2.2.4 Upper Transition Beds

The strata are composed of alternating beds of sandstone and greyish blue shale and the thickness is about 95 m. A shale bed at the lowest part can make a useful key bed for lithostratigraphic correlation because of its characteristic thin rhythmical intercalations of fine-grained sandstone. Impure calcareous thin beds are intercalated in the lower part of the strata but are found locally. The beds tend to occur in the eastern part of the investigated area.

2.3 Geologic Structure

The Karoo Supergroup in Swaziland was deposited in the Lebombo Graben that trends north-south. The Graben is considered to have been formed by continuous faulting and preserves the whole sequence of the supergroup. A N-S trending geologic structure is predominant in the area which has been affected by the said faulting.

The Ecca Group strikes generally north-south but it partially varies between NNW-SSE and NNE-SSW in the Lubhuku area. Relatively gentle folding structures are found in the southern and northern parts. A syncline plunging very gently to the east with its axis near the DD11 to DD31 line is found in the southern part. An anticlinal fold with a similar trend of plunge is found near the LD11 to LD13 line in the northern part, subsidiary folds are found in the eastern side. The beds generally dip around 5° east but some of them dip around 10°. However, the beds have a slightly steeper dip, 10 to 20°E, east of the investigated area. A north-south main fault runs from the west of DD1 to the west of DD11 in the western end of the area. The fault is named the "Lubhuku Fault" as this is the most significant fault in the area.

Several faults with the N-S trend running almost in parallel to the Lubhuku Fault are expected to exist in the area. But they are all small normal faults generally dipping west and their throws are considered to be 10 to 50 m. A NE-SW trending fault ("A" Fault) dipping east and running through LD16 and DD48 is delineated in the present investigation, and the fault seems to have branched from the Lubhuku Fault. NNE-SSW trending fault ("B" Fault) is expected to run through between LD1 to LD2 and DD1 to DD2 because the beds discontinued around these parts. NNW-SSE trending fault ("C" Fault) dipping east is expected to run near LD10, and a wedge-shaped depression was formed on the east side of this fault, where the overlying Nkondolo Group is exposed in a small area (Drawing 1).

Strike and dip of the beds vary in part because of the intrusion of a dolerite sill. Displacement of the beds is also found above an inclined sill due to raising of the overlying beds by intrusion of the sill. As stated in Paragraph 2.4 "Karoo Dolerite", the intrusion of the dolerite sill is considered to be related to the folding because most of the major sills are found near the axes of folds.

2.4 Karoo Dolerite

2.4.1 Modes of Occurrence

The Karoo Dolerite widely intrudes in the Lubhuku area. Various intrusions with thicknesses of 1 cm to over 100 m were penetrated in all boreholes drilled to date. These dolerite intrusions are divided into three types, concordant sill, inclined (discordant) sill and dyke, based on the modes of occurrence.

The concordant sill intrudes almost parallel to bedding of the surrounding strata. The inclined sill intrudes into the strata to cut the bedding and dips generally 10° to 50°. Many of the sills are inclined. The sills show complicated modes of occurrence as they vary in thickness, strike and dip, form gently inclined basin-like or dome-like structures, branch out partially and thin out (Drawing 6).

The dykes strike generally N-S and dip almost vertically. The width are mostly up to 10 m but may reach 50 m (LD7, LD10, etc.). Dolerite intrudes also along faults. The dykes may merge with or cut the sills. Fine- to very fine-textured dykes are occasionally observed in medium- to coarse-textured sills.

As a result of the dolerite intrusions, small fragments of sandstone, shale and coal are found in the dolerites. Small breccias of the dolerite are occasionally found in the invaded strata. Moreover, evidence of contamination of dolerite with invaded strata is rarely found. Indurated zones are occasionally observed in the invaded rocks near contacts with thick intrusions.

The dolerite is considered to have intruded in several episodes. More than one intrusion period has been confirmed in the area because different lithologic characters of the dolerites occur in contact with each other. The intrusion of the dykes appear to have continued on at a slightly later stage because the dykes cut the sills.

The frequency of dolerite sill intrusion (percentage of dolerite sills in the boreholes) shows that the dolerite percentage is as low as 1 to 25% in most parts of the investigated area. But the percentage is generally high, over 25% south of the area (Figure II-4). The high and low percentage regions of dolerite intrusions are distributed alternately in the north to south direction. A very low percentage region of dolerite intrusion is found especially in the central part of the investigated area. The regions showing a relatively high percentage of dolerite tend to distribute at intervals of 6 to 7 km in the north-south direction, and this tendency can give useful information to the planning of future exploration.

Eight major dolerite sills are recognized in the investigated area. The areal extent of the individual sill is 5 to 28 km^2 and its thickness ranges from about 10 m to 140 m (Figure II-5). These sills tend to be concentrated in the above-mentioned high percentage regions of the dolerite intrusion. Since the sills tend to be distributed along the fold axes, it may be inferred that the folding structure may be related to the intrusion of the dolerite sill. Many dolerite sills were recognized during the investigation to date, but no particular horizon in which the sill intrudes has been recognized. Occasionally some of the thick dolerite sills partially replaced the invaded strata.

2.4.2 Petrographic Characters

The dolerite is generally a massive, compact, crystalline, dark greenish grey to dark grey, almost homogeneous rock. Its lithology varies depending upon the modes of occurrence. Lithology of the central parts of the dykes and the thin sills are medium-textured dolerite or basalt and they change to fine-textured basalt at the marginal parts. On the other hand, lithology of the central parts of the thick sills are coarse-textured dolerite and a partially gabbro-like rock and grade into basalt at their margins. A narrow chilled margin is generally observed at the contact with the invaded rocks.

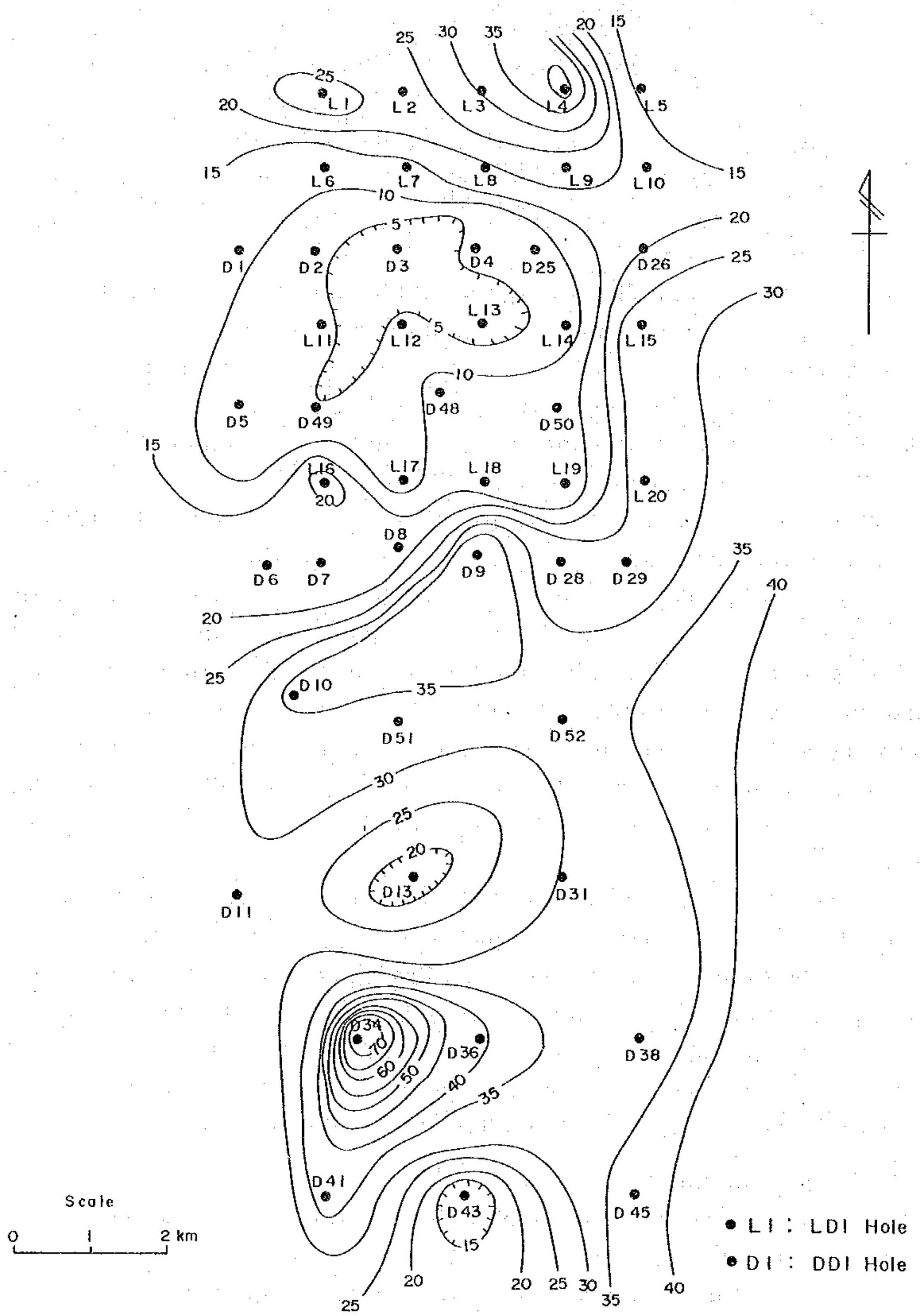

Figure II-4 Dolerite Sill Incidence Map
(Showing percentage of dolerite sill in borehole)

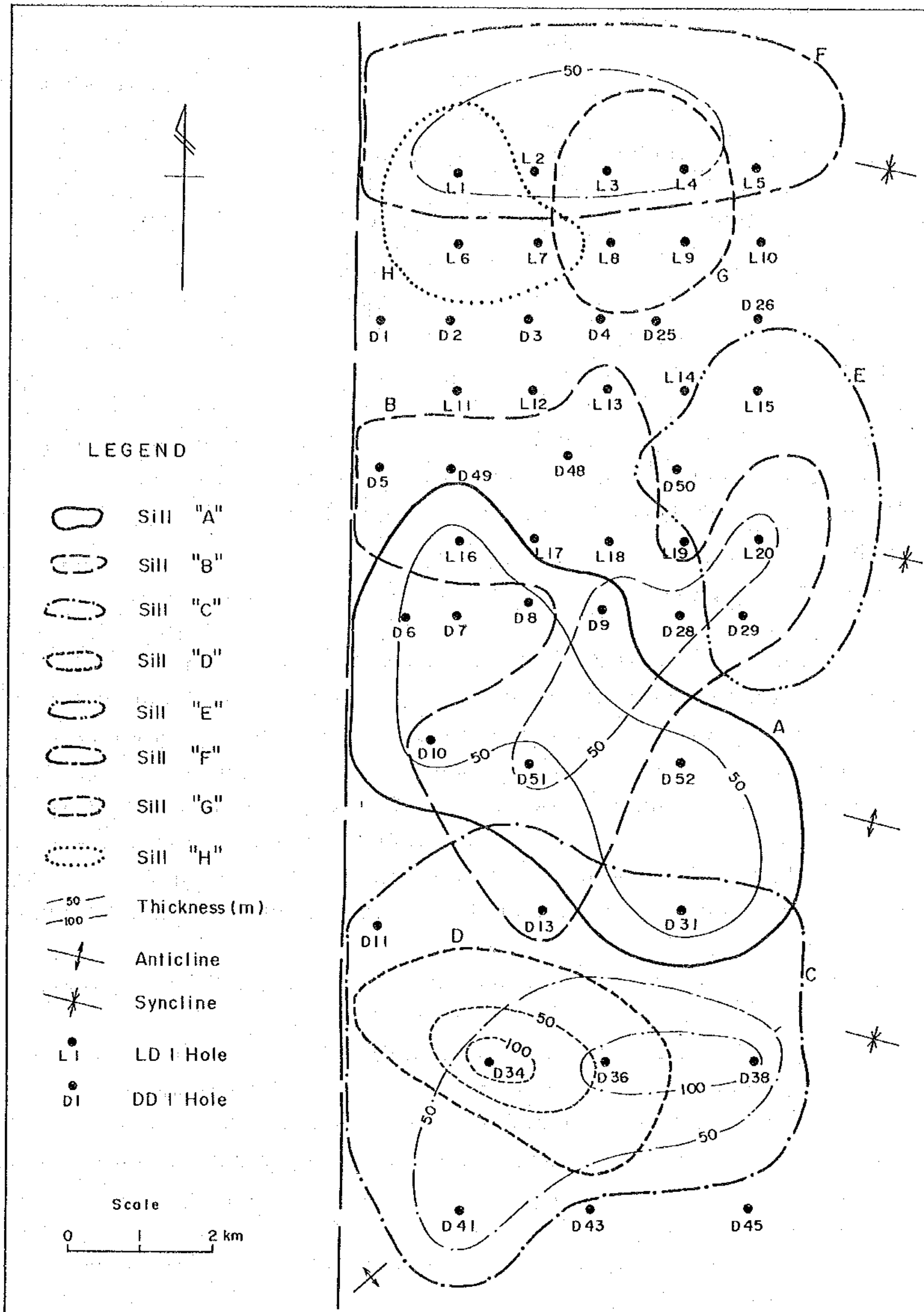

Figure II-5. Distribution of Major Dolerite Sills

The main constituents are plagioclase and pyroxene, and the plagioclase is mainly labradorite (anorthite content: 55 to 70%) with minor amounts of andesine and bytownite. The pyroxene is generally augite and orthopyroxene, and other mafic minerals include very small amounts of olivine, pigeonite and biotite. Small grains of iron minerals such as titaniferous magnetite and ilmenite are scattered throughout the rock as accessory minerals. Chlorite, iron-bearing montmorillonite, calcite, and sericite are found as alteration products.

Combinations of mafic minerals show the following characteristics; the main combinations are augite and augite-orthopyroxene, among other mafic minerals in minor amounts only olivine accompanies both combinations, and others accompany the augite combination. With respect to the major sills, large and thick sills such as "A", "B" and "C" have the combination of augite-orthopyroxene, and the small sills such as "E" and "H" contain the augite combination. These dolerites are all tholeiitic type rocks typically occurring in the continental areas.

Typical photomicrographs of dolerite are shown in Plates 3 and 4.

2.4.3 Effect on the Coal Seams

Coal in the Lubhuku area is all anthracite and semianthracite. These anthracitic characters are considered to be the result of the thermal effect of extensive volcanic activities of the Lebombo Group and the Karoo Dolerite after deposition of the coal seams, and the volcanic activities of the former seem to have played the main role. Direct effect of the dolerite on the coal seam was the thermal alteration such as burnt-out and coking, as well as the replacement of coal seam by sills.

Distance being thermally affected by the intrusion of the dolerite is generally considered to be almost equal to the thickness of the sill and the width of the dyke. However, it generally seems that the dykes had a thermal effects such as burnt-out or coking on the invaded rocks more than sills. The thermal effects of the dykes on the coal seams cannot be exactly recognized from drilling information, but the coal seams in the Mpaka mine are affected by the dykes on both sides of their width.

The thermal effects of the sills on the coal seams are very irregular; sometimes no effects are recognized even near the thick sills, on the contrary the thin sills sometimes have had an effect on the seams. The sill "F" (about 70 m thick) intruded in the LD2 to LD5 line and replaced most of the Main Seam, but burnt-out coal was hardly recognized in the major coal seams above and below the sill. Washability of coal is occasionally poor for Footwall 3 which is about 30 m below the sill, and washability of the Intermediate Marker is normal, which is found about 25 m above the sill. Burnt-out coal is recognized in Footwall 4 about 7 m above a sill which is over 29 m thick at LD13, but no effect is recognized at Footwall 3 about 22 m above the sill. The burnt-out coal is recognized in the Main Seam and Bottom Marker in this borehole, but dolerite of 15 cm thick is found only in the Main Seam.

Plate 3. Photomicrographs of Dolerite (1)

1. Thick dolerite sill "F"
   Dolerite (augite-orthopyroxene-olivine combination).
   LD1, 370 m, open nicol, x35.
2. Thick dolerite sill "F"
   Dolerite (augite-orthopyroxene combination).
   LD3, 262 m, open nicol, x35.
3. Thick dolerite sill "F"
   Dolerite (augite-orthopyroxene-olivine combination).
   LD4, 338 m, crossed nicol, x35.
4. Thick dolerite sill "A"
   Dolerite (augite-orthopyroxene combination).
   LD16, 220 m, open nicol, x35.

Plate 4. Photomicrographs of Dolerite (2)

5. Thin dolerite sill "H"
   Dolerite (augite combination).
   LD1, 139 m, crossed nicol, x35.
6. Thin dolerite sill "E"
   Dolerite (augite combination).
   LD15, 151 m, crossed nicol, x35.
7. Dolerite dyke
   Dolerite (augite-orthopyroxene combination).
   LD7, 201 m, open nicol, x35.
8. Contact between different rock types of dolerite
   Fine-textured dolerite intrudes along cracks of coarse-textured dolerite sill "B".
   LD20, 287 m, open nicol, x35.

Remarks. O: olivine, A: augite, R: orthopyroxene,
P: plagioclase, C: chlorite,
M: iron-bearing montmorillonite.

Plate 3

1

2

3

4

Plate 4

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2.5 Coal Seam

Coal-bearing formations distributed in the Lubhuku area are the Upper Coal Zone which is composed of thin coal seams at the upper horizon and the Lower Coal Zone found about 180 m below the former. The Lower Coal Zone was the target for the recent investigation. Major coal seams including the Main Seam which is considered to be workable exist in the lower part of the Lower Coal Zone. The major coal seams are gently folded reflecting the geologic structure, and they strike almost north-south and dip approximately 5° east (Drawing 8). Outline of the major coal seams observed in boreholes is shown in Table II-6, and borehole logs of coal seams are shown in Drawing 7.

2.5.1 Modes of Formation of Major Coal Seams

Remarkable difference of the modes of formation of the coal seams in the Lower Coal Zone can be found between the upper part above the Top Marker and the lower part below it. Coal seams in the upper part are generally thin with poor continuity and occasionally thin out. An areal extent of the thick part (over 1 m) of the Upper Coal Seam, which is relatively thick among the coal seams in the upper part, is remarkably restricted within a small area.

On the other hand, coal seams in the lower part extend continuously and include several thick seams which exceed 1 m in thickness although they have some variations in coal thickness, partings and clastic interval thickness. The major coal seams, including the Main Seam with thickness which occasionally exceeds 4.0 m, are contained in this part.

Interval thickness between coal seams (thickness of clastics) and the multiple interval thickness measured downward from the Top Marker of these major coal seams (Top Marker to Footwall 3) have the following characteristics: The frequency distribution of the interval thickness between the coal seams has an almost normal distribution and is stable between the Top Marker and Bottom Marker. But it remarkably varies in the horizon below the Bottom Marker, and two peaks are found between the Main Seam and Footwall 1, and Footwall 1 and Footwall 2 respectively (Figure II-6). These characteristics are reflected in the frequency distribution of the multiple interval thickness measured downward from the Top Marker, and the frequency distribution is almost constant up to the Bottom Marker. But it varies remarkably and is irregular in the horizon below the Main Seam (Figure II-7).

As stated above, the interval thickness between the Top Marker and Bottom Marker is about 13 m in average and is mostly stable. In addition, a thin shale bed and coal seam exist immediately above the Top Marker, and the clastics between the seams are mostly coarse-grained sandstone. From these characteristics, these coal seams can be used as a useful key bed for the lithostratigraphic correlation between boreholes.

No great areal variation of the interval thickness between coal seams is found between the Intermediate Marker and Bottom Marker, the interval thickness of which is almost stable (Figure II-8). On the other hand, the areal variation between the Main Seam and Footwall 1 is remarkable

Table II-6a Thickness of Major Coal Seam

		LD1	LD2	LD3	LD4	LD5	LD6	LD7	LD8	LD9	LD10
	Elevation of Collar (m)	+298	+284	+280	+297	+273	+300	+284	+267	+285	+282
INTERMEDIATE MARKER	Depth to Seam (m)		218.95	222.12		351.47	228.75		296.87	349.55	
	Seam Thickness (m)	-	1.33	1.38	-	1.23	0.87	-	1.43	1.10	-
	Coal Thickness (m)		1.33	1.38		0.62	0.87		1.14	1.07	
MAIN SEAM	Depth to Seam (m)		318.13	310.30	379.40	413.65	254.42		319.57		
	Seam Thickness (m)	-	0.55	0.80	0.18	0.89	3.10	-	1.68		-
	Coal Thickness		0.36	0.77	0.18	0.89	2.97		1.50		
FOOTWALL 3 SEAM	Depth to Seam (m)		346.82	341.22	406.98	442.41	287.67	314.12	350.74		
	Seam Thickness	-	0.83	1.93	1.72	1.30	1.34	1.54	1.81		-
	Coal Thickness (m)		0.58	1.43	1.45	1.06	1.34	1.54	1.44		

Table II-6b Thickness of Major Coal Seams

		LD11	LD12	LD13	LD14	LD15	LD16	LD17	LD18	LD19	LD20	
	Elevation of Collar (m)	+271	+280	+264	+260	+255	+272	+277	+269	+251	+242	
INTERMEDIATE MARKER	Depth to Seam (m)	133.91	208.27	229.52	305.40		285.07	278.85	316.61	352.50	391.59	
	Seam Thickness (m)	1.59	1.70	1.08	0.90	-	1.69	1.71	1.86	1.35	1.45	
	Coal Thickness (m)	1.59	1.70	1.08	0.55		1.69	1.71	1.86	1.35	1.45	
MAIN SEAM	Depth to Seam (m)	171.33	234.90	256.51	333.75	395.86	311.78	307.56	341.89	378.91	415.86	421.56
	Seam Thickness (m)	3.06	6.22	2.93	0.39	5.28	0.57	4.88	4.69	5.83	1.64	2.21
	Coal Thickness	2.25	2.68	2.19	0.29	3.14	0.50	4.43	3.99	3.80	0.86	1.78
FOOTWALL 3 SEAM	Depth to Seam (m)	200.26		276.33	357.17	429.47	325.56	332.98	368.34	407.11	447.88	
	Seam Thickness (m)	1.37	-	1.01	1.15	1.35	0.47	1.64	1.45	1.44	1.38	
	Coal Thickness (m)	1.27		1.01	0.84	1.27	0.47	1.58	1.40	1.38	1.18	

as the variation of the intervals is large and two peaks are found in this horizon. The interval thickness in this horizon is small in an extensive area in the northern part and the large interval thickness is found in a scattered pattern surrounding the small interval thickness part mentioned above (Figure II-9). As far as the interval thickness between Footwall 1 and Footwall 2 is concerned, the small interval surrounds the large interval. The large interval is found in the south and north which reflects the sedimentary unit mentioned later (Figure II-10).

The modes of formation of the major coal seams in the northern part of the Lubhuku area are favourable as compared with those in the southern part because the coal seams have excellent thicknesses, continuity and are stably distributed in the northern part.

2.5.2 Intermediate Marker Seam

Among 20 boreholes drilled in the present investigation, 15 boreholes penetrated through the Intermediate Marker Seam, and the remaining boreholes could not confirm the seam because of the intrusion of dolerite. This coal seam has a thickness of over 1.0 m throughout the investigated area, and is thickest at DD5 in the southwestern part, where the seam thickness is 2.06 m (coal thickness of 1.83 m), and the seam gradually thins toward the surrounding area. The seam is thinnest at LD6 in the northern part (0.87 m of both seam and coal thicknesses) and at LD14 in the eastern part (0.90 m of seam thickness and 0.55 m of coal thickness) (Drawing 9a).

This coal seam locally contains shaly coal and rarely has partings of sandstone, shale and coaly shale. The roof of the seam is mostly coarse-grained sandstone, and the floor is coarse-grained sandstone, sandy shale, shale and coaly shale as it varies locally.

2.5.3 Main Seam

The Main Seam is the thickest coal seam among the coal-bearing formations distributed in the investigated area. The Mpaka mine, the sole working colliery in Swaziland, exploits this seam. Among the 20 boreholes drilled this time, 17 boreholes penetrated through this coal seam, and the remaining 3 boreholes could not confirm the seam due to intrusion of dolerite. In some of the holes bored through this coal seam, only a part of the seam was confirmed because of faulting at LD16 and of the intrusions of the dolerite at LD14 and along the LD2 to LD5 line.

As far as the overall occurrence of the seam is concerned, the seam thickness is generally over 2.0 m and attains a maximum of 4.99 m (coal thickness of 4.57 m) at DD5 in the southwestern part of the investigated area, and it thins gradually toward the surrounding area. If the effect of the intrusion of the dolerite is excluded, the seam thickness is a minimum of 1.69 m (coal thickness of 1.51 m) at DD1 in the northern part and followed by a minimum thickness of 2.93 m (coal thickness of 2.19 m) at LD13 in the central part (Drawing 9b).

The Main Seam is generally composed of a thin coal member with a thickness of 10 to 60 cm at the uppermost part, a parting of shale or coaly shale of 30 to 70 cm thick and a thick coal member at the lower part. The lower part occasionally contains a thin layer of shaly coal, coaly shale and

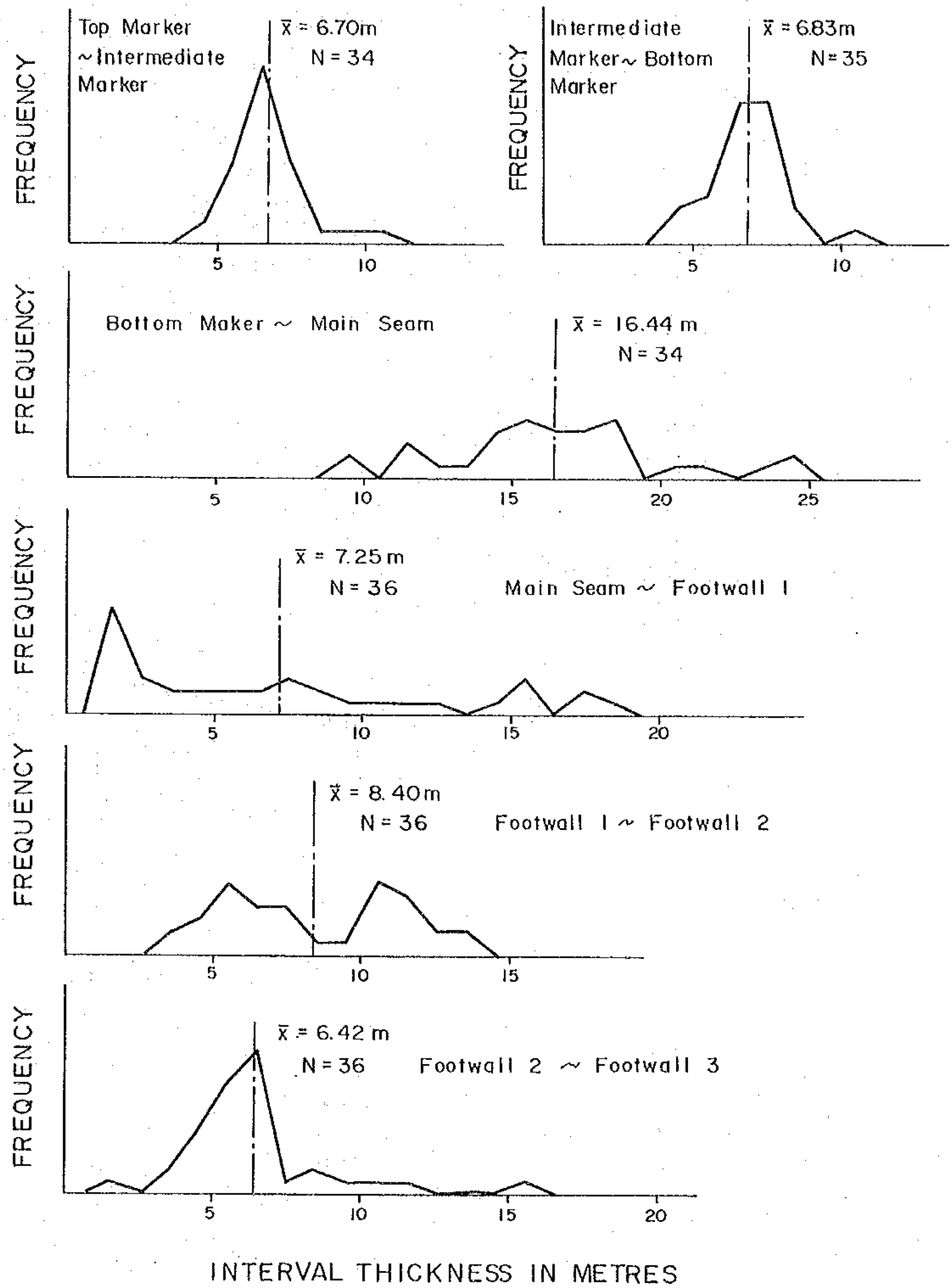


Figure II-6. Frequency Distribution of Individual Clastic Interval Thicknesses Between Major Coal Seams.

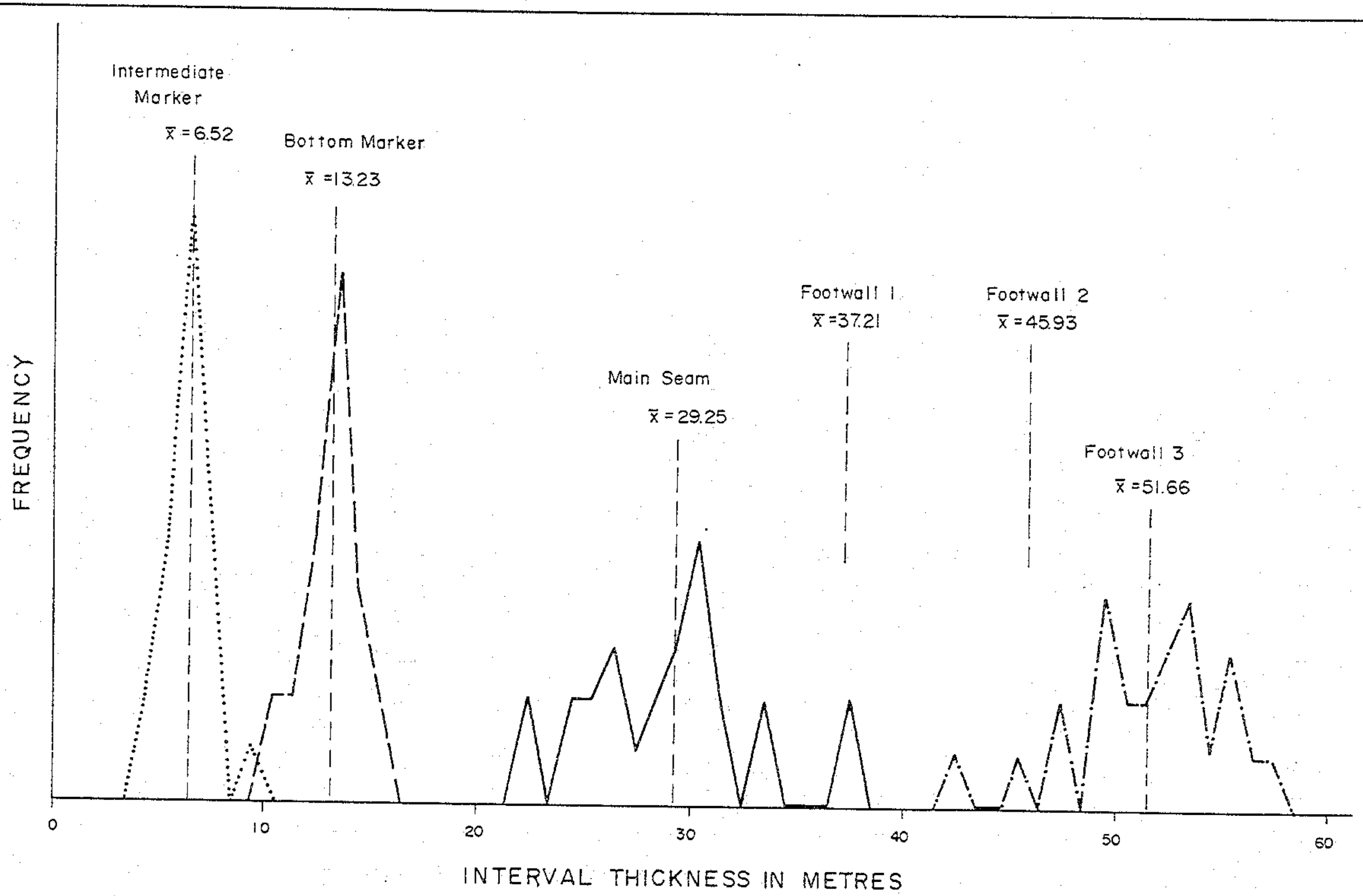


Figure II-7 Frequency Distribution of Interval Thickness Measured from Top Marker.

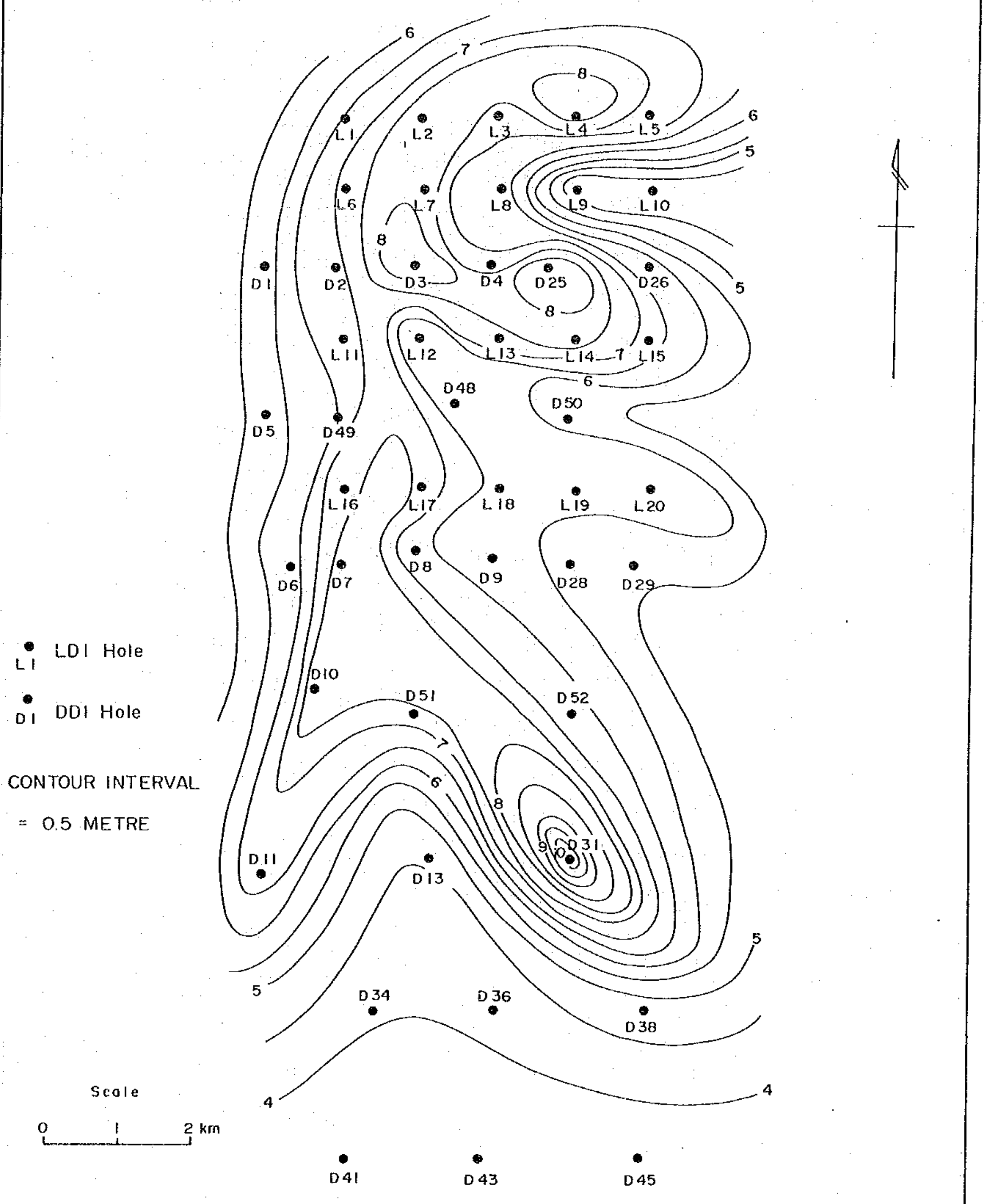


Figure II-8 Clastic Interval Thickness Trends between Intermediate Marker and Bottom Marker

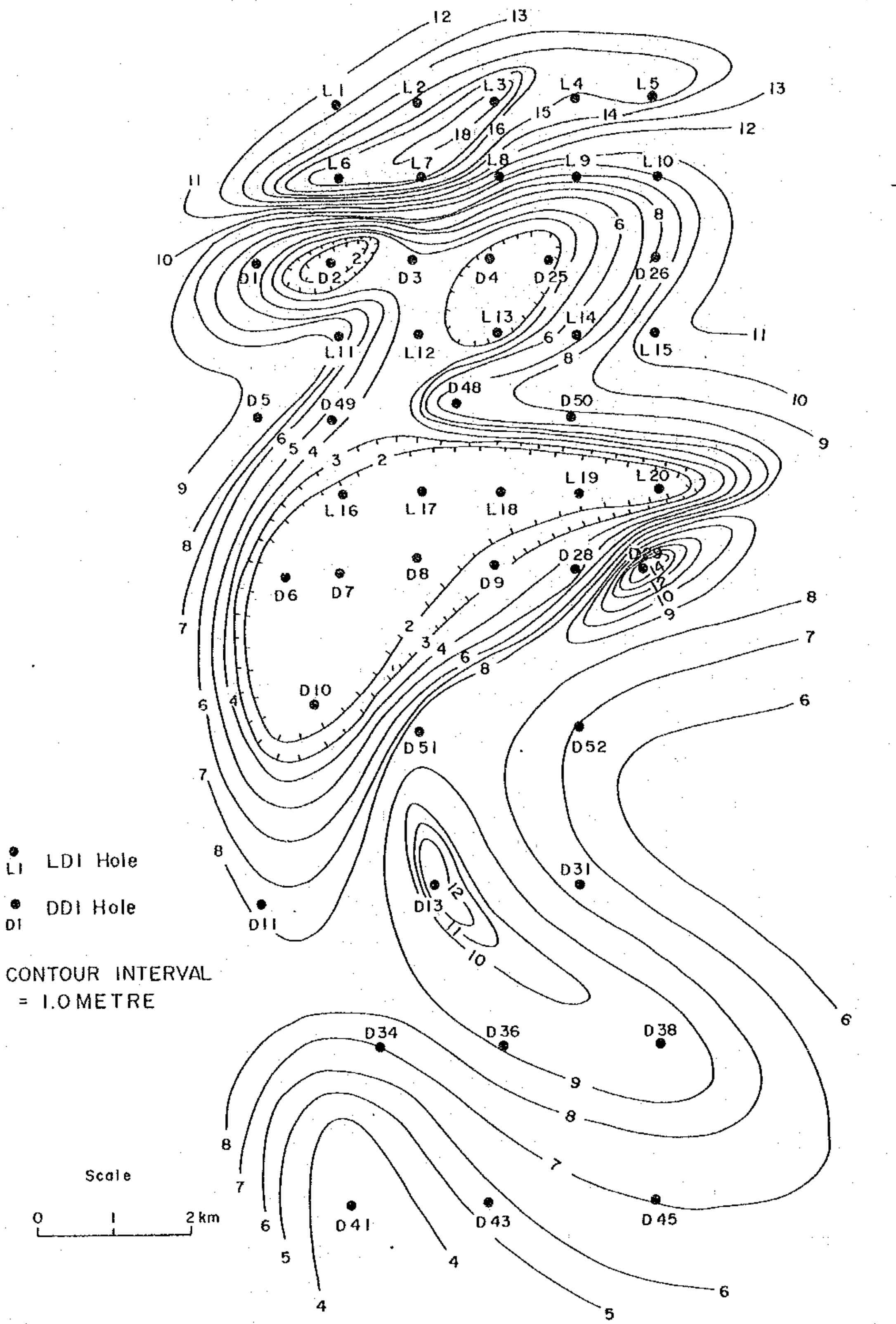


Figure II-9 Clastic Interval Thickness Trends between Main Seam and Footwall I.

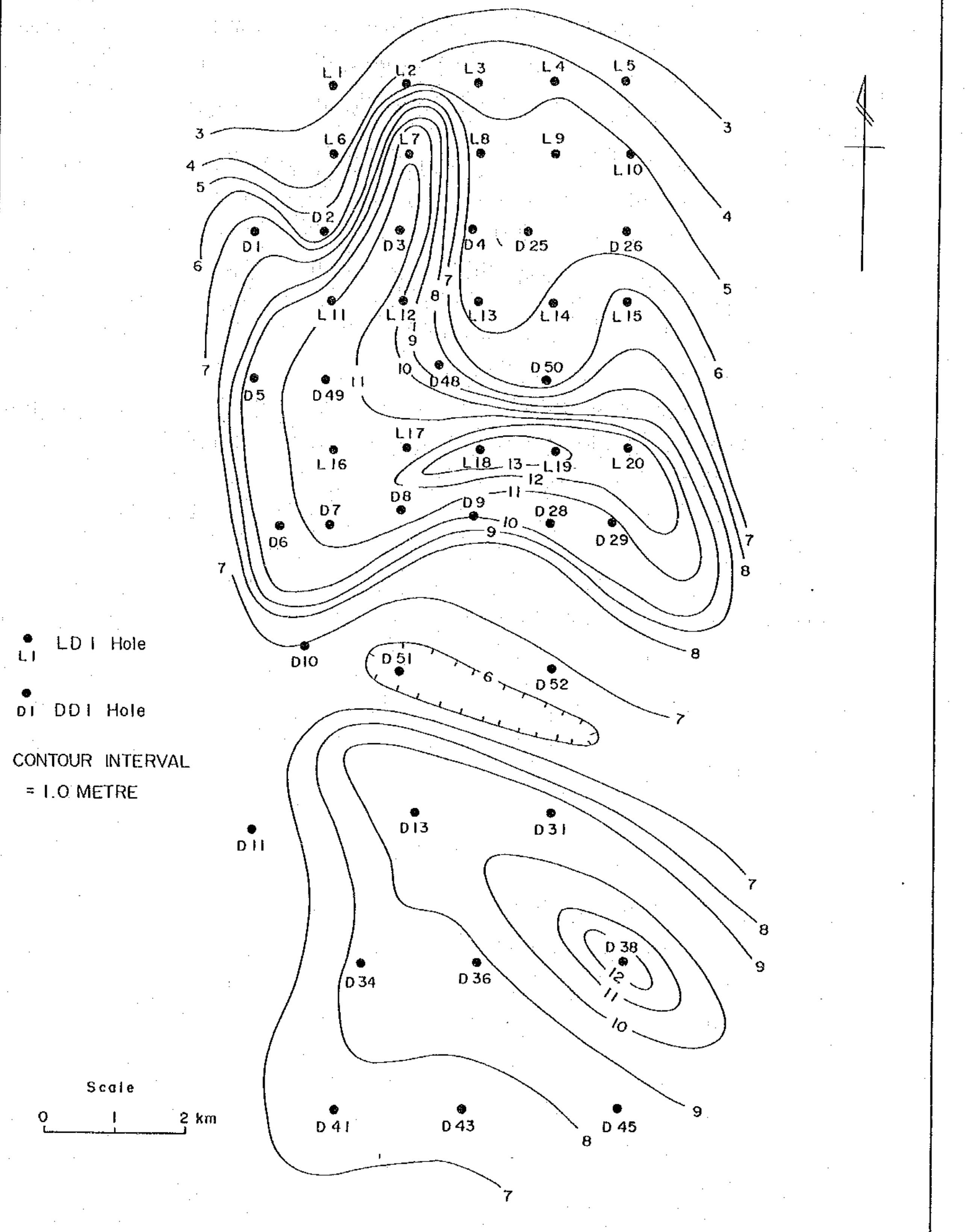


Figure II-10 Clastic Interval Thickness Trends between Footwall 1 and Footwall 2.

shale. The coal seam tends to split into 2 to 3 parts as the parting of shale or sandstone thickens in the eastern part, and this tendency is remarkable especially at LD19 and LD20. The parting thickens at DD3 in the northern part. This coal seam almost stably continues on the DD5 to DD50 line (Drawing 10).

The roof of the Main Seam is generally coarse-grained sandstone continuing up to the overlying Bottom Marker Seam, and a thin layer of shale (0.5 to 4.7 m thick) is locally found immediately above the coal seam. The floor is mostly shale or sandy shale and sandstone is found locally. The thickness of clastics between the Main Seam and Bottom Marker Seam varies between 12 m and 25 m in the investigated area. The thin shale layer immediately above the Main Seam is distributed forming a narrow zone extending nearly north and south in the western and eastern parts (Figure II-11). This shale layer occasionally contains thin coal beds.

Considering the modes of formation and thickness of the Main Seam, distribution of dolerites and faults, and geologic conditions of roof rocks, the most suitable area for the development of the Main Seam can be delineated as follows:

East limit:	"C" Fault and zone provides a dominance of dolerite dyke.
West limit:	Lubhuku Fault and "B" Fault.
South limit:	"A" Fault.
North limit:	Dolerite sill "F".

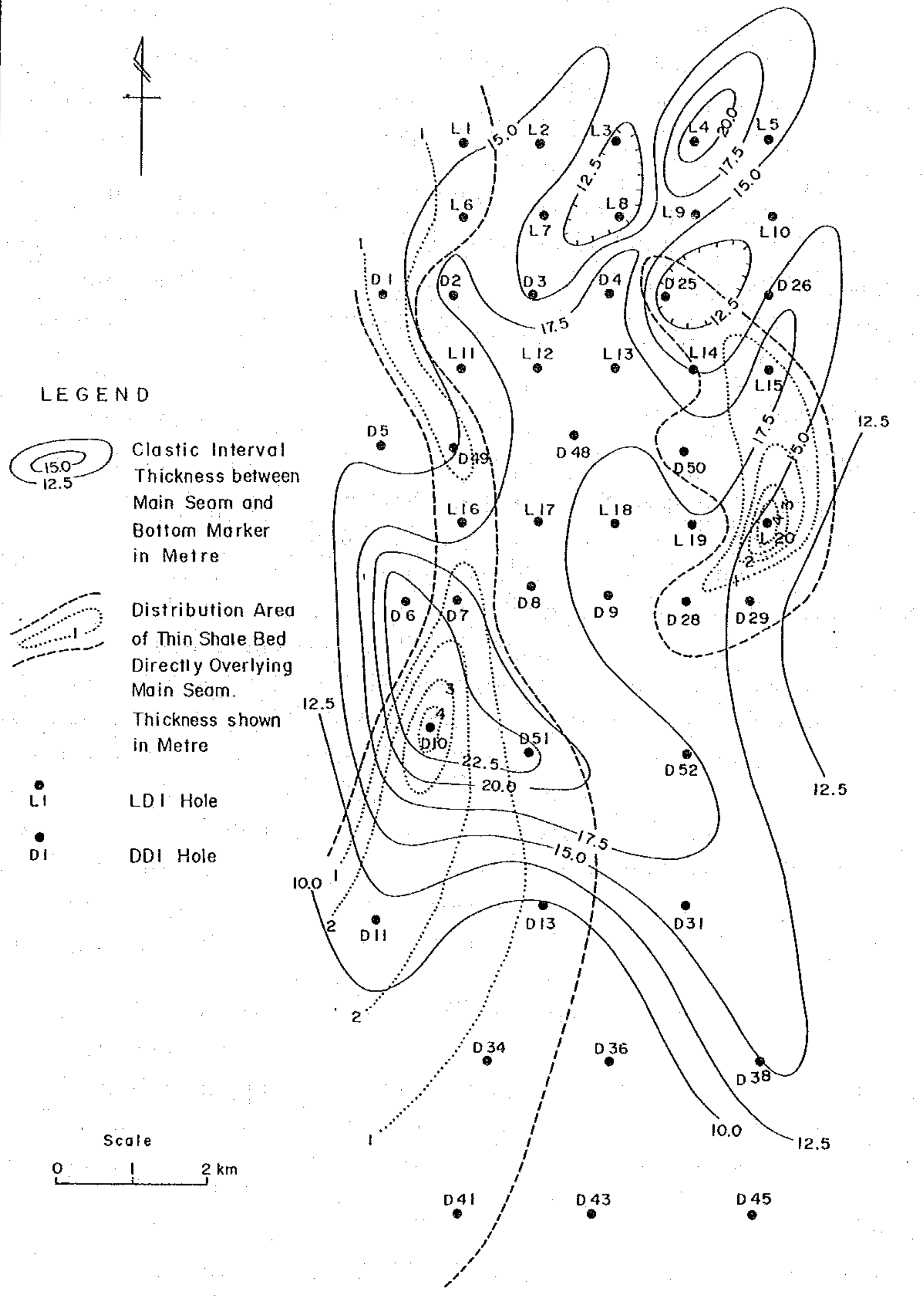


Figure II-11. Clastic Interval Thickness Trends between Bottom Marker and Main Seam.

2.5.4 Footwall 3 Seam

The Footwall 3 Seam exists about 22 m below the overlying Main Seam and the seam thickness generally exceeds 1.0 m. In the present investigation, this coal seam was confirmed in 15 out of the 20 boreholes and the remaining boreholes did not intersect the seam because of the intrusion of the dolerite.

The thickness varies remarkably, it is thickest at DD1 with the seam thickness of 2.20 m (coal thickness of 1.96 m) and thinnest at DD25 with 0.69 m of both the seam and coal thicknesses in the investigated area (Drawing 9c). The seam is occasionally interbedded with thin layers of shaly coal and coaly shale. The roof of the seam is coarse- or medium-grained sandstone and the floor is sandy shale or shale.

2.6 Sedimentary Environment

2.6.1 Sedimentary Environment of the Ecca Group

Considering the lithofacies of the Ecca Group, the Lower Ecca Formation at the lowest horizon is well-sorted argillaceous facies. Arenaceous facies is predominant in the Middle Ecca Formation although argillaceous facies is sometimes found in the lower-most and upper-most parts. Argillaceous facies is predominant again in the Upper Ecca Formation.

A large scale upward-coarsening sedimentation, which is a characteristic feature of a prograding delta formation, is found in the Lower Ecca Formation, Lower Transition Beds and Basal Sandstone, and these seem to correspond to a delta shelf, prodelta and delta-front facies respectively. A thick formation having a small-scale upward-coarsening sedimentary cycle is found in the Lower Coal Zone, and this may represent a stable delta plain or alluvial plain facies, and a lot of coal seams are contained in the formation. A large scale upward-coarsening sedimentation is found again in the Upper Sandstone and Upper Transition Beds, which may indicate further progradation, and these beds may represent the delta-front and prodelta facies respectively. An upward-coarsening sedimentary cycle is found again in the Upper Ecca Formation, which may represent stable lacustrine or swamp sediments. Coal seams are contained in the formation.

Cross-lamination and cross-bedding are frequently found in sandstone of the Middle Ecca Formation, and this indicates that water currents played an important role during sedimentation. The formation was deposited under fresh water fluvial and delta conditions. It is considered that the supply source of the sediments was relatively close to the sedimentary basin judging from ill-sorting and low degree of roundness of constituent particles in the sandstone. Feldspars are mostly fresh and thus the weather is considered to had been continuously cold from the glacial age of the Dwyka epoch.

Trace fossils of U-shaped burrows are occasionally observed in the Upper Sandstone and Upper Transition Beds, and this suggests the sedimentary environment stated above. The Upper Transition Beds locally intercalates with thin calcareous beds and this may indicate transgression for very short periods.

The sedimentary environment stated above is summarized below.

<table>
<tr><th colspan="2">Name of strata</th><th>Sedimentary facies</th></tr>
<tr><td colspan="2">Upper Ecca Formation</td><td>Lacustrine, swamp sediments (deposition of coal seams)</td></tr>
<tr><td rowspan="5">Middle Ecca Formation</td><td>Upper Transition Beds</td><td>Prodelta facies</td></tr>
<tr><td>Upper Sandstone</td><td>Delta-front facies</td></tr>
<tr><td>Lower Coal Zone</td><td>Delta plain facies and alluvial plain facies
(deposition of coal seams)</td></tr>
<tr><td>Basal Sandstone</td><td>Delta-front facies</td></tr>
<tr><td>Lower Transition Beds</td><td>Prodelta facies</td></tr>
<tr><td colspan="2">Lower Ecca Formation</td><td>Delta shelf facies</td></tr>
</table>

2.6.2 Sedimentary Environment of the Middle Ecca Formation

The sedimentary environment of the beds from the Lower Coal Zone to the Upper Transition Beds of the Middle Ecca Formation have the following characteristics:

With respect to the sand-shale ratio, stable sedimentary units (sedimentary basins) in the Lower Coal Zone are recognized in the north and south of the DD10 to DD52 line. The sand percentage of thse units are 85 to 95% in the lower part (from the lower boundary of the Zone to the Top Marker) and 75 to 85% in the upper part (from the Top Marker to the upper boundary), which are relatively high but almost constant. The formation in the units represents structurally stable delta plain facies to alluvial plain facies. The units provided favourable places for the deposition of coal seams, and the lower part of the Zone is more stable (Figure II-12, 13).

These sedimentary units become slightly unstable in the upper part of the Lower Coal Zone and they finally disappeared before deposition of the Upper Transition Beds, becoming irregular during deposition of the Upper Sandstone. The sand percentage of the Upper Sandstone and Upper Transition Beds are 60 to 90% and 30 to 80% respectively. The sand percentage decreases gradually because of a relative increase of shale. Areal variation of the sand-shale ratio of these beds is remarkably large and no sedimentary basin for coal deposition is found in these beds (Figure II-14, 15). The supply source of sediments judging from the sand-shale ratio is considered to have been the highland in the west (bedrocks of Precambrian era).

With respect to the coal ratio (percentage of coal seams in sedimentary beds), high coal ratios of 5.5 to 10% in the lower part of the Lower Coal Zone are found in the northern sedimentary unit, where the present drilling investigation was conducted. Higher coal ratios are distributed in the form of a C-shape in the unit. This reflects the stable sedimentary basin indicated by the sand-shale ratios, in which major coal seams such as the Main Seam were deposited extensively. The coal ratios in the southern sedimentary unit are relatively low, 4.0 to 6.5%, and deposition of the coal seams there was inferior as compared with that in the northern part (Figure II-16). On the other hand, the coal ratio of the upper part of the Lower Coal Zone is low, 1.5 to 5.0%, in both sedimentary units due to the unstable condition of the sedimentary basin, and deposition of the coal seams there is considerably inferior as compared with that of the lower part (Figure II-17).

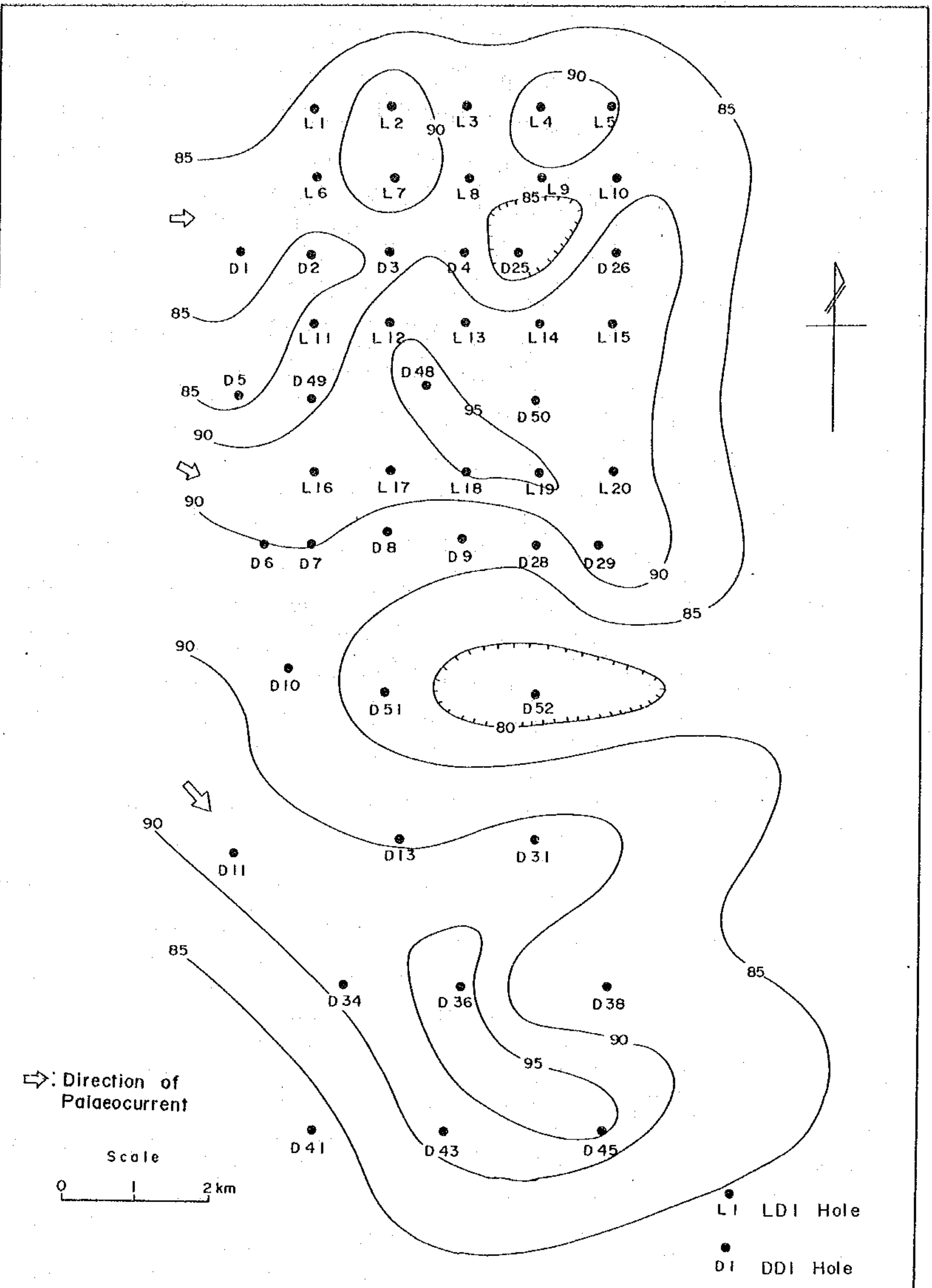


Figure II-12 Sand-Shale Ratio, Lower Part of Lower Coal Zone, Middle Ecca
(Showing sandstone in percent)

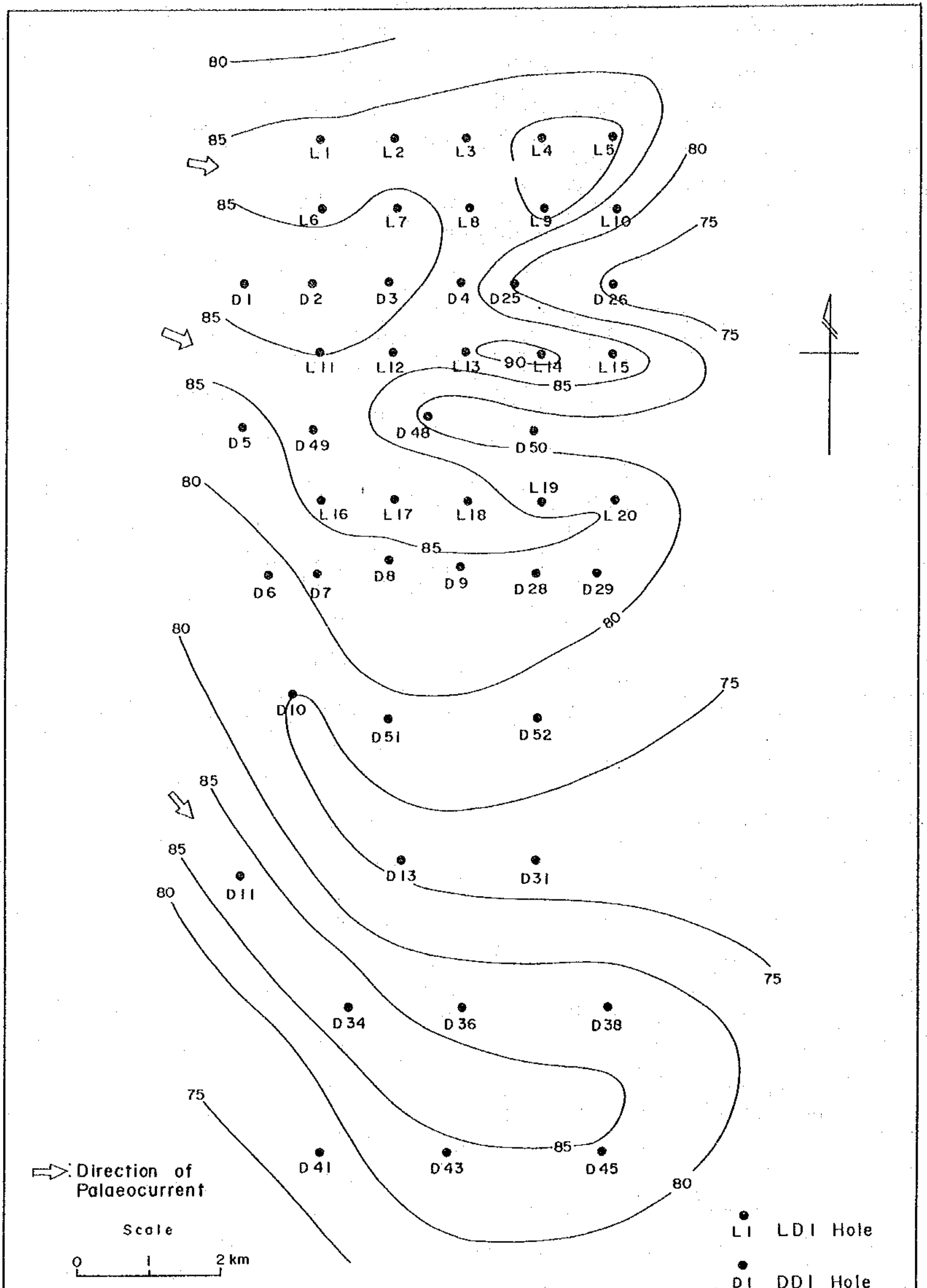


Figure II-13 Sand-Shale Ratio, Upper Part of Lower Coal Zone, Middle Ecca

(Showing sandstone in percent)

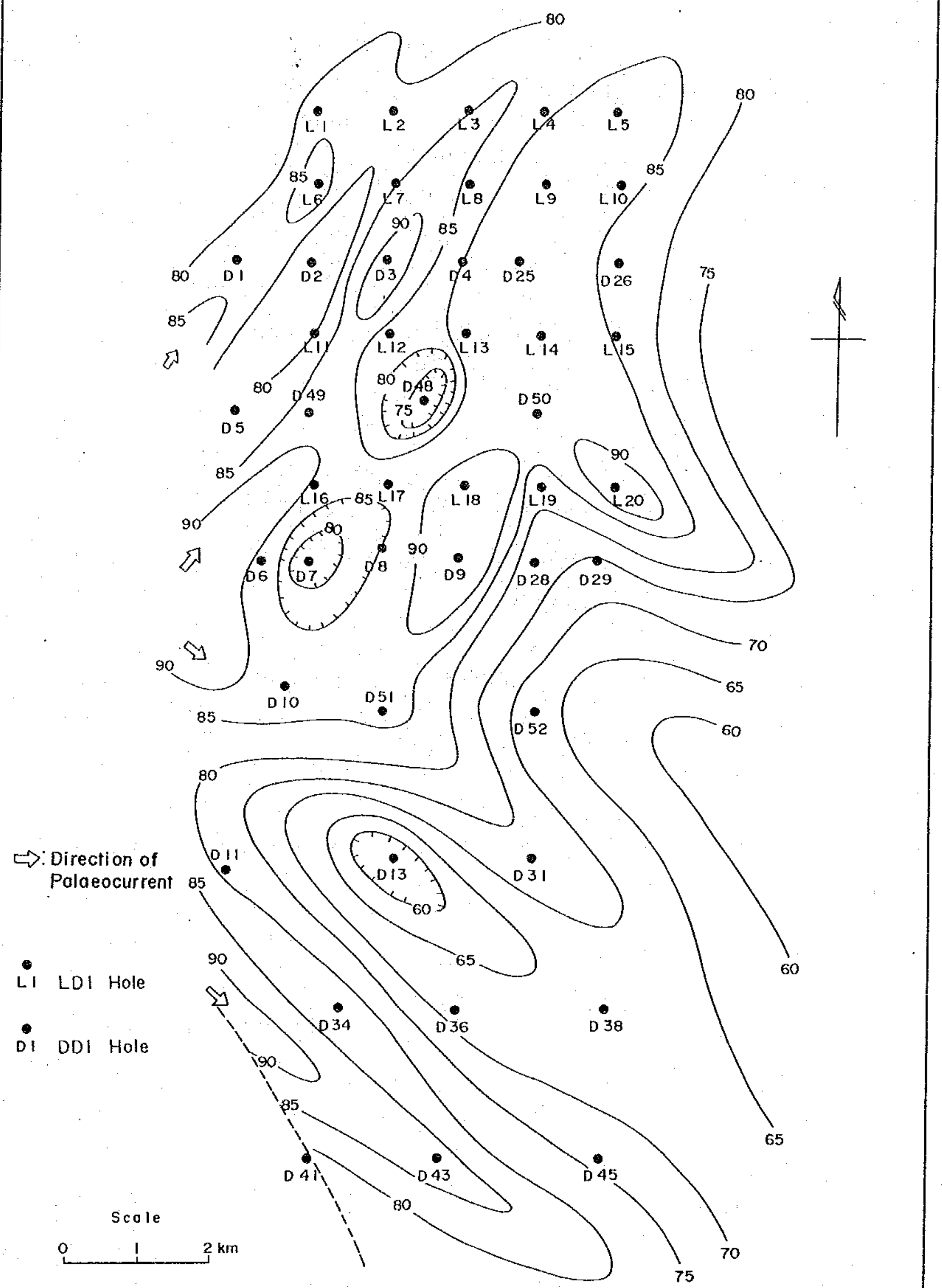


Figure II-14 Sand-Shale Ratio, Upper Sandstone, Middle Ecca
(Showing sandstone in percent)

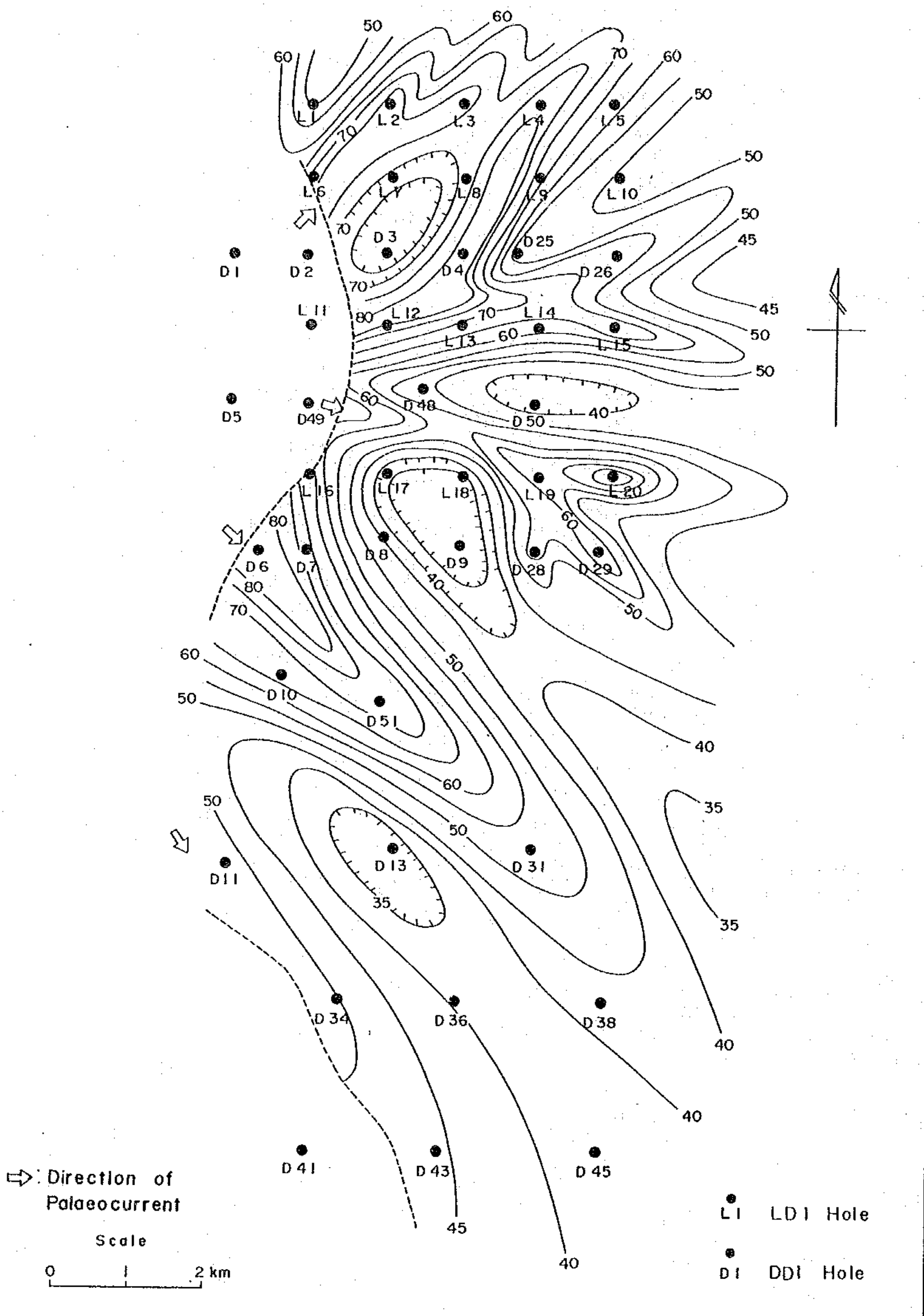


Figure II-15 Sand-Shale Ratio, Upper Transition Beds, Middle Ecca
(Showing sandstone in percent)

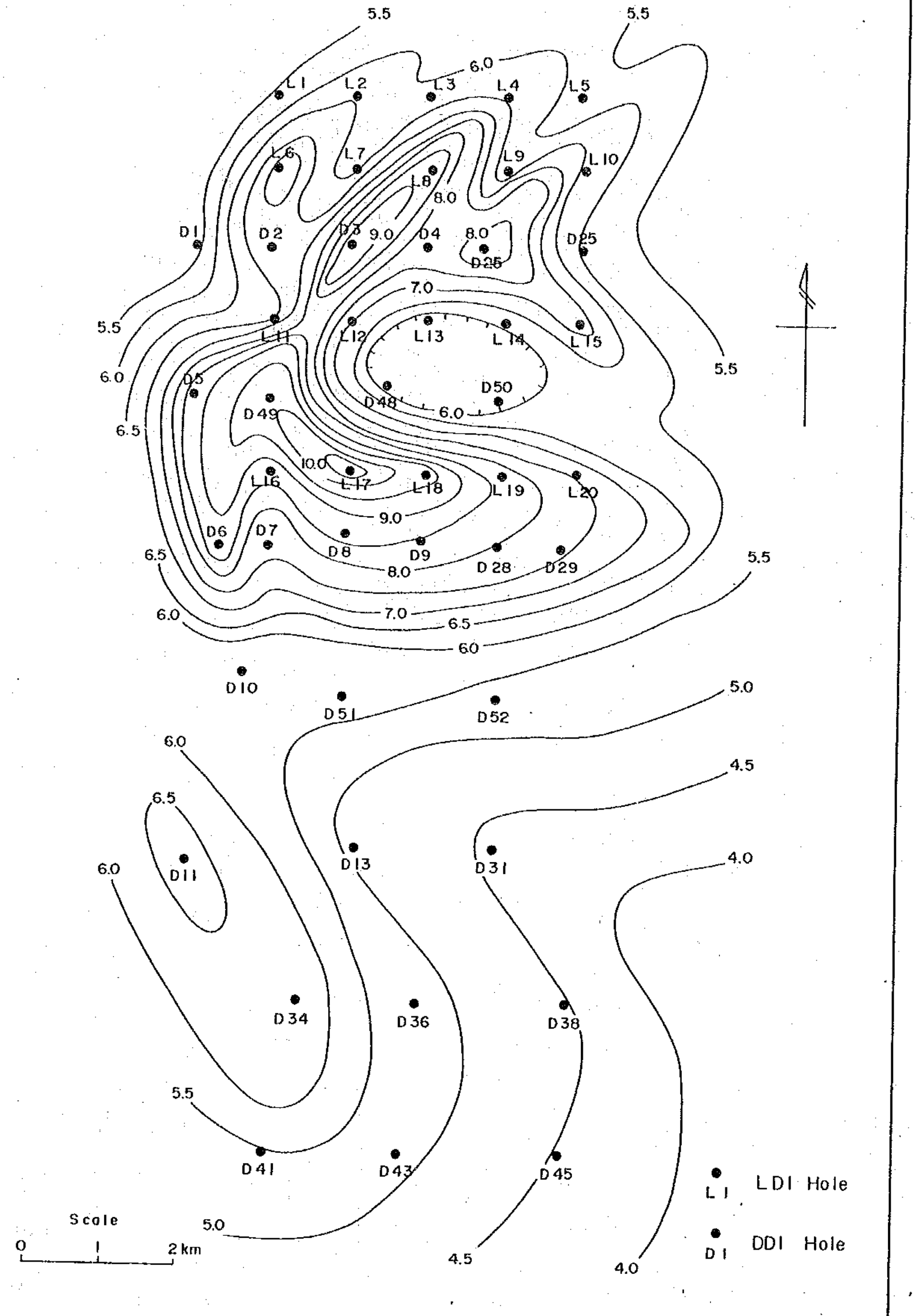


Figure II-16 Coal Ratio, Lower Part of Lower Coal Zone, Middle Ecca

(Showing coal in percent)

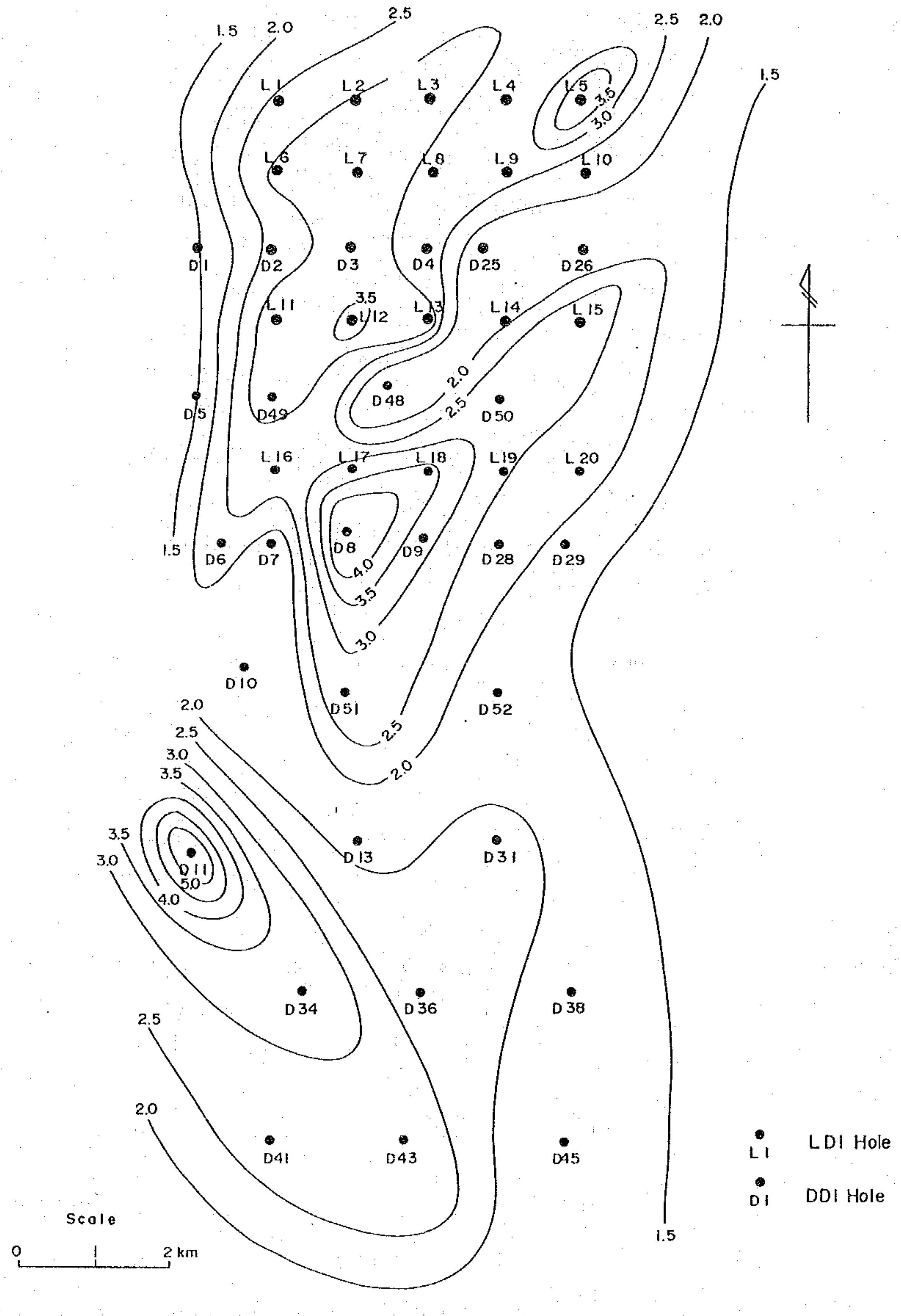


Figure II-17 Coal Ratio, Upper Part of Lower Coal Zone, Middle Ecca (Showing coal in percent)

CHAPTER 3. MAGNETOMETRY SURVEY AND GEOPHYSICAL LOGGING

CHAPTER 3. MAGNETOMETRY SURVEY AND GEOPHYSICAL LOGGING

3.1 Magnetometry Survey

Magnetic intensity was measured using a Chemtron G3 proton magnetometer at 5 m intervals on both 200-m lines in east-west and north-south directions crossing at the planned drilling site. When a remarkable magnetic anomaly had been found on the planned site, further magnetometry surveys were performed in the same manner along lines 50 m apart from the original lines. Based on this data, the actual drilling site was selected in order to drill the minimum amount of dolerite.

Drilling sites of LD3, 5, 11 and 14 were shifted from the planned sites because intrusion of dolerite was expected. A dolerite dyke which has an almost vertical dip was expected at the planned site of LD5, and the drilling was performed at 130 m west of the planned site where no magnetic anomaly was recognized (Figure II-18). As a result, the dolerite was not penetrated up to the depth of 318 m. Dolerite dykes dipping steeply east were expected at the planned site of LD14 and at about 70 m west of it. The hole was drilled at 100 m west of the planned site, and no dolerite was penetrated up to the depth of 316 m (Figure II-19).

On the other hand, dolerite intrusions occasionally appeared in boreholes even though the magnetic anomaly was hardly recognized on the surface. Several dolerite sills and small dykes were found at the depth of 60 to 380 m in LD4, and thick dolerite sills and dykes appeared at the depth of 10 to 280 m in LD7.

Magnetic intensity was measured on 3 lines in the east-west direction extending for 4 km in the area of present investigation. No intensive magnetic anomaly was recognized in the western part, however, several magnetic anomalies suggesting dolerite dykes were recognized in the eastern part (Figure II-20). On the other hand, no significant magnetic anomaly was recognized around LD7 where the dolerite sills were drilled near the surface.

As stated above, it is almost impossible to forecast the intrustion of a dolerite sill by magnetometry survey, but it is possible to identify a dolerite dyke near the surface. Distribution of dolerite dykes can often be determined through systematic magnetometry survey.

3.2 Geophysical Logging

Upon completion of drilling, geophysical logging was performed in several boreholes (carried out by BPB Instruments Ltd.). Two resistivity logs were added to five kinds of previous logging items. Seven logging items are caliper, gamma ray, neutron-neutron, long spacing density (LSD), bed resolution density (BRD), single point resistivity, and micro resistivity.

As a result of the geophysical logging, the following characteristic log responses were recognized for coal seam and dolerite:

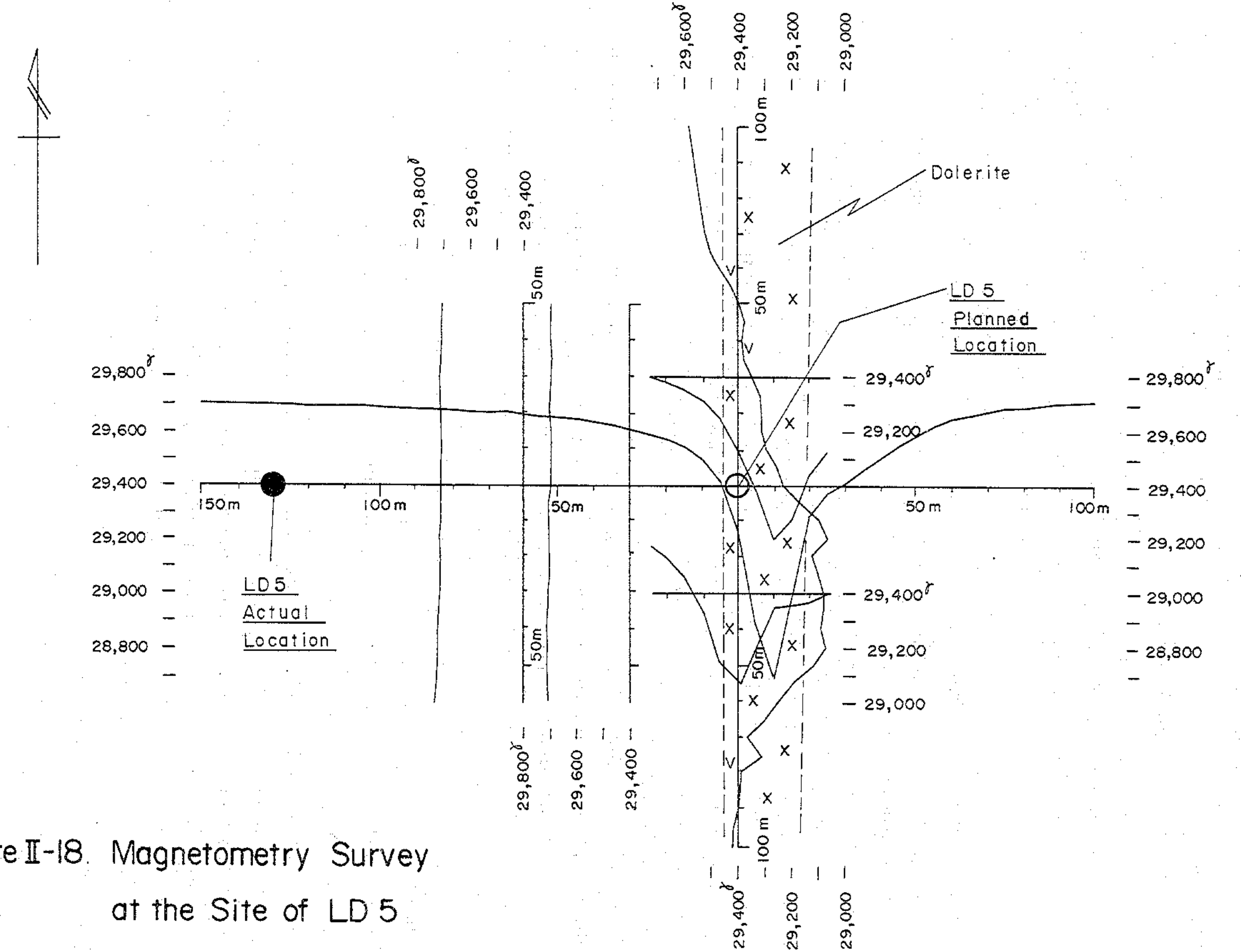


Figure II-18. Magnetometry Survey at the Site of LD 5

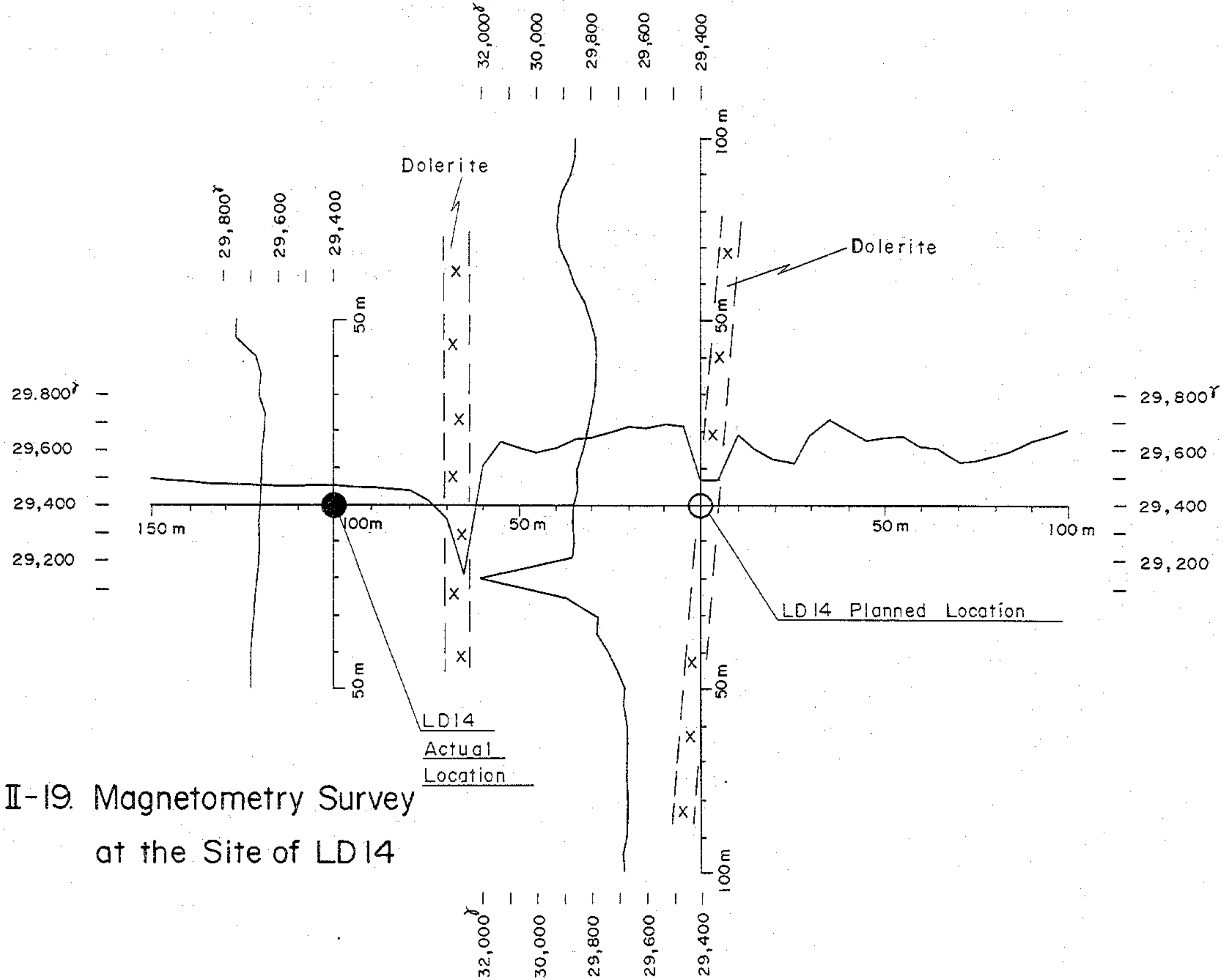


Figure II-19 Magnetometry Survey at the Site of LD14

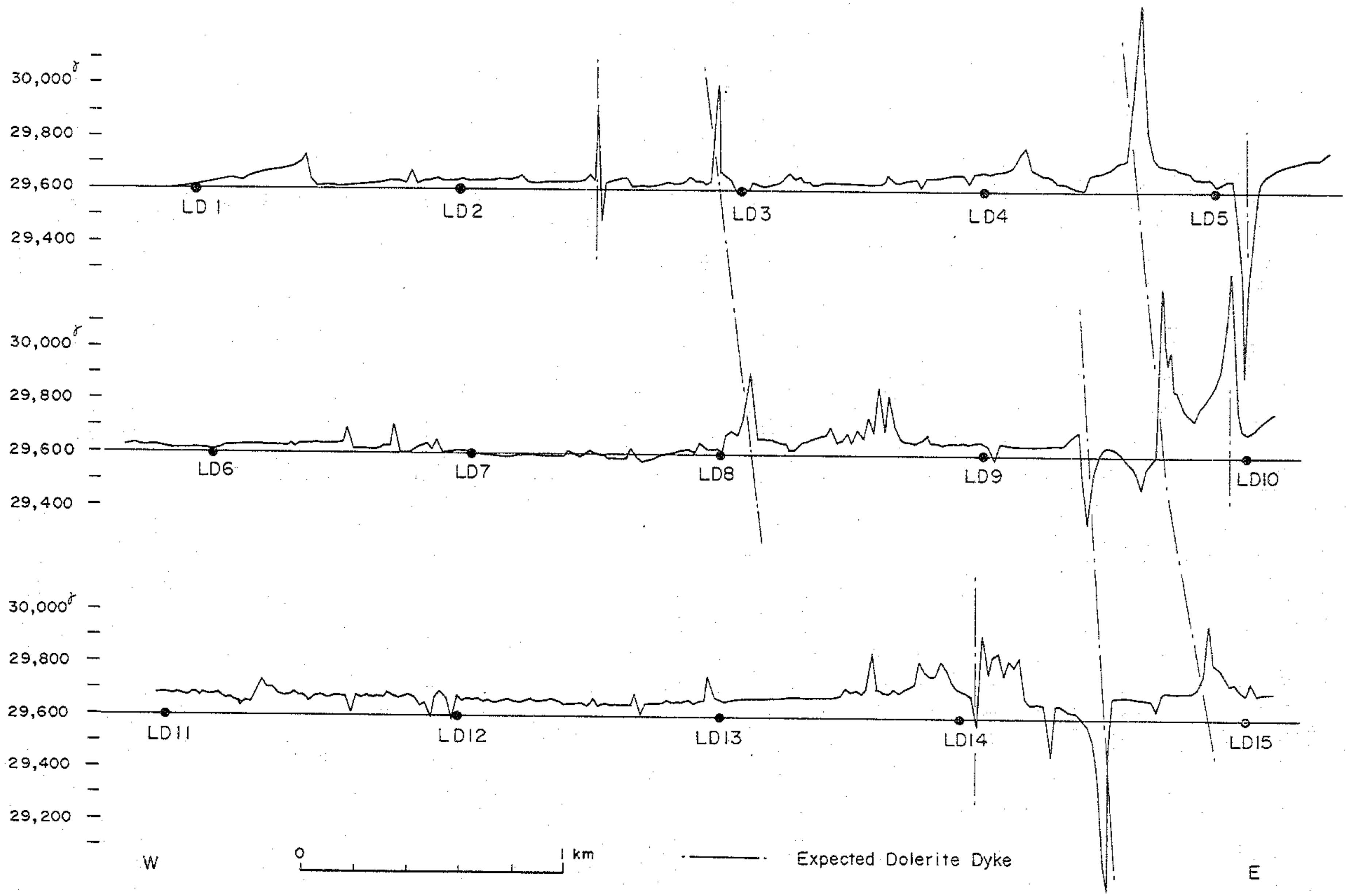


Figure II-20 Magnetometry Survey along E-W Lines

Coal seam:

Gamma ray log: Very low radioactivity.

Density log: Very low bulk density, normally below 1.7 g/cm^3.

Neutron log: Very low value (high porosity).

Resistivity log: High value.

Dolerite:

Gamma ray log: Very low radioactivity.

Density log: Considerably high bulk density, normally over 2.8 g/cm^3.

Neutron log: Stable medium to slightly high value (low porosity).

Resistivity log: Very high value.

The results of the geophysical logging for the horizon from the Main Seam to Footwall 3 in LD15 are shown in comparison with the geologic borehole log in Figure II-21, in which the coal seam and dolerite show significant log responses. Coarse-grained sandstone above the Main Seam shows almost stable log responses.

On the other hand, in the composite log of gamma ray and neutron-neutron for the Middle Ecca Formation, the following characteristic log responses were found at a particular horizon (Figure II-22):

Shale zone (Zone 2) of the Upper Ecca Formation:

Gamma ray log: Stable, slightly high value.

Neutron log: Stable, low value.

Shale bed at the lowermost portion of the Upper Transition Beds:

Gamma ray log: Stable, slightly high value.

Neutron log: Stable, medium value.

Coarse-grained sandstone bed just above the Intermediate Marker and Main Seam:

Gamma ray log: Stable, medium value.

Neutron log: Stable, slightly high value.

Sandstone bed at the uppermost portion of the Basal Sandstone:

Gamma ray log: Stable, medium value.

Neutron log: Stable, slightly high value.

By reviewing the cross plot of gamma ray and bed resolution density logs for the Lower Coal Zone, apparent characteristics depending on the rock types can be recognized as shown in Figure II-23. Bed resolution density for both sandstone and shale is 2.50 to 2.78 g/cm^3, and gamma ray is mostly higher than 200 API for shale and less than 150 API for sandstone. In case a coal seam contains a shaly parting or impurities, specific gravity of the coal increases toward the region of shale, and in case of sandy materials, the specific gravity approaches the region of sandstone.

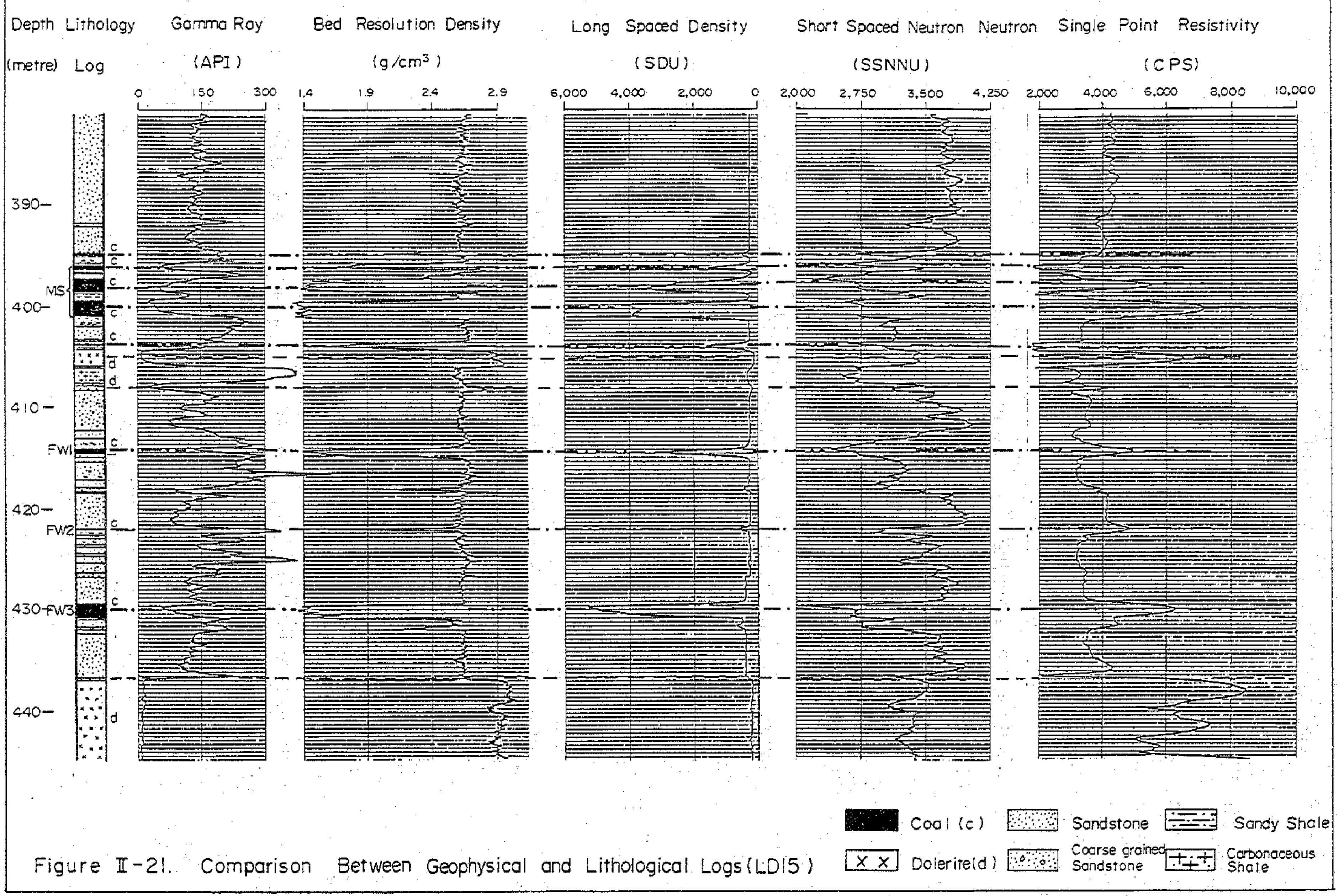
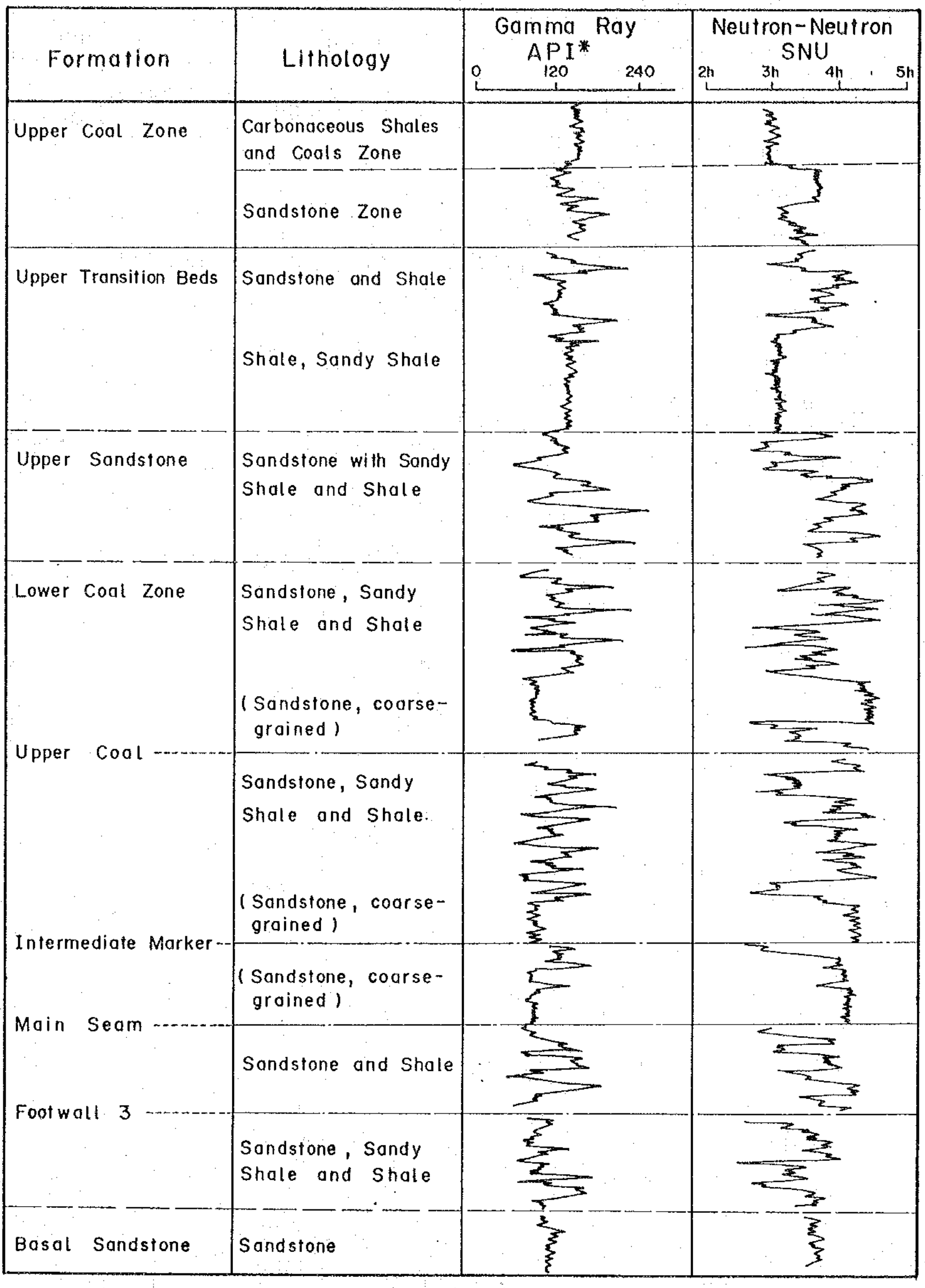


Figure II-21. Comparison Between Geophysical and Lithological Logs (LD15)

Figure II-22 Schematic Profile of Geophysical Logging



Remarks : * Hole Size Corrected

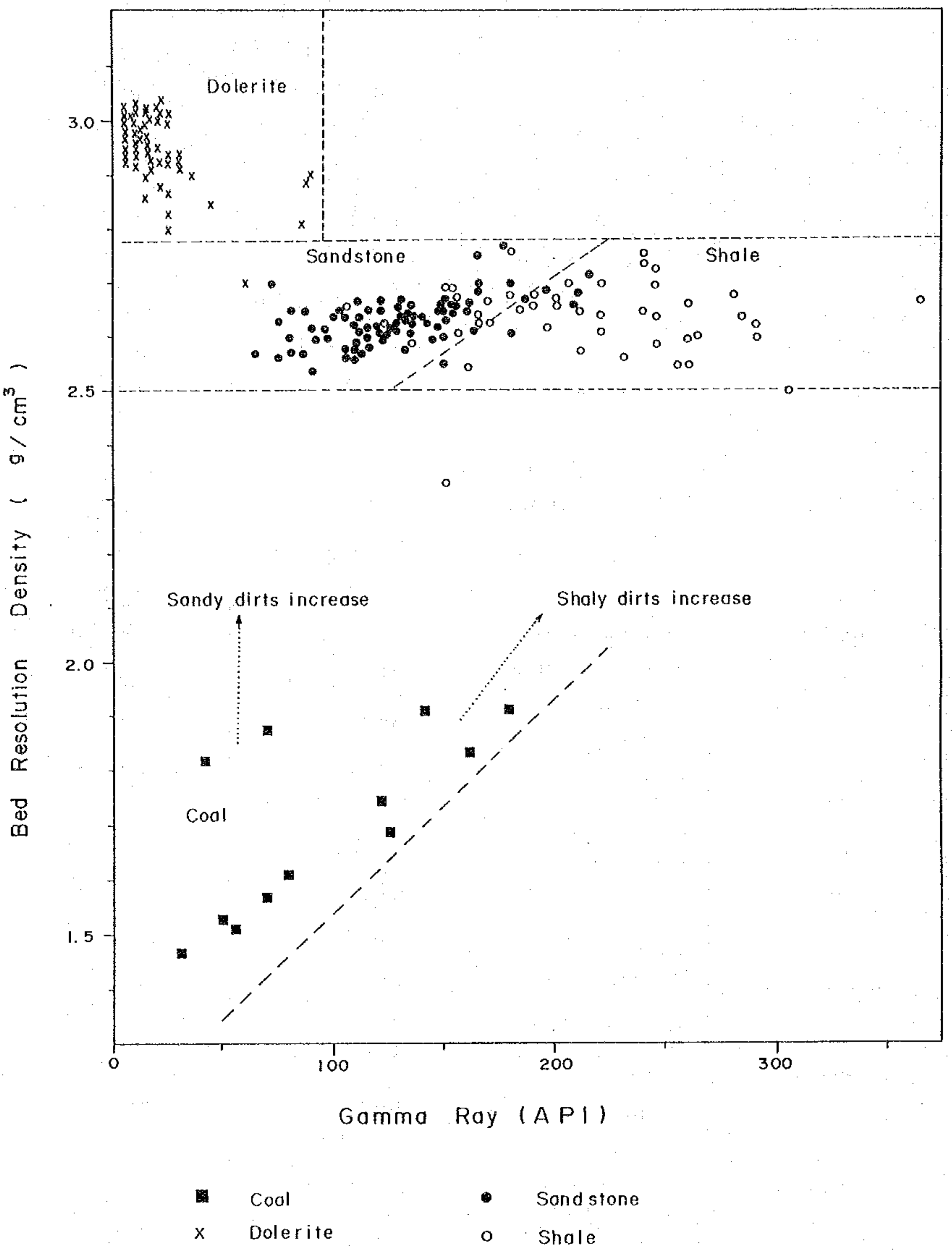


Figure II-23 Cross Plot of Bed Resolution Density and Gamma Ray, Lower Coal Zone (LD15)

As stated above, the geophysical logging is effective in identifying coal seam and dolerite. Moreover, the geophysical logging can be employed as a useful means for the stratigraphic correlation of the beds in the Upper and Middle Ecca Formations. Drawing 11 shows typical characteristic log responses for the Upper and Middle Ecca Formations.

CHAPTER 4. COAL QUALITY AND RESERVES

CHAPTER 4. COAL QUALITY AND RESERVES

4.1 Coal Quality

Analysis of coal samples taken mainly from the Intermediate Marker, Main Seam and Footwall 3 among the major coal seams was performed by the Swaziland Government (analysed by the National Institute for Coal Research, South Africa). The total number of coal samples for analysis was 51. The whole core of the coal seams was sampled for analysis and samples were divided into over two portions in case of thick coal seams.

Items of analysis were proximate analysis, calorific value, total sulphur, sink-and-float test, hard grove grindability index, ash fusion temperature, ultimate analysis, petrographic analysis and reflectance. The results of these analyses are shown in Drawings 12a to 12d.

The results of these analyses are explained below in detail together with the results of analyses performed in the previous investigation.

4.1.1 Analysis of Coal Properties

A. Proximate Analysis and Calorific Value

Average coal quality on dry-ash-free basis of three coal seams, the Intermediate Marker, Main Seam and Footwall 3 distributed in the investigated area, is as shown below. Coal samples, which showed a yield of less than 50% at specific gravity 1.8 in the sink-and-float test, were excluded in the said calculation as these are considered to be apparently affected by dolerite.

Name of Coal seam	Calorific value (Kcal/kg) [MJ/kg]		Fixed carbon (%)		Volatile matter (%)	
Intermediate Marker	8,218 [34.4]	<17>	90.1	<21>	9.9	<21>
Main Seam	8,264 [34.6]	<15>	90.2	<27>	9.8	<27>
Footwall 3	8,266 [34.6]	<11>	90.1	<24>	9.9	<24>

Note: Number of samples are shown in < >.

Based on the calssification of ASTM (American Society for Testing and Materials), most of these coals are semi-anthracite except some of them are anthracite.

Coal quality maps (equal ash-equal volatile matter map) of the Intermediate Marker, Main Seam and Footwall 3 in the investigated area are shown in Drawings 13a to 13c. Ash content of the Intermediate Marker is low in the western part and tends to increase toward the east, and the volatile matter is generally high in the western part and tends to decrease toward the east. The Main

Seam has high ash content in sub-parallel regions trending east-west, however, volatile matter content was found to show no specific trend. The Footwall 3 has low ash content part elongated in NE-SW direction north of the "A" Fault, and high ash content parts are found at both sides of it. The volatile matter content is high in the southwestern part and tends to decrease toward the surrounding area.

The total sulphur content is low as under 1% except in the Intermediate Marker at DD6 (1.67%), the Footwall 3 at DD8 (1.37%) and the Footwall 3 at LD18 (1.25%).

B. Ultimate Analysis

Based on the results of ultimate analysis, the relation between O/C-C and H/C-C is as shown in Figure II-24 and 25. The relation of O/C-C is almost stable, although in some coal samples it deviates from general tendency. On the other hand, with respect to the relation of H/C-C, the value of H/C tends to become low as compared with general tendency. These deviations from general tendency are considered to be caused by the intrusion of dolerite.

C. Ash Fusion Temperature

The ash fusion temperature of the Intermediate Marker is commonly 1,240 to +1,400°C, commonly 1,290 to +1,400°C for the Main Seam and over 1,400°C in all cases for the Footwall 3.

4.1.2 Sink-and-Float Test

The sink-and-float test was done on 15 samples of the Intermediate Marker, 20 of the Main Seam and 14 of the Footwall 3. Samples were crushed to the size between +1/2mm and–25 mm, and then the washability was checked at the specific gravity of 1.5, 1.6 and 1.8. The yield and ash content for the fractions of F1.6, S1.6–F1.8 and F1.8 are shown in Table II-7. On the basis of the above results and those obtained in the previous investigation, average ash content on raw coal basis of the Intermediate Marker, Main Seam and Footwall 3 in the investigated area is calculated at 22.3%, 22.8% and 25.3% respectively. In case samples are separated at the specific gravity of 1.6, average yield of the Intermediate Marker is 81% with average ash content of 16.8%. Average yield of the Main Seam is 78% with average ash content of 13.9%. Average yield of the Footwall 3 is 61% with average ash content of 14.7%.

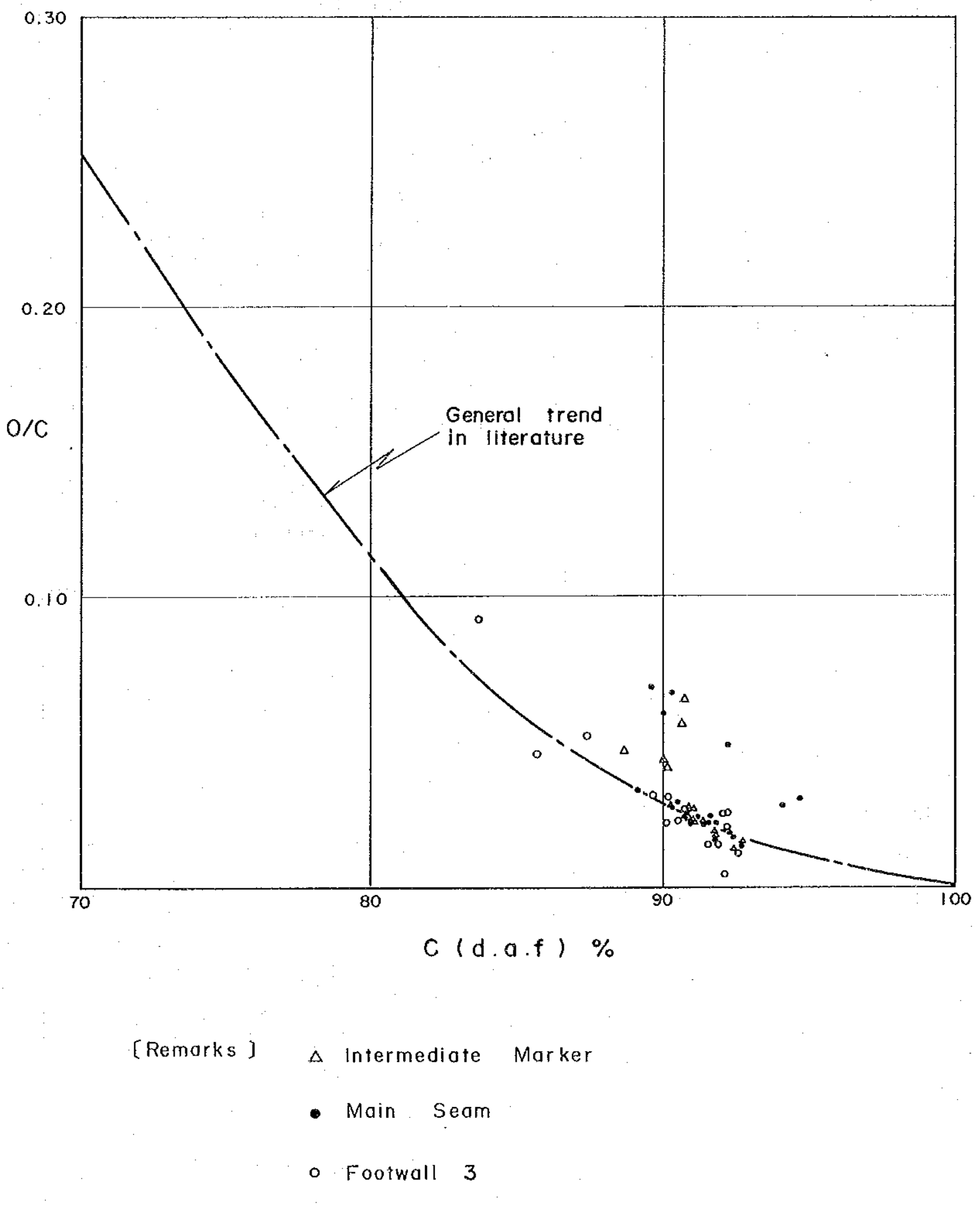
4.1.3 Petrographic Analysis and Reflectance

Results of petrographic analysis of coal show the common characteristics that the exinite group is hardly recognized, and the content of inertinite group as an inactive component is very high at 34.3% in average (Table II-8).

Mean maximum reflectance of vitrinite is low in the coals in the western part of the investigated area and tends to slightly increase toward the east. The reflectance tends to increase remarkably in the cases where dolerite intrusions occur in the vicinity.

The relation between the mean maximum reflectance and "fixed carbon/fixed carbon +

Figure II-24 Relation Between O/C Ratios (in number of atoms) and Carbon Contents (d. a. f.)



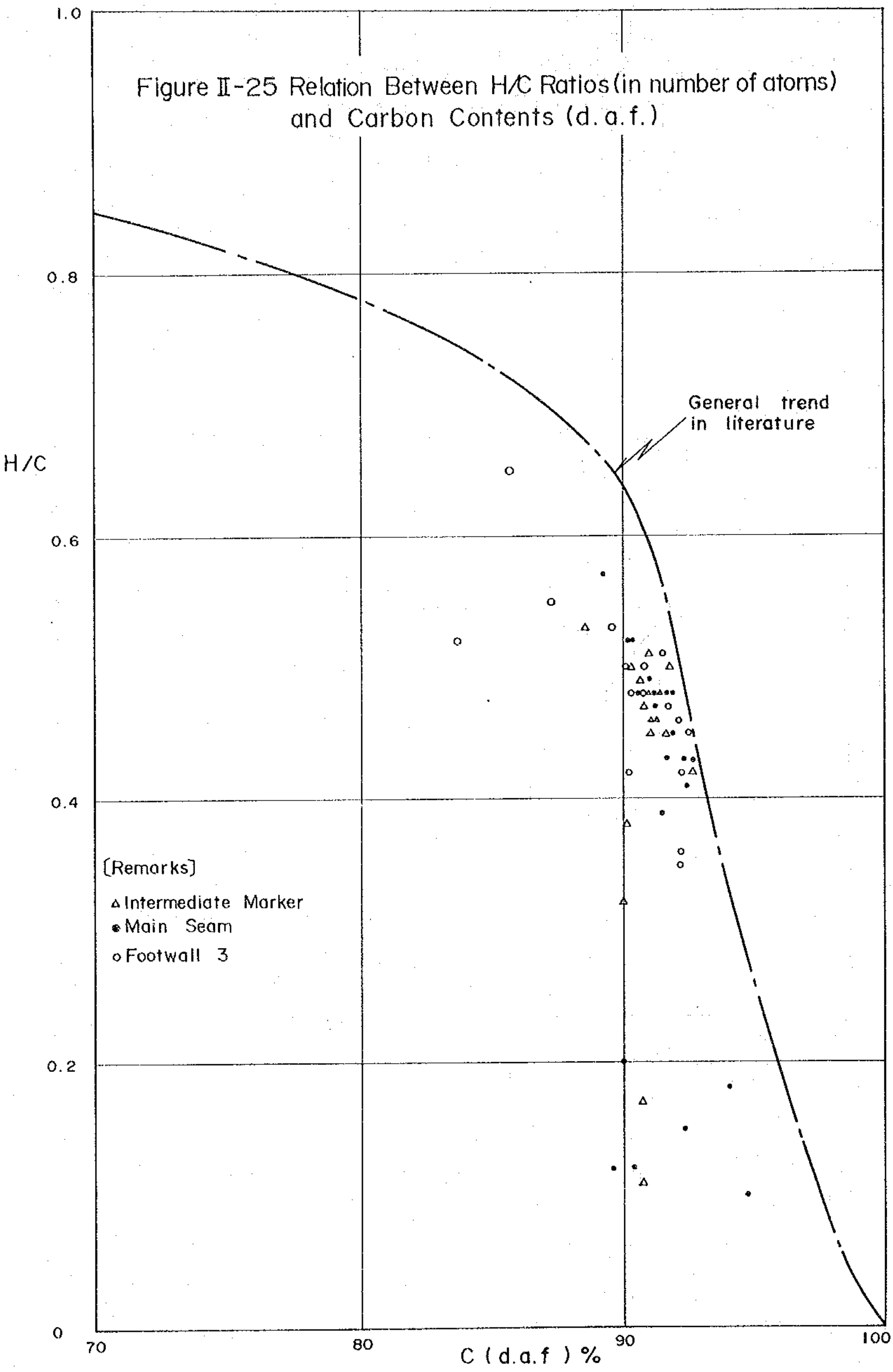


Figure II-25 Relation Between H/C Ratios (in number of atoms) and Carbon Contents (d.a.f.)
General trend in literature
H/C
1.0
0.8
0.6
0.4
0.2
0
70
80
90
100
C (d.a.f) %
[Remarks]
Intermediate Marker
Main Seam
Footwall 3

Table II-7a Result of Sink and Float Analysis

Coal Seam	Fraction / Sample No.	F1.6		S1.6 - F1.8		F1.8	
		Weight (%)	Ash (%)	Weight (%)	Ash (%)	Weight (%)	Ash (%)
Intermediate Marker	LD2-1	76.7	17.1	22.7	-	99.4	-
	LD3-1	90.3	18.0	9.0	-	99.3	-
	LD5-1	76.9	17.5	21.8	31.0	98.7	20.5
	LD6-1	98.6	-	1.4	-	100.0	-
	LD8-1	0.0	0.0	8.7	13.4	8.7	13.4
	LD9-1	28.0	15.7	59.6	25.6	87.6	22.4
	LD11-1	88.7	15.4	10.1	-	98.8	17.1
	LD12-1	96.5	16.5	3.3	-	99.8	-
	LD13-1	0.0	0.0	87.5	24.0	87.5	24.0
	LD14-1	30.2	14.8	12.2	35.4	42.4	20.8
	LD16-1	95.8	17.2	3.9	-	99.7	-
	LD17-1	87.8	16.7	7.4	34.1	95.2	18.0
	LD18-1	72.6	16.4	10.3	32.8	82.9	18.5
	LD19-1	88.6	17.2	10.0	-	98.6	-
	LD20-1	71.7	19.4	13.6	32.7	85.3	21.5
Main Seam	LD6-2	73.3	11.9	17.7	24.6	91.0	14.4
	LD8-2	0.0	0.0	19.0	14.6	19.0	14.6
	LD11-2A	92.8	20.3	5.3	30.4	98.1	20.8
	LD11-2B	0.0	0.0	0.0	0.0	0.0	0.0
	LD12-2A	0.0	0.0	9.1	18.4	9.1	18.4
	LD12-2B	0.0	0.0	15.1	-	15.1	-
	LD12-2C	0.0	0.0	0.0	0.0	0.0	0.0
	LD13-2A	0.0	0.0	86.2	20.9	86.2	20.9
	LD13-2B	0.0	0.0	35.9	25.4	35.9	25.4
	LD15-2A	58.6	14.2	25.7	27.7	84.3	18.3
	LD15-2B	95.2	10.8	4.0	30.5	99.2	11.6
	LD16-2	46.4	17.0	21.5	39.0	67.9	24.0
	LD17-2A	67.3	15.4	20.6	35.4	87.9	20.1
	LD17-2B	87.6	12.1	8.8	33.8	96.4	14.1
	LD18-2A	69.6	14.7	13.2	34.0	82.8	17.8
	LD18-2B	84.1	13.1	9.4	37.1	93.5	15.5
	LD19-2A	70.2	14.0	20.1	36.6	90.3	19.0
	LD19-2B	72.1	12.5	8.8	33.7	80.9	14.8
	LD20-2A	77.2	18.3	14.0	33.2	91.2	20.6
	LD20-2B	80.5	12.7	6.3	30.6	86.8	14.0

* Size of coal washed 1/2 mm - 25 mm
* Tested by National Institute for Coal Research

* **Table II-7b Result of Sink and Float Analysis**

Coal Seam	Fraction / Sample No.	F1.6 Weight (%)	F1.6 Ash (%)	S1.6 - F1.8 Weight (%)	S1.6 - F1.8 Ash (%)	F1.8 Weight (%)	F1.8 Ash (%)
Footwall 3	LD2-3	12.2	14.6	6.6	37.1	18.8	22.5
	LD3-3	53.5	14.9	10.8	33.1	64.3	18.0
	LD4-3	58.1	14.3	16.3	33.1	74.4	18.4
	LD5-3	90.4	13.7	6.0	27.6	96.4	14.6
	LD6-3	43.2	15.0	16.2	27.6	59.4	18.4
	LD7-3	42.2	14.0	46.3	23.9	88.5	19.2
	LD8-3	48.7	16.5	5.6	32.0	54.3	18.1
	LD11-3	75.4	14.2	17.9	33.6	93.3	17.9
	LD14-3	26.1	14.0	59.0	24.3	85.1	21.2
	LD15-3	75.1	13.6	19.0	28.4	94.1	15.7
	LD17-3	82.5	15.5	6.5	31.3	89.0	16.7
	LD18-3	70.5	16.7	9.7	28.6	80.2	18.2
	LD19-3	61.2	13.6	24.9	31.3	86.1	18.7
	LD20-3	69.2	15.7	19.6	31.7	88.8	19.2

* Size of coal washed 1/2 mm - 25 mm
* Tested by National Institute for Coal Research

Table II-8a Petrographic Analysis

Sample No.	LD2-1	LD2-3	LD3-1	LD3-3	LD4-3	LD6-1	LD6-2	LD6-3	LD7-3	LD8-1	LD8-2	LD8-3	LD9-1	LD12-1	LD12-2A	LD12-2B	LD12-2C	LD13-1	LD13-2A	LD13-2B
* Coal Seam	I	F	I	F	F	I	M	F	F	I	M	F	I	I	M	M	M	I	M	M
Maceral Analysis																				
Vitrinite	27.3	9.0	25.4	39.5	43.6	24.1	25.2	32.6	21.4	16.2	30.1	32.4	18.7	24.6	32.2	34.3	22.6	7.9	21.0	18.8
Exinite	4.6	0.2	1.9	0.4	0.5	0.3	17.6	5.0	2.0	2.8	1.9	0.7	0.9	0.2	1.6	1.5	1.0	4.8	2.8	2.5
RSF	24.4	8.1	37.3	21.7	20.5	44.7	17.4	13.2	15.8	17.3	12.2	16.7	7.4	41.8	6.5	10.2	14.9	18.6	8.6	10.1
Semifusinite, Macrinite, Micrinite, Sclerotinite } Inert	41.9	13.0	34.1	23.4	26.4	27.5	38.8	37.2	56.5	57.1	47.5	19.7	65.0	32.0	27.2	37.0	50.5	66.4	64.0	47.9
Visible Minerals	1.8	69.7	1.3	15.0	9.0	3.4	1.0	12.0	4.3	6.6	8.3	30.5	8.0	1.4	32.5	17.0	11.0	2.3	3.6	20.7
Reflectance Mesurements																				
% R̄o	3.586	2.158	2.475	2.439	2.848	2.087		3.844	3.971	5.554	5.563	2.378	4.197	2.348	5.452	6.393	5.129	5.475	4.908	4.665
% Ro max.								7.35	4.85	6.80	7.20		4.90		8.35	8.85	7.90	6.95	5.90	6.25
% Ro Deviation	0.243	0.231	0.213	0.200	0.229	0.149		0.795	0.395	0.525	0.611	0.235	0.447	0.164	0.852	0.979	1.326	0.585	0.468	0.471
% Ro Classes:																				
1.5																				
1.6		2										1								
1.7		2										1								
1.8		4				9		1				1								
1.9		10				9						1		1						
2.0		9	1	1	1	22						2		4						
2.1		16	8	11		22						8		7			1			
2.2		24	8	9		24		1				13		12			2			
2.3		18	15	13	1	11		1				19		24			2			
2.4		11	8	15	4	2						14		29			1			
2.5		1	19	18	6	1		1				17		13						
2.6			20	13	5							7		7			1			
2.7			12	16	13			1				11		2						
2.8		2	7	4	13							5		1						
2.9			2		17			1					1							
3.0	2				21			5	2						1					
3.1	2	1			15			2	1				1				3			
3.2	7				4			6	1											
3.3	10							8	3				2				3			
3.4	7							4	3				4			1				
3.5	12							7	4				4							
3.6	14							10	5				2				2			
3.7	15							4	11		1		2			2	2			
3.8	16							4	4				6		1					
3.9	11							4	9				5		1	2			2	
4.0	4							4	14				5		3	2	5			
4.0 -								36	43	100	99		68		95	93	78	100	98	100

* I ..... Intermediate Marker
M ..... Main Seam
F ..... Footwall 3

Analyzed by National Institute for Coal Research.

Table II-8b Petrographic Analysis

Sample No.	LD14-1	LD14-3	LD16-1	LD16-2	LD17-1	LD17-2A	LD17-2B	LD17-3	LD18-1	LD18-2A	LD18-2B	LD18-3	LD19-1	LD19-2A	LD19-2B	LD19-3	LD20-1	LD20-2A	LD20-2B	LD20-3
* Coal Seam	I	F	I	M	I	M	M	F	I	M	M	F	I	M	M	F	I	M	M	F
Maceral Analysis																				
Vitrinite	28.1	39.0	28.5	28.7	21.5	41.6	38.9	44.0	17.8	37.1	34.7	26.9	20.0	31.6	30.2	42.7	17.6	37.3	28.0	33.0
Exinite	0.5	0.6	0.2	0.3	0.1	0.4	0.1	0.0	0.0	1.2	0.5	0.0	0.3	0.6	0.6	0.0	0.0	0.0	0.0	0.5
RSF	9.9	15.9	45.1	33.7	42.1	31.2	31.8	11.6	28.0	30.9	26.9	34.0	50.7	33.4	35.8	26.0	48.6	31.4	27.2	32.1
Semifusinite } Inert	28.0	38.6	23.3	16.9	30.3	23.0	19.0	40.4	49.4	25.8	21.5	19.9	28.0	27.4	30.8	23.5	27.2	22.3	35.5	28.0
Macrinite																				
Micrinite																				
Sclerotinite																				
Visible Minerals	33.5	5.9	2.9	18.4	6.0	3.6	10.2	4.0	4.8	5.0	16.4	9.2	1.3	7.0	2.6	7.8	6.6	9.0	9.3	6.4
Reflectance Mesurements																				
% Ro	2.616	2.638	2.181	2.147	2.016	2.208	2.075	2.118	2.170	2.297	2.457	2.316	2.396	2.453	2.475	2.430	2.748	2.941	2.712	2.627
% Ro max.			min.1.2																	
% Ro Deviation	0.178	0.190	0.218	0.236	0.153	0.201	0.195	0.164	0.191	0.205	0.375	0.154	0.195	0.200	0.201	0.179	0.245	0.435	0.224	0.226
% Ro Classes:																				
- 1.5			2																	
1.5				1																
1.6						2	1													
1.7				2	4		1	1												
1.8			4	6	14	2	6	4	4											
1.9			5	8	11	6	12	11	10	4	4									
2.0			12	16	19	8	21	15	13	12	8	9	5	1						1
2.1			12	18	33	10	30	22	17	13	12	8	7	9	4	4			2	
2.2		1	29	22	15	26	18	25	20	13	10	8	13	6	7	15	2		3	5
2.3	12	3	19	14	3	25	9	17	17	12	18	28	13	13	20	17	5	3	2	7
2.4	9	12	11	7	1	15		2	13	13	10	27	20	19	10	12	8	3	5	8
2.5	12	20	3	3		4		3	5	23	5	19	19	18	21	25	8	7	6	13
2.6	15	16	2	1		1				8	1		16	13	22	14	9	7	10	13
2.7	23	15					1		1	2	2	1	5	16	9	9	13	8	25	18
2.8	21	18	1	1							7		1	4	5	3	18	10	20	18
2.9	7	11									7			1	1	1	15	24	12	13
3.0	1	3				1					8		1				12	14	9	4
3.1											6						7	10	6	
3.2		1															2	4		
3.3				1			1				2						1	1		
3.4															1			1		
3.5																				
3.6																		2		
3.7																				
3.8																				
3.9																		1		
4.0																		1		
4.0 -																		4		

* I ..... Intermediate Marker
M ..... Main Seam
F ..... Footwall 3

Analyzed by National Institute for Coal Research.

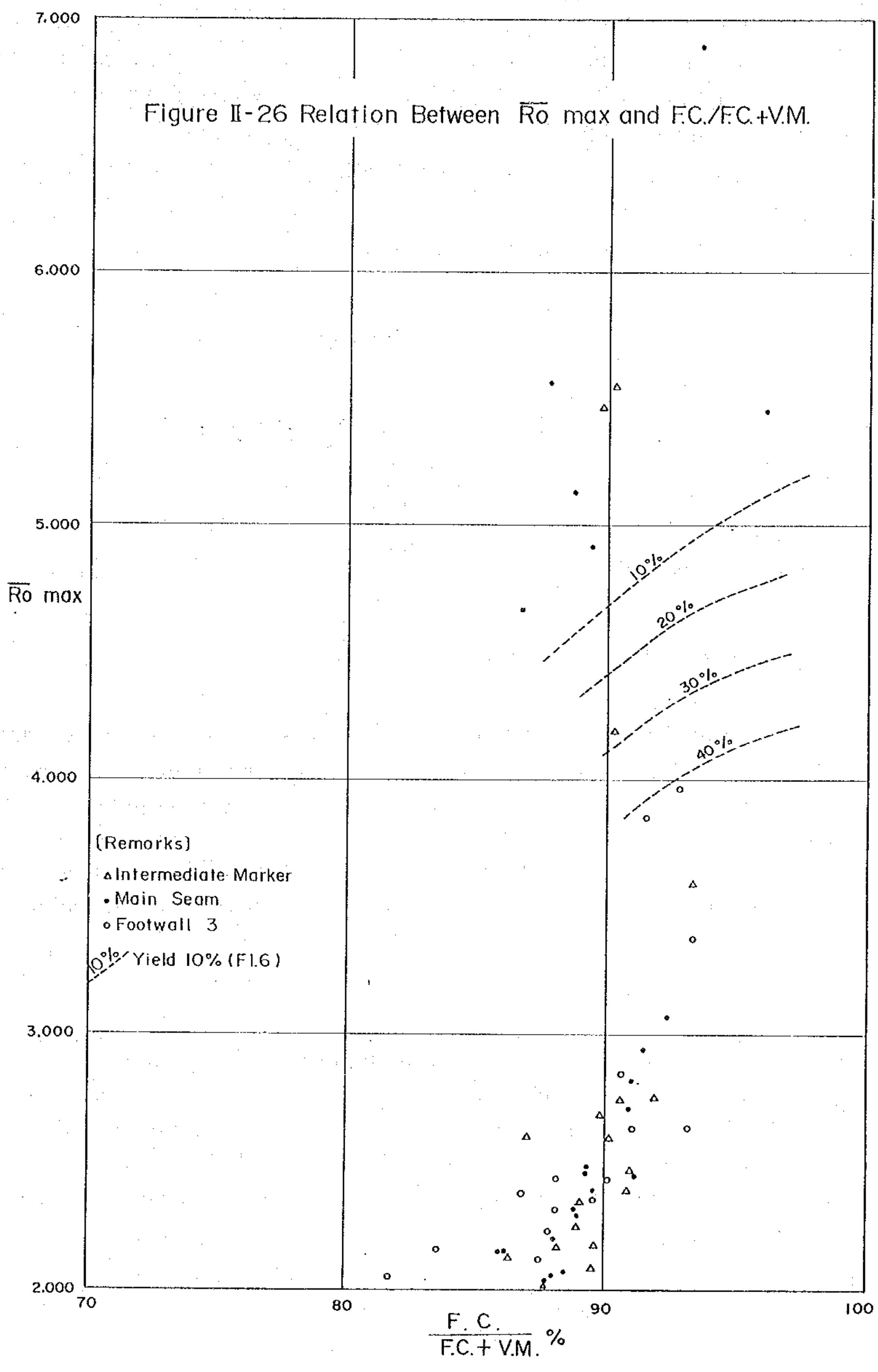


Figure II-26 Relation Between $\overline{Ro}$ max and F.C./F.C.+V.M.
7.000
6.000
5.000
4.000
3.000
2.000
$\overline{Ro}$ max
10%
20%
30%
40%
(Remarks)
△ Intermediate Marker
• Main Seam
○ Footwall 3
10% Yield 10% (F1.6)
70
80
90
100
$\frac{F.C.}{F.C.+V.M.}$ %

volatile matter" is shown in Figure II-26. In this figure, the boundary indicated by a broken line shows the yield at the specific gravity of 1.6 in the sink-and-float test. The mean maximum reflectance tends to increase as the "fixed carbon/fixed carbon + volatile matter" becomes high. The yield at the specific gravity of 1.6 rapidly decreases in the cases where the mean maximum reflectance shows over 4.0, and this apparently indicates the thermal effect of dolerite intrusions.

4.1.4 Evaluation of Coal Quality

Average coal quality of the Main Seam in the investigated area and Mpaka mine is shown below.

	Investigated area	Mpaka mine
Moisture (%)	1.4	1.3
Ash (%)	13.9	14.0
Fixed carbon (%)	76.9	72.0
Volatile matter (%)	7.7	12.7
Total sulphur (%)	0.43	0.35
Calorific value (kcal/kg)	7,108	6,745
Calorific value (MJ/kg)	29.7	28.2
Yield (%)	78.0	75.0

Note: Clean coal is separated at specific gravity of 1.6.

As stated above, coal in the investigated area compares favourably with coal in the Mpaka mine and the calorific value of the former is rather high and its volatile matter is low as compared with those of the latter. Moreover, quality of this coal is almost equivalent to that of anthracite in Natal, the Republic of South Africa.

Therefore, coal in the investigated area is marketable for wide uses such as burning lime, carbide, reductant in a ferroalloy, briquettes, cement kiln fuel, thermal power generation and smokeless fuel.

4.2 Coal Reserves

4.2.1 Standard of Coal Reserves Calculation

The reserves calculated are within the area measuring about 6 km both east-west and north-south directions in the northern part of the Lubhuku area (with southern margin running approximately from DD6 to DD29, see Figure II-1). The western limit is set at the Lubhuku Fault. Coal reserves were calculated in the previous investigation, but minable coal reserves of the Main Seam, which is considered to be the subject for preparation of a draft of the coal mine development plan, are calculated in this study.

1) Classification of Coal Reserves

The coal reserves were classified into the following categories based on the certainty of the occurrence of coal seams. This classification was established as the standard to be applied to the coalfield in Swaziland after having discussions with the Geological Survey and Mines Department of Swaziland by taking into account the complexity of the dolerite intrusions in the coalfield.

(a) Measured Reserves

Coal reserves within the range of 125 m from the observed point, in which occurrence and extent of the coal seam have been confirmed.

(b) Indicated Reserves

Coal reserves within the range of 250 m from the observed point, in which occurrence and continuity of the coal seam are geologically estimated to an appropriate extent.

(c) Inferred Reserves

Coal reserves inferred from overall geological information for the relevant area. The continuity of the coal seam can be expected from the observed point using for calculation of the measured and indicated reserves. In a case where only a part of the coal seam can be confirmed as the seam cuts by fault and/or dolerite at the observed point, the reserves are classified into this category.

2) Range for Calculating Coal Reserves

Coal reserves were calculated for those parts in which the seam thickness exceeds 1.0 m, in principle, for the three major coal seams (Intermediate Marker Seam, Main Seam and Footwall 3 Seam) distributed over the whole investigated area with generally excellent thickness. However, the followings were excluded from the calculation:

i. Areas where the ratio of coal thickness against seam thickness shows under 70%. However, areas where a ratio of coal thickness against seam thickness shows over 50% were included for the calculation, in case the coal seam is considered to be minable taking into account the occurrence of the coal seam in the surrounding area.
ii. Areas where no coal exists along a fault.
iii. Areas where a coal seam cannot be expected due to the intrusion of dolerite.

3) Block

Block was determined based on the classification of coal reserves, major faults, level of the occurrence and the thickness of the coal seams. Boundaries except the western limit were determined by the polygonal method using each drilling point as centre, and their distances from the points were set within 1 km.

4) Level

Five levels were used; shallower than +100 m (above sea level), +100 m to ±0 m, ±0 m to −100 m, −100 m to −200 m and deeper than −200 m.

5) Area

Area was measured using a planimeter on the coal calculation maps; 1/10,000 in scale for the Main Seam which is considered to be minable and 1/25,000 for other coal seams.

6) Dip

No dip conversion rate was used because most of the coal seams dip less than 5 degrees.

7) Thickness of Coal Seam

For the measured and indicated blocks, the seam thickness confirmed in each borehole represents that of corresponding block. For the inferred block, the average thickness of coal seam calculated from the isopach map of the seam was used for the corresponding block. For the block represented by a borehole in which the coal seam did not appear because of a fault or dolerite intrusion, the thickness of the coal seam of the block concerned was determined from the isopach map of the seam. No conversion into true thickness was done because most of the dips were less than 5 degrees.

8) Specific Gravity

Average specific gravity of 1.55, which was measured from coal samples taken up to now, was employed uniformly. Analytical results show that ash content is about 20% at specific gravity of 1.55.

9) Theoretical Coal Reserves

Theoretical Coal Reserves = Area x Thickness of Coal Seam x Specific Gravity (1.55)

10) Safety Factor

Based on the geologic safety factors such as stability of coal seam, geologic structure and dolerite intrusion, and accuracy of the investigation; and dolerite factors such as burnt-out coal which is caused by intrusions, the safety factor was determined as follows:

Coal Reserves Classification	Geologic Safety Factor	Dolerite Factor	Safety Factor
Measured block	95%	80%	76%
Indicated block	85%	80%	68%
Inferred block	70%	80%	56%

11) Coal Reserves

Coal Reserves = Theoretical Coal Reserves x Safety Factor

12) Mining Recovery

Mining recovery of 60%, which was determined from the model case of a mine development plan considering mining method and safety pillar, was uniformly used.

13) Minable Reserves

Minable Reserves = Coal Reserves x Mining Recovery

4.2.2 Coal Reserves

The coal reserves of the three major coal seams in the northern part of the Lubhuku area calculated on the basis of the above-mentioned standard are summarized below. The reserves are calculated to divide the area into the following three parts based on the distribution of faults.

Area I : North of the "A" fault.

Area II : South of the "A" fault.

Area III : East of the 'C" fault.

Details of the calculation are shown in Table II-9.

Area	Category of Reserves	Coal Reserves (x10^3 tons)		
		Intermediate Marker	Main Seam	Footwall 3
I	Measured	1,214	2,181	1,123
	Indicated	3,315	5,852	3,038
	Inferred	25,218	50,877	20,054
	Subtotal	29,747	58,910	24,215
II	Measured	942	2,092	846
	Indicated	2,443	5,604	2,247
	Inferred	22,177	49,105	16,599
	Subtotal	25,562	56,801	19,692
III	Measured	71	–	75
	Indicated	191	–	201
	Inferred	1,802	–	1,920
	Subtotal	2,064	–	2,196
Total		57,373	115,711	46,103

Consequently, total coal reserves in the investigated area sum up to 219,187 x 10^3 tons. About 53% of these reserves are found in the Main Seam and approximately 51% of which are in the area I.

The following coal reserves had been calculated in the southern part of the Lubhuku area (south of the DD10-DD52 line) in the previous investigation.

Intermediate Marker	12,064,000 tons
Main Seam	49,093,000 tons
Footwall 3	9,666,000 tons
Total	70,823,000 tons

The levelwise minable coal reserves of the Main Seam to be developed in the investigated area are shown below. 35,346 x 10^3 tons of the minable reserves are estimated in the area I and 34,078 x 10^3 tons in the area II.

Minable Reserves of the Main Seam

(Unit: 10^3 tons)

Level / Area	above +100 m	+100~ ±0 m	±0~ −100 m	−100~ −200 m	below −200 m	Total
I	5,545	17,335	8,664	3,802	0	35,346
(Measured)	173	646	490	0	0	1,309
(Indicated)	511	1,822	1,122	56	0	3,511
(Inferred)	4,861	14,867	7,052	3,746	0	30,526
II	737	2,003	11,856	17,806	1,676	34,078
(Measured)	0	0	667	588	0	1,255
(Indicated)	737	94	950	1,580	0	3,361
(Inferred)	0	1,909	10,239	15,638	1,676	29,462
Total	6,282	19,338	20,520	21,608	1,676	69,424

Approximately 65% of the minable reserves of the Main Seam in the area I exist above the sea level (within 300 m below the surface), and about 87% of the reserves in the area II exist between the sea level and 200 m below it (about 300 ~ 500 m below the surface). The minable reserves per unit area of the area I is about 2,085 x 10^3 t/km^2 and about 1,720 x 10^3 t/km^2 in the area II.

Consequently, mining conditions of the Main Seam in the area I which is to be developed are superior to those in the area II, considering depth of occurrence and minable reserves per unit area of the seam, and dolerite intrusions.

Table II-9 Coal Reserves Calculation

A. Main Seam

1

Area	Classification	Block No.	Level (metre)	Reserves Area ($x10^3$ m^2)	Thickness (metre)	Volume Calculated ($x10^3$ m^3)	Specific Gravity	Theoretical Reserves ($x10^3$ tons)	Safety Factor (%)	Reserves ($x10^3$ tons)	Mining Recovery (%)	Minable Reserves
I	Measured	D5	>+100	49	4.99	245	1.55	380	76	289	60	173
						(sub total)		(380)		(289)		(173)
		L6	+100 ∿ ±0	49	3.10	152		236		179		107
		D2		49	1.67	82		127		97		58
		D3		49	4.14	203		315		239		143
		L11		49	3.06	150		233		177		106
		L12		49	1.92	94		146		111		67
		D49		49	4.65	228		353		268		162
		L16-A		6	0.67	4		6		5		3
						(sub total)		(1,416)		(1,076)		(646)
		L8	±0 ∿ -100	49	1.68	82		127		97		58
		D4-B		49	4.70	230		357		271		164
		D25-A		49	3.87	190		295		224		134
		L13		49	2.93	144		223		169		101
		D48-A		25	1.36	34		53		40		24
		L16-B		19	0.67	13		20		15		9
						(sub total)		(1,075)		(816)		(490)
	Total							2,871		2,181		1,309

A. Main Seam

2

Area	Classification	Block No.	Level (metre)	Reserves Area ($x10^3$ m^2)	Thickness (metre)	Volume Calculated ($x10^3$ m^3)	Specific Gravity	Theoretical Reserves ($x10^3$ tons)	Safety Factor (%)	Reserves ($x10^3$ tons)	Mining Recovery (%)	Minable Reserves
I	Indicated	D5	> +100	147	4.99	734	1.55	1,138	68	775	60	465
		L11-A		24	3.06	73		113		77		46
						(sub total)		(1,251)		(852)		(511)
		L6	+100∿±0	147	3.10	456		707		481		289
		D2		147	1.67	245		380		258		155
		D3		147	4.14	609		944		642		385
		D4-A		22	4.70	103		160		109		65
		L11-B		123	3.06	376		583		396		238
		L12		147	1.92	282		437		297		178
		L13-A		30	2.93	88		136		92		55
		D49		147	4.65	684		1,060		721		433
		L16-A		56	0.67	38		59		40		24
						(sub total)		(4,466)		(3,036)		(1,822)
		L8	±0∿-100	147	1.68	247		383		260		156
		D4-B		125	4.70	588		911		619		371
		D25-A		124	3.87	480		744		506		304
		L13-B		117	2.93	343		532		362		217
		L14-A		7	0.39	3		5		3		2
		D48-A		74	1.36	101		157		107		64
		L16-B		18	0.67	12		19		13		8
						(sub total)		(2,751)		(1,870)		(1,122)
		D25-B	-100∿-200	23	3.87	89		138		94		56
						(sub total)		(138)		(94)		(56)
	Total							8,606		5,852		3,511

A. Main Seam

Area	Classification	Block No.	Level (metre)	Reserves Area (x10³ m²)	Thickness (metre)	Volume Calculated (x10³ m³)	Specific Gravity	Theoretical Reserves (x10³ tons)	Safety Factor (%)	Reserves (x10³ tons)	Mining Recovery (%)	Minable Reserves
I	Inferred	a_1	> +100	1,154	4.70	5,424	1.55	8,407	56	4,708	60	2,825
		a_2		440	4.25	1,870		2,899		1,623		974
		a_3		375	3.75	1,406		2,179		1,220		732
		a_4		195	3.25	634		983		550		330
						(sub total)		(14,468)		(8,101)		(4,861)
		b_1	+100∿±0	1,330	4.70	6,251		9,689		5,426		3,255
		b_2		8	4.50	36		56		31		19
		b_3		375	4.25	1,594		2,471		1,384		830
		b_4		715	4.25	3,039		4,710		2,638		1,583
		b_5		698	4.25	2,967		4,599		2,575		1,545
		b_6		2,434	3.75	9,128		14,148		7,924		4,754
		b_7		330	3.25	1,073		1,663		931		559
		b_8		940	3.25	3,055		4,735		2,652		1,591
		b_9		255	2.75	701		1,087		609		365
		b_{10}		190	2.75	523		811		454		272
		b_{11}		80	2.25	180		279		156		94
						(sub total)		(44,248)		(24,780)		(14,867)
		c_1	±0∿-100	120	4.70	564		874		489		293
		c_2		7	4.70	33		51		29		17
		c_3		13	4.25	55		85		48		29
		c_4		175	4.25	744		1,153		646		388
		c_5		470	4.25	1,998		3,097		1,734		1,040
		c_6		135	3.75	506		784		439		263
		c_7		270	3.75	1,013		1,570		879		527
		c_8		855	3.75	3,206		4,969		2,783		1,670
		c_9		190	3.25	618		958		536		322
		c_{10}		315	3.25	1,024		1,587		889		533
		c_{11}		1,045	3.25	3,396		5,264		2,948		1,770
		c_{12}		140	2.75	385		597		334		200
						(sub total)		(20,989)		(11,754)		(7,052)
		d_1	-100∿-200	285	3.75	1,069		1,657		928		557
		d_2		1,300	3.25	4,225		6,549		3,667		2,201
		d_3		690	2.75	1,898		2,941		1,647		988
						(sub total)		(11,147)		(6,242)		(3,746)
	Total							90,852		50,877		30,526

A. Main Seam

Area	Classification	Block No.	Level (metre)	Reserves Area ($x10^3$ m^2)	Thickness (metre)	Volume Calculated ($x10^3$ m^3)	Specific Gravity	Theoretical Reserves ($x10^3$ tons)	Safety Factor (%)	Reserves ($x10^3$ tons)	Mining Recovery (%)	Minable Reserves
II	Measured	L14-B	±0∿-100	49	0.39	19	1.55	29	76	22	60	13
		D48-B		24	1.36	33		51		39		23
		L16-C		24	0.67	16		25		19		11
		L17		49	4.88	239		370		282		170
		L18		49	4.69	230		357		272		163
		D6		49	4.26	209		324		246		148
		D7		49	2.84	139		215		163		98
		D8		28	2.11	59		91		69		41
						(sub total)		(1,462)		(1,112)		(667)
		L15	-100∿-200	49	1.77	87		135		103		62
		D50		49	3.03	148		229		174		104
		L19		49	3.71	182		282		214		128
		L20		49	2.21	108		167		127		76
		D28		49	2.35	115		178		135		81
		D29		49	3.93	193		299		227		137
						(sub total)		(1,290)		(980)		(588)
	Total							2,752		2,092		1,255

A. Main Seam

5

Area	Classification	Block No.	Level (metre)	Reserves Area (x10^3 m^2)	Thickness (metre)	Volume Calculated (x10^3 m^3)	Specific Gravity	Theoretical Reserves (x10^3 tons)	Safety Factor (%)	Reserves (x10^3 tons)	Mining Recovery (%)	Minable Reserves
II	Indicated	L18	> +100	147	4.69	689	1.55	1,068	68	726	60	435
		D6-B		112	4.26	477		739		503		302
						(sub total)		(1,807)		(1,229)		(737)
		D6-A	+100∿ ±0	35	4.26	149		231		157		94
						(sub total)		(231)		(157)		(94)
		L14-B	±0 ∿ -100	140	0.39	55		85		58		35
		D48-B		73	1.36	99		153		104		62
		L16-C		73	0.67	49		76		52		31
		L17		147	4.88	717		1,111		755		453
		D7		147	2.84	417		646		439		263
		D8		79	2.11	167		259		176		106
						(sub total)		(2,330)		(1,584)		(950)
		L15	-100 ∿ -200	147	1.77	260		403		274		164
		D50		147	3.03	445		690		469		281
		L19		147	3.71	545		845		575		345
		L20		147	2.21	325		504		343		206
		D28		147	2.35	345		535		364		218
		D29		147	3.93	578		896		609		366
						(sub total)		(3,873)		(2,634)		(1,580)
	Total							8,241		5,604		3,361

A. Main Seam

Area	Classification	Block No.	Level (metre)	Reserves Area ($x10^3$ m^2)	Thickness (metre)	Volume Calculated ($x10^3$ m^3)	Specific Gravity	Theoretical Reserves ($x10^3$ tons)	Safety Factor (%)	Reserves ($x10^3$ tons)	Mining Recovery (%)	Minable Reserves
II	Inferred	e_1	+100～±0	620	4.25	2,635	1.55	4,084	56	2,287	60	1,372
		e_2		275	3.75	1,031		1,598		895		537
						(sub total)		(5,682)		(3,182)		(1,909)
		f_1	±0～-100	660	4.70	3,102		4,808		2,692		1,615
		f_2		20	4.70	94		146		82		49
		f_3		38	4.25	162		251		141		85
		f_4		227	4.25	965		1,496		838		503
		f_5		420	4.25	1,785		2,767		1,550		930
		f_6		550	4.25	2,338		3,624		2,029		1,217
		f_7		815	3.75	3,056		4,737		2,653		1,592
		f_8		165	3.75	619		959		537		322
		f_9		140	3.75	525		814		456		274
		f_{10}		550	3.25	1,788		2,771		1,552		931
		f_{11}		140	3.25	455		705		395		237
		f_{12}		235	3.25	764		1,184		663		398
		f_{13}		465	2.75	1,279		1,982		1,110		666
		f_{14}		160	2.75	440		682		382		229
		f_{15}		100	2.75	275		426		239		143
		f_{16}		60	2.75	165		256		143		86
		f_{17}		50	2.75	138		214		120		72
		f_{18}		20	2.25	45		70		39		23
		f_{19}		275	2.25	619		959		537		322
		f_{20}		465	2.25	1,046		1,621		908		545
						(sub total)		(30,472)		(17,066)		(10,239)
		g_1	-100～-200	30	4.50	135		209		117		70
		g_2		530	4.25	2,253		3,492		1,956		1,174
		g_3		840	3.75	3,150		4,883		2,734		1,640
		g_4		460	3.70	1,702		2,638		1,477		886
		g_5		45	3.70	167		259		145		87
		g_6		1,120	3.25	3,640		5,642		3,160		1,896
		g_7		1,195	3.25	3,884		6,020		3,370		2,023
		g_8		415	2.75	1,141		1,769		991		595
		g_9		165	2.75	454		704		394		236
		g_{10}		1,310	2.75	3,603		5,585		3,128		1,877
		g_{11}		330	2.75	908		1,407		788		473
		g_{12}		380	2.75	1,045		1,620		907		544
		g_{13}		345	2.75	949		1,471		824		494

A. Main Seam

Area	Classification	Block No.	Level (metre)	Reserves Area ($x10^3$ m^2)	Thickness (metre)	Volume Calculated ($x10^3$ m^3)	Specific Gravity	Theoretical Reserves ($x10^3$ tons)	Safety Factor (%)	Reserves ($x10^3$ tons)	Mining Recovery (%)	Minable Reserves
II	Inferred	g_{14}	-100∼-200	2,609	2.25	5,870	1.55	9,099	56	5,094	60	3,056
		g_{15}		170	2.25	383		594		333		200
		g_{16}		300	2.25	675		1,046		586		352
		g_{17}		30	2.25	68		105		59		35
						(sub total)		(46,543)		(26,063)		(15,638)
		h_1	-200>	35	3.00	105		163		91		55
		h_2		625	2.75	1,719		2,664		1,491		894
		h_3		25	2.25	56		87		49		29
		h_4		440	2.25	990		1,535		860		516
		h_5		155	2.25	349		541		303		182
						(sub total)		(4,990)		(2,794)		(1,676)
	Total							87,687		49,105		29,462

B. Intermediate Marker

8

Area	Classification	Block No.	Level (metre)	Reserves Area ($x10^3$ m^2)	Thickness (metre)	Volume Calculated ($x10^3$ m^3)	Specific Gravity	Theoretical Reserves ($x10^3$ tons)	Safety Factor (%)	Reserves ($x10^3$ tons)
I	Measured	D2	> +100	49	1.58	77	1.55	119	76	90
		L11		49	1.59	78		121		92
		D5		49	2.06	101		157		119
		D49-A		40	1.73	69		107		81
						(sub total)		(504)		(382)
		L2	+100∿±0	49	1.33	65		101		77
		L3		49	1.38	68		105		80
		L6		49	0.87	43		67		51
		D3		49	1.30	64		99		75
		D4-A		49	1.54	75		116		88
		L12		49	1.70	83		129		98
		L13		49	1.08	53		82		62
		D49-B		9	1.73	16		25		19
		L16-A		3	1.69	5		8		6
						(sub total)		(732)		(556)
		L8	±0∿-100	49	1.43	70		109		83
		L9		49	1.10	54		84		64
		D25		49	1.53	75		116		88
		L16-C		18	1.69	30		47		36
						(sub total)		(356)		(271)
		D26-A	-100∿-200	4	1.08	4		6		5
						(sub total)		(6)		(5)
	Total							1,598		1,214

B. Intermediate Marker

Area	Classification	Block No.	Level (metre)	Reserves Area (x10^3 m^2)	Thickness (metre)	Volume Calculated (x10^3 m^3)	Specific Gravity	Theoretical Reserves (x10^3 tons)	Safety Factor (%)	Reserves (x10^3 tons)
I	Indicated	D2	> +100	147	1.58	232	1.55	360	68	245
		L11		147	1.59	234		363		247
		D5		147	2.06	303		470		320
		D49-A		100	1.73	173		268		182
						(sub total)		(1,461)		(994)
		L2	+100∼ ±0	147	1.33	196		304		207
		L3		147	1.38	203		315		214
		L6		147	0.87	128		198		135
		D3		147	1.30	191		296		201
		D4-A		147	1.54	188		291		198
		L12		147	1.70	250		388		264
		L13		147	1.08	159		246		167
		D49-B		47	1.73	81		126		86
		L16-A		70	1.69	118		183		124
						(sub total)		(2,347)		(1,596)
		L8	±0 ∼ -100	147	1.43	210		326		222
		L9		147	1.10	162		251		171
		D4-B		25	1.54	39		60		41
		D25		147	1.53	225		349		237
						(sub total)		(986)		(671)
		D26-A	-100∼ -200	47	1.08	51		79		54
						(sub total)		(79)		(54)
	Total							4,873		3,315

B. Intermediate Marker

Area	Classification	Block No.	Level (metre)	Reserves Area ($x10^3$ m^2)	Thickness (metre)	Volume Calculated ($x10^3$ m^3)	Specific Gravity	Theoretical Reserves ($x10^3$ tons)	Safety Factor (%)	Reserves ($x10^3$ tons)
I	Inferred	a_1	> +100	290	2.20	638	1.55	989	56	554
		a_2		3,688	1.75	6,454		10,004		5,602
		a_3		283	1.25	354		549		307
						(sub total)		(11,542)		(6,463)
		b_1	+100∿±0	3,153	1.75	5,518		8,553		4,790
		b_2		6	1.50	9		14		8
		b_3		4	1.50	6		9		5
		b_4		235	1.50	353		547		306
		b_5		5,737	1.25	7,171		11,115		6,224
		b_6		739	1.25	924		1,432		802
		b_7		433	0.75	325		504		282
						(sub total)		(22,174)		(12,417)
		c_1	±0 ∿-100	190	1.50	285		442		248
		c_2		4,905	1.25	6,131		9,503		5322
		c_3		7	1.25	9		14		8
		c_4		50	1.00	50		78		44
						(sub total)		(10,037)		(5,622)
		d	-100 ∿ -200	660	1.25	825		1,279		716
						(sub total)		(1,279)		(716)
	Total							45,032		25,218

B. Intermediate Marker

11

Area	Classification	Block No.	Level (metre)	Reserves Area ($x10^3$ m^2)	Thickness (metre)	Volume Calculated ($x10^3$ m^3)	Specific Gravity	Theoretical Reserves ($x10^3$ tons)	Safety Factor (%)	Reserves ($x10^3$ tons)
II	Measured	D6	+100∼ ±0	49	1.66	81	1.55	126	76	96
		D7		49	1.25	61		95		72
		D8		45	1.43	64		99		75
						(sub total)		(320)		(243)
		L14	±0∼ -100	49	0.90	44		68		52
		L16-B		28	1.69	47		73		55
		L17		49	1.71	84		130		99
		L18		49	1.86	91		141		107
		L19-A		5	1.35	7		11		8
						(sub total)		(423)		(321)
		D26-B	-100∼ -200	45	1.08	49		76		58
		L19-B		44	1.35	59		91		69
		L20		49	1.45	71		110		84
		L28-B		49	1.53	75		116		88
		D29		49	1.37	67		104		79
						(sub toal)		(497)		(378)
	Total							1,240		942

B. Intermediate Marker

12

Area	Classification	Block No.	Level (metre)	Reserves Area (x10^3 m^2)	Thickness (metre)	Volume Calculated (x10^3 m^3)	Specific Gravity	Theoretical Reserves (x10^3 tons)	Safety Factor (%)	Reserves (x10^3 tons)
II	Indicated	D6	+100∿±0	147	1.66	244	1.55	378	68	257
		D7		147	1.25	184		285		194
		D8		110	1.43	157		243		165
						(sub total)		(906)		(616)
		L14	±0∿-100	147	0.90	132		205		139
		L16-B		77	1.69	130		202		137
		L17		147	1.71	251		389		265
		L18		147	1.86	273		423		288
		L19-A		45	1.35	61		95		65
		D28-A		8	1.53	12		19		13
						(sub total)		(1,333)		(907)
		D26-B	-100∿-200	100	1.08	108		167		114
		L19-B		102	1.35	138		214		146
		L20		147	1.45	213		330		224
		D28-B		139	1.53	213		330		224
		D29		147	1.37	201		312		212
						(sub total)		(1,353)		(920)
	Total							3,592		2,443

B. Intermediate Marker

13

Area	Classification	Block No.	Level (metre)	Reserves Area ($x10^3$ m^2)	Thickness (metre)	Volume Calculated ($x10^3$ m^3)	Specific Gravity	Theoretical Reserves ($x10^3$ tons)	Safety Factor (%)	Reserves ($x10^3$ tons)
II	Inferred	e_1	+100 ∿ ±0	1,180	1.75	2,065	1.55	3,201	56	1,793
		e_2		165	1.60	264		409		229
		e_3		1,214	1.25	1,518		2,353		1,318
		e_4		445	1.25	556		862		483
		e_5		40	1.00	40		62		35
						(sub total)		(6,887)		(3,858)
		f_1	±0 ∿ -100	4,085	1.75	7,149		11,081		6,205
		f_2		2,255	1.25	2,819		4,369		2,447
		f_3		935	1.25	1,169		1,812		1,015
		f_4		180	1.00	180		279		156
		f_5		95	1.50	143		222		124
		f_6		75	0.90	68		105		59
		f_7		58	0.90	52		81		45
						(sub total)		(17,949)		(10,051)
		g_1	-100 ∿ -200	125	1.50	188		241		163
		g_2		6,122	1.25	7,653		11,862		6,643
		g_3		720	1.00	720		1,116		635
		g_4		1,068	0.90	961		1,490		834
		g_5		4	1.00	4		6		3
						(sub total)		(14,765)		(8,268)
	Total							39,601		22,177

B. Intermediate Marker

Area	Classification	Block No.	Level (metre)	Reserves Area (x10^3 m^2)	Thickness (metre)	Volume Calculated (x10^3 m^3)	Specific Gravity	Theoretical Reserves (x10^3 tons)	Safety Factor (%)	Reserves (x10^3 tons)
III	Maasured	L5-A	> -100	49	1.23	60	1.55	93	76	71
	Total							93		71
	Indicated	L5-A L5-B	> -100 -100>	137 10	1.23 1.23	169 12	1.55	262 19	68	178 13
	Total							281		191
	Inferred	h_1 h_2 i	> -100 -100>	185 110 1,780	1.00 1.00 1.00	185 110 1,780	1.55	287 171 2,579	56	161 96 1,545
	Total							3,217		1,802

C. Footwall 3

Area	Classification	Block No.	Level (metre)	Reserves Area (x10^3 m^2)	Thickness (metre)	Volume Calculated (x10^3 m^3)	Specific Gravity	Theoretical Reserves (x10^3 tons)	Safety Factor (%)	Reserves (x10^3 tons)
I	Measured	D5-A	> +100	3	1.48	4	1.55	6	76	5
						(sub total)		(6)		(5)
		L6-A	+100～±0	49	1.34	66		102		78
		D2		49	1.14	56		87		66
		D3		49	1.07	52		81		62
		L11		49	1.37	67		104		79
		D5-B		46	1.48	68		105		80
		D49		49	1.23	60		93		71
		L16-A		27	0.47	13		20		15
						(sub total)		(592)		(451)
		L2	±0～-100	49	1.28	63		98		74
		L3		49	1.93	95		147		112
		L7		43	1.54	66		102		78
		L8		49	1.81	89		138		105
		D4		49	1.07	52		81		62
		L13		49	1.01	49		76		58
		D48-A		27	1.24	33		51		39
						(sub total)		(693)		(528)
		L4	-100～-200	49	1.72	84		130		99
		D25		49	0.69	34		53		40
						(sub total)		(183)		(139)
	Total							1,474		1,123

C. Footwall 3

Area	Classification	Block No.	Level (metre)	Reserves Area ($x10^3$ m^2)	Thickness (metre)	Volume Calculated ($x10^3$ m^3)	Specific Gravity	Theoretical Reserves ($x10^3$ tons)	Safety Factor (%)	Reserves ($x10^3$ tons)
I	Indicated	D5-A	> +100	40	1.48	59	1.55	91	68	62
						(sub total)		(91)		(62)
		L6-A	+100 ∿ ±0	139	1.34	186		288		196
		D2		147	1.14	168		260		177
		D3		147	1.07	157		243		165
		L11		147	1.37	201		312		212
		D5-B		107	1.48	158		245		167
		D49		147	1.23	181		281		191
		L16-A		77	0.47	36		56		38
						(sub total)		(1,685)		(1,146)
		L2	±0 ∿ -100	147	1.28	188		291		198
		L3		147	1.93	284		440		299
		L6-B		8	1.34	11		17		12
		L7		147	1.54	226		350		238
		L8		147	1.81	266		412		280
		D4		147	1.07	157		243		165
		D25-A		32	0.69	22		34		23
		L13		147	1.01	148		229		156
		L14-A		9	1.15	10		16		11
		D48-A		75	1.24	93		144		98
						(sub total)		(2,176)		(1,480)
		L4	-100 ∿ -200	147	1.72	253		392		267
		D25-B		115	0.69	79		122		83
						(sub total)		(514)		(350)
	Total							4,466		3,038

C. Footwall 3

17

Area	Classification	Block No.	Level (metre)	Reserves Area ($x10^3$ m^2)	Thickness (metre)	Volume Calculated ($x10^3$ m^3)	Specific Gravity	Theoretical Reserves ($x10^3$ tons)	Safety Factor (%)	Reserves ($x10^3$ tons)
I	Inferred	a_1	> +100	800	1.70	1,360	1.55	2,108	56	1,180
		a_2		110	1.25	138		214		120
						(sub total)		(2,322)		(1,300)
		b_1	+100∿ ±0	800	1.70	1,360		2,108		1,180
		b_2		7	1.50	11		17		10
		b_3		5,641	1.25	7,051		10,929		6,120
						(sub total)		(13,054)		(7,310)
		c_1	±0 ∿ -100	2,648	1.70	4,502		6,978		3,908
		c_2		865	1.50	1,298		2,012		1,127
		c_3		2,969	1.25	3,711		5,752		3,221
		c_4		1,060	1.25	1,325		2,054		1,150
		c_5		740	0.75	555		860		482
						(sub total)		(17,656)		(9,888)
		d_1	-100 ∿ -200	1,144	1.70	1,945		3,015		1,688
		d_2		1,175	1.25	1,469		2,277		1,275
		c_3		57	1.00	57		88		49
		d_4		1,455	0.75	1,091		1,691		947
						(sub total)		(7,071)		(3,959)
	Total							40,103		22,457

C. Footwall 3

Area	Classification	Block No.	Level (metre)	Reserves Area (x10^3 m^2)	Thickness (metre)	Volume Calculated (x10^3 m^3)	Specific Gravity	Theoretical Reserves (x10^3 tons)	Safety Factor (%)	Reserves (x10^3 tons)
II	Measured	L14-C	>-100	16	1.15	18	1.55	28	76	21
		L16-B		22	0.47	10		16		12
		L17		49	1.64	80		124		94
		D6		49	1.11	54		84		64
		D7		49	1.09	53		82		62
		D8		49	1.10	54		84		64
						(sub total)		(418)		(317)
		L14-D	-100∿-200	33	1.15	38		59		45
		L15		49	1.35	66		102		78
		D48-C		22	1.24	27		42		32
		L18		49	1.45	71		110		84
		L19		49	1.44	71		110		84
		D28		49	0.79	39		60		46
						(sub total)		(483)		(369)
		L20	-200>	49	1.38	68		105		80
		D29		49	1.38	68		105		80
						(sub total)		(210)		(160)
	Total							1,111		846

C. Footwall 3

19

Area	Classification	Block No.	Level (metre)	Reserves Area ($x10^3$ m^2)	Thickness (metre)	Volume Calculated ($x10^3$ m^3)	Specific Gravity	Theoretical Reserves ($x10^3$ tons)	Safety Factor (%)	Reserves ($x10^3$ tons)
II	Indicated	L14-C	±0∿-100	38	1.15	44	1.55	68	68	46
		D48-B		3	1.24	4		6		4
		L16-B		70	0.24	17		26		18
		L17		147	1.64	241		374		254
		L18-A		20	1.45	29		45		31
		D6		147	1.11	163		253		172
		D7		147	1.09	160		248		169
		D8		147	1.10	162		251		171
						(sub total)		(1,271)		(865)
		L14-B	-100∿-200	100	1.15	115		178		121
		L15		142	1.35	192		298		203
		D48-C		69	1.24	86		133		90
		L18-B		127	1.45	184		285		194
		L19		147	1.44	212		329		224
		L20-A		9	1.38	12		19		13
		D28		147	0.79	116		180		122
						(sub total)		(1,422)		(967)
		L20-B	-200>	138	1.38	190		295		201
		D29		147	1.38	203		315		214
						(sub total)		(610)		(415)
	Total							3,303		2,247

C. Footwall 3

20

Area	Classification	Block No.	Level (metre)	Reserves Area ($x10^3$ m^2)	Thickness (metre)	Volume Calculated ($x10^3$ m^3)	Specific Gravity	Theoretical Reserves ($x10^3$ tons)	Safety Factor (%)	Reserves ($x10^3$ tons)
II	Inferred	e_1	±0∿-100	85	1.50	128	1.55	198	56	111
		e_2		3,878	1.25	4,848		7,514		4,208
		e_3		180	1.00	180		279		156
		e_4		155	1.00	155		240		134
		e_5		162	0.75	122		189		106
		e_6		55	0.75	41		64		36
						(sub total)		(8,484)		(4,751)
		f_1	-100∿-200	6,369	1.25	7,961		12,340		6,910
		f_2		65	1.00	65		101		57
		f_3		1,935	0.75	1,451		2,249		1,259
		f_4		65	1.00	65		101		57
		f_5		8	1.00	8		12		7
						(sub total)		(14,803)		(8,290)
		g	-200 >	4,099	1.00	4,099		6,353		3,558
						(sub total)		(6,353)		(3,558)
	Total							29,640		16,599

C. Footwall 3

Area	Classification	Block No.	Level (metre)	Reserves Area ($x10^3$ m^2)	Thickness (metre)	Volume Calculated ($x10^3$ m^3)	Specific Gravity	Theoretical Reserves ($x10^3$ tons)	Safety Factor (%)	Reserves ($x10^3$ tons)
III	Measured	l_5	-100>	49	1.30	64	1.55	99	76	75
	Indicated	l_5	-100>	147	1.30	191	1.55	296	68	201
	Inferred	h_1 h_2 h_3	-100>	280 1,105 585	1.50 1.25 0.70	420 1,381 410	1.55	651 2,141 636	56	365 1,199 356
	Total							3,428		1,920

PART III. MINE DEVELOPMENT PLAN

CHAPTER 1. OUTLINE OF MINE DEVELOPMENT PLAN

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CHAPTER 1. OUTLINE OF MINE DEVELOPMENT PLAN

1.1 Outline of Plan

As the present investigation is a pre-feasibility study, this draft of the coal mine development plan is prepared mainly from a technical viewpoint to provide basic data to be used for studying future developments. Therefore, no economic evaluation is made in this draft, but initial investment for development and production cost at the mine site are roughly estimated for the convenience of future reference based on information obtained from those sources such as the Mpaka mine which is now in operation.

Various studies are made based on the results of the investigation, mainly of the drilling, carried out to date in the northern part of the Lubhuku area. The Main Seam is selected as a coal seam to be mined in this plan because the seam shows stable occurrence with overwhelmingly large minable reserves. Mining conditions of the Main Seam are generally favourable with the seam thickness of over 2.0 m. The extent to be mined, based on the distribution of faults and dolerite intrusions, is restricted within an area surrounded by "C" Fault in the east, Dolerite Sill "F" in the north, the Lubhuku Fault and "B" Fault in the west and "A" Fault in the south.

The Intermediate Marker and Footwall 3 are excluded from the coal seams to be mined, since their seam thicknesses are below 2.0 m and their mining conditions are very inferior to those of the Main Seam in view of the intrusion of dolerite and so forth.

Construction of a thermal power plant is currently being studied in the Kingdom of Swaziland, but the market of the coal is not clear at present. Consequently, the scale of clean coal production of approximately 500,000 tons/year is tentatively planned in this study by taking into account the geologic conditions, especially the intrusion of dolerite in the mining area. About 35 million tons of minable reserves of the Main Seam are expected in this area, which are sufficient to operate the mine for over 40 years.

Development is made by incline in view of the depth of the occurrence of the Main Seam, and the pit mouth is opened near the existing Swaziland Railway. Two inclines are planned; a trackless incline for air intake (also for transport of workers and materials) and a belt incline for air return (also for transport of coal). The main entry along the seam is developed almost at the centre of the mining area, from the point where the inclines reach the seam, and main cross entries are developed on both sides of the main entry.

For the coal mining, the room and pillar mining method using a continuous miner is to be employed by taking into account working thickness and dip of the Main Seam and the intrusion of dolerite. Three working faces including the main entry and main cross entries are constantly maintained in this plan. Transportation of raw coal is performed by the belt conveyor method. Ventilation is initially made by a central ventilation system, but when the development reaches far away from the pit mouth in future, a vertical shaft for return air will be sunk and the diagonal ventilation system will be employed.

Raw coal taken out from the mine is crushed, screened, hand picked and separated by a heavy medium for obtaining clean coal in the coal preparation plant to be built near the pit mouth of the belt incline. Railway siding and loading facilities for goods wagons are prepared taking into account the export of coal. Facilities for power distribution and compressors are designed to meet required consumption.

Manpower requirements are planned by taking into account mechanized underground mining and the example of the Mpaka mine for the surface operation.

1.2 Production Plan

Development schedule is planned as follows:

1st year : Detailed engineering.

2nd year : Construction of roads and surface facilities.

3rd year : Driving of inclines and construction of coal preparation facilities.

4th year : Driving of inclines, main entry and cross main entries, and construction of underground facilities.

5th year : Setting up of working faces.

6th year : Full production (510,000 tons/year of clean coal).

Year-wise production plan of clean coal is shown below:

Year	Production
1st	–
2nd	–
3rd	–
4th	100,000 tons
5th	340,000 tons
6th and after	510,000 tons

CHAPTER 2. UNDERGROUND STRUCTURE

CHAPTER 2. UNDERGROUND STRUCTURE

The basic underground structure is established as follows by taking into account the modes of formation of the Main Seam expected from the results of the drilling investigation, topography, mining area and existing railway.

2.1 Incline Design

Since the coal seam to be mined exists in a relatively shallow part about 165 m below the surface, an inclined shaft is to be employed as the development method that fits for shallow coal seam developing.

The pit mouth is planned to be located 600 m to the southwest of DD1 with the direction of incline S 37° E, considering location of the coal seam in the shallowest part to be mined initially, topography around the pit mouth and barrier pillars for the railway (Drawing 15). Two inclines are developed parallel to each other; a trackless incline (grade: 7°) for transporting personnel and materials and for air intake, and the belt incline (grade: 16°) for coal and return air.

A centre-to-centre distance between the two inclines is designed to be 30 m. Both the inclines are lined with concrete for a distance of 20 m from the pit mouth. 4.5 m arch frames are installed to support the roof at the intervals of 1.2 m.

Standard and cross section of the incline are shown in Table III-1 and Figure III-1 respectively. Figure III-2 shows cross section of box cut. Figures III-3 and 4 show longitudinal sections of the trackless and belt inclines respectively.

The following points were considered in designing the inclines:

(1) Barrier pillar for the inclines and railway on the ground is to be as small as possible.

(2) Surface facilities and the pit mouth of the belt incline are located as close as possible to the railway so that the railway siding for loading clean coal should be made as short as possible.

(3) Underground structure should be as simple and rational as possible.

(4) Elevation of the pit mouth should be higher than the surrounding area so as to prevent any inundation in rainy season.

2.2 Incline Driving Method

The inclines are driven as follows:

i) Driving by electric drills and blasting.

ii) Waste is loaded by side dump loader into 2 m^3 tubs which are hauled to the surface by a winding machine at the pit mouth.

Table III-1 Standard of Incline

	Length (m)	Angle of dip (°)	Width (m)	Height (m)	Sectional area (m^2)
Belt Incline	608	16	4.7	3.1	14.46
Trackless Incline	1,287	7	4.7	3.1	14.46

Note: Length of box cut is not included in the length of the inclines.

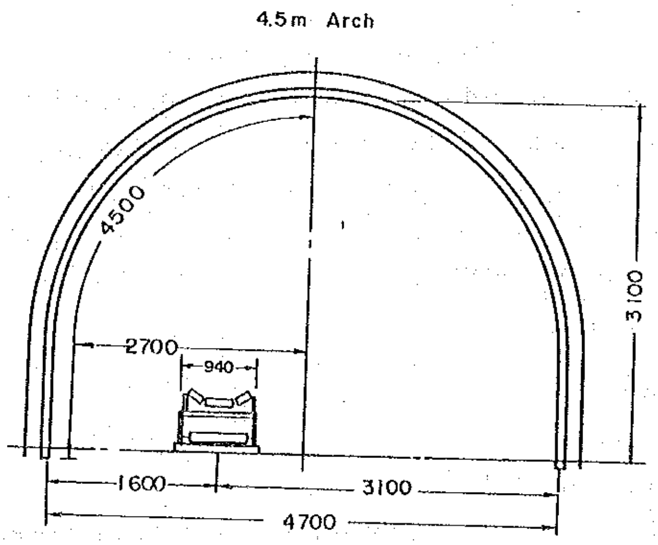


Figure III-1 Cross Section of Incline
(Rail Gauge 610mm)

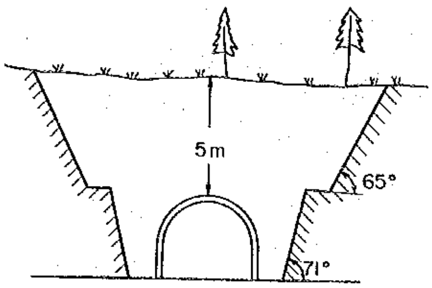


Figure Ⅲ-2 Box Cut Cross Section

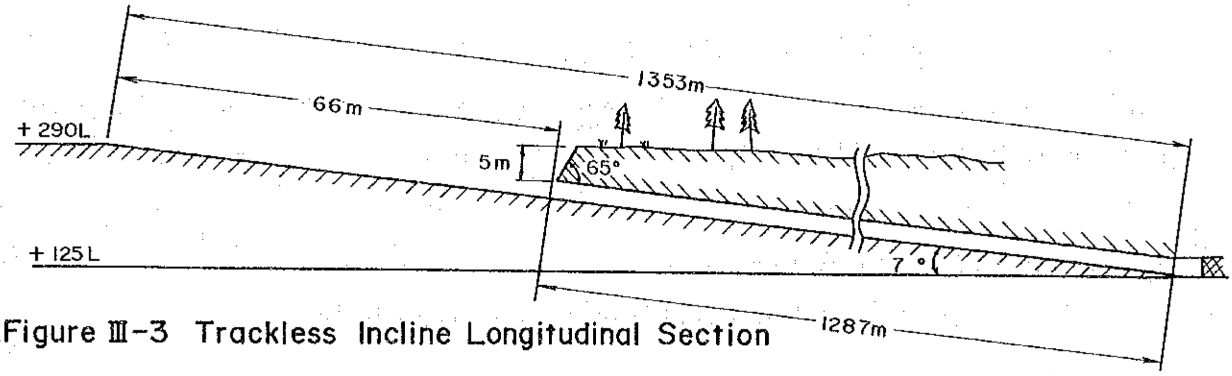


Figure Ⅲ-3 Trackless Incline Longitudinal Section

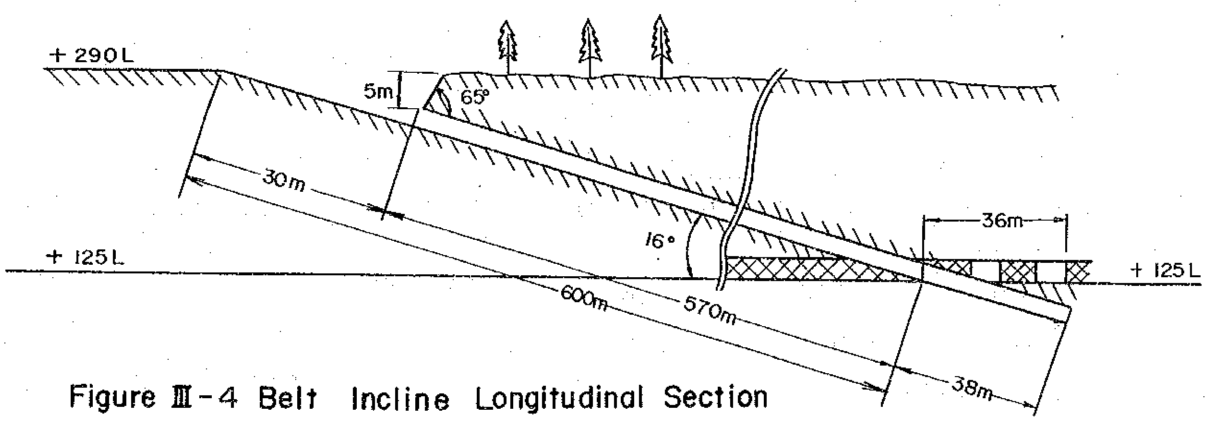


Figure Ⅲ-4 Belt Incline Longitudinal Section

iii) Roof support using 4.5 m arch frames placed at intervals of 1.2 m.

iv) Driving capacity is supposed:

Driving speed: 1.0 m for 1 shift.

4 shifts/day: 1.0 m x 4 shifts = 4 m/day

Quantity of driving: $14.46\ m^2$ (area of cross section) x 1.0 m (driving speed) x 1.7 (volume increment)

= $24.58\ m^3$/shift,

4 shifts/day: $24.58\ m^3$/shift x 4 shifts = $98.32\ m^3$/day

2.2.1 Period of Incline Driving

Period required for incline driving is as follows:

Belt incline: $\frac{608\text{ m}}{4\text{ m/day}} = 152$ days

In case the number of working days per month is 20, about 8 months will be necessary for belt incline driving.

Trackless incline: $\frac{1{,}287\text{ m}}{4\text{ m/day}} = 321.8 \fallingdotseq 322$ days

In case the number of working days per month is 20, about 16 months will be required for trackless incline driving and additionally 5 months will be necessary for the preparations for development.

2.2.2 Procedure of Incline Driving Work

Typical example of incline driving work per shift is shown in Table III-2.

2.3 Driving Manpower Requirements

The following workers for one working face per shift are required for driving incline.

Foreman:	1
Driving workers:	5
Winding machine operator:	1
Tub operator:	1
Total:	8

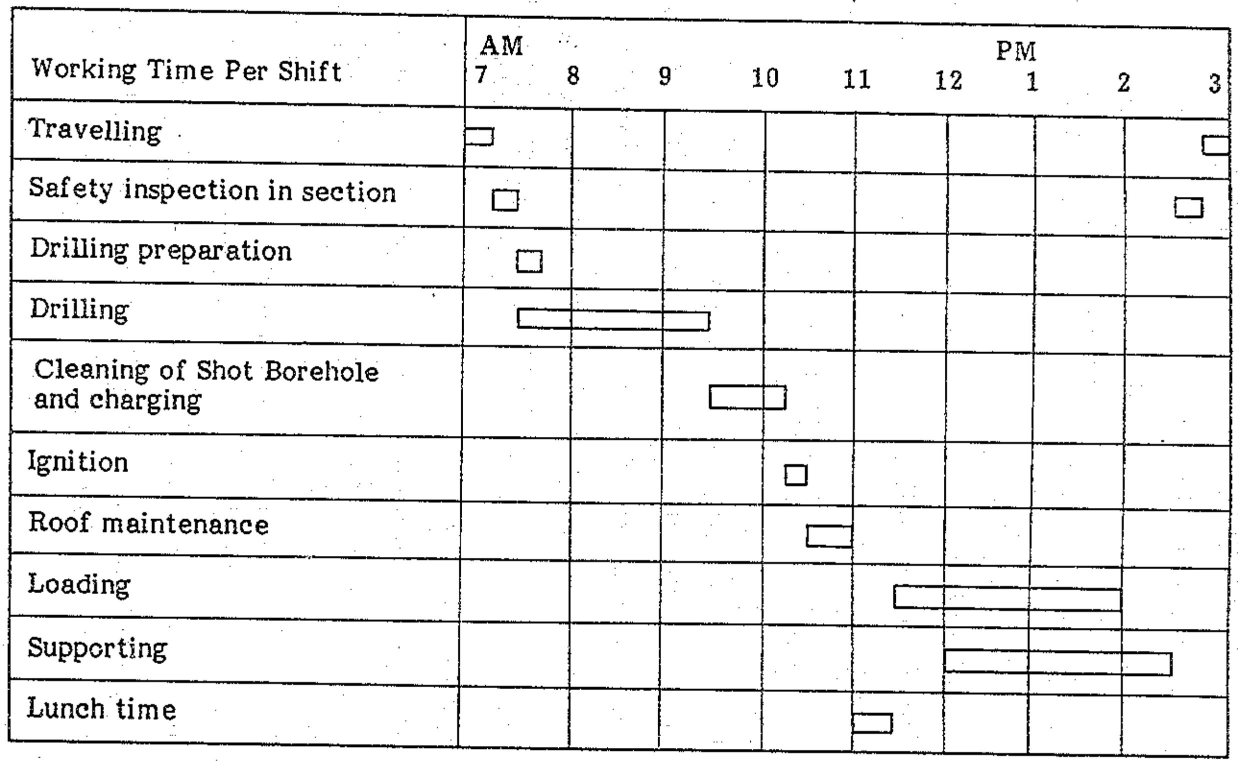
In addition, maintenance workers of 4 persons/day are necessary to extend air ducts, compressed-air pipes and tracks.

Total number of workers required for driving two inclines is as follows:

8 persons x 2 working faces x 4 shifts/day + 4 persons/day x 2 working faces

= 72 persons/day

Table III-2 Incline Driving Sequence



2.4 Transportation Plan of Incline Driving

2.4.1 Waste Transportation

Portable turnout and small hoist are used for moving the tubs at driving faces (Figure III-5). The number of tubs to be used for both inclines is 40. Details of the winding machine for transporting the waste from incline driving are as:

$$P = F + G = n(W+L)\, f \cos\alpha + n(W+L) \sin\alpha$$
$$= n(W+L)\,(f \cos\alpha + \sin\alpha) \quad \text{(2.1)}$$

where,

- P: Total resistance in lifting tubs along slope by winding
- F: Frictional resistance (kg)
- G: Grade resistance (kg)
- n: Number of tubs
- W: Deadweight of tub (kg)
- L: Load of tub (kg)
- f: Coefficient of friction resistance of tub
- α: Grade of incline (°)

(1) . Total resistance of winding machine for belt incline (P)

$\alpha = 16°$, $n = 4$, $W = 1{,}500$ kg, $L = 2 \times 0.83 \times 1.4 \fallingdotseq 2{,}300$ kg, $f = 0.015$

$P = 4 \times (1{,}500 + 2{,}300) \times (0.015 \times \cos 16° + \sin 16°) \fallingdotseq 4{,}400$ kg

(2) Total resistance of winding machine for trackless incline (P)

$\alpha = 7°$, $n = 4$, $W = 1{,}500$ kg, $L = 2 \times 0.83 \times 1.4 \fallingdotseq 2{,}300$ kg, $f = 0.015$

$P = 4 \times (1{,}500 + 2{,}300) \times (0.015 \times \cos 7° + \sin 7°) \fallingdotseq 2{,}080$ kg

(3) Winding rope specifications

If the safety factor of the winding rope against static load is assumed to be 6, the total resistance of the winding rope for belt incline is as follows:

4,400 kg x 6 = 26,400 kg

Therefore, the specifications of the rope to be used are:

Diameter:	22 mm (Class 1: 155 kg/mm^2)
Weight:	1.79 kg/m

The total resistance of winding rope for trackless incline is as follows:

2,080 kg x 6 = 12,480 kg

Therefore, the specifications of the rope to be used are:

Diameter:	16 mm (Class 1: 155 kg/mm^2)
Weight:	0.94 kg/m

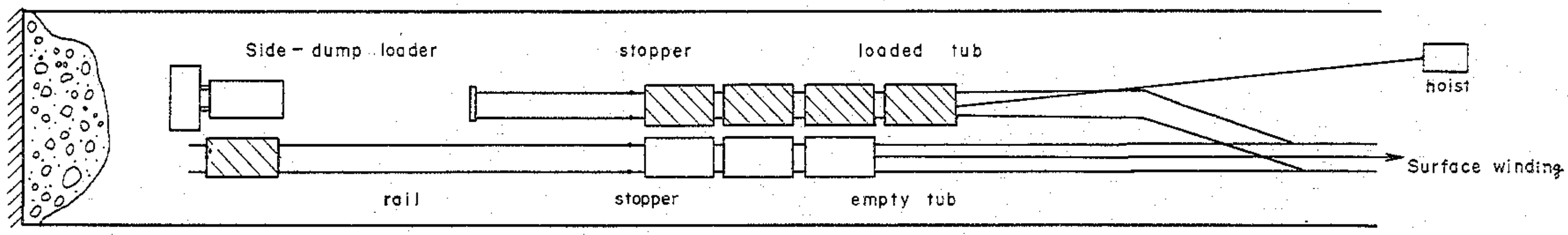


Figure Ⅲ- 5 Method of Haulage at Driving of Incline

(4) Horsepower required for winding machine

Horsepower of the prime mover for belt incline is calculated by the following formula, but winding speed is supposed to be 3 m/sec:

$$N = \frac{PV}{75\eta} \qquad \text{(2.2)}$$

where,

N: Horsepower of prime mover (HP)

P: Haulage load of rope (kg) = total resistance + length of rope x unit weight of rope x (frictional resistance x $\cos\alpha + \sin\alpha$)

V: Hoisting speed (m/sec)

η: Efficiency of machine (0.8)

$$N = \frac{4.4 + 0.638 \times 1.79 \times (0.1 \times \cos 16^\circ + \sin 16^\circ) \quad \times 3}{75 \times 0.8} \times 10^3 = 241 \text{ HP}$$

Therefore, a prime mover of 250 HP is required.

Horsepower of the prime mover for trackless incline is as follows:

$$N = \frac{2.08 + 1.353 \times 0.94 \times (0.1 \times \cos 7^\circ + \sin 7^\circ) \quad \times 3}{75 \times 0.8} \times 10^3 = 118 \text{ HP}$$

Therefore, a prime mover of 120 HP is required.

Results of the above calculations are shown in Table III-3. The track and winding machine installed in the trackless incline are removed when main entry is developed. On the other hand, the track and winding machine installed in the belt incline are left for maintenance of the belt conveyor.

2.5 Transportation of Personnel and Materials

Winding machines are used for transporting workers, materials and waste in each incline. During the development, workers are carried to the working place by personnel carrier which is connected with material carriers at the time of shifting. Each incline requires one personnel carrier for 12 passengers and about 20 tubs for material and waste.

Number of tubs: 18 tubs/shift x 2 working faces = 36 tubs/shift

Number of personnel carriers:

1 carrier/shift x 2 working faces = 2 carriers/shift

2.6 Entry Driving Plan

Since both the main entry and cross main entries are driven within the seam, the driving method for these entries is the same as that for primary coal mining. The main entry and cross main entries are driven by continuous miner and load header. A shuttle car is employed to transport coal from mining machine to belt conveyor.

Table III-3 Specifications of Incline Rail and Hoist

		Belt Incline	Trackless Incline
Rope speed	(m/min)	180	180
Tub weight	(kg)	1,500	1,500
Loaded tub weight	(kg)	2,300	2,300
Hoist tub number	(tub/round)	4 (16)	4 (16)
Rope diameter	(mm)	22 (155 kg/mm^2)	16 (155 kg/mm^2)
Max. haulage	(kg/m)	1.79	0.94
Max. haul age	(ton)	14.4	7.08
Rail gauge	(mm)	610	610
Tub width	(mm)	1,280	1,280
Rail weight	(kg/m)	22	22
Horsepower of Hoist	(HP)	250	120

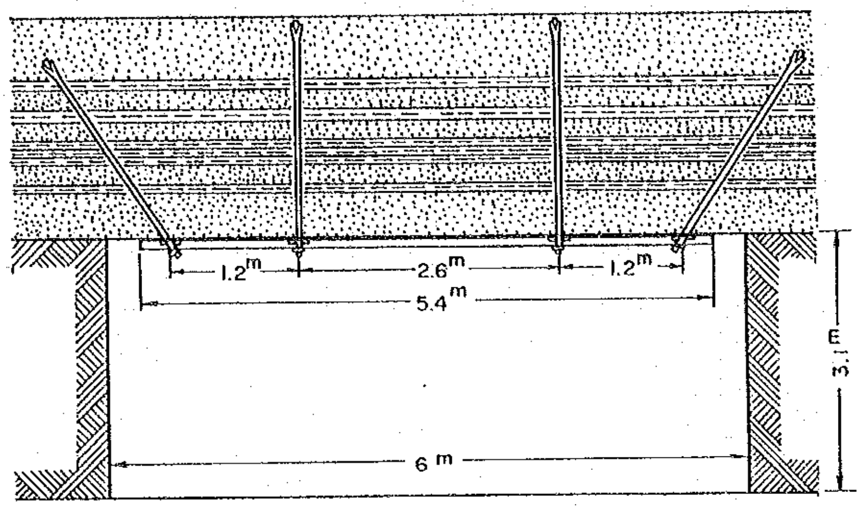


Figure Ⅲ-6 Combination of Vertical and Angle Bolting

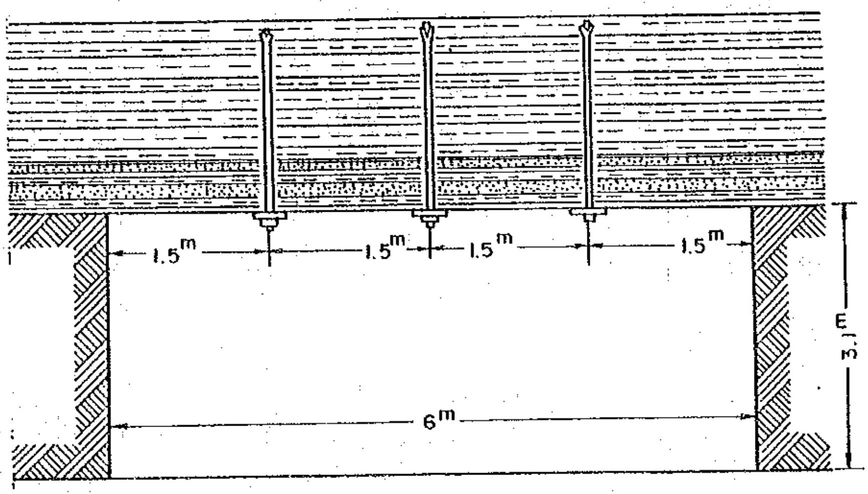


Figure Ⅲ-7 Vertical Bolting

Figure III-8 Standard of Main entry (5-Entries)

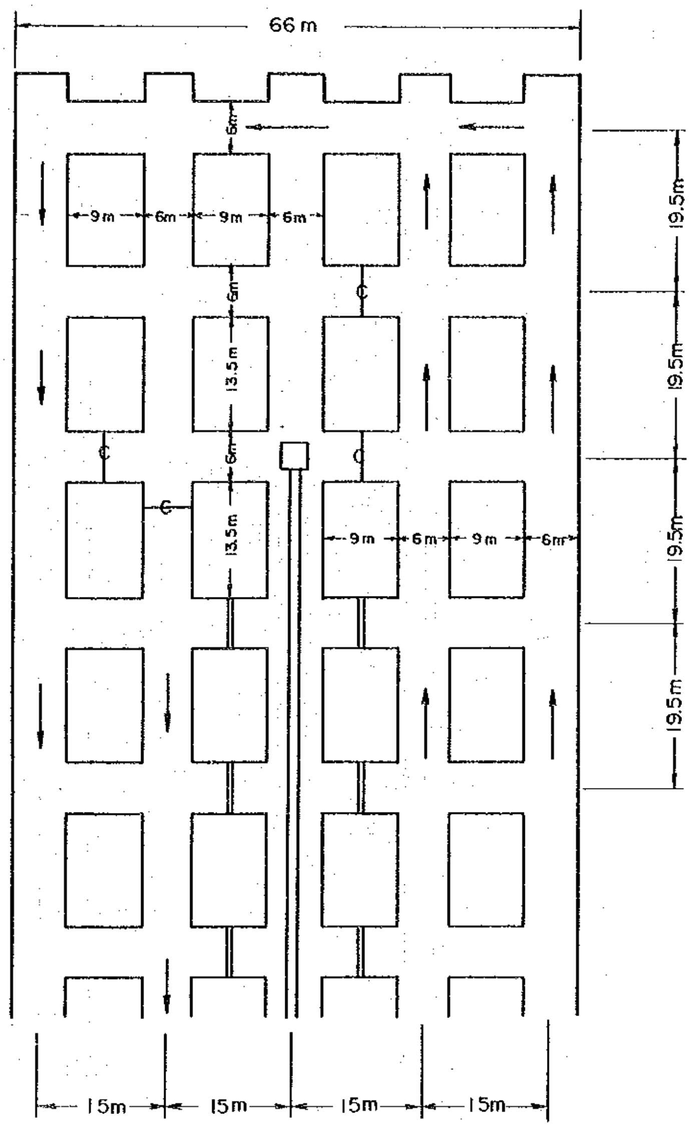
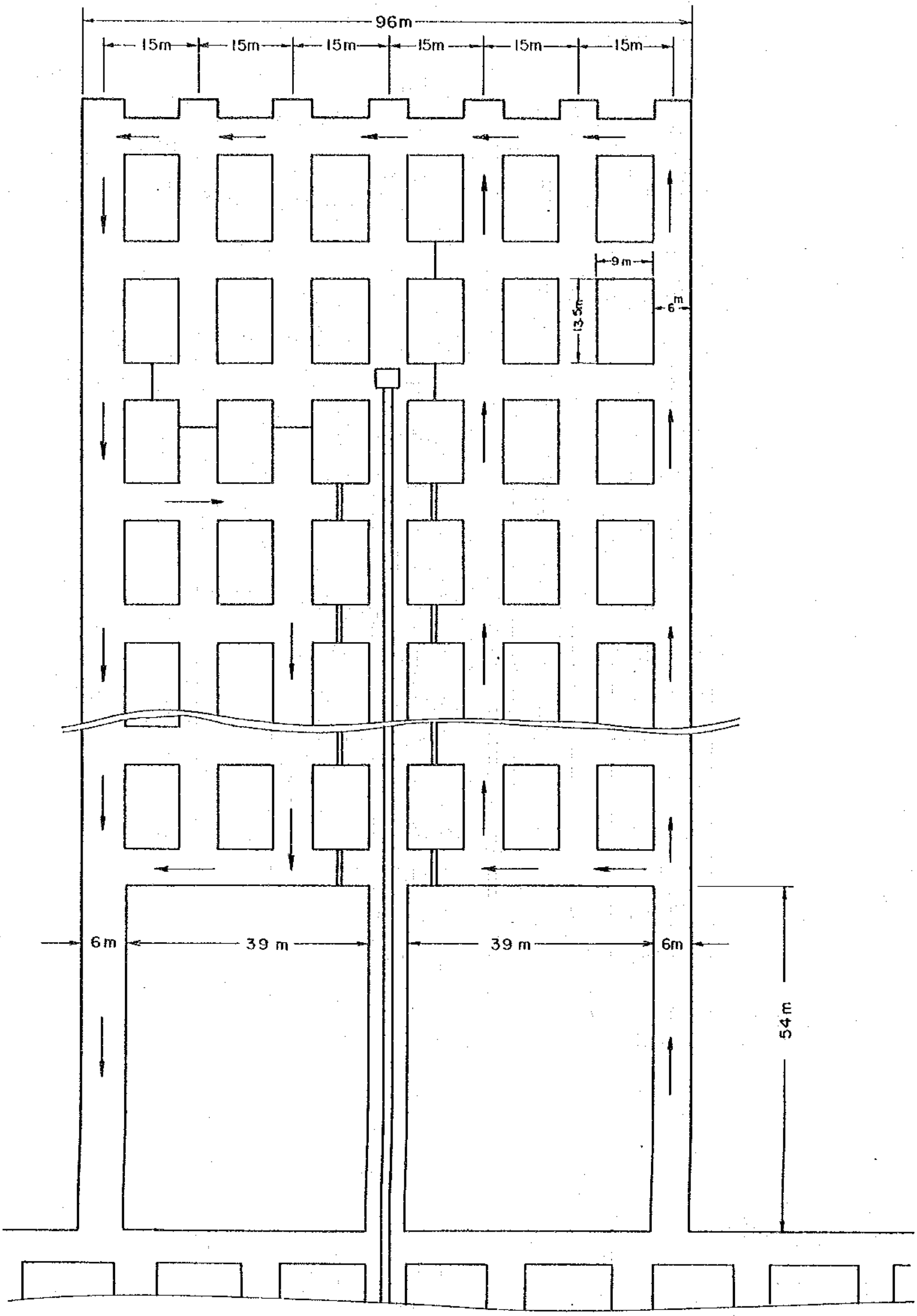


Figure III-9 Standard of Cross Main Entry (7-Entries)



2.6.1 Main Entry

The main entry (with 5 entries) is driven for about 7 km through the central part of the mining area toward LD9, in the north-east direction from the point where the inclines encounter the seam. A belt conveyor is installed in a central entry of the 5 entries.

A vertical shaft for ventilation will be sunk near LD9, about 20 years after commencement of coal production, and the ventilation will be changed to the diagonal system. However, a central ventilation system using the two inclines as stated previously is used until the upcast shaft is completed. In this case, two entries are for air intake and another two entries are for air return among the five entries.

Roof support for entries is made by friction type roof bolts (1.8 m long, 22 m in diameter). Standards of roof bolting and main entries are shown in Figures III-6, 7 and 8 respectively.

2.6.2 Cross Main Entry

The cross main entry (with 7 entries) is driven perpendicularly to the main entry, on both sides of it. The cross main entry is driven up to the boundary of the mining area. Each interval of cross main entires is set at 1,000 m since the size of one mining panel is about 440 m x 186 m. A conveyor belt is installed in the central entry of the 7 entries. This central entry is not used for ventilation by means of stopping every cut-through, 3 entries on one side are used for air intake and the other 3 entries for air return. Friction type roof bolts are used for supporting the roof of entries (Figures III-6 and 7).

The standard of cross main entry is shown in Fig. III-9. Location of the main entry and cross main entries is shown in Drawing 15.

CHAPTER 3. MINING PLAN

CHAPTER 3. MINING PLAN

3.1 Mining Method

3.1.1 Selection of Mining Method

Minable coal reserves of the Main Seam are about 35 million tons in the mining area of about 1,700 hectares surrounded by the Lubhuku Fault and "B" Fault in the west, "A" Fault in the south, "C" Fault in the east, and Dolerite Sill "F" in the north.

The Main Seam in this area has the following characteristics:

(1) The thickness ranges from 3.0 to 4.5 m (1.03 to 2.25 m in some parts where dolerite intrudes the seam).

(2) Dip of the seam is 3° to 5° which is very favourable for mining. Occurrence of the seam is very stable because of the consistent strike and dip.

(3) The seam is so rarely intruded by dolerite that mechanized mining can be employed.

(4) Roof of the seam is generally hard, massive and thick coarse-grained sandstone.

It is decided to employ room and pillar mining method which is best suited for the above-mentioned geologic conditions, and mechanized mining is planned by taking into account the scale of annual production (640,000 tons/year of raw coal) in this study.

3.1.2 Room and Pillar Mining Method

Room and pillar mining method has general advantages such as low initial investment, relatively high productivity and low production cost. In addition, this method can be managed with a simple mining technique and can be introduced easily. The equipment can be shifted more readily from one location to another when unexpected problems such as dolerite intrusion are encountered during operation.

In this plan, coal is mined out up to the end of each panel by 13 parallel entries provided in the panel and cut-throughs which connect with each entry in a grid pattern, and relatively small square pillars remain among them. After completion of the above coal mining, pillars are extracted on retreat.

A continuous miner is employed for coal extraction, transportation of raw coal is by shuttle car, and roof bolts are used for roof supporting. Figure III-10 and 11 show process of the mining.

Pillars are extracted immediately after completion of developing of the entries and cut-throughs except those located near the main entries. The roof is allowed to collapse into the mined out area foming a goaf after pillar extraction. Entrances of the entries are fully closed by blocks and coal extraction is carried out in the next panel. Drawing 16 shows year-wise coal extraction plan.

3.2 Design of Entry and Pillar

3.2.1 Entry Design

Entry span between coal pillars must be determined for the room and pillar mining.

A coaly shale parting of 30 to 60 cm thickness generally exists about 1 m below the immediate roof. This parting, is liable to be peeled off from the roof, must be taken out in order to prevent falling of the roof during the mining.

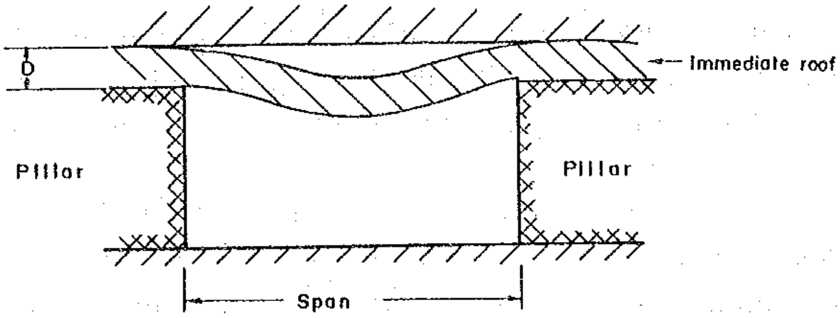
The safety span can be calculated from the following equation, for example, if the lithology of the immediate roof is apparently different from that of main roof or if the thickness of the immediate roof is less than one-half of the width of entry.

$$Sf = \sqrt{\frac{2DSt}{WF}} \quad \ldots\ldots\ldots\ldots\ldots \quad (3.1)$$

where,
- Sf: Safety span (m)
- D: Thickness of immediate roof (m)
- St: Tensile strength (kg/cm^2)
- W: Bulk density of immediate roof (kg/m^3)
- F: Safety factor

Design of safety span is shown in Figure III-12.

Figure III-12 Design of Safety Span



Immediate roof is to remain in situ when the thickness (D) exceeds 1 m in this plan. In this case, the safety span is calculated as follows:

$$Sf = \sqrt{\frac{2 \times 1 \times 500 \times 10^4}{2{,}700 \times 8}} = 21.5 \text{ m}$$

where, $St = 500$ kg/cm² (coaly shale); $W = 2{,}700$ kg/m³ ; $F = 8$.

Therefore, the entry span can be designed less than 21.5 m in case of D = 1 m.

In this study, the width of the entry is designed to be 6 m by taking into account that of the Mpaka mine, which qualifies in the above calculated safety span. The height of the entry is planned to be 3.1 m due to the working-height of the continuous miner and bolter to be employed.

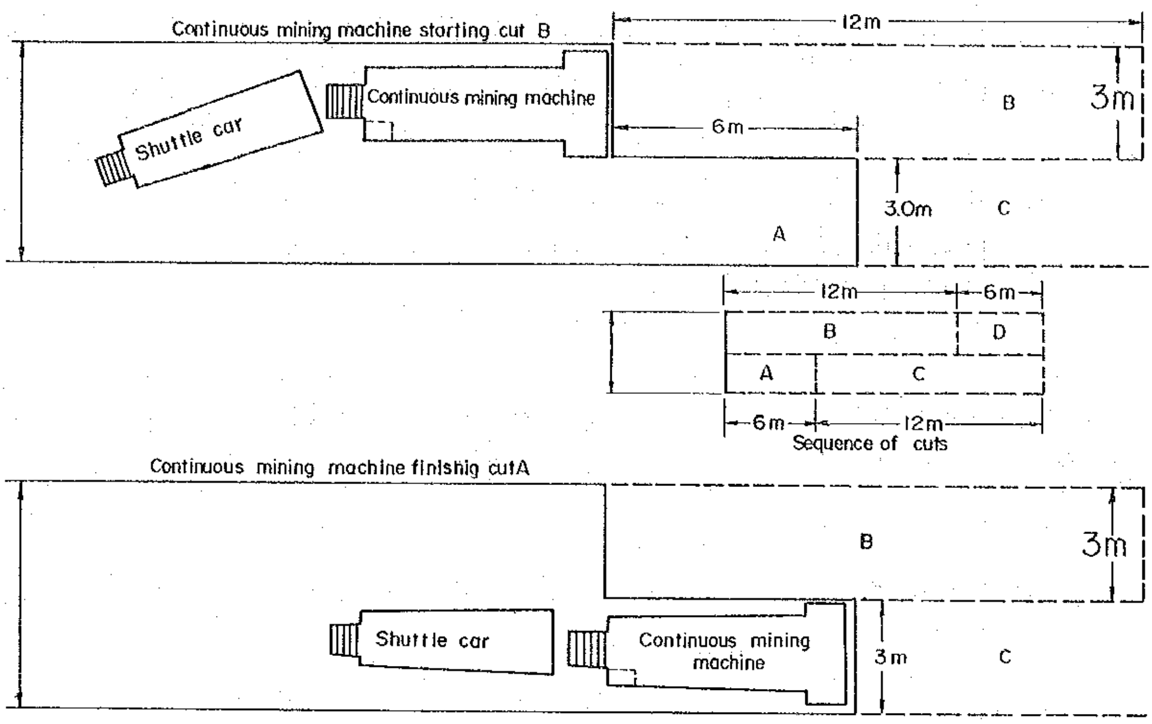


Figure III-10 Sequence of Cuts at Eentry

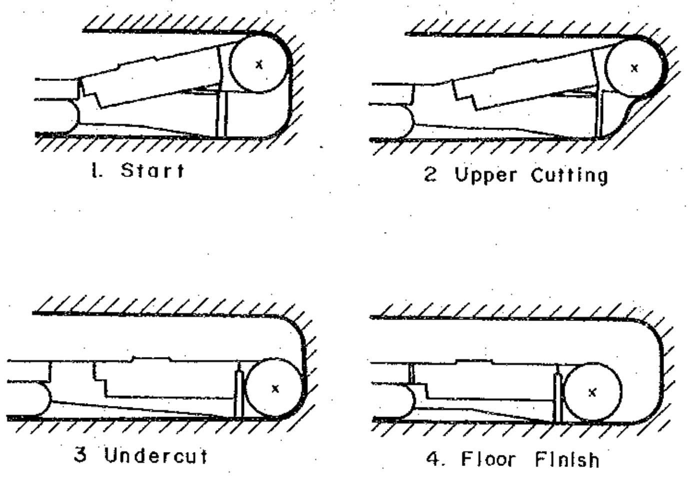


Figure III-11 Continuous Miner Operating Sequence

3.2.2 Pillar Design

Compressive strength (Sp) of the pillar is calculated from the following equation, for example, based on the unconfined compressive strength (Sc) of specimens taken from the pillar.

$$Sp = (Sc + b\ell)\sqrt{\frac{\ell}{h}} \quad \ldots\ldots \quad (3.2)$$

where, Sp: Compressive strength of pillar (kg/cm^2)

Sc: Uniaxial compressive strength of coal (kg/cm^2)

b: Coefficient of coal, b=0 if coal can be considered as a rigid body.

ℓ: Width of pillar (cm)

h: Height of pillar (cm)

In this plan, the size of the pillar is designed; length is 13.5 m, width is 9 m and height is 3.1 m. Compressive strength of the pillar is calculated as follows:

$$Sp = (200 + 0 \times 900) \times \sqrt{\frac{900}{310}} = 341 \text{ kg/cm}^2$$

where, Sc = 200 kg/cm^2

b = 0

ℓ = 900 cm

h = 310 cm

Maximum load W(kg) per pillar can be obtained from the following equation, for example:

$$\begin{aligned} W &= Sp \times A \\ &= 341 \times 10^4 \times 9 \times 13.5 \\ &= 41{,}432 \times 10^4 \text{ kg} = 414{,}000 \text{ tons} \end{aligned}$$

where, A: Cross sectional area of pillar

Therefore, the roof pressure above the pillar at the depth of 300 m below the surface (300 x 15 x 19.5 x 2.7 = 237,000 tons) can be fully supported. The safety factor shows 1.75. The size of the pillar should be increased when the development reaches deeper than 300 m below the surface.

Standard of the mining is shown in Figures III-13 and 14.

3.3 Standard of Primary and Secondary Mining

As above-mentioned, the sizes of pillar and entry are determined as follows:

Primary mining:

Size of pillar: length x width x height = 13.5 x 9 x 3.1 m

Size of entry: width = 6 m, height = 3.1 m, driving length per shift = 12 m.

Secondary mining:

Secondary mining consists of additional extraction at the entry in case the thickness of the seam exceeds 3.1 m and pillar extraction. The additional extraction is planned to dig down to a maximum of 1.5 m with a 4.5 m width. The pillar is designed to be extracted with recovery of 40%.

Outline of the secondary coal extraction is shown in Figure III-15.

Figure III-13 Standard of Underground Mining

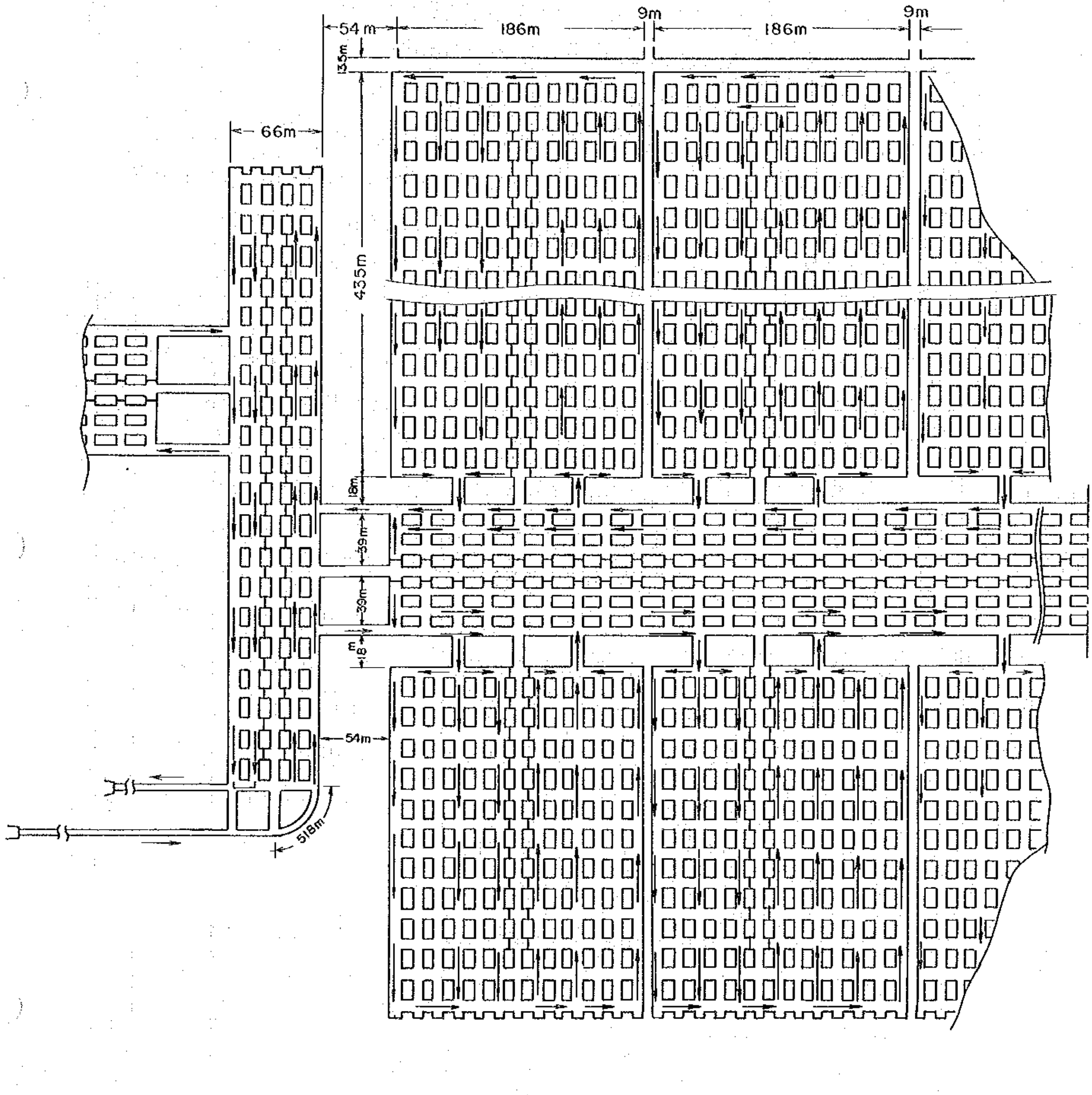


Figure Ⅲ-14 Standard of Development at Minig Panel (13-Entries)

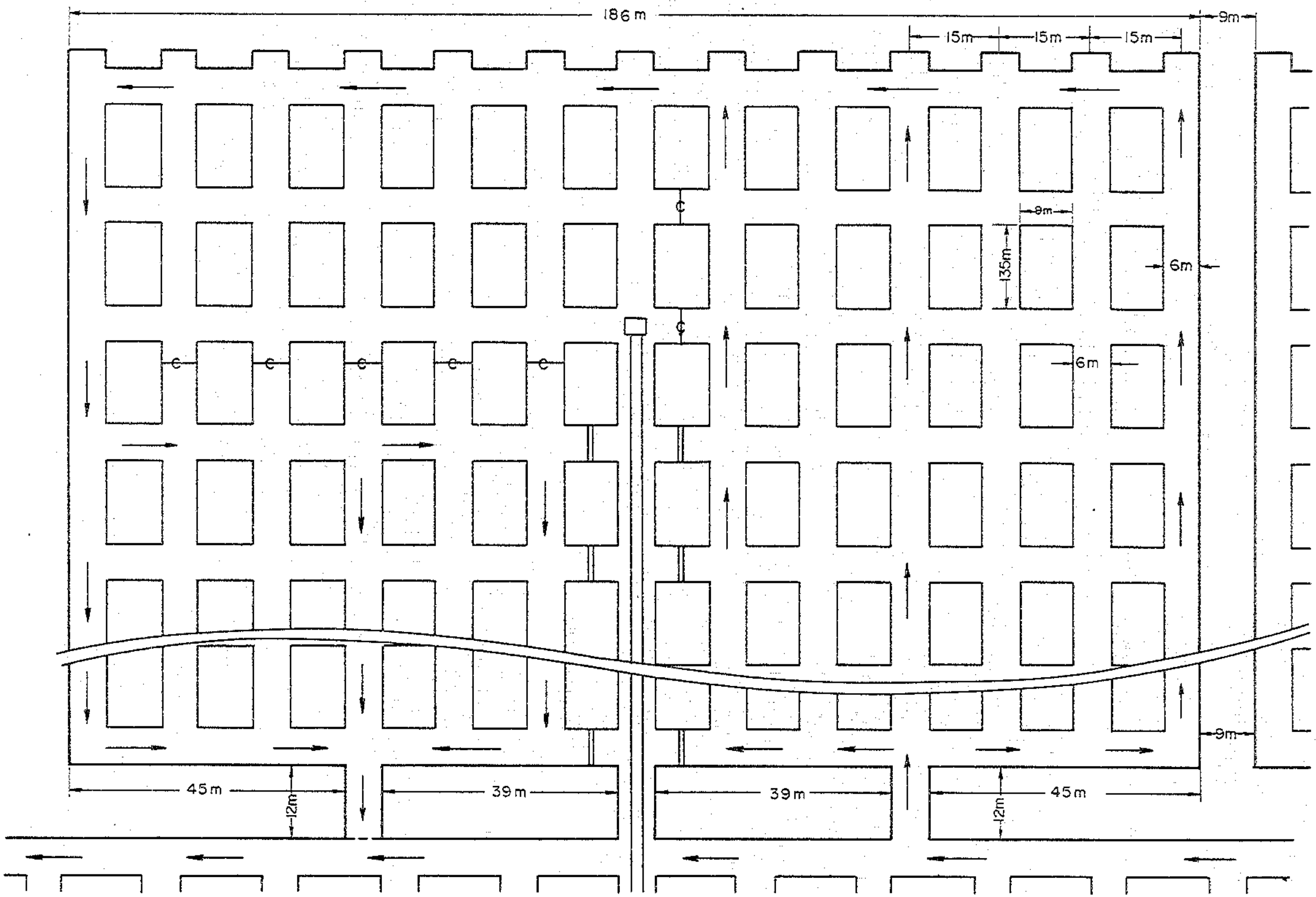
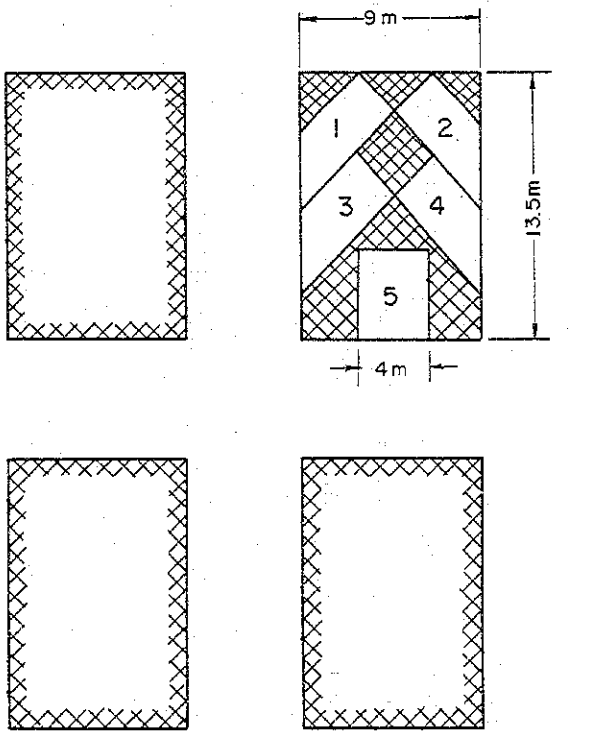
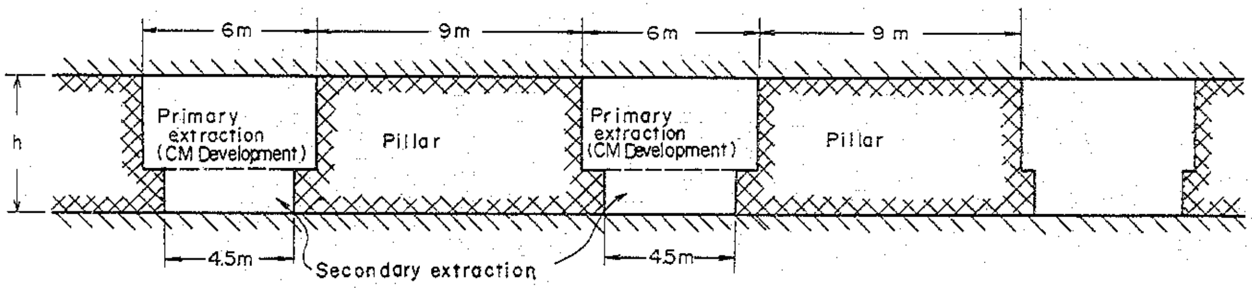


Figure Ⅲ-15 Standard of Pillar and Secondary Extraction

Pillar extraction rate : 40%



Secondary extraction (case of h>3.1m)



3.4 Standard of Roof Support

The roof of the entries is supported by inserting friction type bolts. Four bolts are required for main entry and cross main entries and three bolts for panel entries. The bolt is 1.8 m long and 25 mm in diameter. Bolt setter is shown in Figure III-16 (refer to Figures III-6 and 7 for bolt set standard).

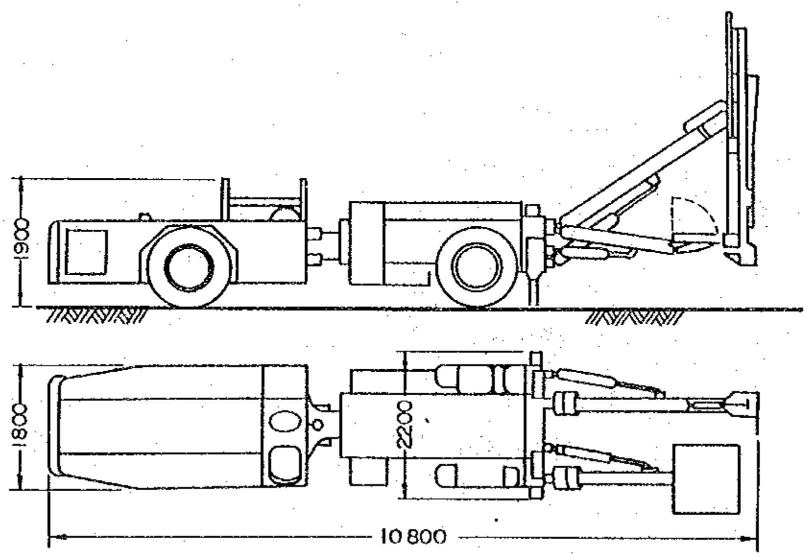


Figure III-16 Roof Bolter

3.5 Barrier Pillar

A barrier pillar is to be designed to prevent subsidence of the surface which may damage various structures on the ground such as houses, railways, roads, etc. In this study, a barrier pillar is designed so that the railway and two inclines can be protected from any damages arising from the mining activities.

Since the critical angle of the barrier pillar is required to be 60°, the width of the pillar for the railway is calculated at 214 m as shown in Figure III-17.

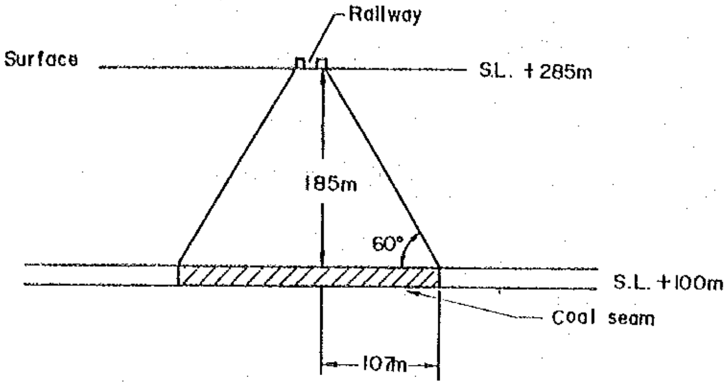


Figure Ⅲ-17 Barrier Pillar for Surface Railway

3.6 Mining Recovery

3.6.1 Mining Recovery of Thinner Coal Seam

Mining recovery of the seam with thickness below 3.1 m is estimated as follows:

· Area (S) of one mining panel is calculated below based on the mining standard:

$$S = (13.5 \text{ m} \times 22 + 6 \text{ m} \times 23) \times (9 \text{ m} \times 12 + 6 \text{ m} \times 13)$$
$$= 80{,}910 \text{ m}^2$$

where,

22 pillars and 23 entries in the face advancing direction.

12 pillars and 13 entries in direction perpendicular to the above.

Area of coal produced by primary mining;

$$13 \times 6 \times 435 + 23 \times 6 \times 186 - 13 \times 6 \times 6 \times 23 = 48{,}834 \text{ m}^2$$

Area of barrier pillars required for setting up of one mining panel:

$$18 \text{ m} \times 195 \text{ m} + 9 \text{ m} \times 435 \text{ m} = 7{,}425 \text{ m}^2$$

Primary mining recovery:

$$\frac{48{,}834}{7{,}425 + 80{,}910} = 0.5528$$

Secondary mining recovery based on 40% pillar extraction:

$$\frac{(80{,}910 - 48{,}834) \times 0.4}{7{,}425 + 80{,}910} = 0.1452$$

Therefore, the total mining recovery of one mining panel is 69.8%.

3.6.2 Mining Recovery of Thicker Coal Seam

Mining recovery of the seam with the thickness of 4.6 m is estimated as follows:

Primary mining recovery:

$$\frac{48{,}834 \times 3.1}{88{,}335 \times 4.6} = 0.3725$$