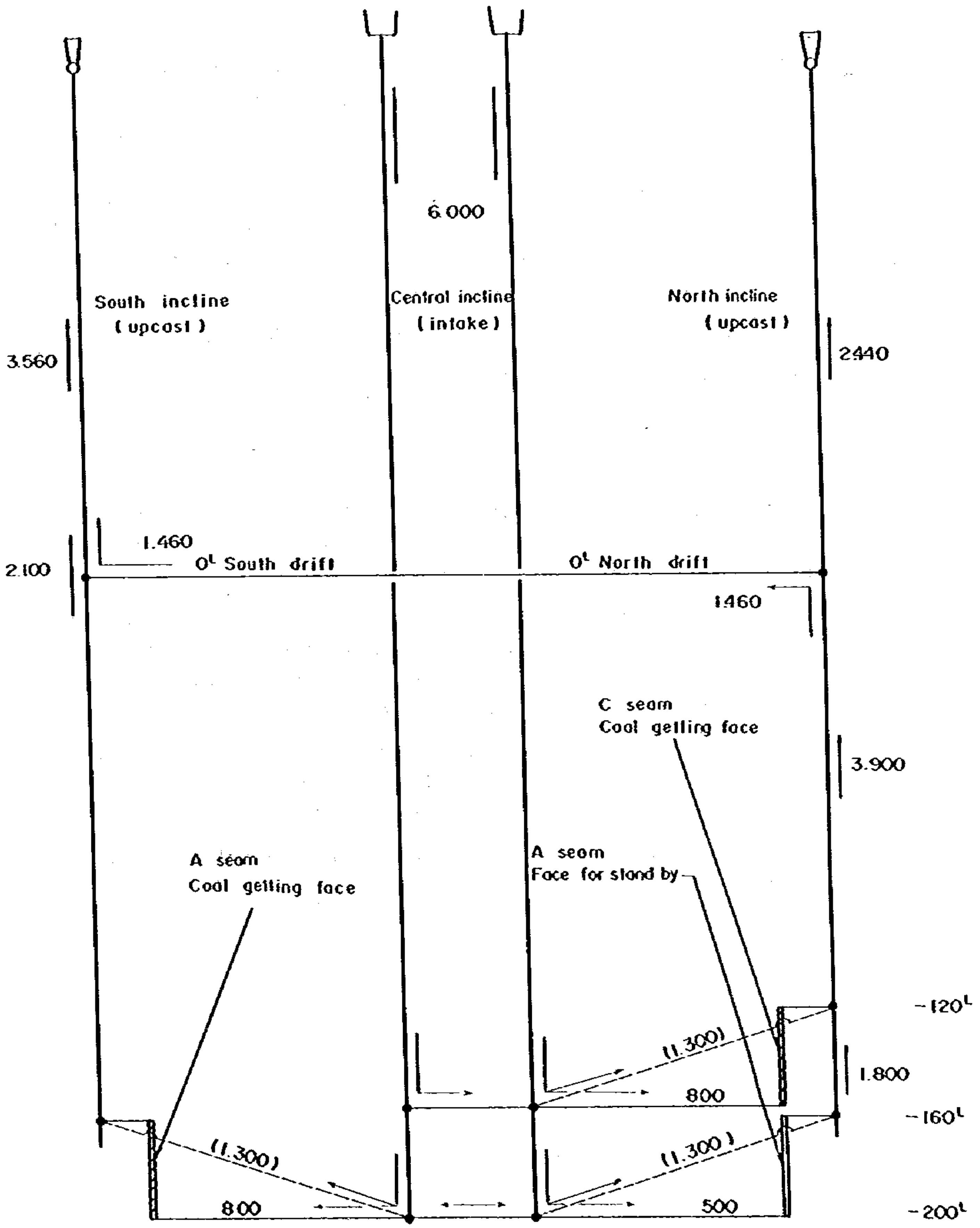


Fig.I-22 VENTILATION NETWORK (CASE-3)

In Japan, air flow of about 800 - 1,000 m^3/min for the self-advancing support face and 600 - 800 m^3/min for the single steel prop face is generally observed in the area where combustible gas exists. The ratio of required air quantity for coal getting faces is 40 to 45 percent of the gross quantity when a ventilation network is relatively simple like centralized ventilation system. When underground development is expanded and diagonal ventilation system is applied, the above ratio should be about 35 percent.

Consequently, gross air quantity for ventilation is required as follows;

Centralized ventilation system : 5,000 m^3/min
Diagonal ventilation system : 6,000 m^3/min

5.1.4 Calculation of ventilation

Required negative pressure is calculated based on the air quantity and the specific resistance of every branch unit road according to the ventilation network provided for each case.

The negative pressure which shows the maximum in total along the certain route from inlet to outlet is taken as the planned amount. To the other routes in the same network, adequate amount of resistance are added to be equal to the maximum amount of the negative pressure.

In this calculation, changes of air density caused by the variation of pressure and temperature, correction of air quantity by water evaporation etc. are not taken into consideration.

Respective specific resistances for drifts and roads are shown in the Table 1-18.

Table 1-18 Specific resistance

Item	Sectional area (m^2)	Effective sectional area (m^2)	Effective ratio	F Effective sectional area for ventilation (m^2)	U Peri-pheral length (m)	K Friction coefficient	μ Specific resistance per 1.000m	
5.0m Arch	18.02	14.82	1.0	14.82	15.0	0.0015	6.91	Central incline (for Hoist)
			0.9	13.338			9.48	Central incline (for B.C.)
4.5m Arch	14.46	12.05	1.0	12.05	13.75	0.0015	11.79	Rock drift
			0.9	10.845			16.17	
4.2m Arch	12.83	10.58	1.0	10.58	12.86	0.0015	16.29	
			0.9	9.522			22.34	Cross cut seam road
2.4m × 2.4m	8.28	5.72	0.9	5.148	9.82	0.0025	179.9	Gate road
			0.6	3.432			607.3	Single steel prop face
3.6m × 2.4m	11.61	8.28	1.0	8.28	12.24	0.0025	53.9	
			0.5	4.14			431.2	Self advancing shield support face

Note) Specific resistance per 1.000m(μ) is obtained by the following formula.

$\mu = 1.000 \times K \frac{1.000U}{F^3}$

K: Friction co-efficient U: Peripheral length (m)
F: Effective sectional area for ventilation (m)

Calculation results are indicated in the following table on the four cases mentioned in 5.1.2.

Table 1-19 Ventilation capacity

Case	Upcast	Gross air quantity (m^3/min)	Negative pressure (mmAq)	Equivalent orifice (m^2)	Ventilation system
1	Central incline	5,000	121.4	2.87	centralized
2A	Central incline	5,000	200.2	2.24	centralized
2B	South incline North incline	3,480 2,520	83.9 84.1	2.41 1.74	diagonal
3	South incline North incline	3,480 2,520	126.6 126.4	1.96 1.42	diagonal.

Note) In case of diagonal system (case 2B and case 3), gross air quantity for respective inclines are chosen by the terms of equal negative pressure for the north and south ventilation fans.

In case 2, both centralized ventilation system (A) and diagonal ventilation system (B) are taken into consideration.

In case 2A, negative pressure of fan is worried to reach 200 mmAq, which is judged to be unfavourable in general. Gross air quantity for ventilation is required to increase following the underground expansion. Ventilation network is thus to be changed to diagonal system by means of opening north and south inclines by about 10th year.

5.1.5 Ventilation equipment

Propeller fans are selected in consideration of making posible to change air quantity and negative pressure following to the expansion of underground development and of being obtainable high efficiency. Two fans with capacity of 4,000 m^3/min x 250 mmAq are suitably introduced with certain allowance.

At the beginning of the operation, two main fans are equipped in parallel, and when diagonal ventilation system starts, the fans are separately installed.

Required shaft power of fan is calculated by the following formula.

$$F = \frac{Q \cdot h}{6,120 \cdot \eta}$$

F : Shaft power (kw)

Q : Air quantity (m^3/min)

h : Pressure (mmAq)

η : Efficiency 0.8

for the referrence

Propeller	0.75 - 0.85
Turbo	0.65 - 0.75
Sirroco	0.45 - 0.55

F is obtained as 192.2, but 250 kw of electromotor power is required based on 0.85 of the electromotor efficiency.

Specification of fan

Type	:	Propeller fan, MITSUI
Air capacity	:	4,000 m^3/min
Max. negative pressure	:	250 mmAq
Pitch	:	10 pitches
Power of electromotor	:	250 kw

5.2 Water Drainage

5.2.1 Quantity of mine water

Accurate quantity of mine water by which drainage system is planned can not be estimated until underground mining operation is actually carried out. Very little mine water is observed in the current mining area, and only water for sand filling is drained off at present. However, since the mining operation is carried out below the Lunto River in the future, about 3 m^3/min of mine water is presumed for drainage system.

5.2.2 Drainage equipment

Main drainage station is placed at four levels, i.e., +160, +40, -80 and -200 L. Three pumps including one spare pump, water sump of 60 m^3 volume and automatic operation device are equipped at the drainage station. Pumps with the same specifications are adopted considering easiness of replacement. As water PH is estimated at 6.5 to 8.0, no particular considerations are taken for the prevention of corrosion etc. to the pumps and pipes.

(1) Required capacity of main pump

Friction loss of pipeline flow

$$h = \lambda \frac{v^2}{2g} \cdot \frac{l}{d}$$

h : = Friction loss (m)

λ : = Coefficient of pipe friction

250 mm ... 0.024

100 mm ... 0.026

v : Flow velocity (m/s)

g : Acceleration of gravity (9.8 m/s^2)

l : Length of pipe line (m)

d : Inner diameter of pipe

On the presumption that 3 m^3/min of mine water is drained off and the operation rate is 60%, the required capacity of a pump is obtained by the following calculation.

$$3\ m^3/min \div 0.6 \div 2 = 2.5\ m^3/min$$

Supporting an additional friction loss for bend pipe and valves equivalent to 10% of pipeline, total pipeline friction loss is calculated as follows;

$$h = 0.024 \times \frac{(1.7)^2}{2 \times 9.8} \times \frac{600 \times 1.1}{0.25} = 9.3\ m \doteqdot 10\ m$$

Consequently, required head of pump is proposed as follows. Required potential head (120 m) + Pipeline friction loss (10 m) + Certain allowance (20 m) = 150 m

Required power of pump is obtained by the following formula.

$$P = \frac{Q \cdot H}{6,120 \cdot \eta}$$

P : Water power (kw)

Q : Water capacity (l/min)

H : Discharge pressure (m)

η : Efficiency (0.6 - 0.7)

i.e.

$$P = \frac{2,500 \times 150}{6,120 \times 0.65} = 94.3\ kw$$

Taking 0.85 of motor efficiency in assumption, following power of electromotor is obtained.

$$94.3 \div 0.85 = 110.9 \doteqdot 110\ kw$$

(2) Specification of main pump

Type	: Volute
Head	: 150 m
Capacity	: 2.5 m^3/min
Electromotor power	: 110 kw

SGP pipes with victoric joints are introduced for the main drainage. Pipe with the diameter of 250 mm is proposed to be laid for main incline and of 100 mm for each level. Valves are provided in each 1,000 m. Some of pipes in the upper levels are shifted to be used for the lower levels after removing them following working places being deepened.

5.3 Miscellaneous Countermeasures for Safety

5.3.1 Combustible gas

Combustible gas is hardly detected at current operating area, Sawah Rassaw V. However, increase of the gas would be predicted following as the working area is deepened.

Thus the following basic countermeasures for the gas should be taken into consideration.

(1) Goaf of each mining panel is closed rapidly so that combustible gas is not leaked into air current.

(2) Independent ventilation to working face is maintained in order to supply fresh air.

(3) Gas automatic warning devices are equipped at around main working places.

5.3.2 Coal dust

Coal dust makes working environment unfavorable, and provide danger of explosion caused by fire and/or shock. Particularly, when combustible gas exists with coal dust, the circumstances are more serious.

Following countermeasures for coal dust are executed.

(1) Water sprays and/or sprinklers are equipped at coal getting faces, road excavation heads, and other working places.

It is also equipped to the drum shearer to depress an occurrence of coal dust during cutting coal.

(2) Besides of the above, sufficient rock powder and water shall be sprayed to the places where much of coal dust is worried to accumulate.

5.3.3 Spontaneous combustion

An occurrence of spontaneous combustion depends much on the coal properties, though it is generally caused by accumulated oxygenation heat of coal.

The countermeasures are commonly carried out as follows;

(1) Complete coal getting so as not to leave residual coal.

(2) To secure the sufficient air flow quantity.

(3) Complete isolation of the air flow.

In this project, however, (1) is difficult to practice becuase of the high variation of coal seam thickness. Accordingly, the countermeasures of spontaneous combustion are to secure the sufficient air flow quantity to the drift and road being used where no oxygenation heat accumulates, and to isolate the goaf completely where lack of oxygen prevents the oxygenation from progressing. Besides, in principle, it is important to adopt retreating longwall mining method and to control proper negative pressure of main ventilation fans.

In Japan, fly-ash slurry packing method is commonly applied for an isolation of goaf and a prevention from spontaneous combustion which contributes to as the almost perfect countermeasures.

However, this method is not taken in an investment of the project.

5.4 Safety Equipment

5.4.1 Pipelines for water sprinkler and fire extinguishing

Fresh water from the surface is supplied for sprinkler and fire extinguishing. A required water quantity is estimated as follows;

Water quantity for sprinkler

Drum shearer	40 l/min x 2 = 80 l/min
Hydraulic pump	60 l/min
Face spray	100 l/min
Total	240 l/min

Water quantity for fire extinguishing (for an emergency)
200 l/min

Pipelines with the capacity of 300 l/min is introduced. Since 7 kg/cm^2 of water pressure is reasonable at the end used, suitable valves for pressure reduction are provided in some places of the pipelines. Pipes with the inner diameter of 100 mm is proposed for main incline and drift, 50 mm for the other level roads and 25 mm for coal getting faces. Victoric joints are used for those pipes, thinking of the convenience for frequent removal. Steel water tunk with the capacity of 200 m^3 is provided near pit mouth on the surface.

5.4.2 Underground gas measure sensor

Several electric gas measure sensors are introduced in order to detect methane gas density in underground. The results with an electric modulation are indicated at the centralized monitoring room on the surface. When the indication exceeds a certain density of gas, supposedly 1.5%, alarm system is worked.

The sensors are introduced at the following places.

Return air side of coal getting face	3 places
Seam road excavation head	10 places
Total	13 places

5.4.3 Communication system

Telephones and induced wireless devices are introduced for the mine communication.

(1) Telephone system

Twenty telephone circuits on the surface and thirty circuits in underground are planned to be equipped for communication between a working place and another. Telephone sets with an explosion proof are used in underground and ordinary ones on the surface, which are operated by an automatic converter with dial system.

(2) Induced wireless devices

Induced wireless devices are expected to be used for communication and for the instruction of evacuation in case of disaster. Central exchange is placed at monitoring room in the mine office, and branch exchanges are carried by managerial persons in underground.

When necessary, communication between the central exchange and the branches is operated by push-talking. Induction wire is laid down in the whole underground. Vinyl covered wire and uncovered one are used for roads and coal getting faces respectively.

5.4.4 Centralized monitoring room

Operating conditions of following equipments in the mine are indicated automatically on a graphic board in the monitoring room;

Drum shearer	2
Incline belt conveyor	1
Main drainage equipment	5
Air compressor	4
Main air door	15
Total	27

Recorders for following machines, which are required to check their electric load and operating conditions, are to be equipped.

Main fan	Negative pressure, revolution
Compressor	Pressure, air quantity
Main drainage pump	Pressure, water quantity

6. SURFACE FACILITIES

6.1 Air Compressor

Considering mine safety, machineries used in underground are needed to be operated with the power of compressed air where density of methane gas is worried to be high. In this study, however, machineries with electric power are mainly to be adopted because combustible gas is hardly detected in the new field. Only drifting hummers and some of the equipments below are operated with compressed air.

The number of those equipments and their required air quantity are shown in the following table.

Table 1-20 Air consumption of pneumatic equipment

	Number of units	Air consumption per unit (m^3/min)	Simultaneous usage ratio	Load factor	Total air consumption (m^3/min)
Auxiliary fan	10	4.2	1.0	0.9	37.8
Leg-hammer	60	3	0.05	1.0	9
Coal pick	20	1.2	0.05	1.0	1.2
Air hoist for scraper loader	5	15	0.2	0.7	10.5
Air auger	10	2	0.05	1.0	1
Boring machine	2	15	0.1	1.0	3
Others	—	—	—	—	10
				Total	72.5

The type of air compressor is selected out of two existing types, i.e. recipro-type and rotary type, based on the required air quantity --- 104 m^3/min here due to the leakage which is estimated 30%.

Since the amount of compressed air is thus not large, the recipro-type is concluded to be preferable than the rotary-type in this project.

The former is changeable in its capacity of compressed air with high efficiency, though it is rather expensive and relatively large in scale.

Two of these air compressors are to be introduced, one is for operation and another for a spare.

Major specifications of air compressor are shown below.

Specifications of air compressor,

Type	: Balanced-opposed recipro-compressor with 2 stages
Capacity	: Piston-displacement 145 m^3/min
	Actual capacity 115 m^3/min
Pressure	: 7 kg/cm^2
Electromotor	: 670 kw
Required cooling water	: 1 m^3/min
Diameter of air outlet pipe	: 200 m
Diameter of air inlet pipe	: 300 m

Compressed air pipelines from surface to working places are laid down under the following conditions.

* Compressed air quantity in the incline is 120 m^3/min.
* While the quantity in the rock drifts and seam road is 60 m^3/min and 20 m^3/min respectively.
* Pressure loss is estimated at about 0.1 kg/cm^2 per one km of pipe length.
* Victoric type joints are used for pipe connection and the valves are equipped at every 1,000 m.

The diameter of pipes laid in the main incline, rock drift, cross cut, seam road and coal getting face are selected in consideration of the pressure loss which is obtained by the following formula.

$$Pr = \lambda \frac{v^2}{2g} \frac{x\ l}{x\ D} x\ r$$

Pr : Pressure loss caused by friction of pipe (kg/cm^2)

λ : Coefficient of friction of pipe

v : Flow velocity (m/sec)

g : Acceleration of gravity (m/sec^2)

l : Length of pipe (m)

D : Diameter of pipe (m)

r : Specific gravity (kg/m^3)

According to the above account, pipeline with diameter of 200, 150 and 100 mm are laid down in the incline, main rock drift and seam road respectively.

6.2 Electricity Equipment

Power supply to the underground and surface facilities is planned to be transmitted through the branch line which is distributed from the main power line with the capacity of 6,000 V located near the pit mouth.

Power for the surface facilities is supplied after transforming it down to 3,000 V, since little amount of voltage loss are expected due to the short distance between the transformer and major facilities which are located mainly around the pit mouth.

Three electric lines are provided for the surface. A line for the ventilation fans is independently distributed in order to prevent it from an accident caused by the power failure. The other two lines supply electricity to the transportation system and air compressors.

Power to the underground is supplied through two lines with the capacity of 6,000 V in order to prevent the voltage drop down. Required voltage and current are supplied to each working place form the underground power pack.

These two lines are provided separately for the major use of production system and of safety system. Skelton diagram of electric power system is shown in Fig. 1-23. Electricity equipments for surface and underground are shown in Table 1-21 and Table 1-22.

The maximum electricity consumption is assumed to be 50% of total estimated capacity of equipments, and 5,000 kw could be obtained based

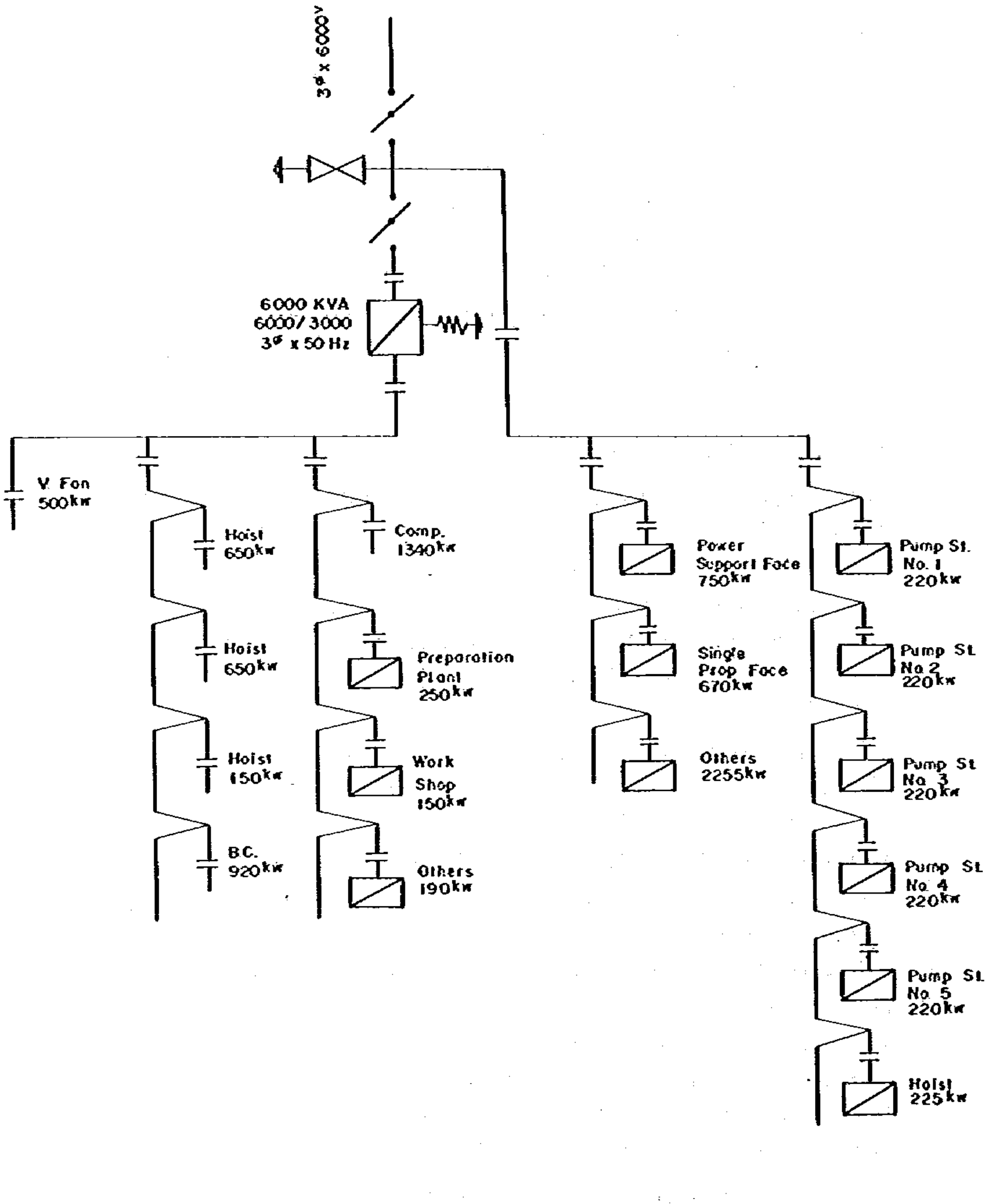


Fig. 1-23 SKELTON DIAGRAM OF ELECTRIC POWER SYSTEM

Table 1-21 Surface electricity equipment

Name	[kw] Capacity	Name	[kw] Capacity
Compressor × 2	1,340	Main ventilation fan × 2	500
Main hoists of incline × 2	1,300	Preparation plant	250
Auxiliary hoist	150	Workshop	150
Incline belt conveyor	920	Others	190
		Total	4,800

Table 1-22 Underground electricity equipment

Name	[kw] Capacity	Name	[kw] Capacity
Self-advancing shield support L.W.	750	Others in underground	3,580
Drum shearer	200	Pump station × 5	1,100
Hydraulic pump	75	Side dump loader × 17	1,190
Face conveyor	180	Battery charger × 5	200
Stage loader	70	Section pump × 10	75
Connection belt conveyor × 2	30	Local ventilation fan × 10	40
Section belt conveyor × 5	150	Lamp	50
Others	45	Hoist 75 kw × 3	225
Single prop L.W.	670	Hoist 10 kw × 40	400
Drum shearer	200	Others	300
Face conveyor	180	Total	5,000
Connection belt conveyor × 2	30		
Section belt conveyor × 5	150		
Others	110		

on the following calculation.

$$(4,800 + 5,000) \times 0.5 = 4,900 \doteqdot 5,000 \text{ kw}$$

6.3 Coal Preparation Plant

Mechanical mining system by drum shearer is proposed in this project and adoption of drum shearer is obliged to cut parts of roof and floor in relation to the coal seam condition. Consequently, coal preparation is required for separating waste rock from raw coal and securing the necessary coal quality. Coal preparation is planned to be mainly carried out with new installed jig since the data of coal preparation at Sawah Rasau V indicates that separation between coal and waste is comparatively easy. The plant is to be worked in three shifts a day, two shifts for operation and one for maintenance.

Capacity of equipment

Raw coal bin : Bin is designed so as to have the capacity of 1,000 t raw coal which is nearly equivalent to coal production per one shift.
(Raw coal exceeding 1,000 t is to be stocked in the open strage yard.)

Jig : Proposing 15 hours per day for jig operation, the capacity of jig is required as follows;

$$\frac{2,200}{15} = 146.7 \doteqdot 150 \text{ t/h}$$

Belt conveyor : 600 mm of belt width of each conveyor is applied and the speed is planned to be changeable according to the required capacity.
Following belt capacity is needed for each object.

Clean coal : 150 t/h
Waste : 50 t/h
Middling : 20 t/h

Crusher : Coal of 20 tons per hour is planned to be crushed into -30 mm by double roll typed crusher.

Waste bin : Waste bin is to have a capacity of 100 m^3 which is nearly equivalent to the total amount of waste per one shift.

Flow sheet of preparation plant is shown in Fig. 1-24. The layout and major specification of preparation equipment are shown in Fig. 1-25 and Table 1-23 respectively.

Table 1-23 Specifications of preparation equipment

Name	Number	Specification
Belt conveyor (R)	1	Capacity 400t/h, width 750mm, length 110m, 22kw, with ironpiece eliminator
Raw coal bin	1	Capacity 1,000m³, 15mϕ×8m, reinforced concrete bin
Raw coal bin feeder	1	Capacity 150t/h, 1.5kw×2, hanger type
Belt conveyor (F)	1	Capacity 150t/h, width 600mm, length 40m, lift 3m, 3.7kw
Belt feeder	1	Capacity 150t/h, width 1,800mm, length 7m, lift 2m, 7.5kw
TACUB jig	1	Two cell , Capacity 150t/h, width 3m, length 7m, valve 3.7kw, gate 0.75kw
Sieve bend	1	Wedge 2x 5mm, mesh size 0.5mm
Roll crusher	1	Double roll type, capacity 20t/h, 15kw
Belt conveyor (M_1)	1	Capacity 20t/h, width 600mm, length 13m, lift 3m, 3.7kw
Belt conveyor (M_2)	1	Capacity 20t/h, width 600mm, length 10m, lift 2m, 3.7kw
Belt conveyor (D)	1	Capacity 20t/h, width 600mm, length 30m, lift 5m, 3.7kw
Waste bin	1	Capacity 200m³, 6mϕ×8m, reinforced concrete bin
Belt conveyor (C)	1	Capacity 150t/h, width 600mm, length 50m, lift 3m, 3.7kw

6.4 Waste Disposal

Valley near the pit mouth (+275 L ~ +315 L) is expected to be filled up with waste rock as a disposal place where capacity of about 500 to 600 thousand cubic meter is expected. Before filling up, culvert is to be

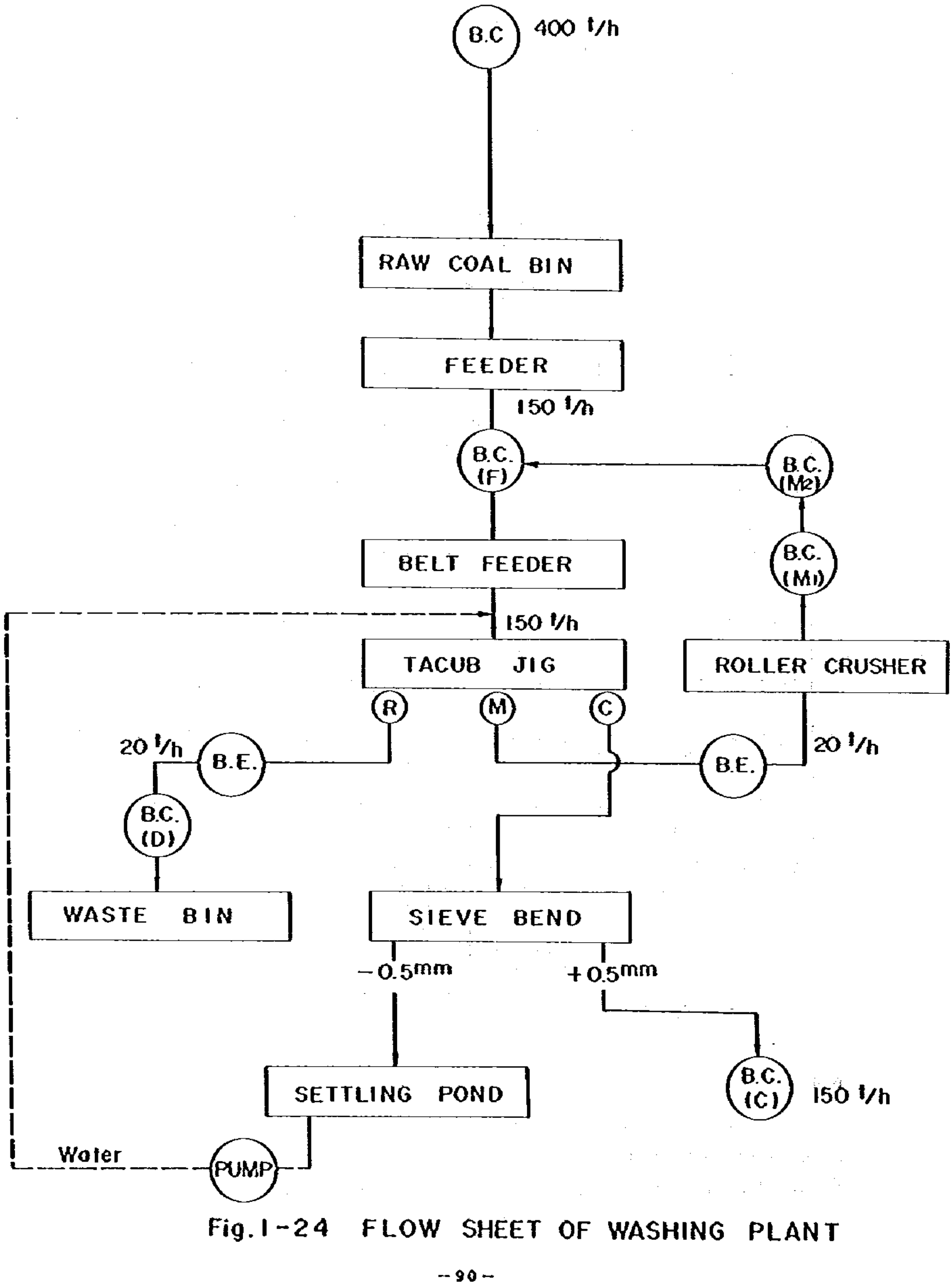


Fig.1-24 FLOW SHEET OF WASHING PLANT

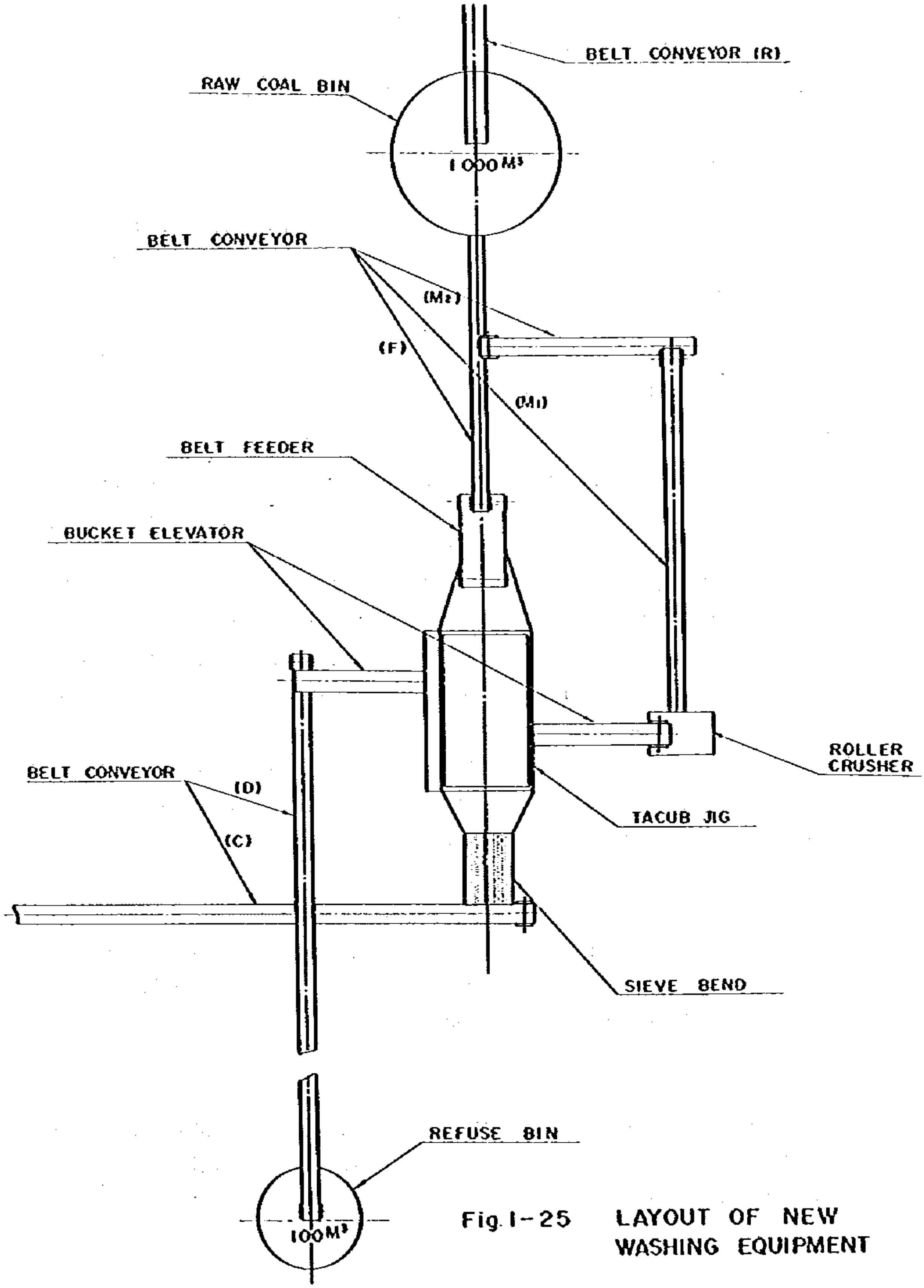


Fig. I-25 LAYOUT OF NEW WASHING EQUIPMENT laid along the bottom line of waste disposal place in order to drain sufficiently and to prevent it from water disaster.

Protector is planned to be constructed at 275 L S.L., and waste rock will be filled up between 275 L and 315 L S.L. so that surface inclination of waste pile is to be 30°. Waste rock from underground is carried by wagons to the waste disposal place, where waste is thrown by tippler and is well leveled with bulldozer. Waste from coal preparation plant is to be carried by dump truck to the same place. As the capacity of this disposal place is limited, new place(s) with proper capacity is to be chosen, if necessary, in the future.

6.5 Other Surface Facilities

6.5.1 Workshop

General maintenance of machineries will be carried out in the presnet workshop. However, the newly introduced equipments for mechanized coal getting are mainly maintained in the new workshop, which is provided around the new pit. Such maintenance is carried out before the equipments are installed in the new mining face after completion of old one.

Workshop is designed in the area of 15 m x 40 m with travelling crane. Welding machine, grinder, water pressure press etc. are introduced in the workshop for assembling and disassembling.

Sufficient space of about 500 m^2 is provided in the adjoining area to stock the machines for a mining face. Maintenance is to be carried out based on certain manual for each equipment.

6.5.2 Material stock yard

Material stock yard of about 10,000 m^2 is proposed, which comprises a yard for steel and wooden material of 5,000 m^2, for locomotive garage of 2,000 m^2 and for others of 3,000 m^2.

6.5.3 Mine office

Expected floor space of mine office is about 1,000 m^2, which includes staff room, centralized monitoring room, mine lamp charging room, rescue apparatus room and so on.

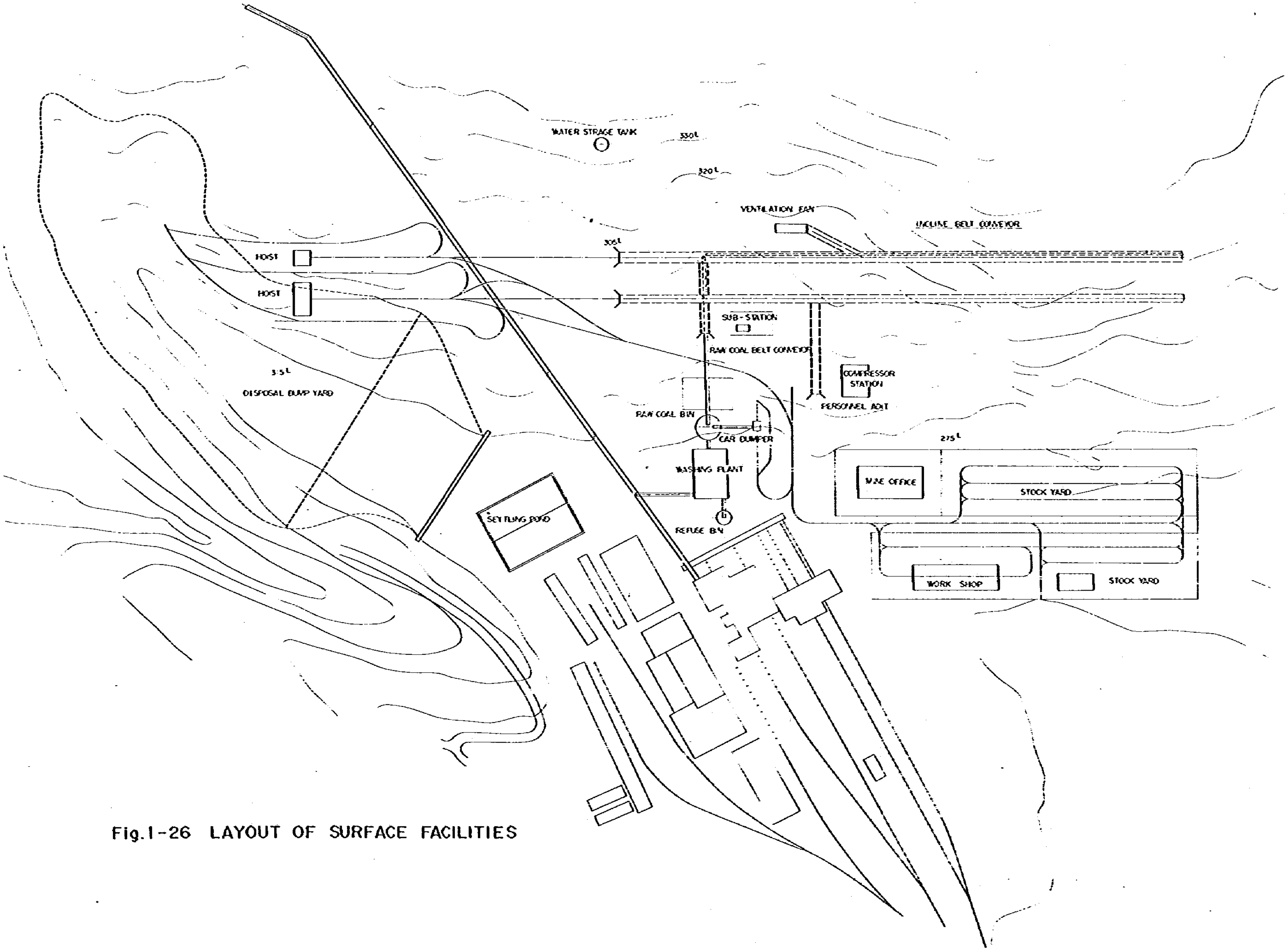


Fig.1-26 LAYOUT OF SURFACE FACILITIES

7. EQUIPMENT INVESTMENT

Plans for an introduction and investment (renewal included) of necessary equipments are made in the period from 1980 to 2005 based on the sectional programs of coal getting, drift and road excavation, transportation, mine safety and surface facilities as aforementioned. It is noted that the following prerequisites are necessary in making such plans.

* Prerequisites

1 The price of each equipment and machinery is estimated based on at the 1980 price. It is expected, however, that the prices will go up year after year due to the inflation and other economic and social factors. The price escalation, therefore, will be duly incorporated in our overall estimates that are shown in the Fourth Chapter, Economic evaluation of this report. (The same applies to the mining cost, harbor expenses, railway cost, head office expenses, selling expences and the sales.)

2 Since most of the equpments, materials and machineries except the mine timber are not available in the domestic market of Indonesia, they have to be imported from abroad. Thus, theoretically speaking, it is expected that high import duties will be imposed on those goods. However, in view of the significance in the current development plan and the huge amount of investment to this project, the Indonesian Government shall take the policy to exempt them from duties. (The same applies to the investments other than the mine investment.)
However, the spare parts incurred in the later section, the production cost, are imposed 30% duties of the C.I.F. price.

3 The price will be shown in US$ in this report, with the rate of conversion as follows.

1 US$ = 620 RP = 220 Yen

4 All the necessary funds will be financed by the parties concerned. Consequently, there will be no loan nor the interest to be paid.

5 As for the amount of investment, the purchase and the capital are to be coincident each other.

6 Contingency and engineering expenses are to be considered in the initial investment.

7.1 Schedule of Equipment Investment

The schedule of equipment investment for the development in the planning area is shown in Table 1-24. Design will be made in 1982, and the purchase of necessary equipments and the installation work will be completed during 1983 to 1987. Then, from 1988 to 2005, only the maintenance investment is expected.

7.2 Amount of Equipment Investment

7.2.1 Yearly amount of equipment investment

The outline of the yearly amount of equipment investment based on the investment schedule is shown in Table 1-25. Details of equipment investment except for the initial expenses and capitalized pit excavation, i.e. incline and major rock drift are shown in Table 1-26.

As an engineering fee and contingency, 3% and 10% of the total amount of investment in 1983-87 are earmarked respectively, which is shown in Table 1-26.

7.2.2 Capitalized pit excavation

The inclines and major rock drifts, which compose a skelton in the excavation plan as mentioned before, are considered as assets and become an object of depreciation. Consequently, the above excavation expenses can be considered as an equipment investment. Below are the bases for calculation of the expenses.

1) Yearly excavation plan

The plan for 5.0 m arch incline, 4.5 m arch incline and 4.5 m arch rock drift is shown in Table 1-13.

2) Excavating expenses

(1) Labor expenses

The number of necessary workers multiplied by average annual wages makes the labor cost.

* Change in the number of workers
  See Table 1-33.
* Average annual wages
  Average annual wages of workers of Ombilin Coal Mine are

Table 1-24 CONSTRUCTION AND INSTALLATION SCHEDULE

		1982	1983	1984	1985	1986	1987	1988	1989	1990
Engineering										
Civil construction	Preparation for mine site									
	Service road									
	Office & others									
	Surface track									
Surface	Sub-station									
	Hoist									
	Trunk belt conveyor									
	Compressor station									
	Work shop, stock yard									
	Ventilation fan									
	Washing plant									
	Disposal yard									
Underground	Tunneling									
	Drifting									
	Coal getting (prop)									
	" " (shield support)									

Table 1-25 SUMMARY OF CAPITAL INVESTMENT

				1980	1981	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	Total
Development drift	Performance	5.0m arch incline	m				360	720	720	720	720	720	720		720	720	720			720	720	720	460							9,460
		4.5m arch incline	"								480	480	480	480	480									480	380					3,260
		4.5m arch drift	"											1,200		600	600	1,800	1,800					600	590	1,200	1,200	1,200	1,200	11,990
		Total	"				360	720	720	720	1,200	1,200	1,200	1,680	1,200	1,320	1,320	1,800	1,800	720	720	720	460	1,080	970	1,200	1,200	1,200	1,200	24,710
	Cost	Labor cost	US$ 10^3				68	168	259	46	68	68	68	68	68	68	68	68	68	46	46	46	46	46	46	46	46	46	46	1,613
		Material cost: 5.0m arch incline	"				160	320	320	320	320	320	320		320	320	320			320	320	320	204							4,202
		Material cost: 4.5m arch incline	"								190	190	190	190	190									190	151					1,291
		Material cost: 4.5m arch horizontal	"											476		238	238	714	714					238	234	476	476	476	476	4,756
		Material cost: Sub total	"				160	320	320	320	510	510	510	666	510	558	558	714	714	320	320	320	204	428	382	476	476	476	476	10,248
		Total	"				228	408	579	366	578	578	578	734	578	626	626	782	782	366	366	366	250	474	428	522	522	522	522	11,861
Construction & equipment	Underground		"				1,639	3,257	6,541	2,530	6,044	61	66	377	2,118	1,895	1,838	1,732	7,169	1,921	1,101	1,429	1,988	1,544	784	875	1,006	741	1,215	47,871
	Civil construction		"				300	450	200																					950
	Surface machineries		"				1,050	1,490	1,675					700	58	58	40		32	32		58	58	1,500	40					7,591
	Coal preparation		"					500	1,445						60	80					60		80					60		2,286
	Contingency		"				379	605	1,010	253	605																			2,852
	Total		"				4,168	6,302	10,871	2,783	6,649	61	66	1,077	2,236	2,033	1,878	1,732	7,201	1,953	1,161	1,487	2,126	3,044	824	875	1,006	801	1,215	61,549
Others	Operational drift	Labor cost	"				30	73	110																					213
		Material cost: 4.2m arch drift	"				43	588	712																					1,343
		Material cost: 4.2m arch seam road	"					178	1,001																					1,179
		Material cost: 2.4m×2.4m seam road	"						78																					78
		Material cost: Sub total	"				43	766	1,791																					2,600
		Total	"				73	839	1,901																					2,813
	Engineering fee		"			428	57	91	151	38	91																			856
	Grand total		"			428	130	930	2,052	38	91																			3,669
Capital investment			"			428	4,526	7,720	13,502	3,187	7,310	639	644	1,811	2,814	2,659	2,504	2,514	7,983	2,319	1,527	1,853	2,376	3,518	1,252	1,397	1,528	1,323	1,737	77,079

Table 1-26 CAPITAL INVESTMENT SCHEDULE

(10^3US$) (1/3)

Category	Name	Specification	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993
Underground coal getting machineries	Drum shearer	KDW-200L, Diameter of drum: 1600mm, Cutting depth: 750mm, Cutting height: 2100mm				(2) 1,500	(1) 750							
	Shield support	HEMSCHEIT, Height: 1.5~2.5m, Setting load: 242t, Yeild load: 320t						(1) 5,340						
	Power pack	HAU HINKO 2At/min x 350kg/cm² x 75kw 2 pump-1 tunk						(2) 160						
	Face chain conveyor	EICKHOFF, 450t/h x 150m x 45m/min x 90kw x 2				(2) 1,180	(1) 590					(1) 590	(1) 590	(1) 590
	Stage loader	MITSUI, 400t/h x 40m x 45m/min x 75kw Advancing unit 3.7kw x 2						(1) 120						(1) 120
	Gate chain conveyor	WESTFALIA 450t/h x 60m x 50m/min x 75kw				(1) 60	(1) 60					(1) 60	(1) 60	
	Light metal hydraulic prop	NIHON KOKI, 40AL 2400 Extension 2400~1500 yield load 40t weight kg				(375) 300	(375) 300				45	45	45	45
	Kappe	NIHON KOKI, 1400mm Pin type KAC-100				(375) 45	(375) 45				7	7	7	7
	Hydraulic pump	HAU HINKO 104/min x 350kg/cm² x 35 kw				(3) 42	(2) 28							
	Gun & hose	MITSUI				(6) 2	(4) 1				2	1		
	Auger drill	FURUKAWA, AA-1				(6) 3	(4) 2					3	2	
Tunnelling machineries	Side dump loader	MITSUI EIMCO ME625 Capacity: 0.7m³		(6) 570	(7) 665	(4) 380						(5) 475	(4) 380	(4) 380
	Air leged drill	TOYO, TY-24		(40) 40	(20) 20	(20) 20			(30) 30	(30) 30	(20) 20			(30) 30
	Scraper loader	TAIKU, Double drum hoist 20PS			(3) 42	(2) 28					(3) 42	(2) 28		

Name	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	Total
Drum shearer		(1) 750	(1) 750	(1) 750									4,500
Shield support		(1) 5,340											10,680
Power pack	(2) 160							(2) 160					480
Face chain conveyor					(1) 590	(1) 590	(1) 590						5,310
Stage loader						(1) 120							360
Gate chain conveyor					(1) 60	(1) 60							360
Light metal hydraulic prop	45	45	45	45	45	45	45	45	45	45	45	45	1,320
Kappe	7	7	7	7	7	7	7	7	7	7	7	7	202
Hydraulic pump			(3) 42	(2) 28									140
Gun & hose		2	1				2	1				2	14
Auger drill				3	2					3	2		20
Side dump loader	(4) 380					(5) 475	(4) 380	(4) 380	(4) 380				4,845
Air leged drill	(20) 20	(20) 20			(30) 30	(20) 20	(20) 20			(30) 30	(20) 20	(20) 20	370
Scraper loader			(3) 42	(2) 28					(3) 42	(2) 28			280

(Note) ( ) shows number of unit.

(10³US$) 2/5)

Description Category	Description Name	Specification	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	Total
	Auger drill	FURUKAWA, AA-1			(18) 9	(12) 6				(10) 5	(10) 5	(10) 5			(10) 5	(10) 5	(10) 5			(10) 5	(10) 5	(10) 5			(10) 5	(10) 5	70
	Chain conveyor				(6) 342	(4) 228						(4) 228	(3) 171	(3) 171					(4) 228	(3) 171	(3) 171						1,710
Haulage	Gate belt conveyor	750mm x 60m x 120m/min x 400t/h x 15kw						(1) 14										(1) 14									28
	Connection belt conveyor	750mm x 15 m x 120m/min x 400t/h x 15kw				(1) 9	(1) 9	(1) 9								(1) 9	(1) 9	(1) 9									54
	Section belt conveyor	750mm x 250m x 120m/min x 400t/h x 27kw				(10) 1,150	(3) 345								(5) 575	(4) 460	(4) 460							(5) 575	(4) 460	(4) 460	4,485
	Electric battery locomotives	NIPPON YUSOKI, BL46-H Pull: 1000kg, Speed: 7km/h Traction motor: 10kw x 2			(4) 116	(5) 145						(4) 116	(5) 145						(4) 116	(5) 145						(4) 116	899
	Battery charger & holder	NIPPON YUSOKI,			(4) 64	(5) 80										(4) 64	(5) 80										288
	Mine car	TOYOHIRA, Capacity: 2m³, Rail gauge: 610mm			(90) 261	(50) 145	(50) 145						(50) 145	(50) 145	(50) 145	(40) 116					(50) 145	(50) 145	(50) 145	(40) 116			1,653
	Dump car	TOYOHIRA, Capacity: 2m³, Rail gauge: 610mm			(90) 342	(90) 342					(45) 171	(45) 171	(45) 171	(45) 171			(45) 171	(45) 171	(45) 171	(45) 171			(45) 171	(45) 171	(45) 171	(45) 171	2,736
	Chock carrier							(30) 57																			57
	Hoist (small)	1000kg x 50m/min x 10kw		(20) 600	(10) 300	(10) 300									(10) 300	(10) 300											1,800
	Hoist	MITSUI, Single drum 3500kg x 10m/min x 75kw			(2) 580	(1) 290																					870
	Personnel car	TOYOHIRA, First and emergency car: 1, Personnel car: 6, Tail end car: 1		(2) 380																							380
Power	Main cable			10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	230
	Distribution cable			10	10	10	10	10	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	140

(Note) ( ) shows number of unit.

(10³US$) 3/5)

Description: Category	Description: Name	Specification	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	Total
Communication	Telephone	NIHON DENKI,			38	9																					47
	Induced wireless telephone	KOKUSAI DENKI, XA331: 1 set, XA306: 50 sets				86							43	43						43	43						258
Safety	Section ventilation fan	MITSUI, MFA 1000P2+SC 6 300m³/min x 400mmaq x 30kw		(2) 29	(2) 29																						58
	Section ventilation fan (small)	100m³/min x 100mmaq x 2.7kw				(5) 35	(5) 35	(5) 35																			105
	Air fan	SANEI, TL-6 260m³/min x 80mmaq x 4.2 m³/min				(5) 10	(5) 10	(5) 10					(5) 10	(5) 10	(5) 10					(5) 10	(5) 10	(5) 10					90
	Gas detector for CH_4	TONKA,			95	95	95					95	95	95					95	95	95					95	950
Drainage	Section pump station				(2) 10	(3) 15	(5) 25									(4) 20	(3) 15	(3) 15									100
	Trunk pump station				240			240				240					240									240	1,200
	Drainage pipe line				34	11	11	34	11	11	11	34	11	11	11	11	34	11	11	11	11	11	11	11	11	34	357
	Hydrant and spray pipe line				50	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	155
Others	Back hoe						(3) 54				(3) 54				(3) 54				(3) 54				(3) 54				270

(Note) ( ) shows number of unit.

(10³US$) 4/5)

Description: Category	Description: Name	Specification	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	Total
Civil construction	Preparation for mine site			200	100																						300
	Service road			100	50																						150
	Office & bath-house				300	200																					500
Surface machineries	Ventilation fan	MITSUI, Axial flow propeller fan 2 stage, 6,000m³/min x 250mm x 250kw				200																					200
	Hoist	SUMITOMO, Single drum, 240m/min x 13500kg x 650kw		(2) 1,200																							1,200
	Trunk belt conveyor	400t/h x 920kw				1,200					700										1,500						3,400
	Hoist for maintenance	MITSUI, Single drum 120m/min x 6500kg x 150kw		(1) 250																							250
	Surface track			50	100																						150
	Compressor station	HITACHI, Balanced type 7kg/cm² x 136m³/min x 670kw			1,000																						1,000
	Work shop			50	200																						250
	Sub station	5000kVA 1 BANK		200																							200
	Power distribution			100	100	100																					300
	Electric battery locomotives	NIPPON YUSOKI, 8L-6AH Pull: 1000kg, Speed: 7km/h			(2) 58	(2) 58						(2) 58	(2) 58						(2) 58	(2) 58							348

(Note) ( ) shows number of unit.

(10³US$) (5/5)

Description: Category	Description: Name	Specification	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	Total
Coal preparation	Battery charger & holder	NIPPON YUSOKI			(2) 32	(2) 32										(2) 32	(2) 32										128
	Log loader					40								40								40					120
	Storage tank					45																					45
	Raw coal bin	Capacity: 1000m³				200																					200
	Washing plant				500	1,000																					1,500
	Refuse bin	100m³				40																					40
	Settling pond					65																					65
	Disposal dump yard	Capacity: 200m³				50							50							50							150
	Dump truck	10t dump truck				(2) 60						(2) 60						(2) 60							(2) 60		240
	Forkend loader					(1) 30							(1) 30							(1) 30							90
Contingency				379	605	1,010	253	105																			2,852
Engineering			428	57	91	151	38	91																			856

(Note) ( ) shows number of unit.

US$ 1,020 per person.

(See Section 9 for the calculation method.)

(2) Material expenses

Material expenses are given by the amount of excavation and unit consumption of materials.

* Unit consumption of materials

The unit consumption of materials for excavation is as shown in Table 1-27, considering the standard of Japanese coal mines.

Table 1-27 Unit consumption of materials

		5.0m arch incline	4.5m arch incline & rock drift
Explosives	(kg)	20.24	16.24
Detonators	(piece)	68	55
Steel frame	(kg)	287.0	258.3
Timber	(m^3)	0.45	0.36
Rail	(m)	4	4
Steel pipe 6 in.	(m)	1	1
Steel pipe 4 in.	(m)	2	2
Sleepers	(piece)	4	4
Others		See the note.	

Note) Standards of materials

Explosives	P.G. Dinamit 40%
Detonators	No. 8
Steel frame	I-type steel, 28.7 kg/m
Rail	22 kg/m

Other materials not included here are such as air duct, hose, cable and so on. 17% of the total sum of major materials from explosives to sleepers will be earmarked for them.

* Unit price of materials

Those materials listed in the data of P.N. Tambang Batubara show their 1979 prices, of which 10% increase will be the prices in 1980 given the current inflation and others. (Sleepers are originally shown in 1980 price.)

As for the materials not listed in the data, their prices will be reckoned by adding 30% increase to the 1980 prices in Japan considering import duties and others.

However, the prices of domestic products such as timber and sleeper are used as they are.

Explosives	: 332.31 Rp/200 g x 5 x 1.1 ÷ 620 Rp = 2.86 US$/kg
Detonators	: 512 Rp/piece x 1.1 ÷ 620 Rp = 0.91 US$/piece
Steel frame	: 98 Yen/kg x 1.3 ÷ 220 Yen = 0.58 US$/kg
Timber	: 8,000 Rp/m^3 ÷ 620 Rp = 12.90 US$/kg
Rail	: 2,364 Yen/m x 1.3 ÷ 220 Yen = 13.97 US$/m
Steel pipe	
(6 in.)	: 2,060 Yen/m x 1.3 ÷ 220 Yen = 12.17 US$/m
(4 in.)	: 1,205 Yen/m x 1.3 ÷ 220 Yen = 7.12 US$/m
(2 in.)	: 590 Yen/m x 1.3 ÷ 220 Yen = 3.49 US$/m
(1 in.)	: 365 Yen/m x 1.3 ÷ 220 Yen = 2.16 US$/m
Sleepers	: 43,750 Rp/m^3 ÷ 620 x 0.02 m^3/piece = 1.41 US$/m

Note) Although 2 in. and 1 in. steel pipes are not used in this section, they are used in the operational road excavation as described later.

(3) Others

Other costs such as power expenses and transportation expenses may also go up, but these are included in the production cost of colliery and not earmarked here.

7.2.3 Initial expenses

Properly speaking, the operational excavation expenses may be managed in the production cost of colliery, but here, the operational excavation expenses incurred during the construction period before entering the production will not be earmarked on the accrual basis. It will be redempted evenly in five years from 1986 when the production starts after being accumulated.

Also the engineering fee, considered a part of initial expenses, will be treated in the same way.

1) Operational excavation

The excavation of 1983-85 is planned in 4.2 m arch rock drift, 4.2 m arch seam road and 2.4 m x 2.4 m gate road shown in Table 1-13.

2) Excavating expenses

(1) Labor cost

Calculation method is the same as in 7.2.2.

(2) Material expenses

Calculation method is the same as in 7.2.2. However, the unit consumption of materials will employ those shown in Table 1-28.

Table 1-28 Unit consumption of materials in the operational excavation

		4.2m arch rock drift	4.2m arch seam road	2.4m×2.4m seam road
Explosives (kg)		14.41	3.53	2.28
Detonators (piece)		48	18	12
Steel frame (kg)		241.1	241.1	
Timber (m^3)		0.32	0.53	1.02
Rail (m)		4.0	2.0	
Pipe	4 in. (m)	1		
	2 in.. (m)	2	1	1
	1 in. (m)		2	2
Sleepers (piece)		4	2	
Others		17% of the above amount.		

8. PERSONNEL ALLOCATION

8.1 Basic Consideration

Personnel allocation was understood at the previous meeting in Jakarta with the project counterparts as follows;

* 200 trained workers are able to move from the current operating area to the new pit.
* Any number of additional workers needed for the new pit operation will be employed whenever required.
* No additional workers are expected for the current operation area including surface personnel, i.e. haulage, workshop, mine office etc.

However, not sticking to the above number of trained workers, 200, the most appropriate number of them would be transfered to the new project in consideration to the proper balance of personnel allocation in the whole Ombilin Coal Mine. Personnel allocation plan in this section indicates only the number of workers for the new pit.

8.2 Personnel Allocation of New Pit

8.2.1 Coal getting

Basic personnel allocation for coal getting operation at self-advancing shield support face and at single steel prop face is shown in 2.2.4 of this chapter. Actual number of workers required for coal getting is shown in Table 1-29.

Table 1-29 Actual number of coal getting workers

	Coal output plan (t/d)		Actual number of workers for coal getting		
	Self-advancing shield support face	Single steel prop face	Self-advancing shield support face	Single steel prop face	Total
1986		530		54	54
1987		530×2		54×2	108
1988	1,060	530	42	54	96
1989∿2005	1,590	530	42	54	96

8.2.2 Drift and road heading

Actual number of workers for heading is obtained by the personnel needed for each excavation head mentioned in 3.2.3 and number of headings. Actual required number of workers for the development and the operation is indicated in Table 1-30.

8.2.3 Road maintenance

With reference to the collieries in Japan which have similar underground conditions, it is assumed that five workers per day are required for coal output of 100 tons on R.O.M. basis. Actual number of workers for road maintenance is shown in Table 1-31.

Table 1-30 ACTUAL NUMBER OF HEADING WORKERS

	5.0m Arch incline		4.5m Arch incline		4.5m Arch horizontal rock drift		4.2m Arch horizontal rock drift		4.2m Arch horizontal seam road		2.4mx2.4m horizontal seam road		Total		
	Number of faces	Number of workers	Number of faces	Number of workers	Number of faces	Number of workers	Number of faces	Number of workers	Number of faces	Number of workers	Number of faces	Number of workers	Capitalized development	Operation	total
	(Capitalized development)														
1983	2	40					1	20					60		60
1984	2	40					3	60	3	48			148		148
1985	2	40					3	60	6	96	2	32	228		228
	(Capitalized development)						(Operation)								
1986	2	40					1	20	6	96	4	64	40	180	220
1987	2	40	1	20			1	20	6	96	4	64	60	180	240
1988	2	40	1	20			1	20	6	96	4	64	60	180	240
1989	2	40	1	20			1	20	6	96	4	64	60	180	240
1990			1	20	2	40	1	20	6	96	4	64	60	180	240
1991	2	40	1	20			1	20	6	96	4	64	60	180	240
1992	2	40			1	20	1	20	6	96	4	64	60	180	240
1993	2	40			1	20	1	20	6	96	4	64	60	180	240
1994					3	60	1	20	6	96	4	64	60	180	240
1995					3	60	1	20	6	96	4	64	60	180	240
1996	2	40					1	20	6	96	4	64	40	180	220
1997	2	40					1	20	6	96	4	64	40	180	220
1998	2	40					1	20	6	96	4	64	40	180	220
1999	2	40					1	20	6	96	4	64	40	180	220
2000			1	20	1	20	1	20	6	96	4	64	40	180	220
2001			1	20	1	20	1	20	6	96	4	64	40	180	220
2002					2	40	1	20	6	96	4	64	40	180	220
2003					2	40	1	20	6	96	4	64	40	180	220
2004					2	40	1	20	6	96	4	64	40	180	220
2005					2	40	1	20	6	96	4	64	40	180	220

Table 1-31 Actual number of road maintenance workers

	Actual number of workers
1986	27
1987	53
1988	80
1989-2005	106

8.2.4 Underground indirect workers

30% of underground workers are regarded as indirect personnel such as haulage workers, mechanics, electrician, safety guards and the like.

8.2.5 Registered number of workers

According to the attendance ratio 90%, which is the actual record in the current operating area, the registered number of workers in each section is determined as shown in Table 1-32.

8.3 Number of Workers Moved from Current Operating Area

Required workers for the new pit are obtained from the current operating area and/or by the new employment.

After a careful examination, 265 workers are decided to be relocated from the old to the new pit. This number exceeds 200 which was previously agreed with the Indonesian counterpart, but our examination reveals that the former is more appropriate for the balance of personnel allocation in the whole Ombilin Mine.

The relocation, however, is only available from the underground and surface sections and not from the open cut one in the current working organization.

Thus, 165 workers from the underground and 100 from the surface section are proposed to be relocated in this project.

Personnel allocation in whole Ombilin Coal Mine is shown in Fig. 1-33.

Table 1-32 REGISTERED NUMBER OF WORKERS IN NEW PIT

	Development		Operation											
			Drifts and roads		Coal getting		Road maintenance		Indirect		Sub total	Sub total registered	Total	Total registered
	actual	registered	actual	registered	actual	registered	actual	registered	actual	registered	actual		actual	
1983	60	67							26	29	26	29	86	96
1984	148	165							64	72	64	72	212	237
1985	228	254							97	108	97	108	325	362
1986	40	45	180	200	54	60	27	30	129	144	390	434	430	479
1987	60	67	180	200	108	120	53	59	172	192	513	571	573	638
1988	60	67	180	200	96	107	80	39	179	199	535	595	595	662
1989	60	67	180	200	96	107	106	118	190	212	572	637	632	704
1990	60	67	180	200	96	107	106	118	190	212	572	637	632	704
1991	60	67	180	200	96	107	106	118	190	212	572	637	632	704
1992	60	67	180	200	96	107	106	118	190	212	572	637	632	704
1993	60	67	180	200	96	107	106	118	190	212	572	637	632	704
1994	60	67	180	200	96	107	106	118	190	212	572	637	632	704
1995	60	67	180	200	96	107	106	118	190	212	572	637	632	704
1996∼2005	40	45	180	200	96	107	106	118	181	202	563	627	603	672

Table 1-33 PERSONNEL ALLOCATION IN WHOLE OMBILIN COAL MINE

			1980	1981	1982	1983	1984	1985	1986	1987	1988	1989~1995	1996~2005
Operation	Existing area	Under ground	565	565	565	517	428	400	400	400	400	400	400
		Open cut	152	152	152	152	152	152	152	152	152	152	152
		Sub total	717	717	717	669	580	552	552	552	552	552	552
	New pit	Under ground				29	72	108	434	571	595	637	627
	Surface		802	802	802	754	702	702	702	702	702	702	702
	Total		1,519	1,519	1,519	1,452	1,354	1,362	1,688	1,825	1,849	1,891	1,881
New pit development						67	165	254	45	67	67	67	45
Grand total			1,519	1,519	1,519	1,519	1,519	1,616	1,733	1,892	1,916	1,958	1,926
Additional personnel required			0	0	0	0	0	97	214	373	397	439	407

9. PRODUCTION COST

Although the mining project of the current study is designed only for the Planning area, the final economic evaluation aims at an overall assessment including the current operating area.

Therefore, a study on the cost of production in the current operating area is also required as well as the one in the planning area.

In mapping out the future plan for the cost of production in the current underground operating area, the plan in the Planning area provided a good example.

As for the surface, attentions are given to present a viable plan in consideration of the current cost of production.

The following prerequisites were considered.

1 In Ombilin coal mine at present, depreciation expenses are not included in the cost of production, but in this plan such expenses is earmarked in it.

2 The plan covers the period from 1980 to 2005, and the operating days are 300 days each year.

3 Prices are calculated on the basis of the 1980 ones throughout the period, with no consideration given to the current inflation.

4 Prices are shown in US$, with the rate of conversion as follows;

1 US$ = 620 Rp = 220 Yen

9.1 Planning Area

Only the direct cost in the underground will be calculated based on the mining plan. Therefore, the indirect cost which may arise in the underground and the surface operation will not be earmarked for cost in the Planning area. Such cost will be covered by the general expenses.

9.1.1 Labor cost

1) Number of workers

Workers required for the development and operation in the Planning area will be moved from the current operating area and/or newly employed. (Refer to section 8, Personnel allocation.)

The changes of number are shown in Table 1-33.

2) Labor cost

The labor cost is worked out by multiplying the number of workers by an average annual income. An average annual income per person regardless of their position and sex will be employed.
According to the data of P.N. Tambang Batubara, the number of workers as of the end of July 1980 was 1,519, and the total amount of wages paid during the period from January to June 1980 was 480,220,000 Rp. Thus, simply making it double, 960,440,000 Rp is assumed to be the annual total wages. From here, an annual average income per person in 1980 is calculated as follows.

960,440,000 Rp ÷ 1,519 ≑ 632,284 Rp

Since the conversion rate of Rp against US$ is 1 US$ = 620 Rp, it will be,

632,284 Rp ÷ 620 ≑ 1,020 US$

In the above annual income, an allowance in kind and overtime is included.

Table 1-34 Labor cost by year in the Planning area

(10^3US$)

Year	1986	1987	1988	1989-1995	1996-2005
Labor cost	443	582	607	650	640

Note) This concerns only the labor cost of underground workers in the Planning area, as the surface workers are considered in the plan of the current operating area.

9.1.2 Maintenance cost

1) Coal getting

As there may be no problem to think that the mechanized mining with self advancing support requires little amount of materials, the material expenses are regarded as zero in this study. What concerns here is thus single steel prop face alone, though, the overhaul expenses of self advancing support and drum shearers may arise. Those expenses are described later.

The cost is calculated based on the following unit consumption of materials and prices.

* Unit consumption of materials

The consumption of materials used for one ton of the coal output is as follows, based on the standard of single steel prop face in the Japanese coal mines.

Explosives	0.020 kg
Timber	0.028 m^3
Others	Equivalent to 20% of the total sum of explosives and timber

The coal mentioned here refers to the raw coal.

* Unit price of materials

The same values used in section 7 are used.

Explosives	2.86 US$/kg
Timber	12.90 US$/$m^3$

Table 1-35 Material expenses for coal getting

Year		1986	1987	1988	1989-2005
Coal output (raw coal)(10^3t)		159	318	159	159
Material expenses (10^3US$)	Explosives	9	18	9	9
	Timber	57	115	57	57
	Others	13	27	13	13
	Total	79	160	79	79

Note) Coal output is based on single steel prop face. (Refer to Table 1-5.)

2) Drift and road excavation

The excavation amount is shown in Table 1-13.
The material expenses in a year are shown in Table 1-36 in the same manner as mentioned in section 7.

Table 1-36 Material expenses for excavation

			4.2m arch rock drift	4.2m arch seam road	2.4m×2.4m seam road	Total
Performance (m)			620	4,040	4,220	8,880
Material expenses (10^3US$)	Explosives		26	41	28	95
	Detonators		27	66	46	139
	Steel frame		87	565		652
	Timber		3	28	56	87
	Rail		35	113		148
	Pipe	4 in.	4			4
		2 in.	4	14	15	33
		1 in.		17	18	35
	Sleepers		3	11		14
	Others		32	145	28	205
	Total		221	1,000	191	1,412

3) Spare parts

As for the spare parts necessary for repair and maintenance of the equipments, annual expenditure is calculated based on the repair factor of each equipment. As the spare parts must be all imported, 30% import duties are imposed.

Table 1-37 Estimation of annual spare parts cost

(10^3US$)

		Repair factor (%)	Amount of investment	Spare parts	Import duty	Total
Coal getting machineries	Drum shearer	12	2,250	270	81	351
	Shield support	10	5,340	534	160	694
	Power pack	10	160	16	5	21
	Face chain conveyor	5	1,770	89	26	115
	Stage loader	5	120	6	2	8
	Gate chain conveyor	5	120	6	2	8
	Light metal hydraulic prop	2	600	12	4	16
	Kappe	0	90			
	Hydraulic pump	5	70	4	1	5
	Gun and hose	0	3			
	Auger drill	5	5			
Tunnelling machineries	Side dump loader	10	1,615	162	49	211
	Air leged drill	20	80	16	5	21
	Scraper loader	5	70	4	1	5
	Auger drill	5	15	1		1
	Chain conveyor	5	570	29	9	38
Haulage	Gate belt conveyor	2	14			
	Connection belt conveyor	2	27	1		1
	Section belt conveyor	3	1,495	45	13	58
	Electric battery locomotives	3	261	8	2	10
	Battery charger & holder	1	144	1		1
	Mine car	3	551	17	5	22
	Dump car	5	684	34	10	44
	Chock carrier	3	57	2	1	3

Table 1-37 (Continued)

(10³US$)

		Repair factor (%)	Amount of investment	Spare parts	Improt duty	Total
Haulage	Hoist (small)	2	1,200	24	7	31
	Hoist	2	870	17	5	22
	Personnel car	1	380	4	1	5
Power	Main cable	0				
	Distribution cable	0				
Communi-cation	Telephone	1	47			
	Induced wireless telephone	10	86	9	3	12
Safety	Section ventilation fan	3	58	2	1	3
	Section ventilation fan (small)	2	105	2	1	3
	Air fan	2	30	1		1
	Gas detector for CH_4	6	285	17	5	22
Drainage	Section pump station	2	50	1		1
	Trunk pump station	2	480	10	3	13
	Drainage pipeline	0				
	Hydrant and spray pipeline	0				
Oth-er	Back hoe	10	54	5	2	7
Civil con-struction	Preparation for mine site	0	300			
	Service road	2	150	3	1	4
	Office and bath house	2	500	10	3	13
Surface machineries	Ventilation fan	2	200	4	1	5
	Hoist	2	1,200	24	7	31
	Trunk belt conveyor	2	1,200	24	7	31
	Hoist for maintenance	2	250	5	2	7
	Surface truck	5	150	8	2	10

Table 1-37 (Continued)

(10³US$)

		Repair factor (%)	Amount of investment	Spare parts	Import duty	Total
Surface machineries	Compressor station	2	1,000	20	6	26
	Work shop	2	250	5	2	7
	Sub station	1	200	2	1	3
	Power distribution	2	300	6	2	8
	Electric battery locomotives	3	116	3	1	4
	Battery charger & holder	1	64	1		1
	Log loader	5	40	2	1	3
	Storage tank	0	45			
Coal preparation	Raw coal bin	0	200			
	Washing plant	5	1,500	75	22	97
	Refuse bin	0	40			
	Settling pond	0	65			
	Disposal dump yard	0	50			
	Dump truck	7	60	4	1	5
	Loader	7	30	2	1	3
Total				1,547	464	2,011

4) Others

Other expenses besides coal getting and excavating should arise in the underground, which will be calculated in the light of the underground expenses in the current operating area. Such expenses in the current operating area are estimated at 113,000 US$ when producing 225,000 tons of coal.

Thus, in the Planning area, in case of 600,000 tons of coal output, it will be,

113,000 US$ x 600,000 tons/225,000 tons ≑ 305,000 US$.

During the period of increase in coal output, the expenses shall go up 10% each.

Table 1-38 Other expeneses

(10^3 US$)

Year	1986	1987	1988	1989-2005
Amount	229	252	277	305

9.1.3 Depreciation and amortization

The objects of depreciation are equipment, installation cost, major rock excavations and initial expenses as described in section 7. The following prerequisites are given in calculating depreciation expenses.

* Prerequisites

1 In principle, all the assets begin depreciating in the next year of acquisition. However, as for the acquisition before 1986 when the production starts, 1987 shall be the first year for depreciation.

2 The depreciation method is uniformly the fixed instalment method, by which the balance shall be zero in the final year of the life of equipment.

3 The life of each equipment was determined within the framework shown by P.N. Tambang Batubara, taking into account the conditions and circumstances where the equipment is used. It is,

therefore, not necessarily consistent with the legal life of each equipment.

4 Contingency shall not fall in the category of depreciation.

5 Although the major rock drifts seem to be treated in the cost of production in the current Ombilin Coal Mine, they are considered as depreciable assets in this study, and to depreciate with the same life as buildings.

6 The initial expenses will depreciate evenly in five years from 1986.

The amount of depreciation calculated by year based on the above prerequisites is shown in Table 1-39.

9.2 Current Operating Area

The cost of production in the current operating area and the management section will be calculated on the basis of "A Review of Mining Studies for the Expansion of Ombilin Coal Mines", hereinafter referred to as "Review-report", which was prepared by the staff of Ombilin Coal Mine in December, 1978.

Table 1-39 DEPRECIATION COST OF PLANNING AREA

(10³US$)

Item			Item / Year	Durable year	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	Total
Development drift				20	65	83	112	141	170	206	235	267	298	337	376	394	413	431	444	467	489	515	541	567	6,551
Construction & equipment	Underground	Coal getting machineries	Drum shearer	10	150	225	225	225	225	225	225	225	225	225	150	150	225	225	225	225	225	225	225	225	4,275
			Shield support	8			668	668	668	668	667	667	667	667	668	668	668	668	667	667	667	667	—	—	10,680
			Power pack	7			23	23	23	23	23	23	22	23	23	23	23	23	23	22	23	23	23	23	412
			Face chain conveyor	7	169	253	253	253	252	252	337	253	252	252	252	253	254	254	253	252	252	252	253	254	5,055
			Stage loader	6			20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	360
			Gate chain conveyor	7	9	18	18	18	17	16	25	26	18	18	17	16	16	17	18	18	18	17	16	16	352
			Light metal hydraulic prop	5	60	120	120	120	120	69	18	27	36	45	45	45	45	45	45	45	45	45	45	45	1,185
			Kappe	5	9	18	18	18	18	10	2	3	5	7	7	7	7	7	7	7	7	7	7	7	178
			Hydraulic pump	11	4	7	7	7	7	7	7	6	6	5	5	6	7	7	7	7	7	7	6	6	128
			Gun & hose	5	1	2				1	2				1	2				1	2				12
			Auger drill	6	1	2	2				1	2	2				1	2	2				1	2	18
		Tunnelling machineries	Side dump loader	8	202	202	202	202	202	202	261	308	155	203	203	203	202	201	200	201	202	203	203	203	4,160
			Air leged drill	5	16	16	16	22	28	16	16	16	16	14	14	14	14	14	14	14	14	14	14	14	316
			Scraper loader	6	12	12	12	12	11	18	12	12	12	12	11	11	12	12	12	12	11	11	12	12	241
			Auger drill	5	3	3	3	3	4	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	60
			Chain conveyor	7	81	81	81	81	82	82	115	57	81	81	80	81	82	83	82	81	81	80	81	82	1,635
		Hauling	Gate belt conveyor	10			1	1	1	1	1	1	2	2	2	2	1	1	1	1	1	1	2	2	24
			Connection belt conveyor	10	1	2	3	3	3	3	3	3	3	2	2	2	3	3	3	3	3	3	3	2	53
			Section belt conveyor	9	128	166	166	166	166	166	166	166	166	103	115	166	166	166	166	166	166	165	167	167	3,169
			Electric battery locomotives	7	37	37	37	37	37	38	55	38	38	38	37	37	36	37	38	38	38	37	37	36	763
			Battery charger & holder	11	13	13	13	13	13	13	13	13	13	13	20	13	13	13	13	13	13	13	13	13	267
			Mine car	8	51	69	69	69	69	69	68	86	55	54	69	69	69	69	69	69	69	68	69	69	1,348

Table 1- 39 (Continued)

(10³US$)

Item			Year	Durable year	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	Total
Construction & equipment	Underground	Railage	Dump car	6	114	114	114	114	114	143	58	87	115	114	113	113	114	115	115	114	113	113	114	115	2,226
			Chock carrier	20			3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	54
			Hoist (small)	10	120	120	120	120	120	120	120	120	120	150	60	60	60	60	60	60	60	60	60	60	1,830
			Hoist	20	44	44	44	44	44	44	44	44	44	44	43	43	43	43	43	43	43	43	43	43	870
			Personnel car	20	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	19	380
		Power	Main cable	20	2	3	4	5	6	7	8	9	10	11	11	11	11	11	11	11	11	11	11	11	175
			Distribution cable	20	2	3	4	5	6	7	8	9	9	9	8	7	6	6	6	6	6	6	6	6	125
		Communication	Telephone	20	2	2	2	2	2	2	2	2	2	2	2	2	2	3	3	3	3	3	3	3	47
			Induced wireless telephone	7	12	12	12	12	12	13	13	6	12	12	12	12	12	13	13	12	12	12	12	12	238
		Safety	Section ventilation fan	20	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	58
			Section ventilation fan (small)	20	2	4	6	6	6	6	6	6	6	6	6	6	6	6	6	5	4	3	3	3	102
			Air fan	7	1	2	3	3	4	5	6	5	4	3	3	4	5	6	5	4	3	3	4	5	78
			Gas detector for CH_4	7	27	41	41	41	41	40	55	41	42	42	41	40	39	40	41	42	42	41	40	39	816
		Drainage	Section pump station	11	2	4	4	4	4	4	4	4	5	6	8	6	4	4	4	4	4	4	5	5	89
			Trunk pump station	9	27	27	54	54	54	54	80	80	79	53	53	54	53	53	53	27	27	26	26	26	960
			Drainage pipe line	20	2	3	5	6	7	8	10	11	12	13	14	16	16	17	17	16	15	16	16	17	237
			Hydrant and spray pipe line	20	3	4	5	6	7	8	8	8	8	8	8	8	8	8	8	7	7	7	7	7	140
		Cutters	Back hoe	4		14	14	13	13	14	14	13	13	14	14	13	13	14	14	13	13	14	14	13	257
	Civil construction		Preparation for mine site	20	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	300
			Service road	20	8	8	8	8	8	8	8	8	8	8	7	7	7	7	7	7	7	7	7	7	150
			Office & bath house	20	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	25	500
	Surface machineries		Ventilation fan	20	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	200
			Hoist	20	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	1,200
			Trunk belt conveyor	20	60	60	60	60	60	95	95	95	95	95	95	95	95	95	95	170	170	170	170	170	2,100

Table 1-39 (Continued)

(10³US$)

Item			Durable year	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	Total
Construction & equipment	Surface facilities	Hoist for maintenance	20	13	13	13	13	13	13	13	13	13	13	12	12	12	12	12	12	12	12	12	12	250
		Surface track	20	8	8	8	8	8	8	8	8	8	8	7	7	7	7	7	7	7	7	7	7	150
		Compressor station	20	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	50	1,000
		Work shop	20	13	13	13	13	13	13	13	13	13	13	12	12	12	12	12	12	12	12	12	12	250
		Sub station	20	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	200
		Power distribution	20	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	300
		Electric battery locomotives	7	17	17	17	17	16	16	24	16	16	16	16	17	18	17	16	16	16	16	17	18	339
		Battery charger & holder	11	6	6	6	6	6	6	6	6	6	5	8	6	6	6	6	6	6	6	6	5	120
		Log loader	8	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	100
		Storage tank	20	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	35
	Coal preparation	Raw coal bin	20	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	200
		Washing plant	20	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	75	1,500
		Refuse bin	20	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	40
		Settling pond	20	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2	2	55
		Disposal dump yard	7	7	7	7	7	7	7	8	7	7	7	7	7	7	8	7	7	7	7	7	7	142
		Dump truck	6	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	200
		Loader	7	4	4	4	4	4	5	5	4	4	4	4	4	5	5	4	4	4	4	4	5	85
	Sub total			1,757	2,093	2,842	2,849	2,858	2,864	2,973	2,887	2,765	2,760	2,625	2,670	2,745	2,755	2,750	2,708	2,786	2,779	2,120	2,123	52,789
Others	Miscellaneous pre-production works		5	563	563	563	562	562																2,813
	Engineering		5	171	171	171	171	172																856
	Sub total			734	734	734	733	734																3,669
Total				2,556	2,910	3,688	3,723	3,762	3,070	3,208	3,154	3,063	3,097	3,001	3,064	3,158	3,186	3,194	3,255	3,275	3,294	2,661	2,690	63,009

9.2.1 Mining area

There may be no necessity of mentioning the mining area in detail, since they are listed in "Review report".

1) Open-cut mining area

    (1) Tanah Hitam
    (2) Kandi
    (3) Sapan Dalam

2) Underground mining area

    (1) Sawah Rassaw V and Sawah Luhung
    (2) Tanah Hitam
    (3) Kandi

9.2.2 Coal output

Coal output in the current operating area is estimated at 400,000 tons of clean coal from both the open-cut and underground in 1984 when the full production starts.

Beginning with 150,000 tons at first (1980), the output is expected to increase year after year.

In "Review report", the yield of clean coal was estimated at 90%, but here in this study it is employed the rate of 85%.

Table 1-40 Coal output in the current operating area

(10^3t)

	Year	1980	1981	1982	1983	1984	1985	1986-2005
Open cut	Raw coal	78	102	155	155	206	206	206
	Clean coal	66	87	132	132	175	175	175
Under-ground	Raw coal	99	133	198	198	265	265	265
	Clean coal	84	113	168	168	225	225	225
Total	Raw coal	177	235	353	353	471	471	471
	Clean coal	150	200	300	300	400	400	400

Note) Self advancing support will be introduced into the underground mining area in 1982 for the trial operation.

9.2.3 Mining system

1) Open-cut mining

The mining method using conventional equipment such as wheel scraper, shovel, truck etc. is adopted. For the coal transportation, 8 ton coal trucks are used. The stripping ratio will be expected 7 m^3 of overburden to 1 ton of raw coal in average.

2) Underground mining

The longwall coal getting method is adopted at present using mainly air pick hammers, but in a few years it is planned to introduce self-advancing support into the same area to realize the mechanization of mining operation using together with the drum shearer.

At the same time, the present sandfilling method will be abolished to be replaced by the non-filling method.

The present manual loading system at the road excavation will be also replaced by the mechanized loading system using the side dump loader from 1981.

9.2.4 Equipment investment

As a matter of course, the area covered by this plan encompasses all the investments, new and maintenance (renewal), in the current Ombilin Coal Mines (including the management section) as a whole. Equipment investment in the underground and equipment and maintenance investment in the open-cut area are discussed here.

1) Open-cut mining equipment

Necessary equipment is given for the production of 175,000 t.p.y. in "Review report" added by one wheel scraper to be purchased. Renewal plan by year is shown in Table 1-41. The time of purchase of the equipments presently owned was inferentially drawn out from the data of P.N. Tambang Batubara.

The transferrable equipments, though they may be rebuilt or overhauled, are also earmarked for the investment.

Table 1-41 INVESTMENT PLAN FOR CURRENT OPERATING AREA (Open cut)

		Number of unit	Purchase time	Unit price	Capital cost	Operating hrs./year	Rebuild				Replacement	
		Unit	Year	US$	US$	Hours	Hours	Years	Formulation	US$/unit	Hours	Years
Overburden removal	35 sht dump truck	7	1976(3 units) 1977(4 units)	202,456	1,417,192	2,310	12,000	5	50%(202,456-90,850)	55,803	20,000	9
	Bulldozer(300HP)	5	1974	215,286	1,076,430	2,543	10,000	3	40%x215,286	86,114	16,000	5
	Wheel scraper(15m³)	3	1977(2 units) 1980(1 unit)	219,803	659,409	2,860	12,000	4	50%(219,803-9,680)	105,062	20,000	7
	Hydraulic shovel(2.5m³)	1	1976	312,439	312,439	2,970	12,500	4	45%x312,439	140,598	25,000	8
	F.E. wheel loader	1	1976	196,886	196,886	2,970	10,000	3	40%(196,886-16,940)	71,978	16,000	5
	Blast hole rig	1	1977	215,656	215,656	—	—	—	—	—	—	10
Coal handling	F.E. crawler loader	1	1979	114,837	114,837	2,300	10,000	4	50%x114,837	57,419	16,000	7
	Bulldozer(62HP)	1	1976	33,630	33,630	1,200	10,000	8	35%x33,630	11,771	16,000	13
	Coal truck(8t)	4	1977	62,058	248,232	2,563	12,000	5	40%(62,058-2,251)	23,923	20,000	8
General	Flat form truck(7t)	1	1975	47,878	47,878	1,500	—	—	—	—	16,000	11
	Personnel carrier	1	1976	46,377	46,377	—	—	—	—	—	—	10
	Tipper truck	1	1976	47,878	47,878	—	—	—	—	—	—	10
	Land rovers	4	1976	15,539	62,156	—	—	—	—	—	—	8
	10t mobile crane	1	1976	127,596	127,596	—	—	—	—	—	—	15
	Lubrication truck	1	1976	71,126	71,126	1,500	—	—	—	—	16,000	11
	Water sprinkler truck	1	1976	51,819	51,819	1,500	—	—	—	—	12,000	8
	Fuel truck	1	1976	55,156	55,156	1,500	—	—	—	—	15,000	11
	Road grader	1	1976	120,152	120,152	2,915	10,000	3	40%(120,152-13,264)	42,755	16,000	5
	Maintenance work shop	1	1977	421,445	421,445	—	—	—	—	—	—	40
	Total				5,386,575							

Investment amount (10³US$)

	1980	1981	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	Total
35 sht dump truck	287	383				607	810			287	383				607	810			287	383				607	810		6,261
Bulldozer(300HP)		431			1,076		431			1,076		431			1,076		431			1,076		431			1,076		7,535
Wheel scraper(15m³)	430			105	440			430			105	440			430			105	440			430			105	440	3,900
Hydraulic shovel(2.5m³)		312		72	141					312			141					312			141					312	1,671
F.E. wheel loader		197					197		72			197		72			197		72			197		72			1,345
Blast hole rig								216										216									432
F.E. crawler loader			57				115			57				115			57				115			57			573
Bulldozer(62HP)				12						34							12						34				92
Coal truck(8t)		56				248				56				248				55				248				56	1,128
Flat form truck(7t)							48											48									96
Personnel carrier							46										46										92
Tipper truck							48										48										96
Land rovers					62								62								62						186
10t mobile crane															128												128
Lubrication truck								71											71								142
Water sprinkler truck					52								52							52							156
Fuel truck								55											55								110
Road grader		120		43			120		43			120		43			120		43			120		43			815
Maintenance work shop																											
Total	717	1,539	57	232	1,771	855	1,815	772	115	1,862	488	1,188	255	478	2,241	810	911	777	968	1,511	318	1,426	34	779	1,991	848	24,758

Table 1-42 INVESTMENT PLAN FOR CURRENT OPERATING AREA (Underground)

(10^3US$) ($10^3US)

Description: Category	Description: Name	Specification	1981	1982	1983	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	Total
Coal getting machineries	Drum shearer	EDW-200L, Diameter of drum 1600mm, Cutting depth: 750mm Cutting height: 2800mm		(1) 750										(1) 750										(1) 750				2,250
	Shield support	HEMSCHEIT, Height: 1.5~2.5m, Setting load: 242t, Yeild load: 320t		(1) 5,340								(1) 5,340								(1) 5,340								16,020
	Power pack	HAU 28l/min x 350kg/m² x 55kw 2 pump-1 tank		(2) 160							(2) 160							(2) 160							(2) 160			640
	Face chain conveyor	EICKHOFF, 450t/h x 150m x 45m/min x 90kw x 2		(1) 590							(1) 590							(1) 590							(1) 590			2,360
	Stage loader	MITSUI, 450t/h x 40m x 45m/min x 75kw Advancing unit 3.7kw x 2		(1) 120						(1) 120						(1) 120						(1) 120						480
	Gate chain conveyor	WESTFALIA, 450t/h x 50m x 50m/min x 75kw		(1) 60							(1) 60							(1) 60							(1) 60			240
Tunnelling machineries	Side dump loader	MITSUI CIMCO, ME 632, Capacity: 0.6m³	(5) 300								(5) 300								(5) 300								(5) 300	1,200
Haulage	Section belt conveyor	750mm x 250m x 120m/min x 400t/h x 27kw		(10) 1,150									(1) 115	(1) 115	(1) 115							(1) 115	(1) 115	(1) 115				1,840
Total			300	8,170						120	1,110	5,340	115	865	115	120		810	300	5,340		235	115	865	810		300	25,030

(Note) ( ) shows number of unit.