A MODERN MINING OPERATION

FOR

THE VIRGINIA SEMIANTHRACITE COAL FIELD

BY

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I. THE INTRODUCTION

Mining the coal of the Virginia Semianthracite field, or the Valley Coal Fields of Virginia as it is commonly referred to, has always been considered a difficult operation. The few operations of any extent that have attempted the mining have changed owners many times. The physical difficulties surrounding the mining of the coal are many. First, the coal lies in pitching synclines making deep mining a necessity in most cases; second, the coal is a high ash fuel analyzing 6%-10% higher than Pennsylvania anthracite and 10%-15% higher than bituminous; third, the coal seam is broken up by numberous bands and partings which vary in occurrence throughout the field and which make the coal washing problem a difficult one. These partings and bands are so extensive that it has been necessary to discard approximately one-third of the seam as impurities; fourth, rail transportation is available to a limited area only; and fifth, explosive gas is present in sufficient quantities to make mining hazardous unless properly controlled.

The apparent physical difficulties and failure of past operations have discouraged the investment of capital for any large scale operation which could make use, in as far as possible, of the newer techniques and newer mining equipment developed in recent years.

The problem then is to select a locality and plan an operation using the newer techniques and equipment where possible and to determine if a better product can be placed on the market at a better financial advantage to a mining company. It is realized that a solution for one locality would not necessarily be the correct one for all areas where the coal outcrops. However, a successful solution of the problems encountered in one area should provide a guide for the solution of similar difficulties encountered in mining operations in other areas.

II. LITERATURE REVIEW

Practically no written material is available dealing with an investigation into the possibilities of a fully mechanized mine in the Valley Coal Fields of Virginia. Maurius R. Campbell and other state geologists published in 1925 a bulletin[9] dealing primarily with the geology and the origin of the Valley Coal Fields. This bulletin traces the occurrence of the coal measures throughout the state and lists coal analyses made from samples taken from the various coal prospects. It was not intended to be an engineering study of the problems involved in mining the coal.

A search was made through the trade literature to find recent mining operations in other areas where similar mining conditions were encountered. At the Royal Gorge Mine[15] where the coal seam dipped 48°, mining was carried on by driving a slope down the dip of the seam and lateral development was obtained by driving drift entries parallel to the strike on 60-ft. centers to the property boundaries. The strike entries were connected by cross-cuts every 80 ft. The working face was the 50-ft. face made by the crosscuts. The working face was retreated from the property boundaries back to the slope. This operation was mechanized to the extent that Sullivan 5-B cutting machines were used to undercut the 50-ft. working face. The cutting machine cut the face up the 48° pitch from one drift entry to the one above, the machine being lowered back to the lower entry after the completion of the cut and before the undercut face was shot.

At the Dines Mine of the Colony Coal Company[13] where the coal dipped only 13°, shaker conveyors, duckbills, and cutting machines were used to great advantage. Double entries were turned off the slope parallel to the strike of the seam and driven to the property boundaries with shaking conveyors and duckbill loading heads. Double track followed the progress of the duckbills up the haulage entry. The shaking conveyor in the aircourse discharged into mine cars on the haulage entry through crosscuts. Driving heads of the conveyors were moved after 350 ft. of advance. Rooms were driven in pairs 25 ft. wide on 50-ft. centers and were turned up the dip on the aircourse side. The pair of rooms were driven up 375 ft. by duckbills, and the pillars were extracted between the rooms by the duckbills using a double swivel. Both conveyor-duckbill units discharged together at the same point by using an angle trough. The mine produced 1200-1500 tons daily. The units on development work in the aircourse and entry were able to average six cuts per shift. Cutting machines remained with each duckbill unit.

In mining the steeply pitching seams in the State of Washington[4] hand loading on shaking conveyors was practiced on dips up to 56°. Development entries were turned parallel to the strike off the main slope, the aircourse being 12 ft. wide and the haulageway 24 ft. The conveyors in both the entry and aircourse were placed on the down pitch side to take advantage of gravity when the faces were shot. The aircourse conveyor discharged through a crosscut at the same point as the haulage entry. The entry and aircourse unit had its own cutting machine. A unique feature in the haulage entry was the use of a short bucket conveyor to elevate the coal from the conveyor up to the mine cars. For production a panel system above the aircourse was laid out by driving a pair of rooms every 300 ft. from one level to the one above. From these rooms other rooms were driven parallel to the strike and were worked by conveyors similar to the entry work. All strike room conveyors discharged on a common coal chute in the dip rooms where the coal slid to the mine cars on the haulage entry. Supplies were hauled to the panel by a small hoist in each of the dip rooms.

III. PREVIOUS MINING OPERATIONS

The history of mining in the Valley Coal Fields of Virginia is connected with some interesting historical events of the state and the country. The following information was taken from a letter to Professor O. C. Burkhart, Virginia Polytechnic Institute, from Buy F. Ellet, attorney at law, Christiansburg, Virginia. The letter was dated January 16, 1931.

A large number of Hessian soldiers were captured by George Washington at the Battle of Trenton during the Revolutionary War and were sent to Charlottesville, Virginia, to prevent their recapture. About the same time the British general, Tarleton, was raiding the Carolinas and was heading northward. The captured Hessian soldiers were sent to the mountains of Montgomery County to prevent their recapture. It seems that some of the Hessian soldiers were iron workers and had had some experience with coal in Germany. They discovered the present Merrimac seam and used some of the coal for blacksmithing and some for smelting iron. There was no extensive mining done in Montgomery County prior to the Civil War. About 1855 a Christiansburg man named Jerry Kyle, who owned the land surrounding the present Merrimac property, conceived the idea of developing his property by building a tram railway from the Christiansburg station, of what was then the Virginia-Tennessee Railway, to the property. Mr. Kyle got the backing of some Pennsylvania capital with the provision that he sink the slopes where they designated. (All three slopes were in the vicinity of the Merrimac Breaker.) Mr. Kyle sent to Wales for three miners to supervise the sinking of the slopes; but by the time the deal was completed, the Civil War had started and the project was dropped. During the progress of the war it became impossible to work the coal of the Midlothian Basin near Richmond because of frequent raids by northern troops. The War Department detailed Mr. I. H. Adams of Lynchburg to come to Montgomery County to mine enough coal to operate the newly raised ship, the Merrimac, which had been made an iron-clad. Mr. Adams mined the coal, with the assistance of the Welsh miners, and hauled it to Christiansburg by mule team to be sent by rail to Norfolk and other points of the Confederacy. In 1864 the Federal General Averill made his raid through the country and completely destroyed the mining operation of Mr. Adams. Except for a little mining for local use, no extensive mining was done in the county until 1902, when an extensive strike in the anthracite fields took place. Col. Payne of Richmond became interested in the coal situation and obtained the lease on the Merrimac property and bought up a large acreage on Brush Mountain from Kanodes Mill on Toms Creek westward to New River. Col. Payne built the railroad from Christiansburg to Merrimac and to Blacksburg, built the cleaning plant, and operated the mines. In 1907 the depression forced the operation into the hands of the receivers, and a flash flood filled the mine with water.

According to Mr. Merrill of Blacksburg, in 1914 the mine was pumped out by the Likens Hill Coal Company of Buffalo and operations were resumed. In 1934 this company gave up the lease, and the property reverted to the owners, the Brush Mountain Coal Company.

The Parrot Mine operated by the Pulaski Anthracite Coal Company was started in 1905 by Wolbridge and Parrot. It seems to have been one of the most successful in the area. It depended on the middle western states for its market. Approximately $600,000 was invested in surface equipment including a briquetting plant for the culm. The mine apparently was

was in continuous operation until 1933. (This information was also obtained from Mr. Merrill.)

The Great Valley Coal Corporation was organized in 1926 by a group of Baltimore and Chicago businessmen with 15,000 shares of common no par stock and $300,000 worth of bonds at 7%. They planned the expenditure of $125,000 in developing the initial unit. Mr. Merrill of Blacksburg was of the opinion that the sum actually spent was in the neighborhood of $700,000. This corporation was bought by the Great Valley Morgan Coal Corporation in 1933. They contracted with the Raleigh Smokeless Coal Company of Beckley, West Virginia, for disposal of the product.

Great Valley Morgan Coal Corporation did not make a success of the venture and the property went into the hands of the receivers in 1935. In 1936 Raleigh Smokeless bid in the property at auction, hoping to recoup some of the losses incurred while working under the agreement with Great Valley Morgan. In 1938 the property was again auctioned off and purchased by R. G. Stevens of Radford, Virginia, who has operated the mine ever since as Great Valley Coal Company. (Mr. Irving McCoy of McCoy related the information concerning the Great Valley Morgan Coal Corporation.)

The Big Vein Mine of Superior Anthracite Coal Company was started in 1919 by C. E. Smith of Pulaski. To capitalize the venture, Smith sold stock, the largest investor being Col. E. Jones of Washington, D. C. After the surface equipment was erected, Smith's interest was purchased in 1922, and the mine was operated as Big Vein Anthracite Collieries. To finance the venture Jones had borrowed money from R. N. Harper of Washington, D. C.; and Jones failing to meet his obligations, Harper took over the operation in 1927 and placed a Mr. Zimmerman in charge. Zimmerman bought the property from Harper in 1931 but was forced to turn it back a year later. In 1934 Harper had the John McCall Coal Company start operating the mine as the Superior Anthracite Company. In 1945 the John McCall Coal Company bought the property from Harper. (The history of this operation was related by Mr. DeHart, superintendent of Big Vein Mine.)

It would be very difficult to find the reason, or reasons, for so many companies failing to make a success of an operation in this area because it is impossible to obtain exact figures and cost statements. Apparently, general business conditions (1928-1934) forced the Merrimac operation to cease on two occasions. Mr. Merrill states that the mine was so deep and the drainage problem so difficult that the Likens Hill Company was forced to stop.

In the case of the Parrot Mine, which enjoyed continuous operation until 1933, termination was attributed by the older mining men of the area to poor management after the elder Parrot died. Some advance the reason that higher wages brought on by unionization caused the cost of mining to rise disproportionately and thus forced the Parrot operation out of business.

The first Great Valley Company collapsed due to the business conditions shortly after its formation. The Raleigh Smokeless Company was apparently more interested in its operations elsewhere and afforded little supervision to Great Valley.

The succession of operators at Big Vein seemed to be the result of undercapitalization in most cases. Not enough capital was available to develop the mine and market to a stage where profits could be made before the operator was forced to stop.

All the mines in the area followed the same pattern in their method of mining. On the steeper pitches rooms were turned off the strike entries up the dip and the coal was mined "off the solid" and allowed to slide down the rooms. It was retained at the mouth of the rooms by batteries where it was loaded into mine cars to be hoisted to the surface. On the shallower pitches conveyors, shaking or chain-flight, were used to convey the coal from the room working faces to the cars on the entry. In most cases four to six men were employed in each room drilling, shooting, and hand loading the coal onto the conveyors. The men at the face began preparation by discarding the impurities that could be visibly recognized.

IV. GEOLOGY OF VIRGINIA SEMIANTHRACITE FIELD

A. Origin and Occurrence

It is generally agreed that coal was formed from an accumulation of vegetal matter in troughs or depressions in the topography of the land during the long past geological period known as the Carboniferous.

In the eastern part of North America the deposition had been primarily marine previous to the Carboniferous Period. During this period the land over most of this part of the continent began to emerge from the sea. With the emergence of this great area there was a difference in rise of the area bordering on the Atlantic and lying between the Atlantic — as it then existed — and a long northeast-southwest sound on its westward side, known as the Appalachian trough. This trough acted as a catch basin for the sediments that were to form the coal measures of the Carboniferous. This flat area, the Appalachian trough, was subjected to slight warping. This warping caused many depressions into which vegetal matter slowly accumulated which were to be the coal beds of the Carboniferous.

B. Geological Structure

The geological columnar section of this area northward from Blacksburg to Gap Mountain and west to New River (Fig. 1) shows the order of deposition of the sediments prior to and during the Carboniferous Period. The columnar section shows that the Valley Coal Fields of Virginia were formed during the Mississippean while the bituminous fields of southwestern Virginia and West Virginia were formed during the Pennsylvanian and Permian.

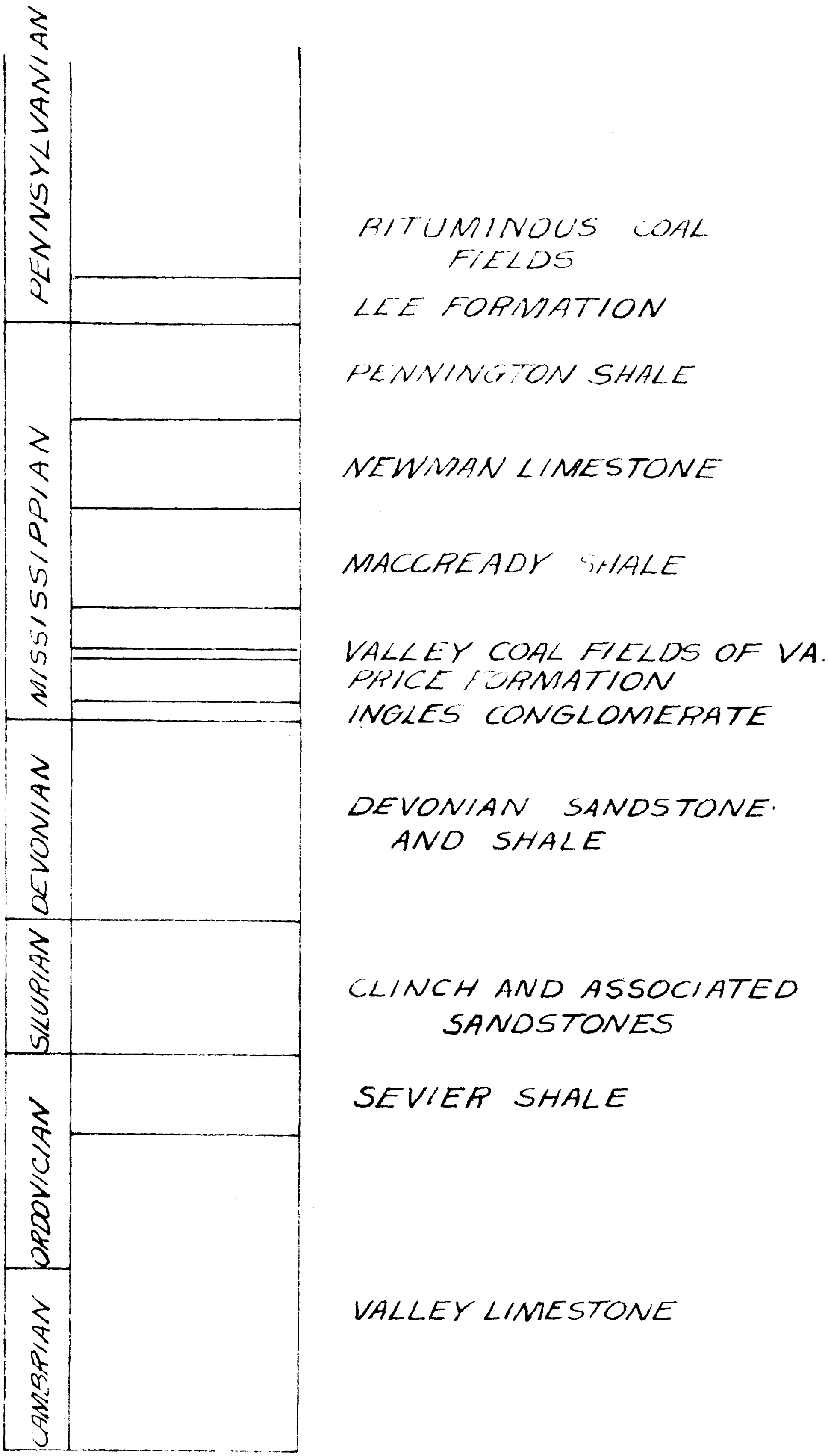


Figure I. Geological Columnar Section Showing Order of Deposition

Comparing the columnar section (Fig. 1) with a cross-section of the strata as it exists today (Fig. 2) indicates that radical changes have taken place since original deposition.

It is not the purpose of this paper to go into a detailed explanation or discussion of the geological events necessary to produce such a change in the order of strata as shown by Figure 1 and Figure 2. Generally, the nearly inverted order of strata resulted from faulting and folding produced by a lateral pressure originating from the southeast. The area containing the Valley Coal Fields is characterized by many faults and folds. The major faults are the overthrust variety where the older beds are thrust over the younger ones. The folded structure takes the form of synclines and anticlines sometimes resulting in the "shingle like" structure shown in Figure.2.

C. Geological Factors Affecting Mining

What effect did the faulting and folding have on the coal beds? In the first place, the great pressure necessary for such changes hastened the transformation of the accumulated vegetal matter into coal by driving off more and more of the volatile matter and leaving a higher percentage of fixed carbon. One would expect then to find coals of highest rank, that is highest ratio of fixed carbon to volatile matter, on the southeast edge of the fields. The Valley Fields are the highest ranking coals in the scale of coal transformation in Virginia. In Pennsylvania the highest ranking coals are eastermost. The rank gradation is not uniform westward since faults have occurred to relieve the pressure. In the second place, the

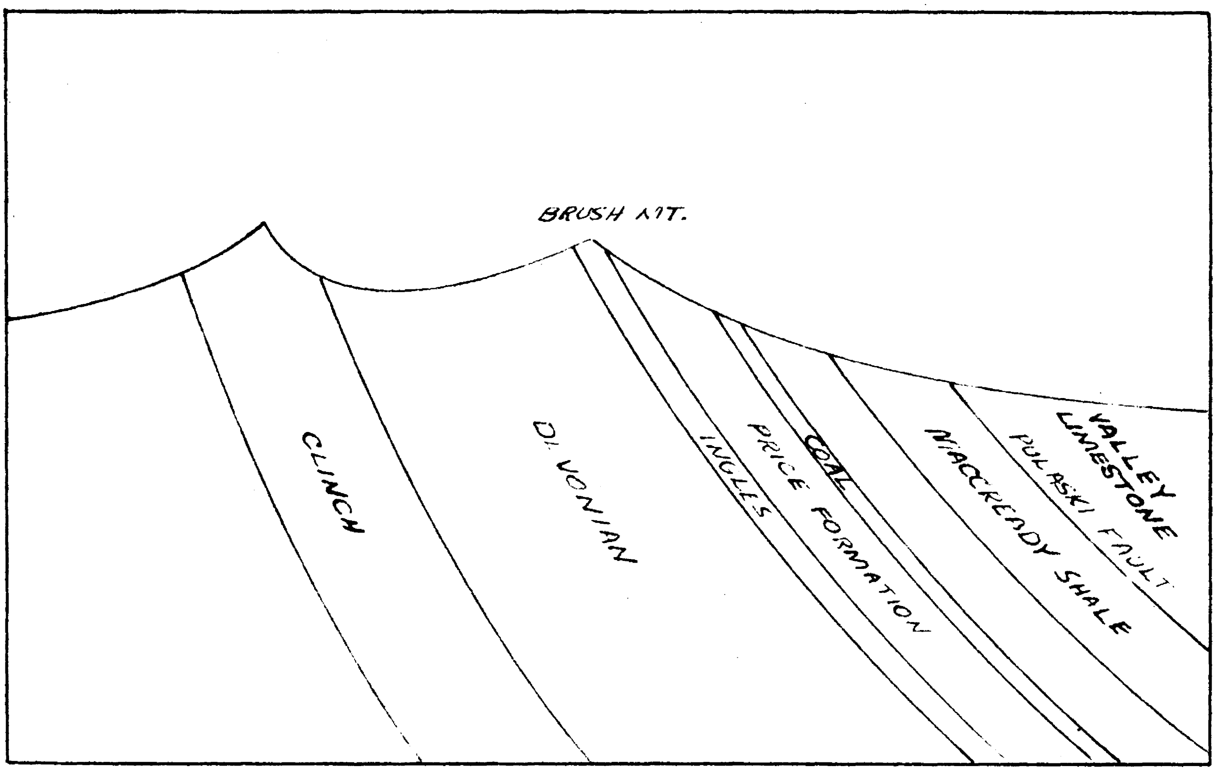


Figure 2. Cross Section of Strata on New River near Parrot

faulting and folding of the coal measures limits mining operations by:

1. Cutting off the coal seam in certain localities.
2. Occurrence of coal beds in steeply pitching synclines with dips of 20^{o}-36^{o}, hence necessitating very deep mining.
3. Displacement of the coal seam a few feet due to normal faulting.

The two most important coal seams in the Price formation are the Merrimac, or "big seam", and the Langhorne seam, or "little seam". The Merrimac seam, although varying in thickness throughout the area, is consistently the thickest and the one most mined. The Langhorne seam underlies the Merrimac seam by a distance of 20-70 ft. and is not thick enough in most places to be commercially important.

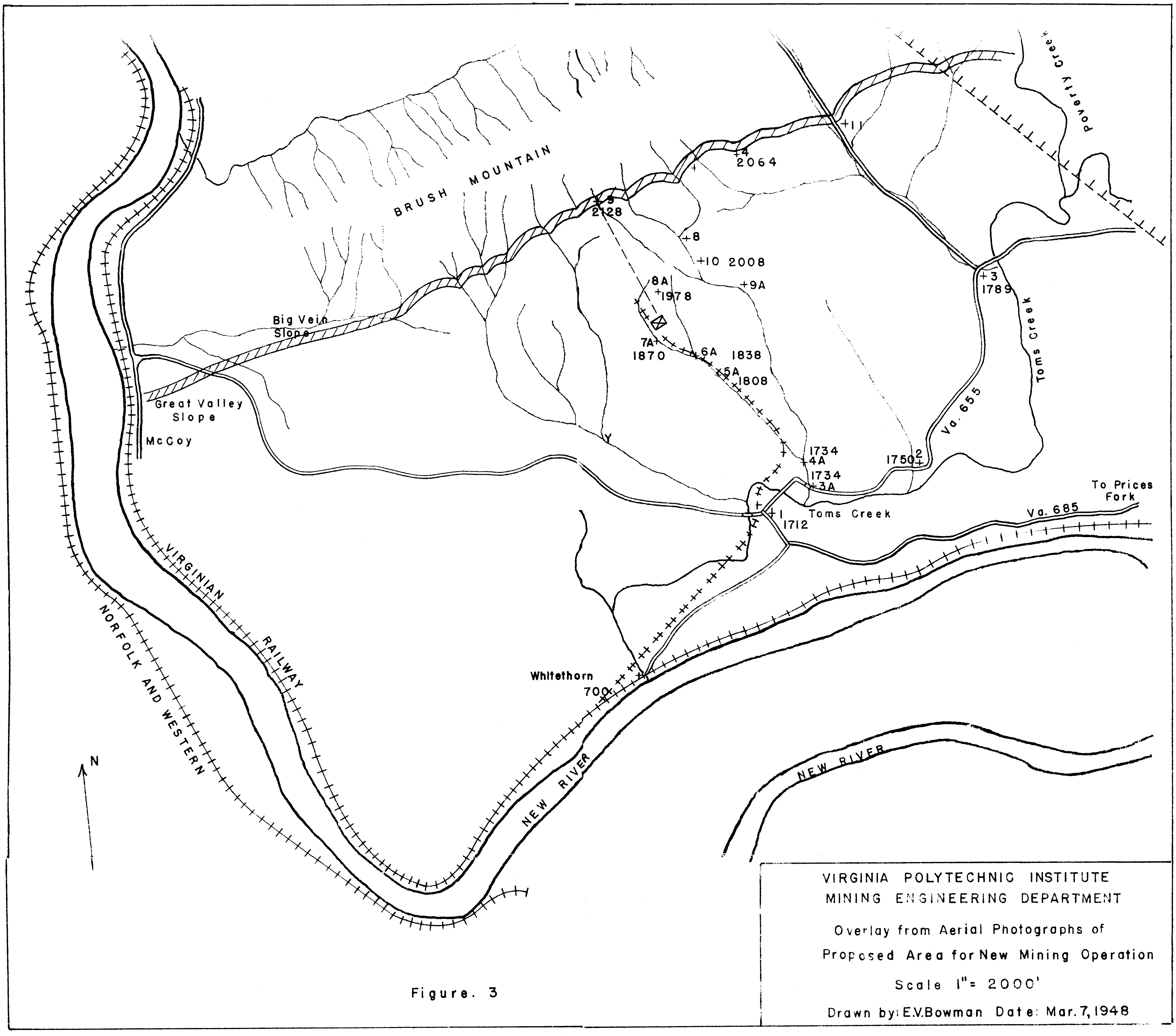
The Valley Coal Fields are located in the counties of Augusta, Bland, Botetourt, Montgomery, Pulaski, Roanoke, Smyth, and Wythe. The fields of Montgomery and Pulaski are most important from the standpoint of thickness of seam and nearness to railroads. In Montgomery County there are two fields, the Brush Mountain field and the Price Mountain field, denoting outcrops of the seams at these two locations. The particular site chosen for this study is in the Brush Mountain field near the location of the two major operating mines in the field — Great Valley Anthracite and Superior Anthracite.

V. LOCATIONS OF SITE FOR POSSIBLE OPERATIONS

The area chosen for a possible operation is the Brush Mountain Field of Montgomery County. It lies east of New River (Fig. 3) and east of the properties of Great Valley Anthracite and Superior Anthracite Coal companies and extends eastward to Poverty Gap in Brush Mountain. This area lies northward of the great bend in New River where part of the Radford Ordnance Works is presently located.

Both the Norfolk and Western and Virginian Railways have roads near the area. The Virginian is on the east bank of New River. Both railroads have taken advantage of the gap in Brush Mountain made by the New River for their roads northward. It is in this cut, where New River crossed the coal beds, that the major companies operating in the Brush Mountain field have located their operations. It made an ideal location being next to the main lines of two railroads and near a river where an ample supply of water was available for coal washing. The Parrot Mine was located on the west bank of New River and was served by the Norfolk and Western; Great Valley on the east bank is served by the Virginian. The Superior Anthracite Coal Corporation is located east of the property of Great Valley and is not on the Virginian, but the coal is transported to the railroad by a surface mine haulage system of approximately one mile in length.

Lying between the east boundary of the Superior property and Poverty Gap is a body of coal three miles long which is untouched except for the few wagon mines that rarely extend down the dip from the outcrop more than 250 ft. If mined to a projected depth of only 4000 ft. down the dip, this tract would furnish a total of 20,000,000 tons of coal before cleaning.



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Overlay from Aerial Photographs of
Proposed Area for New Mining Operation

Scale 1" = 2000'

Drawn by: E.V. Bowman Date: Mar. 7, 1948

Figure. 3

In order to warrant the investment of an appreciable amount of capital in an operation in this area three items must be made available: first, a source of electric power for machinery; second, an ample supply of water, necessary to a varying extent in the majority of coal cleaning processes; and third, access to rail transportation.

The power requirement can easily be met. Referring to the map of the area (Fig. 3), two sources of electrical energy are available. Appalachian Electric Power Company lines cross the northeast corner of the area, and power is available from the same source at both Great Valley and Superior Coal Companies properties.

In seeking access to the property for rail transportation, it is natural to study the drainage pattern of the area. Several small northern tributaries of Toms Creek cut across the area. Overlapping strip aerial photographs of the area and a stereoscope were used in studying the relief of the area. Elevations of critical points were obtained by the Paulin System Altimeter. The Virginian Railway, following New River, is closest to the area at Whitethorn where Toms Creek joins New River. In investigating the gaps cut by the northern tributaries of Toms Creek (See map Fig. 3) the most western tributary was discarded as a possible rail entrance because of the ridge lying between it and Whitethorn making a right of way difficult, if not impossible. Also, Brush Mountain becomes much steeper where the tributary splits at "Y" on the map. The next tributary eastward, joining Toms Creek at Location 3A on the map, was ideal since it divided the area approximately in halves. The valley it makes is 100-150 ft. wide, and the gradient is uniform up to and above location 7A on the map. At 800 ft.

above location 7A Brush Mountain again begins to rise steeply. This point marks the extent of a possible rail contact with the coal outcrop. The same tributary splits at location 4A, the right or eastward fork being very narrow and twisting throughout its length.

Location 7A would be the closest connection to the coal outcrop if a spur line railroad were built from Whitethorn. To make this connection by a right of way up the valley beginning at 3A would require a spur of 2.6 miles long with an average grade from Whitethorn of 1.3% and maximum grade of approximately 3.2%. The projected spur line is shown by a sketch on the map (Fig. 3).

This spur line would bring rail transportation within 3500 ft. of the coal outcrop in the center of the property. The preparation plant could be located above 7A as shown on map, and the remaining distance to the coal outcrop could be covered by a belt conveyor. This arrangement is possible since the necessary tipple height is available. Location 8A (elevation 1978 ft.) is on the point of a ridge overlooking the valley at 7A. The conveyor line would follow the ridge (as shown on Fig. 3) to the coal outcrop and never exceed the limiting slope angle of 18° to the horizontal. Refuse from the washer could be dumped in the next valley eastward from the preparation plant.

The most reliable source of water the year around in Toms Creek, as its northern tributaries become nearly dry during part of the year. Pumping the water from Toms Creek would not be a difficult problem since the vertical head from the tipple site to the creek is less than 200 ft.

VI. THE DEPOSIT

A. Analysis

A 250 lb. channel sample was taken from the Merrimac seam from the Herman Price Coal Mine from the 1st entry (west) off the main slope 100 ft. from the slope. The sample was taken in accordance with ASTM Standards No. D 21-40.

The sample was reduced in size and returned to the laboratory for analysis. The procedure outlined in ASTM Standards No. 271-44 was followed in making a proximate analysis of the seam.[5] The following results were found as a result of the analysis:

	As Received Basis	Dry Basis
Moisture	2.75	
Ash	32.30	33.21
Volatile Matter	12.20	12.55
Fixed Carbon	52.75	54.24
Sulphur	.55	
B. T. U.	10,500.00	

Figure 4

B. Washability

A sample of approximately one ton of coal of the Merrimac seam was procured from the Herman Price Coal Mine. This sample was obtained by taking a one ft. deep section of the entire face. The sample was trucked to the laboratory and screened into the following fractions: plus 3", 3 x 1 1/2, 1 1/2 x 3/4, 3/4 x 3/8, 3/8 x 3/16, 3/16 x 14 mesh, and minus 14 mesh. Each fraction was weighed and results tabulated in figure 5. The plus 3" was crushed by Denver Jaw crusher set at 3". The crushed product was screened over the same screen sizes as original screening and the size fractions were weighed, weights tabulated in figure 5, and the fractions added to original sample. Each size fraction was split into two samples — one for washability, and the other for head ash analysis. Each size, except the minus 14 mesh fraction, was placed in vats containing solutions of the following specific gravities: 1.35, 1.435, 1.55, 1.62, 1.70, and 1.90. The float from each specific gravity solution from each size was collected, dried, and weighed. The sink in each solution was rinsed, allowed to drain, and placed in the next higher specific gravity solution. Each float product from each size was dried, weighed, and weights tabulated in figure 6. The final sink product in the 1.90 solution was also dried and weighed. The solutions for the 3 x 14 mesh size fractions were made by adding calcium chloride or zinc chloride to water depending on whether the specific gravity needed was above 1.55 or below.

Each float product and the sink from the 1.90 specific gravity solution was reduced to size and ground to minus 60 mesh for ash analysis. Each specific gravity fraction was analyzed for ash on the air-dry basis by

	As Received		After Crushing	
Size	Weight lbs.	Weight per cent	Weight lbs.	Weight per cent
Plus 3"	630.0	38.6		
3" x 1 1/2"	203.0	12.4	537.0	32.8
1 1/2" x 3/4"	282.0	17.3	432.0	26.4
3/4" x 3/8"	115.5	7.1	210.5	12.9
3/8" x 3/16"	93.5	5.7	113.5	7.0
3/16" x 14 mesh	176.0	10.8	191.0	11.7
Minus 14 mesh	133.0	8.1	149.0	9.2

Figure 5. Size Distribution of Sample of Merrimac Seam for Washability

the procedure outlined in ASTM Standards No. 271-44 for analyzing coal and coke.[5] Each ash per cent was entered in the float and sink data tables, figure 6. The table was completed by calculating the weight per cent of each specific gravity fraction on each size. The cumulative weight per cent columns are in each case the sum of all the preceding weight per cents.[10] The values listed in the last column as cumulative ash represent the ash analysis on the total float coal on the corresponding specific gravity shown in the specific gravity column. For example, (Fig. 6) on the size fraction 3 x 1 1/2, the total float coal on specific gravity 1.62 would analyze 13.25% ash, although the ash analysis on the 1.55-1.62 fraction shows 46.03% ash. A sample calculation on the 3 x 1 1/2 size fraction will illustrate the method of arriving at the cumulative ash value. Referring to figure 6, the cumulative ash for the float on the 1.35 fraction is the same as the corresponding percentage listed under ash per cent. The next cumulative ash values were calculated.

$$\frac{\text{Weight per cent x Ash per cent}}{100} = \text{units of ash}$$

In the float on 1.35 there is

$$\frac{13.30 \times 6.20}{100} = 0.825 \text{ units of ash}$$

In the 1.35-1.435 fraction there is

$$\frac{15.05 \times 11.06}{100} = 1.663 \text{ units of ash}$$

The sum of these, or 2.488, is the units of ash in the total material lighter than 1.435 specific gravity which, as shown by the cumulative weight per cent column, comprises 28.35% of the total sample. Then $\frac{2.488 \times 100}{28.35} = 8.78\%$,

Size	Sp. Gr.	Wt. K.G.	Wt. %	Ash %	Cum. Wt.%	Cum. Ash %
	Float 1.35	39.8	13.30	6.20	13.30	6.20
	1.35-1.435	45.0	15.05	11.06	28.35	8.78
3 on 1 1/2	1.435-1.55	44.0	14.70	18.47	43.05	12.09
42.45	1.55-1.62	8.91	2.98	30.18	46.03	13.25
	1.62-1.70	6.28	2.10	36.83	48.13	14.30
	1.70-1.90	21.50	7.17	51.30	55.30	19.10
	Sink 1.90	33.80	44.70	76.30	100.00	44.80
	Float 1.35	51.09	11.83	6.57	11.83	6.57
	1.35-1.435	115.12	26.65	9.00	38.48	8.26
1 1/2 on 3/4"	1.435-1.55	65.56	15.18	20.32	53.66	14.90
33.21	1.55-1.62	32.42	7.50	26.85	61.16	16.68
	1.62-1.70	20.01	4.63	33.49	65.79	18.10
	1.70-1.90	30.29	7.01	43.11	72.80	21.00
	Sink 1.90	117.51	27.20	72.38	100.00	36.80
	Float 1.35	8.90	9.13	5.12	9.13	5.12
	1.35-1.435	24.80	25.43	10.77	34.56	9.28
3/4 on 3/8	1.435-1.55	15.60	16.03	16.28	50.58	11.50
32.05	1.55-1.62	13.00	13.37	27.05	63.95	14.73
	1.62-1.70	5.00	5.14	34.90	69.09	16.23
	1.70-1.90	6.30	6.47	47.30	75.56	18.89
	Sink 1.90	23.80	24.43	77.00	100.00	33.08
	Float 1.35	.67	2.53	5.28	2.53	5.28
	1.35-1.435	9.50	35.80	9.06	38.33	8.81
3/8 on 3/16	1.435-1.55	8.65	32.60	15.50	70.93	11.90
30.60	1.55-1.62	1.38	5.21	29.50	76.14	13.20
	1.62-1.70	.66	2.49	35.80	78.63	13.90
	1.70-1.90	1.25	4.53	47.50	83.16	15.65
	Sink 1.90	4.44	16.84	76.20	100.00	25.90
	Float 1.25	.155	.96	7.57	.96	7.51
	1.25-1.35	3.322	20.56	7.76	21.52	7.76
3/16 on 14 mesh	1.35-1.435	5.510	34.10	10.28	55.62	9.31
27.64	1.435-1.55	3.230	19.99	20.49	75.61	12.27
	1.55-1.62	.724	4.48	30.48	80.09	13.29
	1.62-1.70	.423	2.62	36.23	82.71	14.02
	1.70-1.90	.669	4.14	48.37	86.85	15.67
	Sink 1.90	2.124	13.15	76.32	100.00	24.04
	Float 1.35	103.94	13.46	6.33	13.46	6.33
	1.35-1.435	199.93	25.91	9.72	39.37	8.56
Composite	1.435-1.55	137.04	17.77	18.97	57.14	11.80
3 x 14 mesh	1.55-1.62	56.43	7.32	29.19	64.46	13.77
	1.62-1.70	32.37	4.20	34.43	68.66	15.04
	1.70-1.90	60.00	7.80	46.63	76.46	18.26
	Sink 1.90	181.67	23.54	73.85	100.00	31.35

Figure 6. Float and Sink Data on Sample Taken from Merrimac Seam.

which is the ash per cent of the total float on 1.435. These calculations are made for all the sizes on all specific gravity fractions including sink in 1.90.

After the float and sink data for all the individual size fractions are calculated, a composite table is calculated for a 3 x 14 mesh product. This table is made by combining the weight per cent from each specific gravity fraction on each size. Ash per cents are combined by computing the units of ash each specific gravity fraction of each size contributes to the total 3 x 14 mesh product.

To better use the data tabulated in figure 6, four curves for each size fraction, including the composite, were drawn — the cumulative, the elementary ash, the specific gravity, and the ±0.10 specific gravity distribution curve. The curves were drawn with a No. 48 Copenhagen ship curve. Referring to the curve 3 x 1/2 (Fig. 7), it is seen that the abscissa is cumulative ash per cent, elementary ash per cent, and specific gravity. All the curves have a common ordinate; i.e., cumulative weight per cent.

The cumulative curve shows the yield of float coal resulting from a 100% efficient separation at a selected cumulative ash percentage. The curve is outlined by plotting the cumulative weight per cents against cumulative ash per cents.

The elementary ash curve is the derivative of the cumulative ash curve. Although mathematically an approximation, it shows the rate of change of ash content at different yields. For example, from the data for 3 x 1 1/2, the float on 1.35 analyzes 6.20% ash and comprises 13.30% by weight. The 6.20% ash is the average ash content, some particles have a higher ash content, some less. If we assume that half the particles have a higher ash content and half have a lower, then 1/2 x 13.30 = 6.65%. Then the first point on the curve is found by plotting 6.65% cumulative weight against 6.20% elementary ash. The elementary curve shows then the ash percentage in the highest ash particle included in a float coal product of any given cumulative ash percentage. Other points on the curve are found by plotting half the weight per cent of the specific gravity interval involved, plus the cumulative weight per cent of all material of lower specific gravity against the ash content of the specific gravity interval involved.

The specific gravity curve is constructed by plotting the specific gravities against the corresponding cumulative weight per cents. This curve shows the yield of float coal for a perfect separation at any specific gravity within the range of the gravities of the float and sink tests.

The ± 0.10 specific gravity distribution curve shows the percentage of the raw coal feed that lies in the range of +0.10 or -0.10 specific gravity of any given point on the specific gravity curve. For instance, the ± 0.10 value at 1.45 specific gravity is the percentage of total coal that lies within 1.35-1.55 specific gravity. This curve graphically illustrates the relative ease of separation at any given specific gravity. To plot the curve the difference in yield at ± 0.10 of the specific gravity point desired is divided by the yield at 2.00 specific gravity and the adjusted percentage plotted against the specific gravity point desired. This adjustment is necessary to take care of the heavy, faster-settling particles that affect the efficiency of separation very little.

After all four curves are plotted, the problem of separation and the results expected are readily available. For example, on the 3 x 1 1/2" size (Fig. 7) if one wants to know the yield, if the separation is to be

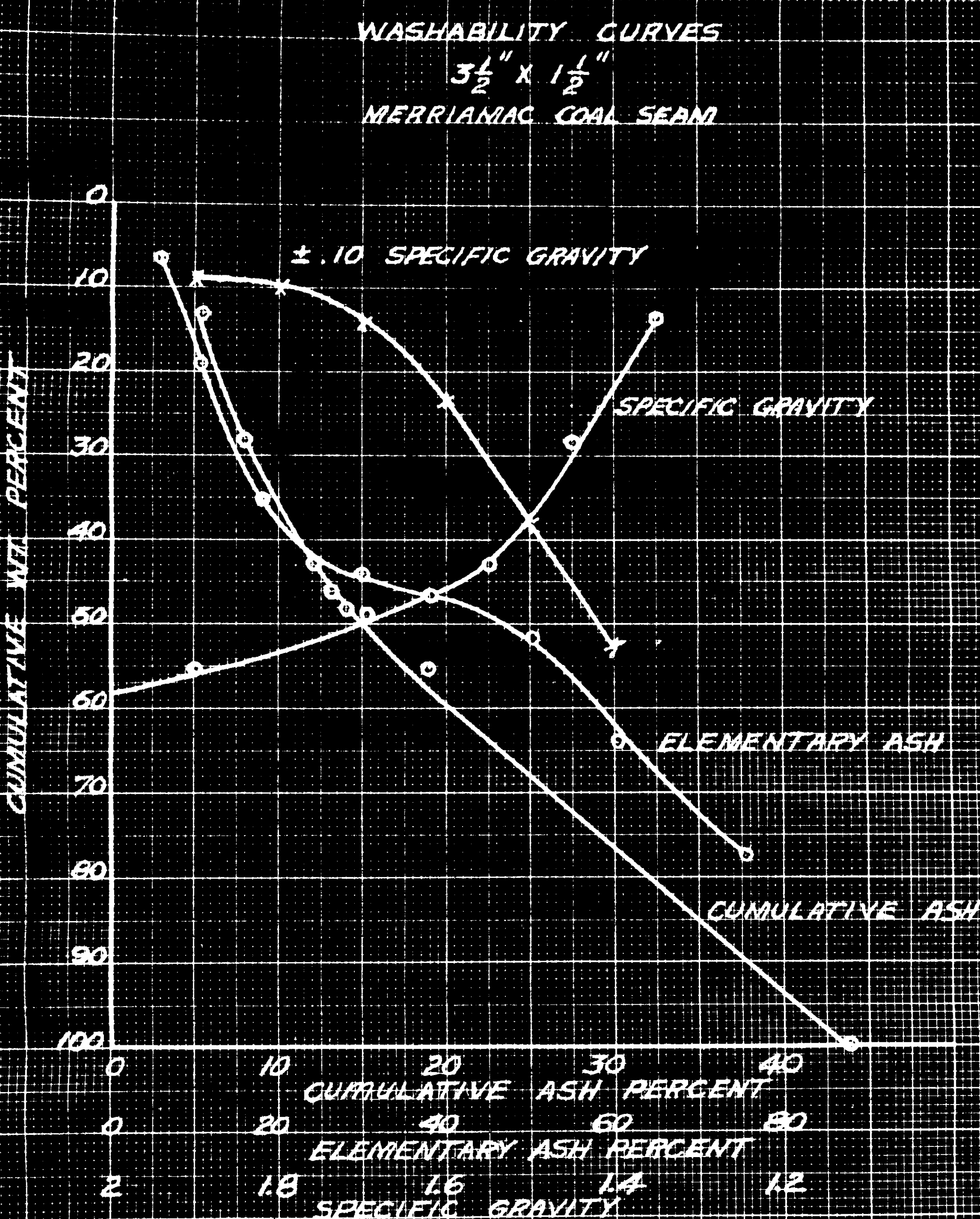

WASHABILITY CURVES
$3\frac{1}{2}'' \times 1\frac{1}{2}''$
MERRIAMAC COAL SEAM
±.10 SPECIFIC GRAVITY
SPECIFIC GRAVITY
ELEMENTARY ASH
CUMULATIVE ASH
CUMULATIVE WT. PERCENT
0
10
20
30
40
50
60
70
80
90
100
0
10
20
30
40
CUMULATIVE ASH PERCENT
0
20
40
60
80
ELEMENTARY ASH PERCENT
2
1.8
1.6
1.4
1.2
SPECIFIC GRAVITY

Figure 7

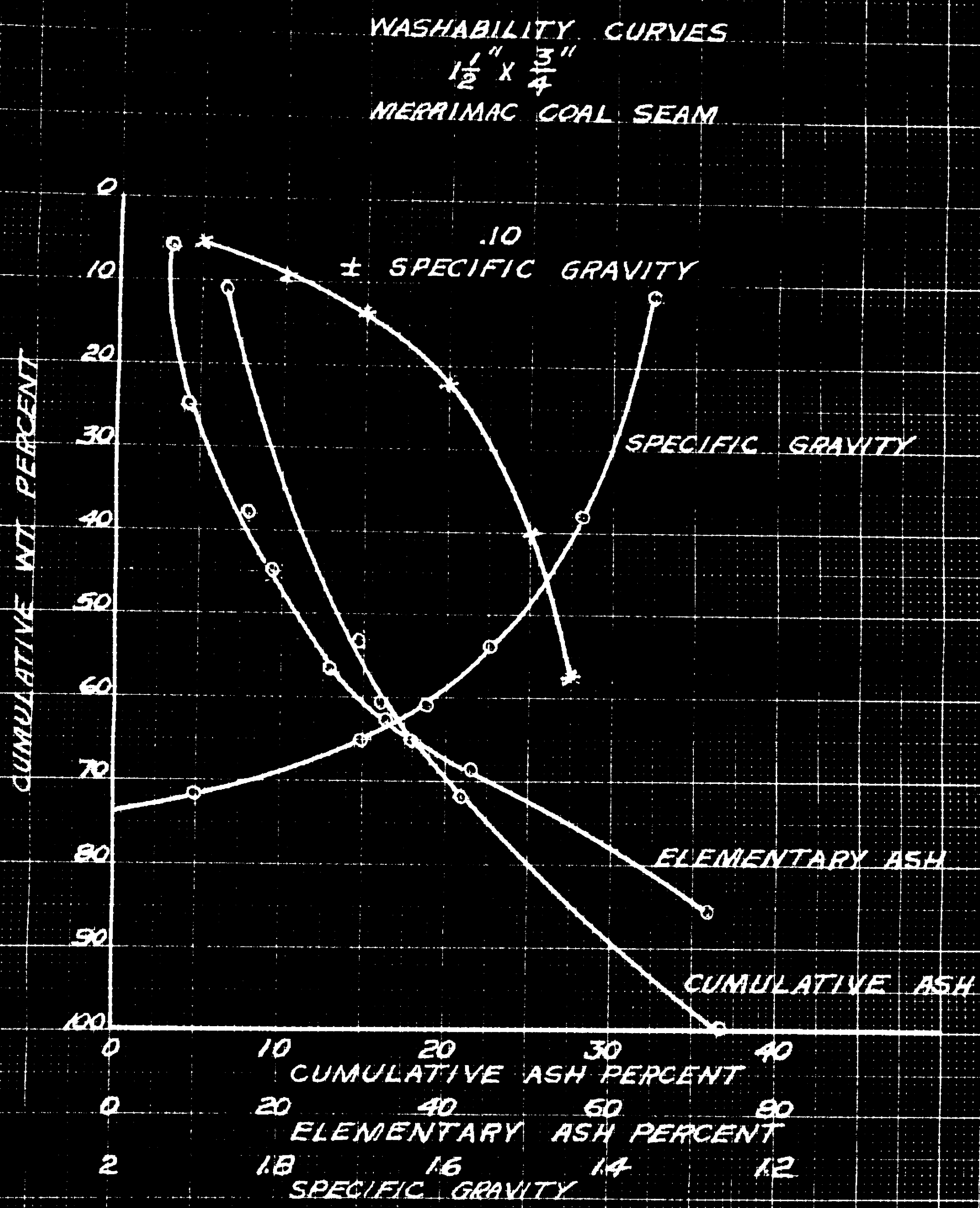
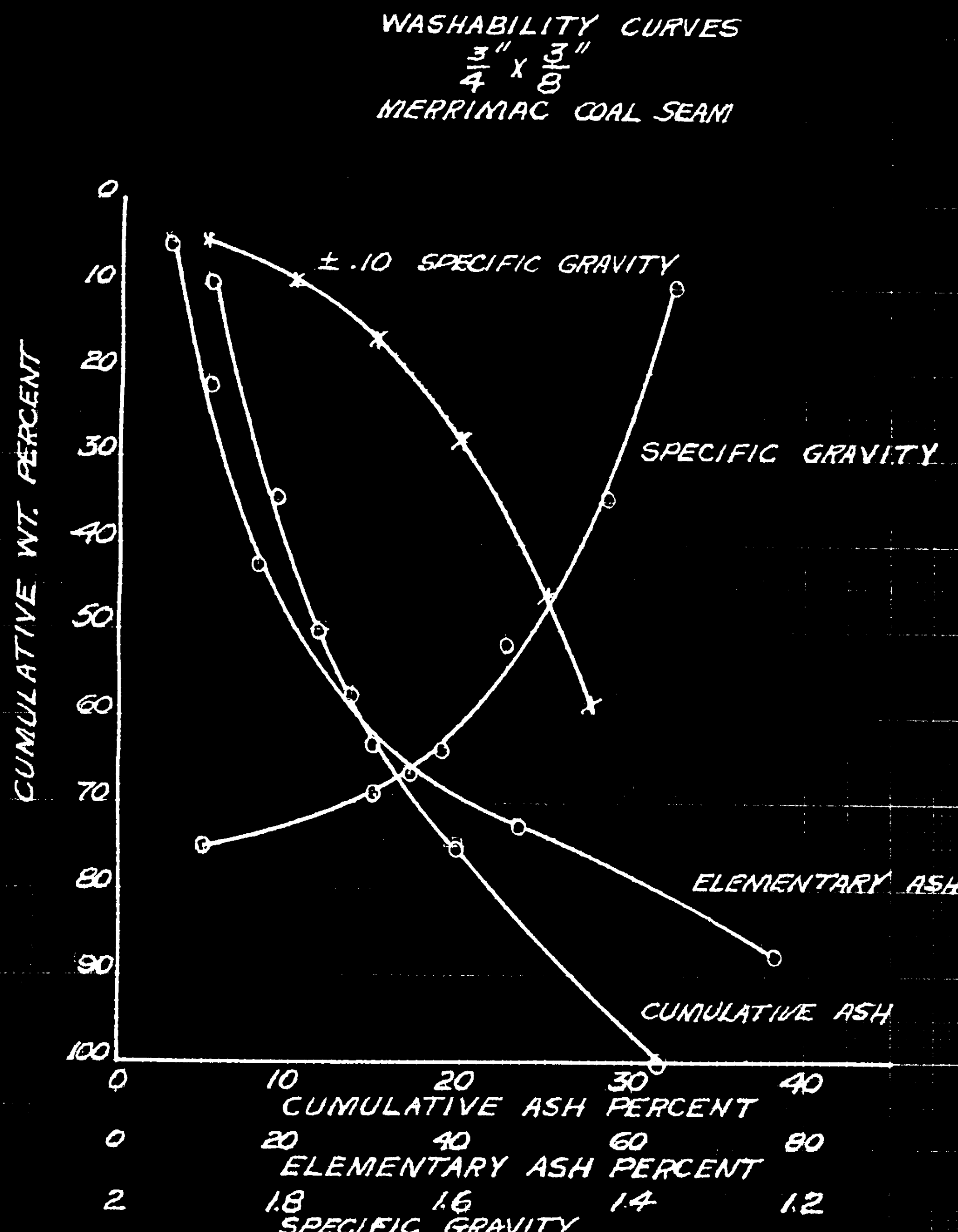

WASHABILITY CURVES
1 1/2" X 3/4"
MERRIMAC COAL SEAM
.10
± SPECIFIC GRAVITY
SPECIFIC GRAVITY
ELEMENTARY ASH
CUMULATIVE ASH
CUMULATIVE WT. PERCENT
0
10
20
30
40
50
60
70
80
90
100
0
10
20
30
40
CUMULATIVE ASH PERCENT
0
20
40
60
80
ELEMENTARY ASH PERCENT
2
1.8
1.6
1.4
1.2
SPECIFIC GRAVITY

Figure 8



WASHABILITY CURVES
3/4" x 3/8"
MERRIMAC COAL SEAM
± .10 SPECIFIC GRAVITY
SPECIFIC GRAVITY
ELEMENTARY ASH
CUMULATIVE ASH
CUMULATIVE WT. PERCENT
0 10 20 30 40 50 60 70 80 90 100
0 10 20 30 40
CUMULATIVE ASH PERCENT
0 20 40 60 80
ELEMENTARY ASH PERCENT
2 1.8 1.6 1.4 1.2
SPECIFIC GRAVITY

Figure 9

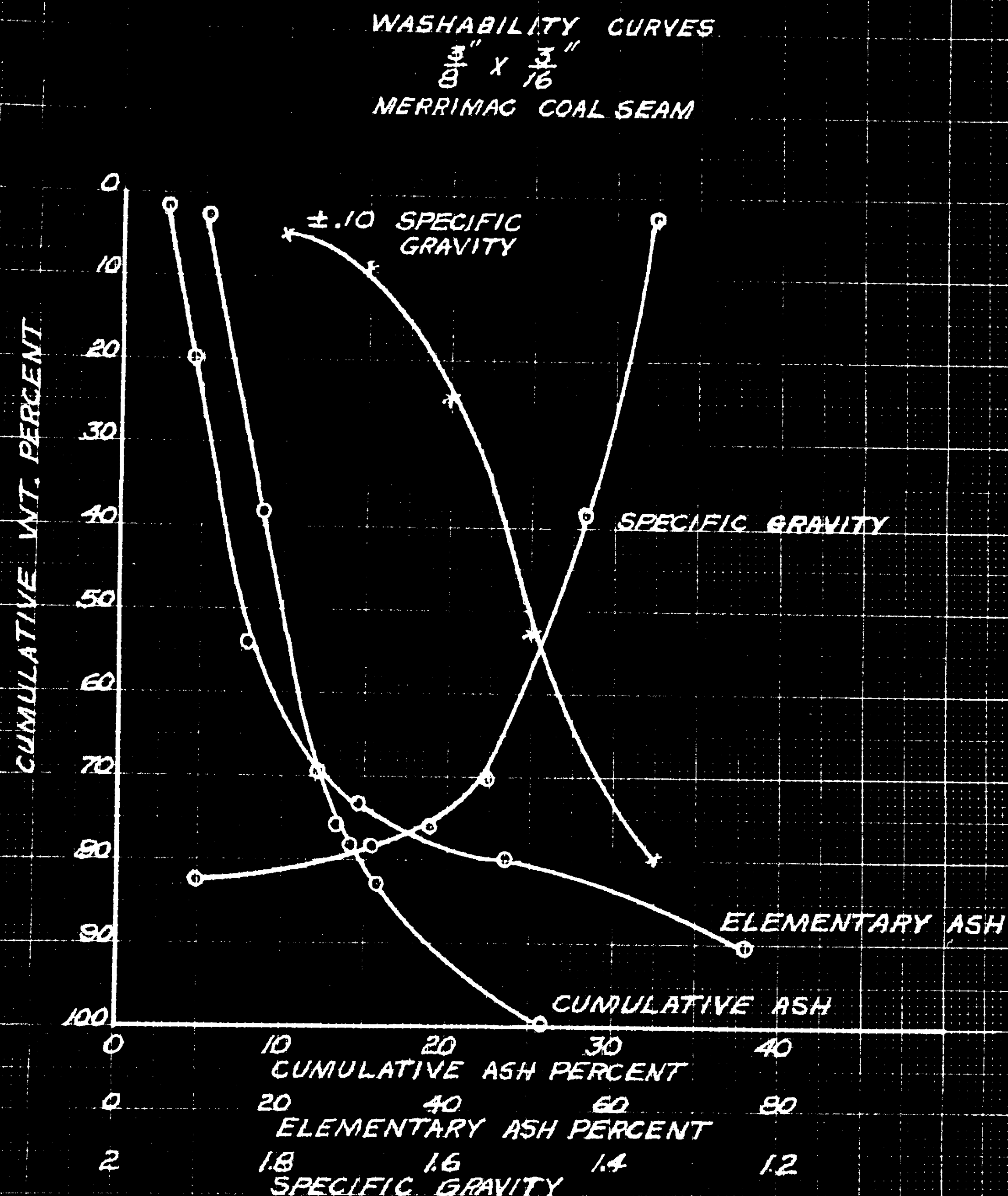

WASHABILITY CURVES
3/8" X 3/16"
MERRIMAC COAL SEAM
±.10 SPECIFIC GRAVITY
SPECIFIC GRAVITY
ELEMENTARY ASH
CUMULATIVE ASH
CUMULATIVE WT. PERCENT
0
10
20
30
40
50
60
70
80
90
100
0 10 20 30 40
CUMULATIVE ASH PERCENT
0 20 40 60 80
ELEMENTARY ASH PERCENT
2 1.8 1.6 1.4 1.2
SPECIFIC GRAVITY

Figure 10

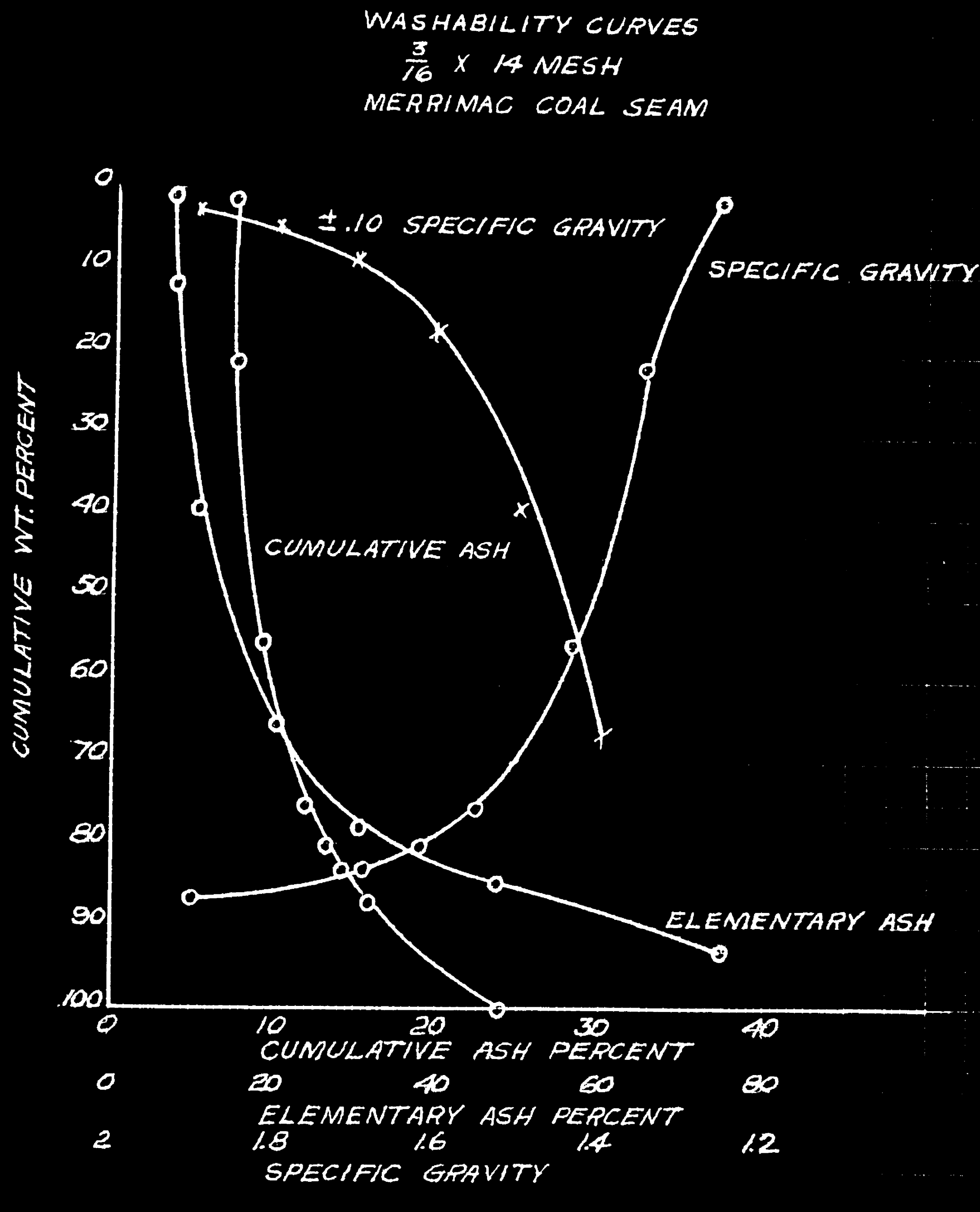

WASHABILITY CURVES
3/16 X 14 MESH
MERRIMAC COAL SEAM
±.10 SPECIFIC GRAVITY
SPECIFIC GRAVITY
CUMULATIVE ASH
ELEMENTARY ASH
CUMULATIVE WT. PERCENT
0
10
20
30
40
50
60
70
80
90
100
0
10
20
30
40
CUMULATIVE ASH PERCENT
0
20
40
60
80
ELEMENTARY ASH PERCENT
2
1.8
1.6
1.4
1.2
SPECIFIC GRAVITY

Figure 11

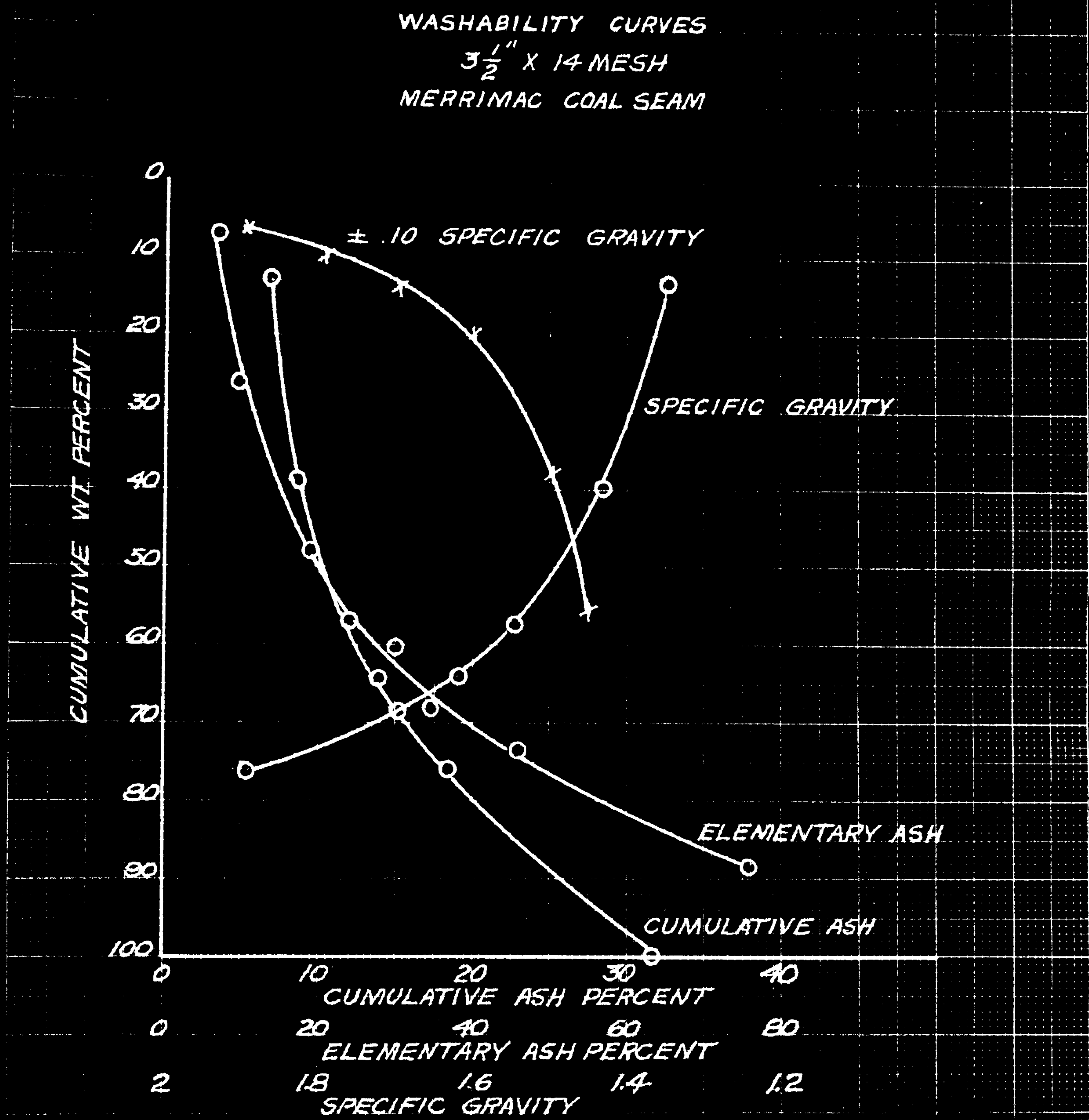

WASHABILITY CURVES
3 1/2" X 14 MESH
MERRIMAC COAL SEAM
± .10 SPECIFIC GRAVITY
SPECIFIC GRAVITY
ELEMENTARY ASH
CUMULATIVE ASH
CUMULATIVE WT. PERCENT
0
10
20
30
40
50
60
70
80
90
100
0
10
20
30
40
CUMULATIVE ASH PERCENT
0
20
40
60
80
ELEMENTARY ASH PERCENT
2
1.8
1.6
1.4
1.2
SPECIFIC GRAVITY

Figure 12

made at 1.60 specific gravity, pick the 1.60 point on the specific gravity curve, follow over horisontally until the same abscissa intersects the cumulative ash curve, and drop a vertical ordinate, and read the cumulative ash on the cumulative ash scale. The height of the point on the specific gravity curve is the cumulative weight or yield. At 1.60 specific gravity on this size the yield expected is 46% at an average ash of 13%. Included in the float would be particles having an ash content of 34% as read from the elementary ash curve.

C. Rank

The coal of this area, although commonly known on the market as Virginia Anthracite, falls in the semianthracite group. The specifications for a semianthracite coal, according to the American Society for Testing Materials, are: dry, mineral matter free, fixed carbon, 86 per cent or more, and less than 92 per cent (volatile matter 14 per cent or less, and more than 8 per cent.)

D. Impurities Occurring Physically Within the Seam

The heterogeneous character of the seam can best be illustrated by a columnar section of the seam. Figure 13 shows the character of the bands that comprise the total seam, their percentage of the seam and their ash content.

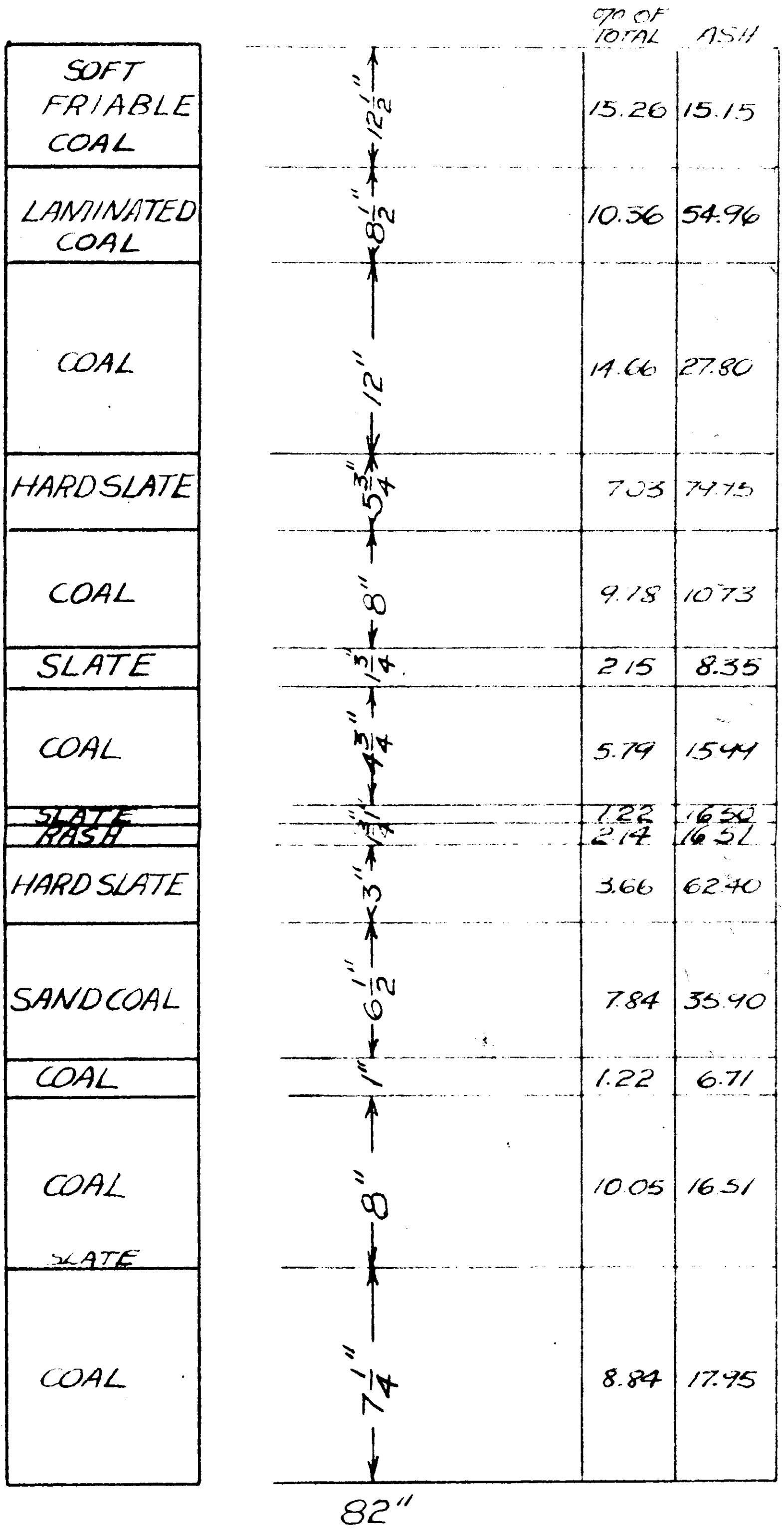


Figure 13. Merrimac Coal Seam

VII. COAL PREPARATION

From the analyses of the bands that comprise the total seam (Fig. 13) it is seen that the smallest percentage of ash in any individual band is 6.71%, comprising approximately 1.22% of the total. The next higher ash band contains 8.35% ash and makes up 9.77% of the total seam. Only 51.5% of the total seam, as revealed by analyses of individual bands, analyzed less than 20% ash.

Washability curves on a composite 3 1/2 x 14 mesh (Fig. 12) shows that if the coal is washed at a specific gravity of 1.60, one can expect to get a product containing 13% ash and a recovery of 62%. If washed at a specific gravity of 1.70, a product containing 15% ash and 68% recovery can be expected.

From analyses of individual bands only 28% of the entire seam analyzed less than 15.5% ash, yet washability tests indicate a yield of 68% with only 15% ash.

It is apparent that in any band in the seam regardless of the total ash analyses there must be some coal that contains a much lower percentage of ash, and that it is possible to separate this coal on a gravity basis if crushed to the proper size. For example, in band 2 (Fig. 13) which comprises approximately 10% of the total and contains 54.96% ash by analysis there is coal that contains a much lower percentage of ash. Obviously this conclusion cannot be applied to homogeneous slate partings occurring in the seam.

In order to expand the market for Virginia Anthracite and to hold present markets, it must be cleaned to at least 18% ash.[24] Referring to the composite washability curve, 3 1/2 on 14 mesh, (Fig. 12) if the separation is to be made at 1.60 specific gravity, a 13% ash product can be expected with a yield of 62%. If the separation is to be made at 1.70, a 15% ash product can be expected with a yield of 68%. From the slope of the specific gravity curve it is apparent that a very little increase in yield would result from making a separation at a gravity higher than 1.70.

If a separation is to be made a the 1.60-1.70 specific gravity range, the desired ash percentage can be obtained with fair recovery for this particular seam. A separation at this range is possible with a number of dense media commercial processes provided there will not be too great a percentage of raw coal feed occurring near the 1.60-1.70 specific gravity range. The current practice[12] assumes that the most efficient separation is possible where 10% or less of the raw coal feed occurs within ±0.10 specific gravity of the separation point desired. Referring to the composite washability curve, 3 1/2 on 14 mesh, (Fig. 12) if a separation is to be made at 1.60 specific gravity, 20% of the raw coal feed lies between ±0.10 of the specific gravity 1.60; if a separation is to be made at 1.70, 13% of the raw coal feed lies between ±0.10 of specific gravity 1.70. Thus it is apparent that the desired separation will be a difficult one and that overlapping of the sink and float products will occur unless the separation gravity is controlled accurately.

Washing at an average specific gravity of 1.65 would theoretically yield a product containing 14% ash and 66% recovery. In order to meet the daily requirement of 1000 tons of cleaned coal, the preparation plant would be required to handle:

$$\frac{1000}{0.66 \times 0.90} = 1680 \text{ tons}$$

(The 90% represents the cleaning efficiency expected as far as recovery is concerned.) 1680 tons of raw coal feed would mean that 680 tons of refuse must be handled daily. If the two homogeneous slate bands, Nos. 4 and 10, (Fig. 13) analyzing 79.15% and 62.40% ash were left underground, the theoretical yield would still be 66% of the entire seam; but the yield from the preparation plant would be 66% + 12.8% = 78.8%, since the two bands make up 12.8% of the seam. The preparation plant would now be required to handle

$$\frac{1000}{78.8 \times 0.90} = 1410 \text{ tons}$$

with 410 tons of refuse to be handled.

From a study of the present selling price of Virginia Anthracite (Fig. 14) it is apparent that the preparation plant must be equipped to prepare for market the greatest percentage of the larger sizes with the minimum production of fines. However, from washability curves (Figs. 7-12) it is apparent that the finer the coal (down to 14 mesh or fines) the cleaner it can be washed. The maximum size is limited to 3 1/2" in order to free the coal from impurities for washing. This is readily seen from the washability curve on the 3 1/2" x 1 1/2" size. If this size were washed at a specific gravity of 1.60, the recovery would be only 46%. The first step then in treating the raw coal feed is to reduce it to minus 3 1/2".

Screen analyses on the sample after crushing to minus 3 1/2" shows 59.2% of the coal falling into the 3 1/2" on 3/4" size bracket. (Fig. 5) Average sales for Virginia Anthracite during a six month period of 1946

Grade	Rail	Truck
Stove	7.79	8.80
Nut	7.79	8.80
Pea	5.60	
Buckwheat	4.65	
Rice	3.94	
Culm	2.10	

Figure 14. Selling Price of Virginia Semianthracite F.O.B. Mine

Size	Market Designation	Per Cent of Total Shipments
3 1/2 x 2 1/8	Stove	8.98
2 1/8 x 7/8	Nut	31.82
7/8 x 1/2	Pea	17.91
1/2 x 1/4	Buckwheat	14.54
1/4 x 1/8	Rice	8.86
Minus 1/8	Culm	17.89

Figure 15. Size Distribution of Sales by Great Valley Anthracite Coal Company (6 months average, 1946)

shows that 40.8% of the coal was sold in that size bracket with only 8.98% falling into the top size bracket. (Fig. 15) It follows then that in the preparation plant some provision must be made for recrushing the Stove size to smaller sizes.

In selecting a washing unit to affect the separation at the 1.60-1.70 specific gravity range, several requirements must be met:

1. a unit using a suspension as the separating fluid to obtain the high gravities necessary,
2. a unit capable of handling a large volume of refuse,
3. a unit capable of fine adjustment since the separation is not a clear cut one.

The heavy density process using finely ground magnetite in suspension appears to answer the above three requirements. Apparent specific gravities above 2.00 are readily obtainable in this process. The heavy density process patents are largely controlled by the American Cyanamid Company, but several equipment companies sell washing plants based on the same principles. Western Electric Machinery Company manufactures a washing plant using the heavy density principle with a conical separatory vessel and an air lift to dispose of the sink or refuse. Link-Belt manufactures a washing unit with a cylindrical separatory vessel and a revolving drum with vanes, similar to a Bradford Breaker, to dispose of the refuse. In view of the large quantity of refuse to be handled, the Link-Belt appears to be better suited for the job. Pittsburg Consolidated Coal Company has found the Link-Belt successful in handling over 200 tons per hour of raw coal feed averaging 33% refuse at the Champior Mine. The Link-Belt heavy density process is capable of adjustment to $\pm$0.01 specific gravity which satisfies the third requirement.

No provision is made for cleaning the fines or minus 14 mesh material. The selling price is only $2.10 per ton while the cleaning costs are four times that of coarse coal.[16] Too, wet cleaning methods on fines are much more efficient than dry methods, but the present market for the fines demands a dry product[24] and this would entail an expensive drying system.

The present size designations, Stove, Nut, Pea, Buckwheat, and Rice, of Virginia Anthracite do not correspond exactly to Standard Anthracite Sizing Specifications -- the greatest difference is in the Stove size. The following table illustrates the difference between Virginia Anthracite designation and Standard Anthracite Sizing Specifications:

	Virginia Anthracite	Standard
Stove	3 1/2 x 2 1/8	2 7/16 x 1 5/8
Nut	2 1/2 x 7/8	1 5/8 x 13/16
Pea	7/8 x 1/2	13/16 x 9/16
Buckwheat	1/2 x 1/4	9/16 x 5/16
Rice	1/4 x 1/8	5/16 x 3/16
Fines	Minus 1/8	

To produce the five grades of cleaned coal, Stove, Nut, Pea, Buckwheat, Rice, and the uncleaned fines, the flowsheet, figure 16, was designed. Referring to the flowsheet, (Fig. 16) first, the raw coal from the belt conveyor is fed to a scalping shaker screen where the plus 3 1/2" coal is scalped off. If found necessary, the plus 3 1/2" slate can be hand picked at this point. The undersize goes to a vibrating screen where the minus 14 mesh is separated for loading directly into railroad cars. The oversize from the 3 1/2" scalping screen is conveyed over a magnetic pulley to remove tramp iron to a roll crusher set at 3 1/2". The crushed product returns to make up part of the feed of the vibrating screen. The 3 1/2" x 14 mesh raw coal is fed to the washing unit. The float product overflows to a dewatering screen under which is two sumps for collecting the magnetite medium. From the first sump the medium is circulated directly back to the washing unit. Over the second sump are watersprays which remove the remainder of the medium from the clean coal. The clean coal is conveyed to a series of shaking screens for final sizing from which the sized coal is conveyed into bins for loading into railroad cars or trucks. The refuse flows over a dewatering arrangement identical to the clean coal. The dewatered refuse goes to a vibrating screen where the plus 1" refuse is scalped off for recrushing in a roll crusher set at 1". The crushed product returns to the washing unit for rewashing. The minus 1" refuse is conveyed to the refuse dump. The magnetite medium and fines from the dewatering screens (sink and float product) are passed through a magnetic block to further magnetize the medium. It is then thickened and pumped to two Crockett type magnetic separators where the medium is recovered from the slime. The clean magnetite flows to an Akins type spiral classifier where it is densified. From the classifier the magnetite flows past a demagnetizing coil and then back to the washing unit as clean medium. The slimes from both the float and sink medium cleaning circuits leave the plant as refuse on the refuse conveyor. Overflow from each thickener flows to a clarifier where the clean water is pumped back into the system. The Stove size coal from the final sizing shakers can be diverted back to the small secondary roll crusher to enable the production of a greater percentage of the smaller sizes.

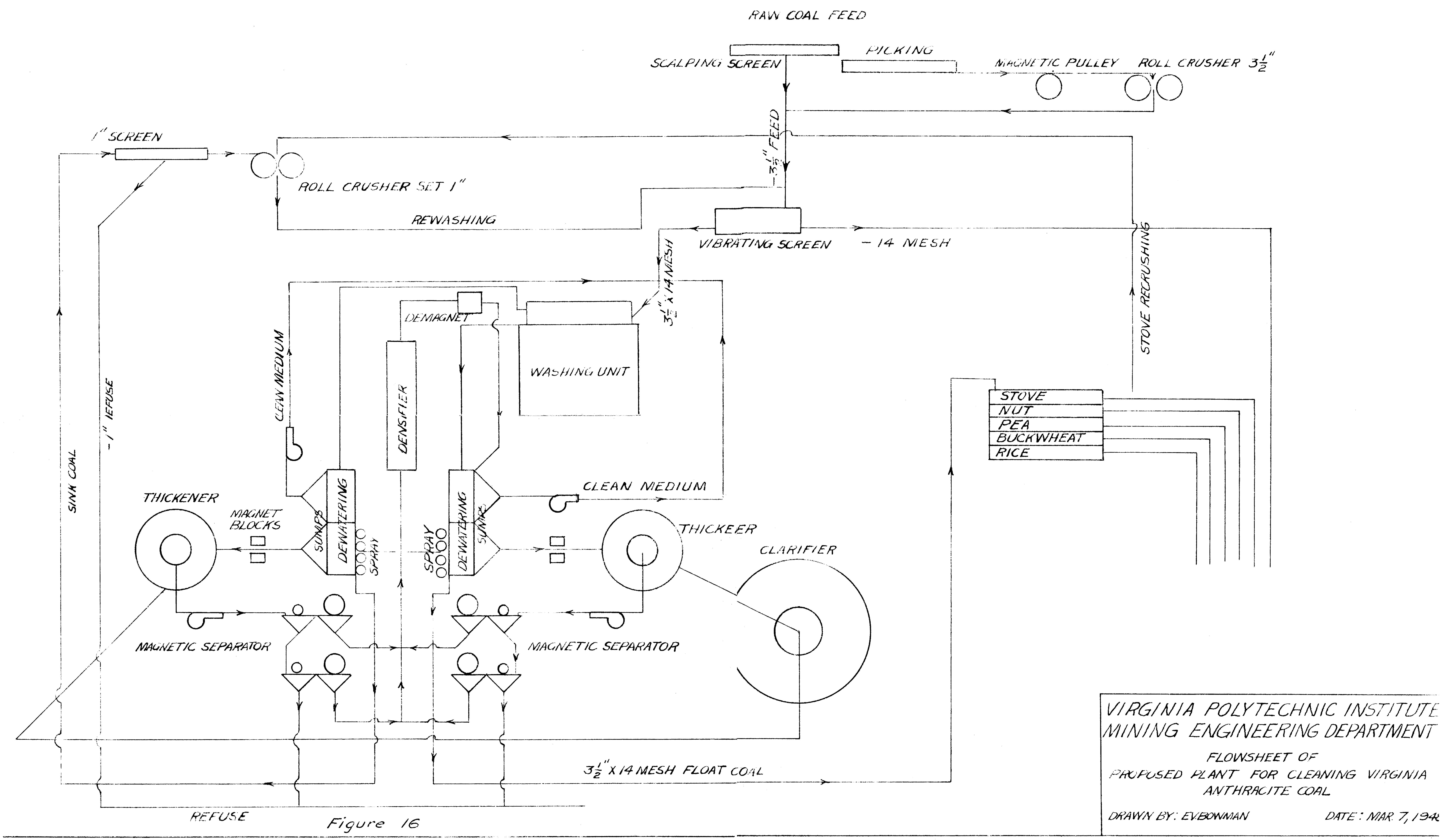

RAW COAL FEED
SCALPING SCREEN
PICKING
MAGNETIC PULLEY
ROLL CRUSHER 3½"
-3½" FEED
1" SCREEN
ROLL CRUSHER SET 1"
REWASHING
VIBRATING SCREEN
-14 MESH
3½" x 14 MESH
STOVE RECRUSHING
DEMAGNET
WASHING UNIT
CLEAN MEDIUM
DENSIFIER
STOVE
NUT
PEA
BUCKWHEAT
RICE
-1" REFUSE
SINK COAL
THICKENER
MAGNET BLOCKS
SUMPS
DEWATERING
SPRAY
CLEAN MEDIUM
THICKEER
CLARIFIER
MAGNETIC SEPARATOR
MAGNETIC SEPARATOR
3½" x 14 MESH FLOAT COAL
REFUSE
VIRGINIA POLYTECHNIC INSTITUTE
MINING ENGINEERING DEPARTMENT
FLOWSHEET OF
PROPOSED PLANT FOR CLEANING VIRGINIA ANTHRACITE COAL
DRAWN BY: E.V. BOWMAN
DATE: MAR. 7, 1942

Figure 16

VIII. MINING METHODS

A. Bringing the Coal to the Surface

1. Hoisting, General Considerations

From the study of the washability curves, 65% recovery can be expected, so in order to obtain 1000 tons of marketable coal 1600 tons must be mined.

It is necessary to approximate a haulage profile from actually measuring the dip at the outcrop and studying the probable relationships of the strata as interpreted by geologists. This approximation is necessary since no bore hole data are available. The profile is estimated to be: first, 1500 ft. down the slope 26°; second, 1500 ft., 18°; and third, 1500 ft., 10°.

Several problems were presented at the beginning. How much coal should be hoisted per trip? What should be the maximum hoisting speed and the rate of acceleration? What should the size and weight of cars be? To avoid excessive rope sizes, it is believed that 20 tons per trip should be the maximum. Using 20 tons as a basis, an additional load of 8 tons could be assumed for cars since the dead-live load ratio for modern mine cars is 60-40. This load could be easily handled in four five-ton mine cars making a total load per trip of 28 tons. Rope speeds vary widely with different operations depending on inclination and condition of track. With a well-balanced track, kept in good state of maintenance, a 1200 ft./min. rope speed would not be excessive. a 1 ft./sec.^{2} acceleration will be taken as maximum. Other items of data to be selected are: How many hours per day will the hoist be in operation, and what time should be allowed for coupling, uncoupling, etc.? Eight hours should certainly be maximum for one shift, although there would be nothing to prevent additional hoisting if storage facilities were available at the slope bottom. With proper track layout one minute should be sufficient to allow for dead time. Calculations will be made to determine the horsepower of a hoist necessary to hoist the required load from different levels. First, the calculations will be based on an unbalanced system.

2. Hoisting, Unbalanced System — 1500 ft. level

Calculations will follow the procedure outlined by H. H. Broughton in his book on electric winders.[8]

Data

Load	40,000 lbs. maximum
Cars	Weight 4,000 lbs., capacity 10,000 lbs.
Slope	1500' at 26°, 1500' at 18°, 1500' at 10°
Output	200 tons an hour for eight hours
Maximum Speed	1200 ft. a minute or 20 ft. per second
Acceleration	1 ft./sec.^{2}
Safety Factor (for rope)	7-10
Dead Time	60 seconds

Subdivision of Cycle

$$a = \frac{v_1 - v_0}{t}$$

$$1 = \frac{20 - 0}{t}, \quad t = 20 \text{ secs. time for acceleration and retardation}$$

$d = 1/2at^2 = 1/2 \times 1 \times 20^2 = 200$ ft. distance travelled during acceleration and retardation

$1500 - (200 + 200) = 1100$ ft. distance to be travelled at full speed

$\frac{1100}{20} = 55$ secs. time for full speed

Acceleration	20 secs.
Full Speed	55 secs.
Retardation	20 secs.
Dead Time	60 secs.
	155 secs. total time one way
	310 secs. total time round trip

$8 \times 60 \times 60 = 28{,}800$ secs. daily hoisting time

$\frac{28{,}800}{310} = 93$ trips per day

$\frac{1600 \times 2000}{93} = 34{,}400$ lbs. per trip can be handled in four cars

This would make a total load per trip of

$4 \times (10{,}000 + 4000) = 56{,}000$ lbs. (assuming full cars)

Try 1 3/4", 6 x 19 hoisting rope (monitor steel)[3]

Breaking strength 248,000 lbs. Weight per ft. 4.90

Gravity load — $[56{,}000 + (1500 \times 4.9)]0.4384$	= 27,800
Car friction load — $56{,}000 \times 0.0270$	= 1,510
Rope friction load — $1500 \times 4.9 \times 0.0540$	= 397
	29,707
Plus 10% acceleration stress	2,971
	32,678

Safety factor $= \frac{248,000}{32,678} = 7.6$ rope is satisfactory.

From the above calculations it is apparent that from the time element alone 1500 ft. down the slope is the extent possible with an unbalanced system using 20 tons as the maximum load per trip. If larger loads were hoisted, a rope even larger than the 1 3/4" would be necessary.

3. <u>Hoisting, Balanced -- 1000 ft. level</u>

Data

Load	40,000 lbs. maximum
Cars	Weight 4,000 lbs. capacity 10,000 lbs.
Slope	1,500' at 26°, 1500' at 18°, 1500' at 10°
Output	200 tons per hour for eight hours
Maximum Speed	1200 ft. a minute or 20 ft. per second
Acceleration	1 ft./sec.^{2}
Safety Factor	7-10
Dead Time	60 seconds

Subdivision of Cycle

$t = \frac{v_1 - v_0}{a} = \frac{20 - 0}{1} = 20$ secs. for acceleration and retardation

$d = 1/2at^2 = 1/2 \times 1 \times 20^2 = 200$ ft. travelled during acceleration and retardation

$1000 - (200 + 200) = 600$ ft. for full speed

$\frac{600}{20} = 30$ secs. time for full speed

Acceleration	20 secs.
Full Speed	30 secs.
Retardation	20 secs.
Dead Time	60 secs.
	130 secs. time per trip

$\frac{3600}{130}$ = 27 trips per hour

$\frac{200}{27}$ = 7.42 tons per trip

Use two five-ton cars or 20,000 lbs. per trip

Making a total load of 28,000 lbs.

Rope Size[3]

Try 1 1/4" 6 x 19 hoisting rope with a breaking strength of 129,200 lbs.

Weight -- 2.5 lbs. per ft.

Gravity load --[28,000 + (1000 x 2.5)]0.4384	= 13,400
Car Friction -- 28,000 x 0.0270	= 757
Rope Friction -- (1000 x 2.5)0.0540	= 135
	14,292
Plus 10% acceleration stress	1,429
	15,721

Safety Factor $\frac{129,200}{15,721}$ = 8.2 rope is satisfactory.

Drum Size

For a 1 1/4" rope a 7 ft. diameter drum is recommended.[19]

Single layer winding is preferred if possible.

Number turns = $\frac{1000}{7\pi}$ = 45.5 turns + 3 (stays on drums) = 48.8

48.8(1 1/4 + 1 1/4) (Assuming the drum is grooved and each turn of rope is separated by 1/4")

= $\frac{73.2}{12}$ = 6.1 ft. size of drum face necessary

The nearest size is an 8 x 6 drum[20] which has a WR^2 of 760,000 ft.-lb.^{2}

Gear for 8 x 6 drum has a WR^2 of 368,000 ft.-lb.^{2}

Fleet Angle

How far should the drum be located from the sheave wheel to keep fleet angle below 2°?

$$\text{Distance} = \frac{3}{\tan 2^\circ} = 76'$$

Total Rope in Service

1000 + 76 + 3 turns on drum = 1152 ft.

Static Moments

Length	Dip	Sin Dip	Moment
Ascending 1000 ft.	26°	0.4384	28000 x 4 x .4384 = 49,200
Descending 1000 ft.	26°	0.4384	8000 x 4 x .4384 = 14,050

Frictional Moments

Rolling Friction

Angle of 2 1/2° is assumed

Coefficient = Cos 26° x tan 2 1/2° = 0.0391

Total load = 36,000 lbs. (up & down)

Moment = 36,000 x 0.0391 x 4 = 5630 lb.-ft.

Rope Friction

1000 x 2.5 = 2500 lbs. Weight of rope in slope

Assume 5 per cent for friction

Moment = 2500 x 0.05 x 4 x Cos 26° = 450 lb.-ft.

Total friction moment = 6080 lb.-ft.

Rope Moments

Graphical solution

At the beginning and end of the cycle there are 1152 x 2.5 = 2880 lbs. of rope suspended from drum. Moment = 2880 x 4 x Sin 26° = 5060 ft.-lb. See figure 17. Lay off ab equal to equivalent full Speed hoisting time (50 secs.). Through a and b and perpendicular to ab lay off the rope amount of 5060 ft.-lbs. ac and bd. Lay off ag = af = 1/2 accelerating period. Lay off bi = bh = 1/2 retarding period. Connect ab and bc. Through f and h lay off vertically the rope moments, fj = ac, hk = bd. Through g and i construct perpendiculars intersecting ad at l and o and bc at m and o. Connect jm, fl, ko, and hn. The area jmalf is the positive resultant torque and the area kozmh is the negative resultant torque.

Accelerating and Retarding Moments

1200 ft./min. $= \frac{1200}{8\pi} = 47.8$ R.P.M. referred to drums

$\frac{47.8 \times 2\pi}{60} = 5$ radians per sec.

Angular acceleration $= \frac{5 - 0}{t} = \frac{5.0}{20} = 0.25$ radians/sec.^{2}

Traveling masses consist of:

Load + 4 cars + 2 ropes = 41,760 lbs.

Moments of inertia

$\frac{41,760}{32.2} \times 4^2 = 20,700$ Traveling parts.

$\left(\frac{760,000}{32.2}\right)^2 + \frac{368,000}{32.2} = 58,600$ Drums & gear

Total 79,300

Accelerating and retarding moment = 79,300 x 0.25 = 19,800 ft.-lbs.

Each of the above moments are plotted against cycle time (Fig. 17) and the resultant torque curve plotted.

Motor Capacity

The torque at the cardinal points of the torque-time curve is converted into Horsepower and Motor Rating necessary calculated by the R.M.S. method.

$$H.P. = \frac{2\pi \times R.P.S.}{550} \times \text{moment}$$

$$= \frac{2\pi \times 0.8}{550} = 0.00914M$$

At A: HP = 0.00914 x 65,500 = 598

B: HP = 0.00914 x 63,500 = 580

C: HP = 0.00914 x 44,000 = 402

D: HP = 0.00914 x 38,000 = 347

E: HP = 0.00914 x 18,500 = 169

F: HP = 0.00914 x 16,500 = 151

The H.P. at each of the cardinal points is squared and the area of the H.P.^{2}-time curve is calculated.

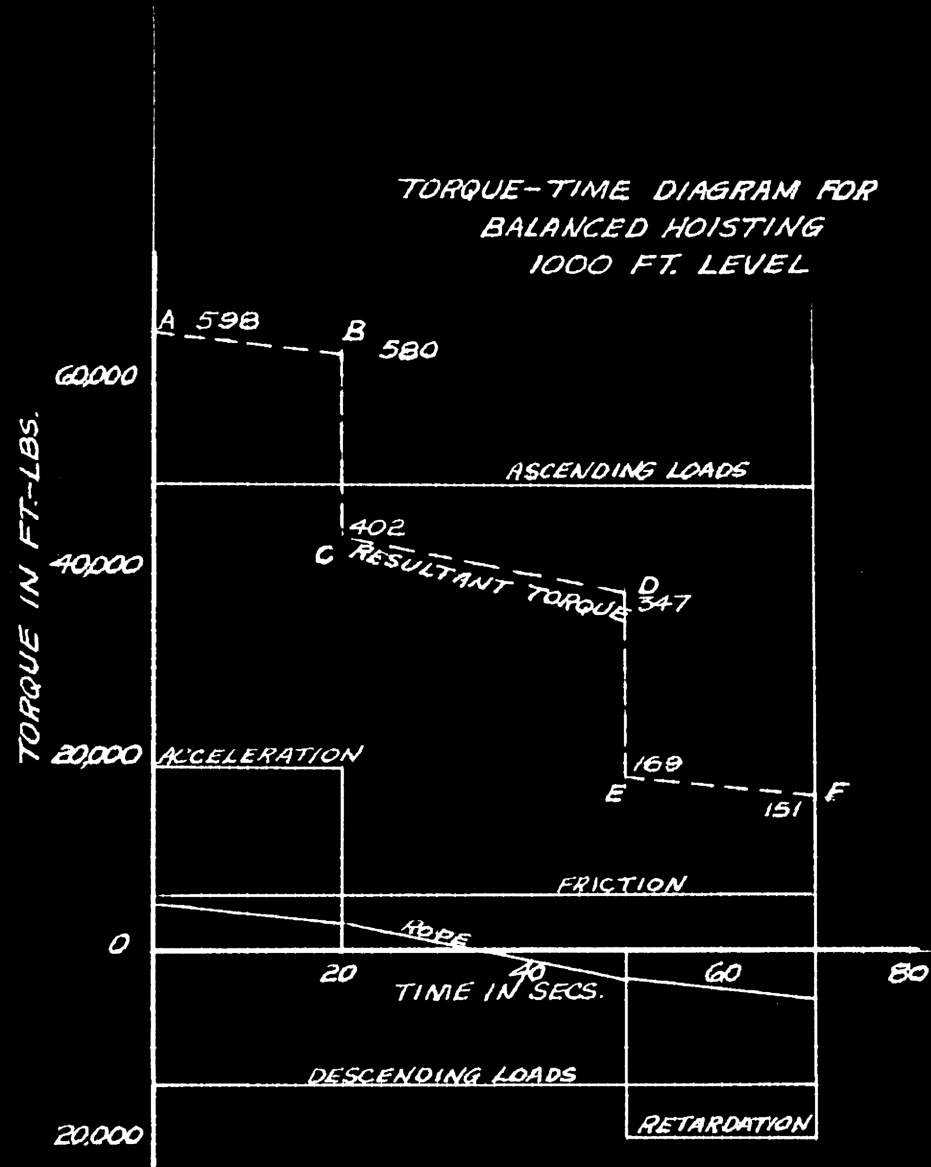
$$\text{Area 1} \quad \frac{\overline{598}^2 + \overline{580}^2}{2} \times 20 = 3,470,000 \text{ H.P.-secs.}$$

$$2 \quad \frac{\overline{402}^2 + \overline{347}^2}{2} \times 30 = 2,200,000 \text{ H.P.-secs.}$$

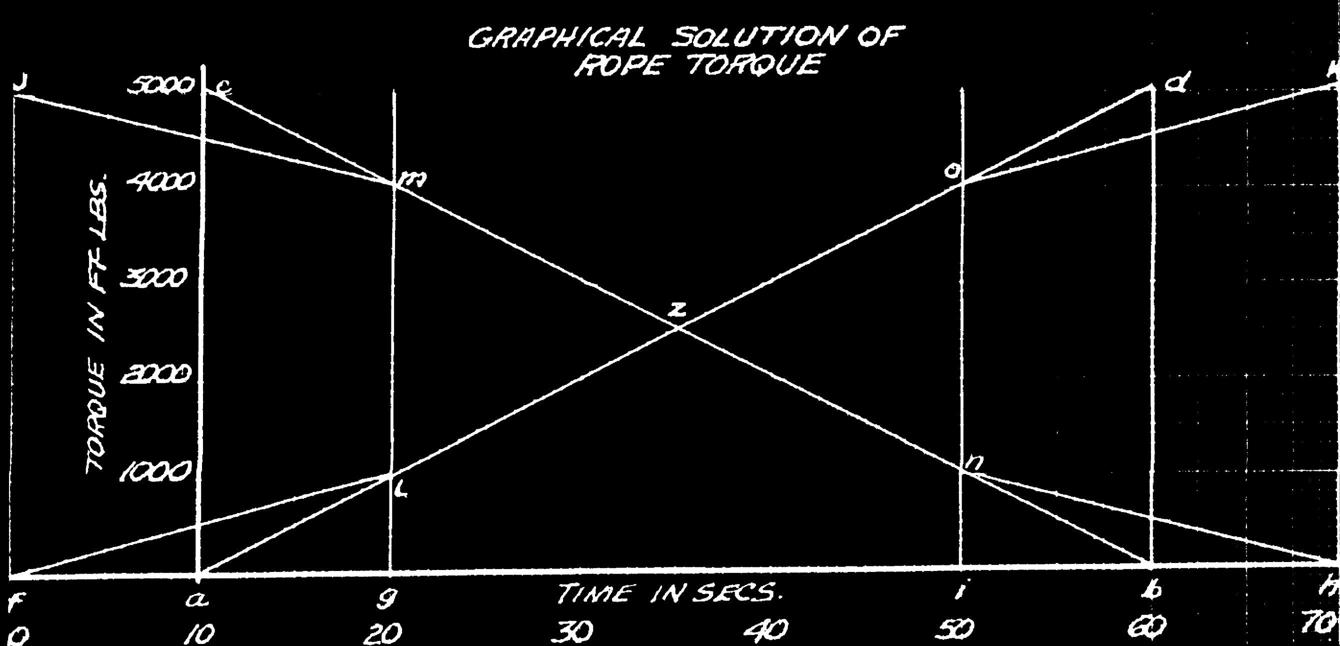
$$3 \quad \frac{\overline{169}^2 + \overline{151}^2}{2} \times 20 = 257,000 \text{ H.P.-secs.}$$

$$5,927,000$$

Equivalent time = 20(0.5) + 30 + 20(0.5) + 60 x (0.25) = 65 secs.



TORQUE-TIME DIAGRAM FOR
BALANCED HOISTING
1000 FT. LEVEL
A 598
B
580
60,000
ASCENDING LOADS
402
C RESULTANT TORQUE
D
347
40,000
TORQUE IN FT.-LBS.
20,000 ACCELERATION
169
E
151
F
FRICTION
ROPE
0
DEAD TIME
20
40
60
80
TIME IN SECS.
DESCENDING LOADS
20,000
RETARDATION



GRAPHICAL SOLUTION OF
ROPE TORQUE
J
5000
c
d
K
4000
m
o
TORQUE IN FT-LBS.
3000
z
2000
1000
n
l
f
a
g
TIME IN SECS.
i
b
h
0
10
20
30
40
50
60
70

Figure 17

$$\text{R.M.S. H.P.} = \frac{5,927,000}{65} = 302 \text{ H.P.}$$

This rating does not consider the power necessary to accelerate the armature of the motor itself. If the induction motor had a pull-out torque of 2, it would just be sufficient to handle the peak of 598 H.P. A 350 H.P. motor is recommended for this duty cycle.

4. Balanced Hoisting — 2000 ft. level

Data

Load	40,000 lbs. maximum
Cars	Weight 4,000 lbs., capacity 10,000 lbs.
Slope	1500' at 26°, 1500' at 18°, 1500' at 10°
Output	200 tons per hour for eight hours
Maximum Speed	1200 ft. a minute or 20 ft. per second
Acceleration	1 ft./sec.^{2}
Safety Factor	7-10
Dead Time	60 seconds

Subdivision of Cycle

$t = \frac{v_1 - v_0}{a} = \frac{20 - 0}{1} = 20$ secs. for acceleration and retardation

$d = 1/2at^2 = 1/2 \times 1 \times 20^2 = 200$ ft. traveled during acceleration and retardation

$2000 - (200 + 200) = 1600$ ft. full speed distance

$\frac{1600}{20} = 80$ secs. time for full speed

Acceleration	20 secs.
Full Speed	80 secs.
Retardation	20 secs.
Dead Time	60 secs.
	180 secs. time per trip

$\frac{3600}{180} = 20$ trips per hour

$\frac{200}{20} = 10$ tons per trip

Use 2-5 ton mine cars making a total load of 28,000 lbs.

Rope Size

Since the load is the same as the 1000 ft. level, the 1 1/4" 6 x 19 rope should be adequate. (Stress is greatest in the 26° portion.)

Gravity load — [28,000 + (1500 x 2.5)] 0.4384	= 13,900
Car Friction — 28,000 x 0.0270	= 757
Rope Friction — 1500 x 2.5 x 0.0540	= 203
	14,860
Plus 10% acceleration stress	1,486
	16,346

Safety Factor $\frac{129,200}{16,346} = 7.9$ rope is satisfactory.

Drum Size

Using the drum selected for the 1000 ft. level, 8' x 6', there would be:

$\frac{2000}{8} = 77.6 + 3 = 80.6$ turns on drum

$80.6 \times (1\ 1/4 + 1/4) = \frac{120.9}{12} = 11$ ft. face on drum which

means that two layers must be wound on drum.

WR^2 of 8' x 6' drum = 760,000 ft.-lb.^{2}

WR^2 of gear for 8' x 6' drum = 368,000 ft.-lb.^{2}

Total Rope in Service

$2000 + 76 + (8\pi \times 3) = 2152$ ft.

Calculation in Moments

The same method was followed in calculating the moments from the 2000 ft. level that was used in calculating the moments from the 1000 ft. level. The torque time diagram for this level is Figure 18.

R.M.S. value at 2000 ft. level = 349

A 375 H.P. hoist is recommended for duty at this level.

5. <u>Balanced Hoisting</u> -- <u>3000 ft. level</u>

Data

Load	40,000 lbs. maximum
Cars	Weight 4,000 lbs., capacity 10,000 lbs.
Slope	1500' at 26°, 1500' at 18°, 1500' at 10°
Output	200 tons per hour for eight hours
Maximum Speed	1200 ft. a minute or 20 ft. per second
Acceleration	1 ft./ sec.^{2}
Safety Factor	7-10
Dead Time	60 seconds

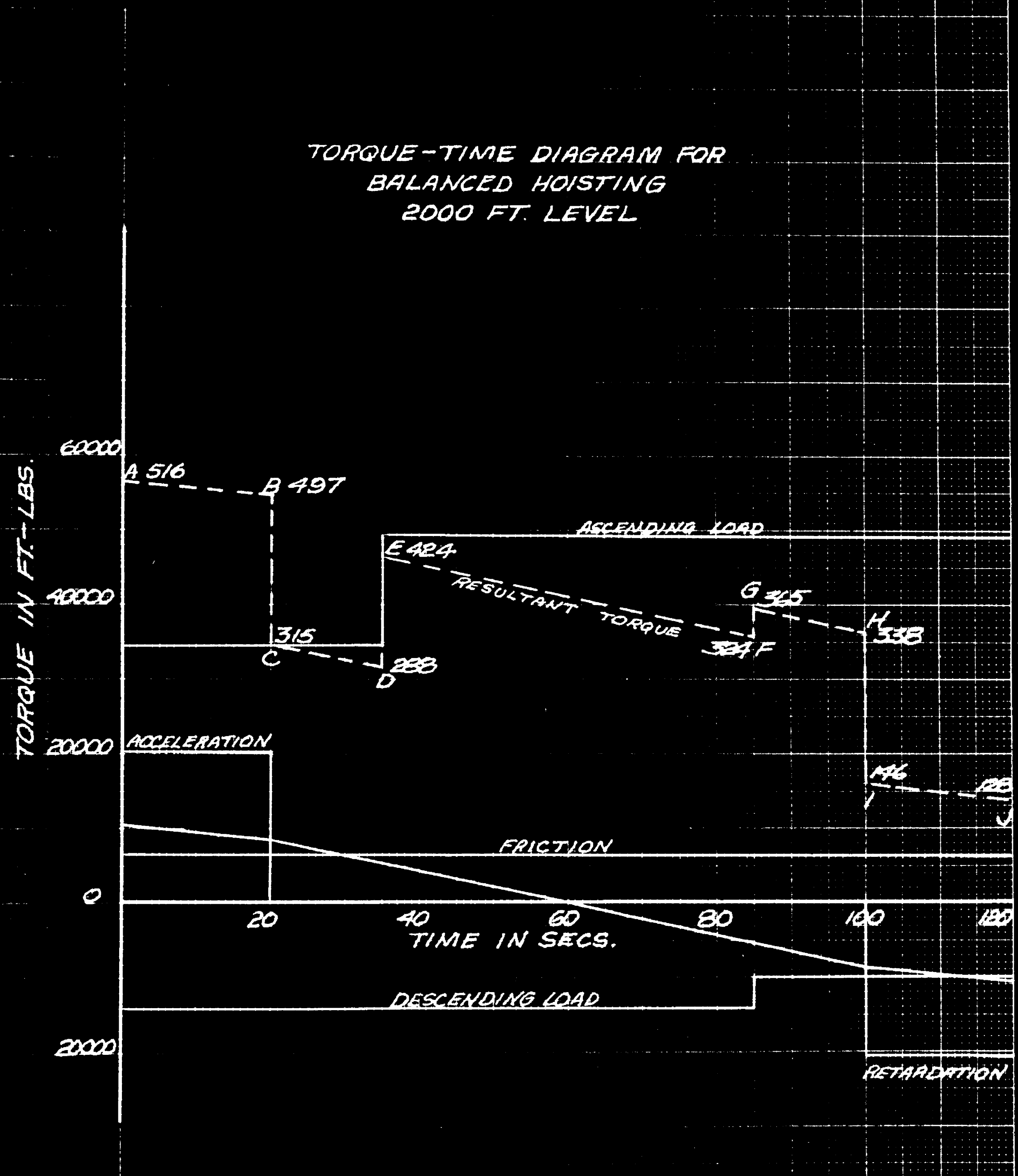

TORQUE-TIME DIAGRAM FOR
BALANCED HOISTING
2000 FT. LEVEL
TORQUE IN FT.-LBS.
60000
40000
20000
0
20000
A 516
B 497
C 315
D 288
E 424
G 365
F 324
H 338
146
128
J
ASCENDING LOAD
RESULTANT TORQUE
ACCELERATION
FRICTION
20
40
60
80
100
120
TIME IN SECS.
DESCENDING LOAD
RETARDATION

Figure 18

Subdivision of Cycle

$t = \frac{v_1 - v_o}{a} = \frac{20 - 0}{1} =$ 20 secs. for acceleration and retardation

$d = 1/2at^2 = 1/2 \times 1 \times 20^2 =$ 200 ft. traveled during acceleration and retardation

3000 - (200 + 200) = 2600 ft. full speed distance

$\frac{2600}{20} =$ 130 secs. time for full speed

Acceleration	20 secs.
Full Speed	130 secs.
Retardation	20 secs.
Dead Time	60 secs.
	230 secs. time per trip

$\frac{3600}{230} =$ 15 trips per hour

$\frac{200}{15} =$ 13.4 tons per trip

Use 3-5 ton mine cars making a total load of 42,000 lbs.

Rope Size

Try 1 1/2" 6 x 19 monitor steel rope, weight 3.6 lbs.-ft.

Breaking strength — 184,000 lbs.

Gravity load — [42,000 + (1500 x 3.6)]0.4384	= 20,800
Car Friction — 42,000 x 0.0270	= 1,135
Rope Friction — (1500 x 3.6) 0.0540	= 292
	22,227
Plus 10% acceleration stress	2,223
	24,450

Safety Factor = $\frac{184,000}{24,450}$ = 7.5 rope is satisfactory.

Drum Size

Use 8' x 6' drum with multiple layer winding.

Number turns = $\frac{3000}{8\pi}$ = 118 + 3 = 121 turns

121 x(1 1/2 + 1/4) = $\frac{211.75}{12}$ = 17.6 ft. making three layer winding necessary

WR^2 of drum = 760,000 ft.-lb.^{2}

WR^2 of gear for drum = 368,000 ft.-lb.^{2}

Total Rope in Service

3000 + 76 + (8π x 3) = 3152 ft.

Calculation in Moments

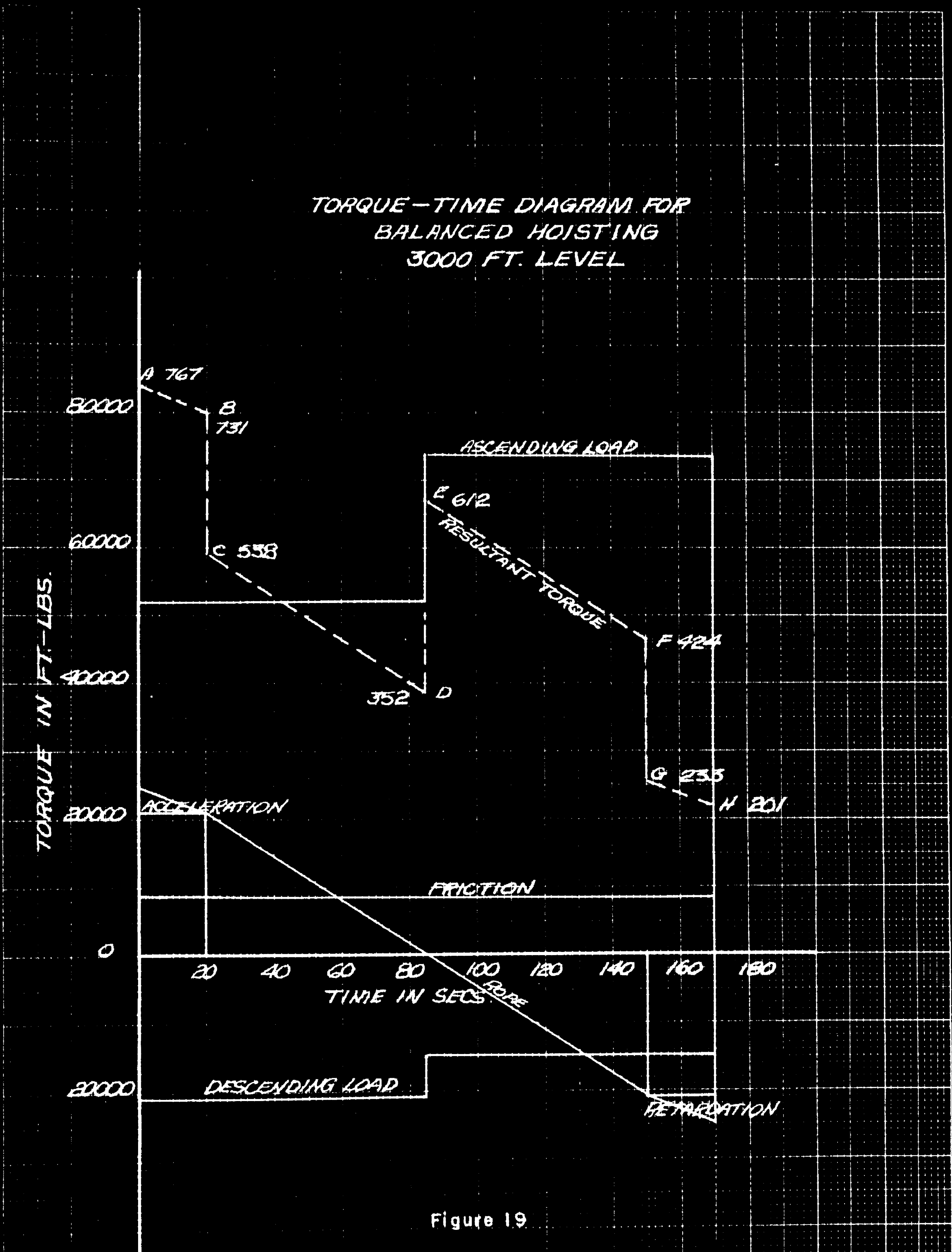
Moments were again calculated and torques plotted to time base for 3000 ft. level (Figure 19).

R.M.S. H.P. = 515

6. <u>Balanced Hoisting — 4500 ft. level</u>

Data

Load	40,000 lbs. maximum
Cars	Weight 4,000 lbs., capacity 10,000 lbs.
Slope	1500' at 26°, 1500' at 18°, 1500' at 10°
Output	200 tons per hour for eight hours
Maximum Speed	1200 ft. a minute or 20 ft. per second
Acceleration	1 ft./sec.^{2}
Safety Factor	7-10
Dead Time	60 seconds



TORQUE—TIME DIAGRAM FOR
BALANCED HOISTING
3000 FT. LEVEL
TORQUE IN FT.-LBS.
80000
60000
40000
20000
0
20000
20
40
60
80
100
120
140
160
180
TIME IN SECS.
A 767
B
731
C 558
352
D
E 612
F 424
G 235
H 201
ASCENDING LOAD
RESULTANT TORQUE
ACCELERATION
FRICTION
ROPE
DESCENDING LOAD
RETARDATION

Figure 19

Subdivision of Cycle

$t = \frac{v_1 - v_o}{a} = \frac{20 - 0}{1} = 20$ secs. for acceleration and retardation

$d = 1/2at^2 = 1/2 \times 1 \times 20 = 200$ ft. distance traveled during acceleration and retardation

$4500 - (200 + 200) = 4100$ ft. full speed distance

$\frac{4100}{20} = 205$ secs. time for full speed

Acceleration	20 secs.
Full Speed	205 secs.
Retardation	20 secs.
Dead Time	60 secs.
	305 secs. time per trip

$\frac{3600}{305} = 11$ trips per hour

$\frac{200}{11} = 18.2$ tons per trip

Use 4-5 ton cars making a total load of 56,000 lbs.

Rope Size

Try 1 3/4" 6 x 19 monitor rope, weight 4.9 lbs.-ft.

Breaking strength — 248,000 lbs.

Gravity Load — $[56,000 + (1500 \times 4.9)]0.4384$	= 27,800
Car Friction — $56,000 \times 0.0270$	= 1,510
Rope Friction — $1500 \times 4.9 \times 0.0540$	= 397
	29,787
Plus 10% acceleration stress	2,979
	32,766

Safety Factor $= \frac{248,000}{32,766} = 7.6$ rope is satisfactory.

Drum Size

Since the rope size has increased to 1 3/4" a 10 ft. diameter drum is necessary. The nearest commercial size is a 10' x 6' drum.

$WR^2 = 1,574,000$ and WR^2 of gear $= 660,000$.

$\frac{4500}{10\pi} = 143 + 3 = 146$ turns

$146 \times (1\ 3/4 + 1/4) = \frac{292}{12} = 24\ 1/3$ft. necessary for drum face

A 6 ft. face is sufficient with multiple winding.

Calculation in Moments

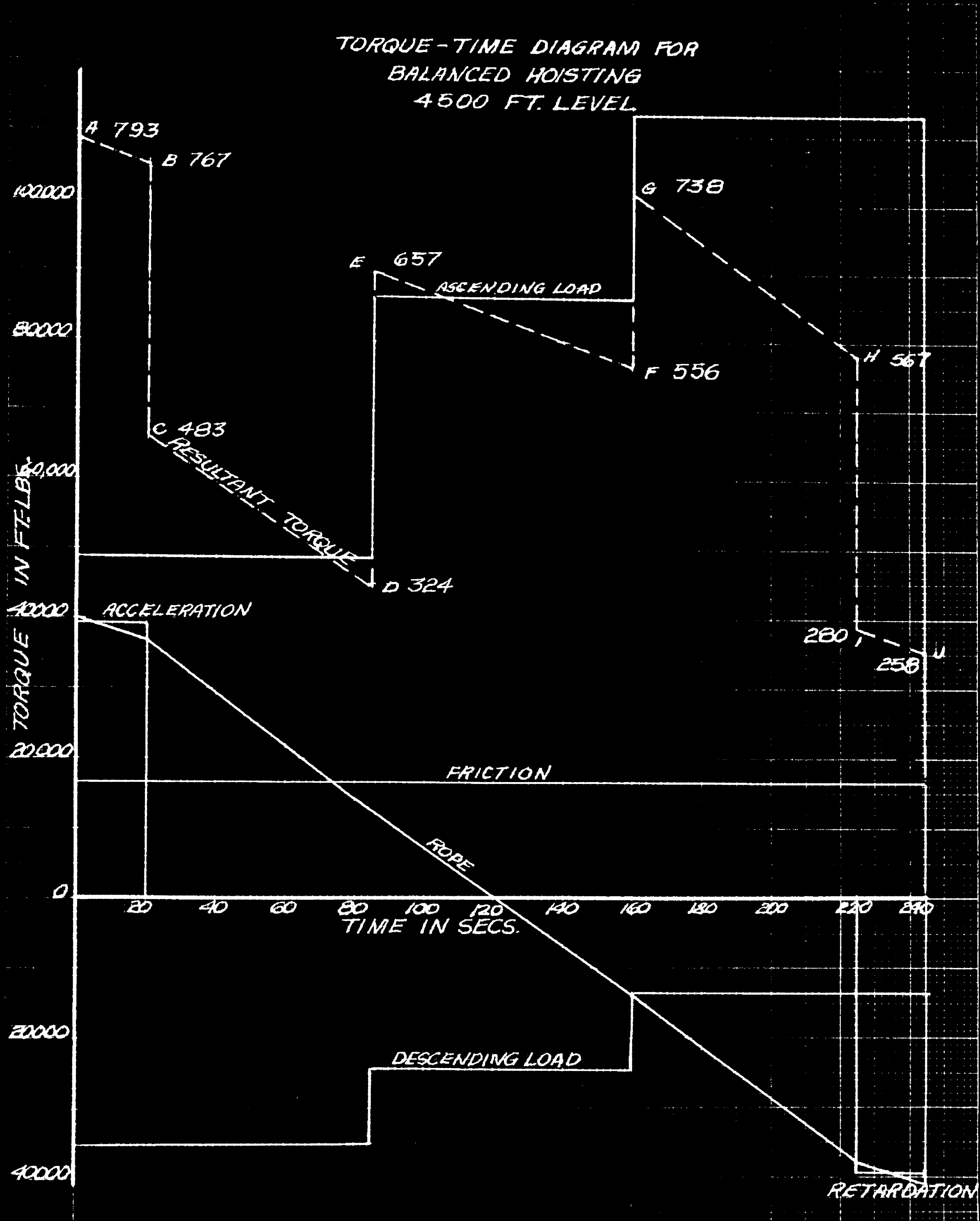
Moments were calculated at each cardinal point of the cycle and plotted against time, Figure 20.

Calculated R.M.S. H.P. = 583

Recommended H.P. = 600

7. Conveyor Belt System

The other alternative in bringing the coal to the surface is the use of belt conveyors. With the estimated profile it is necessary for the first 1500 ft. down the pitch to turn the development slope at such an angle to the strike that the limiting angle of 18° for belt conveyor operation will not be exceeded. This angle is calculated to be horizontal angle of 48°14', measured from the direction of the dip. To get the same possible degree of development with a 1500 ft. hoist, a belt conveyor 2126 ft. long is necessary. If the estimated profile is correct, after the first 1500 ft. the conveyors could be projected directly down the slope without exceeding 18 degrees inclination.



TORQUE-TIME DIAGRAM FOR
BALANCED HOISTING
4500 FT. LEVEL
A 793
B 767
C 483
D 324
E 657
F 556
G 738
H 567
280 I
258
J
ASCENDING LOAD
RESULTANT TORQUE
ACCELERATION
FRICTION
ROPE
DESCENDING LOAD
RETARDATION
TORQUE IN FT-LBS.
100,000
80,000
60,000
40,000
20,000
0
20,000
40,000
20
40
60
80
100
120
140
160
180
200
220
240
TIME IN SECS.

Figure 20

The following table, figure 21, shows the calculated widths, capacities, required horsepower ratings, and maximum allowable horsepower ratings for the necessary conveyor system to convey the required amount of coal from the mine. Calculations were based on roller bearings with a 180° simple drive.[7] It is readily apparent that a coninuous belt in any part of the 18° portion of the slope is impossible. The maximum slope distance with one conveyor drive is 213 ft. with a 30-inch, 5 ply, 28 oz. belt and 250 ft. with a 36-inch, 6 ply, 32 oz. belt. To transport the coal from the 3000 ft. level by belt conveyors would require 15 conveyor drive installations using the 36-inch, 6 ply belt. The remaining 1500 ft. on the 10° portion of the slope would require five conveyor drive installations using the 30-inch, 5 ply belt. This would make a total of 20 conveyor drives having an aggregate horsepower of 432.

8. Belt Conveyor Versus Track Hoisting

Since the design of the hoisting equipment will, in all probability, be based on the ultimate conditions, (hoisting from 4500 ft. on the slope) a comparison will be made with the belt system from that level.

In the first place if a belt system is chosen, it would not eliminate the need for a hoist since some provision must be made for handling men, equipment, and supplies. The supplementary hoist naturally would not have the same duty to perform that a hoist designed for the ultimate conditions, but it would have to be large enough to handle the heavier machinery; i.e., cutting machines which weigh approximately 8000 lbs.

A hoist designed for the ultimate conditions (200 tons per hour from the 4500-ft. level) would require a 600 horsepower motor and would cost approximately $125,000.00 installed.[2] This figure includes all the necessary

18 Degree Slope

Slope Distance	Vertical Lift	Width and Ply of Belts	Weight of Duck	Capacity in Tons/Hr.	Speed f.p.m.	Required Horsepower	Max. H.P. of Belt (Allowable)
2126	658	30"-5	28	225	300	197.0	20.4
1063	329	30"-5	28	225	300	98.5	20.4
213	66	30"-5	28	225	300	19.8	20.4
1000	309	36"-6	32	216	200	90.5	22.0
400	124	36"-6	32	216	200	36.3	22.0
250	78	36"-6	32	216	200	22.8	22.0

10 Degree Slope

1000	174	30"-5	28	225	300	62.3	20.4
300	53	30"-5	28	225	300	18.9	20.4
400	70	36"-6	32	216	200	24.6	22.0
350	61	36"-6	32	216	200	17.5	22.0

Figure 21. Belt Conveyor Data for Operation on Slope

controls and two drums for balanced hoisting. If the current estimate figure of \$30.00 per ft. for belt conveyors installed is taken as a basis for estimating the cost, it would require \$30.00 x 5126 or approximately \$154,000 to transport the coal to the surface from the 4500-ft. level. To the cost of the conveyor system must be added the cost of the supply hoist. From the initial cost standpoint alone the hoist is preferred.

The difference in power consumed between the two systems would be very little since a total of 432 horsepower is required for the belt system against the 600 horsepower hoist. Naturally, peak loads would be eliminated in the belt system, but the power consumed by the supplemental hoist would be charged against the belt system.

Due to the fact that 20 separate conveyor drives are necessary, it is felt that the hoist would be more reliable than the belt system. An elaborate system of controls would have to be maintained in order to stop the entire belt system in the event of trouble at any section. Since the belt system is a continuous system, a breakdown at any point would halt production everywhere since no storage facilities would be available. In the hoisting system a breakdown of the hoist would not instantly stop production since some storage facilities would be available in the form of mine cars.

To transport the coal to the surface a balanced system of hoisting is recommended.

B. Degree of Mechanization

Taking the estimated profile of the coal seam as: first 1500 ft. 26°, second 1500 ft. 18°, and third 1500 ft. 10°, it is apparent that the use of mobile loading equipment is limited to that area of the mine where the seam does not pitch more than 18°. This is so for two reasons: first, all mobile loaders have a conveyor to transport the coal from the loading head to the receiver (if the angle of dip exceeds the angle of repose (18°), the coal will roll from the conveyor); second, in order to gain the full advantage from mobile loaders they must be in operation as much as possible during the shift which means that they must be trammed from face to face as the faces are prepared for loading. Manufacturers do not recommend the operation of loading machines on such pitches. It is true that it would be possible to have all the entries along the strike at any desired grade, but the slope of the entry faces from rib to rib would still be the seam pitch. Travel from level to level would still present the same tramming difficulties. The use then of mobile loaders is not recommended in that part of the seam where the dip exceeds 18°. It would be more desirable to limit their operation to that portion below 10°-12° which would mean below 2500 ft. from the outcrop.

The operation of cutting machines from the standpoint of the seam pitch alone again limits the amount that the machines can be moved around. The actual cutting of the coal face on the pitch presents no great difficulties. After the cutting machine is once in place, the cutting operation could be completed — either up or down the pitch. It is recommended that wherever possible the mine should be laid out in order to have the machine cut up or down the slope — sumping in along the strike — rather than sumping in up the pitch which would be very difficult on the steeper slopes. One of the greatest difficulties in cutting the coal would be the choice of the band to cut in. Several bands would be suitable, but they change their horizon in the seam from place to place. The type of cutting machine recommended for use is the short-wall type, not to be moved from the particular working face it serves and one which the cutter bar is adjustable to cut at different horizons. The manufactured type best suited for the conditions appears to be the Goodman 712. Since the coal seam contains the hard slate bands, it is felt that deep cuts by cutting machines would cause difficulty in shooting down the face. A 6 1/2-foot cut is therefore recommended.

Conveyors can readily be used in driving entries and rooms along the strike. These can be the chain-flight, shaking, or belt type conveyor. The latter is seldom used as a transportation system all the way to the working face. Shaking or chain flight conveyors can be used in driving rooms up the pitch. Below 26° coal will slide with difficulty even when steel chutes are used so it would be advisable not to count on gravity flow below the 800-ft. level. Shaking conveyors with self-loading heads (duckbills) can be used on rooms driven up the pitch or in entries or rooms along the strike where the dip does not exceed 18°. It is not recommended that shaking conveyors with loading heads be used on strike entries, or rooms, since the loading heads would not conform to the slope of the floor when the pan line is level.

C. Slope Development

Driving the slope entries directly down the pitch affords little opportunity for mechanization in those sections where the dip is steepest. Using track and mine cars for haulage, there is no recourse but hand loading. Pit car loaders or elevating conveyors could be used to reduce the lift necessary to get the coal from the bottom up and into the mine car.

In order to keep the various sections of the mine separated or panelized from a ventilation standpoint a triple entry system is advised. This system will permit east and west sections of the mine to be on separate splits of air and also allow a greater number of passages for the return air.

Using a balanced system of hoisting necessitates a very wide entry in order to accommodate the double track down the slope. Using mine cars 6 ft. wide and 42-inch track gauge, an entry 23 ft. wide will be necessary to give the proper clearances. Referring to figure 22, a triple entry system is laid out -- the main entry being 23 ft. wide and the two aircourses 18 ft. wide on 40-ft. centers. Crosscuts are turned 45° from the main entry, 15 ft. wide and 80 ft. apart. Track is maintained in each aircourse through the last open crosscut. A small auxiliary hoist with 250 ft. of rope is used in the slope development to move the cutting machine from place to place. The main hoist could be used for this until the first level is reached. After that point the small hoist would take over in order to prevent interference with the main job of hoisting. By using the small hoist on slope development, one cutting machine can serve a minimum of four working faces on the slope.

As to clearances, referring again to figure 22, clearance between the right hand car and rib is 2 ft., between the left hand car and rib 4 ft. To support the top 8-inch round timbers are set between the tracks on 10-ft. centers. Clearance between the two cars and line of timbers is 2 ft. on each side.

Personnel required for the slope sinking operation are as follows:

1 cutting machine operator

1 cutting machine operator helper

6 loaders (2 main slope, 1 each aircourse, 1 each crosscut)

1 trackman

1 track helper

2 timbermen

2 bratticemen

1 pumpman.

Slope development will be single shifted with cutting, drilling and shooting on the off shift. Since the development of this property will extend laterally to a much greater degree than development down the dip, the rate of advance of the main slope need be only one-fourth that of the cross entries. Considering the difficult conditions surrounding the development of the slope entries, a rate of advance of 3-4 feet per day would be considered a normal performance. This would mean that the loaders would be required to clean up a cut every other day. If a daily task of one complete cut is set as standard, the entire slope would be driven in $\frac{4500}{6.5} = 700$ working days or less than four years. This rate is far above the required amount of development if the mine is to be in operation 20 to 30 years.

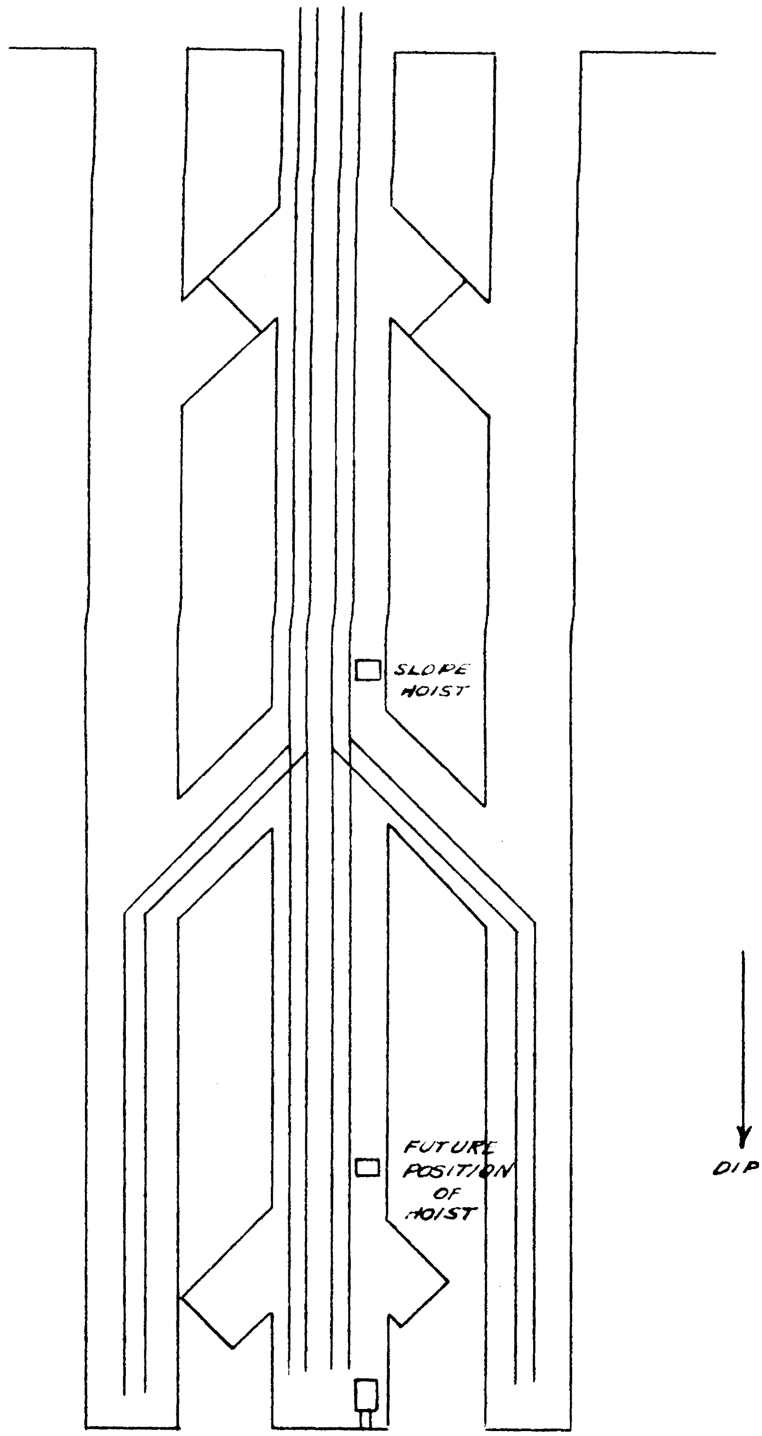


Figure 22. Method of Slope Development

Also, pumping against a high head would be necessary for a period of years when the slope would not be used. If it is found that the personnel listed above are in excess of the number required to maintain the necessary rate of advance, they can be shifted to other parts of the mine. It is true that until the first level is reached it would be advantageous to advance the slope as fast as possible in order to reach full production quickly.

In designing a slope bottom layout using a balanced system of hoisting, several factors are to be considered. First, sufficient storage space should be provided at each of the cross entries so that hoisting will not be interrupted; second, the layout should be flexible enough to permit cars returning from the surface to be spotted in either east or west cross entries; and third, the layout should be simple enough to be easily developed using the same system of development that is used on the main slope.

Referring to figure 23, double track in that portion of the cross entries adjacent to the main slope will provide storage for loads and also o for empties. Having one drum clutched in a balanced system of hoisting permits hoisting from more than one level. In addition, a diamond crossover at each level will insure the empty cars of being placed on the empty storage track where they are needed. The cross entries are turned off the main slope through the projected crosscuts making possible a smooth curve and an even grade for the track. The same auxiliary hoist is used in the initial stage of development of the cross entries that is used for the development of the main slope.

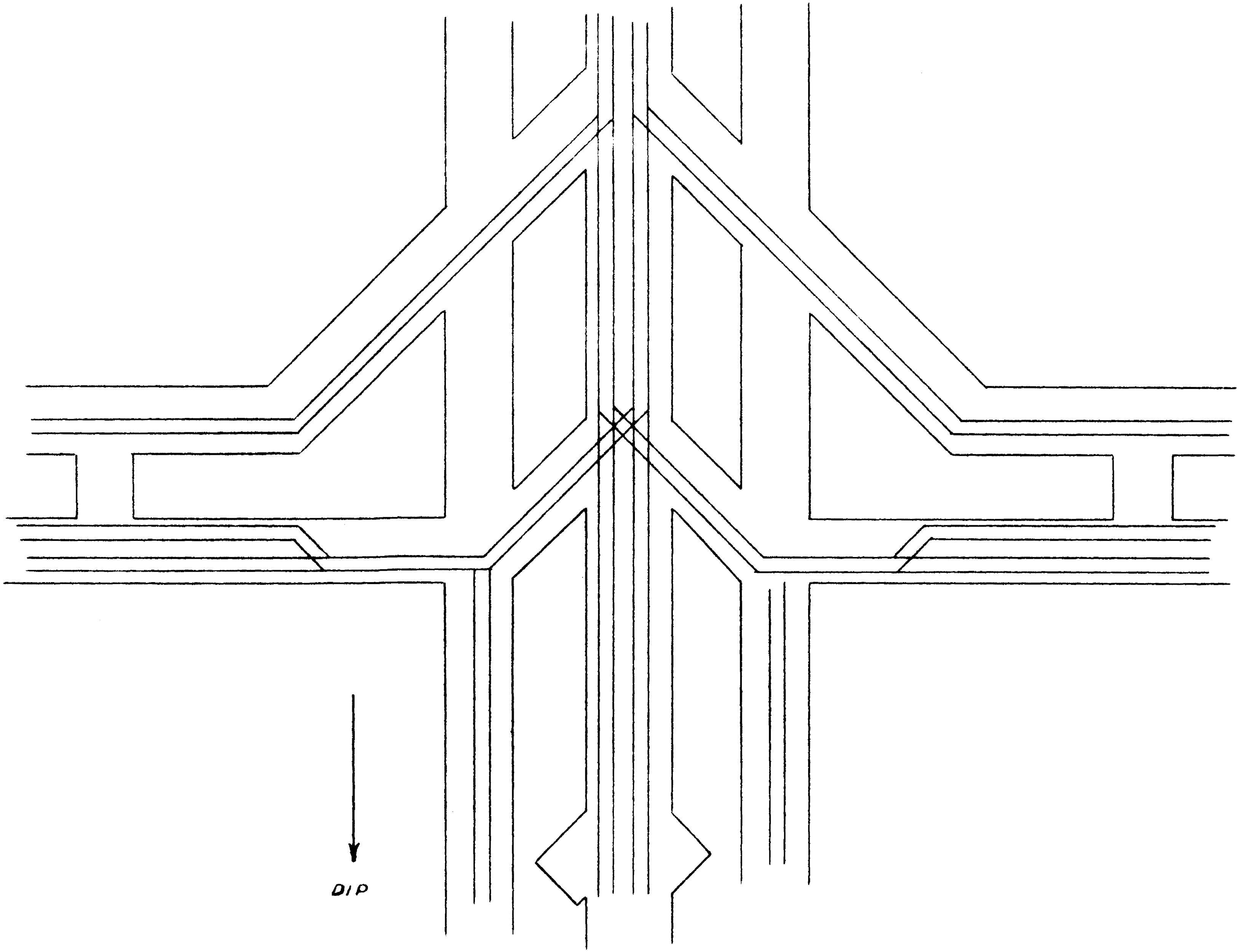


Figure 23. Slope Layout at each Level

D. Lateral Development

Lateral development of cross entries will be accomplished by a double entry system, entries 18 ft. wide on 40-ft. centers. It has been discussed under the section, Degree of Mechanization, that mechanization of entry driving along the strike can be accomplished to a great degree by the use of conveyors and cutting machines. Hand loading on shaking conveyors will be used in developing the cross entries in those sections of the mine where it is impossible to use mobile loaders. One cutting machine will remain in the entry and one in the aircourse.

Referring to figure 23, to avoid taking bottom, the middle line of timbers is lagged with 2-inch lagging to a height of three feet. With this height of lagging, filled in with the two hard slate partings in the seam, sufficient level space is obtained to install the track without taking bottom. Approximately 78 cubic feet of the slate is mined each cut, and approximately 60 cubic feet is required to make the fill.

To take advantage of the slope in loading, the conveyors will be installed on the down pitch side of the entries. Placing the conveyors on the down pitch side necessitates the use of an elevating conveyor to lift the coal into the mine car from the shaking conveyor. See section A-A, figure 23. Coal from the aircourse is chuted through the open crosscut into the mine cars.

Crosscuts are driven by taking two cuts from the entry and one cut in the same crosscut on the aircourse side. If rooms are turned up the pitch, crosscuts will be projected to correspond to room centers. If rooms are not driven up the pitch but along the strike, crosscuts every 300 ft. would be

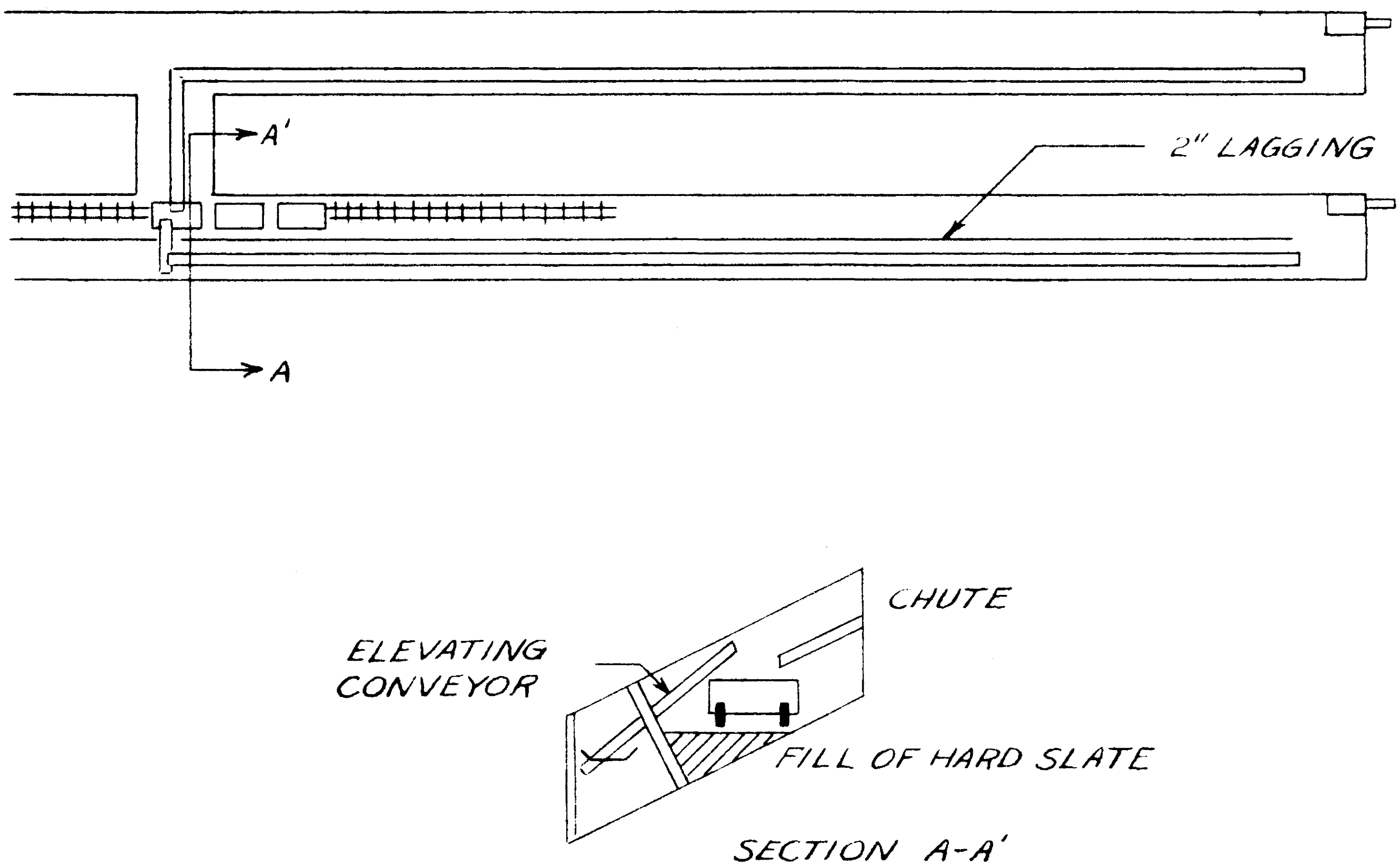


Figure 24. Method of Developing Cross Entries

sufficient. Ventilation will be provided at the face by booster fans in the last crosscut aided by a line brattice along the timber line.

Entries can be advanced 300 ft. with one set-up of the equipment.

Personnel required for developing each of the cross entries are as follows:

2 motormen

2 motorman helpers

1 car trimmer

5 loaders (3 on entry, 2 in aircourse)

1 trackman

1 track helper.

The three loaders on the entry will be required to do cutting, drilling and shooting, keeping the line of timbers up, advancing the pan line, in addition to loading the cut. The two loaders on the aircourse will not be required to do the loading, otherwise their duties will be the same. The car trimmer will operate and maintain the shaker drives in addition to spotting the cars under the loading head.

In entry driving in level coal with similar conveyor system, two to three cuts per shift is a normal performance for two men per face using an eight or nine foot cutter bar. This also includes the auxiliary tasks of cutting, drilling and shooting, timbering, etc. It is felt that a two cut average per shift could be maintained under the conditions of this mine, considering the fact that a short cutter bar will be used and also having the slope as an aid to loading.

Assuming an advance of two cuts per shift, it may become necessary to double shift the entry work to provide the proper balance between development and production. By keeping the entries well ahead of the production sections, it would not be necessary to delay room development when it became necessary to pierce the normal faults. Too, by keeping the entries well ahead, the position of the faults will be known; and if the rooms are projected along the strike, they can be made to fit the seam conditions. Entries will be projected based on an average of two cuts per shift or a 13 foot advance. If for any reason the development of the entries cannot keep ahead of the room production, they will be double shifted.

E. Production

To obtain the desired production of 1600 tons daily there must be

$\frac{wdt}{v} = 1600$ tons of coal mined, where,

w - width of working face in feet

d - depth of cut in feet

t - thickness of seam in feet

v - specific volume of the coal in the seam in cubic feet per ton.

The necessary width would be:

$$w = \frac{1600 \times v}{dt}$$

$$= \frac{1600 \times 23}{6.5 \times 6.83} = 830 \text{ feet daily must be cut.}$$

From slope development there would be:

23 feet - main slope

36 feet - 2 aircourses

<u>30 feet</u> - 2 crosscuts

89 feet

Since 4 ft. is the projected rate of advance of the main slope, there would be $\frac{4}{6.5} \times 89 = 55$ feet available from the slope.

From the development of the cross entries there would be 18' x 8 or 144 ft. of face cut daily since two cuts per shift per heading is expected.

From the room section it would be necessary to work

830 - (144 + 55) = 631 feet of face.

In choosing between the two general methods of mining; i.e., Room and Pillar, and the true Longwall, there are several factors which must be con sidered. First, since the depth of overburden increases as mining progresses down the pitch, tremendous roof pressures would soon be encountered; second, the Merrimac seam has a very strong roof and floor; third, generally the maximum allowable size of opening decreases with increasing pitch and thickness of seam;[17] and fourth, presence of explosive gas indicates the desirability of a panel system. From the known factors then, large unsupported openings should be avoided because of the tremenduous pressure at the depths to which the seam must be mined. Since none of the seam conditions are applicable to Longwall mining, a Room and Pillar system is preferred. In fact, practically all the pitch mining done in the United States is done by Room and Pillar methods.

Since practically no second mining has been done in this area, no attempt will be made to project a Room and Pillar system where the pillars will be extracted on second mining. By excluding the possibilities of pillar extraction, a better comparison can be had between the old and the projected methods.

The average room width in the area is approximately 30 ft. Room centers vary from 40-60 ft. Little roof control difficulties have been encountered with this combination.

One system of mining to fit the seam conditions on all degrees of pitch would be impossible if mechanization is to be practiced to the highest degree possible. Reviewing the capabilities of the various items of mechanical equipment, (see section Degree of Mechanization) between 18°-26° cutting machines can be used if they cut up or down the slope sumping in along the strike; between 18°-26° conveyors can be used if laid out parallel with the strike but loading heads on shaking conveyors cannot; between 12°-18° cutting machines can be used to cut either up the pitch or across the pitch, loading heads on shaking conveyors would still not be advisable loading along the strike but could be used up the pitch; below 12° mobile mechanical equipment can be used provided the layout along the strike is at such an angle to reduce the slope below 10° for tramming. The seam conditions then make it necessary to divide the property into three divisions; i.e., 18°-26° section, 12°-18° section, and below 12° section. An extraction method will be designed for each of the divisions keeping in mind the equipment that can be used and also with the idea of keeping the number of types of equipment at a minimum throughout the life span of the mine.

In the first system (18°-26° section) two 20-foot rooms on 40-ft. centers will be turned directly up the pitch every 300 ft. along the cross entry from the main slope barrier pillar. Off these pitch rooms a panel of rooms 30 ft. wide on 50-ft. centers will be turned and driven 300 ft. until they cut into the next pair of pitch rooms. See figure 25. This development will progress simultaneously on the east and west levels. Hand loading on shaking conveyors will be used in this section of the mine. Cutting machines will be used in each room and will remain in the same room until it is driven up.

From the pitch rooms there would be 160 feet of face cut daily which would leave 631'-160' or 470 ft. for the rooms. If the rooms produce two cuts per shift, there would be necessary $\frac{470}{60}$ or 8 rooms for the required productions. These could be projected four on each side of the main slope and developed together. This would require the pairs of pitch rooms to be 215 ft. long up the pitch.

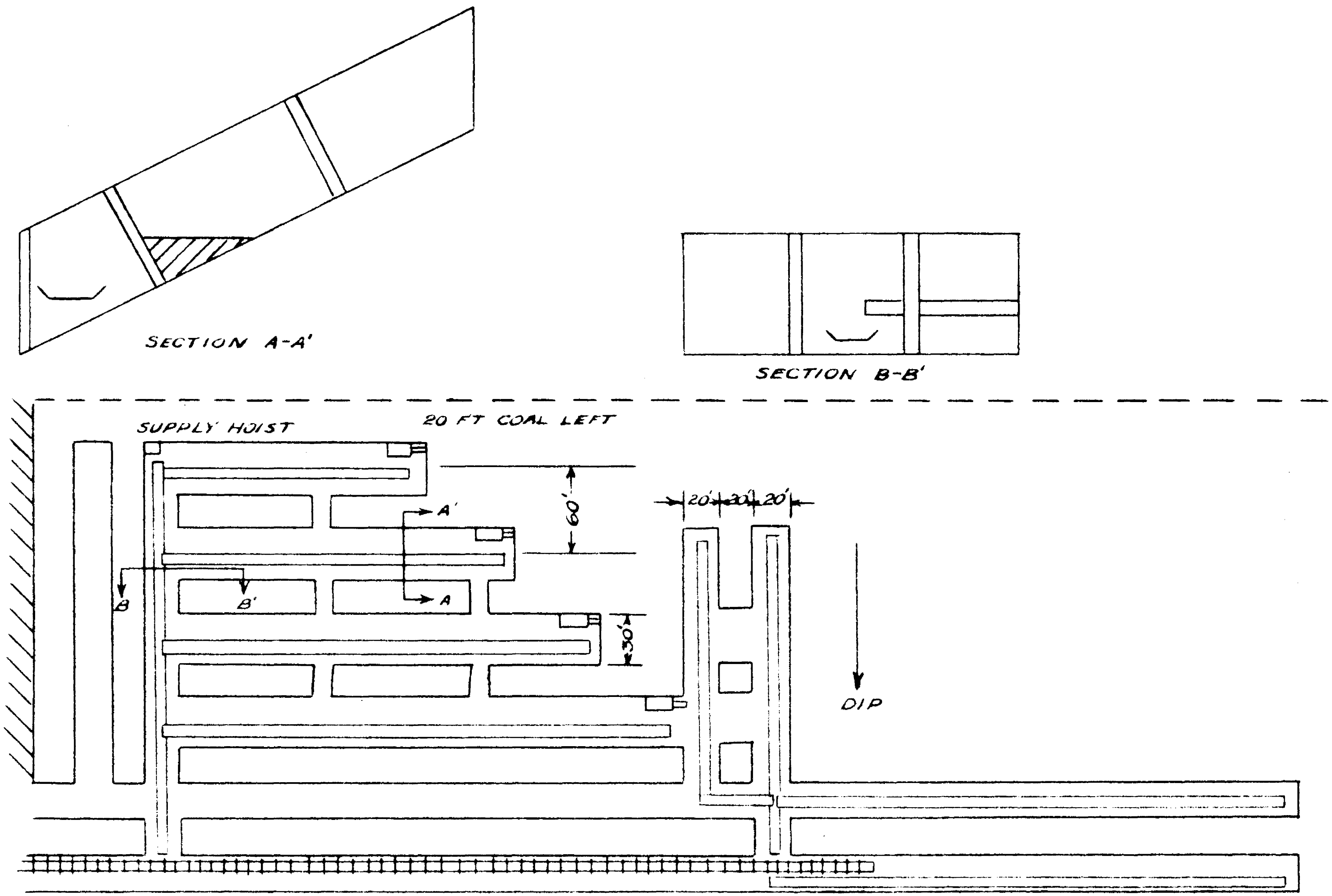


Figure 25. Method of Mining where Coal Seam dips 26-18 Degrees

Figure 25 represents a stage in the development where the first room of the first panel has just cut through to panel No. 2. The cross entry having been developed far enough to have the next pair of pitch rooms partially developed, the pan drive of room 1 of panel 1 can be moved straight across through the crosscut into the room neck of room 1 panel 2. Also cutting machines can begin cutting in room 1 panel 2 with a minimum of movement.

The pan line is placed next to the rib on the down pitch side of the room to facilitate loading. See section A-A', figure 25. The upper side of the timber set supporting the pan line is lagged with boards to provide a space to dispose of the heavy slate bands and also to provide a small level supply road.

Timbers, equipment, pans, etc., will be supplied to the rooms by a small hoist in the pitch room. Supplies will be transferred to the room necks where they will be pushed to the face on small rubber-tired supply trucks.

Personnel required for this method per panel:

16 loaders

2 supply men

1 car trimmer

1 shot firer

1 foreman.

Two cuts per shift would easily be possible with this layout considering the favorable slope for loading, the supplies near at hand, and the shortness of the cut.

The second method of extraction is designed for that section of the mine where the coal dips between 12° and 18°. In this method the rooms are projected up the pitch since no difficulty is expected in cutting up this degree of pitch with short wall cutting machines.

The shaking conveyors used in the steeper portions of the mine are now equipped with loading heads (duckbills).

The same system will be used in developing the cross entries that was used in the previous cross entries with the exception of the position of the crosscuts. The crosscuts are now projected to fit the room neck of every fifth room.

With the loading heads on the conveyors an additional cut per shift is expected. Instead of the 6 rooms in the section above (2 pitch rooms, 4 strike rooms) four rooms each on the east and west cross entries will give the required production. No production is calculated from the main slope since the rate of advance assumed for it is sufficient to have it fully developed.

Figure 26 shows four rooms being driven simultaneously — all discharging at the same loading point on the entry.

The rate of advance of the cross entries will have to be increased to keep up with the rapid development of the rooms. Double shifting the entries will be necessary in this section of the mine.

Personnel required per panel of four rooms:

12 loaders

2 supply men

1 car trimmer

1 shot firer

1 foreman.

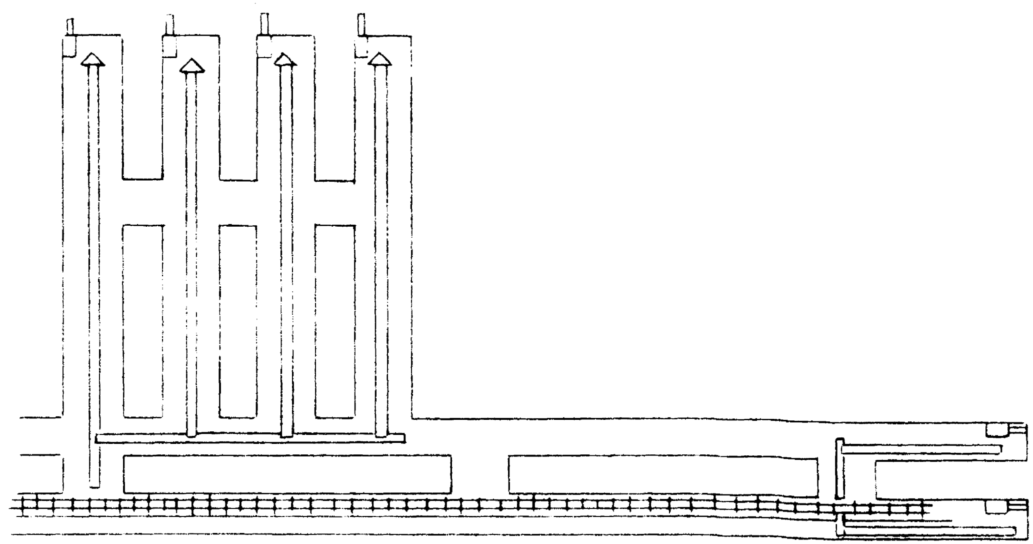


Figure 26. Method of Mining where Coal Seam Dips 18-12 Degrees

The third extraction system is based on the use of mobile loaders and cutters. Below 2500 ft. from the outcrop it is assumed that the pitch of the seam will be below 12°. By projecting the working faces at an angle of 45° to the strike a slope of 8 1/2° would be encountered which would present no great difficulties to the operation of mobile machinery.

Since 1600 tons per day is a relatively small output for mobile loaders, the working faces will be concentrated in as small an area as possible.

Referring to figure 26, with the rooms projected as shown, there will be 15 working places with the six rooms, including the crosscuts. With a similar set-up on the opposite cross entry, the required production can be obtained.

Plan of operation is shuttle car haulage discharging into mine cars on the cross entry. The operations of cutting, drilling and shooting, loading and timbering should be completed once each shift for each working face.

Joy 11-BU crawler type loaders with a rated capacity of four tons per minute are selected. Sullivan 10-RU universal cutters will be used to cut the coal.

Personnel required for this method per panel of six rooms:

1 loading machine operator

1 loading machine operator helper

1 cutting machine operator

1 cutting machine operator helper

1 drillman

1 drill helper

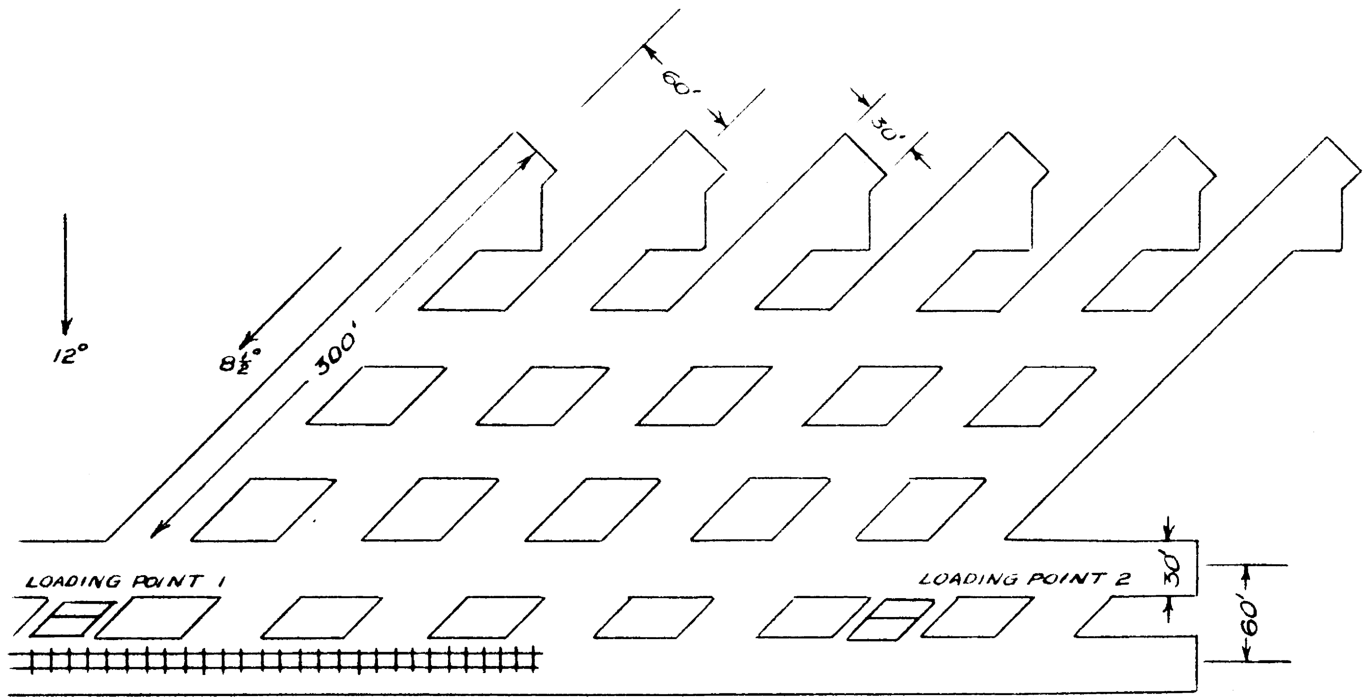


Figure 27. Method of Mining where Dip of Coal Seam is below 12 Degrees

1 trackman

1 track helper

2 shuttle car operators

2 motormen

2 brakemen

1 car trimmer

1 timberman

1 timber helper

1 foreman.

The mine is projected (Fig. 28) based on the three methods previously outlined. The yearly limits of extraction are plotted on the projection based on the estimated rates of advance given in the detailed layout of each method. In plotting the yearly limits of extraction 224 working days per year is taken as an average. In those sections employing conveyor mining, an allowance is made for the production days lost due to advancing equipment. Three man days delay is estimated to be the production lost. No delay in production is assumed for difficulties encountered in passing through the fault sones because the rate of advance of the cross entries is far enough ahead of the room sections to allow sufficient time to pierce the faults.

Beginning when the slope sinking operation starts, the first year's extraction will net approximately 133,000 tons which is below normal production. After the first year of underground operations the expected yearly tonnage of 358,800 tons before cleaning should be realized.

Referring to figure 28, nine years will complete operations on the steeper part of the seam; five years will complete operations on the intermediate slope; and ten years will complete the mine to a depth of 4500 ft. —

making a total life of the mine twenty-four years to the 4500-ft. level.

With the rate of advance estimated for the slope sinking operations, it is apparent from the projection that the slope will be completed to a depth of 4500 ft. in six years of continuous working.

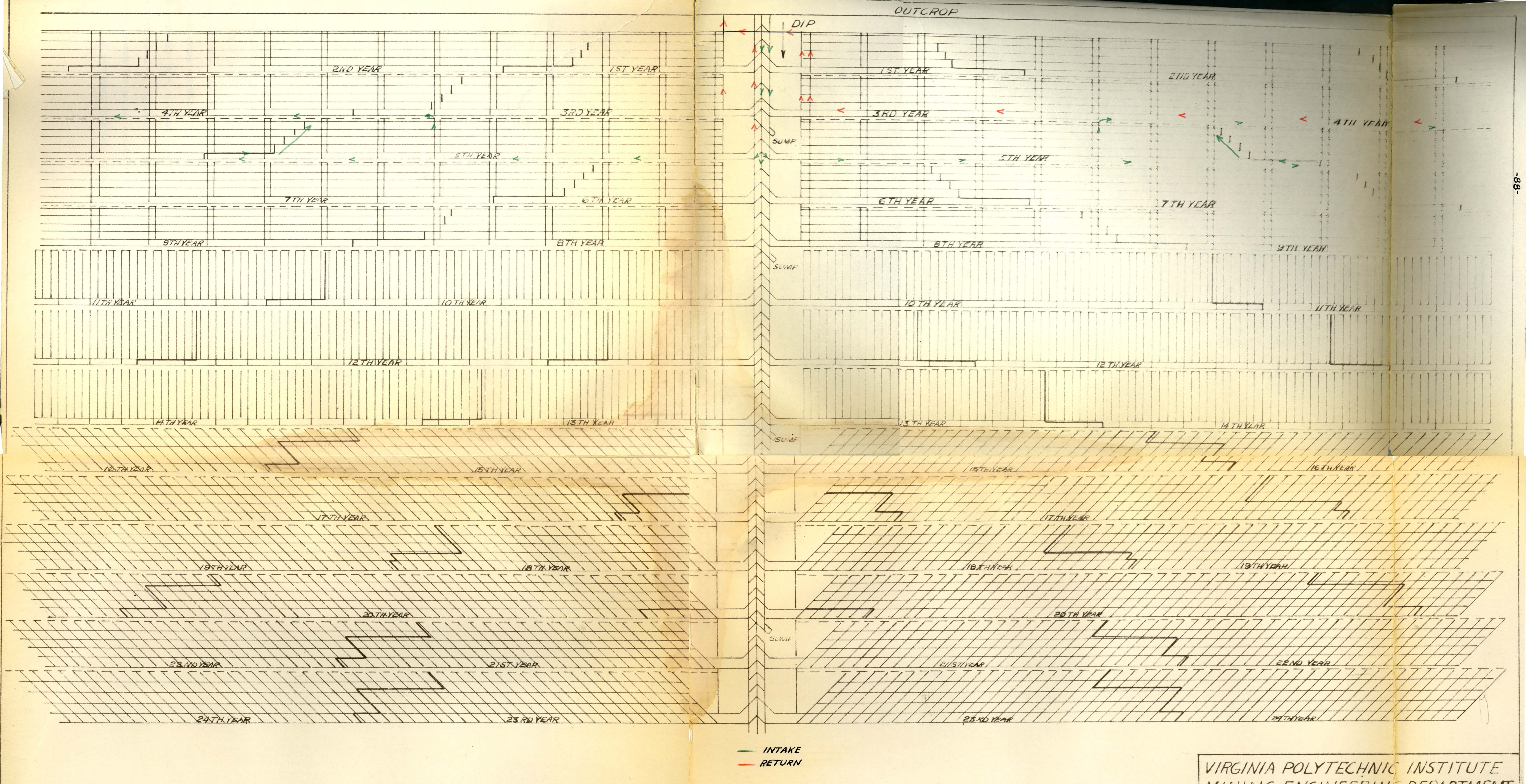


Figure 28

VIRGINIA POLYTECHNIC INSTITUTE
MINING ENGINEERING DEPARTMENT
PROJECTION OF PROPOSED MINE
FOR VIRGINIA ANTHRACITE
SCALE 1" = 400'
DRAWN BY: E.V. BOWMAN DATE: MAR 7, 1948

F. Ventilation

Ventilation for the mine projected in figure 28 will be supplied by a Jeffrey Aerodyne Mine Fan type 8 H. on a split system.

Two of the slope entries will be the intakes. The first two rooms on each entry of the east panel will be one return, while the other slope entry and the first room in the west panel will be the other return. This will make a total of two intakes and four returns. Using connecting rooms from one entry to the next makes possible lower air velocities and pressures without the expense of driving a multiple entry system on the main slope. The triple entry system projected is necessary from the ventilation standpoint in order to provide air for that portion of the slope entry that extends below the last cross entry that is in production.

The flow of air is indicated on the projection, figure 28. Air entering the mine through the two intakes splits at each cross entry the same way. Part of the air goes to the east entry; part, to the west entry and down the slope. From the experience of the Great Valley Anthracite operation, 100,000 cubic feet of air per minute is necessary at the 4500-ft. level. This quantity would be unevenly split at each cross entry since the intake air must also supply those portions of the main slope entries below the last cross entries. The split ratio will be 56,000 cubic feet per minute to the west side and main slope and 44,000 cubic feet per minute to the east side. A regulator in the east side will waste the required pressure to insure the proper quantity in each side of the split.

A permanent type door will not be necessary across the haulageway leading to the west side of the mine. To divert sufficient air to the slope bottom a check curtain will suffice because the slope will extend to a maximum depth of 320 ft. below the last active cross entry.

Since it is required to ventilate the adjacent entry whether it is active or not, another room will be cut through to the level above. This room will be located in the center of the panel of rooms comprising the particular level. In this way the advantages of a panel system are retained, yet the return air is made to traverse as short a distance as possible through the cross entries where the cross sectional area is least.

The velocity of the air down the main slope will be 349 ft. per minute, through the cross entries down to the 9th level 454 ft. per minute, and in the return airways 222 ft. per minute. In the cross entries from the 9th level on the velocity of the air will be 266 ft. per minute. With these velocities and a friction factor of 0.00000002 the pressure necessary to overcome friction is calculated. (Weight of air assumed to be 0.075 lbs. per cubic foot) To these calculated pressures a factor of 0.2" water gauge is added to compensate for velocity changes and shock loss. The calculations were made at the point of highest pressure during each year. The mine pressures were tabulated by year in figure 29.

Referring to the table of mine pressures, figure 29, it is seen that the lowest pressure is a one inch water gauge and the highest is 3.06". The very slight increase in mine pressure from year to year is due to the fact that the greatest proportion of the resistence to the flow of air is in the cross entries and not on the main slope.

For 100,000 cubic feet of air per minute at a pressure of 3.00" of water, a 60" diameter fan with a 75 horsepower motor is selected. It is impossible to select a fan that would exactly fit the duty required at all stages of development. The above fan was selected on the basis of the ultimate conditions. This fan will deliver the required amount of air at the given pressure when operated at 1045 r.p.m. (data from fan curves) This fan when operated at earlier stages of development would supply the required amount of air at a lower speed. It being impractical to change the speed of the fan as the mine characteristic changes, an average condition will be taken as the basis for changing speed. Referring to the table of mine pressures, figure 29, the average pressure for the first seven years is 2.15", between 8 and 16 years the average is 2.39" and the average for the remaining years is 2.81". This would make a total of three speed changes using the fan selected for the ultimate conditions.

Air horsepowers and brake horsepowers are calculated for the fan selected when operated against the average mine pressures. The necessary speeds and static efficiencies are taken from the characteristic curve of the fan and tabulated in figure 29. The motor output is calculated, assuming a drive efficiency of 95%, and results tabulated in figure 29. Motor input power is calculated assuming efficiencies of 80%, 85%, and 90%. These different efficiencies were assumed since the selected 75 horsepower motor would not be operating at peak efficiency except at the ultimate conditions.

Year	Mine Pressure Inches Water Gage	Static Efficiency Range	Fan Speed R.P.M.	Volume C.F.M.	Air H.P.	Average B.H.P.	Motor Output H.P.	Motor Input H.P.	Motor Input kw
1	1.00								
2	2.05								
3	2.16								
4	2.24 2.15	68-74	976	100,000	33.9	47.8	50.4	63.1	63.0
5	2.01								
6	2.29								
7	2.30								
8	2.37								
9	2.43								
10	1.97								
11	2.54								
12	2.40 2.39	70-76	1000	100,000	37.7	51.7	54.3	65.2	65.2
13	2.66								
14	2.71								
15	2.06								
16	2.70								
17	2.41								
18	2.76								
19	2.84								
20	2.74 2.81	71-77	1035	100,000	44.3	60.0	62.2	69.2	69.0
21	2.91								
22	2.97								
23	2.91								
24	3.06								

Figure 29. Fan Data

G. Drainage and Haulage

Little difficulty is expected from water in the mine as far as water at the working faces is concerned. This is so because the cross entries will be projected at such an angle to the strike that the grade will be down hill from the active workings to the slope, allowing natural drainage to the slope bottom. The mouth of the slope will be so located as to preclude the possibility of a large quantity of surface water gaining entrance to the mine via slope.

Estimating the quantity of water that the mine would make at this particular location and at various stages of development is a difficult problem. At the Great Valley operation pumps with a capacity of 200 gallons per minute handle the water problem in two shifts from a depth of approximately 5000 ft. down the slope. The wagon mines in the vicinity of the site selected seldom go below the water table, thus affording little insight into the water problem. The only basis, then, for estimating pumping requirements is from Great Valley.

The ideal situation in dealing with the pumping problems of this mine as projected, would be to collect the water from each level in a sump and pump it outside. This would not permit the water from the upper portion of the mine to drain to the lower portions and thus increase the head against which the water must be eventually pumped. This would entail a sump at each level, and a pump which would lift the water from one level to the one above. It would be more practical to combine the water from several levels and use a larger pump even though some of the water has gained considerable head.

It is not recommended that Single Stage Centrifugal pumps for mine service be used when the head is greater than 100 ft. per stage and when the mine water is highly acid. If the water is clear, the limit is 250 ft.[18] Against the total head, using the highest recommended pump in the Ingersoll-Rand pump catalog,[25] it would require 15 single stage pumps having a total horsepower of 225 to rid the mine of 200 gallons per minute. If two stage pumps are used, it would take four pumps requiring 40 horsepower each and one single stage pump requiring 20 horsepower making a total of only 180 horsepower. (The friction head is calculated on a three inch diameter pipe.)

Using the two stage pumps, they would be located at the levels indicated in figure 30.

Distance down the slope	Type Centrifugal	Catalog Designation	Horsepower
800 ft.	Two stage	2MRV40	40
1600 ft.	Two stage	2MRV40	40
2600 ft.	Two stage	2MRV40	40
3900 ft.	Two stage	2MRV40	40
4500 ft.	Single stage	2RVH20	20

Figure 30. Pump Requirements

Each pump will deliver water to the sump at the pump location above. The single stage pump, being the last permanent installation, can be used in the slope sinking operating until the 4500-ft. level is reached.

To lessen the friction head as much as possible a three inch diameter pipe is used.

The permanent sumps are shown on the projection, figure 28. In addition to the permanent sumps, temporary sumps are provided on the slope below the level of each cross entry to collect the water for pumping into the permanent sump above.

In choosing the haulage equipment for this operation, it is recommended that storage-battery locomotives be used from the standpoint of safety of operation in gaseous atmospheres. Mine cars having a capacity of five tons and weighing two tons have already been selected in connection with the design of thehoisting operation.

Cross entries are projected so that the grades will be 1% in favor of the loads.

Track is to be laid with 40-lb. steel rails throughout the mine.

In designing a haulage cycle for this operation for the ultimate conditions of this mine, the time element is all important. At the 4500-ft. level the hoist will remove cars from one of the cross entries at the rate of four cars every 10 minutes. (Hoisting time per lift is 305 seconds.) Taking the maximum haulage distance laterally of 4300 ft. and assuming an average speed of three miles per hour for a single locomotive, it will take

$$\frac{2 \times 4300 \times 60}{3 \times 5280} = 32.6 \text{ minutes running time for a round trip.}$$

Adding three minutes for coupling, etc., the total time per round trip will be 35.6 minutes. Assuming seven hours per day, a total of $\frac{7 \times 60}{35.6}$ or 11 trips per day will be possible. Since 800 tons must be mined daily from each cross entry, there will be $\frac{800}{11}$ or 72.7 tons per trip. This could be handled in trips of $\frac{72.7}{5}$ or 15 cars. From the time element alone this arrangement will provide the hoist with sufficient cars to maintain its schedule of operation.

However, it will mean that 15 cars must be loaded during the time it takes the locomotive to make the round trip. It will also require a storage track for changing empty cars and loaded cars at each loading point. It is highly improbable that loading at the loading point will proceed at such a uniform rate.

Calculating the size of a storage-battery locomotive necessary for 11 trips of 15 cars each over the 4300 ft. haul, it is found that it would be necessary to have a storage battery having a minimum capacity of 100 kw-hr.[21] This rating is above the recommended battery capacity of a 10-ton locomotive.

From the standpoint of locomotive size and keeping empty cars under the loading point, it is advisable to keep the haulage cycle in two stages -- making the longest haul 2150 ft. To provide more flexibility in the haulage system a side track will be maintained on the cross entries every 1075 ft. from the main slope.

In order to eliminate such a high capacity battery, the same number of cars per trip will be used in calculating the size of locomotive when hauling a distance of 2150 ft.

Calculation of locomotive size:[21]

Distance -- 2150 ft.

Cars per trip -- 15

Tonnage per trip -- 75

Trips per day -- $\frac{800}{75} = 11$

Average speed -- 3 mph. = 264 ft./min.

Running time per trip -- $\frac{2150 \times 2}{264} = 16.3$ minutes

Total time per trip -- 19.3 (3 minutes for coupling

Number trips possible -- $\frac{7 \times 60}{19.3} = 21$ per day.

Cars -- plain bearings

Acceleration -- 0.1 mph per second

Weight of locomotive:

In calculating the weight of the locomotive, it will be advisable to count on starting the trip from level track although the average grade would be minus 1%.

Weight of locomotive $= \frac{105(30 + 100 \times 0.1}{480 - 100 \times 0.1} = 8.94$ or 10 tons.

Capacity of battery: kwhr $= \frac{t \times d(30 + 20G)}{1,760,000}$ [23]

Loaded train $= \frac{115 \times 2150(30 + 20 \times 1)}{1,760,000} = 1.41$ kwhr

Empty train $= \frac{40 \times 2150(30 + 20 \times 1)}{1,760,000} = 2.93$

Total 4.34

Capacity of battery $= 43.4 \times 1.25 = 54.3$ kwhr.

With two 48-cell batteries having a voltage of 96, there will be $\frac{54,300}{96} = 565$ amp-hrs.

With an Exide-Ironclad MVA each plate has a capacity of 34 amp-hrs.

$\frac{565}{34} = 17$ positive plates, or a total of 35 plates will be necessary.

One cell MVA type with 41 plates has a capacity of 1.350 kwhr. A battery made from 48 cells will have a capacity of $48 \times 1.350 = 64.8$ kwhrs. which should be ample.

For each cross entry two 10-ton battery locomotives with Exide-Ironclad MVA batteries (48 cells of 41 plates each) will be required. 30 mine cars on each entry will be required for the operation.

H. Power

From the safety standpoint, battery locomotives are chosen for haulage instead of the normal D. C. electric locomotives with open trolley wire. Since direct current is not to be used in the haulage system, it is advisable to power the entire mine with alternating current.

2300 volt, three phasecurrent will be installed in the main slope with 3-conductor primary cable. At the junction of each cross entry 2300-440 volt transformers will be installed to provide power to the active sections on each cross entry. These transformers will be delta connected on the primary and star connected on the secondary with the neutral grounded. This will provide power to the working face at 440 volts and yet not exceed 220 volts line to ground.

The electrical loading in each section of the mine, according to the method of mining, is as follows:

Method I

(One Panel)

8 cutting machines @ 50 H.P. -	=	400
9 conveyor drives @ 10 H.P.	=	90
2 car spotting hoists @ 5 H.P.	=	10
1 elevating conveyor @ 5 H.P.	=	5
2 booster fans @ 2 H.P.	=	4
1 charging set @ 30 H.P.	=	30
8 drills @ 2 H.P.	=	16
Total		555

Method II

(One Panel)

6 cutting machines @ 50 H.P.	= 300 H. P.
7 conveyor drives @ 10 H.P.	= 70 H.P.
2 car spotting hoists @ 5 H.P.	= 10 H.P.
1 elevating conveyor @ 5 H.P.	= 5 H.P.
2 booster fans @ 2 H.P.	= 4 H.P.
1 charging set @ 30 H.P.	= 30 H.P.
8 drills @ 2 H.P.	= 16 H.P.
Total	435 H.P.

Method III

(One Panel)

1 loading machine @ 50 H.P.	= 50 H.P.
1 cutting machine @ 50 H.P.	= 50 H.P.
1 drill @ 5 H.P.	= 5 H.P.
2 hoists @ 5 H.P.	= 10 H.P.
1 charging set @ 30 H.P.	= 30 H.P.
2 shuttle cars @ 15 H.P.	= 30 H.P.
Total	175 H.P.

Slope

(During Period when Method I is being used)

1 pump @ 20 H.P.	= 20 H.P.
1 pump @ 40 H.P.	= 40 H.P.
1 cutting machine @ 50 H.P.	= 50 H.P.
1 drill @ 2 H.P.	= 2 H.P.
1 hoist @ 75 H.P.	= 75 H.P.
Total	187 H.P.

(During Period when Method II is being used)

1 pump @ 20 H.P.	= 20. H.P.
2 pumps @ 40 H.P.	= 80 H.P.
1 cutting machine @ 50 H.P.	= 50 H.P.
1 drill @ 2 H.P.	= 2 H.P.
1 hoist @ 75 H.P.	= 75 H.P.
Total	227 H.P.

(During Period when Method III is being used)

1 pump @ 20 H.P.	= 20 H.P.
4 pumps @ 40 H.P.	= 160 H.P.
1 cutting machine @ 50 H.P.	= 50 H.P.
1 drill @ 2 H.P.	= 2 H.P.
1 hoist @ 75 H.P.	= 75 H.P.
Total	307 H.P.

With the loading as diversified as it is in normal mine operations 500 watts per horsepower will be the estimated consumption.[6] With this same loading diversification, the size of transformers needed are estimated to be $\frac{KVA}{2}$ per horsepower. The maximum electrical loading is in Method I with 555 H.P. which would require a 280 KVA transformer. The maximum slope load of 307 H.P. is on a separate circuit and requires a 150 KVA transformer.

IX. ORGANIZATION

The organization showing the total labor and supervising personnel requirements can best be illustrated by an organization chart, figure 31. This does not include sales personnel.

- General Manager
  - Mining Engineer
    - Transitman
    - Rodman
  - Superintendent (Surface)
    - **Office**
      - Timekeeper
      - Accountant
      - Shipping Clerk
      - Nightwatchman
    - **Shops**
      - 1 Master Mechanic
      - 2 Electricians
      - 2 General Mechanics
      - 1 Blacksmith
      - 1 Lamphouse
    - **Preparation**
      - 1 Hoistman
      - 1 Car Shifter
      - 2 Washer
      - 1 Car Dropper
      - 1 Boom Operator
      - 2 Raw Coal
      - 2 Refuse
      - 1 Maintainance
      - 1 Oiler
  - Superintenaent (Underground)
    - Foreman (East)
      - Same as West
    - Foreman
      - **Slope**
        - 6 Loaders
        - 2 Machinemen
        - 2 Timbermen
        - 1 Bratticeman
        - 1 Pumpman
      - **West**
        - **15-24 yrs.**
          - 2 Loading Operators
          - 2 Machinemen
          - 2 Trackmen
          - 2 Drillmen
          - 2 Timbermen
          - 2 Motormen
          - 2 Brakemen
          - 2 Shuttle Car
          - 2 Car trimmer
        - **1-9 yrs.**
          - 2 Motormen
          - 2 Brakemen
          - 2 Car trimmers
          - 2 Car shifters
          - 2 Trackmen
          - 21 Loaders
          - 2 Supply men
          - 1 Battery Charger
        - **10-14 yrs.**
          - Same as above except 17 Loaders

Figure 31. Organizational Chart

X. COST

Since the mine is divided into three periods according to the method of mining, total costs will be divided according to the same three periods. The cost estimates follow in outline form.

A. Capital Equipment

1. Underground Face

Type	Number	Life	Period Needed	Unit Cost	Total Cost
Shortwall cutting machines	19	14	1-14	$ 7,430.00	$141,170.00
Drilling machines	19	14	1-14	500.00	9,500.00
Shaking conveyors complete with drives & 18 troughs	18	14	1-14	3,400.00	61,200.00
Curved discharge troughs	3	14	1-14	82.50	246.50
Extra troughs	40	14	1-14	22.75	910.00
Loading heads for shaking conveyors	18	5	9-14	1,263.00	22,734.00
Loading machines	3	10	15-24	17,000.00	51,000.00
Universal cutting machines	3	10	15-24	19,000.00	57,000.00
Drilling machines	3	10	15-24	5,000.00	15,000.00
Locked wood storage boxes for powder	3	4	1-24	30.00	540.00
Wood storage boxes for blasting caps	3	4	1-24	15.00	270.00
Shot firing batteries	3	3	1-24	0.30	7.20
Trailing cable for machines	300 ft. per machine	5	1-24	0.40/ft.	1,284.00
Elevating conveyors	3	14	1-14	1,750.00	5,150.00
Shuttle cars	3	10	15-24	8,000.00	24,000.00

2. Track and Accessories

Type	Number	Life	Period Needed	Unit Cost	Total Cost
Steel rail 40 lbs.	20,000 ft. 238.1 tons	24	1-24	$ ton/60.00	$ 14,286.00
Track turnouts	12	24	1-24	64.00	768.00
Diamond crossovers	2	24	1-24	200.00	400.00
Ties, wood	12,000	24	1-24	0.50	6,000.00
Track gauge bars	4	24	1-24	1.00	4.00
Splice bars	1,500 prs.	24	1-24	0.75	1,125.00
Spikes	150 kegs	24	1-24	6.90	1,035.00
Bolts & nuts	50 kegs	24	1-24	6.90	345.00

3. Ventilation

Type	Number	Life	Period Needed	Unit Cost	Total Cost
75 H.P. Aerodyne fan	1	24	1-24	12,000.00	12,000.00
Air pressure recorder	1	24	1-24	78.00	78.00
Booster fans	3	14	1-14	800.00	2,400.00

4. Power and Transmission

Type	Number	Life	Period Needed	Unit Cost	Total Cost
280 KVA transformers	4	24	1-24	2.75/KVA	2,880.00
150 KVA transformers	2	24	1-24	2.75/KVA	825.00
Charging sets	2	24	1-24	50.00/H.P.	6,000.00
Controls for charging sets	2	24	1-24	5.00/H.P.	600.00
Primary cable 2300 volt	5000 ft.	24	1-24	1.25/ft.	6,250.00
Secondary cable 440 volt	10,000 ft.	24	1-24	0.50/ft.	5,000.00
Accessory equipment, circuit breakers, switches, etc.		24	1-24		12,000.00

5. Coal Preparation

Type	Number	Life	Period Needed	Unit Cost $	Total Cost $
Cleaning unit	1	24	1-24	200,000.00	200,000.00
Tipple structure	1	24	1-24	75,000.00	75,000.00
Tipple equipment	1	24	1-24	90,000.00	90,000.00
Storage bins	1	24	1-24	100,000.00	100,000.00
Railroad tracks & site	1	24	1-24	55,000.00	55,000.00
Refuse disposal	1	24	1-24	50,000.00	50,000.00
Water supply system	1	24	1-24	10,000.00	10,000.00
Conveyor system from slope	1	24	1-24	170,000.00	170,000.00

6. Hoist

Type	Number	Life	Period Needed	Unit Cost $	Total Cost $
Hoist complete with controls and 600 H.P. motor	1	24	1-24	125,000.00	125,000.00
Additional hoist rope	30,000 ft.	24	1-24	2.00	60,000.00
Storage bin & feeder for dumping cars from slope	1	24	1-24	25,000.00	25,000.00

7. Drainage

Type	Number	Life	Period Needed	Unit Cost $	Total Cost $
Pumps 40 H.P. centrifugal	4	24	1-24	2,000.00	8,000.00
Pumps 20 H.P. centrifugal	1	24	1-24	1,200.00	1,200.00
Pipe 3", black, 5,000 ft.	5,000 ft.	24	1-24	0.90/ft.	4,500.00
Elbows 45°-3"	20	24	1-24	4.80	96.00
Elbows 90°-3"	20	24	1-24	2.20	44.00
Suction valves	10	24	1-24	2.00	20.00
Suction strainers	10	24	1-24	2.00	20.00
Valves	20	24	1-24	50.00	1,000.00

8. Haulage Equipment

Type	Number	Life	Period Needed	Unit Cost	Total Cost
10-ton battery locomotives	4	24	1-24	$ 10,000.00	$ 40,000.00
5-ton steel mine cars	60	24	1-24	500.00	30,000.00
Car spotting hoists	4	24	1-24	650.00	2,600.00
Slope auxiliary hoist	1	24	1-24	2,500.00	2,500.00
Supply trucks push type	4	24	1-24	350.00	1,400.00
Supply hoists	2	14	1-14	650.00	1,300.00
Storage boxes for locomotive sand	2	4	1-24	15.00	180.00
Markers for haulage trips	8	3	1-24	0.40	25.60
Charging station	2	24	1-24	1,200.00	2,400.00
Batteries	4	4	1-24	1,500.00	36,000.00

9. Miscellaneous Shop Equipment

$12,882.50

The above total includes one arc welding machine, one power drill, one 50-ton hydraulic press, one machinist lathe, and one milling machine, as well as numerous smaller items.

10. Hand Tools, General

$ 375.00

11. Hand Tools, Track

$ 312.00

12. Hand Tools, Masonry

$ 46.25

13. Hand Tools, Timbering

$ 200.00

14. Illumination

$4,630.00

The above figure includes the cost of 125 cap lamps and batteries and charging racks and charging set for the lamps.

15. Safety and First Aid Equipment

$3,000.00

16. Engineering Supplies

$1,031.00

17. Surface Buildings

Type	Number	Life	Period Needed	Unit Cost	Total Cost
Shop	1	24	1-24	$15,000.00	$15,000.00
Supply warehouse and lamphouse	1	24	1-24	15,000.00	15,000.00
Mine office	1	24	1-24	9,000.00	9,000.00
Powder house	1	24	1-24	3,000.00	3,000.00

B. Supervision and Labor

1. Supervision

	Number	Monthly	Total
General Manager	1	$900.00	$900.00
Superintendent	2	600.00	1200.00
Mining Engineer	1	600.00	600.00
Transitman	1	380.00	380.00
Rodman	1	300.00	300.00
Foreman	2	450.00	900.00
		Total	$4,280.00

2. Labor

a. Surface Labor

Preparation Plant	Number	Daily (8 hrs.)	Monthly
Hoist	1	9.12	$ 182.40
Car shifter	1	10.24	204.80
Cleaning plant	2	9.60	384.00
Car Dropper	1	8.80	176.00
Boom operator	1	8.80	176.00
Raw Coal Handling	2	8.56	342.40
Refuse Disposal	2	8.64	345.60
Maintenance	1	9.60	192.00
Slope oiler	1	10.24	204.80

Shop	Number	Daily (8 hrs.)	Monthly
Master mechanic	1		$ 350.00
Electricians	2	9.20	368.00
General mechanics	2	9.20	368.00
Blacksmith	1	9.20	184.00
Lamp house	1	8.88	222.00
Office			
Timekeeper	1		250.00
Accountant	1		300.00
Shipping clerk	1		250.00
Supply clerk	1		200.00
Night watchman	1	8.96	268.80
General outside	3	8.56	513.60
		Total	$5,482.40

b. Labor Underground

(1) Method I

Loaders	48	15.00	14,400.00
Trackmen	3	9.17	550.20
Track helpers	3	9.17	550.20
Timbermen	2	9.17	366.80
Motormen	4	9.01	720.80
Brakemen	4	9.01	720.80
Car spotters	4	10.24	819.20
Shot firers	2	10.21	408.40
Bratticemen	2	9.17	366.80
Pumpmen	1	9.17	183.40

	Number	Daily (8 hrs.)	Monthly
Supply men	4	10.24	$ 819.20
Machine men	2	15.00	600.00
Battery charger	1	9.57	191.40
		Total	$20,697.20

(2) Method II

Loaders	40	15.00	12,000.00
Trackmen	3	9.17	550.20
Track helpers	3	9.17	550.20
Timbermen	2	9.17	366.80
Motormen	4	9.01	720.80
Brakemen	4	9.01	720.80
Car Spotters	4	10.24	819.20
Shot Firers	2	10.21	408.40
Bratticemen	2	9.17	366.80
Pumpman	1	9.17	183.40
Supply men	4	10.24	819.20
Machine men	2	15.00	600.00
Battery charger	1	9.57	191.40
		Total	$19,297.20

(3) Method III

	Number	Daily (8 hrs.)	Monthly
Loading machine operators	4	16.00	$ 1,280.00
Cutting machine operators	4	16.00	1,280.00
Drilling	4	14.00	1,120.00
Shot firers	2	10.21	408.40
Supply men	4	10.24	819.20
Timbermen	2	9.17	366.80
Car spotters	2	10.24	409.60
Motormen	4	9.01	720.80
Brakemen	4	9.01	720.80
Trackmen	3	9.17	550.20
Track helpers	3	9.17	550.20
Shuttle car operators	4	14.00	1,120.00
Loaders	6	15.00	1,800.00
Bratticemen	2	9.17	366.80
Pumpmen	1	9.17	183.40
Machine men	2	15.00	600.00
Battery charger	1	9.57	191.40
		Total	$12,296.20

C. Maintenance

	1-9 yrs.	10-14 yrs.	15-24 yrs.
Maintenance supplies all types	2,000.00	2,000.00	2,500.00
Preparation plant (0.04 per ton)	800.00	800.00	800.00
Lubrication (0.0071 per ton as mined)	154.00	154.00	154.00
Total	2,954.00	2,054.00	3,454.00

D. Operating Costs (other than labor)

Timber	855.00	855.00	855.00
Magnetite (0.015 per clean ton)	300.00	300.00	300.00
Power, preparation plant (0.02 per clean ton)	400.00	400.00	400.00
Power, hoist (0.02 per kwhr)	672.00	800.00	960.00
Power, surface (0.02 per kwhr)	200.00	200.00	200.00
Power, underground (0.02 per kwhr)	2,075.00	1,755.00	1,051.00
Power, fan (0.02 per kwhr)	2,256.00	3,130.00	3,322.00
Powder costs (0.06 per ton)	1,200.00	1,200.00	1,200.00
Patent Royalties (0.01 per clean ton)	200.00	200.00	200.00
Total	7,858.00	8,840.00	8,488.00

E. Monthly Total Costs

The total capital equipment cost is computed for each of the three periods of mine operation. A capital recovery factor of three per cent is used in computing the monthly payments necessary to amortize the investment over the particular periods.

Estimates for the items, insurance, taxes, compensation and unemployment were made on the current rates that are charged to the Great Valley operation.

Monthly Total Costs

	1-9 yrs.	10-14 yrs.	15-24 yrs.
Labor	26,179.60	24,779.60	17,778.60
Supervision	4,280.00	4,280.00	4,280.00
Maintenance (other than labor)	2,954.00	2,954.00	3,454.00
Operating Costs (other than labor)	7,858.00	8,840.00	8,488.00
Depreciation and interest on investment	7,775.00	8,125.00	7,729.00
Taxes, state, federal	100.00	100.00	100.00
Insurance	500.00	500.00	500.00
Welfare (0.10 per ton)	2,000.00	2,000.00	2,000.00
Royalty on leases (0.20 per ton)	4,000.00	4,000.00	4,000.00
Compensation ($4.50 per $100 of payroll)	1,370.70	1,307.70	992.70
Unemployment, social security (4% of gross payroll)	1,218.39	1,162.39	882.39
Total	$58,235.69	58,048.69	50,204.69

F. Total Monthly Income

The total monthly income will be computed assuming an average of 20 working days per month. The selling price is the current selling price, and the size distribution of sales is the average of a six month's period in 1946. (Fig. 14)

Stove	8.98% x 20,000 = 1791 tons x 7.79 =	$ 13,950
Nut	31.82% x 20,000 = 6370 tons x 7.79 =	49,600
Pea	17.91% x 20,000 = 3580 tons x 5.60 =	20,020
Buckwheat	14.54% x 20,000 = 2910 tons x 4.65 =	13,530
Rice	8.86% x 20,000 = 1770 tons x 3.94 =	6,980
Culm	17.89% x 20,000 = 3579 tons x 2.10 =	7,530
		$111,610

XI. DISCUSSION

A. General

In collecting the material for this thesis, interviews and informal conversations were necessary. The people who have been intimately associated with the Valley Coal Fields for a number of years are miners, superintendents, and foremen — as well as owners. In every case a great deal of optimism was shown regarding the prospects of the success of a modern mine in this area.

B. Recommendations

Although washability studies indicated a fair recovery at a lower ash percentage than is currently available, a study should be made on a pilot plant scale if a heavy density unit is to be used for washing. The results obtained from the washability study are theoretically possible, but on a practical scale lower recoveries may result — how much lower can be ascertained only by pilot plant studies. Since the degree of recovery is of such importance in widening the gap between profit and loss, a practical test with a large sample should be made.

The underground mining methods in this thesis were based on the use of cutting machines at various attitudes with respect to the coal seam. Although manufacturers recommendations were taken regarding the capabilities of the cutting machines, it is felt that actual cutting tests should be made on various slopes and cutting should be done at different horizons in the coal seam. Mechanically cutting the coal increases the tons per man ratio which is very important considering the high cost of labor.

Mobile loading machines, too, should be physically tested under the conditions that they are to be operated.

C. Limitations

The 1000 tons of clean coal per day production figure was chosen since that is the minimum that the railroads will consider in building a spur line track of three or four miles in length. Although not a part of this study, market conditions would certainly determine the daily output which could very possibly be much lower than the output chosen. A lower output would of course increase the burden cost of mining.

In this study it is assumed that continuous operation is possible year in and year out. In an industry noted for its erratic labor contitions this appears to be an optimistic assumption, but one that is necessary considering the unpredictability of the problem.

It is very difficult to obtain estimates of equipment costs at the present time. Some manufacturers will not quote any price and others include escalator clauses. An attempt was made to get estimates on the major items of equipment from the experts in the particular fields. Other cost figures were obtained by taking older estimates and bringing them up to date.

XII. CONCLUSIONS

The following conclusions can be made as a result of this study:

1. The best location for a new mine in this area is in the center of the body of coal lying between New River and Poverty Creek.

2. Material previously discarded underground as worthless by visual inspection contains valuable fuel which can be separated on a specific gravity basis if ground to the proper size. It is possible to place on the market a fuel containing 14%-17% ash in place of the present fuel with 18%-22% ash (comparable recoveries).

3. Using newer equipment and newer methods, the coal can be placed on the market at a cost of $50,000 to $60,000 monthly for 20,000 tons of clean coal against an income of $112,000 for the same period.

XIII. SUMMARY

A study of the possibility of a modern mining operation in the Valley Coal Fields of Virginia begins with a brief history of previous mining operations in the area. Following this is a study of the geological occurrence and a geographical description of the site selected.

The deposit is studied from various angles -- physical characteristics of the seam in place, chemical characteristics and washability are all included. After the amount and type of impurities present in the deposit are ascertained, a coal preparation plant is proposed to clean the coal to the desired level.

Leaving the surface problems, the underground problems of hoisting, mining methods, ventilation, haulage and drainage, and power are taken up separately and a solution recommended for each problem.

To operate the mine an organization is designed to fit the solution of the various problems.

The estimate of the cost of equipping and providing the labor and supervision to operate such a mine is included in the study. This cost is compared with the expected income.

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