CHAPTER 3 CONCEPTS OF MINE PLANNING AND DEVELOPMENT SCHEDULE

3-1 Mine Planning (see Fig. 3-1, 3-2, 3-3 and 3-4)

From the view-point of mine planning, the investigated area of approximately 26 km^2 was divided into three blocks, namely western, central and eastern block respectively. Each block is geologically separated from other blocks by faults/folds and the mining depth as well as the number of mineable coal seams are pretty different between these blocks. Underground mining will be proposed only for the central block where the number of mineable coal seams is the least, the main seam to be mined is the deepest in all blocks and the mining ratio (m^3 of overburden/tonne of as-received coal) may be more than fifteen to one.

In other two blocks, the total mining ratio was calculated at approximately eleven to one and open pit mining will be selected because of the increase in mineable tonnage and easiness of excavation. The surface conditions are suitable for open pit mining owing to the rock desert areas and little rainfall. Accordingly, the mining activity will be reasonable planned mainly by open pit mining both in the western block and the eastern block.

The geological conditions of each block informed by geological investigations are shown in Table 3-1.

The solid volume of overburden indicated in Table 3-1 is the bulk m^3 of overburden limited vertically by the boundary of coal seam to be mined, and the stripping ratio indicated in the same table does not represent the actual mining ratio of open pit mining.

The figures in Table 3-1 were calculated for the spots with stripping ratio of less than 10 to 1 for the western block, less than 17 to 1 for the central block and less than 12 to 1 for the eastern block.

The depths of mineable coal seams vary between the blocks as follows:

Block	Mineable seams	Range in depth
Western block	No. 5, No. 3, No. 2, No. 1	33 m to 85 m
Central block	No. 3	72 m to 85 m
"	No. 1	85 m to 123 m
Eastern block	No. 3, No. 2, No. 1	45 m to 91 m

The principles for basic planning were following:

a) Main production unit will be open pit.

b) Underground mining will be planned only for the central block and the annual production rate is estimated approximately at 200,000 tonnes of air-dried basis coal.

c) The production capacity for the western block which is comparatively favorable for open pit mining will be about 50 percent of total mine capacity.

d) The mining equipment to be selected are to be world-wide popular. easy in maintenance work, parts supply and multipurpose type. Specially the erection and construction at mine site must be simply and economically performed.

The main notions of underground mining method involved in plan are as follows;

Mine opening:	Inclined shaft driven through rock formations toward the coal seam
Main roadway:	Central main level entries in seam
Panel development:	Driving of gate entries for longwall mining panels
Main road heading method:	Blasting, side-tippling loaders and mine car haulage system
Main roadway support:	Arched steel support
Main coal excavating method:	Longwall retreat system, blasting, pick hammer and hand loading
Support in coal face:	Hydraulic steel props and link bars
Face haulage machine:	Double chain armoured conveyor
Gate haulage machine:	Battery locomotive, mine cars
Haulage in main roadway:	Battery locomotive, mine cars
Inclined shaft haulage:	Winding machine for mine car haulage
Ventilation system:	Centralized ventilation with a main exhaust fan

The main notions of open pit mining method involved are as follows;

Overburden stripping:	Overburden removal with shovel/truck
Shape of pit:	Multiple stage bench cut
Coal excavation:	Multiple seam mining with hydraulic excavator/ bulldozer
Preparation of overburden stripping:	Drilling with electric rotary drill and ANFO blasting
Partings removal:	Drilling with hydraulic drill, ANFO blasting, loading and hauling with wheel roader/truck or scraper/dozer
Coal loading and hauling:	Hydraulic excavator and truck
Restoration of mined-out area:	Filling with overburden hauled and grading with bulldozer and scraper

3-2 Production Scale

The mine production rate was determined, in addition to the principles for the basic plan mentionned in Paragraph 3-1, by the major considerations were as follows;

a) A production capacity should be sustained for about 30 years.

b) Each of three blocks contains some barren or thin seam areas and the number of boreholes penetrated in mineable seams is only more or less than ten in each block.

c) Concerning the western block, the northern, eastern and southern parts are limited with the thin seam area or barren area, accordingly the possible extension of mineable coal seams could be expected only toward the western outside of the investigated area.

d) Concerning the central block which is limited with faults in western and eastern parts and barren area in the northern part, the production rate to be planned as underground mining method is restricted. The annual production capacity of 200,000 tonnes air-dried basis coal was reasonably determined by the main reason of mineable reserves and also economical of equipment and mineable reserves and the technical capabilities in mining industry in Pakistan.

e) Concerning the eastern block which is limited with fault/fold in the western part, coal seams are thin in the eastern part and the northern part is barren. Concequently the area to be planned for open pit mining becomes long and narrow shape in the direction of SN and the mining ratio is estimated to be remarkably high. Because of unstable geological conditions, the share in production rate of the eastern block to total mine production capacity cannot be dominant.

As a result of above-mentionned considerations, it was planned that the western block should contribute the production as 50 percent of total mine production. Basically the production scale of each block was determined as follows: (Air-dried basis coal without out-of-seam dilution)

Western block (open pit mining)	500,000 tonnes/year
Central block (underground mining)	200,000 tonnes/year
Eastern block (open pit mining)	300,000 tonnes/year
Total	1,000,000 tonnes/year

In order to design a open pit in the western block as above-mentionned production for 30 years operation period, it was necessary to suppose that the geological conditions of the western boundary limit of investigated area may possibly extend westward.

3-3 Planning of Development and Operation

3-3-1 Fundamental Data for Planning

The following data were prepared prior to commencing the planning:

a) Topographic maps and drillhole information

b) Various kinds of detailed iso-value maps for each block

c) Isopach maps and geological structure maps revised after mining considerations for each coal seam

d) Iso-value maps of overburden/minable coal ratio

The generation of isopach maps and volumes are time-consuming and routine. As a result, computerized techniques were investigated which take input data from a topographic map and a set of drill logs, and create a series of grid surfaces representative of the ground surface and the top and bottom of each seam of coal. The grids also are used to calculate the volumes of coal and overburden associated with any designated area of the investigated area. The grid data can be visualized as maps by using of a X-Y plotter. For underground mine planning, the all dimensions of roadways, longwall panels and specific gravity are used as input data in order to calculate the volume of waste material and tonnage to be excavated.

3-3-2 Underground Mine in the Central Block: "UNDERGROUND MINE"

The portal is planned that shaft bottom will be situated at the center of total recoverable reserves. The inclined shaft will terminate at the point of intersection of No. 1 seam and at the bottom of inclined shaft two main entries will be developed in No. 1 seam in the direction of SN. The maximum gradient of main entries was determined to the 1 : 150, considering the battery locomotive haulage. Because of the restriction in gradient of haulage tracks, the main entry is obliged to bend in the northern and southern area. Two gate entries will be developed from the main entries and a longwall-mining panel will be prepared between the two gate entries. Considering the countermeasure against the spontaneous combustion, the whole mining area is to be divided into several sub-blocks with a certain number of longwall panels and the safety coal pillar of 50 m to 100 m in width is provided between the sub-blocks. For the purpose of excavating the No. 3 seam, inclined shaft is driven upward from the No. 1 seam to the mining area in No. 3 seam. The coal tonnage of each longwall panel was calculated by computer based on the input data of panel dimension, working height, specific gravity and others derived from the drillhole logs.

3-3-3 Open Pit in the Western Block: "WEST OPEN PIT"

The initial box cut will be planned to be located close to the line JT 16, JT 7, JT 9 and PS 19. The working benches are cut as long as possible and advance gradually westward. A central access ramp splits the east side of the pit. The first overburden excavating will be carried out near JT 16 because of the best stripping condition with shallow overburden and low mining ratio.

The pit design will be done using various kinds of iso-value line maps of 1 : 5,000 scale. Bench cut stripes were drawn parallel to the initial box cut line on the No. 5 seam level to make the mining area spreading westward. Based on the stripes drawn on the No. 5 seam level, many kinds of input data were made and the computer simulation was performed using a mathematic model. This simulation made the output data as the amount of overburden and partings and coal tonnage for each bench cut.

Then annual amount of air-dried basis coal of 500,000 tonnes was allocated for every year, and in order to make the annual amount of overburden to be removed as uniform as possible and a certain amount of in-pit coal inventory, the allocation of the amount of overburden to be prepared and removed was revised.

To determine the required number of trucks for overburden hauling and coal hauling, a computerized shovel/truck simulation model was applied.

3-3-4 Open Pit in the Eastern Block: "EAST OPEN PIT"

The intial box cut will be planned to be located parallel to the southern boundary of the licenced area and the first bench cut commence, at the site close to JT 50. The working benches cannot be prepared so long as that of the West open pit because of narrow mining area. The benches advance gradually northward varying the length according to the mining ratio. The procedure of mining plan will be same as that of the West open pit, but the basic bench cut stripes were drawn on the No. 3 seam.

3-3-5 Surface Facilities

The surface facilities for the mine complex will be planned to be well concentrated near the portal of underground mine. The plan involves the railway facilities, work shops, store houses, explosives magazine, offices, substation, roads and other support facilities.

3-3-6 Coal Preparation Plant

For the benefication of coal to be transported to the power station, a preparation plant will be planned to be situated near the underground mine. The run-of-mine coal from open pit hauled in trucks to the ROM hopper and that of underground mine in hauled by mine cars and conveyors to the ROM hopper. The ROM was designed to be treated in hand picking method and waste rock of 4 % will be eliminated by manpower. After the treatment of hand picking the clean coal will be loaded into wagons. The raw coal feed rate is designed at 400 t/h.

3-3-7 Coal Transportation

For the purpose of shipping the clean coal to the power station at Jamshore, the railway was planned to be situated near the underground mine. The run-of-mine coal from open pit will be hauled in trucks to the ROM hopper and that of underground mine hauled by mine cars and

3-3-8 Coal Production and its Design Criteria

The construction period was scheduled to terminate at the end of 1985 and the operating period begins consequently in 1986. Calculated the coal tonnages (air-dried basis and no out-of seam dilution) to be excavated and based on these values, ROM coal and the clean coal tonnages were computed with the following parameters:

Area	Underground mining	Open pit mining
Volumetric calculation	By panels and roadways	By unit bench cuts
Specific gravity of coal	(No. 3 seam) 1.44 (No. 1 seam) 1.41 – 1.64	(West O/P No. 5 seam) 1.55 (West O/P Nos. 3,2,1 seam) 1.54 (East O/P No. 3 seam) 1.45 (East O/P Nos. 2,1 seam) 1.53
Geological safety factor	70 %	80 %
Recovery ratio	95 %	90 %
Out-of-seam dilution	4 %	5 %
Min. mineable thickness of coal	0.90 m	0.50 m
Min. thickness of parting selected	–	0.30 m

In addition to above-mentionned parameters, the total moisture of ROM was estimated at 25 percent and the amount of waste eliminated by hand picking in the preparation plant was defined to the 4 weight percent of total ROM. The tonnages of ROM and clean coal and the quantity of clean coal are summarized in Table 3-3.

3-3-9 Geomechanical Aspects in Design of Lakhra Coal Mine (Refer ANNEX 2, 3)

Geomechanical aspects were carried out for the general consideration of displacement and stress distribution in the vicinity of inclined shaft under the effect of excavation as well as in the vicinity of main entries in seam under the effect of approaching seam excavation in underground mine.

Geomechanical considerations also were applied for the slope stability problem of open pit by same manner as for underground mine structure.

For this purpose, the "Rigid Body Spring Models (RBSM)" was applied as a practical method of limit analysis of general solids and structures by using these new discrete models. Displacement and stress distribution were graphically represented by using computer graphics to obtain the visual concept of the variation in geomechanical behavior of underground structure and surrounding rock.

After examination of proposed problems by the information derived from computerized analysis, it is concluded that the planned structures in underground mine as well as the pit slope designed in open pit mining can be always sustained in safe condition.

3-3-10 Power Supply in Initial Developing Stage

In this initial stage of the mine development, the mine substation is not working yet. Therefore, it is assumed that for the time being temporary power supply for the stripping work in the east and west open pit, 2,000 kVA each, utilizing huge electrically operated shovels and drills will be provided by WAPDA.

However, two units of mobile type diesel generators, 100 kVA 400 V 3 phase each, one in operation and one standby, will be provided at the underground mine portal area to supply the power for the construction and/or installation of the mine substation and workshops concentrated within the portal area and for the lighting for night work and the worker's camp. After the completion of the regular power receiving and distribution system, the generators will be used for emergency purposes, especially for water drainage in the open pit in case of power failure in heavy rain expected every rainy season.

3-3-11 Power Supply in Operating Stage

The total installed motor capacity in all over the mine at this stage is anticipated at approximately 7,000 kW. It is assumed that the power will be supplied by WARDA at 33 kV 3 phase 50 Hz to the mine substation, 4,000 kVA 33 kV/3.3 kV, installed at approximate load centre of the colliery by means of adequate capacity of the overhead line and then, after measuring, distributed to each area either at 33 kV or 3.3 kV. The installed motor capacity and distribution voltage in each area are shown in following table.

Area	Installed Motor Capacity (kW)	Distribution Voltage (kW)
East Open Pit	1,760	33
West Open Pit	1,760	33
Underground	930	3.3
Preparation Plant with Conveyor Belt & Tippler for U/G Raw Coal	600	3.3
Surface	1,950	3.3
Total	7,000	

It is assumed that 33 kV transmission line for the mine and 3.3 kV and/or 400 V distribution line in Khanot area for the residential area and water supply facilities will be provided by WAPDA at cost the same as the other infrastructure, such as the bungalows, colonies and rest house, etc.

3-3-12 Mine Substation

Mine substation will be provided at approximate load Centre of the colliery. The substation, 4,000 kVA 33 kV/3.3 kV 3 phase 50 Hz, will be outdoor type and of dustproof construction except supervisory board and other switchgear, etc. and will be able to operate under severe conditions of high ambient temperature and sand storms.

The supervisory board and all switchgear as well as the storage batteries, etc. are installed in the control room provided in the adjascent power house, and from here the substation will be supervised and controlled by the shift working operators.

3-3-13 Power Distribution Facilities

Overhead power lines will be used for the surface power distribution system. For the open pit mine the power will be supplied at 33 kV to eliminate the voltage drop caused by extremely high load current and long distribution distance, as well as to minimize the used cable size. On the other hand, the power for the underground mine and surface facilities will be distributed at 3.3 kV because of their relatively short distribution distances.

For the underground power distribution either armoured cables or cabtyre cables will be used within the underground mine.

3-3-14 Emergency Power Supply Equipment

It is recommendable to provide the emergency generator in the mine to supply the power only for such safety facilities as the main ventilation fan and lighting equipment in case of power failure from the power source side the same as in the other PMDC coal mines.

Therefore, in addition to aforementioned 2 x 100 kVA of generators, 2 no.s of 500 kVA 3.3 kV 3 phase diesel generators will be installed in the power house constructed adjascent to the mine substation. All switchgear of the mine substation is also installed in the power house for the convenience of the changeover of loads in case of power failure.

3-3-15 Communication Facilities

The colliery communication is carried out by means of telephone, wireless and inductive radio systems.

For the general communication inside and outside of the colliery, 100 circuits of the internal line capacity private telephone system with 2 circuits of office lines will be utilized.

In addition, the wireless system for the open pit mine, the inductive radio system for the underground mine and the paging system for the coal preparation plant will also be provided.

3-3-16 Lighting Facilities

The administration office and other surface facilities will be equipped with adequate lighting fixtures and, in addition, proper outdoor lights will be provided for the underground mine portal area, outdoor stock yard and major roads within the mine, etc.

The mobile lighting towers in open pit and the explosion-proof lighting fixtures for underground will also be installed to improve the working conditions, as well as to prevent accidents.

3-4 Coal Production and Recoverable Coal Reserves (see Fig. 3-5)

The life of underground mine will be ended in 2012 with the operating period of 27 years. Total tonnage of underground mine will be estimated at 5,119,000 tonnes (air-dried basis). This figure is equivalent to about 80 percent of the recoverable geological reserves. The main reason of the reduction in tonnage is that the mineable area was decreased to be about 80 percent of the originally calculated area. The shape of whole area is not suitable to the systematic underground layout with high recovery ratio and the eastern part where loose sand formations cover the coal seams was rejected to be planned.

The East open pit will be planned for a period of 30 years operation. The mining ratio becomes remarkably high in northern part and the whole shape of mining area is generally long and narrow. Therefore, it is not expectable to plan a large scaled open pit with systematic stripping layout. To make up for these disadvantages in open pit mining, it must be necessarily supposed that the geological conditions of the western boundary limit of investigation may extend westward in planning of the West open pit. However the mining area was decreased to be 80 percent of the originally calculated area. In planning of the East open pit, the mining area will be only 30 percent of the originally calculated area.

The total mining area was resulted to be 58 percent of the initially proposed area and the total production tonnage was estimated to be 52 percent of the original calculation of mineable reserves.

The proportion of underground mine open pit in production tonnage was 12/88 in the phase of geological studies and 19/81 in the phase of mine planning. The mineable reserves for unit area was consequently decreased to be about 90 percent of that in the geological studies. By reason of above-mentioned situation, the decrease in production was reflected directly by the decrease in mineable area.

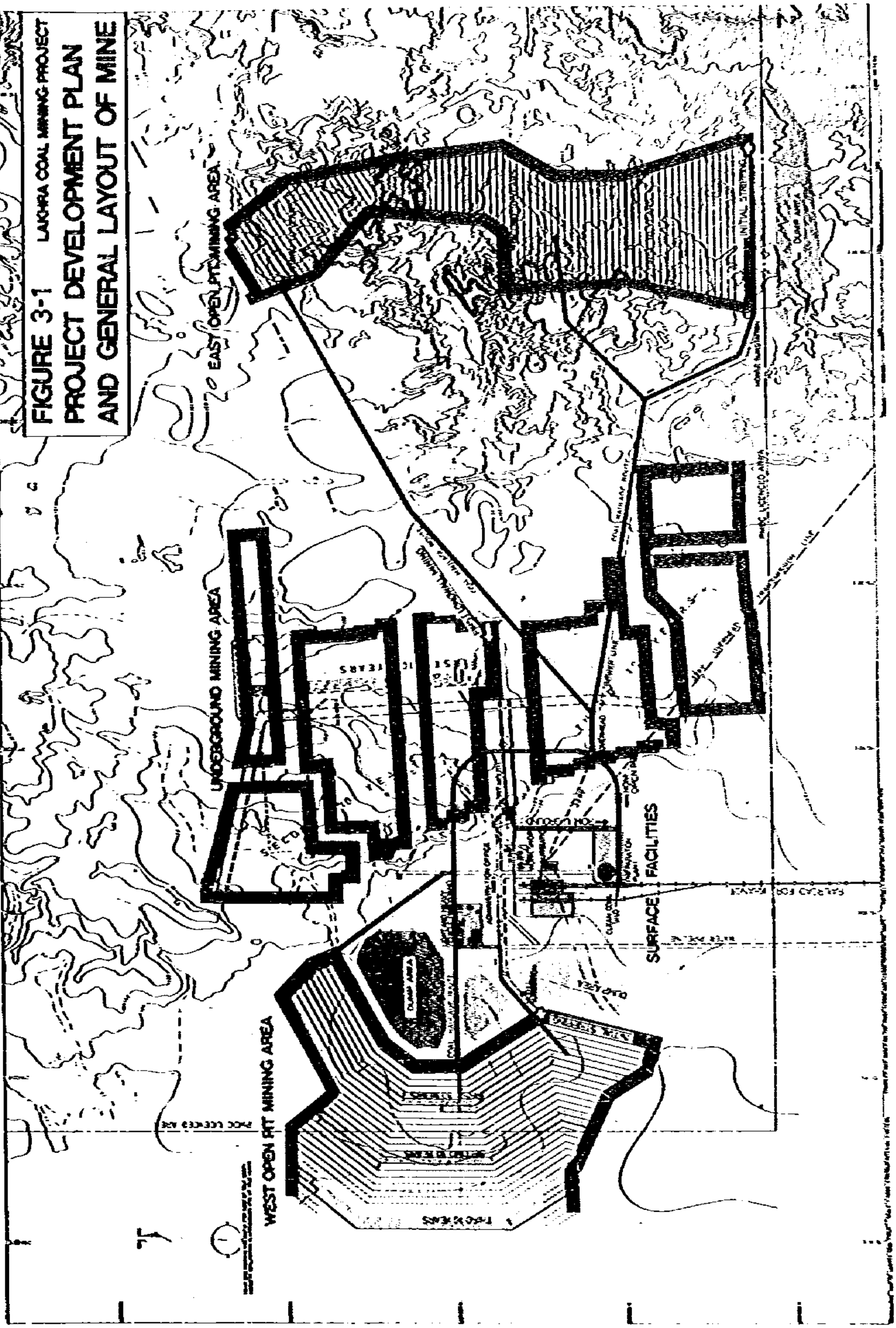


FIGURE 3-1 LAKHRA COAL MINING PROJECT
PROJECT DEVELOPMENT PLAN AND GENERAL LAYOUT OF MINE

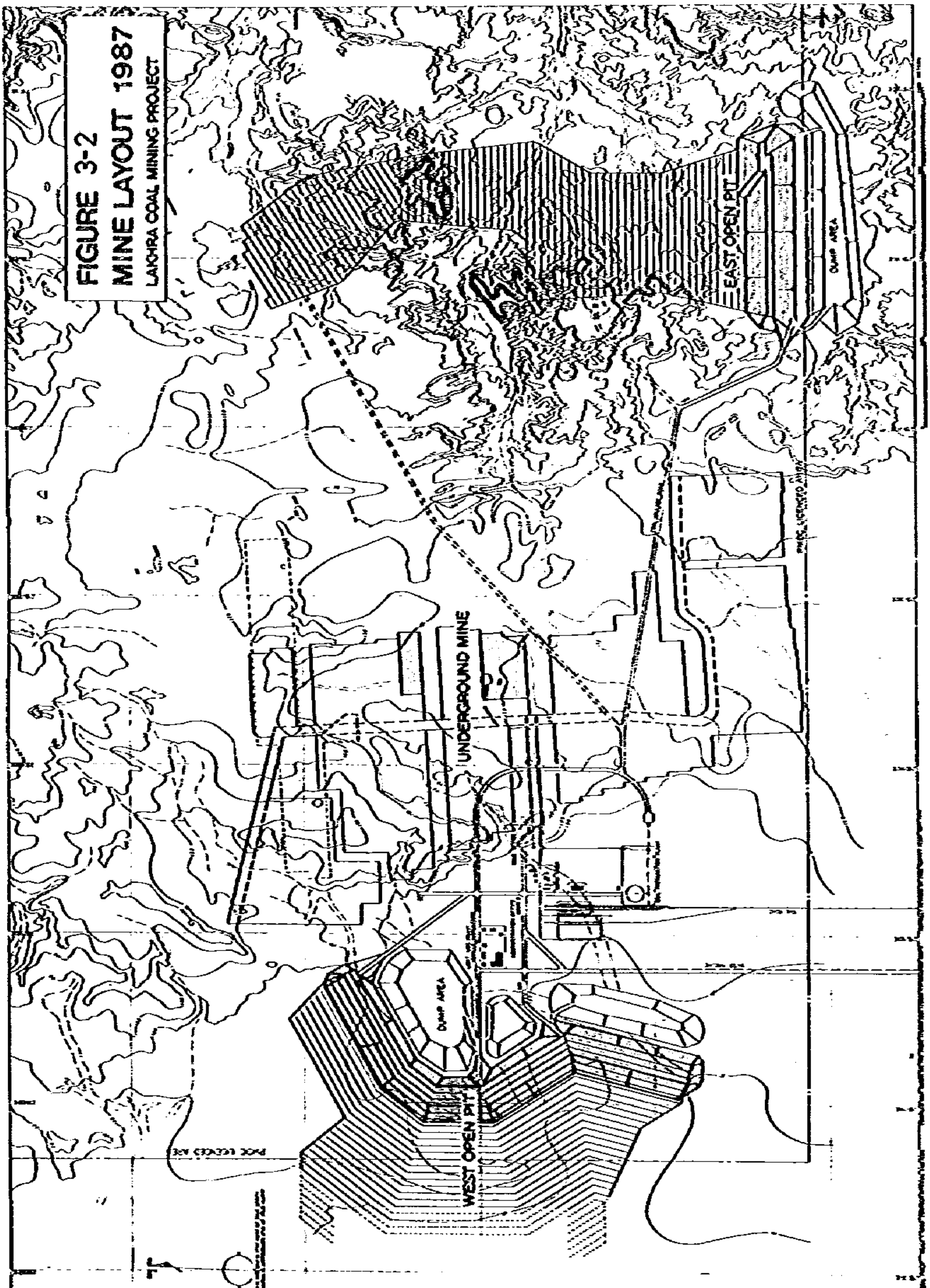


FIGURE 3-2
MINE LAYOUT 1987
LAKHRA COAL MINING PROJECT

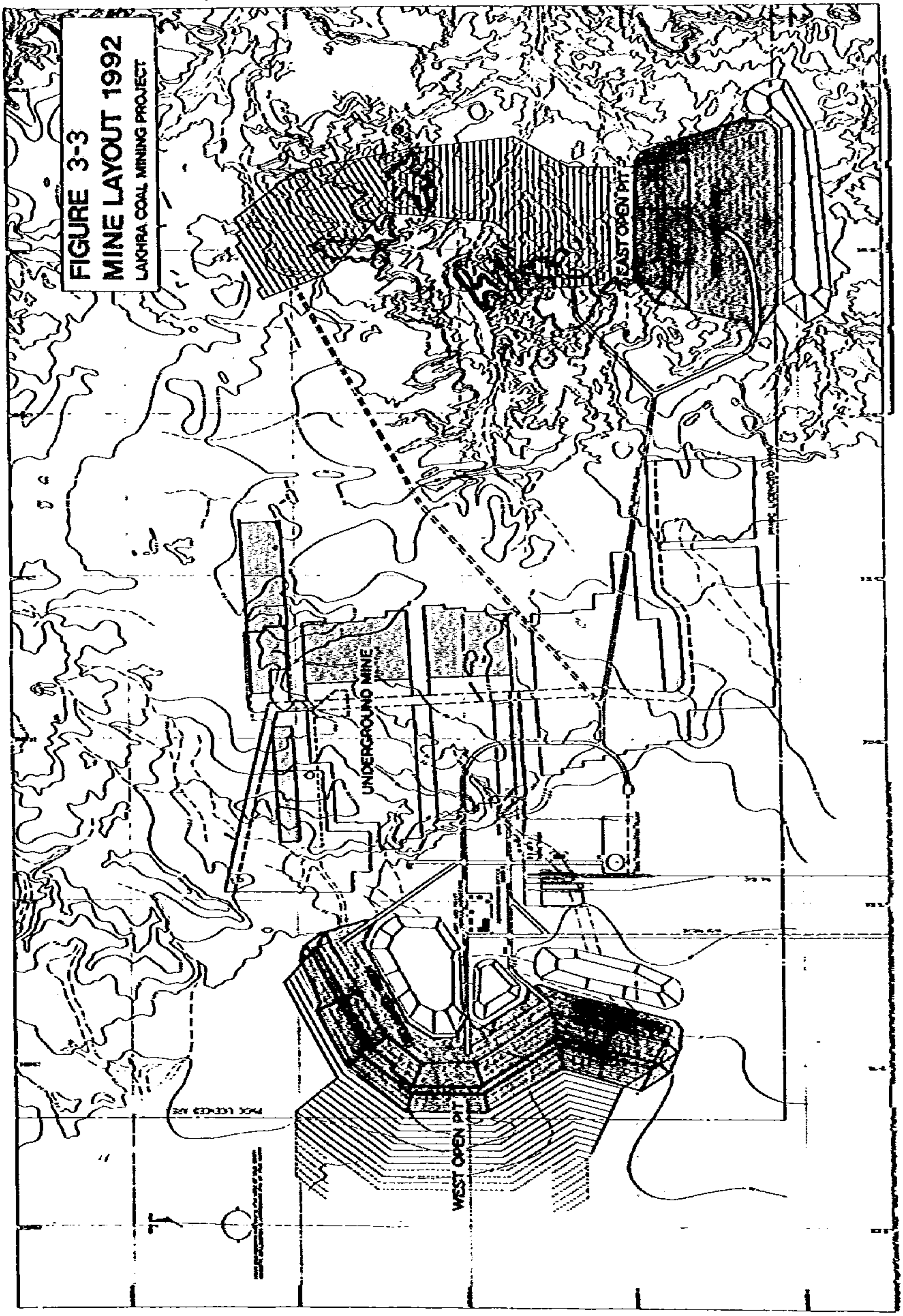


FIGURE 3-3
MINE LAYOUT 1992
LAKHRA COAL MINING PROJECT
EAST OPEN PIT
UNDERGROUND MINE
WEST OPEN PIT

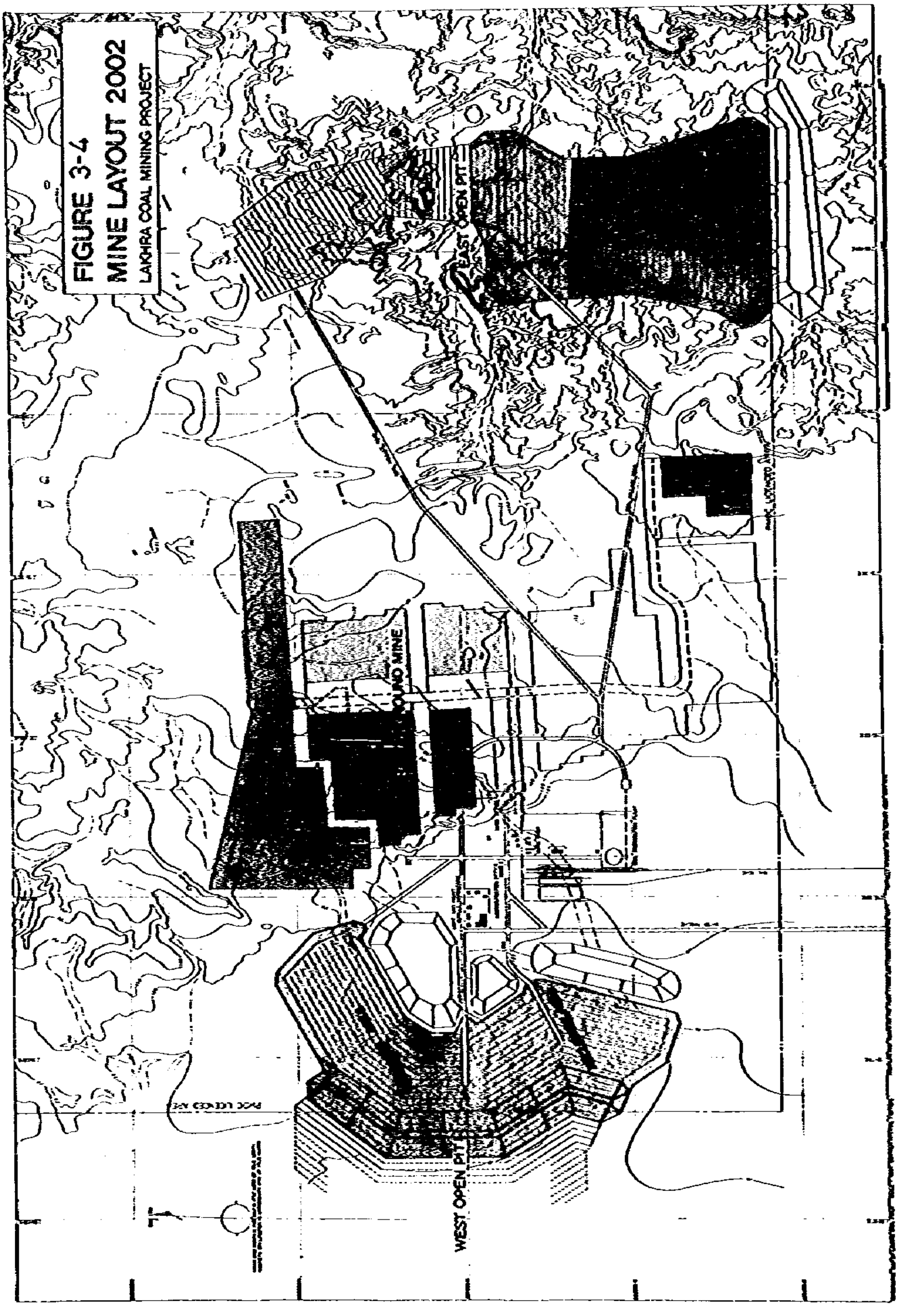


FIGURE 3-4
MINE LAYOUT 2002
LAKHRA COAL MINING PROJECT

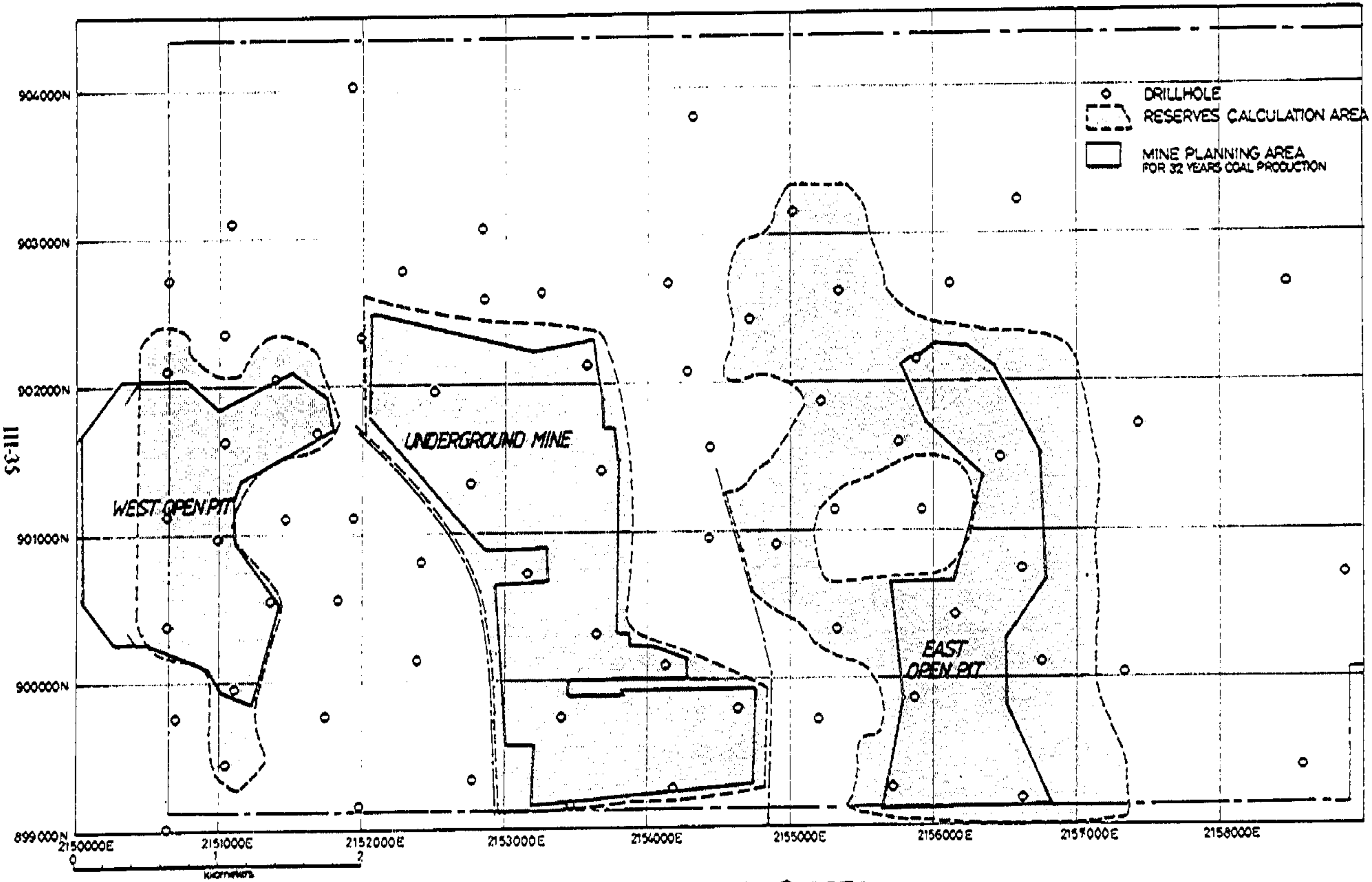


FIGURE 3-5 MINE PLANNING AREA

TABLE 3 – 1

Geological Reserves

Description		West	East	Total	Central	Grand Total
1. Area	10^3 m^2	2,353	6,817	9,170	5,717	14,887
2. Cumulated Seam Thickness	m	6.03	4.17	4.79	-	-
3. Minimum Workable Seam Thickness	m	0.5	0.5	0.5	0.75	-
4. Theoritical Coal Reserves in Raw Coal	10^3 t	21,147	42,478	63,425	14,284	77,709
5. Estimated Recovery Percentage	%	72	72	72	45.5	-
6. Recoverable Coal Reserves in Raw Coal	10^3 t	15,226	31,933	47,159	6,499	53,658
7. Solid Volume of Overburden	10^3 m^2	174,189	519,923	694,112	-	-
8. Stripping Ratio	m^3/t	8.11	12.30	10.94	-	-
9. Average Seam Thickness	m	-	-	-	1.91	-
10. Mining Method		Open Pit	Open Pit	Open Pit	Underground	-

TABLE 3 – 2

Comparison of Coal Reserves

Description		Geology (A)	Mining (B)	Difference (A/B)
Undergorund				
Mining area	$10^3 x m^2$	5,717	4,680	82%
Recoverable Coal Reserves	$10^3 x$ t	6,499	5,119	80
Coal Reserves per area	t/m2	1.1	1.1	-
Production ratio	%	12	19	+58%
Open Pit				
West				
Mining Area	$10^3 x m^2$	2,353	2,100	82
Recoverable Coal Reserves	$10^3 x$ t	15,226	13,703	80
Coal Reserves per ton	t/m2	6.5	6.5	-
East				
Mining Area	$10^3 x m^2$	6,817	1,860	27
Recoverable Coal Reserve	$10^3 x$ t	31,933	9,109	29
Coal Reserves per area	t/m2	4.7	4.9	-
TOTAL				
Mining Area	$10^3 x m^2$	9,170	3,960	43
Recoverable Coal Reserves	$10^3 x$ t	47,159	22,812	48
Coal Reserves per area	t/m2	5.1	5.7	+11
Production Ratio	%	88	81	92
GRAND TOTAL				
Mining Area		14,887	8,640	58
Recoverable Coal Reserves		53,658	28,007	52
Coal reserves per ton		3.6	3.2	89

PRODUCTION (000') tons

Description	as Received	Exclusive of Dilution	Air dried basis
Underground	6,765	6,494	5,195
Open Pit West	18,030	17,129	13,703
East	11,985	11,386	9,109
Total	30,015	28,515	22,812
GRAND TOTAL	36,780	35,009	28,008

TABLE 3-3 SUMMARY OF R.O.M. AND CLEAN COAL PRODUCTION

Year	R.O.M. COAL MINED (10^3 tonnes)			CLEAN COAL									
				Production (10^3 tonnes)			Moisture (Percent)			Ash	Heat Value	Total Sulphur
	U/Ground	Open Pit	Total	U/Ground	Open Pit	Total	Inherent	Surface	Total	(Percent)	(kcal/kg)	(Percent)
1984	4	127	131	4	123	127	8.2	17.9	26.1	20.1	3,797	6.1
1985	19	311	330	18	301	319	7.4	16.2	23.6	20.8	3,911	6.2
1986	135	623	758	131	602	733	7.1	15.5	22.6	21.8	3,854	6.4
1987	239	757	996	232	732	964	7.3	16.0	23.3	20.1	3,921	6.0
1988	238	1,005	1,243	231	971	1,202	7.3	16.0	23.3	19.5	3,963	5.9
1989	241	1,006	1,247	235	972	1,207	7.4	16.1	23.5	19.3	3,959	5.9
1990	248	1,006	1,254	242	972	1,214	7.5	16.3	23.8	18.7	3,981	5.9
1991	241	1,025	1,266	235	992	1,227	8.2	17.8	26.0	17.4	3,918	5.6
1992	249	1,024	1,273	242	991	1,233	8.4	18.2	26.6	16.4	3,960	5.4
1993	280	1,002	1,282	273	968	1,241	8.0	17.5	25.5	20.7	3,688	5.8
1994	265	1,002	1,267	258	969	1,227	8.0	17.3	25.3	21.5	3,669	5.8
1995	260	1,001	1,261	250	967	1,217	7.7	16.7	24.4	22.0	3,726	5.8
1996	257	1,002	1,259	250	969	1,219	7.7	16.8	24.5	21.8	3,728	5.8
1997	254	1,002	1,256	248	968	1,216	7.7	16.8	24.5	21.9	3,724	5.8
1998	250	1,007	1,257	244	973	1,217	7.9	17.1	25.0	20.2	3,839	5.8
1999	258	1,007	1,265	252	973	1,225	7.9	17.1	25.0	19.8	3,857	5.8
2000	266	998	1,264	260	964	1,224	7.8	17.1	24.9	17.7	3,989	5.7
2001	273	1,003	1,276	267	970	1,237	8.2	17.8	26.0	18.0	3,857	5.8
2002	275	1,003	1,278	268	970	1,238	8.2	17.8	26.0	18.0	3,858	5.9
2003	275	1,002	1,277	268	969	1,237	8.1	17.8	25.9	17.9	3,865	5.9
2004	274	1,012	1,286	268	978	1,246	8.4	18.3	26.7	19.0	3,739	5.8
2005	257	1,015	1,272	251	981	1,232	8.0	17.5	25.5	20.2	3,750	6.0
2006	257	1,018	1,275	251	984	1,235	8.1	17.7	25.8	20.0	3,747	5.9
2007	254	1,018	1,272	247	985	1,232	8.1	17.7	25.8	20.0	3,750	5.9
2008	264	1,022	1,286	257	988	1,245	8.4	18.2	26.6	19.1	3,769	5.9
2009	263	1,023	1,286	256	989	1,245	8.4	18.3	26.7	19.0	3,770	5.9
2010	263	1,007	1,270	256	974	1,230	8.0	17.4	25.4	19.6	3,796	5.9
2011	249	1,008	1,257	242	975	1,217	7.8	17.0	24.8	19.9	3,820	6.0
2012	157	994	1,151	153	960	1,113	7.4	16.1	23.5	20.3	3,846	6.0
2013	-	995	995	-	961	961	7.5	16.3	23.8	20.3	3,825	5.9
2014	-	995	995	-	961	961	7.5	16.3	23.8	20.2	3,834	5.9
2015	-	995	995	-	961	961	7.5	16.3	23.8	20.1	3,836	5.9
Total	6,765	30,015	36,780	6,589	29,013	35,602	7.9	17.1	25.0	19.6	3,827	5.9

CHAPTER 4 UNDERGROUND MINING

4-1 Mining Plan

The mining plan developed involves the extraction of 6,589,000 tonnes of clean coal and 6,765,000 tonnes of raw coal (ROM) from No. 1 and No. 3 seams based on exploration study, which 6,202,000 tonnes are obtained from No. 1 seam and 563,000 tonnes are obtained from No. 3 seam. (details in Table 4-1)

The mining area is located at central block between fault A and fault B-1, and the eastern boundary was planned by the thickness of direct roof on the coal seam having more than two meters for the loose sand.

The major considerations for the development of this plan were as follows:

- Seam thickness is considered mineable more than 0.9 m.
- Geological factor of 70 % was used to obtain the coal production.
- Mining recovery factor in the longwall face of 95 %.
- Out of seam dilution of 4 % has been estimated.
- Workable height between 0.9 m and 3.0 m were selected.
- Specific gravity is 1.53 for No. 1 seam and 1.44 for No. 3 seam.

A production rate sufficient to produce approximately 250,000 tonnes of raw coal has been established.

The operation will be based on an operating schedule of 3 shifts per day, 6 days per week, 300 days per year.

The production during operating period from 1986 to 2012 is provided continuously for 27 years at an average of 243,000 tonnes clean coal per year as received base.

The developing work of this mine will commence in 1984, the mining production will be terminated in 2012. (See Table 4-2)

4-2 Mine Development (see Fig. 4-1, 4-2)

(1) Mine Development Schedule

Mine development schedule for initial 2 years and 3 months is shown in Table 4-3.

(2) Inclined Shaft

Two main inclined shafts, each 462 m long, will be the main entries to the mine from the surface plant side. The rock inclined shafts will terminate at the point of intersection with No. 1 seam. Both shafts will be driven at minus twelve degrees of gradient. The shafts will be 4.2 m wide and 3.0 m high (inside), and effective sectional area of 11.2 m^2 and driven on 55 m centers. The shafts will be supported by arched support at one metre centres.

The gradient of inclined shafts is selected considering the availabilities of alternative transportation system, travelling purpose and drifting speed. The location of portals were provided for that the shaft bottom will be at the centre of coal reserves and transportation of coal.

The shafts will be driven conventionally utilizing blasting, side-tipping loaders and mine car winding system. The planned rate of advance will be 2.7 m/day for each shaft with overall time schedule of 8 months. A cross-cut will be driven at the halfway of shaft, and also fan drift of 10 m long will be provided with the concrete structure.

(3) Connecting Entry

Two connecting entries, each 100 m long, will be driven in No. 1 seam connecting the inclined shaft and main entry. The entries will be 4.2 m wide and 3.0 m high (inside), and effective sectional area of 11.2 m^2 and driven on 55 m centres. The entries will be supported by arched support at one metre centres.

The intake connecting entry will be directly connected with the intake main entry, used for mine car switching. The return connecting entry will be connected with the return main entry after overcrossing the intake main entry.

A cross-cut will be driven between two connecting entries at the bottom of inclined shaft provided with the concrete structure, used as the B.L. battery charging station.

(4) Main Entry

Two main entries will be driven on 55 m centres from north to south in the central part of mining area for use as main airway and transportation roadways provided with 11.2 m^2 of effective sectional area supported by arched support at one metre centres.

Two main entries will be connected every 120 m by cross-cuts provided with 8.0 m^2 of effective sectional area and supported by square set.

The maximum gradient of main entries was determined to the 1:150, considering the battery locomotive haulage. Because of the restricion in gradient of haulage tracks, the main entry will be obliged to bend in the northern and southern area.

The main entries will be driven utilizing blasting, side-tipping loaders and mine car haulage system and the planned rate of advance will be 2.7 m/day.

(5) Gate Entry

Gate entries will branch out from main entries to prepare longwall panels. Two gate entries will be driven for one longwall panel, main gate entry for air intake and transportation of coal and materials and tail gate entry for exhaust air. Each main gate entry will be used as the tail gate entry for the next longwall panel.

Gate entries will be supported by square set and effective sectional area will be 8.0 m^2 for single track and 9.5 m^2 for double track. The double track will be 50 m long every 300 m of main gate entry provided for mine car switching.

Gate entries will be driven utilizing balsting, gate-end loaders and mine car haulage system and planned rate of advance will be 2.7 m/day.

(6) No. 3 Seam

Two rock inclined shafts, each 72 m long, will be driven at a positive gradient of twelve degree for No. 3 seam development. The shafts will be driven from the gate entry of No. 1 seam and terminate at the point of intersection with No. 3 seam, then main entries and gate entries will be driven similarly as No. 1 seam development.

(7) Pumping Station

Pumping station and drainage sump, 55 m in length with concrete structure, will be located between intake main entry and return main entry at the end of connecting entries.

(8) Mining Layout

Mining area will be divided into eleven blocks for the purpose of preventing loss of coal reserve due to spontaneous combustion. Each block will be separated by 50 – 100 m width of safety pillars. In each mining block only one longwall face will be operated.

First longwall face will start at A-1 panel nearest to the bottom of inclined shaft and second face at B-1 panel one year later to achieve the full production. The sequence of longwall face is shown in Table 4-4.

4-3 Mining Plan

4-3-1 Mining Method

Longwall retreat system is selected as a mining method based on the consideration of following items;

- Geological structure
- Production scale
- Extraction ratio
- Productivity
- Mine safety
- Capability of equipment maintenance and repair
- Spontaneous combustion

Schematic concept of longwall retreat system is shown in Figure 4-3.

Coal mining in longwall face is planned to commence at the dead-end of each gate entry, advance toward the main entry and terminate leaving safety pillar to protect the main entry. The width of safety pillar is 100 m in the block undeveloped inbye like block A and B, and 50 m in the block develped outbye like block D and G. These safety pillars and coal pillars between the main entries are recovered by pillar splitting after longwall mining.

Pillar splitting is performed from the dead-end of main entries keeping one panel from the operating longwall panel.

Mined out panel is sealed off by fly ash packing in each gate entry, and mined out block is sealed off in each main entry. Safety pillar between the blocks are left intact as a countermeasure to spontaneous combustion.

4-3-2 Longwall Mining

The longwall mining equipment is set up across the 120 m face. Using pneumatic picks or blasting coal is broken and fed onto the face conveyor which is installed parallel and adjacent to the longwall face and is pushed forward by hydraulic shifters as the coal is mined and roof support advanced following the coal face. At the main gate entry the coal from the face conveyor is discharged onto the stage loader which transfers the coal into mine cars. The stage loader is located in the main gate entry adjacent to the longwall block of coal.

When the longwall face is mined back to the main entry leaving 100 or 50 m safety pillar, the longwall equipment is disassembled and removed from the completed longwall face as quickly as possible then moved to the next longwall face and set up.

(1) Mining Method

Face lay-out is shown in Fig. 4-4 and the design criteria is shown in Table 4-5.

One crew consits of two workers, in charge of each 3 m along the face. 120 m face is separated in four 30 m lengths, and 10 crews are in charge for each 30 m. The sequence of mining operation is as follows; (shown in Fig. 4-5)

1) Coal face is broken by use of pneumatic picks or blasting where necessary.

2) Coal is loaded onto the face conveyor.

3) Link bars are connected forward to cover the newly exposed roof.

4) Face conveyor is pushed towards the face.

5) Hydraulic props are set beneath the link bars to support the roof.

6) Props of goaf side are drawn to vacate the roof of goaf and allow it to collapse. The number of workers for one face is as follows. (per shift)

Coal winner	40
Shot firer	2
Prop drawer	6
Stable	4
Packer	4
Prop Counter	1
Salvage	2
Total:	59 men/face/shift

And the number of workers for face maintenance is 11 men/face/shift.

(2) Equipment and Materials

The face will be supported by hydraulic props (shown in Fig. 4-6) and 1.2 m link bars (shown in Fig. 4-7). As the maximum stroke of hydraulic props is 0.9 m, to meet the variation of working height from 0.9 to 3.0 m, two or three types of props in different length and extension tube will be installed. The required number of props will be 500 for one longwall face including for the stable and gate entry. High pressure water will be supplied by a plunger pump (40 kw x 60 ℓ/min x 200 kg/cm², shown in Fig. 4-8).

Face conveyor will be 60 cm wide double chain type (40 kw x 120 m x 100 t/h, shown in Fig. 4-9) and stage loader will be also double chain type (22.5 kw x 60 m x 100 t/h).

Three sets of mining equipment will be provided including one sets for standby.

Mining equipment will be replaced every 5 years and 10 % of FOB investment value of each equipment will be the annual maintenance cost.

Materials for longwall mining are shown in Table 4-9.

4-3-3 Pillar Splitting (see Table 4-7)

Pillar splitting will be performed by room and pillar method. Face will be supported by 3.6 m I-beam square set at one meter centers. One pillar splitting face will be operated by 7 workers in three shifts operation.

Coal will be loaded onto V-type chain conveyor by hand after blasting and loaded into mine cars at gate entry or main entry. The size of safety pillar was planned 25 m x 25 m and the maximum entry length 100 m.

Two sets of mining equipment will be provided including one set for standby. Replacement and maintenance cost will be same as longwall mining.

Materials for pillar splitting face is shown in Table 4-6.

Pillar splitting workers will be employed from 1998. In 1989, 1990, 1993, 1994 a part of longwall workers will be in charge of pillar splitting.

4-4 Road Heading (see Table 4-8)

Two crews each consists of 7 workers will be in charge of driving the inclined shaft, and after completion of the inclined shaft and connecting entry, four crews will be in charge of road heading, two crews for main entry and other two crews for gate entry. Road heading of main entry will be finished in 1997, then two crews will be in charge only for gate entry.

It will take 24 months to develop the first longwall face A-1, 8 months for inclined shaft, 8 months for connecting entry and main entry and 8 months for gate entry and longwall face. Sequence of road heading face is shown in Table 4-3 and annual length of road heading is shown in Table 4-9.

4-4-1 Road Heading Method and Equipment

(1) Drilling

Two air jack hammers will be installed for each face. The drilling depth was planned 1.2 m using 22 mm x 1.6 m rod and 38 mm bit.

(2) Blasting

Ammonium nitrate explosive and mili-second detonators will be used for blasting both for coal and rock. Required amount of explosives was estimated 300 g/m^3 for coal and 1,000 g/m^3 for rock, 200 g for each drill hole. Sand and/or clay will be used for tamping and also water tube for coal.

(3) Loading

Slope type side-tipping loader will be used for inclined shaft driving. After completion of inclined shaft side-tipping loader will be diverted for connecting entry and main entry driving. Gate-end loader will be used for gate entry and cross-cut driving. Coal and rock should be loaded separately into mine cars at the face.

(4) Support

Inclined shaft, connecting entry and main entry will be supported by three pieces I-42 type steel arched support at one meter centres. Cross-cut and gate entry of single track will be supported by 3.0 m I-beam square set and gate entry of double track by 3.6 m I-beam square set at one meter centres.

(5) Road Heading Equipment

Road heading equipment will be replaced every 7 years and 10 % of FOB investment value of each equipment will be the annual maintenance cost.

(6) Others

Fan roadway, B.L. battery charging station and pump station will be provided with 30 cm thick concrete structure.

4-4-2 Materials for Road Heading

Materials for road heading is shown in Table 4-10. Steel materials, some timber and sleepers will be salvaged and re-used for development. Details are shown in Table 4-11.

4-5 Roadway Maintenance

Each main gate entry is used as the tail gate entry of next panel. The sectional area of main gate entry will be decreased due to geomechanical influence of advancing face. The supports of main gate entry will be repaired or re-supported as face advancing by gate maintenance crews stationed in each block on the three shifts/day. Rock dusting, track maintenance and other roadway maintenance are performed by other crews on the one shift/day.

4-6 Ventilation

4-6-1 Ventilation System

Although the emission of inflammable gases are assumed to be very small, for the purpose of cleaning up blasting fumes and keeping working condition suitable, mechanized ventilation system will be proposed.

Main exhaust fan will be installed on the top of No. 2 inclined shaft connected with fan roadway. The maximum air flow is about 5,000 m^3/min and the maximum required ventilating pressure is 175 mm in water column. The main fan is axial flow type and the required power is 300 kW.

During the initial development stage, forcing fan of 15 kW will be used with 30 and 24 inch diameter ventilation tube. This ventilation equipment will provide adequate quantity of air flow to the road heading faces.

The main ventilation system during the whole production period of the underground mine will be a centralized ventilation system. Intake air flows via No. 1 inclined shaft, intake connecting entry and intake main entry then through each face. Exhaust air flows via return main entry, return connecting entry and No. 2 inclined shaft.

The required quantity of air flow was calculated by using computer simulation program for the year 1992, 1997, 2002 and 2007. The input data are shown in Table 4-12, and the result in Fig. 4-15 to Fig. 4-18.

4-6-2 Auxiliary Ventilation

(1) Longwall Face

Required amount of air flow was planned 700 m^3/min for each face. Intake air flows via the main gate entry, exhaust air flows via the tail gate entry and overcrosses intake main entry by air crossing then flows into return main entry.

(2) Road Heading Face and Pillar Splitting Face

Required quantity of air flow was planned 300 m^3/min for each face. Auxiliary forcing fans 15 kW, 7.5 kW, 2.2 kW installed in intake main entry will provide ventilating air for each face via 30 and 24 inch diameter ventilation tube.

(3) Others

In the computer simulation, air leakage of each cross-cut in main entry was assumed to be 50 m^3/min to make the total ventilating efficiency about 50 %. Some cross-cuts might be sealed where necessary. For B.L. battery charging station and pumping station, 150 ~ 250 m^3/min of ventilating air will be provided.

4-6-3 Sealing

Mined out panel will be sealed on gate entry and mined out block also sealed on main entry by 5 m thick fly ash packing. Fly ash recovered by the dust collector at the power plant, is transported underground in mine cars, injected by slurry pump after mixing with water.

For countermeasures to spontaneous combustion, gas patrols go round each sealing and observe gases and temperature, etc. in each sealing.

4-7 Haulage

4-7-1 Haulage System and Method

(1) Raw Coal and Waste Rock

Raw coal and waste rock from each face loaded in 2.0 m^3 steel body mine cars (shown in Fig. 4-21) is transported by 8 or 10 t battery locomotive (shown in Fig. 4-19, 20) through main gate entry and intake main entry, then hoisted up to the surface by 200 kW single-drum hoist through No. 1 inclined shaft. On the surface after dumping by tippler, coal is transported by belt conveyor to the preparation plant, and waste rock is transported by 32 t dump truck to the disposal area. On the surface, 6 t diesel locomotive is also used for mine car switching.

(2) Materials

Materials are loaded into mine cars or material cars on the surface stock yard or workshop and transported into underground.

(3) Workers

No transportation for workers is provided.

4-7-2 Haulage Equipment

The required number of each equipment is, 225 for mine cars, 10 for materials cars, 5 for 8 t battery locomotives, 5 for 10 t battery locomotives and 2 for 6 t diesel locomotives.

Daily amount of transportation is 760 tonnes of coal and 160 tonnes of rock in 12 hours operation.

The unit weight of steel rails is 22 kg/m for inclined shaft and main entry and 15 kg/m for gate entry and surface. The track is double for intake main entry and single for others.

Haulage equipment will be repalced every 7 years and 10 % of FOB investment value of each equipment will be the annual maintenance cost.

4-8 Water Supply and Drainage

Water yield underground is estimated very small. Only during rainy season some water may permeate underground. Water underground flows into the sump and is pumped up through No. 2 inclined shaft to the surface. The quantity of drainage was estimated 1 m^3/min.

The quantity of water to be supplied for pneumatic jack hammer plunger pump, fly ask packing and spinkling was estimated 0.5 m^3/min.

(1) Drainage

During the initial development stage, two submersible pumps (15 kW x 100 m x 0.4 m^3/min, 1.5 kW x 15 m x 0.2 m^3/min) will be installed. For main drainage, two turbine pumps (55 kW x 180 m x 1.4 m^3/min) will be installed at the pumping station and the capacity of sump will be 100 m^3.

(2) Water Supply

One turbine pump (45 kW x 280 m x 0.5 m^3/min) will be installed in the bottom of inclined shaft and provide water to each face.

(3) Piping

The diameter of piping is as follows.

	Drainage	Supply
Inclined shaft	4 in.	4 in.
Main entry	2 in.	3 in.
Gate entry	2 in.	2 in., 1 in.

4-9 Compressed Air

Pneumatic equipment and air consumption are shown in Table 4-13. The required quantity of compressed air is 90 m^3/min in total, provided by two units of 240 kW and one unit of 75 kW compressor installed near the portal of No. 2 inclined shaft. Compressed air will be supplied via 8 inch pipe in No. 2 inclined shaft and 6 inch pipe in main entry and 4 or 2 inch pipe in gate entry.

4-10 Electrical Equipment

(1) Power Distribution System

The total installed motor capacity within the underground mine is estimated at 930 kW and the detail is shown in Table 4-14.

The power will be supplied by means of 3.3 kW overhead line up to No. 2 inclined shaft portal and beyond that, by armoured cable through No. 2 inclined shaft to the main switch room provided at the bottom on inclined shaft.

The main switch room consists of some dry type H/T line switches, from which the power will be distributed to the underground substations provided at the load centre of each section by means of armoured cable.

The underground substation is composed of 300 kVA, 200 kVA or 150 kVA 3.3 kW/400 V mine power centre with H/T and L/T air circuit breakers, L/T earth relay and gate and boxes, and the power will be fed to each machine by cabtyre cable.

All electrical equipment used in the undergound mine will be explosion proof or intrinsically safe construction.

Typical one line power distribution diagram in the underground mine is shown in Fig. 4-22.

(2) Operational Control and Supervisory System

No remote supervisory system is used for the underground equipment, and the operation of all equipment will be supervised and inspected by the foremen and electrical workers concerned.

In underground no operator is posted to control and supervise the pumps and ventilation fans which are operated automatically or continuously. However conveyors, hoists and battery chargers for the battery locomotives are controlled and supervised by individual operators.

(3) Communication System

The communication for the underground mine is carried out by the combination of telephone and inductive radio systems.

The telephones will be installed not only by the entrance of each longwall face but also at the other key points such as the portal and bottom of the inclined shaft and the battery charging station, etc. The system will be used for inter-colliery only and cannot to used for outside calls. All telephones used in the underground mine will be of explosion proof construction.

The inductive radio communication system consists of the fixed station installed in the mine office on the surface and the intrinsically safe mobile stations (portable transceivers) carried by the underground personnel such as foremen and, in addition, gas patrols, electricians and mechanics who have the chance to patrol every nook and cranny underground. The audio communication is carried out between fixed station and mobile station as well as mobile stations with each other using the antenna wire spread out throughout the underground. The system, now being used in all Japanese coal mines more than 10 years, is very useful for the intimate information on safety and operation control aspects.

(4) Signal System

The operating signal system within the longwall face will be conbination of explosion proof signal bells and push buttons. The power, 24 V AC, will be stepped down at the underground substation by means of the signal transformer from 400 V AC and fed into the face by cabtyre cable.

The signal bell system will also be used for the inclined shaft and underground hoists. For the battery locomotive the electrical or manual siren mounted on the locomotive, and the whistles carried by the operator and other workers concerned will be used for the warnings and operating signals respectively.

(5) Lighting System

Not only No. 1 inclined shaft and main haul road but also the main switching room, the pumping station and the battery charging station will be illuminated by means of flameproof lamps. However no lighting facilities are provided in any faces but only the safety lamps carried by all face workers are used for the lighting purpose.

The lighting voltage used in the underground will be 50 V AC and stepped down from 400 V AC at the underground substation concerned by the lighting transformer.

The safety lamp will be of flameproof construction and self service type alkalin storage battery. The battery charger will be silicon rectifier and installed in the safety lamp room provided near by the portal together with 700 numbers of the safety lamps.

(6) Maintenance

The daily inspection and maintenance for underground electrical equipment will be carried out by shift working staff, one electrical foreman and one each electrician and apprentice in northern and southern areas. The periodic inspection and maintenance will be carried out by the non-shift staff together with the shift working staff.

The annual maintenance cost includes one year use of spare parts for the imported equipment and materials and 7 % of previously purchased cables to replenish the wear and tear so the foreign currency, and also 1 % of investment value of foreign currency in C&F exclusive of spare parts as the local currency.

4-11 Safety

(1) Gas Detector

All foreman in the underground always carry portable methan gas detector and measure the concentration of CH_4 frequently and the stationary gas alarm will be installed in each face.

(2) Self-rescue

All workers in the underground always carry the CO filter mask for self-rescue and the oxygen breathing apparatus will be installed at each working face.

TABLE 4-7 COAL PRODUCTION UNDERGROUND MINE

Sheet 1

Description		Longwall Face Coal				Pillar Splitting Coal			Development Coal			Total
Seam	Panel	Working Height m	Specific Gravity tonne/m3	Entry Length m	R.O.M. Coal Mined 000's tonnes	Unit Reserve tonne/m	Entry Length m	R.O.M. Coal Mined 000's tonnes	Unit Reserve tonne/m	Entry Length m	R.O.M. Coal Mined 000's tonnes	R.O.M. Coal Mined 000's tonnes
No.1	A - 1	2.00	1.54	378	115	-	-	-	9.0	1,885	17	132
No.1	A - 2	2.03	1.56	402	126	9.7	310	3	8.0	794	7	136
No.1	A - 3	2.11	1.59	426	142	9.7	310	3	9.8	819	8	153
No.1	A - 4	2.19	1.61	449	157	9.7	310	3	9.8	819	8	168
Total		2.09	1.58	1,655	540	9.7	930	9	9.4	4,317	40	589
No.1	B - 1	2.30	1.64	429	160	9.0	445	4	10.1	1,585	16	180
No.1	B - 2	2.25	1.63	390	142	9.0	445	4	9.2	759	7	153
No.1	B - 3	2.07	1.61	373	123	9.0	445	4	8.7	692	6	133
No.1	B - 4	1.82	1.57	373	106	9.0	445	4	7.5	799	6	116
No.1	B - 5	1.58	1.53	383	92	9.0	445	4	6.7	749	5	101
Total		2.06	1.60	1,948	623	9.0	2,225	20	8.7	4,584	40	683
No.1	C - 1	1.16	1.51	338	59	6.7	3,125	21	4.9	1,423	7	87
No.1	C - 2	1.33	1.51	364	73	6.4	625	4	5.6	897	5	82
Total		1.24	1.51	702	132	6.6	3,750	25	5.2	2,320	12	169
No.1	D - 1	1.75	1.54	624	167	6.6	910	6	6.6	2,721	18	191
No.1	D - 2	1.69	1.52	700	178	6.6	910	6	6.8	1,178	8	192
No.1	D - 3	1.74	1.51	670	174	6.6	910	6	7.0	1,149	8	188
No.1	D - 4	1.83	1.50	387	105	6.6	910	6	6.9	866	6	117
No.1	D - 5	1.82	1.49	350	94	6.6	910	6	6.0	829	5	105
No.1	D - 6	1.71	1.50	312	79	6.6	910	6	6.3	790	5	90
Total		1.75	1.51	3,043	797	6.6	5,460	36	6.7	7,533	50	883
No.1	E - 1	1.55	1.52	233	54	9.0	445	4	6.1	1,312	8	66
No.1	E - 2	1.71	1.51	611	156	9.0	445	4	6.4	937	6	166
No.1	E - 3	1.66	1.53	618	155	9.0	445	4	6.6	1,206	8	167
No.1	E - 4	1.56	1.53	787	186	9.0	445	4	6.1	1,156	7	197
No.1	E - 5	1.56	1.55	780	187	9.0	445	4	6.4	1,098	7	198
Total		1.61	1.53	3,029	738	9.0	2,225	20	6.3	5,709	36	794
No.1	F - 1	1.34	1.55	672	138	9.5	315	3	5.3	1,711	9	150
No.1	F - 2	1.40	1.56	623	135	9.5	315	3	5.3	941	5	143
No.1	F - 3	1.40	1.56	577	125	9.5	315	3	5.6	896	5	133
Total		1.38	1.56	1,872	398	9.5	945	9	5.4	3,548	19	426

Sheet 2

Description		Longwall Face Coal				Pillar Splitting Coal			Development Coal			Total
Seam	Panel	Working Height m	Specific Gravity tonne/m3	Entry Length m	R.O.M. Coal Mined 000's tonnes	Unit Reserve tonne/m	Entry Length m	R.O.M. Coal Mined 000's tonnes	Unit Reserve tonne/m	Entry Length m	R.O.M. Coal Mined 000's tonnes	R.O.M. Coal Mined 000's tonnes
No.1	G - 1	1.57	1.41	523	115	8.0	625	5	5.8	1,728	10	130
No.1	G - 2	1.57	1.42	518	114	8.0	625	5	6.0	995	6	125
No.1	G - 3	1.64	1.43	516	120	8.0	625	5	6.1	984	6	131
No.1	G - 4	1.73	1.43	515	126	8.0	625	5	7.1	983	7	138
Total		1.63	1.42	2,072	475	8.0	2,500	20	6.3	4,690	29	524
No.1	H - 1	1.82	1.49	518	139	8.0	625	5	7.4	1,494	11	155
No.1	H - 2	1.71	1.50	477	121	8.0	625	5	6.3	946	6	132
No.1	H - 3	1.62	1.52	394	96	8.0	625	5	6.4	944	6	107
No.1	H - 4	1.53	1.53	411	95	8.0	625	5	5.6	894	5	105
No.1	H - 5	1.44	1.53	417	91	8.0	625	5	5.6	888	5	101
No.1	H - 6	1.38	1.53	421	88	8.0	625	5	5.6	900	5	98
No.1	H - 7	1.40	1.53	432	92	-	-	-	5.5	908	5	97
No.1	H - 8	1.53	1.55	428	101	-	-	-	6.3	948	6	107
Total		1.58	1.52	3,489	823	8.0	3,750	30	6.2	7,922	49	902
No.1	I - 1	1.63	1.59	136	35	-	-	-	6.6	1,060	7	42
No.1	I - 2	1.62	1.58	169	43	9.0	445	4	6.9	580	4	51
No.1	I - 3	1.62	1.58	212	54	9.0	445	4	6.4	622	4	62
No.1	I - 4	1.62	1.56	253	63	6.5	310	2	6.0	663	4	69
No.1	I - 5	1.62	1.55	295	73	6.5	310	2	5.7	705	4	79
No.1	I - 6	1.66	1.55	337	86	6.5	210	2	6.8	740	5	93
No.1	I - 7	1.67	1.55	371	95	6.5	310	2	6.8	740	5	102
Total		1.64	1.56	1,773	449	7.1	2,130	16	6.4	5,110	33	498
No.1	J - 1	1.50	1.58	334	78	9.0	445	4	6.4	1,572	10	92
No.1	J - 2	1.88	1.52	740	209	9.0	445	4	6.3	1,110	7	220
No.1	J - 3	1.56	1.49	524	121	6.3	315	2	5.9	843	5	128
No.1	J - 4	1.36	1.48	427	85	6.3	315	2	5.4	744	4	91
No.1	J - 5	1.43	1.48	414	87	6.3	315	2	5.5	733	4	93
No.1	J - 6	1.66	1.51	415	103	6.3	315	2	6.8	732	5	110
Total		1.62	1.51	2,854	683	7.5	2,150	16	6.1	5,734	35	734
No.1 Total		1.70	1.53	22,446	5,658	7.7	26,065	201	6.7	51,467	343	6,202

Sheet 3

Description		Longwall Face Coal				Pillar Splitting Coal			Development Coal			Total
Seam	Panel	Working Height m	Specific Gravity tonne/m^3	Entry Length m	R.O.M. Coal Mined 000's tonnes	Unit Reserve tonne/m	Entry Length m	R.O.M. Coal Mined 000's tonnes	Unit Reserve tonne/m	Entry Length m	R.O.M. Coal Mined 000's tonnes	R.O.M. Coal Mined 000's tonnes
No.3	3 - 1	0.98	1.44	1,120	157	3.8	4,715	18	2.7	3,738	10	185
No.3	3 - 2	0.98	1.44	1,070	150	3.2	310	1	3.1	1,588	5	156
No.3	3 - 3	0.98	1.44	700	98	3.2	315	1	2.5	1,973	5	104
No.3	3 - 4	0.98	1.44	750	105	3.8	2,625	10	2.5	1,218	3	118
No.3 TOTAL		0.98	1.44	3,640	510	3.5	7,965	30	2.7	8,517	23	563
GRAND TOTAL		1.64	1.52	26,086	6,168	7.7	34,030	231	6.4	59,984	366	6,765

TABLE 4-2 R.O.M. COAL MINED FROM UNDERGROUND MINE

(000's Tonnes)

Year	Longwall	Pillar Splitting	Road Heading	Total
1984	-	-	4	4
1985	-	-	19	19
1986	110	-	25	135
1987	221	-	18	239
1988	221	-	17	238
1989	221	7	13	241
1990	224	8	16	248
1991	229	-	12	241
1992	237	-	12	249
1993	246	14	20	280
1994	244	1	20	265
1995	239	-	21	260
1996	236	-	21	257
1997	236	-	18	254
1998	235	6	9	250
1999	236	14	8	258
2000	242	14	10	266
2001	249	14	10	273
2002	249	14	12	275
2003	249	14	12	275
2004	249	14	11	274
2005	234	14	9	257
2006	234	14	9	257
2007	234	11	9	254
2008	240	14	10	264
2009	240	14	9	263
2010	240	14	9	263
2011	232	14	3	249
2012	141	16	-	157
Total	6,168	231	366	6,765

TABLE 4-3 DEVELOPMENT SCHEDULE

Description			
Entry	Support	Rock or Seam	Length
No.1 Inclined Shaft	Arch	Rock	462 m
No.2 Inclined Shaft	Arch	Rock	462
Inclined shaft X-cut	Arch	Rock	50
Connecting Entry (Intake)	Arch	Seam	104
Connecting Entry (Return)	Arch	Seam	156
Con.Entry X-cut (BL Charging Stn.)	Arch	Seam	50
Main Entry (Intake)	Arch	Seam	528
Main Entry (Return)	Arch	Seam	528
Main Entry X-Cut	Sq. Set	Seam	304
Gate Entry (A-1)	Sq. Set	Seam	477
Gate Entry (A-2)	Sq. Set	Seam	502
Longwall Face (A-1)	Sq. Set	Seam	120
Pumping Station	Arch	Seam Rock	52
Fan Roadway	Arch	Rock	10

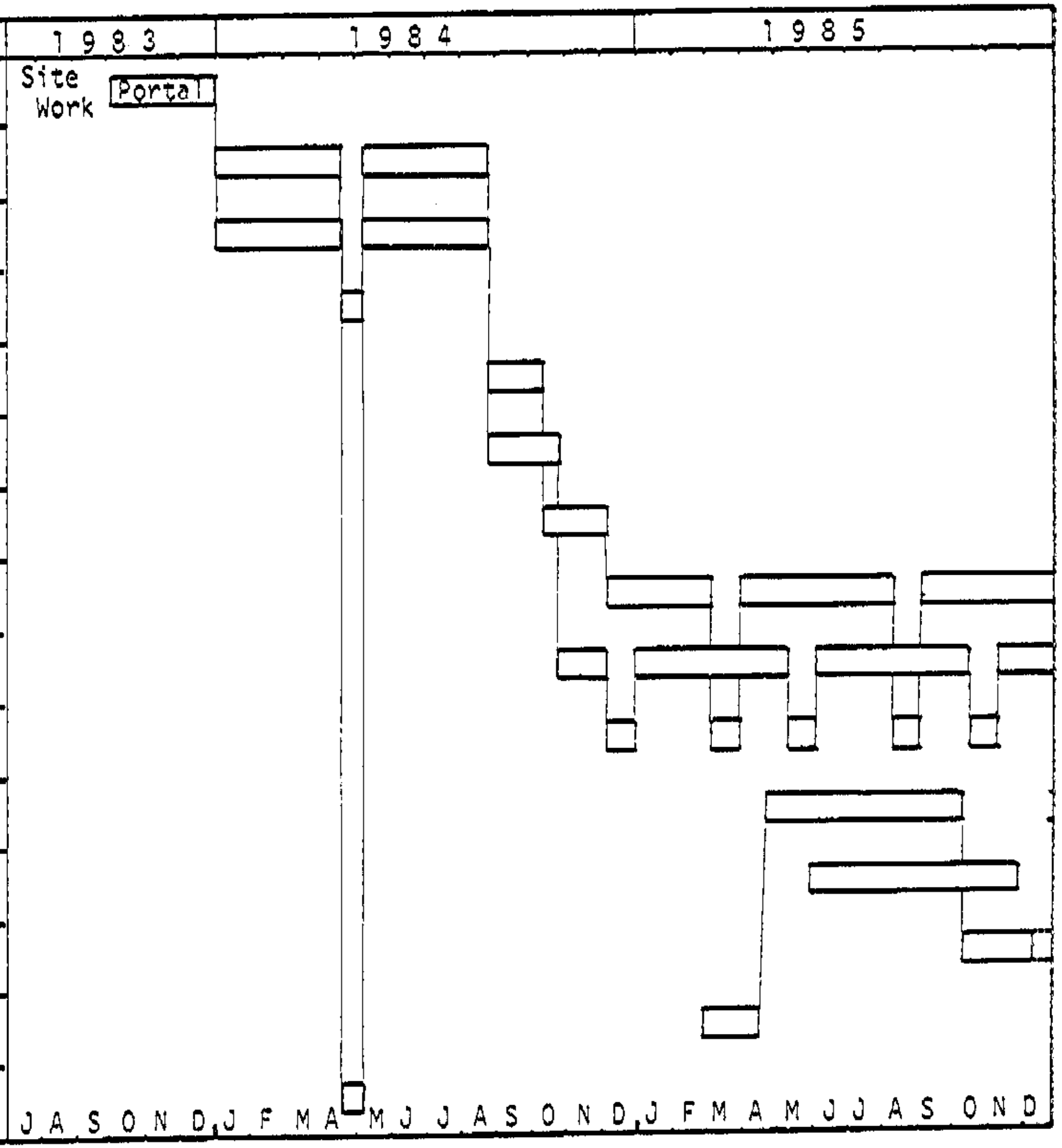


TABLE 4-4 SEQUENCE OF MINING FACE

YEAR	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993
LONGWALL No.1 FACE			A - 1	A - 2	A - 3	A - 4	3 - 1	3 - 2		C-1
LONGWALL No.2 FACE				B - 1	B - 2	B - 3	B - 4	B - 5	3 - 3	3 - 4
PILLAR SPLITTING FACE						3-1 North				3-1, 3-2, 3-3, 3-4

YEAR	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003
LONGWALL No.1 FACE	C-2, E-1, E - 2		E - 3	E - 4		E - 5	G - 1	G - 2	G - 3	
LONGWALL No.2 FACE	D - 1	D - 2	D - 3		D - 4	D - 5, D - 6	F - 1	F - 2	F - 3	
PILLAR SPLITTING FACE					C/D, D-1	D-2, D-3	D-4, D-5	D-6, C ~ D	C-2, C-1	E-1, E-2

YEAR	2004	2005	2006	2007	2008	2009	2010	2011	2012	2013
LONGWALL No.1 FACE	G - 4, I-1, I-2	I-3, I - 4	I - 5, I - 6	I - 7	J - 1	J - 2	J - 3	J-4, J-5	J-6	
LONGWALL No.2 FACE	H - 1	H - 2	H - 3	H - 4, H - 5	H - 6	H - 7	H - 8			
PILLAR SPLITTING FACE	E - 3, E - 4	E - 5, F-1	F-2, F-3	G-1, G-2	G-3, G-4, H-1	H-2, H-3, H-4	H-5, H-6, J - 1	J - 2, J-3	J-4, J-5,6	

TABLE 4-5

DESIGN CRITERIA OF LONGWALL FACE

Number of Faces	2 (1 in Year 1986, 2011, 2012)
Face Length	120 m
Working Height	0.9 - 3.0 m (Ave. 1.64 m)
Face Inclination	0 - ± 3°
Mining Method	Blasting and/or Coal Pick
Operation	8 hours/shift 3 shifts/day (2 shifts for coal winning & 1 shift for maintenance) 6 days/week, 300 days/year
Face Advance	1.7 m/day (average)
Production	400 t/day/face *
Face Support	Hydraulic Props with Link Bars
Prop Arrangement	Lattice
Raw of Props	3 rows
Length of Link Bar	1.20 m
Packing	Partial Fly Ash Packing with Wooden Pack-wall
Face Equipment	Pneumatic Pick and Auger Drill
Coal Transportation	Face; Armoured Face Conveyor Gate; Stage Loader & Battery Locomotive
Number of Workers	Mining; 59 men/face/shift Maintenance; 11 men/face/shift
Productivity (all in face)	3.1 O.M.S.

* 120 x 1.64 x 1.7 x 1.53 x 0.7 x 0.95 x 1.2 ≑ 400 tonnes
(120 m; 1.64 m; 1.7 m; 1.53 S.G.; 0.7 Geol. Fac.; 0.95 Min. Fac.; 1.2 Moisture Fac.)

TABLE 4 – 6

MATERIALS FOR COAL MINING

(per R.O.M. Tonne)

Description	Unit	Longwall	Pillar Splitting
Square Set (I-Beam 3.6 m)	Set	-	0.15
Rod *	kg	0.003	0.003
Bit & Pick *	No.	0.002	0.002
Other Steel	kg	0.03	0.03
Hose 2" *	m	0.005	0.005
1" *	m	0.015	0.015
Blasting Cable *	m	0.03	0.03
Blasting Code *	m	0.03	0.03
Machine Oil	ℓ	0.03	0.02
Ventilation Tube *	m	-	0.17
Explosive	kg	0.15	0.3
Detonator	No.	1.5	1.5
Timber	m^3	0.03	0.36

* consumption

TABLE 4 — 7

DESIGN CRITERIA OF PILLAR SPLITTING FACE

Number of Face	1
Mining Method	Room & Pillar Method with Blasting
Support	Square Set I - Beam 3.6 m and Wooden legs
Working Height	0.9 - 3.0 m (Ave. 1.7 m)
Coal Transportation	V - Type Conveyor and Battery Locomotive
Operation	3 shifts/day
Support Spacing	1.2 m
Face Advance	3.6 m (Ave.)
Coal Production	50 t/day/ face *
Number of Workers	21 men/day/face
Productivity	2.4 O.M.S.
Pillar Size	25 x 25 m
Entry Length	Max. 100 m

* 7.6 x 3.6 x 1.53 x 1.2 ≑ 50 tonnes
m^2 m S.G. Moisture Factor

TABLE 4-8 DESIGN CRITERIA OF ROAD HEADING

Drilling	Air Jack Hammer (2 sets/face) Rod 22 mm x 1.6 m Bit 38 mm
Blasting	Ammonium Nitrate 300 g/coal 1 m^3 & H.S.D. 1 kg/rock 1 m^3 Tamping by sand or clay and water tube in coal
Loading	Side Dump Loader for inclined shaft & main entry (shown in Fig. 4-10) Gate End Loader for gate entry & X-cut (shown in Fig. 4-11)
Support	3 pieces arch frame for slope & main entry (shown in Fig. 4-12) Squre set (I-beam 3.0m or 3.6m) for gate entry and X-cut (shown in Fig. 4-13)
Separation of Support	1.0 m
Operation	3 shifts/day
Haulage	2.0 m^3 mine car & battery locomotive
Face Advance	67.5 m/month (2.7 m/day)
Number of Worker	21 men/face/day

TABLE 4-9 SUMMARY DEVELOPMENT LENGTH REQUIRED BY YEAR

(UNIT: METER)

	1984	1985	1986	1987	1988	1989	1990	1991	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2008	2009	2010	2011	Total Length
CAPITAL																													
INCLINED SHAFT	924	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	924
CONNECTING ENTRY	260	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	260
MAIN ENTRY	104	952	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1,056
X - CUT	154	200	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	354
GATE ENTRY	-	979	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	979
LONGWALL FACE	-	120	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	120
OTHERS	60	52	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	112
SUB-TOTAL	1,502	2,303	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	3,805
OPERATING																													
INCLINED SHAFT	-	-	-	144	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	144
MAIN ENTRY	-	-	952	1,618	526	600	1,485	614	324	215	875	1,210	1,295	560	-	-	-	-	-	-	-	-	-	-	-	-	-	-	10,274
X - CUT	-	--	200	300	50	-	318	103	100	30	173	340	255	150	-	-	-	-	-	-	-	-	-	-	-	-	-	-	2,019
GATE ENTRY	-	-	1,583	928	2,423	1,942	1,177	2,143	2,316	2,571	1,806	1,430	1,204	2,017	1,310	1,430	1,310	1,239	1,292	1,279	1,190	1,270	1,197	1,463	1,255	1,245	1,310	222	38,552
LONGWALL FACE	-	-	240	120	120	360	120	240	360	240	240	120	346	254	240	120	240	311	258	271	360	280	353	87	295	305	240	240	6,360
SUB-TOTAL	-	-	2,975	3,100	3,119	2,902	3,100	3,100	3,100	3,056	3,094	3,100	3,100	2,981	1,550	1,550	1,550	1,559	1,550	1,550	1,550	1,550	1,550	1,550	1,550	1,559	1,559	462	57,349
TOTAL																													
INCLINED SHAFT	924	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1,068
CONNECTING ENTRY	260	-	-	144	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	262
MAIN ENTRY	104	952	952	1,618	526	600	1,485	614	324	215	875	1,210	1,295	560	-	-	-	-	-	-	-	-	-	-	-	-	-	-	11,330
X - CUT	154	200	200	300	50	-	318	103	100	30	173	340	255	150	-	-	-	-	-	-	-	-	-	-	-	-	-	-	2,423
GATE ENTRY	-	979	1,583	928	2,423	1,942	1,177	2,143	2,316	2,571	1,806	1,430	1,204	2,017	1,310	1,430	1,310	1,239	1,292	1,279	1,190	1,270	1,197	1,463	1,225	1,245	1,310	222	39,531
LONGWALL FACE	-	120	240	120	120	360	120	240	360	240	240	120	346	254	240	120	240	311	258	271	360	280	353	87	295	305	240	240	6,480
OTHERS	60	52	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	62
GRAND TOTAL	1,502	2,303	2,975	3,100	3,119	2,902	3,100	3,100	3,100	3,056	3,094	3,100	3,100	2,981	1,550	1,550	1,550	1,550	1,550	1,550	1,550	1,550	1,550	1,559	1,559	1,550	1,559	462	61,154

TABLE 4-10

MATERIALS FOR ROAD HEADING (per m)

Description	Unit	Arch	Sq. Set	Sq. Set L
Arched Support	Set	1	-	-
I-Beam 3.0 m	Set	-	1	-
3.6 m	Set	-	-	1
Rail 22 kg/m	m	2 or 4	-	-
15 kg/m	m	-	2	4
Pipe Air	m	(8" or 6") 1	(4" or 2") 1	(4" or 2") 1
Water	m	(4" or 3") 1	(2" or 1") 1	(2" or 1") 1
Drainage	m	(4" or 2") 1	(2") 1	(2") 1
Rod *	kg	1.0	1.0	1.0
Bit *	No.	0.014	0.014	0.014
Other Steel *	kg	0.5	0.5	0.5
Blasting Cable *	m	0.5	0.5	0.5
Blasting Code *	m	0.5	0.5	0.5
Machine Oil	ℓ	10.0	10.0	10.0
Ventilation Tube*	m	0.1	0.1	0.1
Explosive (Coal)	kg	6.0	4.0	4.8
(Rock)	kg	12.0	-	-
Detonator (Coal)	No.	30	20	24
(Rock)	No.	40	-	-
Timber	m^3	0.23	0.36	0.36

* Consumption

TABLE 4-11

MATERIALS SALVAGE

Description	Salvaged (%)	Available (%)	Installed (%)
Arched Support	80	80	64
I - Beam	80	90	72
Wooden Leg	40	50	20
Rail, Pipe	100	90	90
Spike, Joint, etc.	90	90	81
Sleeper	100	50	50
Timber	40	50	20

TABLE 4-12

ESTIMATED VENTILATION CHARACTERISTICS OF ROADWAYS

1. Resistance Factor

Description		Support	F	U	k	R
No.1 Inclined Shaft		Arch	10.0	13.0	0.0012	0.016
No.2 Inclined Shaft		Arch	10.0	13.0	0.0012	0.016
Main Entry	Intake	Arch	10.0	13.0	0.0013	0.018
	Return	Arch	9.5	13.0	0.0013	0.020
Gate Entry	Intake	Sq.Set	6.4	10.7	0.0018	0.074
(X-cut)	Return	Sq.Set	5.4	10.7	0.0018	0.12

F; Cross-sectional area of airway (m)
U; Perimeter of airway (m)
k; Friction factor ($kg \cdot sec^2/m^4$)
R; Resistance factor (murgue; 0.001 $kg \cdot sec^2/m^8$)

2. Cross-sectional Area

Description		Initial (m^2)	Effective (m^2)
Arch	Intake	11.15	10.0 = 11.15 x 0.9
	Return	11.15	9.5 = 11.15 x 0.9 x 0.95
Sq.Set	Intake	8.0	6.4 = 8.0 x 0.8
	Return	8.0	5.4 = 8.0 x 0.8 x 0.85

3. Pressure Loss

$$P = \frac{k \cdot L \cdot U \cdot V^2}{F} = R \cdot Q^2$$

P; Pressure loss (kg/m^2)
L; Velocity of flow (m/sec)
V; Length of airway (m)
Q; Quantity of flow (m^3/sec)

4. Required Quantity of Flow for Each Face

(1) Longwall Face

Kata degree is one index to judge the working condition in underground and dried kata degree of 8 $mcal/cm^2$ is estimated favorable for longwall mining.

Dried kata degree is calculated as follows;

$$Kd = (0.13 + 0.47\sqrt{w})(36.5 - \theta)$$

Kd; Dried kata degree
w; Velocity of flow (m/sec)
θ; Temperature in face (Centigrade)

θ is estimated 25°

$$8 = (0.13 + 0.47\sqrt{w})(36.5 - 25)$$
$$w \fallingdotseq 1.4 \text{ m/sec}$$

Cross-sectional area of face is estimated 6.0 m^2. Therefore required quantity of flow for longwall face is;

$$6.0 \times 1.4 \times 60 \fallingdotseq 500 \text{ m}^3/\text{min}$$

In anticipation of 30 % leakage;

$$500 \div 0.7 \fallingdotseq 700 \text{ m}^3/\text{min}$$

(2) Road Heading and Pillar Splitting Face

A minimum velocity of 18 m/min in working places is required and respirable dust levels (-5μ) cannot exceed 3 mg/m^3. The maximum cross-sectional area of roadway is 11.2 m^2.

Therefore required quantity of flow is;

$$18 \times 11.2 \fallingdotseq 200 \text{ m}^3/\text{min}$$

In anticipation of 30 % leakage;

$$200 \div 0.7 \fallingdotseq 300 \text{ m}^3/\text{min}$$

5. Ventilating Efficiency

The total ventilating efficiency is estimated about 50 %, in anticipation of 50 m^3/min of leakage for each cross-cut of main entry.

TABLE 4-13

PNEUMATIC EQUIPMENT AND AIR CONSUMPTION

(per shift)

Description	Equipment	No.	Rated Air Consumption m^3/min	Total m^3/min	Load Factor %	Actual Air Consumption m^3/min
Coal Mining	Pick	20	0.9	18.0	40	7.2
	Auger	12	2.3	27.6	30	8.3
Road Heading	Hammer	9	2.7	24.3	30	7.3
	Pick	6	0.9	5.4	10	0.6
	Gate End Loader	3	5.1	15.3	63	9.7
	Side Tipping Loader	2	18.0	36.0	56	20.2
Sub-Total				126.6		53.3
Surface	Tippler					10.0
Total						63.3
Required Air Quantity (efficiency 70 %)						90 m^3/min

TABLE 4-14

DETAILED INSTALLATION kW IN UNDERGROUND

Description	Unit (kW)	Q'ty	Total (kW)
Armourd Face Conveyor	40	3	120
Gate Conveyor	22.5	3	67.5
Plunger Pump	40	3	120
Hoist	15	2	30
Ventilation Fan	11	2	22
ditto	7.5	2	15
ditto	2.2	5	11
Drainage Pump	55	2	110
ditto	15	1	15
ditto (submersible)	1	2	2
Water Feed Pump for Face	45	1	45
B.L. Battery Charger for 10 t	33	4	132
ditto for 8 t	26	5	130
Lighting		1 lot	25
Miscellaneous		1 lot	85.5
Total			930

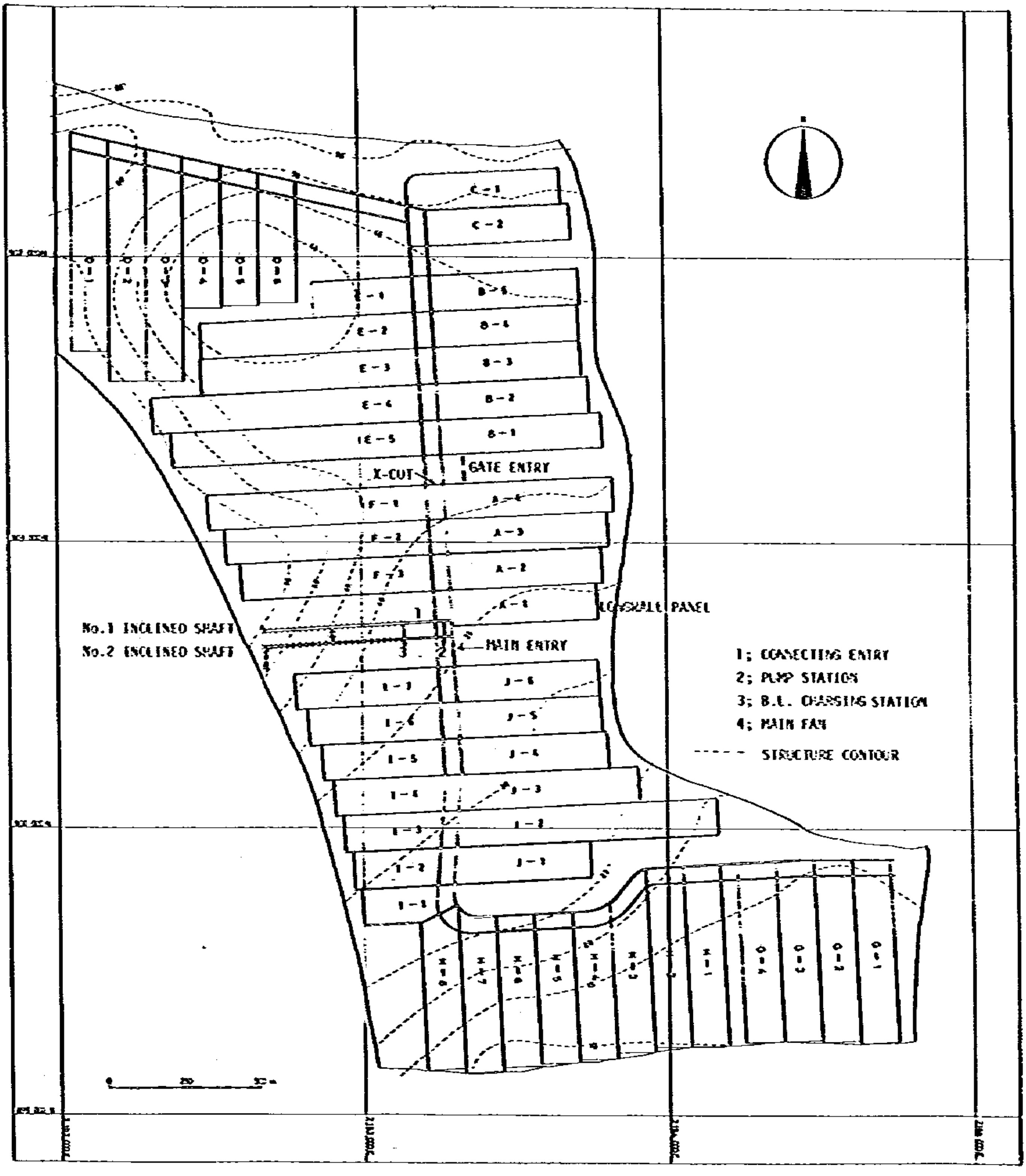


FIGURE 4-1 No. 1 SEAM MINING LAYOUT

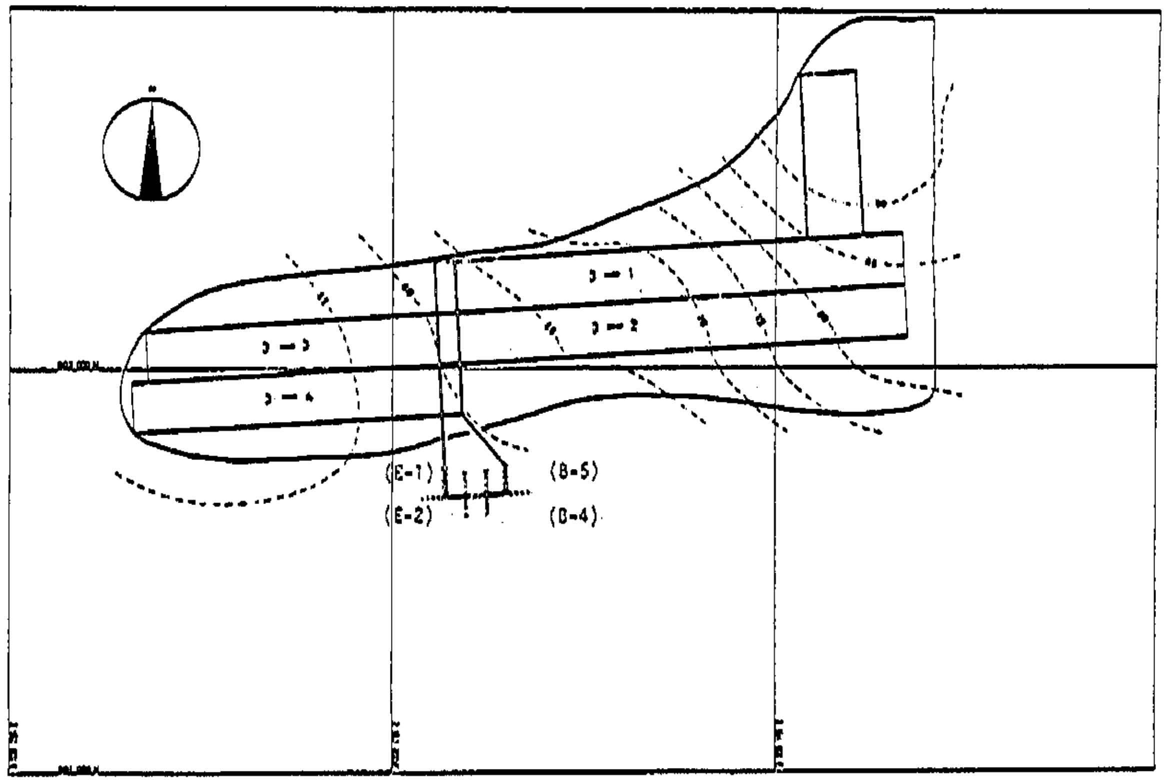


FIGURE 4 - 2 No. 3 SEAM MINING LAYOUT

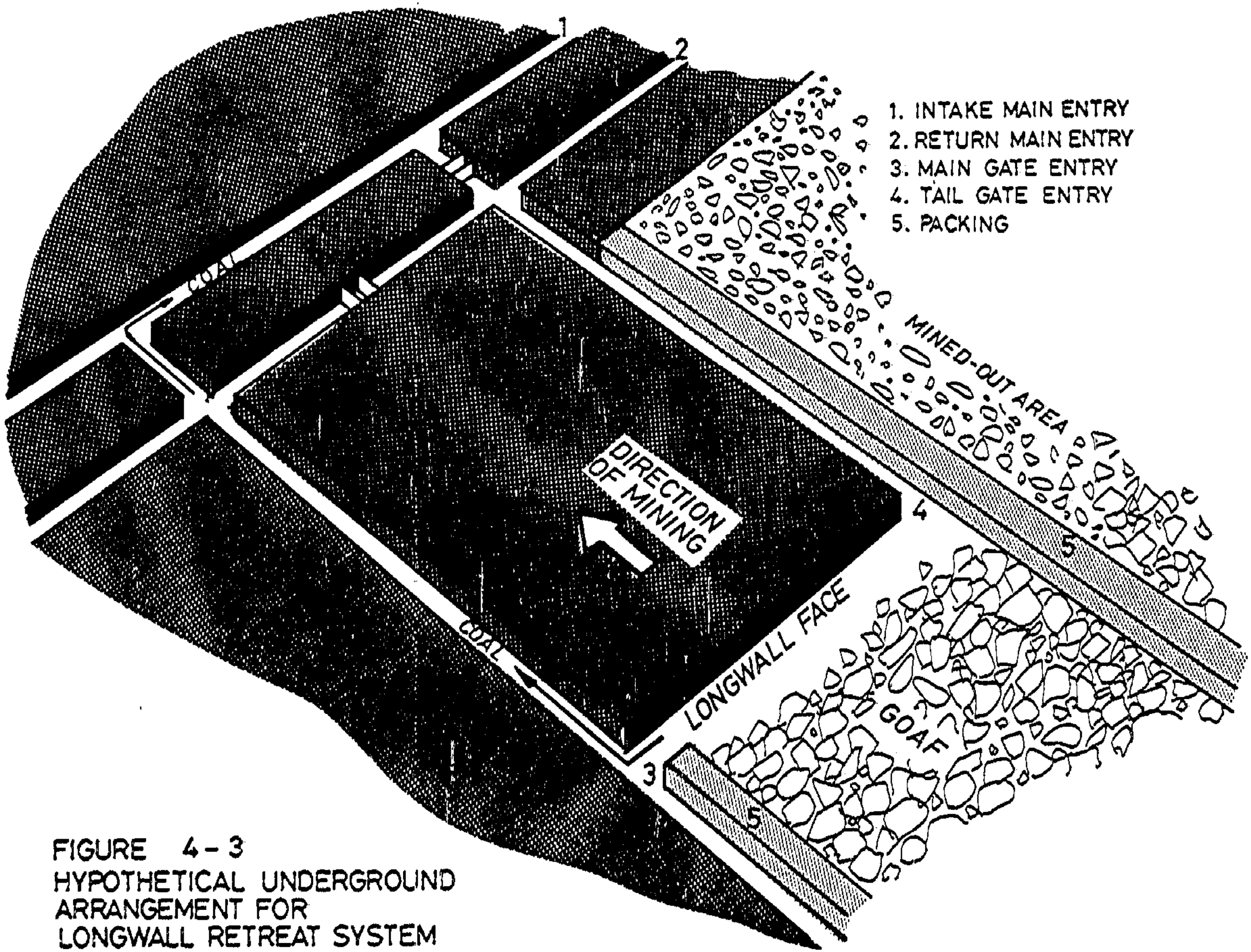
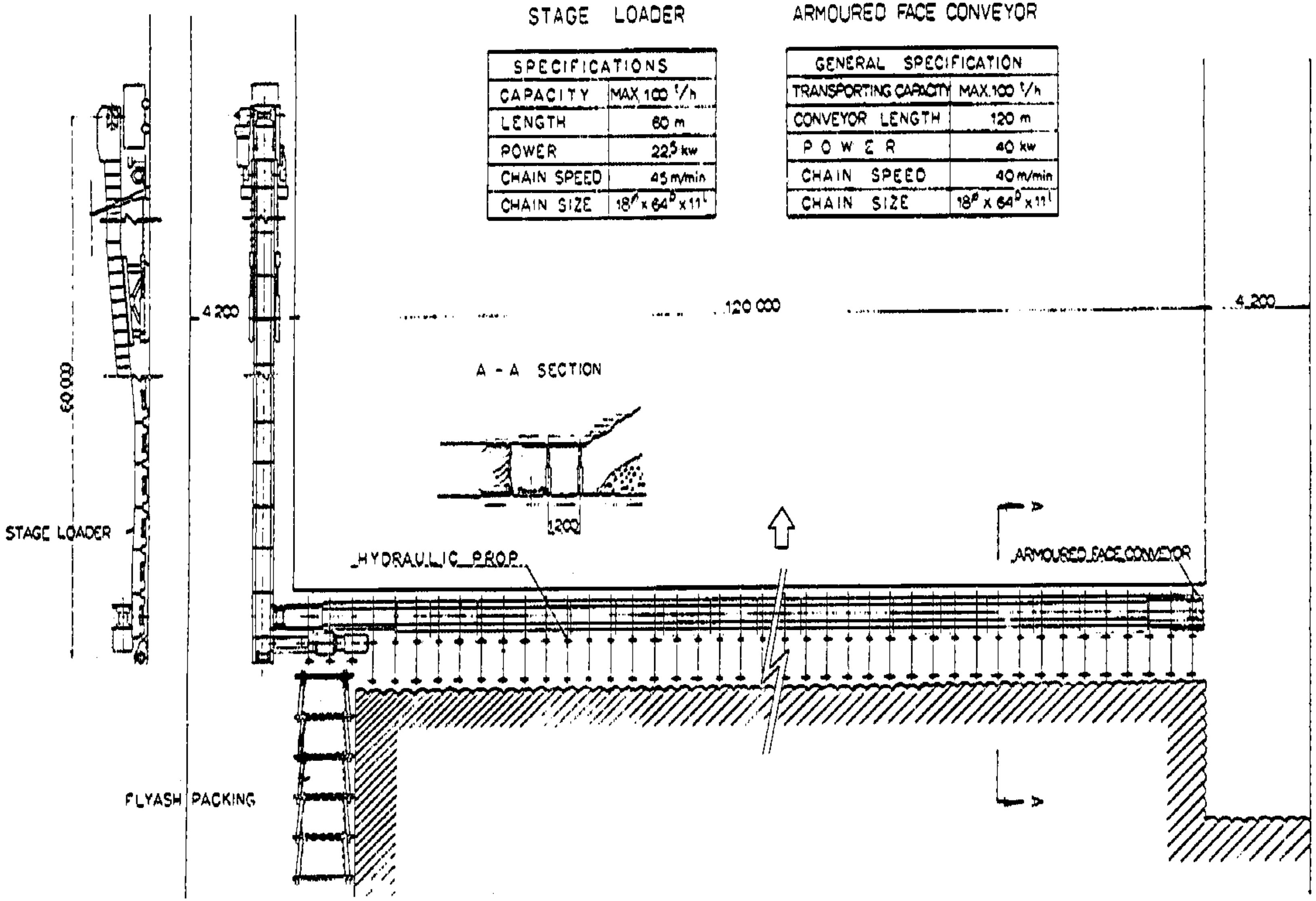


FIGURE 4-3
HYPOTHETICAL UNDERGROUND
ARRANGEMENT FOR
LONGWALL RETREAT SYSTEM



STAGE LOADER

SPECIFICATIONS	
CAPACITY	MAX 100 t/h
LENGTH	60 m
POWER	22.5 kw
CHAIN SPEED	45 m/min
CHAIN SIZE	18φ x 64P x 11L

ARMOURED FACE CONVEYOR

GENERAL SPECIFICATION	
TRANSPORTING CAPACITY	MAX.100 t/h
CONVEYOR LENGTH	120 m
POWER	40 kw
CHAIN SPEED	40 m/min
CHAIN SIZE	18φ x 64P x 11L

FIGURE 4-4 MINING PLAN LAY - OUT

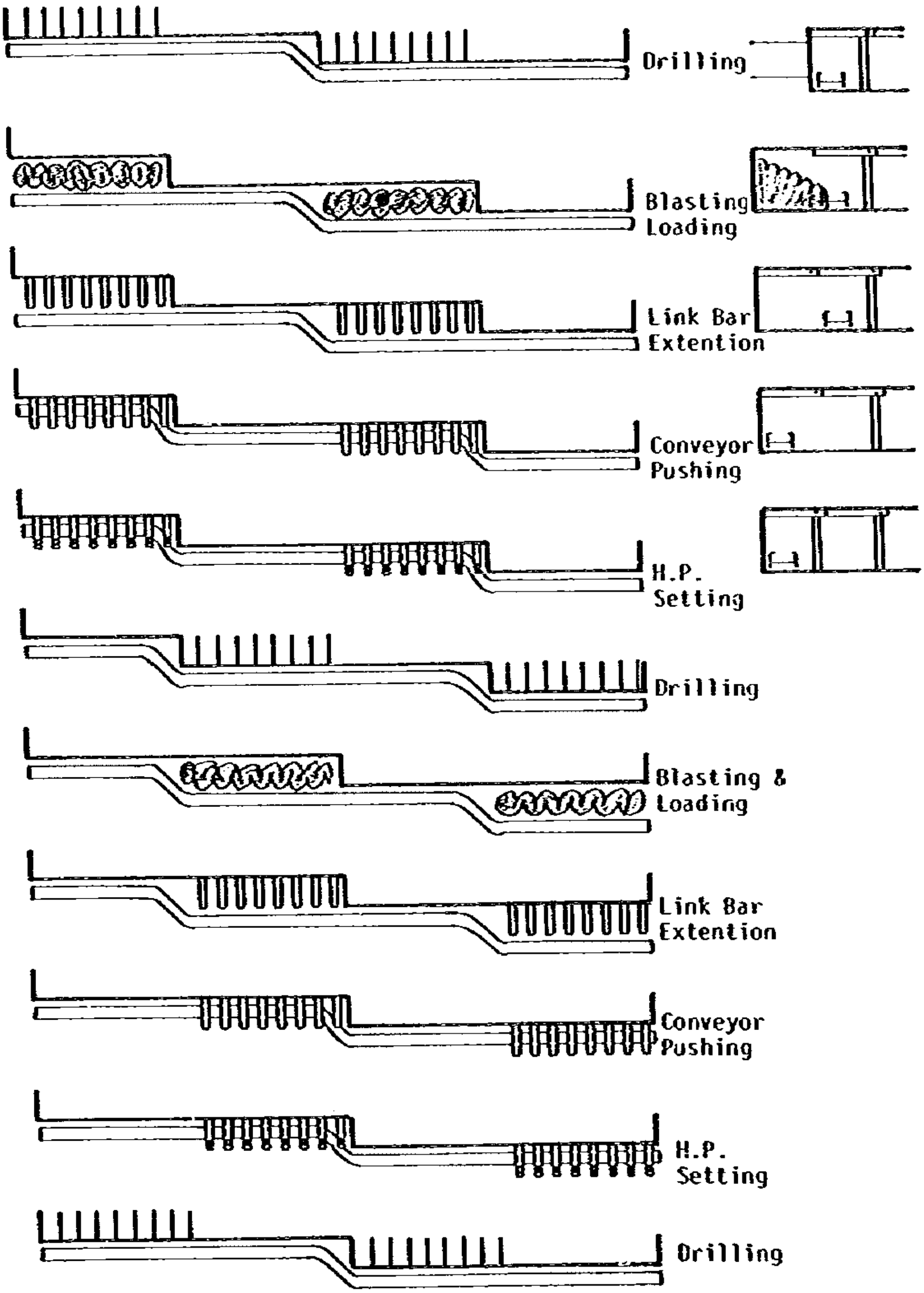
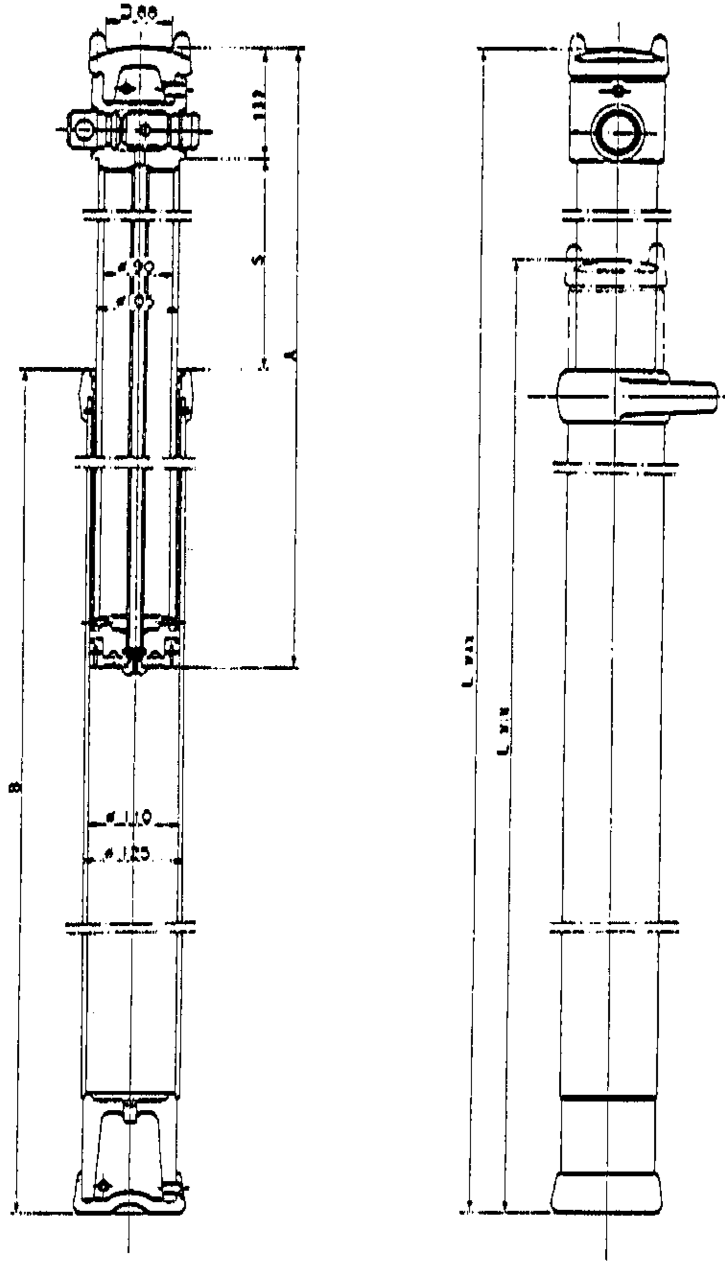


FIGURE 4-5 LONGWALL MINING METHOD



GENERAL SPECIFICATIONS								
NOMINAL SIZE (m)	L MAX (mm)	L MIN (mm)	S (mm)	A (mm)	B (mm)	WEIGHT (kg)	YIELDING LOAD (t)	SETTING LOAD (t)
1.8	1 800	1 155	445	990	1 018	61.0	40	28.5 (26.6)
2.0	2 000	1 280	720	1 115	1 143	66.0		
2.2	2 200	1 400	800	1 235	1 263	71.0		
2.4	2 400	1 500	900	1 335	1 363	75.0		
2.6	2 600	1 700	900	1 535	1 563	83.0		
2.8	2 800	1 900	900	1 735	1 763	91.5		

REMARK

1. SETTING LOAD ON DRAWING IS IN THE CASE OF WORKING PRESSURE 300 kg/cm².
2. SETTING LOAD IN ( ) IS IN THE CASE OF WORKING PRESSURE 280 kg/cm².

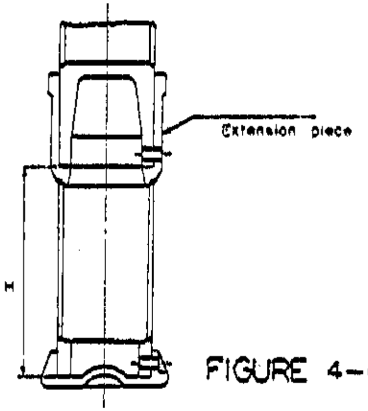


FIGURE 4-6 HYDRAULIC PROP

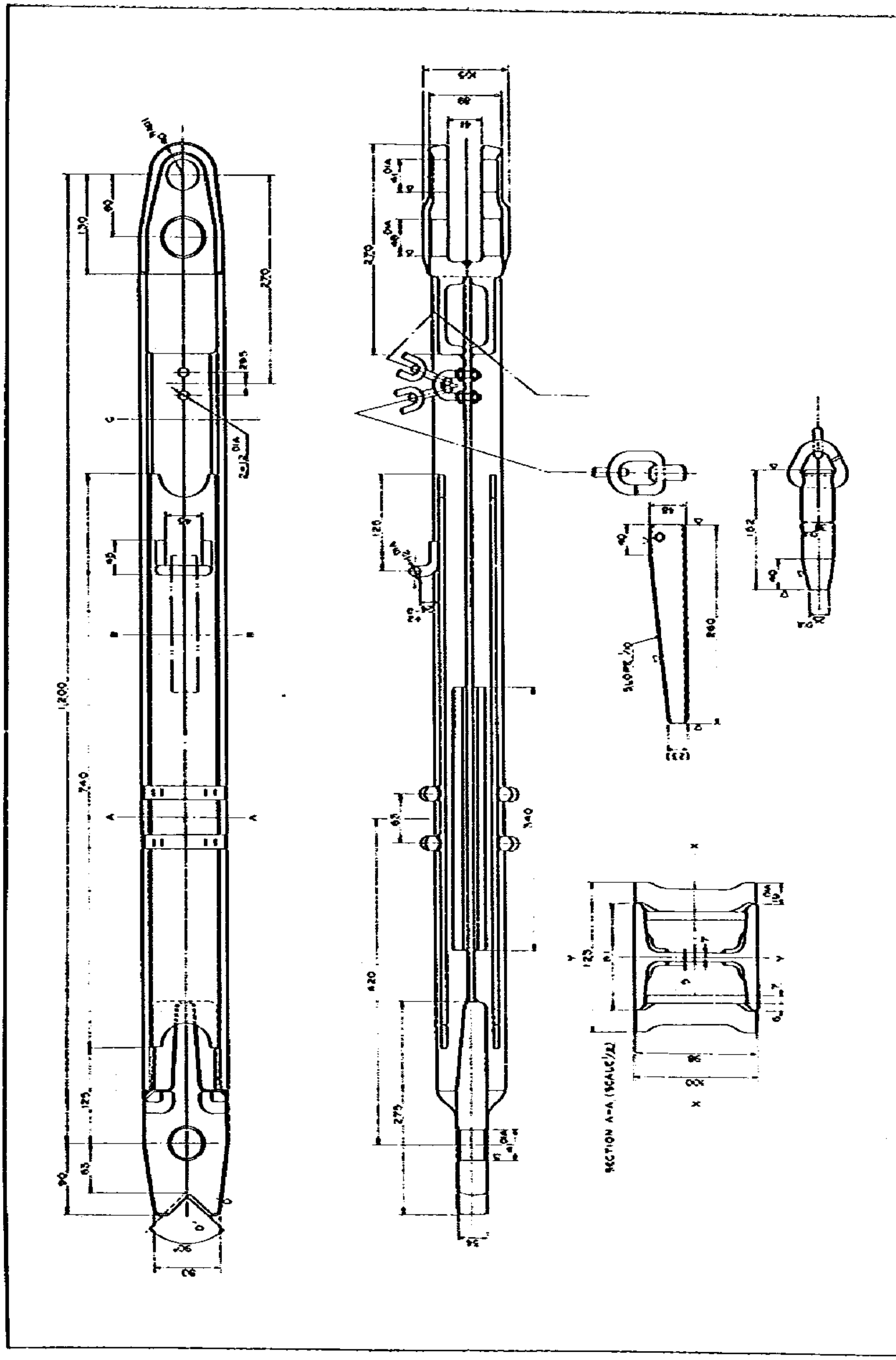
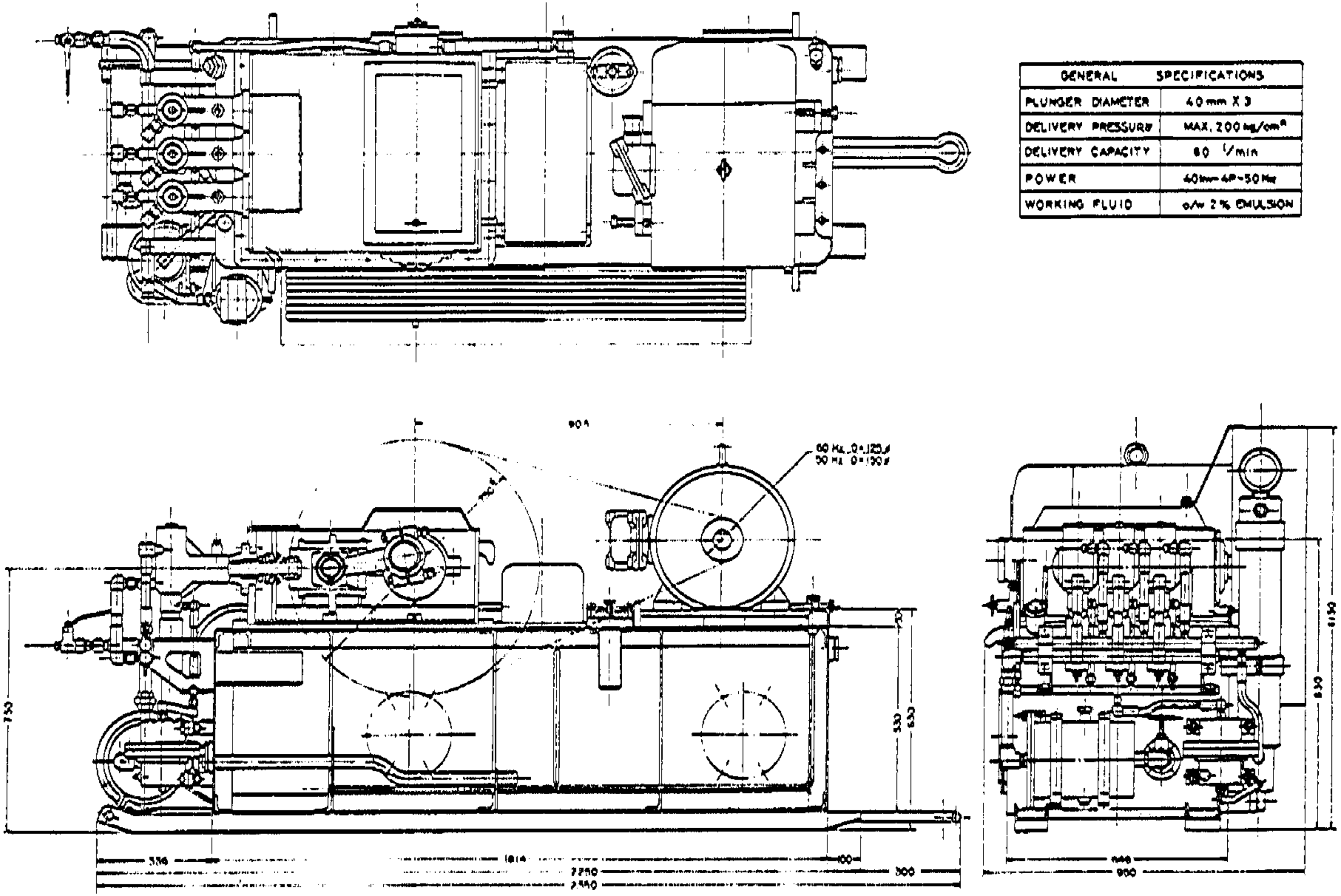
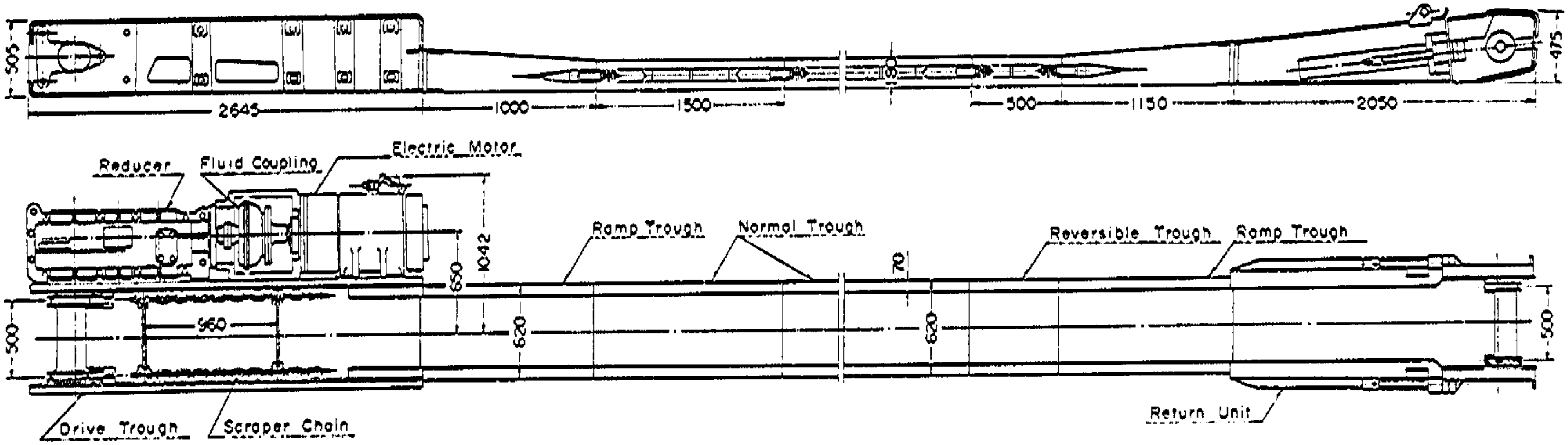


FIGURE 4—7 LINK BAR (PIN TYPE)



GENERAL	SPECIFICATIONS
PLUNGER DIAMETER	40 mm X 3
DELIVERY PRESSURE	MAX. 200 kg/cm²
DELIVERY CAPACITY	60 l/min
POWER	40kw-4P-50Hz
WORKING FLUID	o/w 2% EMULSION

FIGURE 4-8 PLUNGER PUMP



GENERAL	SPECIFICATIONS
TRANSPORTING CAPACITY	MAX 100 t/h
CONVEYOR LENGTH	120 m
POWER	40 kw
CHAIN SPEED	40 m/min
CHAIN SIZE	18$^{\phi}$ X 64^{P} X 11^{L}

FIGURE 4-9 ARMOURED FACE CONVEYOR

GENERAL SPECIFICATIONS	
BACKET CAPACITY	0,6 m^3
TRAVELING SPEED	0~3 km/h
WORKING AIR PRESSURE	5,5~7,0 kg/cm^2
AIR CONSUMPTION	15~22 m^3/min
TOTAL WEIGHT	6,600 kg

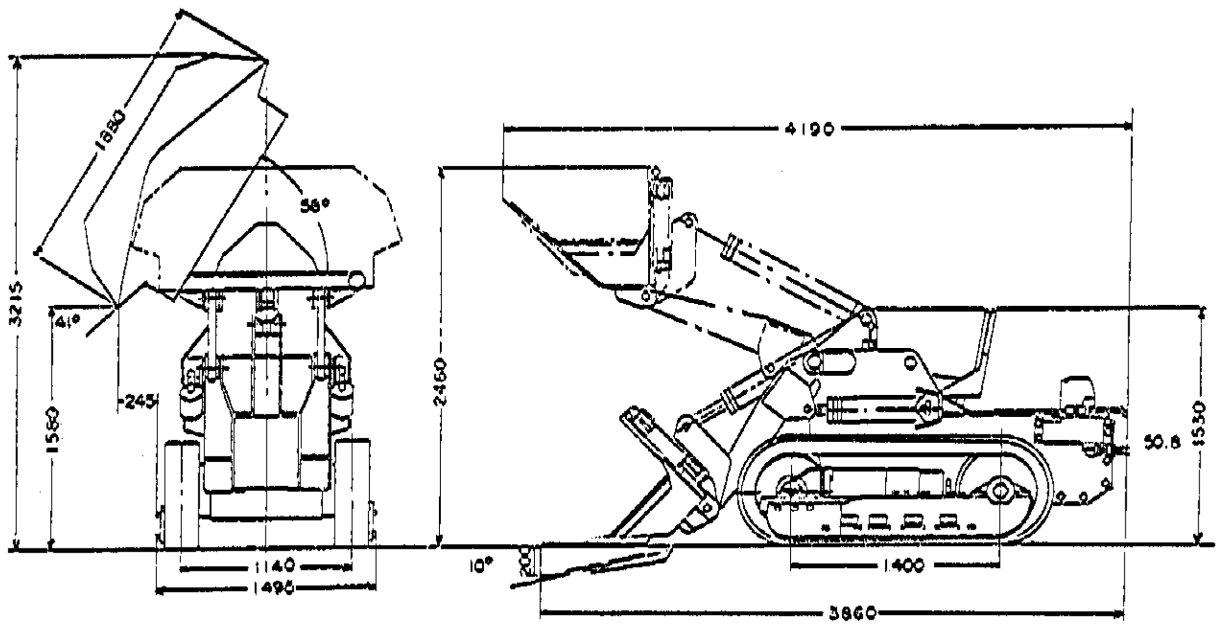
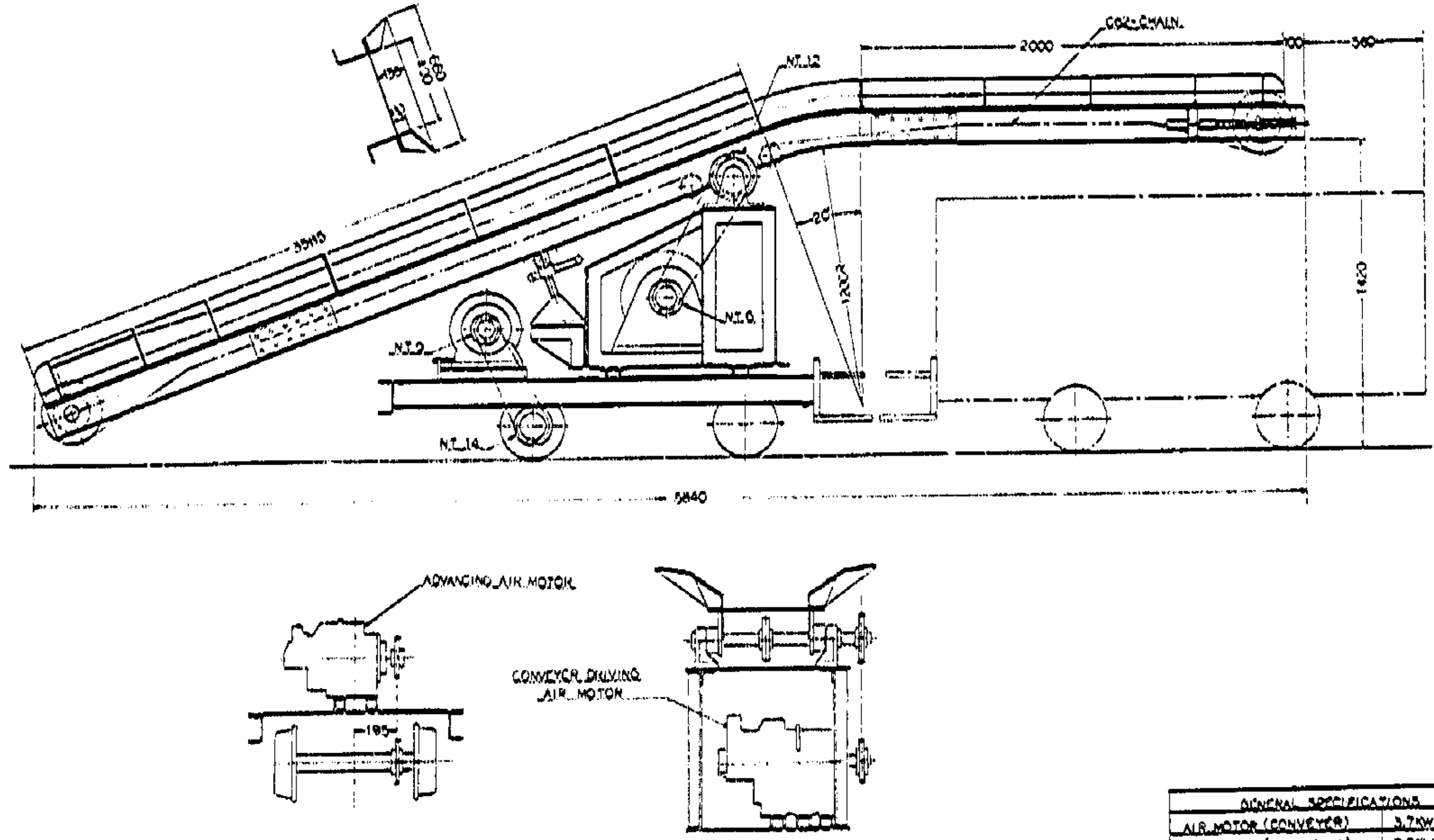


FIGURE 4-10 SIDE TIPPING LOADER



GENERAL SPECIFICATIONS	
AIR MOTOR (CONVEYER)	3.7KWx40RPM
AIR MOTOR (ADVANCING)	2.2KWx35RPM
CONVEYER SPEED	30 M/MIN
LOADER SPEED	25 M/MIN
TRANSPORTING CAPACITY	100T/H

FIGURE 4-11 GATE END LOADER

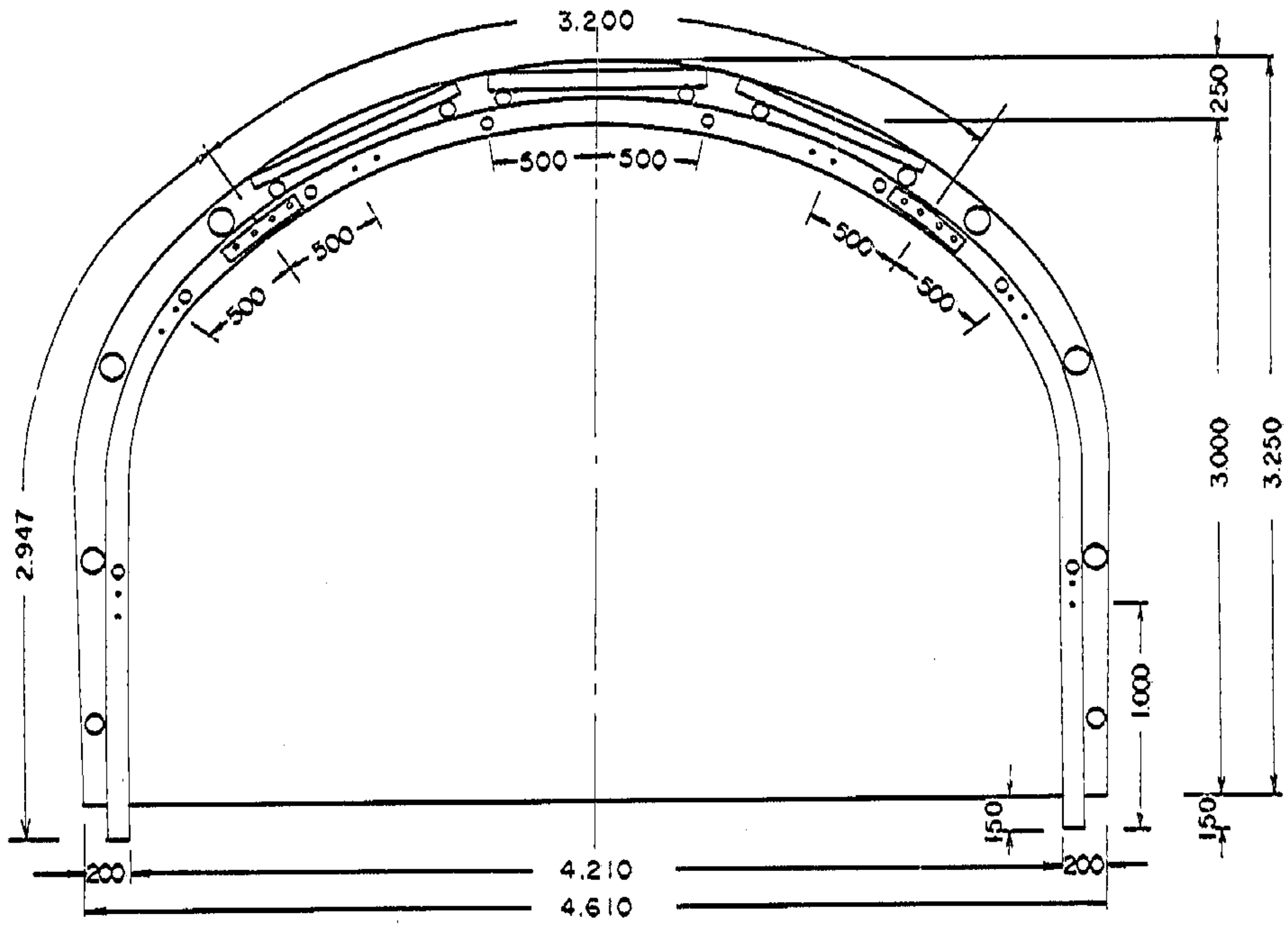


FIGURE 4-12 MAIN ENTRY (INCLINED SHAFT) 11.15 m^2

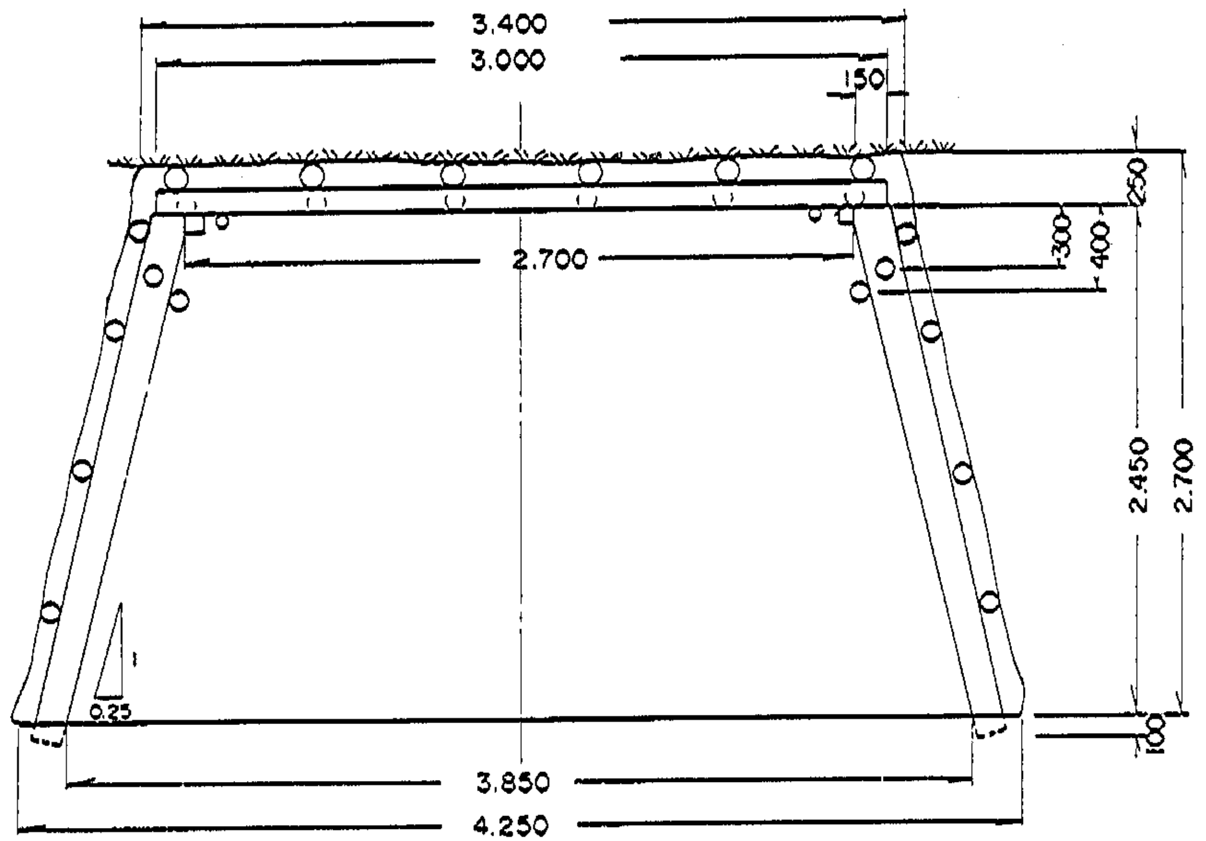


FIGURE 4-13 GATE ENTRY AND X - CUT (SINGLE TRACK) 8.0 m^2

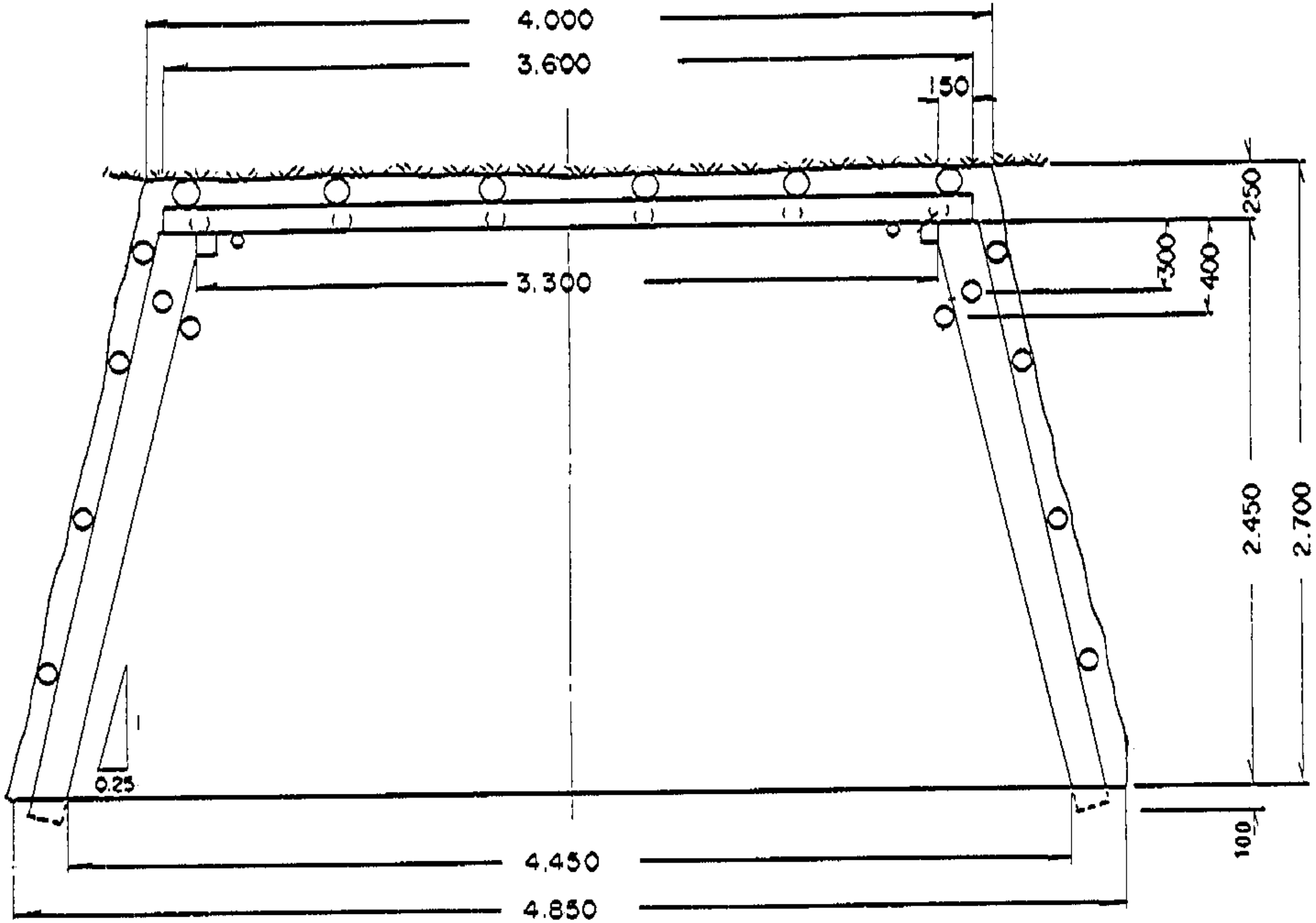


FIGURE 4-14 GATE ENTRY (DOUBLE TRACK) 9.5 m^2

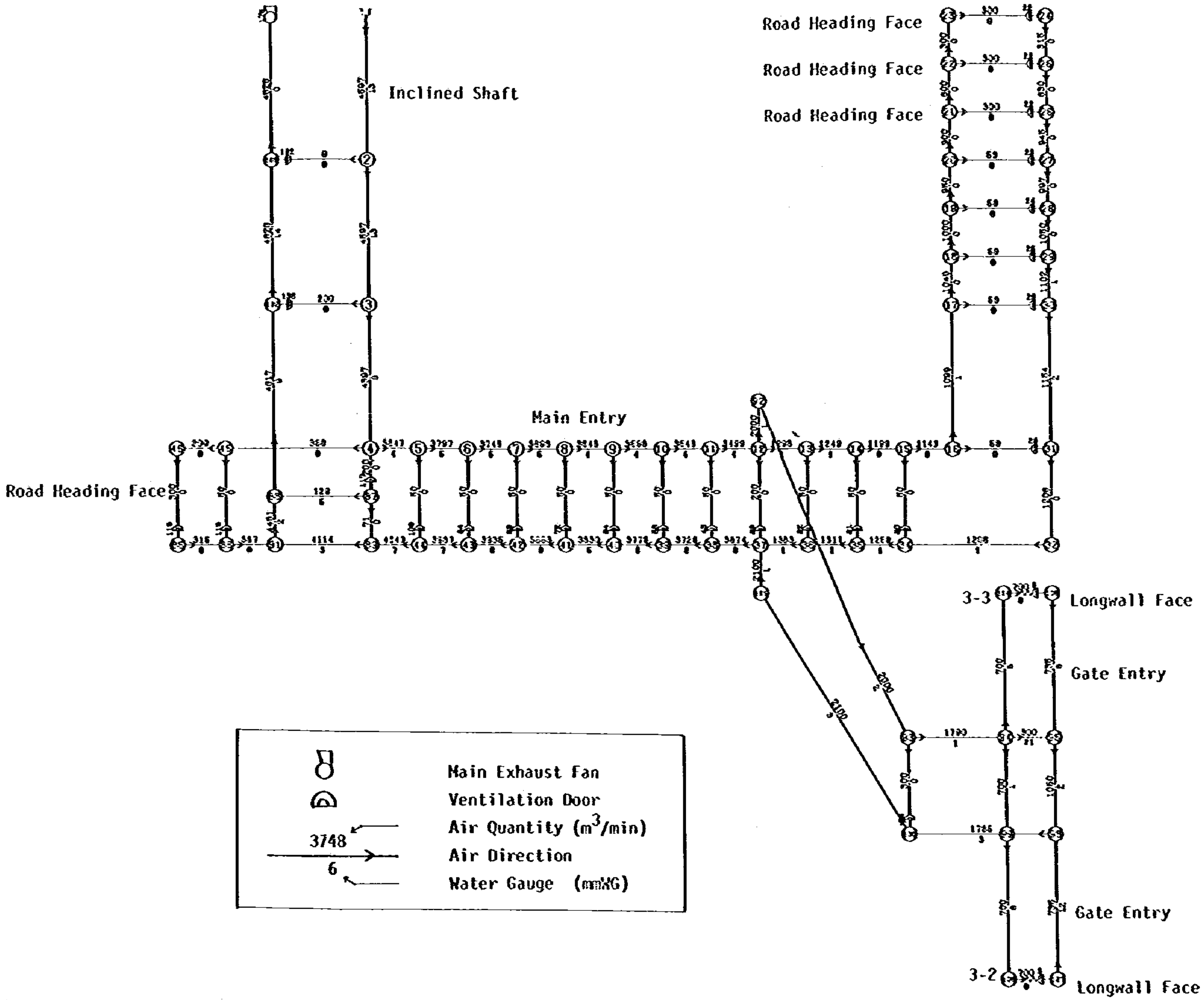


FIGURE 4-15 VENTILATION SIMULATION RESULT No.1 YEAR 1 992

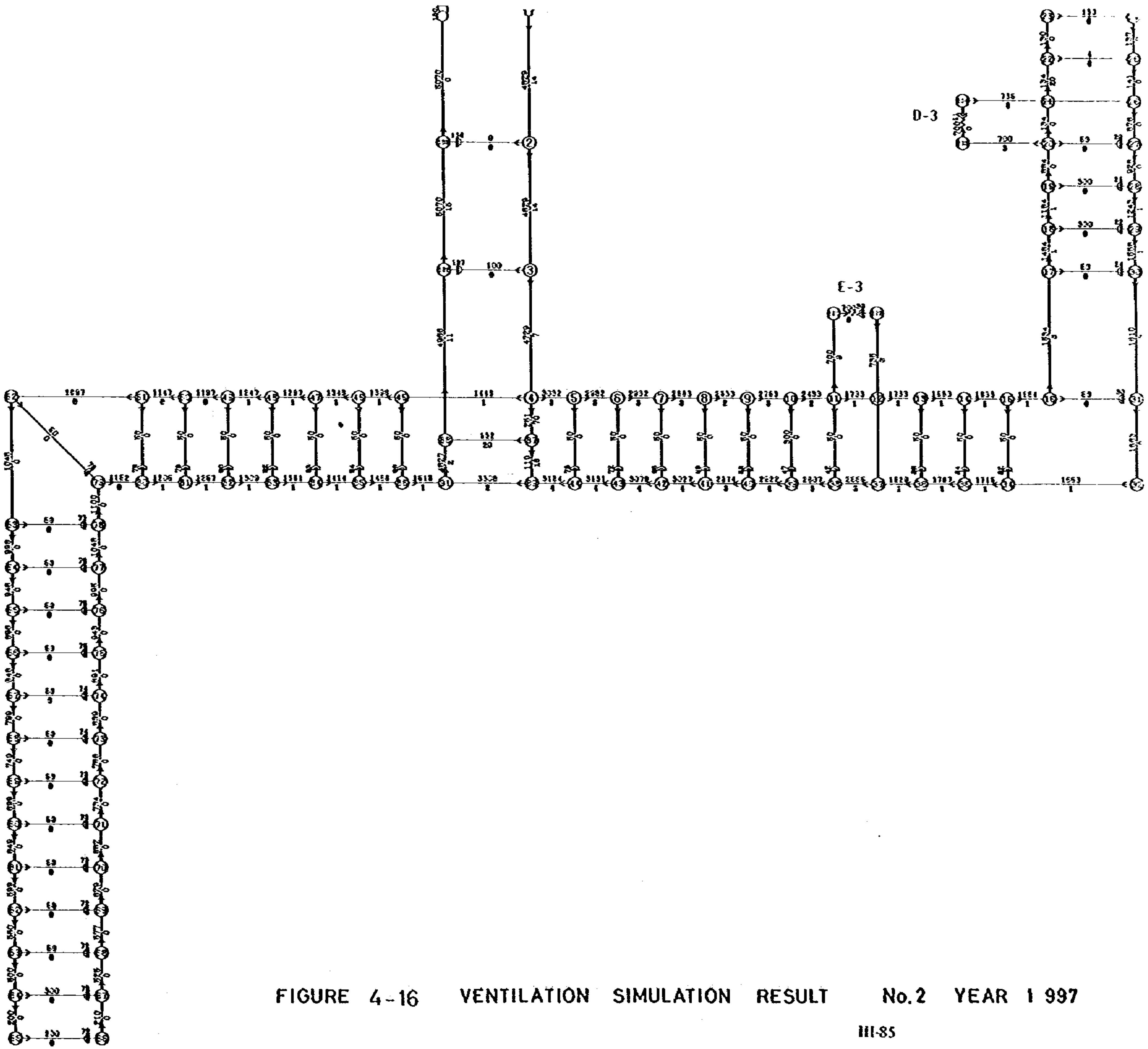


FIGURE 4-16 VENTILATION SIMULATION RESULT No.2 YEAR 1 997

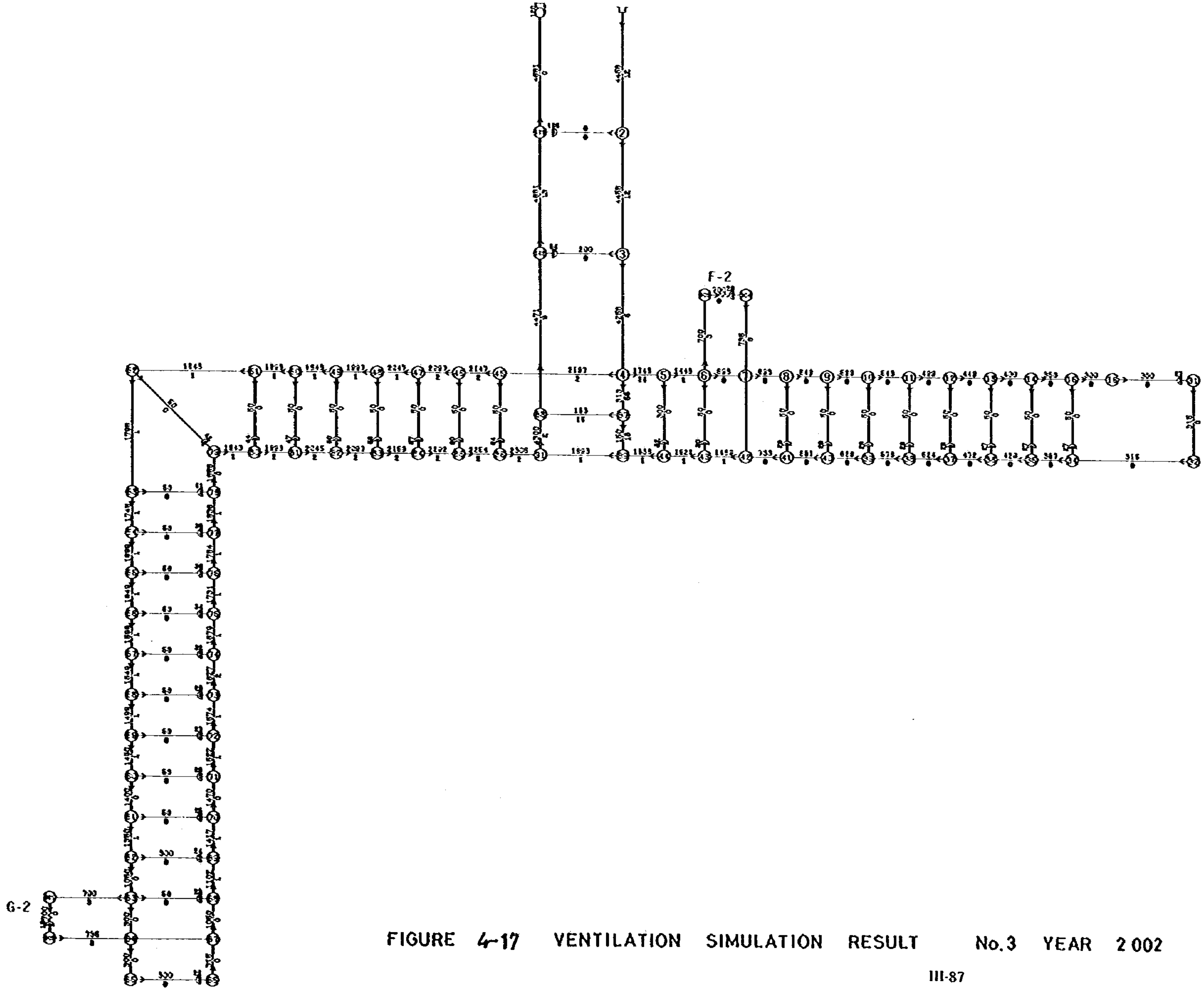


FIGURE 4-17 VENTILATION SIMULATION RESULT No.3 YEAR 2002

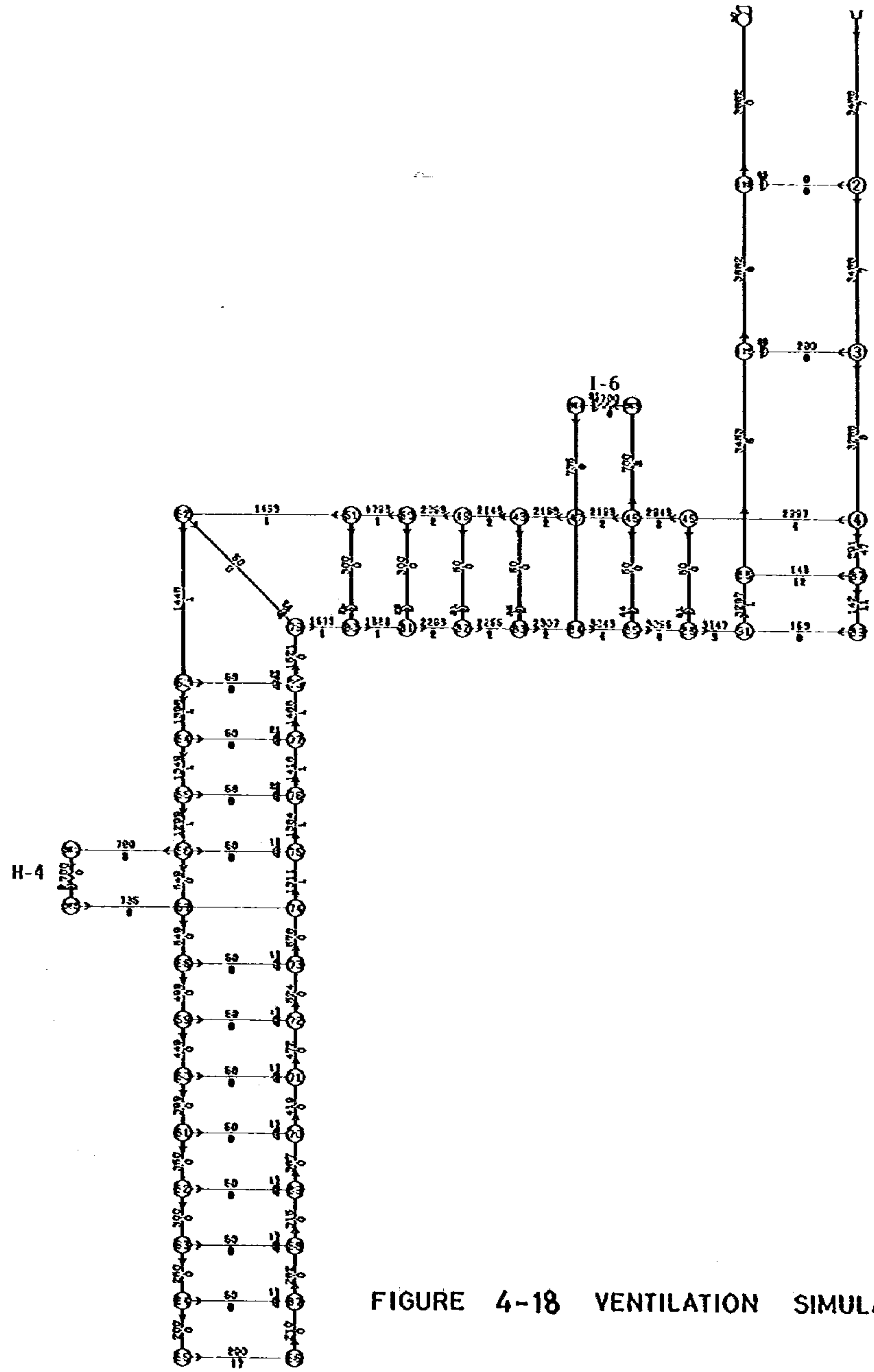
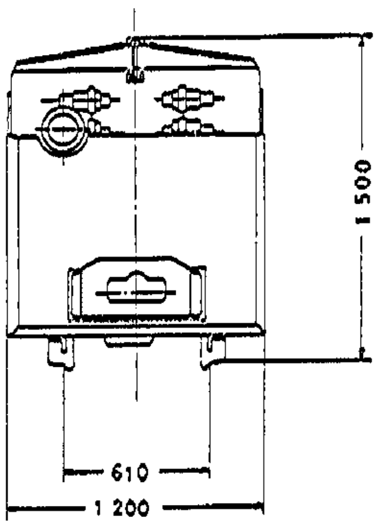


FIGURE 4-18 VENTILATION SIMULATION RESULT No.4 YEAR 2007

GENERAL SPECIFICATIONS		
NOMINAL WEIGHT		10 t
GAUGE		610 mm
RATING	TRACTION	1,700 kg
	SPEED	8,9 km/h
TRACTION MOTOR		22 kw x 2
		180 V
BRAKE		HAND, AIR
BATTERY CAPACITY		406 AH/5h



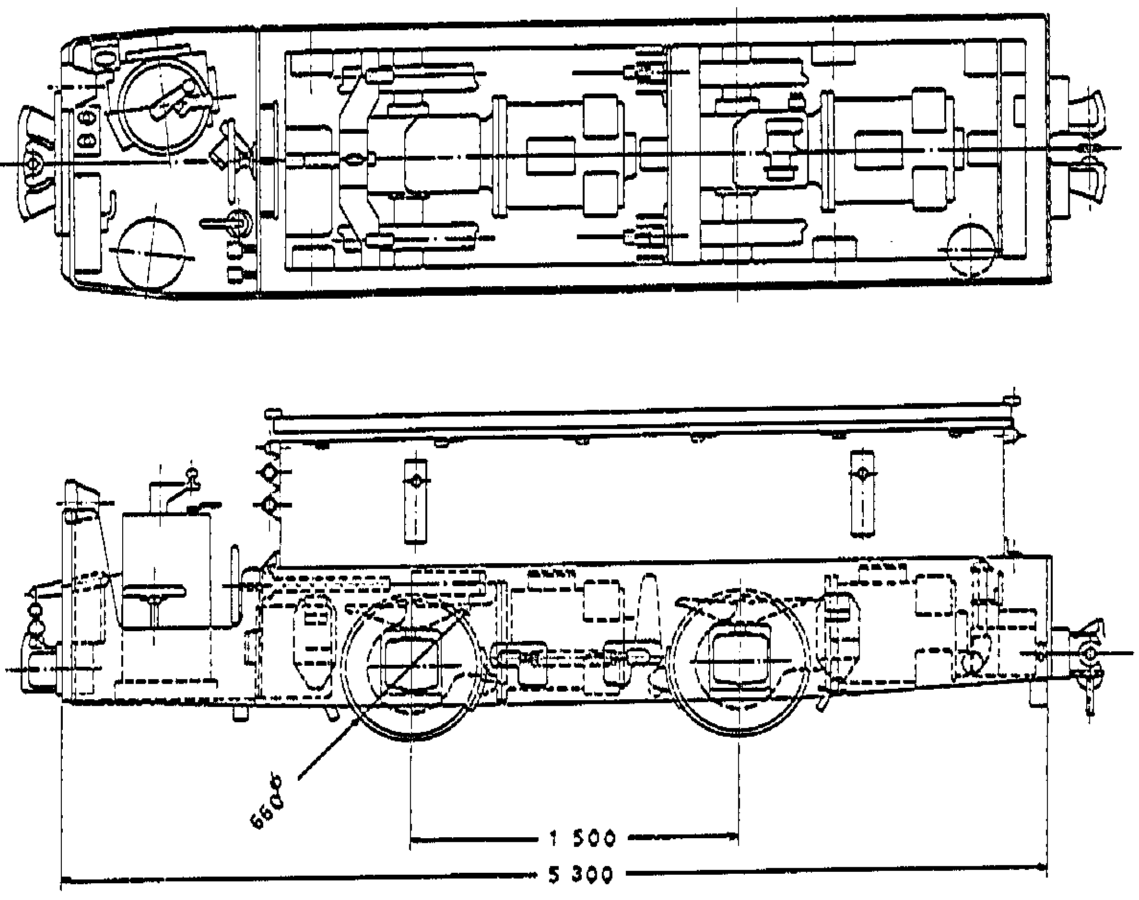
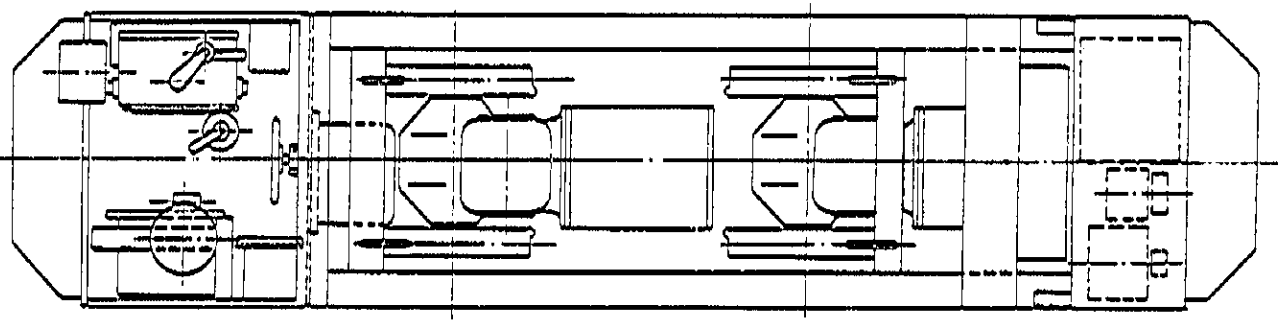


FIGURE 4-19 BATTERY LOCOMOTIVE (10 t)

GENERAL SPECIFICATIONS		
NOMINAL WEIGHT		8 t
GAUGE		610 mm
RATING	TRACTION	1,300 kg
	SPEED	8 km/h
TRACTION MOTOR		15 kw x 2
		180 v
BRAKE		HAND, AIR
BATTERY CAPACITY		372 AH/5h



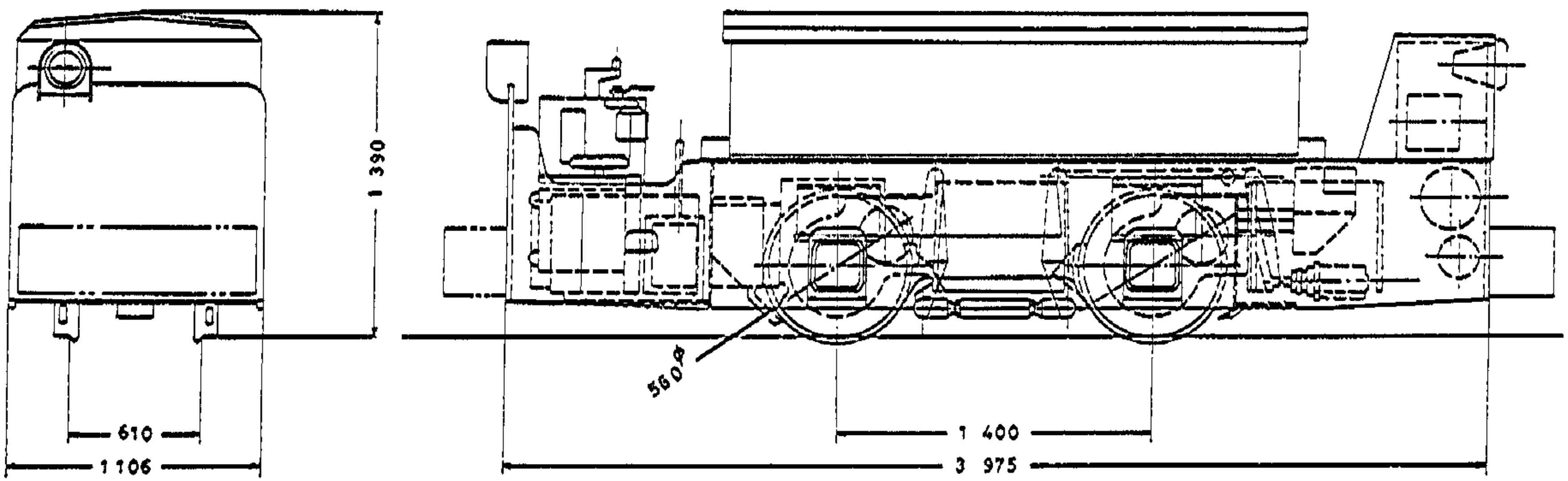


FIGURE 4-20 BATTERY LOCOMOTIVE (8 t)

LOADING CAPACITY	Coal 1,980 kg Rock 2,460 kg
WEIGHT	960 kg
BREAKING STRENGTH	90,000 kg

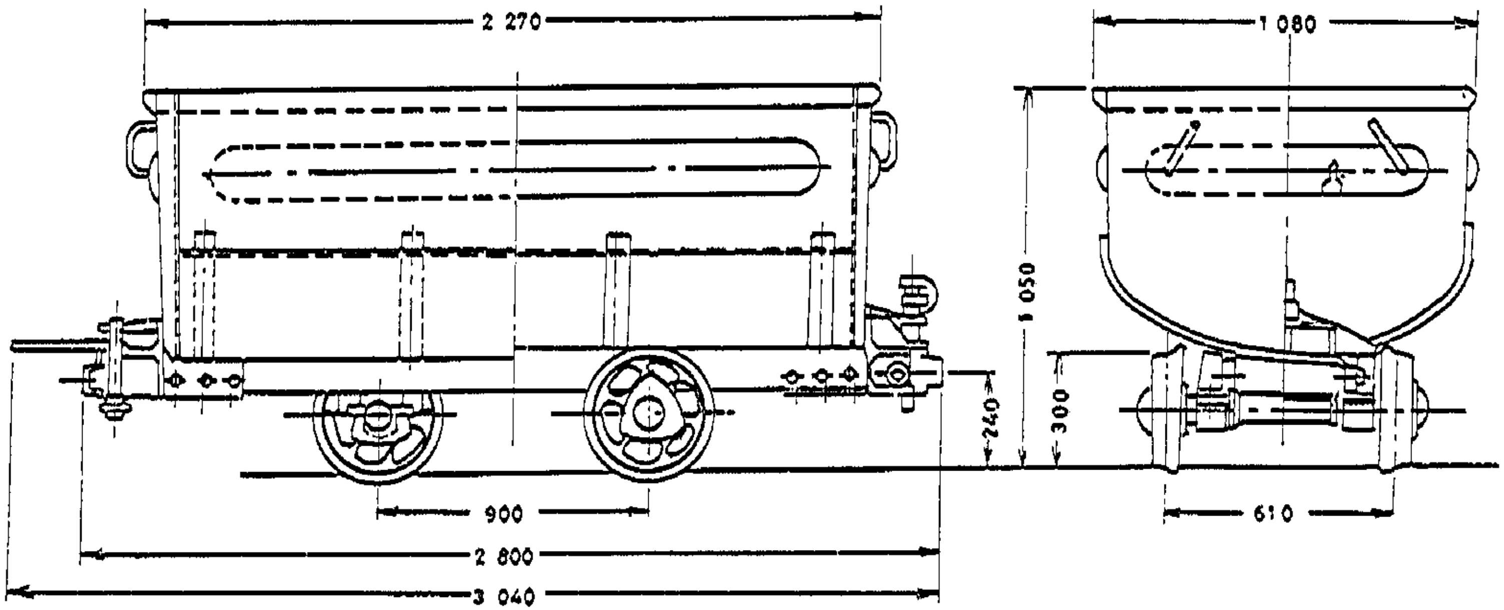


FIGURE 4-21 MINE CAR (2 m³)

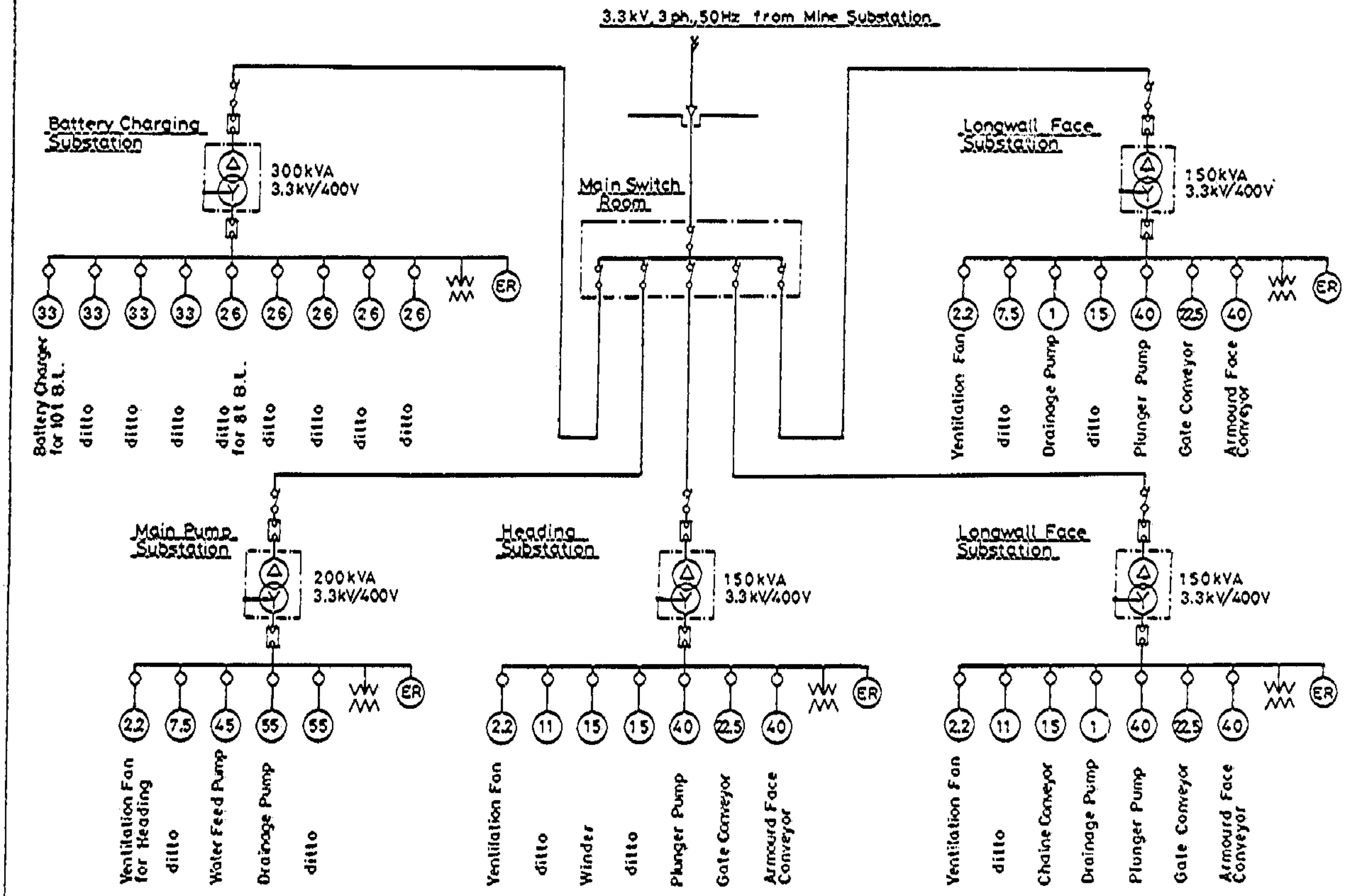


FIGURE 4-22 TYPICAL ONE LINE POWER DISTRIBUTION DIAGRAM IN UNDERGROUND

CHAPTER 5 OPEN PIT MINING

5-1 Assessment of Open Pit Mining Area

5-1-1 General Aspects

Examination of the geological structure of the investigated area indicated the most favorable areas to consider for open pit mining operations. In spite of lack of geological information in the western outside of the investigated area, it was necessary to assume that the geological conditions of the western boundary limit of investigation may possibly extend westward. This estimation is required in order to design and open pit in the western block as a main production unit for 30 years operation period.

In-situ mining ratio is comparatively high everywhere in Lakhra coal field. It is impossible to allocate the open pit mining area of the annual output of 800,000 tonnes coal (air-dried basis) for 30 years with a mining ratio of less than 10 m^3 of overburden per tonne of coal. A variety of isopach maps and contour maps have been created by the aid of computer techniques for selecting the open pit mining area. As a result, two open pit production units, namely West open pit and East open pit were separately allocated to the western block and the eastern block respectively.

5-1-2 West Open Pit

This pit covers an area in the south-western corner of the investigated field and has an effective mining ratio of 9.5 m^3/tonne of clean coal. Northern and southern limit are determined by the barren area proved by the borehole findings and eastern limit is also determined by the barren area and the fault line. Four recoverable coal seams are in this pit, namely No. 5, No. 3, No. 2 and No. 1 seam in descending order. This pit will be a main production unit of Lakhra coal mine. Then, it was necessarily assumed that the coal seams of the western limit may extend westward with same geological conditions. Accordingly, the coal production tonnage from West open pit has been estimated including the hypothetical reserves outside of the investigated area.

5-1-3 East Open Pit

This pit has a high effective mining ratio of 14.2 m^3/tonne of clean coal, because No. 5 seam is not recoverable in the eastern block and the overburden is deeper. Western and eastern limit are determined by the extremely high mining ratio and/or the barren area. From the view-point of mining condition, this area is suitable for underground mining rather than open pit mining, but for the purpose of keeping sufficient production planned for coal firing power generation the open pit mining method was planned for this block, and there is another reason why the open pit mining method is adopted that the recoverable seam is covered widely by a loose sand bed which may cause difficulties in roof support in the case of underground mining operations. The mining area is very restricted specially in western and eastern sides and is long and narrow area in the direction of north-south, which increases the mining ratio more than a roundly arranged area with same structural conditions.

5-2 Selection of Mining Method

5-2-1 Basic Considerations

In technical and economical points of view, the selection of mining method for the open pit was done based on the following peculiarities of the proposed area and considerations:

(1) The field for open pit mining is separated into two individual areas, namely the western block and the eastern block.

(2) More than one coal seam exists at the depth of 33 to 85 meters in the western block and 45 to 91 meters in the eastern block. Accordingly, the quantity of overburden to be removed is relatively large compared with the tonnage of coal seam uncovered.

(3) In order to assure an effective and stable operation, the mining area must be concentrated as much as possible and it is necessary to abandon reasonably juts and narrow or isolated zones.

(4) In order to keep coal production stable as well as reliable and to minimize the influence by machine trouble to coal production, more than one stripping unit is to be provided.

(5) It is recommendable in the first case of introducing the mechanized coal extraction system on a large scale that the open pit mining method commonly applied in the world would be selected and the type and size of heavy machinery would be world-wide popular and of reasonable purchase price.

(6) Considerations were focused also on important factors as maintenance capability, operational experiences, availability of related techniques and engineering, easiness of parts supply and capacity of repair work for heavy machinery to be applied.

5-2-2 General Description of Open Pit Mining

Three basic methods for large-scale open pit mining can be selected in overburden stripping and/or removal, the most expensive part of the operation for excavating moderate coal seams overlaid by thick overburden.

Three basic methods for overburden stripping operation are as follows:

(1) Overburden removal with bucket wheel excavators.

(2) Overburden stripping with walking draglines.

(3) Overburden removal with shovels or loaders and trucks.

Combination of any two of the above mentioned will be adopted if necessary.

(1) Overburden Removal with Bucket Wheel Excavators (BWE)

It is well that BWE's were first built to handle only the easy digging materials. But more sophisticated steel alloys have been developed to meet the problems of wear, subzero temperatures and steel fatigue. Electric drives have improved, with the Ward Leonard System and shovel, bucket and teeth manufacturers are continually improving the digging characteristics and the tooth and bucket designs. The conveying industry developed steel cord belts, new synthetic coverings, and improved hardware which have promoted belt speeds of over 300 metres per minutes. These improvements now permit the BWE's to dig medium to medium-hard materials.

Compared with shovels or draglines of equal output, the BWE is physically smaller because it is a continuous excavator. In contrast with shovels or draglines BWE has:

a) Lower instantaneous power demands,
b) Less weight for greater output, and
c) No shock loading.

Basically, a BWE equipped with high-speed belt conveyor system can handle effectively a large amount of uniformly soft materials which fragment well without blasting. However, the proposed area of Lakhra coal field includes the following disadvantages in using BWE:

a) Comparatively hard overburden which consists of limestone of about 20 metres in average thickness.

b) Interbedded hard partings and more than one coal seam to be mined which slows the production unduly and increases maintenance cost.

c) The relatively high initial capital cost.

d) Necessity of excellent technical levels in engineering, operation and maintenance, which are indispensable for keeping the stable availability of a total BWE system.

e) Difficulties in selecting and establishment of design of BWE suitable for the conditions of Lakhra coal field, because no past examples.

f) Extremely high cost of special design and additional refinement required by job complexity.

g) Necessity of various large-size equipment and engineering technique for erection, reconstruction and removal of BWE in mine site.

h) Necessity of immense quantity of overburden removal for initial installation of BWE and belt conveyor system and preparation of working benches.

i) Current delivery and erection schedules for BWE's are reported to be 48 months.

(2) Overburden Stripping with Walking Dragline

Draglines have been used for area stripping in surface mining operations for about 50 years and they have emerged as the dominant stripping tool in the past 20 years or so. The mining method in which dragline is the leading equipment is strip mining uncovering a single coal seam by side casting overburden. As mine operators have been faced with ever-increasing depths of overburden, bigger machines have been employed by reason of economical advantages.

With the development of walking mechanisms, a whole new world in dragline size opened up. The design, which allows the walking mechanisms to be lifted out of the way so that the dragline sits on its massive tub, combines low enough ground pressure to make even the heaviest dragline suitable for use in soft or wet conditions.

The trend toward larger and heavier draglines, however, has leveled off. Dragline manufacturers seem ready to concede that walking draglines have reached the limit of their upward growth curves. The trend has shifted toward longer booms with about the same size bucket, or even a bit smaller.

The dragline used for stripping the overburden of more than 30 meters in depth, like as Lakhra's condition, must be equipped with a long boom and a comparatively small size bucket and forced to rehandle a large part of overburden. Generally, walking draglines are advantageous stripping machines from the view-points of digging ability, travelling capability and capacity of land reclamation. However, the application of draglines under Lakhra's conditions includes the following disadvantages:

a) Open pit operations must be done in two individual areas. Accordingly, the stripping operation cannot be concentrated in a large-scale production unit or efficiently performed.

b) Anyway, upper part of overburden should be removed by shovel/truck system to make a suitable depth of overburden for dragline operation. This causes and increase in capital cost.

c) More than one coal seam is to be recovered using either the basic or rehandle approaches. But Lakhra's conditions require that the parting between two coal seams is necessarily removed to make multiple seam mining possible and coal haulage ramps are always maintained. This causes a complexity in operation and a decrease in productivity.

d) Large excavating equipment is in short supply. Lead time for delivery of large draglines presently is reported to be as long as three to five years. The planners facing difficulties in acquiring equipment needed to start a new project must give consideration to any other mining systems for which equipment is now available, even if they may be less optimal.

e) Erection work of dragline require skillful engineers, capable contractors, large transportation vehicles and large cranes. Normally this work takes more than ten months, even in the United States.

(3) Overburden Removal with Shovels or Loaders and Trucks

The principle of the shovel is widely used in open pit mining operations. Generally, the term "open pit mining" is applied to a mining activity quite popular in surface copper mines and surface iron mines. The activity is often referred to as a quarry operation. The material to be handled in Lakhra coal field is generally hard and/or consolidated overburden as limestone. It must be prepared for digging to drill and blast the overburden. In order to effectively and efficiently dig the prepared material, a heavy duty, close-coupled shovel is required. In the market the machines are known as quarry-mine shovels.

In the below-the-surface-type excavation as Lakhra coal project, the average side slopes vary between jobs depending upon pit geometry and the rock mechanics aspects of materials. To provide working faces and haulage ways, benches are developed in the side of the excavation. For economic and technical reasons, the bench widths are usually restrictive and limit the maneuvering of the equipment.

The operating ranges of a shovel are usually controlled, in part at least, by the clearance required to conveniently load into the matched haulage units. Generally a quarry-mine shovel is operated in conjunction with haulage trucks. At present, this type of shovel ranges in dipper size from about 5 m^3 up to about 20 m^3. The excavating unit is small and compact in comparison with BWE or walking dragline.

The overburden removal with shovels and trucks (shovel/truck system) is said to be unfavorable compared with BWE or walking draglines in following aspects:

a) Lower productivity and more manpower required.
b) Expensive operating cost.
c) Large repair shop and expensive maintenance cost.

However, this method is employed at many open pit coal mines where seams are pitched. Multiple seam mining can be done effectively by shovel/truck system. The application of this mining method to Lakhra coal project includes the following definitive advantages:

a) The lead time for delivery of equipment is remarkably short, compared with that of BWE or walking dragline. That is the unique key which will permit Lakhra coal excavation to start earlier.

b) Capital cost is less expensive and erection work of equipment in mine site is very little.

c) Maintenance work is technically easy because no specially designed facility is required and parts are mostly popular and available in the world.

d) Machine trouble has not so much influence as in the case of dragline operation, because more than one shovel is generally employed at one open pit.

e) Adaptability of total operation system is flexible under the variation of geological conditions.

f) Long term and expensive costs for training of operators and maintenance workers are not necessary.

5-3 Applicable Mining Method

5-3-1 Conception of Mining Method

Overburden removal with shovels and trucks was selected as a mining method to be applied to Lakhra open pit planning. The decision to employ this mining method was based on the above-mentionned advantages of shovel/truck system and additionally the following reasons:

a) The coal seams to be excavated are mostly horizontal. However, the overburden is relatively thick in whole area for surface coal mining so that the simple side casting with a large sized dragline is impossible.

b) Multiple seam mining operation with partings removal is proposed.

c) A systematic overburden removal must start early 1983 based on the development schedule of Lakhra coal mining and power station project by the request of Pakistani authorities. From the viewpoint of delivery time of stripping machine, the shovel/truck system is the only applicable method satisfying the development schedule.

A hypothetical pit arrangement for open pit mining is shown in Fig. 5-1. The coal seam is mined in successive parallel strips while the coal is uncovered by a shovel filling a fleet of haulage trucks which transport the overburden to back fill the area behind the working pit. The sequence of operation in the pit is as follows:

(1) Top soil removal
(2) Overburden drilling and blasting
(3) Overburden loading
(4) Overburden hauling
(5) Partings drilling and blasting
(6) Partings removal
(7) Coal drilling and blasting
(8) Coal loading and hauling
(9) Restoration of excavated area
(10) Pit services and road maintenance

5-3-2 Basic Extraction Methodology

Prior to commencing the detailed extraction design the basic extraction methodology had first to be determined. Examination of the open bit mining areas indicates the following seam characteristics:

(a) The mining areas offered a multi-seam deposit with four mineable seams in the western block and three in the eastern block including some with multiple splits.

(b) The thickest coal seam is generally the lowest in the sequence.

(c) The partings and coal seams vary remarkably in thickness.

(d) Seams are generally moderate and almost horizontal.

A flexible, selective mining method is therefore essential to ensure maximum extraction of reserves and to permit the reliable and constant supply of clean coal to the power station.

Before finalizing the mining system to be adopted, the following alternative methods of operation were considered:

(1) Total Overburden Removal with one Dragline

Generally, it is said that a dragline usually gives the lowest cost for overburden removal but the following difficulties would occur:

a) Dragline geometry cannot be optimized to adequately deal with the multi-seam situation and the thickness of overburden to be removed.

b) Scheduling the dragline application would be difficult.

c) The method would be inflexible for selective exposure of the multiplicity of thin seams.

(2) Total Overburden Removal with two Draglines

Because of the thick overburden and multiple seam situation, there may be in excess of five working horizons and the application of a double dragline method of working was considered impracticable. Rehandling quantities would be large and due to the restricted pit length available, inpit operation would be extremely difficult to schedule.

(3) Combined Shovel/Truck and Dragline Operation

This system of mining could be used for all two pits. Technically speaking, this combined system is applicable and flexible to multiple seam operation and advantage would be taken of the productive benefits of moving a large percentage of the overburden with the dragline. However there is a difficulty of scheduling the dragline application and the initial capital expenditure will be extremely large, additionally maintenance work and parts supply will also be difficult.

As before-mentioned, it was therefore concluded that the Lakhra open pit should be mined as a shovel/truck operation.

5-3-3 Determination of Extraction Sequence

(1) Topsoil Removal

The area to be mined is cleared of brush and trees in advance where necessary and topsoil is removed forming a working bench for blast hole drilling. In case of Lakhra mine site, topsoil removal may be practically cut down, because trees are rare and the area looks like a desert of rocks. Bulldozers are mainly used for surface grading.

(2) Overburden Drilling and Blasting

The overburden is drilled and blasted, as required, along a strip according to the width of the panel being excavated, with sufficient lead time to avoid interference with the advance of the shovel. Fundamentally multiple-stage of working benches is formed. Electric rotary drills capable of producing blastholes 9-7/8 in. in diameter are used for overburden drilling and the holes are loaded with ANFO explosives. ANFO is capable of breaking the overburden for easy digging.

(3) Overburden Loading

The prepared overburden is stripped using an electric mining shovel to make a working bench. The operating ranges of a shovel are usually controlled, in part at least, by the clearances required to conveniently load, into the matched haulage unit. This is significant because generally a quarry-mining shovel is operated in conjunction with haulage trucks. The level of working bench is cleaned up by a bulldozer and this groundman's operation is always associated with the stripping and loading operation with shovel.

(4) Overburden Hauling

The overburden is hauled out of the pit in dump trucks and dumped into a backfill area behind the working pit. For this purpose, rear dump trucks of conventional type are used. They have a body mounted on the truck chassis that is raised by means of an integrally mounted hydraulic hoist system. These units cannot be used for any but off-highway service since they exceed legal width and weight limits.

(5) Partings Drilling and Blasting

After the top seam uncovered is loaded, the parting overlaying the next lower seam must be removed. Partings are drilled and blasted prior to loading to facilitate the digging operation, if necessary. A hydraulic drill capable of producing blastholes 80 mm in diameter is used for this purpose.

(6) Partings Removal

Partings prepared by blasting are removed by wheel loader/dump truck or motor scraper/bulldozer method, depending upon the operating condition. A scraper can be very productive on short hauls. Because of the short loading time of a scraper (as compared to a truck), a scraper can complets an entire short cycle before the truck is loaded. Dozers can push and remove the parting down into the mined-out floor, while scrapers haul the parting to the dump area (Fig. 5-5).

(7) Coal Drilling and Blasting

The exposed coal seams may be easily ripped by a bulldozer or a hydraulic excavator. Coal seams seem to be so soft that blasting is not necessary, but thick part of coal seam is recommended to be drilled and blasted for easy digging. A hydraulic rotary drill used for partings drilling is available for coal drilling.

(8) Coal Loading and Hauling

The multiple seam mining can be generally done in the lowest bench using a hydraulic excavator, dozer or scraper. The coal is loaded into rear dump trucks using a hydraulic excavator equipped with a 6.0 m^3 bucket. The trucks leave the pit by way of ramps through the spoil pile and discharge ROM (run-of-mine) coal into a ROM hopper near the preparation plant.

(9) Restoration of Excavated Area

The restoration of area disturbed by mining should be planned to complete the final surface contours. Overburden tips outside of the mining areas are to be designed for permanence and all restored ground levels and contours will be formed to blend in with existing topographical features in the undisturbed areas. Progressive restoration of the excavated areas will commence at the boundary of initial pit opening and steadily westward in West open pit and northward in East open pit during the life of the mine.

(10) Pit Services and Road Maintenance

For the maintenance, repair and supply work of the heavy equipment operating in pit, it is always necessary that a variety of service equipment and vehicles as crane, welding unit, cargo truck should be available. Road maintenance is also very important because a large quantity of material must be hauled through the surface of mined-out area.

5-4 Pit Layout and Major Equipment Selection

5-4-1 Pit Layout

Pit layout is dictated by the geometry of the major extraction equipment chosen and method of its operation. Shovel/truck system is applied to make the multiple stage of working benches and loader/truck or dozer/scraper system is expected to be suitable for coal and partings removal.

(1) West Open Pit

Initial box cut is oriented as close as possible to the eastern limit of in-situ mining ratio of 10/1 in the direction of NS. The cut width and height were determined by reference to the shovel and truck dimensions and the system of working adopted (See Fig. 5-4). Three stages of working bench are necessary to be prepared to uncover the top coal seam by using of quarry-mine type shovel (See Fig. 5-2). Main ramps into the pit run generally in a E-W direction. The pit advances westward being filled back the excavated area.

(2) East Open Pit

Initial box cut is located close to the southern limit of the planned area with in-situ mining ratio of 12/1 and parallel to the southern boundary of PMDC licenced area. Pit operation was intended to perform in same concept as that of West open pit. The length of pit is comparatively short because the mining area is limited both in western and eastern side. The pit is allowed to advance only northward and ramps run into the pit generally from the western side of pit.

5-4-2 Design Criteria

Main design criteria of open pit mining were determined as belows:

(1) Operating Time

8 hours/shift, 3 shifts/day, 300 days/year

(2) Factors concerned with Reserves

Geological safety factor:	80 %
Recovery factor:	90 %

(3) Geometry of Pit

Height of bench:	14.0 m
Slope of bench:	60 degrees
Final slope of pit:	45 degrees
Max. width of bench:	60 m
Min. width of bench:	20 m
Width of bench where shovel operates:	40 m
Max. slope of ramp:	8 degrees
Width of haul road:	14 m

(4) Overburden Drilling and Blasting

Diameter of drill hole:	250 mm (9-7/8 in.)
Drill hole spacing:	7 m x 8 m
Average ratio of penetration:	23.5 m/h
Powder factor:	ANFO 0.44 kg/bank m^3

(5) Overburden Removal (Overburden Loading and Hauling)

The average productivity of overburden removal with shovel/truck was estimated to be 137 bank m^3/truck/h, considered the results of computer simulation for shovel/truck system (See ANNEX 5).

(6) Coal Loading and Hauling

The average productivity of coal loading and hauling with shovel/truck was estimated as belows:

104.4 tonnes/truck/h	for West open pit
95.7 tonnes/truck/h	for East open pit

(7) Mechanical Availability of Equipment

Electric mine shovel:	75 %
Dump truck:	70 % (120 t truck), 75 % (46 t truck)
Drill:	75 %
Hydraulic excavator:	72 %
Wheel loader:	70 %
Bulldozer:	70 %

(8) In-pit Inventory of Coal

In-pit inventory coal was scheduled at least for about three months and the annual amount of overburden to be prepared was arranged so that the inventory of coal can be sufficiently assured during the whole period of operation.

5-4-3 Major Equipment Selection

Since both pits have similar operating conditions, pit dimensions and geological structure except the number of mineable seams, the type of equipment as well as the number of machines to be employed is not much different between both pits. Annual overburden stripping and coal production schedules are shown in Table 5-1 to Table 5-3 and annual ROM coal mined from open pits is summarized in Table 5-4.

(1) Overburden Drilling and Blasting

The overburden which consists of rock formations stratified above the top coal seam, is drilled with two electric rotary drills of crawler-mounted type (Fig. 5-6) in each pit. The drill is capable of producing blast hole 250 mm (9-7/8 in.) in diameter and prepares the overburden to be blasted for shovel loading.

Topsoil removal, surface clean-up and grading on the drilling site are performed with three bulldozers in each pit.

(2) Overburden Loading

Two 11.5 m^3 electric shovels (Fig. 5-7) are used for overburden loading in each pit, because two shovels equipped with 11.5 m^3 dipper are sufficient and suitable for overburden removal of approximately 5 million bank m^3 per year and chosen as a main overburden stripping machine. A groundman operates bulldozer to grade and clean up on each shovel working bench.

(3) Overburden Hauling

Diesel electric rear dump trucks of 120 t max. payload (Fig. 5-8) are selected as overburden hauling equipment. The number of trucks was determined for each pit based on the result of "loading machines/truck simulation" (Refer ANNEX 5). Each pit is equipped with 9 trucks operating on the three shifts/day.

(4) Handling of Coal and Partings

A hydraulic rotary drill is used to drill coal seams and partings according to the necessity. Bulldozers, 5.6 m^3 wheel loaders, 24 m^3 scrapers and 46 t trucks are applied to remove partings.

(5) Coal Loading and Hauling

A 6.0 m^3 hydraulic excavator fills 46 t trucks with coal. The number of trucks for coal hauling also was determined for each pit by applying the simulation model (Refer ANNEX 5).

4 trucks for West open pit and 3 trucks for East open pit are considered to be sufficient for planned ROM production for two shifts/day operation.

The main equipment used in West and East open pits is summarized as belows: (The number of equipment is represented for the period of full production capacity.)

Operation and main equipment		Life of equipment (years)	Number of units	
			West open pit	East open pit
Overburden removal				
Electric drill	(9-7/8 in.)	8	2	2
Electric shovel	(11.5 m^3)	Life of mine	2	2
Bulldozer		6	5	5
Dump truck	(120 t)	8	9	9
Partings removal				
Hydraulic drill	(80 mm)	8	1	1
Wheel loader	(5.6 m^3)	5	1	1
Motor scraper	(24 m^3)	5	2	1
Bulldozer		6	3	2
Dump truck	(46 t)	9	2	1
Coal loading and hauling				
Hydraulic excavator	(6.0 m^3)	6	1	1
Bulldozer		6	1	1
Dump truck	(46 t)	9	4	3
Restoration				
Wheel loader	(5.6 m^3)	5	1	1
Motor scraper	(24 m^3)	5	2	2
Bulldozer		6	2	2
Dump truck	(46 t)	9	1	1
Pit service, road maintenance				
Grader		6	2	2
Bulldozer		6	2	2
ANFO truck		6	1	1
Water truck		6	1	1
Crusher		10	1	1
Table top store truck		6	1	1
Fuel truck		6	1	1
35 ton mobile site crane		10	1	1
6 ton mobile site crane		10	1	1
Site personnel vehicle		3	5	5
Drainage pump		10	2	2
Mobile welding unit		6	1	1
Lighting set	(4 kW/unit)	6	2	2
Cargo truck	(10 t)	6	2	2

5-5 Electrical Equipment

5-5-1 Power Distribution System

The power supplied by WAPDA at 33 kV will be received and measured at the mine substation and distributed to the east and the West open pit mine at same voltage by means of each overhead line respectively. The power will then stepped down from 33 kV to 3.3 kV by each mobile substation provided at adequate places in both mines to avoid the danger of flying stones caused by storms. Subsequently, the power will be supplied to the vicinity of the operating site by the 3.3 kV overhead line installed along the site. The mobile switching stations are installed along the overhead line and connected with it, then the power to each machine will be fed from the nearest switching station via trailing cable respectively.

Total installed motor capacity in each open pit mine is estimated at 1,760 kW including distribution loss and miscellaneous and shown in following table.

Description	Specification	Required No.	Installed Capacity (kW)
Shovel	3.3 kV 600 kW	2	1,200
Electric Drill	3.3 kV 200 kW	2	400
Loss & Misc.			160
Total			1,760

The mobile substation used to supply the power to aforementioned machine consists of a 2,500 kVA 33 kV/3.3 kV 3 phase oil immersed transformer, each one of N/T and L/T side switchgear and lightening arresters. The substation is outdoor type and mounted on the sled for the convenience of the shifting hauled by the bulldozer when required.

Mobile switching stations will be provided at every 300 m to 500 m of the 3.3 kV overhead line and the power for each machine will be fed from the nearest station through the trailing cable accordingly with the shifting of its operating position. The switching station consists of 3.3 kV disconnecting switch and protecting steel box and mounted on the sled. The front door of the protecting box will be interlocked mechanically to be able to open only when the switch is in off position. The switching station is also able to shift its position hauled by the bulldozer when required.

In case of the power failure in heavy rain, each one of 100 kVA mobile generator for east and west mine, previously used for the power supply in the initial development stage, will be used to supply the power for drainage pumps.

Typical one line power distribution diagram in open pit mine is shown in Fig. 5-10.

5-5-2 Communication System

The communication system for the open pit mine is the combination of telephone and wireless.

One telephone each is installed in the east and west mine office, for inter-colliery use only and unusable for outside calls.

The wireless system consists of fixed station installed in the mine office room in the administration office and mobile stations on the site personnel vehicles used by officers, and is used to make intimate directions and information on safety and operation control aspects. In addition, carefully selected different frequency will be used for east and east mine respectively to avoid interference.

5-5-3 Lighting System

Considering the shift work in the open pit ample lighting is necessary to make suitable operating conditions and to avoid accidents. Therefore, in addition to the headlights and/or floodlights equipped with each machine, mobile lighting towers will be provided, namely local lighting by headlight and/or floodlights and general lighting by lighting tower.

The movable lighting tower consists of the telescopic mast with adequate number of mercury arc lamps and a diesel generator installed on the truck, and is shifted by bulldozer or truck when required.

5-5-4 Maintenance

General inspection and maintenance of the electrical equipment will be carried out by shift working staff, one foreman and one each of electric an and apprentice. However, the periodic inspection and maintenance as well as large scale work such as extribution line, etc. will be the responsibility of the electrical and mechanical section because of no arrangements for non-shift electrical staff in the open pit mine.

The annual maintenance cost includes one year use of spare parts for mobile substation, mobile switching station, as well as 10 % of C & F investment value for the trailing cable and its accessaries such as cable couplers and cable reel, etc. in foreign currency, and also each 1 % of C & F value for all imported equipment excluding the spare parts in local currency.

TABLE 5-1 ANNUAL STRIPPING AND PRODUCTION SCHEDULE
WEST OPEN PIT

Year	O/Burden Drilling (000's m)	O/Burden Prepared (000's m³)	Partings Prepared (000's m³)	TOTAL Prepared (000's m³)	O/Burden Removed (000's m³)	Partings Removed (000's m³)	TOTAL Removed (000's m³)	ROM Mined (000's t)	Clean Coal As-Received (000's t)	Mining Ratio m³/t
1983	67.3	3296.5	3.5	3300.0	2500.0	0.0	2500.0	0.0	0.0	0.0
1984	86.7	4247.5	32.5	4280.0	4261.2	18.8	4280.0	62.3	60.2	71.1
1985	102.1	5002.1	155.9	5158.0	5047.1	110.9	5158.0	183.5	177.2	29.1
1986	96.1	4707.6	450.5	5158.1	4756.5	401.5	5158.0	367.9	355.3	14.5
1987	90.6	4438.2	719.8	5158.0	4441.0	716.9	5157.9	373.6	361.0	14.3
1988	90.1	4415.3	742.6	5157.9	4422.5	735.5	5158.0	621.3	600.3	8.6
1989	89.2	4371.3	786.7	5158.0	4376.4	781.7	5158.1	622.7	601.7	8.6
1990	88.7	4344.5	813.5	5158.0	4343.8	814.2	5158.0	622.7	601.7	8.6
1991	89.0	4360.2	797.7	5157.9	4348.5	809.6	5158.1	622.7	601.7	8.6
1992	92.0	4510.1	759.9	5270.0	4520.9	749.1	5270.0	621.3	600.3	8.8
1993	90.6	4437.6	832.3	5269.9	4449.6	820.3	5269.9	600.2	579.2	9.1
1994	89.2	4371.8	898.2	5270.0	4370.6	899.4	5270.0	600.2	579.2	9.1
1995	89.6	4389.6	880.4	5270.0	4378.8	891.1	5269.9	599.0	578.0	9.1
1996	91.2	4468.8	816.3	5285.1	4462.8	822.2	5285.0	600.2	579.2	9.1
1997	91.7	4491.4	793.6	5285.0	4502.0	782.9	5284.9	600.2	579.2	9.1
1998	90.0	4407.8	877.3	5285.1	4431.3	853.7	5285.0	605.3	584.3	9.0
1999	87.7	4299.4	985.6	5285.0	4275.9	1009.2	5285.1	606.6	585.6	9.0
2000	90.4	4427.6	857.4	5285.0	4431.4	853.7	5285.1	602.8	581.8	9.1
2001	89.9	4402.9	882.1	5285.0	4406.7	878.3	5285.0	607.9	586.9	9.0
2002	89.3	4378.1	906.8	5284.9	4381.8	903.3	5285.1	607.9	586.9	9.0
2003	88.9	4357.3	927.7	5285.0	4358.4	926.7	5285.1	606.6	585.6	9.0
2004	88.5	4338.9	931.2	5270.1	4339.5	930.6	5270.1	607.9	586.9	9.0
2005	87.9	4307.1	927.9	5235.0	4306.8	928.2	5235.0	607.9	586.9	8.9
2006	87.3	4280.1	920.0	5200.1	4280.1	920.0	5200.1	607.9	586.9	8.9
2007	87.3	4280.1	920.0	5200.1	4280.1	920.0	5200.1	607.9	586.9	8.9
2008	87.3	4280.1	920.0	5200.1	4280.1	920.0	5200.1	607.9	586.9	8.9
2009	87.3	4280.1	920.0	5200.1	4280.1	920.0	5200.1	607.9	586.9	8.9
2010	87.3	4280.1	920.0	5200.1	4280.1	920.0	5200.1	607.9	586.9	8.9
2011	87.3	4280.1	920.0	5200.1	4280.1	920.0	5200.1	607.9	586.9	8.9
2012	87.3	4280.1	920.0	5200.1	4280.1	920.0	5200.1	607.9	586.9	8.9
2013	87.1	4266.3	933.8	5200.1	4280.1	920.0	5200.1	607.9	586.9	8.9
2014	76.8	3765.2	989.8	4755.0	3831.3	923.8	4755.1	607.9	586.9	8.1
2015	25.0	1225.0	469.2	1694.2	1803.5	690.7	2494.2	607.9	586.9	4.2
TOTAL	2856.9	139988.1	25612.2	165600.1	139988.5	25612.3	165600.6	18029.7	17408.1	9.5

TABLE 5-2 ANNUAL STRIPPING AND PRODUCTION SCHEDULE
EAST OPEN PIT

Year	O/Burden Drilling (000's m)	O/Burden Prepared (000's m³)	Partings Prepared (000's m³)	TOTAL Prepared (000's m³)	O/Burden Removed (000's m³)	Partings Removed (000's m³)	TOTAL Removed (000's m³)	ROM Mined (000's t)	Clean Coal As-Received (000's t)	Mining Ratio m³/t
1983	66.9	3278.0	21.3	3299.9	2497.1	2.9	2500.0	0.0	0.0	0.0
1984	68.8	3373.6	126.5	3500.1	3419.3	80.7	3500.0	64.8	62.7	55.8
1985	71.2	3489.8	310.2	3800.0	3495.1	304.8	3799.9	127.7	123.5	30.8
1986	70.7	3463.7	336.3	3800.0	3469.7	330.3	3800.0	254.9	246.5	15.4
1987	103.3	5059.7	540.4	5600.1	5071.4	528.6	5600.0	383.2	370.6	15.1
1988	101.4	4971.0	629.1	5600.1	4989.1	610.9	5600.0	383.2	370.6	15.1
1989	99.0	4849.6	750.3	5599.9	4862.6	737.4	5600.0	383.2	370.6	15.1
1990	97.3	4765.9	834.1	5600.0	4772.0	828.0	5600.0	383.2	370.6	15.1
1991	96.5	4727.8	872.2	5600.0	4729.8	870.2	5600.0	402.7	390.1	14.4
1992	96.3	4717.7	882.3	5600.0	4715.8	884.2	5600.0	402.7	390.1	14.4
1993	96.6	4733.3	866.7	5600.0	4729.1	870.9	5600.0	401.7	389.1	14.4
1994	97.2	4760.8	839.1	5599.9	4758.3	841.7	5600.0	401.7	389.1	14.4
1995	97.6	4780.8	819.2	5600.0	4775.9	824.1	5600.0	401.7	389.1	14.4
1996	98.3	4815.3	784.8	5600.1	4810.5	789.5	5600.0	401.7	389.1	14.4
1997	87.5	4287.4	662.7	4950.1	4281.5	668.6	4950.1	401.7	389.1	12.7
1998	80.2	3930.3	569.6	4499.9	3925.5	574.5	4500.0	401.7	389.1	11.6
1999	80.7	3954.3	545.7	4500.0	3952.5	547.5	4500.0	400.6	388.0	11.6
2000	75.4	3693.6	506.3	4199.9	3698.7	501.4	4200.1	395.6	383.0	11.0
2001	74.9	3671.5	528.6	4200.1	3672.3	527.8	4200.1	395.6	383.0	11.0
2002	75.1	3679.9	520.1	4200.0	3667.9	532.2	4200.1	395.6	383.0	11.0
2003	76.2	3735.2	464.8	4200.0	3730.9	469.1	4200.0	395.6	383.0	11.0
2004	100.5	4924.8	575.2	5500.0	4915.3	584.7	5500.0	403.8	391.2	14.1
2005	102.5	5024.3	520.7	5545.0	5021.2	523.8	5545.0	407.0	394.4	14.1
2006	102.8	5036.3	508.8	5545.1	5042.8	502.3	5545.1	410.3	397.7	13.9
2007	102.1	5000.9	544.2	5545.1	4997.8	547.2	5545.0	410.3	397.7	13.9
2008	102.4	5016.5	528.6	5545.1	5019.0	526.0	5545.0	413.7	401.1	13.8
2009	102.1	5001.6	543.3	5544.9	5001.4	543.7	5545.1	414.8	402.2	13.8
2010	102.2	5008.7	536.3	5545.0	5003.4	541.7	5545.1	399.6	387.0	14.3
2011	103.0	5049.2	495.8	5545.0	5040.6	504.4	5545.0	400.6	388.0	14.3
2012	104.3	5108.7	436.3	5545.0	5100.5	444.6	5545.1	386.0	373.4	14.9
2013	105.4	5165.2	379.8	5545.0	5157.8	387.2	5545.0	386.9	374.3	14.8
2014	103.4	5065.8	334.2	5400.0	5073.1	326.9	5400.0	386.9	374.3	14.4
2015	48.5	2378.6	178.1	2556.7	3122.9	233.9	3356.8	386.9	374.3	9.0
TOTAL	2990.2	146519.8	17991.5	164511.0	146520.3	17991.7	164511.6	11985.6	11605.5	14.2

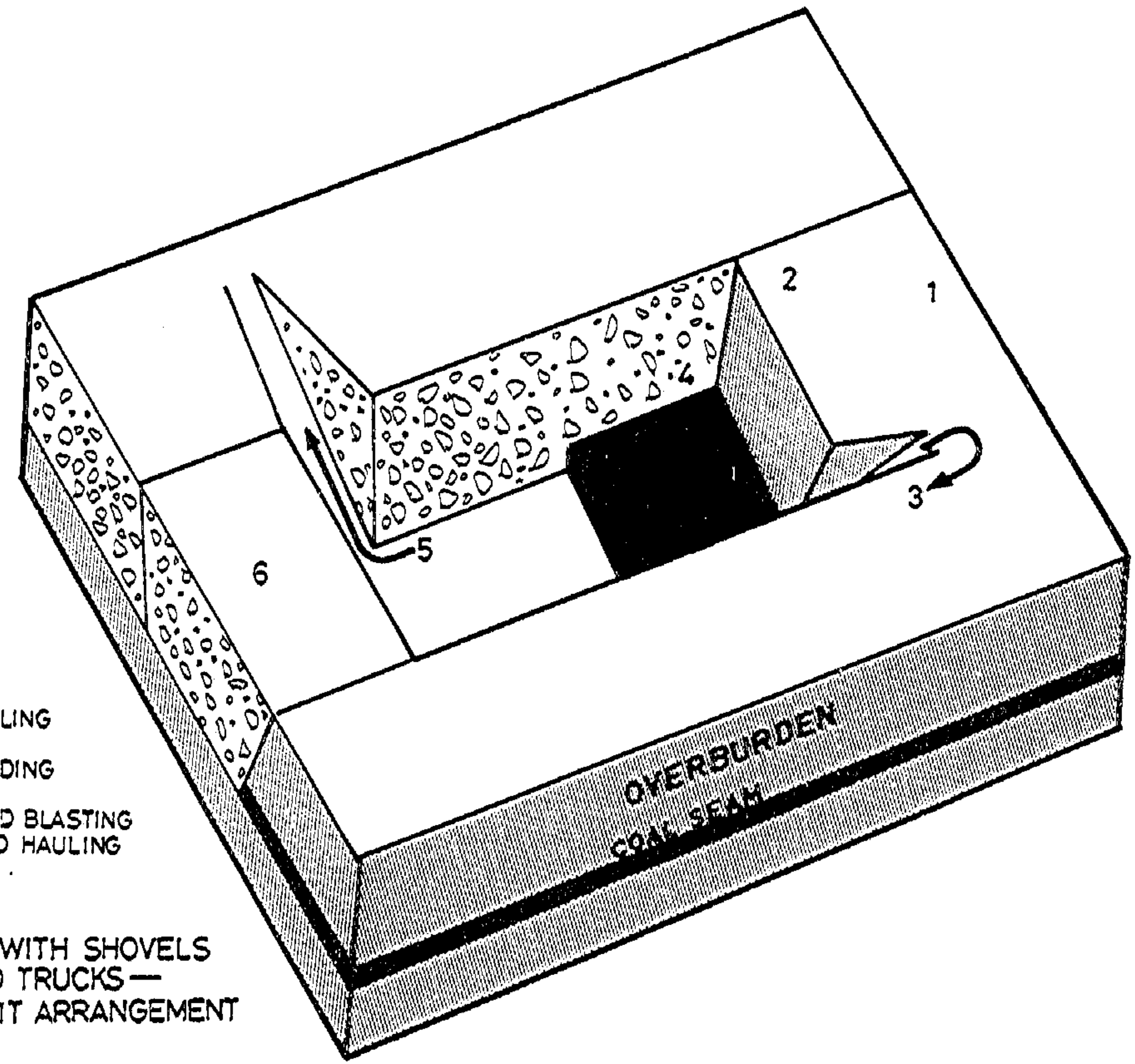
TABLE 5-3 ANNUAL STRIPPING AND PRODUCTION SCHEDULE
TOTAL — EAST O.P. AND WEST O.P.

Year	O/Burden Drilling (000's m)	O/Burden Prepared (000's m³)	Partings Prepared (000's m³)	TOTAL Prepared (000's m³)	O/Burden Removed (000's m³)	Partings Removed (000's m³)	TOTAL Removed (000's m³)	ROM Mined (000's t)	Clean Coal As-Received (000's t)	Mining Ratio m³/t
1983	134.2	6575.1	24.8	6599.9	4997.1	2.9	5000.0	0.0	0.0	0.0
1984	155.5	7621.1	159.0	7780.1	7680.5	99.5	7780.0	127.1	122.9	63.3
1985	173.3	8491.9	466.1	8958.0	8542.2	415.7	8957.9	311.2	300.7	29.8
1986	166.8	8171.3	786.8	8958.1	8226.2	731.8	8958.0	622.8	601.8	14.9
1987	193.8	9497.9	1260.2	10758.1	9512.4	1245.5	10757.9	756.8	731.6	14.7
1988	191.6	9386.3	1371.7	10758.0	9411.6	1346.4	10758.0	1004.5	970.9	11.1
1989	188.2	9220.9	1537.0	10757.9	9239.0	1519.1	10758.1	1005.9	972.3	11.1
1990	185.9	9110.4	1647.6	10758.0	9115.8	1642.2	10758.0	1005.9	972.3	11.1
1991	185.5	9088.0	1669.9	10757.9	9078.3	1679.8	10758.1	1025.4	991.8	10.8
1992	188.3	9227.8	1642.2	10870.0	9236.7	1633.3	10870.0	1024.0	990.4	11.0
1993	187.2	9170.9	1699.0	10869.9	9178.7	1691.2	10869.9	1001.9	968.3	11.2
1994	186.4	9132.6	1737.3	10869.9	9128.9	1741.1	10870.0	1001.9	968.3	11.2
1995	187.2	9170.4	1699.6	10870.0	9154.7	1715.2	10869.9	1000.7	967.1	11.2
1996	189.5	9284.1	1601.1	10885.2	9273.3	1611.7	10885.0	1001.9	968.3	11.2
1997	179.2	8778.8	1456.3	10235.1	8783.5	1451.5	10235.0	1001.9	968.3	10.6
1998	170.2	8338.1	1446.9	9785.0	8356.8	1428.2	9785.0	1007.0	973.4	10.1
1999	168.4	8253.7	1531.3	9785.0	8228.4	1556.7	9785.1	1007.2	973.6	10.1
2000	165.7	8121.2	1363.7	9484.9	8130.1	1355.1	9485.2	998.4	964.8	9.8
2001	164.8	8074.4	1410.7	9485.1	8079.0	1406.1	9485.1	1003.5	969.9	9.8
2002	164.4	8058.0	1426.9	9484.9	8049.7	1435.5	9485.2	1003.5	969.9	9.8
2003	165.2	8092.5	1392.5	9485.0	8089.3	1395.8	9485.1	1002.2	968.6	9.8
2004	189.1	9263.7	1506.4	10770.1	9254.8	1515.3	10770.1	1011.7	978.1	11.0
2005	190.4	9331.4	1448.6	10780.0	9328.0	1452.0	10780.0	1014.9	981.3	11.0
2006	190.1	9316.4	1428.8	10745.2	9322.9	1422.3	10745.2	1018.2	984.6	10.9
2007	189.4	9281.0	1464.2	10745.2	9277.9	1467.2	10745.1	1018.2	984.6	10.9
2008	189.7	9296.6	1448.6	10745.2	9299.1	1446.0	10745.1	1021.6	988.0	10.9
2009	189.4	9281.7	1463.3	10745.0	9281.5	1463.7	10745.2	1022.7	989.1	10.9
2010	189.6	9288.8	1456.3	10745.1	9283.5	1461.7	10745.2	1007.5	973.9	11.0
2011	190.4	9329.3	1415.8	10745.1	9320.7	1424.4	10745.1	1008.5	974.9	11.0
2012	191.6	9388.8	1356.3	10745.1	9380.6	1364.6	10745.2	993.9	960.3	11.2
2013	192.5	9431.5	1313.6	10745.1	9437.9	1307.2	10745.1	994.8	961.2	11.2
2014	180.2	8831.0	1324.0	10155.0	8904.4	1250.7	10155.1	994.8	961.2	10.6
2015	73.5	3603.6	647.3	4250.9	4926.4	924.6	5851.0	994.8	961.2	6.1
TOTAL	5847.1	286508.2	43603.7	330111.9	286509.0	43603.9	330112.9	30015.2	29013.5	11.4

TABLE 5-4 SUMMARY OF ANNUAL ROM COAL MINED FROM OPEN PIT

(000's TONNES)

YEAR	WEST OPENPIT					EAST OPENPIT					TOTAL (EAST & WEST)				
	COAL SEAM					COAL SEAM					COAL SEAM				
	No.5	No.3	No.2	No.1	Total	No.5	No.3	No.2	No.1	Total	No.5	No.3	No.2	No.1	Total
1984	60	2	-	-	62	-	61	4	-	65	60	63	4	-	127
1985	93	57	19	15	184	-	63	57	7	127	93	120	76	22	311
1986	47	83	114	124	368	-	110	123	22	255	47	193	237	146	623
1987	39	55	114	166	374	-	189	167	27	383	39	244	281	193	757
1988	77	121	143	280	621	-	190	168	26	384	77	311	311	306	1,005
1989	62	96	197	268	623	-	193	163	27	383	62	289	360	295	1,006
1990	66	78	198	281	623	-	192	148	43	383	66	270	346	324	1,006
1991	70	76	184	293	623	-	194	137	71	402	70	270	321	364	1,025
1992	86	107	154	274	621	-	190	122	91	403	86	297	276	365	1,024
1993	62	87	165	286	600	-	184	114	104	402	62	271	279	390	1,002
1994	63	54	207	276	600	-	182	105	115	402	63	236	312	391	1,002
1995	66	36	217	280	599	-	183	104	115	402	66	219	321	395	1,001
1996	79	49	196	276	600	-	181	106	115	402	79	230	302	391	1,002
1997	79	56	177	288	600	-	173	120	109	402	79	229	297	397	1,002
1998	68	53	204	280	605	-	166	135	101	402	68	219	339	381	1,007
1999	56	5	290	256	607	-	157	141	102	400	56	162	431	358	1,007
2000	90	43	193	277	603	-	147	151	97	395	90	190	344	374	998
2001	92	38	199	278	607	-	135	165	96	396	92	173	364	374	1,003
2002	93	33	202	280	608	-	115	179	101	395	93	148	381	381	1,003
2003	93	27	207	279	606	-	72	233	91	396	93	99	440	370	1,002
2004	93	24	210	281	608	-	84	225	95	404	93	108	435	376	1,012
2005	94	22	213	279	608	-	112	196	99	407	94	134	409	378	1,015
2006	93	21	214	280	608	-	134	169	107	410	93	155	383	387	1,018
2007	93	21	214	280	608	-	140	157	113	410	93	161	371	393	1,018
2008	93	21	214	280	608	-	165	133	116	414	93	186	347	396	1,022
2009	93	21	214	280	608	-	178	121	116	415	93	199	335	396	1,023
2010	93	21	214	280	608	-	185	102	112	399	93	206	316	392	1,007
2011	93	21	214	280	608	-	205	82	113	400	93	226	296	393	1,008
2012	93	21	214	280	608	-	224	52	110	386	93	245	266	390	994
2013	93	21	214	280	608	-	239	36	112	387	93	260	250	392	995
2014	93	21	214	280	608	-	252	23	112	387	93	273	237	392	995
2015	93	21	214	280	608	-	254	8	125	387	93	275	222	405	995
TOTAL	2,558	1,412	5,943	8,117	18,030	-	5,249	3,946	2,790	11,985	2,558	6,661	9,889	10,907	30,015



STEPS:
1. TOPSOIL REMOVAL
2. OVERBURDEN DRILLING AND BLASTING
3. OVERBURDEN LOADING AND HAULING
4. COAL DRILLING AND BLASTING
5. COAL LOADING AND HAULING
6. RECLAMATION

FIGURE 5-1
AREA STRIPPING WITH SHOVELS OR LOADERS AND TRUCKS— HYPOTHETICAL PIT ARRANGEMENT

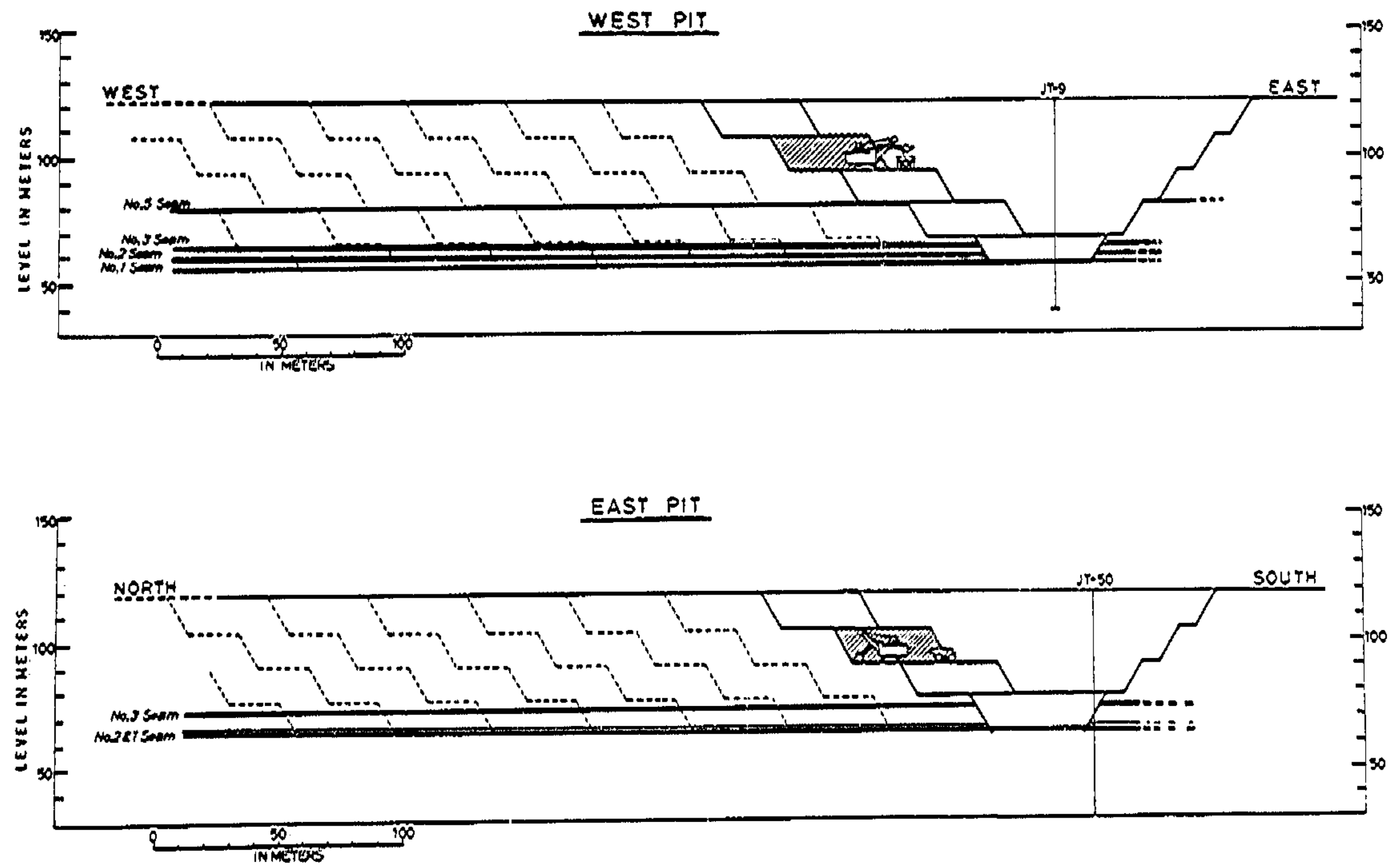


FIGURE 5-2 SCHEMATIC PROFILE SECTION —OVERBURDEN STRIPPING WITH SHOVEL/TRUCK

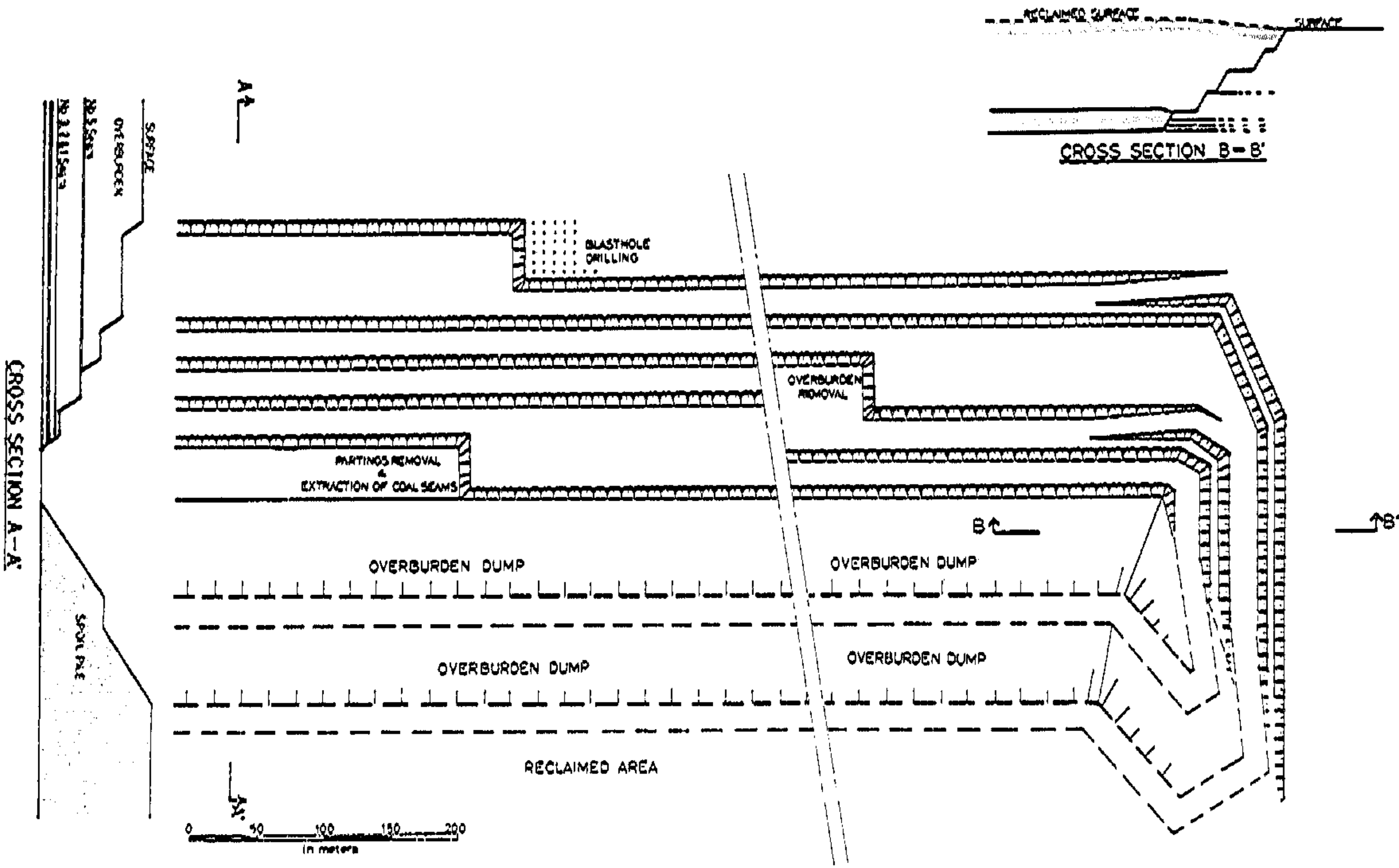


FIGURE 5-3 SCHEMATIC VIEW OF PIT END

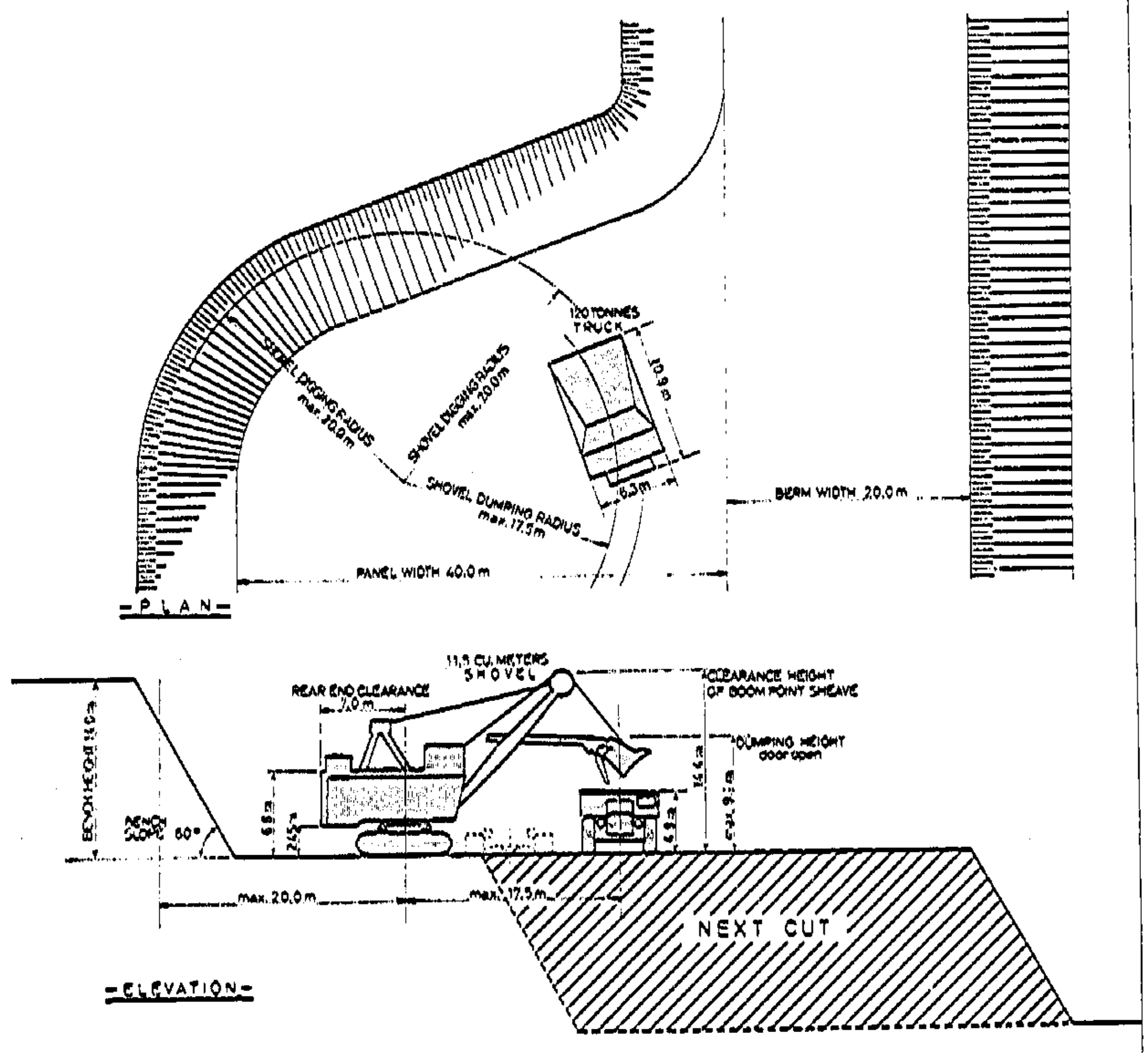


FIGURE 5-4 RELATIONSHIP OF EQUIPMENT SIZE, BENCH SPACING AND OPERATING ROOM IN OVERBURDEN STRIPPING

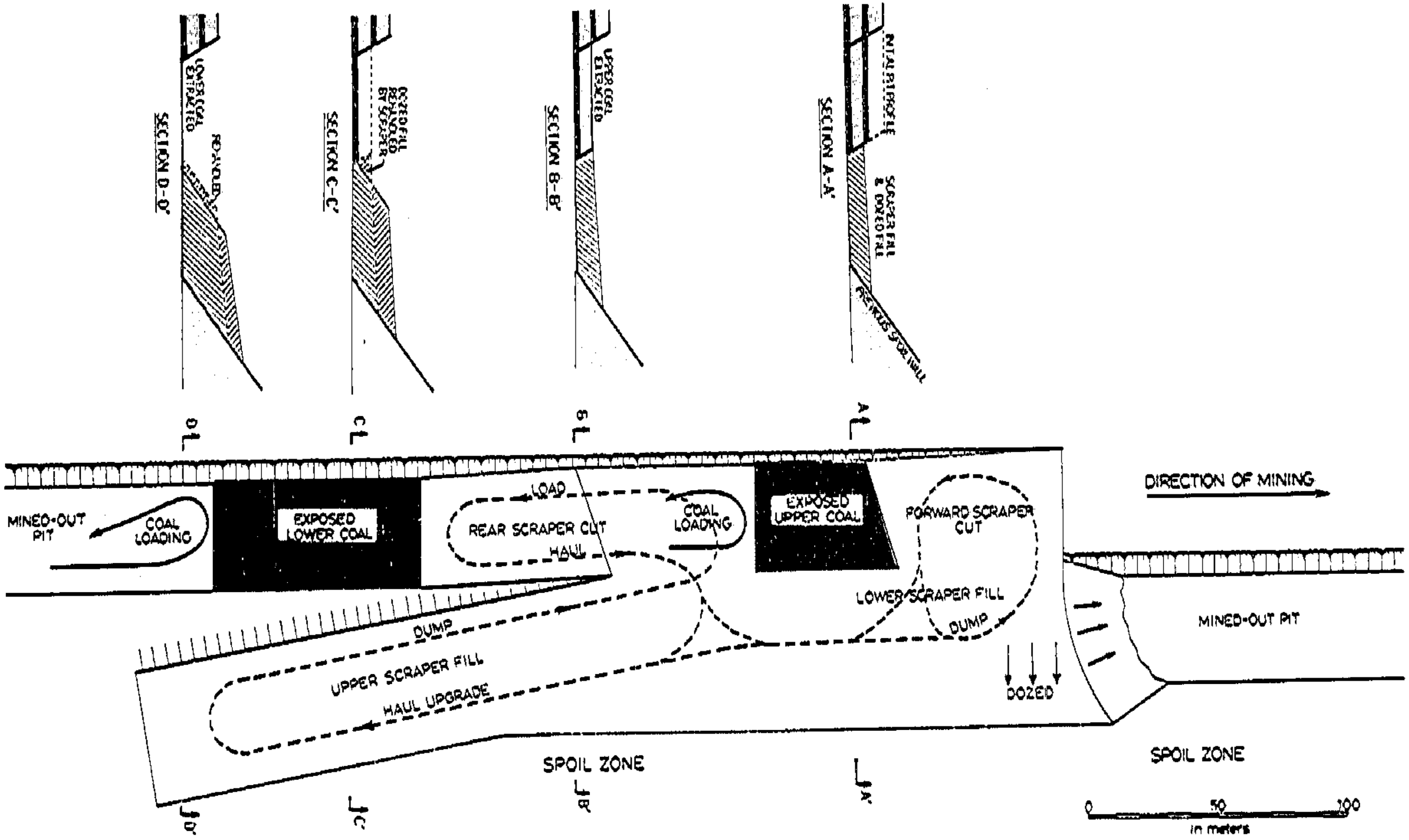


FIGURE 5-5 MULTIPLE SEAM MINING AT LOWEST BENCH WITH DOZER/SCRAPER METHOD

MAIN SPECIFICATIONS

MAST	9.91 m — 16.76 m
WORKING WEIGHT	68 040 kg
HYD. PULLDOWN	31 751 kg
HYDRAULIC HOIST	24 947 kg
ELECTRIC ROTARY MOTOR	37 kw @ 460V
COMPRESSOR	27.81 m³/min. @ 2.81 kg/cm²

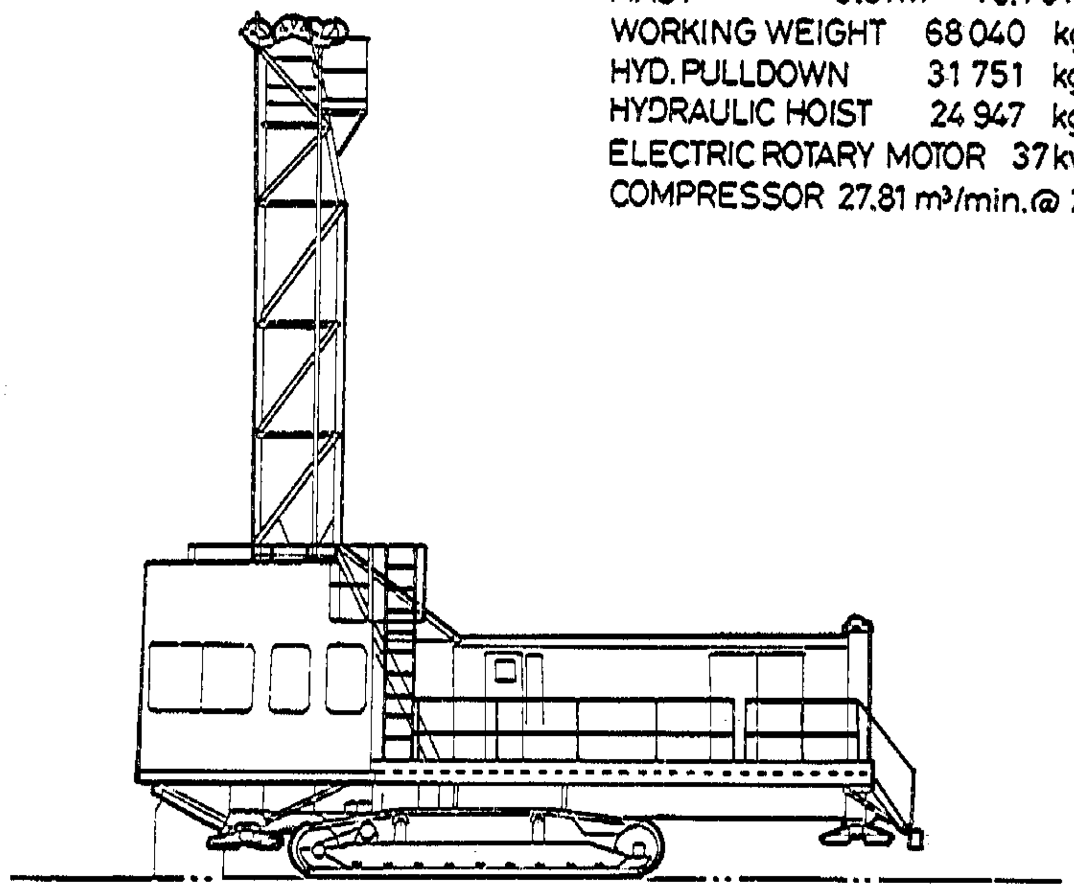


FIGURE 5-6 ROTARY BLAST HOLE DRILL 9⅞" Hole Size

SHOVEL POWER

Electric, A.C.	3 Phase, 60 Cycle
Rated Operating Voltage	2200/3810 Volts
For 50 Cycle	Consult Milwaukee
Main A.C. Motor	750 H.P. Continuous 1875 H.P. Intermittent
Speed — 60 Cycle	590 R.P.M.
Swing Generator — 250 Volts, 1550 R.P.M.	168 K.W.
Crown Generator — 250 Volts, 1550 R.P.M.	168 K.W.
Cab Blower Motor H.P. (2)	Each 15
Crowd Motor Blower H.P.	1.5
Swing Motor Blower H.P. (2)	Each 1.5
Propel Motor Blower H.P.	2
Voltage of A.C. Auxiliary Units	440
Hoist Magnetorque H.P.	700
Air Compressor Motor H.P.	10
Dipper Trip Motor H.P.	15

Propel Motor (Mill Type)

H.P. at 475 Volts D.C. (one-half hour)	325
Motor Ventilation	Blown

Swing Motor (Mill Type — Vertical — Two Used)

H.P. at 475 Volts D.C. (continuous)	Total 355
Motor Ventilation	Blown

Crowd Motor (Mill Type)

H.P. at 475 Volts D.C. (continuous)	205
Motor Ventilation	Blown
Insulation — A.C. Motors	Class F
D.C. Generators and Motors	Class F
Bearings	Ball or Roller
Bearing Lubrication	Grease

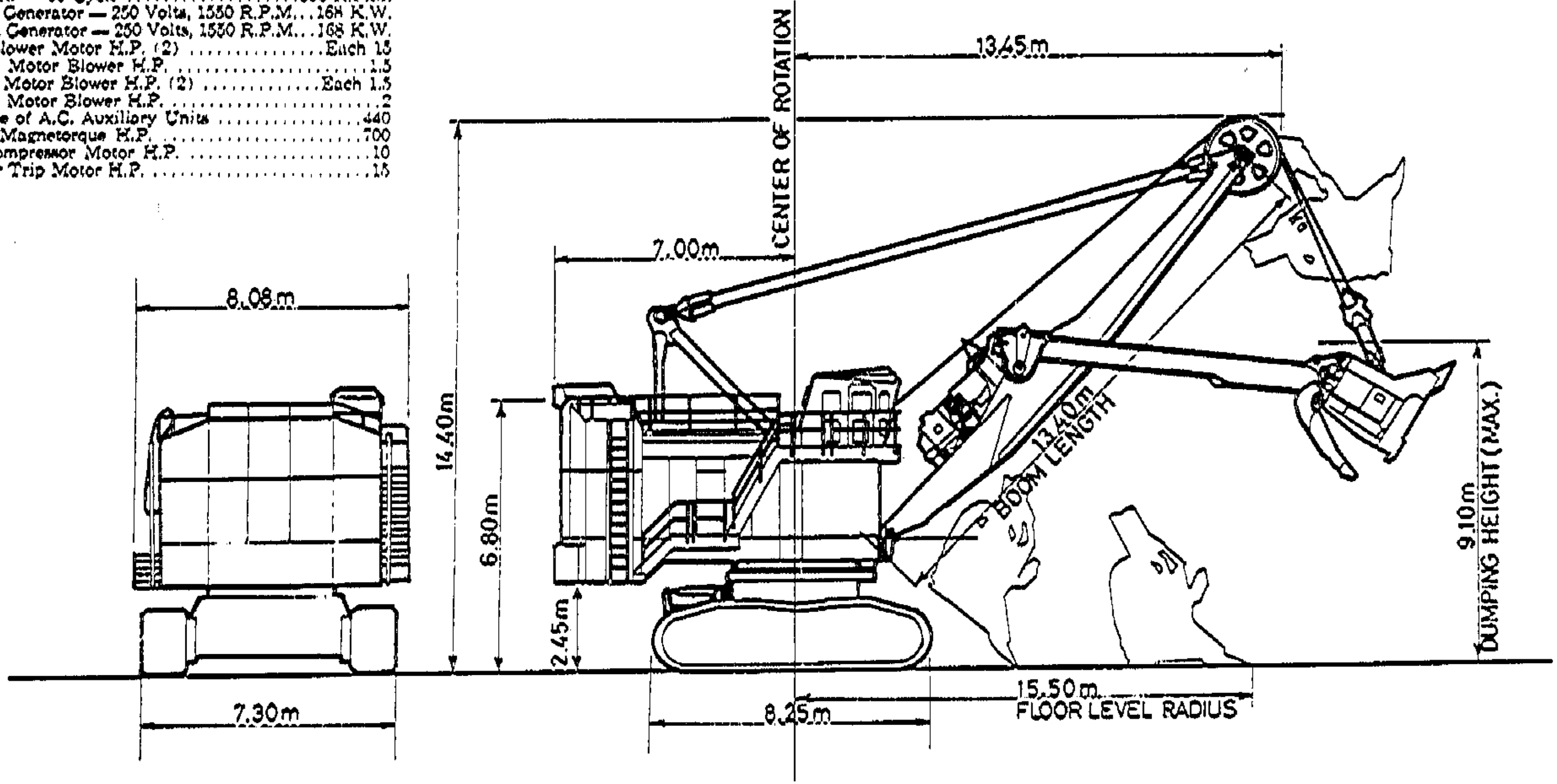


FIGURE 5-7 11.5 m³ ELECTRIC MINING SHOVEL FOR OVERBURDEN LOADING

MAIN SPECIFICATIONS

GROSS HORSEPOWER	1200 HP/2100 RPM
NET WEIGHT	84500 kg
CAPACITY : HEAPED	70 m^3
MAX. PAYLOAD	120000 kg
TRANSMITTING SYSTEM	ELECTRIC DRIVE
MAX. TRAVEL SPEED	57.5 km/h

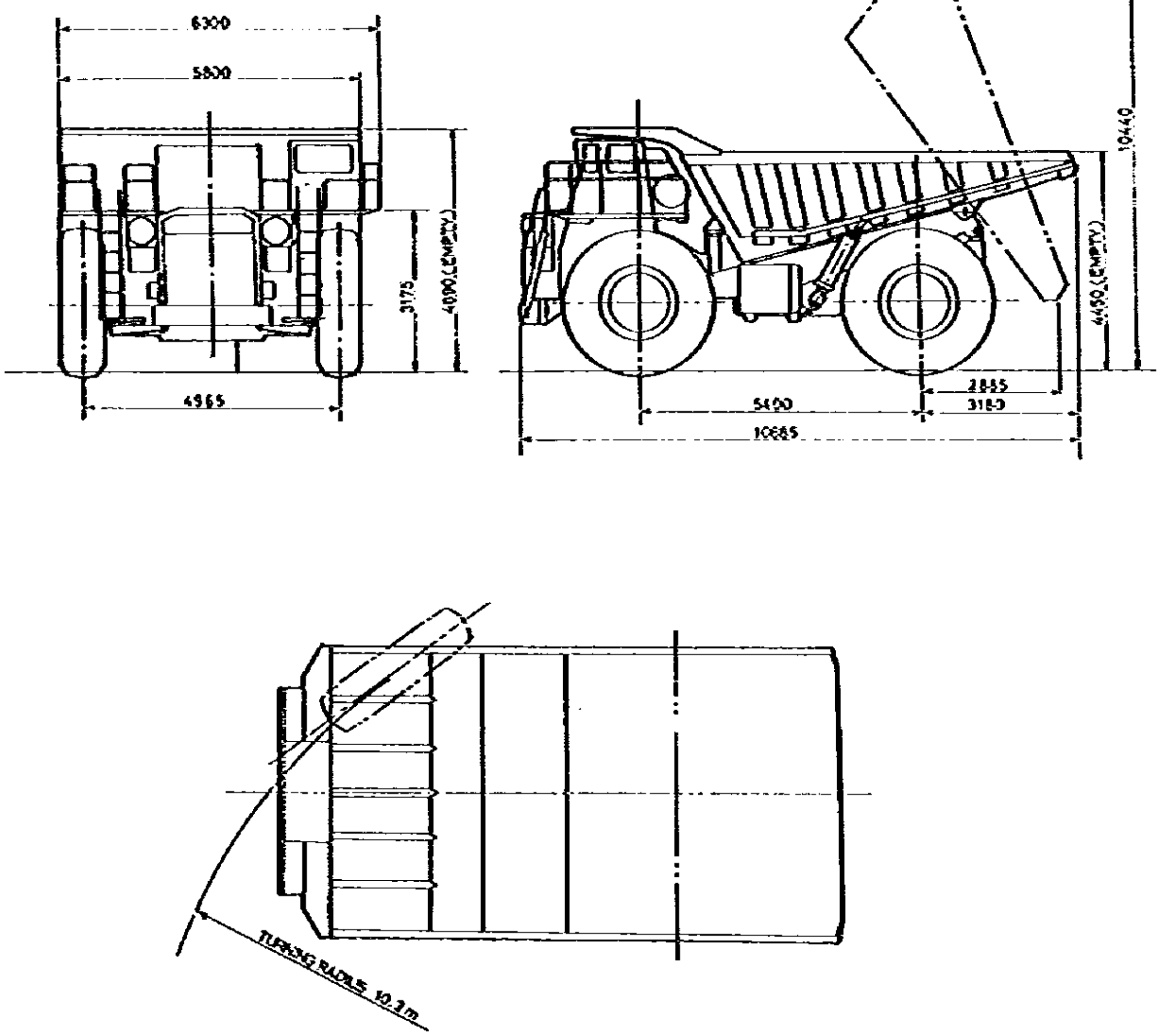
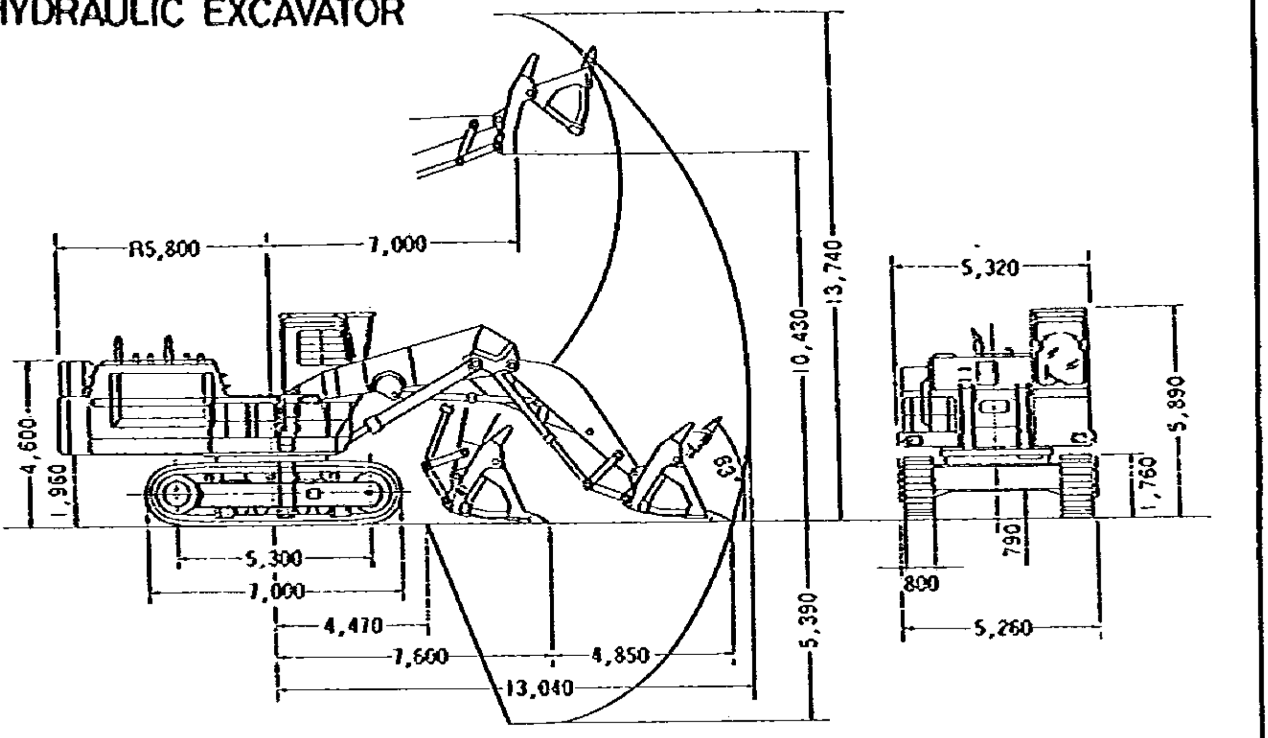


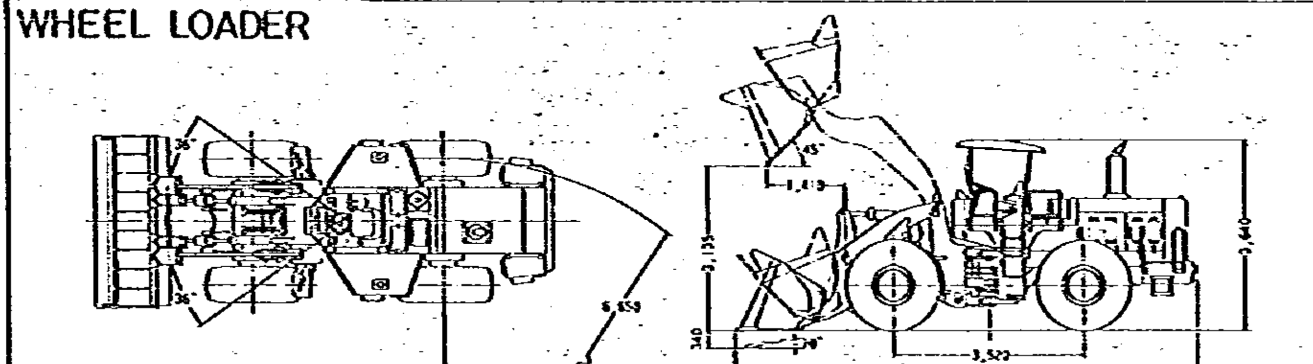
FIGURE 5-8 OFF-HIGHWAY 120t DUMP TRUCK FOR OVERBURDEN HAULING

HYDRAULIC EXCAVATOR



BUCKET CAPACITY . . . . 3.7 - 6.2 m^3
RATED HORSEPOWER OF DIESEL ENGINE . . . . . 2 x 147 kW (2 x 200 PS)
OPERATING WEIGHT. . . . . 73,000 kg
MAXIMUM REACH . . . . 10.70 m
MAXIMUM DEPTH . . . . 4.88 m
MAXIMUM DUMPING HEIGHT. . . . . 8.30 m

WHEEL LOADER



BUCKET CAPACITY . . . . 4.5 - 7.0 m^3
RATED OUTPUT OF DIESEL ENGINE . . . . 310 PS / 2,100 rpm
NO. OF TRAVEL SPEEDS (REVERSE). 4 (2)
MAXIMUM TRACTIVE EFFORT . . . . 21,000 kg
OPERATING WEIGHT. . . . . 25,900 kg
DUMPING CLEARANCE . . . . 3.14 m

FIGURE 5-9 HYDRAULIC EXCAVATOR AND WHEEL LOADER

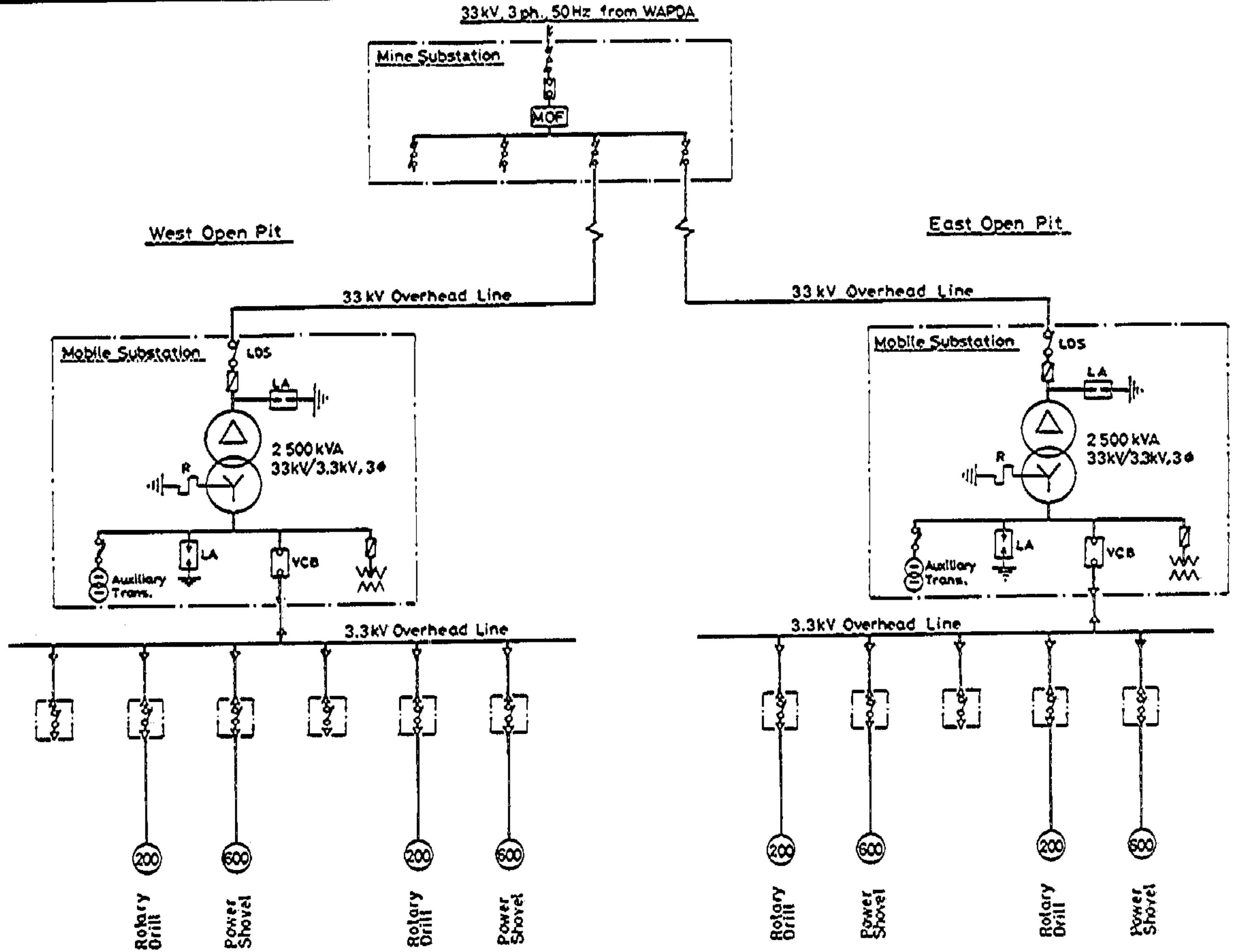


FIGURE 5-10 TYPICAL ONE LINE POWER DISTRIBUTION DIAGRAM IN OPEN PITS

CHAPTER 6 SURFACE FACILITIES

6-1 Outline of Surface Facilities

The surface facilities will be provided in the barren area between the west and the central blocks concentrically to avoid damage caused by the mining operations in open pit and underground mines.

Namely, the mine office, the air compressor room, the winding machine house, the safety lamp room, the mine substation, the power house, the electrical and mechanical workshops and warehouse, etc. will be provided in the central part of this area near the underground portal.

On the other hand, the administration and the heavy equipment workshop in the western part of the area, and the coal preparation plant and the emergency stockpile in the southern part of the area near the railway terminal will also be provided respectively. The detailed surface layout is shown in Fig. 6-1.

The transportation of coal from the colliery to the power station as well as of the colliery personnel between the mine site and Khanot will be carried out by the railway. On the other hand the transportation of equipment and materials will be carried out mainly by road.

The surface water of the River Indus will be used for the industrial and living water after being purified.

6-2 Civil Engineering and Construction

6-2-1 Civil Engineering Work

The main civil engineering work concerning the surface facilities contains land grading work of sites for various facilities, road work inside and outside the mine yard and so on.

(1) Grading Work

Grading work is necessary for the sites prepared for buildings, offices, workshops, pit head facilities for underground mine, preparation plant, roads and railway. An area of approximately 20,000 m^2 is required for buildings including the areal surplus of 20 percent and approximately 190,000 m^2 for surrounding area. The lands to be graded for the construction of roads inside and outside the mine yard and 610 mm gauge tracks inside the mine yard are (Length x Width = Area)

a)	Coal haulage roads inside the mine yard:	6 km x 20 m = 120,000 m^2
b)	General purpose roads inside the mine yard:	3 km x 10 m = 30,000 m^2
c)	General purpose roads outside the mine yard:	30 km x 10 m = 300,000 m^2
d)	610 mm gauge tracks:	4 km x 5 m = 20,000 m^2

The total area to be graded is approximately 680,000 m^2. However, this area does not include the land for railway construction nor the site of colony in Khanot district. The former will occupy the area of 325,000 m^2 (32.5 km x 10 m) and the latter will need flat land of approximately 700,000 m^2.

(2) Road Construction Work

Two kinds of roads are planned to be constructed, one is the coal haulage road inside the mine yard and another is the general purpose road inside and outside the mine yard. For road construction, the CBR for road design is estimated considering the nature of roadbed.

The soil of the site for road construction consists of mainly fine earth and partly contains clay soil with minute particles. The existing gravel roads also can be widened, improved and utilized for mining purposes. Based on the site conditions, the CBR for road as 10, 20, 30 and 40 percent respectively.

(3) Coal Houlage Roads Inside the Mine Yard

These roads are exclusively used for the travel of 46 tonne dump trucks hauling ROM coal from West open pit and East open pit to the 100 tonne dump hopper of the preparation plant. Therefore, the surface layout was so designed that the coal haulage roads do not intersect the other roads, taking traffic safety into consideration.

In addition to the CBR for road design, the traffic loads are to be assumed as hypothetical conditions for design work. The assumed values are as follows:

a) Quantity of coal hauling per year: 800,000 x 1.1 = 880,000 tonnes (max.)
b) Operating days per year: 300 days
c) Quantity of coal hauling per day: 2,700 tonnes
d) Weight of truck: 38 tonnes
e) Max. payload per truck: 46 tonnes
f) Number of passes (one way): 59

The roads of approximately 6 km will be initially constructed before pit opening. Then, 1 km will be added after 10 years and 3 km after 25 years. These additional road constructions will be budgeted in replacement and improvement costs for operating period.

The effective width of road is 10 m and the 2 m wide shoulder is prepared by both sides of road, then the total width of road is 14 m.

(4) General Purpose Roads Inside and Outside the Mine Yard

These roads are initially used for the transportation of equipment and materials during the mine construction period. The traffic of 20 tonne trucks and 30 tonne trailers is taken into consideration.

The length of road is 25 km between the mine site and Khanot, 3 km between Khanot and the water-intake at The River Indus, 2 km between the existing railway and the power station and 3 km in the mine yard, then the total length is 33 km.

The width of road is designed to be 7.5 m (the effective width of 3.5 m and the 2 m wide shoulder by both sides).

(5) Other Civil Engineering Work

Civil engineering work is also necessary for the construction of following facilities:

a) Foundation, fence and ballast work for substation (50 m^3 in volume, 310 m in length and 5,350 m^2 in area or 1,050 m^3 in volume respectively).

b) Fuel storage tank in maintenance shops for heavy equipment (capacity: 50 kℓ).

c) Loading dock for maintenance shops for heavy equipment (concrete: 125 m^3 in volume, pit: 50 m^3 in space).

d) Yard fence for prohibition of intruders (8 km in length).

e) Fuel tank for open pit machines (capacity: 2 x20 kℓ).

f) Banking of explosive stores (1,440 m^3 in volume).

g) Sewage disposal (cesspool capacity: 150 m^3).

The construction of 610 mm gauge tracks of approximately 4 km inside the mine yard, tippler pits for coal and waste and foundation of belt conveyors, etc. will be stated later.

6-2-2 Building Work

The building work involves the pit head facilities for underground mine, offices, mechanical and electrical workshops, explosives magazine and other accessory structures and buildings.

(1) Pit Head Facilities for Underground Mine

Substation, power house, compressor rooms, posting rooms, safety lamp rooms, main winding room, sub-winding room and fan room have to be planned taking the building area of approximately 2,700 m^2. Main structures are constructed of reinforced concrete and brick. All buildings are one-storied.

(2) Offices

An administration office will be established between the West open pit area and the portal of underground mine. A foremen office will be located close to each open pit and a mine office will be situated near the portal of underground mine. A laboratory will be built in the vicinity of preparation plant. The total building area is estimated to be approximately 4,140 m^2.

Main structures are constructed of reinforced concrete and brick. Only the administration office is two-storied building and the others are one-storied.

(3) Work Shops

Maintenance shops of heavy equipment for open pit mining (see Fig. 6-2), mechanical and electrical work shops for underground mining equipment and warehouses (see Fig. 6-3) and workshops for railway facilities (see Fig. 6-4) are to be planned. Garages will be built adjoining the administration office. The building area for above-mentionned structures is estimated to be approximately 12,200 m^2. Main structures are one-storied construction of steel and reinforced concrete.

(4) Accessory Building

An explosives magazine will be situated about 300 m north of the coal haulage road between West open pit and underground mine. An explosives control for underground mine will be located near the portal. They will be constructed of reinforced concrete and brick occupying the total building area of 130 m^2.

The maintenance cost for buildings is generally estimated to be 1 percent of total construction cost.

6-3 Water Supply and Sewage

6-3-1 Water Supply

(1) General Aspects

Two kinds of water, industrial water and living water must be supplied for mine activities. The industrial water is used for washing the heavy equipment, sprinkling, charging the hydraulic props and so on. The living water is consumed in shower rooms, offices at mine site and the colony in Khanot district as drinking water.

The surface water of the River Indus is planned to be taken at the site located about 3 km south-east of Khanot as a water source for mine complex. This water intake consists of pumps mounted on the steel structure pontoon, discharge pipes and water conveyance pipes with flexible joints, considering countermeasures against flooding of the River Indus.

The water taken at the water source is pushed up by the pump and sent to the transit point at Khanot where a sand basin is constructed. Then, the water from the sand basin is separated into the colony and the mine site. The water for the colony is treated at the water purification facilities and sent to the service reservoir of colony at Khanot. The water for the mine site is pushed up from the sand basin at Khanot to the water purification facilities at Lakhra mine site and then to the service reservoir. The water is distributed by booster pumps from the service reservoir to where necessary both at the colony and the mine site.

For the water supply, cast iron pipes are used for water intake, conveyance and distribution, and galvanized steel pipes for service pipelines. All pipes are buried in the shoulder of main roads.

(2) Estimated Water Consumption for Design

The number of inhabitants supplied are estimated for the mine site and the colony at Khanot to be 1,824 people and 9,000 people respectively.

The maximum water consumption per day is estimated as follows:

	Industrial water	Living water	Total (m^3)
Lakhra mine site	770	210	980
Colony at Khanot	–	2,850	2,850
Total	770	3,060	3,830

(3) Supply System

Schematic water supply system is shown below:

Water source → Sand basin at Khanot → Purification plant at Khanot

Sand basin at Khanot ↓ Purification plant at mine site ↓ Service reservoir at mine site ↓ Water distribution

Purification plant at Khanot ↓ Service reservoir at Khanot ↓ Water distribution

(4) Altitude and Distance

Colony at Khanot (+ 40 m)
| 1.54 km
Water source (+ 9 m) —— 3 km —— Transit point at Khanot (+ 30 m) —— 10.5 m —— Intermediate point (+ 168 m)
| 14 km
Lakhra mine site (+ 123 m)

(5) Design of Facilities

a) Water intake facilities

Water source: Surface water of the River Indus

Water intake equipment: Pumping house (wooden structure 3 m x 6 m) built on the steel structure pontoon moored at the intake site on the River Indus

Estimated quantity of water taken:

$3,830 \ m^3/day \times 1.1 = 4,213 \ m^3/day \doteq 176 \ m^3/h \doteq 2.93 \ m^3/min \doteq 0.0488 \ m^3/sec$

Sand basin: Dimension; 4 m x 30 m x 2 m (height)

Effective capacity; 240 m^3

Number of sand basins; 2

b) Water conveyance facilities

Estimated quantity of water conveyance: 4,213 m^3/day

Water conveyance system: booster pumps and pipeline

Diameter of pipe: 200 mm

Total pipe length: approximately 3 km

c) Water purification facilities

Estimated qunatity of purified water: 3,830 m^3/day

Pruification system: Rapid filtration

Well for flow of water control:

Effective capacity; 3,830 m^3/day x 1.5 min./(24 x 60) ≑ 4 m^3 over

Dimension; 2 m x 6 m x 3 m (height) = 36 m^3

Chemical treatment facilities:

Mixing basin; Effective capacity

3,830 m^3/day x 1~5 min/(24 x 60) = 2.7~13.3 m^3

Dimension of mixing basin; 2 m x 2 m x 3 m (height) = 12 m^3

Floc basin; Effective capacity

3,830 m^3/day x 20~40 min/(24 x 60) = 84~106 m^3

Dimension of floc basin; 4 m x 12 m x 3 m (height) = 144 m^3

Coagulation basin; Effective capacity

3,830 m^3/day x 3~5 h/24 = 478~798 m^3

Dimension of coagulation basin; 10 m x 30 m x 3 m (height) = 900 m^3

Number of coagulation basins; 2

Rapid filter:

Rate of filtration; 120~150 m/day

Filtration area; $\frac{3{,}830 \ m^3/day}{120\sim150 \ m/day} ≑ 32\sim26 \ m^2$

Dimension; 4 m x 8 m x 3 m (height) = 96 m^3

Number of filters; 2

Clear water reservoir:

Effective capacity; 3,830 m^3 x 1 h/24 h = 160 m^3 over

Dimension; 10 m x 10 m x 3.5 m (height) = 350 m^3

Disinfection equipment:

Chlorinator room; 2 m x 3 m = 6 m^2 (timber construction)

d) Water distribution facilities

For planning of water distribution facilities, the water quantity of delivery is estimated to be normally as maximum hourly water delivery and in case of fire the sum of maximum hourly water delivery and water for fire fighting.

Lakhra mine site:

Estimated maximum water consumption; 980 m^3 /day, 41 m^3/h

Estimated maximum water delivery; 41 m^3/h x 200 % = 82 m^3/h

Water delivery in case of fire; 41 m^3/h + 1 m^3/min x 60 min/h) = 101 m^3/h

Colony at Khanot:

Estimated maximum water consumption; 2,850 m^3/day, 119 m^3/h

Estimated maximum water delivery; 119 m^3/h x 200 % = 238 m^3/h

Water delivery in case of fire; 119 m^3/h + (2 m^3/min x 60 min/h) = 239 m^3/h

Water distribution system: Booster pump system

Service reservoir:

Service Reservoir

Lakhra mine site;

Effective capacity $(41\ m^3/h \times 12\ h) + 50\ m^3 = 542\ m^3$ over

Dimension $10\ m \times 10\ m \times 5.5\ m\ (height) = 550\ m^3$

Colony at Khanot

Effective capacity $(119\ m^3/h \times 12\ h) + 100\ m^3 = 1,526\ m^3$ over

Dimension $10\ m \times 28\ m \times 5.5\ m\ (height) = 1,540\ m^3$

e) Pumping equipment

Main specifications of each kind of pump are shown in Table 6-1 to Table 6-3.

6-3-2 Sewage

Storm drainage can be considered to be the main drainage system, however it is not available due to the lack of rainfall at Lakhra mine site. The sanitary sewage of offices and work shops only may be taken into consideration. The dirty water from water closets runs off through the septic tank into the cess pool and the overflowing water of the cess pool is allowed to percolate into the earth. The total capacity of cesspools is planned to be $150\ m^3$

6-4 Mechanical Equipment

6-4-1 Heavy Equipment Maintenance Shop

The conception of maintenance is gradually changing from corrective maintenance to preventive maintenance since open pit equipment is becoming bigger and bigger and productivity is getting higher and higher. Furthermore, the maintenance work has become very complicated because of the development of modern equipment, therefore, preventive maintenance is usually executed by the service engineers of each manufacturer.

However, Lakhra is located far from town so that it is expected to take a lot of time for the service engineers to be available at the colliery, therefore, the production loss due to the interruption of operation will be tremendous. In addition relatively numerous troubles are expected because a lot of machines will be used in the open pit.

Therefore, the heavy equipment maintenance shop will be provided to make the maintenance and repair work for all mining equipment used in the open pit.

From the aforementioned point of view, the shop will consist of the general workshop inclusive of such rooms as engine repair, assembling, welding, power, radiator repair, battery charging and hydraulic testing, and, in addition, the under carriage repair shop, preventive maintenance shop, tire service shop, painting and cleaning bays and warehouse, etc. The shop will also be equipped with necessary machineries, such as overhead crane, hydraulic press, electric and gas welders and milling machine, etc.

The annual maintenance cost includes the spare parts at 1 % of investment value as the foreign currency and also 1 % of C & F investment value exclusive of spare parts for the repair cost in local currency.

The typical layout of the shop is shown in Fig. 6-2.

6-4-2 Mechanical Workshop

The shop is used to produce the required parts of the machines used all over the colliery as well as to repair the machines especially used underground. Therefore the shop will consist of wood working, foundry, blacksmith, metal working and mechanical shops to be able to fulfil aforementioned tasks, and equipped with 2 no.s of lathes, 1 no. of shaper, 4 no.s of bench grinders, 1 no. of air hammer, each 2 no.s of electric and gas welders and other necessary machines and tools. In addition, the mine car repair shop and the battery locomotive shop will also be provided.

The shop is planned to be located near the underground portal for the convenience of transportation to and from the underground mine. The layout of the workshops is shown in Fig. 6-3. The conception of the maintenance cost of the mechanical workshop is the same as the heavy equipment maintenance shop.

6-5 Electrical Equipment

6-5-1 Mine Substation

As mentioned before it is necessary to provide the mine substation to receive, transform and distribute the power transmitted by WAPDA to all facilities in the colliery.

The total installed load capacity in the colliery is expected to be approximately 7,000 kW and, exclusive of the open pit supplied at 33 kV, the total load capacity to be supplied at 3.3 kV is estimated at 3,480 kW. Therefore, the required transformer capacity to be installed in the substation will be 4,000 kVA.

The areas and their load capacities supplied at 3.3 kV are shown in the following table.

Area	Installed capacity (kW)
Underground mine	930
Surface	2,000
Coal preparation plant	550
Total	3,480

The major equipment of the substation will be outdoor type and of dustproof construction, and will be able to operate under severe conditions of high ambient temperature and sand storms. However, some facilities such as switchgears and supervisory and control board will be of indoor type and installed in the control room provided at the corner of the power house constructed adjacent to the substation, from where the operation of the substation will be supervised and controlled by the shift working operators. Adequate capacity of the static capacitor will also be installed to compensate the expected reactive power within the distribution system.

The details of the surface facilities are shown in Table 6-4 and the prospecting one line power receiving and distribution diagram is shown in Fig. 6-5.

6-5-2 Emergency Power Generating Equipment

For a mechanized coal mine power failure causes not only interruption of the mining operations, but also the danger of the presence of mine gas and coal dust underground. Therefore it is recommendable to provide an emergency generator in Lakhra colliery the same way as in the other PMDC's coal mines to supply the power immediately at least to the safety facilities in case of power failure from the power source.

The total load capacity of the safety facilities to be supplied power by the generator is estimated at 750 kW as shown in following table.

Description	Q'ty	Installed capacity (kW)
Main ventilation fan	1	300
Air compressor	1	240
Cooling pump for above	1	15
Water supply pump	1	55
Sewage treatment facilities	1 lot	20
Lighting & office fixture	1 lot	50
Miscellaneous & loss	1 lot	70
Total		750

Therefore, 2 no.s of 500 kVA, 3.3 kV, 50 Hz and 3 phase diesel generators will be installed in the power house provided adjacent to the mine substation.

6-5-3 Power Distribution Facilities

The power distribution system on the surface will be carried out by the overhead lines at 33 kV for the open pits and at 3.3 kV for the other facilities.

The title of each overhead line and facilities and/or areas to be supplied with power by the line concerned are as follows:

(1) East open pit line: east open pit
(2) West open pit line: west open pit
(3) Office line: administration office, foremen office, heavy equipment maintenance shop.
(4) Ventilation fan line: main ventilation fan, air compressor, mine office and safety lamp room.
(5) Winder line: Winding machine
(6) Workshop line: Electrical & mechanical workshops, mine car repair shop, battery locomotive shop.
(7) Plant line: Coal preparation plant, tippler and conveyor belt for U/G raw coal receiving.
(8) Underground line: Underground mine

Among the aforementioned overhead lines, only the office line and the ventilation fan line will be supplied with power by the emergency generators in case of an incoming power failure.

6-5-4 Operational Supervision and Control

In general each surface facility will be supervised and controlled by the shift working operators, however, no operator is envisaged for the main ventilation fan which will be supervised from the main winding machine room remotely since the ventilation fan is to be operated continuously.

6-5-5 Communication Facilities

The private telephone system will be utilized for the internal and external communication of the colliery. The automatic crossbar exchanger with 2 circuits of office lines and 100 circuit of internal lines will be used for internal calls and, on the other hand, the exchange to and from the offices line will be carried out by the shift working telephone operator to control external calls.

All facilities such as automatic exchanger, the operator switch board for the office line, the storage batteries with automatic rectifier and other necessary accessories will be installed in the telephone exchange room provided in the administration office. Approximately 80 no.s of telephones will be installed in the colliery, 50 in the administration and mine offices, each 1 in the east and west open pits, 10 in the underground mine and the remainder in the other surface facilities.

The fixed stations of the wireless system for the open pit and of the inductive radio system for the underground mine will be installed in each office room in the mine office respectively to carry out the pit and mine management smoothly. The antenna tower for the open pit wireless system will be provided by the mine office.

6-5-6 Electrical Workshop

Adjacent to the mechanical workshop the electrical workshop will be provided. The workshop will be equipped with an overhead travelling crane, the coil winding machine, the test operating facility at various voltages and other necessary equipment, instrument and tools and used for the maintenance and repair work of all electrical equipment in the colliery. However H/T coil rewinding and other special work will be done by outside professional factories. The welding machines, lathes and the other equipment installed in the mechanical workshop which are possible to use in common will not be installed in the electrical workshop.

6-5-7 Lighting Equipment

All surface buildings and facilities will be equipped with adequate room lighting fixtures. In addition the major roads within the colliery, especially their crossings, will be illuminated by means of the fluorescent lamps and the stock yards and the portal area of incline by mercury vapour lamps to prevent accidents and larcenies.

6-5-8 Maintenance

The daily inspection and maintenance of the surface electrical equipment will be carried out by the operators, however, periodic inspection and maintenance will be carried out by the electrical workshop staff.

The annual maintenance cost includes one year's use of spare parts for the mine substation, power house and overhead lines, etc. and 2 % of investment value for the electrical workshop to replenish the damaged fixtures as foreign currency, and also 1 % of investment value of the foreign currency in C & F excluding the spare parts in local currency.

6-6 Others

6-6-1 Drilling Machine

The mining plan in this report is established based on not only the results of the previous drilling work, 19 no.s by PMDC and 3 no.s by GSP, but also the result of 50 no.s of drillings with approximately 100 m of average depth carried out by the JICA survey team in 1979. However, at the stage of the definite study and, more over, at the stage of the commencement of the mining operation requiring more detailed and tangible yearly mining plan, it is considered that further drilling work to confirm the condition of coal seams, overburden and faults, etc. will be required.

Therefore, 2 drilling machines will be provided for the drilling team established in the planning section to carry out further drilling work. The machines will be replaced every 10 years.

The annual maintenance cost includes the spare parts for one year's use as foreign currency and, in addition, 1 % of investment value of the foreign currency in C & F excluding spare parts in the local currency.

6-6-2 Vehicle

For the transportation vehicles within the colliery 2 units of staff cars, 6 units of 4 wheels drive pickups and 1 ambulance will be provided.

The annual maintenance cost will be paid in local currency, 5 % of C & F investment value, same as the patrol vehicles in the open pit.

6-6-3 Fixture

(1) Computer

The calculation of the operating cost in the colliery is very complicated as the coal is mined from 3 areas, 2 open pits and 1 underground mine. In addition it is very troublesome to control the enormous numbers of spare parts as various machines are used in the colliery.

Therefore 1 set of computer, 256 kB of operating capacity, will be installed in the system section room in the administration office not only to calculate the operation cost in each mine and wages and salaries for all personnel but also to control the spare parts and materials for all equipment and facilities in the colliery. As the annual maintenance cost in foreign currency 10 % of investment value is provided.

(2) Others

Not only the administration office but also mine office, hospital and dispensary will be equipped with necessary furniture and fixtures from the local market. The annual maintenance cost in local currency will be 1 % of investment value.

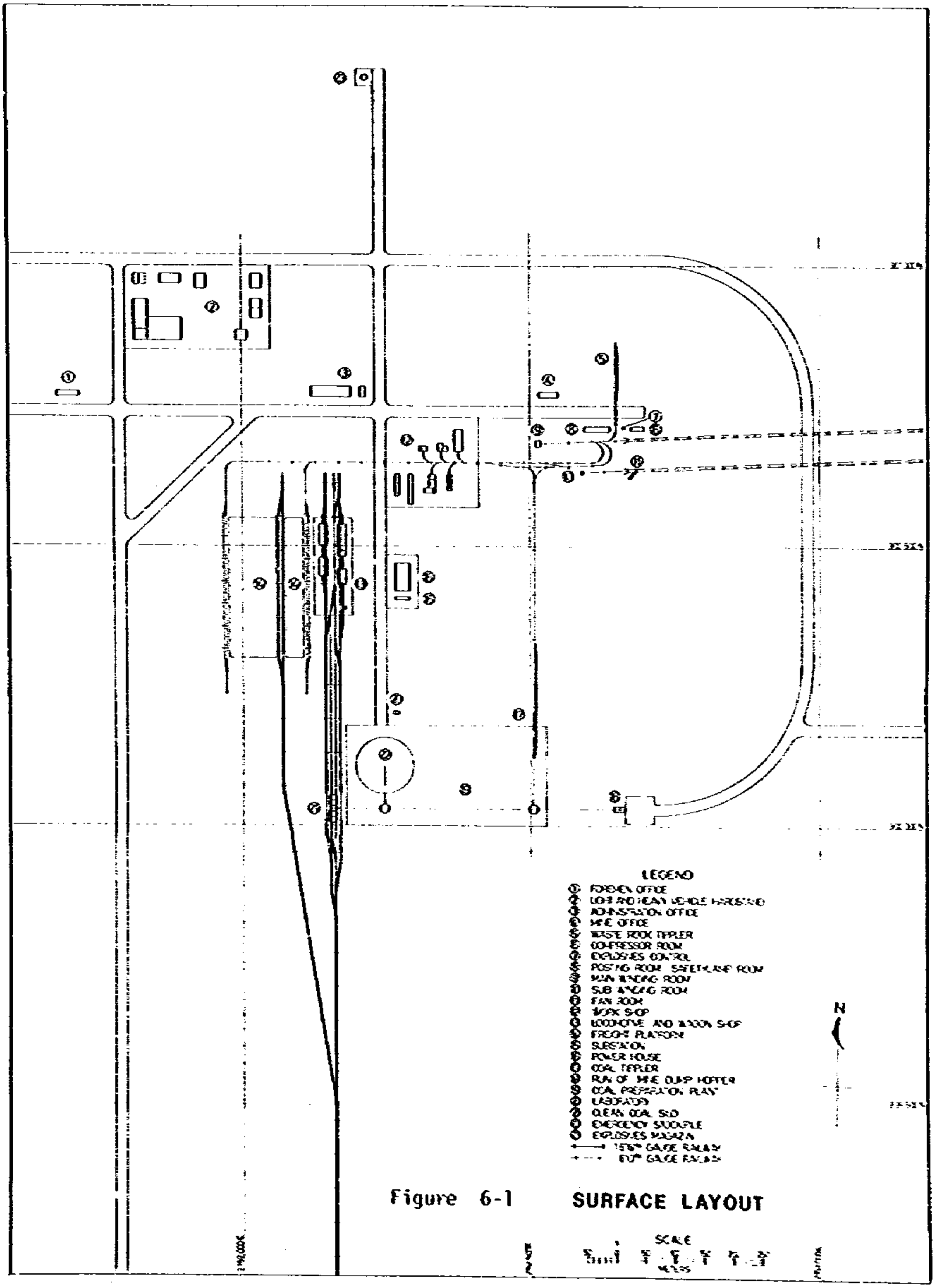


Figure 6-1 SURFACE LAYOUT

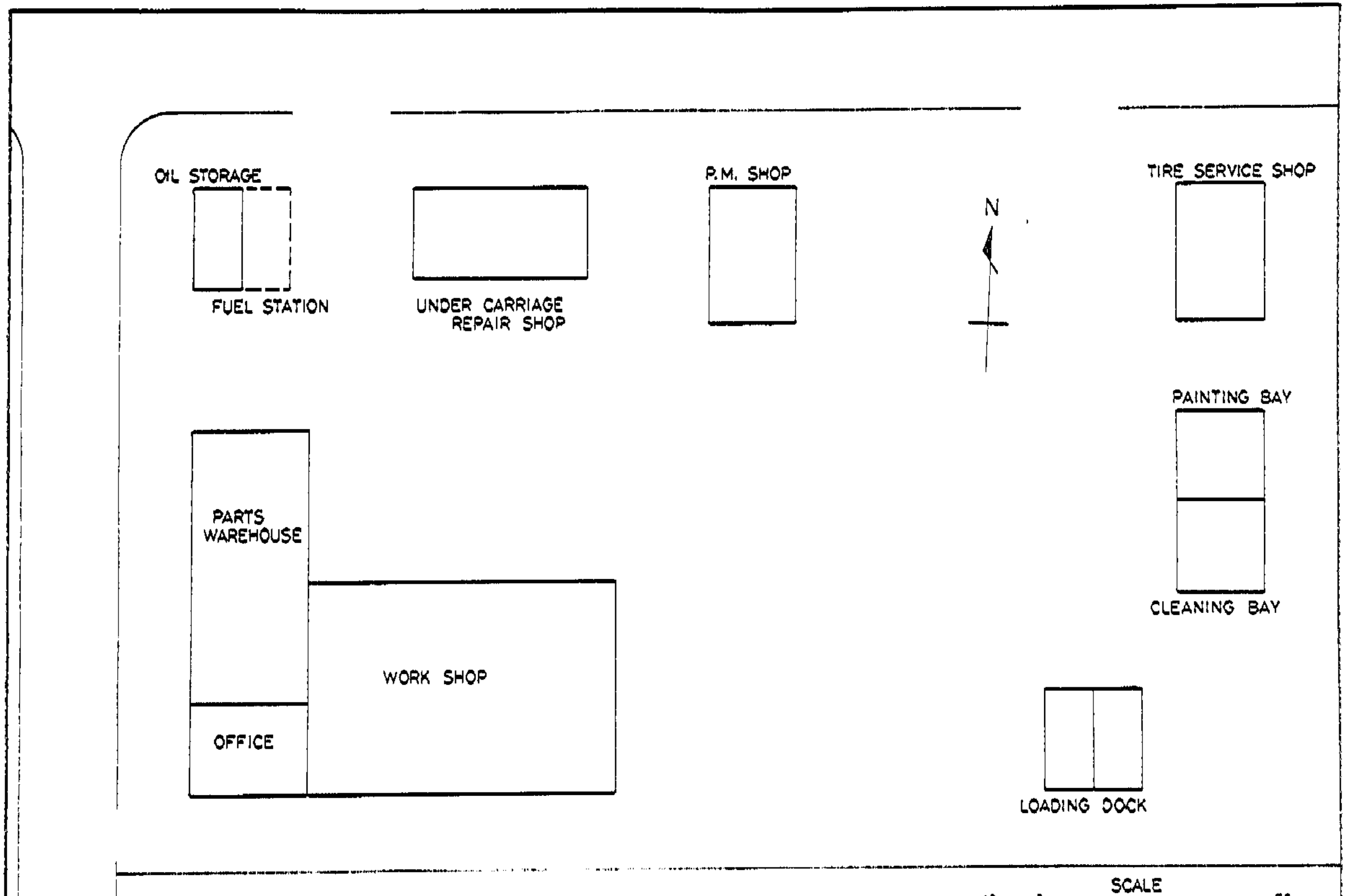


Figure 6-2 HEAVY EQUIPMENT MAINTENANCE SHOP

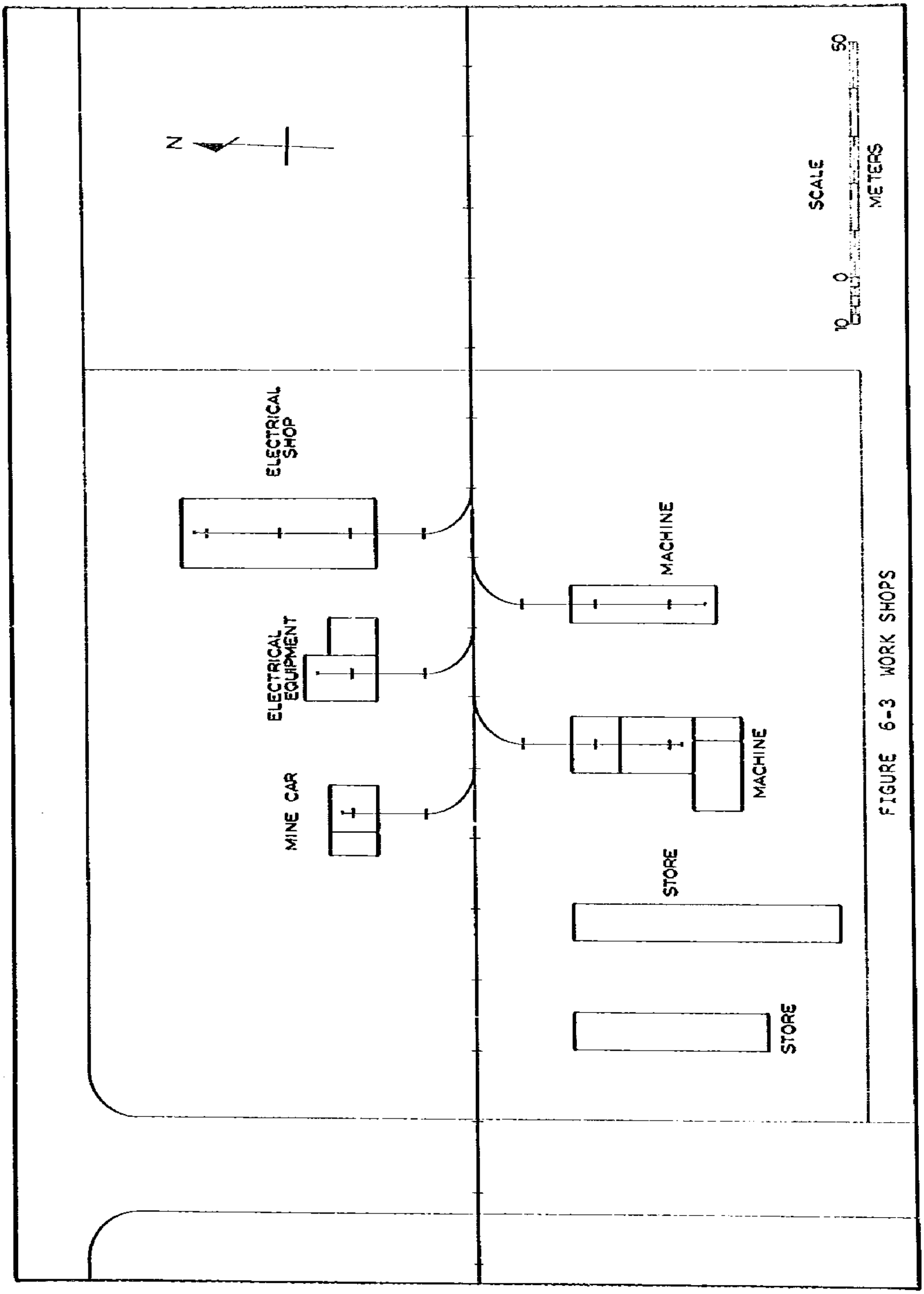


FIGURE 6-3 WORK SHOPS

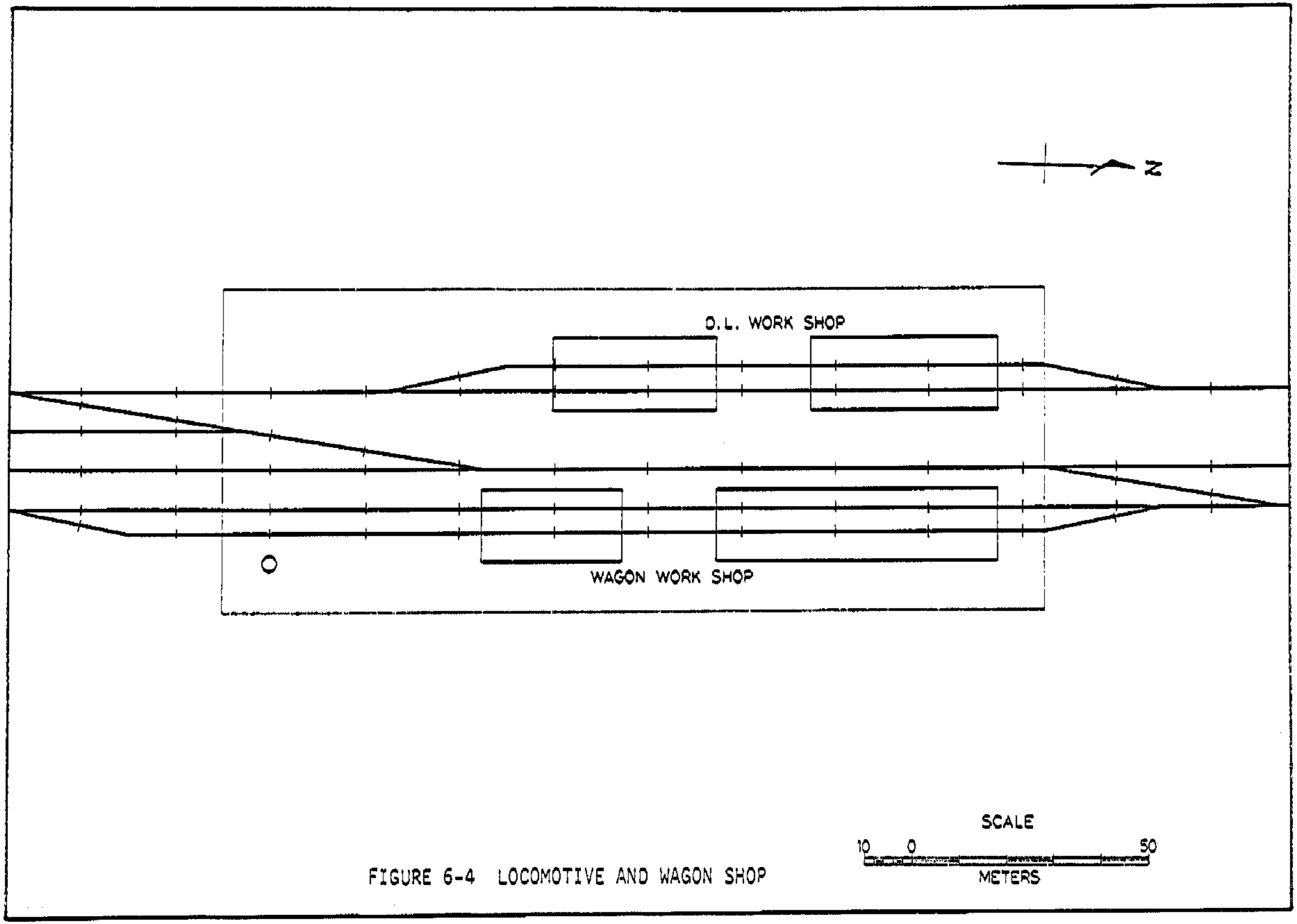


FIGURE 6-4 LOCOMOTIVE AND WAGON SHOP

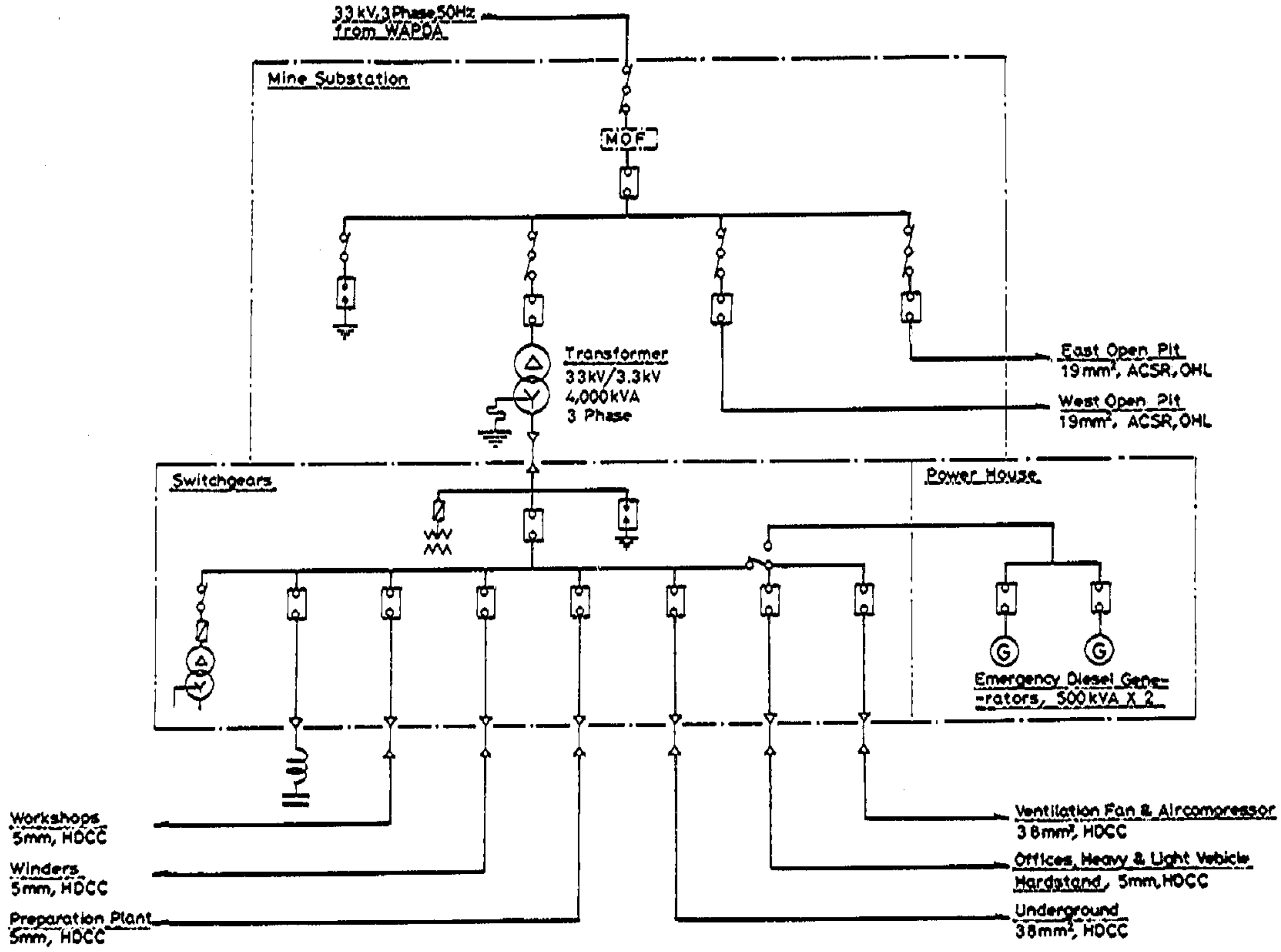


FIGURE 6-5 TYPICAL ONE LINE POWER RECEIVING AND DISTRIBUTION DIAGRAM

TABLE 6-1 WATER INTAKE PUMP

Description		Note
Location		Moored Pontoon on River Indus app. 3 km from Khanot
Service Distance		River Indus to Sand Basin
Pipe Length	(km)	3
Water Quantity	(m^3/min.)	2.93
Total Head	(m)	90
Suction Velocity	(m/sec.)	2.0
Suction Diameter	(mm)	125
Shaft Horsepower	(kW)	31
Required Motor Output	(kW)	36
Required Pump Specification		1.7 m^3/min. x 90 m H x 45 kW
Required No.		3 (2-Operation, 1-Standby)
Operating Motor Output	(kW)	90
Pipe Diameter	(mm)	200
Water Velocity in Pipe	(m/sec)	1.57

TABLE 6-2 WATER DELIVERY PUMP

Description	Note	
Location	Khanot Sand Basin	Khanot Purification Plant
Service Distance	Settling Pond to Colliery Distribution Pond	Khanot Cleaning Station to Colony Service Reservoir
Pipe Length (km)	24.5	1.5
Water Quantity (m^3/min)	0.75	2.18
Total Head (m)	180	30
Suction Velocity (m/sec)	2.0	2.0
Suction Diameter (mm)	100	125
Shaft Horsepower (kW)	32	8
Required Motor Output (KW)	37	10
Required Pump Specification	1.1 m^3/min x 185 m H x 55 kW	1.7 m^3/min x 30 m H x 15 kW
Required No	2 (1 - Operation, 1 - Standby)	3 (2 - Operation, 1 - Standby)
Operating Motor Output (kW)	55	30
Pipe Diameter (mm)	200	200
Water Velocity in Pipe (m/sec)	0.4	1.18

TABLE 6-3 WATER SUPPLY PUMP

Description	Note	
Location	Colliery Service Reservoir	Colony Service Reservoir
Service Distance	Distribution Pond to Facilities	Distribution Pond to Colonies
Pipe Length (km)	1.0	1.0
Water Quantity (m^3/min)	1.68	3.98
Total Head (m)	75	55
Suction Velocity (m/sec)	2.0	2.0
Suction Diameter (mm)	150	150
Shaft Horse Power (kW)	30	26
Required Motor Output (kW)	35	30
Required Pump Sepecification	2.6 m^3/min. x 75 m H x 55 kW	2.6 m^3/min. x 55 m H x 37 kW
Required No.	2 (1 - Operation, 1 - Standby)	3 (2 - Operation, 1 - Standby)
Operating Motor Output (kW)	55	74
Pipe Diameter (mm)	150	250
Water Velocity in Pipe (m/sec)	1.62	1.42

TABLE 6-4 INSTALLED MOTOR CAPACITY ON SURFACE

Description	Unit capacity (kW)	Installed No.	Total installed capacity (kW)
Main winding machine	160	1	160
Sub winding machine	20	1	20
Air compressor	240	2	480
ditto	75	1	75
Cooling pump for above	15	1	15
Main ventilation fan	300	1	300
Elec., mech., mine car & work-shop		1 lot	400
Heavy equipment maint. shop		1 lot	200
Booster pump for water supply	55	1	55
Sewage treatment facilities		1 lot	20
Surface lighting & office fixture		1 lot	50
Cool preparation plant		1 lot	465
Tippler and belt conveyor for U/G raw coal receiving		1 lot	80
Miscellaneous			230
Total			2,550