Secondary mining recovery:

$$\text{Entry: } \frac{4.5/6 \times 48{,}834 \times 1.5}{88{,}335 \times 4.6} = 0.1352$$

$$\text{Pillar: } \frac{(80{,}910 - 48{,}834) \times 4.6 \times 0.4}{88{,}335 \times 4.6} = 0.1452$$

Therefore, the total mining recovery of one mining panel is 65.3%.

3.7 Mining Machine

3.7.1 Continuous Miner

The continuous miner is able to cut the seam up to the thickness of 3.1 m. Specifications of continuous miner are shown in Table III-4 and the outline in Figure III-18.

3.7.2 Shuttle Car

A shuttle car is used for conveying coal from the continuous miner to the belt conveyor. Outline of the shuttle car is shown in Figure III-19.

3.8 Raw Coal Production

Annual raw coal production is planned to be 640,000 tons under full operation in this plan by taking into account the various relevant elements such as minable reserves, modes of formation of coal, mining method, technical level of mining, etc.

Number of working days per year:	240 days
Operating system:	4 shifts per day, 3 shifts for operation and 1 shift for maintenance.
Number of working faces:	3 with 1 standby face.
Daily production of raw coal:	2,670 tons (280 tons per face per shift)
Annual production of raw coal:	640,000 tons
Mine life:	40 years

Daily raw coal production plan is as follows:

(Working thickness) x (Width of entry) x (Driving length per shift) x (Specific gravity) x (Mining safety ratio) x (Number of shifts) x (Number of faces)

= 3.1 m x 6 m x 12 m x 1.55 x 0.86 x 3 x 3 ≒ 2,670 tons

2,670 tons/day x 240 days ≒ 640,000 tons

Thus, annual raw coal production is 640,000 tons.

Table III-4 Specifications of continuous miner (12CM 11-9A & -10A)

GENERAL			
Mining Capacity	8-12TPM		
Cutter head Diameter	0.914m		
Conveyor Width	0.762m		
Conveyor Depth	0.203m	Seam Height to	3.556m
Conveyor Chain Pitch	0.635/0.203m	Basic Chassis Height	0.914m
Crawler Width	0.457m	Height at Boom Pivot	0.965m
Crawler Pitch	1.295/0.203m	Ground Clearance	0.229m
Ground Bearing Pressure	24.5PSI		
AC/DC SCR Electric Tram Drive			
Weight	87,000 lbs.		
Machine length Overall	10.008m		
Dust suppression Water Sprays			

	60HZ	50HZ
Conveyor Speed	400FPM	385FPM
Cutter Speed	64RPM	53RPM
Bit Tip Speed	605 FPM	500FPM
Gathering Arm speed	51RPM	50RPM

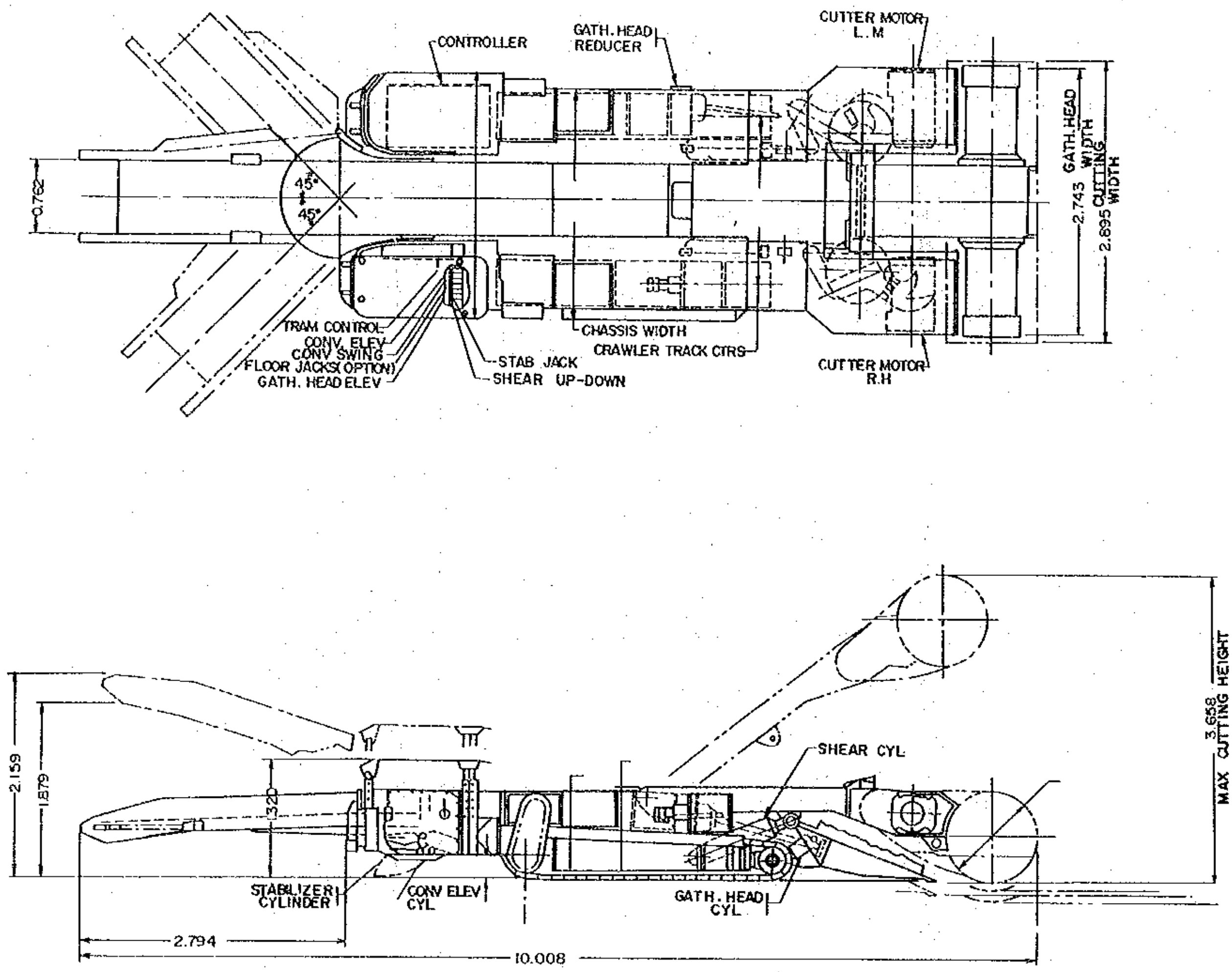


Figure Ⅲ-18 Continuous Miner

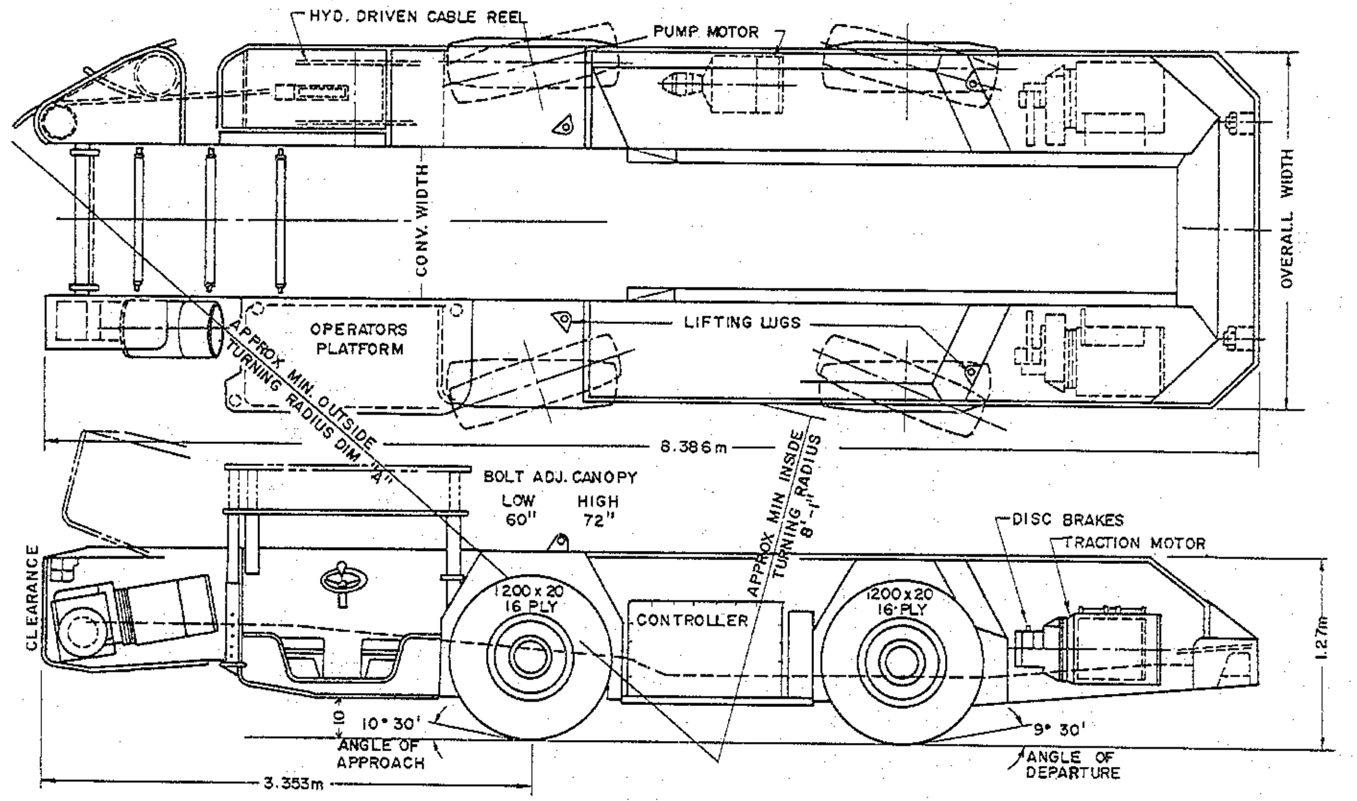


Figure Ⅲ-19 Shuttle Car

3.9 Manpower Requirements

Personnel arrangement per shift of each working face is as follows:

Foreman:	1
Continuous miner operator:	1
Assistant for the above:	1
Shuttle car operators:	2
Roof bolters:	3
Mechanician:	1
Electrician:	1
Belt man:	1
Total:	11

Therefore, personnel at the faces is:

11 persons x 3 faces x 3 shifts/day = 99 persons/day

In addition, personnel required for maintenance is:

4 persons x 3 faces x 1 shift/day = 12 persons/day

Thus, the total number of the face-workers is:

99 persons/day + 12 persons/day = 111 persons/day

From the formula shown below, the productivity is 24 tons/person.

$$\frac{2{,}670 \text{ tons}}{111 \text{ persons}} = 24 \text{ tons/person}$$

3.10 Daily Working Time

Daily time schedule is as follows based on 8 working hours per shift and 4 shifts per day:

1st shift:	7:00 to 15:00
2nd shift:	13:00 to 21:00
3rd shift:	19:00 to 3:00
4th shift:	1:00 to 9:00

The time schedule for 3 mining shifts and 1 maintenance shift is as shown in Figure III-20.

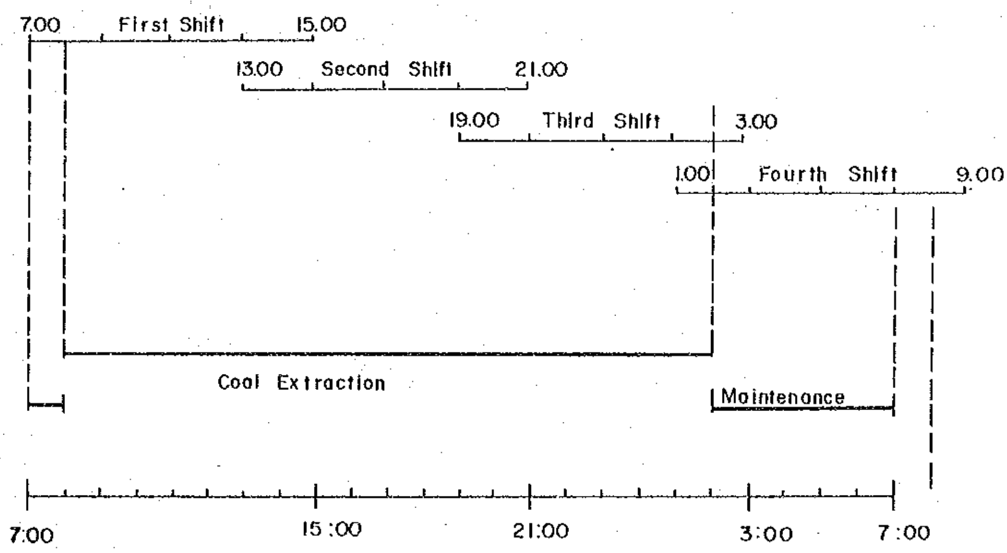


Figure Ⅲ-20 Working Cycle of Mining

CHAPTER 4. TRANSPORTATION PLAN

CHAPTER 4. TRANSPORTATION PLAN

4.1 Raw Coal Transportation

Compared with the mine tub method, belt conveyor method is employed because of its advantage in both safety and economic standpoints.

The flow of raw coal from the working face to the surface raw coal pocket is as follows:

Shuttle car → feeder breaker → panel BC → coal storage bunker (100 tons) → cross main entry BC → main entry BC → coal storage bunker (200 tons) → belt incline BC → surface pocket

where, BC: Belt conveyor

4.2 Design of Belt Conveyor

Raw coal to be transported is as follows:

1 shift : 890 tons (3 working faces)

1 day : 2,670 tons

Belt conveyor operating hours: 12 hours/day

Belt speed : 135 m/min.

Maximum slope of belt conveyor: 16°

Since 3 working faces are concentrated in the same mining block, the belt conveyor capacity of the belt incline BC, main entry BC and cross main entry BC is the same. The capacity of working face BC is calculated to be able to handle 600 tons per shift for covering two working faces.

The belt conveyor capacity is calculated by the following formula:

$$T = 60\ AVr \qquad (4.1)$$

$$A = K\ (0.9B - 0.05)^2 \qquad (4.2)$$

where,
- T: Belt conveyor capacity (tons/hr)
- A: Cross sectional area of materials carried (m^2)
- V: Belt speed (m/min.)
- r: Apparent specific weight of materials carried ($tons/m^3$) (1.00 for coal)
- K: Coefficient of carrier type
- B: Width of belt (m)

(1) In case the trough angle of belt conveyor is 20° and 3 roll trough carriers (K = 0.1245) are used, belt width of the belt incline BC, main entry BC and cross main entry BC is calculated by the formula (4.1) as follows.

$$A = \frac{2{,}670/12}{60 \times 135 \times 1.00} = 0.02747\ m^2$$

Therefore, belt width is given by the formula (4.2):

$$B = \left(\sqrt{\frac{A}{K}} + 0.05\right)/0.9 = 0.5775\ m$$

The belt width of 900 mm is employed by taking into account the safety factor.

Actual transport capacity (T) is as follows by the formula (4.1):

$$T = 60 \times 0.1245 \times (0.9 \times 0.9 - 0.05)^2 \times 135 \times 1.00 = 582.5 \text{ tons/hr}$$

Thus, transport capacity is 500 tons/hr.

(2) Belt conveyors for panels

Required quantity (Q) to transport coal from two working faces to the cross main entry BC within 4 hours is calculated as follows:

$$Q = \frac{890 \text{ tons}}{3 \text{ faces}} \times 2 \text{ faces} \times \frac{1}{4 \text{ hours}} = 148 \text{ tons/hr}$$

Thus, in case the trough angle of belt conveyor is 20° and 3 roll trough carriers are used (K = 0.1245), then the belt width of panel BC is calculated by the formula (4.1) as follows:

$$A = \frac{148}{60 \times 135 \times 1.00} = 0.01827 \text{ m}^2$$

Therefore, the belt width is given by the formula (4.2):

$$B = \left(\sqrt{\frac{0.01827}{0.1245}} + 0.05\right)/0.9 = 0.481 \text{ m}$$

For safety purposes, the belt width of 600 mm is employed. Actual transporting capacity (T) is about 200 tons/hour as follows by the formula (4.1):

$$T = 60 \times 0.1245 \times (0.9 \times 0.6 - 0.05)^2 \times 135 \times 1.00 = 242.1 \text{ tons/hr}$$

(3) Power required for belt conveyor

Power required for operating the belt conveyor is calculated from the following formula:

$$P = P_1 + P_2 + P_3 + Pt \quad \cdots\cdots \quad (4.3)$$

$$P_1 = \frac{f(\ell + \ell_0) WV}{6,120} \quad \cdots\cdots \quad (4.4)$$

$$P_2 = \frac{f(\ell + \ell_0) Qt}{367} \quad \cdots\cdots \quad (4.5)$$

$$P_3 = \frac{HQt}{367} \quad \cdots\cdots \quad (4.6)$$

where,

- P : Power required (kw)
- P_1 : Horizontal no-load power (kw)
- P_2 : Horizontal load power (kw)
- P_3 : Lifting load power (kw) (given with negative sign for descending belt)
- Pt : Operating power for tripper or stacker (kw)
- f : Friction coefficient of roller bearing
- W : Weight of moving portions other than carried materials (kg/m)
- V : Belt speed (m/min)
- H : Lift (m) (vertical height of ascending or descending belt including the height of tripper)
- ℓ : Length of conveyor (m) (horizontal centre distance of head and tail belt pulley)

ℓ_0 : Corrected horizontal length of belt conveyor (m). This length is experimentally given by the following equation:

$$\ell_0 = \frac{0.77931}{f - 0.006436} + 15.93$$

Qt : Transport quantity at carrier side (t/hr)

i) Required power (P) for belt incline BC is given by the above formulas:

where,

f = 0.022

$$\ell_0 = \frac{0.77931}{0.022 - 0.006436} + 15.93 = 66.001 \text{ m} \fallingdotseq 66 \text{ m}$$

W = 63 kg/m (width of belt: 900 mm)

Pt = 1.25 kw in case fixed tripper is used.

Qt = 500 t/hr

$$P = \frac{0.022 \times (600 + 66) \times 63 \times 135}{6{,}120} + \frac{0.022 \times (600 + 66) \times 500}{367} + \frac{169 \times 500}{367} + 1.25$$

$$= 271.8 \text{ kw}$$

Therefore, 300 kw motor is to be used.

Effective tension (Fp) of belt is:

$$Fp = \frac{6{,}120P}{V} = \frac{6{,}120 \times 300}{135} = 13{,}600 \text{ kg} = 13.6 \text{ tons}$$

ii) Required power (P) for the main entry BC is:

where, f = 0.022, ℓ_0 = 66 m, W = 63 kg/m (width of belt: 900 mm),

Pt = 1.25 kw (fixed tripper is used), Qt = 500 t/hr.

$$P = \frac{0.022 \times (1{,}000 + 66) \times 63 \times 135}{6{,}120} + \frac{0.022 \times (1{,}000 + 66) \times 500}{367} + \frac{135 \times 500}{367}$$

$$+ 1.25 = 249.7 \text{ kw}$$

Therefore, 300 kw motor is to be used.

Effective tension (Fp) of belt is:

Fp = 13.6 tons

iii) Required power (P) for cross main entry BC is:

where, f = 0.022, ℓ_0 = 66 m, W = 63 kg/m (width of belt: 900 mm),

Pt = 1.25 kw (fixed tripper is used), Qt = 500 t/hr.

$$P = \frac{0.022 \times (1{,}000 + 66) \times 63 \times 135}{6{,}120} + \frac{0.022 \times (1{,}000 + 66) \times 500}{367} + \frac{75 \times 500}{367} + 1.25$$

$$= 168.0 \text{ kw}$$

Therefore, 200 kw motor is to be used.

Effective tension (Fp) of belt is:

$$Fp = \frac{6{,}120 \times 200}{135} = 9{,}067 \text{ kg} \fallingdotseq 9.1 \text{ tons}$$

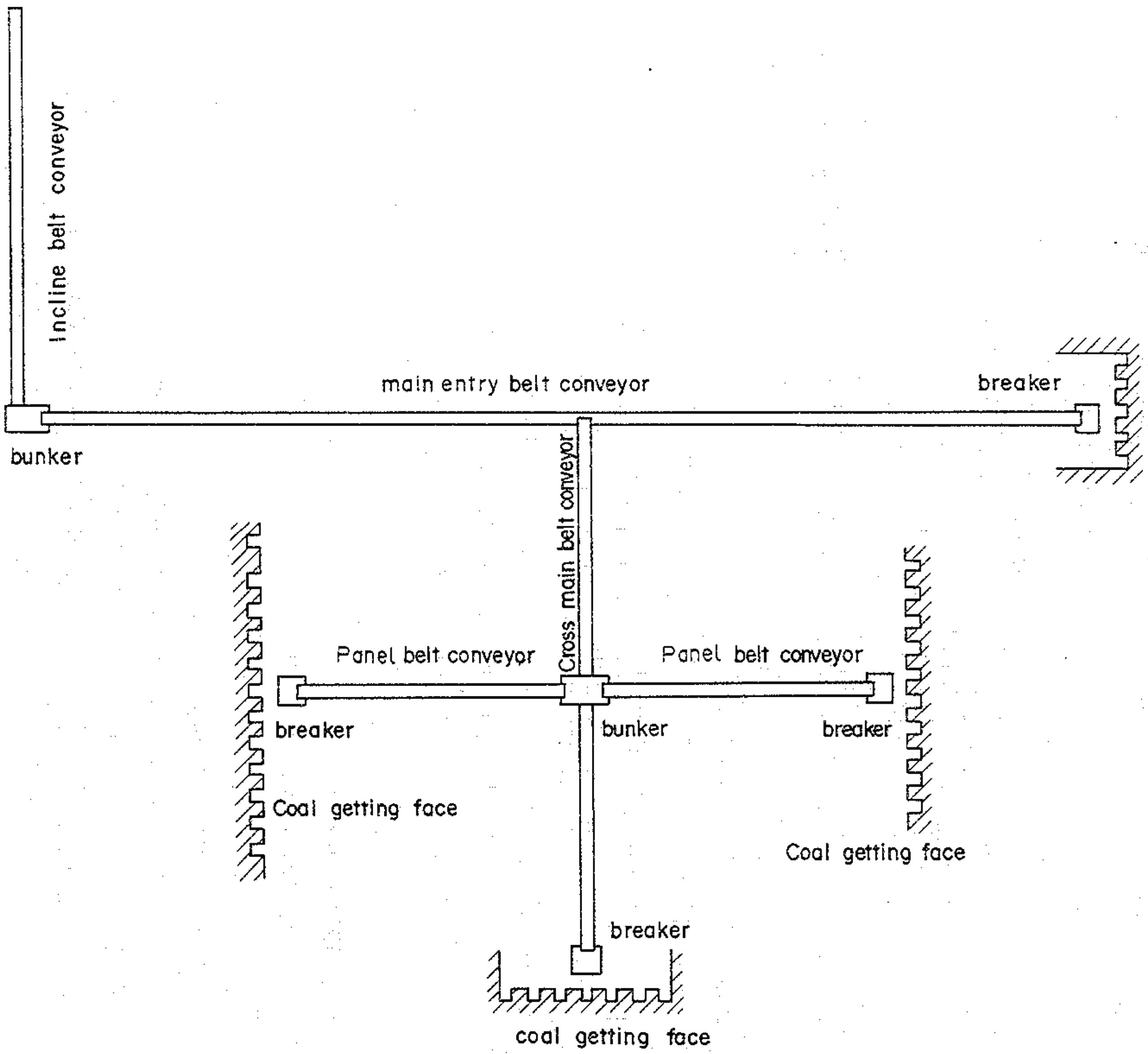


Figure Ⅲ-21 Layout of Underground Belt Conveyor

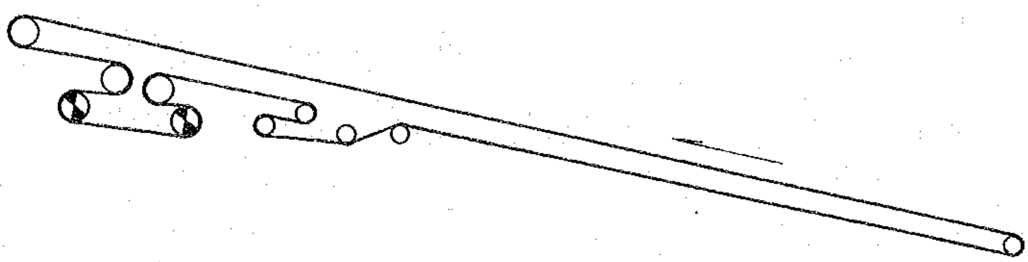


Figure Ⅲ-22 Incline and Main Entry Belt Conveyor

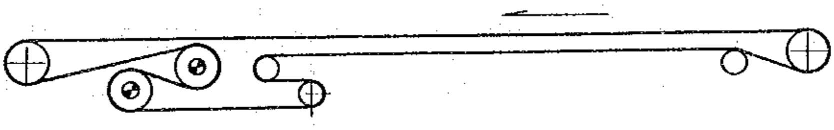


Figure Ⅲ-23 Cross Main Entry Belt Conveyor

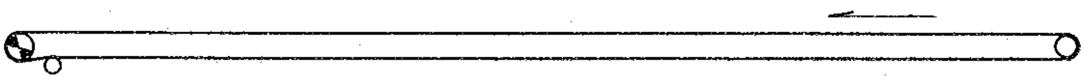
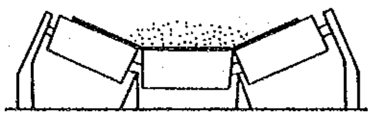


Figure Ⅲ-24 Panel Belt Conveyor

Table III-5 Specifications of Belt Conveyor

		Incline belt BC	Main entry BC	Cross main BC	Panel BC
Transport capacity	(T/H)	500	500	500	200
Belt width	(mm)	900	900	900	600
Belt speed	(m/min)	135	135	135	135
Horizontal length	(m)	600	1,000	1,000	500
Vertical length	(m)	169	135	75	10
Valid tension	(ton)	13.6	13.6	9.1	1.36
Drive unit size	(KW)	150 x 2	150 x 2	200	30



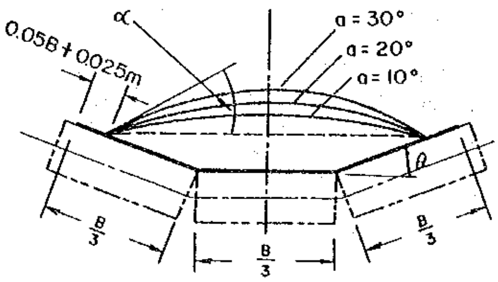


Figure III-25 Three Roll Trough Carrier (B: belt width)

iv) Required power (P) for panel BC is:

where, $f = 0.022$, $\ell_0 = 66$ m, $W = 35.5$ kg/m (width of belt. 600 mm),

$Pt = 1.25$ kw (fixed tripper is used), $Qt = 200$ t/hr.

$$P = \frac{0.022 \times (500 + 66) \times 35.5 \times 135}{6,120} + \frac{0.022 \times (500 + 66) \times 200}{367} + \frac{10 \times 200}{367} + 1.25$$

$= 23.2$ kw

Therefore, 30 kw motor is to be used.

Effective tension (Fp) of belt is:

$$Fp = \frac{6,120 \times 30}{135} = 1,360 \text{ kg} = 1.36 \text{ tons}$$

Specifications of belt conveyors determined from the above calculations are shown in Table III-5. The layout plan of underground belt conveyors system is shown in Figure III-21, outline of each belt conveyor is shown in Figures III-22, 23 and 24, and a section of 3 roll trough carriers is shown in Figure III-25.

4.3 Underground Coal Storage Bunker

Coal is produced by 3 shifts per day from three working faces in this plan. The capacity of the cross main entry belt conveyor is possibly insufficient to transport coal from 3 faces simultaneously since those faces are planned to be concentrated in one mining block.

Therefore, 100-ton and 200-ton coal storage bunkers are installed at the tails of the panel BC and main entry BC respectively in order to smoothen the transport of coal. The bunkers are automatically operated.

Figure III-26 shows the schematic drawing of the coal storage bunker.

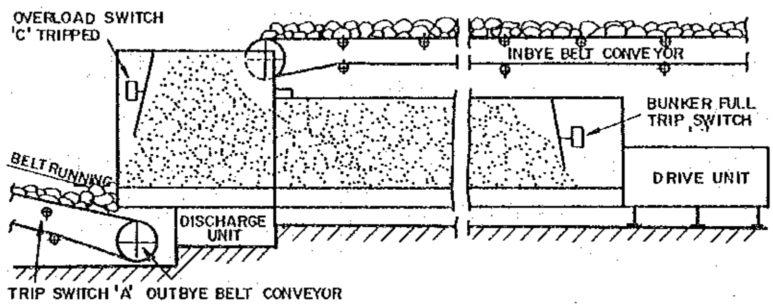


Figure III-26 Underground Bunker

4.4 Personnel Transportation

Since the time required for transporting the personnel directly affects the working hours, the transport plan is made for providing the shortest time for personnel transportation. Mine workers carry safety devices (such as cap lamp), wait in a waiting-room at the pit mouth, and are transported by a trackless personnel carrier to the working place.

Trackless transport is selected for the following reasons:

1) Flexibility: Fixed facilities are less equipped, and the vehicles can be concentrated at a required time in a required place.
2) Simplification of transport system: Workers can be transported without transfer from the pit mouth to the destination.
3) Speediness: Maximum speed of about 30 km/ hr can be easily operated.
4) Labour-saving: Personnel for transport operation can be greatly reduced since a transfer of load is almost not required.

Diesel engine driven type vehicles are to be employed for the trackless transport.

In the selection of trackless vehicles, low-floor type, articulated, all-wheel driven type vehicles, which can be used even for a narrow width, small curvature, rough road surface and steep slope, are employed.

Outline of the trackless personnel carrier is shown in Table III-6 and Figure III-27.

One trackless carrier can generally carry about 14 persons, and 4 carriers including standby are necessary to transport 39 persons required for one shift.

Table III-6 DIMENSIONS OF PERSONNEL CARRIER

A.	Height	1,524 mm
B.	Height,	
	over canopy	1,829 mm
C.	Height, rear chassis with canopy	1,829 mm
D.	Length, overall	5,982 mm
E.	Length, rear chassis	3,404 mm
F.	Width, power unit	1,956 mm
G.	Width, rear chassis	1,956 mm
H.	Wheel base	2,997 mm
I.	Turning radius,	
	outside	5,588 mm
	inside	3,404 mm

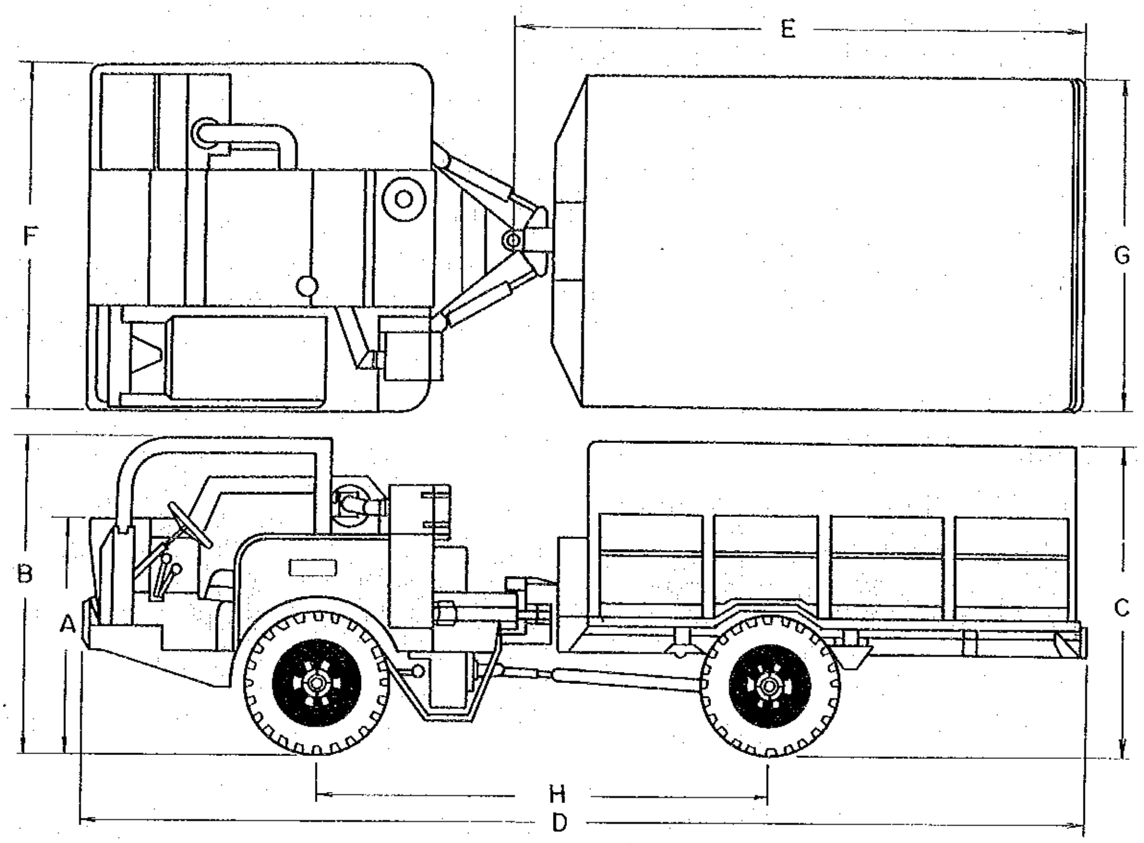


Figure Ⅲ-27 Personnel Carrier

4.5 Material Transportation

Materials used daily are roof bolts and materials for stopping. In addition, materials for belt conveyors, electric cables, steel pipes and explosives are used as required. These materials are transported by 3 trackless material carriers.

Trackless material carrier is outlined in Table III-7 and Figure III-28.

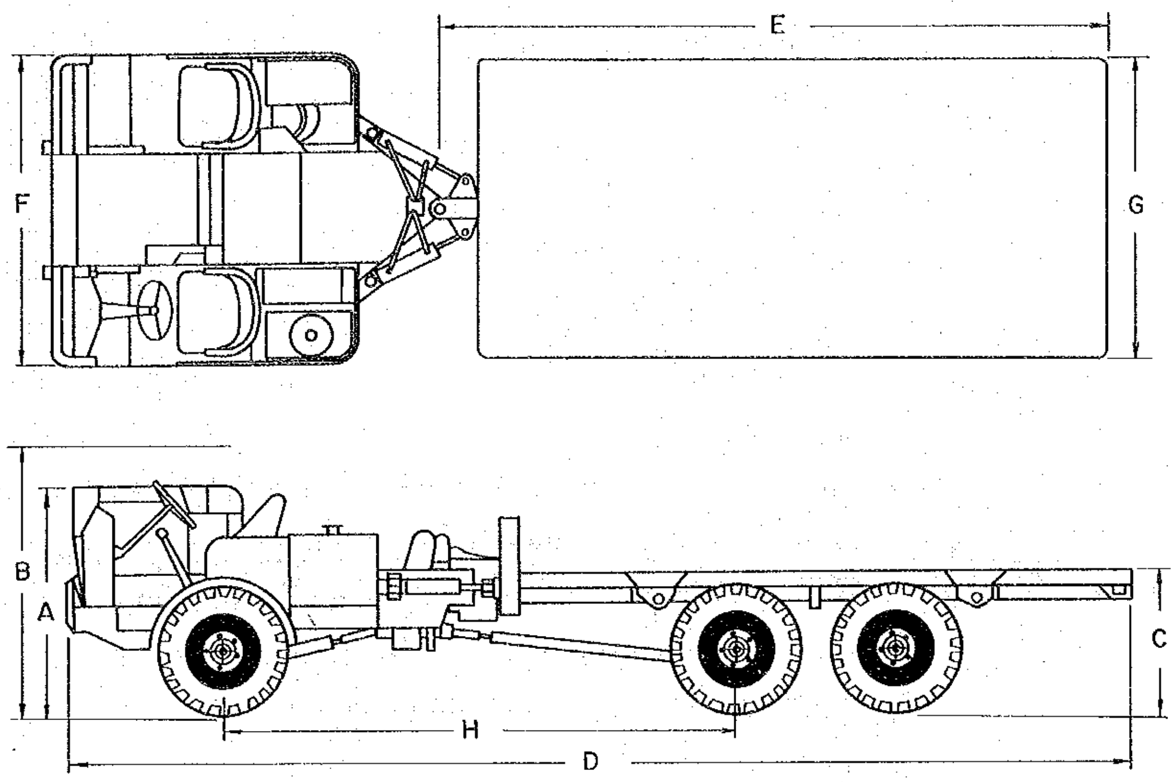


Figure Ⅲ-28 Material Carrier

Table III-7 DIMENSIONS OF MATERIAL CARRIER

A.	Height	1,524 mm
B.	Height, over canopy	1,829 mm
C.	Height of bed	965 mm
D.	Length, overall	6,400 mm
E.	Length, rear chassis	3,658 mm
F.	Width, power unit	1,956 mm
G.	Width, rear chassis	1,956 mm
H.	Wheel base	2,997 mm
I.	Turn radius,	
	outside	6,096 mm
	inside	3,886 mm

CHAPTER 5. MINE SAFETY

CHAPTER 5. MINE SAFETY

5.1 Safety Plan

Accidents in coal mines can be divided into frequently occurring accidents and serious accidents. The former includes the accidents caused by roof fall, collapse, blasting, explosives, transport, overturning and mishandling of equipment. For preventing these accidents, it is required to establish detailed standards and procedures of the work. Accidents can be reduced through strictly observing those standards and procedures in the course of safety training and management of daily work.

On the other hand, serious accidents are gas or coal dust explosion, spontaneous combustion, mine fire, and so forth. The following are required to prevent these accidents; establishment of countermeasures, introduction of central monitoring system for finding underground situations, enforcement of monitoring and quick response in the event of abnormal conditions.

1) Main countermeasures are:

   i) Gas or Coal Dust Explosion

      To detect the behavior of CH_4 gas. To spray water for controlling coal dust. To install stone dust barriers and to perform stone dusting.

   ii) Spontaneous Combustion

      To fully seal and close the goafs.

   iii) Mine Fires

      To control combustible substances and electric wires.

2) The central monitoring system should consist of the following items:

   i) Monitoring devices for mine gases

      CH_4 and CO gases

   ii) Belt conveyor monitoring devices

      Slip, deviation and breakage of belt, fire (CO gas), water spray, weighing raw coal, running conditions and checking coal quantity in bunker.

   iii) Monitoring devices for power distribution

      Electric current, voltage, earth resistance, and controlling switch.

   iv) Monitoring devices for main fan

      Air pressure, air volume, operating conditions, etc.

   v) Monitoring devices for drainage pump

      Water level and operation.

   vi) TV monitoring

      Graphic monitoring of main underground places.

3) Inductive radio system is as follows:

   Foremen and deputies should carry inductive radio sets in order to communicate smoothly and to make quick response in case of emergency.

5.2 Ventilation Plan

A central ventilation system using air intake and return inclines is employed in this plan. A diagonal ventilation system by vertical shaft will be required in future when working faces move far away from the pit mouth. However, this system is not studied in the report as the system will be employed about 20 years after commencement of coal production.

Required quantity of ventilation is determined as follows:

(1) Required quantity of ventilation for the underground workers:

Required quantity of ventilation per person:	$3\ m^3/min.$
Maximum number of workers:	78 persons
Effective air quantity:	35%

$$\frac{3 \times 78}{0.35} = 668.5\ m^3/min.$$

(2) Required quantity of ventilation for diluting the gases ejected from mining area:

Daily required ventilation quantity for the dilution is supposed to be $2\ m^3/min.$ for every 1 ton of raw coal. Daily production of raw coal is 2,670 tons.

$$2{,}670 \times 2 = 5{,}340\ m^3/min.$$

Therefore, total required quantity of ventilation is:

$$668.5 + 5{,}340 \fallingdotseq 6{,}000\ m^3/min.$$

5.2.1 Ventilation Calculation Method

Calculation of ventilation is as follows:

Pressure loss by Atkinson's formula is given by:

$$h = k\,\frac{L \cdot U \cdot V^2}{F} = k\,\frac{L \cdot U \cdot Q^2}{F^3} \qquad \cdots\cdots (5.1)$$

where,

- h: Pressure loss due to friction (mm in water column)
- F: Cross sectional area of entry (m^2)
- L: Length of entry (m)
- Q: Air quantity (m^3/sec)
- U: Peripheral length of cross section of entry (m)
- k: Coefficient of frictional resistance of entry
- V: Mean wind velocity (m/sec)

Required power of a fan is given by the following formula:

$$W = \frac{Q \times h}{6{,}120 \times \eta} \qquad \cdots\cdots (5.2)$$

where,

- W: Axial power (kw)
- Q: Air quantity (m^3/min)
- h: Negative pressure (mmAq)
- η: Efficiency of machine (0.85)

5.2.2 Central Ventilation System by Inclines

Central ventilation system using inclines is shown in Figure III-29.

(1) Ventilating conditions for each entry

i) Ventilating conditions for incline

Length of trackless incline: 1,287 m

Length of belt incline: 570 m

Cross sectional area of each entry: 14.46 m^2

ii) Ventilating conditions for main entry (5 entries)

Ventilation is necessary for 4 entries (2 entries for air intake and 2 entries for air return).

Total length: 10,000 m (intake: 5,000 m, return: 5,000 m)

Cross sectional area of entry: 6 m x 3.1 m = 18.6 m^2

iii) Ventilating conditions for cross main entry (7 entries)

Ventilation is necessary for 6 entries (3 entries for air intake with 2,000 m and 3 entries for air return with 2,000 m).

Total length: 4,000 m

Cross sectional area of entry: 6 m x 3.1 m = 18.6 m^2

iv) Ventilating conditions for mining panel (13 entries):

Since the room and pillar mining method is employed, the coal mining panel is divided into two sections by stopping. One section is used for air intake and the other for air return, and all the working faces are ventilated.

Maximum ventilation length: 1,450 m x 2 = 2,900 m

Cross sectional area of entry: 6 m x 3.1 m = 18.6 m^2

v) Air flow resistance (R) of each entry is given by:

$$R = k\,(L{\cdot}U/F^3) \qquad (5.3)$$

$$R = \frac{h}{Q^2} \qquad (5.4)$$

Entries are designed to be parallel to each other.

Parallel composition of airflow resistance is given by the following formula:

$$R = \frac{1}{\left(\frac{1}{\sqrt{R_1}} + \frac{1}{\sqrt{R_2}} + \frac{1}{\sqrt{R_3}} + \cdots\cdots\right)^2} \qquad (5.5)$$

Airflow resistance of main entry (R'_3) is given by the formula (5.3):

$$R'_3 = 0.00092 \times \frac{10{,}000 \times 18.2}{18.6^3} = 0.02602$$

where, k = 0.00092 (for bare entry), L = 10,000 m, U = 18.2 m, F = 18.6 m^2.

Therefore, composite airflow resistance is:

$$R_3 = \frac{1}{(2/\sqrt{R'_3}\,)^2} = \frac{R'_3}{4} = \frac{0.02602}{4} = 0.00651$$

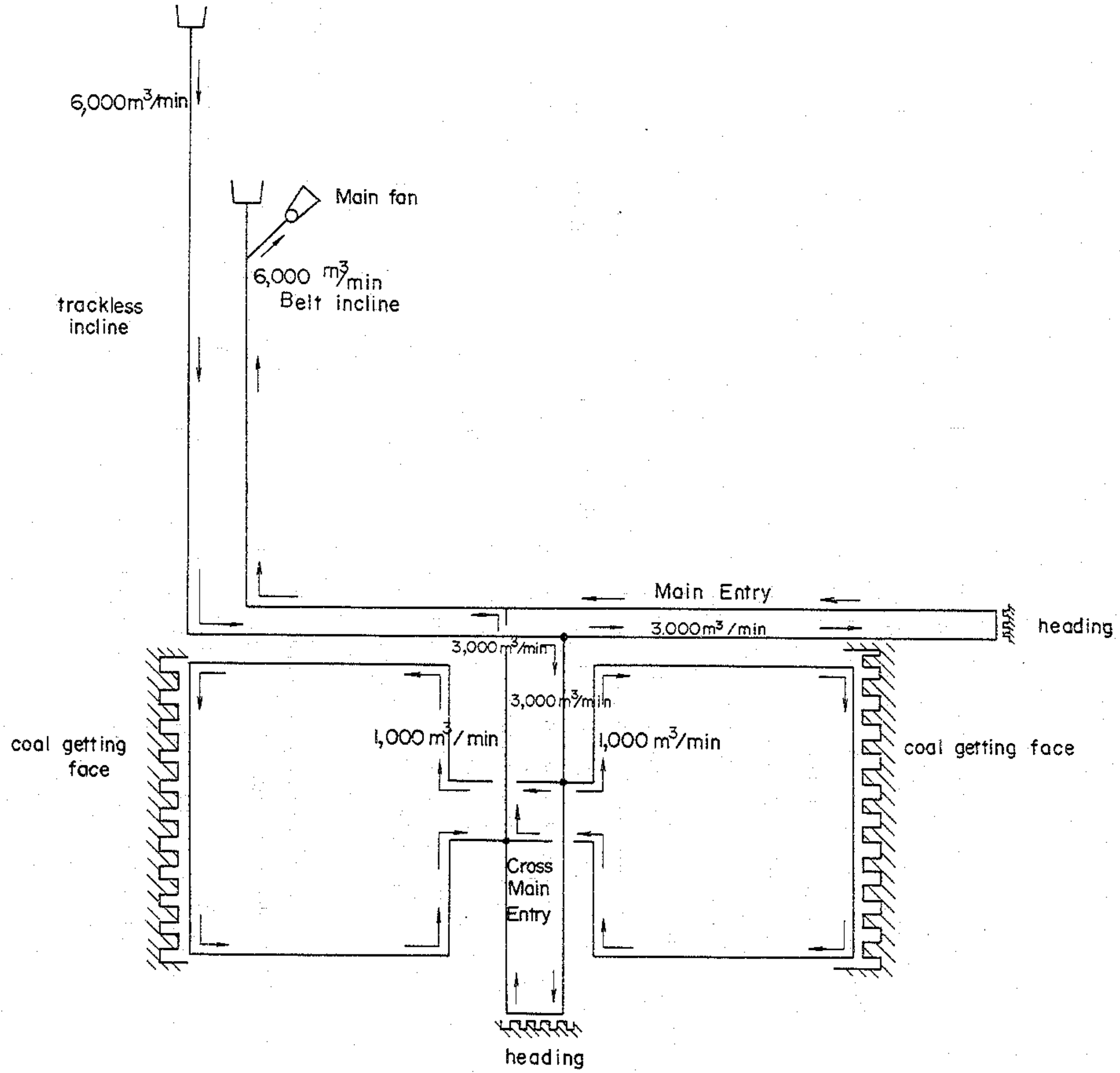


Figure Ⅲ-29 Ventilation Network

Airflow resistance of cross main entry (R'_4) is given by the formula (5.3):

$$R'_4 = 0.00092 \times \frac{4{,}000 \times 18.2}{18.6^3} = 0.01041$$

where, k = 0.00092 (for bare entry), L = 4,000 m, U = 18.2 m, F = 18.6 m^2

Therefore, composite airflow resistance is:

$$R_4 = \frac{1}{(3/\sqrt{R'_4})^2} = \frac{R'_4}{9} = 0.00116$$

(2) Calculation of pressure loss (h) of each entry

i) Pressure loss of belt incline

Air velocity (V) of belt and trackless inclines is 414.94 m/min. Pressure loss of belt incline is calculated as follows by the formula (5.1):

$$h_1 = 0.00068 \times \frac{570 \times 13.7 \times 414.94^2}{14.46 \times 60^2} = 17.56 \text{ mmAq}$$

where, k = 0.00068 (arched, straight, bare entry), L = 570 m, U = 13.7 m, F = 14.46 m^2.

ii) Pressure loss of trackless incline is given by the formula (5.1):

$$h_2 = 0.00068 \times \frac{1{,}287 \times 13.7 \times 414.94^2}{14.46 \times 60^2} = 39.66 \text{ mmAq}$$

where, k = 0.00068 (arched, straight, bare entry), L = 1,287 m, U = 13.7 m, V = 414.94 m/min, F = 14.46 m^2.

iii) Pressure loss of main entry is calculated by the formula (5.4) as follows:

$$h_3 = 0.00651 \times (6{,}000/60)^2 = 65.1 \text{ mmAq}$$

where, R = 0.00651, Q = 6,000 m^3/min.

iv) Pressure loss of cross main entry is given by the formula (5.4):

$$h_4 = 0.00116 \times (3{,}000/60)^2 = 2.90 \text{ mmAq}$$

where, R = 0.00116, Q = 3,000 m^3/min.

v) Pressure loss of mining panel is calculated by the formula (5.1) as follows:

$$h_5 = 0.00092 \times \frac{2{,}900 \times 18.2 \times 1{,}000^2}{18.6^3 \times 60^2} = 2.10 \text{ mmAq}$$

where, k = 0.00092 (bare entry), L = 2,900 m, U = 18.2 m, Q = 1,000 m^3/min, F = 18.6 m^2.

vi) Total pressure loss (H) is given by:

$$H = h_1 + h_2 + h_3 + h_4 + h_5$$
$$= 17.56 + 39.66 + 65.1 + 2.90 + 2.10 = 127.32 \text{ mmAq}$$

The specifications of central ventilation system are shown in Table III-8.

(3) Required power (W) for fan to be used in the central ventilation system is given by the formula (5.2):

Table III-8 Central Ventilation System

	Sectional area (m^2)	Peripheral length (m)	Ventilation length (m)	Friction coefficient	Gross air quantity (m^3/min)	Air flow resistance	Negative pressure (mmAq)
Belt incline	14.46	13.7	570	0.00068	6,000	0.00187	17.56
Truckless incline	14.46	13.7	1,287	0.00068	6,000	0.00397	39.66
Main entry	18.6 (5 entries)	18.2 (5 entries)	10,000	0.00092	6,000	0.00651	65.10
Cross main entry	18.6 (7 entries)	18.2 (7 entries)	4,000	0.00092	3,000	0.00116	2.90
Coal getting face	18.6	18.2	2,900	0.00092	1,000	0.00754	2.10

$$W = \frac{6,000 \times 127.32}{6,120 \times 0.85} = 146.85 \text{ kw}$$

Therefore, maximum negative pressure is 150 mmAq and the power for the fan is 200 kw by taking into account the safety factor.

An adjustable pitch axial propeller fan with a high efficiency and air pressure-adjusting function is to be employed. The specifications of the fan are given in Table III-9.

5.2.3 Overcast

Overcasts are constructed at the intersections between intake and return airways to secure fresh air at the working faces. Overcasts are constructed by steel frames and prestressed concrete blocks. Overcasts are outlined in Figure III-30.

5.3 Drainage Plan

5.3.1 Quantity of Mine Water

The quantity of mine water as the basis of determination of capacity of drainage facilities can be accurately known only after the starting of mining. The quantity of mine water in the Mpaka mine is relatively small. Consequently, the quantity of mine water in the mining area is assumed to be 0.4 m^3/min. (576 m^3/day) in the drainage plan.

5.3.2 Drainage Facilities

A main drain pump with 50 m^3 water sump is planned to be installed at the bottom of the belt incline (125 m above sea level) and local pumps are installed at cross main entry and panel entries near working faces.

Series of the drainage system is as follows:

working faces → 3" pipe → water sump at cross main entry → 4" pipe → water sump at the bottom of the belt incline → 4" pipe → surface

The drainage system is shown in Figure III-31.

5.3.3 Pressure Loss of Drain Pipe and Horsepower Required for Pump

Pressure loss of drain pipe and horsepower required for the pump are calculated based on the following assumption.

Quantity of mine water: 288 m^3/day for each mining face, total 576 m^3/day

Working hours of pump: 12 hours/day

Inside wall roughness of cast iron drain pipe (ϵ): 0.0005

Table III-9 Specifications of Fan

Air quantity (m³/min)	6,000
Max. negative pressure (mm Aq)	150
Pitch of fan	1-10
Power of electromotor (kw)	200

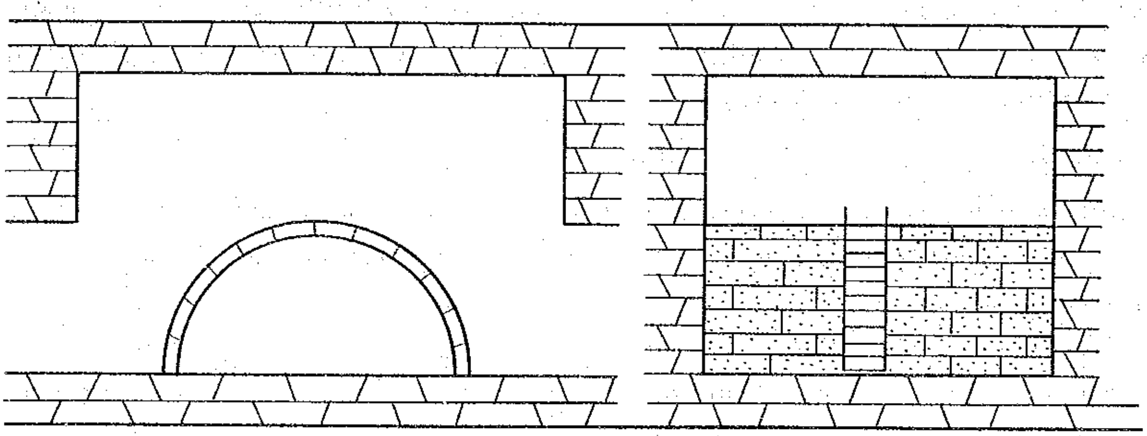


Figure III-30 Ventilation Overcast at Intersection

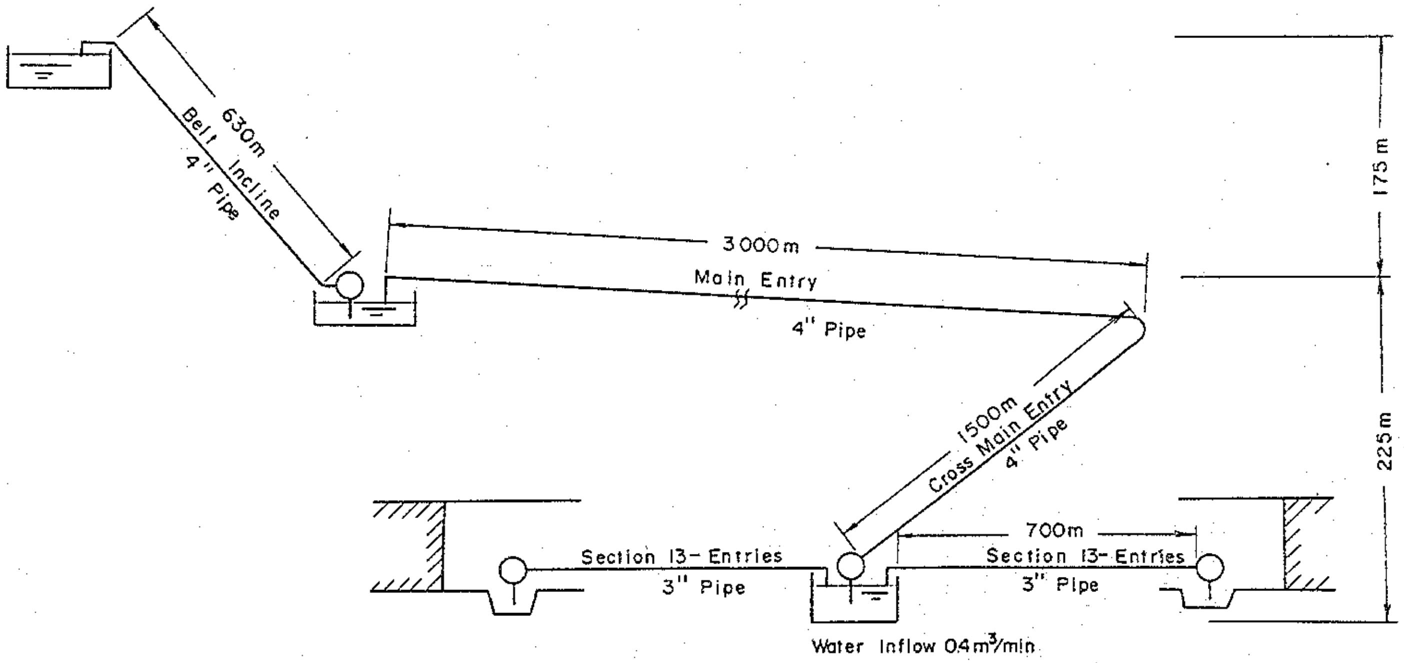


Figure III-31 Schematic Drawing of Drainage System

Total pressure loss (H) of drain pipe is calculated by the following formula:

$$H = h_1 + h_2 + Ha \quad \text{.......... (5.6)}$$

where,

h_1 : Pressure loss of drain pipe (m)

h_2 : Pressure loss without resistance of drain pipe (m)

Ha : Pressure head (m)

$$h = \lambda \cdot \frac{V^2}{2g} \cdot \frac{\ell}{d} \quad \text{.......... (5.7)}$$

where,

h : Resistance of pipeline (m)

λ : Friction coefficient of pipe

V : Velocity (m/sec)

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

ℓ : Length of pipe (m)

d : Inside diameter of pipe (m)

Horsepower required of pump (Np) is given by:

$$Np = \frac{r \cdot Q \cdot H}{75 \times \eta} \quad \text{.......... (5.8)}$$

where,

r : Specific weight of fluid (kg/m^3)

Q : Flow rate (m^3/sec)

H : Pressure head (m)

η : Efficiency of pump (0.8)

(1) Velocity and coefficient of friction of pipe

Velocity (V) and friction coefficient of pipe (λ) are calculated by the following formulas:

$$V = \frac{Q}{A} \quad \text{.......... (5.9)}$$

$$\lambda = \frac{1}{(2 \log \frac{r}{2\epsilon} + 1.75)^2} \quad \text{.......... (5.10)}$$

where, A : cross sectional area of pipe (m^2), r : radius of pipe (m).

i) For 4" (105 mm) pipes

$$V_1 = \frac{576/(12 \times 60 \times 60)}{\frac{0.105^2}{4} \times 3.14} = 1.54 \text{ m/sec}$$

$$\lambda_1 = \frac{1}{(2 \log \frac{0.0525}{0.0005} + 1.75)^2} = 0.0298$$

ii) For 3" (81 mm) pipes

$$V_2 = \frac{288/(12 \times 60 \times 60)}{\frac{0.081^2}{4} \times 3.14} = 1.30 \text{ m/sec}$$

$$\lambda_2 = \frac{1}{(2 \log \frac{0.0405}{0.0005} + 1.75)^2} = 0.0323$$

(2) Total pressure loss from working face to water sump and required horsepower of pump

The head loss is assumed to be 70 times the pipe diameter without the consideration of the friction factor of the pipe in the calculations.

Total pressure loss (H_1) and required horsepower (Np) are calculated by the formulas (5.6), (5.7) and (5.8) respectively:

where,

Pressure head:	25 m
Drainage length:	700 m
Inside diameter of drain pipe:	81 mm
Velocity:	1.30 m/sec.
Friction coefficient of pipe:	0.0323

$$H_1 = 0.0323 \times \frac{1.30^2}{2 \times 9.8} \times \frac{700}{0.081} + 0.081 \times 70 + 25 = 54.74 \text{ m}$$

$$Np = \frac{1{,}000 \times \frac{0.081^2}{4} \times 3.14 \times 1.30 \times 54.74}{75 \times 0.8} = 6.11 \text{ HP}$$

Therefore, two 10-HP pumps are required.

(3) Total pressure loss from water sump in cross main entry to the bottom of belt incline and required horsepower of pump

Total pressure loss (H_2) and required horsepower (Np) are:

where,

Pressure head:	225 m
Drainage length:	4,500 m
Inside diameter of drain pipe:	105 mm
Velocity:	1.54 m/sec.
Friction coefficient of pipe:	0.0298

$$H_2 = 0.0298 \times \frac{1.54^2}{2 \times 9.8} \times \frac{4{,}500}{0.105} + 0.105 \times 70 + 225 = 386.88 \text{ m}$$

$$Np = \frac{1{,}000 \times \frac{0.105^2}{4} \times 3.14 \times 1.54 \times 386.88}{75 \times 0.8} = 85.94 \text{ HP}$$

Therefore, one 90-HP pump is required.

(4) Total pressure loss from water sump at belt incline to surface water sump and required horsepower for pump

Total pressure loss (H_3) and required horsepower (Np) are:

where,

Pressure head:	175 m
Drainage length:	600 m
Inside diameter of drain pipe:	105 mm
Velocity:	1.54 m/sec.
Friction coefficient of pipe:	0.0298

$$H_3 = 0.0298 \times \frac{1.54^2}{2 \times 9.8} \times \frac{600}{0.105} + 0.105 \times 70 + 175 = 202.95 \text{ m}$$

$$Np = \frac{1{,}000 \times \frac{0.105^2}{4} \times 3.14 \times 1.54 \times 202.95}{75 \times 0.8} = 45.08 \text{ HP}$$

Therefore, one 50-HP pump is required.

Specifications of drain pipe and pump are shown in Table III-10.

Table III-10 Drainage Pipe and Pump Specifications

	Vertical distance (m)	Drainage pipe length (m)	Pipe diameter (mm)	Flow rate (m^3/sec)	Friction coefficient of pipe	Pressure loss (m)	House power of drainage pump (HP)
Belt incline	175	600	105	0.0133	0.0298	202.95	50
Main entry	200	3,000	105	0.0133	0.0298	386.88	90
Cross main entry	25	1,500	105	0.0133	0.0298		
Coal getting panel	25	700	81	0.0067	0.0323	54.74	10 x 2

CHAPTER 6. COAL PREPARATION PLAN

CHAPTER 6. COAL PREPARATION PLAN

Sizing and washability tests and analysis of each size range of the raw coal are required for selecting the optimum preparation method. However, the coal preparation method is determined in this plan by referring to that being used in the Mpaka mine because the above data is not available for this study.

6.1 Preparation Capacity

Capacity of the coal preparation facilities per hour is as follows:

Annual working days:		240 days
Number of shifts:		
	8:00 to 16:00 16:00 to 24:00	2-shifts operation
	24:00 to 8:00	1-shift standby & maintenance
Operating hours:		7 hours/shift
Annual feed of coal:		640,000 tons
Clean coal production:		510,000 tons/year (yield: 80%)

1) Feed per hour

Since the quantity of raw coal transported from underground is not constant, it is necessary to store the coal in a raw coal storage bin after primary crushing and hand-picking so that a constant quantity of the coal can be fed to the preparation plant. Scattering of the quantity of raw coal from underground is assumed to be 30%.

Quantity of raw coal fed to primary crusher:

$$\frac{2{,}670 \text{ t/day} \times 1.3}{12 \text{ hours/day}} = 289 \text{ t/hr} \fallingdotseq 300 \text{ t/hr}$$

The quantity of raw coal taken out from the raw coal bin is as follows:

Capacity of preparation plant:

$$\frac{2{,}670 \text{ t/day}}{14 \text{ hours/day}} = 190 \text{ t/hr} \fallingdotseq 200 \text{ t/hr}$$

6.2 Coal Preparation Process

Coal preparation flow sheet is shown in Figures III-32 and 33.

The raw coal is transported from underground, screened and large refuse is picked out at the picking band and stored in the raw coal storage bin after primary crushing. The coal, sized at 22 to 75 mm, is separated into clean coal and refuse by a heavy medium vessel. The coal of –22 mm is sent to clean coal storage bin since its ash content is low as shown in the material balance of Figure III-34. Clean coal is stored in the clean coal bin by its size range.

The sink-and-float tests of the Main Seam show that an average yield of the clean coal at the specific gravity of 1.6 is 78.0% as described in the Paragraph 4.1 in the Part II. As the size fraction

Figure III-32 CONCEPTUAL DIAGRAM

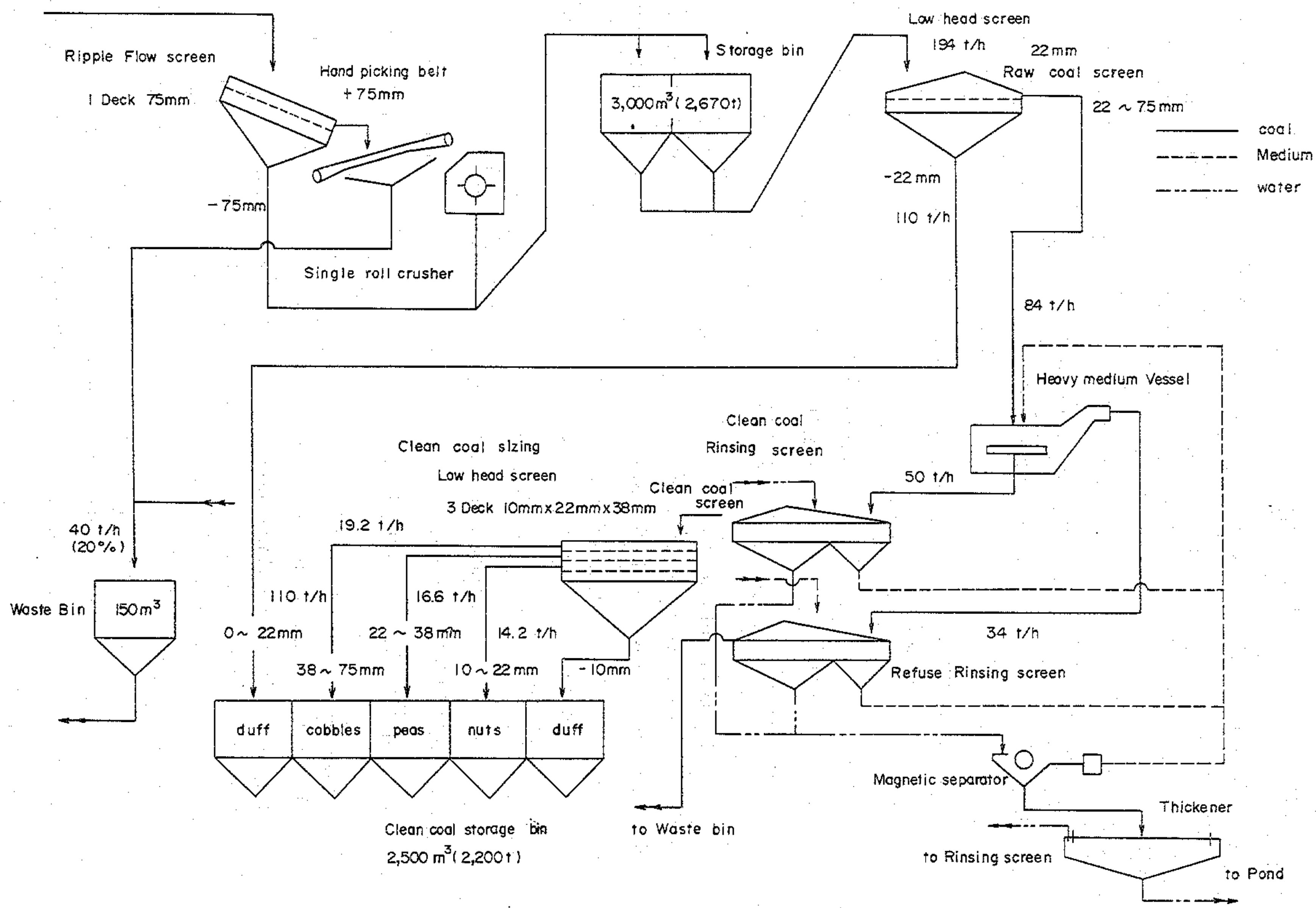


Figure III-33 Flow Sheet of Coal Preparation

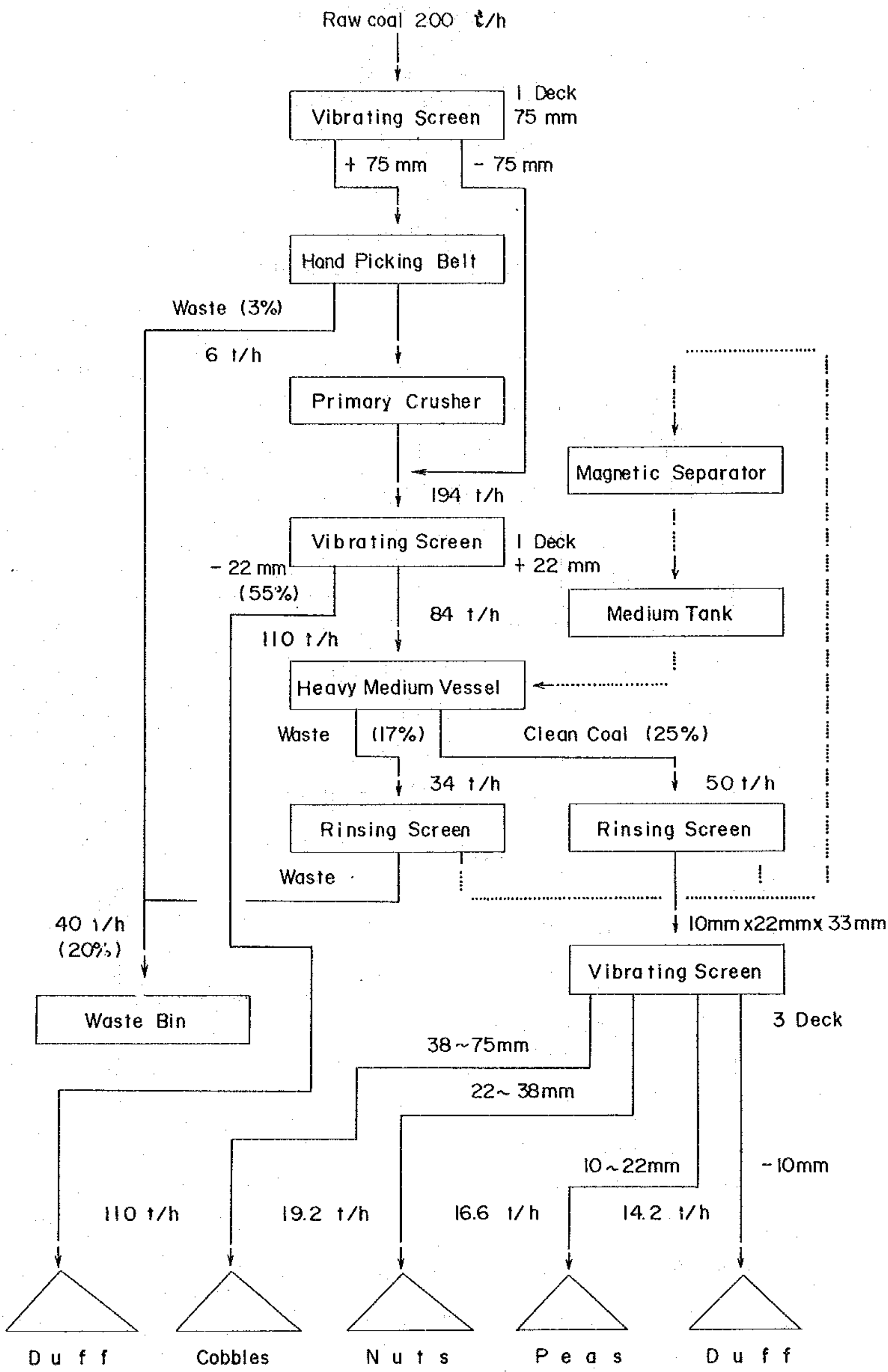
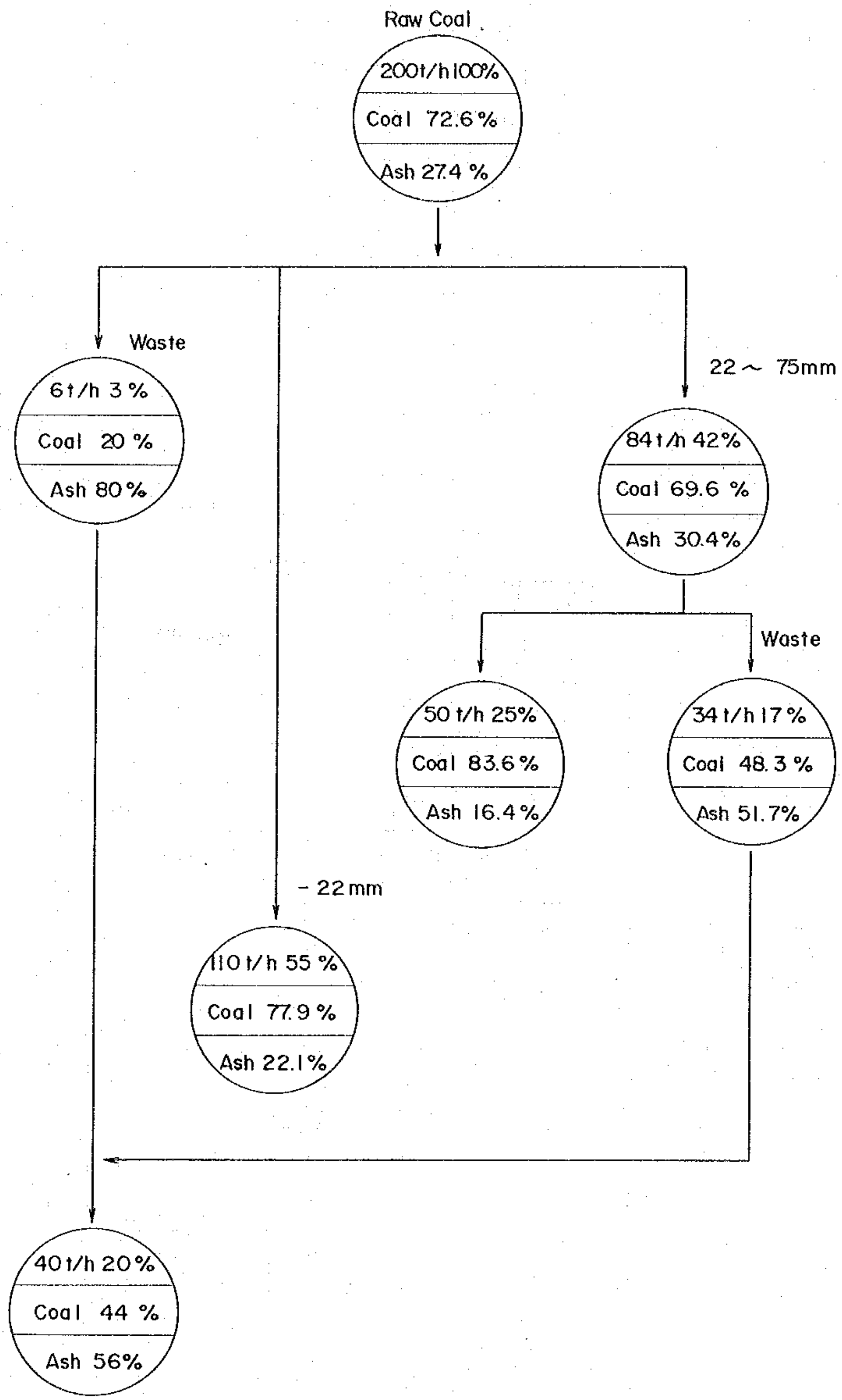


Figure III-34 Material Balance of Coal



of –22 mm, which is 55% of the feed in weight, is directly sent to the clean coal storage bin after screening in the actual operation, the remaining 45% of the raw coal are fed to the heavy medium separation. Consequently, the total yield of the clean coal is planned to be 80% in this study.

6.3 Main Facilities

6.3.1 Primary Crushing Facilities

As the coal is produced by a continuous miner and crushed by a breaker, maximum size of raw coal is expected to be 250 mm. The raw coal is fed to the ripple flow screen with an aperture of 75 mm by the connected belt conveyor from the surface pocket at the pit mouth. This screen is designed to separate the coal into +75 mm and –75 mm. +75 mm coal is conveyed by the picking band which travels at 15 m/min. Hand pickers stand on both sides of the picking band and pick out visual refuse.

Then +75 mm raw coal is reduced to –75 mm by a primary crusher. The type of the crusher to be employed is a single roll crusher which is popularly used in primary crushing for coal.

The raw coal sized at –75 mm from the ripple flow screen and primary crusher is carried to the raw coal storage bin by belt conveyor. The storage bin, made of steel plate, has enough capacity to meet one day's raw coal production. The inside of the bin is divided into three rooms which have a 250 t/hr Syntron feeder at each bottom.

6.3.2 Coal Preparation Facilities

The raw coal is transported by belt conveyor at a rate of 194 t/hr from the raw coal storage bin to the single deck low head screen with 22 mm mesh through the Syntron feeder.

–22 mm coal is sent directly to the clean coal storage bin and 22 ~ 75 mm coal is fed to heavy medium vessel. The vessel separates the coal into clean coal and refuse. The type of the vessel is planned to be simple to produce a consistent range in quality of clean coal. The clean coal then passes over a drain and rinse screen for washing heavy liquid medium. The refuse is handled by the same process as the above-mentioned.

Heavy liquid medium is supplied by slurry pumps from its storage tank to the vessel. A magnetic separator is employed to recover magnetite in the heavy liquid medium after washing it out at the drain and rinse screens.

The clean coal is then carried to a three deck screen to separate into 38 to 75 mm (cobbles), 22 to 38 mm (peas), 10 to 22 mm (nuts) and –10 mm (duff). The screen employs a low head type which can be maintained easily. The above four products are carried by belt conveyors to each clean coal storage bin. The storage bins are made of steel plate and have a capacity to meet one day's clean coal production. The inside of each storage bin is divided into two rooms which have a 250 t/hr Syntron feeder at each bottom.

The refuse is transported by belt conveyor from the drain and rinse screen to the refuse bin, which is made of steel plate and designed to store the refuse for one shift. The refuse is transported by dump truck from the bin to a refuse dumping yard.

6.3.3 Other Facilities

1) Laboratory

In order to control the quality of the clean coal, required equipment for analysis is prepared in a laboratory. At the least, proximate analysis and size distribution tests are carried out in the laboratory.

2) Loading facilities

The clean coal is planned to be transported by both trucks and railway from the mine site to consumers. A railway spur line is constructed in accordance with the standard of the Swaziland railway.

Rails are designed to have zero gradient and be straight around the loading facilities to make the loading easier. Wagons currently being used have a capacity of 41 tons and 33 tons. Maximum train length with one locomotive and 18 wagons is 230 m (23.16 m of locomotive + 11.28 m x 18). Therefore, a straight railway is constructed for about 230 m before and behind a loading bin. Total length of the railway spur line is 3,000 m.

The loading bin is made of steel plate and has a storage capacity of 1,600 tons, which is sufficient to load 1,476 tons of coal for two trains (41 tons x 18 wagons x 2 trains). Since coal is loaded to the wagon being connected to the locomotive, wagon-driving equipment is not used.

3) Refuse dumping yard

Refuse dumping yard of 800 x 500 m is prepared near the coal preparation facilities. The refuse is handled by bulldozer in the dumping yard. This work is done by a contractor.

6.3.4 Specifications of Preparation Facilities

Description	Specification
Connected belt conveyor	Width: 900 mm, Length: 30 m, Capacity: 400 t/hr, Motor: 30 KW
Ripple flow screen	Width: 1.5 m, Length: 3.6 m, Single deck, 75 mm aperture, Punch plate Capacity: 400 t/hr, Motor 11 KW
Picking band	Width: 1,200 mm, Length: 14.5 m, Capacity: 100 t/hr, Motor: 2.2 KW, Speed: 15 m/min.
Primary crusher	Roll diameter: 36" (915 mm), Width: 48" (1,219 mm), Open set: 75 mm, Motor: 45 KW, Capacity: 325 t/hr
Belt conveyor (to raw coal storage bin)	Width: 900 mm, Length: 140 m, Lift: 24 m, Motor: 40 KW, Capacity: 400 t/hr
Raw coal storage bin	Capacity: 2,670 t

Syntron feeder	Motor: 2.2 KW, 11 sets
Belt conveyor (to single deck low head screen)	Width: 750 mm, Length: 85 m, Lift: 15 m, Motor: 22 KW, Capacity: 300 t/hr
Single deck low head screen	Width: 2.4 m, Length: 6.0 m, Mesh: 22 mm, Single deck, Capacity: 200 t/hr, Motor: 30 KW
Heavy medium vessel	Diameter: 4.5 m, Length: 7.5 m, Type: Wemco, Motor: 15 KW
Slurry pump	3 sets (1 for standby), Lift: 30 m, Capacity: 3 m^3/min., Motor: 30 KW
Drain and rinse screen	Width: 1.8 m, Length: 5.4 m, Mesh: 0.5 mm, Motor: 15 KW, 2 units
Belt conveyor (to 3 deck low head screen)	Width: 600 mm, Length: 11 m, Capacity: 50 t/hr, Motor: 2.2 KW
3 deck low head screen	Width: 1.8 m, Length: 3.6 m, Mesh: 38 mm, 22 mm, 10 mm, 3 decks, Capacity: 50 t/hr, Motor: 11 KW
Clean coal belt conveyors (to clean coal storage bins)	Width: 600 mm, Total Length: 300 m, Total power of motors: 70 KW
Refuse belt conveyor (to refuse storage bin)	Width: 600 mm, Length: 100 m, Motor: 20 KW
Clean coal storage bin	Capacity: 2,200 tons
Refuse storage bin	Capacity: 150 m^3
Railway spur line	Maximum gradient: 1 to 120, Minimum radius: 600 m, Rail weight: 40 Kg/m, Gauge: 1,435 mm

CHAPTER 7. SURFACE FACILITIES

CHAPTER 7. SURFACE FACILITIES

7.1 Water Supply Plan

Average annual rainfall is about 1,000 mm in Swaziland, but is 500 to 900 mm in the investigated area. However, required water quantity for development of the mine is expected to be completely secured. There are dry and rainy seasons, and the rainy season is between November and March. Streams running through the area flow usually in the rainy season but most of them are dried up in the dry season. There are small water reservoirs near the mine site but these cannot be used to supply the mine since the water is used for agriculture and livestock. Therefore, groundwater or water from a large river must be used for the mine.

Two water supply sources can be considered for the mine; to supply water from the Great Usutu River, about 20 km south of the mine site, which has enough flow even in the dry season; to dig several wells and to pump up groundwater with deep-well or piston pumps in the initial stage, and mine water can be purified and used if required when the mining area expands to some extent.

In this plan, water wells are used the same as the Mpaka mine.

7.1.1 Water for Mining Operation

Water for mining operation is mainly used for coal preparation, cooling of continuous miners and spraying for belt conveyor and dust control.

Water for cooling and spraying is supplied through a water purifier in order to prevent clogging of nozzles and pipes.

(1) Coal preparation

Water for coal preparation is used in a closed cycle. Water loss caused by adhering to clean coal and refuse is additionally supplied. Adherent moisture content of clean coal and refuse from the preparation plant is supposed to be 15%, and about 3% of water is assumed to be lost in the process of the heavy medium separation.

Raw coal of 1,120 tons is handled in the heavy medium separation per day, and water adhering to the clean coal and refuse is calculated at 198 m^3/day. Water loss in the heavy medium spearation is estimated at 135 m^3/day as the water is used at the rate of 450 m^3/h (10 hrs/day actual operation). Therefore, supply of water for the heavy medium separation is calculated at 333 m^3/day.

(2) Continuous miner

One continuous miner consumes 50 litres/min. (3 m^3/hr) of water, and water for 3 miners for the operation of 2 hours/shift is calculated at 72 m^3/day.

(3) Spray for belt conveyor

Water is sprayed at the transferring place of the belt conveyor. Volume of water to be sprayed is equivalent to about 2% of transported raw coal. Required water is calculated at 53 m^3/day.

(4) Others

Required water for other uses is expected to be 10% of total water stated above, which is 46 m^3/day.

Total water to be supplied is calculated at 504 m^3/day. Mine water of 0.35 m^3/min. is required in case the above water is supplied from underground.

7.1.2 Potable Water

Potable water is pumped up from deep wells and used for shower bath in the changing house, offices and houses.

According to the record of the Mpaka mine, the well water contains a very small amount of impurities and is made potable by chlorination.

(1) Shower water

Shower water for the changing house is 100 litres per person and thus 19 m^3/day is necessary for 183 underground workers.

(2) Offices

Water for offices is 40 litres per person, and 5 m^3/day is necessary for 115 workers.

(3) Houses

Required water is 0.5 m^3/day per house. 60% of 288 employees excluding managers are supposed to be singles and every four of these share one house.

The number of houses is 10 for managers, 115 for employees with their families and 44 for single. The following quantity of water is required for these houses.

$$0.5\ m^3/day \times 169\ houses \fallingdotseq 85\ m^3/day$$

(4) Others

Required water for other uses is expected to be 10% of the total water stated above, which is 11 m^3/day.

Total potable water is calculated at 120 m^3/day (0.083 m^3/min). Considering the present situation of the Mpaka mine, 6 water wells and water tanks with the total capacity of 60 m^3 are required.

Lifting pump:	Six sets (2.2 KW motor)
Delivery pump:	Two sets (5.5 KW motor)

7.2 Electric Power Supply

According to the statistics of Swaziland, the annual total power consumption is 600 million KWH and the power consumption per capita is 980 KWH. The recent capacity of power generation facilities (Swaziland Electricity Board) is about 40% of the total consumption, and the remaining is received from South Africa. The power transmission lines are well arranged.

A transmission line for this study is planned to be branched from the Mbabane-Manzini-Siteki 66 KV high voltage trunk line. Transmitting capacity from South Africa is 97 MW, and the maximum demand is 80 MW. The allowance is enough to supply required power to the projected coal mine.

The only problem is an unexpected power failure due to occurrences of thunderbolts. From a standpoint of mine safety, a standby power generator will be required to operate ventilation fans constantly in case of a power failure caused by thunderbolts. However, such a generator is not planned to be installed in this study (the same as the Mpaka mine).

(1) Branch point of power line

As shown in Figure III-35, the power line is planned to be branched from the transmission line between Manzini and Siteki (66 KV, 50 Hz, 3 phases) at a point near the Malindza Clinic. 66 KV power line will be installed for 9 km to the mine site.

(2) Power receiving capacity

Power consumption in this plan is calculated as follows; (based on the total capacity of electric facilities shown in Figure III-36 and the rate of operation times load factor equals 0.5.)

$$\text{Total capacity} \times (\text{operation rate} \times \text{load factor}) = 5{,}740 \text{ KW} \times 0.5 = 2{,}870 \text{ KW} \fallingdotseq 2{,}900 \text{ KW}$$

where, total capacity is shown in Table III-11. And 20% of the above is used on a non-working day. Maximum power demand is calculated on the basis of 30% fluctuation, which is:

$$2{,}900 \text{ KW} \times 1.3 = 3{,}770 \text{ KW} \fallingdotseq 3{,}800 \text{ KW}$$

Therefore, required capacity is 5,000 KVA.

(3) Power distribution

Power distribution is for the surface and underground. Power is distributed through 11,000 V overhead lines to the surface facilities such as the winding machine, main fans, coal preparation facilities, work-shop, coal handling facilities. A transformer is installed for each facility, and the power is stepped down to 3,000 V for a motor of over 100 KW and to 380 V for a motor of less than 100 KW.

Power is distributed through two lines of 11,000 V armoured cable to underground; one line is for mining machines and the other for belt conveyors. A 800 KVA power pack is installed at the working place for the continuous miner, shuttle car, breaker, ratio feeder and air compressor.

7.3 Surface Facilities

7.3.1 Roads

(1) Ground leveling

The total area of 375,000 m^2 for surface facilities is leveled prior to initial construction.

(2) Access road

A new access road of a Class 3 district road is planned to be constructed from the mine site running to the west for 1.4 km to connect to an existing road, which runs parallel to the railway and joins the Manzini-Siteki national road near Mfelafutsi.

In addition, the existing road is graded up to a Class 3 district road for 12 km. The standard of a Class 3 district road is:

Design speed:	80 km/hr

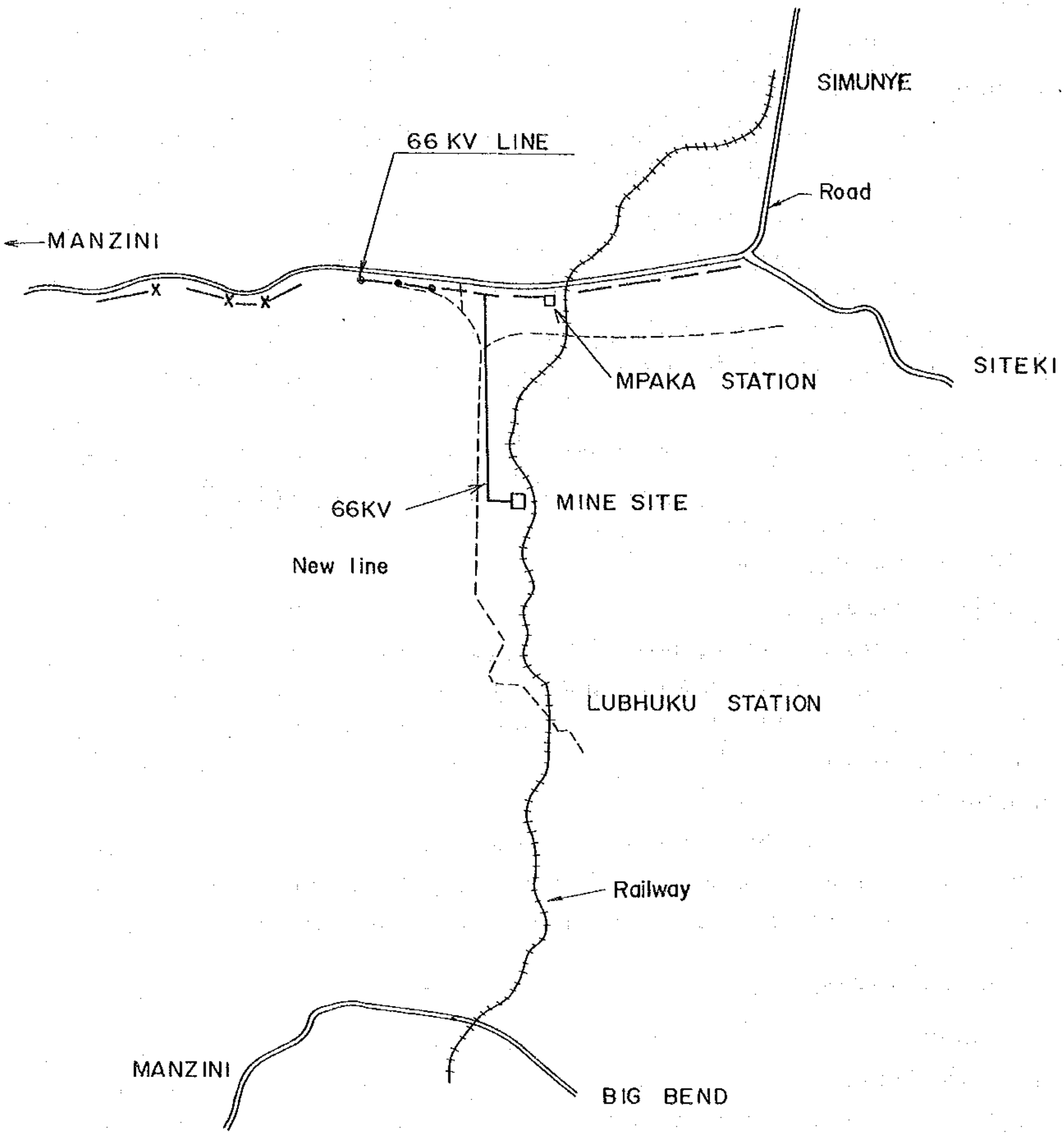


Figure Ⅲ-35 Power Line Map

Figure III-36 Schematic Layout of Power Supply

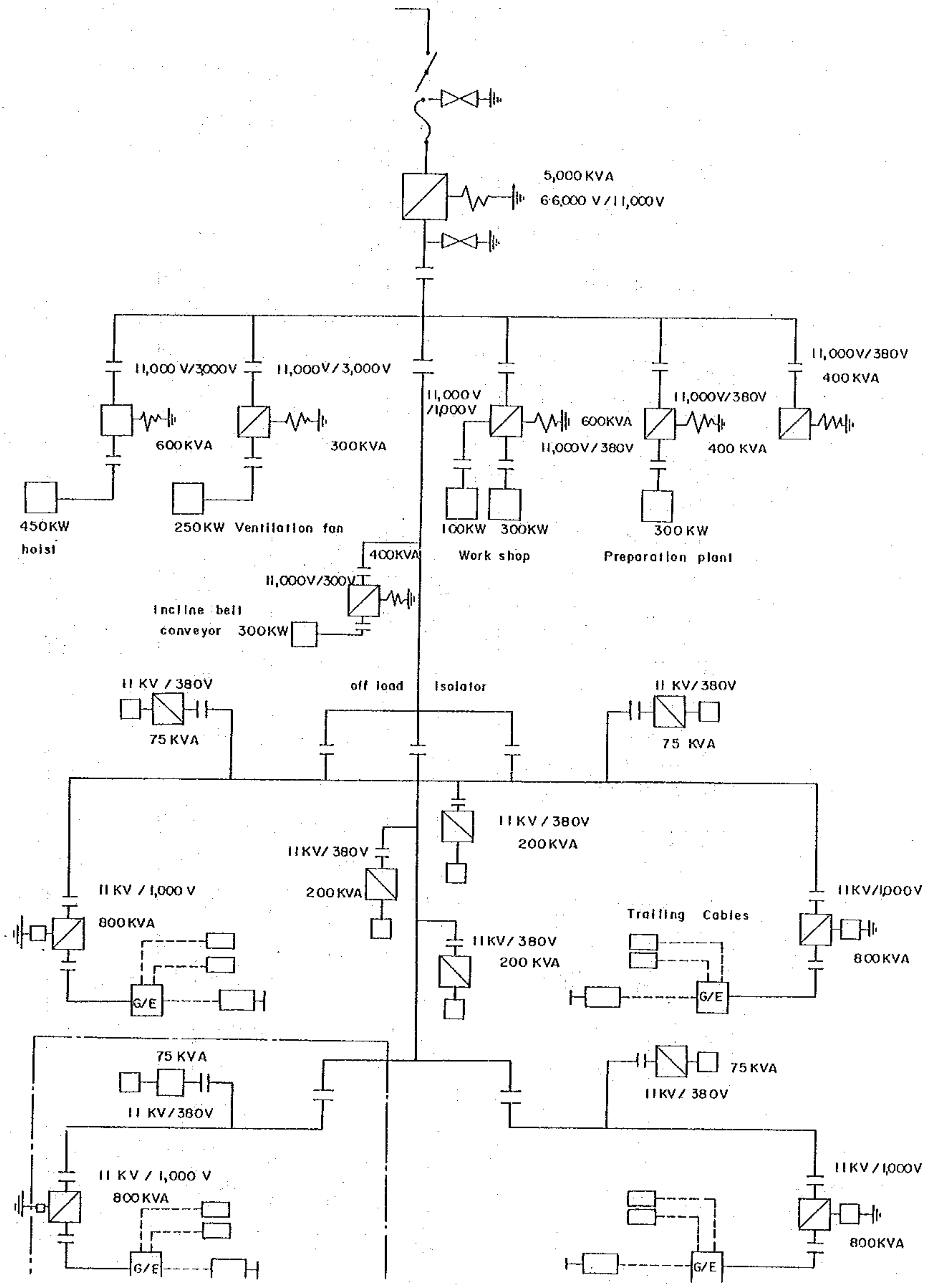


Table III-11 Electric Equipment

	Item	Unit Capacity (KW)	Number	Capacity (KW)	Remark
Office	Office		1	30	
	Waiting room		1	20	
	Storage		1	30	
	Maintenance works			300	
	Others		1	20	
	Sub total			400	
Surface Electric Equipment	Ventilation fan		1	200	
	Winding		1	250	
	Pump equipment		1	30	
	Sub total			480	
Washing Plant	Primary crusher		1	150	
	Washing plant		1	150	
	Clean coal bin		1	50	
	Waste bin		1	20	
	Forkend loader		1	20	
	Others		1	110	
	Sub total			500	
Coal Getting Machineries	Continuous miner	320	4	1,280	
	Shuttle car	45	8	360	
	Breaker	15	4	60	
	Ratio feeder	20	4	80	
	Compressor station	75	4	300	
	Others		1	100	
	Sub total			2,180	
Raw Coal Haulage	Incline B.C		1	300	
	Main entry B.C	150	3	450	
	Cross mainentry B.C	130	3	390	
	Panel B.C	30	4	120	
	Bunker		1	100	
	Sub total			1,360	
Underground Drainage Equipment	Incline main pump	50	2	100	
	Main pump	75	2	150	
	Local pump	10	5	50	
	Sub total			300	
Surface Waterworks	Water pump	30	2	60	
	Filtration equipment	10	2	20	
	Drainage equipment	10	2	20	
	Water spray, fire fighting pump	10	2	20	
	Sub total			120	
Houses & Common Facility	Houses		1	300	
	Common facility		1	50	
	Others			50	
	Sub total			400	
Total				5,740	

Formation width:	9.7 m
Running surface:	6.4 m
Maximum gradient:	6%
Minimum horizontal radius:	150 m

(3) Service road

A service road connecting surface facilities is planned to be paved to control dust. Standard of the road is the same as that of the access road, and a running surface of 6.4 m width is paved.

(4) Parking lot

Parking lot with roof: 10 x 30 m, for managers, paved with asphalt.

Open parking lot: 45 x 110 m, for workers, paved with crushed stone.

7.3.2 **Buildings**

(1) Mine office

The mine office is of one-story brick construction with a size of 15 x 45 m and has the managers' offices, meeting rooms and offices for general affairs and clerical divisions.

(2) Clinic

A clinic for first aid is of one-story brick construction with a size of 10 x 15 m.

(3) Service facilities

The service facilities building is of one-story brick construction with a size of 25 x 50 m.

a. Changing house

200 sets of lockers, 30 sets of showers, 5 sets of lavatories and 10 sets of washers are installed in the changing house.

b. Lamp room

200 sets of safety lamp battery chargers and shelves are provided in the lamp room.

c. Rescue apparatus

Oxygen masks for 5 workers are provided for emergency rescue operation.

d. Boiler

A boiler of 300,000 Kcal/h capacity and storage tank of 4 m^3 are installed for the shower.

(4) Work-shop

A work-shop is installed for daily maintenance of equipments. Required tools for general maintenance work are prepared.

Heavy machinery such as continuous miners is served by a contractor. The shop is a size of 15 x 50 m and an overhead travelling crane is to be installed.

(5) Explosives magazine

An explosives magazine with a floor size of 5 x 6 m meeting explosive handling standards is built in a place surrounded by a protection wall at a considerable distance from the surface facilities in order to minimize damage in the event of accident.

(6) Personnel carrier shed

Eight personnel carriers including two standby carriers are provided. The shed for carriers has a 5 x 20 m concrete floor.

(7) Sewage plant

The waste water from showers and toilets is treated through activated sludge process in the sewage plant.

(8) Materials warehouse and machine storage yard

A materials warehouse with a management room is built with a size of 15 x 25 m. Stock control with a personal computer is required.

A machine yard is provided adjacent to the warehouse. The machine yard has the size of 60 x 100 m and is surrounded by 2 m high wire fences. A gasoline tank (20 m^3) for cars and a light oil tank (100 m^3) are installed in the same yard. These tanks have fireproof construction.

(9) Housing

a. Manager's houses: 10 houses, house for the general manager has 150 m^2 floor.

b. Houses for married persons: 115 houses with 75 m^2 floor.

c. Houses for singles: 44 houses with 50 m^2 floor. One house is shared by four persons.

d. Common bathroom houses: 4 houses with 4 shower rooms and lavatories. The floor area is 80 m^2.

e. Common cookhouse: The floor area is 300 m^2.

f. Lunch room: 4 rooms with 70 m^2 floor.

(10) Welfare facilities

a. Assembly hall

b. Soccer fields

c. Grocery

d. Others

7.4 Other Facilities

(1) Compressed Air Plan

Compressed air is normally used as a power source in the coal mines where a large amount of gases is ejected. However, electric power is planned to be used for the underground facilities and equipment excluding a roof bolter in this study since the amount of gases ejected from anthracite is little in the projected mine. Compressed air is more advantageous only for roof bolters so that a small air compressor is planned to be installed near the power pack at each working face. The compressor discharges compressed air to the roof bolters through soft vinyl chloride pipes and rubber hoses 80 mm in diameter.

Specifications of compressor.

Pressure:	7.5 kg/cm^2
Discharge:	10 m^3/min.

Motor output: 75 KW

Cooling method: Air cooling with circulating water

Specifications of vinyl chloride pipe:

80 mm diameter, 200 m length, 10 kg/cm^2 rated pressure

Specifications of rubber hoses:

80 mm diameter, 50 m length, 10 kg/cm^2 rated pressure

(2) Communications Equipment

Thirty telephone sets are installed for the offices and workshops and ten telephone sets for underground. Automatic telephone exchanges are employed. 5 public lines are connected to the offices.

(3) Service Motor Vehicles

The following vehicles are necessary to assist the surface work and to transport the materials which are required immediately:

Passenger cars:	2
Wagon type vehicles:	3
Small trucks.	2
Wheel type crane.	1

Layout of the main surface facilities is shown in Drawing 17.

CHAPTER 8. PERSONNEL PLAN

CHAPTER 8. PERSONNEL PLAN

Manpower requirements at the full scale operation are made by taking into account the work, operating conditions and present condition of the Mpaka mine. The manpower gradually increases depending on the phases of the mine development. Three senior managers, business manager, chief engineer and colliery manager, are assigned under the general manager and five chiefs manage the departments concerned at the full scale operation.

Personnel plan of each department is outlined below. Manpower requirements for the working face are shown in detail in Paragraph 3.9.

Senior manager:	4 (including general manager)
Administrative Department:	Manager 1, staff 3, labourer 8
Accounts Department:	Manager 1, staff 2, labourer 4
Planning Department:	Manager 1, staff 10, labourer 14
Equipment Department:	Manager 1, staff 12, labourer 44
Coal Mining Department:	Manager 1, staff 34, labourer 149

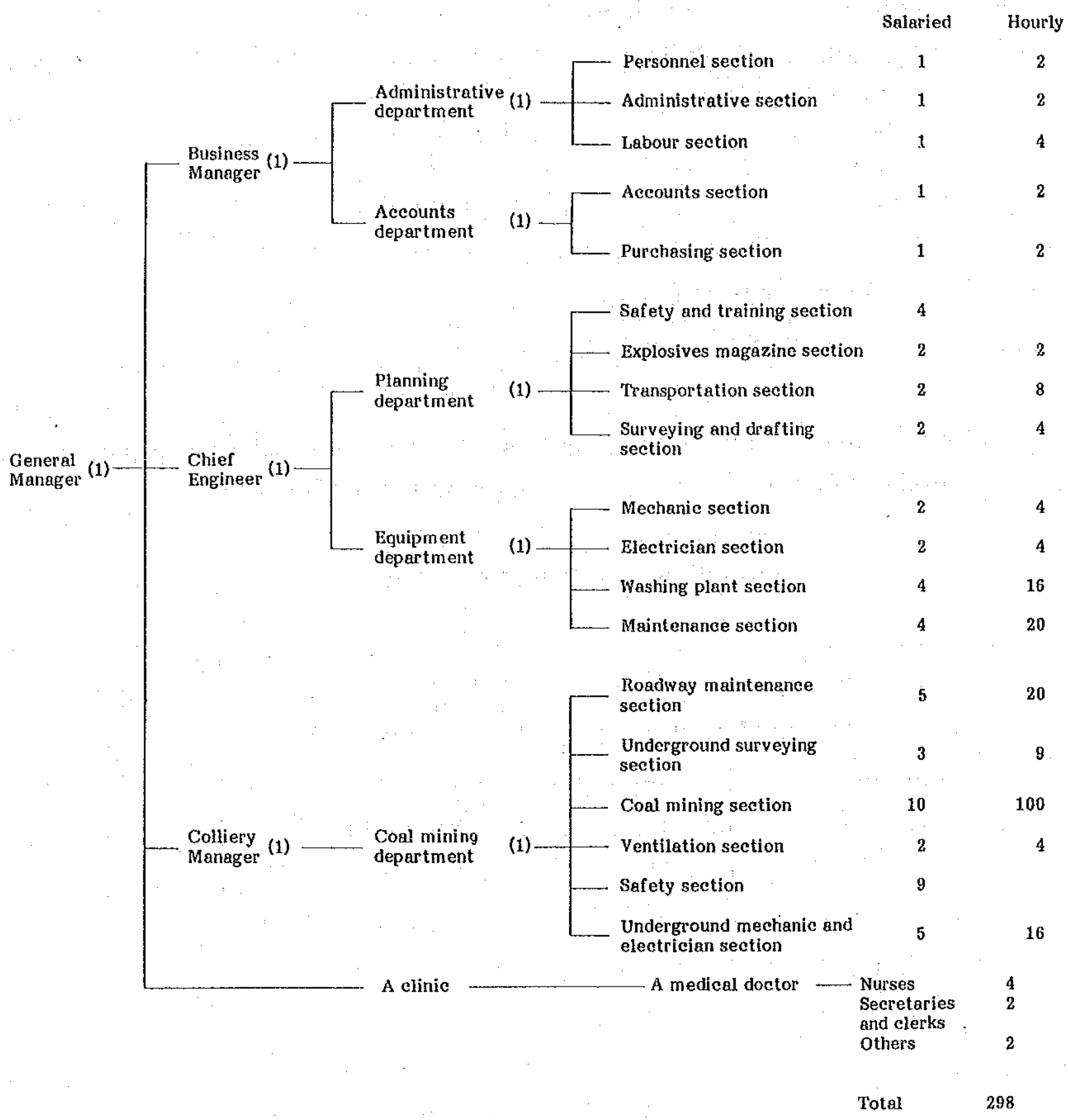
In addition to the above, one medical doctor, four nurses and four clerks and others are stationed at the clinic.

Total manpower requirements of the mine are 298 persons. The manpower requirements are summarized in Table III-12. Figure III-37 shows organization chart of the mine.

Table III-12 Manpower Requirements

	Surface workers	Underground workers	Tatal
Salaried	41	34	75
Hourly	74	149	223
Total	115	183	298

Figure III-37 Mine Organization Chart



CHAPTER 9. INITIAL INVESTMENT AND PRODUCTION COST

CHAPTER 9. INITIAL INVESTMENT AND PRODUCTION COST

9.1 Initial Investment

An initial investment plan for six years is made based on the following factors:

(1) The price estimate is based on the prices of the year 1985.

(2) Interest and inflation are not estimated.

(3) Estimated prices of the facilities, equipment and materials are purchase prices in Japan. Import tax is not taken into account.

(4) Labour costs are based on the wages in Swaziland.

9.1.1 Construction Schedule

A construction schedule of facilities required for the development of the planned area is shown in Table III-13. Design work is performed in the first year, and the procurement of required equipment and materials for facilities and installation works are completed by the 5th year. Only maintenance cost is taken into consideration from the 6th year.

9.1.2 Initial Investment

Initial investment includes civil engineering works, surface facilities and underground facilities. An investment plan by fiscal year is shown in Table III-14. The plan is based on the full production of 510,000 tons/year (clean coal) achieved within three years after production is commenced.

The total initial investment amounts to US Dollars 26.9 million.

9.2 Production Cost

The full scale production cost of clean coal at mine site is estimated based on the following factors:

(1) Labour cost

Labour cost is calculated as follows:

(Average annual wages) x (The number of workers)

Average wages of the Mpaka mine are:

Staff:	25,000 US$/year
Worker:	3,120 US$/year

(2) Material costs, royalty and taxes

Material costs, royalty and taxes are estimated by taking into account those of the Mpaka mine.

Surface material cost:	0.936 US$/ton
Underground material cost:	1.600 US$/ton
Royalty and taxes:	0.052 US$/ton

(3) Electric power cost

Based on the power distribution plan stated in Paragraph 7.2, the power cost is calculated from the present tariff of Swaziland as shown below.

Table III-13 CONSTRUCTION AND INSTALLATION SCHEDULE

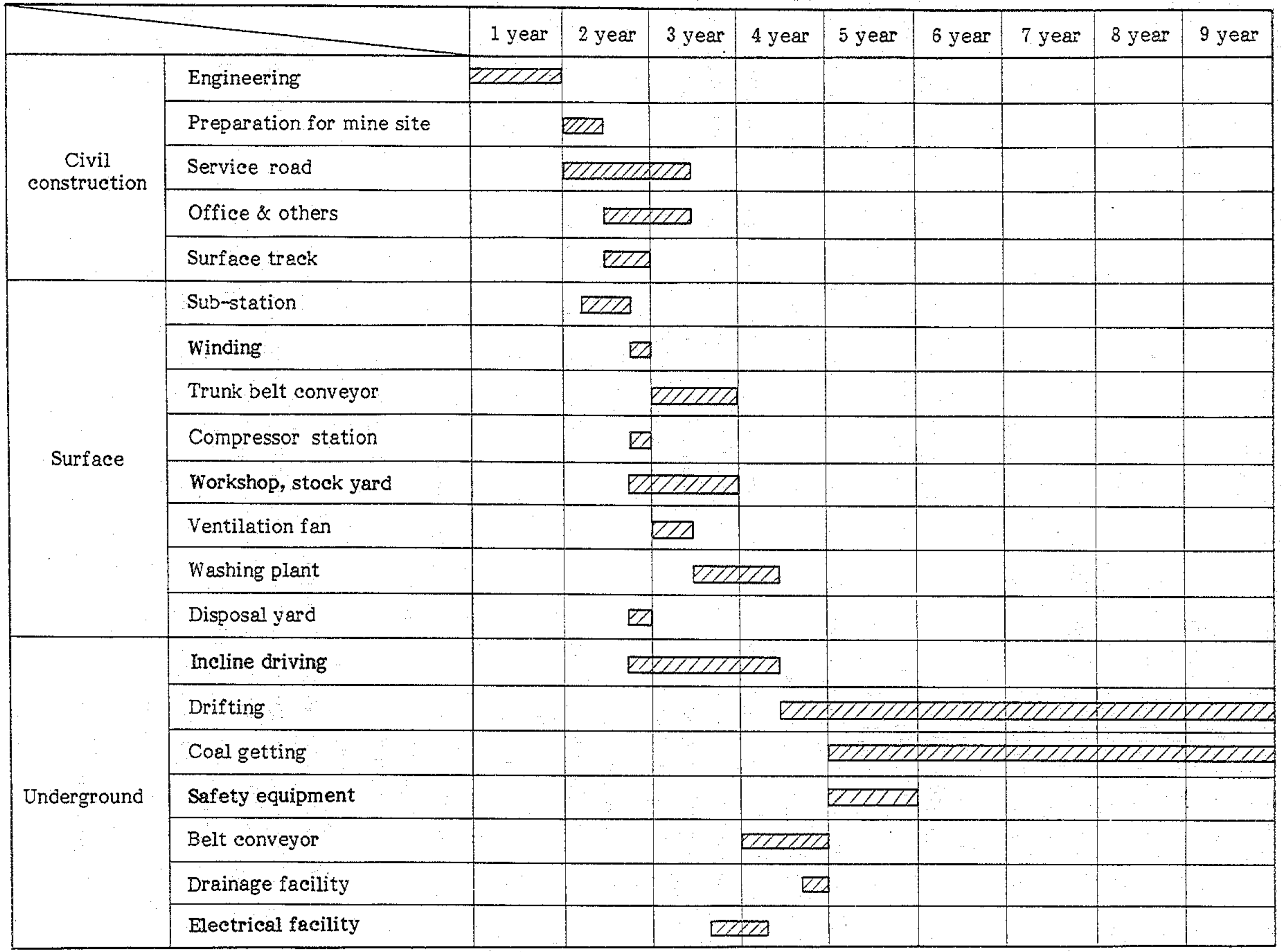


Table III-14 Initial Investment and Depreciation Schedule

(unit; US$1,000.00)

Item		Durable Year	1 year	2 year	3 year	4 year	5 year	6 year	Remark
Operation cost in first 3 years		5	(2,586)			517	517	517	
Engineering		5	(650)			130	130	130	
Civil Construction	Preparation for mine site	20		(225)		11	11	11	
	Service road	20		(216)	(108)	16	16	16	
	Office & others	20		(378)	(378)	38	38	38	
	Surface track	20		(114)		6	6	6	
Surface	Sub-station	20		(891)		45	45	45	
	Incline winding	20		(360)		18	18	18	
	Trunk belt conveyor	10			(405)	41	41	41	
	Compressor station	20		(279)		14	14	14	
	Work shop, stock yard	20		(297)	(594)	45	45	45	
	Ventilation fan	10			(207)	21	21	21	
	Washing plant	20			(3,240)	(3,240) 162	324	324	
	Disposal yard	7		(1,170)		167	167	167	
Underground	Incline driving	20		(587)	(950)	(475) 77	101	101	
	Continuous miner and face equipment	7				(1,450)	207 (5,100)	207 729	
	Safety equipment	7					(913)	130	
	Belt conveyor	10				(354)	35	35	
	Drainage facility	7				(1,300)	189	189	
	Electrical facility	20			(200)	(235) 10	22	22	
	Total		(3,236) 0	(4,517) 0	(6,082) 0	(7,054) 1,318	(6,013) 1,947	2,806	

( ): investment amount

Demand:	4.50 US$/month-KW
Consumption:	0.0164 US$/KWH

(4) Spare parts

A replacement plan is prepared by the assumption that spare parts are required every year from the following year of facilities introduction. Spare parts costs are calculated from repair factors representing a percentage of purchase prices.

Results are shown in Table III-15.

(5) Depreciation

Depreciation schedule is made based on the following factors:

1) All assets are depreciated from the following year of acquisition of assets. The assets acquired before the start of operation are depreciated from the year of production commencement.
2) Fixed installment method is employed for depreciation.
3) Pre-production cost is depreciated 5 years after production commencement.

Results are shown in Table III-14 of Paragraph 9.1.

(6) Contingency

15% of the total cost of the above-mentioned items is provided as contingency.

Full scale production cost at mine site is calculated at approximately US$ 16.27 per ton of clean coal.

Cost is expected to be slightly reduced after the 9th year since the depreciation of pre-production is completed.

Production cost at mine site is shown in Table III-16.

Table III-15 Spare Parts Costs

(Unit: US$1,000)

Item		Repair factor (%)	Amount of investment	Annual amount of spare parts	1 year	2 year	3 year	4 year	5 year	6 year
Civil construction	Service road	2	216	4			4	4	4	4
			108	2				2	2	2
	Office and others	2	378	8			8	8	8	8
			378	8				8	8	8
	Surface track	5	114	6			6	6	6	6
Surface	Sub-station	1	891	9			9	9	9	9
	Incline hoist	2	360	7			7	7	7	7
	Trunk belt conveyor	2	405	8				8	8	8
	Compressor station	2	279	6			6	6	6	6
	Workshop, stock yard	2	297	6			6	6	6	6
		2	594	12				12	12	12
	Ventilation fan	2	207	4				4	4	4
	Washing plant	5	3,240	162				162	162	162
			3,240	162					162	162
	Disposal yard	0	1,170	-						
Underground	Continuous miner and	10	1,450	145					145	145
	face equipment		5,100	510						510
	Safety equipment	5	913	46						46
	Section belt conveyor	3	354	11					11	11
	Drainage facility	2	1,300	26					26	26
	Electrical facility	2	200	4				4	4	4
		2	235	5					5	5
	Total						46	246	595	1,151

Table III-16 Production Cost

(Unit: US$)

Item			1 year	2 year	3 year	4 year	5 year	6 year
Wage	Staff			250,000	250,000	250,000	250,000	250,000
	Worker	S/F		24,800	158,100	275,900	300,700	325,500
		U/G				213,900	390,600	567,300
	Sub total			274,800	408,100	739,800	941,300	1,142,800
Material	S/F		6,000	10,000	34,000	93,600	318,200	477,400
	U/G					161,200	548,100	822,100
	Sub total			10,000	34,000	254,800	866,300	1,299,500
Royalty and Tax						5,200	17,700	26,500
Power			6,600	47,300	122,900	154,400	527,600	788,200
Depreciation			0	0	0	1,318,000	1,947,000	2,806,000
Spare parts					46,000	246,000	595,000	1,151,000
Total			12,600	332,100	611,000	2,718,200	4,894,900	7,214,000
Contingency (15%)			1,900	49,800	91,700	407,700	734,200	1,082,100
Grand Total			14,500	381,700	702,700	3,125,900	5,629,100	8,296,100
Production cost per ton clean coal						31.26	16.56	16.27

CONCLUSION

CONCLUSION

The present study reveals that the Main Seam, which is the most promising coal seam in the Swaziland coalfield, in the northern part of the investigated area (north of the "A" fault) exists in shallow part and has large minable reserves per unit area and also is intruded by little dolerite as compared with that which occurs south of the fault. Consequently, the Main Seam in this northern part is selected for development of a new coal mine in this study. Seam thickness of the Main Seam in this part ranges from 3.0 to 5.0 metres and the minable reserves are calculated at approximately 35 million tons.

Annual production is planned to be at the scale of 510,000 tons of clean coal (640,000 tons of raw coal). Lead time for the development is set for five years. Production of coal is commenced in the 4th year and is in full scale in the 6th. The mine is developed by two inclined shafts and room and pillar mining method with continuous miner is employed in this plan by taking into account the modes of occurrence of the coal seam and dolerite intrusions. Total number of the personnel is about 300 and the productivity is expected to be about 180 tons/man/month (raw coal) as mechanized mining is employed.

As the present investigation is pre-feasibility study, this draft of the mine development plan is prepared mainly from technical point of view. However, initial investment for the development and production cost at mine site are roughly estimated at about 26.9 million US dollars and around US$16.00 per ton of clean coal respectively (based on the price in the fiscal year 1985, and interest is not included). This production cost is considered to be able to compete in the present world-wide coal market.

From this fact it is concluded that the development of the coal mine in the northern part of the Lubhuku area is a hopeful project to supply coal for the domestic market such as thermal power plant and for export.

It is preferable to carry out a feasibility study on the coal development in the area provided the concepts on the following points are clarified;

Policies on the national coal development.

Marketing of the coal products.

Reinforcement and/or establishment of the coal development organization.

REFERENCES

REFERENCES

PART I and II.

Adams, S.C. and Kullerud, G. (1984): Depositional patterns of Pennsylvanian sediments in Sullivan County, west-central Indiana, U.S.A. Coal Geology, Vol. 3, pp. 349-373.

American National Standard Institute (1970): American national standard.

Coal Unit (1978): A note on the coal resources of Swaziland. Geological Survey and Mines Department, Swaziland (GSMDS).

Davies, D.N. (1961): The sedimentation features of the Karroo System in Swaziland. GSMDS, Bulletin No. 1, pp. 14-22.

Fichtner Consulting Engineers, et al. (1982): Swaziland energy masterplan, Interim report, Abridged version.

Fichtner Consulting Engineers, et al. (1984): Swaziland energy masterplan, Final report, recommendations.

Hunter, D.R., editor (1962): The mineral resources of Swaziland. GSMDS, Bulletin No. 2.

Hunter, D.R. (1968): Geologic map (1:50,000) and explanatory notes, Sheet 2631 BC. GSMDS.

Hunter, D.R. (1968): Geologic map (1:50,000) and explanatory notes, Sheet 2631 DA. GSMDS.

Japan International Cooperation Agency (1982): Geological exploration report on the Lubhuku coalfield development project in the Kingdom of Swaziland. MPN, CR (3), 82-147.

Kent, L.E. (Compiled) (1980): Stratigraphy of South Africa, Part 1: Lithostratigraphy of the Republic of South Africa, South West Africa/Namibia and the Republics of Bophuthatswana, Transkei and Venda. Geological Survey, South Africa, Handbook 8.

McKeown, M.C. (1981): A note of the coal resources of Swaziland. GSMDS.

Reeves, D.R. (1976): Application of wireline logging techniques to coal exploration. Proceedings of the first international coal exploration symposium, pp. 112-128.

Reeves, D.R. (1978): Some improvements and developments in coal wireline logging techniques. Proceedings of the second international coal exploration symposium, pp. 468-486.

Scogings, J.L. and Lenz, C.J. (1961): The Karoo System of sediments of eastern Swaziland. GSMDS, Bulletin No. 1, pp. 84-91.

Sogabe, M., Sato, Y. and Fujii, K. (1981): World coal resources, 3.10 The Kingdom of Swaziland, I.S.U., pp. 384-395 (in Japanese).

Urie, J.G. and Hunter, D.R. (1963): The geology of the Stormberg volcanics. GSMDS, Bulletin No. 3, pp.28-44.

Urie, J.G. (1967): Geologic map (1:50,000) and explanatory notes, Sheet 2631 BD. GSMDS.

Urie, J.G. (1967): Geologic map (1:50,000) and explanatory notes, Sheet 2631 DB. GSMDS.

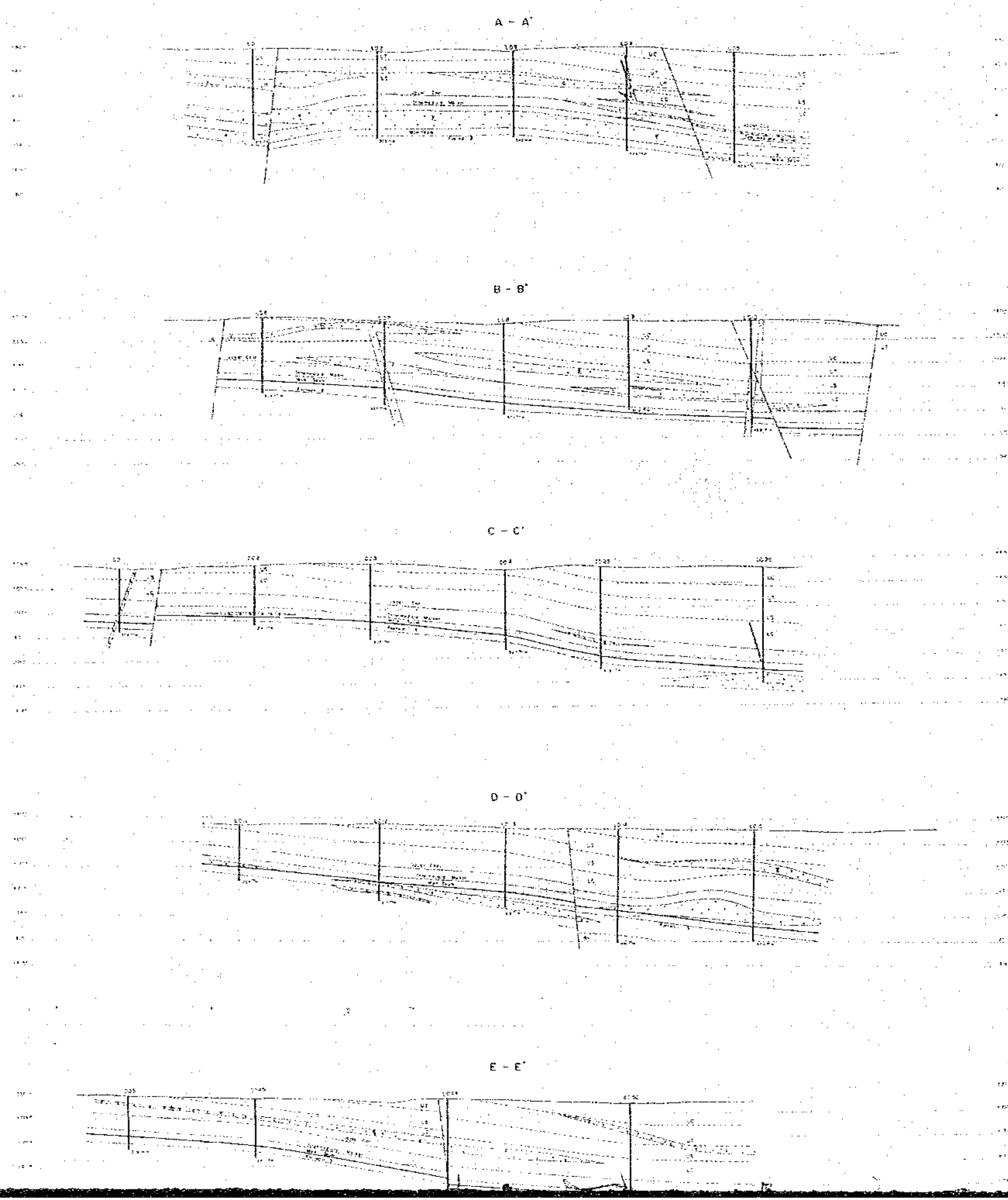
Visser, J.N.J., Loock, J.C. and Jordaan, M.J. (1980): Permian deltaic sedimentation in the western half of the Karroo Basin, South Africa. Karroo Basin symposium, Geological Society of South Africa, pp. 415-424.

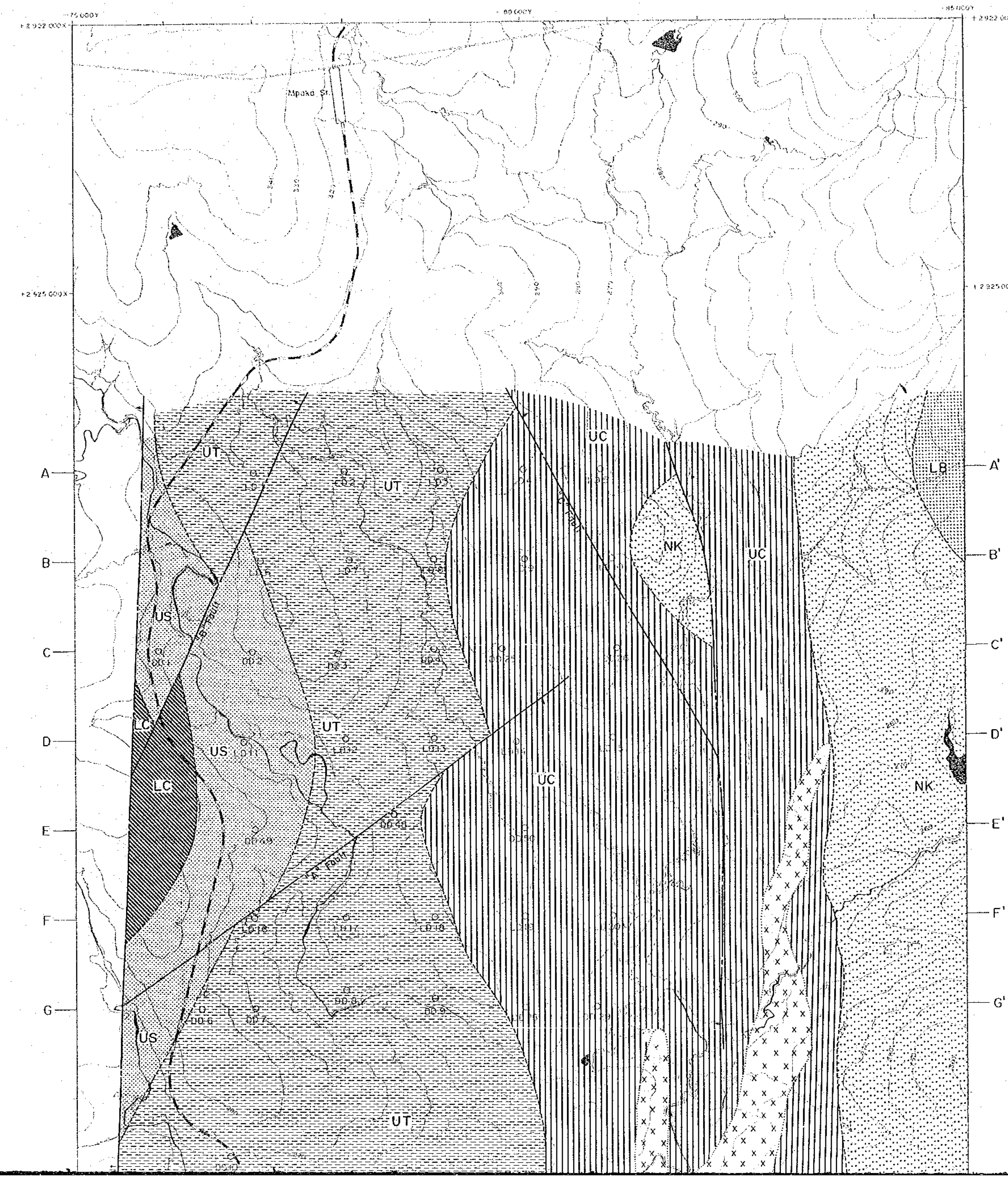
Way, H.J.R. (1961): Major structure in Swaziland. GSMDS, Bulletin No. 1, pp. 94-95.
Whateley, M.K.G. (1980): Deltaic and fluvial deposits of the Ecca Group, Nongoma Graben, northern Zululand. Karroo Basin symposium, Geological Society of South Africa, pp. 345-352.
Wilson, A.C. (1982): Geological map (1:250,000). GSMDS.

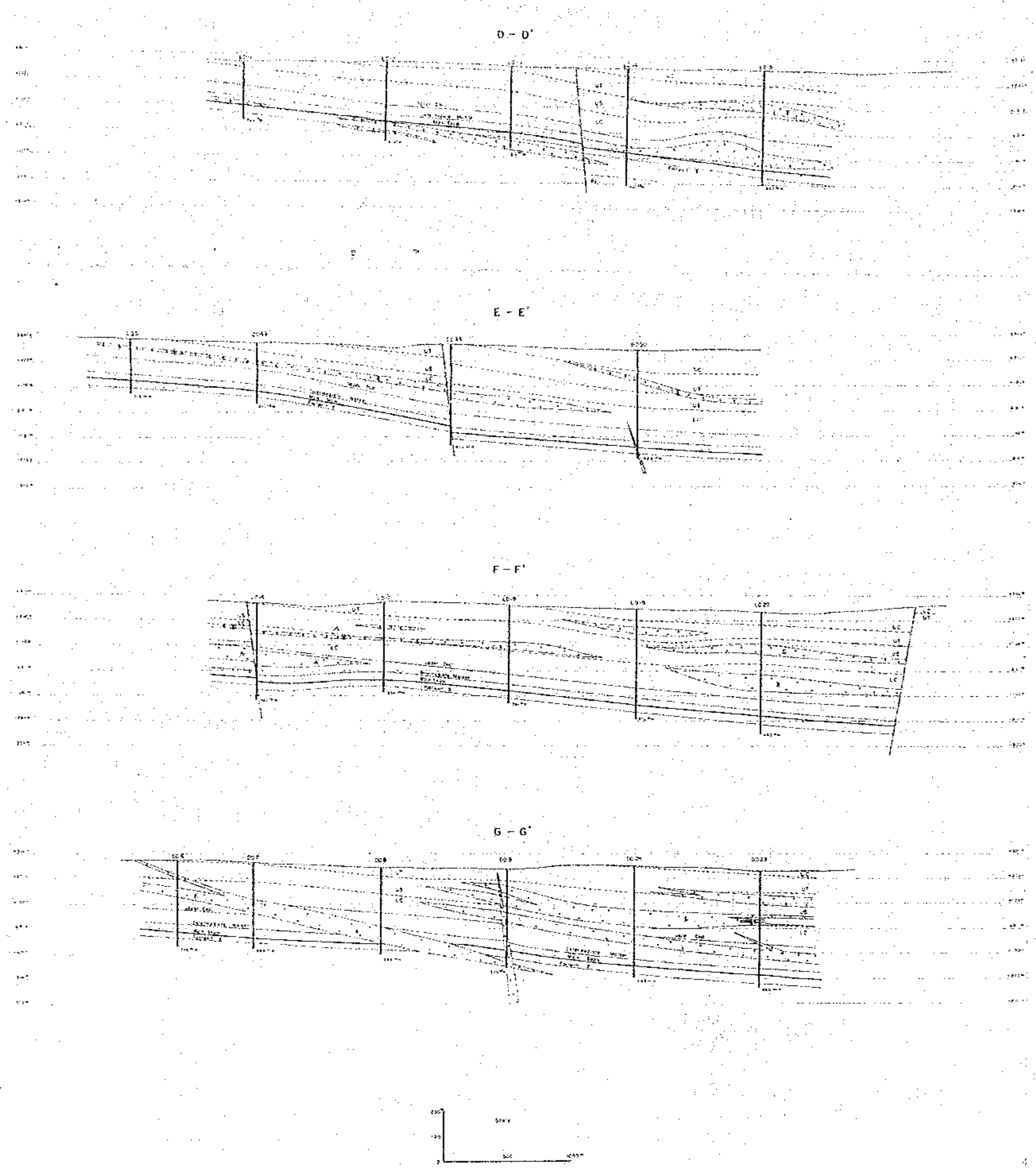
PART III.

Anglo American Corporation of South Africa Limited (1983): Environmental practice and principals – board and pillar workings. pp. 30.1-30.10.
Cummins, A.B. et al. (1973): SME Mining engineering handbook. vol. 1, pp. 12.2-12.71.
Hunter, D.H. et al. (1977): Coal age operating handbook of underground mining. pp. 2-6.
Kirsk, N. et al. (1984): The twenty-second annual report and statement of accounts, the year ended 31st March, 1984. Swaziland electricity board.
Mining and Metallurgical Institute of Japan (1983): Mining plan and recent situation of large mines. pp. 91-116 (in Japanese).
Murphy, J.S. et al. (1983): Annual report and accounts, the year ended 31st March, 1983. Swaziland railway board.
Salamon, M.D.G. and Oravecz, K.I. (1976): Rock mechanics in coal mining. pp. 65-70, Chamber of Mines of South Africa.
Sayama, S. editor (1965): Handbook of mining (in Japanese). Asakura Publishing Co., Ltd.
Suziki, H. et al. (1979): Plan and control of belt conveyor. pp. 199-282, Mining and Metallurgical Institute of Japan (in Japanese).
Woodruff, S.D. (1966): Methods of working coal and metal mines. vol. 3, pp. 73-152, Pergamon Press.

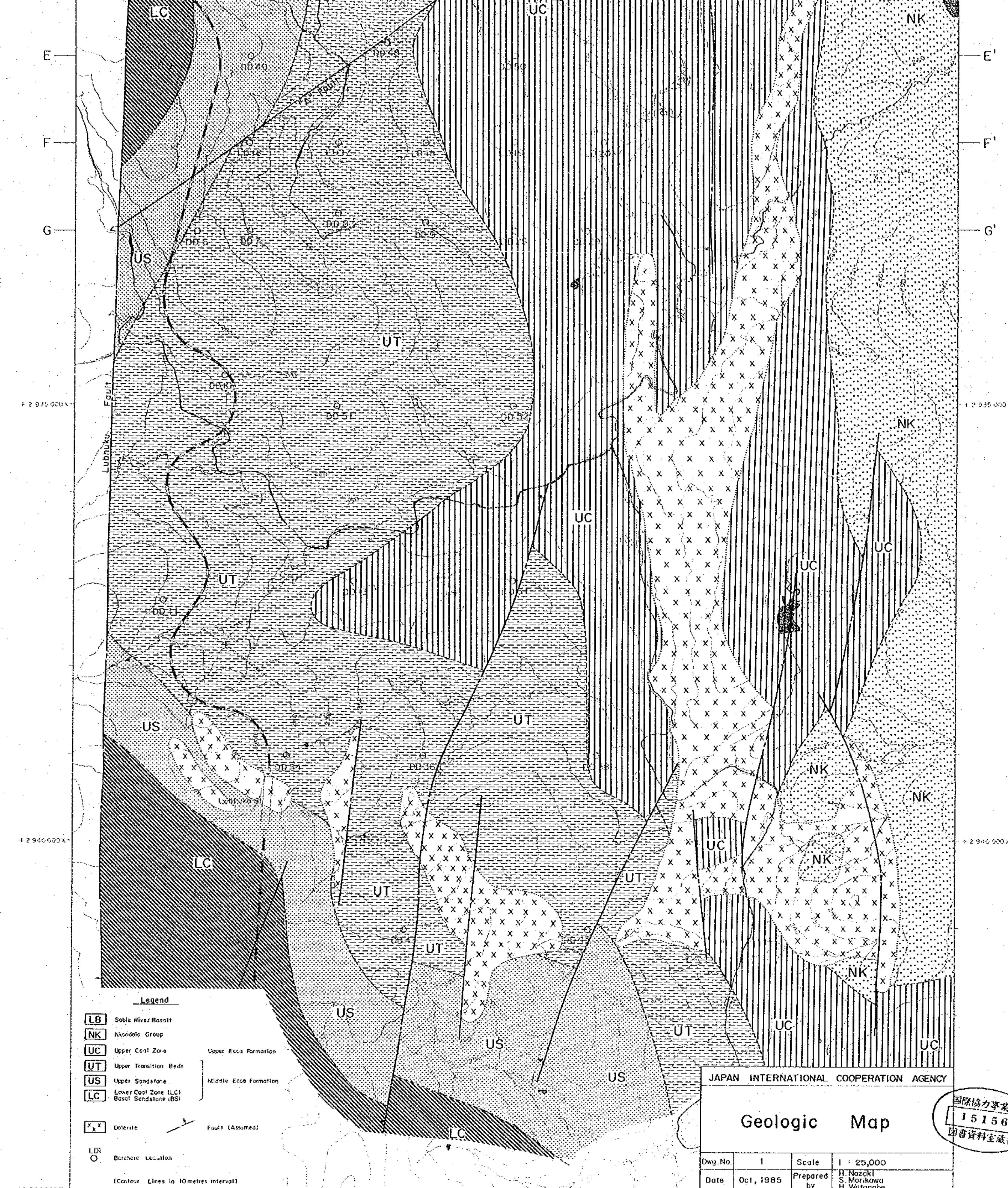
GEOLOGIC PROFILE





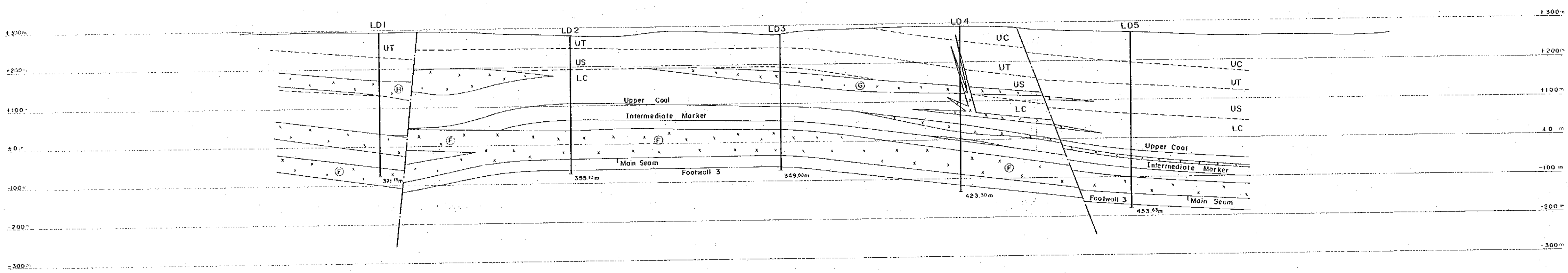


D - D'
E - E'
F - F'
G - G'

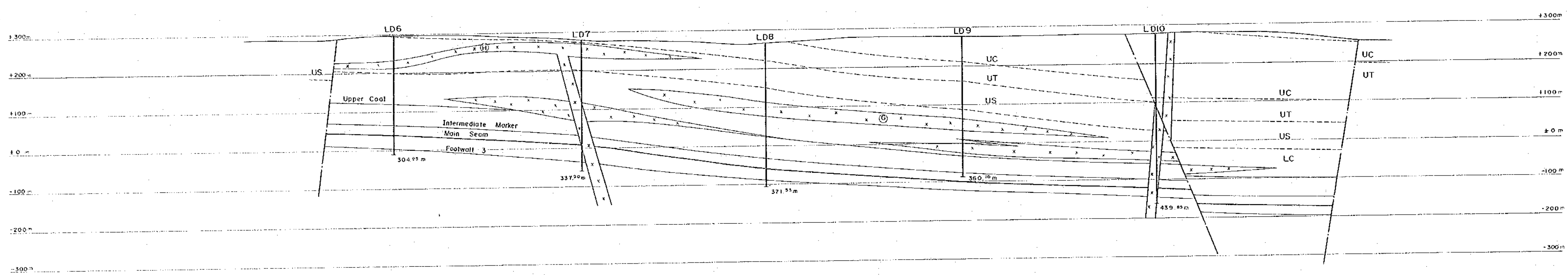


E
F
G
E'
F'
G'
LC
US
UT
UC
NK
Lubhuku Fault
Legend
LB Sable River Basalt
NK Nkondolo Group
UC Upper Coal Zone
Upper Ecca Formation
UT Upper Transition Beds
US Upper Sandstone
LC Lower Coal Zone (LC) Basal Sandstone (BS)
Middle Ecca Formation
Dolerite
Fault (Assumed)
LD1 Borehole Location
(Contour Lines in 10metres interval)
JAPAN INTERNATIONAL COOPERATION AGENCY
Geologic Map
Dwg. No. 1
Scale 1 : 25,000
Date Oct, 1985
Prepared by H. Nozaki S. Morikawa H. Watanabe

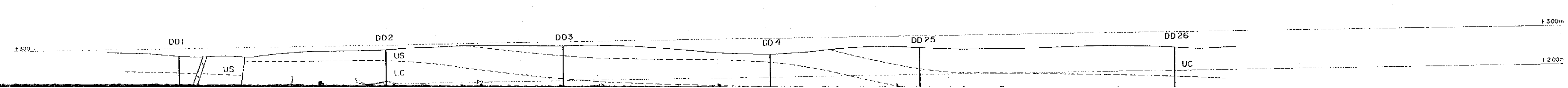
A – A'

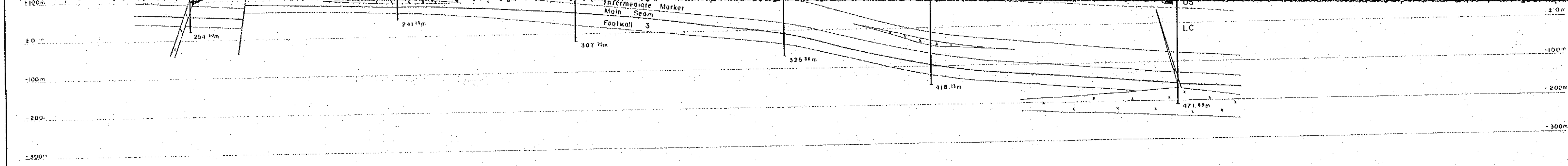

LD1
LD2
LD3
LD4
LD5
UT
US
LC
UC
Upper Coal
Intermediate Marker
Main Seam
Footwall 3
+300m
+200m
+100m
±0m
-100m
-200m
-300m

B – B'


LD6
LD7
LD8
LD9
LD10
US
UC
UT
LC
Upper Coal
Intermediate Marker
Main Seam
Footwall 3
+300m
+200m
+100m
±0m
-100m
-200m
-300m

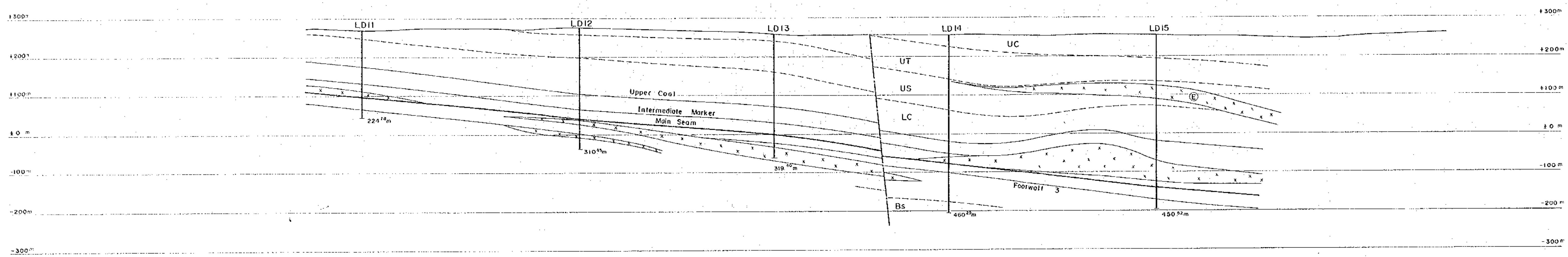
C – C'


DD1
DD2
DD3
DD4
DD25
DD26
US
LC
UC
+300m
+200m



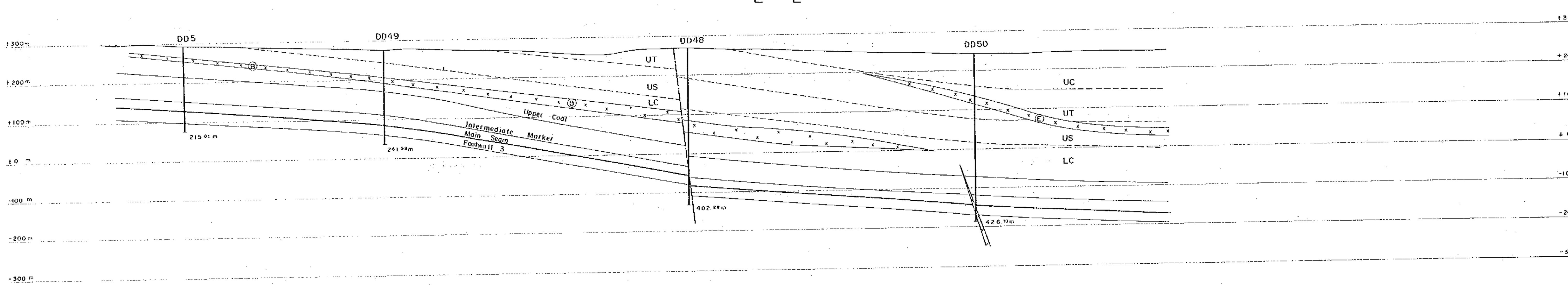
Intermediate Marker
Main Seam
Footwall 3
US
LC
254 30m
241 15m
307 73m
325 36m
418 13m
471 48m
+100m
± 0m
-100m
-200m
-300m

D – D'



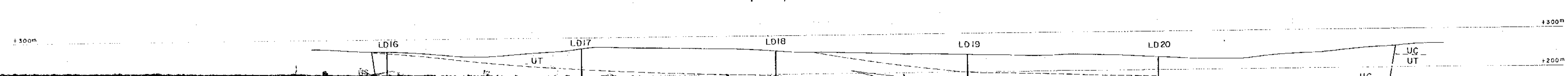
LD11
LD12
LD13
LD14
LD15
UC
UT
US
LC
Upper Coal
Intermediate Marker
Main Seam
Footwall 3
Bs
224 78m
310 55m
460 25m
450 52m
+300m
+200m
+100m
± 0 m
-100m
-200m
-300m

E – E'



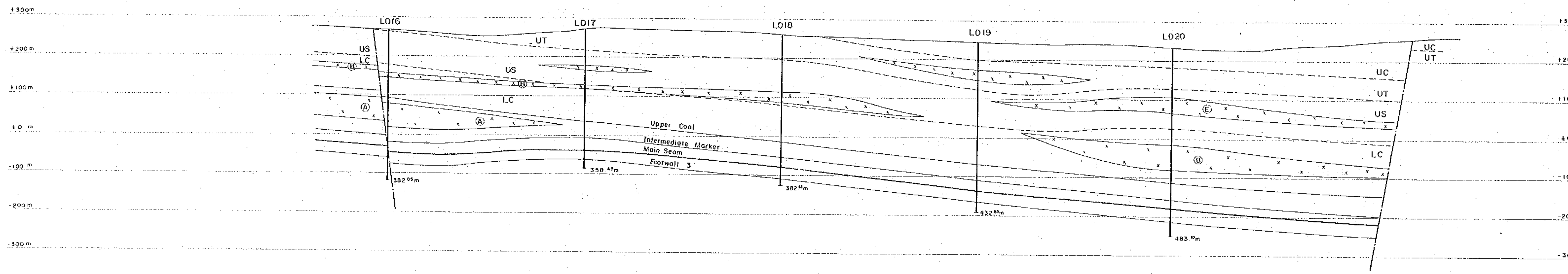
DD5
DD49
DD48
DD50
UT
US
LC
UC
Upper Coal
Intermediate Marker
Main Seam
Footwall 3
215 05 m
241 99m
402 98m
426 19m
+300m
+200m
+100m
± 0 m
-100 m
-200 m
-300 m

F – F'

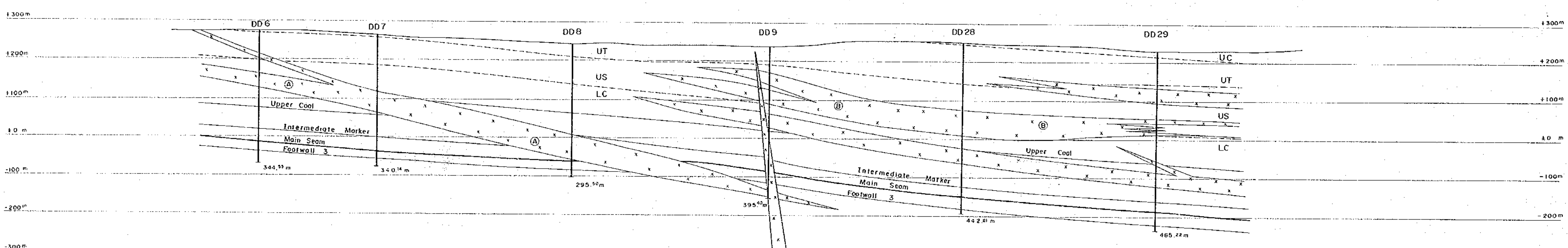


LD16
LD17
LD18
LD19
LD20
UT
UC
+300m
+200m

F – F'



G – G'



LEGEND

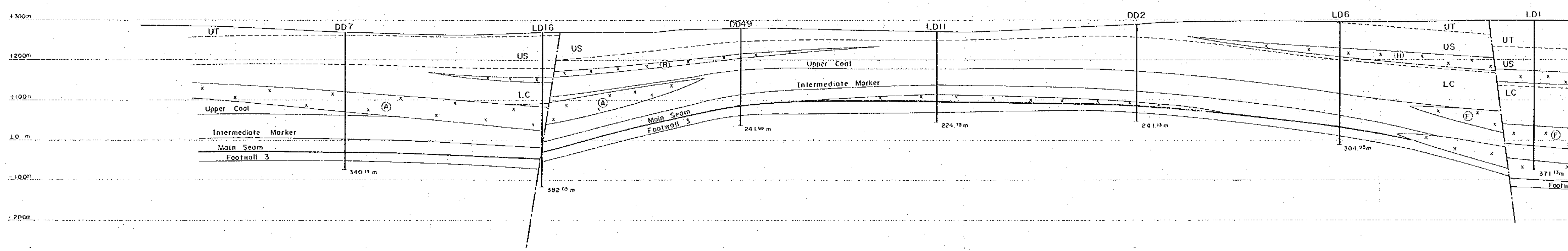
UC	Upper Coal Zone
UT	Upper Transition Beds
US	Upper Sandstone
LC	Lower Coal Zone
BS	Basal Sandstone
x(A)x	Dolerite
/	Fault, Assumed

Scale

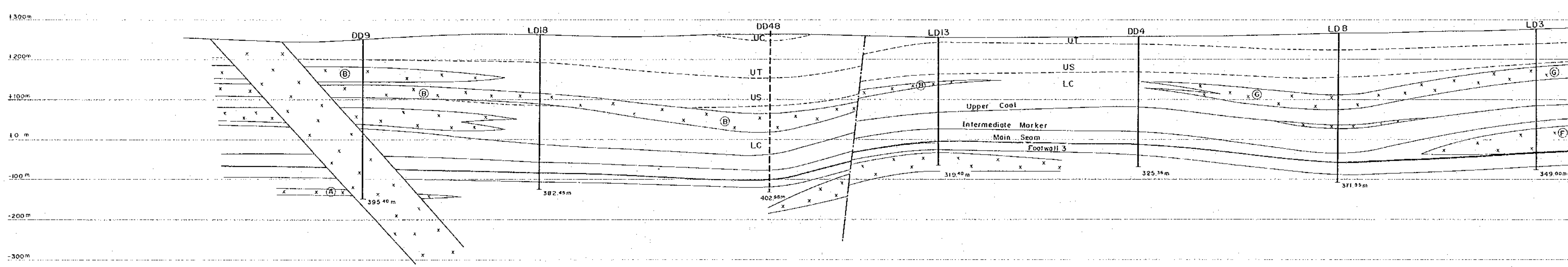
200m / 100 / 0 — 500 — 1000m

JAPAN INTERNATIONAL COOPERATION AGENCY			
Geologic Profile			
Dwg. No	2a	Scale	H=1:10,000 V=1:5,000
Date	Oct, 1985	Prepared by	H. Nozaki S. Morikawa H. Watanabe

H – H'

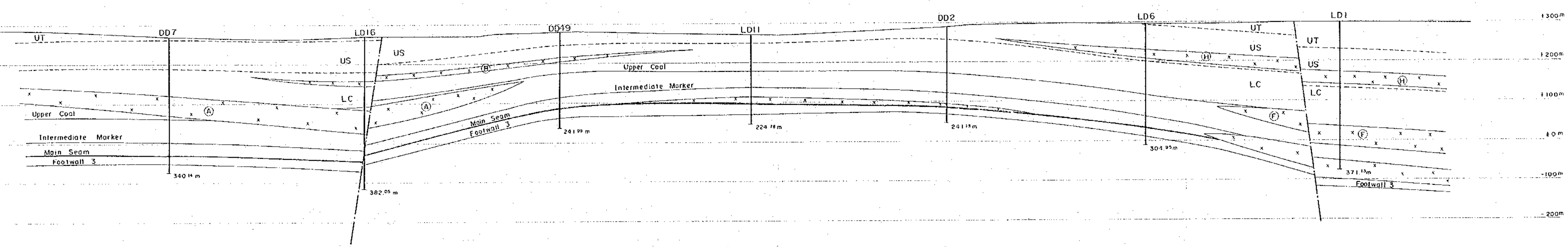

+300m
+200m
+100m
± 0 m
-100m
-200m
-300m
DD7
LD16
DD49
LD11
DD2
LD6
LD1
UT
US
LC
Upper Coal
Intermediate Marker
Main Seam
Footwall 3
340.18 m
382.63 m
241.92 m
224.78 m
241.13 m
304.95 m
371.13 m
Footwall

I – I'


+300m
+200m
+100m
± 0 m
-100m
-200m
-300m
DD9
LD18
DD48
LD13
DD4
LD8
LD3
UC
UT
US
LC
Upper Coal
Intermediate Marker
Main Seam
Footwall 3
395.40 m
382.45 m
402.98 m
319.40 m
325.36 m
371.95 m
349.00 m

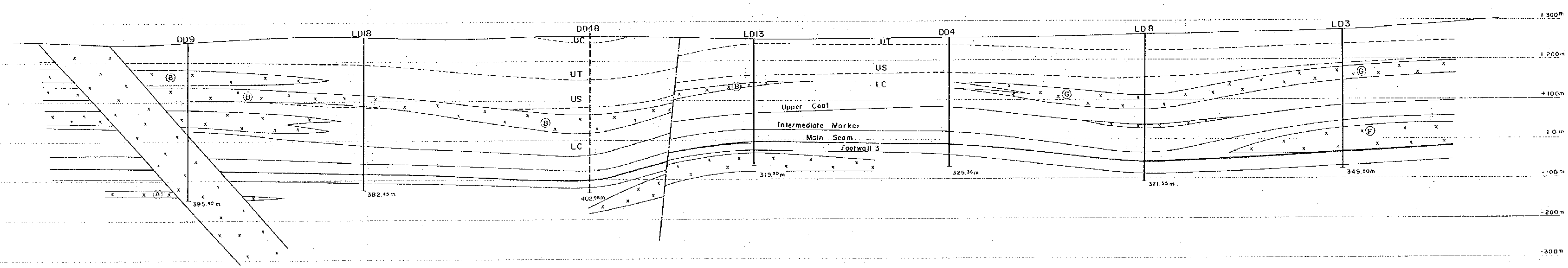
J – J'

H – H'



DD7
LD16
DD49
LD11
DD2
LD6
LD1
UT
US
LC
Upper Coal
Intermediate Marker
Main Seam
Footwall 3

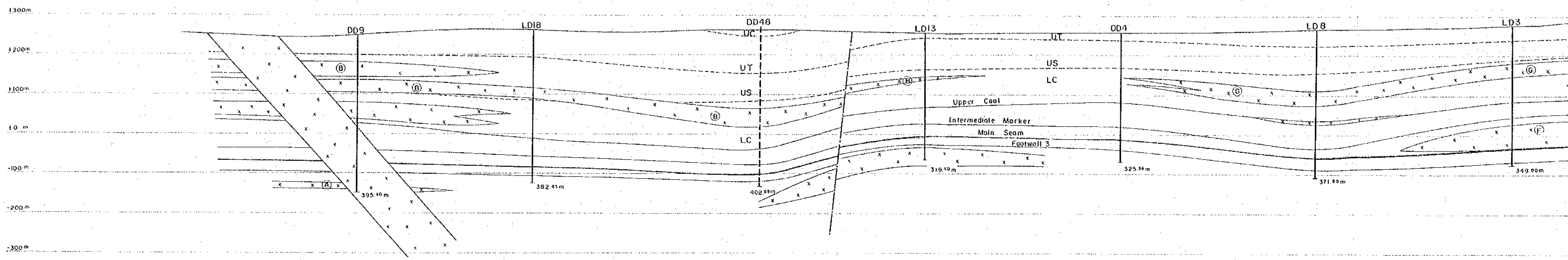
I – I'



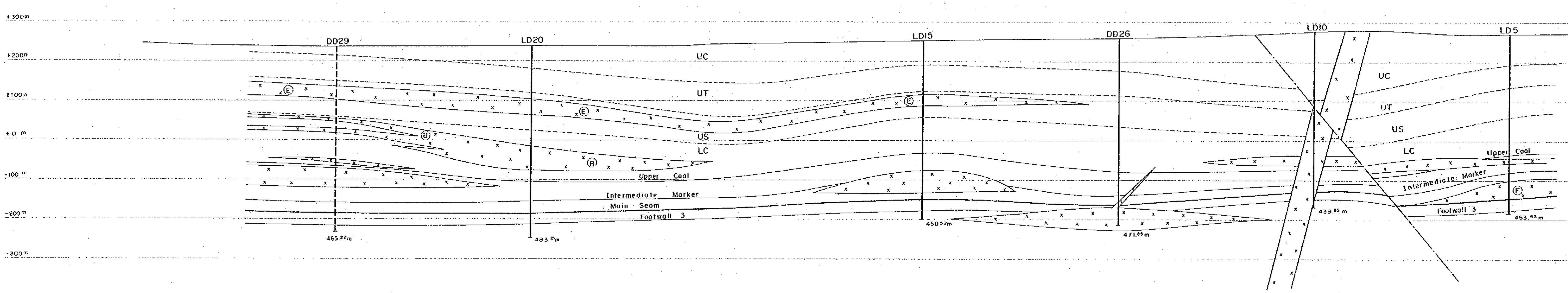
DD9
LD18
DD48
LD13
DD4
LD8
LD3
UC
UT
US
LC
Upper Coal
Intermediate Marker
Main Seam
Footwall 3

J – J'

I - I'

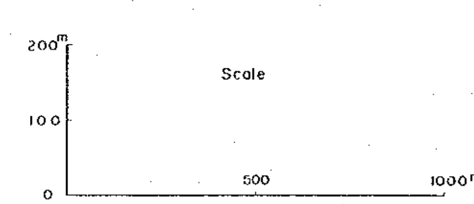


J - J'



LEGEND

UC	Upper Coal Zone
UT	Upper Transition Beds
US	Upper Sandstone
LC	Lower Coal Zone
BS	Basal Sandstone
x(A)x	Dolerite
/	Fault, Assumed

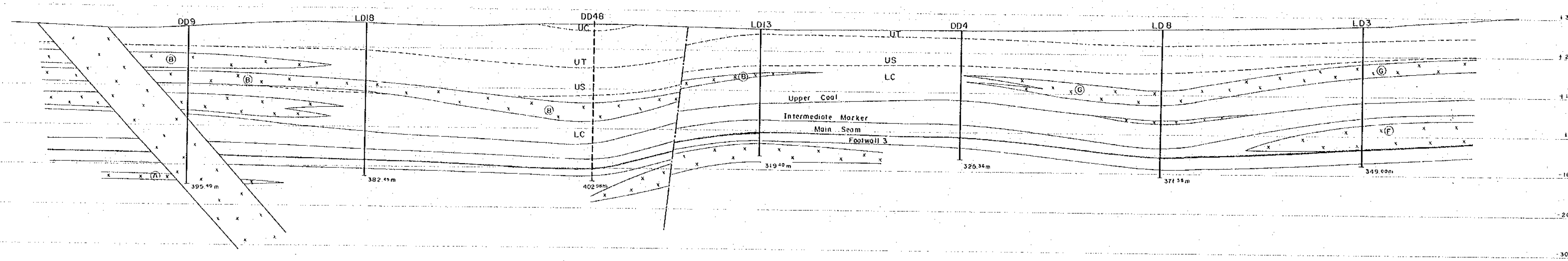


JAPAN

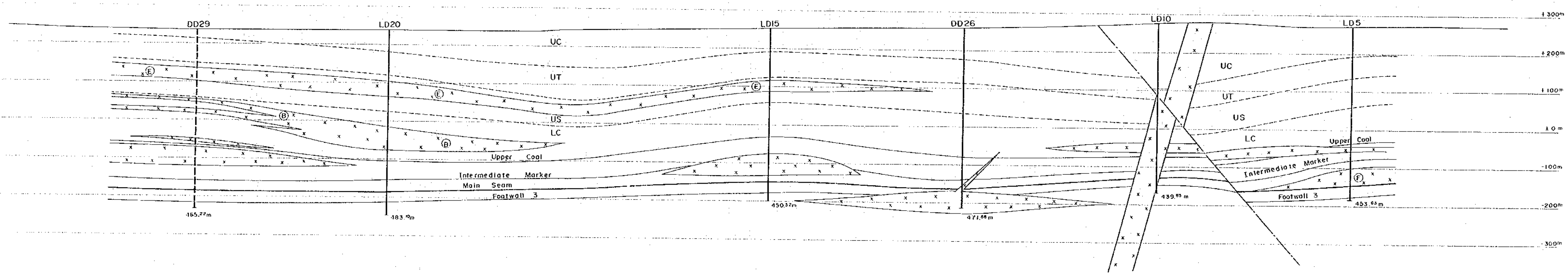
Dwg.No.

Date

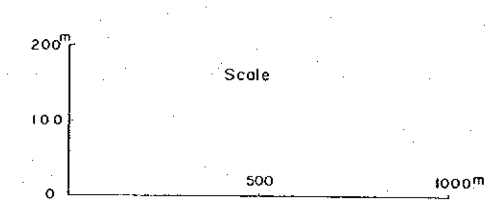
I – I'



J – J'



Zone
tion Beds
tone
Zone
tone
med



JAPAN INTERNATIONAL COOPERATION AGENCY			
Geologic Profile			
Dwg.No.	2 b	Scale	H = 1 : 10,000 V = 1 : 5,000
Date	Oct., 1985	Prepared by	H. Nozaki S. Morikawa H. Watanabe