CHAPTER 4 ECONOMIC GEOLOGY

(Table 1, 3, 10, 11, 12, 13, 14, 15, 16 & 17. Figures 2, 8, 11, 12, 13, 14, 15, 16, 17, 18, & 19)

4-1 Summary

The exploration and drilling programme was undertaken from June, 1979 over 26 square kilometres area in the western part of the compact block of PMDC's licences measuring 52 square kilometres. 50 drill holes (JT1 to JT50) with a total depth of 5,203.06 metres were drilled from June 27 to November 22, 1979.

In addition to the data collected through these 50 holes, the data of 3 holes (L18, 22 and 28) with total depth of 334.36 matres executed by GSP, and 19 holes (PS1, 2, 4, 5, 6, 7, 8, 9, 10, 11, 13, 14, 15, 18, 19, 20, 22, 23 and 24) with total depth of 1,844.93 metres drilled by PMDC were used for the evaluation of the investigated area. Coal and rock samples were collected and tested in the laboratory for determining quality and suitability of coal deposits as fuel for power generation.

As previously mentioned, the coal seams lie in the Upper and Lower Coal-bearing Beds of Lower Ranikot formation. The main coal seams which were previously called Lailian, Dhanwari and Kath exist in Upper Coal-bearing Beds. The Upper Coal-bearing Beds contain a maximum of 11 seams, and Lower contains upto 8 seams including thin and poor ones. The seams have been grouped into zones depending on their mode of occurrence, rock facies bearing them, and other characteristics. Five coal zones have been recognized in the Upper Coal-bearing Beds. They are referred as coal zone No. 1, No. 2, No. 3, No. 4 and No. 5 in ascending order, in the present report. Similarly, three coal zones – L1, L2 and L3 in descending order occur in the Lower Coal-bearing Beds.

It is very difficult to correlate the seams of the Upper Coal-bearing Beds due to following reasons:–

1 – Lack of key bed,
2 – Excessive lateral variation of rock facies,
3 – Existence of secondary altered rocks such as strata of 'red zone',
4 – Excessive lateral variation in thickness of coal seams, mingling and thinning, and fishing out.

Inspite of this correlation has been attempted since in case of Lower Coal-bearing Beds it is rather easy to correlate the seams. The Lower Shell Beds, which were affected by marine transgression underlie the Upper Coal-bearing Beds. Moreover, No. 1 zone is persistent almost over whole of the area. However, there is a possibility that different coal seams, particularly interfingering coal lenses may be considered as one continuous seam.

Each coal zone occurring in the Upper Coal-bearing Beds could be traced, except in red zone, by the presence of carbonaceous matter in rock layers.

The coal seams belonging to Lower Coal-bearing Beds are likely to continue over the entire area although the existence of continuity could not be confirmed. It is expected that further exploration will enable proper correlation.

Thickest coal seam found in the Lower Coal-bearing Beds is 0.92 metre in JT39. Generally thin coal seams in the area have not been considered as reserve. Mostly each coal zone contains more than one seams in the Upper Coal-bearing Beds, but it is difficult to evaluate them by taking into account their real thickness for the open-cut and underground mining because of different working heights in both the methods. The minimum workable thickness of minable seams are 0.50 metre for open-cut and 0.75 metre in case of underground mining. The coal zones No. 1, 2 and 3 with minable seams spread over most of the area, while zones No. 4 and 5 are confined only in the western part of the barren area. Maximum workable thickness of seams reaches upto 3.0 metres. The northern part and an area in the western part are either barren or consist of thin seams, which can not be mined.

4-2 Detailed Description of Coal Zones.

4-2-1 Lower Coal-bearing Beds

The coal seams are classified into zones L1, L2 and L3 in descending order based on confirmed data although the whole area has not been explored for the Beds and their bottom could not be studied properly. The seams are thinner and less significant than those of the Upper Coal-bearing Beds. Most important seam is only 0.92 metre thick found in drill hole JT39. Total number of seams encountered in the Beds are 8, but none of the seams are considered as reserve, because of low quality, poor thickness and insufficient data.

(1) Coal Zone L3

The zone occurs in drill-holes JT12, JT13, JT39, and JT50. It has been confirmed to be the oldest in the Beds. It contains 1-4 thin and insignificant seams of bad coal/coaly shale ranging in thickness from 0.10 to 0.40 metre. Shales and claystones are main rock units of the zone, which is from trace to 5 metres thick.

Roof of the zone consists of shale, siltstone and sandstone, whereas the floor comprises of shale, claystone and siltstone. Depth of the zone as encountered in drill holes is from 105 to 130 metres from the surface.

(2) Coal Zone L2

The zone has been encountered in 9 drill holes JT11, 18, 20, 28, 33, 37, 45, 49 and PS15 in addition to above mentioned 4 drill holes (JT12, 13, 39 and 50). It lies 5–23 metres above the zone L3, and also consists of 1–4 small seams of bad coal to black shale ranging in thickness from 0.10 to 0.92 metre. The zone itself is from trace to 4 metres thick. It is composed mostly of claystones, which also form the roofs and floors of coal seams. Immediate roof of the zone consists of sandstone, siltstone and shale, whereas the floor is made up of siltstone, sandstone, shale and claystone. Its depth ranges from 83–120 metres as encountered in drill holes. Some samples were collected from the drill holes JT11, 13, 20, 28, 39 and 49 for analysis.

(3) Coal Zone L1

It occurs in 20 drill holes-JT6, 17, 19, 23, 42, 44 and 46, and the 13 holes in which zone L2 has been encountered. It lies 5 to 20 metres above the zone L2. It contains 1–2 small seams of bad coal/coaly shale ranging in thickness from 0.30 to 0.91 metre, whereas its thickness ranges from trace to 3.0 metres.

Immediate roof of the zone mostly consists of siltstone, claystone and sandstone, while the floor is made up of shale, claystone, sandstone, and siltstone. Depth of the zone is from 70–122 metres from surface. Few samples of the zone were collected from the cores of drill holes JT13 and JT45 for study in the laboratory.

4-2-2 Upper Coal-bearing Beds

They bear five coal zones, which have been intersected in all the drill holes. All the zones from No. 1 to No. 5 are well developed in the western part of the investigated area. However, No. 4 and No. 5 zones are poorly developed in the eastern part.

As mentioned above there is either no coal or poorly developed insignificant coal seams in the northern part and over a small area in the western part of the investigated area. Therefore the investigated area is divided into small western block, and vast eastern area, which is further divided into two blocks – eastern and western lying respectively to the east and west of the fault B1. Thus the coal bearing area is divided into three blocks – western, central and the eastern. The minimum and maximum depths of coal seams in the blocks are as under:–

Block	Minimum	–	Maximum depths (m)
Western	32	–	75
Central	60	–	126
Eastern	43	–	107

(1) Coal Zone No. 1

It is the oldest zone of the Upper Coal-bearing Beds, and consists of 1–5 seams ranging in thickness from 0.2 to 3.0 metres. The zone itself is from 0.2 to 7.0 metres thick.

Besides coal seams, the zone consists of shale, claystone and occasionally siltstone and sandstone. Roof of seams consists of shale, claystone and sandstone, whereas the immediate floor is constituted by shale, siltstone, claystone and rarely sandstone.

The zone varies in thickness locally but it extends over the entire area. It approaches gradually towards No. 2 zone and finally merges into it on the eastern side of the lines of JT33-JT34, JT28-PS7, JT27-JT29, JT12-JT24, and JT25-JT26.

Coal seams of the zone could be correlated with Lailian and Kath seams.

(2) Coal Zone No. 2

This zone merges into zone No. 1 on the eastern side, but lies 1.0–13.5 metres above it in the western part of the investigated area. It bears 1 to 4 seams ranging from 0.1 to 2.9 metres in thickness. Its thickness is 0.1–8.0 metres. It consists of shale, claystone, siltstone and sandstone between the seams. Roof is nostly composed of siltstone, shale, sandstone and rarely claystone. Loose sandstone appears in the roof towards east. Therefore, much attention should be paid on safety for underground mining. Floor rocks generally consist of siltstone, claystone, shale and rarely sandstone. The zone persists in the entire area, but the coal seams thinout in the barren portions. Its seams can be correlated with Lailian and Dhanwari.

(3) Coal Zone No. 3

It is well developed in the eastern part. Save few places where it becomes thin and bears coaly shale, the zone is fairly persistent after No. 1 and No. 2 zones. It lies 1 to 21 metres above the zone No. 2, and contains 1–3 seams, which are from 0.05 to 2.56 metres thick. Its composite thickness has been noted from 0.05–4.5 metres. Coal seams in the zone are interbedded with shale, claystone, siltstone and rarely sandstone. Their roofs consist of shale, siltstone and sandstone, and floors – siltstone, claystone and rarely sandstone. The zone can be correlated with Lailian or Dhanwari seams.

(4) Coal Zone No. 4

It is fairly developed in the western block, but the seams are not important from mining point of view. It lies 1–10 metres above the zone No. 3, and bears 1–2 coal seams ranging from trace to 1.1 metres in thickness. It comprises of shale, claystone, siltstone, and sandstone all of which form the roofs and floors of coal seams. The zone can be correlated with Dhanwari seam.

(5) Coal Zone No. 5

The zone, except in the western part, is poorly developed. It lies 3 to 27 metres above the zone No. 4. It contains only one coal seam from trace to 1.2 metres, which also is the thickness of the zone itself. In the central and eastern blocks the seam is replaced by siltstone and sandstone with some carbonaceous material. The roof and floor of the seam consists of siltstone, shale and sandstone. Also the roof of the zone is the top of Upper Coal-bearing Beds.

4-3 Studies for Coal Quality

Many samples were collected from the cores of 50 drill holes executed during the course of work. Amongst others they include 189 samples from the coal seams and thin partings, and 32 samples from the thick partings. Tests and analyses have been carridout in Japan to determine proximate constituents, calorific value, total sulphur and specific gravity of the 189 samples on air dried basis. Also 24 samples, and 11 combined samples (prepared by mixing more than 2 samples from the batch of 189) were tested for incombustible/combustible, organic and inorganic sulphur content, ash fusion temperature, ultimate analysis (dry ash free basis), ash analysis and Hardgrove grindability index. 32 samples from the thick partings were tested for determination of moisture, ash, total sulphur and specific gravity on air dried basis.

In addition to above 10 samples were subjected to float and sink, and spontaneous combustibility tests. Further, 11 combined samples, each with more than 2 original samples, were also checked for moisture and sulphur content. The analytical data were affected by moisture content. Therefore, the data were calculated on dry basis too.

4-3-1 Coal Characteristics observed Megascopically

The seams belonging to Ranikot formation contain mostly lignite and rarely sub-bituminous coal in rank. Coalification does not depend on the stratigraphical depth as the characteristics of seams in Upper and Lower Coal-bearing Beds appear to be same megascopically. Colour of coal is dark brown to dull black, sometimes bright black. Some coal samples are rather hard

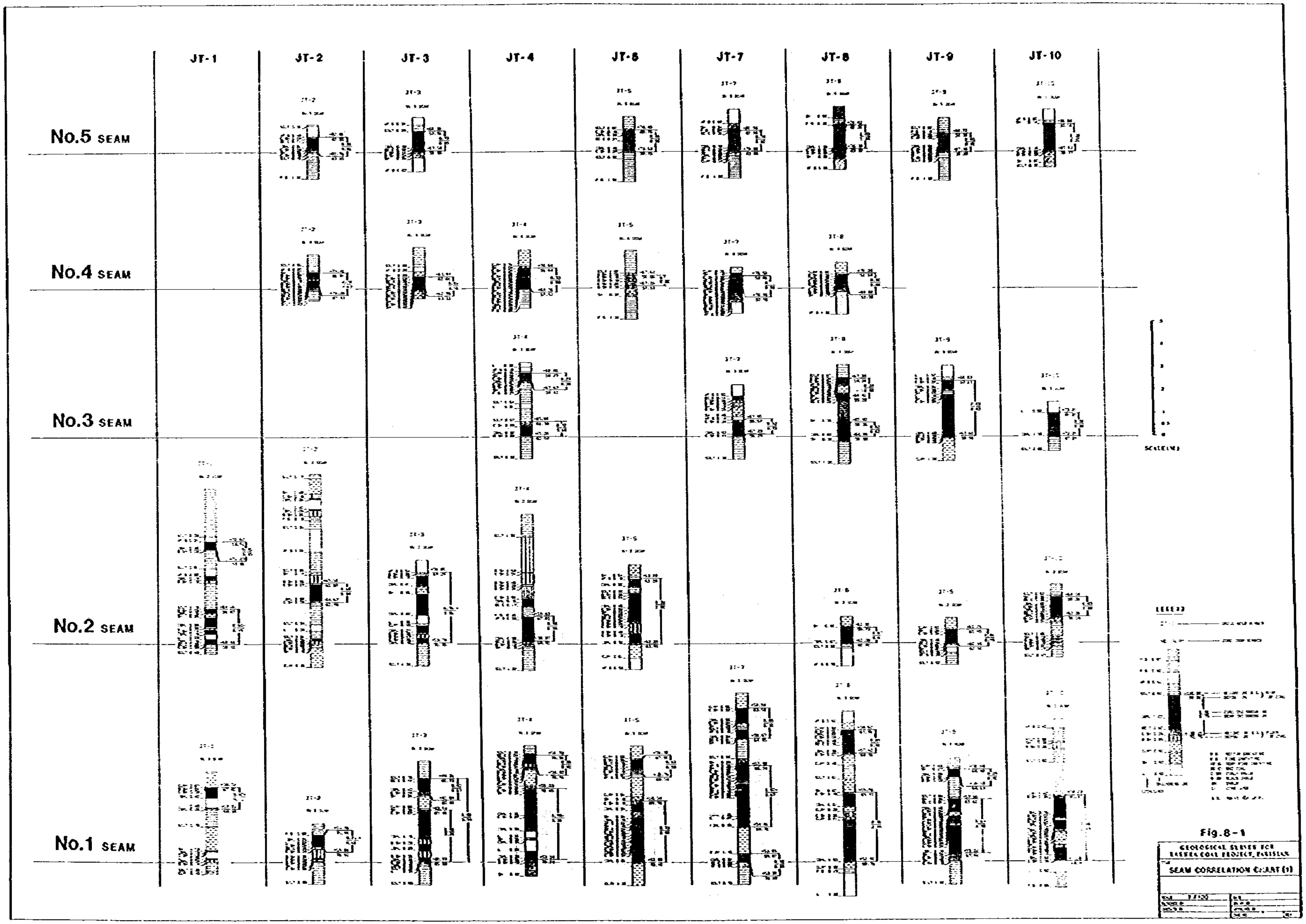


Fig.8-1

SEAM CORRELATION CHART (1)

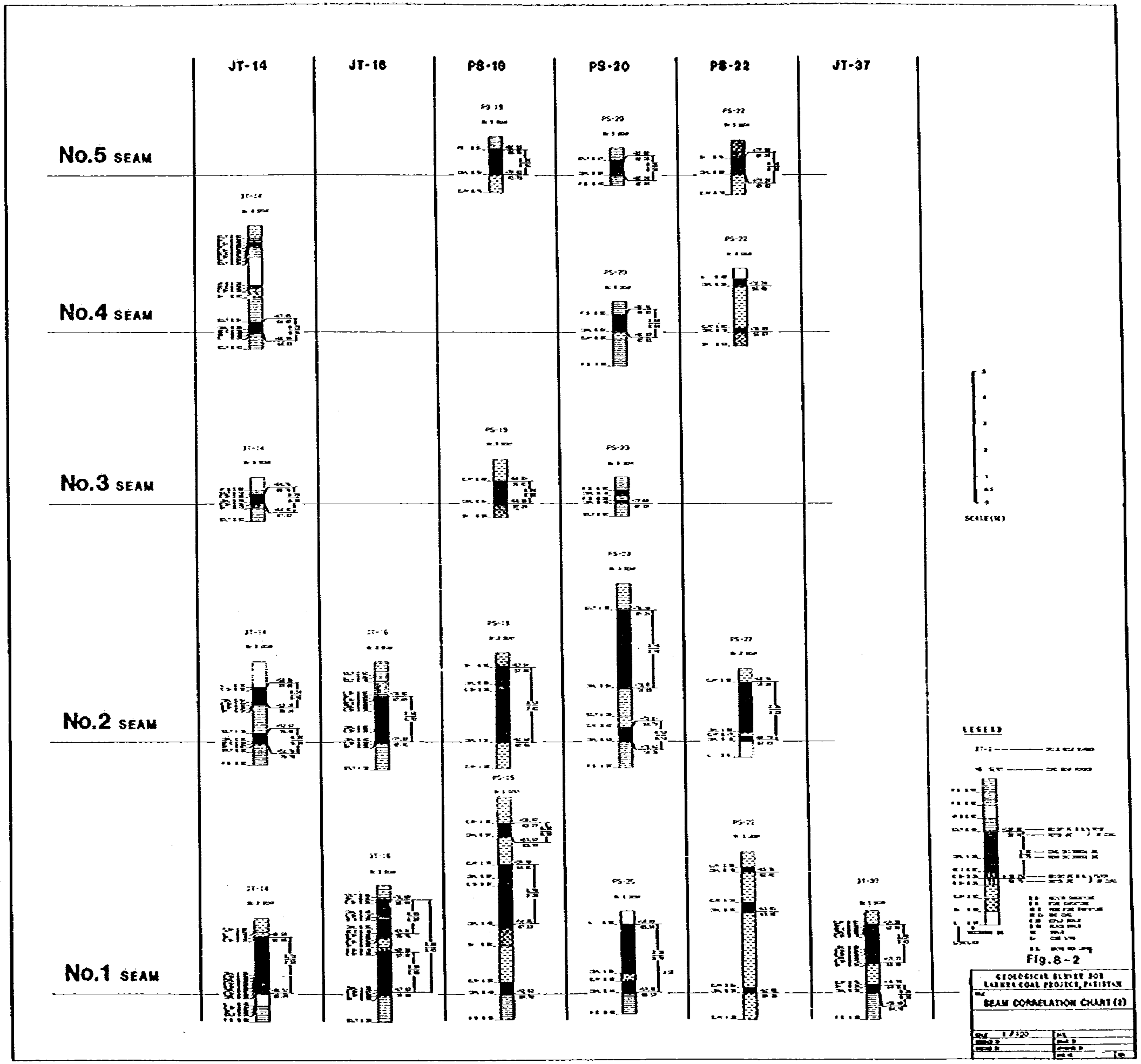

JT-14
JT-16
PS-19
PS-20
PS-22
JT-37
No.5 SEAM
No.4 SEAM
No.3 SEAM
No.2 SEAM
No.1 SEAM
LEGEND
SEAM CORRELATION CHART (2)

Fig.8-2

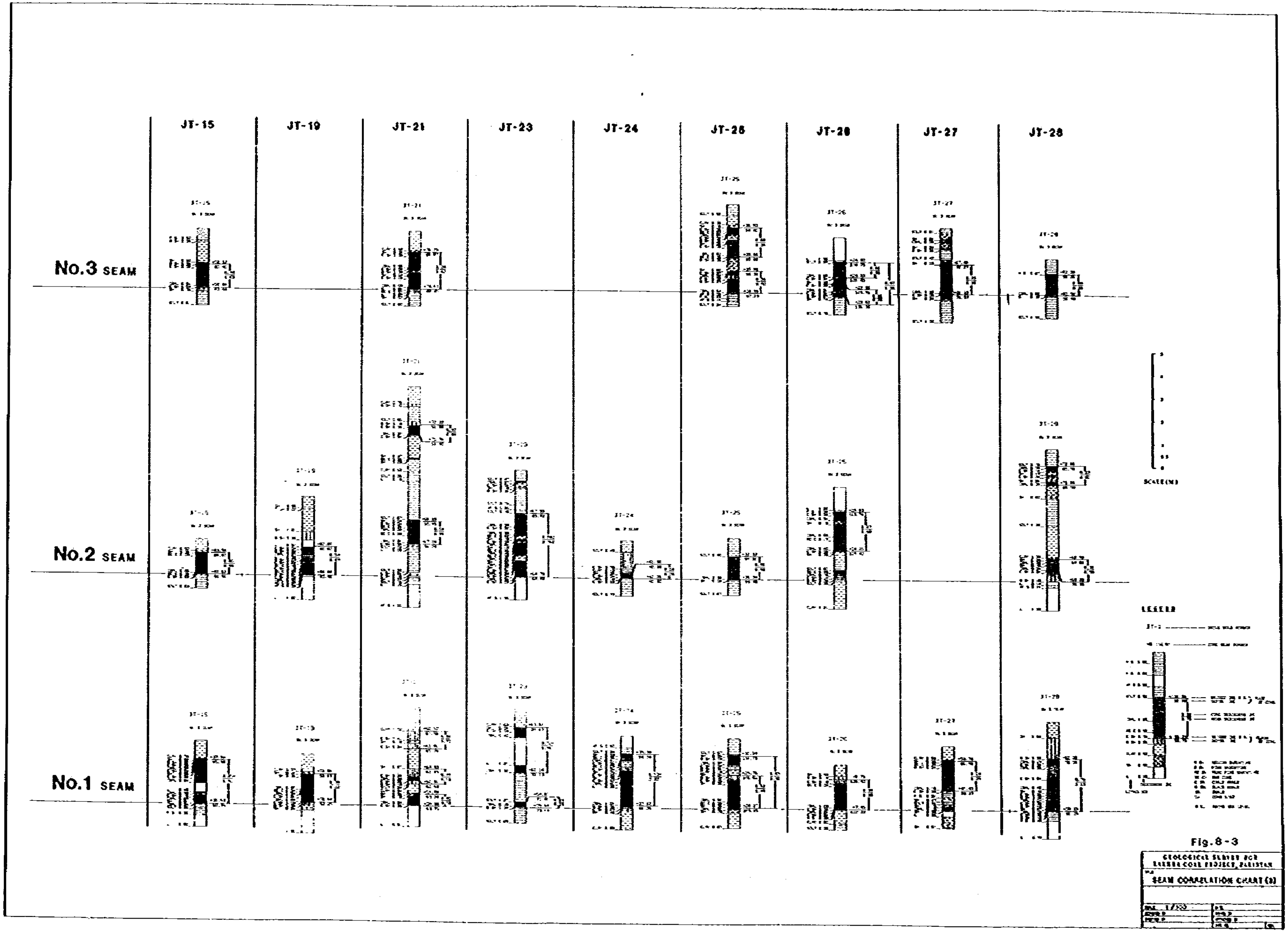

JT-15
JT-19
JT-21
JT-23
JT-24
JT-25
JT-26
JT-27
JT-28
No.3 SEAM
No.2 SEAM
No.1 SEAM
SEAM CORRELATION CHART (3)

Fig. 8-3

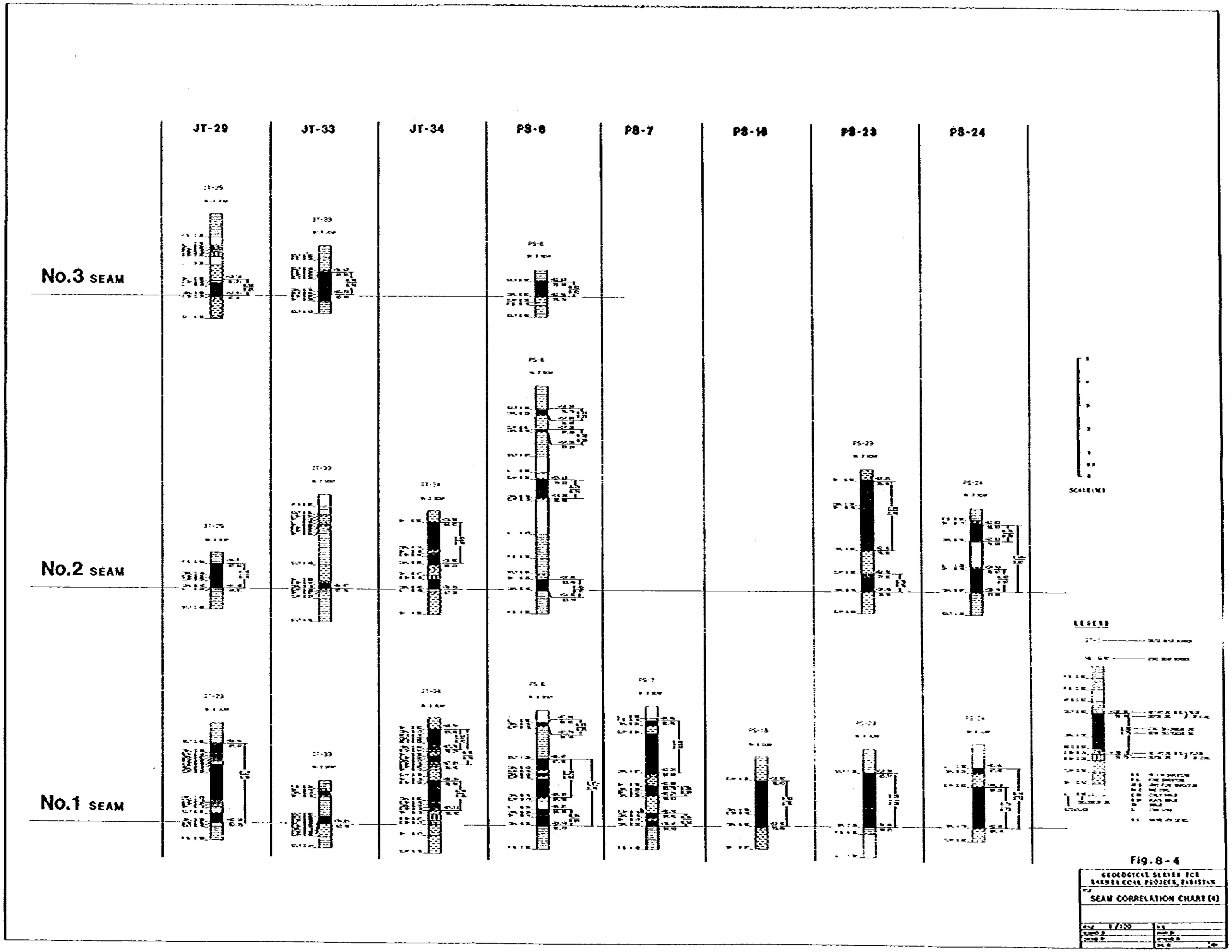


Fig. 8-4

GEOLOGICAL SURVEY FOR LAKHRA COAL PROJECT, PAKISTAN

SEAM CORRELATION CHART (4)

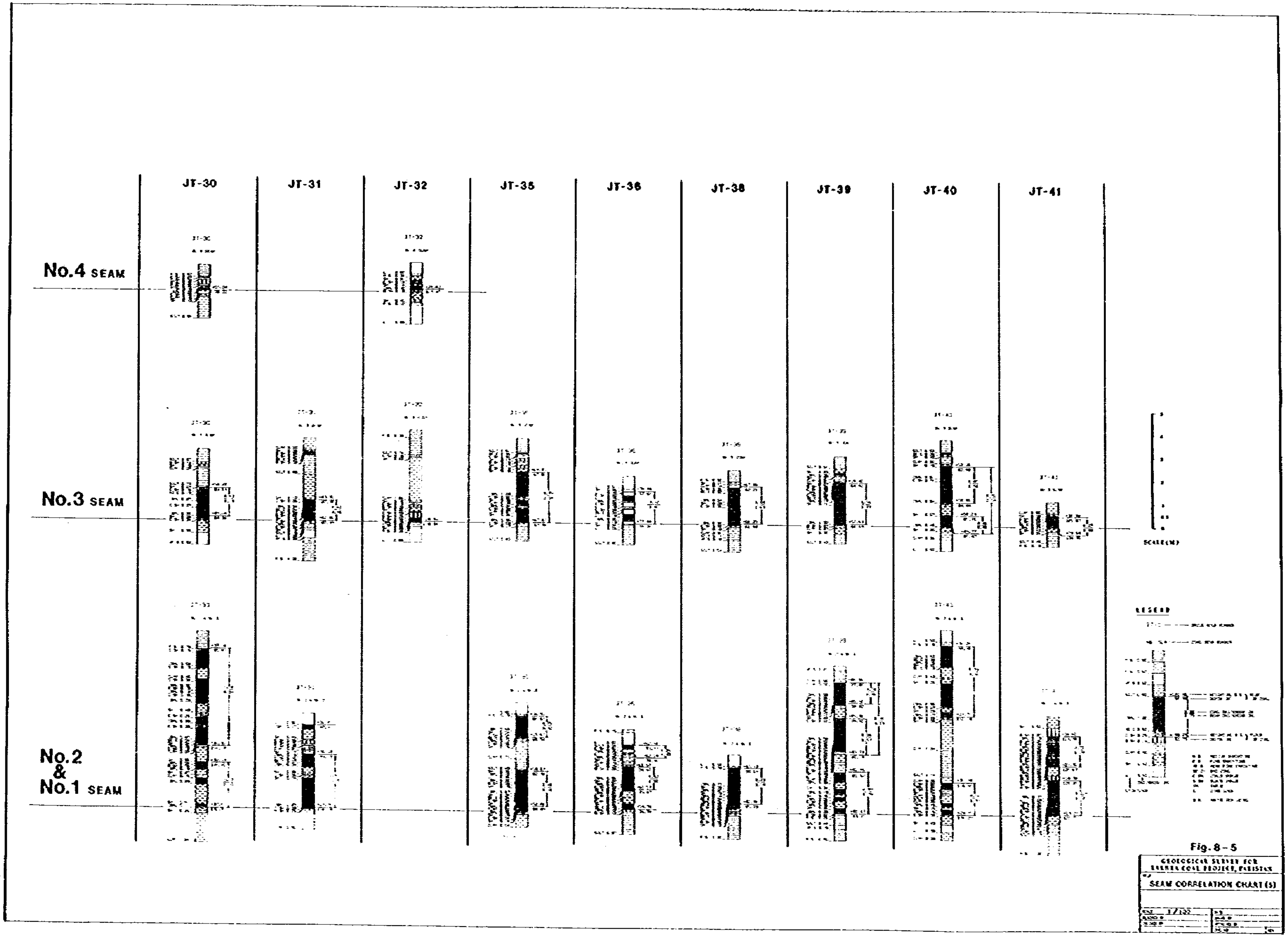

JT-30
JT-31
JT-32
JT-35
JT-36
JT-38
JT-39
JT-40
JT-41
No.4 SEAM
No.3 SEAM
No.2
&
No.1 SEAM
LEGEND
SEAM CORRELATION CHART (5)

Fig. 8-5

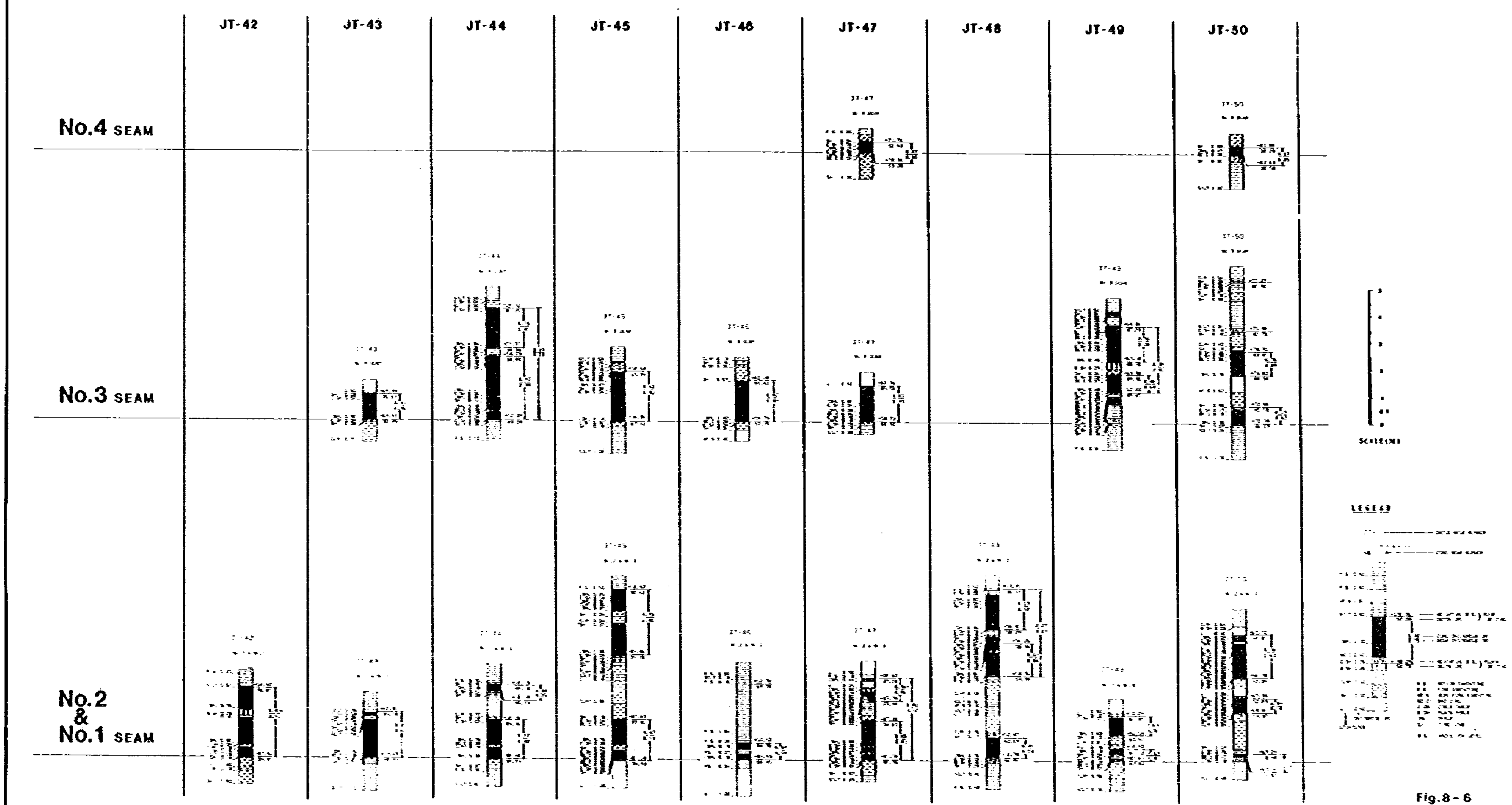


Fig.8-6

GEOLOGICAL SURVEY FOR LAKHRA COAL PROJECT, PAKISTAN

SEAM CORRELATION CHART (8)

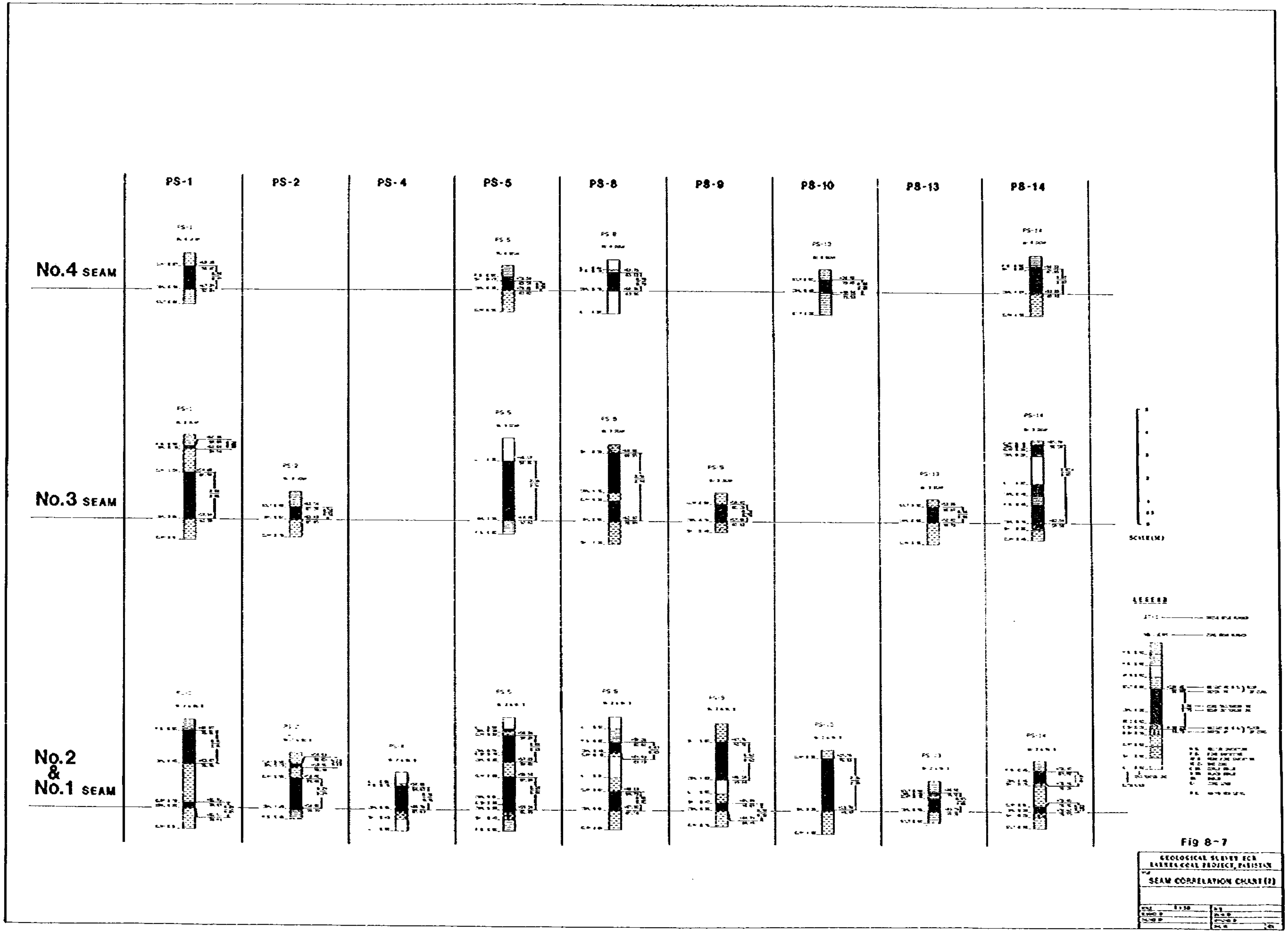

PS-1
PS-2
PS-4
PS-5
PS-8
PS-9
PS-10
PS-13
PS-14
No.4 SEAM
No.3 SEAM
No.2 & No.1 SEAM
SEAM CORRELATION CHART

Fig 8-7

immediately after the core sampling, but they generally crumble and become powdery and friable like shale after the exposure to air. In general partings of coaly shale, shale and claystone are harder than coal.

A number of grains, crystals and films of pyrite and marcasite are irregularly distributed in the seams. Gypsum veins are also common generally along and parallel to the foot wall. Some coal seams are fairly resinous. Intercalations of fine grained sand, silt and clay in the shape of pipe, patch, lense and lamination have been noted in quite a few seams near the roof. These impurities bear upon the quality of coal.

4-3-2 Quality of Coal

The proximate analysis and measurements of calorific value have been undertaken on air dried basis. Results are detailed in Tables 10 & 12. Analytical data of coal for reserve calculation on air dried, and dry (moisture free) bases are given in Table 11. (The moisture content of samples decreases considerably from 30 percent (PMDC, 1976) to 5.5 to 14.0 percent when dried in air.) It is difficult to fix the limits of ash content and calorific value for the useful coal. Assuming that the minimum desired calorific value is 3,500 kcal/kg on air dried basis (4,000 kcal/kg or 6,300 BTU/lb on dry basis), the ash content of the coal would be around 35 percent (37.5% on dry basis). The minimum ash content of Lakhra coals is 7.6 percent (8.7 on dry basis) with the calorific value of 5,860 kcal/kg (6,690 kcal/kg or 10,550 BTU/lb on dry basis).

The total sulphur content is fairly high i.e. 3.3 to 18.1 percent (3.7 to 19.87% on dry basis). The ratio of inorganic to organic sulphur is 66 to 34. Thus the inorganic sulphur is higher than organic sulphur. According to the ultimate analysis on dry ash free basis the composition of coal is as under.

Carbon	58.5% – 72.4%	Hydrogen	4.5% – 5.8%
Oxygen	14.4% – 22.3%	Nitrogen	0.9% – 1.4%
Sulphur	2.4% – 16.7%		

Regarding the ash fusion it has been determined that the initial deformation's temperature is 1,250° – 1,425°C, hemispherical from 1,300° – 1,450°C and flow 1,350° – 1,450°C or more. Hardgrove grindability index is measured from 59 to 88 (97 for coal in zone L1 in the Lower Coal-bearing Beds) which is favourable for use in boiler.

The ash consists of following constituents.

Fe_2O_3	17.30 – 70.76%	
SO_3	1.93 – 7.47%	
SiO_2	8.0 – 44.42%	(relatively small)
Al_2O_3	9.49 – 28.80%	(relatively small)
CaO	1.29 – 12.54%	
Na_2O	0.42 – 2.38%	
MgO	0.93 – 5.00%	
K_2O	0.33 – 1.99%	

Table II-1 ANALYTICAL DATA OF COAL FOR RESERVE CALUCULATION (1)

D.H.NO.	SEAM NO.	SAMPLE NO.	PROXIMATE ANALYSIS MOIST. (%)	ASH (%)	VOL. (%)	F.C. (%)	CAL. V. (Kcal/Kg)	CAL. V. (D. B.) (Kcal/Kg)	TOTAL SUL. (%)
JT-1	NO.2	1+2+3	7.6	28.9	32.6	30.9	4,500	4,892	6.8
	NO.1	4	8.3	20.4	35.2	36.1	5,000	5,453	8.0
JT-2	NO.5	1	7.3	34.0	30.5	28.2	4,020	4,337	7.1
	NO.2	2	8.9	24.4	33.1	33.6	4,660	5,115	6.2
	NO.1	3	8.5	27.1	34.1	30.3	4,230	4,623	12.3
JT-3	NO.5	1	9.5	20.3	33.7	36.5	4,830	5,337	7.5
	NO.4	2	7.6	40.2	28.8	23.4	3,360	3,636	9.0
	NO.2	3+4+5	8.9	28.7	33.3	29.1	4,147	4,568	5.4
	NO.1	6+7+8	8.5	33.9	30.9	26.7	3,774	4,131	9.9
JT-4	NO.4	1	6.4	44.3	28.1	21.2	3,190	3,408	6.6
	NO.3	2	7.0	36.4	28.7	27.9	3,960	4,258	5.5
	NO.2	3	8.6	16.6	40.4	34.4	5,160	5,646	7.0
	NO.1	4+5	8.3	21.9	36.2	33.6	5,003	5,460	9.5
JT-5	NO.5	1	8.8	24.2	33.9	33.1	4,830	5,296	6.7
	NO.2	2+3+4+5+6	7.6	31.6	32.1	28.7	4,050	4,404	7.5
	NO.1	8+9+10	6.8	32.7	32.7	27.8	4,065	4,374	9.0
JT-7	NO.5	1	8.3	25.5	34.4	31.8	4,660	5,082	7.6
	NO.3	2	8.5	18.5	36.3	36.7	5,290	5,781	3.9
	NO.1	3+4+5+6+7	8.1	22.0	35.8	34.1	4,840	5,283	7.0
JT-8	NO.5	1	8.2	19.4	35.9	36.5	4,900	5,338	8.3
	NO.3	2	8.9	9.7	39.1	42.3	5,810	6,378	3.3
	NO.2	3	8.2	15.0	37.6	39.2	5,390	5,871	5.6
	NO.1	4+5+7	7.4	20.4	37.0	35.2	5,046	5,452	7.6

Table 11-2

ANALYTICAL DATA OF COAL FOR RESERVE CALUCULATION (2)

D.H.NO.	SEAM NO.	SAMPLE NO.	PROXIMATE ANALYSIS MOIST. (%)	ASH (%)	VOL. (%)	F.C. (%)	CAL. V. (Kcal/Kg)	CAL. V. (D. B.) (Kcal/Kg)	TOTAL SUL. (%)
JT-9	NO.5	1	9.2	28.0	34.1	28.7	4,450	4,901	8.9
	NO.3	2+P1+3	8.1	27.8	37.6	26.5	4,245	4,625	7.3
	NO.2	4	7.6	24.7	34.6	33.1	5,240	5,671	8.9
	NO.1	6+7+8	6.2	34.5	31.7	27.6	3,951	4,247	8.6
JT-10	NO.5	1	8.7	25.6	31.4	34.3	4,600	5,038	5.6
	NO.3	2	8.9	14.5	37.5	39.1	5,440	5,971	6.8
	NO.2	3	9.0	12.3	38.2	40.5	5,500	6,044	4.2
	NO.1	4+5	7.4	25.0	34.8	32.8	4,590	4,958	8.5
JT-14	NO.2	1	10.2	10.1	38.0	41.7	5,930	6,604	5.8
	NO.1	2+3	9.6	21.5	35.3	33.6	4,828	5,358	9.8
JT-15	NO.3	1	8.9	9.0	38.8	43.3	5,900	6,476	6.7
	NO.2	2	9.7	16.0	35.1	39.2	5,390	5,969	8.4
	NO.1	3+4	9.3	20.0	36.0	34.7	5,101	5,625	5.7
JT-16	NO.2	1	9.9	18.2	35.7	36.2	4,910	5,450	7.2
	NO.1	2+3	10.3	16.6	36.5	36.6	5,248	5,856	7.3
JT-19	NO.2	1	8.5	29.6	29.9	32.0	4,090	4,470	8.9
	NO.1	2	8.8	20.7	35.0	35.5	4,760	5,219	10.3
JT-21	NO.3	1	9.5	28.0	31.7	30.8	4,310	4,762	6.7
	NO.2	2	9.9	17.2	35.2	47.6	5,150	5,716	7.4
	NO.1	3	9.0	25.1	32.0	33.9	4,180	4,593	7.6
JT-23	NO.2	1+P1+2+P2+3	7.5	28.0	34.7	29.8	4,292	4,662	8.1
	NO.1	4	9.3	13.8	36.5	40.4	5,290	5,832	7.9

Table 11-3

ANALYTICAL DATA OF COAL FOR RESERVE CALUCULATION (3)

D.H.NO.	SEAM NO.	SAMPLE NO.	PROXIMATE ANALYSIS MOIST. (%)	ASH (%)	VOL. (%)	F.C. (%)	CAL. V. (Kcal/Kg)	CAL. V. (D. B.) (Kcal/Kg)	TOTAL SUL. (%)
JT-24	NO.1	3	8.5	14.7	38.4	38.4	5,420	5,923	8.0
JT-25	NO.3	2+3	9.5	10.7	38.8	41.0	5,588	6,174	6.6
	NO.2	4	8.4	20.6	34.7	36.3	4,900	5,349	6.4
	NO.1	6	6.6	29.8	32.6	31.0	4,360	4,668	8.2
JT-26	NO.3	1+P1+2	8.8	29.2	36.7	25.3	4,124	4,608	4.5
	NO.2	3	11.0	21.1	33.6	34.4	4,530	5,090	8.8
	NO.1	4	11.4	19.1	34.2	35.3	4,900	5,530	6.9
JT-27	NO.3	1	12.9	11.0	35.7	40.4	5,310	6,096	5.8
	NO.1	2	12.2	17.6	34.8	35.4	4,820	5,490	6.6
JT-28	NO.3	1	7.2	22.0	35.5	35.3	4,390	4,731	7.7
JT-29	NO.3	1	7.7	8.2	40.8	43.3	5,800	6,284	3.9
	NO.2	2	7.3	22.0	41.1	29.6	4,680	5,049	7.3
	NO.1	3+4+5	6.1	32.0	33.4	28.5	4,071	4,369	7.7
JT-30	NO.3	1	9.5	11.9	38.0	40.6	5,520	6,099	4.1
	NO.2 & 1	2+3+4+5+6	6.8	28.9	33.7	30.6	4,147	4,451	8.3
JT-31	NO.3	1	9.2	26.3	32.8	31.7	4,190	4,615	5.6
	NO.2 & 1	2+4	9.5	18.1	37.6	34.8	4,978	5,512	8.0
JT-34	NO.2	1+2+3	7.7	19.7	36.4	36.2	4,397	4,787	5.6
	NO.1	5+6+7+8	7.1	21.5	37.0	34.4	4,930	5,316	7.5

Table II-4 **ANALYTICAL DATA OF COAL FOR RESERVE CALUCULATION (4)**

D.H.NO.	SEAM NO.	SAMPLE NO.	PROXIMATE ANALYSIS MOIST. (%)	ASH (%)	VOL. (%)	F.C. (%)	CAL. V. (Kcal/Kg)	CAL. V. (D. B.) (Kcal/Kg)	TOTAL SUL. (%)
JT-35	NO.3	1	11.4	22.6	33.1	32.9	4,660	5,260	6.6
	NO.2 & 1	2+3+P1+4	10.6	22.1	36.6	30.7	4,725	5,309	6.0
JT-36	NO.3	1	9.0	23.0	31.0	37.0	4,490	4,934	6.5
	NO.2 & 1	2+3+4	9.6	31.6	31.2	27.6	3,859	4,328	7.5
JT-38	NO.3	1	12.8	12.5	36.3	38.4	5,400	6.193	5.7
	NO.2 & 1	2	11.1	20.9	35.0	33.0	4,820	5,422	8.1
JT-39	NO.3	1	10.2	17.7	35.9	36.2	5,130	5,713	6.2
	NO.2 &1	2+3	10.3	20.3	36.4	33.0	5,036	5,611	7.4
JT-40	NO.3	1	13.3	11.6	38.7	36.4	5,620	6,482	5.0
	NO.2 & 1	3+4	12.4	21.9	34.5	31.2	4,715	5,386	5.4
JT-41	NO.2 & 1	1+2	10.6	23.5	34.6	31.3	4,410	4,942	7.0
JT-42	NO.2 & 1	1+P1+2	10.3	26.1	35.6	28.0	4,170	4,658	7.7
JT-43	NO.3	1	12.2	11.0	35.8	41.0	5,660	6,446	3.8
	NO.2 & 1	2+3	9.9	22.9	35.0	32.2	4,662	5,217	4.7
JT-44	NO.3	1+2	11.4	14.3	37.5	36.8	5,265	5,942	5.9
	NO.2 & 1	3	9.5	28.3	33.2	29.0	4,320	4,773	5.6
JT-45	NO.3	1	16.0	14.5	34.7	34.8	5,300	6,310	5.1
	NO.2 & 1	2+3+4	11.5	21.9	33.9	32.7	4,676	5,283	6.8

Table 11-5

ANALYTICAL DATA OF COAL FOR RESERVE CALUCULATION (5)

D.H.NO.	SEAM NO.	SAMPLE NO.	PROXIMATE ANALYSIS MOIST. (%)	ASH (%)	VOL. (%)	F.C. (%)	CAL. V. (Kcal/Kg)	CAL. V. (D. B.) (Kcal/Kg)	TOTAL SUL. (%)
JT-46	NO.3	1	11.7	12.2	38.4	37.7	5,560	6,297	4.9
JT-47	NO.3	1	13.1	15.7	35.5	35.7	5,210	5,995	5.8
	NO.2 & 1	2+3+4	11.5	26.8	32.5	29.2	4,351	4,935	6.4
JT-48	NO.2 & 1	1+P1+2+3+4	10.5	24.7	33.4	31.4	4,426	4,967	6.5
JT-49	NO.3	1+2+3	10.9	22.0	35.9	31.2	4,668	5,278	4.6
	NO.2 & 1	4	11.7	15.2	36.6	36.5	5,230	5,923	7.1
JT-50	NO.3	1+2	11.0	19.8	35.7	33.5	4,963	5,583	5.7
	NO.2 & 1	3+4+5	9.6	23.5	34.2	32.7	4,852	5,369	7.9

Table 12

COAL ANALYTICAL DATA

D.H.NO.	SEAM NO.	SAMPLE NO.	TOTAL SUL.	NON-CONBUS. SULFUR	CONBUS. SULFUR	INORG. SUL.		ORG. SUL.	H.G.I.*	ULTIMATE ANAL. (daf)					ASH FUSION TEMP.			ANALYSIS ON ASH							
						SULPHATE	PYRITE			C	H	O	N	S	DEFORM.	HEMIS.	FLOW	SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO	Na_2O	K_2O	SO_3
			(%)	(%)	(%)	(%)	(%)	(%)		(%)	(%)	(%)	(%)	(%)	°C	°C	°C	%	%	%	%	%	%	%	%
JT-3	No.5	1	7.53	0.64	6.89	0.60	5.34	1.59	61	66.6	4.9	17.4	1.3	9.8	1270	1390	1450+	26.66	13.08	39.60	5.34	2.61	2.03	0.41	7.84
	No.2	3+4+5	5.42	0.48	4.94	1.00	2.29	2.13	80	63.7	4.8	22.3	1.1	8.1	1325	1390	1405	44.42	23.95	17.30	3.18	1.52	1.09	0.81	3.91
	No.1	6+7+8	9.78	0.45	9.33	1.61	6.32	1.84	88	59.2	5.2	17.9	1.0	16.7	1355	1375	1385	39.22	19.62	30.89	2.13	1.44	0.79	0.80	3.04
JT-9	No.5	1	8.93	1.89	7.04	2.28	4.68	1.97	67	65.4	5.5	16.6	1.3	11.2	1215	1300	1375	22.08	9.61	35.40	10.80	2.07	1.64	0.42	16.88
	No.3	2+3	7.58	1.55	6.03	2.09	2.32	2.17	69	64.0	4.9	21.2	1.1	8.8	1320	1350	1450+	13.20	12.73	36.95	12.54	3.96	0.99	0.59	17.47
	No.2	4	8.94	0.37	8.57	1.64	5.59	1.71	70	61.4	4.9	19.1	1.1	12.7	1255	1375	1395	32.04	15.44	40.24	3.10	1.96	0.84	0.60	3.76
	No.1	6+7+8	8.62	0.33	8.28	1.43	4.84	2.35	82	58.5	5.3	20.7	0.9	14.6	1395	1450+	1450+	36.92	26.87	26.09	2.75	0.93	0.44	0.48	2.29
JT-10	No.5	1	5.64	0.56	5.08	0.68	3.53	1.43	68	67.1	4.9	19.0	1.3	7.7	1235	1270	1340	41.60	15.40	24.82	4.18	3.06	2.33	0.64	5.47
	No.3	2	6.85	0.49	6.36	0.87	3.31	2.66	59	67.7	5.2	17.6	1.2	8.3	1290	1450+	1450+	17.96	13.34	44.94	6.02	4.61	2.38	0.42	8.42
	No.2	3	4.25	0.51	3.74	0.68	2.34	1.22	63	70.8	5.5	17.6	1.4	4.7	1230	1310	1380	26.12	14.28	32.59	5.52	3.62	3.17	0.52	10.39
	No.1	4+5	8.72	0.35	8.37	0.70	4.79	3.23	67	64.5	5.2	15.7	1.1	12.5	1355	1380	1405	34.18	22.38	31.92	3.12	1.94	1.14	0.86	3.43
JT-15	No.3	1	6.71	0.10	6.62	0.76	4.24	1.72	66	68.2	4.8	17.6	1.3	8.1	1380	1450+	1450+	8.00	11.10	70.76	1.29	0.99	1.10	0.85	2.64
	No.2	2	8.36	0.36	8.00	0.89	5.96	1.51	63	66.4	4.8	16.7	1.3	10.8	1370	1450+	1450+	15.38	9.49	59.18	4.54	1.55	0.97	0.33	5.63
	No.1	3+4	5.84	0.41	5.42	0.47	3.08	2.29	60	67.9	5.5	17.8	1.2	7.6	1250	1380	1385	36.90	22.03	27.74	3.86	1.42	0.76	0.63	5.20
JT-26	No.3	1+2	5.35	0.63	4.18	0.57	2.71	2.07	65	69.8	5.2	17.5	1.3	6.2	1325	1450+	1450+	19.44	14.50	37.80	8.42	2.94	1.30	0.38	12.18
	No.2	3	8.80	0.39	8.41	1.42	4.88	2.50	74	63.0	4.8	18.7	1.1	12.4	1325	1410	1450+	25.84	18.05	42.69	4.26	1.63	0.84	0.55	4.60
	No.1	4	6.87	0.70	6.17	0.88	1.26	4.73	67	66.8	5.0	18.1	1.2	8.9	1285	1370	1395	33.84	13.81	32.70	5.28	2.27	0.88	0.56	9.20
JT-29	No.3	1	3.91	0.20	3.71	0.69	2.44	0.77	67	70.4	5.2	18.6	1.4	4.4	1350	1450+	1450+	22.52	20.87	42.59	4.24	1.44	1.10	1.99	2.93
	No.2	2	7.28	0.23	7.06	0.54	4.58	2.17	63	65.5	4.8	18.5	1.2	10.0	1358	1380	1390	38.20	19.75	32.98	2.49	1.57	0.67	0.71	2.57
	No.1	3+4+5+6+7	9.02	0.28	8.74	0.98	5.41	2.63	81	60.3	5.6	18.3	1.0	14.8	1390	1450+	1450+	37.92	27.15	26.03	1.92	0.96	0.42	0.64	1.93
JT-31	No.3	1	5.58	0.41	5.17	0.64	2.83	2.11	73	65.6	5.5	19.7	1.2	8.0	1360	1450+	1450+	38.68	28.80	19.07	3.23	1.40	0.55	0.49	3.90
	No.2&1	2+4	7.80	0.49	7.39	0.89	3.39	3.60	67	66.3	5.3	17.1	1.1	10.2	1285	1365	1400	28.58	20.56	32.75	5.36	2.00	0.93	0.53	6.28
JT-35	No.3	1	6.58	0.40	6.18	0.57	4.03	1.99	71	67.2	5.3	16.9	1.2	9.4	1345	1385	1395	35.14	23.48	28.11	3.75	2.11	0.76	0.62	4.37
	No.2&1	2	5.59	0.54	5.05	0.58	1.97	3.05	65	69.3	5.7	17.0	1.1	6.9	1330	1360	1400	31.68	25.40	23.00	6.20	1.96	1.14	0.34	9.08
		3+4	5.84	0.35	5.49	0.63	1.75	3.45	68	68.0	5.4	17.5	1.1	8.0	1425	1450+	1450+	39.36	28.27	18.40	3.63	1.73	0.84	0.94	3.90
JT-39	No.3	1	6.15	0.61	5.55	0.52	3.57	2.07	69	68.8	5.3	17.0	1.2	7.7	1350	1450+	1450+	30.10	18.60	32.05	5.51	2.50	1.43	0.44	8.56
	No.2&1	2	7.64	0.50	7.13	0.73	3.69	3.21	71	66.0	4.9	17.4	1.2	10.5	1360	1380	1390	31.24	23.19	30.02	4.88	2.68	0.82	0.42	5.79
		3	7.21	0.57	6.64	0.63	2.85	3.73	64	64.5	5.8	19.2	1.1	9.4	1315	1350	1400	29.20	21.93	27.39	6.30	2.85	0.84	0.81	7.28
		4	6.56	0.63	5.94	0.58	2.89	3.10	73	69.9	4.5	15.8	1.2	8.6	1275	1310	1350	34.96	19.32	23.91	7.14	2.18	0.98	0.78	7.35
JT-45	No.3	1	5.10	0.44	4.66	0.52	2.79	1.79	68	70.6	5.4	16.0	1.3	6.7	1285	1345	1375	29.16	18.40	30.93	5.69	2.72	2.06	0.50	7.63
	No.2&1	2+3	6.03	0.51	5.53	0.51	3.15	2.37	76	68.0	5.1	17.4	1.2	8.3	1290	1365	1385	35.88	23.66	24.75	4.53	2.65	1.16	0.49	5.80
		4	7.62	0.58	7.04	0.87	4.16	2.59	72	68.3	5.4	14.4	1.2	10.7	1285	1345	1380	37.00	18.97	29.18	4.03	2.05	1.10	0.64	6.41
		5	8.50	0.41	8.08	1.41	3.65	3.44	97	58.9	5.4	18.9	1.1	15.7	1355	1390	1400	43.04	23.40	23.08	2.06	1.32	0.47	1.25	2.64
JT-50	No.3	1	3.33	0.52	2.81	0.32	2.00	1.00	60	72.4	5.5	16.9	1.4	3.8	1280	1320	1405	30.78	23.46	22.45	6.10	5.00	1.73	0.38	9.39
	No.2&1	3+4	9.78	0.56	9.22	0.39	6.30	3.10	72	65.9	5.0	14.4	1.1	13.6	1250	1400	1450	25.56	18.42	41.16	3.68	2.93	0.87	0.48	6.10

* H.G.I : HARDGROBE GRINDABILITY INDEX

Specific gravity of coals with different ash contents is as under:—

Specific gravity	Ash
1.5	20%
1.7	35%
2.44	Thick partings in coal seams

The minimum specific gravity is 1.31

After reviewing the above data it has been observed that Lakhra coal in the investigated area is suitable for power generation though it has high ash content and is liable to spontaneous combustion. Iso-ash maps, iso-calorific value maps, and iso-sulphur content maps of important seams of the coal zones are shown in Fig. 11, 12, and 13. The No. 1 and No. 2 zones merge in the eastern part of the area. Therefore, separate iso-grade maps are drawn for these zones in the western part, whereas in eastern part the maps have been prepared only for the minable seams by taking their average thickness.

The coal seams with following characteristics have not been considered while calculating reserves:-- (a) Coal seams less than 0.5 metre thick (b) low ratio of coal thickness to thickness of zone (c) poor coal with calorific value less than 3,500 kcal/kg.

The iso-grade maps will have to be revised depending upon the results of mining feasibility study, which would decide the mining methods (open-cut and underground), and the working heights.

4-3-3 Washability

During the course of exploration it was noted that the quality of coal varies widely, and sulphur content was high. Keeping this in view 10 typical samples were taken from the drilling cores for studying the washability. On these samples float and sink tests were carriedout. Each sample because of small quantity was crushed under 10 mm size. The tests were carriedout with solution of 1.30, 1.35, 1.40, 1.50 and 1.60 specific gravity. The results of tests are shown in Table 13, sheets 1-11, and washability curves in Fig. 14, sheets 1-11. The Table 13-11, and Fig. 14-11 show composite data of all the 10 samples.

The raw coal ash content for the washability tests fluctuates widely from 9.41 to 33.69 percent. Therefore it is considered that high specific gravity and high ash content materials, such as plus 1.6 fraction in each raw coal sample vary widely from 7.3 to 50.6 percent. After reviewing the specific gravity of coal portion, it is clear in general that the intermediate fractions between 1.3 to 1.6 in specific gravity are more common in the raw coal. Accordingly it is difficult to beneficiate the raw coal with a medium having low specific gravity. Moreover, characteristics of this coal indicate that the ash contents in each fraction upto 1.5 specific gravity resemble each other. This also suggests that the ash reduction by low specific gravity medium will not be anticipated so much.

For example in case of the separation of raw coal by float and sink tests with medium of 1.6 and 1.8 specific gravity, the clean coal ash content, yield and the near gravity materials are shown in Table 14.

ISO-ASH MAP OF NO.1 SEAM (D.B.)

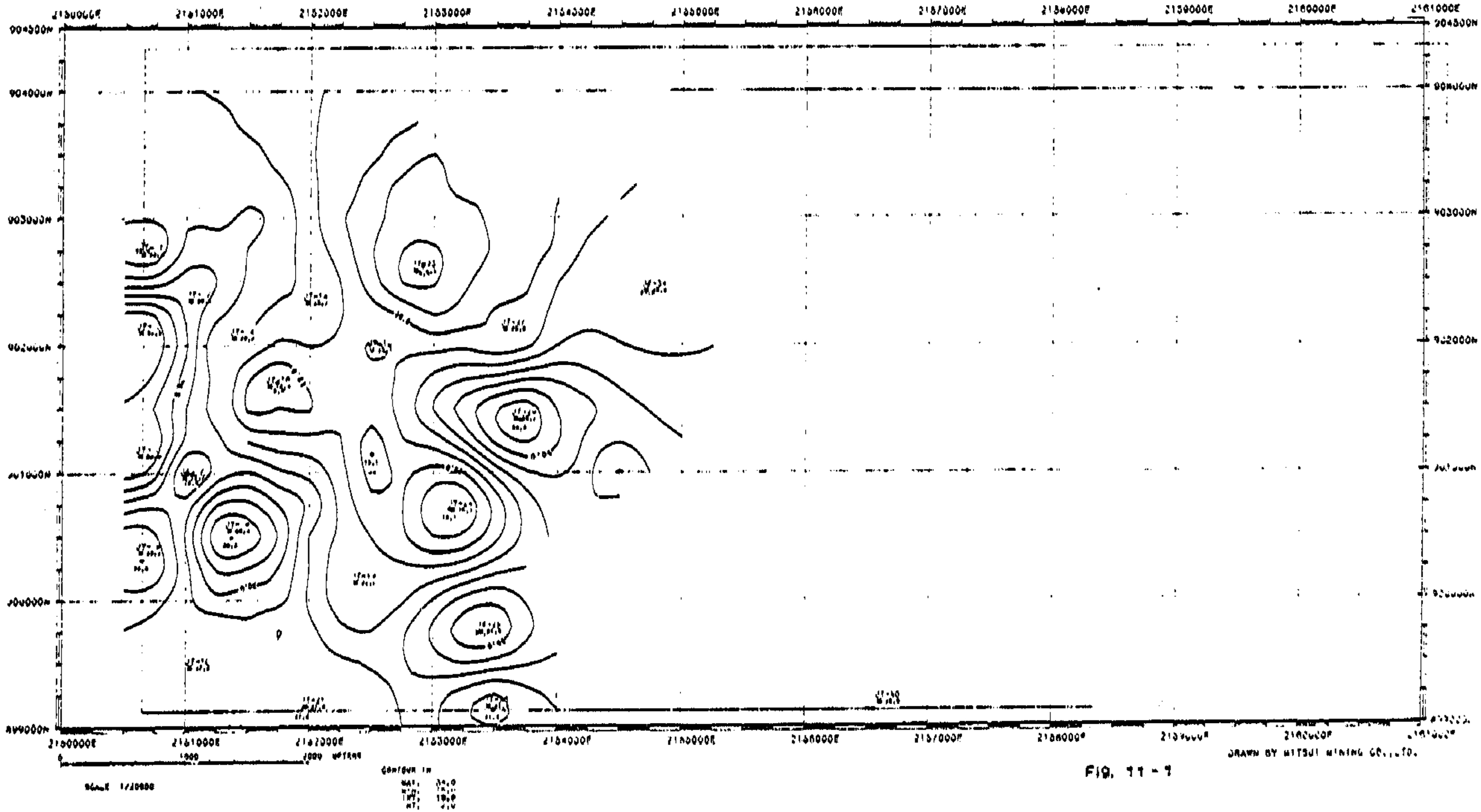


FIG. 11-7

ISO-ASH MAP OF NO.2 SEAM (D.B.)

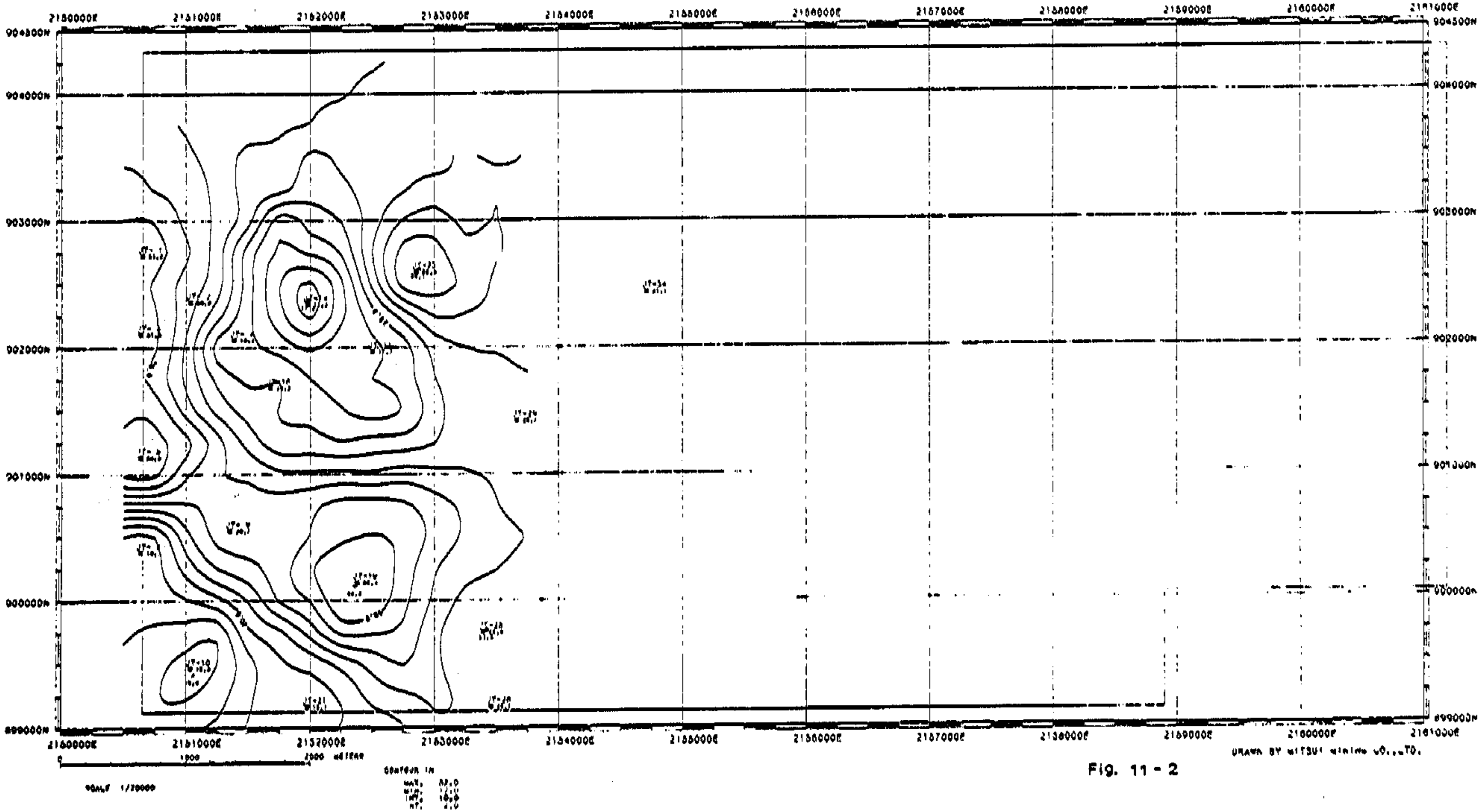


Fig. 11 - 2

ISO-ASH MAP OF NO.2&1 SEAM (D.B.)

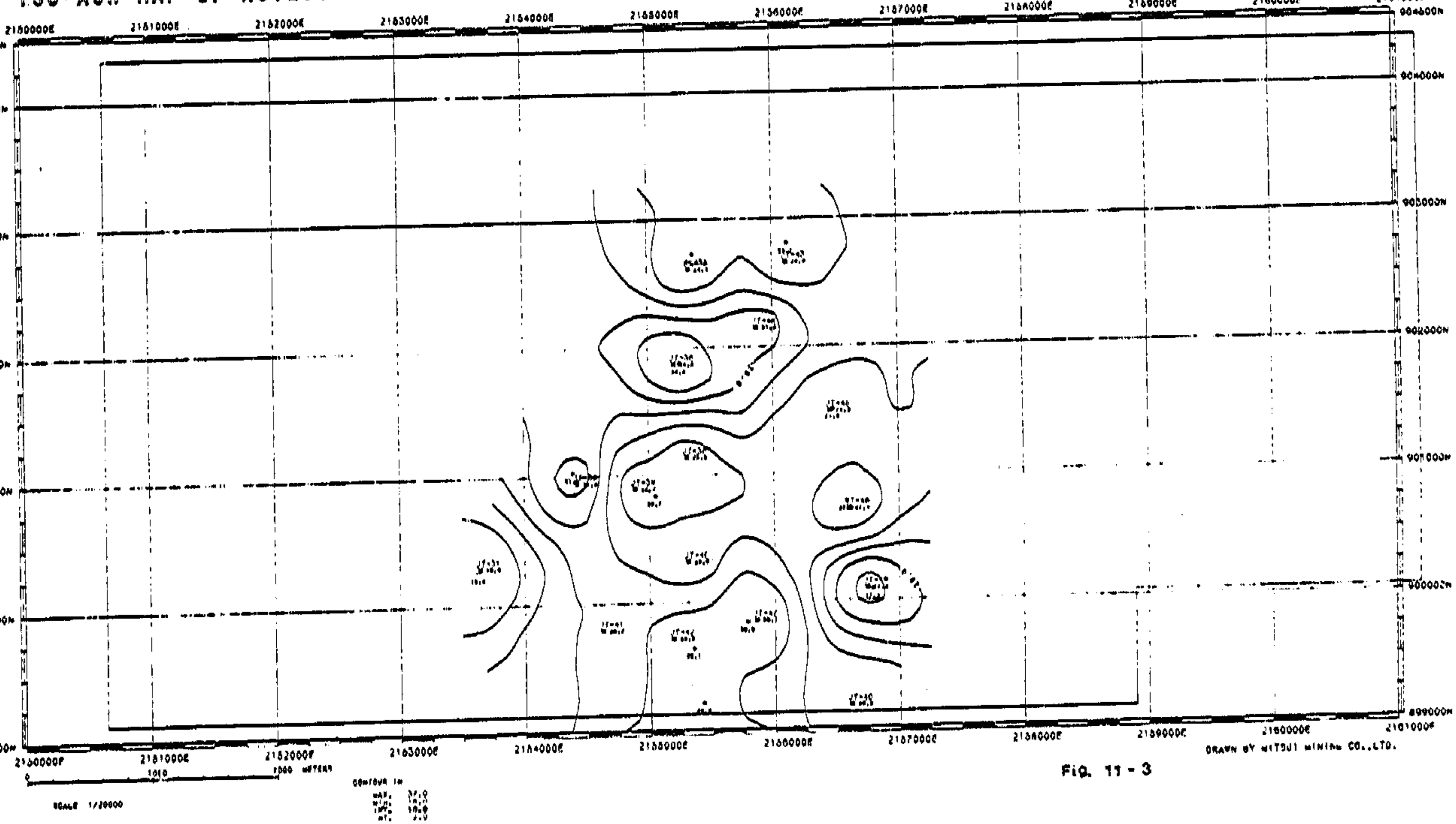
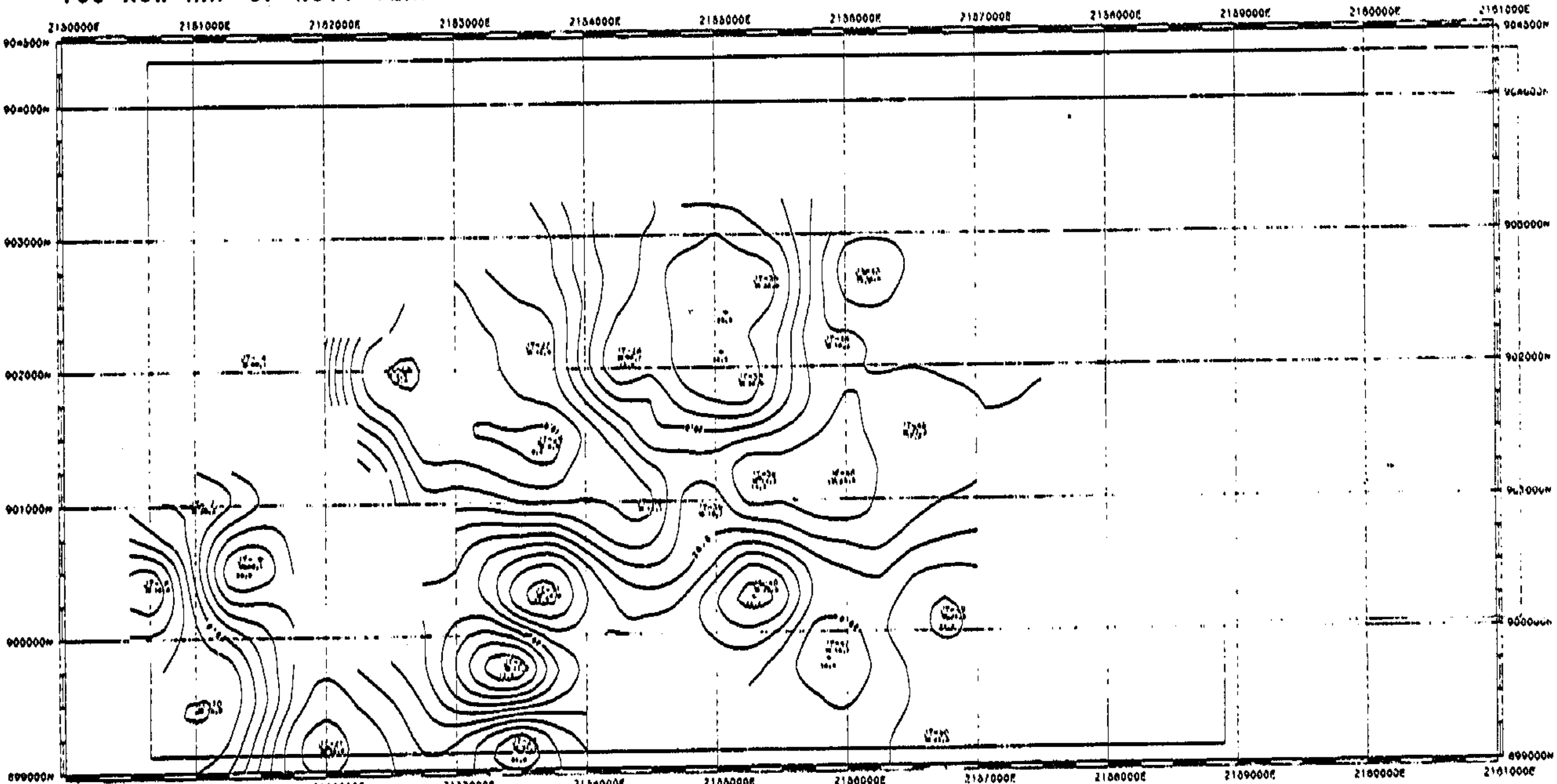


FIG. 11 - 3

ISO-ASH MAP OF NO.3 SEAM (D.B.)



SCALE 1/20000

CONTOUR IN

DRAWN BY MITSUI MINING CO., LTD.

Fig. II-4

ISO-ASH MAP OF NO.5 SEAM (D.B.)

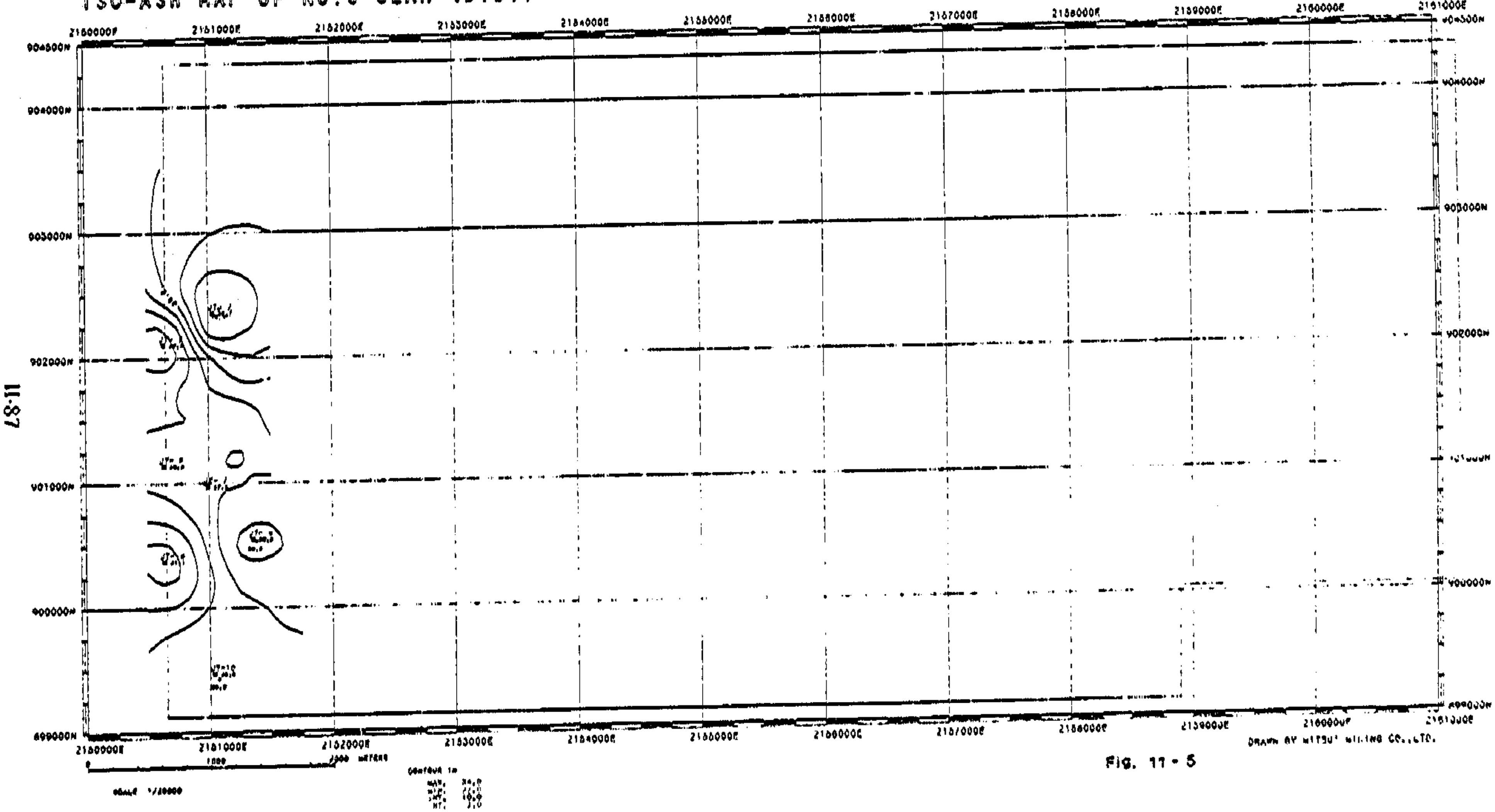


FIG. 11-5

ISO-CALORIFIC V. MAP OF NO.1 SEAM (D.B.)

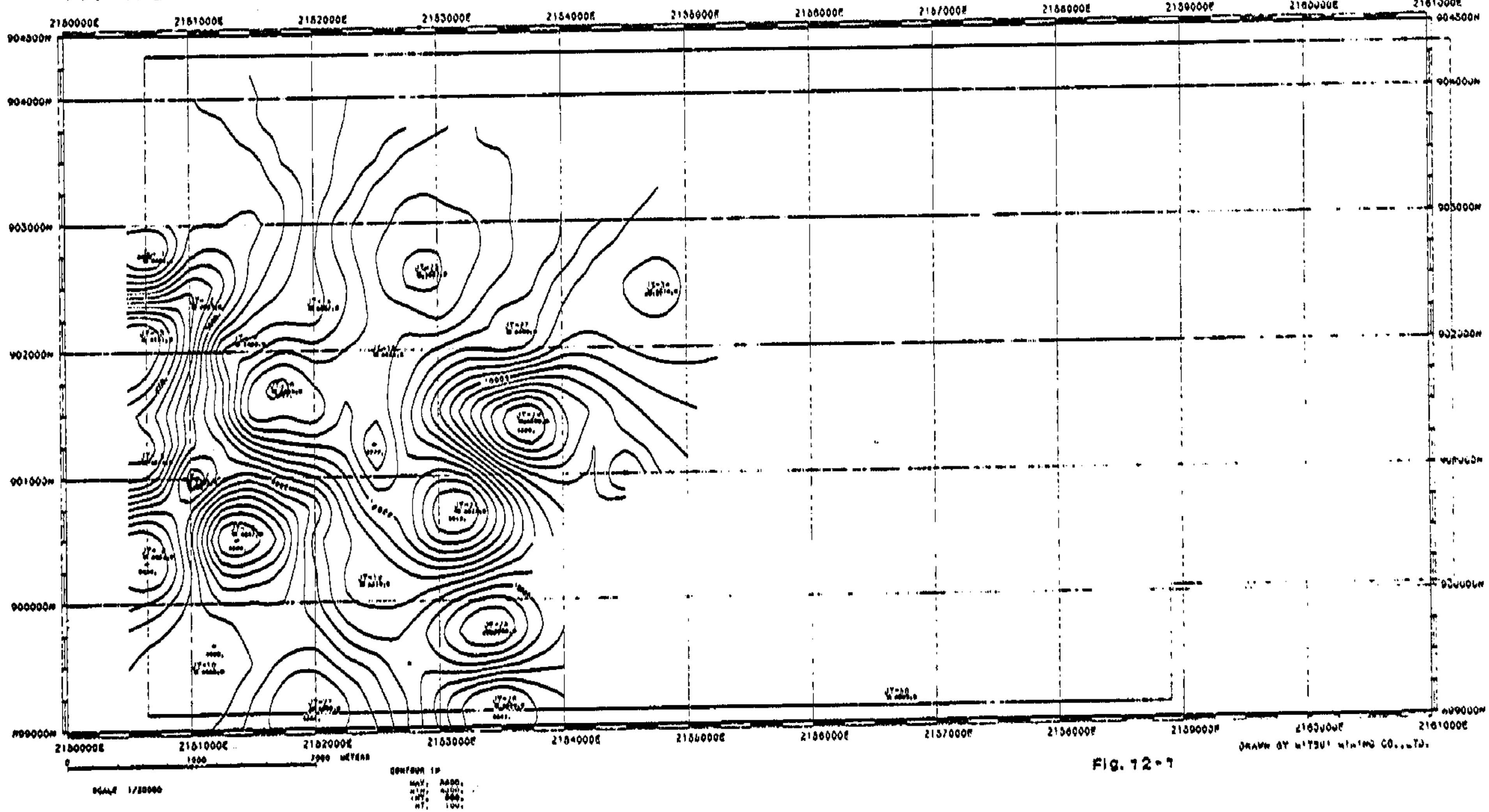


Fig. 12-7

ISO-CALORIFIC V. MAP OF NO.2 SEAM (D.B.)

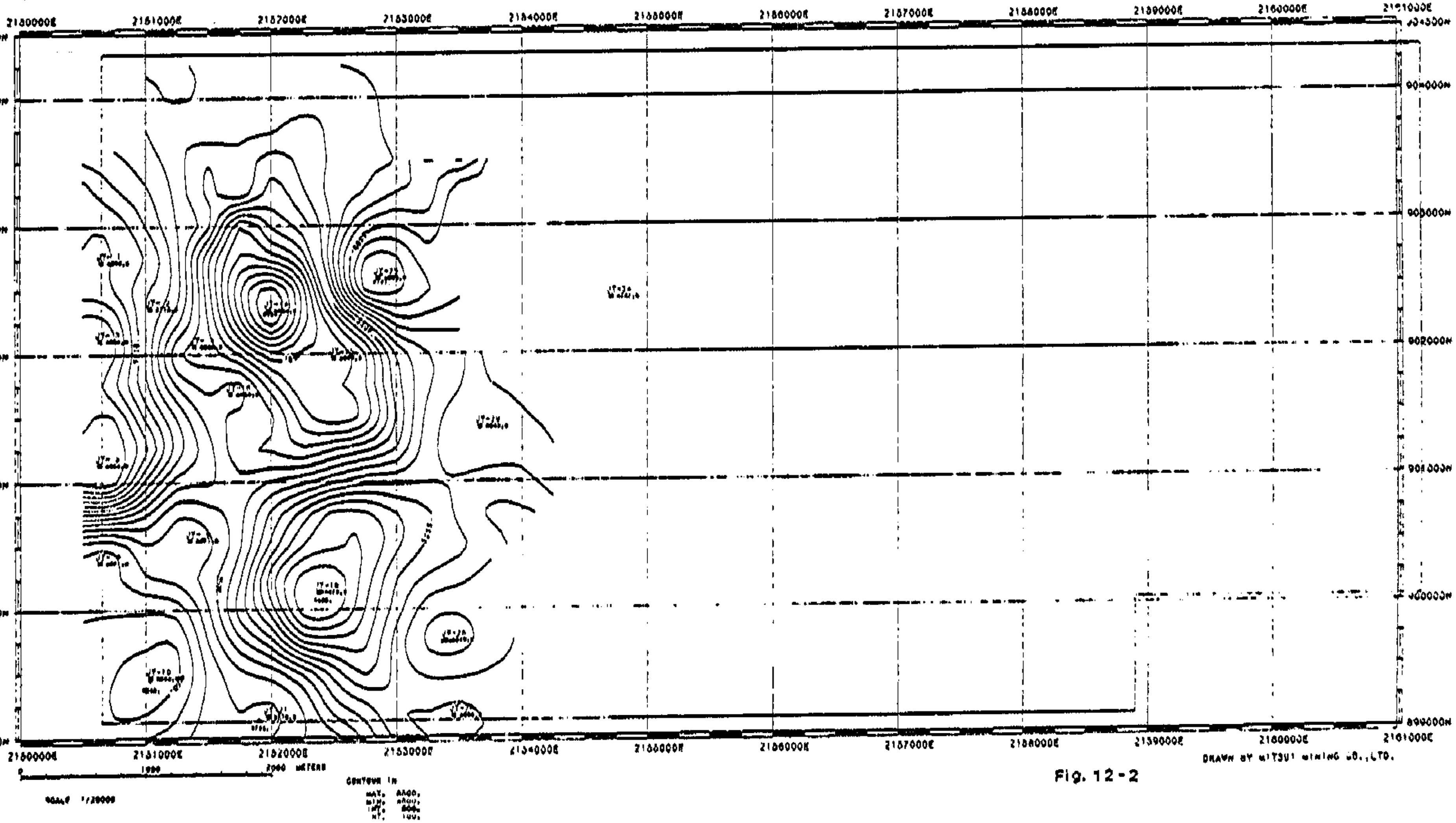


Fig. 12-2

ISO-CALORIFIC V. MAP OF NO.2&1SEAM(D.B.)

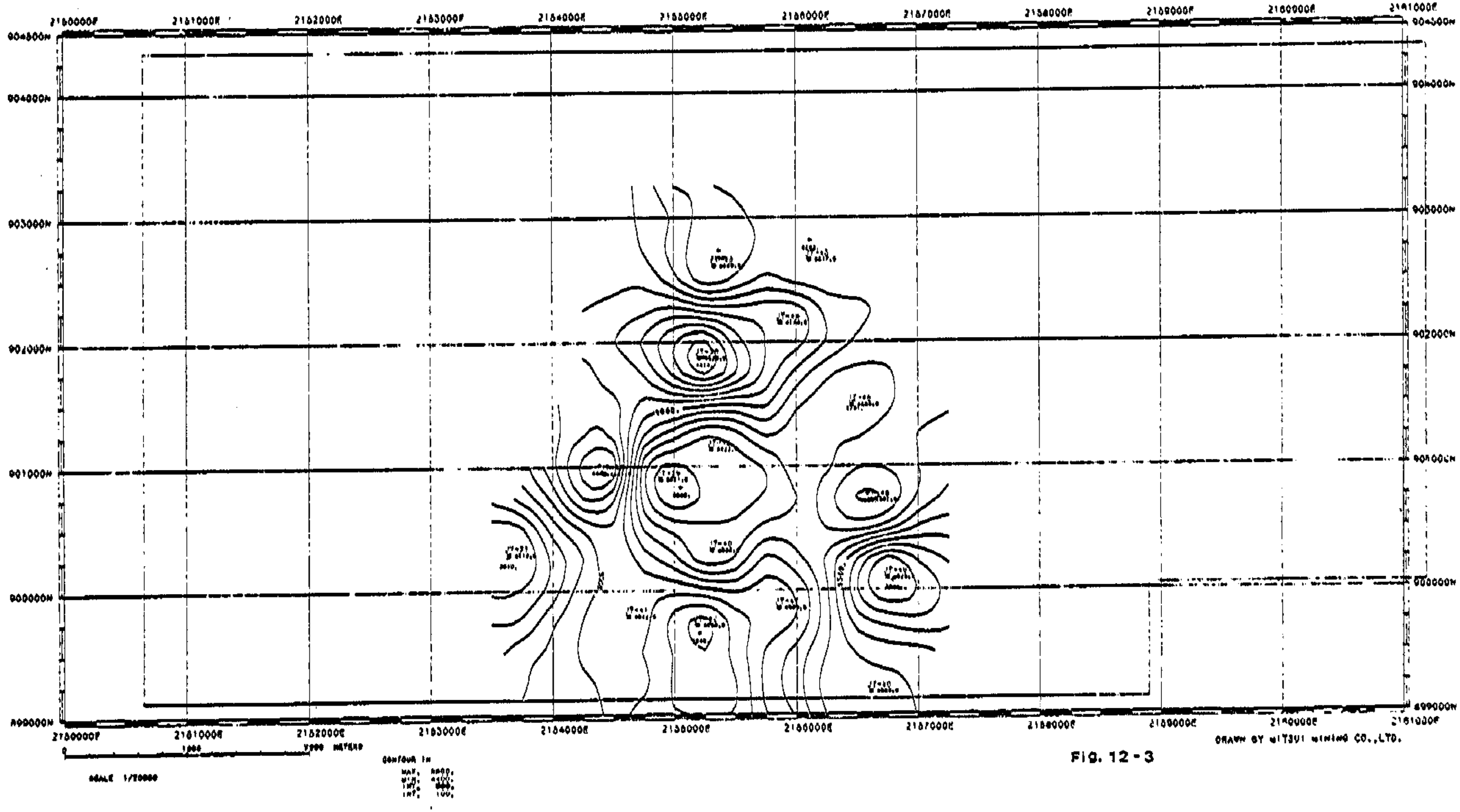


FIG. 12-3

ISO-CALORIFIC V. MAP OF NO.3 SEAM (D.B.)

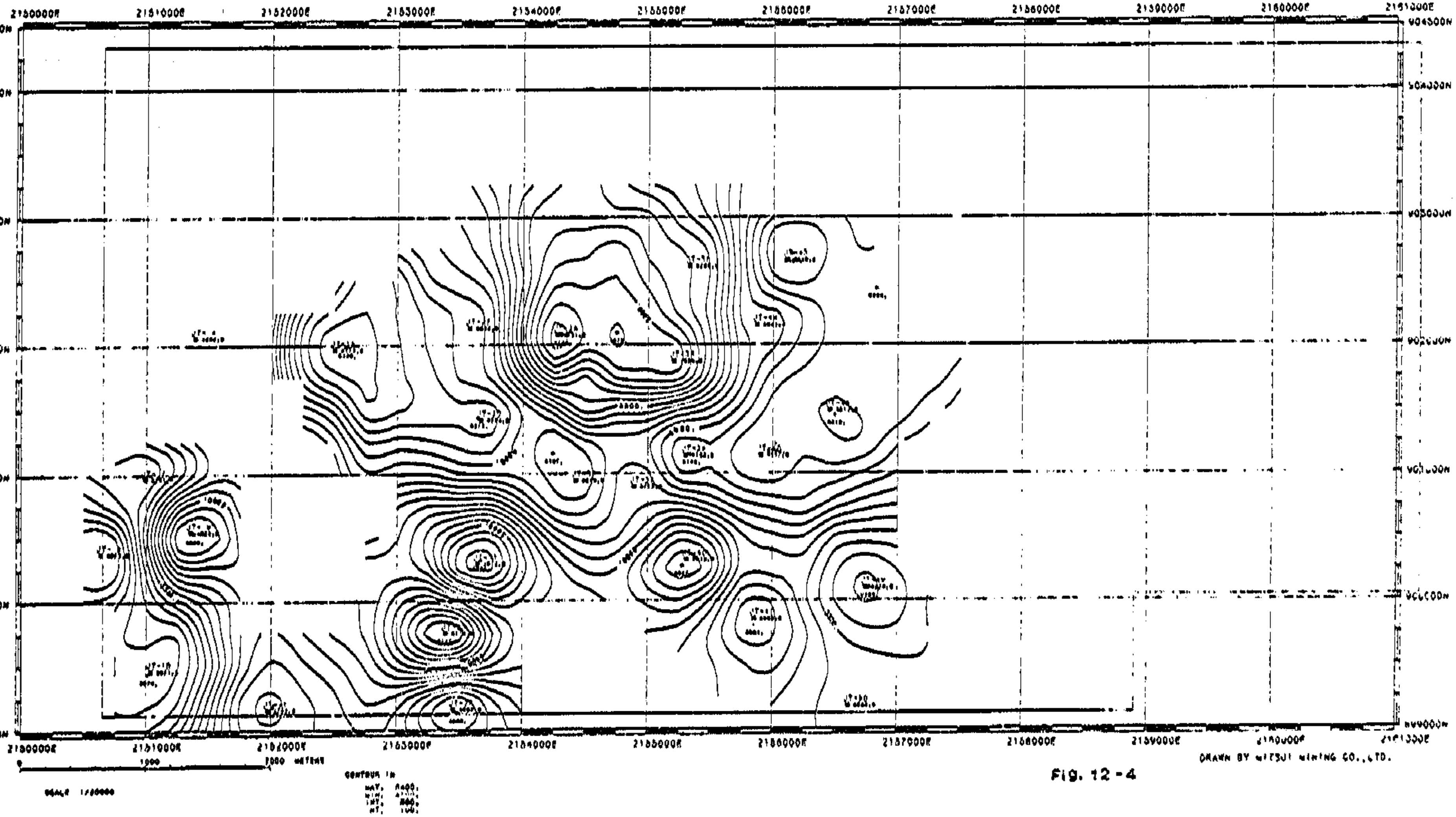


Fig. 12-4

ISO-CALORIFIC V. MAP OF NO.5 SEAM (D.B.)

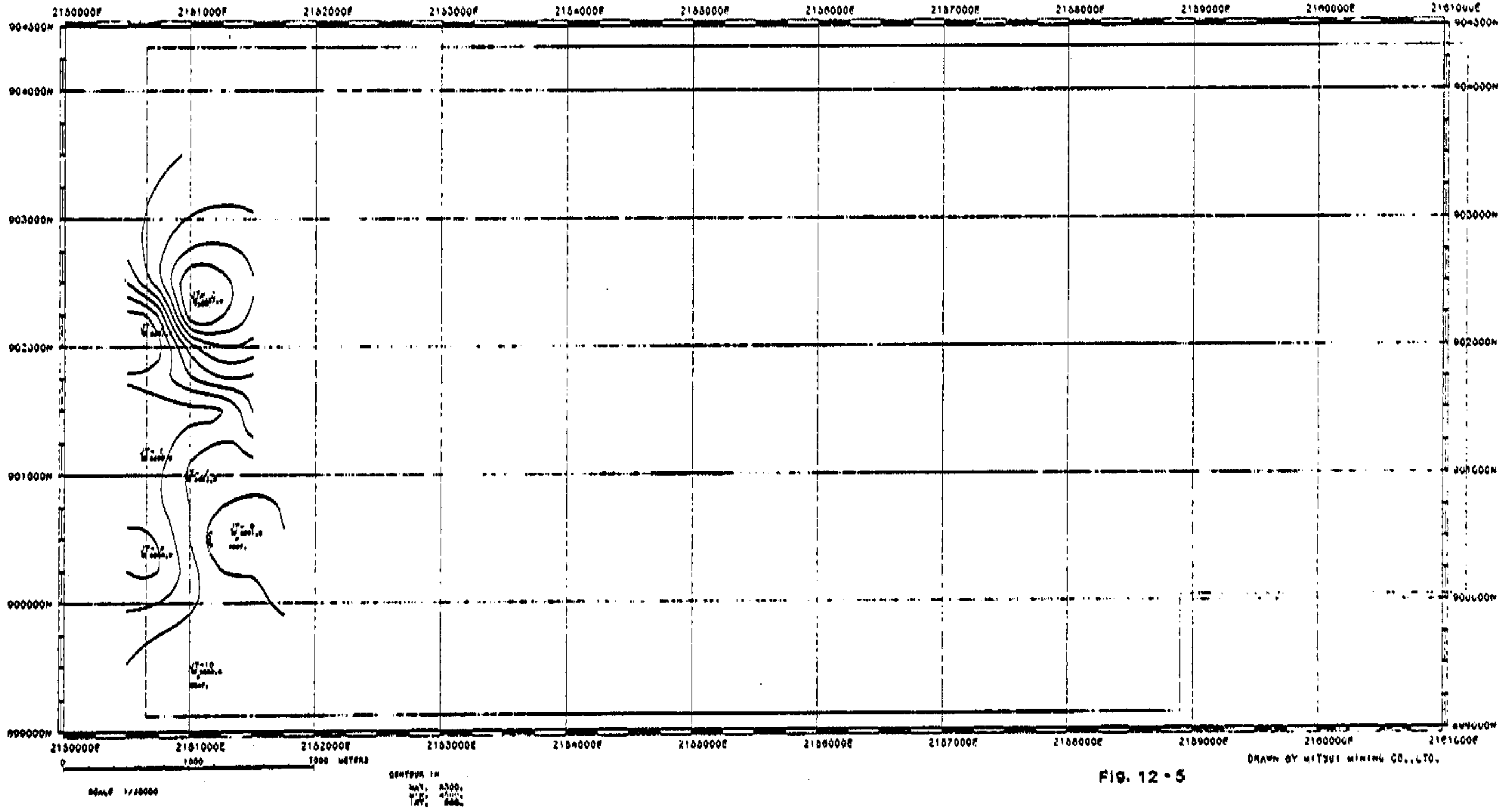


FIG. 12-5

ISO-TOTAL SULFUR MAP OF NO.1 SEAM (D.B.)

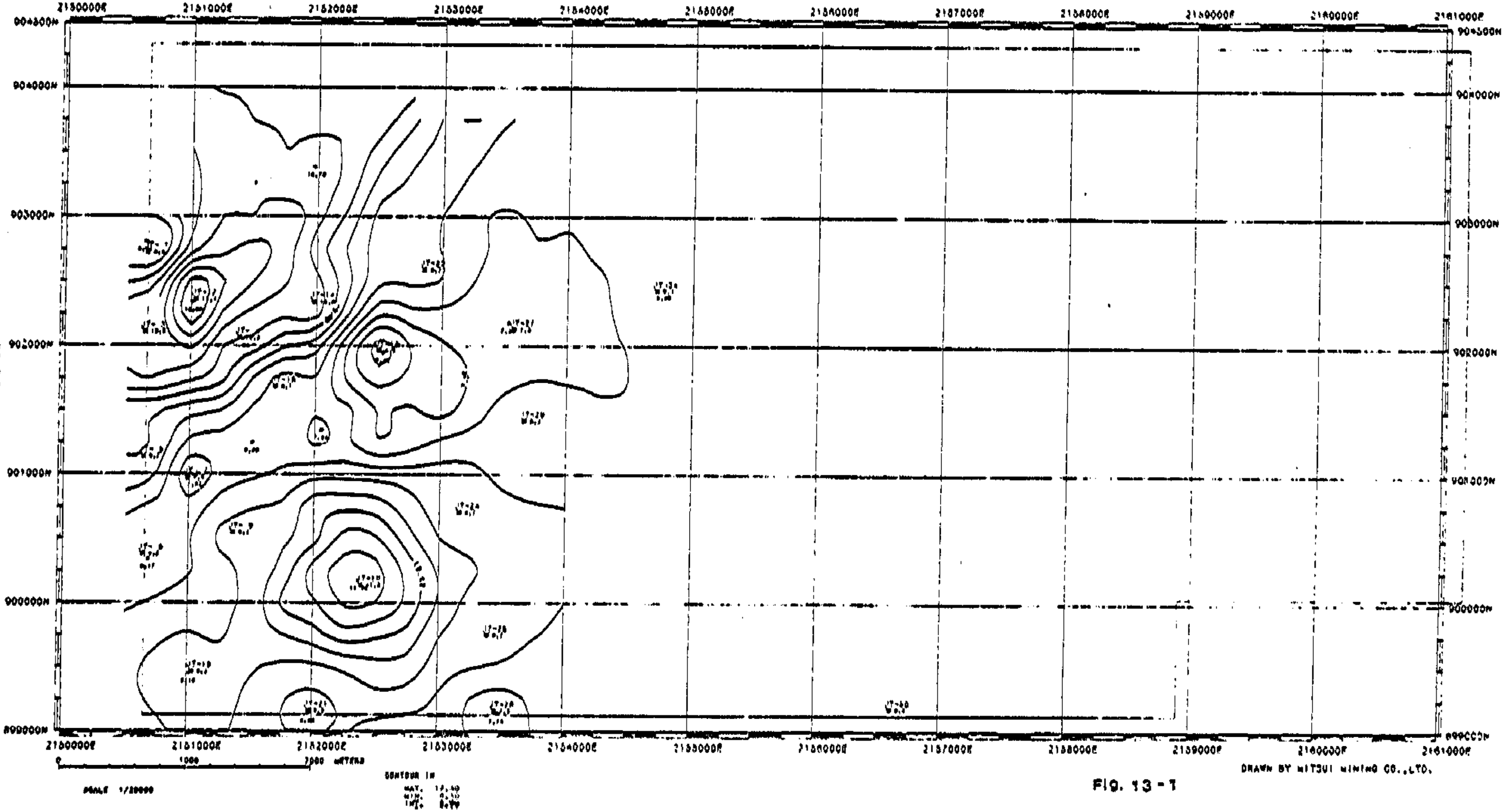


FIG. 13-1

ISO-TOTAL SULFUR MAP OF NO.2 SEAM (D.B.)

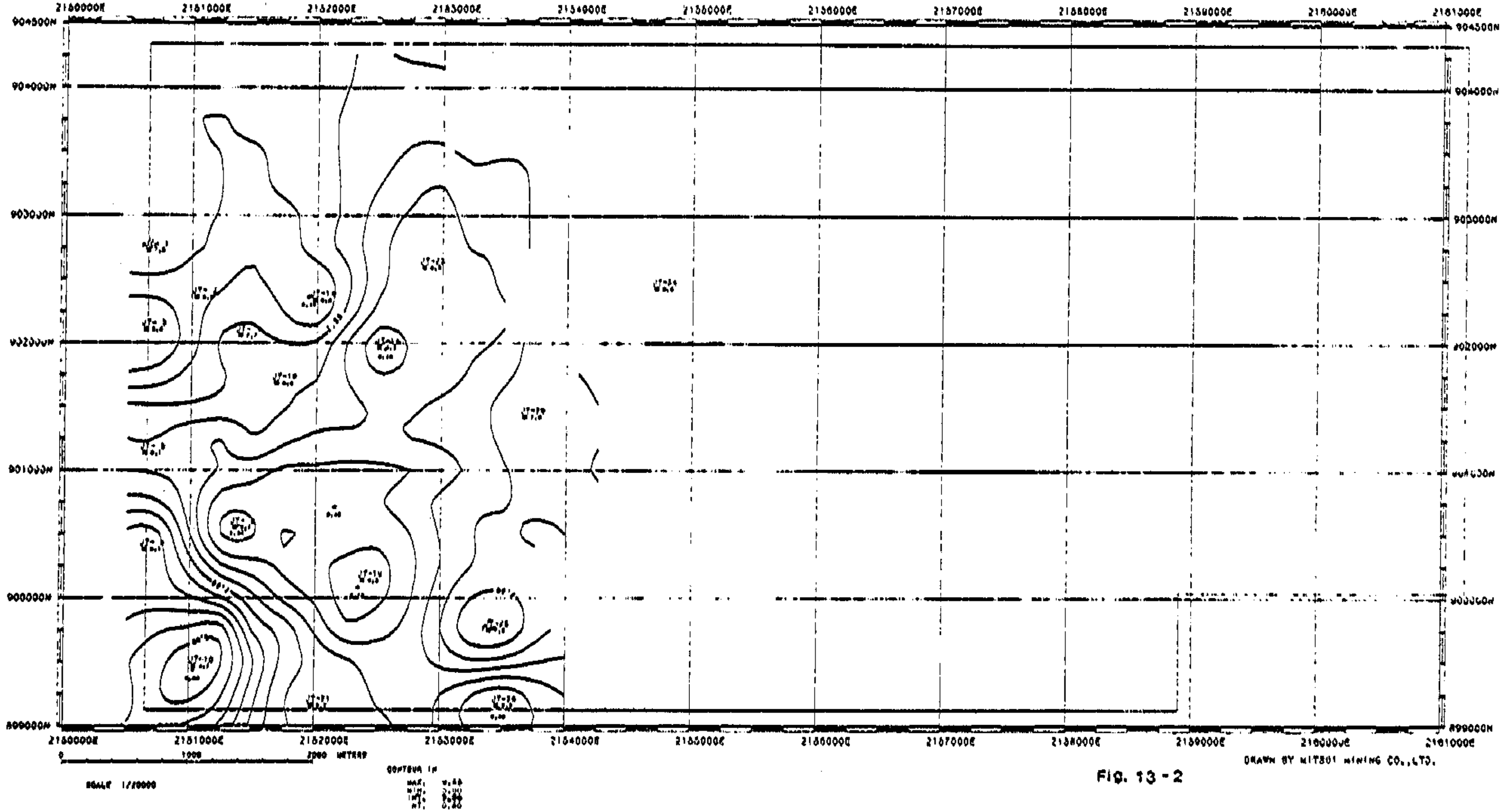


Fig. 13 - 2

ISO-TOTAL SULFUR MAP OF NO.2&1SEAM(D.B.)

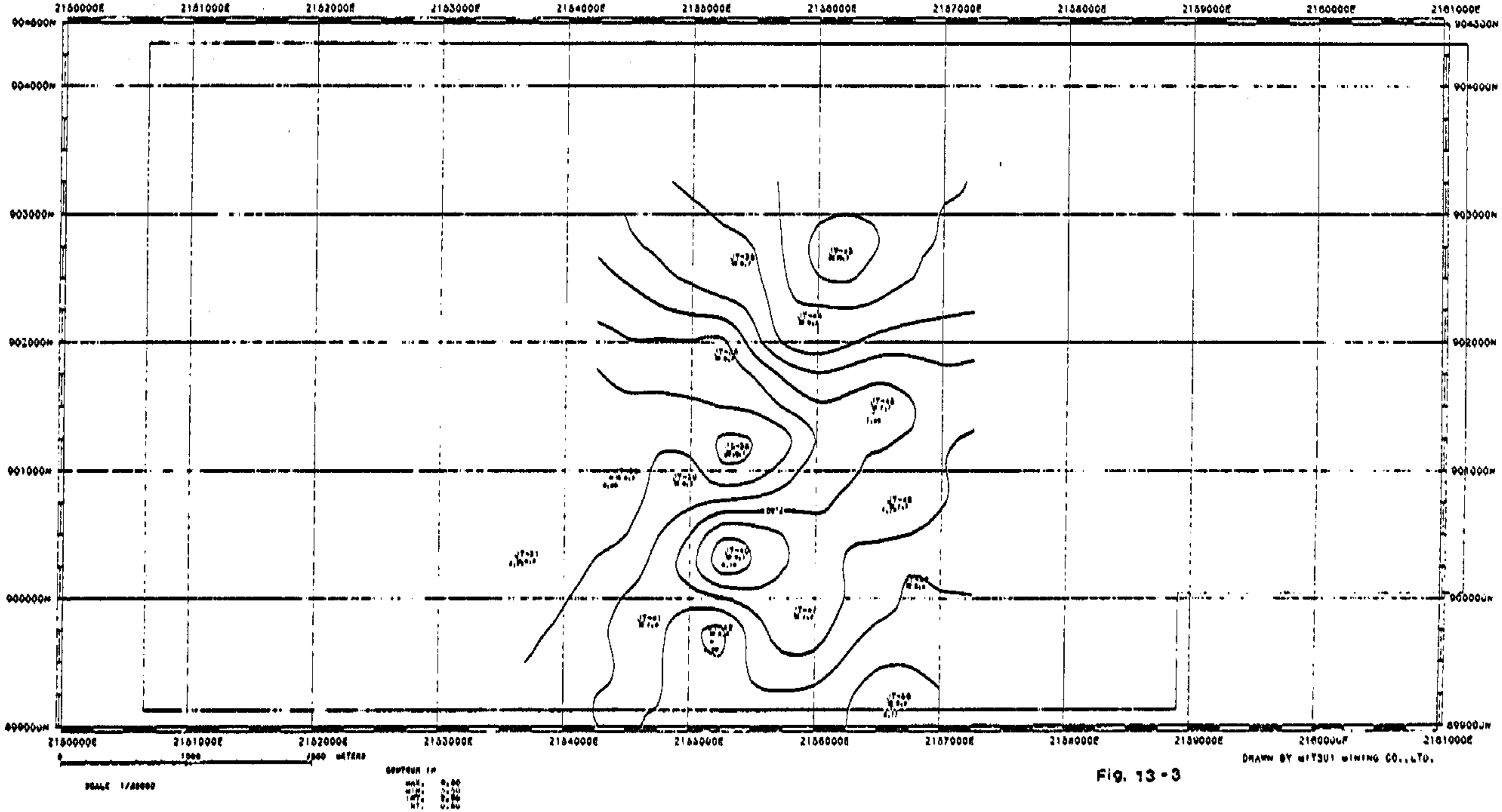


Fig. 13-3

ISO-TOTAL SULFUR MAP OF NO.3 SEAM (D.B.)

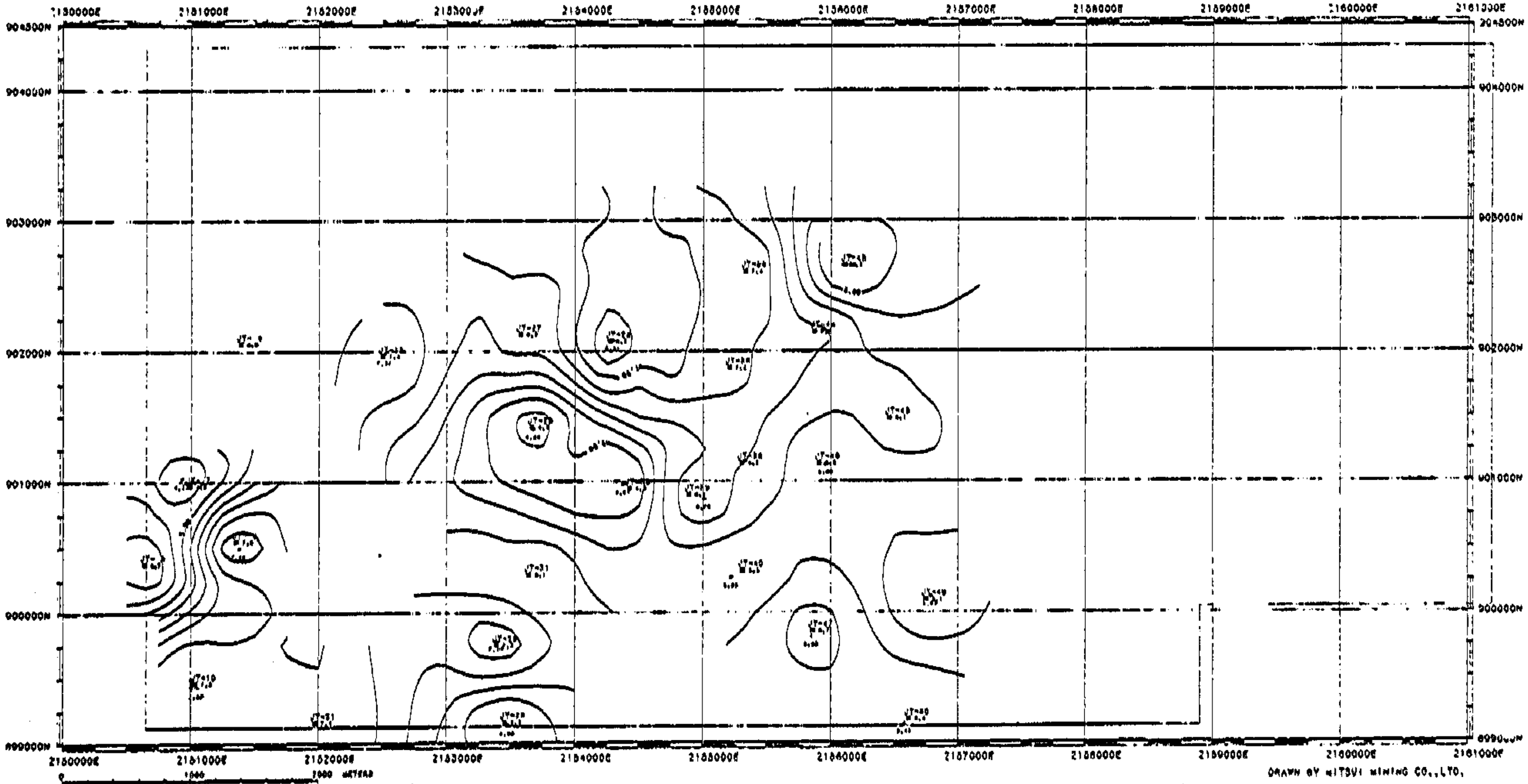


Fig. 13-4

ISO-TOTAL SULFUR MAP OF NO.5 SEAM (D.B.)

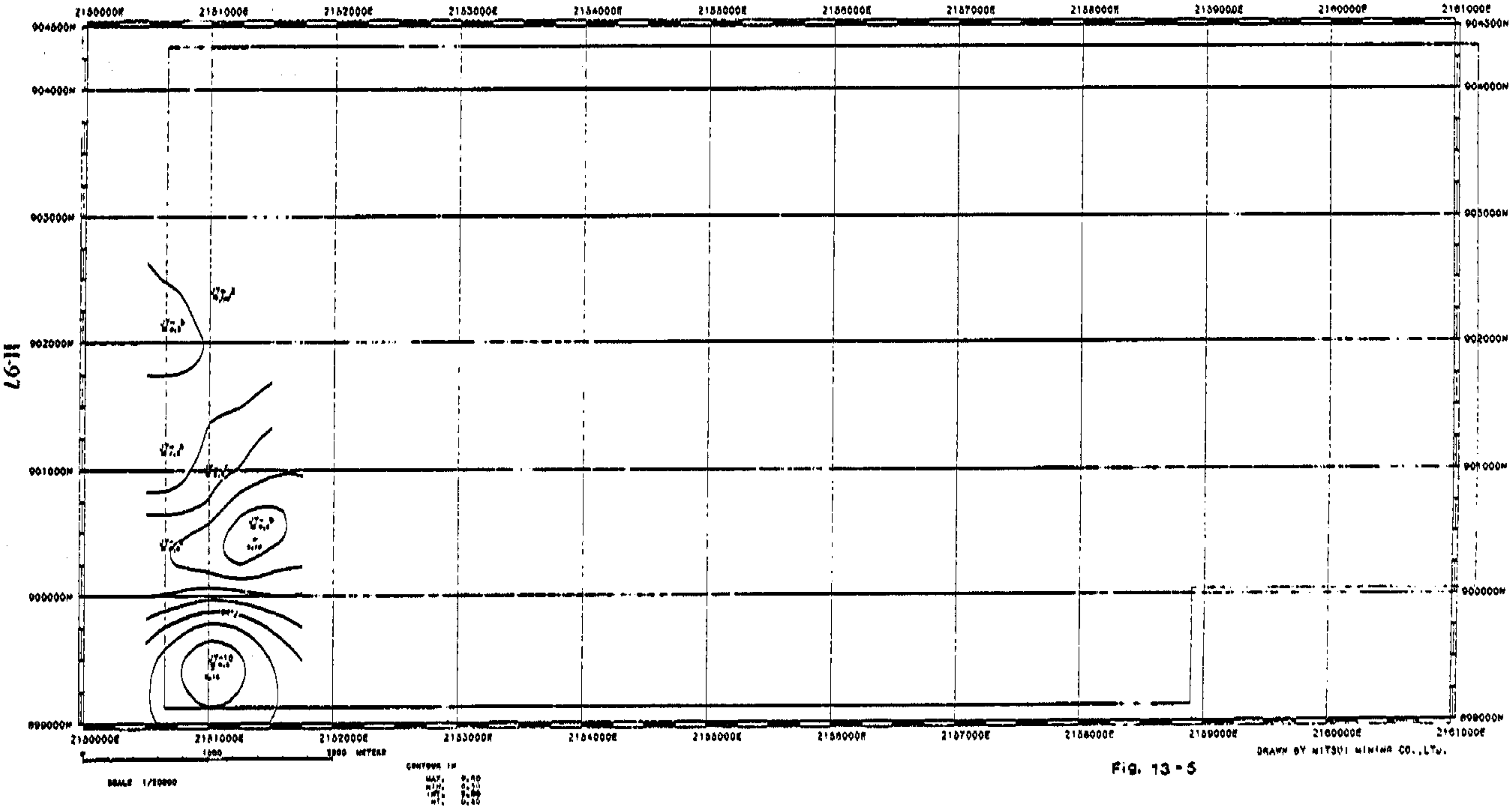


Fig. 13-5

Table 13-1

FLOAT AND SINK TEST

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT 8

SAMPLE NUMBER S-7

DATE: JULY 1979

SIZE: UNDER 10MM

Specific gravity	a		b	c	d	e	f	g	h	i	j
	Weight (%)	Ash (%)	$\sum W_{n-1} + \frac{W_n}{2}$	W·A	$\sum W \cdot A$	$\sum W$	$\frac{\sum W \cdot A}{\sum W}$	Total W·A $-\sum W \cdot A$	100 $-\sum W \cdot A$	$\frac{g}{h}$	±0.1 SG
- 1.30	11.70	5.80	5.85	67.86	67.86	11.70	5.80	3,301.30	88.30	37.39	
1.30 ~ 1.35	2.90	6.10	13.15	17.69	85.55	14.60	5.86	3,283.61	85.40	38.45	
1.35 ~ 1.40	12.50	6.30	20.85	78.75	164.30	27.10	6.06	3,204.86	72.90	43.96	29.60
1.40 ~ 1.50	14.20	12.10	34.20	171.82	336.12	41.30	8.14	3,033.04	58.70	51.67	22.30
1.50 ~ 1.60	8.10	19.00	45.35	153.90	490.02	49.40	9.92	2,879.14	50.60	56.90	
1.60 ~ +	50.60	56.90	74.70	2,879.14	3,369.16	100.00	33.69	.00	.00	.00	
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Fig. 14-1

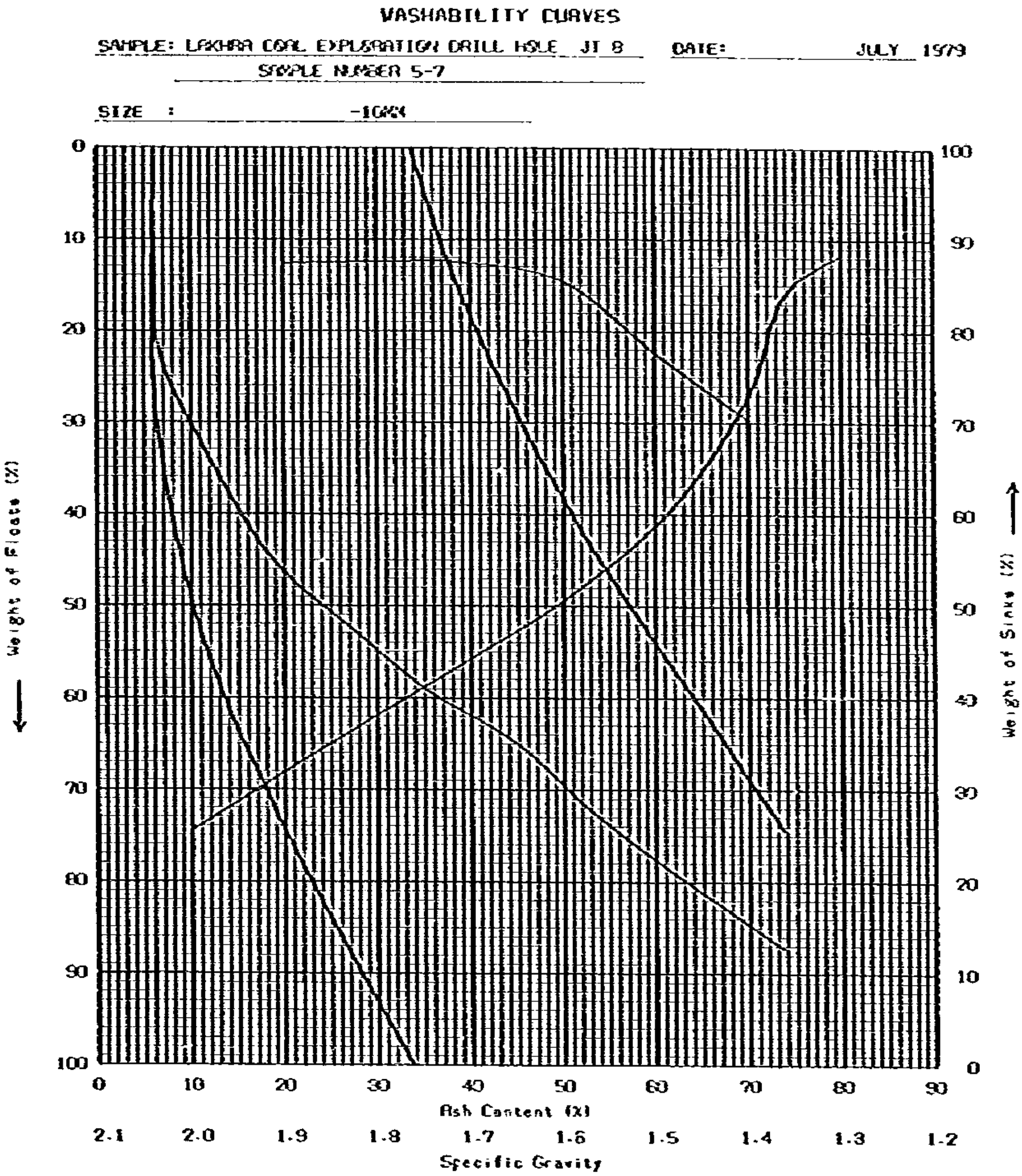
VASHABILITY CURVES
SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT 8
DATE: JULY 1979
SAMPLE NUMBER 5-7
SIZE : -10MM
0
10
20
30
40
50
60
70
80
90
100
Weight of Floats (%)
Weight of Sinks (%)
0
10
20
30
40
50
60
70
80
90
Ash Content (%)
2.1
2.0
1.9
1.8
1.7
1.6
1.5
1.4
1.3
1.2
Specific Gravity

Table 13-2

FLOAT AND SINK TEST

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT16

SAMPLE NUMBER 1

DATE: SEPTEMBER 1979

SIZE: UNDER 10MM

Specific gravity	a		b	c	d	e	f	g	h	i	j
	Weight (%)	Ash (%)	$\frac{\Sigma W_{n-1} + W_n}{2}$	W·A	ΣW·A	ΣW	$\frac{\Sigma W \cdot A}{\Sigma W}$	Total W·A −ΣW·A	100 −ΣW·A	$\frac{g}{h}$	±0.1 SG
- 1.30	7.00	5.20	3.50	36.40	36.40	7.00	5.20	1,973.62	93.00	21.22	
1.30 ~ 1.35	13.20	5.50	13.60	72.60	109.00	20.20	5.40	1,901.02	79.80	23.82	
1.35 ~ 1.40	28.10	6.30	34.25	177.03	286.03	48.30	5.92	1,723.99	51.70	33.35	55.90
1.40 ~ 1.50	14.60	10.50	55.60	153.30	439.33	62.90	6.98	1,570.69	37.10	42.34	21.10
1.50 ~ 1.60	6.50	18.50	66.15	120.25	559.58	69.40	8.06	1,450.44	30.60	47.40	
1.60 ~ +	30.60	47.40	84.70	1,450.44	2,010.02	100.00	20.10	.00	.00	.00	
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Fig. 14-2

WASHABILITY CURVES

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT 16 DATE: SEPTEMBER 1979

SAMPLE NUMBER 1

SIZE : -10MM

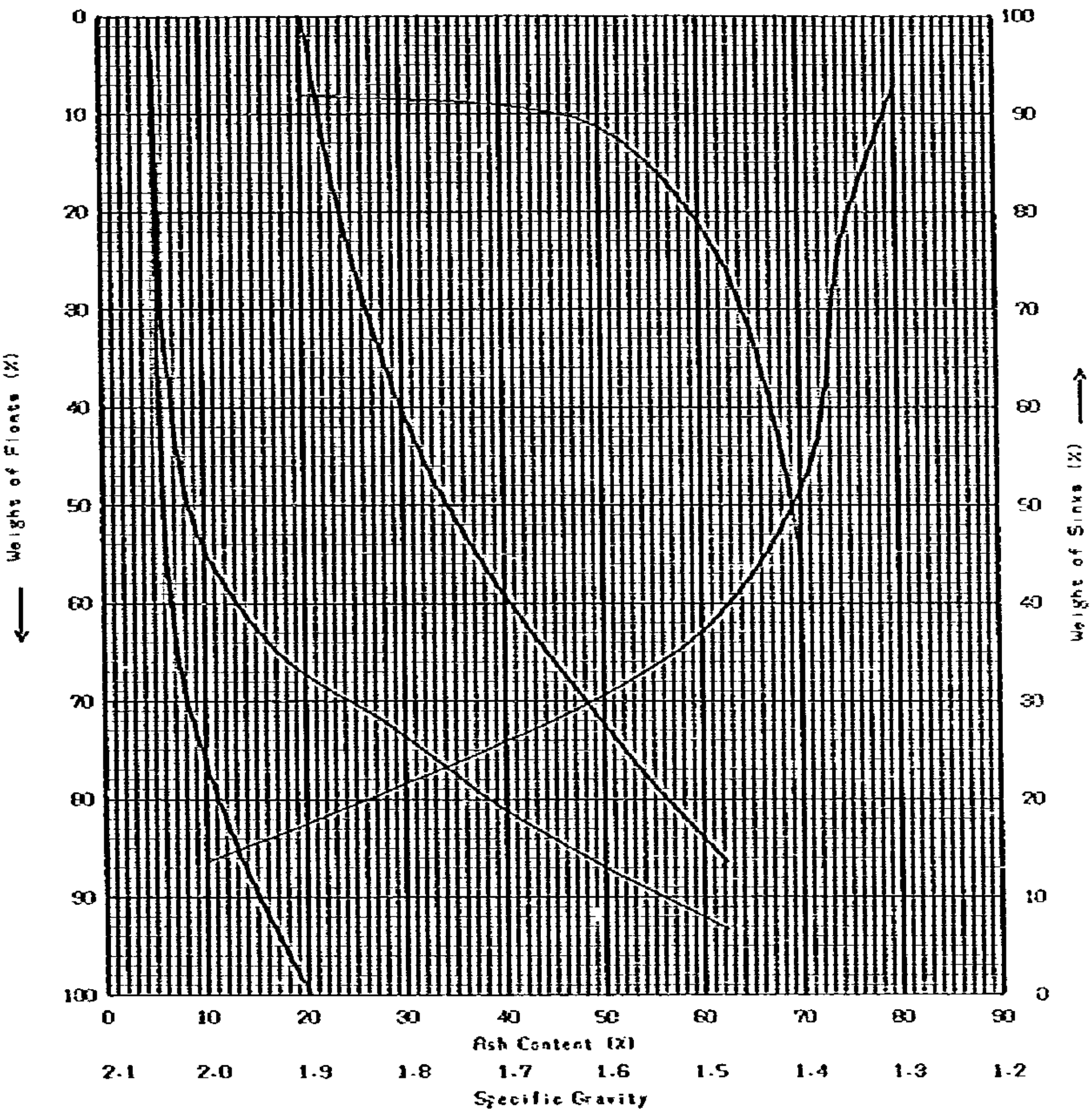

Table 13-3

FLOAT AND SINK TEST

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT16

SAMPLE NUMBER 2

DATE: SEPTEMBER 1979

SIZE: UNDER 10MM

Specific gravity	a		b	c	d	e	f	g	h	i	j
	Weight (%)	Ash (%)	$\Sigma W_{n-1} + \frac{W_n}{2}$	W·A	ΣW·A	ΣW	$\frac{\Sigma W \cdot A}{\Sigma W}$	Total W·A −ΣW·A	100 −ΣW·A	$\frac{g}{h}$	±0.1 SG
− 1.30	5.00	6.00	2.50	30.00	30.00	5.00	6.00	2,452.76	95.00	25.82	
1.30 ~ 1.35	6.50	6.50	8.25	42.25	72.25	11.50	6.28	2,410.51	88.50	27.24	
1.35 ~ 1.40	17.00	7.30	20.00	124.10	196.35	28.50	6.89	2,286.41	71.50	31.98	47.50
1.40 ~ 1.50	24.00	10.30	40.50	247.20	443.55	52.50	8.45	2,039.21	47.50	42.93	32.10
1.50 ~ 1.60	8.10	17.30	56.55	140.13	583.68	60.60	9.63	1,899.08	39.40	48.20	
1.60 ~ +	39.40	48.20	80.30	1,899.08	2,482.76	100.00	24.83	.00	.00	.00	
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Fig. 14-3

WASHABILITY CURVES

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT 16 DATE: SEPTEMBER 1979

SAMPLE NUMBER 2

SIZE : -10MM

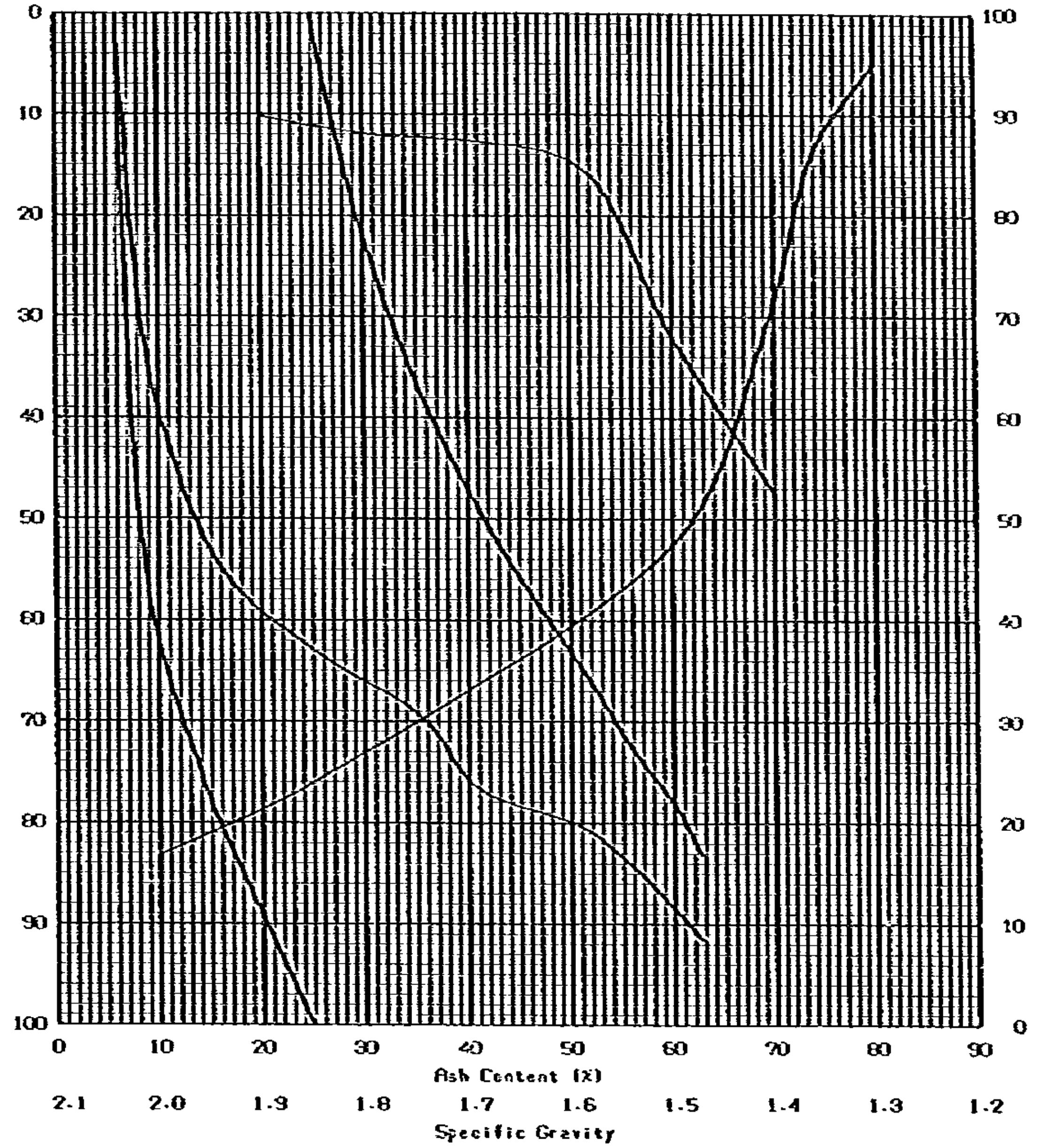

Table 13-4

FLOAT AND SINK TEST

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT16

SAMPLE NUMBER 3

DATE: SEPTEMBER 1979

SIZE: UNDER 10MM

Specific gravity	a		b	c	d	e	f	g	h	i	j
	Weight (%)	Ash (%)	$\Sigma W_{n-1} + \frac{W_n}{2}$	W·A	ΣW·A	ΣW	$\frac{\Sigma W \cdot A}{\Sigma W}$	Total W·A −ΣW·A	100 −ΣW·A	$\frac{g}{h}$	±0.1 SG
− 1.30	32.70	5.00	16.35	163.50	163.50	32.70	5.00	777.81	67.30	11.56	
1.30 ~ 1.35	15.00	5.60	40.20	84.00	247.50	47.70	5.19	693.81	52.30	13.27	
1.35 ~ 1.40	27.10	6.20	61.25	168.02	415.52	74.80	5.56	525.79	25.20	20.86	56.10
1.40 ~ 1.50	14.00	9.80	81.80	137.20	552.72	88.80	6.22	388.59	11.20	34.70	17.90
1.50 ~ 1.60	3.90	19.90	90.75	77.61	630.33	92.70	6.80	310.98	7.30	42.60	
1.60 ~ +	7.30	42.60	96.35	310.98	941.31	100.00	9.41	.00	.00	.00	
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Fig. 14-4

WASHABILITY CURVES

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT 16 DATE: SEPTEMBER 1979

SAMPLE NUMBER 3

SIZE : -10MM

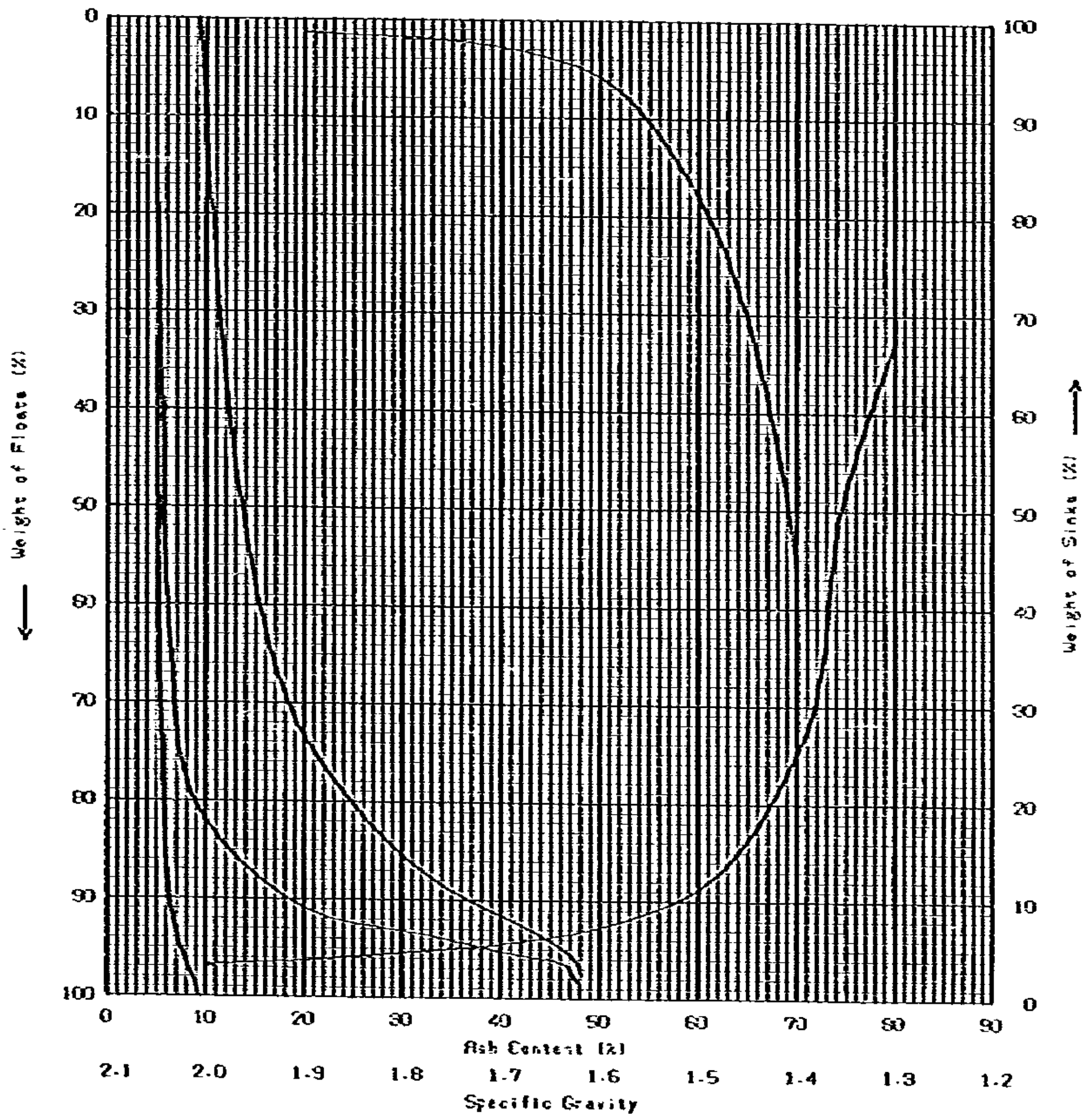

Table 13-5

FLOAT AND SINK TEST

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT38

SAMPLE NUMBER 1

DATE: OCTOBER 1979

SIZE: UNDER 10MM

Specific gravity	a		b	c	d	e	f	g	h	i	j
	Weight (%)	Ash (%)	$\Sigma W_{n-1} + \frac{W_n}{2}$	W·A	ΣW·A	ΣW	$\frac{\Sigma W \cdot A}{\Sigma W}$	Total W·A −ΣW·A	100 −ΣW·A	$\frac{g}{h}$	±0.1 SG
− 1.30	17.50	5.40	8.75	94.50	94.50	17.50	5.40	1.174.38	82.50	14.23	
1.30 ~ 1.35	8.50	5.70	21.75	48.45	142.95	26.00	5.50	1.125.93	74.00	15.22	
1.35 ~ 1.40	25.50	6.10	38.75	155.55	298.50	51.50	5.80	970.38	48.50	20.01	64.10
1.40 ~ 1.50	30.10	11.10	66.55	334.11	632.61	81.60	7.75	636.27	18.40	34.58	36.80
1.50 ~ 1.60	6.70	20.40	84.95	136.68	769.29	88.30	8.71	499.59	11.70	42.70	
1.60 ~ +	11.70	42.70	94.15	499.59	1.268.88	100.00	12.69	.00	.00	.00	
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Fig. 14-5

VASHABILITY CURVES

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT 38 DATE: OCTOBER 1979

SAMPLE NUMBER 1

SIZE : -10MM

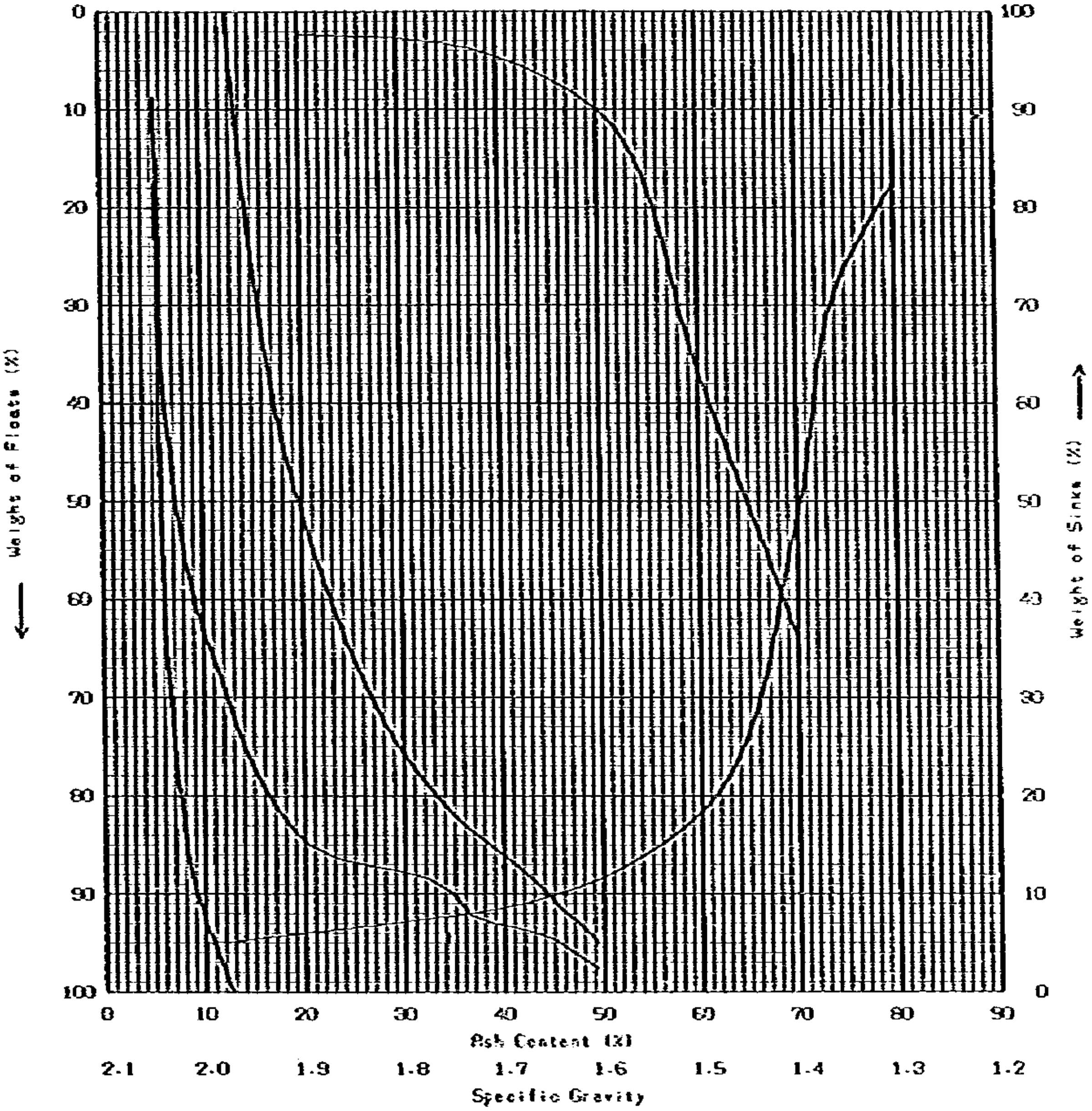

Table 13-6

FLOAT AND SINK TEST

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT38

SAMPLE NUMBER 2

DATE: OCTOBER 1979

SIZE: UNDER 10MM

Specific gravity	a		b	c	d	e	f	g	h	i	j
	Weight (%)	Ash (%)	$\frac{\Sigma W_{n-1} + W_n}{2}$	W·A	ΣW·A	ΣW	$\frac{\Sigma W \cdot A}{\Sigma W}$	Total W·A −ΣW·A	100 −ΣW·A	$\frac{g}{h}$	±0.1 SG
− 1.30	6.80	5.30	3.40	36.04	36.04	6.80	5.30	2.220.90	93.20	23.83	
1.30 ~ 1.35	9.00	5.70	11.30	51.30	87.34	15.80	5.53	2.169.60	84.20	25.77	
1.35 ~ 1.40	23.40	7.20	27.50	168.48	255.82	39.20	6.53	2.001.12	60.80	32.91	52.10
1.40 ~ 1.50	19.70	13.40	49.05	263.98	519.80	58.90	8.83	1.737.14	41.10	42.27	27.30
1.50 ~ 1.60	7.60	21.40	62.70	162.64	682.44	66.50	10.26	1.574.50	33.50	47.00	
1.60 ~ +	33.50	47.00	83.25	1.574.50	2.256.94	100.00	22.57	.00	.00	.00	
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Fig. 14-6

WASHABILITY CURVES

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT 38 DATE: OCTOBER 1979

SAMPLE NUMBER 2

SIZE : -10MM

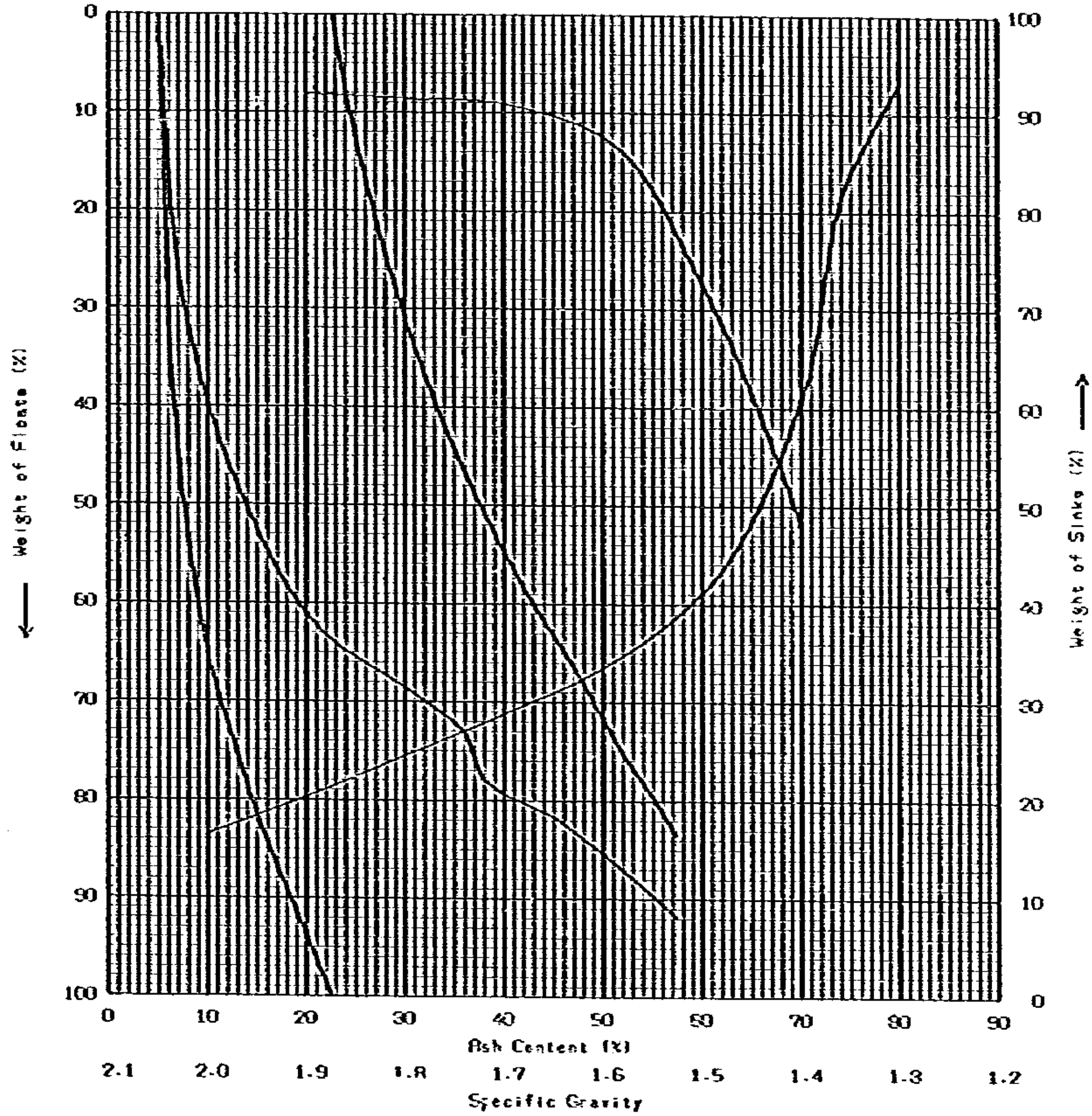

Table 13-7

FLOAT AND SINK TEST

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT42

SAMPLE NUMBER 1

DATE: OCTOBER 1979

SIZE: UNDER 10MM

Specific gravity	a		b	c	d	e	f	g	h	i	j
	Weight (%)	Ash (%)	$\frac{\Sigma W_{n-1} + W_n}{2}$	W·A	ΣW·A	ΣW	$\frac{\Sigma W \cdot A}{\Sigma W}$	Total W·A −ΣW·A	100 −ΣW·A	$\frac{g}{h}$	±0.1 SG
− 1.30	6.20	5.90	3.10	36.58	36.58	6.20	5.90	2.241.12	93.80	23.89	
1.30 ~ 1.35	5.50	6.30	8.95	34.65	71.23	11.70	6.09	2.206.47	88.30	24.99	
1.35 ~ 1.40	19.00	7.40	21.20	140.60	211.83	30.70	6.90	2.065.87	69.30	29.81	53.20
1.40 ~ 1.50	28.70	10.50	45.05	301.35	513.18	59.40	8.64	1.764.52	40.60	43.46	38.90
1.50 ~ 1.60	10.20	19.80	64.50	201.96	715.14	69.60	10.27	1.562.56	30.40	51.40	
1.60 ~ +	30.40	51.40	84.80	1.562.56	2.277.70	100.00	22.78	.00	.00	.00	
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Fig. 14-7

WASHABILITY CURVES

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT 42 DATE: OCTOBER 1979

SAMPLE NUMBER 1

SIZE : -10MM

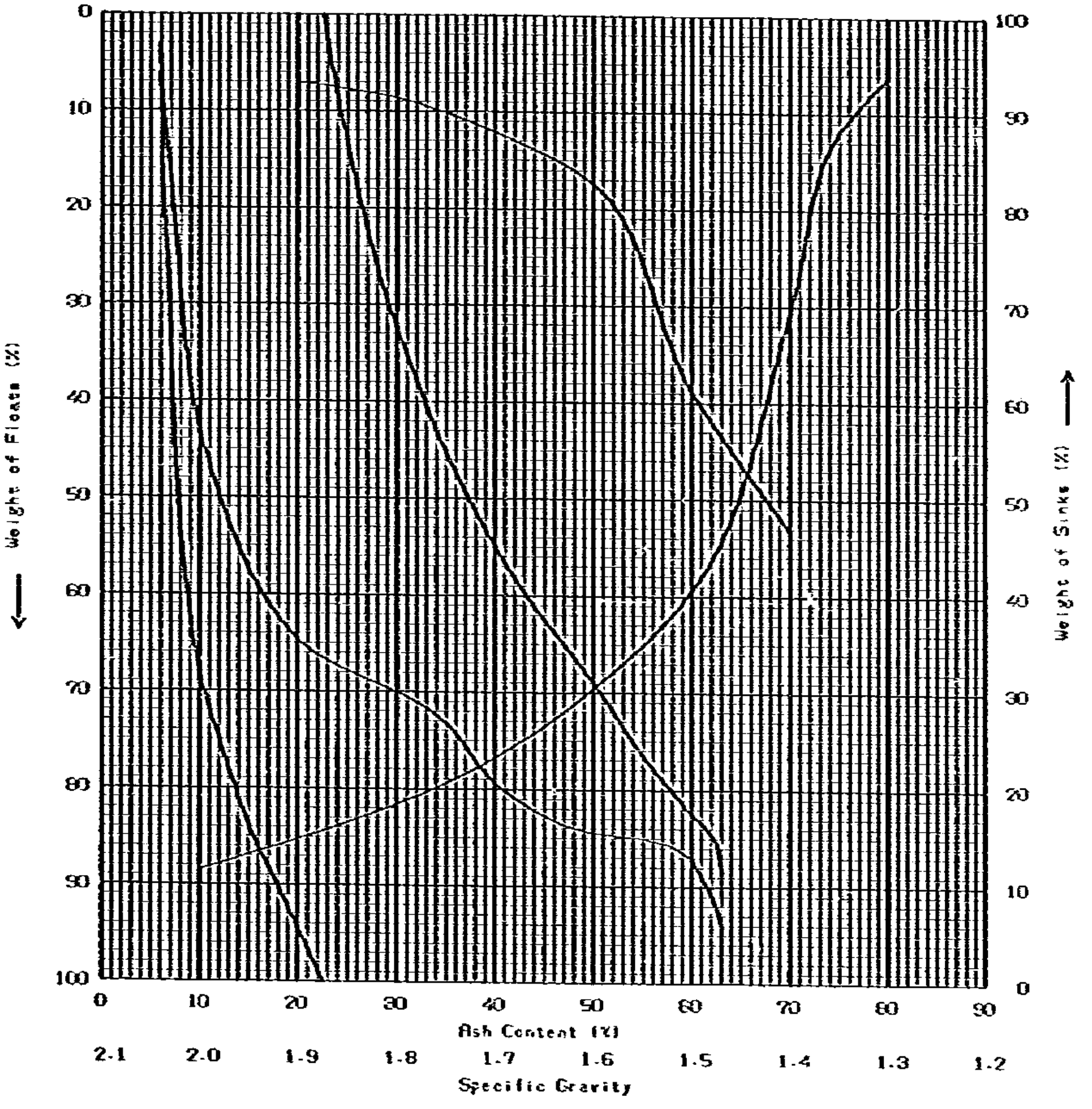

Table 13-8

FLOAT AND SINK TEST

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT44

SAMPLE NUMER 1

DATE: NOVEMBER 1979

SIZE: -10MM

Specific gravity	a		b	c	d	e	f	g	h	i	j
	Weight (%)	Ash (%)	$\Sigma W_{n-1} + \frac{W_n}{2}$	W·A	ΣW·A	ΣW	$\frac{\Sigma W \cdot A}{\Sigma W}$	Total W·A −ΣW·A	100 −ΣW·A	$\frac{g}{h}$	±0.1 SG
− 1.30	15.00	5.60	7.50	84.00	84.00	15.00	5.60	1.217.42	85.00	14.32	
1.30 ~ 1.35	14.70	5.90	22.35	86.73	170.73	29.70	5.75	1.130.69	70.30	16.08	
1.35 ~ 1.40	23.90	6.50	41.65	155.35	326.08	53.60	6.08	975.34	46.40	21.02	67.10
1.40 ~ 1.50	28.50	12.40	67.85	353.40	679.48	82.10	8.28	621.94	17.90	34.75	34.40
1.50 ~ 1.60	5.90	20.60	85.05	121.54	801.02	88.00	9.10	500.40	12.00	41.70	
1.60 ~ +	12.00	41.70	94.00	500.40	1.301.42	100.00	13.01	.00	.00	.00	
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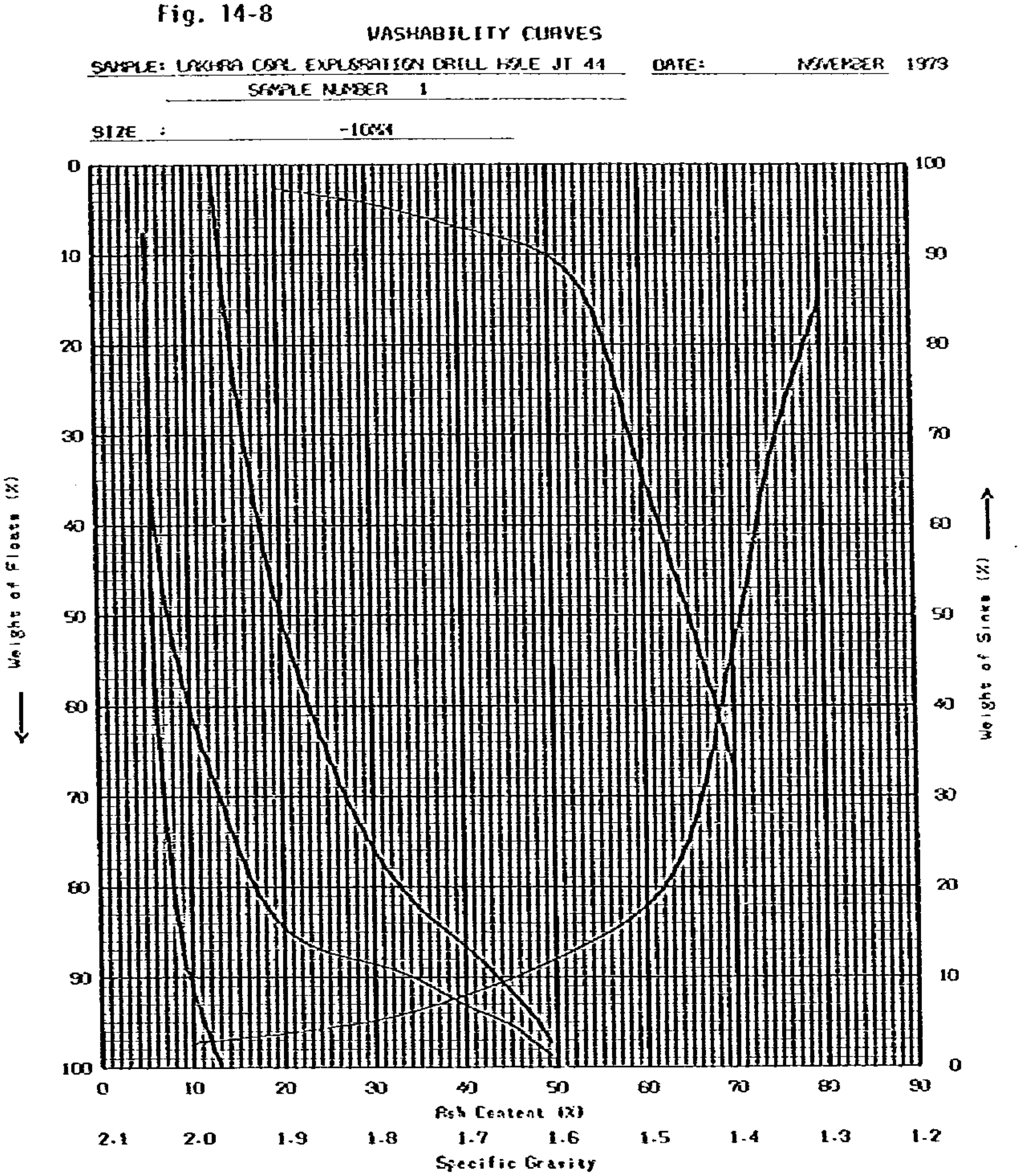
Fig. 14-8



WASHABILITY CURVES
SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT 44
DATE: NOVEMBER 1979
SAMPLE NUMBER 1
SIZE : -10MM
0
10
20
30
40
50
60
70
80
90
100
Weight of Floats (%)
100
90
80
70
60
50
40
30
20
10
0
Weight of Sinks (%)
0
10
20
30
40
50
60
70
80
90
Ash Content (%)
2.1
2.0
1.9
1.8
1.7
1.6
1.5
1.4
1.3
1.2
Specific Gravity

Table 13-9

FLOAT AND SINK TEST

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT44 — DATE: NOVEMBER 1979

SAMPLE NUMBER 2

SIZE: -10MM

Specific gravity	a		b	c	d	e	f	g	h	i	j
	Weight (%)	Ash (%)	$\Sigma W_{n-1} + \frac{W_n}{2}$	W·A	ΣW·A	ΣW	$\frac{\Sigma W \cdot A}{\Sigma W}$	Total W·A −ΣW·A	100 −ΣW·A	$\frac{g}{h}$	±0.1 SG
- 1.30	7.80	5.30	3.90	41.34	41.34	7.80	5.30	1.498.63	92.20	16.25	
1.30 ~ 1.35	6.70	5.50	11.15	36.85	78.19	14.50	5.39	1.461.78	85.50	17.10	
1.35 ~ 1.40	32.30	6.10	30.65	197.03	275.22	46.80	5.88	1.264.75	53.20	23.77	65.90
1.40 ~ 1.50	26.90	11.50	60.25	309.35	584.57	73.70	7.93	955.40	26.30	36.33	34.50
1.50 ~ 1.60	7.60	20.40	77.50	155.04	739.61	81.30	9.10	800.36	18.70	42.80	
1.60 ~ +	18.70	42.80	90.65	800.36	1.539.97	100.00	15.40	.00	.00	.00	
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Fig. 14-9

WASHABILITY CURVES

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT 44 DATE: NOVEMBER 1979

SAMPLE NUMBER 2

SIZE : -10MM

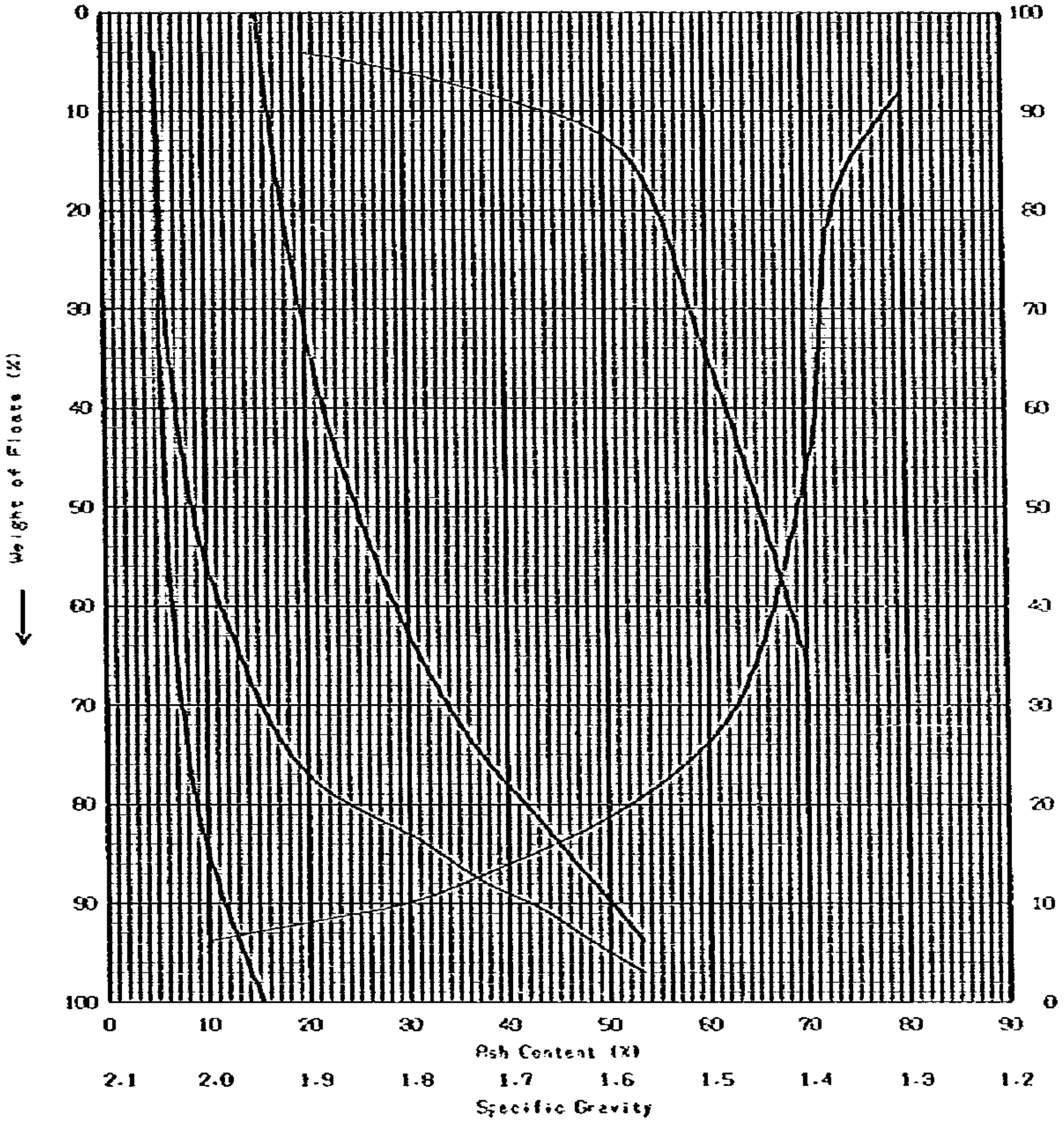

Table 13-10

FLOAT AND SINK TEST

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT44

SAMPLE NUMBER 3

DATE: NOVEMBER 1979

SIZE: -10MM

Specific gravity	a		b	c	d	e	f	g	h	i	j
	Weight (%)	Ash (%)	$\Sigma W_{n-1} + \frac{W_n}{2}$	W·A	ΣW·A	ΣW	$\frac{\Sigma W\cdot A}{\Sigma W}$	Total W·A −ΣW·A	100 −ΣW·A	$\frac{g}{h}$	±0.1 SG
− 1.30	9.60	4.70	4.80	45.12	45.12	9.60	4.70	2,771.06	90.40	30.65	
1.30 ~ 1.35	8.90	4.90	14.05	43.61	88.73	18.50	4.80	2,727.45	81.50	33.47	
1.35 ~ 1.40	21.90	6.10	29.45	133.59	222.32	40.40	5.50	2,593.86	59.60	43.52	47.90
1.40 ~ 1.50	17.10	15.10	48.95	258.21	480.53	57.50	8.36	2,335.65	42.50	54.96	26.10
1.50 ~ 1.60	9.00	23.90	62.00	215.10	695.63	66.50	10.46	2,120.55	33.50	63.30	
1.60 ~ +	33.50	63.30	83.25	2,120.55	2,816.18	100.00	28.16	.00	.00	.00	
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Fig. 14-10

WASHABILITY CURVES

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE JT 44 DATE: NOVEMBER 1979

SAMPLE NUMBER 3

SIZE : -10M

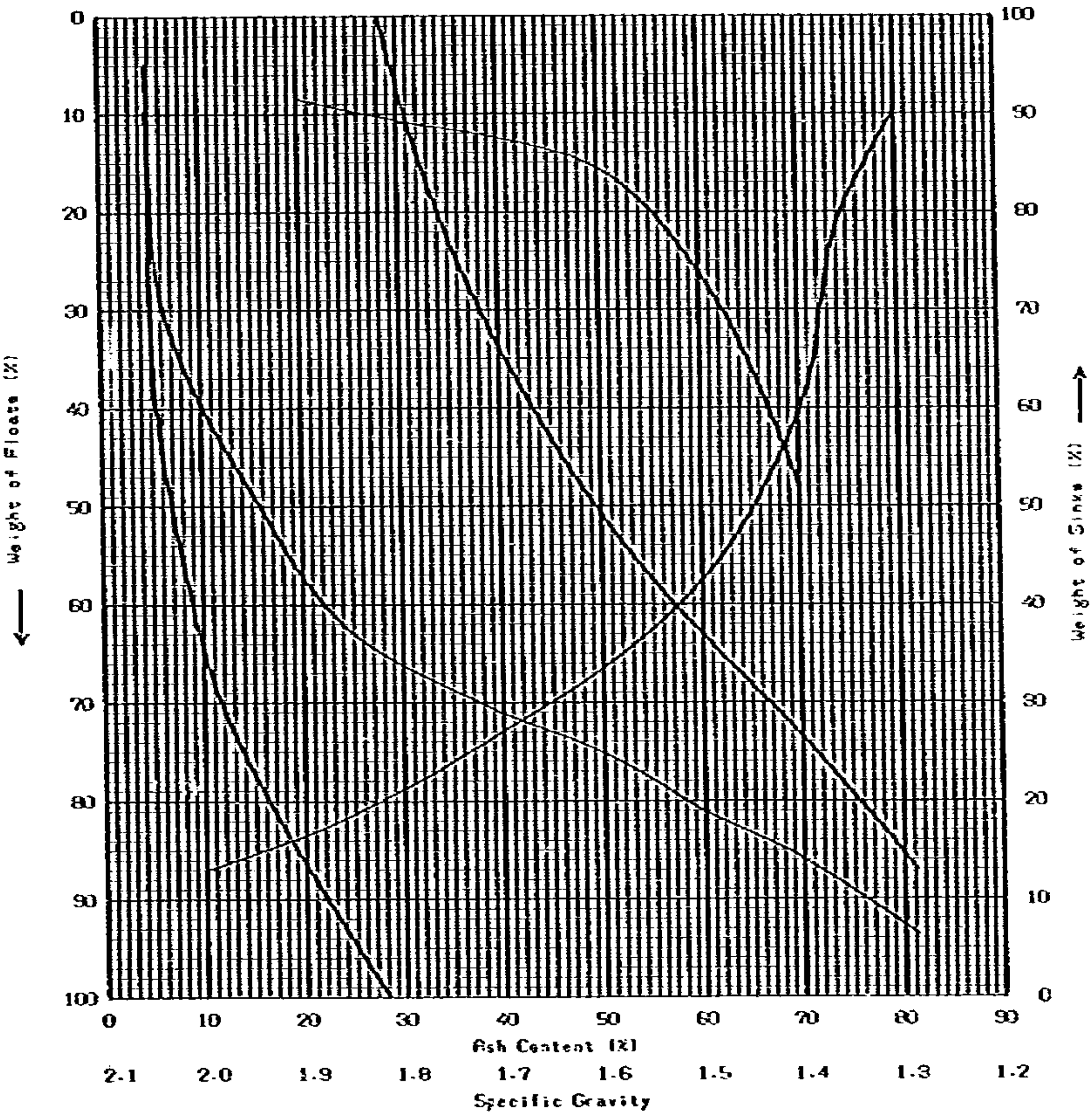

Fig. 13-11

FLOAT AND SINK TEST

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE
10 HOLES COMPOSITE

DATE: 1979

SIZE: -10MM

Specific gravity	a		b	c	d	e	f	g	h	i	j
	Weight (%)	Ash (%)	$\Sigma W_{n-1} + \frac{W_n}{2}$	W·A	ΣW·A	ΣW	$\frac{\Sigma W \cdot A}{\Sigma W}$	Total W·A −ΣW·A	100 −ΣW·A	$\frac{g}{h}$	±0.1 SG
− 1.30	11.93	5.42	5.97	64.66	64.66	11.93	5.42	1.904.37	88.07	21.62	
1.30 ~ 1.35	9.09	5.77	16.47	52.45	117.11	21.02	5.57	1.851.92	78.98	23.45	
1.35 ~ 1.40	23.07	6.55	32.55	151.11	268.22	44.09	6.08	1.700.82	55.91	30.42	53.94
1.40 ~ 1.50	21.78	11.67	54.98	254.17	522.39	65.87	7.93	1.446.64	34.13	42.39	29.14
1.50 ~ 1.60	7.36	20.12	69.55	148.08	670.47	73.23	9.16	1.298.56	26.77	48.51	12.28
1.60 ~ 1.70	4.92	31.72	75.69	156.06	826.54	78.15	10.58	1.142.50	21.85	52.29	8.97
1.70 ~ 1.80	4.05	39.25	80.17	158.96	985.50	82.20	11.99	983.54	17.80	55.25	7.46
1.80 ~ 1.90	3.41	43.95	83.90	149.87	1.135.37	85.61	13.26	833.67	14.39	57.93	6.41
1.90 ~ 2.00	3.00	51.38	87.11	154.14	1.289.51	88.61	14.55	679.53	11.39	59.66	
2.00 ~ +	11.39	59.66	94.30	679.53	1.969.03	100.00	19.69	.00	.00	.00	
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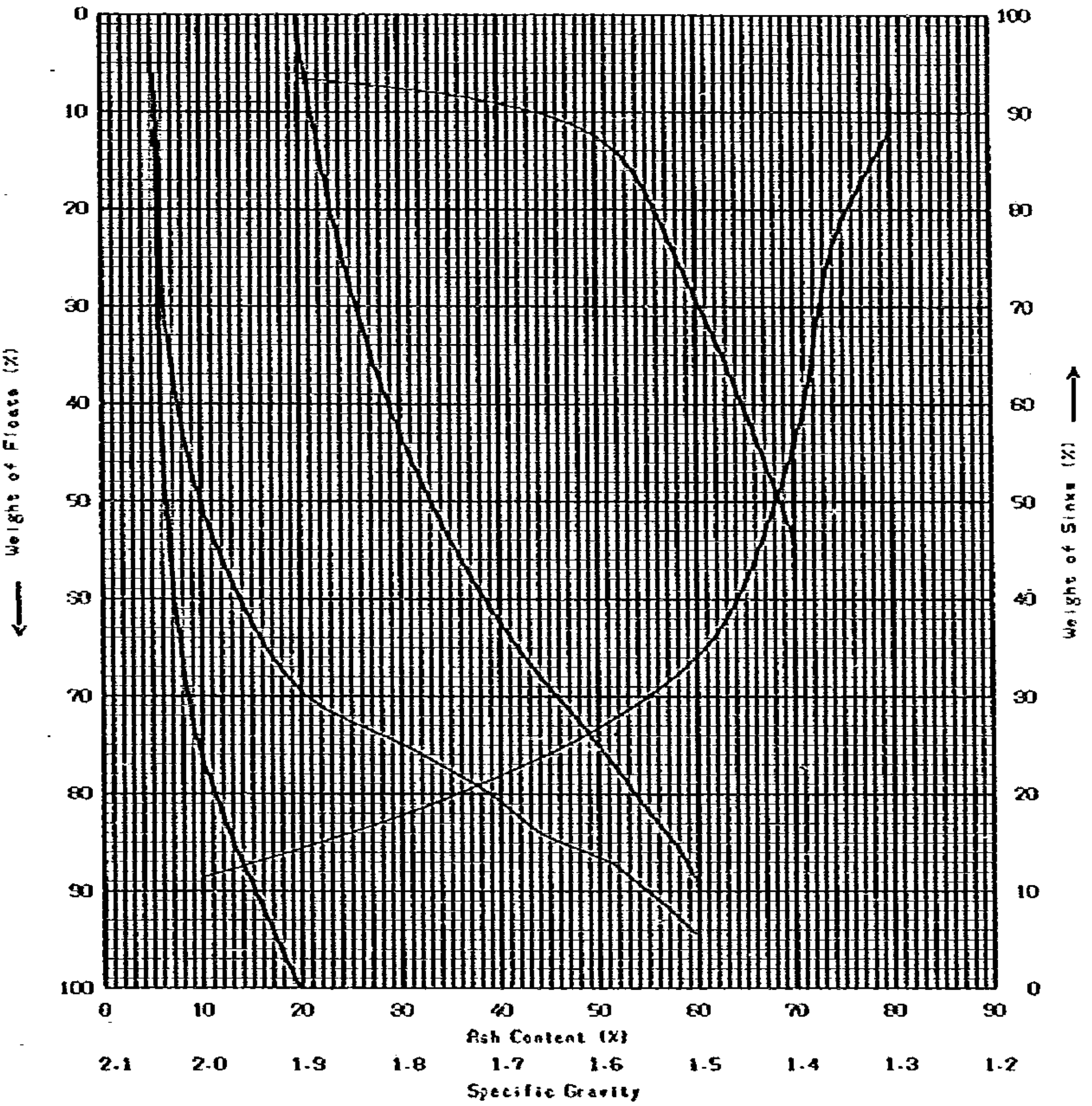
Fig. 14-11

WASHABILITY CURVES

SAMPLE: LAKHRA COAL EXPLORATION DRILL HOLE DATE: 1979

10 HOLES COMPOSITE

SIZE : -10MM



ISOPACH MAP OF NO.1 SEAM (O/C)

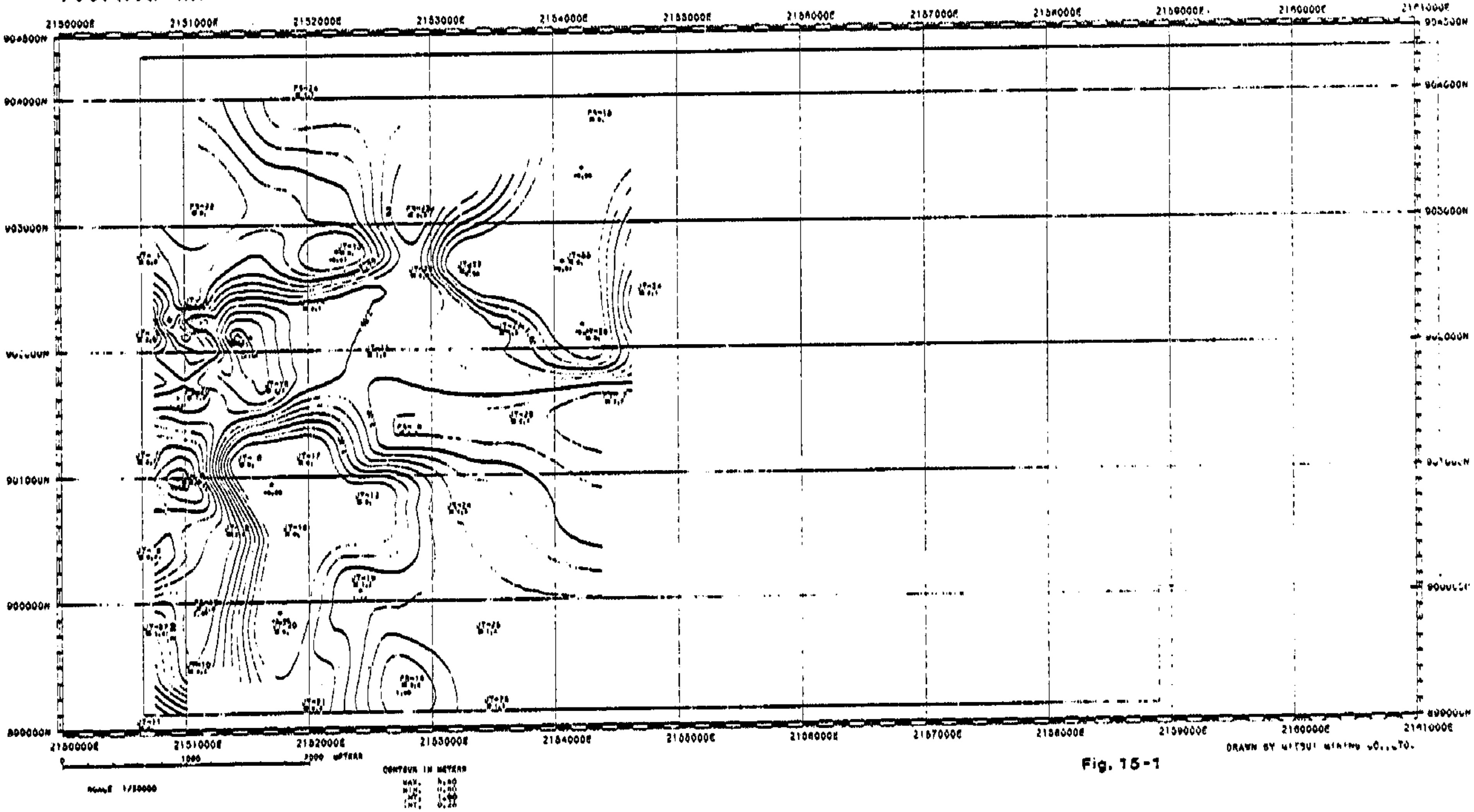


Fig. 15-1

ISOPACH MAP OF NO.2 SEAM (O/C)

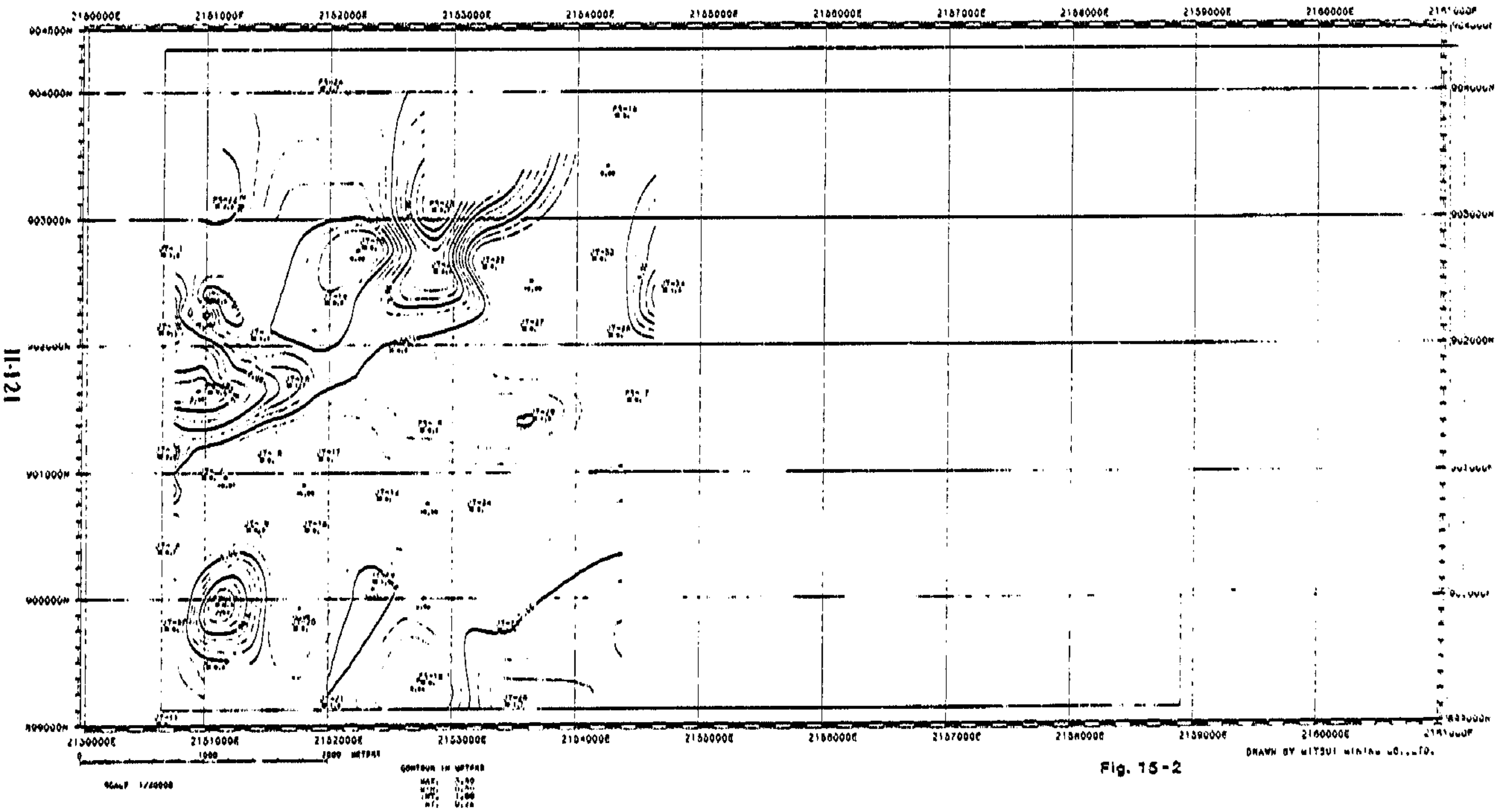


Fig. 15-2

ISOPACH MAP OF NO.2&1 SEAM(G/C)

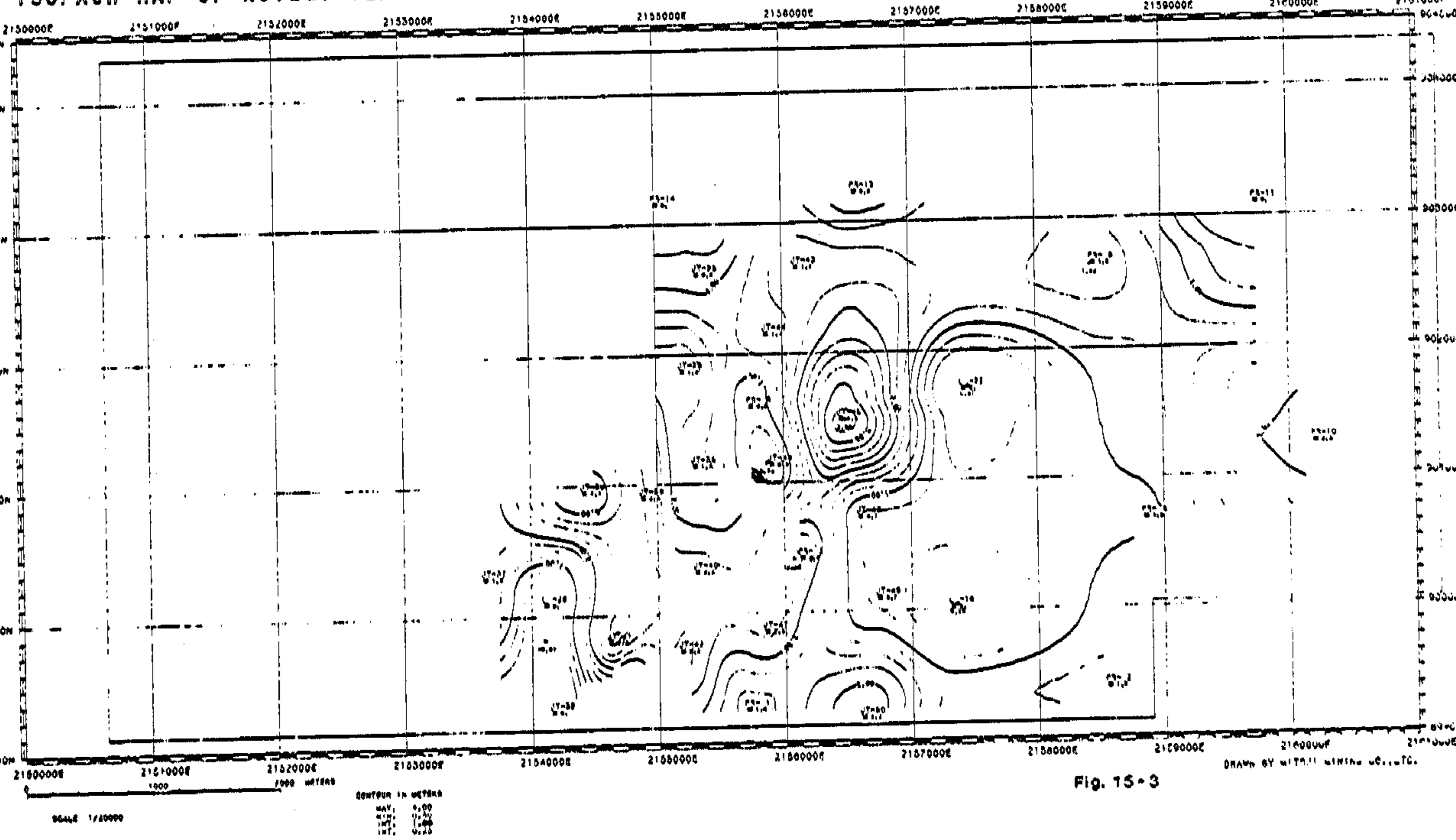


Fig. 15-3

ISOPACH MAP OF NO.3 SEAM (O/C)

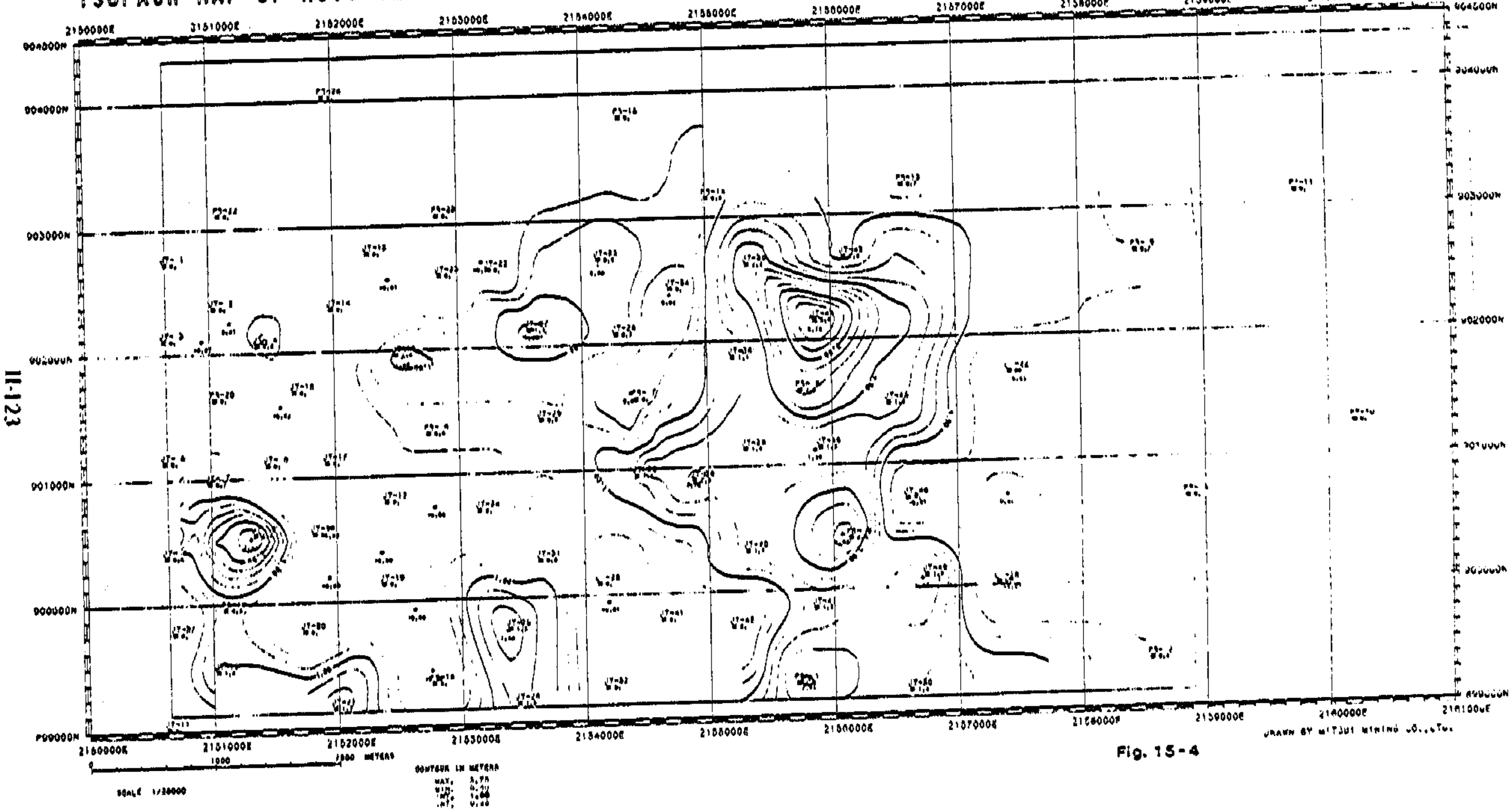


Fig. 15-4

ISOPACH MAP OF NO.4 SEAM (O/C)

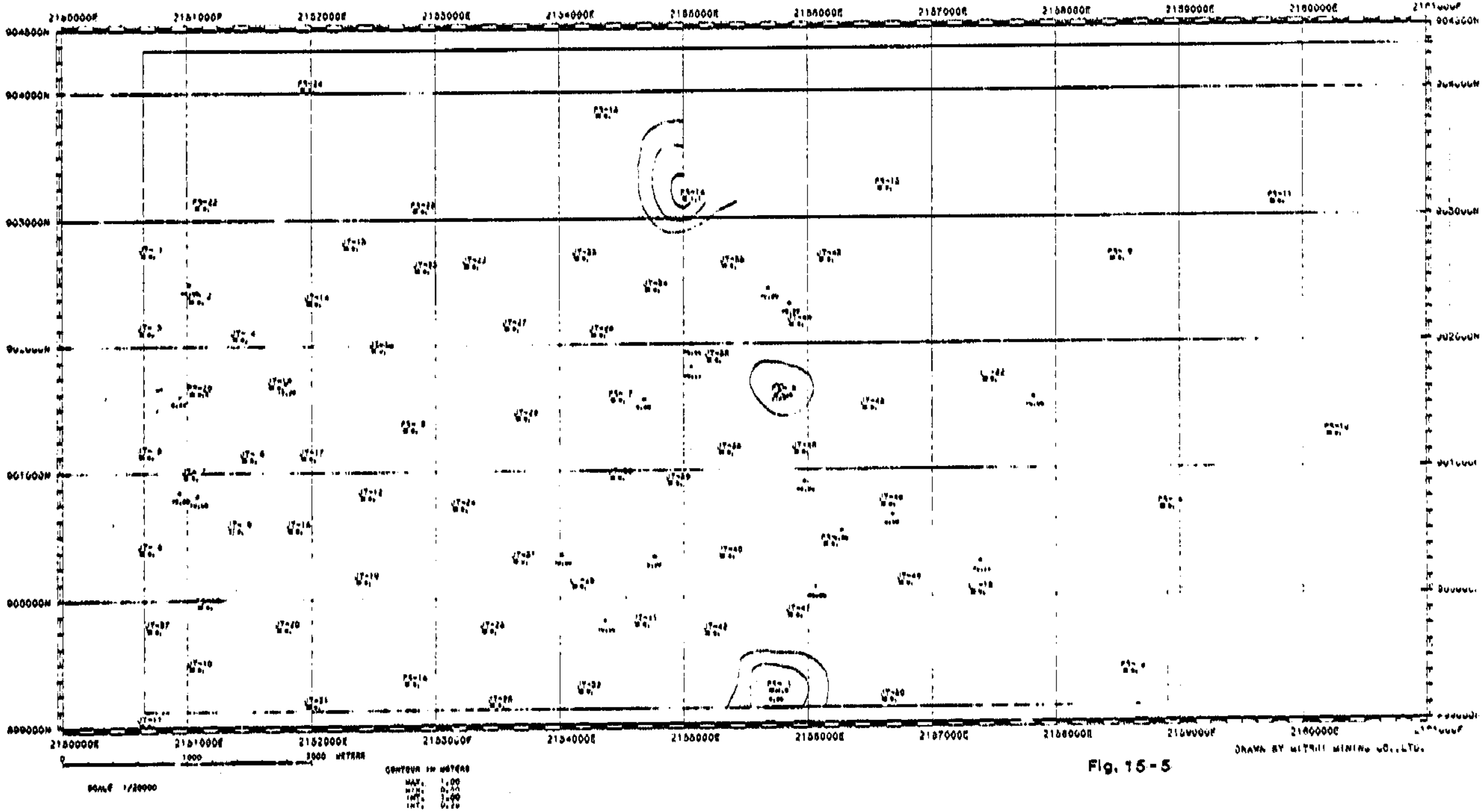


Fig. 15-5

ISOPACH MAP OF NO.5 SEAM (O/C)

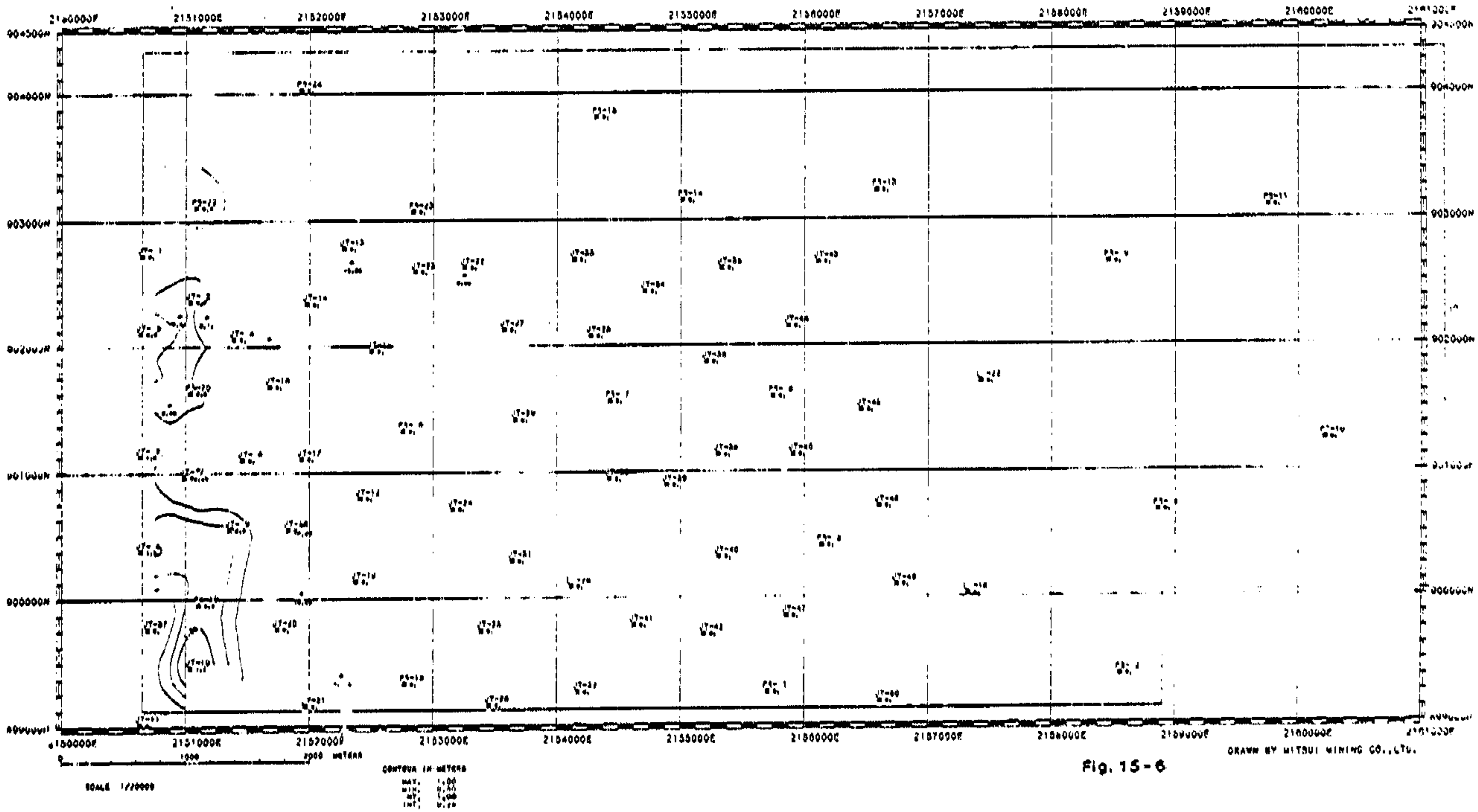
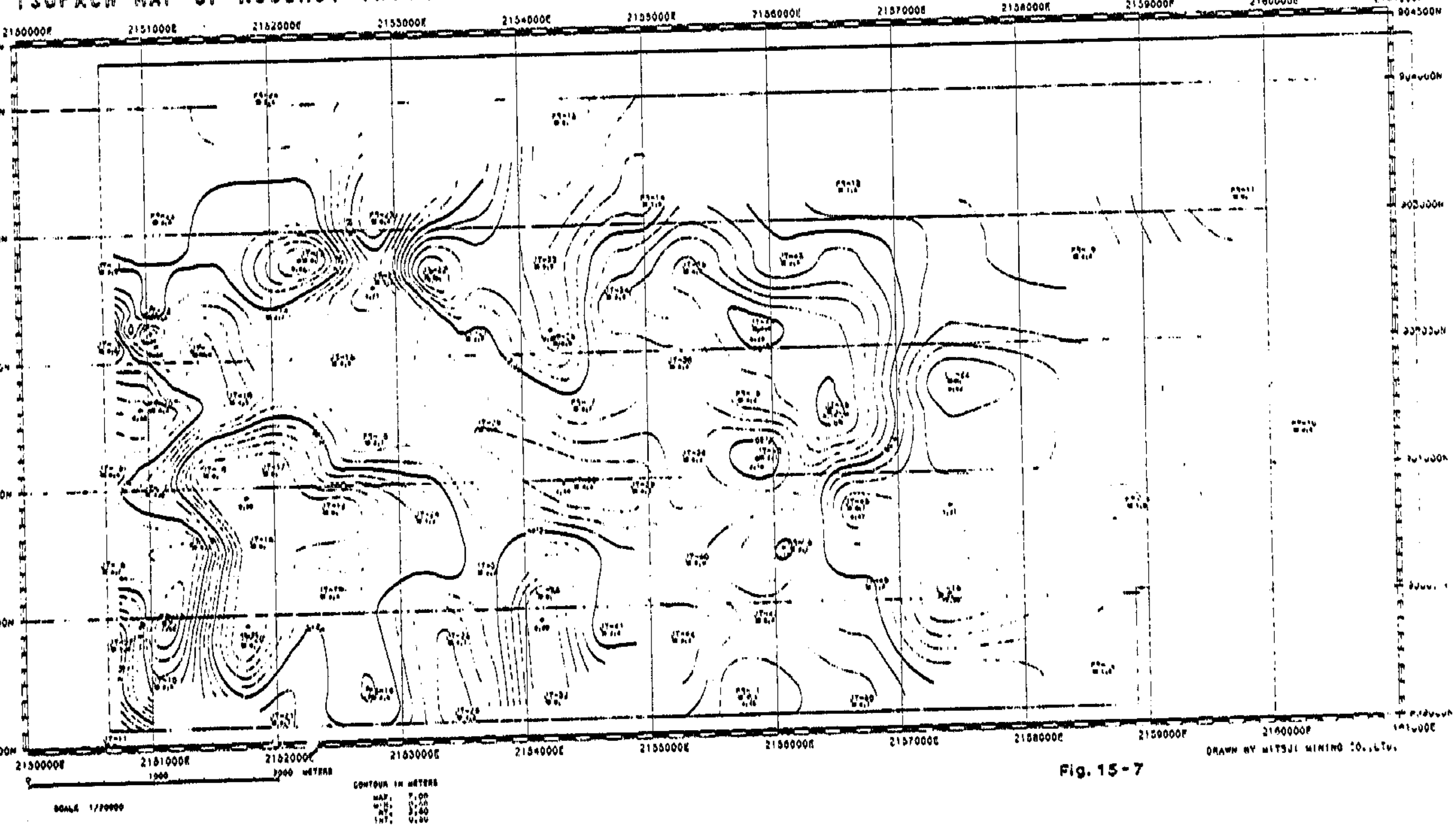


Fig. 15-6

ISOPACH MAP OF ACCUMU. THICKNESS (O/C)



DRAWN BY MITSUI MINING CO.,LTD.

Fig. 15-7

STRIP RATIO CONTOUR MAP

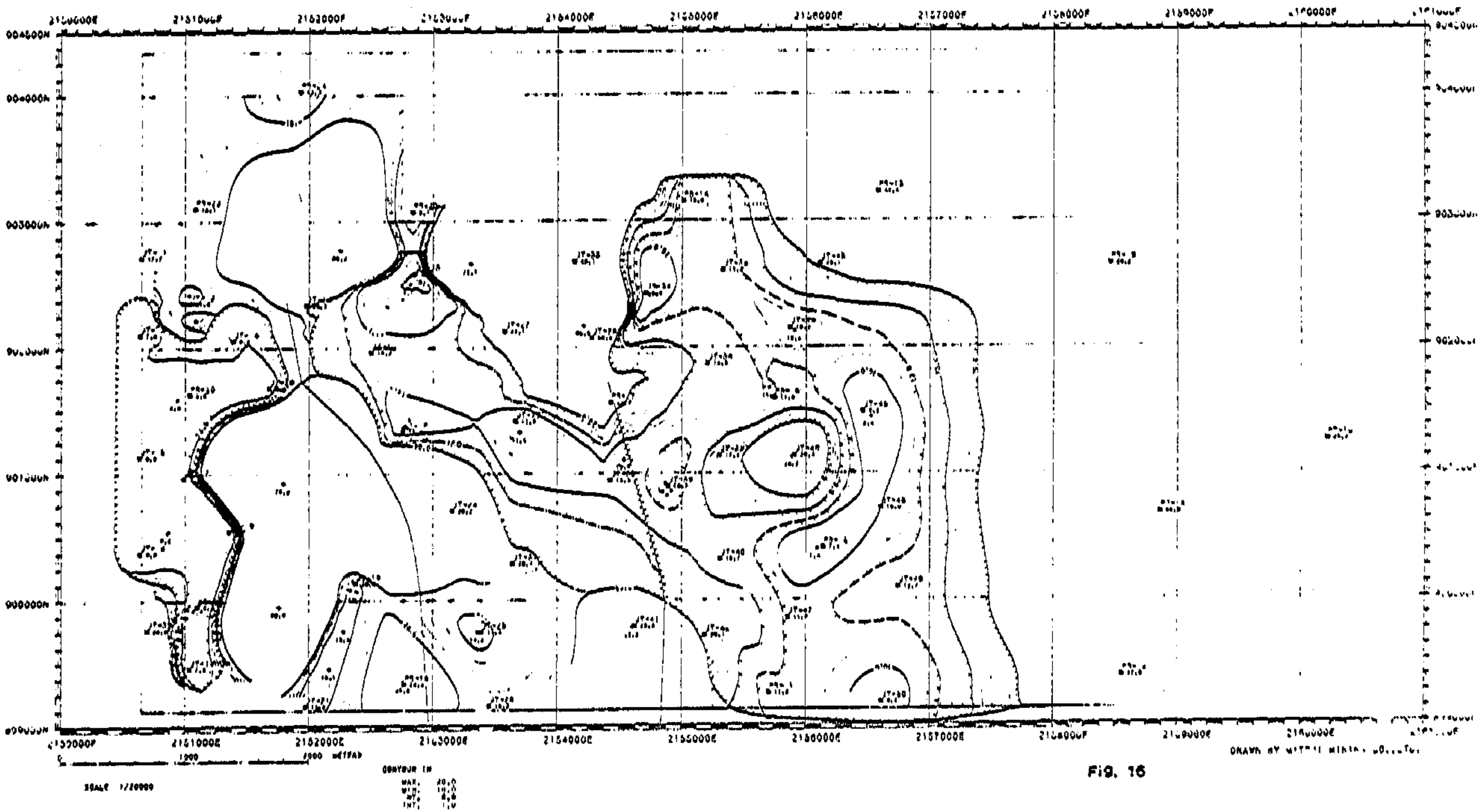


FIG. 16

ISOPACH MAP OF NO.2&1 SEAM (U/G)

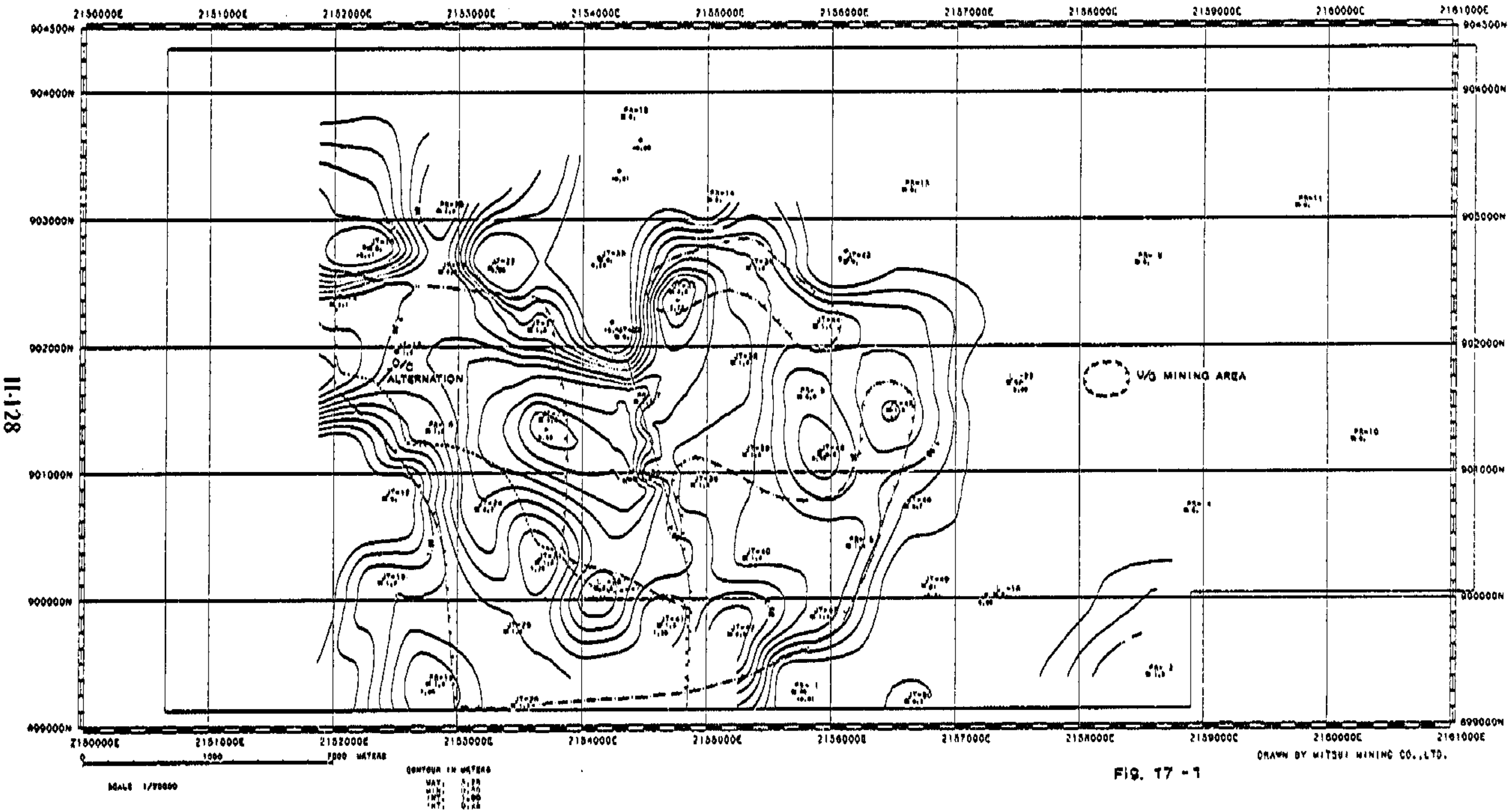
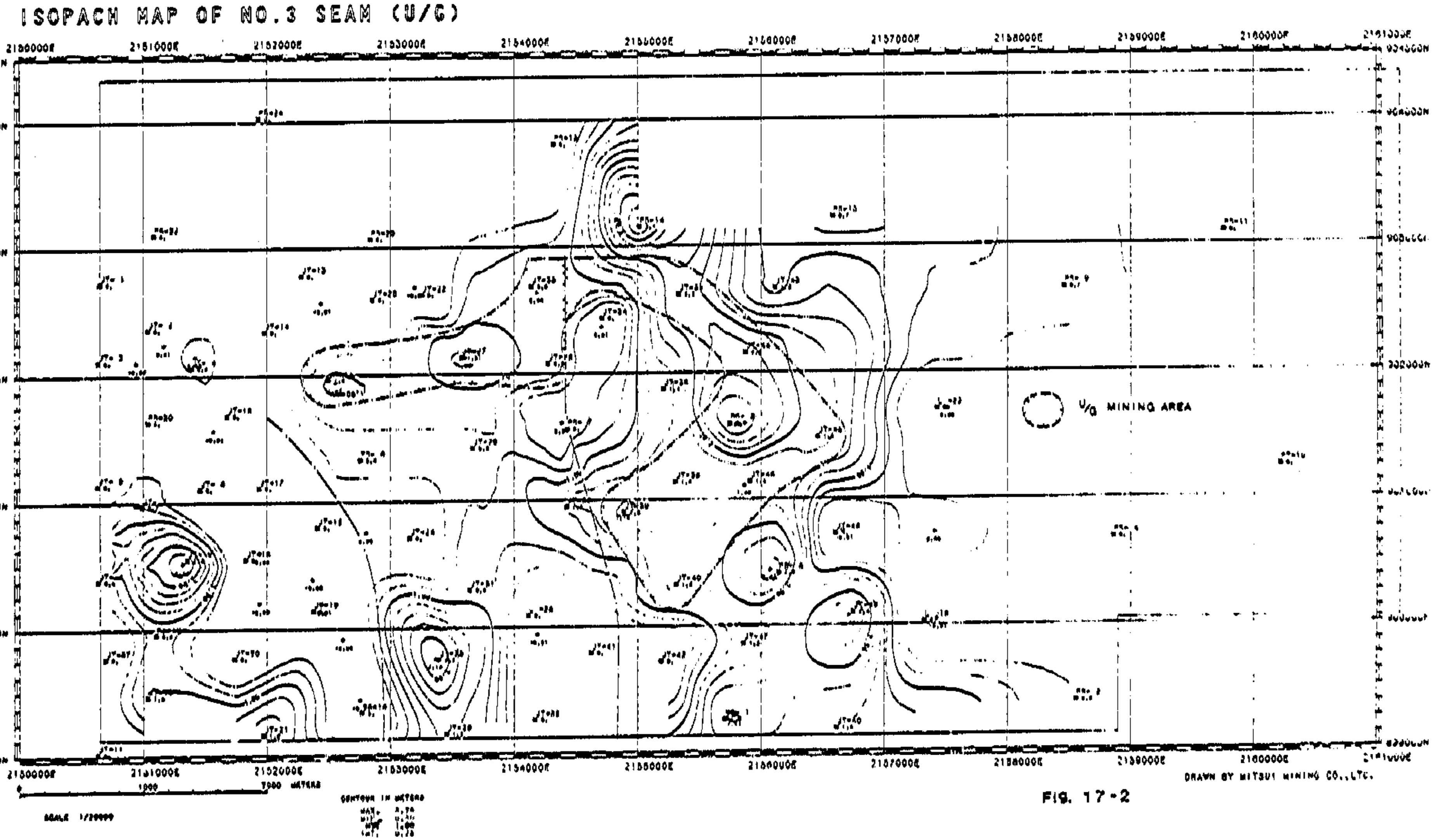


FIG. 17 -1



ISOPACH MAP OF NO.3 SEAM (U/G)
U/G MINING AREA
DRAWN BY MITSUI MINING CO.,LTD.

FIG. 17-2

FLOOR DEPTH CONTOUR MAP OF NO.1 SEAM

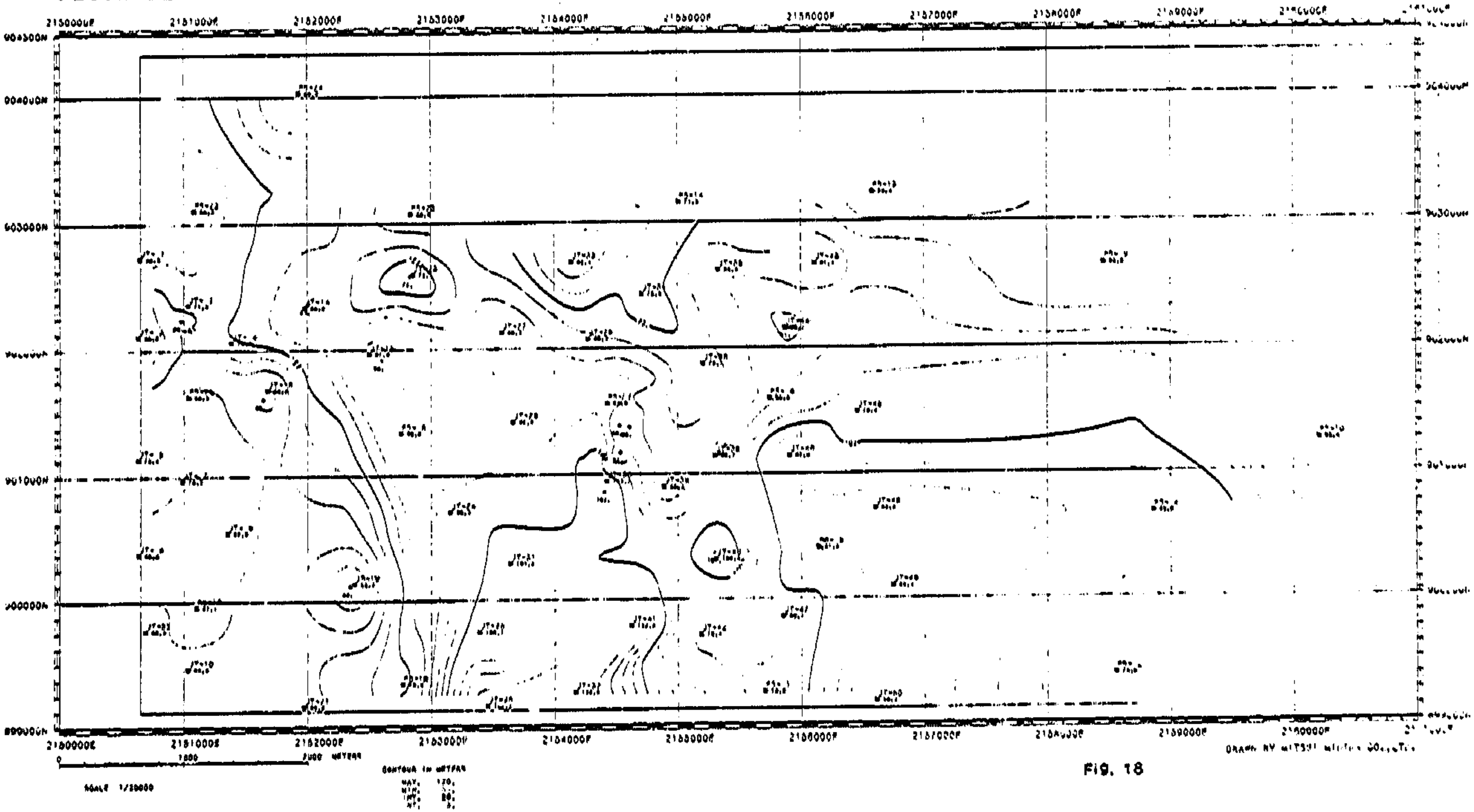


FIG. 18

ISOPACH MAP OF LAKI LIMESTONE

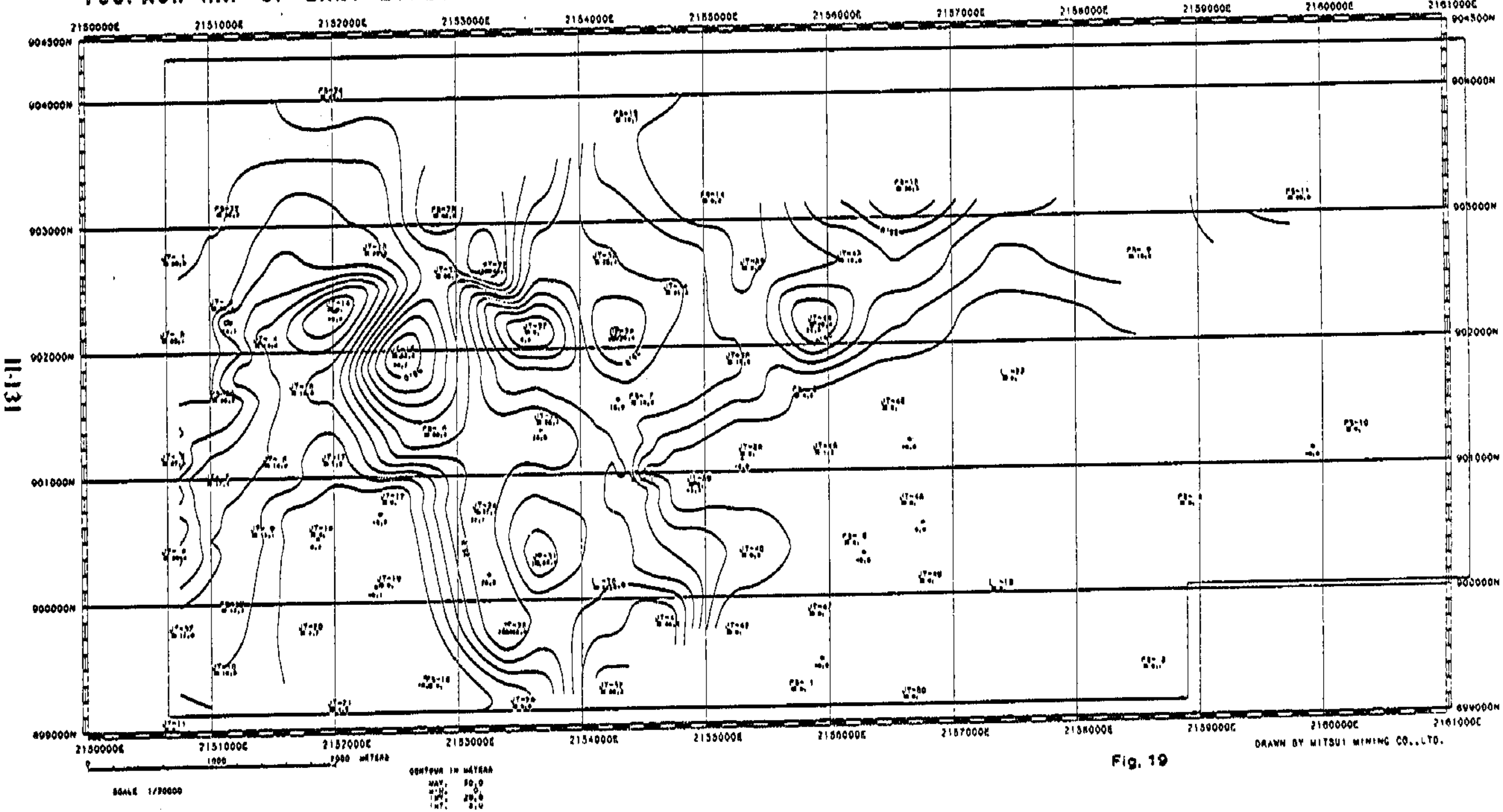


Fig. 19

Table 14 Comparison of separation

Specific gravity of separation medium	1.6	1.8*
Raw coal ash %	19.7	19.7
Clean coal ash %	9.2	12.0
Calorific value (Clean) kcal/kg	6,556 (dry basis)	6,293 (dry basis)
Yield %	73.2	82.2
±0.1 near gravity %	12.3	7.5

* estimated from washability curves

In the above Table, it appears that when cleaning at 1.6 the near gravity material is over 10 percent, and the washability is somewhat difficult and yield is lower, but while cleaning at 1.8 the near gravity material is improved to around 7 percent, and the washability indicates a simple and good separation so that it can be satisfactorily cleaned with a Jig washing system and the clean coal yield will be increased upto 80 percent.

Although this method will give a slightly higher ash content and lower calorific value, the middlings of this coal will give good results and can be utilized effectively. It is therefore desirable to beneficiate with a medium of 1.8 specific gravity by using Jig washing system.

Generally, Lakhra coal has high sulphur content. Pyrite in total sulphur indicates high values such as 50 to 60 percent. It is anticipated that the considerable portion of the pyritic sulphur in the raw coal will be reduced by washing. The tests indicate that the total sulphur at 6.84 percent in the raw coal will be reduced to 4.24 percent in the clean coal, and the average rate of removal will be 38 percent.

The results of tests showing sulphur content in the raw and clean coal samples are detailed in Table 15.

Table 15 Sulphur content and removal rate

Sample	Total Sulphur %		Removal
	Raw Coal	Clean Coal *	Rate %
JT 5-1	6.70	3.88	42.1
JT 7-5	9.50	7.31	23.1
JT24-1	13.21	7.30	44.7
JT29-7	14.43	8.10	43.9
JT10-4	8.02	5.72	28.8
JT34-7	10.00	5.86	41.4
JT45-1	5.10	3.27	35.9
Average	9.57	5.92	38.1

* Separation medium with less than 1.8 specific gravity

After the results of mining study and design of boiler for the power plant, the washability for the Lakhra coal will be studied in detail.

4-4 Mining Conditions

For open-cut mining strip ratio, hardness of overburden materials and slope stability are the main factors, which should be considered, whereas for underground mining continuity of coal seams, characteristics of roof and floor rocks, and structure are important.

4-4-1 Characteristics of Roof and Floor Rocks

Most of the immediate roofs and floors in the investigated area consist of clay, shale, and siltstone, which are very fine grained rocks, but sometime sandstone replaces them. Claystone and shale are favourable as roof of coal, whereas soft and loose sandstone is unfavourable as it can easily cave in and may cause some accident. The areas, where such poor types of roof conditions exist, could not be recommended for underground mining. From the mining point of view it is desirable that there should be more than 2 metres thick claystone, siltstone, shale or relatively hard sandstone interposed between the loose and friable sandstone if it occurs in the roof rocks, and the coal seam to be mined. In consideration of this the eastern and central blocks of the investigated area, where loose and friable sandstone exists along the immediate roof could not be recommended for underground mining. Therefore, underground mining is very limited in the investigated area. Only coal zone No. 1 can be mined to a considerable extent spreading over almost whole of the central block. Furthermore, if the open-cut mining is considered for the eastern and central blocks, the strip ratio will be more than 1:10.

Regarding the characteristics of floor, it is experienced that loose sandstone does not pose serious problems except the sinking of supports into the floor. Presence of shale or claystone in floor generally causes rock swelling.

4-4-2 Thickness and Characteristics of Overburden

Strip ratio is calculated as coal weight [working height (m) x specific gravity] per unit area divided by volume of rock (m^3) per unit area above the coal.

Generally smaller strip ratio is more suitable for open-cut mining. The overburden rocks consist of Laki limestone and Basal Laki laterite of Laki group, Upper Shell Beds and Upper Coal-bearing Beds of Ranikot formation, which consists of sandstone, siltstones, claystones and shales in the investigated area. The southeastern part of the area lacks limestone and laterite.

The results of rock tests are discussed in chapter 6. However, uniaxial compressive strength of limestones is high and the values range widely. The results of rock tests for limestone, sandstone, siltstone and claystone are shown in Table 16.

Table 16 Rock Test

Rock	Specific Gravity	Porosity %	* S.S.W.P.V. m/sec	** U.C.S. kg/cm²	Tensile Strength kg/cm²
Limestone	1.93 - 2.59	57.61 - 3.53	2,650 - 5,250	122.5 - 957.3	6.1 - 70.5
Sandstone	1.70 - 2.39	60.98 - 19.96	1,010 - 4,160	5.2 - 323.0	4.0 - 24.5
Siltstone and Clay	1.79 - 2.43	65.20 - 29.13	885 - 3,550	9.2 - 312.6	3.9 - 13.1
***	2.69+ - 2.65	5.20+ - 6.74	4,890 - 5,470	1,142.3 - 1,780.4+	67.2+ - 89.0

* Propagation velocity of super-sonic wave

** Unaxial compressive strength

*** Compact and hard sandstone ..... very rare and very special

The shallowest overburden is 32.56 metres over zone No. 5 in JT10. The thickest overburden is 120.92 metres over zone No. 1 in JT26.

As previously mentioned, many coal seams lie in the western block with low strip ratio upto 1:10 (average 1:8.24), while to the east the ratio calculated at 1:13.04 is slightly higher, because of lack of a number of coal seams.

The strip ratio in the central block, particularly in the southern portion, is over 1:17 because of thick overburden. Therefore, at present, open-cut mining in this area is uneconomic. The ratio in the northern portion of the central block is also high (average 1:15.67).

4-4-3 General Condition of Water, Gas and Combustion

(1) Water Drainage and Gas

According to PMDC report, formation water is very little, and there will be no water problem in underground mining. However, for open-cut mining much attention should be paid on rain water during rany season. Gas emission is also reported to be very little. Therefore, no series problem will be faced during mining.

(2) Spontaneous Combustion

It is evident from the past experience that Lakhra coal is liable to spontaneous combustion. The tests for the spontaneous combustibility were carriedout after the blending of 10 samples which had been used for the float and sink tests.

Proximate analysis of the samples shows inherent moisture 9.4 percent, volatile matter 36.1 percent, and calorific value 6,930 kcal/kg on dry basis.

TABLE 17 - 1 SUMMARY OF COAL SEAMS IN EACH DRILL HOLES (1)

Drill Hole Number	Base of Limestone Depth • Height (m)	Red Zone Boundary Depth • Height (m)	No. 5 Seam Roof Depth • Height (m)	No. 5 Seam Thickness Coal (m)	No. 5 Seam Thickness Seam (m)	No. 5 Seam Floor Depth • Height (m)	No. 4 Seam Roof Depth • Height (m)	No. 4 Seam Thickness Coal (m)	No. 4 Seam Thickness Seam (m)	No. 4 Seam Floor Depth • Height (m)
JI- 1	35.53 • 99.68									
JI- 2	32.96 • 93.6?		52.35 • 82.46	0.31	0.79	53.05 • 81.76				
JI- 3	32.55 • 93.95	68.91 • 77.18	57.70 • 68.39	0.72	0.85	58.55 • 67.54				
JI- 4	18.58 • 112.68	39.67 • 87.81								
JI- 5	37.31 • 86.97	64.15 • 88.13	59.79 • 74.69	0.64	0.96	58.75 • 73.53				
JI- 6	14.95 • 109.31	65.26 • 58.95								
JI- 7	17.20 • 104.32	39.03 • 89.19	52.96 • 80.56	0.76	0.90	53.86 • 79.64				
JI- 8	24.58 • 95.20	34.61 • 85.07	42.26 • 79.52	1.13	1.96	43.50 • 78.38				
JI- 9	11.18 • 112.66	33.19 • 89.76	34.39 • 88.77	0.61	0.81	35.20 • 87.96				
JI-10	10.88 • 109.49	32.56 • 88.86	32.56 • 86.84	1.16	1.21	33.77 • 85.63				
JI-11		22.76 • 66.19								
JI-12		37.36 • 68.27								
JI-13	28.59 • 108.39	60.19 • 58.88								
JI-14		58.52 • 72.94								
JI-15	53.57 • 79.86	81.37 • 52.86								
JI-16	14.80 • 112.52	39.66 • 85.65								
JI-17	7.60 • 119.45	30.51 • 88.91								
JI-18		59.46 • 69.41								
JI-19		37.57 • 86.72								
JI-20	2.78 • 128.78	50.69 • 73.98								
JI-21	6.88 • 119.88	48.26 • 76.58								
JI-22	58.31 • 67.25	95.59 • 42.35								
JI-23	50.39 • 98.37	58.03 • 73.87								
JI-24	22.72 • 105.66	60.49 • 85.80								
JI-25	28.93 • 102.37	60.53 • 80.97								

Drill Hole Number	No. 3 Seam Roof Depth • Height (m)	No. 3 Seam Thickness Coal (m)	No. 3 Seam Thickness Seam (m)	No. 3 Seam Floor Depth • Height (m)	No. 2 Seam Roof Depth • Height (m)	No. 2 Seam Thickness Coal (m)	No. 2 Seam Thickness Seam (m)	No. 2 Seam Floor Depth • Height (m)
JI- 1					54.67 • 69.86	0.84• Σ 0.84•	1.55 1.55	56.02 • 68.11
JI- 2					61.10 • 65.51	0.68 Σ 0.68	0.84 0.84	61.96 • 64.67
JI- 3					69.29 • 54.86 70.25 • 55.84	0.41 1.35 Σ 1.76	0.58 2.12 2.70	69.87 • 54.22 72.35 • 53.75
JI- 4	61.58 • 65.90	0.32 Σ 0.32	0.64 0.64	62.02 • 65.26	67.05 • 59.63	0.98 Σ 0.98	0.84 0.84	68.93 • 58.37
JI- 5					61.10 • 63.09	1.30 Σ 1.30	2.08 2.08	63.99 • 60.29
JI- 6								
JI- 7	55.57 • 67.95	0.52 Σ 0.52	0.72 0.72	56.29 • 67.23				
JI- 8	54.26 • 65.42	0.62 Σ 0.62	0.62 0.62	54.88 • 64.80	59.73 • 61.95	0.67 Σ 0.67	0.67 0.67	60.40 • 61.28
JI- 9	57.13 • 66.63	2.03 Σ 2.03	2.43 2.43	59.56 • 63.61	61.95 • 61.21	0.46 Σ 0.46	0.55 0.55	62.50 • 60.66
JI-10	56.15 • 73.25	1.00 Σ 1.00	1.00 1.00	57.15 • 72.25	55.26 • 64.18	0.65 Σ 0.65	0.88 0.88	56.16 • 63.20
JI-11								
JI-12								
JI-13								
JI-14					77.64 • 53.84	0.54 Σ 0.54	0.50 0.50	78.24 • 53.28
JI-15	82.41 • 51.12	0.94 Σ 0.94	1.03 1.03	83.44 • 50.09	89.97 • 43.56	0.65 Σ 0.65	0.93 0.93	90.90 • 42.63
JI-16					67.49 • 78.82	1.51 Σ 1.51	2.72 2.72	59.41 • 77.10
JI-17								
JI-18								
JI-19					51.82 • 82.47	0.59 Σ 0.59	1.19 1.19	53.03 • 80.88
JI-20								
JI-21	56.83 • 76.25	1.26 Σ 1.26	1.42 1.42	58.25 • 74.63	55.65 • 68.03	0.59 Σ 0.59	0.99 0.99	56.64 • 67.04
JI-22								
JI-23					65.72 • 67.55	1.84 Σ 1.84	2.76 2.76	68.48 • 64.79
JI-24								
JI-25	95.03 • 34.99 96.51 • 35.81	0.62 0.56 Σ 1.18	0.75 0.92 0.65	95.66 • 35.16 97.35 • 34.60	99.06 • 32.32	0.58 Σ 0.58	0.58 0.58	99.56 • 31.56

Drill Hole Number	No. 1 Seam Roof Depth • Height (m)	No. 1 Seam Thickness Coal (m)	No. 1 Seam Thickness Seam (m)	No. 1 Seam Floor Depth • Height (m)
JI- 1	47.53 • 54.72	0.47• Σ 0.47•	0.93 0.93	48.36 • 57.79
JI- 2	70.99 • 55.71	0.25• Σ 0.25•	0.61 0.61	71.51 • 55.18
JI- 3	81.73 • 44.56 83.06 • 43.03	0.62 1.48 Σ [illegible]	0.67 0.76 2.33	82.48 • 43.69 85.52 • 40.77
JI- 4	72.05 • 55.33	1.90• Σ 1.90•	3.09 3.09	75.16 • 52.18
JI- 5	69.75 • 54.54	0.73 Σ 0.73	2.64 2.64	72.40 • 51.88
JI- 6				
JI- 7	63.54 • 59.58 66.24 • 57.48 69.87 • 53.65	0.55 0.82• 0.32 Σ 2.69•	0.61 2.67 0.35 3.63	64.15 • 59.37 68.71 • 54.81 70.22 • 53.30
JI- 8	63.56 • 58.82 65.77 • 55.91 56.96 • 54.72	0.89 0.50 0.80 Σ 3.20	1.06 0.49 1.06 3.65	64.86 • 57.62 66.30 • 55.34 68.76 • 52.92
JI- 9	65.61 • 57.55	1.69 Σ 1.69	2.33 2.33	67.94 • 55.22
JI-10	61.17 • 58.25 63.33 • 56.07	1.18 0.50 Σ 1.68	0.57 0.67 2.26	62.75 • 56.66 64.00 • 55.48
JI-11				
JI-12				
JI-13				
JI-14	82.68 • 48.84	1.72 Σ 1.72	2.12 2.12	85.00 • 46.52
JI-15	95.98 • 37.55	1.25• Σ 1.25•	1.92 1.92	97.70 • 35.63
JI-16	55.66 • 71.07 57.60 • 69.21	0.83 1.39 Σ 2.22	1.27 1.69 2.76	56.71 • 69.59 58.09 • 67.72
JI-17				
JI-18				
JI-19	51.03 • 72.86	1.98 Σ 1.98	1.22 1.22	52.25 • 71.64
JI-20				
JI-21	63.25 • 59.84	0.50 Σ 0.50	0.51 0.51	63.75 • 59.33
JI-22				
JI-23	78.17 • 63.18	0.72• Σ 0.72•	1.96 1.96	72.13 • 61.11
JI-24	95.77 • 38.61	1.43• Σ 1.43•	1.49 1.49	96.26 • 37.12
JI-25	101.61 • 29.71	1.12 Σ 1.12	0.44 0.44	103.05 • 28.27

Σ: Total

TABLE 17-2 SUMMARY OF COAL SEAMS IN EACH DRILL HOLES (2)

DRILL HOLE NUMBER	BASE OF LIMESTONE Depth / Height (m)	RED ZONE BOUNDARY Depth / Height (m)	NO. 5 SEAM Roof Depth (m)	Roof Height (m)	Thickness Coal (m)	Thickness Seam (m)	Floor Depth (m)	Floor Height (m)	NO. 4 SEAM Roof Depth (m)	Roof Height (m)	Thickness Coal (m)	Thickness Seam (m)	Floor Depth (m)	Floor Height (m)
JT-26	8.93 +113.97													
JT-27		61.72 +74.08												
JT-28	31.37 +100.97	55.52 +83.02												
JT-29	28.50 +105.93	78.87 +55.46												
JT-30	10.10 +115.38	39.82 +101.58												
JT-31	13.00 +117.80	53.02 +77.70												
JT-32	36.52 +95.76													
JT-33	22.65 +121.10	66.85 +96.67												
JT-34	21.22 +116.72	62.19 +75.00												
JT-35	9.00 +120.15	59.88 +110.15												
JT-36	16.00 +120.44	56.71 +79.93												
JT-37	12.60 +109.26	59.90 +81.36												
JT-38		36.65 +97.24												
JT-39		21.65 +102.52												
JT-40	9.00 +126.24	31.90 +103.34												
JT-41	16.38 +117.59	55.64 +80.93												
JT-42		61.65 +85.57												
JT-43	55.00 +126.12	66.86 +34.20												
JT-44	29.39 +110.10	56.53 +85.24												
JT-45		25.22 +99.19												
JT-46	1.50 +115.30	12.40 +104.00												
JT-47														
JT-48		36.01 +77.05												
JT-49		34.06 +75.25												
JT-50														

DRILL HOLE NUMBER	NO. 3 SEAM Roof Depth (m)	Roof Height (m)	Thickness Coal (m)	Thickness Seam (m)	Floor Depth (m)	Floor Height (m)	NO. 2 SEAM Roof Depth (m)	Roof Height (m)	Thickness Coal (m)	Thickness Seam (m)	Floor Depth (m)	Floor Height (m)	NO. 1 SEAM Roof Depth (m)	Roof Height (m)	Thickness Coal (m)	Thickness Seam (m)	Floor Depth (m)	Floor Height (m)
JT-26	104.75	+24.92	0.97 Σ0.97	1.31 1.31	106.06	+23.61	113.26	+15.93	0.27 Σ0.27	0.48 0.48	115.42	+14.25	120.92	+8.75	1.01 Σ1.01	1.30 1.30	122.22	+7.65
JT-27	70.87	+67.45	1.25 Σ1.25	1.30 1.30	71.57	+66.15							84.95	+50.71	1.11 Σ1.11	1.29 1.29	88.27	+49.85
JT-28	54.09	+82.39	0.89 Σ0.89	0.89 0.89	56.98	+81.50												
JT-29	80.75	+53.58	0.50 Σ0.50	0.65 0.65	81.40	+52.93	87.44	+46.89	0.96 Σ0.96	0.61 0.61	88.05	+45.68	89.57	+44.96	1.91 Σ1.91	2.30 2.30	91.75	+42.58
JT-30	83.13	+58.27	0.63 Σ0.63	1.23 1.23	84.36	+55.04	95.01 97.10 98.64	+57.51 +56.31 +54.76	0.39 0.67 0.75 Σ1.81	0.46 0.82 0.43 3.20	96.73 98.21 100.07	+56.67 +55.19 +53.55						
JT-31	87.00	+43.72	0.61 Σ0.61	0.91 0.91	87.91	+42.81	99.23 100.28	+31.57 +30.52	0.37 1.23 Σ1.60	0.54 0.26 1.80	99.77 101.54	+31.03 +29.26						
JT-32																		
JT-33	64.06	+96.67	0.76 Σ0.76	0.95 0.95	65.01	+95.73												
JT-34							64.47	+73.54	1.03 Σ1.03	1.78 1.78	66.25	+71.34	49.56 71.64	+52.59 +66.37	0.64 0.96 Σ1.60	0.83 1.26 2.09	70.29 72.92	+67.76 +65.13
JT-35	67.82	+61.35	1.79 Σ1.79	2.10 2.10	69.91	+67.17	80.62	+54.55	0.75 Σ0.75	0.92 0.92	81.52	+55.63	82.90	+54.17	1.01 Σ1.01	1.51 1.51	84.19	+52.64
JT-36	57.60	+78.84	0.52 Σ0.52	1.40 1.40	59.00	+77.44	73.95	+62.51	1.03 Σ1.03	1.56 1.94	75.67	+60.57						
JT-37													62.96	+59.20	1.01 Σ1.01	1.43 1.43	63.19	+57.72
JT-38	60.21	+67.84	0.44 Σ0.44	0.63 0.63	61.04	+66.25	84.19	+43.96	1.23 Σ1.23	1.54 1.54	85.73	+42.36						
JT-39	64.92	+59.48	1.56 Σ1.56	0.77 0.77	66.69	+57.63	77.46 79.02	+46.06 +45.50	0.76 1.16 Σ1.92	0.96 1.57 2.53	78.42 80.59	+45.90 +43.73						
JT-40	82.88	+52.56	1.45 Σ1.45	1.56 1.56	84.44	+50.00	98.55 100.62	+34.26 +34.62	0.67 1.12 Σ1.79	0.90 1.45 2.35	95.38 112.05	+35.36 +33.29						
JT-41							119.34 121.35	+16.61 +14.71	0.73 3.22+ Σ3.95+	1.20 1.58 2.79	120.63 122.75	+15.32 +13.21						
JT-42							76.85	+58.37	0.83+ Σ0.83+	2.55 2.55	79.40	+57.82						
JT-43	71.15	+69.99	0.94 Σ0.94	0.94 0.94	72.09	+69.05	79.44	+61.58	1.22 Σ1.22	0.65 0.65	81.31	+59.73						
JT-44	83.05 84.85	+56.52 +54.75	1.34 1.93 Σ3.27	1.45 2.34 3.79	84.59 87.18	+55.07 +52.51	89.31	+50.26	1.17 Σ1.17	1.42 1.42	90.73	+48.84						
JT-45	65.79	+54.66	1.01 Σ1.01	1.82 1.82	67.61	+54.84	72.60 73.91 77.50	+51.07 +50.56 +47.07	0.67 1.06 0.87 Σ2.60	0.76 0.10 1.53 3.39	73.36 75.01 78.93	+51.11 +49.14 +45.54						
JT-46	55.60	+61.20	0.42 Σ0.42	1.49 1.49	57.09	+59.79	66.56	+50.52	0.37 Σ0.37	0.59 0.59	66.95	+49.93						
JT-47	65.54	+60.06	0.16 Σ0.16	0.33 0.33	66.01	+58.73	77.04 78.63	+48.40 +47.01	0.56 1.05 Σ1.61	0.86 1.43 2.29	77.90 80.06	+47.71 +45.58						
JT-48							54.59 62.07	+57.95 +52.17	2.27 0.70 Σ2.97	3.10 0.73 3.82	57.70 62.80	+54.76 +51.76						
JT-49	67.23 69.01	+64.84 +63.00	0.90 0.49 Σ1.39	1.20 0.64 1.92	68.51 69.65	+63.56 +62.44	80.00	+51.29	0.63 Σ0.63	0.70 0.70	81.59	+50.59	82.82	+50.07	0.00 Σ0.00	0.00 0.00	82.56	+49.63
JT-50	63.63 65.51	+74.72 +72.84	0.76 0.35 Σ1.11	0.89 0.62 1.51	64.34 66.13	+73.03 +72.04	54.15 52.03	+67.72 +63.54	0.98 0.39 Σ0.52	1.64 0.54 0.58	52.89 53.37	+64.98 +64.00						

Σ: Total

TABLE 17-3 SUMMARY OF COAL SEAMS IN EACH DRILL HOLES (3)

DRILL HOLE NUMBER	BASE OF LIMESTONE Depth / Height (m)	RED ZONE BOUNDARY Depth / Height (m)	NO. 5 SEAM Roof Depth (m)	Roof Height (m)	Thickness Coal (m)	Thickness Seam (m)	Floor Depth (m)	Floor Height (m)	NO. 4 SEAM Roof Depth (m)	Roof Height (m)	Thickness Coal (m)	Thickness Seam (m)	Floor Depth (m)	Floor Height (m)
FS- 1		15.57 +141.65							51.99	+68.28	0.99	0.99	52.98	+67.29
FS- 2		19.80 +107.88												
FS- 4		28.10 +93.29												
FS- 5		18.50 +91.15												
FS- 6	53.51 +140.86	47.33 +87.02												
FS- 7	18.88 +118.01	58.53 +76.50												
FS- 8	4.10 +128.10	23.92 +108.93							63.11	+69.75	0.79	0.79	63.90	+68.96
FS- 9	12.79 +128.98	55.65 +85.93												
FS-10		29.91 +77.17												
FS-11	23.08 +53.79	66.59 +72.28												
FS-13	36.23 +116.91	39.81 +103.35												
FS-14	5.33 +138.86	38.25 +107.92							47.95	+96.22	1.07	1.07	49.02	+95.15
FS-15	10.67 +132.73	51.80 +81.60												
FS-18		59.35 +67.55												
FS-19	12.18 +109.16	37.40 +83.86	50.29	+80.35	0.94	0.94	51.23	+79.41						
FS-20	20.61 +97.82	37.50 +80.20	48.20	+85.60	0.56	0.56	48.76	+85.04	44.97	+69.91	0.64	0.64	45.61	+69.27
FS-22	35.59 +91.97	38.25 +88.92	49.24	+77.92	0.58	0.58	49.82	+77.34						
FS-23	18.00 +89.30	70.21 +87.98												
FS-24	12.66 +90.16	54.18 +78.42												
L -18														
L -22														
L -20	26.91 +150.18													

DRILL HOLE NUMBER	NO. 3 SEAM Roof Depth (m)	Roof Height (m)	Thickness Coal (m)	Thickness Seam (m)	Floor Depth (m)	Floor Height (m)	NO. 2 SEAM Roof Depth (m)	Roof Height (m)	Thickness Coal (m)	Thickness Seam (m)	Floor Depth (m)	Floor Height (m)
FS- 1	60.50	+59.60	2.01 Σ 2.01	2.01 2.01	62.51	+57.59	75.51	+44.67	1.20 Σ 1.20	1.20 1.20	76.71	+43.47
FS- 2	65.34	+61.54	0.51 Σ 0.51	0.51 0.51	65.85	+61.03	73.48	+53.40	1.29 Σ 1.29	1.29 1.29	74.77	+52.11
FS- 4							64.47	+59.66	0.96 Σ 0.96	0.96 0.96	65.43	+58.70
FS- 5	50.06	+64.57	2.56 Σ 2.56	2.56 2.56	52.62	+62.01	58.29 60.39	+58.34 +56.54	1.11 1.10 Σ 2.21	1.11 1.10 3.20	59.40 61.49	+57.23 +55.44
FS- 6	73.18	+41.17	0.61 Σ 0.61	0.61 0.61	73.79	+40.56	79.22	+55.13	0.79 Σ 0.79	0.79 0.79	80.01	+54.34
FS- 7												
FS- 8	68.55 70.43	+64.36 +62.35	1.70 0.84 Σ 2.54	1.70 0.84 2.54	70.25 71.19	+62.61 +61.37	84.73	+48.13	0.81 Σ 0.81	0.81 0.81	85.54	+47.32
FS- 9	81.36	+58.62	0.76 Σ 0.76	0.76 0.76	82.12	+57.86	92.02	+47.76	1.62 Σ 1.62	1.62 1.62	93.64	+46.14
FS-10							91.11	+15.96	2.26 Σ 2.26	2.26 2.26	93.37	+13.70
FS-11												
FS-13	99.17	+53.58	0.56 Σ 0.56	0.56 0.56	99.73	+53.02	98.53	+54.10	0.65 Σ 0.65	0.81 0.81	99.34	+53.29
FS-14	58.82	+85.35	0.76 Σ 0.76	0.76 0.76	59.58	+84.59						
FS-15												
FS-18												
FS-19	56.45	+64.89	0.84 Σ 0.84	0.84 0.84	57.29	+64.05	57.84	+61.50	2.57 Σ 2.57	2.82 2.82	60.66	+58.68
FS-20							49.29 53.73	+76.59 +72.15	2.94 0.53 Σ 3.47	2.94 0.53 3.47	52.23 54.26	+73.65 +71.62
FS-22							58.20	+66.96	2.06 Σ 2.06	2.23 2.23	60.43	+64.73
FS-23							73.18 77.80	+64.27 +60.31	2.84 0.76 Σ 3.60	2.96 0.76 3.72	76.14 78.56	+61.31 +59.55
FS-24							70.17 72.87	+61.83 +59.93	0.71 0.97 Σ 1.68	0.71 0.97 1.68	70.88 73.84	+61.12 +58.96
L -18												
L -22												
L -20												

DRILL HOLE NUMBER	NO. 1 SEAM Roof Depth (m)	Roof Height (m)	Thickness Coal (m)	Thickness Seam (m)	Floor Depth (m)	Floor Height (m)
FS- 1						
FS- 2						
FS- 4						
FS- 5						
FS- 6	87.98 90.08	+46.37 +44.27	0.38 0.55 Σ 0.93	0.60 0.69 2.29	88.58 90.77	+45.77 +43.58
FS- 7	90.18 93.41	+45.44 +43.51	2.18 0.35 Σ 2.23	2.22 0.52 2.74	92.40 94.53	+42.24 +39.99
FS- 8						
FS- 9						
FS-10						
FS-11						
FS-13						
FS-14						
FS-15						
FS-18	71.13	+55.77	1.83 Σ 1.83	1.96 1.96	73.09	+53.81
FS-19	63.27 64.65 65.65	+58.07 +56.69 +55.69	0.50 0.50 1.48 Σ 2.48	0.50 0.50 1.48 2.48	63.77 65.15 67.13	+57.57 +55.99 +54.21
FS-20	65.79	+59.69	1.64 Σ 1.64	1.64 1.64	67.43	+58.05
FS-22						
FS-23	85.77	+52.34	2.28 Σ 2.28	2.28 2.28	88.05	+50.06
FS-24	88.21	+54.59	0.75 Σ 0.75	0.75 0.75	88.96	+53.84
L -18						
L -22						
L -20						

Σ; Total

Ultimate analysis indicates carbon 68 percent, oxygen 16.1 percent and total sulphur 6.98 percent on dry ash free basis. On the basis of above analytical data, Lakhra coal is classified into lignite with low coalification. Therefore, this coal is easily oxidized and liable to spontaneous combustion. Besides the content of pyrite in total sulphur is high ranging from 50–60 percent, and its presence activates the spontaneous combustion. As a result of the oxygen absorption test it has been determined that the K value, the absorption speed of this coal is 1.21 (O_2 generation 0.208 cc/gram/hour). The K value is much higher than that of general sub-bituminous coals and it is considered that this coal is combustible spontaneously. The coal also gives 0.0395 cc/gram/hour CO_2, but it is difficult to judge the spontaneous combustibility from the amount of CO_2 only. During the tests under temperature conducted in a vacuum flask, it is noted that the temperature rises 14.6°C at a condition of inherent moisture of 9.2 percent. This shows higher temperature rise than other coals under the same condition, thus has much possibility for spontaneous combustion. After drying the coal sample to 3–5 percent of inherent moisture the temperature of coal rises upto 25–30°C. Consequently the coal will be liable to combustion.

The thermal conductivity of coal is 0.131 kcal/m.hr.°C at 9.2 percent of inherent moisture. It is generally considered that when thermal conductivity of coal lowers, the coal gives off some heat from the latent heat and it becomes combustible finally. When the coal is dried to 3–5 percent inherent moisture, its thermal conductivity decreases to more dangerous range of 0.07–0.09 kcal/m.hr.°C.

The relative ignition temperature of this coal is low i.e. 172.3°C. In general there is a relation between relative ignition temperature and carbon or oxygen content, and the temperature is low for the coals low in rank and coalification, but this cannot be applied in all the cases, because it is sometimes impossible to judge the spontaneous combustibility only from the relative ignition temperature.

On the basis of above tests for spontaneous combustibility it has been proved that Lakhra coal is liable to spontaneous combustion and therefore some countermeasures should be taken.

4-5 Coal Reserves

As mentioned above the investigated area is divided into three blocks – the western, central and eastern blocks also referred as parts in the report. Mining methods for each block will be decided by mining feasibility study and on the basis of coal reserves. At this stage the reserves have been estimated on the basis of following factors and assumptions:–

a. Thickness of coal seams,
b. Number of coal seams,
c. Condition of roof, and
d. Strip ratio.

The western and eastern blocks can be mined by open-cut method, and the central by both open-cut in northern area and underground mining in southern area or entire block. Reserves of coal in the eastern block have also been calculated for underground mining only for future reference.

4-5-1 Coal Zone and Seam Thickness, Specific Gravity and Strip Ratio

(1) Coal Zone and Seam Thickness

A coal zone consists of either one coal seam or more interbedded with one or more partings. Its thickness is an aggregate of the thickness of coal seam(s), and total thickness of impure coal and interbedded rock layers.

Calculation of reserves is done by taking into account seam thickness including parting. At this stage it is presumed that the standard grade of coal to be mined by open-cut method would be required to have ash content under 35 percent, and calorific value over 3,500 kcal/kg on air dried basis. Minimum workable seam thickness is assumed over 0.5 metre. The partings over 0.3 metre would be removed at the time of stripping. Therefore, such partings are omitted from the thickness of minable seams.

In the underground mining area, the reserves have been calculated for coal seams over 0.75 metre. However, thin coal seams ranging in thickness from 0.5 to 0.75 metre have also been considered for the calculation of total reserves only for reference.

The standard coal grade for underground mining is same as in case of open-cut mining.

(2) Specific Gravity

Apparent specific gravity of all the 221 samples has been measured. In calculation of coal reserve specific gravity of group seams, which can be mined together could be determined for each sample, but in this case specific gravity of each seam or group of seams will be different and in future it will create problem in calculations if the sample is not subjected to studies. Therefore, for open-cut mining the average specific gravity determined for each block as under is suggested to be considered:—

Western block	1.56
Central block	1.54
Eastern block	1.49

It is also suggested that some safety coefficient may be taken. The standard specific gravity taken for calculation of coal reserves, and for estimation of strip ratio is 1.50.

In the same way for underground mining the average specific gravity is 1.43 for the eastern block, and 1.50 for the central block. The standard specific gravity for both the area is taken as 1.45

(3) Strip Ratio

Strip ratio is the amount of waste (m^3) that must be removed to gain access to unit weight (metric ton) of coal. Economically, strip ratio should be under 1:10 although it depends on the natural conditions. The economical limit of strip ratio in the investigated area will be examined in mining feasibility study.

4-5-2 Method of Calculation of Coal Reserves

Theoretical coal reserves = Seam thickness x specific gravity x existing area

Recoverable coal reserves = Theoretical coal reserve x geologic safety factor x recovery

Presently theoretical coal reserves are calculated by computer. Geologic safety factor is based on accuracy of the investigations and geological conditions such as faults, fluctuation in thickness of seams, dip and continuity of coal seam, etc. In underground mining safety factor of a coal seam surrounded by gallery would be 100 percent, while in newly developed area, such as investigated area, it is 60 to 90 percent generally.

Coal recovery depends on artificial conditions such as mining technique, working method etc. These conditions will be determined in the mining feasibility study. However, generally it is from 50 to 90 percent.

If coal is to be washed, the reserves will be revised as the working height, and specific gravity will change. Whether coal should be washed or not will be investigated alongwith washing cost in the mining feasibility study.

4-5-3 Coal Reserves for Open-cut Mining

Coal reserves for open-cut mining are calculated separately for the western, central (northern side), and eastern blocks. The boundary of the western and central blocks is barren area and fault A, whereas the boundary of central and eastern block is fault B1, and boundary for that of northern and southern parts in the central block is given artificially using the limit line of 1:20 strip ratio.

Open cut-mining is suggested for these areas due to the following reasons.

(1) In the western block there are many coal seams having thickness over 0.5 metre and suitable strip ratio.

(2) In the eastern block underground mining is not advisable, because loose sandstone forms roof of coal seams.

(3) The extension of coal seam is limited, and multiple seam mining is impossible, because the extension area of each seam is different. Therefore in spite of higher strip ratio than in western block, the coal reserves in eastern block are calculated for open-cut mining as well as underground mining only for further references in future.

(4) The central block can be the underground mining area, because No. 1 zone here enjoys favourable roof conditions. However, its northern part, where the strip ratio is under 1:20, has been evaluated for open-cut mining. The selection of mining method will be made during the mining feasibility study.

Coal reserves for open-cut mining are calculated on the basis of the strip ratios of 1:10 and

Table 18

ROW COAL RESERVES FOR OPEN CUT MINING

(in metric ton)

Specific Gravity 1.5

STRIP RATIO	DESCRIPTION		THE WEST	THE CENTRAL	THE EAST	TOTAL
1 : 10 and less (The West)	Area	10^3m^2	1,744			1,744
	Average Total Thickness	m	6.30			6.30
	Theoretical Reserves	10^3t	16,487			16,487
	Recovery Percentage	%	72			72
	Recoverable Reserves	10^3t	11,871			11,871
	Bank Overburden Volume	10^3m^3	119,358			119,358
	Average Strip Ratio		7.23			7.23
1 : 12 and less (The East)	Area	10^3m^2			3,103	3,103
	Average Total Thickness	m			4.54	4.54
	Theoretical Reserves	10^3t			21,139	21,139
	Recovery Percentage	%			72	72
	Recoverable Reserves	10^3t			15,220	15,220
	Bank Overburden Volume	10^3m^3			221,916	221,916
	Average Strip Ratio				10.50	10.50
1:10 to 1:15 for the West	Area	10^3m^2	609		3,714	4,323
	Average Total Thickness	m	5.10		3.79	3.98
	Theoretical Reserves	10^3t	4,660		21,139	25,799
1:12 to 1:15 for the East	Recovery Percentage	%	72		72	72
	Recoverable Reserves	10^3t	3,355		16,713	20,068
	Bank Overburden Volume	10^3m^3	54,831		298,007	352,838
	Average Strip Ratio		11.77		14.10	13.68
1 : 17 and less (The Central)	Area	10^3m^2		1,874		1,874
	Average Total Thickness	m		4.27		4.27
	Theoretical Reserves	10^3t		11,990		11,990
	Recovery Percentage	%		72		72
	Recoverable Reserves	10^3t		8,633		8,633
	Bank Overburden Volume	10^3m^3		174,509		174,509
	Average Strip Ratio			14.55		14.55
1:17 to 1:20 The Central	Area	10^3m^2		1,111	3,039	4,150
	Average Total Thickness	m		3.76	3.63	3.66
	Theoretical Reserves	10^3t		6,270	16,536	22,806
1:15 to 1:20 The East	Recovery Percentage	%		72	72	72
	Recoverable Reserves	10^3t		4,514	11,906	16,420
	Bank Overburden Volume	10^3m^3		111,745	273,804	385,549
	Average Strip Ratio			17.82	16.56	16.90
Total	Area	10^3m^2	2,353	2,985	9,856	15,194
	Average Total Thickness	m	5.99	4.08	4.12	4.40
	Theoretical Reserves	10^3t	21,147	18,260	60,887	100,294
	Recovery Percentage	%	72	72	72	72
	Recoverable Reserves	10^3t	15,226	13,147	43,839	72,212
	Bank Overburden Volume	10^3m^3	174,189	286,254	793,727	1,254,170
	Average Strip Ratio		8.24	15.67	13.04	12.50

1:15 in the western block, 1:17 and 1:20 in the central, and 1:12, 1:15 and 1:20 in the eastern block. The results are shown in Table 18.

Open-cut mining is not so restricted by natural conditions as underground mining. Safety coefficient and recovery in case of open-cut mining are presumed to be 80 percent and 90 percent respectively.

4-5-4 Coal Reserves in Underground Mining (Table 19)

Underground mining is not so restricted by the depth of coal seam as in open-cut mining. On the other hand it is much restricted by roof and floor conditions, continuity of seams, geologic structure including faults, etc.

During the present investigations it is found that in some parts of the investigated area medium to fine grained loose sandstone ranging in thickness from 1 to 10 metres occurs as roof rock.

It is possible that in underground mining, mechanized long wall mining method be employed for mass production, but in case of above mentioned condition it will be difficult to control loose sandstone in roof, and it may cause problems.

It is, therefore, suggested that underground mining in such areas should be avoided if possible.

Keeping this in view investigations about the roof were carried out throughout the investigated area, and it has been confirmed that in the eastern block no coal seam could be mined extensively, but the lower parts of No. 1, No. 2 and No. 3 zones are minable over fairly large areas. The upper part of the zones is minable over a small area in the north of the block. The coal seams do not overlap except in this north-eastern corner. Therefore, underground mining may be difficult in the eastern block, but because of rather high strip ratio for open-cut mining, the reserves have also been calculated for underground mining only for the purpose of future reference.

The central block is also influenced by loose sandstone. No. 2 zone is not minable in the entire block, whereas No. 1 zone is minable extensively, and No. 3 zone is partially minable. Therefore, the reserves are calculated in the central block for two purposes. First, for mining the entire block by underground method, and the second to mine northern part by open-cut and southern area by underground methods.

Reserves for underground mining are detailed in Table 19.

When open-cut mining will be applied to the eastern block, and the northern part of the central block, then only the southern extension of No. 1 zone may be the subject of underground mining. Coal reserves in this case are shown in Table 20.

Table 19

ROW COAL RESERVES FOR UNDERGROUND MINING

(in metric ton)

Specific Gravity 1.45

		DESCRIPTION		WORKABLE THICKNESS (metres) 3.5 to 3.0	3.0 to 2.5	2.5 to 2.0	2.0 to 1.5	1.5 to 1.0	1.0 to 0.75	0.75 to 0.5	TOTAL
The Central	No.1 & No.2	Area	(10^3m^2)	120	499	1,128	1,803	922	112	70	4,654
		Average Workable Thickness	(m)	3.18	2.70	2.22	1.76	1.37	0.88	0.63	1.89
		Theoretical Reserves	(10^3t)	554	1,956	3,623	4,595	1,830	143	64	12,765
		Recovery Percentage	(%)	45.5	45.5	45.5	45.5	45.5	45.5	45.5	45.5
		Recoverable Reserves	(10^3t)	252	890	1,648	2,091	833	65	29	5,808
	No.3	Area	(10^3m^2)					318	815	140	1,273
		Average Workable Thickness	(m)					1.11	0.91	0.67	0.93
		Theoretical Reserves	(10^3t)					510	1,073	135	1,718
		Recovery Percentage	(%)					45.5	45.5	45.5	45.5
		Recoverable Reserves	(10^3t)					232	488	61	781
	Total	Area	(10^3m^2)	120	499	1,128	1,803	1,240	927	210	5,927
		Average Workable Thickness	(m)	3.18	2.70	2.22	1.76	1.30	0.90	0.65	1.69
		Theoretical Reserves	(10^3t)	554	1,956	3,623	4,595	2,340	1,216	199	14,483
		Recovery Percentage	(%)	45.5	45.5	45.5	45.5	45.5	45.5	45.5	45.5
		Recoverable Reserves	(10^3t)	252	890	1,648	2,091	1,065	553	90	6,589
The East	No.1 & No.2	Area	(10^3m^2)		70	354	1,088	1,621	166	22	3,321
		Average Workable Thickness	(m)		2.63	2.21	1.71	1.32	0.93	0.66	1.54
		Theoretical Reserves	(10^3t)		267	1,134	2,691	3,095	223	21	7,431
		Recovery Percentage	(%)		45.5	45.5	45.5	45.5	45.5	45.5	45.5
		Recoverable Reserves	(10^3t)		121	516	1,225	1,408	102	10	3,382
	No.3	Area	(10^3m^2)		278	888	1,373	185	110	65	2,899
		Average Workable Thickness	(m)		2.65	2.22	1.71	1.31	0.91	0.63	1.88
		Theoretical Reserves	(10^3t)		1,067	2,858	3,407	351	145	59	7,887
		Recovery Percentage	(%)		45.5	45.5	45.5	45.5	45.5	45.5	45.5
		Recoverable Reserves	(10^3t)		485	1,300	1,550	160	66	27	3,588
	Total	Area	(10^3m^2)		348	1,242	2,461	1,806	276	87	6,220
		Average Workable Thickness	(m)		2.64	2.22	1.71	1.32	0.92	0.63	1.70
		Theoretical Reserves	(10^3t)		1,334	3,992	6,098	3,446	368	80	15,318
		Recovery Percentage	(%)		45.5	45.5	45.5	45.5	45.5	45.5	45.5
		Recoverable Reserves	(10^3t)		606	1,816	2,775	1,568	168	37	6,970
Sum of The Central and The West		Area	(10^3m^2)	120	847	2,370	4,264	3,046	1,203	297	12,147
		Average Workable Thickness	(m)	3.18	2.68	2.22	1.73	1.31	0.91	0.65	1.69
		Theoretical Reserves	(10^3t)	554	3,290	7,615	10,693	5,786	1,584	279	29,801
		Recovery Percentage	(%)	45.5	45.5	45.5	45.5	45.5	45.5	45.5	45.5
		Recoverable Reserves	(10^3t)	252	1,496	3,463	4,866	2,633	721	127	13,559

Table 20. Coal Reserves for Underground Mining
(Southern part of central block)

Block	Description	Workable Thickness (m)					
		3.0-2.5	2.5-2.0	2.0-1.5	1.5-1.0	1.0-0.75	0.75-0.50
Central	Area ($10^3 m^2$)	130	469	1,102	1,051	123	64
	Average Workable Thickness	2.63	2.21	1.70	1.36	0.88	0.63
	Theoretical Reserve (10^3 t)	496	1,505	2,711	2,067	158	58
	Safety factor x Recovery (%)	70x65	70x65	70x65	70x65	70x65	70x65
	Recoverable Reserve (10^3 t)	226	685	1,234	940	72	26
	Total Theoretical Reserve (10^3 t)	496	2,001	4,712	6,779	6,937	6,995
	Total Recoverable Reserve (10^3 t)	226	911	2,145	3,085	3,157	3,183

Underground mining as against the open-cut mining is affected by faults. Moreover, underground mining is controlled by condition of roof and continuity of coal seam. Therefore, recoverable reserves are calculated by assuming geologic safety factor at 70 percent and recovery at 65 percent.

4-5-5 Summary of Coal Reserves

Coal reserves for open-cut and underground mining areas are calculated separately, but in the central and eastern blocks both the areas are overlapping. Therefore, suitable mining method will be chosen during the mining feasibility study, and at this stage it is difficult to state the coal reserves definitely.

Reserves under discussion have been calculated by assuming mining method, and combination of open-cut and underground mining. However, these reserves may be changed by mining cost, strip ratio and working height which will be calculated during the mining feasibility study.

Reserves calculated on the basis that the strip ratio is under 1:10 in case of open-cut mining, and working height is over 1.5 metres in underground mining are detailed in Table 21.

Table 21 Summary of coal reserves (1)

For Open Cut Mining		For Underground Mining		Total	
Western block		Central and eastern blocks			
Theo. * Res. *** (10^3 t)	Rec. ** Res. (10^3 t)	Theo. Res. (10^3 t)	Rec. Res. (10^3 t)	Theo. Res. (10^3 t)	Rec. Res. (10^3 t)
16,487	11,871	22,152	10,078	38,639	21,949

* Theoretical
** Recoverable
*** Reserve

These coal reserves will not be sufficient for the operation of power plant upto 30 years.

Coal reserves calculated on the basis of strip ratio under 1:15, and working height over 0.75 metre in underground mining are shown in Table 22.

Table 22 Summary of coal reserves (2)

For open cut mining		For underground mining		Total	
The eastern & western blocks		Central block			
Theo. * Res. ** (10^3 t)	Rec. *** Res. (10^3 t)	Theo. Res. (10^3 t)	Rec. Res. (10^3 t)	Theo. Res. (10^3 t)	Rec. Res. (10^3 t)
65,498	47,159	14,284	6,499	79,782	53,658

* Theoretical ** Reserve *** Recoverable

These reserves will be sufficient for the operation of power plant.

The reserves as detailed in Table 23 have also been calculated on the condition that strip ratio is under 1:20, and the working height is over 0.75 metre in the southern part of central block, which can be mined only by underground method.

Table 23 Reserves in case of strip ratio upto 1:20

For open-cut mining		For underground mining		Total	
The western, eastern and northern half of central block		Southern half of central block			
Theo. * Res. *** (10^3 t)	Rec. ** Res. (10^3 t)	Theo. Res. (10^3 t)	Rec. Res. (10^3 t)	Theo. Res. (10^3 t)	Rec. Res. (10^3 t)
100,294	72,212	6,937	3,157	107,231	75,369

* Theoretical
** Recoverable
*** Reserve

Among the above calculated reserves under three conditions the theoretical reserves are 79,782,000 tons and the recoverable reserves are 53,658,000 tons as shown in Table No. 22. They appear to be reasonable and will be examined again during the mining feasibility study.

CHAPTER 5 LABORATORY TESTS ON ROCK SAMPLES

Fifty rock samples were collected from the cores of 14 holes, and examined in the laboratory. The tests conducted are, supersonic velocity (P & S wave), uniaxial compressive strength, tensile strength, Poisson's ratio, static modulus of elasticity, specific gravity, coefficient of water absorption, effective porosity, and stability test for aggregate.

Results of rock tests are summarized in Table 24, data in respect of each stratigraphic unit are detailed in Table 25, and the relationships amongst the data regarding each rock type obtained through different tests are given in Figs. 20–24. Following is a brief explanation regarding the methods of each rock test:–

5-1 Supersonic Velocity

Propogation velocity of supersonic wave in a rock is measured in this test. Generally, supersonic velocity is in proportion to compressive strength and much influenced by fractures and cracks in rocks. Therefore, the extent of fracturing and cracks in a solid rock can be measured by comparing the supersonic velocity of the rock samples in laboratory, and results of elastic wave investigation of layer represented by samples in the field.

5-2 Uniaxial Compressive Strength and Tensile Strength

Both tests measure yielding or breaking point of rock under pressure. Generally tensile strength is less than the compressive strength by 1/10 – 1/20. It was very difficult to prepare the samples for tensile test. Therefore radial compression test was performed. In this test, as shown in Fig. 25, the samples are compressed along diameter. Measured values of the test usually differ from the tensile strength by ±20 to 30 percent.

5-3 Specific Gravity (Weight per Unit Volume), Coefficient of Water Absorption, and Effective Porosity

Some of the samples crumbled when soaked in water for the tests. Therefore, the specific gravity was measured by Caliper method, which involves calculation of volume of cylindrical sample by measuring diameter, three times each, in lower, middle, and upper parts. Average diameter is then determined. Usually the specific gravity is calculated from the difference between the weights in air and water.

5-4 General Results of Tests

(a) Laki limestones

Pure limestone widely exposed in the investigated area show high strength in each test. But chalk and sandy limestone or marl show lower strength similar to strata in Ranikot group.

(b) Upper Shell Beds and Upper Coal-bearing Beds

The Beds underlie Laki group and need due consideration from mining point of view as the workable coal in the investigated area is borne by Upper Coal-bearing Beds. Except the siliceous and hard parts of sandstones, compressive strengths of all the strata are measured

Table 24 - 1

DATA OF ROCK TESTS

DRILL HOLE	SAMPLE NUMBER	LOCATION (Depth) (m)	ROCK	CONDITION OF MEASUREMENT	SUPERSONIC VELOCITY P WAVE (m/sec)	SUPERSONIC VELOCITY S WAVE (m/sec)	DYNAMIC POISSON'S RATIO	DYNAMIC MODULUS OF ELASTICITY (Kg/cm2)	UNIAXIAL COMPRESSIVE STRENGTH (Kg/cm2)	TENSILE STRENGTH (Kg/cm2)	POISSON'S RATIO	STATIC MODULUS OF ELASTICITY (Kg/cm2)	SPECIFIC GRAVITY (Natural State)	COEFFICIENT OF WATER ABSORPTION (%)	EFFECTIVE POROCITY (%)	STABILITY TEST FOR AGGREGATE (%)	NOTE
JT-1	R-4 (1)	10.0	Limestone	Natural	3780	1970	0.31	2.48×10^5	141.9	36.7	0.27	9.24×10^4	2.39	4.42	10.36	52.2	
JT-1	R-4 (2)	10.0	"	"	4340	2070	0.35	2.85×10^5	434.9	-	0.25	1.55×10^5					
JT-1	R-5	23.0	Chalky Limestone	"	2960	1470	0.33	1.29×10^5	142.1	11.4	0.26	9.60×10^4	2.11	14.17	26.53	63.1	
JT-1	R-6 (1)	33.6	Limestone	"	4180	1990	0.35	2.56×10^5	366.1	30.5	0.33	8.03×10^4	2.36	7.10	15.84	84.3	
JT-1	R-6 (2)	33.5	"	"	4320	2070	0.35	2.90×10^5	667.3	-	0.29	2.02×10^5					
JT-3	R-1	22.0	Chalky Limestone	"	2840	1610	0.26	1.25×10^5	122.5	6.1	0.27	1.49×10^5	1.93	49.76	57.61	100.0	
JT-3	R-2 (1)	28.5	Limestone	"	4730	2290	0.36	3.65×10^5	957.3	53.7	0.28	3.83×10^5	2.51	2.37	5.99	49.3	
JT-3	R-2 (2)	28.5	"	"	3920	2000	0.33	2.73×10^5	574.1	-	-	-					
JT-3	R-3	38.5	Very Fine Limestone	"	1240	690	0.27	2.32×10^4	63.5	0.3	0.15	1.07×10^4	1.82	60.66	60.98	100.0	Sample from Red Zone
JT-3	R-5	57.0	"	"	1040	520	0.33	1.43×10^4	37.1	14.5	0.19	5.30×10^3	1.80	58.09	58.67	100.0	
JT-3	R-6	60.0	Fine Sandstone	"	1010	-	-	-	5.2	-	-	5.29×10^2	1.76	17.92	28.56	100.0	
JT-3	R-7	33.0	Conglomerate	"	2590	1360	0.31	1.15×10^5	127.5	-	0.24	3.39×10^4	2.33	14.48	29.13	100.0	ditto
JT-3	R-8	35.0	Sandy Claystone	"	1080	640	0.33	1.57×10^4	98.5	3.9	0.21	1.03×10^4	1.94	43.47	52.83	100.0	ditto
JT-5	R-1 (1)	37.0	Foraminifera Limestone	"	4870	2750	0.27	4.82×10^5	933.6	40.8	0.32	3.89×10^5	2.46	5.45	12.99	6.3	
JT-5	R-1 (2)	37.0	"	"	4790	2780	0.25	4.79×10^5	511.8	-	0.27	4.27×10^5					
JT-5	R-2 (1)	40.5	Medium to Fine Sandstone	"	4040	2290	0.26	3.22×10^5	320.7	11.8	0.21	4.58×10^5	2.39	8.71	20.63	65.8	ditto
JT-5	R-2 (2)	40.5		"	4090	2200	0.27	3.24×10^5	311.6	-	0.22	3.67×10^5					ditto
JT-5	R-3	46.0	Siltstone	"	3190	1920	0.22	2.48×10^5	312.6	10.4	0.07	1.86×10^5	2.43	32.81	48.58	100.0	
JT-5	R-5	64.5	Claystone	"	1370	630	0.35	2.18×10^4	89.7	3.9	0.32	2.36×10^4	1.99	45.40	55.30	100.0	
JT-5	R-5'	65.5	Very Fine Sandstone	"	2950	1600	0.29	1.34×10^5	-	13.2	-	-	1.98	14.71	27.26	100.0	
JT-7	R-1 (1)	25.5	Very Fine Sandstone	"	1550	640	0.40	2.18×10^4	22.7	5.3	0.12	2.18×10^4	1.79	21.60	35.4	100.0	ditto
JT-7	R-1 (2)	25.5	"	"	1350	700	0.31	2.42×10^4	38.7	-	0.18	4.35×10^4					ditto
JT-7	R-3	49.5	(Silty) Fine Sandstone	"	1370	-	-	-	59.6	4.6	0.23	1.30×10^4	2.04	41.2	53.9	100.0	
JT-7	R-5	62.0	"	"	1500	-	-	-	46.8	5.9	0.20	1.95×10^4	2.06	44.5	55.0	100.0	
JT-8	R-1 (1)	30.0	Fine Sandstone	"	2910	1580	0.29	1.44×10^5	190.1	19.1	0.08	1.09×10^5	2.20	14.0	25.6	80.3	ditto
JT-8	R-1 (2)	30.0		"	4160	2420	0.24	3.63×10^5	323.0	-	0.21	3.13×10^5					ditto
JT-8	R-3	49.0	Very Fine Sandstone	"	1500	-	-	-	57.9	4.2	0.12	1.68×10^4	2.02	34.8	49.9	100.0	
JT-10	R-2	36.0	Siltstone	"	1030	540	0.31	1.49×10^4	85.6	5.7	0.18	9.40×10^3	1.88	72.3	65.2	100.0	

Table 24 - 2

DRILL HOLE	SAMPLE NUMBER	LOCATION (Depth) (m)	ROCK	CONDITION OF MEASUREMENT	SUPERSONIC VELOCITY P WAVE (m/sec)	SUPERSONIC VELOCITY S WAVE (m/sec)	DYNAMIC POISSON'S RATIO	DYNAMIC MODULUS OF ELASTICITY (Kg/cm^2)	UNIAXIAL COMPRESSIVE STRENGTH (Kg/cm^2)	TENSILE STRENGTH (Kg/cm^2)	POISSON'S RATIO	STATIC MODULUS OF ELASTICITY (Kg/cm^2)	SPECIFIC GRAVITY (Natural State)	COEFFICIENT OF WATER ABSORPTION (%)	EFFECTIVE POROCITY (%)	STABILITY TEST FOR AGGREGATE (%)	NOTE
JT-15	R-2	57.5	Siltstone	Natural	1080	570	0.31	1.44×10^4	43.0	6.9	0.17	6.00×10^3	1.79	24.15	39.45	100.0	Sample from Red Zone
JT-15	R-3	63.3	Fine to Midium Sandstone	"	3380	1760	0.31	1.91×10^5	260.1	24.5	0.21	3.61×10^5	2.32	9.04	19.96	55.5	ditto
JT-22	R-1 (1)	26.7	Limestone	"	4920	2880	0.24	5.40×10^5	448.1	46.2	0.21	3.30×10^5	2.59	1.35	3.53	5.1	
JT-22	R-1 (2)	26.7	"	"	5250	2790	0.30	5.32×10^5	870.2	-	0.24	4.75×10^5					
JT-22	R-3	44.8	Sandy Limestone	"	2650	1580	0.22	1.42×10^5	125.2	-	0.16	1.07×10^5	2.26			100.0	
JT-22	R-4	47.1	Limestone	"	5140	2790	0.29	5.26×10^5	757.6	70.5	0.21	2.65×10^5	2.53	1.99	5.05	37.0	
JT-23	R-2	38.5	Fine Sandstone	"	2350	1380	0.23	8.83×10^4	25.6	14.6	0.28	8.00×10^3	1.70	34.61	46.61	100.0	ditto
JT-23	R-2'	38.8	"	"	2710	1500	0.28	1.13×10^4	89.1	10.1	0.16	2.23×10^5	1.81			100.0	ditto
JT-27	R-1 (1)	44.1	Medium to Coarse Sandstone	"	4890	2620	0.30	5.06×10^5	1142.3	89.0	0.37	4.50×10^5	2.65	2.45	6.74	48.0	ditto
JT-27	R-1 (2)	44.1	"	"	3290	1570	0.35	1.66×10^5	155.8	-	0.05	7.79×10^4					ditto
JT-27	R-3 (1)	46.0	Fine Sandstone	"	2280	1330	0.24	9.83×10^4	132.6	4.0	0.10	7.36×10^4	2.07	15.69	29.50	73.8	ditto
JT-27	R-3 (2)	46.0	"	"	2440	1350	0.28	9.61×10^4	111.9	-	0.12	8.61×10^4					ditto
JT-37	R-4	59.8	Siltstone	"	1090	-	-	-	9.2	4.2	0.09	4.80×10^3	1.95	62.08	60.17	100.0	ditto
JT-37	R-6 (1)	83.7	(Silliceous) Medium Sandstone	"	5470	2900	0.30	6.03×10^5	1790.4	67.2	0.22	6.58×10^5	2.60	2.07	5.20	0.3	
JT-37	R-6 (2)	83.7	"	"	4980	2640	0.30	4.02×10^5	949.6	-	0.19	6.18×10^5					
JT-39	R-4	63.0	Siltstone	"	1390	840	0.21	3.64×10^4	141.3	13.1	0.19	1.47×10^4	2.04	37.02	49.62	100.0	
JT-39	R-5	68.0	"	"	1090	470	0.39	1.31×10^4	69.9	10.4	0.14	8.50×10^3	2.07	37.06	48.78	100.0	
JT-39	R-6	75.5	Fine Sandstone	"	1820	930	0.32	5.01×10^4	-	9.5	-	-	2.11	41.19	51.17	100.0	
JT-39	R-7	90.5	Siltstone	"	3550	1820	0.32	2.42×10^5	270.9	-	0.21	3.64×10^5	2.70	20.87	38.50	100.0	
JT-39	R-12	122.5	Fine Sandstone	"	2140	1200	0.24	8.81×10^4	164.6	18.2 17.2	0.33	4.57×10^4	2.28	12.92	26.42	100.0	
JT-43	R-2	72.5	Claystone	"	805	380	0.38	8.50×10^3	-	6.8	-	-	2.01	69.76	60.73	100.0	
JT-46	R-2	73.0	Fine Sandstone	"	1210	661	0.29	2.34×10^4	82.1	6.0 8.9	0.11	1.08×10^4	2.05	41.00	51.61	100.0	

Table 25 SUMMARY OF ROCK TESTS IN STRATIGRAPHIC SUCCESSION

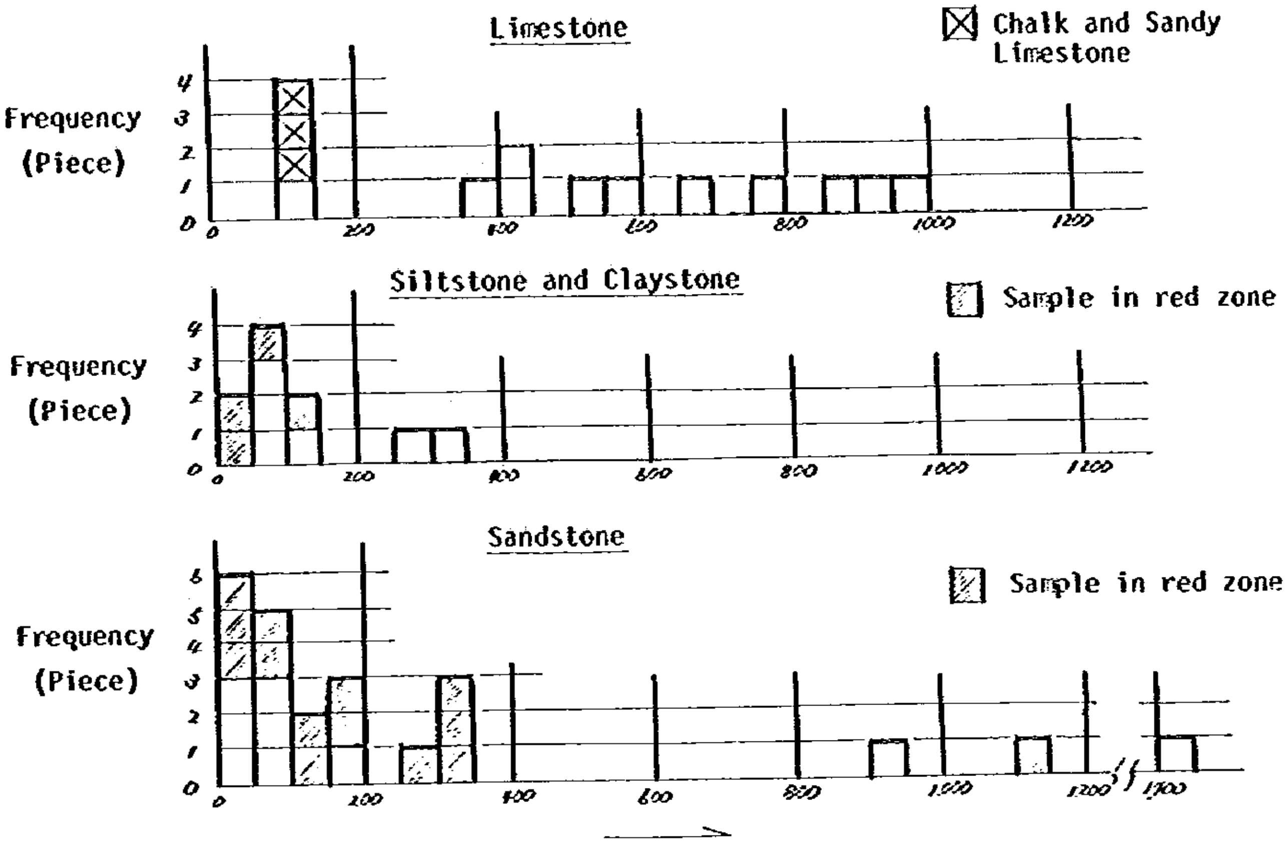
STRATIGRAPHY		ROCK	WEIGHT OF THE UNIT CUBIC VOLUME (t/m3)	EFFECTIVE POROCITY (%)	SUPER SONIC VELOCITY (m/sec)	UNIAXIAL COMPRESSIVE STRENGTH (Kg/cm2)	TENSILE STRENGTH (Kg/cm2)	NUMBERS OF SAMPLES
Laki Limestone		Limestone	2.36 - 2.59	15.84 - 3.53	3,780 - 5,250	141.9 - 957.3	30.5 - 70.5	11
		Chalk and Sandy Limestone	1.93 - 2.26	57.61 - 5.05	2,650 - 2,960	122.5 - 142.1	6.1 - 11.4	3
Upper Ranikot	Upper Shell Beds	Claystone and Siltstone	1.79 - 2.43	52.13 - 29.13	1,080 - 3,190	43.0 - 312.6	3.9 - 10.4	4
		Sandstone	1.70 - 2.39 *(2.65)	60.99 - 19.96 *(6.74)	1,240 - 4,160 *(4,890)	22.7 - 323.0 *(1,142.3)	4.0 - 24.5 *(89.0)	14
Lower Ranikot	Upper Coal-bearing Beds	Claystone and Siltstone	1.88 - 2.07	65.2 - 38.2	885 - 1,390	9.2 - 141.3	3.9 - 13.1	6
		Sandstone	1.76 - 2.11	58.7 - 27.26	1,010 - 2,950	5.2 - 59.6	4.2 - 9.5	7
	Lower Shell and Coal-bearing Beds	Claystone and Siltstone	2.70	38.5	3,550	278.9	-	1
		Sandstone	2.05 - 2.26 (2.60)	51.6 - 26.4 (5.20)	1,210 - 2,140 (5,470)	82.1 - 164.6 (1,790.4)	6.8 - 18.2 (67.2)	4

* Conglomeratic Sandstone

Siliceous Sandstone (Compact and hard)

Fig. 20

UNIAXIAL COMPRESSIVE STRENGTH (Kg/cm^2)



TENSILE STRENGTH (Kg/cm^2)

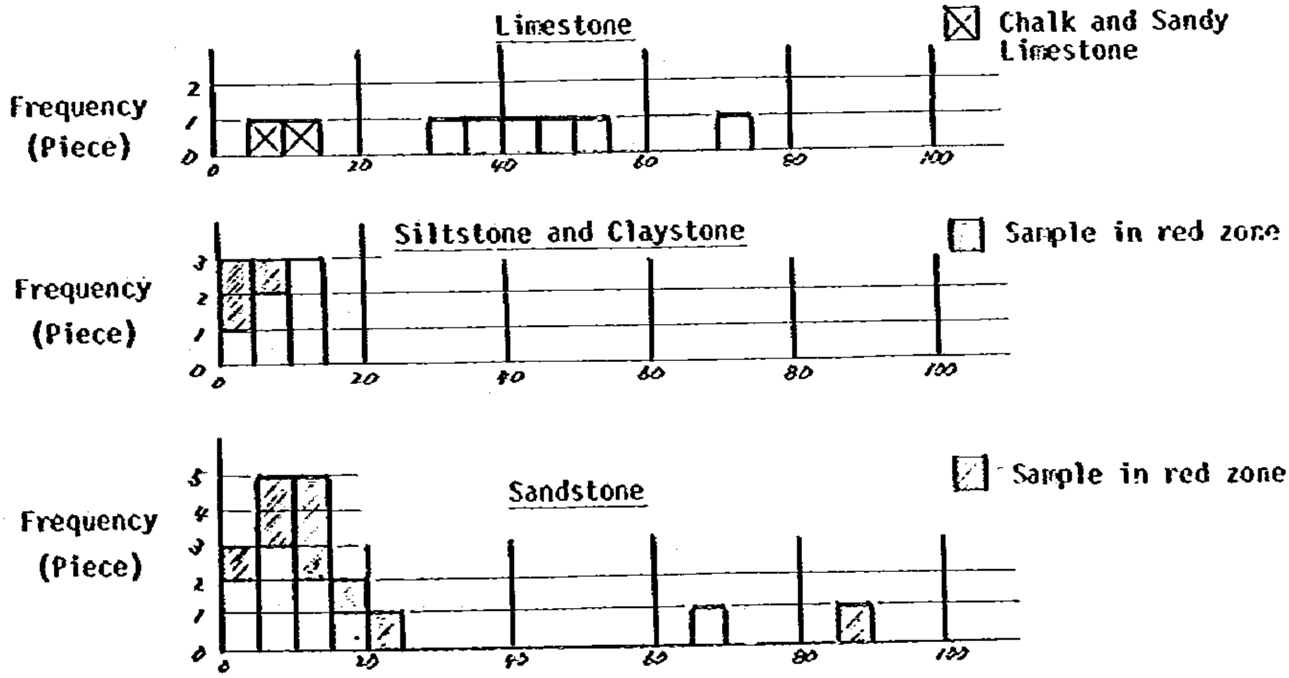
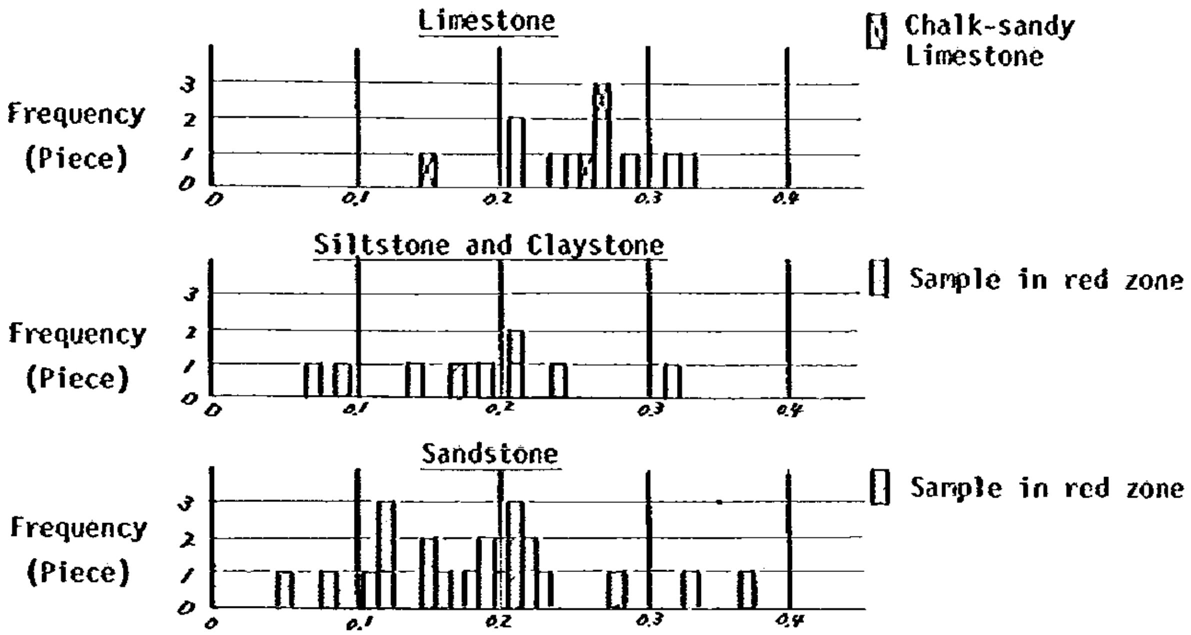


Fig. 21

STATIC POISSON'S RATIO



WEIGHT OF THE UNIT CUBIC VOLUME

(t/m^3)

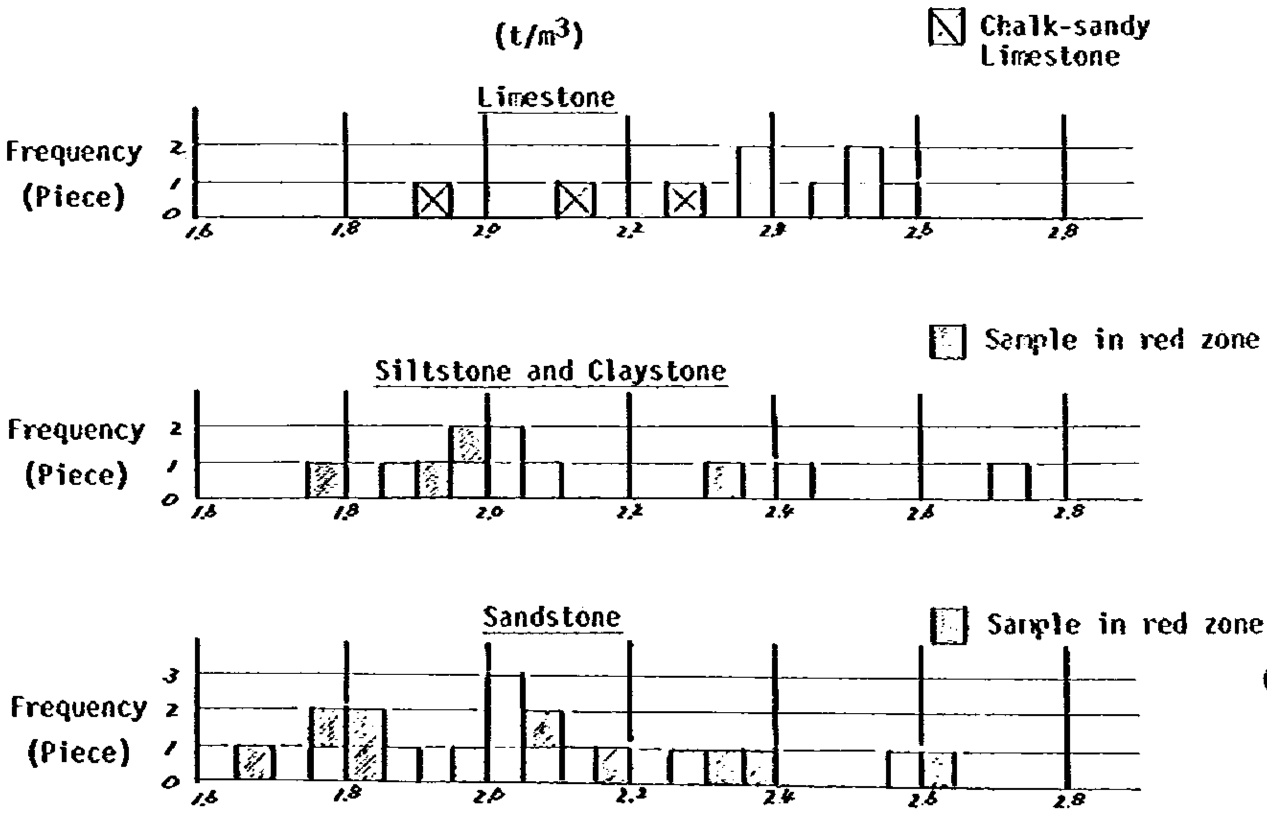


Fig. 22

RELATIONSHIP BETWEEN
TENSILE STRENGTH AND UNIAXIAL COMPRESSIVE STRENGTH

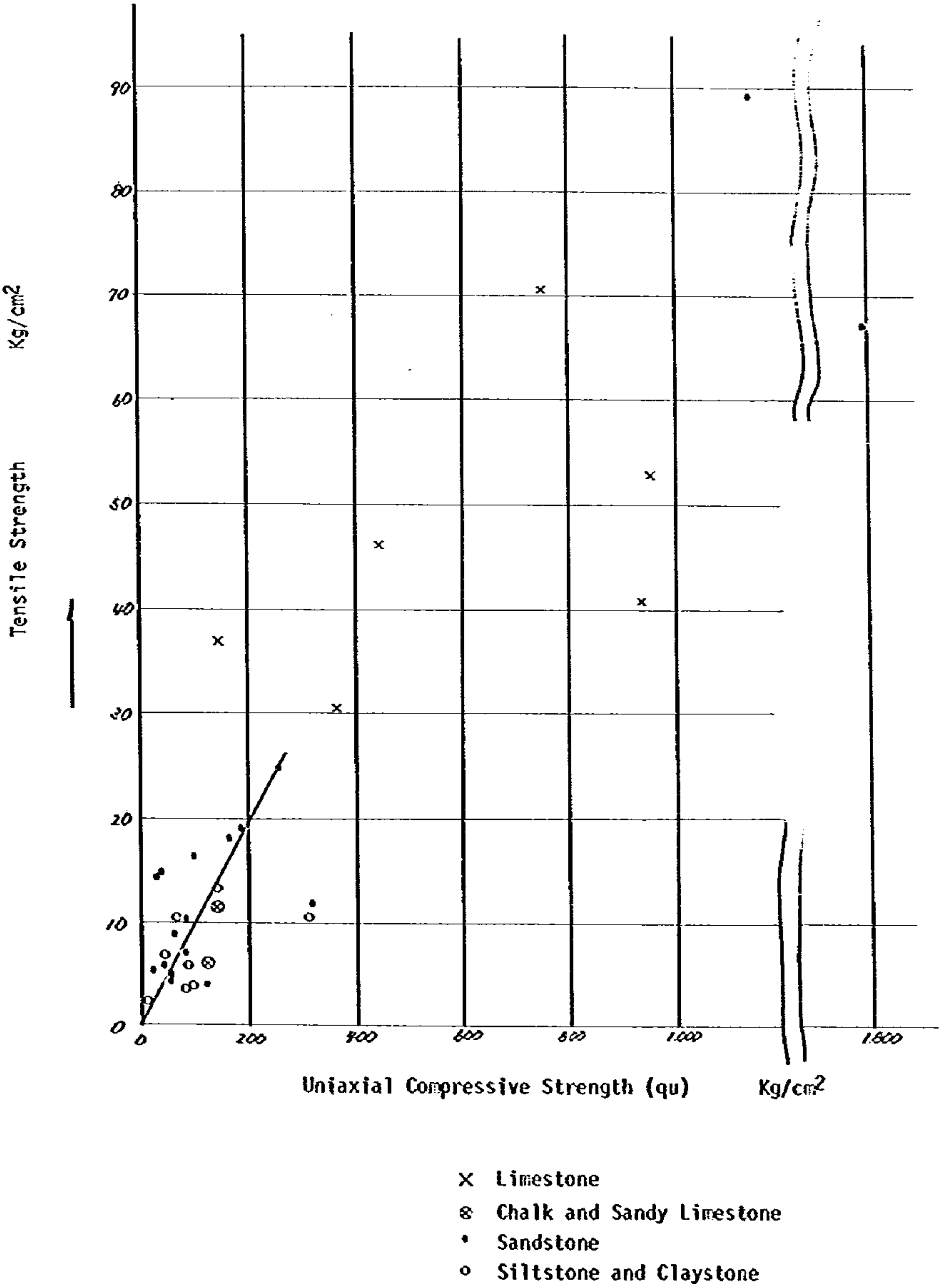
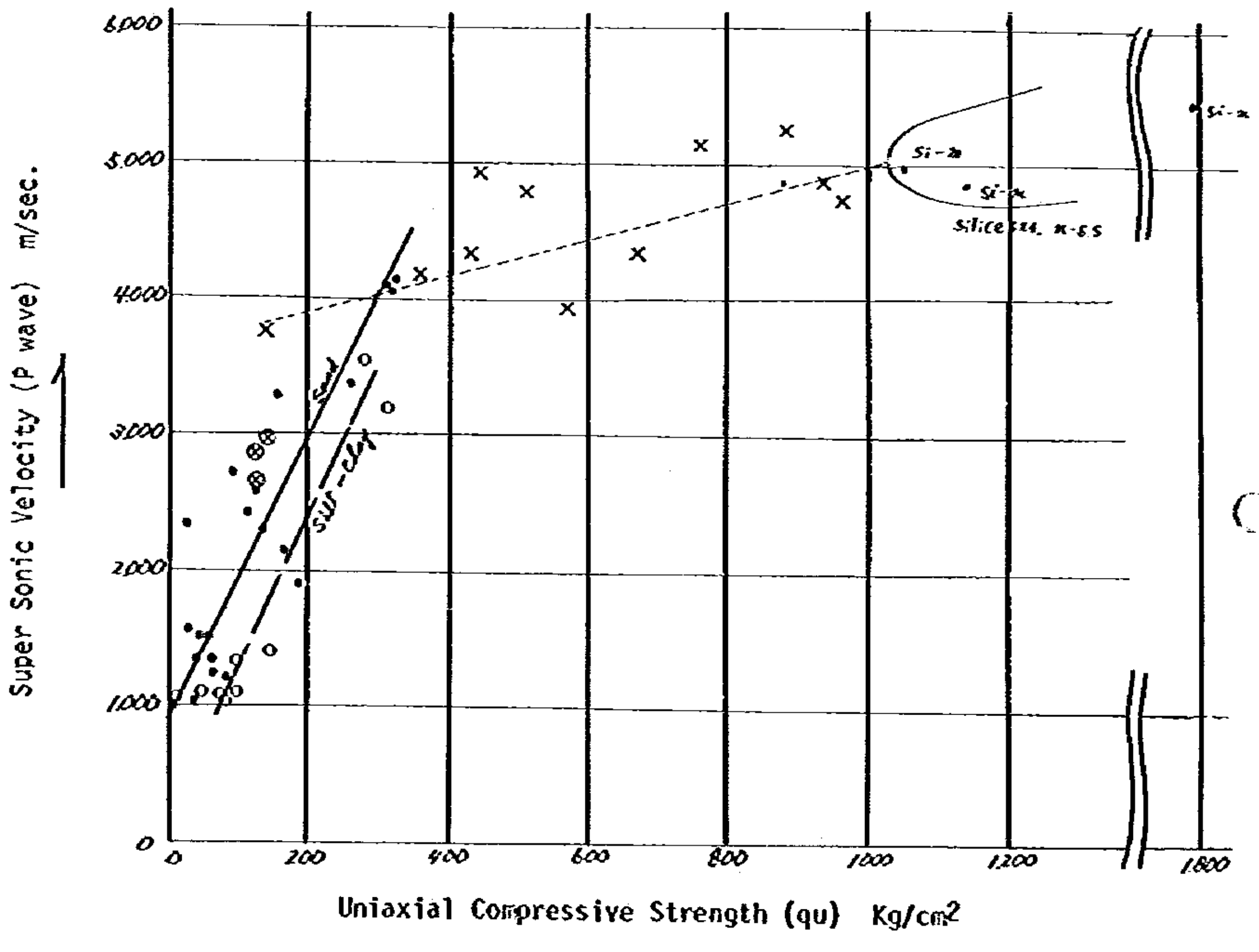


Fig. 23

RELATIONSHIP BETWEEN SUPER SONIC VELOCITY AND UNIAXIAL COMPRESSIVE STRENGTH



× Limestone
⊗ Chalk-sandy Limestone
• Sandstone
o Siltstone and Claystone

Fig. 24

STABILITY TEST FOR AGGREGATES

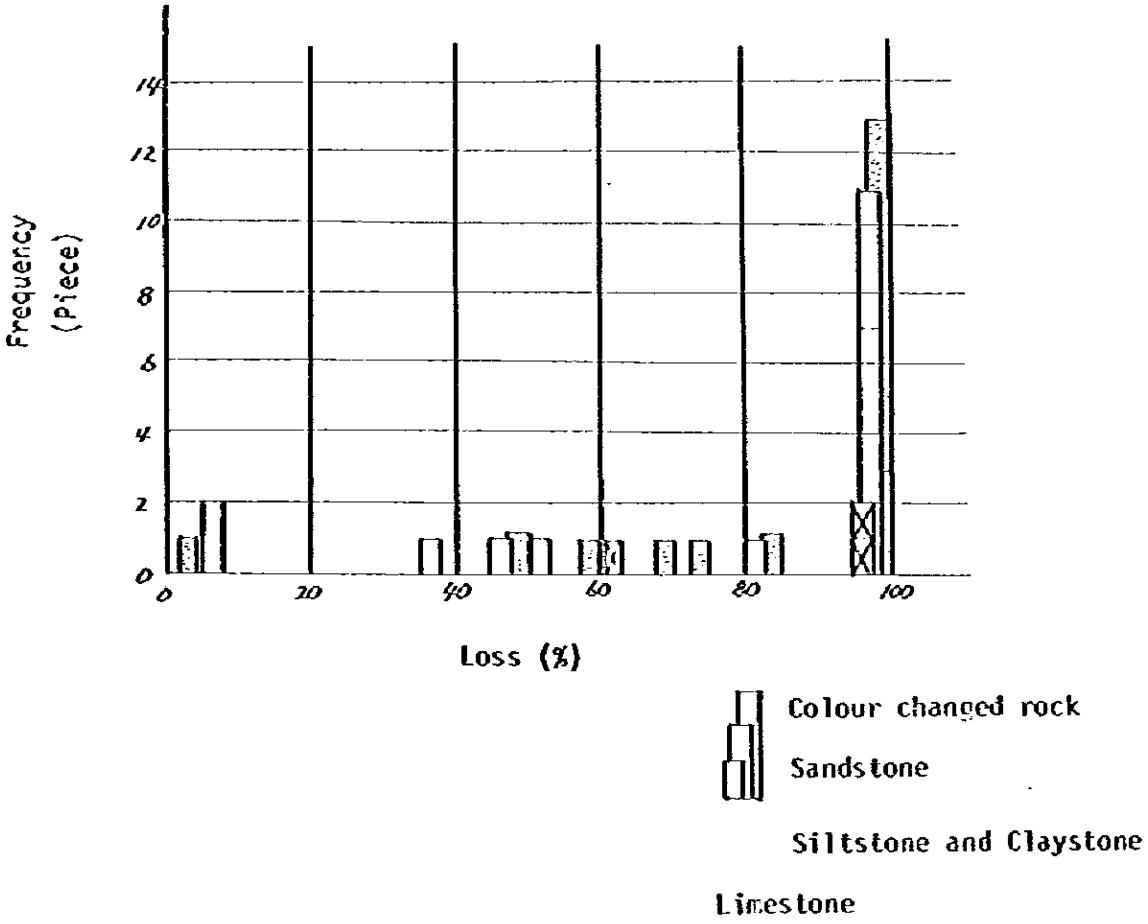


Fig. 25 SHAPE OF TEST PIECES

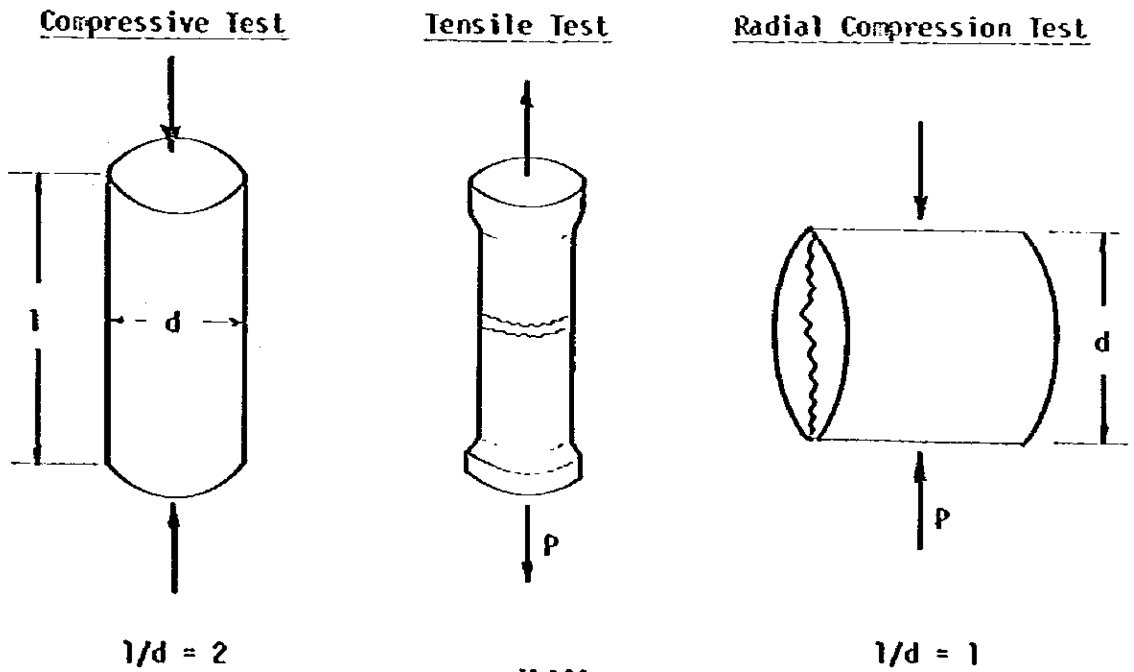
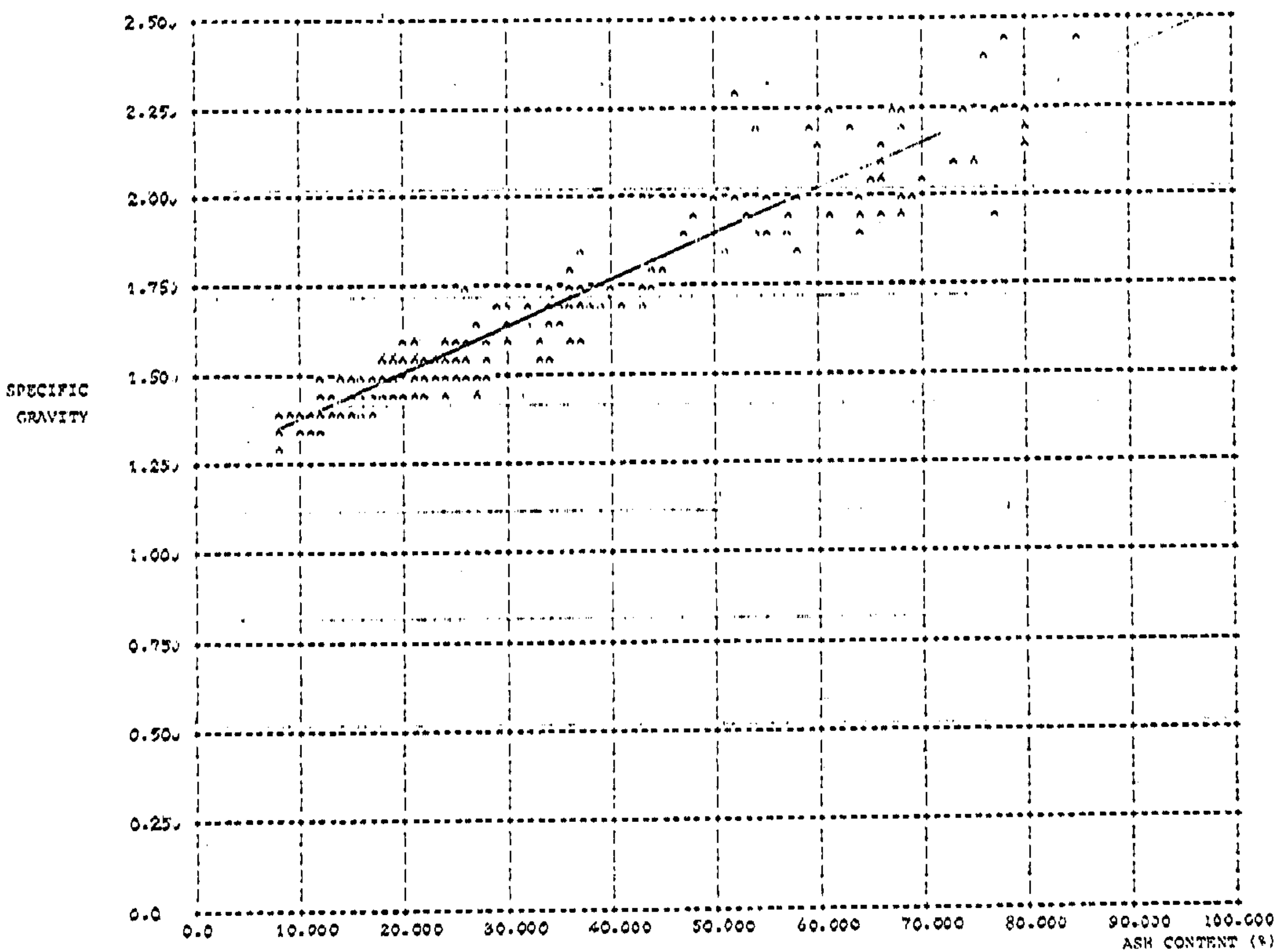


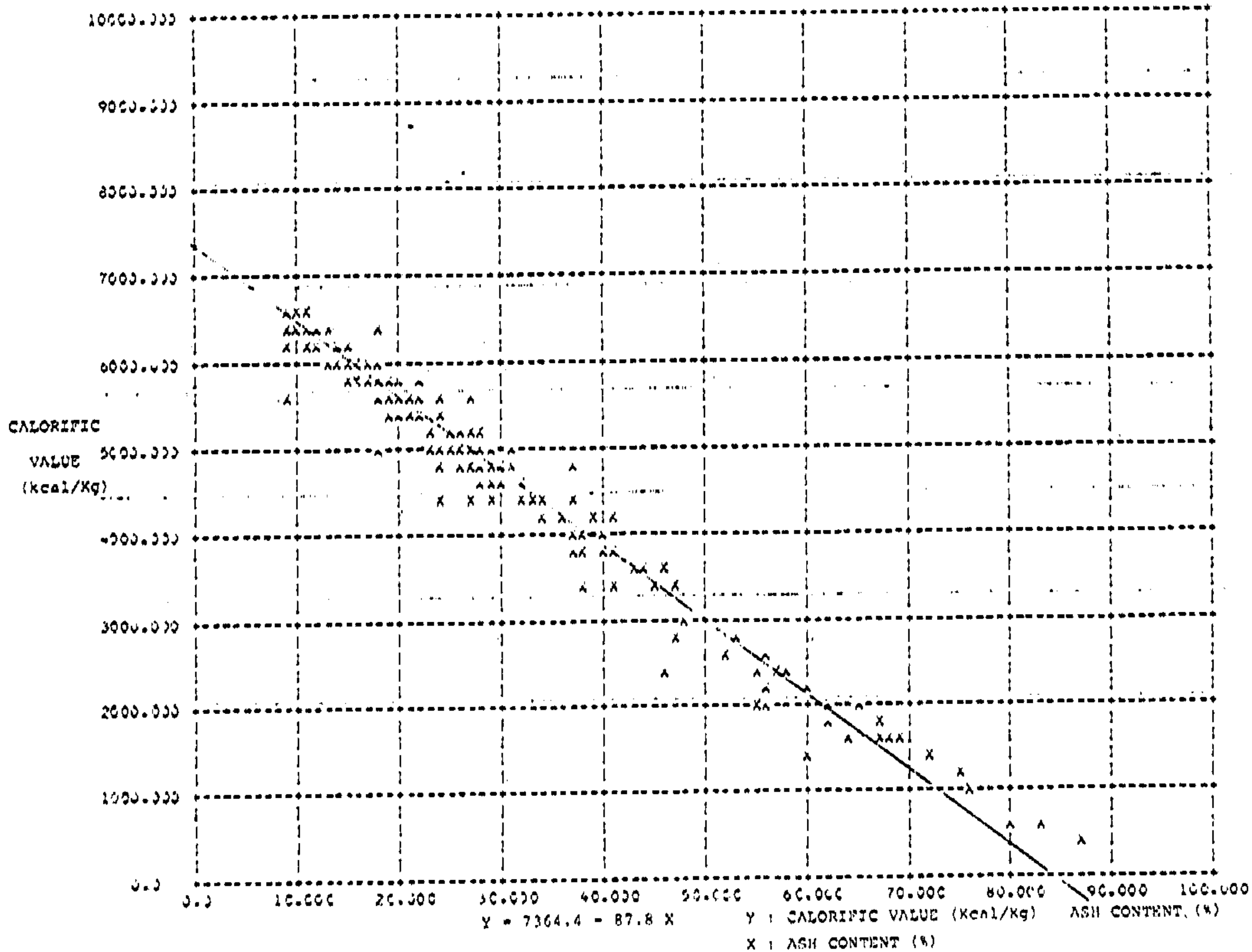
Fig.26 RELATIONSHIP BETWEEN SPECIFIC GRAVITY AND ASH CONTENT (AIR DRY BASIS)



Y = 1.2506 + 0.012632 X

Y : SPECIFIC GRAVITY
X : ASH CONTENT (%)

Fig. 27 RELATIONSHIP BETWEEN ASH CONTEN AND CALORIFIC VALUE (DRY BASIS)



under 320 kg/cm^2, and supersonic velocities under 4,000 m/sec. It is interesting to note that measured values in respect of sandstones in 'red zone' are higher than the corresponding sandstones in normal zone.

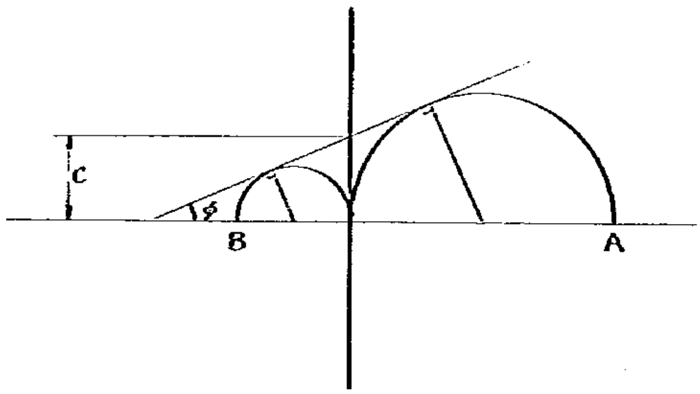
Compressive strengths of all the samples of normal zones of the Beds are under 100 kg/cm^2. Such low values may be because the samples were dried up under high temperatures during daytime, and weakened.

(c) Lower Shell Beds and Lower Coal-bearing Beds

These beds are overlain by the main coal zone No. 1. Their characters could not be judged thoroughly, because of the small number of samples as they were drilled through in some holes. However, strength of these Beds has been measured to be more than the Upper Beds.

5-5 Shearing Strength, Angle of Internal Friction and Cohesion

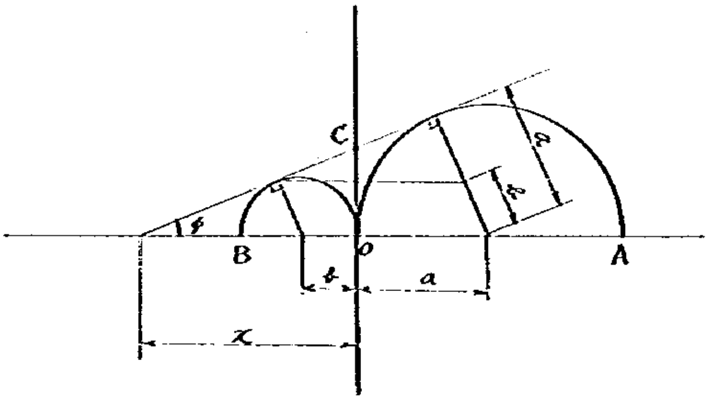
To calculate shearing strength of rock samples angle of internal friction and cohesion are determined from the values of uniaxial compressive strength and tensile strength. If C(kg/cm^2) is cohesion, and ϕ(degree) is angle of internal friction, then



where A (kg/cm^2) = uniaxial compressive strength
B (kg/cm^2) = tensile strength

Cohesion and angle of internal friction of samples are given in Table 26.

Generally, cohesion and angle of internal friction are calculated by drawing figures as mentioned above. This can also be determined by calculations as follows.



$$\frac{A}{2} = a, \frac{B}{2} = b$$

$$\sin\phi = \frac{a-b}{a+b}$$

$$C = x\tan\phi$$

$$x = b + \frac{b}{\sin\phi}$$

$$C = (b + \frac{b}{\sin\phi}) \tan\phi$$

5-6 Stability Test for Aggregate

This test gives data about rock stability for using as aggregate. It involves soaking of rock sample by putting it in saturated solution of sodium sulphate and then drying it in air. The process is repeated five times and the sample is weighed. Percentage loss in weight due to breaking off of pieces of sample in the process indicate stability. If the weight loss percent is under 12, the rock is classified as coarse aggregate.

Only three out of 39 samples from the investigated area have been determined as coarse aggregates. Limestones show stable values, such as 6.3% (JT-5, R-1) and 5.1% (JT-22, R-1). Therefore, almost all the limestones near surface are presumed to have similar stability. Rock samples of Ranikot group are very weak. Stability values of coarse to fine grained sandstones have been determined at 50 percent,fine grained sandstones at 70 to 80 percent, and in case of very fine grained silty sandstones, siltstone, etc. at 100%. But hard and compact sandstones in the group show value of 0.3%.

Table 26 MECHANICAL CHARACTER OF TEST PIECE

DRILL HOLE NO.	SAMPLE NO.	ROCK	UNIAXIAL COMPRESSIVE STRENGTH (Kg/cm^2)	TENSILE STRENGTH (Kg/cm^2)	COHESION	ANGLE OF INTERNAL FRICTION (degrees)
JT-1	R-4	Limestone	141.9	36.7	36.1	36°05'
JT-1	R-5	Chalky Limestone	142.1	11.4	20.1	58°22'
JT-1	R-6	Limestone	366.1	30.5	52.8	57°48'
JT-3	R-1	Chalky Limestoen	122.5	6.1	13.7	64°50'
JT-3	R-2	Limestone	957.3	53.1	112.0	63°30'
JT-3	R-3	Very Fine Sand-stone	63.5	8.3	11.5	50°15'
JT-3	R-5	"	37.1	14.5	11.6	25°58'
JT-3	R-8	Sandy Claystone	98.5	3.9	9.8	67°29'
JT-5	R-1	Foraminifera Limestone	933.6	40.8	97.6	66°23'
JT-5	R-2	Medium to Fine Sandstone	320.7	11.8	30.8	68°17'
JT-5	R-3	Siltstone	312.6	10.4	28.5	69°19'
JT-5	R-5	Claystone	89.7	3.9	9.4	66°27'
JT-7	R-1	Very Fine Sand-stone	22.7	5.3	5.5	38°25'
JT-7	R-3	Fine Sandstone	59.6	4.6	8.3	58°57'
JT-7	R-5	"	46.8	5.9	8.3	50°54'
JT-8	R-1	Fine Sandstone	190.1	19.1	30.2	54°52'
JT-8	R-3	Very Fine Sand-stone	57.9	4.2	7.8	59°51'
JT-10	R-2	Siltstone	85.6	5.7	11.0	61°04'
JT-15	R-2	Siltstone	43.0	6.9	8.6	46°20'
JT-15	R-3	Fine to Medium Sandstone	260.1	24.5	39.9	55°53'
JT-22	R-1	Limestone	448.1	46.2	71.9	54°24'
JT-22	R-4	"	757.6	70.5	115.5	56°04'
JT-23	R-2	Fine Sandstone	25.6	14.6	9.7	15°53'
JT-23	R-2'	"	89.1	10.1	15.0	52°47'
JT-27	R-1	Conglomeratic Medium to Coarse Sandstone	1,142.3	89.0	159.4	58°48'
JT-27	R-3	Fine Sandstone	132.6	4.0	11.5	70°18'
JT-37	R-4	Siltstone	9.2	4.2	3.1	21°54'
JT-37	R-6	Siliceous Medium Sandstone	1,790.4	67.2	173.4	68°04'
JT-39	R-4	Siltstone	141.3	13.1	21.5	56°08'
JT-39	R-5	"	69.9	10.4	13.5	47°49'
JT-39	R-12	Fine Sandstone	164.6	18.2 17.2	27.4 26.6	53°13' 54°10'
JT-46	R-2	Fine Sandstone	82.1	6.8 8.9	11.8 13.5	57°53' 53°33'

PART III DEVELOPMENT OF COAL MINE

PART III DEVELOPMENT OF COAL MINE

CHAPTER 1 SUMMARY AND RECOMMENDATION

1-1 Summary

(1) General Description of Study

The feasibility study has been divided into four main separate studies.

- Mine planning study
- Surface facilities study
- Coal preparation study
- Railway study

The report presented herein represents the study of the mining plan which has been based upon the development of mine plan inclusive of support facilities capable of sustaining an annual production rate of 1.2 million tonnes of clean steaming coal (as received base) for a period of 30 years on the basis of the geological exploration study, site survey, and study in Japan.

The mining plan developed in this report involves the extraction of 36,780,000 tonnes of raw coal from No. 1, 2, 3, and 5 seams.

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

The coal reserves area for this study has been divided into 3 blocks as shown in the geological report, and three mines i.e. an underground mine provided with longwalling in central block, and two open pits provided with truck-shovel method in west and east block.

The annual average coal production (as received base) of clean coal for a period of 30 years will be 1,170,000 tonnes of which 220,000 tonnes of coal will be produced at an underground mine and 950,000 tonnes at open pits. The average calorific value of clean coal is estimated at 3,827 kcal/kg, and its sulphur content is estimated at 5.9 %. The layout of the underground mine will be developed by inclined shaft system provided with the longwall mining method. The open pits will be developed by the average stripping ratio of 11/1.

Office buildings, factory buildings, and factories, etc. as the support facilities on the surface inclusive of service and welfare facilities will be provided. In order to reduce a part of sulphur content and ash content, coal preparation plant provided with a picking belt of the capacity of 400 t/h will be planned.

The railway of 27.5 km between mine site and Khanot, and also spur tracks of five km from existing railway to plant will be laid to meet the main purpose of coal transportation and also commuting train from colony in Khanot to mine.

The operating employee requirements will be 1824 during the main production period, and average employee requirements will be 1689 for a period of 30 years.

Total mine productivity exclusive of railway will be 2.3 tonnes per man per shift.

Preproduction schedule has been made by a request in response to the Pakistani delegation in Tokyo, November 1980, and so construction work will commence in April, 1983. However, in order to achieve this start-date it will be necessary to perform certain pre-engineering and procurement activities, and preparation work inclusive of detailed drilling.

The production schedule indicates longwalling production commencing in 1986, and total production inclusive of open pit production will be 733,000 tonnes in 1986.

The total capital costs calculated for this study inclusive of railway amount to 2,146,000,000 Rupees based on June 1980 price levels. But interest during construction and escalation have not been included.

The average operating costs with freight will be estimated at 381 Rupees per clean coal ton based on June 1980 price levels. But escalation has not been included.

(2) Layout of Mining Plan

1) Mining Area and Production Scale

The investigated area of about 26 square kilometers has been divided into three blocks; namely central, western, and eastern defined by foldings and fault zones at boundary.

In central block, only two seams of No. 1 seam and No. 3 seam are considered mineable. No. 1 seam which covers main part of production lies at a depth of 85 m to 123 m. Therefore, open cut method will be uneconomical due to the high stripping ratio which indicates more than 15/1, an underground mining method will be applicable for this block.

In western block, five coal seams considered minable, and No. 1 seam, bottom seam, lies 33 m to 85 m deep which is shallower compared with eastern block. Accordingly, stripping ratio has been calculated at 8.6/1 and west pit will be planned to cover the half part of production for total production.

In eastern block, three seams considered mineable, and the bottom seam lies 45 m to 91 m deep and in this block there exists loose sand on the direct roof of No. 1 & No. 2 coal seams. Therefore an underground mining method is not applicable. And also even in open cut system the stripping ratio will be calculated at 18/1, so coal reserves will be foundamentally estimated uneconomical at the present time. However coal production has been calculated by both mines with consideration of average stripping ratio keeping to 11/1 or below.

The geological mineable reserve figure as calculated in geological report is approximately 6,500,000 tonnes raw coal (air dried base) in central block, approximately 15,000,000 tonnes raw coal (air dried base) in western block, and approximately 32,000,000 tonnes raw coal (air dried base) in eastern block.

The annual coal production rate has been calculated at 200,000 tonnes of clean coal (air dried base) in central block and 500,000 tonnes of clean coal (air dried base) in western block based on a period of 30 years of power station life.

In eastern block as abovementioned there are not economical reserves. However, by diluting the high stripping ratio with the low stripping ratio in western block so as to reduce the total stripping ratio of 11/1 annual coal production will be calculated at the 300,000 tonnes of clean coal (air dried base).

Overall coal production figure will be one million tonnes on air dried basis and 1.2 million tonnes as received base.

2) Underground Mine

The underground mining plan developed involves the extraction of 6,589,000 tonnes of clean coal as received base. Major access between the mining areas and surface will be via two inclined shafts in rock. No. 1 inclined shaft will be used for coal transportation, personnel and materials transportation, and intake airway, while No. 2 inclined shaft will be used for return airway only.

The portal of No. 1 inclined shaft is located at 2152590 E of X axis and 900685 N of Y axis in co-ordinates, and No. 2 shaft will be driven on 50 m centres.

The inclined shaft will terminate at the point of intersection with No. 1 seam which is on the centre of this mining block. Both shafts will be driven at minus 12 degrees of gradient. Two parallel headings which are intended for use as main intake airway and main transportation roadways provided with 11.2 m^2 of effective sectional area will be driven from north to south in the central part of block via access inseam entry from shaft bottom to east. The panel entry will be driven on 120 m centres. Longwall face will be installed in this section. The main entries and inclined shafts will be driven conventionally blasting, side-tipping loaders, and mine car system with arched support, and other entries will be driven blasting, gate-end-loader, and mine car system with square set support. The developing rate has been calculated at 2.7 m/day at maximum

Mining area will be divided into eleven blocks for the purpose of preventing loss of coal reserves due to spontaneous combusting, and each block will be sealed off after completion of extraction of coal. Pillars will be left along main entries, gob areas and faults, and 50 – 100 m in length between panels.

Retreating longwall method of 120 m long with caving is selected. Coal will win conventionally using blasting, hand loading to the double chain conveyor in the face supported with the hydraulic steel prop and link bar. The packing in gob side of gate entry will be done with fly ash in order to re-use the gate entry for return entry. This method is very advisable for both prevention of spontaneous combustion and increasing the recovery percentage. Coal winning will be operated by two shift system and one shift will be used for preparation and maintenance of face. The coal production from a longwall face has been calculated at 400 tonnes/day, manpower required inclusive of foremen will be 129, and average productivity will be calculated at 3.1 tonnes per man per shift in face operation. In areas which cannot be worked by systematic mining, room and pillar method will be planned.

Mine car transportation system has been selected to handle coal produced, waste rock, and materials transportation. To sustain this mine car transportation underground, the two type of battery locomotives of 8 t and 10 t are selected. The winding machine with the capacity of 200 kW in inclined shaft will be installed.

Drainage quipment with capacity of 50 kW, 180 m head, 1.4 m^3/min will be installed at the bottom for rainy season. And also small capacity supply water pump and drainage pump are provided.

The total consumption of compressed air has been calculated at approximately 90 m^3/min. To cover this demand the compressor with a capacity of 2 sets of 240 kW and 1 set of 75 kW will be installed on the surface.

The centralized ventilation system has been selected to meet this mining plan, main fan of 300 kW will be installed at the entrance to No. 2 inclined shaft. The fan will be capable of 5,000 m^3 at 200 m/m water gauge.

Site work for portal will be completed in a three month period to permit the inclined shafts. Mine development will require a total time of 24 months, including 8 months for inclined shafts, 8 months for main entries, 8 months for gate entry and preparation flor longwalling.

3) Open Pit

A shovel and truck system in both pits to remove the overburden has been selected with the consideration of multi-seam mining, early production, and techniques of workers. The shape in bench represents the multi-bench system, and the bench is designed at 14.0 m high, maximum 60 m wide, and at an angle of 45 degrees from a rock stability points of view. Truck road is designed at 14 m wide with the maximum inclination of 8 degrees.

The total steps of bench will be 3 to 4 benches to reach the upper most coal seam due to the deep mine.

Overburden is drilled with the two sets of drills of electric drive type of 9-7/8 in. diameter in both mine, and drilling operation will be done by two shift system in a day. Three units of bulldozer will be arranged for cleaning the surface of bench and treating the soil.

Spacing of drill holes is designed for 7 m x 8 m, and penetrating speed is designed at 23.3 m/h. Ammonium Nitrate Fuel Oil is used for blasting the overburden, and powder factor is designed at 0.44 kg/m^3.

After blasting the overburden,loose overburden is loaded by two units of electric power shovel with a dipper capacity of 11.5 m^3 in each pit, and a bulldozer for a shovel aid in cleaning the surface. Mechanical efficiency of shovel is estimated at 75 %.

Loosed overburden is removed in 120 t trucks provided 9 nos. in each pit and/or dumped into the space from where the coal has been mined out. The truck speed in pit area is designed at 13 km/h, and in up grade road is designed at 10 km/h, and in flat and good condition road is designed at 30 km/h. This is designed by simulation method of computer system.

The parting rock between coal seams is drilled by rotary type drilling machine of 80 m/m in diameter, and also for the loading and transportation bulldozer, wheel loader, scraper, and 46t trucks are provided. This operation will be done after coal mining and before stripping work. The rock blasted is removed in trucks or dumped in mined area by bulldozer and/or scraper. A bulldozer is used for discharging the rock from the highwall to mined out space and cleaning the surface of coal before mining. A scraper is used for loading and hauling the parting rock and cleaning the surface of coal.

Coal seams are usually broken by the aid of ripper of bulldozer, but in hard coal seam blasting will be done. Coal is loaded by hydraulic excavator using a 6 m^3 dipper and is transported by a 46 ton truck.

In pit service and road maintenance road grader, bulldozer, crusher, water tank, and truck crane, etc. are provided.

Dewatering equipment is equipped with the pump capacity of 2 units of 22 kW capable of 30 m head and 2 m^3/min of water volume during rainy season.

Rock desert area spreads in the surface, and so damage due to pollution need not be considered, but some extent of reclamation work will be necessary. In cleaning the mined out area one unit of wheel loader with bucket capacity of 5.6 m^3, one unit of scraper with the capacity of 24 m^3, two unit of bulldozer, and a 46 t dump truck will be provided.

Layout of pit is designed at 40 m wide, and 300 m to 1,000 m long based on geological factor of 80 % and mining recovery of 90 %.

The clean coal production in open pit has been calculated at 29,013,500 tonnes with the stripping volumes of 330,112,900 m^3 in solid for a period of 30 years, and stripping ratio will be 11/1 which will be 9/1 in west pit and 14/1 in east pit respectively. Productivity in open pit is estimated at 5.7 tonnes per man per shift.

The open pit planning study provides for a continuing intensive coal development schedule. In order for this to be reliably achieved, it is necessary for certain preparation work to be performed prior to the commencement of the project.

The total volume of bank overburden to be removed will be 5,000,000 m^3 in 1983, 7,780,000 m^3 in 1984, 8,958,000 m^3 in 1985 and 1986, approximately 10,000,000 m^3 after 1977.

The coal production will be planned at 123,000 tonnes in 1984, 301,000 tonnes in 1985, 602,000 tonnes in 1986, 732,000 tonnes in 1987, full production in 1988.

Stripping work in the west pit will commence at the line connecting the drill hole JT 16, JT 7, JT 9, and PS19. Bench cut will be developed to west. The total volume of bank overburden to be removed will involve of 11,938,000 m^3 from 1983 to 1985. In east pit stripping work will commence at JT 50 of drill hole. The stripe type of bench cut will be provided from east to west, and mining procedure will be developed to north with a constant angle of highwall and width of bench. The total volume of bank overburden to be removed will be 9,799,900 m^3 from 1983 to 1985.

4) Surface Facilities

Integrated surface layout system has been selected and the buildings and factories are located at the surface of barren area between west block and central block with no influence on mining operation.

In the central part of surface lay-out, mine office, air compressor house, winding machine house, work allocation room, safety lamp room, mine substation, mechanical and electrical workshops and store house, etc. will be provided. In west side of this block head office, heavy vehicle maintenance shop, etc. will be provided. In the south side, preparation plant and emergency stock yard near railway station will be provided.

Coal transportation from mine to power station and commuting service from colony in Khanot to mine will be done by railway. Both railway and road will be utilized for the transportation of materials and heavy machines, etc.

The water near surface in the River Indus will be delivered by in-take pump to sediment pond, and purified. This purified water will be utilized for industrial and living water.

Site work for the area of building inclusive of supporting area will be provided for approximately 210,000 m^2, and for railway inside mine and road outside mine of 43 kilometers long will be provided for approximately 470,000 m^2. Total of 680,000 m^2 for site work will be provided. Other civil work including foundation work of mine substation, and construction of oil tank, protective fence around mine, and embankment work for explosive magazine, and construction work of sewage treatment plant will be provided. Road construction work of 6 kilometers long with 14 m wide for dump truck and other road work of 33 km long with 7.5 m wide will be provided.

Structural work for production facilities consisting of mine substation, power house, air compressor house, and work allocation room, etc. will be provided for approximately 2,700 m^2 in area. These are structural reinforced concrete or brick buildings of single story. Head office building, open pit office building, and underground office building will be provided for approximately 4,140 m^2 in area. These are structural reinforced concrete or brick buildings. Head office building will only be constructed as two storied building. The workshops consisting of heavy vehicle maintenance shop, mechanical and electrical shops for underground mine, etc. will be provided by structural reinforced concrete and steel building with the total area of approximately 12,200 m^2 and also with one story building. Other building will be provided with powder magazine, and powder handling house.

The industrial and living water required for the mine site and colonial area will be delivered from the River Indus to two sand basins in Khanot with each capacity of 240 m^3 by means of two water intake pumps provided with the capacity of 36 kW and 2.93 m^3/min. with the head of 90 m. The water will be purified at the purificating stations in Khanot, and conveyed to the distributing reservoir in colonial area and in mine site respectively, and also distributed by the distributing pump from the reservoir to the consumer. The water conveyance pump to the mine site is provided with the capacity of 37 kW and 0.75 m^3/min. with the head of 180 m. The water conveyance pumps of two units to colonial area are provided with the capacity of 10 kW and 2.18 m^3/min. with the head of 30 m. The pipes used for will be 150 mm, 200 mm and 250 mm in diameter of cast iron pipes for water delivery and conveyance. Especially galvanized steel pipes is used for water distribution. The septic tanks will be provided for the sewage treatment of administration office and other major surface facilities.

The mine substation is provided for receiving the required power from WAPDA at 33 kV and distribute it to all areas at 33 kV and/or 3.3 kV. The total installed motor capacity within the mine has been calculated at approximately 7,000 kW and, excluding the open pit supplied at

33 kV, the total capacity to be supplied at 3.3 kV is estimated at 3,480 kW. Therefore, the required transformer capacity to be installed will be 4,000 kVA.

Two units of emergency diesel generator with the capacity of 500 kVA at 3.3 kV in three phase will also be provided for supplying the power only to safety facilities in case of the power failure. For the communication system within the colliery a private telephone system will be provided. The other major facilities for the mine are two sets of drilling machines, vehicles, a computer, and a hospital.

5) Coal Preparation Plant

The plant feed coal is screened at 50 mm in size and 50 mm oversize waste with 75 % ash content, 4 % of total raw coal, will be removed by hand picking.

The quality of clean coal delivered at the power station will be estimated at total moisture of 25 %, ash content of 19.7 %, volatile matter of 27.8 %, total sulphur of 5.9 %, calorific value of 3,840 kcal/kg, and plant yield of 96 % on as received basis.

The quality indicates inherent moisture of 9.3 % and calorific value of 4,640 kcal/kg on air dried basis, ash fusion temperature is more than 1,300°C, and Hardgrove grindability index is 70, and these characteristics are favourable for steaming coal. The electric resistance of ash is 3.5×10^{13} ohm-cm at 130°C, and it is rather high for the maximum limit of 1×10^{13} ohm-cm, accordingly some considerations are required for the design of a electric dust collector.

The anticipated operating schedule of plant is established at a rate of 16 hours on 2 shift system per day, 300 days per year and availability of 80 % and average plant feed is designed on 400 tonnes of raw coal per hour. The main process is provided for removal of over 50 mm waste in size by hand picking system, and the main plant facilities are as follows:

A 100 t dump hopper and a 1,500 t raw coal bin is included in the raw coal receiving and stocking equipment, and over 300 mm wastes in size are removed at this section.

Raw coal from the underground mine is stored in the above mentioned raw coal bin through a tippler and a raw coal conveyor. Raw coal drawn out from the raw coal bin is fed to the raw coal screen through the plant feed conveyor to be screened at 50 mm in size.

Over 50 mm raw coal is conveyed onto hand picking conveyor at a rate of 40 t/h, while 50 mm undersize of the screen is stored in the clean coal silos through the clean coal conveyors at a rate of 384 t/h.

Over 50 mm waste is delivered to a rock bin through rock belt conveyors and dumped on to a waste area by truck. Over 50 mm coal is crushed by a single roll crusher and stored in the clean coal silos.

The clean coal storage and loading equipment is provided with two 2,000 t clean coal silos, a 20,000 t emergency clean coal stockpile, clean coal reclaiming system, a 110 t loading hopper and a railway track scale.

Reinforced concrete structure will be built for a raw coal dump hopper, a raw coal bin and clean coal silos, and a rock bin and a loading hopper will be constructed with steel structure, and the housing of the hand picking and electrical room will be provided with steel structure.

The total capacity of the motors in the plant will be provided with 460 kW, and power will be supplied with 3 kV and 3 phase from the mine substation.

The open-air stockpilling system of the Lakhra coal will be recommendable by the following methods to prevent spontaneous combustion.

The compacted stockpilling should be less than a week in storage period and under 3 m in height. In order to build and keep a safe stockpile, the stockpile must be compacted perfectly at every layer of piles by bulldozer of loaded truck, and the height of each layer should be kept at 40 – 50 cm, and the final stockpile will be built and piled up by repeating of the above methods.

6) Railway Transportation

In order to supply the coal tonnages required for power plant in Jamshoro of approximately 1,200,000 tonnes per annum i.e. 4,000 tonnes per day the railway system between mine site and power station will be provided for the distance of 64.5 km. For this purpose, the new railway of 27.5 km long having the same gauge with existing Pakistan National Railway from preparation plant to Khanot will be constructed and connected with the existing one at Khanot, and new spur track of 5 km long near power station site will be provided.

Equipment of 3 units of diesel electric locomotives, 50 wagons, 4 passenger cars, tracks of 32.5 km long, and all support facilities inclusive of loading and unloading facilities, and also repair shop will be provided. However operation and management will be left to the Pakistan National Railway.

The freight charge will be estimated at 26 Rs/t. In this case, the freight charge of existing railway is estimated at 5 Rs./t, and total of depreciation cost, amortization cost, and interest is calculated at 9 Rs/t.

Coal is loaded from clean coal silo to wagon through vibration feeder. A train comprised of 24 wagons loaded with the 840 t of coal is pulled by the locomotive with the capacity of 2 units of 825 kW motor and the ownweight of 84 tonnes.

The time required for round trip between mine and power plant will be approximately 202 minutes. Two formation of trains and five round trips per day will be scheduled.

Commuting train between Khanot and mine will be scheduled at four round trips in a day, and other materials will be mainly transported by road, and railway transportation of materials will be available at slack time in middle of the night.

7) Mine Development Schedule

The mining plan development schedule indicates the commencement of the stripping work of open pit and construction work of road, etc. after the date of approval of the project. In order to achieve this starting date it will be necessary to perform detailed drilling, ground investigation, topographic survey, repairing a part of road, preparation work near initial box cut, and certain pre-construction and procurement activities prior to the date of commencement.

The construction work will commence in April 1983. Site work, road construction work, preparatory work, and construction work of main substation and heavy vehicle maintenance shop in the surface facilities will commence in April and be completed at the end of 1983, and other surface facilities will be completed by the end of 1984.

Construction work of railway will commence in April 1983, and be completed in June 1986.

Construction work of coal proparation plant will commence in October 1983, and be completed at the end of 1985.

Construction work of portal in underground mine will commence in October 1983, and development work in underground commence in the beginning of 1984, and one longwall face will be prepared at the end of 1985. Another face will be prepared one year later.

Stripping work by the heavy machines will commence in April 1983. Full production of coal scheduled will commence in 1988.

8) Manpower Requirements and Organization

The operating manpower requirements for this project are planned upon modification of manpower and organization commonly observed in Japan and also the world, and also in prevailing coal mines controlled by PMDC and in PC-1 Form for Lakhra project submitted by PMDC in 1976.

The total numbers of officers and wage workers represent the jobs and/or positions to be filled each day, and to not include persons not working due to sickness, accident or any other reason. Costs associated with a 20 % level of absenteeism for underground wage workers have been included in the plant.

The manpower requirements will be 662 men inclusive of 47 officers and 615 wage workers during the main production period in underground mine.

The manpower requirements in open pit will be 410 men inclusive of 50 officers and 360 wage workers during the main production period.

The manpower requirements in surface facilities will be 629 men inclusive of 111 officers and 518 wage workers.

In preparation plant the manpower requirements will be 123 men inclusive of 10 officers and 113 wage workers.

The total will be 1,824 men inclusive of 218 officers and 1,606 wage workers during the main production period. The average number will be 1,689 for a period of 30 years.

Organization for this project is planned on the basis of prevailing system of coal mines in Japan and Pakistan. The new and special sections of safety, training and system which are not organized in Pakistan coal mine will be added.

Productivity has been calculated at 1.3 t/man/shaft in underground mine, 7.8 t/man/shift in open pit, and 2.3 t/man/shift in overall mine.

9) Capital Costs

The capital costs in this study are based on June 1980 values. The total capital costs calculated for this study amount to Rs. 2,522,000,000 inclusive of interest during construction for the first three years of the project. The total capital costs are composed of direct costs and indirect costs.

The direct costs have been estimated at 1,555 million Rupees inclusive of 1,290 million Rupees for mine development, 191 million Rupees for railway construction, and 74 million Rupees for a 4 % contingency to direct costs.

The indirect costs have been estimated at 591 million Rupees, inclusive of 425 million Rupees, for import duty of 40 % on C & F price of equipment and materials supplies, 77 million Rupees for engineering fees of 5 % to direct costs and 62 million Rupees for administration cost of 4 % to direct costs. Interest during construction is calculated at 376 million Rupees.

The total currency will be divided into 1,433 million Rupees for the foreign currency portion and 1,089 million Rupees for the local currency portion. Escalation has not been included.

The capital costs include the following:

Production facilities –

- site work, and construction cost of road, factory buildings and preparation plant.
- mechanical and electrical equipment purchased and installation costs applicable to the mine operation.
- machines and materials for mine development and installation cost applicable to the mine operation.
- maintenance and power cost during the initial 36 months of the project.
- all miscellaneous costs during the initial 36 months of the project.

Ancillary facilities –

- construction cost of office buildings and furniture,etc.

Service & welfare facilities –

- officers salaries and workers wages during the initial 36 months of the project.

Railway –

- construction of tracks, and mechanical and electrical equipment purchased and installation cost.

- loading and unloading facilities.

The capital cost for development of open pit will be calculated at maximum expenditure of 67 % in total capital costs and secondarily railway of 12 % inclusive of import duty but exclusive of contingencies.

10) Operating Costs

The operating costs have been calculated as follows:

- Depreciation cost calculated at the average 30 years life of equipment and installation cost with the consideration of residual value of 10 %.

- Interest is based on 12.5 % for local currency and 8.75 % for foreign currency. Interest for local currency will be paid for 5 years, and for foreign currency will be paid for 10 years according to repayment schedule. All these calculations are based on PC-1 Form submitted by PMDC.

- Salaries and wages are based one PMDC PC-1 Form and escalation is added to the 1976 base cost.

- Power costs calculated at Ps. 49 per kWh based on WAPDA's Tariff C-3 for bulk supply at 33 kV.

- Replacement and improvement costs of equipment calculated on the basis of life of machine.

- Materials and supplies include explosives, mine timbers, oils, fuel oils, cables, etc.

- Maintenance cost calculated maintenance costs of equipment and buildings. The maintenance costs of machine are estimated at less than 10 % of machine to be used.

- Administration costs are estimated at 3 Rs/t for outside service, management fee of head office, travelling fee, etc.

- A wage personnel absenteeism rate of 20 % has been used in the study. Equivalent additional wage workers will have to be employed to counteract the effect of this absenteeism rate, in order to ensure that all the jobs are manned in underground workers.

- Freight charge will be estimated at 17 Rs/t inclusive of 5 Rs/t in existing freight charge.

- Coal preparation costs calculated for 7 Rs/t.

Estimated Capital Cost

(In Million Rupees)

Description	Foreign Currency	Local Currency	Total
Production Facilities	1,022	222	1,244
Ancillary Facilities	3	20	23
Service, Welfare Facilities	–	23	23
Sub-Total	1,025	265	1,290
Railway Facilities	106	85	191
Contingency	56	18	74
Sub-Total	162	103	265
Direct Cost Total	1,187	368	1,555
Import Duty	–	452	452
Engineering Fee	58	19	77
Administration Cost	–	62	62
Indirect Cost Total	58	533	591
Total	1,245	901	2,146
Interest During Construction	–	376	376
Grand Total	1,245	1,277	2,522

Freight, taxes and duty included.
The estimates reflect June 1980 values.
No escalation.

Estimated Capital Cost by Facilities

(In Million Rupees)

Description	Foreign Currency	Local Currency	Total	%
Underground Mine	85	49	134	7
Open Pit	803	493	1,296	67
Surface Facilities	71	80	151	8
Preparation Plant	66	52	118	6
Sub-Total	1,025	674	1,699	88
Railway	106	128	234	12
Grand Total	1,131	802	1,933	100

Freight, taxes and duty included.
Contingency in direct cost not included.
The estimates reflect June 1980 value.
No escalation.

Estimated Operating Cost

(Rupees per clean tonne)

Description	Foreign Currency	Local Currency	Total
Salaries	–	9	9
Wages	–	3	3
Power	–	5	5
Replacement and Improvement	46	20	66
Materials and Supplies	45	91	136
Maintenance	28	14	42
Administration	–	3	3
Sub-Total	119	145	264
Depreciation	–	32	32
Amortization	–	27	27
Interest	–	36	36
Mine Total	119	240	359
Freight	2	15	17
Depreciation	–	3	3
Amortization	–	2	2
Interest	–	4	4
Railway Total	2	24	26
Deduction	–	△4	△4
Grand-Total	121	260	381
Underground Mine	8	21	29
Open Pit	108	111	219
Preparation Plant	1	6	7
Surface	2	7	9
Total	119	145	264

The estimate reflect June 1980 value.
No escalation.

1-2 Recommendation

(1) Training of Personnel

Since WAPDA will go into the performance of definite studies, construction supervision, maintenance and operation, etc. of such a large-scaled project as the Lakhra Coal Mining and Coal-fired Thermal Power Station Project, it is recommended that WAPDA as the implementing agency of said Project train personnel who will be engaged in the aforementioned work and related assignments in advance to fully meet the requirements of the Project.

(2) Investigation of Open Pit Mining Area

It is judged necessary that prior to overburden stripping, detailed surveys – together with drilling work be conducted by WAPDA on its own responsibility. In this context, it is recommended that WAPDA take prompt action for the performance of the said work. It should also be borne in mind that the first priority should be placed drilling of the western blocks of Lakhra Coal Mine.

(3) Measurements of Surface Moisture and Size Distribution of Coal

There still exist vague points to be calorified regarding surface moisture of raw coal from Lakhra Coal Mine. In this regard it is considered necessary that re-measurements be made as soon as possible.

(4) Geological Surveys and Preparation of Topographic Maps

It is recommended that in view of the urgency of the Project implementation, WAPDA undertake geological surveys on the proposed Project sites and prepare topographic maps thereof which will be definite studies for the Project. It is advisable for WAPDA to take prompt action for this purpose.

CHAPTER 2 THE PRESENT STATUS OF COAL MINING IN PAKISTAN

2-1 General

The production of coal in Pakistan started at the end of the 19th century. However, with the discovery of superior coal in West Bengal (India) coal mining in Pakistan declined steadily and production decreased remarkably. After Independence the coal mining attracted nation-wide attention suddenly and numerous small-scaled coal mines started operation simultaneously. The government of Pakistan, with the intention of increasing coal production as the major energy source of a newly-born country, began the reorganization of small-scaled coal mines from the economical point of view. The government also entrusted four major mines in the country, Makerwal in Punjub Province and Sharigh, Sor-Range and Degari in Baluchistan Province, with the management of PIDC and later, with the independence of PMDC – as the new organization to execute extensive and intensive investigation and development of mineral resources inclusive of coal and rock salt in 1974, transferred these four mines to PMDC's control.

As a result, the whole coal production in Pakistan increased gradually and reached 1,395,000 tonnes in 1968–69, an increase of approximately 1.9 times compared with the 735,000 tonnes in 1959–60.
(Source: Pakistan Economic Survey 1978–79, published by Government of Pakistan Finance Division)

However, the share of coal in the demand for energy was gradually replaced by the abundance of natural gas discovered in the country; and, moreover, the production decreased by degree due to out-dated mining techniques and the shortage of equipment, and now stands at 1,260,000 tonnes annually – a mere 5 % of total energy supply.

2-2 Occurrence and Quality of Coal

The major coal fields in Pakistan are Khost-Sharigh, Mach and Sor-Range in Baluchistan Province, Makerwal and Salt Range in Punjub Province, and Lakhra and Meting-Thimpjr in Sind Province. Total reserves are estimated at approximately 442 million tonnes, and about half of the reserves are concentrated in the Lakhra coal field.
(Source: Pakistan Economic Survey 1978–79)

The quality of coal in Pakistan is mainly lignite and sub-bituminous and partially bituminous and almost all coal is of poor quality with high sulphur and high ash contents.
(Source: STRATIGRAPHY OF PAKISTAN, published by Geological Survey of Pakistan in 1977)

The major items for each coal field mentioned in the DIRECTORY OF MINERAL DEPOSITS OF PAKISTAN, published by the Geological Survey of Pakistan in 1969, are shown in table 2-1. The data is not up-to-date so that it is inevitable that there are some discrepancies and differences in comparison with the results of recent investigations – especially with those carried out by the JICA survey team for Lakhra coal field in 1979. However, generally speaking, the coal in Pakistan is of poor quality and the coal seams are very thin and steep.

2-3 Production

The coal production in Pakistan has been decreasing steadily over the last few years due to the existence of poor quality coal seams, out-dated mining technology, the shortage of various items of equipment and the poor quality of the coal. In addition, demand has decreased as coal has been replaced by natural gas.

The following table shows the production of the PMDC coal mines and of private mines after 1974–75 fiscal year as given in the Monthly Statistical Bulletin Vol. 28 issued by Statistics Division, Government of Pakistan, in February 1980 as well as in the PMDC Performance Review, issued by PMDC in April 1980.

Unit: thousand tonnes

Sector	1974/75	1975/76	1976/77	1977/78	1978/79
PMDC	312	266	271	241	237
Private	1,002	872	876	1,038	1,024
Total	1,314	1,138	1,147	1,279	1,261

Approximately 60 % of total production comes from Baluchistan, 27 % from the major consuming Provinces of Punjub and North West Frontier, and the remainder from Sind.

The figures show that the production in PMDC coal mines is less than 20 % of total production and more than 80 % is being produced by numerous small-scaled coal mines. However, according to the fifth 5 years plan in PMDC started in 1978, the modernization and expansion of existing coal mines with the introduction of more up-to-date technique is scheduled, as well as production increases with the development of new mines in Punjub and Sind. Further an increased of demand in the coal producing centres is anticipated.

As a result, the production in PMDC is scheduled to increase from an annual 240,000 tonnes as at present to 1,800,000 tonnes by the end of this period.

For this purpose, the following projects are planned or under way.

(1) Development of Sharigh mine to increase the production from an annual 50,000 tonnes to 100,000 tonnes.

(2) The installation of a coal washing plant at Sharigh mine to supply 75,000 tonnes of blended coal for the steel plant at Karach – to mix with imported coking coal.

(3) The development of the Makerwal mine from its annual production of 120,000 tonnes to an output of 300,000 tonnes.

(4) The development of a new coal mine in Lakhra with an annual production of 1 million tonnes and the installation of a new coal fired thermal power station in cooperation with WAPDA.

2-4 Uses

In spite of the rapid increase of the total energy supply in Pakistan (approximately 30 % over the past 5 years) the amount of coal supply is almost the same as before and shares only 5 % of the total energy supply (a decrease of 1.51 % over the same period). These figures apparently show that the increase in energy consumption has been taken up by such other energy resources as natural gas, electricity and oil.
(Source: Pakistan Economic Survey 1978–79)

Approximately 95 % of total coal production in Pakistan is now consumed in the Punjub and the North West Frontier, and most of it is used mainly in brick burning, lime burning, briquette plants, small industries, military and domestic fuel and partially in railway locomotives and power stations.

The major reasons now disturbing the development of the coal mining industry in Pakistan are the easy friability and spontaneous combustibility of mined out coal produced in Baluchistan, which is the major coal producing area in Pakistan, and, in addition, the poor availability of railway wagons provided by the Pakistan National Railway. These are impeding the smooth transportation of coal from Baluchistan to each consuming province and are also the major reasons why the demand for coal is being eroded by natural gas and oil, etc.

2-5 PMDC Coal Mines

As mentioned before, PMDC possesses four major coal mines in Pakistan, Makerwal, Sharigh, Sor-Range and Degari, which are now producing 240,000 tonnes of coal annually. They are the typical coal mines in each coal field in this country and are relatively well mechanized as far as Pakistani coal mines go. Among them Makerwal and Degari mines were investigated by a JICA survey team and general contribution of them are shown in Table 2-2 and Table 2-3.

2-5-1 Makerwal Coal Mine

Makerwal is one of the oldest mine in Pakistan. After its nationalization in 1949, the mine came under the control of PIDC in 1954. However, when PMDC became independent from PIDC in 1974, the Makerwal mine was transferred to PMDC.

The annual production at the time of transfer to PIDC was only 60,000 tonnes but as the result of efforts for the improvement of the mining method and mechanization, the production increased up to 200,000 tonnes in 1966. However, the development is making slow progress due to the presence of gigantic faults with 200 m of displacement, so that the production is decreasing gradually and remains at only 400 tonnes per day against 1,000 tonnes as planned at present. Economically speaking, however, Makerwal coal is still gaining 90 Rupees per tonne of profit.

There is only one seam yielding coal. Its height ranges from 0.6 m to 2.3 m, and it lies with an inclination of 10 to 32 degrees. Many faults, especially the aforementioned big fault with a displacement of 200 m and its secondary faults are severely disturbing the underground development. The coal winning system is mainly room and pillar with manual extraction and only one longwall with 55 m of face is also under operation. The coal winning and road heading works are carried out under the contractor system and the workers are employed and distributed by the contractors. Except the steel arch supports in some main entries, wooden supports are in use in general. In addition, the quality of wooden supports is poor and the humidity underground is very high. Therefore, the roof and side walls are pushed out frequently.

In addition to the small ventilation fan, natural ventilation is also being utilized in this mine. However, the quantity of air is remarkably insufficient. Therefore, the temperature and humidity are very high and ventilation is bad.

The transportation in Makerwal is by manpower and donkey for the underground level roadway, small hoists for inclines, and gravity powered ropeways, diesel locomotives and camels for the surface. No man riding vehicle system is utilized.

The biggest problem in Makerwal coal mine is the ventilation. It is apparently impossible to supply fresh air sufficient for all the underground workers. Furthermore, the underground temperature is high due to the subterranean heat and high humidity. Floating coal and rock dust also affect the condition underground. Therefore, it is recommended that the ventilation be improved as urgently as possible with the installation of a fan of adequate capacity and thorough reconstruction of ventilation circuits. Otherwise the operation of the mine will be unavoidably interrupted in the near future by ventilation problems.

Reconsideration of the supporting system is also very important subject. It is recommended that arch type and/or other steel supports be used for all the main entries which are expected to remain in use for many years. In the other roadways supports with wooden materials, continual inspection and repair work will be necessary.

2-5-2 Degari Coal Mine

Degari is also a very old mine and was transferred repeatedly to and from the private and public management and in 1960 passed under the control of PIDC. After that it was transferred to PMDC when the latter became independent in 1974.

From February 1961 up to March 1971 Japanese engineer(s) had stayed at Degari and planned and conducted the development of the mine and the construction of equipment. As a result the production was increased to 70 tonnes per day in 1964 from only 30 tonnes per day in 1961. Moreover, it reached 500 tonnes per day in March, 1968. However, the production decreased gradually from the annual peak of 100,000 tonnes in 1969–70 to 70,000 tonnes in 1978–79 because of the gradual thinning of coal seams and the increasing difficulty of transporting coal and material due to the increasing depth of the mining faces as well as the decline of the coal market. However, Degari is also enjoying approximately a profit of Rupees 90 per tonne the same as the Makerwal mine.

Two yielding coal seams are approximately 1 m each in height and 45 to 50 degrees in inclination, so-called steep seams. The coal winning system is the retreating long wall system with 60 m of face length and manually won, and extracted coal flows along the face by gravity. The coal winning and road heading works are the same as Makerwal, carried out under the contractor system. With the deepening of mining faces the travelling time of workers from the portal to the face has increased remarkably, and at present it takes approximately one hour through 45 degrees of steep incline because of no transportation facilities. It causes, therefore, the decrease of actual working hours at the face and the increase of weariness of workers.

The supporting materials are all wood. However, no problems appear because of the very stable roof and wall. Only a small ventilation fan is used but it is now supplying an adequate quantity of fresh air for the working faces. Transportation is carried out by manpower for the horizontal roadway and by small hoists for the inclines.

The biggest problem in Degari is transportation. Particularly, as mentioned before, the lack of transportation for workers has caused a decrease in productivity. Therefore, it is urgently recommended that the man riding vehicle be provided to maintain the working hours at the faces, to increase productivity and prevent fatigue of the workers. In addition, reconsideration will be required for transportation of the coal and materials to achieve a mine efficient and cheaper operation.

Furthermore, it will be necessary to endeavour to increase productivity with sufficient consideration for the conbination of coal winning and packing works and also most proper manning and operation plans.

TABLE 2-1 MAJOR COAL FIELD IN PAKISTAN

Province	Coal Field	Classification	Seam Thickness (m)	Note
Baluchistan	Khost-Sharigh	Sub Bituminous	0.1 - 1.5	Area: app. 210 km^2 Probable Recoverable Reserves: app. 40 million tonnes Dip: Steepy Quality: Moisture; 10%, VM; 35-45%, Ash: 9-35%, S; 5-7%, Cal. Value; 4,700-6,900 kcal/kg
	SorTange-Degari	Sub Bituminous	0.1 - 3.0	Area; app. 50 km^2 Probable Recoverable Reserves; app. 5.3 million tonnes Dip: more than 45° Quality: Moisture; 16-19%, VM; 34-40%, S;0.5-6%, Cal. Value; 5,000-6,000 kcal/kg
Punjub	Makerwal	Sub Bituminous	0.6 - 3.0	Area: app.80 km^2 Estimated Reserves: app. 1.9 million tonnes Dip: 30° Quality: Moisture; 4-7%, VM; 37-45%, Ash 7-21%, S; 4-6%, Cal.Value; 5,300-6,600 kcal/kg
	Salt Range	Sub Bituminous	0.1 - 1.5	Area: app. 260 km^2 Estimated Reserves: app. 7.5 million tonnes Dip: Steepy Quality: Moisture; 3-7%, VM; 26-39%, Ash; 12-38%, S: 4-11%, Cal. Value; 3,900-6,000 kcal/kg
Sind	Lakhra	Lignite	0.1 - 1.0	Area: app. 200 km^2 Estimated Reserves: 2,540 million tonnes Dip: less than 2° Quality: Moisture; 32-36%, VM; 28-31%, Ash; 7-11%, S; 3-6%, Cal. Value; 3,900-4,300 kcal/kg
	Meting-Jhimpir	Lignite	0.1 - 1.0	Area: app. 900 km^2 Estimated Reserves: 28 million tonnes Dip: Horizontal Quality: Moisture; 15-20%, VM; 30-40%, Ash; 8-15%, S; 3-7%, Cal.Value; 4,100-5,400 kcal/kg

TABLE 2-2 MINING CONDITION IN MARKERWAL AND DEGARI

Description	Makerwal	Degari
Annual Production (t)	103,000	80,000
Worker		
Underground	200	600
Surface	600	200
Total	800	800
O.M.S.		
Face	1.14	0.75
Total	0.43	0.33
Seam Thickness (m)	0.6 - 2.3	1.0
Inclination (degree)	10 - 35	45 - 50
Quality (%)		
Fixed Carbon	35 - 40	45 - 48
Volatile Matter	35 - 40	47 - 48
Ash	14 - 30	3.2 - 6.8
Sulphur	4.7 - 30	1.9 - 2.2
Calorific Value	5,600	6,800
Coal Classification	Sub Bituminous	Sub Bituminous
Coal Mining Method	Room & Pillar	Longwall with Packing
User	Brick Maker	Brick Maker
Selling Price (Rs/t)	333.46	294.42
Cost (Rs/t)		
Wage	162.96	134.50
Material & Equipment	66.15	44.16
Administration	19.80	22.35
Total	248.91	201.01

TABLE 2-3 MAJOR FACILITIES IN MAKERWAL AND DEGARI

Description	Markerwal	Degari
Workshop	Mechanical & Electrical	Mechanical
Ventilation Fan	50,000 cfm x 1 30,000 cfm x 1 Others	26,000 cfm x 2 Others
Air Compressor	1,200 cfm x 4 660 cfm x 2 Others	3,200 cfm x 2
Winder	100 kW x 1 75 kW x 1 50 kW x 1	300 HP x 1 75 kW x 1 50 HP, 20 kW
Locomotive	10 t Diesel Locomotive x 5	
Coal Cutter	60 HP x 7 93 kW x 5	
Drainage Pump	High Pressure Pump x 1 Low Pressure Pump	Low Pressure ·
Substation	66 kV/11 kV, 1,500 kVA	11 kV/3.3 kV, 1,500 KVA
Power House	1,150 HP x 1 325 kW x 1	500 kVA x 2
Others	Safety Lamp Telephone Exchanger	Safety Lamp Telephone Exchanger