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A method of mining the Kumtepe lignite deposit of Turkey

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A METHOD OF MINING

THE KUMTEPE LIGNITE DEPOSIT OF TURKEY

BY

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A

THESIS

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INTRODUCTION

The Agacle lignite deposit, as a potential source of fuel, near Istanbul, Turkey has been used in this study to apply the principles of strip mining operations. The details of the deposit upon which this thesis is predicated will be presented later in this paper.

The fuel requirement of the city of Istanbul with a population of nearly a million is considerably greater than can easily be met. Some of the fuel needs are accommodated by electrical energy by producer gas, by bituminous coal brought from Zonguldak (about 200 miles from the city), by charcoal, and by fuel wood. During the winter months, the lack of readily available fuel becomes critical and semi-rationing is necessary. This paper is devoted to an analysis of the lignite deposit which is presently undeveloped but which, being only 30 miles northeast of Istanbul and containing 8 million metric tons, may serve ultimately to supplement the fuel supplies of the city. A small portion of this deposit has been studied with a view toward exploitation by open-cut mining methods.

There are other scattered lignite deposits in Turkey. The problem of fuel in central Turkey is similar to that of Istanbul. The dearth of private capital and the seeming lack of initiative on the part of the government has delayed the development of these deposits. The government recently took positive steps toward the utilization of domestic lignite.

The underlying reasons for this recent movement are (1) to provide inexpensive fuel, (2) to substitute lignite for wood in cities and towns in order to conserve forests, (3) to supply raw material for the other industrial uses which might be stimulated if sufficient quantities of lignite are assured, (4) to substitute lignite for coking coal in domestic uses. This substitution saves coking coal for industrial uses and possible export in the event there is a surplus to secure foreign exchange and stimulate foreign trade.

All units will be expressed in this paper, unless otherwise stated, in terms of British units. For the Turkish reader conversion factors will be given. It should also be stated that in this study only the major phases of stripping methods, disposal of waste, transporting of coal, method of determining the cost per cubic yard of production, and the general maintenance of the mine will be discussed.

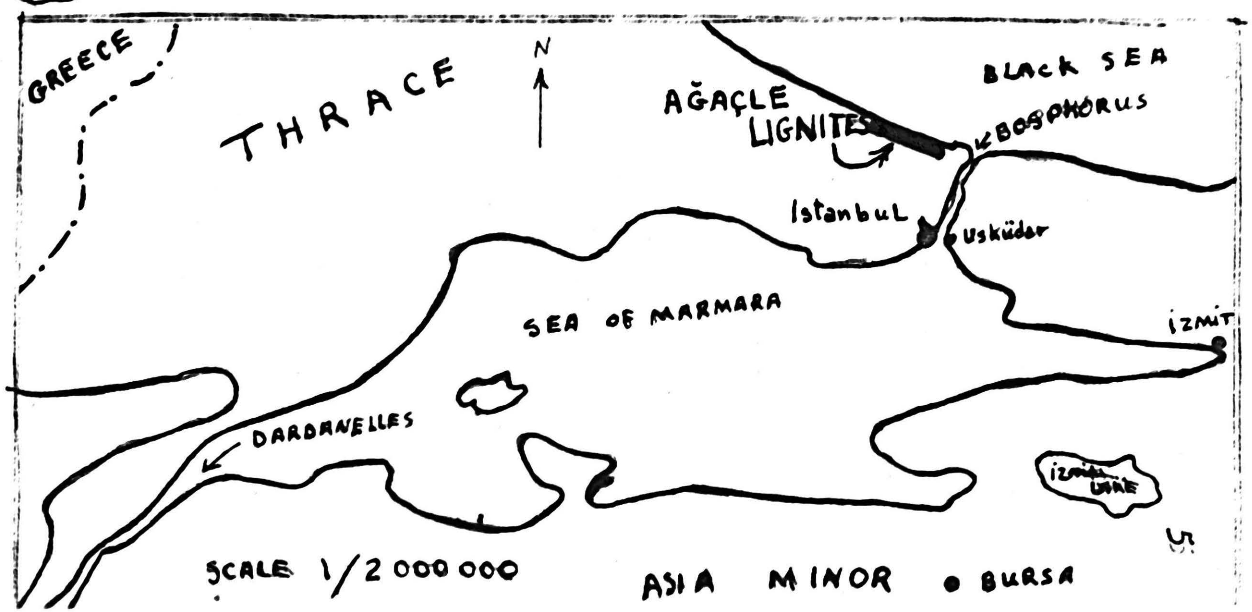
REVIEW OF LITERATURE

Most of the literature published by leading mining magazines is of a descriptive nature. The magazines which frequently feature articles on strip mining are Coal Age, Mechanization, The Excavative Engineer, and The Mining Congress Journal. The discussion of current strip mining problems by leading coal operators at the Mining Congress Journal Convention are published in the "Coal Mine Modernization Year Book", which also contains papers on other phases of the coal industry such as underground operations, safety, maintenance, and management.

The United States Bureau of Mines publications on strip mining are also descriptive, but study of these publications gives only a general picture. The Bureau of Mines Bulletin 298 "Methods, Costs, and Safety in Stripping and Mining Coal, Copper Ore, Iron Ore, Bauxite, and Pebble Phosphate" by E. R. Cash and M. W. Von Bernewitz discusses in detail the development of strip mining in the United States. Albert L. Toenges and Frank A. Jones in Report of Investigations 3416 compare truck and rail haulage in bituminous coal strip mines. J. R. Thoemen and his associates made a series of fine studies on shovel loading, truck haulage, and drilling in quarries. The results of these studies, which are suggestive of methods of increasing efficiency in strip mine operations, are published in Reports of Investigations numbers 3461, 3467, and 3502.

A technical manual TM 5-252 "Use of Road and Airdrome Construction Equipment" by the War Department and the unpublished thesis "Principles and Practices Controlling the Use of Earthmoving Equipment" by W. T. Latvala are excellent reference works on performance, limitations, and quick estimation of output of heavy equipment.

Map. I
INDEX MAPS
of
AĞAÇLE LIGNITES



THE DESCRIPTION OF AGACLE LIGNITE DEPOSIT

Location

The Agacle lignite deposit is located between 30 and 35 miles northeast of Istanbul (formerly called Constantinople), Turkey. The deposit lies along the shore of Black Sea, as shown on map no. 1.

Geography

In general, the regional topography of the district consists of small gently rolling hills which are cut by small, seasonal creek channels. A portion of the Agacle lignite deposit which is suitable for stripping is located on one of these hills. This hill, Kumtepe, (meaning sandy hill) is in the center of the Agacle deposit.

Map no. 2 shows the general topography of Kumtepe. It can be seen from the map that the trend of the hill is northeast, and it is bordered on the west by Agacle Creek and on the east by Havakadin Creek. Both creeks discharge into the Black Sea. The average slope of the hill toward these creeks and toward the Black Sea is about 8 per cent. The slope of the hill is somewhat less in the westerly and northerly direction than in the easterly direction. The maximum elevation of the hill at triangulation point 6A is 94.47 meters (310 ft.).

The vegetation on the hill consists of grass, small bushes, and scattered trees. In the left lower and left upper corner of the area shown on the map the trees become more dense.

Climate

In general, the climate in Istanbul and vicinity is not severe enough to hinder stripping operations, except during the months of December, January, February and the first part of March. The rain and snow of these months will seriously hamper, if not stop, the operations. Throughout the rest of the year, mining operations can continue without serious interruptions. The mine should be able to operate 250 days annually working 6 days a week.

Geology

A few studies have been made of the geology of Thrace, the Euopean portion of Turkey, in which the Agacle lignites are found. Thrace is a comparatively flat basin bounded on the north and south by mountains. The central plains area is almost entirely covered by continental Pliocene deposits. It is believed that subsidence occurred at the end of Cretaceous lime. Miocene formations lie discordantly on the Oligacene formations. A second subsidence which took place during the Miocene was regional and less pronounced. Shallow seas with lacustrine and continental conditions prevailed, and thus deposits of sandstone, lignite, and varicolored clays and shales were formed. The Pliocene deposits are mostly continental.

Prospecting at Kumtepe

There is no record of the exact date of the discovery of lignite in this area. The Mining Research and Exploration Institute of Turkey, here after referred to as M. T. A., completed the prospecting at Kumtepe in March 1950. Total

Table II

Summary of Prospecting Results and Stripping

Ratios at Each Hole (Elevations, Thickness, are in meter)

Block	Hole no.	Elev. of Collar of Hole	Bed no. 1	R_1	Bed no. 2	R_2	Bed No. 3	R_3	Remarks
1	225	8.16	1.01/0.80	8.9					
	202	9.68	2.68/1.60	4.4					
2									Insufficient data
3.	221	18.54	.24/ .27	68					
	222	12.14	5.44/ .15	44.5	4.50/1.75	.54			
	223	15.26	1.46/.65						
4.	173	21.90	6.90/1.00	15					
	141	23.73	12.23/1.00	11.5	9.73/1.00	2.5			
5.	121	28.71	19.21/1.50	6.3	12.21/ .50	14	8.71/1.00	3.5	
	122	31.13	18.63/3.50	3.5	10.13/1.00	8.5			
	125	33.69	15.69/2.00	9	9.19/ .20	28			
	133	24.60	15.60/2.00	4.5	8.60/1.00	7	.60/1.00	6.5	
	137	28.93	15.43/1.50	9	9.43/ .70	8.5	3.43/ .50	11	
	136	18.48	7.48/1.00	11	3.48/ .50	8			
	135	29.15	16.15/2.50	5.2	9.15/ .50	16			
	163	17.20	7.70/ .75	12.5	3.20/ .50	8			
	164	23.65	7.65/ .75	21	3.65/ .50	8			
	166	29.72	15.72/2.00	7	8.72/ .50	14			
	134	28.93	15.43/1.50	9	9.43/ .70	8.5	3.92/ .50	11	
	154	21.67	7.67/ .50	26	1.51/ .50	13			
	124	28.06	17.06/3.00	2.7	10.06/ .50	14	5.56/ .50	11	
6.	158	32.06	15.06/ .50	34	9.06/ .50	12	5.06/ .50	8	
	167	32.11	17.11/ .50	30	11.11/ .50	12			

Table III (cont'd)

Block	Hole no.	Elev. of Collar of Hole	Bed no. 1	R_1	Bed no. 2	R_2	Bed no. 3	R_3	Remarks
7	113	32.52	24.52/2.50	3.2	11.02/1.50	9			
	102	37.33	29.33/2.50	3.2	22.33/2.00	3.5	16.33/ .50	12	
	106	31.09	22.59/2.50	3.8	12.59/3.00	5.0	7.59/1.50	3.3	
	105	27.36	20.36/1.50	4.6	13.36/1.00	7.0	13.36/1.00	3.0	
	111	28.73	15.73/1.50	8.7	11.23/ .50	9			
	229	39.99	24.69/1.85	8.3	17.62/ .60	11.8	10.49/ .40	18.8	
8	138	35.64	26.64/ .25	36	19.64/2.00	3.5	11.64/ .50	16	A fourth Bed at 8.14/1.00
	140	33.40	20.00/3.00	4.6	13.00/ .50	14	8.50/1.50	3	
	104	31.81	21.31/2.00	5.0	9.81/2.00	5.5			
	139	40.42	19.42/2.50	8.3	11.92/ .50				
	126	39.84	21.81/2.50	7.3					
132	132	37.13	22.13/2.50	6.0	13.13/1.50	6.7			
	172	41.44	20.44/2.00	10.5	13.44/0.50	14			
	127	44.77	25.77/2.00	8.5	10.77/1.00	15.00			
	129	40.45	22.45/2.00	9.0	16.95/1.50	3.6	10.45/ .50	13	
	230	40.25	24.45/2.10	7.5	15.97/ .18	53	12.20/ .45	8.4	
	228	41.98	24.93/1.75	15.5	16.68/ .20	41	12.58/ .49	8.4	
	227	40.06	25.41/1.70	8.6	17.56/ .50	15.7	13.06/ .35	41.5	
	128	51.42	26.42/3.00	8.3	18.42/1.00	8.0			
	179	48.56	31.56/1.00	17.0	26.06/ .50	11	23.56/ .50	5	A fourth bed at 20.06/ .50
9	165	37.17	20.67/2.00	8.2					
	170	32.68	18.18/2.00	7.3	5.68/ .50	25			
	171	30.98	10.98/ .50	40					
	177	41.80	20.80/2.00	10.5	13.30/ .50	15			
	174	41.20	22.20/2.00	9.5	14.20/ .50	16			
	175	39.91	24.41/2.00	7.5					
	173	36.02	22.02/ .25	56	17.17/ .50	9.6	13.77/ .50	6.8	
	176	47.18	25.18/2.00	11	18.68/ .50	13			

Table III (cont'd)

Block	Hole no.	Elev. of Collar of Hole	Bed no. 1	R_1	Bed no. 2	R_2	Bed no. 3	R_3	Remarks
10	Insufficient data available								
11	115	37.15	26.15/1.00	11	24.65/1.00	1.5	20.65/1.00	4	
	145	43.67	35.17/2.00	8.2	27.17/ .50	16	22.67/ .50	9	
12	114	39.39	32.39/2.50	2.8	23.89/ .50	17	19.89/ .50	4	
	131	42.53	27.50/2.50	6	19.53/ .25	32	14.73/ .25	19	
	116	43.93	32.93/3.00	3.7	23.93/1.00	9	20.93/1.00	3	
	142	44.80	33.80/2.50	4.4	26.80/ .50	14	25.80/ .50	2	
	143	49.20	34.20/2.50	6	26.70/1.00	7.5	21.20/ .50	11	
	144	49.63	31.63/2.50	7.4	23.13/1.00	8.50	18.13/ .50	10	
	117	52.33	31.33/3.00	7.0	23.83/1.00	7.50			
	148	51.05	37.55/2.00	6.7	30.05/1.00	7.50	25.05/ .50	10	
	153	64.71	36.71/2.00	14.0	30.71/3.00	2.00	16.71/ .50	28	
13	227	46.06	25.01/1.70	12	17.56/ .50	15	13.06/ .35	13	
	155	51.94	28.44/1.50	15	26.44/ .50	4	20.44/1.00	6.0	A fourth Bed at 15.94/ .50
	152	51.13	27.13/3.00	8	19.63/2.50	3	15.13/ .50	8	
	180	53.01	33.01/ .50	40	26.51/ .50	12			Not in stripping limits
	130	54.43	28.93/2.00	13	22.43/1.50	4.3	16.93/ .50	11	
	182	60.20	32.20/ .50	56	28.20/2.00	2	14.20/ .50	28	
	183	66.49	34.49/2.00	16	27.49/ .50	14	22.99/ .50	9	
14	178	49.77	29.77/2.00	10	22.27/ .50	15			
	181	56.77	41.77/1.00	15	30.77/ .50	22			
	215	61.80	43.96/ .15	112	40.31/ .95	3.8	32.93/1.74	4.2	
	213	52.97	40.67/1.05	11.7	32.97/1.90	4.6	22.77/0.45	16	
	212	45.40	30.95/1.85	7.8	23.52/ .34	22	18.16/0.30	18	
	205	45.00	28.10/2.08	8.1	20.55/ .80	9.6	15.65/ .60	8.3	

Table III (cont'd)

Block	Hole no.	Elev. of Collar of Hole	Bed no. 1	R_1	Bed no. 2	R_2	Bed no. 3	R_3	Remarks
15	203	35.50	25.15/1.65	6.3	18.96/ .26	24	14.37/ .19	24	
	204	32.13	25.87/2.10	3	32.97/ .17	17	19.79/ .60	5.2	
16									
17	150	62.96	42.46/ .50	41	35.96/2.00	3.2	27.96/ .50	16	
	151	70.73	34.73/1.00	36					
18	184	77.27	39.77/2.00	18					
	185	72.67	40.67/1.00	32	33.67/2.00	3.5	25.67/ .50	16	
	217	68.79	41.49/ .15	182	39.39/ .80	2.6	33.39/1.75	3.4	
19	216	62.40	34.10/2.10	13.5	25.80/ .50	16.6	20.40/ .30	18	
	214	54.00	42.00/ .60	13	34.30/1.80	4.3	26.46/ .25	31.6	
	233	47.67	33.82/1.95	7.1	27.37/ .50	12	22.09/ .67	8	
	218	62.09	43.69/ .60	32	35.63/1.59	5.1	28.29/ .60	12	
20	Insufficient data available								
21	189	32.24	24.24/1.00	8	15.74/ .50	17	10.24/ .30	18	
	194	32.39	16.39/ .40	40					
	188	45.52	31.02/1.50	9.7	29.2/1.15	1.5	23.52/ .80	7.2	
22	198	52.04	34.74/2.15	7.8	27.54/ .30	24			
	197	52.72	30.34/2.02	11	23.48/ .50	14	17.16/ .25	25	
	199	66.63	45.73/ .50	41.8	32.24/1.55	8.7			
	250	64.67	50.37/ .50	29.6	48.01/ .10	23.6	38.41/1.45	6.6	

lignite reserves in the Agacle Region, including Kumtepe, is estimated to be 8 million metric tons. The prospect holes were drilled with Bravo hand drills and gasoline driven Sullivan and Longyear type drills.

The holes were placed at random, but their coordinates can be determined from the 200 meter grid system of map no. 3. In all, 239 holes were drilled. The most promising grid squares were chosen from the study of log of drill holes. Each square is designated by a number in its NW corner. For convenience these squares will be referred to as Block no. 1, Block no. 2, . . . Block no. 20.

Map no. 2 shows the area of Kumtepe which was most thoroughly prospected. The rest of the Kumtepe was inadaquately explored, so that the resulting data do not justify an estimation of reserves. The pertinent drill hole data are summarized in Table (I). An explanation of Table (I) follows, using hole no. 121 in Block no. 5 as an example:

28.71 m	Elevation of the collar of the hole
19.21/1.50	Bed no. 1 at an elevation 19.21 meter is 1.50 meter thick
6.3	Ratio of overburden thickness to the thickness of coal (stripping ratio)
12.21/.50	Bed no. 2 at an elevation 12.21 meter is .50 meter thick
14	Stripping ratio for Bed no. 2 (it does not include the overburden of Bed no. 1)
8.71/1.00	Bed no. 3 at an elevation 8.7 1 meter is 1.00 meter thick

3.5 Stripping ratio for Bed no. 3 (it does not include the overburden of Bed no. 1 and Bed no. 2.)

Bed no. 1 is the uppermost bed; Bed no. 2 is the intermediate bed, and Bed no. 3 is the lower bed. The average thicknesses of the beds, computed from the data given in the Tables(I) are 1.6 meter (5 ft.) 1.25 meter (4.2 ft.), and 0.88 meter (2.9 ft.), respectively.

In some of the holes the presence of one or two of the three known beds may not be indicated although the existence of these beds is evident from the adjacent holes. This is probably the result of poor core recovery. Considerable difficulty was experienced in attaining good core recovery because of hole cavings.

The average dip of the beds is about one degree to the NE. Though on the accompaning cross-sections, the apparent dip is shown as five degrees because the horizontal and vertical scales are different. However, for the purpose of simplifying the computations the beds have been considered to be horizontal. An additional feature worth noting on the cross-sections is the undulation of the lignite beds. This undulation is more or less similar in each bed. This feature can be detected also if the isopachs of the overburden for Bed no. 1 on map no. 4 are studied. The island-like closures correspond almost identically with the basins or small hills formed by the undulations.

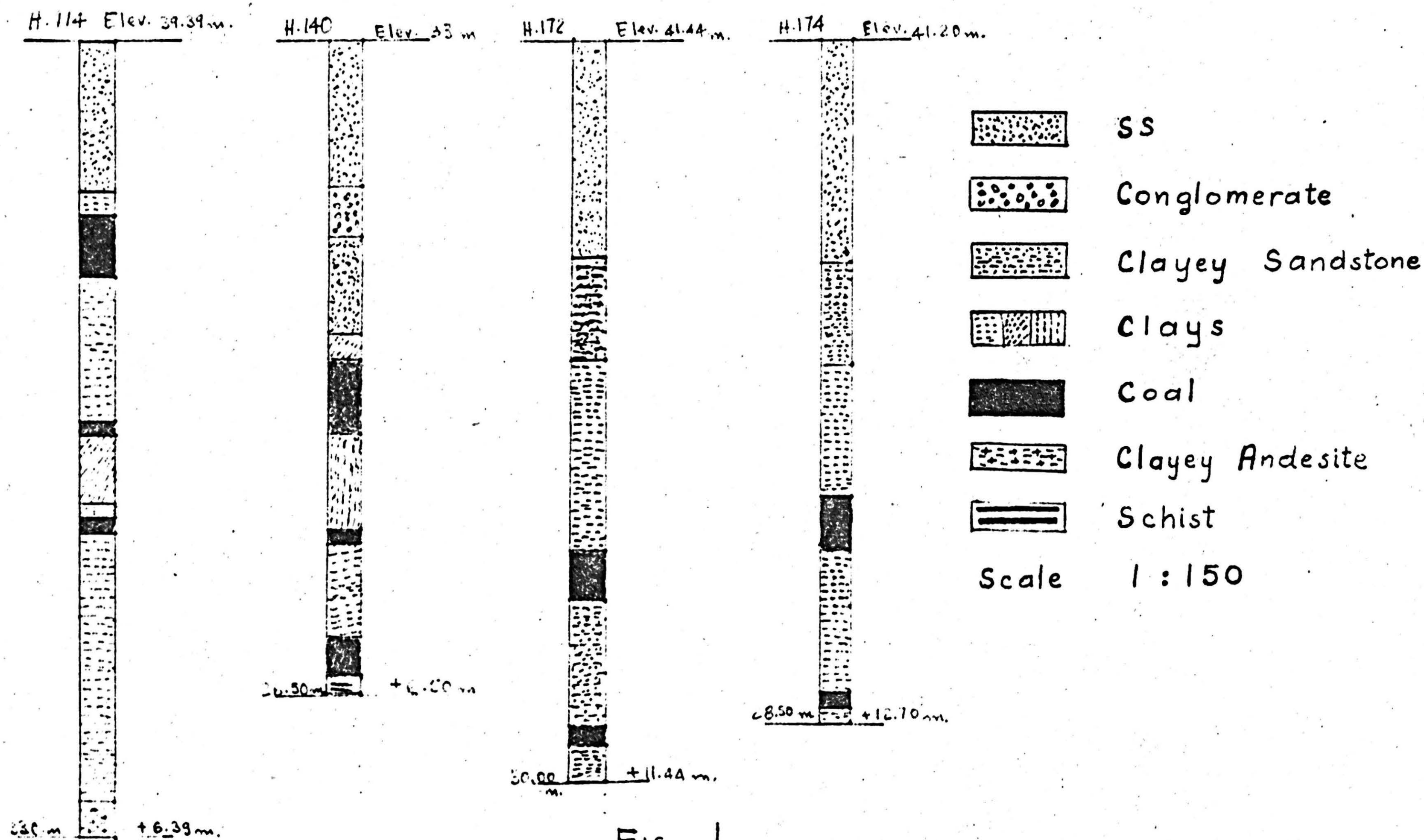


Fig. 1
Typical Columnar Sections

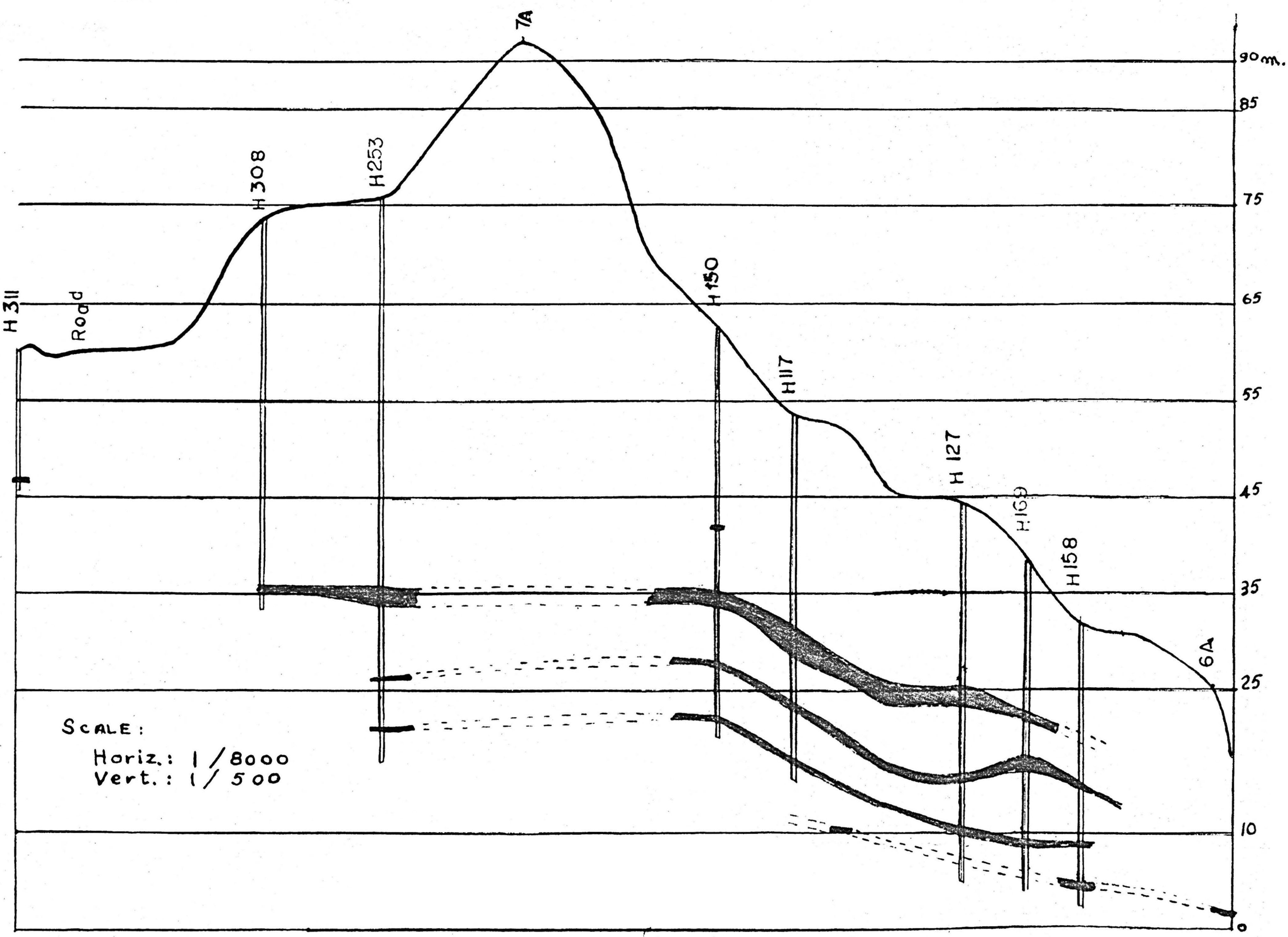


FIG. 2
Crossection Through 6A - 7A
Looking NW

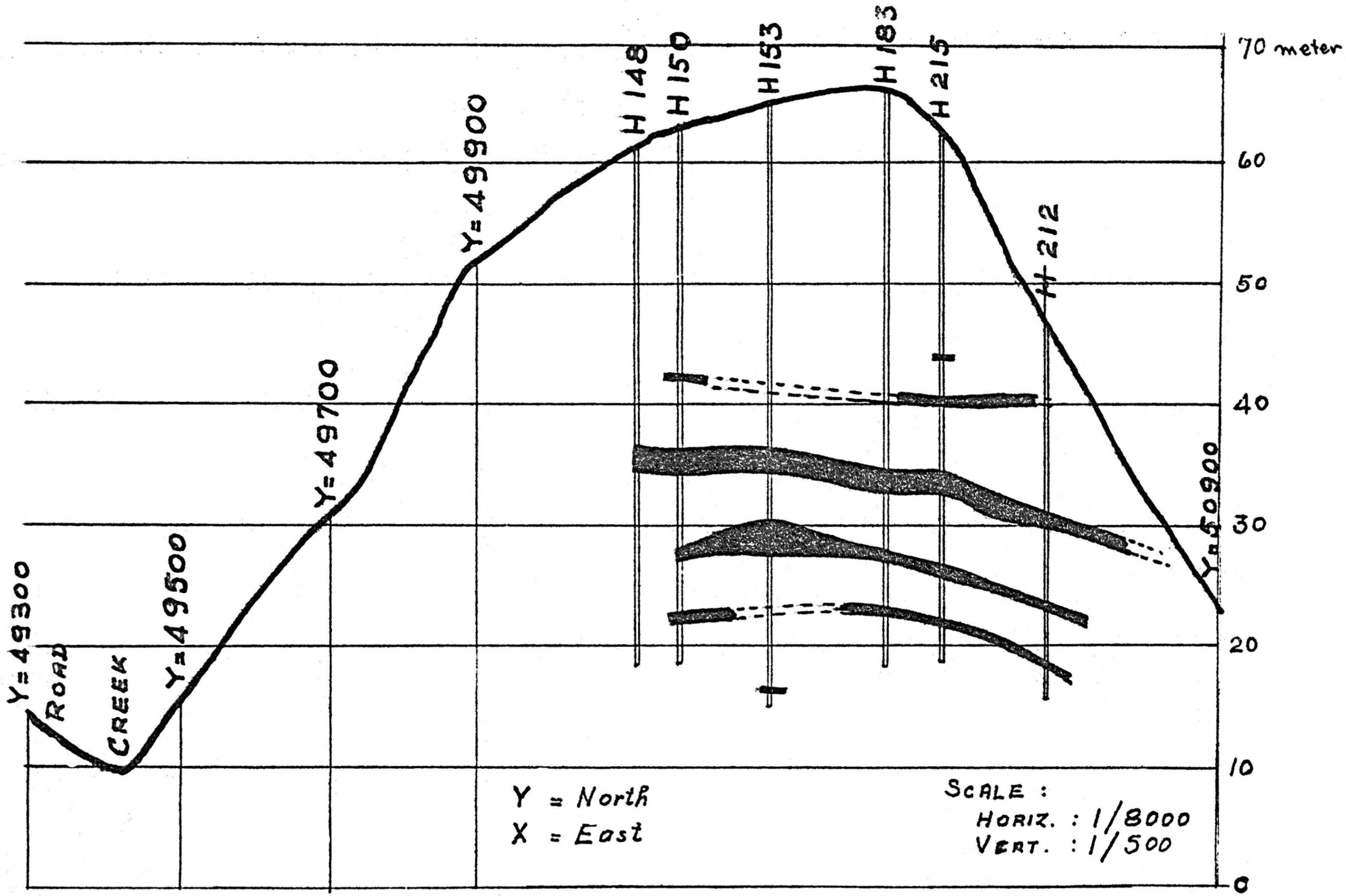


FIG. 3

CROSS-SECTION THROUGH X = 50300
LOOKING NORTH

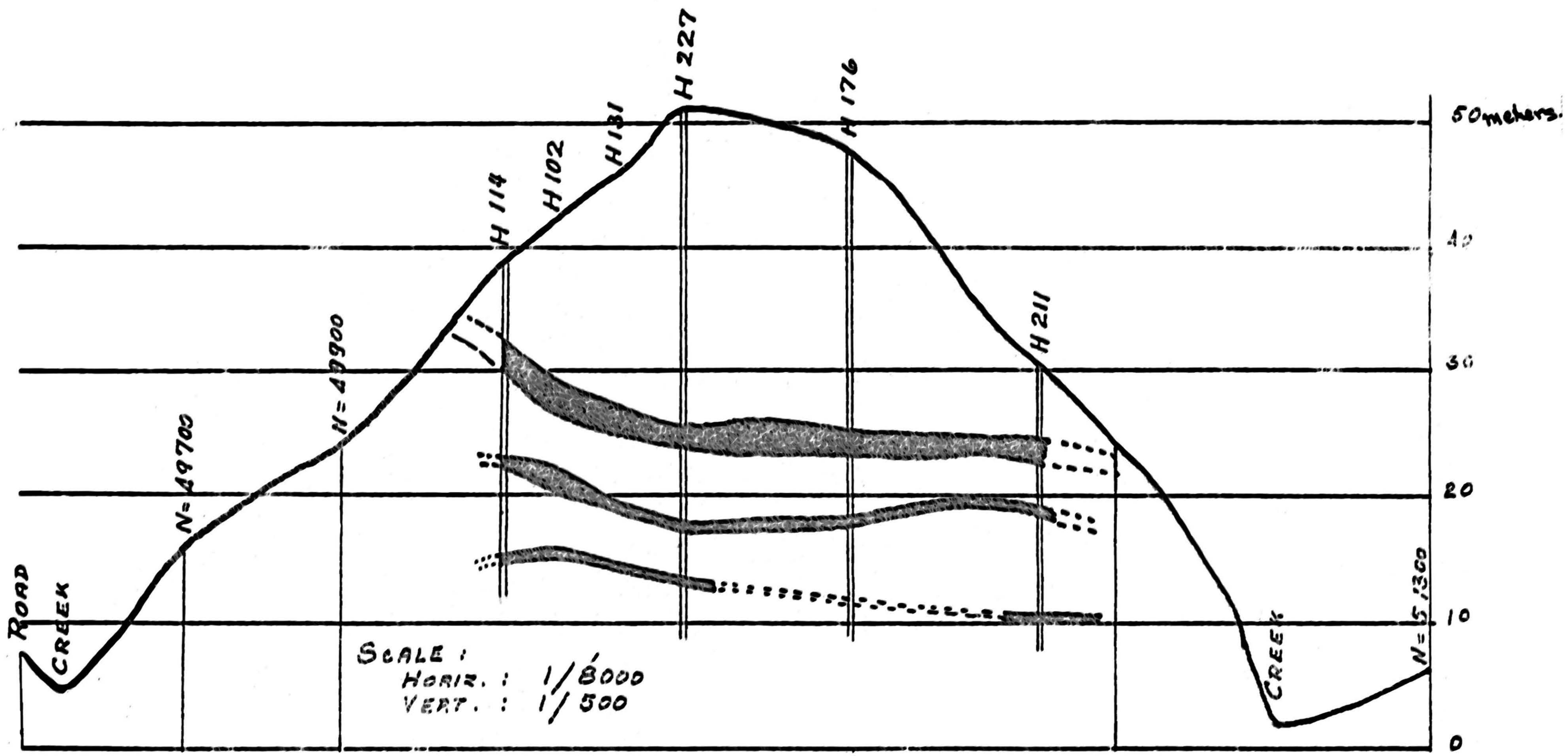


FIG 4
CROSS-SECTION THROUGH
EAST=50500 LOOKING NORTH

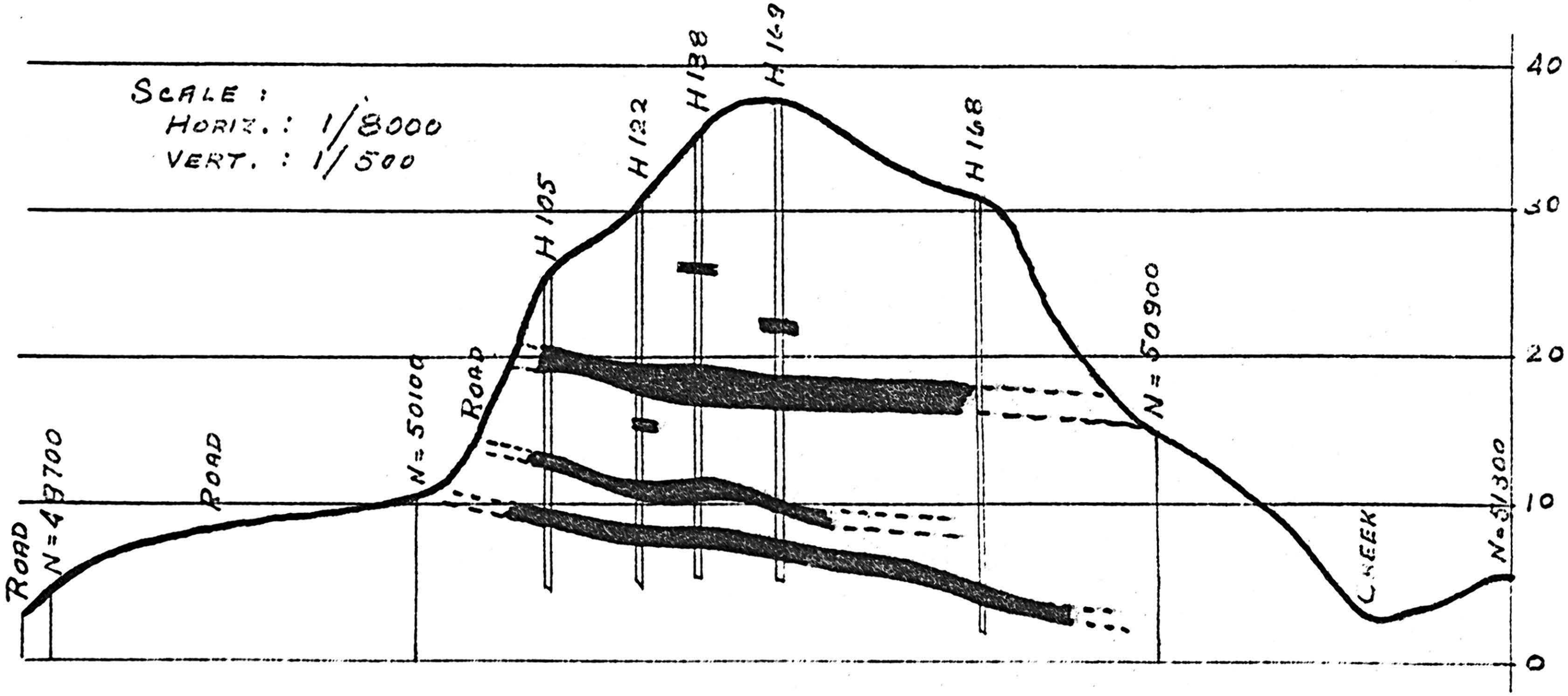


FIG. 5

CROSS-SECTION THROUGH EAST=50700
LOOKING NORTH

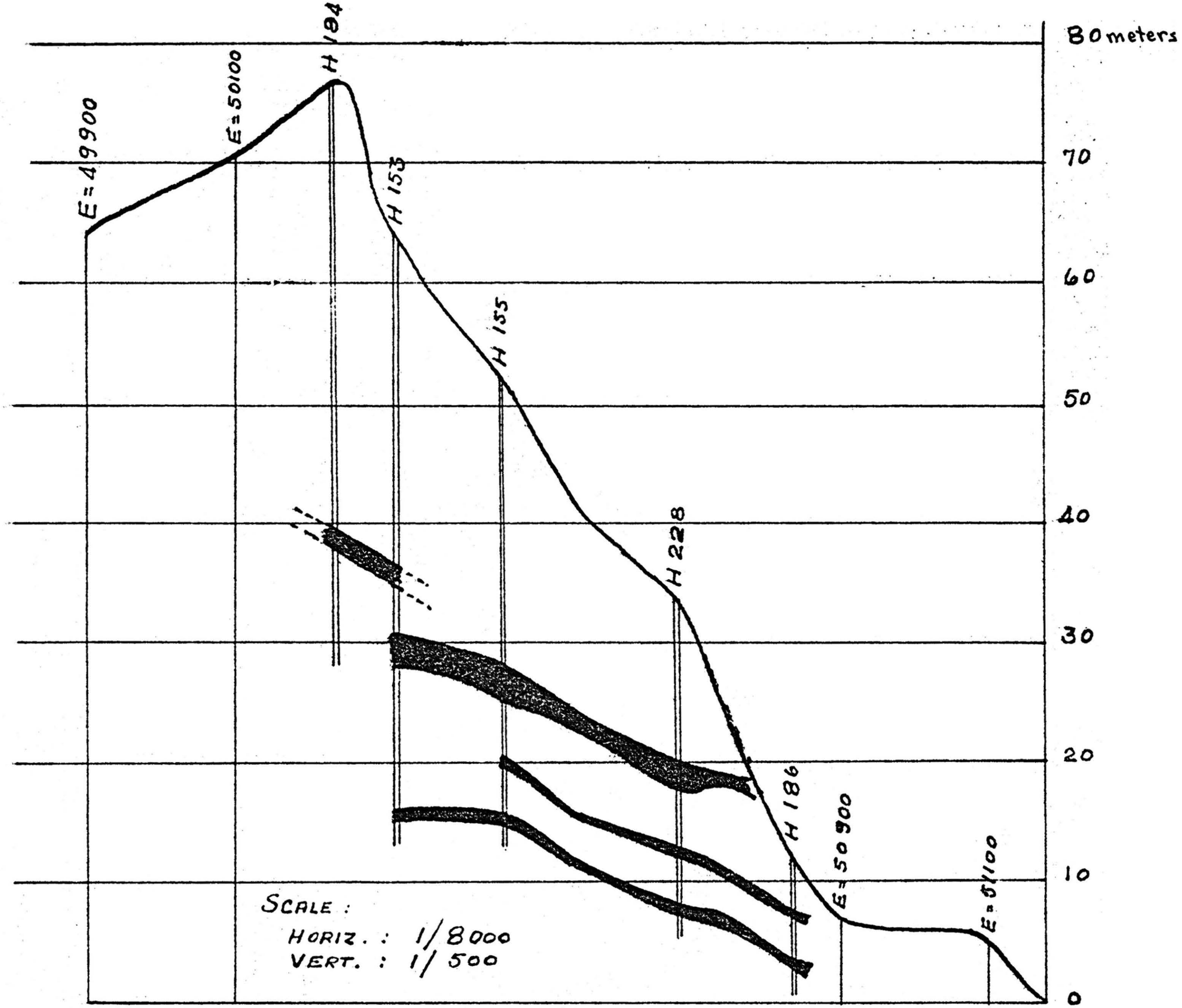


FIG. 6

CROSS-SECTION THROUGH NORTH=50300

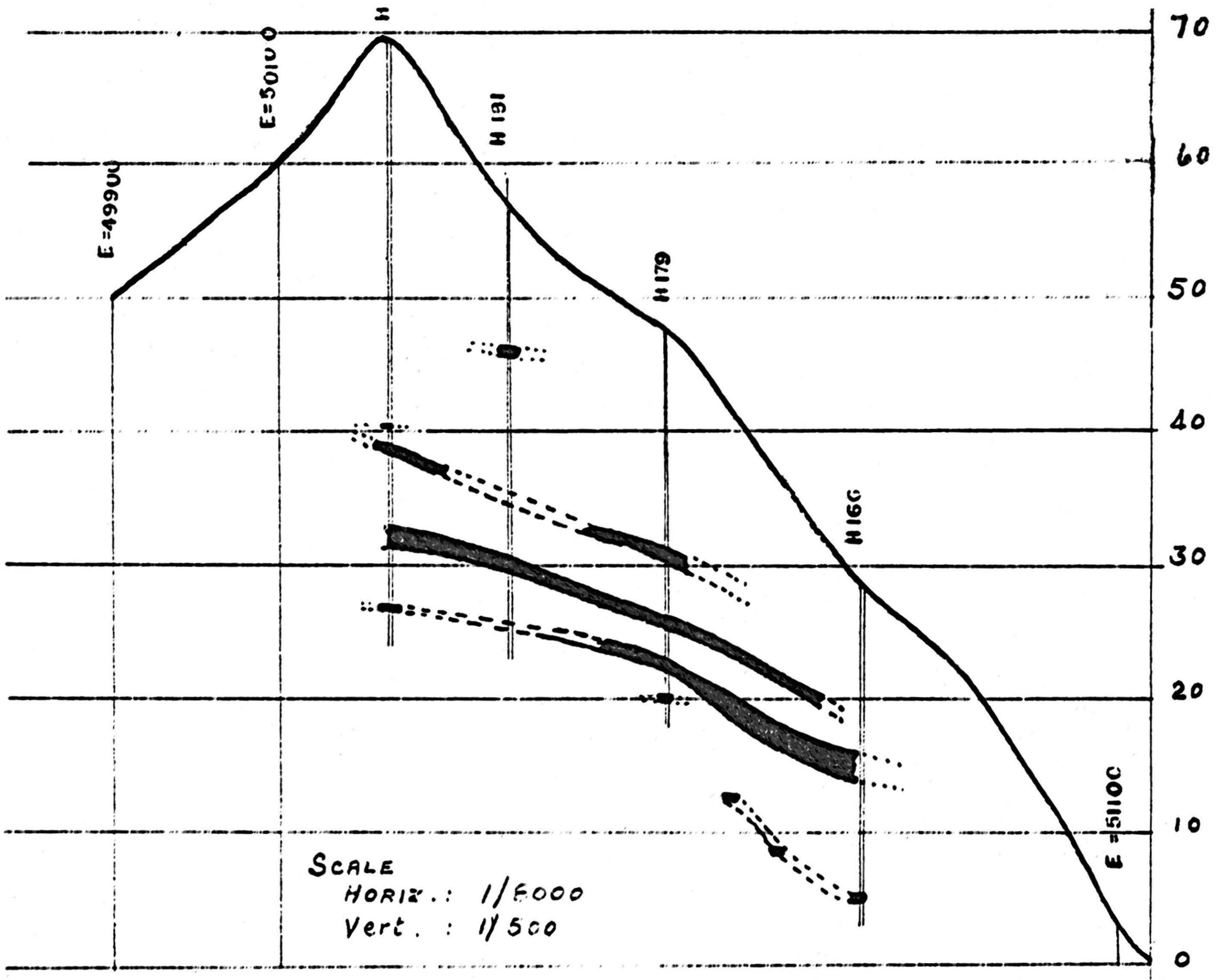


Fig. 7
Cross-Section Through North=50500
Looking West

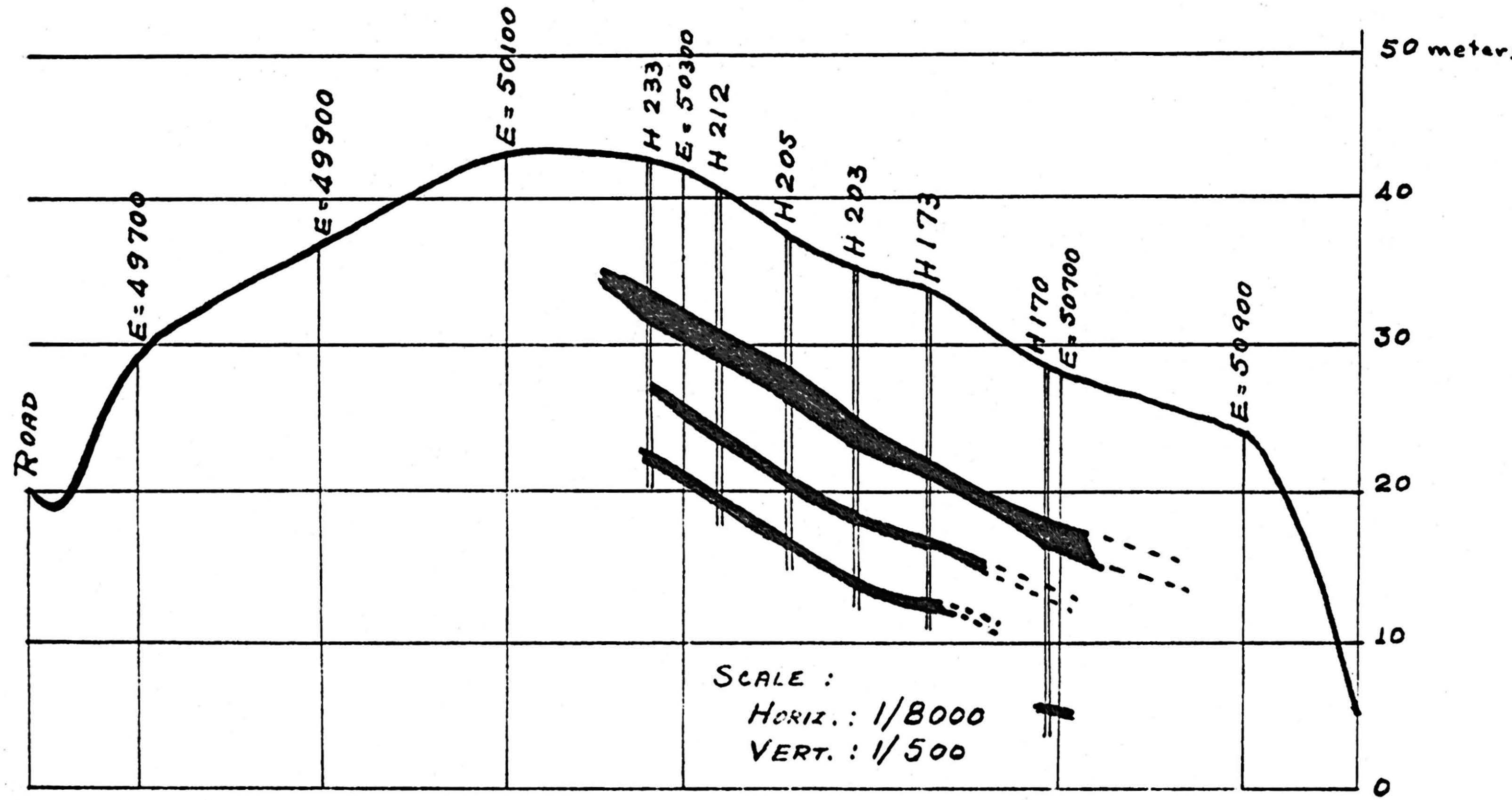


FIG. 8
CROSS-SECTION THROUGH NORTH=50700
LOOKING WEST

The overburden consists of semi-consolidated sand, sandy clay and vari-colored clays. In a few holes, conglomerate has been encountered. The thickness of the burden over Bed no. I varies between 5.0 ft. to 75.0 ft. and averages about 35 ft. The average thickness of burden for Bed no. 2 is 25 ft. and for Bed no. 3 14 ft. Typical columnar sections are shown in Fig. (1). The thickness of sand is approximately equal to the thickness of clay and sandy clay in the overburden of Bed no. 1. The overburden of Bed no. 2 and Bed no.3 consists of clay and sandy clay.

LIGNITE AND ITS USES

In general, the classification of coals is based on the fixed carbon and volatile matter content. Coal with 20 to 40 per cent fixed carbon and 20 to 30 per cent volatile matter is considered lignite. The following Table II shows the complete classification:

Table (II)+

Classification of Coal

Type of Coal	Fixed Carbon	Volatile Matter
Anthracite	75-90 %	2 -7.5 %
Semi anthracite	70-80	7.5-12
Semi bituminous	60-80	12 -25
Bituminous (low moisture)	50-70	25 -38
Bituminous (high moisture)	40-50	30 -40
Subbituminous	30-45	30 -35
Lignite	20-40	20 -30

+(From Coal Miner's Handbook)

Unfortunately there are not many data available on the chemical composition of the Kumtepe lignite. The following statistics were supplied by M. T. A.:[1]

Raw lignite contains	45% water
Air dry lignite contains	10-15% water
Air dry lignite contains	13% ash
Air dry lignite contains	2-3% Sulphur
Air dry lignite gives	11900 B.T.U. (3000 Cal)
Specific gravity of raw lignite	1.25

(1) Personal Communication, December, 1950

Lignite, especially in large blocks, breaks up easily when exposed to air. This disintegration is due primarily to the rapid evaporation of the water which constitute from 20 to 35 per cent of the lignite in place.

The cost of transporting coal containing so much moisture is excessive. Furthermore, the high moisture content causes a severe reduction in the efficiency of the fuel. It is, therefore, important that moisture be removed before shipment. "However, the tendency of coal to form slack when burned, even after the moisture has been removed, presents another difficult problem. As a result, the utilization of lignite has been confined to comparatively narrow regions near the deposits."[2]

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(2) Babcock, E. J., Economic Methods of Utilizing Western Lignites, U. S. Bureau of Mines Bulletin 89, 1915, p. 8.

The utilization of lignite as a fuel can be stimulated if it can be satisfactorily and economically converted to a fuel that is free from moisture and is of a size and strength suitable for general commercial use. Instead of burning the lignite in a crude state, it should be treated so as to yield several products, each of which can be adopted to a particular commercial need, as follows:

1. Dried lignite - This may be used in automatic stokers and fuel-gas producers.
2. Pulverized lignite - Pulverization of highly gaseous lignite produces fuel with properties similar to those of crude petroleum or natural gas. Dry lignite when pulverized is fed into a furnace with an air blast, the result is a gaseous fuel. The supply of coal and air can be regulated in order to secure the desired temperature.

Pulverized lignite can be used in oil burners in conjunction with fuel oil. "An oil mixture containing 30 per cent dried and finely pulverized lignite will still keep fluidity and it would decrease oil cost"(3)

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(3) Darling, S. M., Notes on Lignite, U. S. Bureau of Mines T. P. 178, 1919, p. 19.

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3. Producer Gas - Producer gas is derived from the incomplete combustion of a thick bed of lignite in a specially designed combustion and gas producing chamber. Unlike the production of ordinary coal gas, there is little or no residuum left in the producer chamber. The gas yield from one ton of lignite varies between 6,000 and 70,000 cubic feet and has a heating value of approximately 140 B.Y.U. per cubic foot. Thus, the gas from one ton of lignite can produce between 700 and 800 H. P.

4. Briquetted lignite - Slack or crushed lignite often is briquetted with special binders under pressure. An additional revenue may be derived by combining the gas from the briquetting plants and with the producer gas. Advantages of briquetting may be summarized as follows: (a) A large gain in heating value, (b) Prevention of slacking. (c) Mass is held together during combustion, (d) Uniform size and cleanliness.

The location is the most important point to be considered when contemplating the erection of a plant to treat lignite. The plant should be near the mine and should be designed to take raw lignite directly from the mine cars in order to eliminate intermediate handling. The plant could then profitably utilize slack accumulations resulting from mining operations and from weathering for the manufacture of briquettes.

The briquetting plant should be considered as subsidiary to the lignite mine since, at most mines, only a portion of the output would be briquetted.

Lignite briquettes should be shipped into areas that will be competitive with high grade coal shipped from a distance. In this respect the city of Istanbul is considered an ideal market.

Industrial plants located at a distance from bituminous coal mines should find lignite economical as a fuel. It has been pointed out that there are many industrial uses for lignite (producer gas, tar, and ammonium sulphate) which can be developed profitably by the mine operation, if a careful study has been made of the fuel needs of the industrial area. There are about 8 million metric tons of lignite available for mining in the Agacle Region and any industrial efforts which depend on a supply of lignite as fuel can look to the future with confidence.

ADVANTAGES OF STRIP MINING

The ideal in any mining operation is to obtain the most efficient use of man-power, machinery, and capital in an effort to secure maximum production at the lowest possible cost. Because there is a wide variation in the conditions under which coal is mined, and, because mining engineers and executives have individual preferences for equipment and methods of extraction, there are many bases which can be set up in order to obtain the desired results.

One important difference between underground mining operations and surface mining operations is that surface methods make possible the use of equipment that yields a much higher output per man shift. This advantageous use of man-power in strip mine operation makes it attractive where conditions are suited to this method.

Some emphasis on underlying causes of the present trend toward strip mining may provide a better understanding of the problems to be solved in effecting a successful operation.

"The price of labor is at an all time high with the result that the widest differential on record exists in terms of purchased energy between the cost of man-power and other forms of power such as electric, diesel, and explosives. The relationship (of these energy sources) to each other in terms of cost per horsepower - hour, if the cost of electricity is taken as one then diesel energy would cost from 1 to 3 times as much, explosives energy 10 to 20 times as much, and human energy 5,000 to 20, 000 times as much."(4)

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(4) Bailey, Harold L., Hillside Stripping in West Virginia; <u>Coal Mine Modernization Year Book</u>, 1948, p. 216

Man is outclassed by machinery as a producer of energy. In strip mining greater quantities of low cost energy (electric, diesel, explosives) and lesser quantities of high cost energy (man) are utilized.

Thin beds of coal 14 to 24 inches thick are being mined successfully by strip mining methods. The recovery of coal in underground mining methods in the United States ranges from approximately 50 per cent to 60 per cent. Strip mining recovery in a given area varies from 70 per cent to 95 per cent, and as a result it conserves a natural resource that can not be replaced.

Shut-down expenses are small in strip mining and cost of getting back into production is much less than with under ground methods.

Underground mining may cause subsidence which can be severe and dangerous at shallow depth and may permanently despoil the land. With stripping the ground can be restored so that there is only slight settlement to the finished surface and often the land is improved in value.

"(Strip mining) does not attempt to, and never can, compete with deep mining in this country (England), but for the winning (recovering) of coal from beneath shallow cover, it possesses many advantages over that of normal mining (underground mining) ground mining)."[5] This statement might be true for England,

(5) Hindley, Ralph, Open Cut Coal Production, The Colliery Guardian, Nov. 14, 1947, p. 651.

but in the United States of America, strip mining for over twenty years has been in competition with underground mining and the number of strip mines increases each year. The elements of cost saving in stripping operations can be summarized as follows:

a) The heavy item of cost of timber for support is eliminated.

b) Ventilation is unnecessary.

c) Larger hauling units can be used in hauling coal.

d) The possibility of moving the shovel or other heavy equipment when the coal deposit is exhausted confers a higher salvage value upon investment.

e) Production per man per day in strip mines is about 13.0 tons, as compared with all underground mines which has an average of 4.6 tons. See Table III for comparative statistics in terms of man - hours per ton in a few coal producing states.

The disadvantages of strip mining are stated below:

a) Topography and location of deposits often limit the stripping operation, and creates problems of drainage and disposal of overburden.

b) The initial investment is high.

c) Operations are subject to more frequent delays because of weather.

d) The size of the equipment available often limits the ratio of overburden to coal.

Table III[6]

States	Strip Mines man-hrs/net ton	Underground Mines man-hrs/net ton	Percent of Strip Mine to Underground Mines
Missouri	0.81	3.69	22
Kansas	0.60	3.33	18
Illinois	0.52	1.51	34
Indiana	0.56	1.51	37
Montana	0.21	1.47	14

(6) Kiessling, O. E., and Davis, J. A., Mining bituminous Coal by stripping Methods, Bureau of Mines, I. C. 6383, p. 6

SELECTION OF STRIPPING RATIO

In general, economic stripping ratios are dependent upon the character and thickness of the overburden, the thickness and price of coal, and accessibility of the market. Under ideal conditions, such as in the Kansas fields, this ratio runs as high as 32 to 1. In other areas it does not commonly run beyond 15 to 1 and averages 12 to 1.

The conditions at Kumtepe deposit for mining by open-cut operation are favorable. The overburden is of a type that does not require blasting. The beds have a uniform thickness. In recent years, price of lignite has been high, averaging $8.80 per ton in Istanbul and $5.30 F. O. B. mine. Istanbul is readily accessible from the mine over a state maintained highway. Although the transportation cost per metric ton is high, $2.60 per ton to Istanbul (30 miles), it is expected that this cost can be reduced somewhat. The economic stripping ratio 12 to 1 has been chosen for this deposit. This ratio is comparable to the ratios used in the United States under like conditions. The ratio allows a safety factor in computation of amount of overburden and lignite.

The influence of the stripping ratio on the output per man per day is shown in Fig. 9. It will be noted that the curve of output per man falls rapidly as the stripping ratio increases. Higher ratio mines have a much lower output per man and, therefore, except as offset by lower wages, a much higher labor cost.

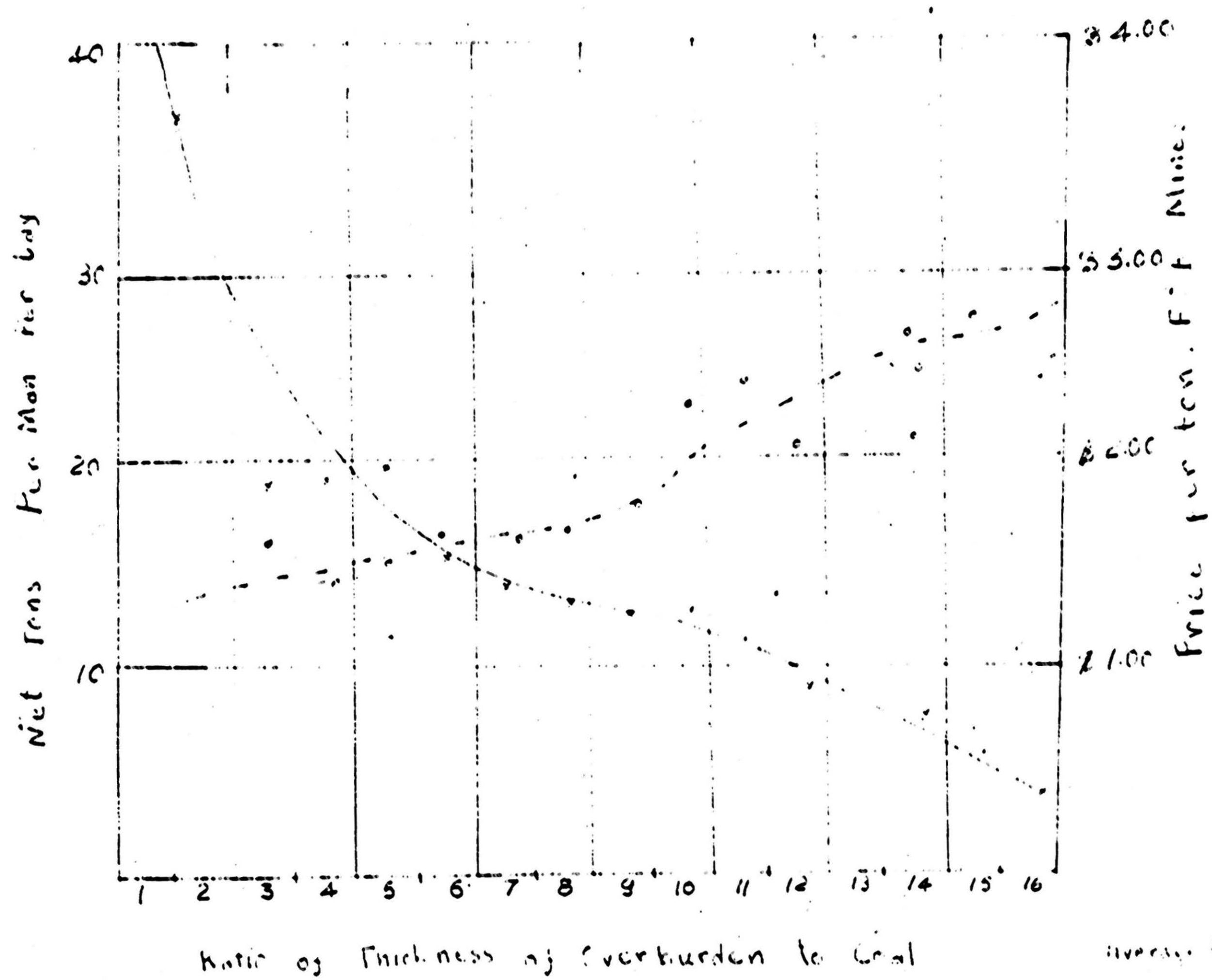

Fig. 9

Stripping versus Production of Coal per Man per Day

(After Kiessling, O.E., S.C. 6363)

COMPUTATION OF OVERBURDEN

Comparison of Earth work Computations Methods:

There are several methods that may be used in computing the volume of earth in place. Some of such methods are the End - Area Formula, the Prismoidal formula and the Method of Unit Areas.

End - Area Formula: $$V = \frac{L}{6} (A_1 + 4M + A_2)$$

V, L, A, and A_2 are the same as in the end - area formula. The area M is not the mean of A_1 and A_2, but is the area determined from the average of the linear dimensions composing A_1, and A_2. In a series of cross sections with equal spacings and with nearly equal areas, "each alternate cross section may be taken as a middle section whose area is M, if L is the distance between sections, the length of the prismoid will be 2L. For a continuous line of earth work, the volume in cubic feet (cubic meter) obtained by the prismoidal formula then becomes"[7]

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(7) Tracy, J. C., Surveying Theory and Practice, John Wiley and Sons, Inc., New York, 1947, pp. 699 - 715.

$$V = \frac{L}{3} (A_1 + 4A_2 + 2A_3 + 4A_4 + 2A_5 + 4A_6 + . . . + A_n)$$

The error in volume due to the use of the end area formula is generally small, often less than 2 %. Moreover, the end - area formula gives volumes that are generally too great and, therefore, the operator suffers no injustice from its use. It involves less computation than is required by the prismoidal formula, particularly when the linear measurements for middle sections must be calculated. For this reason the end -

area formula is commonly used for ordinary earth work computations.

The Method of Unit Areas:

In this method it is assumed that the area in question is divided into a series of squares, rectangles or triangles whose corners are at different elevations, but lie in the same plane. In Fig. 10 abcd is a rectangle which has been staked out on the ground. The surface of the ground within the rectangle is an inclined plane. The earth to be removed is a right trûncated prism, with vertical edges at a, b, c, and d. The rectangle abcd represents the horizontal projection of the upper inclined base of prism and also the lower base (in this study it is the top of the lignite bed). The earth to be removed is a truncated prism, the right section of which is the rectangle abcd and the volume which is found by following formula:

$$V = A \frac{a + b + c + d}{4}$$ V is in cu. ft. or Cubic meter

If there are many equal rectangles or squares, the total volume will be the sum of the individual volumes. The following formula is used for this calculation:

$$V = \frac{A(h_1 + 2\ h_2 + 3\ h_3 + 4\ h_4)}{4}$$

V, in cubic feet or cubic meter

A, area of one rectangle or square; in square feet or square meter

h, Corner height, subscripts indicate number of times a corner height is used. h_2 means that this corner

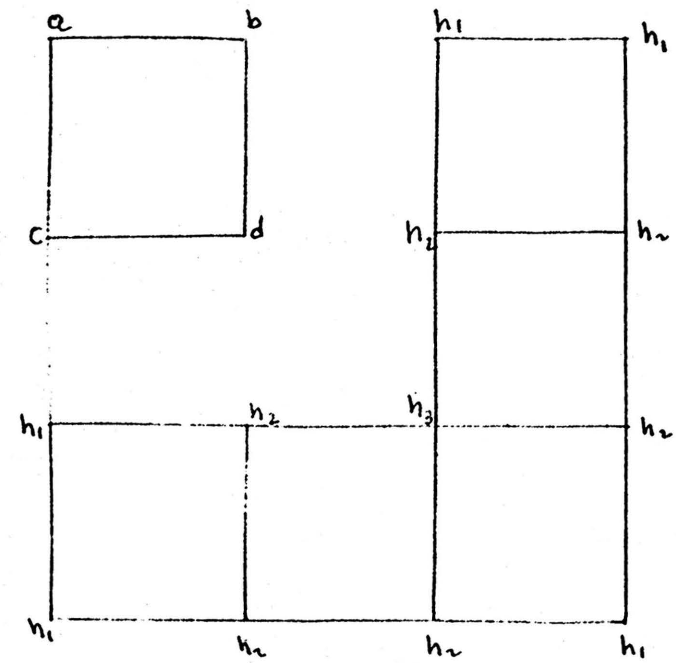

FIG. 10
Earth Computation by Unit Area Method.

is between two rectangles or squares. h_4, h_3, and h_1 are joint corners for four, three, and one squares or rectangles respectively

Before calculating the amount of overburden the stripping limits of each bed must be determined. The ratios of overburden to coal for each hole where determined from the data in Table I. The stripping limits were determined by including the areas of influence of holes with overburden to coal ratio of 12 to 1 or less. These limits are shown on map no. 3.

The volume of overburden over Bed no. 1 was calculated using the unit area method, because the necessary data obtained easily from drill holes. It can be seen that each block is subdivided into 25 squares of 20 m x 20 m. The use of this smaller square gives a more reliable result. Wherever possible the overlapping areas were added to the incomplete squares by estimates. The heights of the corners of small squares within the stripping limit are tabulated in Table IV.

To illustrate the application of the unit area formula, an area bounded by the following corners on map no. 3 is used: OM' = 10m., OL' = 10m, NL' = 11m, ML' = 13m, MM' = 10m, MN' =10m, NN' = 9m, and NM' = 9m. Fig. 11 illustrates this block.

Table (4)

Height of Corners to Be Used in the Method of Unit Areas from Map No. 3

n

Location of Corners	h_1 Height of Corners
AF'	6
Ag'	6
DD'	8
CB'	10
CH'	10
DA'	12
EL'	14
EN'	13
FA'	10
GB'	10
IO'	15
IH'	20
KG'	22
LQ'	10
ME'	24
MQ'	9
OD'	20
TC'	15
TK'	7
UD'	8
UJ'	7
VE'	7
VJ'	7
Total h_1	270 m.

Location of Corners	h_2 Height of Corners
BE'	8
CC'	10
DI'	12
EA'	12
EJ'	14
EK'	14
EM'	13
GD'	12
GE'	15
HG'	18
HH'	16
HJ'	16
IJ'	15
JI'	20
KP'	11
LF'	25
MQ'	8
ME'	24
OP'	9
QC'	20
QN'	7
RC'	15
SC'	15
SK'	7
VK'	7
VG'	7
VH'	7
Total h_2	357 m.

Table 4 (cont'd)

h_3

Location of Corners	Height of Corners
BF'	8
BG'	8
CD'	10
CG'	10
DB'	12
DH'	12
FB'	10
GC'	10
GF'	16
GM'	17
HJ'	16
JJ'	14
JO'	15
KH'	25
LP'	9
MF'	23
MP'	9
OE'	20
PD'	21
PN'	7
RM'	7
TD'	12
NE'	8
UI'	7

Total $h_3 = 306$ m.

h_4

Location of Corners	Height of Corners
CE'	10
CF'	10
DC'	12
DD'	12
DE'	12
DF'	12
DG'	12
EB'	14
EC'	14
ED'	14
EE'	15
EF'	15
EG'	15
EH'	15
FC'	10
FD'	15
FE'	15
FF'	15
FG'	15
FH'	15
FI'	15
FJ'	15
FK'	15
FL'	16
FM'	15
FN'	15
GG'	20
GH'	19
GI'	20
GJ'	17
GK'	17
GL'	17
HK'	19
HL'	18
HM'	17
HN'	16
IK'	18
IL'	17
IM'	17
IN'	16

Table 4 (cont'd)

h_4 (cont'd)

Location of Corners	Height of Corners
JK'	18
JL'	18
JM'	17
JN'	16
KI'	22
KJ'	19
KK'	18
KL'	16
KM'	14
KN'	13
KO'	12
LG'	20
LH'	13
LI'	13
LJ'	13
LK'	13
LK'	15
LM'	13
LN'	12
LO'	10
LP'	9
MG'	20
MH'	15
MI'	15
MJ'	15
MK'	12
ML'	13
MM'	10
MN'	10
MO'	10
MP'	10
NF'	21
NG'	20
NH'	18
NI'	17
NJ'	16
NK'	12
NL'	11
NM'	9

h_4 (cont'd)

Location of Corners	Height of Corners
NN'	9
NO'	9
OF'	17
OG'	16
OH'	15
OI'	15
OJ'	14
OK'	12
OL'	10
OM'	9
ON'	9
OO'	9
PE'	15
PF'	14
PG'	12
PH'	12
PI'	11
PJ'	10
PK'	9
PL'	8
PM'	8
QD'	15
QE'	13
QF'	12
QG'	11
QH'	10
QI'	9
QJ'	9
QK'	8
QL'	8
QM'	7
RD'	14
RE'	12
RF'	12
RG'	12
RH'	10
RI'	9
RJ'	8
RK'	8

Table 4 (cont'd)

h_4 (cont'd)

location of Corner	Height of Corners
RL'	7
RM'	7
SD'	12
SE'	11
SF'	10
SG'	10
SH'	9
SI'	9
SJ'	9
TE'	10
TF'	10
TG'	10
TH'	8
TI'	7
UF'	8
UG'	8
UH'	8
	1798.0 m.

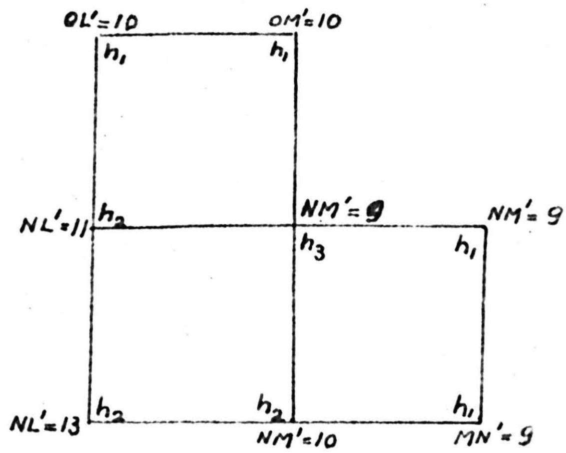

FIG. 11.

APPLICATION OF THE UNIT AREA METHOD

$$V = \frac{A\ (h_1 + 2\ h_2 + 3\ h_3 + 4\ h_4)}{4}$$

$A = 1600 m^2$

h_1 (m)	h_2 (m)	h_3 (m)	h_4 (m)
$OM^1 = 10$	$NL^1 = 11$	$NM^1 = 90$	0
$OL^1 = 10$	$NM^1 = 10$		
$ML^1 = 13$	Total 21		
$MN^1 = 10$			
$NN^1 = 9$			
Total 52			

$$V = \frac{1600\ (52 + 2 \times 21 + 3 \times 9 + 0)}{4} = 48000\ m^3$$

V = 62,600 cubic yards

By using data from Table IV, the total overburden over Bed no. 1 was found as follows:

A = 1600 m

$2h_1$	= 270 m	$2h_1$	= 270 m
$2h_2$	= 357 m	$22h_2$	= 714 m
$2h_3$	= 306 m	$32h_3$	= 918 m
$2h_4$	= 1798 m	$42h_4$	= 8990 m
		Total	10892 m

$$V = \frac{1600 \text{ X } 10892}{4} = 4\ 345\ 514\ m^3$$

V = 5,700,000 Cubic yards overburden to be stripped t recover Bed no. 1.

As stated in General Description, the beds are nearly horizontal. The average thickness of overburden of Bed no. 2 multiplied by average area of Bed no. 2 will give the average volume of overburden of Bed no. 2. It should be noted that Bed no. 2 is divided into two strippable areas. The areas are defined by block numbers as follows:

Block no.	Area (m^2)	Average Thickness (m)	Volume (m^3)
4, 5, 7, 8, 11, 12 and 13	163,010	7.65	1,243,026
14 and 19	12,177	7.59	92,423
		Total	1,339,449

V_{Total} = 1,700,000 cubic yards of overburden to be stripped to recover Bed no. 2.

The same method was used in calculating overburden for Bed no. 3 as was used for Bed no. 2:

Block no.	Area (m^2)	Average Thickness (m)	Volume (m^3)
4, 5, 7, 8, 11, 12 and 13	63,518	4.30	273, 127

V = 358,000 Cubic yards

The total amount of earth to be stripped during the life of the mine is:

For Bed no. 1	5,700,000 cu. yds.	in place
For Bed no. 2	1,700,000 cu. yds.	in place
For Bed no. 3	358,000 cu. yds.	in place
	7,758,000 cu. yds.	in place

AVAILABLE LIGNITE RESERVE FOR STRIP MINING

There is no specific rule as to size of reserves needed to support investment in equipment of stripping operation. The reserves of strip mines now in operation vary from twenty-five thousand to fifteen million tons, although under average conditions the lower limit for reserves is approximately two million tons. "A good estimate of the available reserve, daily and monthly output may be charted. For each one thousand tons daily output, an investment in plant of about $500,000 will be required. This figure includes machinery, buildings, and inventory. Under present day equipment costs this figure, if not exact, may serve as a guide as to the minimum capital required to start a sizable job."[8]

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(8) Bailey, Harold L., op. cit. p. 217

Areas of coal that are suitable for stripping are generally limited to locations adjacent to the outcrop of the coal seam. There are two main types of surface characteristics at and beyond the outcrop which influence the amount of stripping reserves available to a single operation:

1) Where coal beds occur under broad areas of relatively level surface, the strippable reserves per unit length of outcrop are also relatively large.

2) Where the coal beds outcrop along the slope of the ridge and valley, the line of maximum thickness of removable overburden is relatively close to the line of outcrop. Its nearness depends largely on the surface

slope above the outcrop. In general, the shape of strippable area resembles a narrow sinuous ribbon parallel with the surface contours. The strippable reserves per unit length of outcrop are relatively small in this case.

The lignite deposits at Kumtepe fit the second type of surface characteristics. The change in the width of strippable area from west to east is especially noticeable. In the west the general slope of surface is less than in the east, and so the width of this ribbon is larger in the west than in the east. The reserves in each block are shown in Table V.

Assuming 85% recovery will be possible, there will be 780,000 short tons or 705,000 metric tons of lignite available. The proposed production per year is about 100,000 metric tons, therefore, the life expectancy of the mine is about 7 years.

In Agacle Region there are other locations which are suitable for stripping, but only the Kumtepe reserves are being considered in this study. The equipment with longer life than the mine itself can be utilized, if desired, at the other localities. As there is not enough information available on the other locations, however, depreciation of all equipment will be predicted on a 7 year life.

Table V

The Available Reserves

Block no.	Area (m^2)	Ave. Thickness (M)	Tons (metric)
Bed no. 1			
4	13,088.37	1.00	16,360.46
Between 11 and 7	10,936.51	.98	13,397.23
7	39,262.12	1.20	58,893.18
11	20,151.54	1.50	37,784.15
12 + 7	36,531.00	1.15	52,513.32
8	37,012.00	1.42	65,696.30
13	10,063.10	2.07	26,038.05
9	29,645.04	2.00	74,112.50
5	26,278.00	1.83	60,110.95
6	4,961.94	1.50	9,302.62
15 + 20 + 10	18,581.94	1.87	43,435.30
14 + 19	32,151.32	1.51	60,685.62
		Total	518,330.00
Bed no. 2			
4, 5, 7, 8, 11, 12 and 13	163,010.00	1.06	215,988.25
14 and 19	12,177.00	1.58	24,049.58
		Total	240,037.83
Bed no. 3			
4, 5, 7, 8, 11, 12 and 13	63,518.00	.88	69,870.00

Total Reserves:

Bed no. 1	518,330	metric tons
Bed no. 2	240,038	metric tons
Bed no. 3	69,870	metric tons
Total	828,238	metric tons
	915,500	short tons
	840,000	long tons

STRIPPING OVERBURDEN

The coal industry has utilized strip mining for many years as a method of quick recovery. In recent years, the increase of output from strip mines has been 80% as compared with a 14% increase in production from underground operations.

As the production of coal from strip mines depends upon the mine operators' ability to uncover the coal, the function of stripping unit has become of utmost importance. Before deciding what mining method is to be employed and what stripping unit is to be used, the following factors should be considered carefully:

a) total quantity of overburden

b) type of overburden

c) depth of overburden

d) disposal of stripped materials

The amount of overburden to be excavated has an important bearing on the size and number of machines necessary to insure the required rate of removal. The required rate is one that will uncover a sufficient tonnage of coal for the proposed production. The type of overburden is of major importance in determining the type of excavating equipment and the method of excavating.

The opening of a coal deposit by stripping is influenced generally by the problem of drainage. After proper drainage has been established, the development should preferably begin at the low point of coal bed and be carried up the slope. A detailed working map of the operation should be made. It should show the proposed cuts and the location of the spoil material taken from each cut.

The tipple should be located so that a minimum average haulage distance can be realized.

There are numerous types of equipment used for stripping operations, such as, shovels, draglines, scrapers, trucks, bulldozers, roadgraders, and rooters. Each type of equipment is subdivided according to the kind of power supply with which it is equipped (diesel, electric, or gasoline) and according to the mounting type (wheel mounted or crawler mounted). It is not the purpose of this study to discuss this equipment in detail. The limitations and applications of the more important equipment to be used at Kumtepe will be discussed, namely, shovels, scrapers, and trucks.

A detailed study on the application of equipment used in open pit mines was made by Mr. W. J. Latvala.[(9)]

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(9) Latvala, W. J., Principles and Practices Controlling the Use of Earthmoving Equipment, Thesis, Missouri School of Mines and Metallurgy, Rolla, Mo., 1950. 279pp.

In general, the selection of the size and capacity of equipment is predicated on the expected daily, monthly, or annual production. The selection of the type of equipment to be used is based on the physical conditions indicated by prospecting. The principal item of equipment in strip mine is the stripping unit. The required capacity and size of this unit depends largely upon the overburden ratio which is the number cubic yards of overburden that must be removed to uncover a cubic yard of coal. For example, a ratio of 12 indicates that the removal of 1,200,000 cubic yards of overburden is required to produce 100,000 cubic yards which is equivalent to 120,000 tons of coal.

Important Factors in Selection of Equipment

1. The character of overburden:

a) Hardness of material--

Each piece of equipment has been designed to withstand the effect of stresses set up in it when digging into indurated material. If the maximum limit is exceeded repeatedly, life of the equipment is shortened and break-downs will eventually occur. No piece of excavative machinery should be considered as a substitute for good blasting technique. In many cases, therefore, some type of preparation of banks is necessary. An illustration of the effect of this condition is shown in Table VI.

Table VI(10)

Power Shovel Hourly Output, Bucket Capacity in Cubic Yards

Type of Material	3/8	1/2	3/4	1	1 1/4	1 1/2	2	2 1/2
Moist loam or light sandy clay	85	115	165	205	250	285	355	405
Sand and gravel	80	110	155	200	230	270	330	390
Good common earth	70	95	135	175	210	240	300	350
Clay, hard, tough	50	75	110	145	180	210	265	310
Rock, well blasted	40	60	95	124	155	180	230	275
Common, with rocks and roots	30	50	80	105	130	155	200	245
Clay, wet and sticky	25	40	70	95	120	145	185	230
Rock, poorly blasted	15	25	50	75	95	115	160	195

(10) Proper Sizing of Excavators and Hauling Equipment, Crane and Shovel Association, 74 Trinity Place, New York 16, N. Y., 1949, p: 3

b) Swell factor --

The digging process loosens earth and increases the per-centage of air spaces between the solid particles. For example, common earth has an average swell factor of 25%, that is one cubic yard of common earth in place will occupy 1.25 cubic yards when broken. In order to apply the swell factor to measurement by weight the following method of determination is used.

$$\frac{100}{100 + 25} = 0.80 \text{ which is the swell factor.}$$

If common earth weighs 3000 lbs per cubic yard in place, it will weigh

$$3000 \times 0.80 = 2400 \text{ lbs. per cubic yard, loose.}$$

See Appendix A for swell factors of different materials. This factor affects the size of the excavating unit and the size of the haulage unit.

c) The breaking characteristics of the material --

If the blasted material is excessively oversize, it will be difficult to get a full dipper load. However, if the material is clayey the loading time may also be increased, because of the cohesiveness of the material.

d) The bearing capacity of the soil --

This factor will limit the maximum size and weight of the excavating machine which can be used at the mine.

e) The angle of repose of stripped material --

This factor affects the height to which stripped material may be placed in the spoil bank. This spoil bank height affects the discharge height of the shovel.

f) The height of the bank --

Each power shovel has a maximum cutting height. If the bank is higher than the equipment is capable of handling, auxiliary equipment will be necessary to aid in reducing the height of the bank. Bulldozers or scrapers may be used for such purpose.

2. The equipment power supply:

At the present time, the three main sources of power for stripping equipment are, electric power, diesel power, and gasoline power. In general, power shovels and draglines can be equipped to use any of these three sources of power. For this purpose electricity is the cheapest power source and gasoline is the most expensive. From a power standpoint, diesel powered shovels up to 4 cubic yards capacity are competitive with electrically powered shovels. However, above this dipper capacity diesel powered equipment becomes more costly than electrically powered equipment. Although the original cost of the diesel equipment is greater than the elictric equipment, there are several advantages which outweigh the higher initial cost. The advantages of diesel powered shovels are: (1) no necessity for an electric power generating plant, (2) no necessity for maintenance of power cables, (3) may be used on other properties where electricity is not available.

At Kumtepe, diesel - powered shovels are preferred, because electric power is not available. As the life of the mine is only 7 years, it is not feasible to invest in a power generating plant.

3. The mobility of equipment:

Shovels are usually crawler mounted as this type of equipment does not have to be as mobile as scrapers and trucks. The mine owners favor the scraper and truck with high speed ranges. However, at many mines, scrapers and trailers are drawn by crawler mounted tractors. Rubber tired wheel mounted tractors have attained considerable favor, in recent years, among mine operators. "Crawler type of equipment has the advantage of being able to negotiate severe conditions of terrain without the expense of road building, but unfortunately, to date, the speeds at which this equipment is designed to operate are relatively low and the carrying capacity of material - hauling vehicles now in use is also low."(11)

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(11) Berry J. G., Rubber in Open-pit Mining, Mining Congress Journal, June, 1946, p. 51-52

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4. The control mechanism of earthmoving equipment:
In general, the two types of operational controls of heavy equipment are mechanical and hydraulic. "Hydraulic power as applied today to earth-moving equipment is still in its early stages of development. Research work in engineering departments and laboratories of manufacturers has resulted in lowering costs, improving dependability, and facilitating higher output."(12) Most mine operators,

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(12) Hrdlicka, E. J., Hydraulic Control of Earth-moving Equipment, Excavating Engineer, January, 1950, p 17

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however, still prefer mechanical controls on their equipment. At the present time, maintenance cost of the mechanical controls is less as hydraulic controls are relatevly new in their application to earth-moving equipment.

Selection of Power Shovel and Calculation of Its Output

Before attempting to select a shovel for the Kumtepe lignite deposit, the following assumptions were made:

1. The overburden is a sandy-clay and has a 30% swell (0.77 swell factor).
2. The heap capacity of equipment is used in calculations and designated in loose measure.
3. The operators of the equipment will be experienced.
4. The mine will operate 250 days a year.

The use of draglines in strip mines is becoming popular especially where the overburden ratio is high. However, draglines have not been proposed for use at this operation because three seams are to be mined and so the overburden of the first two beds must be stripped and removed beyond the limits of the minable area. This operation can be best done efficiently by shovel-truck system.

The total overburden to be stripped at Kumtepe was calculated as 7,758,000 cubic yards in place. As the per cent of swell is 30% the quantity of overburden to be handled by the stripping shovel will be ;

$$7{,}758{,}000 \text{ X } 1.30 = 1{,}000{,}000 \text{ cubic yards loose.}$$

Assuming that stripping operations will continue about 7 to 8 years, then the power shovel will strip annually approximately 1,250,000 cubic yards of loose material. The daily output of the stripping unit will be:

$$\frac{1{,}250{,}000 \text{ cu. yds.}}{250 \text{ days}} = 5{,}000 \text{ cu. yds. per day}$$

An estimate of the output to be expected from various sizes of shovels is shown in Table VI and is expressed on an hourly basis. A 2 - yd. shovel in an 8 - hr. shift will strip about 2800 cu. yds. whereas a 2 1/2 yd. shovel in the same period will strip about 3200 cu. yds. These two shovels together will strip more than the minimum daily proposed output. Each shovel working a double shift will strip also more than the minimum proposed. These output capacities are somewhat arbitrary, and occur under favorable conditions. Other factors must be also considered in determining the output.

The detailed discussion of these factors has been made by Mr. W. J. Latvala in his previously mentioned paper and therefore a brief review of those factors should be sufficient:

1. Multipliers (See Appendix B)
   a) Type of material.
   b) Size of dipper.
   c) Depth of cut.
   d) Type of operation.

2. Effect of angle of swing on shovel output for shovels from 3/8 - cu. yd. to 2 - cu. yds. In general, the output of any shovel can be ascertained by using a "work output formula" based on the cycle of operations.(13)

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(13) War Department Technical Manual T M 5-252, Use of Road and Airdrome Construction Equipment, War Department, Washington, D. C., January, 1945, p 91

Work output formula is:

$$\text{Output} = \frac{36{,}000 \times f \times E \times K}{Cm}$$

3600 = seconds (one hour)

Q = dipper capacity of shovel, struck

f = soil conversion factor (Appendix A)

E = shovel efficiency factor, Average 0.80

K = dipper efficiency factor (Appendix B)

Cm = Cycle time (seconds), (Appendix B)

It was shown in the original estimate that the capacity of the stripping shovel dipper should be larger than 2 - cu. yds. A more exact estimate of the size of the shovel can now be made based on the additional factors shown in Appendix B.

The expected job conditions at Kumtepe are assumed as follows:

a) material to be stripped is sandy clay.

b) average depth of cut will be 30 ft.

c) trucks to be loaded by side casting, based on an average of a 90° swing.

Case 1, Output for 2 -yd. shovel

Resultant multiplier = 2.00 X 0.90 X 0.68 X 1.25 = 1.61

2.00 = Size of dipper

0.90 = Multiplier for sandy-clay (Appendix B-b)

0.68 = Multiplier for depth of cut (found by interpolation, Appendix B-c)

1.25 = Multiplier for type of operation (Appendix B-d)

Output = 120 X 1.61 = 193 Cu. yd./hr. in place

= 250 cu. yd./hr. loose

Case 1, (cont.)

120 = Basic assumption (See Appendix B-a)

Output = 2000 cu. yd./ 8-hr. shift

= 4000 cu. yd./ 2 - 8-hr. shifts

Case 2, Output for 2 1/2 - yd shovel

$$\text{Output} = \frac{3600 \times Q \times f \times E \times K}{Cm.}$$

Q = 2 1/2 = 2.50 cu. yds.

f = 1.27, Averaging sand and clay factors, (Appendix A-c)

E = 0.80 Average field experience

K = 0.85 Medium digging, (Appendix B-e)

Cm = 22 sec.,(Appendix B-f)

$$\text{Output} = \frac{3600 \times 2.50 \times 1.27 \times .80 \times .85}{22} = 305 \text{ cu. yd./hr. loose}$$

Case 3, Output for 3 - yd. shovel

Cm = 24 seconds

Q = 3.00

$$\text{Output} = \frac{3600 \times 3.00 \times 1.27 \times .80 \times .85}{24} = 388 \text{ cu. yd. / hr. loose}$$

Output per 8-hr. shift = 3100 cu. yd., loose

Output per two 8-hr. shifts = 6200 cu. yd., loose

Case 4, Output for 3 1/2 - yd. shovel

Q = 3.50 cu. yd.

Cm = 25 seconds

$$\text{Output per hr.} = \frac{3600 \times 3.50 \times 1.27 \times .80 \times .85}{25 \text{ seconds}} = 435 \text{ cu. yd. loose}$$

A study of the above output figures for the four different sizes of shovels indicates that a 3 - yd. shovel working two 8-hr. shifts will deliver more than the required amount (5000 cu. yds. loose material) of overburden. The results of these computations are summarized in Table VII.

Table VII

Calculated Power Shovel Output
(Loose Cubic Yards)

Shovels	Output		
	Per hour	One 8-hr. shift	Two 8- hr. shifts
2 - yd.	250	2000	4000
2 1/2 - yd.	305	2440	4880
3 - yd.	388	3100	6200
3 1/2 - yd.	435	3480	6960

The decision to purchase one or two shovels based on above computed results is not completely conclusive. The ownership and operating costs of the various shovels of different capacity must also be considered.

There are advantages and disadvantages attending the use of one or two shovels and they are shown as follows:

Shovels	Advantages	Disadvantages
One shovel working on two shifts	1. Low initial investment 2. Low maintenance cost 3. Less supervision required 4. Less labor cost	1. A breakdown will stop all stripping operations 2. Require supervision on two shifts
Two shovels working on one shift	1. Breakdown of one shovel will not completely disrupt stripping cycle 2. No shift differential payment 3. Two sites can be stripped simultaneously	1. High initial investment 2. High maintenance cost 3. Two haulage fleets required 4. High labor cost

Cost Analysis of Power Shovels

The method of cost analysis to be used here has been developed by the "Power Crane and Shovel Association". The terms used in this method of cost analysis are stated in Appendix C. The analysis will be made for a 2 1/2 - cu. yd. shovel and a 3 - cu. yd. shovel.

The analysis for 2 1/2 - cu. yd. shovel follows:

Case 1. The shovel is Marion Type 93-M, crawler mounted chain crowd diesel shovel, weighs 173,000 lbs.

A. 1) F.O.B. New York price	$71,125.00
2) Freight charges, unloading erecting 10% of F.O.B. price to Istanbul, Turkey	7,113.00
3) Custom duties at $10.80 per metric ton	850.00
Total cost or investment	$79088.00

B. This shovel will be depreciated in 7 years and no salvage value is considered.

C. Average yearly investment = 57% of total investment = $45,000 (Appendix C-d)

	Per Year	Per Hour One Shift 2000 hrs/yr	Per Hour Two Shifts 4000 hrs
D. Depreciation 14.3% of total cost	$11,300	$ 5.54	$ 8.21
E. Interest, taxes, insurance 10% of average investment	4,500	2.25	2.25
F. Maintenance 12.5% of total investment	9,450	4.74	4.74
G. Total Fixed cost	$25,250	$12.53	$15.20

H. Engine Fuel and lubricating cost — Per hour

Average 7 1/2 gallons per hour at $0.31		$2.31
Lubricating oil about .2.gal. per hour at $1.60		0.32
	Total	$2.63 per hr.

J. Labor cost

1 shovel operator	$1.00 per hr.
1 oiler	.41 per hr.
Total labor	1.41 per hr.

L. No overtime

M. Total direct cost

	Per year	One shift	Double shift
Total fixed cost	$25,250.00	$12.53	$15.20
Fuel cost	5,260.00	2.63	2.63
Labor cost	2,820.00	1.41	1.41
	$33,330.00	$16.57	$18.24

N. Supervision and overhead charges will be included after haulage cost have calculated.

O. Output per hour was 305 cu. yd. per hour

P. Cost per cu. yd. = $0.054 for single shift

$0.060 for double shift

Case 2. The shovel is Marion Type 111-M, 3 cu. yds. Diesel powered, weighs 3.0,000 lbs.

A. 1) F.O.B. New York price estimated $90,400

2) Freight, unloading, erecting 9,040

3) Custon duties 1,530

Total investment $100,970

B. This shovel will be depreciated in 7 years, no salvage value considered.

C. Average yearly investment

57% of total investment $ 62,500

	Per yr.	Per hr. One shift 2000 hr/yr	Per hr. Two shifts 4000 hr/yr
D. Depreciation (Appendix C-c and e) 14.3% of total investment	$15,700	$ 7.85	$11.78
E. Interest, taxes, insurance	10,097	5.05	5.05
F. Maintenance, estimated 15% of total investment	16,400	8.20	8.20
G. Total fixed cost	$42,197	$21.10	$25.03
H. Fuel and lubricating cost ave.			
10 gal/hr at $0.31		3.10	3.10
Lub. oil about 0.3 gal/hr at 1.60		0.48	0.48
Total $3.58			
I. Labor cost			
1 shovel operator $1.00 per hr.			
1 oiler .41 per hr.			
Total $1.41		1.41	1.41
M. Total direct cost		$26.09	$30.02

P. Output per hour (Table VII) = 435 cu. yds.

R. Cost per Cu. Yd. $0.06 $0.068

The results of these cost analyses have been summarized in Table VIII.

Table VIII

The result of shovel cost analysis

Size of Shovel	Output, loose material			
	One Shift		Two Shifts	
	cu. yd.	Cost / cu. yd.	cu. yd.	Cost / cu. yd.
2 1/2 - cu. yd.	2440	$0.054	4880	$0.060
3 - cu. yd.	3100	$0.060	6200	$0.068

A study of Table VIII shows that a 2 1/2 - cu. yd. shovel is more economical than the 3 - cu. yd. shovel. This does not mean that the greater the size of shovel the higher the cost per cubic yard. Such a generalization cannot be made. For this particular mine, under the given job conditions, and at the present cost of equipment the 2 1/2 - cu. yd. shovel is preferable to the 3 - yd. shovel.

It can be seen that the 2 1/2 - cu. yd. shovel does not produce the required output in two shifts, but the 2 1/2 - cu. yd. and the 3 - cu. yd. shovel produce more than the minimum required amount of overburden. It is a good policy to have total shovel capacity which will give more than the required output.

The average cost per cubic yard of material produced by two shovels is $0.057 which is less than that of 3 - cu. yd. shovel. Therefore, use of two shovels combined in one shift--in the long run--is more economical than the use of one shovel

(3 - cu. yd.) in two shifts. Furthermore, if the operator wishes to accelorate the stripping operation, he can do so by using the shovels in second and third shifts. Therefore, he almost doubles and triples the stripping output.

The working dimensions of these two shovels may be found in Appendix B.

Truck Haulage

Track haulage will not be considered, because it is not applicable to a rough topography. The cost of laying, maintaining, and shifting track in the pit is high as are also, labor and material costs.

Automotive equipment has taken over a large part of the transportation job at strip mines, because of its lower cost, greater flexibility, and elimination of complications in the pit. Indeed, experience has shown that truck haulage has its limitations, although these limitations are rather flexible depending upon natural conditions, length of haul, size of units, and other factors. Some operators, when the hauling distance is much over three or four miles, have found a combination of rail and automotive equipment, connected by field transfer stations to be the most satisfactory and economical method.

Power plants for today's trucks are based on engines using any of three kinds of liquid-fuel, gasoline, fuel oil and butane. Gasoline has two draw-backs (1) inflammability and (2) the production of carbon monoxide exhaust gases. Both fuel oil and butane deposit but little carbon in the cylinders and rings, therefore, maintenance is lower on engines using these fuels than gasoline engines. Butane must have special transportation and storage facilities. It is so volatile that it must be kept under 50 lb. per sq. in. pressure. "The growing popularity of the diesel engine for stationary power and its low operating cost have attracted truck users. Now that statisfactory engines are available, they are becoming very popular and are gradually replacing gasoline engines."[13]

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(13) Richart, Fred W., Truck Haulage, Coal Age, July, 1944 p. 42

One of the material advantages of the diesel engine, besides using less expensive fuel, is the absence of spark plugs, wires and other ignition system parts. Use of recently developed super chargers on diesels increases the horsepower by approximately one-third or from 150 H.P., to 200 H.P., rated. Reduced to basic principles, a "supercharger is merely an air compressor that puts more air into the engine cylinder and increases the pressure and power."[14] However, superchargers

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(14) Ibid., p. 43

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are not very popular, though there are situations where they are not only justified but necessary. For example, as in converting 15-ton trucks to 35-ton trailer trucks, the added power required may be obtained by the use of superchargers without changing the size of the engine. When the elevation becomes greater than 2,000 feet, the power of the engine decreases, The power loss can be overcome by the use of superchargers.

The size of trucks to be used at a given mine depends to a great extent on the condition of haulage roads for the time required to make a complete cycle. The larger unit, without doubt, requires somewhat better constructed roads and slightly higher maintenance costs. The initial cost of road construction should be proportioned to the life of the mine, and road maintenance should be proportioned to an annual or daily output. It should be pointed out that haulage equipment trends have been toward the use of trucks of larger capacity. The larger the size of truck the less is the uhit cost of maintenance, and labor because fewer trucks are used. "It may be argued that an economic limit to truck size has been, or is being reached but it is a matter of record that every increase in unit size up to the present time has been accompanied by a marked reduction in unit cost of labor, fuel, tires, and road maintenance (this is somewhat doubtful, because large size trucks cause more damage to the road due to their weight.)"[15]

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(15) Coddington, A. E., Progress in Strip Mine Haulage, Coal Mine Modernization, Year Book 1949, p. 310.

An interesting study was made by L. Russel Kelce[16] of the

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(16) Kelce, L. Russel, Development of 80-ton Haulage Trucks, Coal Mine Modernization Year Book 1940, p. 126.

Hume-Sinclair Coal Mining Co. on truck haulage of coal. In this study Kelce compared 15-ton, 20-ton, and 80-ton trucks under similar conditions, which were: (1) the mine producing 4000 tons of coal per day (2) the working time 7 hrs per day (3) the coal loaded with a shovel of 4 tons of capacity, loading cycle 20 secs. (4) one way 3 miles trip. The results of this study are shown in Table IX.

Table IX

Comparison of 15-ton, 20-ton, and 80-ton Capacity Trucks

	15-ton	20-ton	80-ton
Loading time	1:20	1:40	6:40
Loaded trip, 15 MPH	12:00	12:00	13:00
Dumping	:30	:40	1:00
Returning, 21 MPH	8:40	8:40	9:20
Total time (min)	22:30	23:00	30:00
Trips per 7 hrs.	18	18	18
Tons per day	270	360	1120
Units required	15	12	4
Approx price per unit	$7,500	$9,000	$22,000
Total capital investment	$112,500	$108,000	$88,000
Saving over	$ 24,500	$20,000	$00,000

In general, the foregoing discussion on trucks indicates that mine operators prefer the largest capacity diesel powered trucks suitable to their mines, but the final selection of the most suitable size should be made after considering all basic factors and making a cost analysis on the proposed haulage fleets.

Before entering a discussion of particular phases of truck haulage some important definitions will be made:[17]

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(17) Thoenen, J. R. and E. J., Lintner, Time Study Analysis Progress Report 2: Quarry Haulage, Bureau of Mines R. I. 3467, 1939, p. 3.

"Haulage System" is defined as the design, maintenance, and operation of the haulage route and equipment.

"Haulage Unit" is defined as one or more carriers combined with, or attached to, a means of locomotion.

"Haulage Cycle" is defined as the time required for a unit to make a round trip from the shovel to the dumping point, return, and load.

In the opinion of J. R. Thoenen and E. T. Lintner, the haulage system has no control over the yardage or tonnage delivered to it, therefore, it cannot be considered as productive equipment, but functions as service equipment. Here, it should be remembered that although haulage systems cannot control the shovel output, a delay in a haulage system effects on the efficiency of the power shovel. Thus, the importance of coordination of shovel operation and haulage cycle is obvious.

Haulage System for Kumtepe Lignites:

The important points in designing a haulage system are:

1. The route for trucks should be clearly designated.
2. The speeds at each section of the route should be so adjusted so that units will not interfere with each other, either at loading, dumping, or intermediate points.
3. The speeds should be less than the maximum of which the equipment is capable.

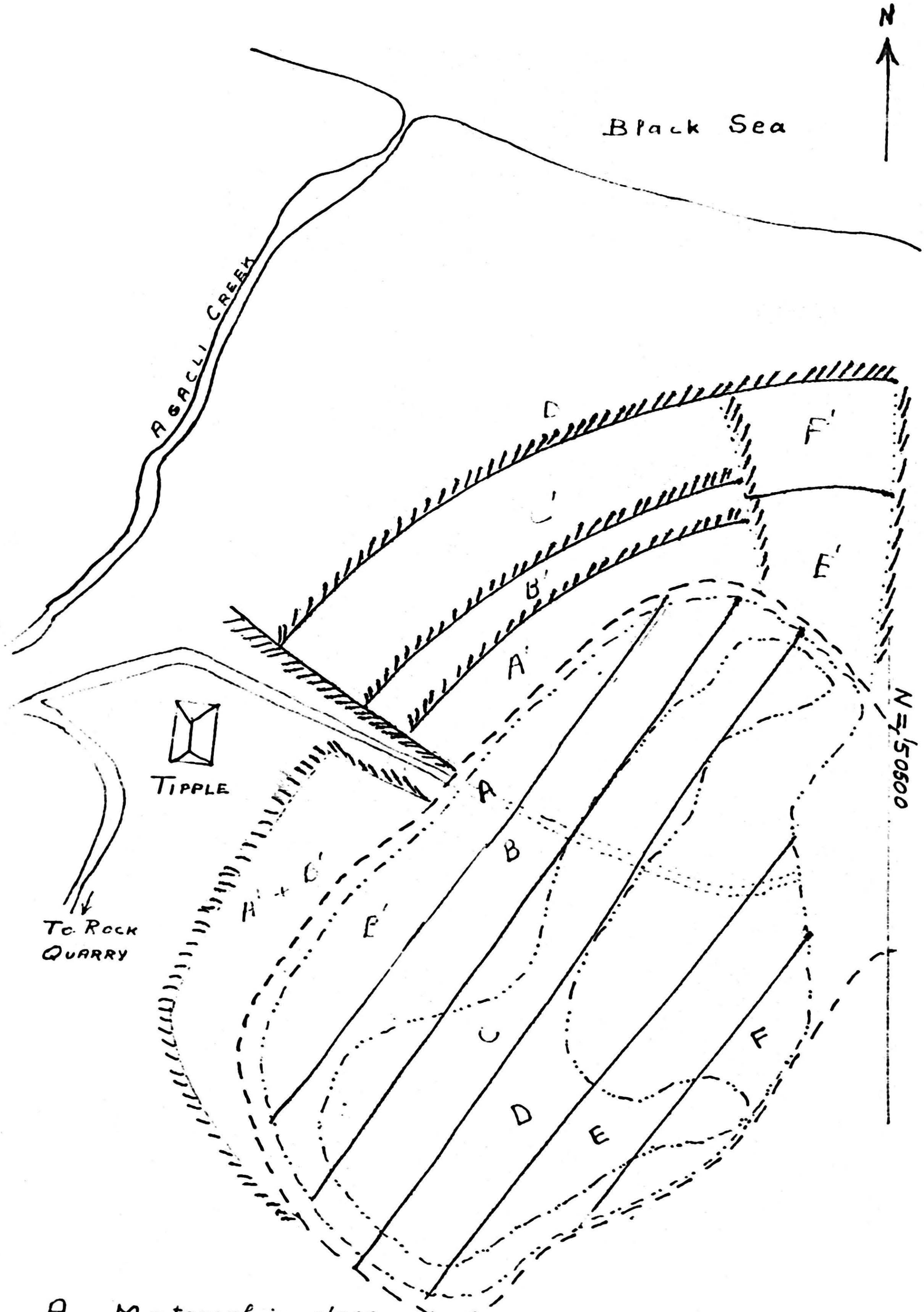
A study of the stripping limits on Map no. 2 shows that most of the earth to be moved with trucks in an area East of N = 50,500 grid line. The estimated amount of overburden of Bed no. 1 and Bed no. 2 in this area is about 4,000,000 cubic yards in place, or 5,200,000 cubic yards loose. As power shovels will deliver 1,250,000 cubic yards loose, per year, the stripping operation in this area will continue a little over 4 years.

In order to recover Bed no. 3, all material above Bed no. 1 and Bed no. 2 must be hauled outside of the stripping area. At Kumtepe, the topography and boundary of the area provide enough spoil space between Agacle Creek and the Black Sea. If more spoil area is required the material can be dumped into the Black Sea.

In the proposed haulage system, trucks will carry spoil material outside of the stripping area and it will form a crescent shape spoil bank. The bank will be kept inclined at about 3 degrees away from the stripping area by a bulldozer. As loaded trucks run over the spoil bank there will be slight compaction.

Map 5 shows the general outline of the lignite beds, sketches of the stripping sections, and the corresponding overburden disposal areas for Bed no. 1. Figures 12 and 13 show the same thing for Bed no. 2 and Bed no. 3, respectively.

As each stripping section advances southward, the dumping points on the spoil banks advance northward. Therefore, the haulage distance during stripping is not constant. There is a sinusoidal change in haulage distances. Starting a stripping section from the north end and going south, the haulage distance increases and it is at a maximum at the south end. Starting a new stripping section at the south end and advancing northward the haulage distance decreases and it is at a minimum at the north end of that section. As the capacity of a stripping shovel is constant, the capacity of the trucks must be changed according to the haulage cycle. The one way maximum haulage distance is about 3000 feet and the average minimum distance



A   Material in place

A'   Spoil Material

Fig 12
Disposal of Stripped Material over Bed No. 2 by Truck.

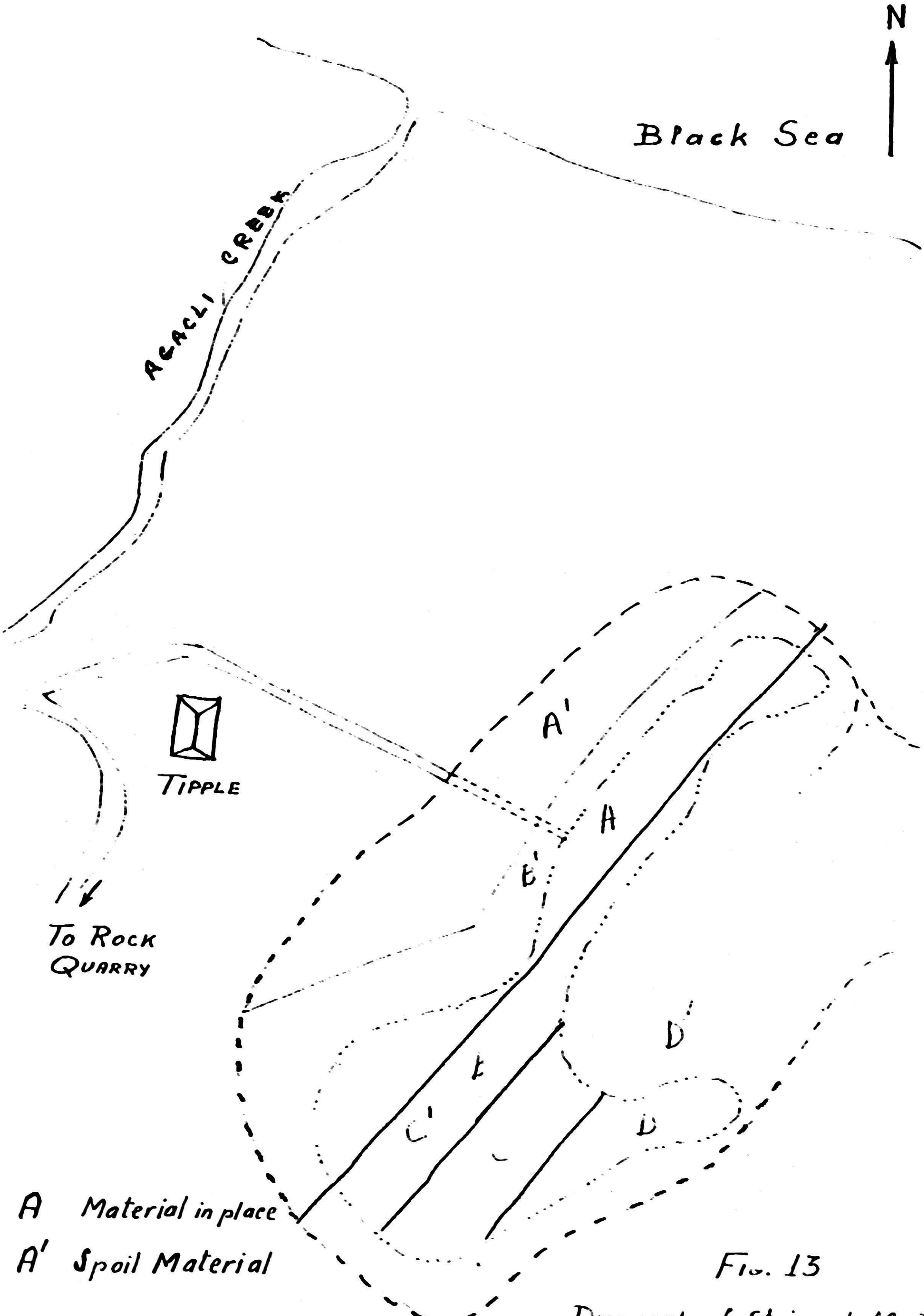


Fig. 13

Disposal of Stripped Material over Bed No. 3 by Shovel is about 200 feet.

Before going into a discussion of haulage cycles a few words should be said about haulage roads. The haulage roads should be smooth, solid roads that stand up under rains, freezes, and thaws. It is highly desirable to eliminate dust, which causes accidents by obscuring the vision of the driver. A great deal of experimenting has been done with road building and surface water proofing in the United States. Most of these efforts were based on using local material as far as possible. "Standard tests for highway road materials do not meet the needs of the mine truck roads. Some of these roads are short lived and there is no time to build a series of road sections for best purposes."[18] As the life of haulage

(18) Richart, Fred W., Good Roads in Strip mine Truck Haulage, Coal Age, August, 1944, p. 97.

roads depends on the life of the mine, the investment in these roads should be returned during the life of the mine. Depending on this life factor, cost of construction of the road and its maintenance should be considered in determining the type of road to be built. Toenges[19] and his associates who

(19) Toenges, Albert L. and Frank A. Jones, Truck Versus Rail Haulage in Bituminous-Coal Strip Mines, Bureau of Mines, R. I. 3416, 1938, p. 20.

studied transportation problems in strip mines gives a few cost figures on the construction of haulage roads. A review of their examples might be suggestive in selecting the type of road to build in this area.

Case 1:

Length of road, ft.	5700
Width of road, ft.	36
Thickness of sub-base, tipple refuse	18
Thickness of crushed limestone, inch	6

Wearing surface, - 2 inch crushed limestone and treated with calcium chloride, average 0.17 # being used per square foot of surface.

Cost of road per foot

Grading, clearing	$1.25
Surfacing	3.75
Total	$5.00

30-ton trucks use the road.

Case 2:

Topography	rough
Length of road	3150 ft.
Length of main road	300 ft.
Width of main road	40 ft.
Width of surface road	30 ft.
Width of surfacing lateral road	16 ft.
Sub-base consist of	
Clay	6 in.
Crushed limestone (-2 - 1/2 in.)	3 in.
Top dressing, clay	

Each layer was rolled before another was put on.

Cost per foot

Grading	$0.62
Surfacing	2.34
Miscellaneous	0.28
Total	$3.24

Case 3:

Topography	flat
Length of road	1.0 mile
Width of surface road	24 ft.
Thickness of sub-base (burnt shale)	18 to 24 inches
Thickness of crushed rock	1 1/2 to 2 inches

Surface is treated w/ 70% asphalt oil, an average 3 quarts of oil being used per square yard of surface.

Cost per foot	$0.95

Shale was hauled 30 miles by freight.

The maximum capacity of truck is 20 tons on the road.

As there is a rock quarry on the property which can supply crushed stone, and considering the cheap cost of labor for the region, the estimated cost per foot of haulage roads should not exceed $2.50 per linear foot with a roadbed depth of 9 inches. About 2,000 ft. of haulage road both for stripping and for hauling coal must be built each year and about 3,000 ft. of haulage road must be maintained daily. Therefore, the yearly expense for building road is $5,000. The maintenance cost has not been included in this estimation but will be included in the estimated profit and loss statement.

Haulage Cycle

The haulage cycle is one of the most important elements of the haulage system. As a haulage cycle is measured by time, other things being equal, the smaller the cycle the larger the output of the truck fleet. The haulage cycle is divided into 5 major divisions:

1. Loading time --

This depends entirely upon operation of the shovel. The approximate loading cycles can be found in Appendix B.

2. Time for hauling from shovel to dump --

There are several factors effecting this time, namely: (1) Time spent for acceleration and deceleration, (2) Time spent for running the haulage distance, and (3) Grade and condition of the road.

3. Dumping time --

This is the time spent by the truck to dúmp material. A proper design at dumping point may reduce this time to a minimum. (See Appendix D)

4. Returning time --

The factors mentioned in item 3 also apply here.

5. Spotting time --

This is the time spent by truck to come into loading position.

As previously mentioned, the haulage system cannot control loading time, and dumping and spotting time is a matter of design which in many cases can be improved. However, a mine operator should give considerable attention to reducing the hauling and returning time. It is obvious that the speeds used on the road sections have a direct effect on the hauling and returning times. It should be remembered, too, that speeds are dependent on the grades, and rolling resistance and design of the trucks. In any truck haulage problem it is essential that the mine operator be aware of the effect of these factors on the haulage cycle. Mr. W. J. Latvala,[20] in his previously

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(20) Latvala, W. J., op. cit.

mentioned thesis, studied those factors (grade ability, grade resistance, and rolling resistance) in great detail with numerous examples. The pertinent information for calculation of truck haulage is given in Appendix D. This information has been used for the calculations which follow shortly.

It is a good idea to have trucks with capacities of at least four times greater than the capacity of the shovel dipper. The ratio of truck size to dipper size should approach a whole number. The shovels selected for stripping operation were 2 1/2 - cu. yd. and 3 - cu. yd. dipper capacity. Trucks with capacities of 15 - cu. yd. and 20 - cu. yd. can be used with these shovels.

The next things to be considered are the grades and the rolling resistances. The grades will be kept to zero degree in the pit and about 3 degrees on the spoil dump. A study of Appendix D-b and c shows that rolling resistance and grade resistance can be added algebraically. Therefore, if the grade of the road is - 3% and rolling resistance is 8% the net grade resistance is equal to

(- 3% + 8%) Gross vehicle weight = 5% of G.V.W.

Performance charts published by leading truck manufacturers can be used for quick estimations of the minimum speeds attainable. In Appendix D performance charts for 15-ton rear-dump, 22-ton rear-dump, and 20-ton bottom-dump Euclid trucks are given. In the example cited above, the resistance to be overcome by the truck is:

5% - 2% = 3% of Gross Vehicle weight

(2% rolling resistance was allowed in calculation of the charts)

The maximum speeds of the trucks under this condition are:

For 15-ton truck, empty 28.0 MPH; loaded 17.8 MPH

For 22-ton truck, empty 32.0 MPH; loaded 20.6 MPH

The above method will be used in cycle calculations, but there are other methods applicable to all kinds of trucks. The speeds and grade ability of all trucks can be found by using the reference formulae given in Appendix D-g.

In some cases, the total of grade resistance and rolling resistance might be a negative figure. For instance, grade of the road is - 6% and rolling resistance is about 2%, their algebraic is equal to - 4% of gross vehicle weight. Under this particular condition, a truck driver must use brakes or must run in a low gear to use the motor as a brake. Cost of maintenance and fuel consumption are high if this situation is allowed to exist.

In designing haulage roads, grades and rolling resistance should be so adjusted that the net result will be approximately equal to zero for loaded trucks. For this proposed haulage system, the adjustment of grades is less expensive than the adjustment of rolling resistance. Rolling resistances for seven types of road surfaces are listed in Appendix D-b. There are two types of road surfaces in this list which might closely resemble the haulage roads at Kumtepe, namely: (1) Soft unplowed dirt or poorly maintained dry dirt, rutted surface, and (2) Soft plowed dirt or unpacked dirt fills. The estimated rolling resitances of the roads in the pit and on the spoil bank are 4% and 8% of gross vehicle weight, respectively.

Haulage Cycle Calculations:

The cycle will be calculated to determine the most economical haulage distances for 15-ton and 22-ton trucks under similar road conditions. References are given only for the calculations of Case 1.

Case 1: 15-ton truck, 2 1/2 - cu. yd. shovel, haulage distance 500 ft. (one way), - 3% grade, R_0R_1 = 6% of GVW, R_0R_1 = rolling resistance, GVW = gross vehicle weight, Haulage Cycle = loading time + hauling time + dumping time + returning time + Spotting time at the shovel.

A. Loading time = 4 X 22 sec. = 88 secs. = 1.47 min. (Appendix B-f)

B. Hauling time

Haul Road	R_0R_1	Grade %	Average Speed	Average Time
500 feet	6%	-3%	14.0 MPH	.40 min.

(Notes on calculations:

1) 6% - 3% - 2% = 1% corresponds 28.0 MPH max. Appendix D-c, h.
2) Average speed, Appendix D-d, factor = .50
3) Average time, Appendix D-f)

C. Dumping time, Appendix D-e 2.0 min.

D. Returning time:

Haul Road	R_0R_1	Grade	Average Speed	Average time
500 ft.	6%	+3%	8.9	.60 min.

E. Spotting time (Appendix D-e) .30 min.

F. Total time per Hauling Cycle (A+B+C+D+E) 2.77 min.

G. Trips per 50 min. hr. 18.0 trips per hour

H. Hourly production (10.0 loose cu. yd. per truck)

10.0 X 18.0 = 180 loose Cu. yd. per hour.

J. Production per shift per truck

$$180 \frac{\text{cu. yd.}}{\text{hr.}} \times 7.25 \times \frac{\text{hrs}}{\text{shift}} = 1310 \text{ cu. yd. / shift / truck}$$

I. No. of 15-ton trucks needed (with one spare truck): 3

Note: 2 1/2 - cu. yd. shovel output per shift is 2240 cu. yds. loose

3 - cu. yd. shovel output / shift 3100 cu. yds. loose

Case 2: Hauling Distance 1000 ft.

Conditions:

Road in the pit Average: 700 ft. R_0R_1 = 4%, Grade = 0%

Road on spoil bank Ave.: 300 ft. R_0R_1 = 8%, Grade = 3%

Truck 15-ton Rear-dump

Shovel 2 1/2 - cubic yard.

A. Loading time 1.47 min.

B. Hauling time

Haul Road Section	Length	R_0R_1	% Grade	Average Speed	Hauling time
In the pit	700	4%	0	10.5	.75
On Spoil Bank	300	8%	- 3%	12.5	.27

C. Dumping time 2.00

D. Returning time

Haul Road Se Section	Length	R_0R_1	% Grade	Ave. Speed	Returning time
On Spoil Bank	300	8%	+3%	6.8	.50
In the Pit	700	4%	0	19.6	.40

E. Spotting Time .40

F. Total time per cycle 5.69

G. Trips per 50 min. hr. 8. 95 trips / hr.

H. Hourly Production = 10.0 X 8.9 = 89.5 Loose cu. yd. per hr.

J. Production per shift:

89.5 X 7.25 = 650 Loose Cu. Yd.

I. No. of 15-ton trucks needed: 5

Case 3: Hauling Distance 1500 ft.

Conditions:

Road in the pit, 1000 ft., R_0R_1 = 4%, Grade = 0%

Road on spoil bank, 500 ft., R_0R_1 = 8%, Grade = -3%

15-ton trucks, 2 1/2 - cu. yd. shovel

A. Loading time 1.47 min

B. Hauling time

Haul Road Section	Length	R_0R_1	% Grade	Ave. Speed	Hauling time
In the pit,	1000 ft.	4%	0%	12.9	.79
On the spoil bank,	500 ft.	8%	-3%	12.5	.54

C. Dumping time 2.00

D. Returning time

Haul Road Section	Length	R_0R_1	% Grade	Ave. Speed	Hauling time
On spoil bank,	1500 ft.	8%	+3%	6.7	.85
In the pit,	1000 ft.	4%	0%	22.4	.50

E. Spotting time .30

F. Total time per cycle (A+B+C+D+E) 6.45

G. Trips per 50 min. hr. 7.75 trips / hr.

H. Hourly production = 10.0 X7 = 77.5 Loose cu. yd. per hr.

I. Production per shift = 77.5 X 7.25 = 564 Loose cu. yd. per shift

J. No. of 15-ton trucks needed: 6

Case 4: Hauling Distance 2000 ft.

Conditions:

In pit, 1000 ft., $R_0R_1 = 4\%$ Grade = 0%

On spoil bank, 1000 ft., $R_0R_1 = 8\%$ Grade = 0%

A. Loading time

B. Hauling time 1.47 min.

Haul Road Section	Length	R_0R_1	% Grade	Ave. Speed	Hauling time
In the pit	1000 ft.	4%	0%	12.9	.89
On spoil bank	1000 ft.	8%	-3%	14.2	.82

C. Dumping time 2.00

D. Returning time

Haul Road Section	Length	R_0R_1	% Grade	Ave. Speed	Hauling time
On spoil bank	1000 ft.	8%	+3%	10.7	1.09
In the pit	1000 ft.	4%	0%	22.4	.50

E. Spotting time .30

F. Total time per cycle 7.07

G. Trips per 50 min. hr. 7.06 trips per hr.

H. Hourly production 10.0 X 7.06 = 70.6 Loose cu. yd. per hr. per truck

I. Production per shift 70.6 X 7.25 = 512 cu. yd. per shift

J. No. of 15-ton trucks needes: 6

Case 5: 22-ton trucks, and 3 - cu. yd. shovel

500 ft. hauling distance, R_0R_1 = 6% Grade = -3%

A. Loading time 2.00 min.

B. Hauling time

Haul Road Section	Length	R_0R_1	Grade	Ave. Speed	Hauling time
Haul Road	500 ft.	6%	-3%	16.0MPH	.34

C. Dumping time 2.00

D. Returning time

Haul Road Section	Length	R_0R_1	Grade	Ave. Speed	Hauling time
Haul Road	500 ft.	6%	+3%	12.3 MPH	.90

E. Spotting time .30

F. Total time per cycle 5.54

G. Trips per 50 min. hr. 9 trips

H. Hourly production 15 $\frac{\text{cu. yd.}}{\text{trucks}}$ X $\frac{\text{9 trips}}{\text{hour}}$ = 135 loose cu. yd. per hour

I. Production per shift 135 X 7.25 = 980 loose cu. yd. per shift per truck

J. No. of trucks needed: 5

Case 6: 22-ton trucks and 3 - cu. yd. shovel

Haulage distances

700 ft. in the pit R_0R_1 = 4% and Grade = 0%

300 ft. on spoil bank, R_0R_1 = 8% and Grade = -3%

A. Loading time 2.00

B. Hauling

Haul Road Section	Length	R_0R_1	Grade	Ave. Speed	Ave. time
In the pit	700 ft.	4%	0%	12.3	.60
On spoil bank	300 ft.	6%	-3%	25.8	.13

C. Dumping time 2.00

D. Returning time 300 ft

On spoil bank	300 ft.	6%	+3%	12.3	.27
In the pit	700 ft.	4%	0%	16.2	.49

E. Spotting time .30

F. Total time per cycle 5.79

G. Trips per 50 min. hr. 8.65 trips per hour

H. Hourly production 15.0 X 8.65 = 130 Loose cu. yd.

I. Production per shift 130 X 7.25 = 940 cu. yd. per hour per truck

J. No. of trucks needed: 5

Case 7: 22-ton truck and 3 - cu. yd. shovel

Haul Distances

1000 ft. tin the pit, R_0R_1 = 4%, Grade = 0%

500 ft. on the spoil bank, R_0R_1 = 8%, Grade = -3%

A. Loading time 2.00 min.

B. Hauling time

Haul Road Section	Length	R_0R_1	% Grade	Ave. Speed	Ave. time
In the pit	1000 ft.	4%	0%	14.8	.77
On spoil bank	500 ft.	8%	-3%	10.3	.54
C. Dumping time					2.00
D. Returning time					
On Spoil bank	500	8%	+3%	10.3	.54
In the pit	1000	4%	0%	26.2	.43
E. Spotting time					.30
F. Total time per cycle					6.58

G. Trips per 50 min. hr. 76 trips per hour

H. Hourly production 15 X 8.6 = 114 loose cu. yd. per hour per truck

I. Production per shift 114 X 7.25 = 826 cu. yd. per shift per truck

J. No. of trucks needed: 5

Case 8: 22-ton truck and 3 - cu. yd. shovel

Hauling Distances

1000 ft. in the pit, R_0R_1 = 4%, Grade = 0%

1000 ft. on spoil bank, R_0R_1 = 8%, Grade =-3%

A. Loading time 2.00

B. Hauling time

Haul Road Section	Length	R_0R_1	% Grade	Ave. Speed	Ave. time
In the pit	1000 ft.	4%	0%	14.8	.77
On spoil bank	1000 ft.	8%	-3%	12.3	.93

C. Dumping time 2.00

D. Returning time

On spoil bank	1000 ft.	8%	+3%	12.3	.93
In the pit		4%	0%	26.2	.43

E. Spotting time .30

F. Total time per cycle 7.36

G. Trips per hour 6.8 trips per hour

H. Hourly production 6.8 X 15.0 = 102 loose cu. yd. per hour / truck

I. Production / shift 7.25 X 102 = 740 cu. yd. / shift / truck

J. No. of trucks needed: 6

Cost Analysis for Truck Haulage

Trucks will be depreciated in four years, or in 8,000 hours. Although an average life for a truck is about 15,000 hours, no salvage value will be considered in depreciation schedules. The cost of the original tires will be included in operating cost, not in depreciation charges.

The method of cost analysis for trucks is similar to that method of cost analysis for shovels. The cost analysis for 15-ton and 20-ton Rear-dump Euclid Trucks follows.

Ownership Costs

	15-ton	22-ton
A. Depreciation		
Purchase price	$23,190	$28,400
Freight, to Istanbul		
10% of purchase price	2,320	2,840
Delivered price	$ 25,510	$31,240
Less cost of tires (6 tires)	3,888	4,764
Total amount to be depreciated	$ 21,622	$26,476
Hourly depreciation cost (8000 hrs. dep. period)	$2.70	$ 3.32
B. Interest, taxes, insurance (% of yearly investment) $\frac{\text{(Delivered price)(62.5\%)}}{\text{2000 hrs. per yr.}}$ X 10%	.80	.98
Total hourly ownership cost	$3.50	$4.30

Operating Costs

	15-ton	22-ton
C. Hourly tire cost (Estimated tire life 2500 and 3500 hrs.)	$1.55	$1.36
D. Tire repairs 15% of hourly tire cost	.23	.20
E. Repairs (including parts and labor)(est. by manufacturer)	.73	1.16
F. Fuel 4.0 and 4.5 gal/hr at $0.31	1.24	1.40
G. Oil, grease, including labor per hour	.25	.25
H. Truck operator, labor	.50	.50
Total hourly operating cost	$4.50	$4.87
Total estimated hourly ownership and operating cost	$8.00	$9.17

Table X

Summary of Cycle Calulations
and
Cost Analysis of Truck Haulage

Haulage Distance (ft.)	15-ton trucks and 2 1/2 cu. yd. shovel				22-ton trucks and 3 cu. yd. shovel			
	500	1000	1500	2000	500	1000	1500	2000
Cycle time, min.	2.77	5.69	6.45	7.07	5.54	5.79	6.58	7.36
Trips per 50 min. hour	18	8.95	7.75	7.06	9.00	8.65	7.60	6.80
Cu. yds. per hr. per truck	180	89.5	77.5	70.6	135	130	114	102 66
Cu. yds. per shift per truck	1310	650	564	512	980	940	826	740
No. trucks needed (with one spare)	3	5	6	6	5	5	5	6
Total ownership and operating cost of each fleet per hr.	$19.50	$35.50	$43.50	$43.50	$41.10	$41.10	$41.10	$50.15
Cu. yds. per fleet per hr.	360	358	388	353	540	520	456	510
Cost per cu. yd.	$0.054	$0.099	$0.112	$0.123	$0.076	$0.079	$0.090	$0.098

The results of cycle calculations and cost analysis of truck haulage are summarized in Table X. A study of this table indicates that the rate of decrease in production is especially rapid as distances go beyond 1000 ft. The production rate for a 22-ton Rear-dump truck decreases slowly for the distances between 500 ft. and 1500 ft. For extents greater than 1500 ft, the decrease in production becomes more rapid. It follows, therefore, that 15-ton trucks are more economical for short distances, but that 22-ton trucks are more economical for hauls around 1500 ft.

The average hauling distance of Kumtepe is to be 1500 ft. Therefore, ten 22-ton Rear-dump trucks are needed to serve the stripping shovels. A 5-truck fleet will be assigned to 2 1/2 - cu. yd. shovel and another 5-truck fleet will be assigned to 3 - cu. yd. shovel. This allows one spare truck per fleet.

Proposed Stripping and Mining Method

The average depth of overburden of Bed no. 1 is about 35 ft. The maximum cutting height is 37' 0" at 60 degrees for a 3 - cu. yd. shovel and 48' 0" at 55 degrees for a 2 1/2 - cu. yd. shovel.

It is not always possible nor advisable to operate the shovel at maximum cutting height. There is an "optimum depth of cut" for each shovel. This depth depends upon the physical character of the bank and can be determined in the field. If the height of the bank is much greater than the optimum depth, some auxiliary equipment must be used to reduce the height to the optimum depth of the shovel.

It is thought that during the major part of the operation, preparation of the bank by blasting will not be necessary. One or two bulldozers working on the bank ahead of the shovel, can reduce the height of bank. For this reason, preparation of "high wall" will not be discussed at this stage but in the interest of completeness, it will be mentioned in a latter part of this study.

The average angle of swing should by kept as low as possible when using a stripping shovel. The shovel production figures used previously were based on 90 degree angle of swing. To lower the loading time, the average angle of swing should be less than 90 degrees. Fig. 14 shows a method of reducing, swing angle and eliminating spotting delays.

The position of the shovel with respect to the bank depends upon the height of the bank and physical character of the bank. If caving and sliding occur frequently, the shovel should operate at a safe distance from the bank. The maximum width of the banks should be equal to the maximum cutting radius of the shovel to eliminate unnecessary forward and backward movements of shovel.

The loading cycle of shovel and the position of the truck are shown in Fig. 15. Fig. 16 is sketch of general stripping plan.

Haulage will be unnecessary in some places (see figures 12 and 13). The shovel can cast spoil into the area from which lignite has been removed. Fig. 17 illustrates a method of casting using two shovel in tandem. This method is used when the height of spoil pile depends upon three basic factors, namely: (1) swell factor of material, (2) angle of repose of loose material, and (3) maximum dumping height of the shovel. Occasionally the maximum dumping height of the shovel will not be high enough to spoil all of the overburden from a given cut. Where this is the case, a small bulldozer can be used to spread the overburden, or trucks can be used to transport a little portion of the overburden in the normal manner.

Stripping shovel leaves a six inches thick overburden. The reasons for this protective blanket may be summarized as follows:

1. To reduce crushing of the lignite by trucks and the shovel.
2. To prevent long-time exposure of lignite to air which would increase slacking.

That thin layer of overburden may be stripped by means of bulldozer, before loading shovel start to load.

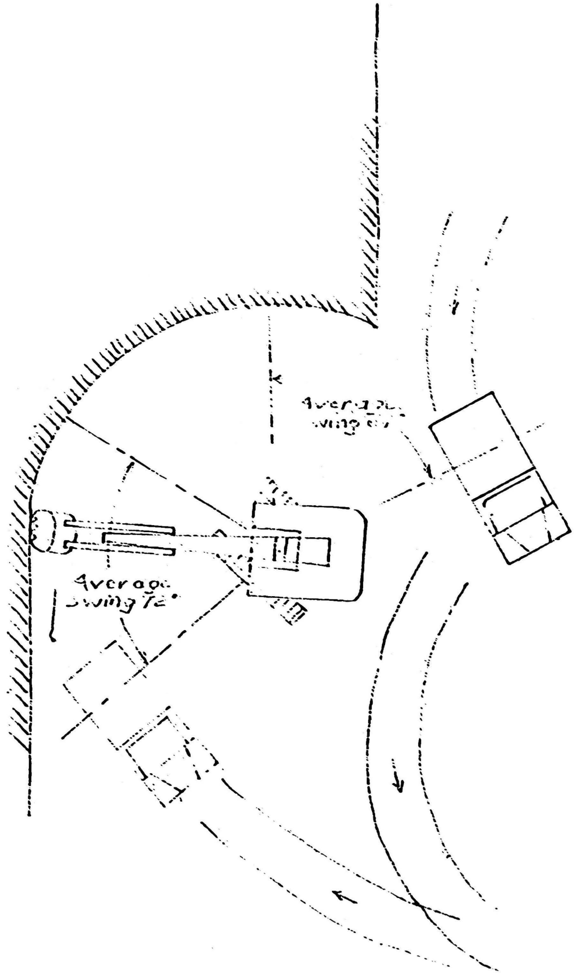

Fig. 14

Double spotting to decrease swing angle

(After W. J. Lutvala)

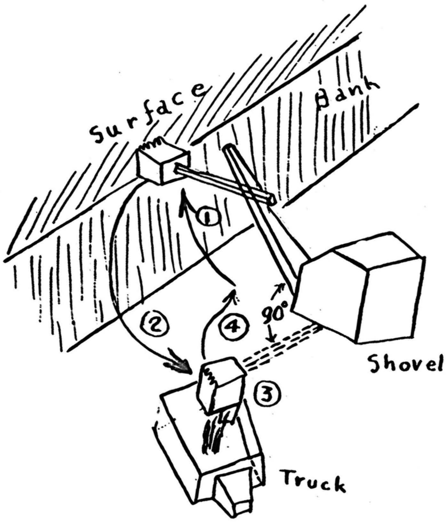

Fig. 15
Loading Cycle of a Shovel
and Position of the Truck

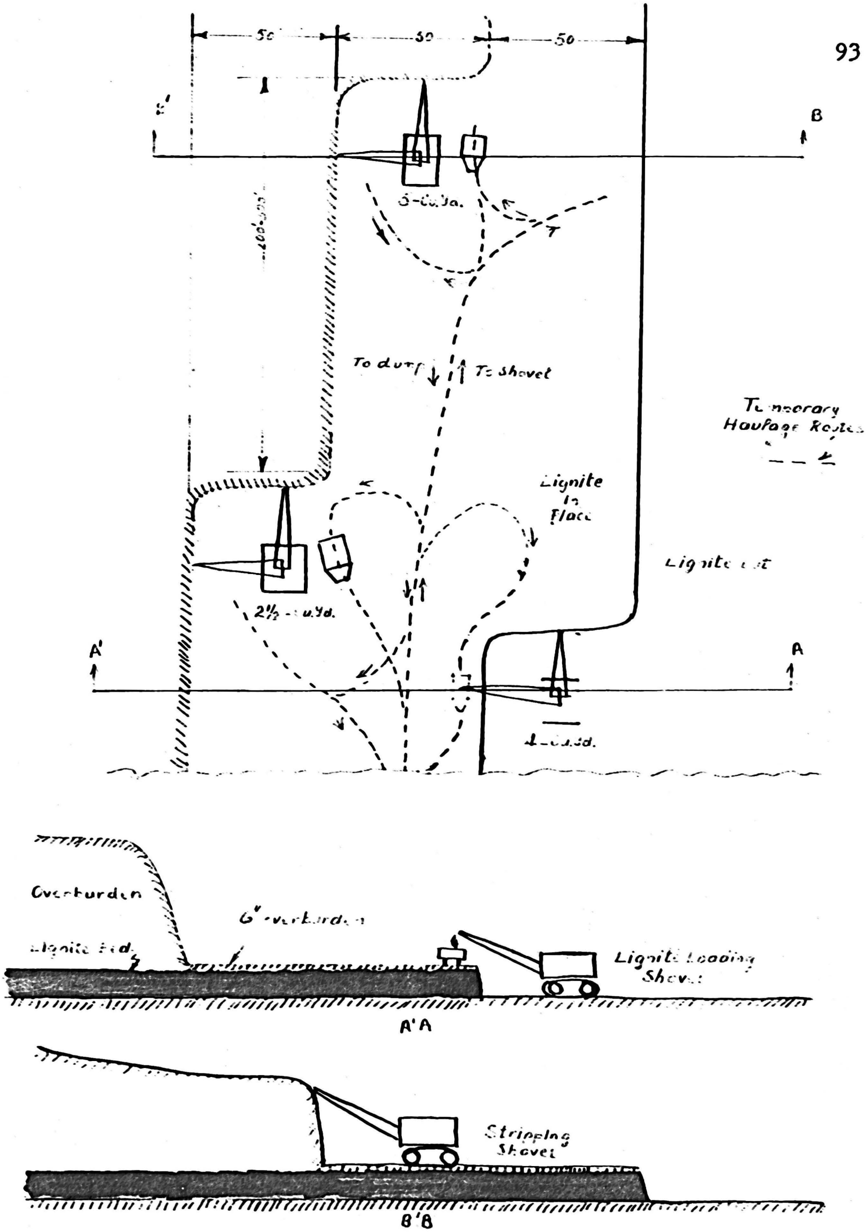

Fig. 16.

Sketch of Stripping and Mining Method by Shovel and Truck.

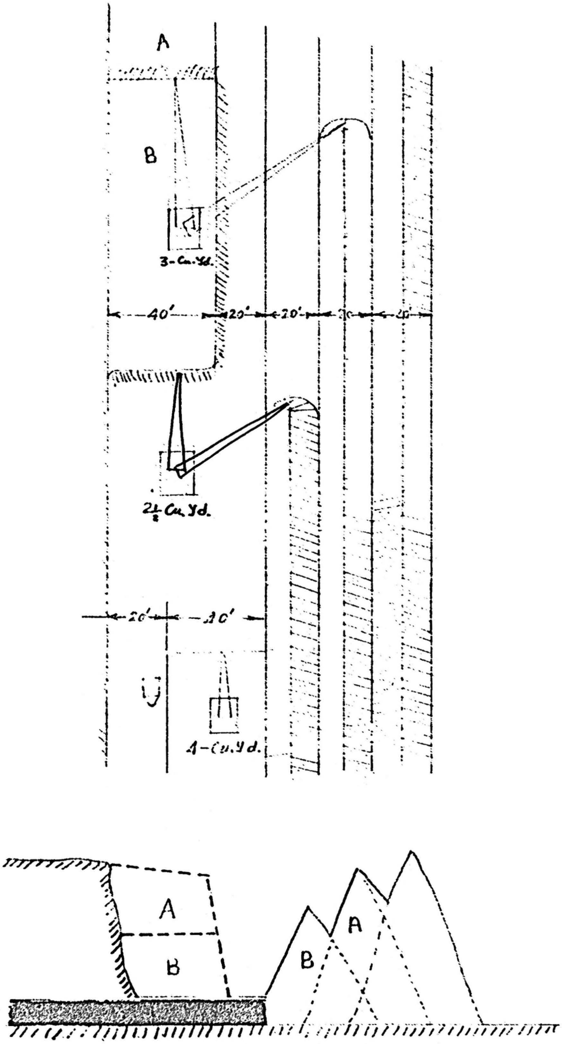

FIG. 17

Stripping and Casting by two Shovel in Tandem.

Coal loading equipment should be balanced against the daily output requirement of the number of working places, and the size of transportation units. As the specific gravity of lignite (1.25) is much lower than that of overburden (about 2.6) it is possible to equip a small shovel with an oversized light weight dipper and acheive a 25 to 60 per cent increase in capacity. For example, with a Marion Type 111 - M three sizes of dippers, 3 1/2 - cu. yd., 4 - cu. yd., and 5 - cu. yd., can be used. Fig. 18 shows the standard loading practice.

The hauling distance for the lignite varies from 300 ft. to 3,000 ft. The average distance will be about 500 ft. It was calculated that a Marion type 111 - M Diesel powered shovel equipped with a light weight 4 - cu. yd. dipper can produce 2,480 metric tons of lignite per shift. As daily production is to be 4,000 m. tons. this shovel must be operated two shifts per day. The estimated cost per ton for loading lignite is $0.07.

For the average haul distance of 1500 ft., three trucks (20-ton Bottom-dump, Euclid Model) are needed per shift. The estimated hourly ownership and operating cost is $9.00 per truck.

Hourly cost of owning and operating of three trucks:	$27.00
Hourly cost of owning one spare truck (estimated):	4.00
Hourly cost of fleet of four trucks:	$31.00

$$\text{Estimated hauling cost per ton for average distance of 1500 ft.} \quad \frac{\$31.00}{2000 \text{ tons}} = \$0.155 \text{ per ton}$$

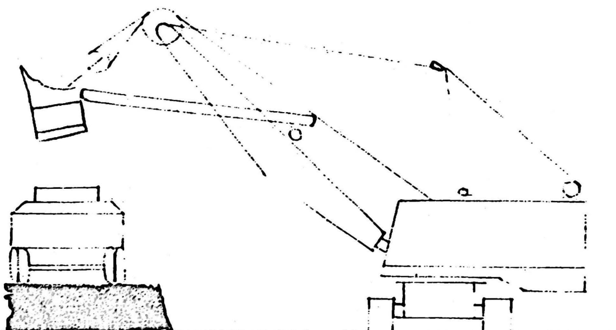

Fig. 18

Marion Type 111-M Coal Loader

(After W. J. Latvala)

Other Auxiliary Equipment Used at Strip Mines

Scrapers

The use of scrapers at strip mines are numerous. They dig, haul, and spread the spoil. Scrapers are independent stripping units and are operated by one man.

The typical applications of scrapers to coal stripping are:

a) Stripping overburden on small properties where amount of recoverable coal and life of property do not warrant investment in large stripping machines.

b) Stripping light overburden that does not present an adequate face for economical shovel operation.

c) Auxiliary stripping, where overburden is so thick that the top material must be removed to allow shovel to work within its limitations.

d) Widening a bench at side of the pit to provide greater casting area for a shovel.

e) Hauling coal from the pit, in conjunction with shovel loadings.

f) Construction and maintenance of drainage ditches and haul roads.

At many mines in the United States, scrapers are being used successfully in stripping operations especially since manufacturers have marketed larger size scrapers by rubber-tire-mounted tractors. For the following reasons, scrapers are not economical to use on Kumtepe lignites:

a) There are three lignite beds to be mined. At many places casting and piling of spoil material by shovels is less expensive than to use scrapers.

b) In general, even rubber mounted scrapers do not have as much speed as trucks.

c) Although the topography is favorable for hauling spoil material down grade, the return trip, upgrade must be accomplished with the scraper unloaded.

d) As the material is mostly clay and sandy clay, when the material is wet loading the bowl of scrapers even with pusher will be difficult.

e) Scrapers usually are applied to stripping shallow overburden, at Kumtepe the average depth of overburden is from 30 to 40 ft.

Although scrapers cannot replace the shovel-truck system at this mine, they may be used to reduce thickness of overburden to the limits of the stripping shovel.

Bulldozers: There are many kinds of dozers and bulldozers generally are very useful equipment at strip mines. They can be used to clean loose material around shovels, to maintain haulage roads and ditches, and to help the stripping shovels by reducing the height of the bank. Bulldozers, if used within 100-200 ft. hauling distance, are very economical stripping units.

At Kumtepe lignites each stripping shovel will receive help from two bulldozers, one operating on the bank, the other operating in the pit around the shovel. Another bulldozer will remove the 6" of overburden left on the lignite by stripping shovel ahead of the lignite loading shovel.

The estimated hourly owning and operating expenses of a Bucyrus-Eric Bulldozer with TD-9 crawler tractor, and that of a Bucyrus-Erie Bullgrader with same tractor are $3.75 $4.00 each. In calculating hourly owning and operating expenses, the percentage figures for "total ownership expense" are tenken from "Contractor's Equipment, Ownership Expense" published by the Associated General Contractors of America, Inc., and hourly operating cost is estimated at $2.50 an hour.

Kumtepe Lignites
Estimated Profit and Loss

Sales: 10,000 m. tons / yr.				
at $5.30 F.O.B. mine				530,000
Cost of Lignite Sold				
Stripping costs (Ave.)	$/cu. yd.		$ / shift	
Power shovel operation	$0.057			
Hauling overburden	0.090			
Five bulldozers	0.020	$0.167		
(at $2.50 / hr. each)				
Total stripping cost / shift			$ 835	
(5000 cu. yd. per shift)				
Mining cost	$ /ton			
Power shovel	$0.070			
Hauling lignite	0.155			
One bulldozer in pit	0.010	$0.237		
Mining cost per shift		$ 474		
Mining cost per second shift		$ 474		
Total mining cost			948	
Maintenance cost per shift				
(Estimated for grading				
roads, cleaning ditches, etc.)			50	
Total Mining and Stripping cost				
per shift			$1833	
Total cost per ton of Lignite			$0.46	
produced				
Total cost per year ($0.46 X 100,000)				46,000
Gross profit per year				$ 484,000
General Administrative Expense				
(Salaries, office supplies,				
depreciation on bldgs. and				
furniture, light, heat, tel-				
ephone, etc.)			$125,000	
Financial Management				
Prorated development expense			$ 20,000	
Royalty $0.20 / ton lignite				
produced			$ 20,000	
Total operating expense				$165,000
Net operating profit before taxes for year				$319,000
Income and other taxes at 37% of net profit				$118,000
Profit for the year				$201,000
Profit per ton of lignite produced				$2.01

Note:

The cost figures are taken from previous calculations

The administrative expenses, the amount of money invested for prospecting, royalty rate per ton coal produced and the percentage of income tax were supplied by the M. T. A., Ankara, Turkey.

There are some hidden expenses which depend, undoubtedly, upon the policy of management and/or existing labor laws, such as recreation facilities for employees, vacation with pay, and funds for accidents, etc. Therefore, the anticipated profit of $2.00 per ton may drop to $1.75 per ton of lignite produced.

Estimated Initial Investment

For Equipment Delivered at Istanbul, Turkey

Stripping Units		
2 1/2 - cu. yd. shovel	$ 79,088	
3 - cu. yd. shovel	100,970	
10 22-ton Rear-dump Euclids	312,400	$492,458
Mining Units		
4 - cu. yd. shovel, coal loader	102,000	
4 20-ton Bottom-dump trucks	90,000	$192,000
Auxiliary Equipment		
6 Bulldozers	51,600	
1 Bullgrader	8,900	$ 60,500
Total Investment		$744,958

MAINTENANCE AT STRIP MINES

According to Webster's dictionary, "Maintain" means "to hold or keep state or condition, especially in a state of efficiency or validity". Equipment maintained in first-class condition eliminates delays due to breakdowns, and thus renders invalid one of the most overworked excuses for low production.

It is always necessary to devote money and time in the maintenance of equipment and roads in coal mines. When a mine is highly mechanized, it becomes increasingly important to keep equipment in good operating condition.

The costs due to a breakdown of mine equipment can be considered as the following example of a shovel breakdown:

1. Hourly cost of ownership of shovel.
2. Hourly cost of labor, (shovel operator, oiler).
3. Hourly cost of truck-fleet serving the shovel.
4. Hourly cost of repairmen.
5. Cost of time spent by foreman.
6. Cost due to loss of production, overtime payment, if any.
7. If the breakdown is of major importance hourly cost of operating washing plant also has to be considered.

Organization of Maintenance Department

An efficient mine manager understands the need for a well organized and equipped maintenance department. Mr. Chas. R. Nailler of Hanna Coal co., Neffs, Ohio says "Maintenance and its proper organization has long been a neglected problem in the mining industry. The reason maintenance has lagged behind other branches of the industry is that the greatest emphasis has always been placed on operation. Now, mechanical mining has forcibly brought to our attention the need of maintenance into the coal production system." (21)

(21) Nailler, Chas. R., Organization of Maintenance Crews in Mechanical Loading, Coal Mine Modernization, Year Book, 1940, p. 157.

A maintenance organization should have some system of reporting and recording machine condition, work needed, and repairs accomplished, Preventive measures should take first place in maintenance work, rather than fixing equipment after breakdown has occurred. Maintenance programs must be supported by an adequate supply of parts and materials to meet normal day-to-day demands. Ample shop facilities for repairs and overhauls should be available. There should be enough tools of the right types. A maintenance system should provide for the following: (22)

(22) Nailer, Chas. R., ibid. p. 158

1. Field inspection
2. Work scheduling
3. Proper handling of maintenance personnel.

1. Field Inspection:

The maintenance system should have inspector mechanics who check on functioning of equipment and its lubrication. Through the inspector's reports more effective scheduling of repairs can be made.

2. Work Scheduling:

a) Unit maintenance: The modern complex machine is made up of various units such as, motors, hydraulic system, gearing, etc. Each unit differs greatly in its length of useful service and should be replaced or repaired accordingly.

The necessity for regular rebuilding of all types of mining equipment cannot be over stressed, for continuous and economical operation of such equipment depends on keeping the equipment in good shape.

b) Adequate supply of materials:

The need for materials should be promptly reported to the purchasing agent or department to provide adequate time for delivery, thus avoiding unnecessary delays.

c) Running and break-down repairs:

Here "running repairs" refers to those of a comparatively minor nature, which do not take the machine out of service for any extended time. If the equipment inspection is conducted efficiently and action is taken promptly on the basis of the inspectors' reports a great number of break-downs repairs may be treated as running repairs. Thus waste of valuable time and expense is greatly reduced.

3. Proper Personnel Handling:

The mester mechanic should be selected not only because of his ability to perform maintenance work, but also because he is production-minded. One of the constant aims of management should be building up personnel and training the mechanics.

<u>Diesel Engines</u>:

Proper functioning of the cooling system of the diesel engine is vital to its efficient operation. Initial problems caused by overheating may be operational in character, such as, loss of power, or increased fuel and oil consumption. Operation of a diesel engine at excessively high temperatures will often lead to more serious consequences. All metals loose strength when they are heated. They undergo considerable expansion during heating, and contraction during cooling. The forces due to expansion and contraction are opposed by shape of parts and by part confinement, and thus high internal stresses are developed. These stresses increase as the temperature rises while at the same time the strength of material is reduced. When the resultant strain becomes greater than the allowable strain of the metal, mechanical failure will result. Therefore, the importance of efficient cooling systems is obvious.

In most cases, water is used as a coolant. The water in many localities contains chemical impurities that produce sediments in the cooling system. The sediment acts as an insulation between the heated surfaces and the coolant. Clear rain water or soft water are preferable as coolants. Antifreeze coolants should have a high boiling point, as diesel engines operate in a range of about 180° F to 200° F. A diesel engine should not be operated before the cooling system is filled. Mr. Walter W. Black, manager of service and parts dept. of International Harvester Co. suggests the following procedure to find the cause of overheating of diesel engines.(23)

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(23) Black, Walter W., Diesel Cooling System Maintenance, <u>Mechanization</u>, September 1948, P. 97

1. Check fan belt for slippage and tension.
2. Check water pump impeller and impeller shaft.
3. Check rubber hose connection for leaks.
4. Check engine block for scaling and sediment.
5. Check radiator fins for obstruction that will prevent free air flow.
6. Keep cooling system full.
7. Check occasionally for aerated coolant while engine is running. If the coolant appears cloudy or filled with small air bubbles, it is very likely that air is entering the cooling system. Aerated coolant will cause quick rusting.

Wire Ropes:

One of the important cost items of strip mines probably is the cost of replacing wire ropes. "Manufacturers have always preached rightly that a wire rope is a machine and deserving of the same consideration (as equipment itself). If long life is to be secured, the ropes must not be subjected to numerous indignities such as kinking, sharp bending around an angular post, and plain jerking."[24]

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(24) Wire Rope and Cable, Coal Age, October, 1942, p. 110

Lubrication, in general, greatly extends the life of wire ropes. However, lubrication of a rope which is wet inside is ineffective, and may even damage the rope by confining the water inside.

Changing the ends of the rope to move points of stress to new places, increases life of the wire considerably. Often switching one type of wire to another also increases the life of the wire. As an example, "Life of the 1/2 - in. 6 X 19 wire ropes in controlling Le Tourneau scraper was increased 400 per cent by changing from regular lay and plain to longlay and preformed."[25]

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(25) Strip Cost Halved at Blue Bell "Cash and Carry Mine", Coal Age, April 1939, p. 75

The cause of rope failure often can be determined by appearance of individual broken wires. A broken wire which shows no reduction in cross section might indicate that the maximum bending strength of the rope has been reached. A reduction in sectional area may indicate that the maximum safe stress has been exceeded, particularly if the break is cup-shaped on one side. The bending stresses can be reduced by using proper size drums and sheaves. Overloading should be prevented to reduce internal stresses in individual wires.

Lubrication:

Each part of the equipment should be lubricated periodically with recommended lubricants. The period of lubrication depends upon working conditions. Type of lubricants are usually recommended by the equipment manufacturers. Storing and handling lubricants is important, and steps should be taken to prevent contamination. Definite schedules of lubrication should be established. A good mine operator will not extend the periods of lubrication to save immediate cost of lubricants.

Trucks and Tires:

Cost of tires are treated as operating costs in a truck haulage system. The total cost of tires for each truck varies from $2,000 to $4,000. The average life of tires depends upon several factors. These factors and their effects are shown in Fig. 19. The maintenance of tires should be considered important because of this high initial investment. "Examination of many discarded tires indicates that they are cut to pieces rather than worn out. Good road, therefore, are an absolute necessity."[26] One foreman states that reducing pressure

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(26) Trucks and Trailers, Coal Age, October, 1942, p. 128

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from 90 to 70 pound per square inch reduced tire cuts by 75 per cent.

The development of tires for military use during World War II has taught the rubber industry many things about the operation of rubber tired vehicles over adverse terrain. It was found that a load could be carried on a tire with lower pressure, if their section width were increased, rather than increasing the diameter.

Low pressure tires, in turn, required fewer number of plys in the tire. J. G. Berry, Field Engineer of U. S. Rubber Co. summarized the advantages and disadvantages of low inflation tires:[27]

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(27) Berry, J. G., The Future Use of Rubber in Open-Pit Mining, Coal Mine Modernization Year Book, 1946, P. 208

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a) It permits for greater mobility and consequently

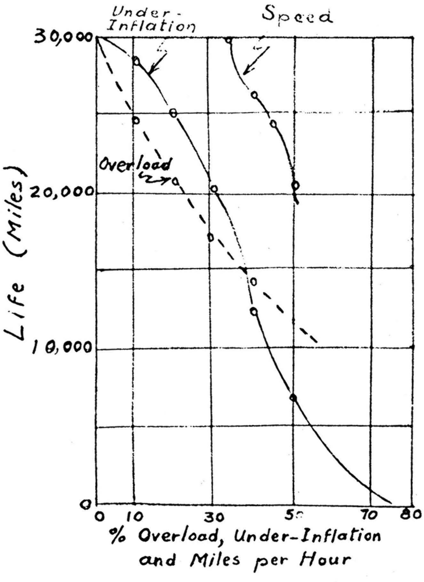

FIG. 19

How Certain Factors affect Tire Life.

("Coal Age". Oct. 1942)

reduces the demand for good roads.

b) It permits the operation of the tires over rock and coal beds with less fabric breaks.

c) Less pounding vibration, therefore less road and vehicle maintenance.

d). The cost the larger tire with fewer plys may approach or equal the cost of the smaller tire with more plys.

e) The only disadvantage of the low pressure tire is the higher power consumption of the trucks.

Longer tire life can be secured, if the following points are considered:

1) Maintain proper tire inflation, The proper pressure to be used on the type of tire and the gross weight of the truck.
2) Keep valve caps on the tubes.
3) Inspect tires regularly and remove all foreign materials within the treads.
4) Keep haulage roads in good condition.
5) Make regular check of front wheel alignment.
6) Replace new tires in pairs on dual wheels.
7) Do not use oil or grease on rims. Solution of soapy water is suggested as best lubricant.
8) Check air pressure regularly. Don't reinflate tires which show signs of air leakage.
9) Keep pressures in dual tires equal on both side of axle.

Effects of Pit Conditions On Maintenance

Preparation of the Bank:

The importance of good high-wall preparation lies in the need to prevent damage to the expensive shovels that operate in strip mines. The shovels will not dig solid limestone, hard shale, or tough sandstone. Explosives are used to reduce these handicaps. If proper bank preparation is not made, high shovel maintenance costs, loss of time, and curtailed output will result.

As in every other problem where the conditions are never duplicated, no hard and fast rules can be established for bank preparation. As with the medical profession, there are general principles that point the way but a diagnosis of the case must determine the kind and size of the dose.

"The preparation of a high-wall for stripping is not a science, it is an art. The blasting artist takes a look at the cross-section of the high wall the wheels of his brain buzz a few turns and he has an answer. The best may fall short. So he changes hole spacing, type of explosives or size of the shot and tries again."[28]

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(28) Richart, Fred W., "High-Wall Preparation," Coal Age February, 1944, p. 88

Whatever the explosives used, the drilling pattern and the weight of charge must be adequate to produce a bank that can be dug with the equipment at hand.

In general, at strip mines, holes are drilled horizontally with anger type drills. If a hard formation lies immediately over the coal bed vertical drills give the best results.

Gelatine dynamites and liquid oxygen explosives may be used in wet holes.

Group shooting of holes generally gives the best results. Blasting problems atyany mine must be solved on the job.

Maintenance of Roads:

In building haulage roads for high capacity trucks, the following conditions should be maintained for low-maintenance costs:

1) compacted, firm subgrade
2) deep drainage ditch on each side of the road, especially in flat areas
3) waterproof road surface
4) grade within the grade ability of truck.

The only way to keep the heavy traffic roads in the mine, in shape is to keep the subgrade and foundation dry. Road graders are widely used to keep road surfaces smooth. Undoubtedly they are the most valuable piece of equipment on the market for maintaining haulage roads. Bulldozers are essential, also, in road maintenance, especially for building a new stretch of road, and rebuilding a section that has begun to deteriorate.

The Drainage Problem:

This subject falls into two distinct classifications, namely: Surface drainage and Pit drainage. The prime object of surface drainage is to prevent surface water from entering the pit. This is accomplished by gravity flow ditches located around the stripping area. Once established, surface drainage ditches ordinarily require little attention or maintenance. In locations where the pit bottom is above natural drainage the problems of both surface and pit drainage is simplified and is less costly. Even in these pits, however, flow control on the high-wall side is necessary in order to divert heavy run offs.

Erratic weather conditions are a menace to stripping operations, therefore, in such areas, drainage is as important problem. The topography is the governing factor in planning surface drainage. Natural water ways on the property should be used until they interfere with mining operations,

A small diesel-driven dragline is usually used as a ditch digging machine. In general, its capacity may vary from less than a cubic yard up to three cubic yards. Other tools useful for drainage work are the road patrol and the bulldozer for making shallow contour ditches.

As one practical operator summed it up: "Failure to arrange proper drainage can be one of the most expensive items around a strip mine. All ditches and other drainage projects should be done during summer when sunshine and weather are favorable."(29)

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(29) Richart, Fred W., Handling Water to Save Time and Money at Strip Mines, Coal Age, January, 1944, p. 60

SUMMARY AND CONCLUSIONS

At Kumtepe of Agacle Region, near Istanbul, Turkey, there are an estimated 700,000 metric tons of lignite which may be mined by stripping.

It has been found that two shovels with capacities of 2 1/2 - cu. yds. and 3 - cu. yds. dipper capacity working one 8-hr. shift each can uncover the minimum required amount of lignite of 4,000 metric tons per day. The average stripping cost, when casting per cubic yard for these shovels is $0.057. The shovels are capable of delivering over 5,000 cu. yds. loose material daily.

During the major part of operation stripping material must be hauled by truck an average distance of 1,500 ft. For this operation, ten 22-ton Rear-dump Euclid trucks are found to be the most satisfactory. The average hauling cost per cubic yard of overburden is $0.090.

The mining of lignite can be best accomplished by a Marion type 111-M diesel shovel equipped with a light weight 4 - cu. yd. dipper. The estimated loading cost of the coal is $0.070 per ton while the cost of transporting the coal from the pit to the tipple is $0.155 per ton.

The use of bulldozers is suggested to facilitate the shovel operation and to maintain roadways.

A strip mine operator must recognize the importance of maintenance of equipment as well as roads and ditches, for a smooth and efficient mine operation.

Because of the speed at which strip mining operations progress, serious mistakes may easily be made. Consequently the detailed engineering must be sound, adequate, and timely.

In this study, the efficiency of management has not been included, because it is a factor which cannot be determined before a mine begins to operate. It is, therefore, necessary for management to obtain a complete cost analysis of various types of equipment along with the equipment limitations in order to pre-determine the most suitable type for each use. In the computation it was assumed that the operators of equipment were experienced. It may be necessary, however, for management to provide for the training of inexperienced personnel while on the job.

The mine operators should plan a series of time studies of all equipment on each shift. From these data the production standards can be established and management will have a gauge for measuring productivity. "This is particularly important in strip coal mining where production is dependent on a relatively small number of large producing machines."(30)

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(30) Utterback, H. Gene, Time Studies in Strip Mining, Mining Congress Journal, June, 1944, p. 43

To obtain the maximum production the efficiency of labor should be determined. The best method would be based on a certain number of units produced per man hour for a given job. The records of this method are always comparable because they are unaffected by working day or wage rates. Other aspects of industrial engineering might well be applied.

It has been shown that Kumtepe lignites can be mined at a profit of $2.01 per ton. To achieve this end, the full responsibility to develop this deposit must be given to a profit-minded, efficient management.

Appendix A

Data On Materials
(a)

Weight of Materials

Material	Pounds per Cubic Foot	Pounds per Cubic Yard
Clay, Dry Excavated	70	1890
Clay, Wet Excavated	110	2970
Coal, Broken, Anthracite	57	1539
Coal, Broken, Bituminous	52	1404
Earth, Excavated Common Loam, Dry	80	2160
Earth, Excavated Common Loam, Moist	90	2430
Earth, Excavated Common Loam, Wet	110	2970
Granite, Broken	96	2592
Gravel, Screen 1/4" to 2"	105	2835
Gravel and Sand, Pit Run	120	3240
Lignite, Broken	52	1404
Limestone, Broken	100	2700
Marl, Wet Excavated	140	3780
Peat, Moist	50	1350
Peat, Wet	70	1890
Phosphate Rock, Broken	110	2970
Sand, Slightly Damp	105	2835
Sand, Wet	120	3240
Sulphur	125	3375
Trap Rock, Broken	105	2835

Appendix A (Cont'd)

(b)

Swell Factors of Material

Material	Weight in Bank per Cubic Yard	Percent of Swell	Swell Factor
Sand or Gravel, Dry	3250	12%	.89
Sand or Gravel, Wet	3600	14%	.89
Sand or Gravel with Earth	3100	18%	.85
Loam	2700	20%	.83
Clay - Light	2800	30%	.77
Clay - Dense - Tough	3000	40%	.71
Earth	2800	25%	.80
Shale or Soft Rock-Blasted	3000	45%	.69
Hard Rock - Well Blasted	4000	50%	.67
Rubbery Clay, Hard Pan or Poorly Blasted Rock	--	80%	.56
Coal - Anthracite	2200	35%	.74
Coal - Bituminous	1900	35%	.74

(c)

Soil Conversion Factors

Soil Type	Present Condition	Converted to In Place	Loose	Compacted
Sand	In place	1.00	1.11	0.95
	Loose	.90	1.00	0.86
	Compacted	1.05	1.17	1.00
Common Earth	In place	1.00	1.25	0.90
	Loose	0.80	1.00	0.72
	Compacted	1.11	1.39	1.00
Clay	In place	1.00	1.43	0.90
	Loose	0.70	1.00	0.63
	Compacted	1.11	1.59	1.00

After War Department Technical Manual TM5-252, p. 46

Appendix B

Tables for Shovel Operations

(a)

The following multipliers (b) and (c) are based on certain assumptions and determined from many years of experience:

1) Under favorable working conditions.

2) 1 - yd. shovel is used and

3) Ordinary earth excavated from

4) 8 ft. - cut (depth of cut) and shovel loaded trucks from side so that

5) angle of swing of shovel was an average of 90 degrees

6) Under these conditions that shovel ordinarily should be able to produce 120 cu. yd. per hour, place measurement.

(b)

Output Coefficients for Shovels

Material	Multiplier
Hard shale and other rocky formations poorly blasted	0.40
Fairly well blasted rock or hardpan, and tough rubbery clay	0.50
Clay boulders	0.60
Heavy clay not sticky	0.70
Clay gravel	0.80
Wet, sandy clay	0.90
Ordinary earth	1.00
Light, dry loam or clay, loose sand and gravel, cinders, ashes	1.10
Light, moist clay and loam	1.25

Appendix B, (Cont'd)

(c)

Size of Dipper and Depth of Cut

Depth of Cut	3/8 .38	1/2 .50	5/8 .63	3/4 .75	1 1.00	1 1/4 1.25	1 1/2 1.50	1 3/4 1.70	2 2.00
0'-3"	.40	.39	.38	.38	.35	.33	--	--	--
0'-6"	.57	.56	.55	.55	.52	.50	.47	.44	.40
1'	.67	.66	.65	.65	.63	.62	.59	.56	.52
1'-6'	.76	.75	.74	.74	.72	.72	.69	.66	.62
2'	.83	.82	.81	.81	.80	.80	.77	.74	.70
3'	.89	.88	.87	.87	.85	.85	.82	.79	.75
4'	.94	.93	.92	.92	.90	.89	.86	.83	.80
5'	.97	.96	.95	.95	.93	.93	.88	.88	.85
6'	1.00	1.00	.98	.98	.97	.96	.94	.91	.88
7'	.97	.97	1.00	1.00	.98	.98	.96	.93	.90
8(	.94	.94	.97	.97	1.00	1.00	.98	.96	.93
9'	.91	.91	.94	.94	.97	.97	1.00	.98	.96
10'	.88	.88	.91	.91	.94	.94	.97	1.00	.98
11'	.85	.85	.88	.88	.91	.91	.94	.97	1.00
12'	.82	.82	.85	.85	.88	.88	.91	.94	.97
13'	.79	.79	.82	.82	.85	.85	.88	.91	.94
14'	.76	.76	.79	.79	.82	.82	.85	.88	.91
15'	.74	.74	.76	.76	.79	.79	.82	.85	.88
16'	.72	.72	.74	.74	.96	.76	.79	.82	.85
18'	.70	.70	.72	.72	.74	.74	.76	.79	.82
20'			.70	.70	.72	.72	.74	.76	.79
22'					.70	.70	.72	.74	.76
24'							.70	.72	.74
26'								.70	.72

Appendix B (Cont'd)

(d)

Type of Operation	Multiplier
Side casting	1.25
Loading trucks in rear, 80° swing from cut	0.80

Halcomb, A. E., Output Factors for Excavating and Material Handling Equipment, Koehring General Excavator Sales Manuel, pp 1-3.

Appendix B (Cont'd)

(e)

Dipper Efficiency Factor (k)
(For Shovels)

Condition	Factor	Remarks
Easy digging	95% to 100%	Loose, soft, free running materials, often provide heaped load Dry sand or small gravel, moist sand loose earth, muck, sandy clay, cinders, ashes, well blasted materials
Medium digging	85% to 90%	Harder materials that do not require blasting, but causing voids in dipper. Dry or wet clay, coarse gravel, packed earth
Medium-hard digging	70% to 80%	Materials requiring some breaking up by light blasting or shaking, hard to penetrate, causing voids in dipper. Well broken limestone, sandstone Blasted shale, heavy wet sticky clay Gravel with large boulders Cemented gravel
Hard digging	50% to 70%	Blasted rock, hard pan, and other materials which are difficult to penetrate and leave large voids in dipper Hard tough shale, limestone trap rock, granite, sandstone, conglomerate. Tough rubbery clay the shaves from bank

Appendix B (Cont'd

(f)

Cycle time (Cm) For Shovels

(90° Shovel Swing)

Capacity Cu. Yd.	Cycle time (Seconds) Easy digging	Medium digging	Hard digging
1/2	15	18	24
3/4	18	20	26
1	18	20	26
1 1/4	18	20	26
1 1/2	18	20	26
2	18	20	26
2 1/2	20	22	28
3	22	24	30
4	24	26	32

Note: For each increase of 10° in swing, add 2 seconds to cycle time; for each decrease of 10°, subtract 2 seconds from cycle time.

Modified after "War Department Technical Manual TM 5-252, pp. 92-93

Appendix (Cont'd)

(g)

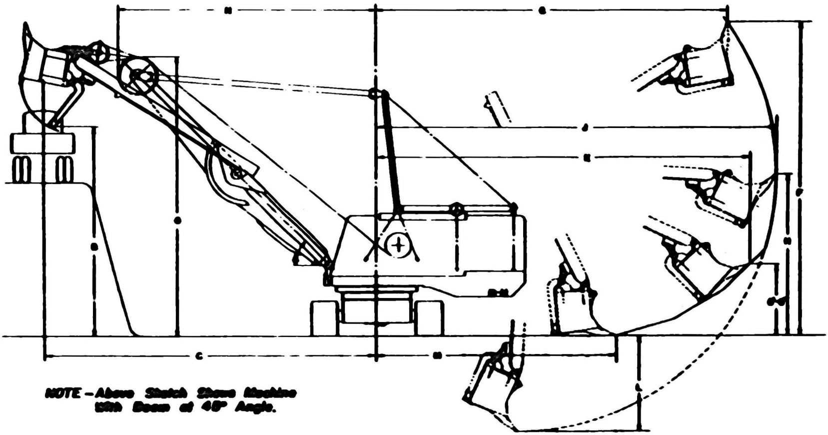
Approximate Loading Cycles of Shovels

(in seconds)

	Degrees of Swing	1/2	1	1 1/2	2	2 1/2
Easy digging	45°	12	14	15	17	18
	90°	16	18	19	21	22
	135°	19	21	23	25	27
	180°	22	25	27	30	32
Medium digging	45°	15	17	18	20	21
	90°	19	21	23	25	26
	135°	23	25	27	29	31
	180°	26	29	31	34	36
Hard digging	45°	19	21	22	24	25
	90°	24	26	28	30	31
	135°	29	31	33	35	37
	180°	33	36	38	41	43

MARION
Type 111-M
SHOVEL
(CLUTCH TYPE)

SPECIFICATIONS
723



NOTE – Above Sketch Shows Machine With Boom at 45° Angle.

BOOM-HANDLE-DIPPER

Boom Length – Effective	32'-0"
Boom Length – Foot Pin to Point Sheave Shaft	30'-0"
Point Sheave Diameter	[illegible]" P.D.
Boom Foot Pin to Shipper Shaft	14'-6"
Dipper Handle Length – Overall	29'-8"
Dipper Handle Length – Effective	22'-0"
Capacity – Heavy Duty Dipper	3-1/2 cu. yds.
Capacity – Light Weight Dipper	4 cu. yds.

WORKING RANGES **

A – Boom Angle	40°	45°	50°	55°	60°
B – Dumping Height – Maximum	20'-9"	23'-6"	25'-9"	[illegible]	30'-0"
C – Dumping Radius @ Maximum Height	38'-6"	37'-0"	35'-0"	33'-0"	30'-6"
* D – Dumping Height @ Maximum Radius	13'-6"	14'-6"	15'-3"	16'-0"	16'-9"
* E – Dumping Radius – Maximum	40'-0"	39'-3"	38'-3"	37'-3"	36'-3"
F – Cutting Height – Maximum	32'-0"	35'-3"	38'-0"	40'-6"	42'-9"
G – Cutting Radius @ Maximum Height	41'-0"	38'-9"	36'-3"	33'-9"	30'-9"
H – Cutting Height @ Maximum Radius	17'-3"	18'-3"	19'-3"	19'-9"	20'-6"
J – Cutting Radius – Maximum	45'-3"	44'-3"	43'-6"	42'-6"	41'-3"
K – Cutting Radius @ 8'-0" Elevation	41'-6"	41'-6"	41'-6"	40'-0"	38'-6"
L – Cutting Depth Below Grade	11'-9"	10'-9"	9'-9"	9'-0"	8'-3"
M – Radius of Clean-up	26'-9"	26'-6"	26'-0"	25'-3"	24'-6"
N – Clearance Radius – Boom Point	30'-6"	28'-9"	26'-9"	24'-8"	22'-6"
O – Clearance Height – Boom Point	29'-6"	31'-3"	33'-0"	34'-9"	36'-0"

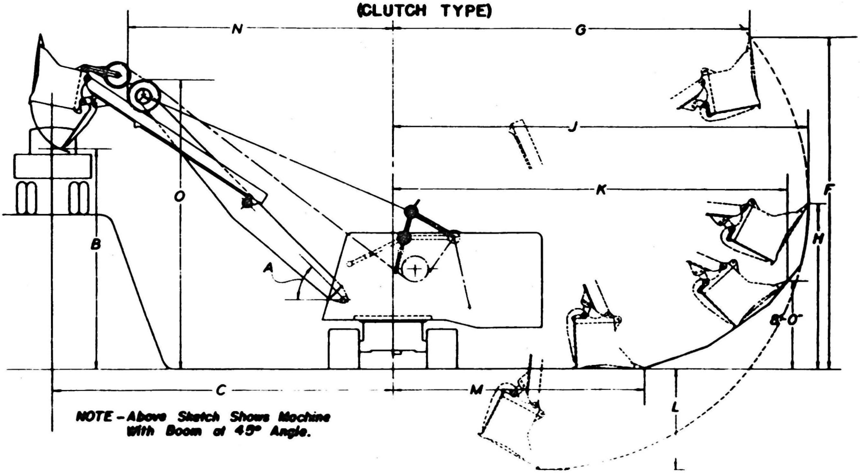
*Not Shown On Sketch

**All Ranges Shown With 3-1/2 Yd. Dipper

MARION Type 93-M SHOVEL

(CLUTCH TYPE)

SPECIFICATIONS 730



NOTE - Above Sketch Shows Machine With Boom at 45° Angle.

BOOM-HANDLE-DIPPER

- Boom Length - Effective .... 27'-4"
- Boom Length - Foot Pin to Point Sheave Shaft ...... 26'-0"
- Point Sheave Diameter .... 32" P.D.
- Boom Foot Pin to Shipper Shaft .... 12'-6"
- Dipper Handle Length - Overall .... 20'-0"
- Dipper Handle Length - Effective .... 18'-5"
- Dipper Capacity .... 2-1/2 cu. yds.

WORKING RANGES

A - Boom Angle	40°	45°	50°	55°	60°
B - Dumping Height - Maximum	17'-9"	20'-0"	22'-0"	23'-9"	25'-6"
C - Dumping Radius at Maximum Dumping Height	32'-6"	31'-0"	29'-6"	27'-6"	25'-6"
*D - Dumping Height at Maximum Dumping Radius	11'-6"	12'-3"	13'-0"	13'-9"	14'-3"
*E - Dumping Radius - Maximum	34'-0"	33'-0"	32'-6"	31'-6"	30'-6"
F - Cutting Height - Maximum	28'-0"	30'-6"	33'-0"	35'-0"	37'-0"
G - Cutting Radius at Maximum Height	34'-3"	32'-3"	30'-0"	27'-9"	25'-0"
H - Cutting Height at Maximum Radius	14'-3"	15'-0"	15'-9"	16'-9"	17'-3"
J - Cutting Radius - Maximum	38'-6"	37'-9"	37'-0"	36'-0"	35'-3"
K - Cutting Radius at 8'-0" Elevation	36'-0"	36'-3"	35'-6"	34'-6"	33'-3"
L - Cutting Depth Below Grade	9'-9"	9'-3"	8'-6"	8'-0"	7'-3"
M - Radius of Clean Up	23'-3"	23'-0"	22'-6"	22'-0"	21'-3"
N - Clearance Radius - Boom Point	25'-9"	24'-3"	22'-6"	20'-9"	18'-9"
O - Clearance Height - Boom Point	24'-9"	26'-3"	27'-9"	29'-3"	30'-3"

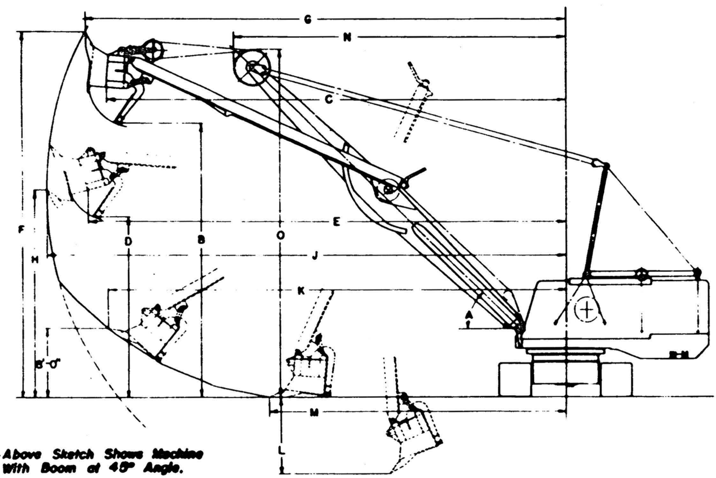
* Not Shown On Sketch.

MARION Type 111-M

SPECIFICATIONS 727

LONG RANGE SHOVEL

(CLUTCH TYPE)



NOTE - Above Sketch Shows Machine With Boom at 45° Angle.

BOOM-HANDLE-DIPPER

Boom Length - Effective	45'-0"
Boom Length - Foot Pin to Point Sheave Shaft	43'-0"
Point Sheave Diameter	48" P.D.
Boom Foot Pin to Shipper Shaft	22'-0"
Dipper Handle Length - Overall	34'-0"
Dipper Handle Length - Effective	33'-0"
Dipper Capacity	3 Cu. Yds.

WORKING RANGES

A - Boom Angle	45°	50°	55°
B - Dumping Height - Maximum	32'-0"	34'-9"	37'-0"
C - Dumping Radius @ Maximum Height	52'-3"	50'-3"	48'-0"
D - Dumping Height @ Maximum Radius	21'-3"	22'-6"	23'-9"
E - Dumping Radius - Maximum	54'-6"	52'-6"	51'-0"
F - Cutting Height - Maximum	42'-6"	45'-6"	48'-0"
G - Cutting Radius @ Maximum Height	54'-9"	52'-3"	49'-9"
H - Cutting Height @ Maximum Radius	24'-3"	25'-6"	26'-9"
J - Cutting Radius - Maximum	59'-3"	57'-9"	56'-0"
K - Cutting Radius @ 8'-0" Elevation	52'-9"	52'-6"	51'-6"
L - Cutting Depth Below Grade	15'-0"	13'-9"	12'-6"
M - Radius of Clean-up	34'-0"	33'-3"	32'-0"
N - Clearance Radius - Boom Point	37'-9"	34'-9"	31'-9"
O - Clearance Height - Boom Point	40'-6"	43'-3"	45'-6"

Appendix C
Data for Cost Analysis of Shovel

(a)

Elements of Cost Analysis

Fixed Costs:

A. Determine total investment considering the following:

1. Price of equipment, F.O.B. factory
2. Freight charges, unloading and erecting
3. Custon duties (if exported to a foreign country)

B. Determine Economic life of equipment

C. Determine average yearly investment

D. Find depreciation per year and per hour

E. Interest, taxes, insurance on average investment per year and per hour

F. Maintenance cost on total investment per year and per hour

G. Total fixed costs = D+E+F per year and per hour

Operating Costs:

H. Engine fuel and lubricating costs per hour

I. Direct Labor Costs per hour

L. Add overtime, workers fund per year and per hour

M. Total direct costs = G+H+I+L (if any)

Other Costs:

N. Indirect costs (overhead) per year or per hour

O. Total costs = M+N per year or per hour

Cost per cubic yard:

P. Output per hour

R. Cost per cu. yd. = O/P

Appendix C (Cont'd)

(b)

Average Useful Life of Shovels

3/8 - 3/4 cu. yds.	5 years	or	10,000 hours
1 - 1 1/2 cu. yds.	6 years	or	12,000 hours
2 cu. yds. and over	8 years	or	16,000 hours

(c)

Depreciation charges (Straight line method)

$$\frac{100\%\ \text{total investment}}{\text{Life in years or hours}} = \begin{array}{l} -\ \%\ \text{total investment per year} \\ -\ \%\ \text{total investment per hour} \end{array}$$

(d)

$$\text{Average Investment} = \%\ \text{of total investment} = 100\%\ \frac{(n+1)}{2n}$$

n = number years, life of equipment

(e)

"It has been suggested that a rate or charge for depreciation be established per hour for the first 2000 hours (normal usage) per year and that 1/2 this rate be added for each hours use beyond the first 2000 hours up to 4000 hours per year, and that 1/4 of the first rate or charge be used for the hours use beyond 4000 hours per year."

Operating Cost Guide, Power Crane and Shovel Association, N.Y. p. 6

Appendix (C) (Cont'd)

(e)

Repairs, Maintenance and Supplies

(Including labor associated with them)

Size of Shovel	Per Year	Per Hour
3/8 - 3/4	20%	0.01%
1 - 1/1/2	16.67%	0.0083%
2 - 2 1/2	12.5%	0.00625%

(f)

Estimating Fuel and Lubricating Costs

Consumption (Diesel Engines)

Size of Shovel	Fuel Consumption Per Hour		Lubrication Oil consumption Per Hour	
	US Gallons	Liters	US Gallons	Liters
1/2 cu. yd.	1.6 - 1.9	6.0 - 7.2	.07	.26
3/4	2.4 - 2.9	9.0 -11.0	.10	.38
1	3.1 - 3.8	11.7-14.3	.10	.38
1 1/4	3.7 - 4.5	14.0-17.0	.16	.61
1 1/2	4.6 - 5.5	17.4-20.8	.18	.68
2	5.8 - 7.0	22.0-25.5	.24	.91
2 1/2	7.0 - 8.5	26.4-32.1	.26	1.00

Appendix D

Data on Truck Haulage

(a)

Haulage Cycle = Loading time + Hauling time + Dumping time + Returning time + Spotting time

(b)

Rolling Resistance

Definition: The rolling is the resistance between the tires and level ground that must be overcome before the tires can roll. It is normally expressed in pounds per ton of gross vehicle weight or in percent of gross vehicle weight.

Type of Road Surface	lbs. /ton of Gross Vehicle	Per cent
1. Smooth, hard, dry dirt and gravel Well maintained. Free of loose material	40 lbs.	2%
2. Dry dirt and gravel, not firmly packed. Some loose material	60 lbs.	3%
3. Soft unplowed dirt or poorly maintained dry dirt, rutted surface	80 lbs.	4%
4. Wet, muddy surface on firm base	80 lbs.	4%
5. Soft, plowed dirt or unpacked dirt fills	160 lbs.	8%
6. Loose sand and gravel	200 lbs.	10%
7. Deeply rutted, sticky or muddy, soft spongy base	320 lbs.	16%

(c)

Appendix D (cont'd)

(c)

Grade Resistance: If the road has 6% grade, grade resistance equal to 6% of gross vehicle weight. If grade of road is 8% with 4% rolling resistance, total resistance to be overcome by the vehicle is equal to (8% + 4%) = 12% of Gross Weight Vehicle.

(d)

Average Speeds Factors

Length of Haul Road Section	Unit Starting from Shop	Unit Entering Haul Road Section after Accelerating
500 ft.	.50	.70
1000 ft.	.60	.80
2000 ft.	.70	.80
3000 ft.	.75	.80
4000 ft. and up	.80 - .85	.80 - .85

Average speed = Maximum speed X Factor

(e)

Total Turning and Dumping Time Per Hauling Cycle

Operating Condition	Euclid Bottom Dump tractor Semi-Trailer	Euclid Rear-Dump	Euclid Side-Dump tractor Semi-trailer
Favorable	.7 min.	1.5 min.	1.0 min.
Average	1.0 min.	2.0 min.	1.5 min.
Unfavorable	2.0 min.	2.5 min.	2.0 min.

Spot at Loading Machine

Operating Condition	Euclid Bottom Dump tractor Semi-trailer	Euclid Rear-Dump	Euclid Side-Dump tractor Semi-trailer
Favorable	.15	.15	.15
Average	.50	.30	.50
Unfavorable	1.00	.50	1.00

Appendix D (Cont'd)

(f)

Travel time in Minutes

Speed MPH #	Feet 100	200	300	400	500	600	700	800	900	1000
3	.379	.757	1.136	1.515	1.893	2.272	2.65	3.03	3.41	3.79
4	.284	.568	.853	1.136	1.42	1.70	2.00	2.27	2.55	2.84
5	.227	.454	.681	.908	1.136	1.363	1.59	1.82	2.04	2.27
6	.189	.378	.568	.757	.946	1.136	1.325	1.51	1.70	1.89
7	.162	.324	.487	.649	.811	.974	1.136	1.30	1.46	1.62
8	.142	.284	.426	.568	.710	.852	.994	1.136	1.28	1.42
9	.126	.252	.378	.505	.631	.757	.883	1.00	1.136	1.26
10	.113	.227	.341	.454	.568	.681	.795	.909	1.02	1.136
12.5	.091	.182	.273	.363	.454	.545	.636	.727	.818	.909
15.0	.075	.152	.227	.303	.378	.454	.530	.605	.681	.757
17.5	.065	.129	.194	.259	.324	.389	.454	.519	.584	.649
20	.057	.113	.170	.227	.284	.341	.397	.454	.511	.568
22.5	.050	.101	.151	.202	.253	.303	.353	.404	.454	.505
25.0	.045	.090	.136	.181	.227	.272	.317	.363	.408	.454
27.5	.041	.082	.124	.165	.206	.248	.289	.330	.371	.412
30.0	.038	.076	.113	.151	.189	.227	.265	.303	.341	.379
32.5	.035	.070	.104	.139	.174	.209	.244	.279	.314	.349
35.0	.032	.065	.097	.129	.162	.184	.227	.259	.291	.324

Estimated Production and Costs of Material Movement with Euclids, The Euclid Road Machinery Co., Cleveland 17, Ohio, 1946, p. 6.

Appendix D (Cont'd)

(h)

Performance Charts of Typical Euclid Trucks

Model 84 FD Rear-Dump Euclid 15-ton

Gear Selection	Travel Speeds 1800 RPM	Grade Ability Empty %	Grade Ability Loaded %
Low Range			
1 st	2.8	35	30
2 nd	4.5	35	18
3 rd	7.7	20	9
4 th	13.5	10	4
5 th	21.3	6	2
Reverse	3.5	35	23
High Range			
1 st	3.6	35	23
2 nd	5.9	27	13
3 rd	10.2	15	7
4 th	17.8	7	3
5 th	28.0	4	1
Reverse	4.6	35	17

Capacity: 9.7 cu. yds. Struck Measure; 11.4 cu. yds. Heaped

Net Weight 35,600 lbs.

Gross Weight 65,600 lbs.

Appendix D (Cont'd)

(h) (Cont'd)

Model 31 TD Rear-Dump Euclid, 22-ton

Gear Selection	Speed at 2100 RPM	Grade Ability Empty %	Grade Ability Loaded %
Low:			
1 st	3.1	35.0	32.5
2 nd	5.2	35.0	18.7
3 rd	9.0	23.3	10.1
4 th	15.6	12.4	4.9
5 the	24.6	7.1	2.3
Reverse	4.1	35.0	24.5
High:			
1 st	4.1	35.0	24.5
2 nd	6.9	31.0	13.7
3 rd	11.7	17.3	7.2
4 th	20.6	9.0	3.2
5 th	32.4	5.0	1.3
Reverse	5.4	35.0	18.0

Capacity: 14.8 cu. yds. struck and 16.2 cu. yds. heaped

Net Weight: 41,300 lbs.

Gross Weight: 85,300 lbs.

Appendix E

Conversion Factors

Length Measure

English to Metric	Metric to English
1 Mile = 1.609 Kilometer	1 Kilometer = 0.6214 miles
1 Yard = 0.9144 Meter	1 meter = 39.37 inches
1 Foot = 0.3048 Meter	= 3.2808 feet
1 Inch = 2.54 Centimeter	= 1.0936 yards

Square Measure

1 Sq. Mile = 2.5899 Sq. Kilometers	1 sq. kilometer = 0.3861 sq. miles
1 Acre = 0.4047 Hectare	= 247.1 acres
1 Sq. Yd. = 0.836 Sq. Meters	1 Hectare = 2.471 acres
1 Sq. Foot = 0.0929 Sq. Meters	1 Are = 0.0247 Acre
	1 Sq. meter = 10.764 sq. ft.
	= 1.196 sq. yd.

Cubic Measure

1 cu. yd. = 0.7645 cu. meters	1 cu. meter = 35.314 cu. ft.
1 cu. yd. = 0.02832 cu. meters	= 1.308 cu. yd.
= 28.317 Liters	1 liter = 0.0353 cu. ft.
1 U.S. gallon = 3.785 Liters	1 liter = 0.2642 U.S. gallon
1 U.S. quart = 0.946 Liters	= 1.196 sq. yd.

Measure of Weight

1 long ton = 1.0161 metric ton	1 metric ton = 2204.6 lbs.
1 short ton = 0.9072 metric ton	1 kilogram = 2.204 lbs.
1 pound = .452 kilogram	

Money Measure

American to Turkish	Turkish to American
1 dollar = 2.83 Turkish lira	1 Turkish lira = $0.46
1 cent = 0.0283 kurus	1 Kurus = .46 cent

Heat

1 B.T.U. = .252 Calorie	1 Calorie = 3.968 B.T.U.

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