COAL MINING COSTS

COAL MINING COSTS

BY

A. T. SHURICK
MEMBER AMERICAN INSTITUTE OF MINING ENGINEERS,
FORMERLY ASSOCIATE EDITOR "COAL AGE," ETC.

FIRST EDITION

McGRAW-HILL BOOK COMPANY, INC.
NEW YORK: 370 SEVENTH AVENUE
LONDON: 6 & 8 BOUVERIE ST., E. C. 4
1922

PREFACE

THERE are books on costs in all the important branches of engineering except coal mining. The author has waited patiently the advent of a similar work in his chosen field and none having been forthcoming he has made bold to venture the effort himself.

Obviously a work of this character cannot be up-to-date as to costs in dollars and cents because of the wide fluctuations in the purchasing power of the dollar in labor, equipment and material, particularly during the last five years. There has been no hesitancy, therefore, in using data of a number of years back so long as the subject discussed is still in general use as in the case of the comparative costs of wood and masonry brattices, etc. The reader will have no difficulty in interpolating the figures given to conform to current standards and to facilitate this a table giving all the wage scales in the Central Competitive District since 1898 has been given on page 158. Thus if a certain piece of work required the services of three men for a certain number of days, say a decade ago, it is a relatively simple matter to estimate the cost in terms of prevailing wage scales. Care has been exercised to give the year during which the different examples cited occurred in order that this interpolation can readily be effected.

A great deal of valuable data of an abstract nature has been obtained from the various State and Federal government reports but in general it has been the endeavor to hold more to specific costs. Thus the cost of haulage or the cost of doing a piece of work under certain well defined conditions has been accepted as of more value than the average cost of mining for a certain district or the capital investment per ton of capacity, etc. In other words it has been the aim to make the work essentially practical.

Only a few of the best systems of mining have been discussed but these, it is believed, have been covered in greater detail than in works dealing with this subject alone. The reason for this is that unless full particulars concerning all phases of working under any system are made clear beyond all peradventure the cost figures are worthless.

To insure a thorough treatment of all subjects taken up it was deemed advisable to limit the present volume to underground costs alone. A great deal of valuable data on outside costs has been assembled in the course of the present work and it is thought the publication of this in a separate volume at a later date will enhance the value of the completed work more than if an attempt were made to straddle the two fields in the present volume.

It is thought, if anything, the book errs on the side of conservatism; old and tried methods only have been accepted, newer systems and appliances, some of which are very promising at this time, having been used with caution. As an example the use of the underground loading machine or the combined mining and loading machine has not become sufficiently general as yet to justify the amount of space devoted to say, the mine motor which is now an accepted part of nearly every mining operation.

In concluding, the author wishes to make sincere acknowledgment to the many friends who have furnished him with much valuable material and still more valuable suggestions and advice; to the various engineering societies, the papers of which have been quoted so freely throughout the book; to the large engineering companies, several of which in particular have been untiring in their efforts to furnish certain special requests for material; and to the technical press which has been drawn upon liberally.

A. T. SHURICK.

DEARBORN, MICH.
January, 1922.

CONTENTS

SECTION I

MINING COSTS

SECTION II

SHAFT SINKING

SECTION III

HAULAGE COSTS

PAGE

SECTION IV

TIMBERING COSTS

SECTION V

MISCELLANEOUS INSIDE COSTS

COAL MINING COSTS

SECTION I

MINING COSTS

The U. S. Census report for the year of 1909 shows that the value of the Pennsylvania anthracite produced that year was $148,957,894. The total gross expenses amounted to $139,110,444, from which should be deducted $4,864,844 made from charges to miners for explosives, oil and blacksmithing, making the net expenses $134,245,600.

The gross expenses are itemized as follows:

Services:		
Salaries	$ 4,572,489	
Wages	92,169,906	$ 96,742,395
Supplies:		
Fuel and power	3,189,279	
Other supplies	23,472,809	26,662,088
Royalties		7,969,785
Miscellaneous		7,736,176
Total gross expenses		$139,110,444
Deductions		4,864,844
Net expenses		$134,245,600

The total production in 1909 amounted to 72,215,273 long tons, so that the average value per ton for the output in that year was $2.06; the average cost per ton was $1.86; and the net returns on the operations for the year were $14,712,294, or an average of 20c per ton. This at first glance looks like a fair return, but attention must be called to the fact that the Census figures of cost make no allowance for interest on capital invested or borrowed, and no offsetting charges for amortization or depreciation.

According to the returns to the Bureau of the Census, the entire capital invested in anthracite mining in 1909 was $246,-700,000, which may appear rather inadequate when one considers the magnitude of the industry, and an annual production of $150,000,000 (in 1911 the output was valued at $175,189,392 and in 1912 it was $177,622,626), but these are the figures reported by the Census Bureau. If on this capitalization an allowance of 4 per cent be made for interest, the net returns for the year amounted in round numbers to $4,844,000.

If new breakers and other equipment are charged into operating expenses no allowance need be made for depreciation, but the exhaustion of from 75,000,000 to 80,000,000 tons from the reserves every year should have some amortization charged against it and if 5c. a ton be allowed the margin of $4,800,000 is practically wiped out.

The figures covering the cost and value of bituminous coal show even more striking comparisons. There are some slight differences in the statistics of production between the Census figures and those published by the United States Geological Survey for the reason that the Census investigations excluded mines having a production of less than 1000 tons, whereas the Survey includes every small country bank from which it can secure a report. For 1909 the Survey showed a bituminous coal production of 379,744,257 short tons valued at $405,486,777, and the Census Bureau showed a production of 376,865,510 tons valued at $401,577,477, the difference being about 3,000,000 tons in quantity and $4,000,000 in value—less than 1 per cent in either case. As the Census figures for cost of mining are the basis of this discussion, the Census figure of production is also used.

The total value of the bituminous production, as already stated, was $401,577,477, and the mining expense of producing this value, including salaries of officers, was $378,159,282. As in the case of anthracite, the expenses of production do not include any charges for depreciation, amortization, or interest on capital invested or borrowed. The expenses are divided as follows:

Salaries	$ 20,417,392
Wages	282,378,886
Supplies	45,345,932
Royalties	12,035,900
Miscellaneous	17,961,172
Total	$378,159,282

From this it appears that 75 per cent of the total cost and 70 per cent of the total value was spent in wages. Salaried officials got less than 5.5 per cent.

The total capital invested in the bituminous coal mines of the United States in 1909 was, according to the Census bulletin, in round numbers $960,000,000 ($960,289,465), and this does not appear as if there were very much over-valuation, whatever the capitalization may be as represented by stock issue. The difference between the value of the product and the expense of producing it was $23,440,000 in round numbers or a fraction over 2.5 per cent on the capital.

According to the figures compiled by the Bureau of the Census, the amount paid in wages was, in 1909, above 80 per cent of the total selling value of coal at the mine mouth. From 1909 to 1913 there were two wage increases granted—one in 1910 of 5.55 per cent and another in 1912 of 5.26 per cent. These increases brought the wage cost per ton of coal produced to 92.44c. in 1913.

In 1913 the average selling price of coal at the mines in Illinois was $1.14 and in Indiana $1.11 per ton. This leaves a margin of only 21.6c. in Illinois and 18.6c. in Indiana. Out of this must be paid the cost of material used at the mines; the cost of making sales; all officers' salaries; general expenses; insurance (liability, fire, storm, etc.); taxes (including tax on plant and mineral rights); interest on the investment; depreciation of plant; royalties or charges for the exhaustion of coal.

The report of the Bureau of the Census for 1909 showed that, without allowing for any interest charge on the investment or for amortization of property, the so-called net returns in Illinois and Indiana were only 3c. per ton in Illinois and less than 1c. per ton in Indiana.

The average royalty paid, however, in these two states on coal recovered under lease is 5c. per ton, and the average present valuation of coal land is such as to require a minimum amortization charge of 3c. per ton to recover such land value within the period of the mine's life. It will therefore be seen that in even so good a year as 1913 an actual profit return was impossible. As existing facts show, the industry sustained a substantial deficit in these two states.

The average value per ton of all the bituminous coal produced in the United States was $1.07 and the costs averaged a fraction of a cent over $1, so that the margin of profit to cover interest, depreciation and amortization was a little less than 7c. a ton. In some states the expenses exceeded the returns. Take Arkansas, for instance, where the expenses totaled $3,630,526 and the value of the product was $3,508,590. Other instances were:

	Value of Product	Expenses
Iowa	$12,682,106	$12,816,076
Kentucky	9,940,485	10,127,987
Tennessee	6,548,515	6,691,482
Oklahoma	6,185,078	6,536,441
Virginia	4,336,185	4,392,440

Pennsylvania, by long odds the most important producer, with an output of 137,300,000 tons, showed a total of expenses of $117,440,000 and of value of $129,550,000 a balance on the profit side of a little over $12,000,000, or about 3 1/3 per cent on the capital invested, $358,600,000. The four competitive states, West Virginia, Illinois, Ohio and Indiana, which rank second, third, four and fifth, respectively, in producing importance, all show such narrow margins between income and outlay that profits are infinitesimal. The figures follow:

	Value of Product	Expenses	Difference
West Virginia	$ 44,344,067	$ 43,024,716	$1,319,351
Illinois	53,030,545	51,697,504	1,333,041
Ohio	27,353,663	27,153,497	200,166
Indiana	15,018,123	14,906,831	111,292
	$139,746,398	$136,782,548	$2,963,850

DISTRIBUTED COST AND SELLING PRICE OF COAL WITH OTHER DETAILS, UNITED STATES CENSUS OF ILLINOIS AND INDIANA, 1909

State	Number of Operators	Gross Capital and Capital per Ton of Product	Expenses						Number of Wage Earners	Coal Produced, Including Coal Coked at Mines	
			Total	Salaries	Wages	Supplies	Royalties	Miscellaneous Expenses		Value Including Minor Products	Tons (2000 Pounds)
Illinois	470	$75,257,667	$51,697,504	$2,083,668	$41,991,246	$4,944,371	$744,860	$1,933,359	74.445	$53,030,545	50,570,503
Per ton	....	$1.488	$1.024	$0.041	$0.832	$0.0977	$0.014	$0.038	.....	$1.05	
Indiana	223	35,937,961	14,906,831	604,111	12,273,544	1,198,974	240,494	589,708	22,357	15,018,123	14,723,231
Per ton	....	$2.441	$1.012	$0.041	$0.834	$0.0815	$0.0163	$0.040	.....	$1.02	

COMPARATIVE DISTRIBUTED COST AND SELLING PRICE OF COAL IN 1889 AND 1909 IN ILLINOIS AND INDIANA

State	Census	Gross Capital and Capital per Ton of Product	Expenses				Value at Mines	Tons (2000 Pounds)	Wage Ratio to Value
			Total	Wages	Supplies	Contract Work			
Illinois	1909	$75,257,667	$51,697,504	$41,991,246	$4,944,371	$51,480	$52,999,918	50,570,503	79%
Per ton	....	$1.488	$1.024	$0.832	$0.0977	$0.0001	$1.048		
	1889	17,630,351	10,366,069	8,111,253	966,927	26,662	11,755,203	12,104,272	69
Per ton	....	$1.457	$0.856	$0.670	$0.0799	$0.002	$0.097		
Indiana	1909	35,937,961	14,906,831	12,273,544	1,198,974	10,674	14,984,616	14,723,231	81
Per ton	....	$2.441	$1.012	$0.834	$0.0815	$0.0007	$1.018		
	1889	3,435,703	2,581,669	2,045,641	241,094	5,807	2,887,852	2,845,057	70
Per ton	....	$1.207	$0.906	$0.718	$0.0847	$0.0002	$1.015		

These four states with an aggregate production of a little more than the bituminous output of Pennsylvania, showed a total of less than $3,000,000 as the excess of receipts over expenses. The capital invested in the coal-mining industry in these states was something over $310,000,000, so that the returns on the capital were less than 1 per cent.

The United States Fuel Administration, organized during the war emergency, was empowered to exact the most intimate details as to operating costs and profits in the coal industry under severe penalities for omissions or incorrect returns. The Engineers Committee of the Fuel Administration, headed by some of our most prominent engineers and equipped with an excellent organization for assembling and correlating this mass of material compiled a report on general production costs unexcelled in the history of the industry for its authenticity and accuracy.

The accompanying table is a summary showing reported and adjusted costs, prices fixed and tonnage for all the principal districts to August 12, 1918. The diagram Fig. 1 shows these, in general, before the labor increase of November, 1917, compensated for by the 45c. general advance in coal prices, giving the average costs, ''bulk lines,'' and prices fixed for practically all districts in the country as of August and September, 1917, and covers about 95 per cent of the total output of bituminous coal for the period stated.

The costs for each district in the proportion of its output to the total tonnage studied are shown in heavy lines, the ''bulk lines'' are shown by medium lines, and the prices fixed are indicated by light lines. The diagram also shows the weighted average costs, ''bulk lines,'' and prices fixed for the tonnage included, and effectively disposes of the widely circulated aspersions of profiteering, of which the industry was so freely accused by people having no knowledge of the facts or willfully misrepresenting them.

Diagram Fig. 2 shows the same data for the principal districts for the full year 1918, giving, however, only reported costs and prices fixed, the prices having been fixed on the August-September, 1917, data, and changed only by the 45c. allowed November 1, 1917, to compensate for the labor increase of that date, reduced May 24, 1918, to 35c. in consideration of equal car distribution ordered at that time.

The weighted average margin between costs and prices for practically the entire bituminous coal production of the country was but 45.6c., and between the "bulk line," which represents

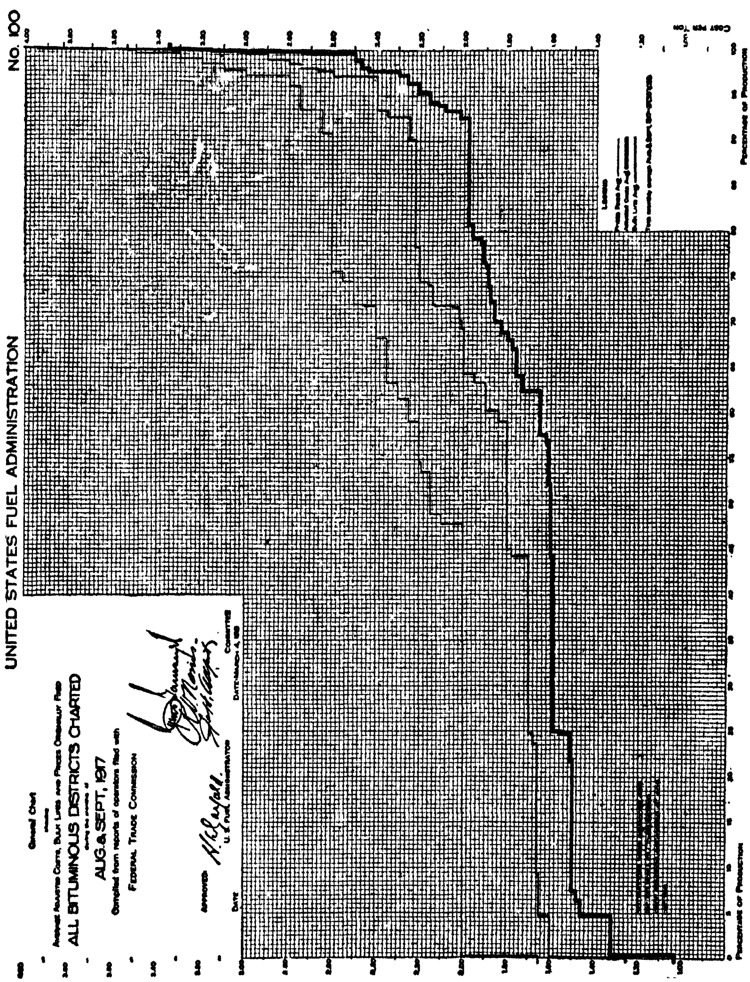


FIG. 1.—Adjusted costs and prices fixed by the Fuel Administration for all bituminous districts previous to November, 1917.

the higher-cost necessary coal and the price fixed by the Fuel Administrator, was but 26c.

As it is known that the capital invested per ton of yearly output in bituminous mines ranges from \$2 to nearly \$8, interest on which is included with other mines in the "margin" not included in the charted costs, it is evident that taking the

SUMMARY OF REPORTED AND ADJUSTED COSTS, PRICES FIXED, TONNAGE AND WEIGHTED AVERAGES TO AUGUST 12, 1918

No.	State or District	Month	Monthly Tons	Average Costs		Price Fixed	Tons Above Price Fixed	Per Cent Above Price Fixed
				Reported	Adjusted			
1	Colorado lignite	August	150,000	$1.80	$1.75	$2.35	1,500	1.0
2	Colorado lignite	September	140,000	1.89	1.81	2.35	17,950	12.7
3	Colorado domestic	August	475,000	2.03	1.98	2.55	28,500	6.0
4	Colorado domestic	September	450,000	2.06	1.95	2.55	45,000	10.0
5	Colorado Trinidad	August	340,000	1.88	1.80	2.19	6,800	2.0
6	Colorado Trinidad	September	300,000	1.88	1.80	2.19	8,200	3.0
7	Iowa, Des Moines	August	410,000	2.14	2.16	2.80	14,400	4.0
		September	410,000	2.14	2.16	2.80	14,400	4 0
8	Iowa, Appanoose	August	140,000	2.42	2.44	3.00	1,700	1.3
		September	140,000	2.42	2.44	3.00	1,700	1.3
9	Central Pennsylvania	August	4,120,000	1.93	1.96	2.60	123,600	3.0
10	Central Pennsylvania	September	4,500,000	1.98	2.00	2.60	198,000	4.4
11	Montana, Wyoming, Utah	August	950,000	1.89	1.89	2.60	9,500	1.0
		September	950,000	1.89	1.89	2.60	9,500	1.0
12	Upper Potomac	August	690,000	1.75	1.81	2.40	24,200	3.5
13	Upper Potomac	September	500,000	1.94	1.95	2.40	30,000	6.0
14	West Virginia, Tug River	August	230,000	1.83	1.76	2.40	6,900	3.0
15	West Virginia, Tug River	September	230,000	1.83	1.80	2.40	13,200	5.7

16	Virginia, Upper Clinch	August	30,000	2.07	2.10	2.50	900	3.0
17	Virginia, Upper Clinch	September	25,000	2.24	2.15	2.50	1,100	4.4
18	West Virginia, Preston	August	55,000	1.87	1.89	2.40	5,000	9.0
19	West Virginia, Preston	September	50,000	1.97	1.91	2.40	4,600	9.3
20	Pennsylvania, southwestern	August	7,450,000	1.54	1.57	2.00	74,500	1.0
21	Pennsylvania, southwestern	September	7,000,000	1.60	1.61	2.00	91,000	1.3
22	Ohio, District No. 8	August	1,230,000	1.49	1.49	2.00		
23	Ohio, District No. 8	September	1,000,000	1.51	1.53	2.00		
26	Illinois, District No. 1	August	340,000	2.26	2.24	2.75	1,700	0.5
27	Illinois, District No. 1	September	280,000	2.31	2.29	2.75	21,000	7.5
28	Illinois, Districts Nos. 2 and 5	August	425,000	1.86	1.84	2.40	3,200	0.7
29	Illinois, Districts Nos. 2 and 5	September	390,000	1.86	1.82	2.40		
30	Illinois, Districts Nos. 3, 4 and 6	August	5,700,000	1.47	1.48	2.00	19,000	0.3
31	Illinois, Districts Nos. 3, 4 and 6	September	5,350,000	1.50	1.53	2.00	69,500	1.3
32	Oklahoma and Arkansas	August	305,000	2.43	2.43	3.30	4,600	1.5
33	Oklahoma and Arkansas	September	240,000	2.57	2.58	3.30	6,000	2.5
34	Oklahoma, McAlester District	August	80,000	2.94	3.02	3.95		
35	Oklahoma, McAlester District	September	60,000	3.17	3.29	3.95	1,800	3.0
38	West Virginia, Pocahontas	August	1,690,000	1.38	1.30	2.00	30,400	1.8
39	West Virginia, Pocahontas	September	1,560,000	1.42	1.33	2.00		
40	New Mexico, Raton District	August	261,000	1.62	1.73	2.35		
41	New Mexico, Raton District	September	261,000	1.49	1.73	2.35		
42	East Kentucky and East Tennessee	November	900,000	2.02	1.91	2.65	18,000	2.0
		November	900,000	2.02	1.91	2.65	18,000	2.0
43	Kentucky, Tennessee and Virginia	November	1,135,000	1.54	1.60	2.20	3,400	0.3
		November	1,135,000	1.54	1.60	2.20	3,400	0.3
44	West Virginia, District No. 10	August	370,000	1.65	1.70	2.30		

SUMMARY SHOWING REPORTED AND ADJUSTED COSTS, PRICES FIXED, TONNAGE AND WEIGHTED AVERAGES TO AUGUST 12, 1918—*Continued*

No.	State or District	Month	Monthly Tons	Average Costs		Price Fixed	Tons Above Price Fixed	Per Cent Above Price Fixed
				Reported	Adjusted			
45	West Virginia, District No. 10	September	360,000	1.73	1.75	2.30	2,700	0.8
46	West Virginia, Logan District	August	885,000	1.61	1.65	2.15	6,200	0.7
47	West Virginia, Logan District	September	800,000	1.70	1.65	2.15		
48	West Virginia, Kanawha District	August	950,000	1.56	1.57	2.25	4,800	0.5
49	West Virginia, Kanawha District	September	830,000	16.1	1.63	2.25	24,900	3.0
50	West Virginia, New River	August	1,035,000	1.73	1.72	2.35	24,700	2.4
51	West Virginia, New River	September	1,150,000	1.76	1.79	2.35	25,300	2.2
52	West Virginia, Thacker and Kenova	August	300,000	1.62	1.65	2.30		
53	West Virginia, Thacker and Kenova	September	270,000	1.57	1.60	2.30		
54	West Virginia, Fairmont	August	985,000	1.54	1.59	2.15	4,900	0.5
55	West Virginia, Fairmont	September	910,000	1.58	1.60	2.15		
57	Kansas, Cherokee and Crawford	August	475,000	2.16	2.21	2.75	3,561	0.7
		September	475,000	2.16	2.21	2.75	3,561	0.7
58	Missouri, District No. 1	August	240,000	2.03	2.09	2.75	2,400	1.0
		September	240,000	2.03	2.09	2.75	2,400	1.0
59	Missouri, District No. 2	August	130,000	2.47	2.47	3.20	900	0.7
		September	130,000	2.47	2.47	3.20	900	0.7

63	Indiana	August	1,770,000	1.58	1.61	2.00	35,540	2.0
		September	1,770,000	1.58	1.61	2.00	35,540	2.0
66	Ohio and West Virginia—Pomeroy	August	94,000	2.15	1.92	2.45	400	0.4
		September	94,000	2.15	1.92	2.45	400	0.4
68	North Dakota lignite	August	40,000	1.47	1.55	2.10	1,608	4.0
	South of twelfth standard parallel	September	40,000	1.47	1.55	2.10	1,608	4.0
69	North Dakota lignite	August	17,000	2.38	2.17	2.65		
	North twelfth standard parallel	September	17,000	2.38	2.17	2.65		
70	Kentucky, western	August	658,000	1.43	1.46	2.00	1,645	0.3
		September	658,000	1.43	1.46	2.00	1,645	0.3
71	Texas lignite	August	80,000	0.83	1.03	1.65		
		September	80,000	0.83	1.03	1.65		
72	Texas, District No. 1, bituminous	August	12,000	3.38	3.35	4.35		
		September	12,000	3.38	3.35	4.35		
73	Texas, District No. 2, bituminous	August	73,000	2.62	2.71	3.50		
		September	73,000	2.62	2.71	3.50		
74	Ohio, Districts Nos. 2 and 7	August	165,000	2.38	2.30	3.10	1,236	0.7
		September	165,000	2.38	2.30	3.10	1,236	0.7
75	Ohio, District No. 5	August	360,000	1.61	1.65	2.35	1,800	0.5
		September	360,000	1.61	1.65	2.35	1,800	0.5
	Ohio, District No. 3	August	591,000	1.61	1.65	2.20	14,764	2.5
		September	591,000	1.61	1.65	2.20	14,764	2.5
76	Ohio, Districts Nos. 4 and 6	August	296,000	2.11	2.02	2.60	5,920	2.0
		September	296,000	2.11	2.02	2.60	5,920	2.0
77	Alabama, District No. 1	August	380,000	1.47	1.48	2.10		
		September	380,000	1.47	1.48	2.10		

SUMMARY SHOWING REPORTED AND ADJUSTED COSTS, PRICES FIXED, TONNAGE AND WEIGHTED AVERAGES TO AUGUST 12, 1918—*Continued*

No.	State or District	Month	Monthly Tons	Average Costs		Price Fixed	Tons Above Price Fixed	Per Cent Above Price Fixed
				Reported	Adjusted			
78	Alabama, District No. 2	August	185,000	2.49	2.47	3.10	12,900	7.0
		September	185,000	2.49	2.47	3.10	12,900	7.0
79	Alabama, District No. 3	August	790,000	1.87	1.86	2.50	3,200	0.4
		September	790,000	1.87	1.86	2.50	3,200	0.4
80	Alabama, District No. 4	August	90,000	1.86	1.96	2.50		
		September	90,000	1.86	1.96	2.50		
	Totals and weighted averages	....	74,714,000	1.696	1.706	2.262	1,197,400	1.603
	Less reduction of May 24, 1918	....	....	......	......	.10	....	....
			74,714,000	1.696	1.706	2.162	1,197,400	1.603

industry as a whole no excessive price allowances were given. If prices had been fixed at a point high enough to even cover the highest costs reported in each district, the result would have been to add over a billion dollars to the price paid for coal,

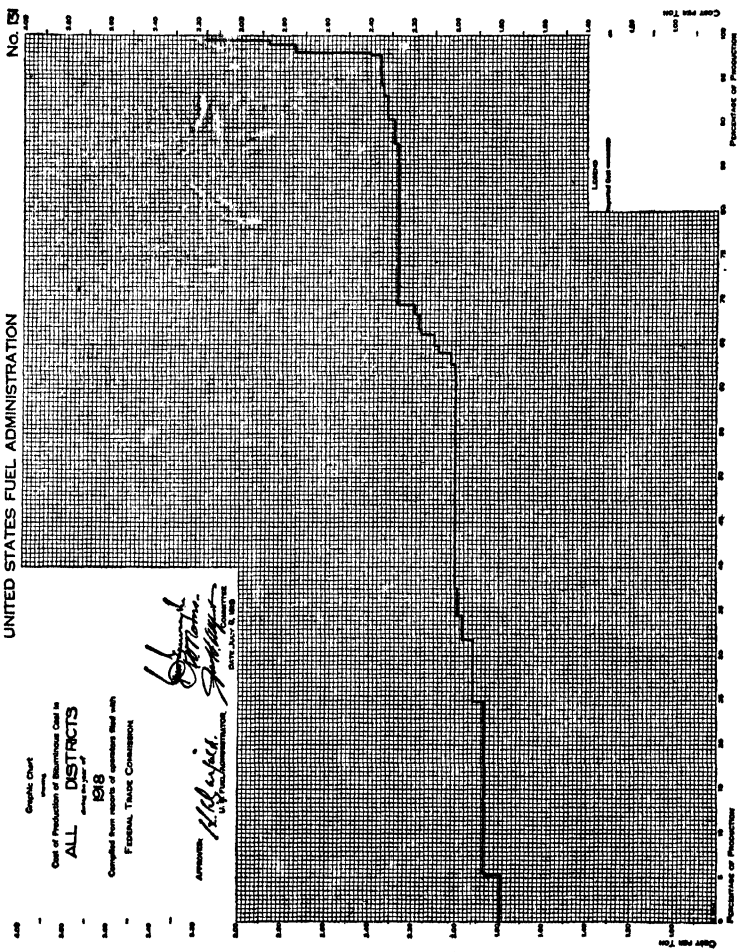


FIG. 2.—Reported costs and prices fixed by the Fuel Administration for all bituminous districts for the year 1918.

with probable labor disturbances, in an effort to obtain some of the abnormal profits which would have gone to the great majority of the tonnage, so serious as to have probably decreased rather than increased the tonnage, which was in fact ample for all the needs of the country.

The prices fixed from this complete investigation of costs have shown in many cases a remarkable compliance with economic laws. For instance, in Illinois the cost of coal from the different price districts delivered in Chicago was found to be practically identical, showing that the mining of the higher-cost coal is due to its proximity to the principal market and the lower resulting transportation costs. High-grade coal shipped by lake and rail to Minneapolis was found to cost precisely the same per heat unit as a lower-grade coal shipped a much less distance all rail.

Anthracite prices as fixed by the President Aug. 23, 1917, with an adjustment for the labor increase of Dec. 1, 1917, were the subject of an intensive study by the committee immediately after the first charting of bituminous costs was completed.

A technical paper giving the methods adopted and the results of this analysis was presented before the American Institute of Mining Engineers, February, 1919, and the following is abstracted from this paper:

The adjustments of cost from a reported to a price-fixing basis, as described for the bituminous methods, were applied but showed only minor adjustments as necessary.

The great spread in anthracite prices on the varying sizes, which for the 6-month period under review ranged in average from \$5.244 for nut to \$2.074 for barley coal, makes the question of the percentage of sizes produced at the different collieries a vital one. The realization with the same prices for each size must be within very wide limits, when it is considered that the percentage of prepared coal reported from different collieries varied from over 80 per cent to below 30 per cent for fresh-mined coal. Hence, as the spread in prices for the various sizes must be predicated on some percentage, it is essential to find some method of adjustment to allow for this variation. The logical method of adjustment is to calculate actual costs to costs as of the standard percentage of sizes, so that the margin between the adjusted costs and the average realization shall be the actual margin for each colliery between its actual costs and actual realization due to its particular percentage of sizes. As a base for realization the actual percentage of sizes for fresh-mined coal for the 6 month period was adopted. This percentage is given in the following table.

PERCENTAGE OF SIZE OF FRESH-MINED COAL

Size of Coal	Mesh, in Inches: Through, Round		Over, Round		Percentage of Sizes: Fresh-mined	Washery	Fresh-mined and Washery
Broken		$4\frac{1}{2}$	$3\frac{3}{8}$	$3\frac{1}{4}$	6.8	0.4	6.2
Egg	$3\frac{3}{8}$	$3\frac{1}{4}$	$2\frac{5}{16}$	$2\frac{1}{4}$	14.6	1.2	13.5
Stove	$2\frac{5}{16}$	$2\frac{1}{4}$	$1\frac{5}{8}$	$1\frac{1}{2}$	19.6	2.3	18.2
Nut	$1\frac{5}{8}$	$1\frac{1}{2}$		$\frac{3}{4}$	24.7	10.1	23.5
Pea		$\frac{3}{4}$		$\frac{1}{2}$	9.1	10.0	9.2
Buckwheat		$\frac{1}{2}$	$\frac{5}{16}$	$\frac{1}{4}$	11.6	21.4	12.4
Rice	$\frac{5}{16}$	$\frac{1}{4}$	$\frac{3}{16}$	$\frac{5}{32}$	3.2	14.9	4.2
Barley	$\frac{3}{16}$	$\frac{5}{32}$	$\frac{3}{32}$	$\frac{1}{16}$	4.9	27.5	6.8
Boiler	$\frac{5}{16}$	$\frac{1}{4}$	$\frac{3}{32}$	$\frac{1}{16}$	3.9	8.8	4.3
Screenings	$\frac{3}{32}$	$\frac{1}{16}$	...	...	1.6	3.4	1.7

For adjustment as a base for fixing a spread of prices the percentages used were, taken at even figures, prepared 65 per cent, pea 9 per cent, buckwheat 12 per cent, and smaller 14 per cent.

The adjustment finally arrived at after long study was tested on actual reports from collieries having percentages that varied from over 80 per cent to under 30 per cent prepared coal and was found to be correct within a maximum variation of less than 1½ per cent. It was as follows:

For Each 1 Per Cent Variation	Above Standard, Per Cent Deduction	Below Standard, Per Cent Addition
Prepared	1.20	1.20
Pea	0.85	0.85
Buckwheat	0.75	0.75
Smaller	0.50	0.50

As examples of the working of this adjustment with prices assumed at about the average for the 6 months and taking mines well away from average percentage of sizes, the following may be cited:

Size	Base Sizes, Per Cent	Base Price	Realization	Mine A Sizes, Per Cent	Correction, Per Cent	Actual Realization	Mine B Sizes, Per Cent	Correction, Per Cent	Realization
Prepared	65	$5.10	$3.315	73.1	−9.72	$3.730	55.1	+11.880	$2.810
Pea	9	3.70	0.333	6.4	+2 21	0.237	15.3	− 5.355	0.566
Buckwheat	12	3.20	0.384	10.4	+1.20	0.333	13.7	− 1.275	0.438
Smaller	14	2.20	0 308	10.1	+1.95	0.222	15.9	− 0.950	0.350
Total	100	....	$4.340	100.0	−4.36	$4 522	100.0	+ 4.30	$4.164

Assume cost for each mine	$4.000		$4.000
Actual margin	0.522		0.164
Standard realization	$4.340		$4.340
Calculated cost as of standard per cent sizes, $4×0.9564% =	3.826	$4×104.30% =	4.172
Calculated margin	$0.514		$0.168

The correction for mine A is then —4.36 per cent and the adjusted cost $3.826, showing 51.4c. margin on the $4.34 standard realization against 52.2c. actual margin. Similarly, for mine B, the correction is +4.30 per cent, giving an adjusted cost of $4.172 and a margin of 16.8c. as compared with the actual margin of 16.4c. Thus the adjusted costs on the chart bear a true relation to the realization received from a scale of prices for the various sizes based on the standard or average percentage of sizes adopted as a base, regardless of the actual percentage of sizes produced by each operation, and prices can be fixed from the chart line of adjusted costs which will result in giving each mine its intended margin. The correction, of course, is an allocation based on realization from the different sizes and could be made more accurately by taking into account each size produced, but at the cost of more time than was available for the work. With a material variation in price, different factors of correction should be calculated.

A large percentage of the anthracite coal is owned in fee by operators, who also lease tracts contiguous to their fee holdings. As all report royalties on the basis of tonnage produced, the general average 15.5c. per ton reported is misleading. The actual average royalty reported by operators mining generally from leased lands was 33.25c. and by those generally mining from fee lands, 5.5c. As relatively few operators mine exclusively from either class of lands, no data are available to show the actual average royalties paid, but it is believed that the present average would be approximately 40c. per ton.

A few leases, notably those made by the trustees of the Girard estate, owned by the city of Philadelphia, base the royalty payments on a percentage of the sale price of the coal at the mines instead of requiring fixed royalties. This percentage varies from 15 per cent to as high as 28 per cent of the price. As the labor war bonuses materially add to the sale price, these have resulted in excessive royalties and serious embarrassment to the operators, who were not allowed to increase the price of coal sufficiently to even fully absorb this additional labor cost and by whom the extra royalties must be paid out of already narrow margins.

Cost charts were made from averages of the 6 months, showing both the reported and the adjusted costs for standard fresh-mined white ash anthracite, both by collieries and by operating companies. As, in the prices fixed by the President Aug. 23, 1917, a differential of 75c. per ton on pea size and above, equivalent to 52.95c. per ton on all sizes, was established for the independent operators over certain companies with railroad affiliation, generally known as the "companies."

AVERAGE AND BULK LINE COSTS OF WHITE ASH COAL

Description	Costs, Averages Returned	Costs, Adjusted	Cost, 90 Per Cent Bulk Line
Excluding washery coal:			
All operations, each colliery separate	$3.85	$3.91	$4.80
All company operations, each colliery separate	3.71	3.79	4.65
All independent operations, each colliery separate	4.37	4.36	4.97
All operations, each company operating two or more collieries consolidated	3.85	3.91	4.38
Including washery coal:			
All operations, each company operating two or more collieries consolidated	3.57	3.77	4.36

AVERAGE PRICES RECEIVED FOR WHITE ASH COAL

Size	Fresh-mined Coal		Bank Coal		Total, Including Banks	
	Per Cent	Average Price	Per Cent	Average Price	Per Cent	Average Price
Broken	6.8	$4.889	0.4	$4.416	6.2	$4.886
Egg	14.6	5.028	1.2	4.815	13.5	5.027
Stove	19.6	5.161	2.3	5.060	18.2	5.160
Nut	24.7	5.244	10.1	5.246	23.5	5.244
Pea	9.1	3.687	10.0	3.696	9.2	3.698
Total and weighted average, prepared and pea	74.8	$4.959	24.0	$4.544	70.6	$4.947
Buckwheat	11.6	$3.342	21.4	$3.213	12.4	$3.324
Rice	3.2	2.482	14.9	2.452	4.2	2.473
Barley	4.9	2.231	27.5	1.767	6.8	2.074
Boiler	3.9	2.341	8.8	2.123	4.3	2.304
Screenings	1.6	2.202	3.4	1.555	1.7	2.162
Total and weighted average, small sizes	25.2	$2.795	76.0	$2.339	29.4	$2.697
Grand total	100.0	$4.414	100.0	$2.868	100.0	$4.285

The prices received by the companies and independents have not been separately averaged, but, calculating on the differential and assuming the percentages the same for companies and independents which is only approximately the case, the selling price of fresh-mined coal would average for companies \$4.287, and for independents, \$4.817. Margins over reported

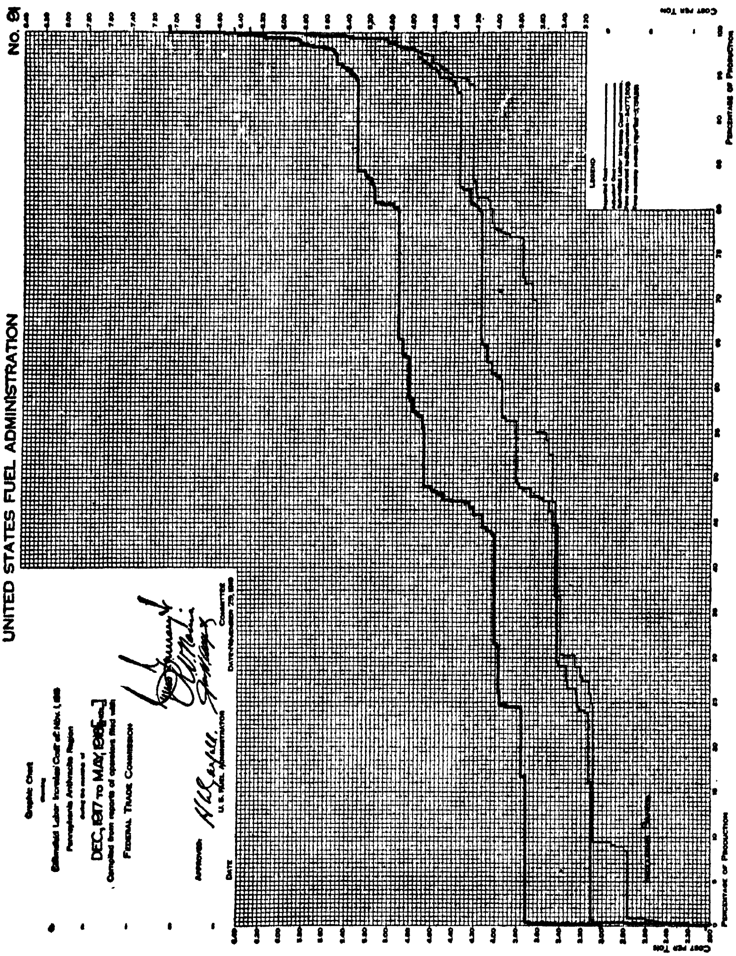


FIG. 3.—Calculated increase in the cost of anthracite after the second War Bonus, November 1, 1918.

costs of companies would be 58c., and for independents 45c., with a general average margin for all fresh-mined coal of 56c., and for all coal, including washery, of 71c. per ton, and under "bulk line" costs, fresh-mined companies, 36c.; independents, 15c.; total, 39c., including washeries consolidated sheets total of 7.5c.

These margins include all expenditures for Federal income and excess-profit taxes, selling expenses, interest charges, expenditures for improvements and developments to increase output, excess of capital expenditures over normal cost, and all profit on the investment of about $8 per ton annual output.

Effective Dec. 1, 1917, a labor war bonus, ranging from 60c. to $1.10 per day for labor and 25 per cent for contract miners was granted over and above the wage scales effective by agreement Apr. 1, 1916, expiring Apr. 1, 1920, and the prices fixed Aug. 23, 1917, and modified Oct. 1, 1917, by reducing pea coal 60c. per ton, were increased by 35c. per ton to compensate for this labor increase. The actual reported increase in labor costs due to this advance was figured by the Federal Trade Commission from the operators' reports to be 60.3c. From the actual pay-roll figures later obtained by the United States Fuel Administration, this increase was found to be 76.3c. per ton.

Effective Nov. 1, 1918, a second labor war bonus was granted. The calculated increase in cost due to this is shown in Fig. 3, on which the increases for each operator are found by figuring from the pay rolls for the 6 months the actual increase in pay which would have been given, applying the Nov. 1, 1918, increases, and dividing by the 6 months' tonnage of the colliery. This line, adjusted to per cent of sizes, and compared with the adjusted cost, shows an increase in cost of 74.1c. As this was necessarily applied to the prepared and pea sizes, 70.6 per cent of the total, the increase on those sizes was $1.05 per ton, which increase was allowed to balance the increased cost of labor.

Except for the two increases to compensate for labor increases just noted and the reduction Oct. 1, 1917, of the pea coal price, the anthracite prices are as fixed by the President on Aug. 23, 1917. The present realization, all companies and all sizes, including washery coal and both the labor increases, is calculated to average $5.13 per ton, while the bulk line of the chart shown in Fig. 3, plus the Nov., 1918, labor increase, would be $5.32.

The capital invested per ton output in the larger and better equipped collieries ranges from $5 to $11, with an average investment of from $7.50 to $8.

Prices Fixed by the President, August 23, 1917

	White Ash		Red Ash		Lykens Valley	
	Company	Independent	Company	Independent	Company	Independent
Broken	$4.55	$5.30	$4.75	$5.50	$5.00	$5.75
Egg	4.45	5.20	4.65	5.40	4.90	5.65
Stove	4.70	5.45	4.90	5.65	5.30	6.05
Chestnut	4.80	5.55	4.90	5.65	5.30	6.05
Pea	4.00	4.75	4.10	4.85	4.35	5.10

FIXED PRICES, DECEMBER 31, 1918

	White Ash		Red Ash		Lykens Valley	
	Company	Independent	Company	Independent	Company	Independent
Broken	$5.95	$6.70	$6.15	$6.90	$6.40	$7.15
Egg	5.85	6.60	6.05	6.80	6.30	7.05
Stove	6.10	6 85	6.30	7.05	6.70	7.45
Chestnut	6.20	6.95	6.30	7.05	6.70	7.45
Pea	4.80	5.55	4.90	5.75	5.15	5.90

The prices fixed by the President, Aug. 23, 1917, are given in the accompanying table. No price was fixed on sizes smaller than pea, which was decreased 60c. per ton Oct. 1, 1917. There was a general increase of 35c. per ton Dec. 1, 1917, and one of $1.05 per ton Nov. 1, 1918. Sizes smaller than pea were limited to a maximum 50c. per ton below pea coal by order of Nov. 15, 1918.

AVERAGE COST PER TON, DECEMBER, 1917, TO MAY, 1918

	Fresh-mined coal, 35,256,550 Tons	Washery Operations, 3,431,916 Tons	Total, Including Washeries, 38,688,466 Tons
Labor	$2.593	$0.687	$2.423
Supplies	0.616	0.260	0.584
Transportation, mine to breaker	0.004	0.007	0.004
Royalty, current	0.153	0.102	0.148
Royalty, advance	0.002	......	0.002
Depletion	0.099	0.077	0.097
Amortization of cost of leasehold	0.014	0.024	0.016
Depreciation	0.091	0.86	0.090
Pro rata suspended cost of stripping	0.023	......	0.021
Contract stripping and loading	0.009	......	0.009
Taxes, local	0.054	0.034	0.052
Insurance, current	0.016	0.014	0.016
Insurance, liability	0.058	0.018	0.055
Officers' salaries and expenses	0.030	0.019	0.029
Office salaries and expenses	0.048	0.024	0.045
Legal expenses	0.005	0.003	0.005
Miscellaneous	0.026	0.023	0.026
Total	$3.841	$1.378	$3.622
Increase over May to November, 1917	0.764	0.365	0.719

The present fixed prices Dec. 31, 1918, per ton of 2240 lbs. f.o.b. mines, are given in the accompanying table. Smaller than pea is not to be sold within 50c. of maximum pea-coal price. Thus the selling price of anthracite has been increased but 30.5 per cent over the pre-war price, while the cost of production has gone up 52 per cent, the difference having been absorbed by the operators.

The average cost as reported for the six months, Dec., 1917, to May, 1918, inclusive, prior to the increase of Nov. 1, 1918, but including that of Dec. 1, 1917, is given in the accompanying table.

Chief among the factors causing fluctuations in mining costs are the changes in wage scales and the tonnage produced. The labor cost per ton forms 70 to 80 per cent of the total f.o.b. mine cost. In 73 mining districts for which detailed statistics for 1918 were published by the Federal Trade Commission the distribution was as follows:

Per Cent of f.o.b. Mine Cost that goes to Labor	Number of Districts	1918 Production, Tons	Per Cent of Total Tonnage
60 to 64	1	247,000	
65 to 69	10	175,880,000	35
70 to 74	33	158,877,000	32
75 to 79	18	129,913,000	26
80 to 84	11	32,502,000	7
Totals......	73	497,419,000	100

It will be noted that the difference between the labor cost proportions from district to district are much less than between the total costs themselves, which, excluding lignite, ranged from $1.62 per ton in a West Virginia field to $4.45 in an Arkansas field.

The principal reasons for the difference in cost of the various fields are the physical conditions under which mining must be carried on, chief of which is the thickness of seam and the extent of the use of modern machinery for mining and transporting coal to the mouth of the mine, as compared with the old fashioned pick mining and mule haulage. It is a mistake, however, to try to measure the advantage one district has over another by a direct comparison of labor or even total f.o.b. mine costs. Allowance must be made for the much heavier investment necessary in the fields where machinery is used to cut down the manual labor for mining and transporting the coal.

Nor does it necessarily follow in a given field that the thicker the seam, the lower will be either the labor cost or the total f.o.b. mine cost per ton. The analyses of cost by thickness of seam mined shown in the Federal Trade Commission reports indicate that after a certain thickness of seam is reached—in most districts between five and six feet—the mining of thicker seams involves higher costs. In other words both labor and total f.o.b. mine costs per ton in a given field are likely to decrease as the thickness of seam increases from two feet up to between five and six feet, and then to rise as the thickness increases still further. Apparently in such cases it is the greater amount of labor and supplies required in timbering the thicker seams which increases the costs.

Wage scales for each field are fixed with relation to the particular mining conditions of the field. Common or uniform increases in existing wage scales, however, have widely different results in their effect on the per ton labor costs of the different fields. Thus it was that the wage increase granted in November, 1917, for which a uniform price increase of 45c. per ton was allowed, increased the cost about 28c. per ton in the Illinois mines of F. S. Peabody, according to his testimony before Senator Reed's committee. On the other hand, it has been found that this wage increase in some other fields increased the labor cost as much as 70c. per ton.

The second important factor in causing changes in the per ton cost is the fluctuation in the tonnage produced. Obviously the greater the divisor, the less per ton will be the regular upkeep expenses—whether in supplies or in general overhead charges. But also the proportion of so-called "non-productive" labor employed in the mine is so large with relation to the labor paid on a per ton basis that an increase in the production will often materially decrease the total labor cost. In fact the increase in production may be so great as to obscure, for a time, the direct effect of an increased wage scale.

There is little authentic information available as to costs prior to 1916. There is reason to believe that for the period immediately preceding the war, while costs had been gradually increasing (disregarding the effect of fluctuations in production), there was no sudden jump, the increase taking place in the wages from time to time having been relatively small as compared to total cost. The 1916 costs, therefore, can be regarded as high water mark for a number of years previous. The experience of the anthracite field, where published labor costs are available as far back as 1913, supports this conclusion. There labor costs on fresh-mined coal of operators who produced about 60,000,000 tons annually were $1.62 per gross ton in 1913, $1.62 in 1914, $1.63 in 1915 and $1.75 in 1916.

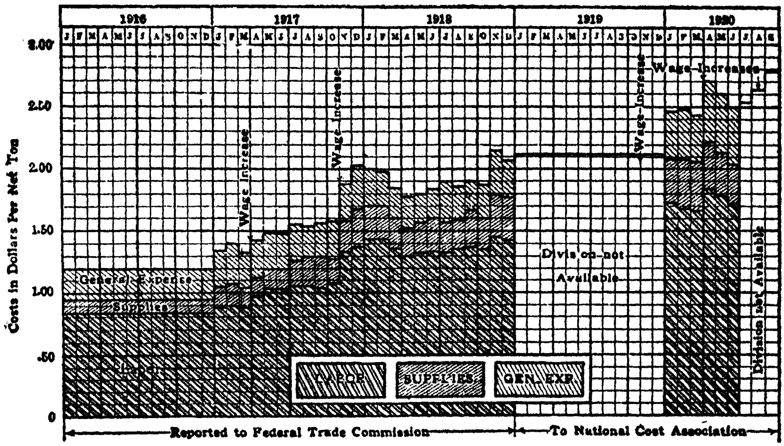


FIG. 4.—Production costs in Southwestern Pennsylvania for the years 1916 to 1920.

The accompanying charts, Figs. 4 to 9, inclusive, for all of the principal producing fields in the United States—these fields produced about 275,000,000 tons in 1918—show the rapid rise of costs since 1916 and also give some measure of the distribution of costs. The figures for 1916–1918 are taken from the Federal Trade Commission reports, those for 1919 and 1920 from the reports made by operators to the National Coal Association, and tabulated by the Senate Committee on Reconstruction ("Calder committee"). The allocation of these costs to labor, supplies and general expense for January-June, 1920, has been compared on the basis of the distribution shown in the Federal Trade Commission bulletins which covered the first half of 1920. The fields or districts are those established by the Engineer Committee of the Fuel Administration, and are defined as follows:

(1) *Southwest Field, Pennsylvania:* The counties of Allegheny, Westmoreland, Fayette, Greene and Washington, in the State of Pennsylvania, except (1) that portion of Allegheny County from the lower end of Taren-

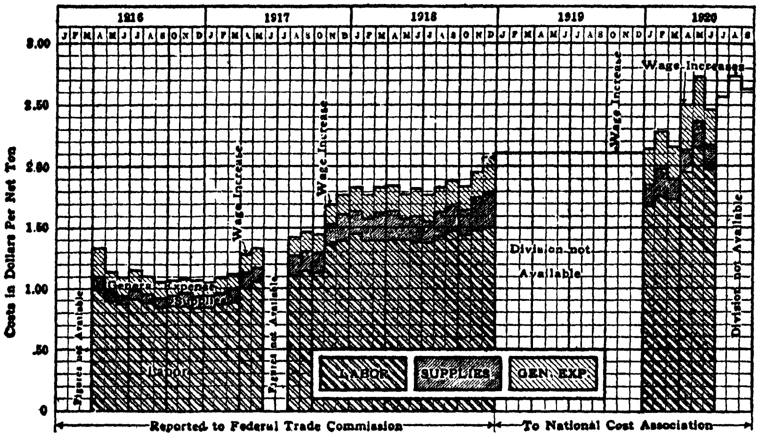


FIG. 5.—Production costs in the Indiana district for the years 1916 to 1920.

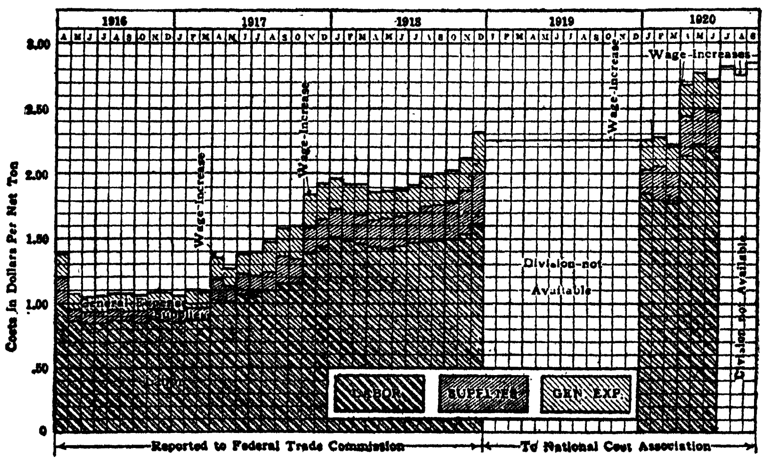


FIG. 6.—Production costs in the Illinois No. 6 district for the years 1916 to 1920

tum Borough north of the county line; (2) the territory in Westmoreland County from a point opposite the lower end of Tarentum Borough, north along the Allegheny River to the Kiskiminitas River, along the Kiskiminitas River eastward to the Conemaugh River, and continuing along the Conemaugh River to the county line of Cambria County; (3) operations on Indian Creek in Westmoreland County; and (4) operations in the Ohio Pyle district of Fayette County. See Fig. 4.

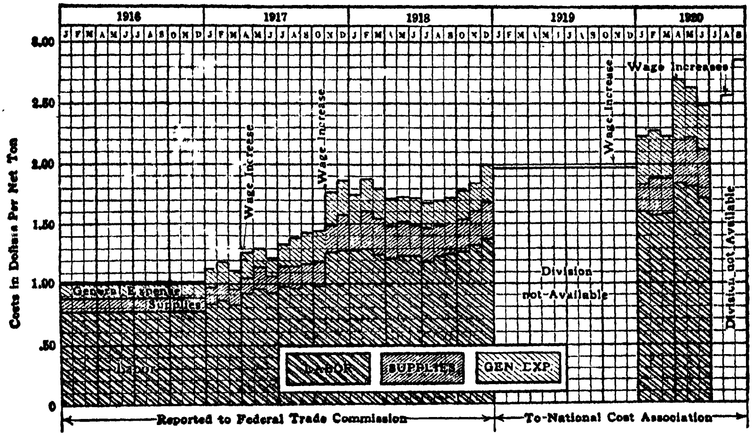


FIG. 7.—Production costs in the Ohio No. 8 district for the years 1916 to 1920.

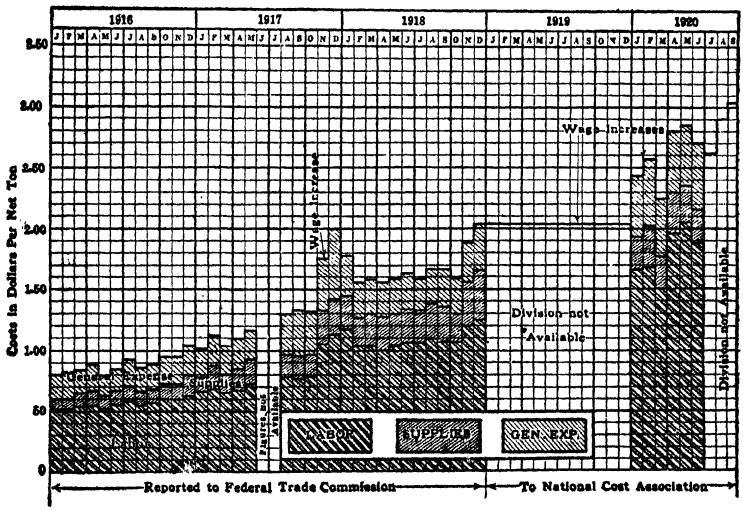


FIG. 8.—Production costs in the Pocahontas Field for the years 1916 to 1920.

(2) *Central Field, Pennsylvania:* The counties of Tioga, Lycoming, Clinton, Center, Huntingdon, Bedford, Cameron, Elk, Clearfield, Cambria,

Blair, Somerset, Jefferson, Indiana, Clarion, Armstrong, Butler, Mercer, Lawrence and Beaver, and operations in Allegheny County from the lower end of Tarentum Borough north to the county line, and in Westmoreland County from a point opposite the lower end of Tarentum Borough north along the Allegheny River to the Kiskiminitas River and along the Kiskiminitas River eastward to the county line of Cambria County, operations on the Baltimore & Ohio R.R. from the Somerset County line to and including Indian Creek and the Indian Creek Valley branch of the Baltimore & Ohio R.R. See Fig. 9.

(3) *Pocahontas Field, West Virginia:* Operations on the Norfolk & Western Ry. and branches west of Graham, Va., to Welch, W. Va., including Newhall, Berwind, Canebrake, Hartwell and Beech Fork branches; also operations on the Virginian R.R. and branches, west of Rock to Herndon, W. Va. See Fig. 8.

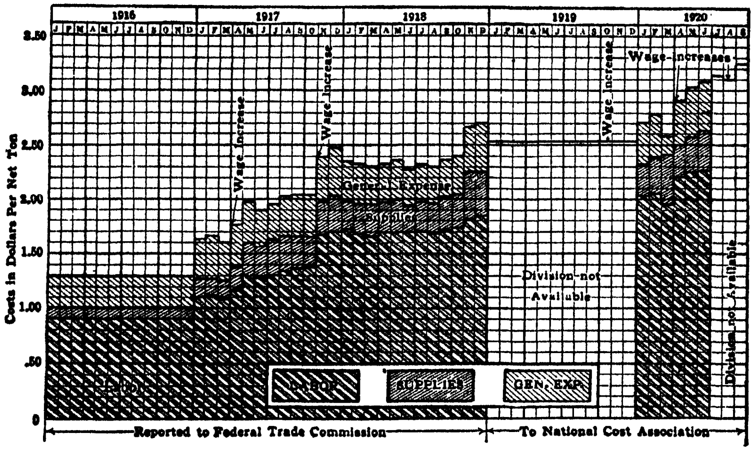


FIG. 9.—Production costs in the Central Pennsylvania district for the years 1916 to 1920.

(4) *District No. 8, Ohio:* The County of Monroe, the County of Belmont, except the township of Warren and operations in the 8-A vein in Flushing and Union Townships, the County of Harrison except the townships of Monroe, Franklin, Washington and Freeport, and the County of Jefferson except the townships of Brush Creek, Saline, Ross, Knox and Springfield. See Fig. 7.

(5) *Bituminous Field, Indiana:* Coal mined in the State of Indiana other than Brazil Block coal. See Fig. 5.

(6) *District No. 6, Illinois:* Including Marion, Jefferson, Franklin, Williamson, Johnson, Hamilton, Saline, White, Gallatin, and mines along the main line of the Illinois Central R.R. between Vandalia and Carbondale in Clinton, Washington, Perry and Jackson Counties. See Fig. 6.

The charts show clearly what has happened to costs since 1916 in some of the principal fields of the United States. The increases in these fields, based on the 1916 cost, are as follows:

COSTS PER NET TON

(Federal Trade Commission Figures Used Exclusively)

	Pennsylvania				West Virginia			
	Southwest		Central		Pocahontas		New River	
	Labor	F.o.b. Mine	Labor	F.o.b. Mine	Labor	F.o.b. Mine	Labor	F.o.b. Mine
1916 (base)........	$0.82	$1.19	$0.92	$1.32	$0.56	$0.87	$0.74	$1.00
Jan.–March, 1920...	1.50	2.13	1.97	2.56	1.31	1.89	1.79	2.44
April–June, 1920..	1.88	2.62	2.17	2.82	1.51	2.11	1.85	2.39
Per Cent of Increases over 1916								
Jan.–March, 1920 ..	84	179	104	94	138	118	142	124
April–June, 1920...	130	120	136	114	175	143	150	120

COSTS PER NET TON

	Ohio				Indiana		Illinois			
	No. 1		No. 8		Bituminous		No. 3		No. 6	
	Labor	F.o.b. Mine	Labor	F.o.b. Mine	Labor	F.o.b. Mine	Labor	F.o.b. Mine	Labor	F.o.b. Mine
1916 (base)....	$0.84	$1.17	$0.78	$1 02	$0.87*	$1.09	$0.89†	$1.10†	$0.86*	$1.07*
Jan.–Mar., 1920	1.65	2.27	1.54	2.14	1.63	2.00	1.57	1.96	1.63	1.98
Apr.–June, 1920	1.64	2.09	1.79	2.47	1.94	2.41	1.77	2.21	1.85	2.29
Per Cent of Increases over 1916										
Jan.–Mar., 1920	97	94	97	110	87	83	77	78	89	85
Apr.–June, 1920	96	78	130	142	123	121	100	101	115	114

* April–December, 1916.
† July–December, 1916.

In the foregoing table the 1920 figures, while not strictly comparable because not obtained from the same operators as the 1916 figures, are probably representative enough to show in a general way the change in conditions since 1916. If the September, 1920, total cost figures of the Calder committee be compared with the 1916 total f.o.b. mine cost, the increases shown would be yet more marked, as the effect of the wage increase late in the summer of 1920 was to increase costs. Such increase, however, cannot be as exactly measured because the 1916 figures are "revised" costs and exclude selling expense, while the 1920 figures are "reported" costs, and include selling expense, etc.

As the events of the past few years have shown, labor in coal mines, just as on railroads, holds the strategic advantage of being able to tie up the whole country through an effective strike, it is not likely that costs will be materially lessened through any immediate reduction of wages. On the other hand, the necessary writing off of some of the heavy investment charges caused by high cost development during the past few years, in order to get the investment down to present day values, will in many cases increase the overhead charges.

Method of computing tax returns.—The intent of the law, according to an article on this subject in Black Diamond, is clearly that the cost of the coal in the ground shall be considered as part of the cost of the same coal when removed and sold. The regulations fail to carry out that intent, because of the unwarranted assumption that all the tons of coal in the mine cost the same amount per ton. This is what gives the Treasury Department's method its simplicity, but at the same time robs it of its reasonableness, because the assumption is contrary to fact.

It is self-evident that a ton of coal near the surface or exposed by present workings is worth much more than a ton buried far down in the earth that cannot be removed for many years. If no improvements in mining methods were expected, the value of deeply buried coal would be further reduced by the greater expense that would be required to bring it to the surface. There is a possibility, of course, that improvements in methods may keep pace with the difficulties encountered in the majority of cases.

But whether or not inventions may be expected to offset in some measure the increasing difficulties of greater depth, there is nothing to compensate for the time element. And the time element is always an important factor. No extensive deposit, whether coal or ore or other minerals, can be mined in a day or a year. Almost invariably before a deal in mining properties is consummated the purchaser has, through the aid of specialists, made exhaustive studies as to the extent and quality of the deposit, and the most economical methods of its exploitation.

It would not be economical to mine one ton a year, and it would ordinarily be impossible to mine the whole deposit in a year. But a plan is adopted between these extremes usually based on the annual production of a certain definite quantity, and the equipment and machinery to be provided in order to maintain that output is elaborated. Ordinarily the purchaser not only has these plans, but has concrete evidence of them that would convince any fair-minded and disinterested person that in purchasing the property the price he was willing to pay was based upon these engineering reports. And in arriving at that price he always does, either mathematically or intuitively, take into consideration the time that must elapse before his investment can be realized in cash by operations.

Let us suppose that the mine is known to contain 1,000,000 tons of coal, and that the plans call for the mining of 20,000 tons per annum, indicating a life of 50 yr. for the mine, and that with these facts in mind the coal lands are purchased for $100,000. This is the cost of the entire deposit. It indicates the average cost per ton is 10c., but although the average is 10c., that figure does not apply to the tons near the mouth nor to those deeply buried.

To determine the cost of the several tons let us indicate by V the value of a ton exposed and minable to-day, and assume an interest rate of 6 per cent, and, to avoid unnecessary intricacies of computation, let us further assume that each year's production comes at the end of the year.

The cost of the coal to be mined the first year would be

$$20{,}000V\left\{\frac{1}{1.06}\right\}.$$

The cost of the coal to be mined the second year would be

$$20{,}000V\left\{\frac{1}{(1.06)^2}\right\}.$$

The cost of the coal to be mined the third year would be

$$20{,}000V\left\{\frac{1}{(1.06)^3}\right\},\ \text{etc.}$$

The sum of this series for 50 terms constitutes the cost of the entire deposit. Therefore:

$$\$100{,}000 = 20{,}000V\left\{\frac{1}{1.06}+\frac{1}{(1.06)^2}+\frac{1}{(1.06)^3}+\ \ldots\ +\frac{1}{(1.06)^{50}}\right\}.$$

It is at once seen that the series in the bracket is equivalent to the present value of an annuity of \$1 for 50-yr. at 6 per cent, which is readily computed at \$15.761. Our equation is therefore reduced to:

$$\$100{,}000 = 20{,}000V(15.761).$$

Solving for V, we find that the cost of one ton minable to-day is 31.72c.

From this the cost of the 20,000 tons to be mined each year may be computed as follows:

Tonnage		Cost per Ton	Cost of 20,000 Tons
1st 20,000 tons	\$0.3172 ÷ 1.06	= \$0.2992	\$5984
2nd 20,000 tons	.3172 ÷ $(1.06)^2$	= 0.2823	5646
3rd 20,000 tons	.3172 ÷ $(1.06)^3$	= 0.2865	5326
4th 20,000 tons	.3172 ÷ $(1.06)^4$	= 0.2512	5024
5th 20,000 tons	.3172 ÷ $(1.06)^5$	= 0.2370	4740
10th 20,000 tons	.3172 ÷ $(1.06)^{10}$	= 0.1772	3544
15th 20,000 rons	.3172 ÷ $(1.06)^{15}$	= 0.1324	2648
20th 20,000 tons	.3172 ÷ $(1.06)^{20}$	= 0.09897	1979
30th 20,000 tons	.3172 ÷ $(1.06)^{30}$	= 0.05528	1106
40th 20,000 tons	.3172 ÷ $(1.06)^{40}$	= 0.03088	618
50th 20,000 tons	.3172 ÷ $(1.06)^{50}$	= 0.01725	345

If this table were filled in complete for the 50 yr. the last column would total up to \$100,000, which is the cost of the entire deposit.

If an interest rate of 8 per cent were adopted the cost of a ton minable to-day would be 40.87c., and the cost of the respective groups of 20,000 tons would range from 37.84 to 0.871c. per ton.

Other properties besides mines are acquired for lump sums, and it is conceded in those cases that the purchaser has the right and the duty to analyze his cost and set up in separate accounts a fair apportionment of it. Thus a taxpayer may buy for a lump sum a going store with all the assets and liabilities attached to it. He is expected to apportion this cost and set up separately in his books, the land, buildings, furniture, merchandise, accounts receivable, good will, accounts payable, etc. The apportionment must be fair, and the net total must agree with the aggregate cost.

As another example a merchant buys a shipment of miscellaneous hides for a lump sum. He then sorts them into numerous grades. The best hides may be worth many times as much as the poorest ones. He is expected to apportion the cost on the basis of quality and value. The apportionment must be fair and the total of the costs thus allocated to the several grades must agree with the aggregate cost. If the hide merchant sold all the poorer grades, but had the best hides on inventory at the end of the year, and priced them at the average cost, there is little doubt but that the Treasury Department would compel him to use a higher figure and would assess an additional tax.

With the mining company the situation is reversed. In the nature of its operations it mines and sells first the most accessible coal—the coal that really cost it most—and is then asked to carry the less accessible coal on its balance sheet at the average cost. The law permits a reasonable allowance for depletion, and any mining company that makes a reasonable and fair apportionment of the cost of its mineral deposits should be accorded the same fair treatment that is accorded to the merchant of hides.

Comparison of costs and distribution of revenue of German and Middlewestern mines.—A study of mining costs and the distribution of revenue at the mines of the Westphalian Syndicate in Germany as compared with the mines of Illinois and Indiana throws some interesting light on these questions.

The mine operators of the Westphalian district in Germany suffered from severe competition resulting from overproduction, and various efforts were made to find relief as early as 1850. Price agreements, which were forbidden by the German law, were disregarded, notwithstanding the heavy penalties imposed for violations. Finally in 1885 the Westphalian Syndicate was established, and continues to the present date. It is a selling organization without any property and only a nominal working capital.

Its affairs are administered by an official who has no financial interest in the mines and acts as chairman of a board made up of one representative from each participating company. The function of the syndicate is to sell the product of the mines, coke ovens and briquetting plants and to allot to each company the tonnage which it should produce.

Twice each year an estimate of the probable requirements is made, and a tonnage is allotted to each company based upon previous production after allowance has been made for the ton-

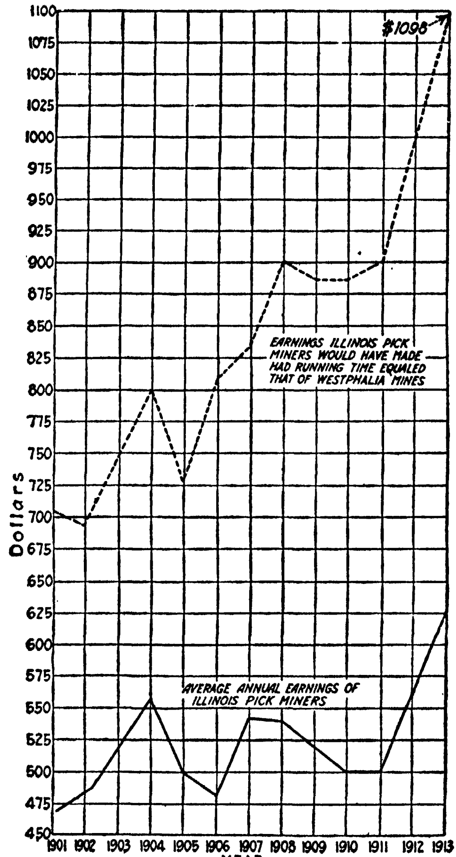


Fig. 10.—Wage losses to miners due to intermittent work.

nage of companies, such as railroad, furnace and other such corporations, which consume a part of their own production.

On May 1 each company is notified how much coal it will be called upon to furnish during the second half of the calendar year, and each mine can make its arrangements for the most economical production of the tonnage called for. Any company falling short in its supply, if market conditions continue as anticipated, must pay damages for the shortage unless the deficit can be made up by another company.

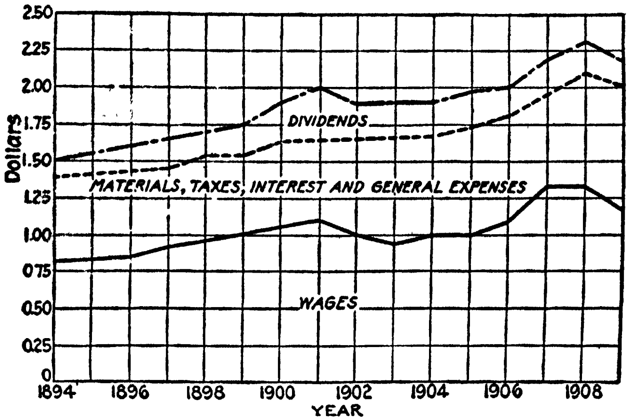


FIG. 11.—Distribution of gross revenue at Westphalia mines.

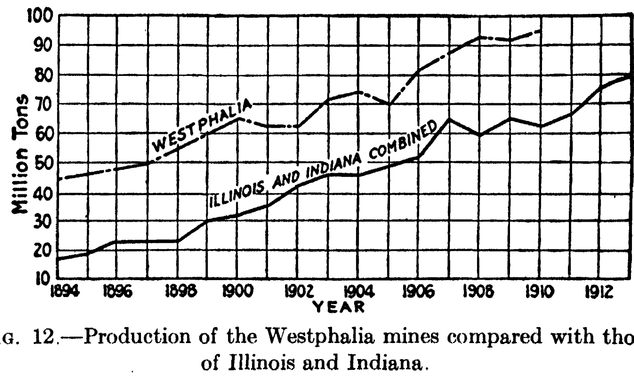


FIG. 12.—Production of the Westphalia mines compared with those of Illinois and Indiana.

Losses due to inferior preparations are borne by the company responsible for the defect. Prices are agreed upon and fixed in advance semi-annually and take into account the quality of coal produced from each mine, making it immaterial to the purchaser where the coal comes from, because of the adjustment of price to the intrinsic value of the material sold.

It has sometimes happened that by some unforeseen condition the syndicate was not able to market through its ordinary trade channels the estimated quantities of coal, and other markets had to be entered in order to permit the mines to operate under the most economical conditions. Losses due to these difficulties are borne alike by all, the syndicate paying to the participants the price agreed upon, having retained a commission, from which all deficits are paid.

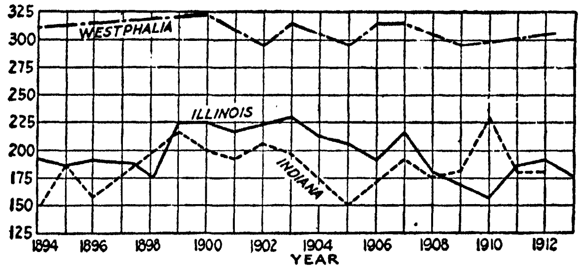


FIG. 13.—Comparison of the average number of shifts worked per annum at the Westphalia and Middle Western mines.

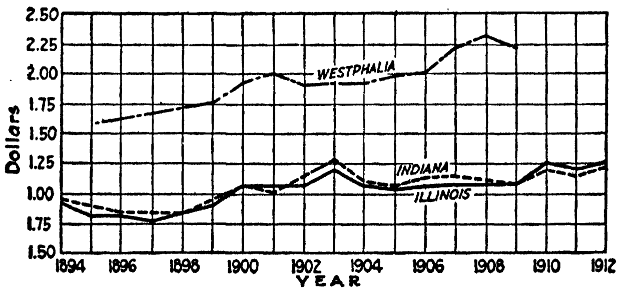


FIG. 14.—Average price per ton of coal at Tipple, Westphalia and Middle Western mines.

The advantages of a single seller marketing 50,000,000 tons of coal a year are apparent. Markets are available to the syndicate which individual operators could not reach. Its contracts are made for five-year periods, and this assures an income to the operators and enables them to finance their properties and engage in business which, while more remunerative, requires larger investments. Thus they have erected large coking, by-product and briquetting plants. Such financing would be impossible with the uncertainties of ordinary competition.

The higher returns have made possible an expenditure of money for improved equipment, safety measures and labor-saving devices quite unknown in this country. Complete extraction of coal is required by the government, and it is estimated that the cost of flushing to sustain overlying strata and to permit of the removal of all coal adds 25c. per ton to the production cost.

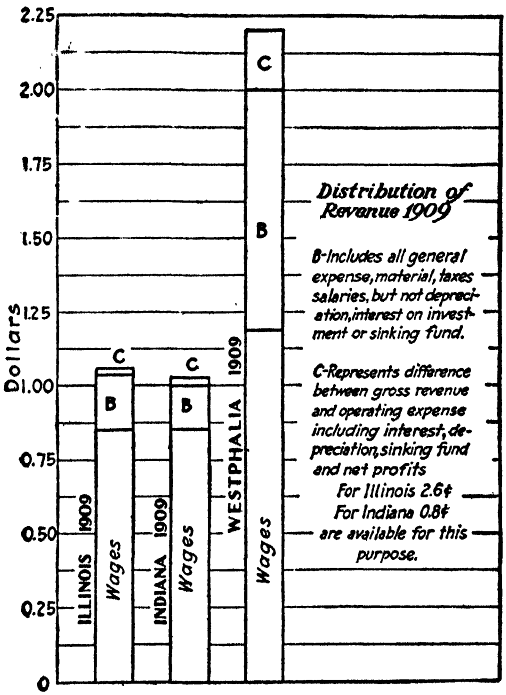


FIG. 15.—Comparison of revenue at Illinois, Indiana and Westphalia mines.

The coal operators are enabled to provide funds for the protection of the injured employees and for the support of the families of those fatally injured. They also provide pensions for the incapacitated and the aged. The cost of this social insurance in 1909 added 20c. per ton to the cost of production.

No protest has been made by the consumer against the higher coal prices which have followed the establishment of the syndicate. The increase in price has been generally accepted as the best expedient for solving a most vexatious question. Undoubtedly it induced more care and economy in the use of coal and resulted in the adoption of more economical engines and improved boiler settings.

The Westphalia production increased from 1,665,000 in 1850 to 81,000,000 tons in 1907; at the same time the number of companies was reduced from 100 to 76, indicating growth of individual companies and concentration of capital. The 17 companies in the syndicate the output of which was sold for commercial use and which were not allied with the fuel-consuming industries had an aggregate annual production of 28,000,000 tons and a capitalization of $72,450,000 which is an average of $4,200,000 each.

This indicates an investment for plant and equipment of $2.50 per ton of annual production. The capital account does not include any outpay for coal land, as all the coal belongs to the government. For Illinois the capital invested in 1909 was $1.49 and in Indiana $2.44 per ton of annual production. This latter, however, includes the coal rights, which represent the major portion of the investment.

The accompanying diagrams, Figs. 10 to 15, inclusive, show graphically the points developed in this discussion.

Conditions where operations may be conducted at an apparent loss.—Maintenance charges for drainage, timbering, ventilation taxes, etc., are so heavy at mines that it may be more economical to continue operations in the face of an apparent loss than to shut down the mines entirely. Particularly is this the case with older mines having long underground hauls and high pumping heads. Some interesting data on this subject will be found in the report of a committee appointed to investigate the receivership of a prominent coal corporation about 1911.

This committee found that if the coal properties were shut down, the annual loss will be $420,000. If they were operated at the standardized cost per ton of $0.857 and for an output of 3,000,000 tons and the coal sold at the price realized the previous year, $0.8097, the loss will be $141,900. The standard cost includes a charge for interest of $0.067 and for depreciation of $0.058, a total of $0.125 per ton. The standard costs are 14.8 per cent lower than 1909–10 corresponding costs, 17.4 per cent lower than July and August, 1910, corresponding costs.

This short report was amplified into the following:

The coal lands have been injudiciously acquired.

Money has been injudiciously spent in equipping the plants. Overhead charges for interest, maintenance and depreciation are therefore high.

The current market selling price for coal was so low as to make profitable coal mining difficult, if not impossible, even if the coal lands had been secured without price, and had been equipped with rigid reference to economical operation.

To shut down the mines and wait for better prices would entail an annual expense for power, maintenance, supervision, depreciation and interest of $420,000. This does not include an annual charge of $104,494 on book value of coal lands not immediately identified with the plants to be operated.

The cost of mining coal if operations are standardized, will be $0.857 per ton for a daily output of 12,000 tons, a monthly output of 250,000 tons and a yearly output of 3,000,000 tons.

The loss from continued operation will depend on the price obtained for coal sold as follows:

At $0.66 loss will amount to	$561,000
At $0.70 loss will amount to	420,000
At $0.70 loss from operations and loss from suspension of operations will be equal.	
At $0.79 loss will amount to	200,000
At $0.8097, price netted by coal sales in 1909–10, loss from operation will be	141,900
At $0.857 there is neither loss nor profit from operation.	
At $0.921, profit above operation	192.000
This is sufficient to pay interest on obligation. Coal should therefore continue to be mined.	
At $0.948, profit from operation	272,0000
This pays for operation, for moneys owed and for present administration charges.	

While waiting for better coal prices, costs of operations were to be standardized as follows:

By revaluing all the lands and equipment, thus reducing future operating overhead charges.

By putting the management in the hands of a competent and experienced man of reliable character.

By concentrating operations at that plant, or those plants, where coal could be mined most cheaply.

By investigating the advantages, if any, to be derived from coking the product of these mines.

By investigating the advantages, if any, of establishing a washery at the mines.

In making its investigations the committee attempted to determine a standard cost per ton of mined coal for a standard output, which was assumed at 3,000,000 tons each year. The standards adopted for immediate use were, per ton:

The existing wage scale for mining labor, $0.485.

Current rates of wages for a minimum amount of other efficient working labor, $0.175.

Moneys for supervision, supplies and other bills, taxes, insurance, etc.; an efficient minimum, $0.07.

Depreciation charges based on revaluations, on experience, and on the present ascertained coal reserve tributary to operating plants, $0.06.

Interest at 6 per cent per annum on reappraised values of coal reserves, mining buildings, equipment, etc., actually used for mining operations, $0.067.

Other expenses not standard and not directly appertaining to mining operations were:

Interest and other charges on investments at present inoperative, $0.029.

Excessive interest load, due partly to investment in elaborate and unnecessary plants, partly to deficits accumulated from former years, and partly to other causes, $0.035.

High costs of administration of the company's business.

Costs for 1910

Operation	$77,294	
Maintenance	14,156	
General expense, excluding insurance	37,912	
	$129,362	
Less allowance for mining operation	48,000	
	$81,362	
Cost per ton		$0.0271

The output of coal can fluctuate from no tonnage, if the mines are closed, to a maximum daily tonnage of 17,000 tons.

If this maximum of 17,000 tons daily could be attained it would reduce mining costs about as follows:

OUTPUT PER YEAR, 4,250,000 TONS

	Costs per Ton
Mining labor	$0.455
Other labor	0.15
Operation	0.06
Depreciation	0.06
Interest	0.045
Total	$0.77

TABLE ON BASIS OF 3,000,000 TONS ANNUALLY

Daily Output, 12,000 Tons

	Costs per Ton	
1. Mining labor	$0.485	
2. Other labor	0.175	
3. Total working pay-roll (1 and 2)		$0.66
4. Operations	$0.07	
5. Depreciation	0.06	
6. Interest	0.067	
7. Total overhead charge (4, 5, 6)		$0.197
8. Total standard cost per ton of coal (3 and 7)		$0.857

Systems of mining.—A thorough grasp of the economics of the various systems of mining can be obtained best by a brief review of the various stages of development that have lead up to the adoption of the systems now in use. A study of the faults that were found in these older methods and the remedies that were applied to overcome the difficulties is of prime importance. The changes in the systems of mining in the Georges Creek field, one of the oldest bituminous districts in the country, was described in a paper presented before the West Virginia Mining Institute in 1908, from which the following has been excerpted, disregarding the methods that were used prior to 1870 when there was apparently little attempt at systematized effort.

Fig. 16 illustrates two methods followed during the years between 1870 and 1880. These workings are inaccessible to surveys at the present time owing to the creeps and squeezes induced by the irregular method of robbing the small pillars. The sketch was constructed from some old projection drawings

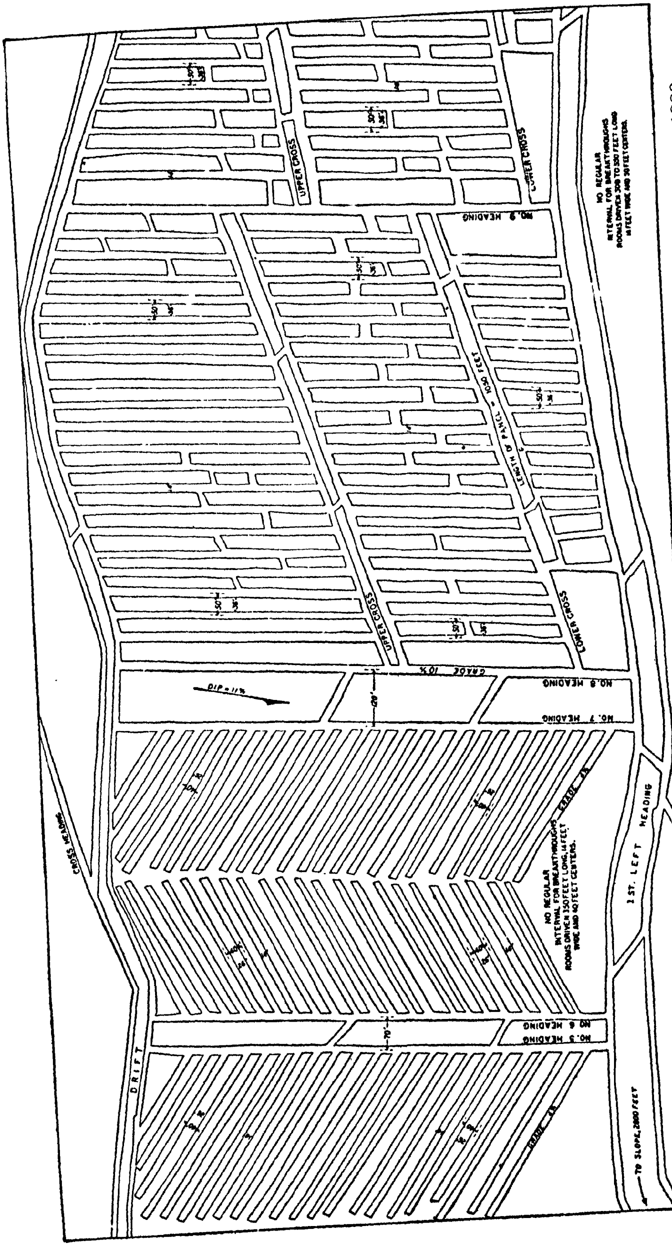


FIG. 16.—Two general methods of working the Georges Creek Big-Vein Seam during the years 1870 to 1880.

and from information obtained from a number of men actually engaged in the work. The main headings consisting of haulage road and airway were driven on the strike of the coal. In the first method the room headings were driven in pairs from the main entry at intervals of 600 ft. and on the rise of the coal on about 10-per cent grade. From these headings approximately 25 rooms were driven to the right and left with 40-ft. centers on a grade of 4 per cent, giving an average length of about 350 ft. The rooms were 14 ft. wide and pillars 26 ft. These pillars were found to be totally inadequate and extracting them impossible. Cross-cutting the pillars at frequent intervals was then attempted after completion of the rooms, but this was generally accompanied by creeps closing a whole district at a time. The maximum height of the superincumbent strata in this territory is 200 ft.

The second method, shown in Fig. 16, was adopted later. The maximum thickness of the overlying strata is 150 ft. By this method headings were driven from the main entry on the rise of the seam, at intervals of 1000 ft., to the level above, and two pairs of cross-headings turned to the right. The rooms were driven from these cross-headings at 50-ft. intervals and 14 ft. wide, leaving a pillar of 36 ft. The length of rooms varied from 300 ft. to 550 ft. These pillars were also of insufficient size; robbing was conducted spasmodically, and, although more coal was recovered than in the adjoining districts, a great deal was lost. In addition to the small pillars, the method of robbing them was calculated to promote squeezes. It appears to have been the method to hold the strata with props until sufficient coal had been removed to enable the weight to break the props. As a general rule, however, before this was attained the weight had induced a creep, which is well known to have no limits within a territory of small pillars.

Fig. 17 represents a method in use in 1890. The main entries were driven from the slope on the strike of the seam, sufficient grade being allowed for drainage. Cross-headings were driven on an angle of about 35 degrees to the main entry and headings turned off these parallel to the main entry. Rooms were turned, as shown, from all headings on 100-ft. centers, and pillars split by half-rooms. The length of rooms varied from 300 ft. to 600 ft., and were 15 ft. wide, leaving pillars 42½ ft.

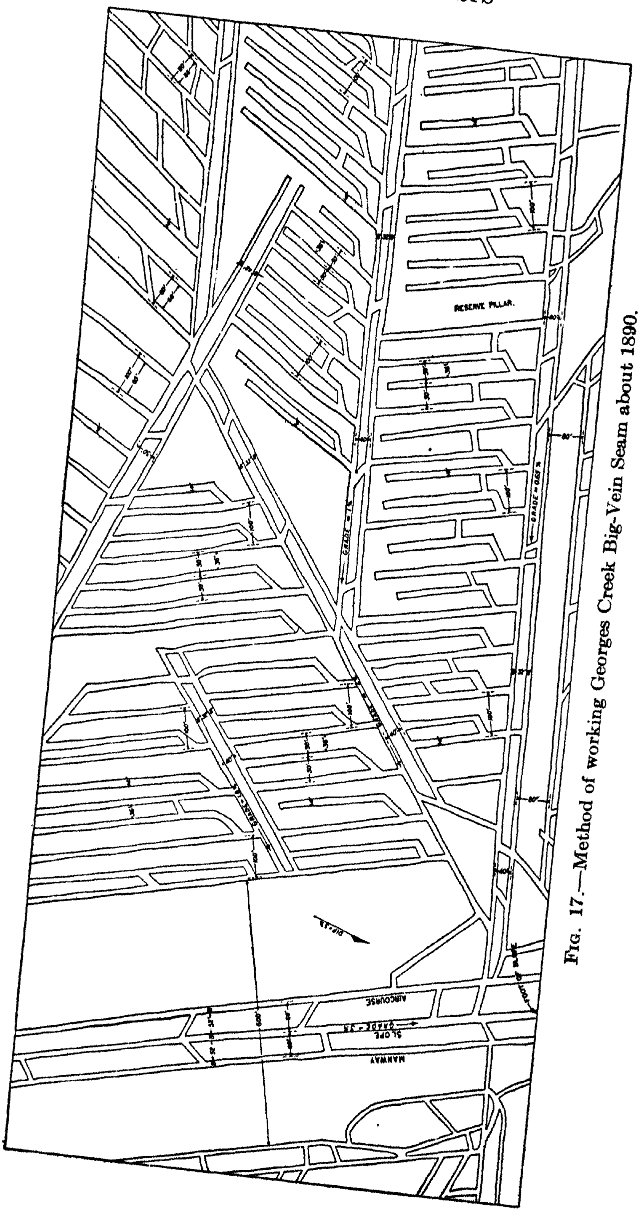


FIG. 17.—Method of working Georges Creek Big-Vein Seam about 1890.

wide. These pillars were not strong enough to support the overlying strata, 500 ft. high, and the usual creep or squeeze resulted when pillar drawing commenced.

This half room method has the advantage of facilitating gathering of coal and doubling the support of the haulway. The squeeze in this district could have been prevented by turning the rooms from the haulway on 200-ft. centers and, after driving the half-rooms, the resultant pillars would be 85 ft. wide. While this would have avoided a squeeze, the great weight to be supported by this pillar of soft coal would not have permitted a very high percentage of recovery.

Fig. 18 shows a method adopted in 1900. The maximum dip is 15 per cent, and the greatest thickness of superincumbent strata 425 ft. The slope, together with parallel air-course and man-way, are sunk on the heaviest dip of the coal, and double entries turned off to right and left at intervals of 1000 ft. on a grade of 1¼ to 2¼ per cent in favor of the loads. From these haulways cross-headings are deflected at intervals of 240 ft. at an angle of about 25 deg. and driven on a grade of 4 per cent to 7 per cent. Rooms varying in length from 100 to 800 ft. are turned on the rise of the coal from these cross-headings. The rooms are driven 15 ft. wide on 65-ft. centers, leaving pillars 50 ft. wide. Twenty-five rooms are driven in each of these diagonal panels. Unusually large protecting pillars are left along the main haulage roads. This system has been found to be especially adapted to rapid gathering of cars thus ensuring a large tonnage. It has been found, however, that a very large recovery from the pillars is impossible, owing to the many sharp angles, which, in a thick seam of soft coal, are always difficult and ofttimes impossible to extract. This sharp-angle method was even resorted to formerly in cross-cutting the pillars preparatory to drawing them, but this has been changed to a rectangular method, thereby increasing the actual percentage of pillar coal recovered from 80 per cent to 83 per cent. The distance of rooms apart has also been increased in the last few years to 100-ft. centers giving pillars 85 ft. thick. It is expected that the extraction of these will show a further increase in the percentage of yield from pillars. The present yield from headings, rooms, and pillars under this system is about 90 per cent,

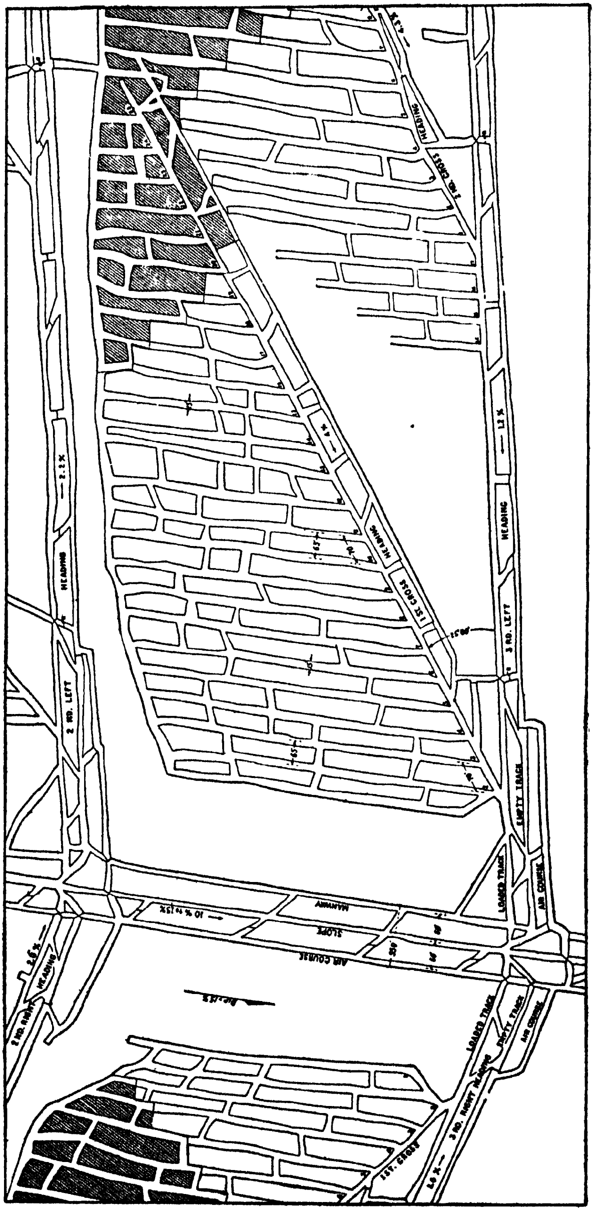


FIG. 18.—Method of working the Georges Creek Big-Vein Seam about 1900.

considering the recovery from headings and rooms at 100 per cent.

Fig. 19 illustrates a method instituted in the latter part of 1904. The main haulway is an extension of the slope from the opposite side of the basin. Double entries are turned off from this entry, on 1½-per-cent grade, 400 ft. apart, from which rooms are driven directly on the rise of the coal. Rooms are from 13 ft. to 15 ft. wide and practically no barrier pillar left between the room face and the air-course of the panel above. They are driven at 100-ft. intervals, leaving a pillar 85 ft. wide. The length of a panel is about 2500 ft., containing 22 rooms. There are five such panels in this district and when completed it is proposed to draw the pillars in a retreating fashion with the line of pillar work on an angle of 45 deg. across the whole district. A similar method in another district, but with rooms on a deflection of 35 deg. from a right angle, is yielding 88½ per cent from the pillars with a total recovery of 94 per cent from headings, rooms, and pillars, and it is believed that this can at least be duplicated if not exceeded in the case of Fig. 19. The maximum dip in this district is 6½ per cent with the greatest height of the overlying strata 250 ft.

With this résumé of systems used at different periods under conditions now known in view, a suggested method of extracting the coal from thick soft seams with a brittle top and a height of superincumbent strata of 400 ft. or less is presented in Fig. 20. The general design of the mine for haulage, drainage, and ventilation is not given, because they are variable quantities, depending on conditions which change with the locality, and the method suggested is therefore limited to the ultimate recovery of the coal. By this method a territory under development is divided up into rectangular panels of 10 rooms each. The room headings are driven on easy grades favorable to drainage and haulage and the panels worked in pairs. When the upper heading has been driven to its end the rooms are turned at intervals of 100 ft. with the drawing of pillars following the retreating method. The rooms are 400 ft. long and 13 ft. wide, leaving pillars 87 ft. in width. The rooms in the upper panel are limited by a barrier pillar separating them from the heading above, and those on the lower panel are driven through to the gob of the upper panel. The line of pillar work extending

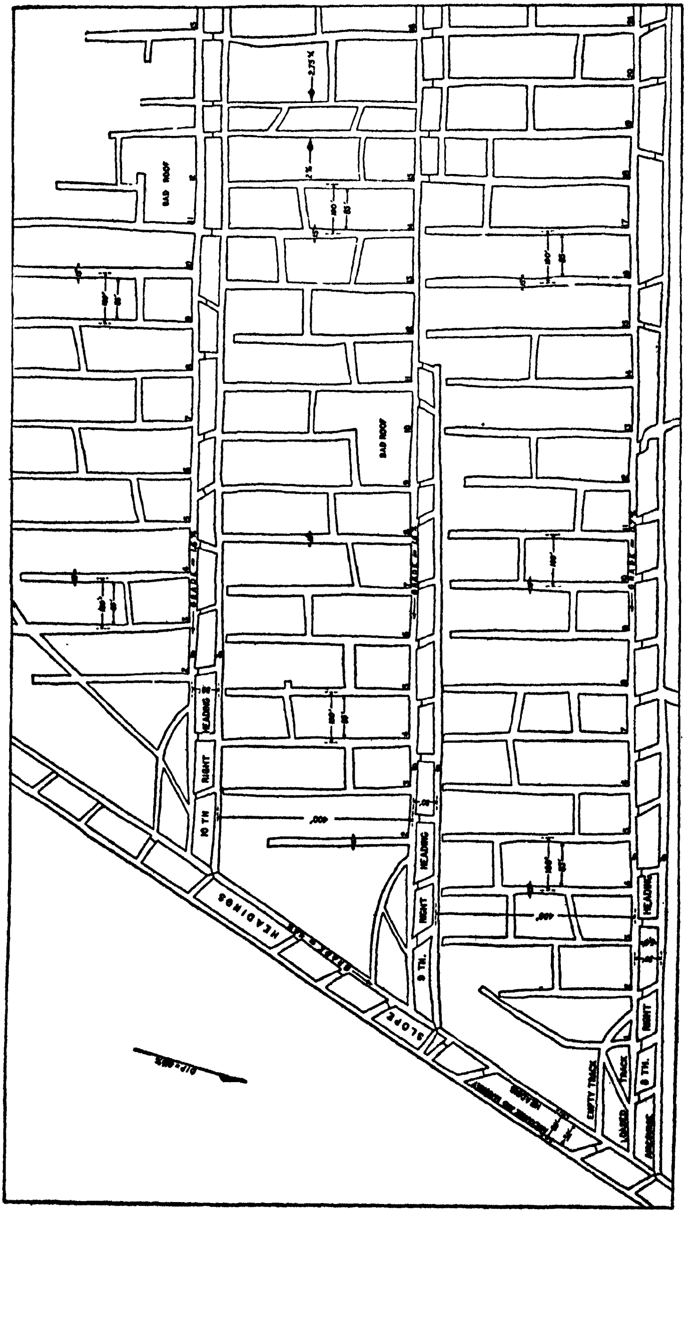


FIG. 19.—Method of working the Georges Creek Big-Vein Seam in 1904.

over the two panels should have an angle of about 45 deg. The length of rooms can be varied to suit the conditions, and, when the height of the overlying measures exceeds 400 ft., the thickness of pillars should be increased accordingly.

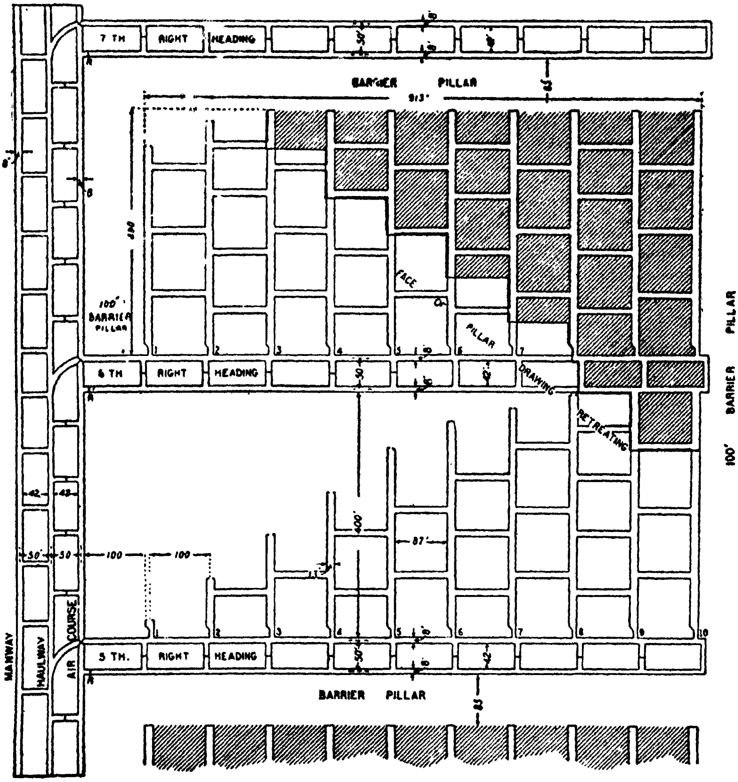


FIG. 20.—Suggested method of working thick, soft seams, having a brittle top and 400 ft. cover.

Fig. 21 shows the method of drawing the pillars in detail. The rectangular method should be carefully adhered to and all sharp angles avoided. When the work commences a cut is made separating a block of coal 30 ft. wide and 87 ft. long. This piece is further subdivided into blocks varying from 6 × 12 ft. to 15 × 12 ft. in size, which are cut off and extracted successively as shown. In no case should the small blocks cut off exceed in size the distance a man can shovel under average conditions, which is about 15 ft. The largest of the small blocks should be removed last because it is there that the greatest pressure manifests itself in the removal of the original block cut from the pillar. When this block has been removed another cut is made and the process repeated. The line of gob should be approximately on an angle of 45 deg. If it is found impossible to take out the small blocks clean, on account of the gob from the preceding work running in on the new cut, a row of props can be set on the upper side of the large block to hold the gob when the small blocks below are removed. This method

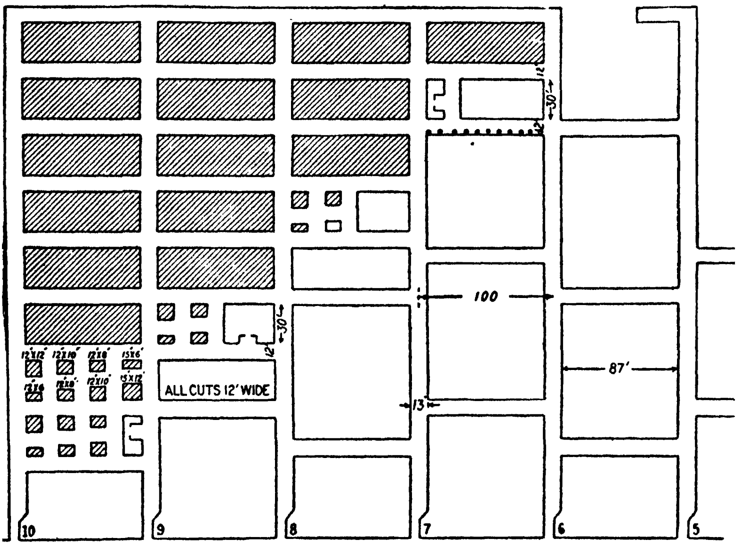


FIG. 21.—Method of drawing pillars to be used in the system shown in Fig. 20.

of working should result in a total average recovery of fully 95 per cent of the seam from headings, rooms, and pillars and ensure the workings against creeps and squeezes.

One of the most interesting studies of costs as applied to the different systems of mining flat seams of coal was included in a paper presented before the February, 1915 meeting of the Am. Inst. of Mining and Metallurgical Engineers, excerpts from which are given herewith.

It may be stated that the unit with which we have to deal in studying mining costs is the room; what takes place at its face is the real productive work of the mine, and all else underground is for the purpose of serving best the worker at the room face. Fig. 22 shows several typical methods of procedure. They are of particular interest in that one may see them in mines following the same plan, working the same seam, under conditions which admit of comparison. The features of these methods are given in Table I.

In all of these methods variations may be seen, from entries driving with no rooms turned to entries driving with two or more rooms turned and driving as the entries advance; in respect to the robbing, one may see variations from robbing following

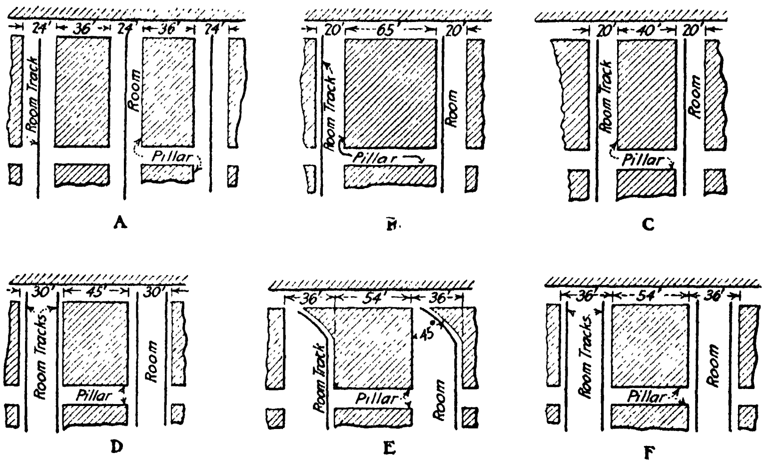


FIG. 22.—Typical methods of procedure in working rooms.

immediately upon the completion of the first two rooms to the robbing following at an indefinite date after the completion of the first workings of the panel. Where continuous paneling, or advancing robbing, is in effect, robbing is not compelled to wait until the completion of all the entries of the panel.

The number of rooms per entry varies from about 12 to an indefinite number, and the depth of the room varies from about 300 to about 800 ft. The amount of timber and the manner and time of placing same depend largely upon the individual miner, and as a rule there are no specific instructions for his guidance; also, in general, no effort is made to recover the timber in robbing.

TABLE I

METHODS OF PROCEDURE IN DRIVING ROOM

Sketches	A	B	C	D	E	F
Width of room in feet	24	20	20	30	36	36
Width of pillar in feet	36	65	40	45	54	54
Location of track	In center of room	Along robbing rib	Along robbing rib	Along robbing rib	Along robbing rib	Along robbing rib
Location of gob	Along both ribs	Opposite robbing rib	Opposite robbing rib	Between tracks	Opposite robbing rib	Between tracks
Number of men per room	1 to 2 rooms	1	1 or 2	2	6	4
Feet of room face per man	48	20	20 or 10	15	8.5	9
Feet of entry per man	120	85	60 or 30	37.5	15	22.5

A method of procedure observed at one mine (but which has not as yet been sufficiently tested out in the matter of recovering the pillar to warrant its unreserved adoption), is shown in Fig. 22, *E*. Here it is intended that rooms shall be driven 36 ft. wide on centers 90 ft. apart, carrying a room face at an angle of 45 deg. and a single track along the robbing rib but curved to parallel and follow the length of the room face. It is intended to work six men to the room, the gathering motor receiving and placing three cars at a time. Immediately upon the completion of the room the pillar is to be withdrawn. By this method of procedure a high degree of concentration will be effected and the efficiency of the gathering motors, mining machines, and miners will be increased. It is also hoped that by carrying the working face on a diagonal, fewer unexpected falls of top will occur than at present, because the fracture will generally be partly exposed before the entire coal support is removed from beneath it.

In most mines the miner at the face is responsible for the safe working conditions of his room. In the above methods of procedure, therefore, one might say that, for the same expenditure of time, energy, and watchfulness, the relative degree of security which the miner may feel as a result of his efforts is inversely proportional to the room space he occupies. It is also true that for cars of the same height the energy expended by the miner, or the work done in loading the coal, is much less where two or more men work per room and the room space per miner is low, than where one man works per room and the room space per miner is high.

In the grouping of rooms as outlined in Fig. 22 many arrangements were made from which have matured certain well-defined plans. Probably the consensus of opinion favors the panel system, but even with it there are differences of opinion. Many men think that the entries should be driven to the inside lines of the property and the coal extracted retreating; others think that half of the property should be worked advancing and the remainder retreating; yet others think that all or nearly all of the coal should be extracted as the entries advance. It is probable that it is best to extract the coal in such a manner that the present worth on the returns from the mining venture will be greatest, both to the property owner and the operator, leaving only such coal during the advance of the entries as will permit of profitable mining until the final exhaustion of the property. Figs. 23 and 24 show typical plans on the panel system. Fig. 23 is the square or rectangular panel, Fig. 24 the continuous panel.

For purposes of discussion and demonstration a typical property of 1000 acres of the approximate shape indicated in Fig. 25 will be assumed from which it is desired to produce 2800 tons per day when running at maximum. The dip is 2.5 per cent and the strike lies at an angle of about 10 deg. to the long axis of the property, its general direction being from the upper right hand corner of the property to the lower left hand corner. The coal is fairly clean, 6 ft. thick, and the condition of grades, top and bottom, are fair. It is also assumed that the rate of loading per man per day will be 16 tons. The questions to be decided are what method of procedure and what plan are best, to determine which the following information is desired:

1. What period of time will be required to reach the desired output?

2. How many day laborers, mining machines, mine cars, mules for gathering, and main-haulage motors will be required?

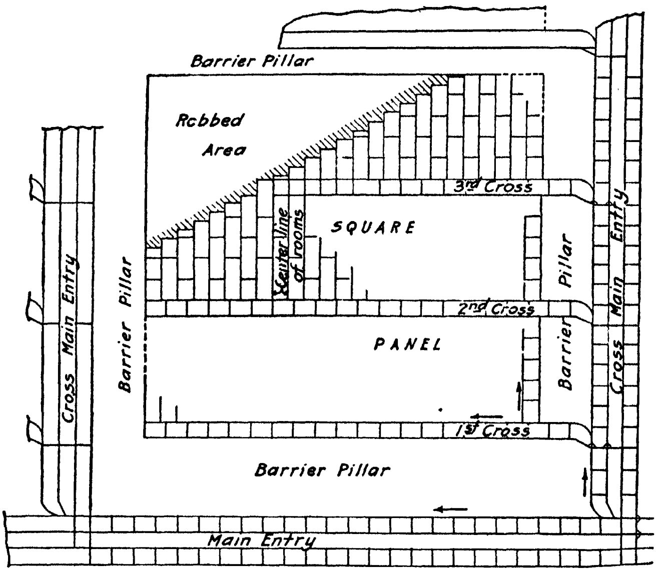


Fig. 23.—Typical plan of mining on the square or rectangular panel system.

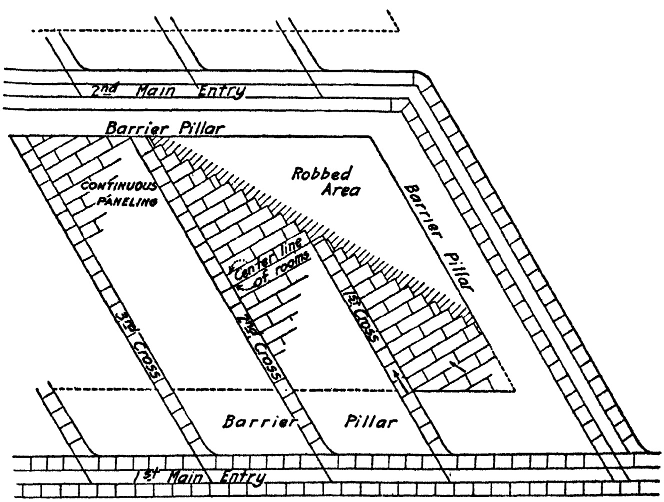


Fig. 24.—Typical plan of mining on the continuous panel system.

3. How much main entry, main entry track and trolley wire; cross main entry, cross main entry track and trolley wire; room entry, room entry track, and rooms and room track will be required?

4. What is the length of the average car haul?

5. What is the relative amount of power for ventilation?

6. What is the acreage of standing pillars, the estimated relative cost of production, and the estimated percentage of recovery?

The methods referred to in Fig. 22, *C*, and the plans of mining in Figs. 23 and 24 will be applied to the problem as follows:

First Form.—Drive the third entry of the panel, turn the last two rooms on this entry first, start removing the pillar immediately upon the completion of the next to the last room, and continue to drive all the rooms in the panel only fast enough to provide for the uninterrupted advance of the robbing. Work two men to the room and in the air-courses and on the pillars. Only this method of procedure will be applied to the square and continuous panels, and the following methods to the square panel.

Second Form.—Drive the rooms of the panel as they are encountered, turning the first entry of the panel when it is reached, and start robbing immediately upon the completion of the last room on the third entry of the panel. Work one man to each working place.

Third Form.—Drive the rooms and entries of the panel as they are encountered, start robbing immediately upon the completion of the last room in the third entry, and continue the robbing until the completion of the panel. Work one man to every other room, but advance all rooms and work one man to each pillar.

The accompanying table shows the comparison, as well as some other figures to which further reference will be made.

From these data it may be concluded that the first form of procedure and the plan of mining, Fig. 24, are best.

The period of time required to reach the desired output was determined for the several methods, as shown in Figs. 25, 26, 27, 28 and 29; the location of the working faces from day to day, as determined by the assumed rate of advance of 16 tons per man, was plotted on a map, and the total number of faces at the time the desired output was reached were counted, from which data the tonnage curves were plotted as shown in Figs. 25A, 26A, 27A, 28A and 29A.

	First Form	Continuous Panel	Second Form	Third Form	Advancing Method
Output reached, months	53	42	62	92	7
Day laborers	60	65	82	102	31
Advancing Method Uses 8 Assistant Foremen					
Mining machines	9	9	14	18	8
Mine cars	275	310	335	465	155
Mules	18	22	24	32	10
Motors	4	4	5	6	2
Main entry					
Main-entry track	7,850	6,500	9,300	13,950	600
Main-entry trolley					
Cross main entry					
Cross main-entry track	5,550	5,150	8,850	13,500	1,000
Cross main-entry trolley					
Room entry and room-entry track	10,700	12,700	33,900	50,400	7,000
Room track	15,840	20,500	96,800	230,300	18,100
Average car haul	6,180	5,333	7,420	10,230	3,640
Ventilation power, kilowatt hours	40	42	125	175	20
Acreage of standing pillars	62.8	65.7	168.2	277.0	13.8
Relative cost of production	1.33	1.24	1.76	2.1	1
Percentage of recovery	94	95.5	83	80	97

In arriving at the relative number of men and mules required, rates for performing certain tasks, taken from time study observations were used. The amount of rolling stock required is based on the assumption that the equipment will travel at the same rate of mileage per day; the other items compared were taken direct from the maps. In the absence of facts for comparison, opinions of creditable authorities were sought in very instance.

These methods of procedure and both plans of mining have been designed to meet certain wants. In some instances certain features of the plan have been prescribed by the land owners in order to safeguard their interests from "squeezes" and losses of coal due to lack of proper supervision. Were the proper supervision supplied and better methods of procedure adopted, the restrictions in the plan of mining might very properly be

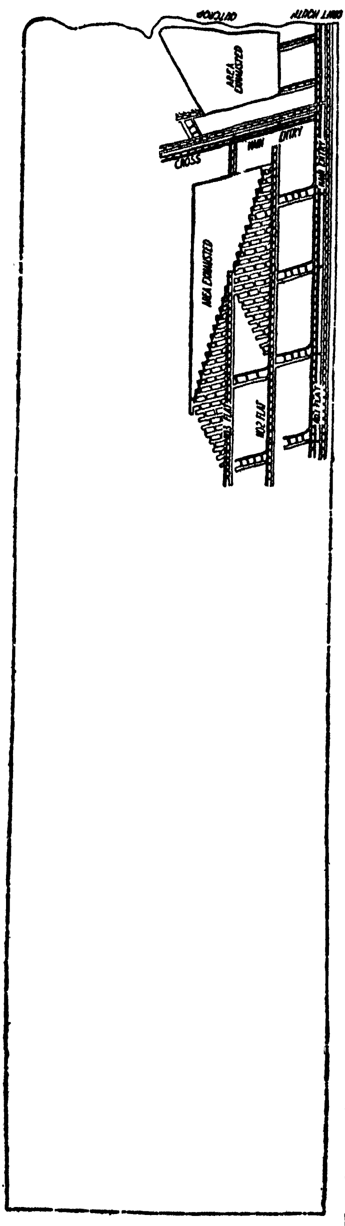


FIG. 25.—Advancing system of working with 4 men to a room showing position of workings at the time the output is reached

removed. Other details of design have been the result of accepting certain "rules of thumb" which have since been proved wrong, and yet other details, although admittedly wrong and expensive, have been introduced rather than combat the wrongs which they are designed to circumvent.

In the plan of mining shown in Fig. 23, the frequent interposition of barrier pillars is for the purpose of confining a squeeze and limiting its range of destructive action. The use of these barriers is imperative under the methods of procedure

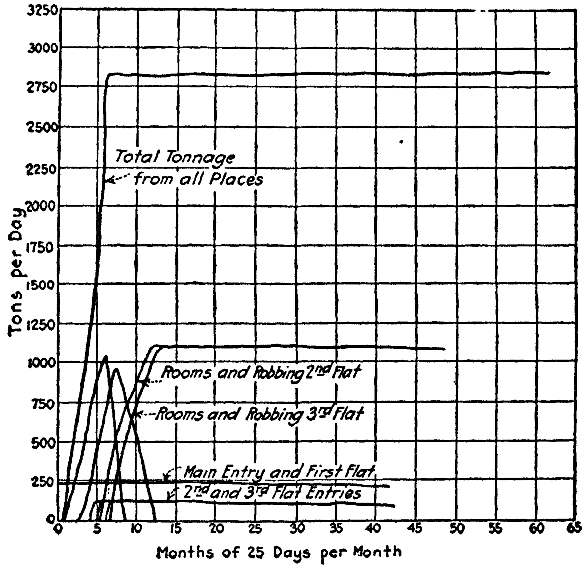


FIG. 25A.—Tonnage curves under the method of procedure shown in Fig. 25.

that involve large areas of long-standing pillars and where the degree of supervision is low. It is to be regretted that their use is so common, for they tend to interfere seriously with the maximum degree of concentration because one is seldom, if ever, able to provide a satisfactory output from a single panel, and then only for a short period of time. Where two or more panels are required to produce the output, the further the workings advance the more distantly seperated they become, or other important considerations must be sacrificed. Disadvantages of the unit-panel plan may be seen in the curve in Fig. 30 which shows the great variation in the tonnage obtained daily, vary-

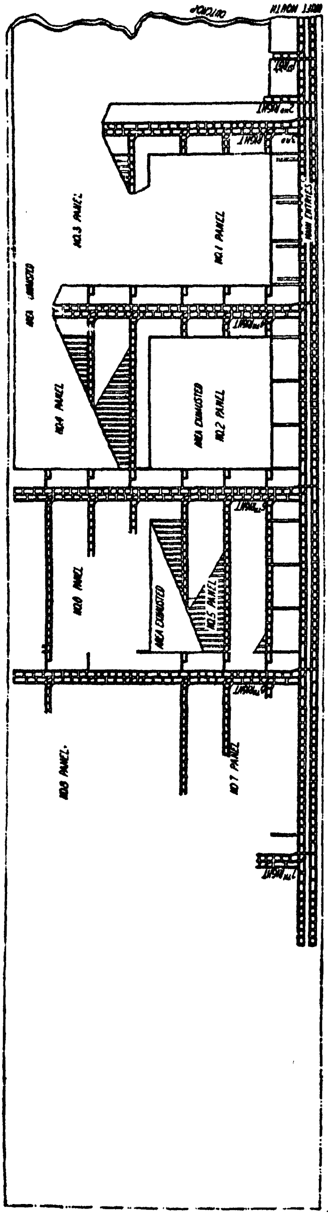


FIG. 26.—Square panel system, robbing retreating immediately upon the completion of the room, showing position of the workings at the time the output is reached.

ing from zero at the opening of the panel, augmented by a more or less constant rate of increase, to a certain maximum number of tons, and then a gradual decline to zero again. If a certain number of tons per day gathered from the panel is accepted as 100 per cent efficiency for a gathering motor, as shown in Fig. 30, it will be noticed that the motor is at first working at a very low efficiency, which gradually increases until the maximum is reached, at which time another motor must be added, and the average efficiency of the two motors is

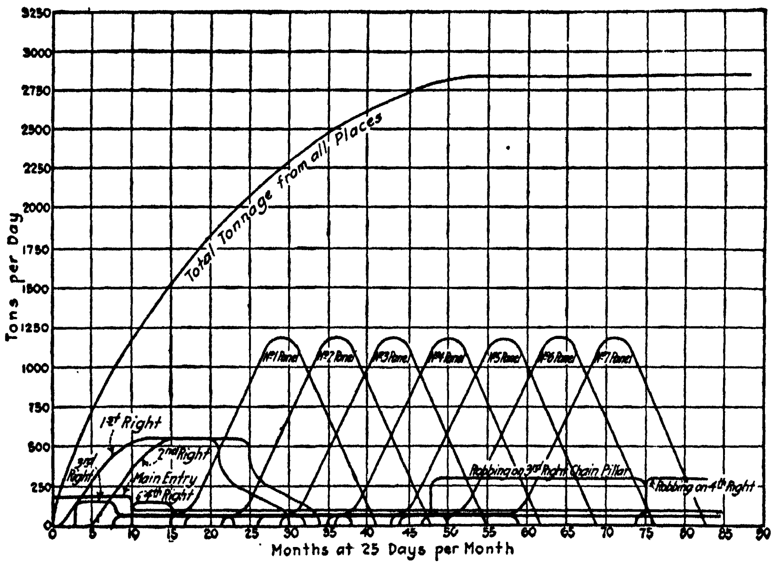


FIG. 26A.—Tonnage curves under the method of procedure shown in Fig. 26.

about 50 per cent; there is a similar drop in efficiency with each motor that is added, until the maximum tonnage from the panel is reached, after which the process of removing motors from the panel is begun. In some measure this degree of efficiency may be increased by working the motors over more than one panel, as is often done, and a better efficiency curve might be obtained more nearly in accordance with the full line shown, but in practice a rigid watch must be kept on this detail, or more often than otherwise a lower degree of efficiency than that shown will result.

If, in the preceding paragraph, instead of considering the

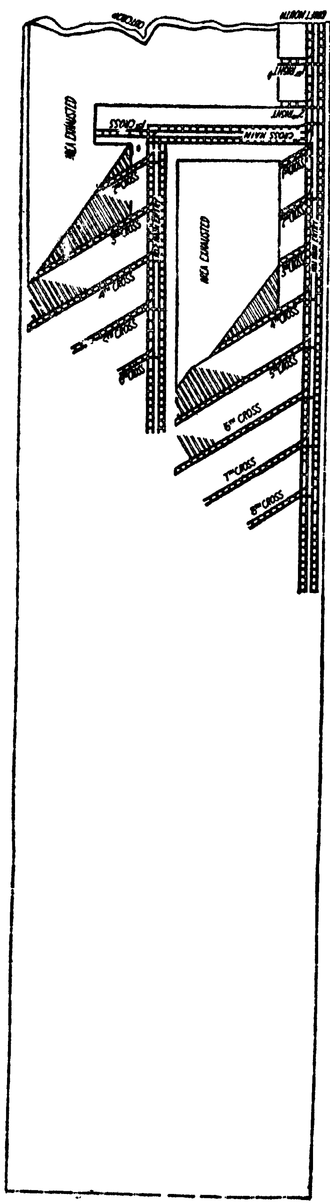


FIG. 27.—Continuous panel system, robbing retreating following immediately upon the completion of the room, showing position of workings at the time the putput is reached.

efficiency of the gathering motor the efficiency of day laborers or the tons produced per unit of material and equipment in use had been considered, the same general discussion would apply. Where low efficiencies are obtained from day laborers, material, and equipment, low efficiencies are also obtained from the miners at the room face. For these reasons it is difficult, and in practice well-nigh impossible, to establish any constant relation between a given tonnage desired to be uniformly pro-

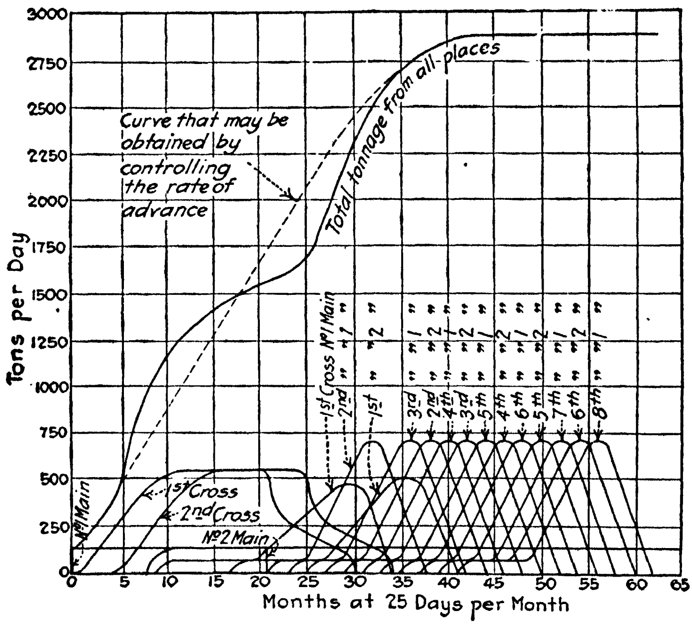


FIG. 27A.—Tonnage curves under method of procedure shown in Fig. 27.

duced, and the amount of material, equipment, and day laborers required to produce that tonnage; the efficiency of these quantities rises and falls with the rise and fall of the tonnage curve, although in an erratic manner.

Thus it would appear that the square panel, while designed to meet certain requirements, does so at the loss of much that is to be desired, and introduces new complications. The barrier pillars are, as the term implies, for the purpose of barricading against some impending danger, such as an unforeseen squeeze. Since no one can predetermine where or when these squeezes

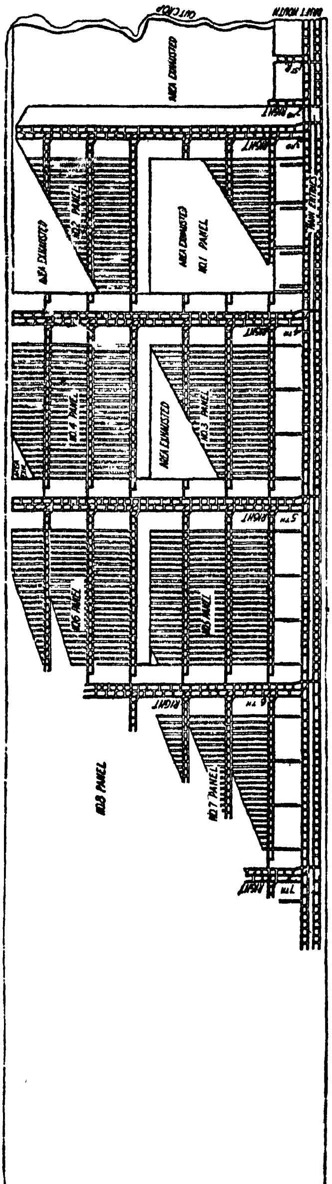


FIG. 28.—Square panel system, rooms driven as they are encountered and robbing conducted retreating immediately upon the completion of the panel, showing position of the workings at the time the output is reached.

will occur it sometimes happens that the barrier pillars are provided where they are not needed; yet experience has shown the wisdom and necessity of their use under certain conditions.

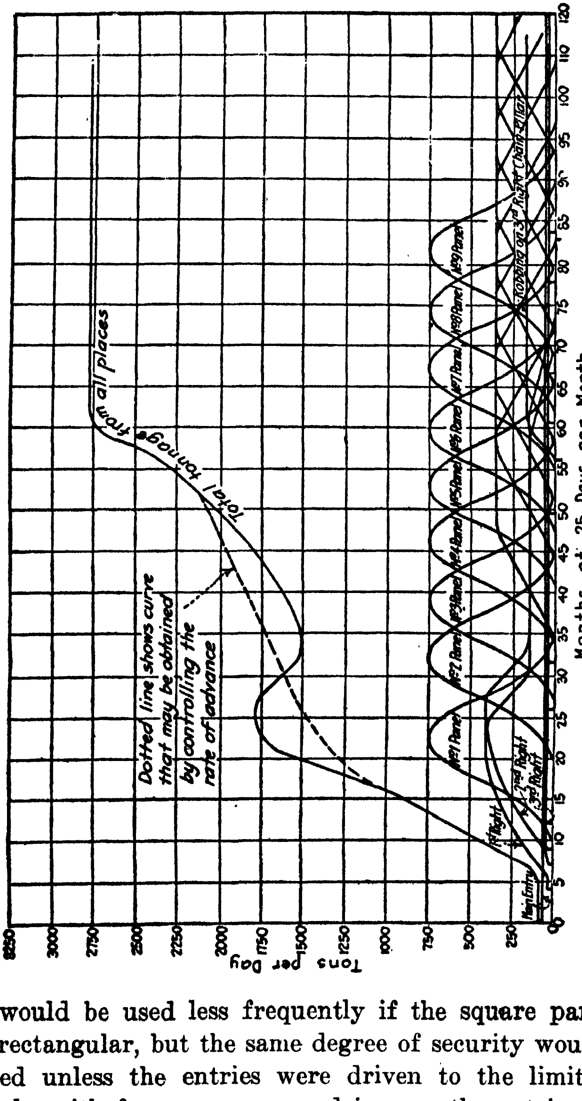


FIG. 28A.—Tonnage curves under method of procedure shown in Fig. 28.

They would be used less frequently if the square panels were made rectangular, but the same degree of security would not be obtained unless the entries were driven to the limit of the rectangle, with few or no rooms driven as the entries advance. If we accept it as axiomatic that when a room is driven to com-

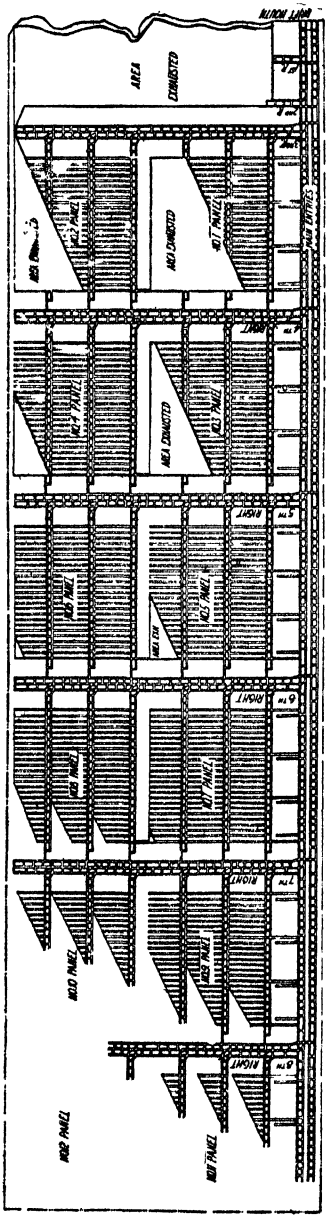


FIG. 29.—Square panel system with 2 rooms to a man, rooms driven as they are encountered and robbing conducted retreating immediately upon the completion of the panel, showing position of the workings at the time the output is reached.

pletion its pillars should be immediately removed in order to obtain the best results, or that it is equally as fundamental to open up no new entries until ready to mine from them, and

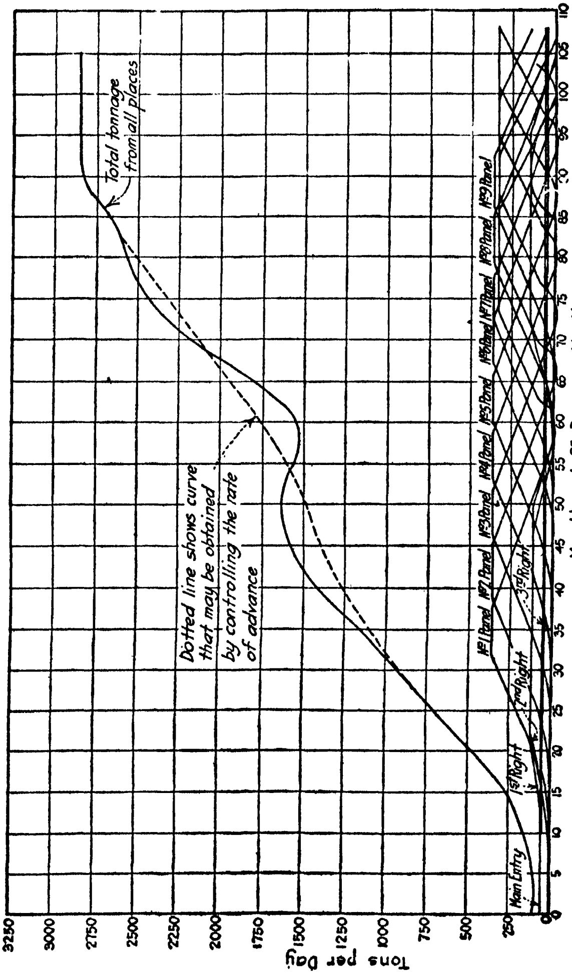


FIG. 29A.—Tonnage curves under the method of procedure shown in Fig. 29.

that mining should then be conducted at the maximum rate of production, the rectangular panel that involves either long-standing pillars or long-unproductive entries must be rejected.

The continuous panel obviates the necessity for frequently interposing a barrier pillar and it is especially well adapted to a property where the main entries are driven to the dip. However, the tonnage from a single continuous panel is limited, and where the main entries of a property go to the rise the maximum degree of concentration cannot be obtained or the rooms off the cross entry will go to the dip. Advancing robbing is impracticable because the pockets in the pillars go to the dip. The rate of production from a single room entry rises and falls in the same manner as the rate of production in the room entries of the square panel and the general discussion above in reference to the square panel applies to the continuous panel.

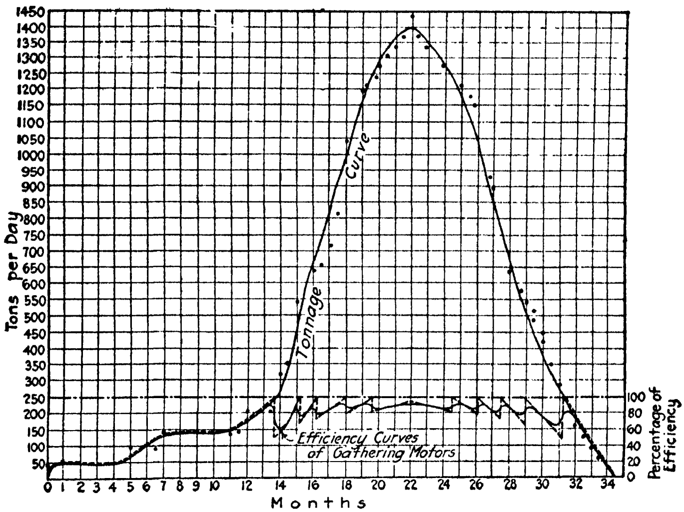


FIG. 30.—Variation in the tonnage daily obtainable from the unit panel and the efficiency of gathering motors working in the panel when proceeding as outlined in Fig. 26.

However, if one follows the history of the development of mining methods from the early-day single-entry system to the present-day panel system, it will be found that the square or nearly square panel meets sound mining practice more closely than any of the plans which have preceded it. Until methods of procedure are adopted which make the restrictions of the panel unnecessary, or until a plan of mining is devised without the objectional features of the panel, but retaining its may favorable features, the square panel will be accepted by many operators as the standard plan of mining.

For many years it has been the common belief that coal could be most economically cut and blasted by using a depth of cut equal to the height of seam. This erroneous idea frequently resulted in blasting down more coal than could be loaded in one day, and was the cause of allotting more than one room to a miner. That the height of seam does not bear any direct relation to economical cutting or blasting was demonstrated by the United States Coal & Coke Co. at Gary, W. Va., working with a Sullivan shortwall machine, and it has been found that miners are pleased to work two or more to a room, provided their earnings are as great as when they work in rooms by themselves.

Probably the most marked results in devising more economical mining methods has been achieved by the officials of the above-mentioned company. They have realized the objections to the mining methods outlined above and applied themselves to working out a plan which would be simple, direct and efficient. They accepted it as axiomatic that any change in the prevailing plans of mining must be beneficial to the property owner, operator and miner alike, for any change that would benefit one or more of the interested parties at the expense of the others would not last.

In this study difficulty was experienced because of the entire lack of systematized knowledge as to the proper relative rate of advance of room to retreat of pillar, the most economical width of room, and in fact what might be considered 100 per cent efficiency for any man, animal or machine about the mines. In order to determine these data, which were absolutely essential to an intelligent solution of the problem, a series of time studies was instituted and extended over a period of weeks, covering all the motions that make up certain underground operations that have to do with getting the coal from the working face to the railroad car. Thousands of observations were taken, properly checked, tabulated, collated, and used as a basis for a method of procedure, which has been put to the rigid test of practical use with remarkably good results.

This method of procedure has for its object the maximum degree of safety, sanitation and opportunity to the miner and of security to the property owner, while at the same time offering the greatest advantage to the operator. It combines a maximum degree of concentration with a minimum expenditure for labor, material and equipment, in such a manner that these quantities bear a constant relation to the output. Its use has resulted in a marked reduction in fatalities, increased earnings to the miners, decreased costs per ton for labor, material, equipment, and capital, and the recovery of practically all the coal in the seam.

At Gary, W. Va., mules are used for gathering, and as a result of concentration their efficiency has, in some instances, been increased to over 200 per cent. At one of the mines, fewer day laborers are employed underground than are employed about the tipple.

For the purpose of comparing the results obtained under this method with those from the several methods of procedure in the panel system, the method has been applied to the property and the problem under consideration. Fig. 31 shows the arrangement of the workings at the time the desired maximum output is reached. It also shows the details of the method of procedure; the other data desired are given in Table II. Fig. 32 shows the tonnage curve and, for comparison, the total tonnage curve from the unit entry shows that the tonnage rises rapidly curve, from the unit entry shows that the tonnage rises rapidly until the maximum is reached and then continues indefinitely at that rate. By using available data, the proper length of room, angle of breakline and angle of advancing faces may be predetermined, so that the total daily tonnage from the entry is the multiple of the tons that can be hauled by a mule or motor; thus, the mules or motors are always working at the maximum efficiency. It is equally true that when the workings have advanced for a short distance, after reaching the maximum tonnage from the entries, the estimated minimum number of day laborers required, may readily be confirmed, and once the entry reaches its maximum tonnage, and the quantities of labor, material and equipment have been accurately determined, these quantities remain constant throughout the entire extent of the entry, which may be as great as the property is long.

The room space occupied per miner is less than in any of the other methods now in effect, which is an index of the relative degree of safety a miner obtains for a given expenditure of time and energy. The excellent manner in which the rooms

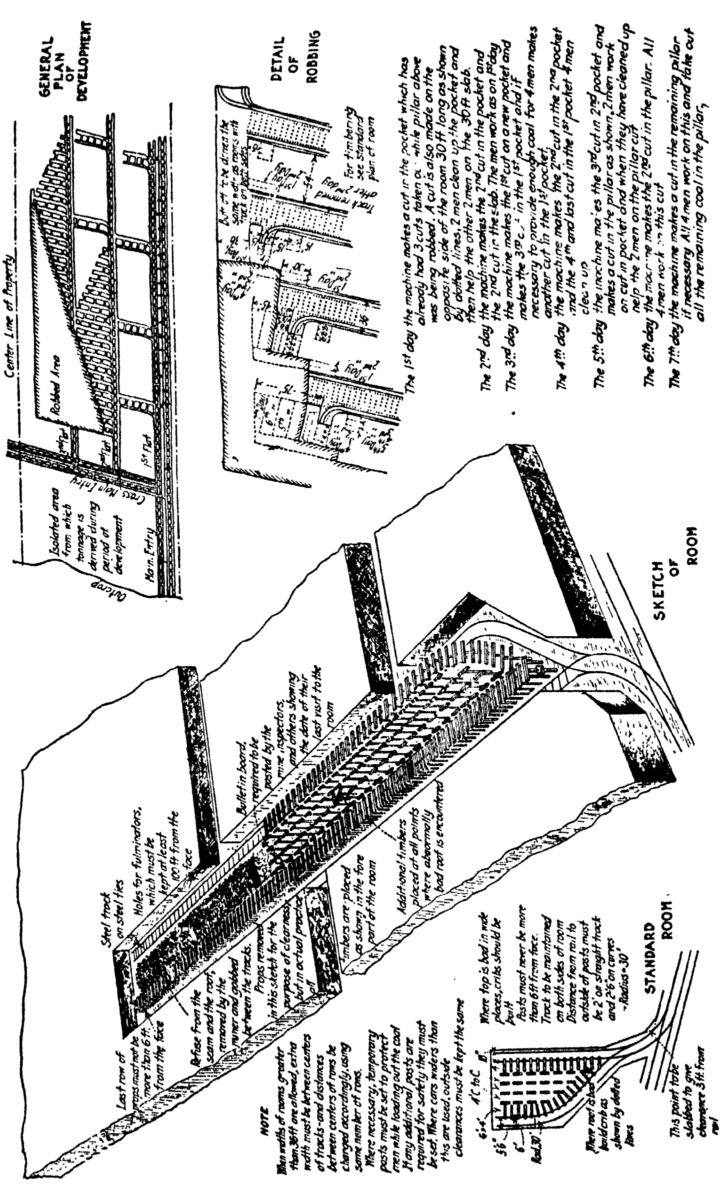


FIG. 31.—General plan and details used in the advancing system of mining, with rooms 36 ft. wide and 400 ft. long.

are timbered, shown in Fig. 31, is the minimum required; where the mine foreman or miner has reason to believe that additional timber is required to make the place safe, the miner must place additional timber before doing anything else.

As the entries advance, all rooms are driven and robbed immediately upon their completion, and rooms are opened up only fast enough to provide for the uninterrupted advance of

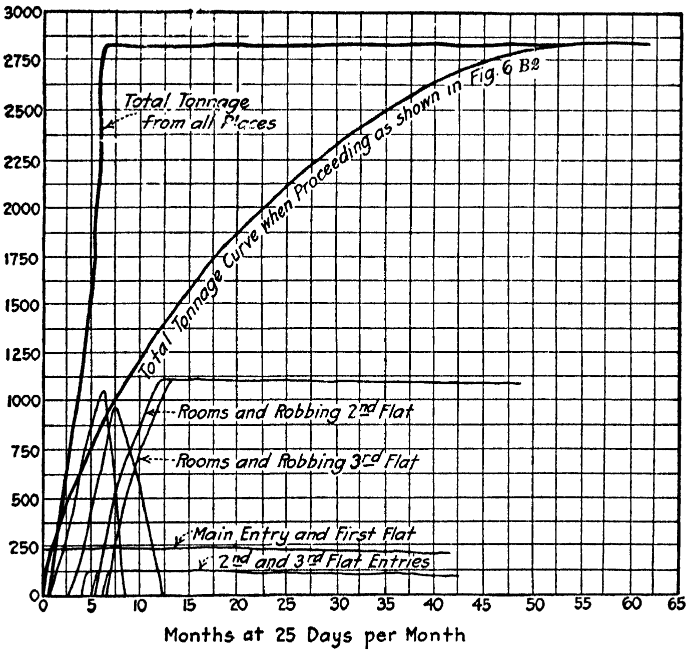


FIG. 32.—Curve showing rate of development to the desired output, under the method of procedure, sketch F, Fig. 22, and the advancing plan of mining, Fig. 31.

the robbing. Thus no barrier pillars are required, for the virgin coal protects the workings on three sides and the weight of the roof is resting on the bottom of the robbing. If a disturbed area of coal is encountered, or for some reason it is desired to discontinue the panel, a barrier pillar may be introduced at any time exactly where it is needed and the entries continued for the purpose of exploration.

In order that the different methods of mining may be readily compared, Fig. 33, showing the relative amount of labor, material, and equipment required to produce the tonnage desired from the property shown in Fig. 25; also the acreage of standing pillars, the relative cost of production, and the estimated percentage of recovery.

Any method of procedure that does not provide for the removal of pillars immediately on the completion of a room is fundamentally wrong, because it involves long-standing pillars open to the unfavorable influence of atmospheric agencies and other forces of nature; the duplication of track work; the clean-

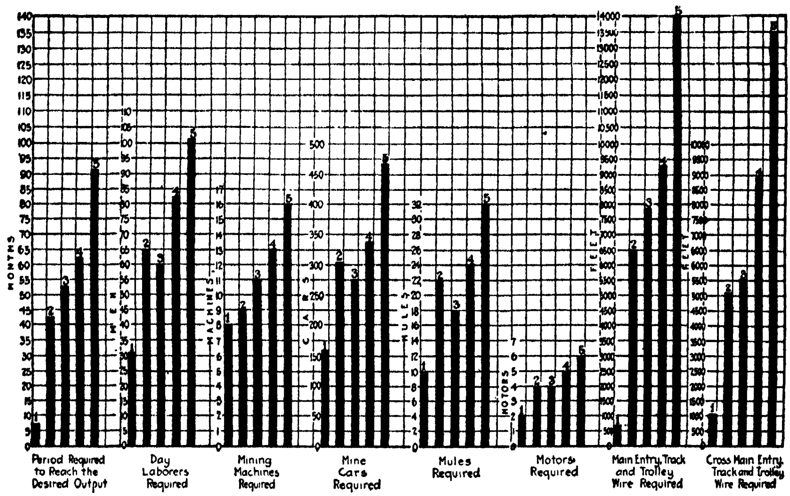


FIG. 33.—Comparison of the amount of labor, material and property required when following the methods of procedure shown in Fig. 25 and the relative cost per ton, the recovery, and the period of time required.

ing up of many slate falls that might otherwise have been avoided; and the scattering of workings, all of which increase the cost per ton for labor, material, and equipment, and cause the pillar coal to be badly disintegrated and low in domestic lump sizes.

It sometimes happens in practice, however, that fundamentals must be sacrificed to adapt the method to peculiar conditions encountered, often resulting in lack of concentration and large areas of standing pillars. Where considerable tonnage is desired and a new property is being opened, skilled miners, experienced in robbing pillars, are hard to get and frequently the officials, mine foreman, and underbosses are not experienced. In order to keep up the tonnage under these conditions, the workings must necessarily become distantly separated, because coal can only be obtained from room workings. It frequently happens also that the rates for mining pillar coal and room coal are not properly adjusted, so that the men can earn more in room work than in pillar work, naturally causing the pillars to lag behind, and requiring the introduction of barrier pillars to safeguard against squeezes; these barriers in turn cause a further separation of the workings, and a decrease in the efficiency of labor,

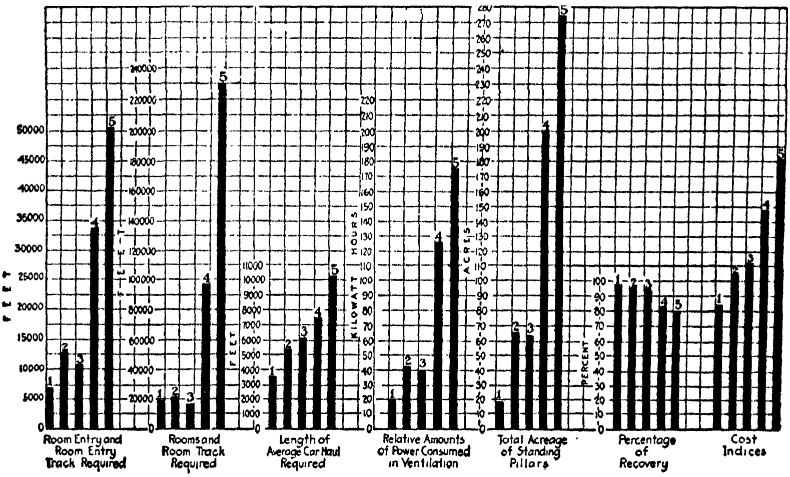


FIG. 34.—Equipment required to produce an output of 2800 tons per day from the different plans of mining outlined. Also the acreage of standing pillars to reach the output.

material, and equipment. The natural impulse of the mine foreman, therefore, is to open up more rooms in advance of the robbing in order to increase the efficiency to something like a proper standard.

For these reasons the territory for a given output during the development period should be as isolated as possible, and no greater in extent than is practicable. After the development period is passed and the organization perfected, there is no good reason why a mine operation should not be conducted with much the same regularity as a blast furnace or an industrial railroad.

The fallacy that the average miner will load only so much coal and no more has long since been exploded, and it is a matter of every-day observation that the miners are pleased to load coal if the mine cars are given to them with some degree of regularity and with some relation to the time required to load a car. When one considers that a coke loader, working under the heat of the sun, and of the coke ovens, will

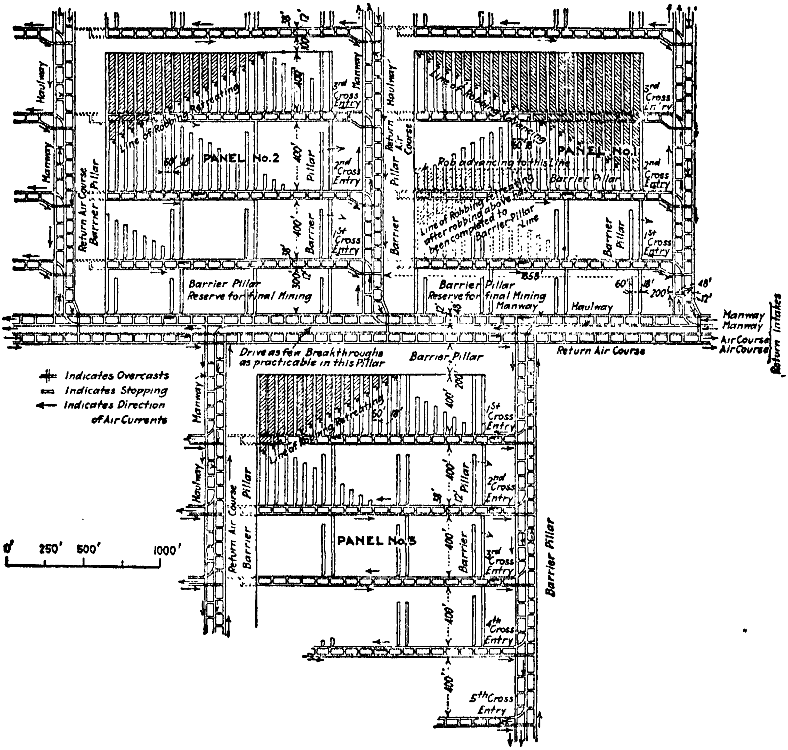


FIG. 35.—Standard plan of mine development adopted by the Pocahontas Coal & Coke Co.

load from 35 to 40 tons as an ordinary day's work, there is no reason why a miner working under so much more favorable circumstances should not load at the same rate. In this connection the following observations that have to do with loading coal underground are interesting.

These figures show that less than 47 per cent of the time spent underground was consumed in loading coal and over 12 per cent of the time was lost waiting for the empty mine cars. It may be stated further that these men were loading at the rate of 35 tons per day of 8 hr., and actually did load at the rate of 16 tons per man per day per year.

Methods of working in the Pocahontas field.—The entire Pocahontas field proper is practically all leased out on royalty

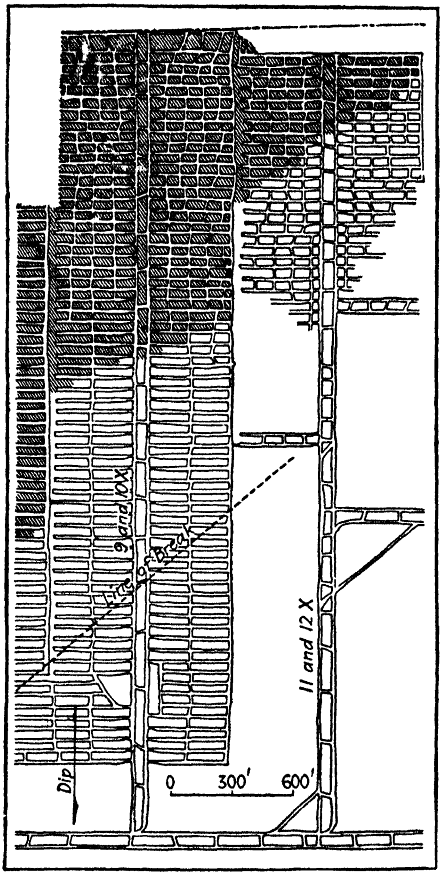


FIG. 36.—Double-entry system of mining used by the Upland Coal & Coke Co.

by two large holding companies, the Pocahontas Coal & Coke Co. and the Crozer Land Association. Under the lease contracts, the holding companies have reserved the right to define the method of working, and the result has been satisfactory.

A standard plan of mining, by Thomas H. Clagett, chief engineer of the Pocahontas Coal & Coke Co., is shown in Fig. 35; this is largely followed by their lessees, although in instances materially modified, due to local conditions. This large holding company owning or controlling some 275,000 acres of Pocahontas coal, has in active operation some 45 leases (in 1913), covering about 145,000 acres.

One of the special advantages of the system of mining adopted by the Pocahontas Coal & Coke Co., is the relatively quick recovery of the pillars, and the panels are so driven that the rooms and all entries split the pitch; thus if the maximum pitch is 3 per cent, then the maximum for the working will not exceed 2 per cent and may be even slightly less. Further particulars of this system appear on page 77.

The method of working adopted by the Upland Coal & Coke Co., on one of the Crozer leases, is shown in Fig. 36. Were the crop line shown on this plan it would be evident that the break line is carried in from the crop and does not involve, strictly speaking, breaks in the solid. There may be several "lifts" where the width of the lease is too great to admit of one lift only, as shown. This plan of mining was evolved from a number of years of revisions and has been found satisfactory under all conditions. The main entry is to be driven as near the line of strike as possible, in order that the reverse grade against the loads may be negligible. If the cross entries are turned off at more than 90 deg. from the main, and the rooms are less than 90 deg. of the cross entries, grades in favor of the loads may be obtained.

The Pocahontas Consolidated Collieries Co.'s Angle colliery, as of July, 1912, is shown in Fig. 37. Soon after taking up the work of the old Norfolk Coal & Coke Co. (which was essentially the nucleus of the Pocahontas Consolidated Collieries Co.) in 1904, the work of revising the systems of mining was taken up in detail. One of the first improvements adopted was the introduction of the multiple air-course system.

Further interesting particulars regarding mining methods in this field were given in a paper presented before the West Virginia Coal Mining Institute in 1913, from which the following has been excerpted.

It is bad practice to drive up a room and allow the pillars to stand in the expectation of drawing them later. A better

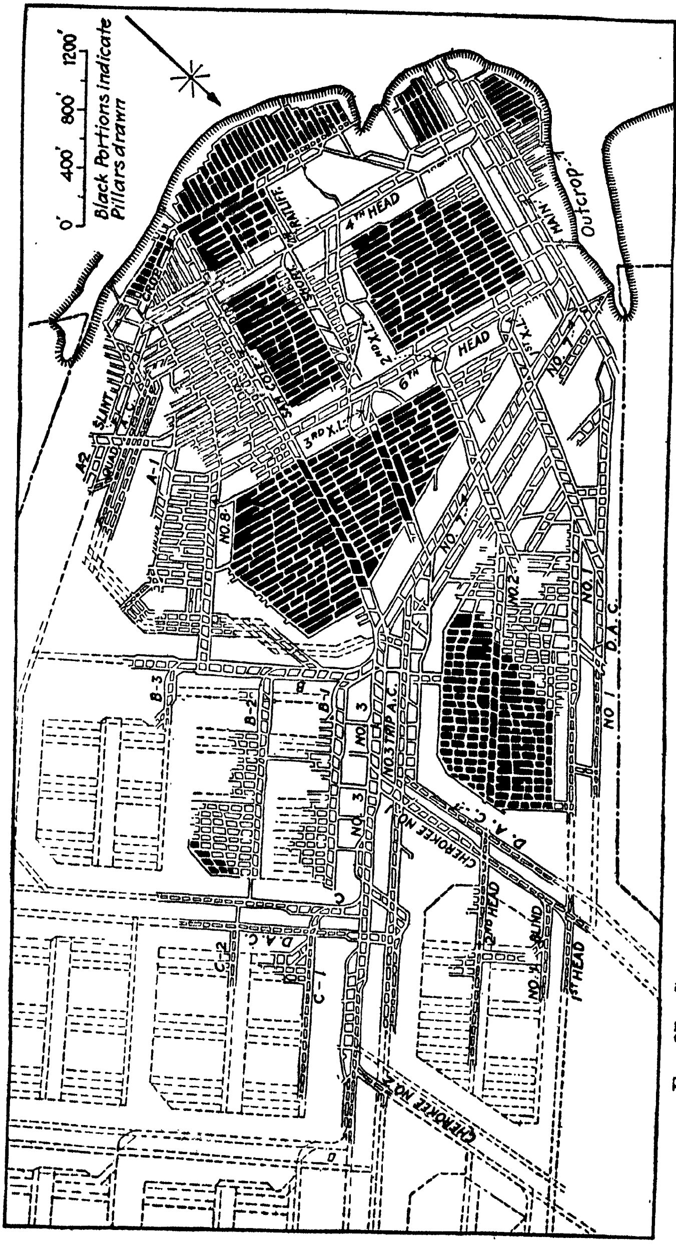


FIG. 37.—System of working used at the Pocahontas Consolidated Collieries Co.'s Angle Mine.

method is to start to "stump off" the pillars as soon as the room is completed by cutting across the rib as at *A*, in Fig. 38. The track should be laid so that cars can be placed conveniently for loading out both the coal in the cut and also that mined when the stump is drawn back.

In this method, the miner is always protected by solid coal and the losses are reduced to a minimum. Room No. 6 shows the pillar and the heading stumps completely removed; room No. 5, a pocket just starting in; room No. 4, a pocket finished and a stump partly drawn back. Room No. 3 shows the pocket finished and work just starting on the stump; room No. 2 shows the pocket being driven, followed by a second pocket, which

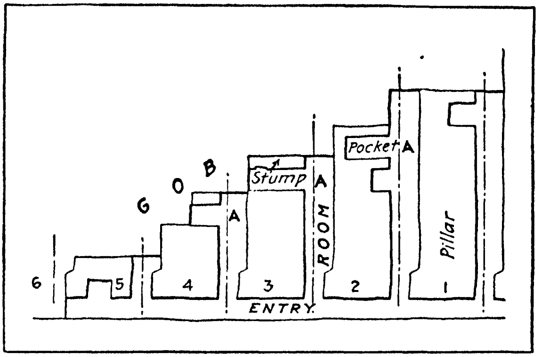


FIG. 38.—Method of splitting the pillars used in the Pocahontas Field.

is only extended as far as a man can conveniently load the coal without a track turn, in order to avoid the necessity for frogs and switch points. Room No. 1 shows the pocket just starting.

The width of this pocket and the thickness of the stump depend largely on the nature of the roof and the mine equipment. With poor roof, which falls unexpectedly or within a few hours after the removal of the coal, the thickness of the stump should be such that a miner can reach all the coal safely and easily without venturing too far beyond the rib line of the pocket. If the roof is good and does not fall soon after the removal of the stump the thickness of the small pillar may be increased and the number of track turns required per pillar may be reduced.

In the application of mining machines to the robbing of pillars, the distance between the centers of pockets should be such that the thickness of stump left will form, under bad roof, one machine cut, or under good roof, two cuts of the machine.

The more common practice where the roof falls soon after the extraction of the stump is to leave a small shell of coal to protect the miner from the gob and also prevent him from loading fine slate into the car of coal. This results in a loss of coal that can be avoided at an expense for timber, which, under ordinary circumstances, is less than the value of the coal.

A practice which has been advocated and proved successful, is to place a row of props on the lower rib of the pocket, before the removal of the pillar stump has begun. When the next pocket to the outby is driven, it will be found that practically the entire stump may be loaded out without any admixture of gob and a greater percentage of lump coal will be obtained. This precautionary row of timbers is especially desirable where machines are used on the pillars.

The roof over a robbing line exceeding 2400 ft. in length sometimes begins to sag in the middle and renders the removal of the pillars in that immediate section difficult.

The breakline should be kept as uniform as possible at all times. A method in practical every-day use, which is to be recommended, is as shown in Fig. 39.

The engineers, as they take their monthly measurements, mark the pocket centers on the robbing rib of the room, and the foremen are required to drive their crosscuts on the line of a pocket as at *A*. The object in keeping the breakline uniform is to insure against pillars extending back into the gob and acting as a fulcrum, or the knife-edge of a scale beam, upon which the roof teeters. This almost invariably causes additional timber expense and sometimes losses of coal, both of which could have been avoided had the break line been kept uniform.

The essential features of the Pocahontas Coal & Coke Co.'s plan of mining, shown in Fig. 35 are: Provisions for tonnage during the development period; provision for meeting the market demand; large barrier pillars, insuring against squeezes and rendering impossible the destruction of coal over an extended area; four-entry system for all extensive main entries, using two as intakes and two as returns with breakthroughs between only at points where the cross entries turn off, rendering unnecessary the building of expensive masonry brattices every 80 ft. and insuring the maximum quantity of air for ventilation at a minimum cost for brattices and ventilating power; cross entries with narrow chain pillars, permitting the rapid advance of the entry.

In general all robbing must be done retreating with rooms driven after the entry is nearing completion, insuring against slate falls and rendering possible the extraction of all of the

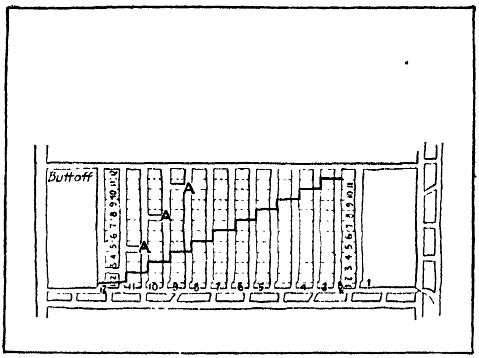


FIG. 39.—Locating crosscuts so as not to interfere with lifts from pillar.

coal in one operation, thus combining first development and robbing.

The depth and number of rooms on an entry vary greatly at different mines, depending on local conditions of the seam; whether the haulage is by mule or gathering motor, whether the undercutting is performed by pick or machine, and not infrequently on the personal equation of the mine executive, for sometimes the manager of a plant will contend that he obtains the best results when he drives 25 rooms, 500 ft. deep to the entry, and another manager working on an adjoining property under identically the same physical conditions and with the same type of equipment, not 1000 ft. away, will say that he gets the best results when his rooms do not exceed 300 ft. in depth and when there are only 15 rooms to the entry. The better policy is to encourage individual initiative and allow freely such modifications in any plan of mining as may be desired, provided that the modified plan embodies all the principles of modern methods and sound mining practice.

In the successful operation of any mine some general scheme of mining must be agreed upon, subscribed to by all parties in any way concerned with the matter, including the land owner, if the property is a leased one. Then no omissions in, additions to, or deviations from that plan of mining should be permitted without the written consent of the lessee and lessor.

RECOVERY OF COAL IN MINES OF POCAHONTAS COAL & COKE CO.

Plant	Thickness of Seam in Feet	Acres of Entry-mined	Acres of Rooms Mined	Acres of Pillars Mined	Total Acres Mined	Total Tonnage Mined	Tons Mined per Acre	Theoretical Tons per Acre	Percentage of Recovery	Proportion of Seam Rejected
1	6.15	3.06	4.57	11.03	18.66	165,254	8,856	9,922	89.3	0.24
2	5.65	4.40	4.80	14.80	24.00	188,391	8,185	9,115	89.79	0.22
3	5.16	2.68	6.52	15.80	25.00	180,386	7,215	8,325	86.6	0.22
4	4.42	5.88	8.65	13.09	27.62	192,437	6,960	7,131	97 6	0.23
5	5.94	7.00	10.09	19.20	36.29	334,005	9,203	9,582	96.0	0.22
6	4.32	2.11	3.64	9.20	15.04	94,427	6,278	6,969	90.0	0.31
7	5.34	3.31	6.34	0.00	9.65	83,000	8,601	8,614	99.8	0.20
8	5.42	3.72	6.06	9.72	19.50	144,769	8,181	8,777	93.2	0.20
9	4.65	8.10	16.80	2.34	27.24	201,044	7,380	7,534	98.0	0.18
10	8.03	5.20	8.47	10.09	23.76	262,975	11,068	12,923	85.6	0.23

After the general plan of mining has been decided and operations begun, its success or failure will depend largely upon the degree of watchfulness exercised. The mine should be accurately surveyed and mapped at least once every 90 days. Frequent inspections should be made of the mine, minute attention being given to the conditions of the working faces and the robbing line. At least once a year the theoretical yield of the property should be balanced against the actual tonnage delivered at the tipple. Accurate and complete records should be kept of the number of acres of entries, and rooms driven and pillars drawn each year and of both the percentage of recovery per acre and the state of exhaustion of the property.

That the above method of mining will yield the maximum recovery is indicated in the accompanying table, the figures in which are typical of the results obtained by the Pocahontas Coal & Coke Co., which are probably unexcelled anywhere. In this connection it should also be noted that the percentages of recovery are based on the total seam, including the portion rejected.

The lower percentages of recovery in the table result from the fact that in some instances, pillars were being robbed that had been standing for many years. In the mines of the United States Coal & Coke Co., at and near Gary, W. Va., where all the work has been opened in recent years, the average percentage of recovery per acre since the beginning of operations, has been better than 95 per cent, and of the area mined, over one-third has been final mining. The cost of production of room and entry coal by this method is the same as in other methods of mining, while pillar coal is produced at less expense than is incurred in other methods.

Most operating companies have a statement showing the revenue derived from operations per ton of coal mined based on the net receipts from operations divided by the tonnage. By placing a value, which could be closely approximated, on the recoverable coal lost, and adding it to the net receipts a figure could be obtained showing what revenue would have been derived from the operations had the coal been mined without unnecessary waste. Dividing this by the tonnage a figure could be obtained for the statement which would show the profit that would have been derived per ton produced if the coal in the seam had been worked by the most conservative methods.

Statements of the above nature have a further value from a financial point of view for if it can be shown to a bonding concern that a property contains, let us say $500,000 worth of coal in the ground, 90 per cent or more of which will be mined, it is certain that a greater asset value will be placed on the property than would be credited to it if the engineers of the bonding house report that under the methods of mining pursued, only 50 per cent of the coal in the ground will be mined and the rest irretrievably lost.

Connellsville Systems.—The system of mining practiced by the H. C. Frick Coke Co. in the Connellsville region as described in a paper presented before the Engineers Society of Western

Pennsylvania in 1916, is the application of shortwall mining machines to the extraction of rib coal. The two salient factors effecting this result were, first, the effort to reduce accidents and second, the desire to obtain an increased output of coal per man per day.

It has long been realized that the more intense the supervision of working places and workmen the less liability there is to accident. In order to obtain the desired supervision without making the cost prohibitive, it was seen that the time spent by mine officials in traveling from working place to working place must be reduced to the minimum and the time actually spent in working places increased to the maximum. To obtain this result the working places were concentrated gradually, and it was soon found that, under the old method of pick mining, a limit was quickly reached, and it was realized that to obtain the desired intensive supervision it was necessary to decrease the number of working places and workmen. This could only be accomplished by an increased production from each miner and a consequent reduction in the number of working places without affecting the total output of the mine.

On account of the conditions in the Connellsville region, where it is necessary to drive narrow headings, narrow rooms and have large room centers, it was found that machines in the narrow work would not accomplish the result since the bulk of the coal comes from rib extraction.

The use of electrically driven mining machines and the blasting of coal on the very rib line itself requires a system of ventilation that will insure gob gas being found only on the return. Such a system of ventilation necessitates ample, reliable fan equipment, airways of sufficient size and number, a generous provision of upcast openings, wise coursing of the air current and the existence of numerous bleeders from every gob into a return airway. It also demands the elimination of danger from dust by keeping it sprinkled and removing it before any dangerous accumulations are found. In addition it necessitates the use of permissible explosives and these only in the hands of selected competent shotfirers.

It has been proved that working places cannot be concentrated to as great an extent by any system yet tried in the Connellsville region as by the use of the H. C. Frick Coke Co.'s system of machine mining in rib coal. On account of the intense concentration of working places and the output that is obtained it is necessary that the haulage arrangements and equipment be perfected beyond anything that had previously been necessary, and the transportation of coal cannot well be handled except by the use of electric gathering locomotives.

The general plan by simple modifications can be made to suit all conditions; depth of cover; presence or absence of drawslate; nature of coal, and the nature of bottom and roof. This is divided into what we know as maximum, medium and minimum plans.

The maximum plan is applicable where thickness of overlying cover does not exceed 125 ft. and where the coal is hard and the general physical condition of roof and bottom is good.

The medium plan is applied where the cover does not exceed 250 ft. with the same physical conditions of coal and bottom and roof as obtain under the maximum plan.

The minimum plan may be applied to coal underlying any depth or thickness of cover, and whether or not the coal is hard or soft and the physical condition of roof and bottom good or bad, provided, of course, that mining machines in any form can be used.

The H. C. Frick Coke Co. has always worked its mines according to a projection, carefully prepared, for the field of coal to be worked before actual excavations have been started. In this plan of concentrated mining it has been found of great advantage to supplement these general projections with a schedule, prepared on a scale of 20 ft. to the inch, showing in detail the daily operations.

It should be understood that in the shortwall plan of mining the development is made on the face and the butt of the coal. After it has been determined as to what plan is to be followed for a given tonnage, the mining section is projected and developed and a fracture line established.

Let us first consider the minimum plan of extraction. The main haulage headings are driven on the face as are also the return airways while the producing headings are on the butt. Off these producing headings main face rooms are turned, generally on 112-ft. centers. From these main face rooms, butt rooms are driven on 25-ft. centers. As the main face rooms advance the necks of the butt rooms to be driven are excavated to a depth of three machine cuts. After this main face room has been advanced 50 ft. there are available two places for the machine to cut that will yield 40 tons, and when it advances to a point where the first crosscut is turned off, there are three places to cut in each main face room, yielding 60 tons. This main room may continue to the end of the section or to the end of the coal field, turning butts or producing headings off at projected distances. The main face rooms being driven on 112-ft. centers and 12 ft. wide leave a pillar 100 ft. in thickness between the rooms. This pillar is considered ample to support any thickness of cover with a floor or bottom under the coal seam of any nature that may be found in the Connellsville region.

On this minimum plan of extraction, where main rooms are advanced sufficiently far to begin the extraction of main face room pillars, the butt rooms are advanced in succession so that each room is 50 ft. behind the one next preceding. This plan provides for a tonnage output from three working places—two butt rooms advancing furnish 40 tons and one butt rib retreating provides an additional 40 tons, or a total of 80 tons of coal while retreating; and the main face room advancing is yielding 60 tons, or a total of 140 tons of coal from one main face room properly prepared and developed on this minimum plan of production. A sketch of this method is shown in Fig. 40.

Along the same lines the medium plan will not yield any greater tonnage from the advancing main rooms, but on the retreat the butt rooms are so driven as to maintain each face 30 ft. behind that of the preceding room. This allows three butt rooms to advance at a time, producing 60 tons, and necessitates two butt ribs retreating at the same time, giving a production of 80 tons, or a total from the butt rooms of 140 tons. This with the production of 60 tons from the advancing main room totals 200 tons for each main room. This method is shown in Fig. 41.

In the maximum plan the main face rooms advancing produce 60 tons while the butt rooms are so driven that the face of one is 15 ft. behind the face of the preceding room, thus necessitating four advancing butt rooms and the simultaneous

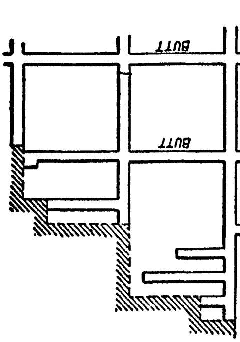


FIG. 40.

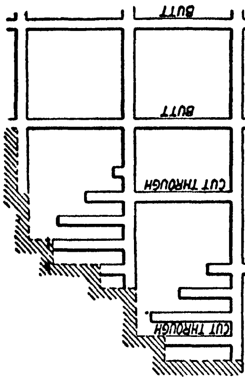


FIG. 41

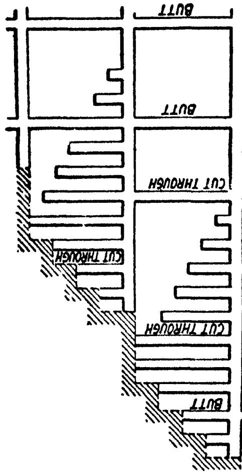


FIG. 42.

FIGS. 40 to 42.—Sketches showing minimum (Fig. 40) medium (Fig. 41) and maximum (Fig. 42) plans of extraction used by the H. C. Frick Co. in the Connellsville region.

withdrawal of four butt ribs. The four advancing butt rooms will produce 80 tons while the four retreating butt rooms will produce 160 tons. The sum of these, together with the 60 tons produced by the advancing main room, gives a total tonnage of 300 for each main room. The maximum plan is shown in Fig. 42.

The work is thoroughly systematized and the routine can be described as follows: After the miner has cleaned up his place and the day's run is completed the machine crew enters and cuts the place to a depth approximately 7 ft. The timbermen follow the machine crew, resetting any posts that it has been necessary for the machine men to remove. They post up any crossbars that have been notched in the coal over the machine cut, and generally put the place in good condition, following out a prescribed system of timbering. The timbermen are followed by the driller, who bores the holes for blasting with an electrically operated power drill. The driller is followed by the shotfirer, who charges the hole, tamps it, and after his own personal examination of conditions, explodes the charge, blasting down the coal ready for loading. After the coal has been blasted empty cars are placed by the gathering locomotives preparatory for the next day's work, so that when the loader arrives at his working place in the morning it is in a safe condition and every facility has been given him to load a maximum tonnage. Especial pains are taken through the day to see that wagons are changed as soon as loaded, thereby eliminating all unnecessary loss of time and allowing the men to load a maximum tonnage in the minimum time.

Actual results obtained regularly with miners loading under these conditions are 18 to 20 tons per man per shift; the average of all the loaders behind shortwall mining machines in all mines of the company for the month of August, 1916, was approximately 19 tons per shift.

At mines where there is a full equipment of mining machines the proportion of machine-coal amounts to from 80 to 95 per cent of the total output.

The concentration of work that has been obtained by this method resulted in a decrease in the cost of transportation, ventilation, track work and drainage because of the smaller area in active operation. There is also a considerable saving in the amount money invested in track and materials generally.

Some further interesting data on this system appeared in a paper presented before the Coal Mining Institute of America from which the following has been excerpted.

Preparations for the adoption of this system are made well in advance by subdividing the panels into blocks 90 ft. square

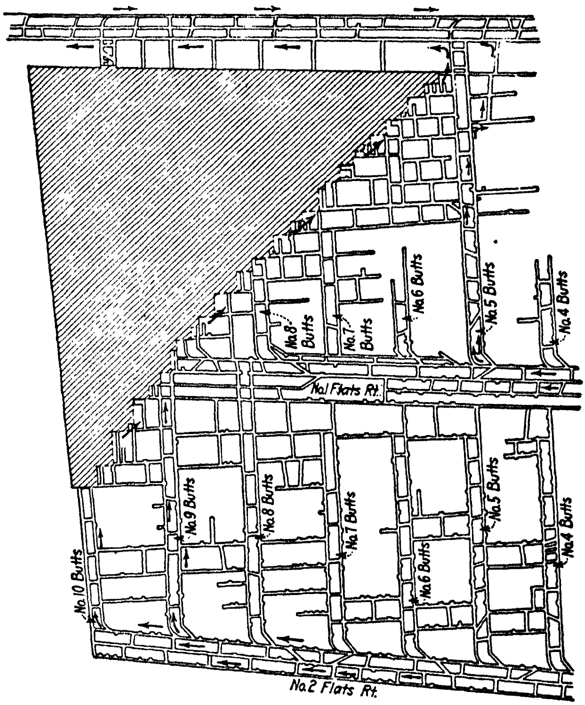


FIG. 43.—Plan showing how the Connellsville system is used at the Continental Mine No. 2.

as shown in Fig. 44. Double butt entries on 50-ft. centers, 10 ft. in width and 1200 ft. long are driven in parallel across the panel with cutthrough connections every 100 ft. Other and similar butt entries are driven 350 ft. apart, dividing the panels into blocks 350 by 1200 ft. and the chain pillars between the double entries into blocks 40 by 90 ft.

The 350 by 1200-ft. blocks are then subdivided into blocks 90 ft. square by driving rooms 10 ft. wide and 350 ft. long at right angles to the butt entries, the rooms being connected by cutthroughs at intervals of 100 ft. In this manner a whole panel can be developed and prepared to a reasonable distance in advance for the work of concentration in quickly withdrawing the pillar coal.

Fig. 45 illustrates the concentration method, showing a part of a section when in full operation as worked in the mines of the lower Connellsville district. This shows the coal in 90-ft.-square blocks and oblong pillars 15 ft. by 90 ft., also the entries,

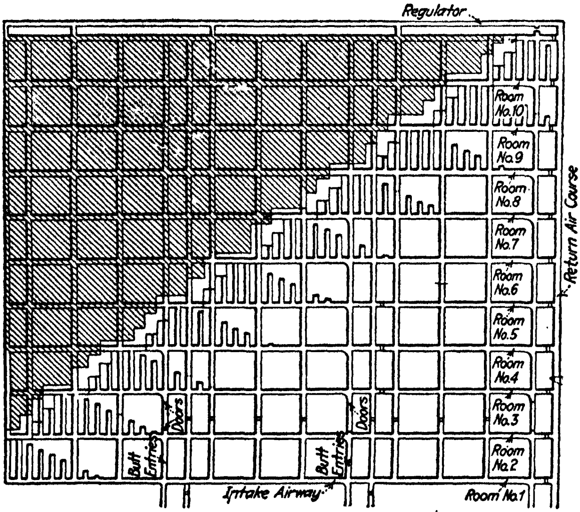


Fig. 44.—Method of laying out mine in 90-ft. blocks used in the Connellsville region.

rooms, crosscuts and cutthroughs. The section shown crosshatched represents that portion from which the coal has all been withdrawn and the overlying strata have subsided, or the "gob" section.

This section of coal is developed by driving on the right hand side a pair of butt entries, 10 ft. in width, 500 ft. in length, on 50 ft. centers and connected by cutthroughs every 100 ft. Rooms 10 ft. wide and 350 ft. long, on 100 ft. centers, are driven at right angles to the butt entries, the rooms being connected by a straight line of cutthroughs at distances of 100 ft. apart for ventilation. Thus, the room pillars are divided into 3½ blocks 90 ft. square, as shown on the plan in preparation for the final operations of driving the crosscuts and the withdrawal of the pillar coal.

The selection of the place to commence on the pillar work is important and must be determined by the persons directly interested largely from the local conditions—such as the inclinations or pitch of the coal bed, convenience of transportation or haulage, the size, area or extent of the panel or section available for operations or the required daily tonnage.

Preparations are now completed for the essential part of the concentration work. It will be noticed on the map that the pillar work at the end of the last room, which was in the

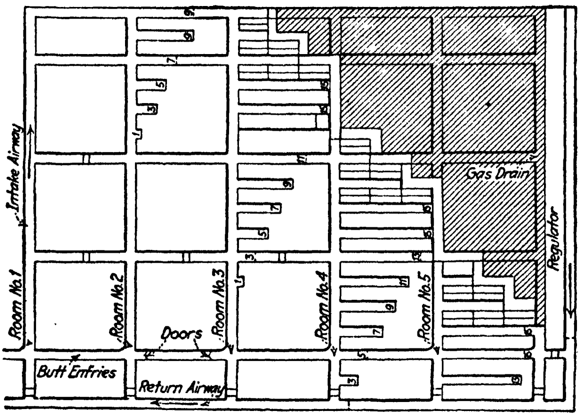


FIG. 45.—The concentration method used in the Lower Connellsville region.

upper left-hand corner of the plan, has been started. Each room having been divided into $3\frac{1}{2}$ blocks, 90 ft. square, almost three and a half blocks from the last, or No. 5 room have been worked out; nearly one and a half blocks from No. 4 room; one fourth of a block from No. 3 room and crosscuts started in No. 2 room. This makes the angle of the gob line about 45 deg. with the butt entry.

The plan shows that the pillar work of each room is 75 ft. in advance of the pillar work or gob fall of the room following. The room pillars are kept in this position for the purpose of breaking the roof falls in the advancing pillars and offering more resistance to thrust caused by the breaking of the strata; also for the purpose of affording better protection against crushing in the pillars of the room following.

The system so far as this pair of butt entries is concerned is now in full operation. The pillar work or gob line, however, may be, and often is, extended across several pairs of butt entries (see Fig. 45), leaving an offset of about 75 ft. as a breaker at each pair. In this manner the gob line may be extended clear across the panel at an angle of 45 deg. or even at 35 deg.

The work on this plan was commenced at the top of the last room in the block, which was in the upper left-hand corner of the map. New crosscuts were started at intervals of two days, thus making each crosscut two days' work, or about 12 ft. in advance of the one following, until the crosscut first started has been driven through the 90-ft. block, for which 15 days were allowed at 6 ft. per day.

The crosscuts being 10 ft. in width and turned off the rooms at distances of 25 ft. between centers makes the pillars, for the final operation, 15 ft. in width and 90 ft. in length. These pillars are divided and subdivided by lines drawn across and lengthwise of the pillar. The cross division lines divide the pillar into three sections of 15 × 30 ft. each—the amount of coal to be taken out in one fall by three or four men working two days—which makes three falls to each pillar in six days' work, as shown.

Three of these pillars in each room are being worked at the same time and are started at intervals of two days, thus placing each pillar two days' work, or 30 ft. in advance of the one following.

The proper time to start the first crosscut at the top of the next room in order that the 75-ft. offset may be maintained can be ascertained as follows: It takes 15 days to complete the first crosscut at the top of the last room, six days to withdraw the pillar, two days to finish the next pillar and two more days to finish the next one, making a total of 25 days to complete the withdrawal of the three pillars or cover a horizontal distance of 75 ft. Therefore, by starting the first crosscut in the next room 10 days later, the offset of 75 ft. will be maintained as shown on the plan.

By continuing the work on this schedule, leaving intervals of two days between the starting time of each crosscut and 10-day intervals between the starting time of the crosscut in the next room, we shall have three room pillars of 90-ft. thickness retreating in a diagonal line on each pair of butt entries and three of the crosscut pillars in each room pillar or 9 working places when in full operation.

In estimating the amount of daily output from this section, there are in operation 15 crosscuts and 9 pillars. Allowing one man in each crosscut and three men to each pillar makes a total of 42 loaders. The coal being undercut to a depth of 6 ft. by the mining machines, the tonnage of each crosscut 6 ft. undercut, 10 ft. in width and 7 ft. in height of seam, will be $6\times10\times7=420$ cu. ft. Allowing 80 lb. to each cubic foot and 2000 lb. to the ton, we have,

$$\frac{420\times80}{2000}=16.8 \text{ tons,}$$

and 15 crosscuts equals 16.8×15 or 252 tons. Assuming a two-ton car, this gives us $252/2=126$ loaded cars.

From the 9 pillars having 6-ft. depth of undercut, 30 ft. in length of pillar and 7 ft. in thickness of seam, there will be

$$\frac{6\times30\times7\times80}{2000}=50.4 \text{ tons,}$$

and for 9 pillars,

$$50.4\times9=453.6 \text{ tons, or } 226.8, \text{ 2-ton loaded cars,}$$

which makes the total number of tons from the whole section

$$252+453.6=705.6 \text{ tons, or } 352.8 \text{ loaded cars from 42 loaders,}$$

being an average of 8.4 cars for each loader. This average may seem rather high until we take into consideration the facilities afforded by the concentration method and the preparations made to enable the miner or loader to attain a high efficiency.

During the previous night all working places are undercut to a depth of 6 ft. by mining machines. The shot holes are drilled with power drills by men employed especially for that purpose and are charged, tamped and fired by shotfirers using permissible explosives, clay for tamping and electric batteries for firing.

By this system each miner, when he arrives at his working place, has about eight or nine carloads of loose coal, which he can at once begin to load, provided no unusual difficulties arise to prevent him. He is kept constantly supplied with empty cars by the driver, who can also attain a high efficiency by reason of having the loaders within a comparatively small area and only a short distance from the side track. The tracks are kept in good condition and laid with steel rails, even in the miner's places.

The trackmen, timbermen and laborers are also enabled to do more effective work, as there is no lost time in traveling long distances from place to place. For the same reason much better supervision can be given by mine foreman, assistant mine foreman and firebosses to the workmen and working places. They can make frequent visits and keep in close touch with the workmen and other matters influencing the success of the operations—such as the machinery, transportation or haulage, ventilation, trackmen and laborers, timbering and timbermen, miners and drivers—and see that defects interfering with work or causing delays are remedied immediately.

Comparative cost of mining different thickness of coal.—An interesting study of the determination of the minimum thickness of anthracite coal that can be economically mined was presented in a paper read before the Engineering Society of Northeastern Pennsylvania in 1914.

In deciding which beds are and which are not minable, we face, at once, the question of profitable operation, and it may be conceded that other things being equal, beds which are 6 ft. and over in thickness are more cheaply mined than those which are thinner. If we eliminate all variations other than cutting and loading in making our calculations, the relative cost of mining for varying thicknesses is a matter of simple calculation.

Let a = allowance for rock in cents per inch per yard;
h = normal required thickness, in inches, on which allowance is based;
x = thickness of coal, in inches, as measured for allowance;
x^1 = net thickness of coal, in inches;
f = capacity of mine car, cubic feet;
w = width of chamber in feet;
s = thickness, in inches, which will give one mine car per yard of chamber. Assume loose coal occupies $1\frac{1}{2}$ times the volume of an equal weight of solid;
c = cents per car allowance;
m = mining price per car, no allowance.

Then

$$(h-x)a = \text{allowance per yard of chamber;}$$

$$f=\frac{3ws}{12}\times 1\tfrac{1}{2}.$$

$$\therefore\ s=\frac{8fx^1}{3ws}=\text{number cars per yard of chamber width;}$$

$$c=\frac{(h-x)a}{\frac{x^1}{s}}=\frac{has-asx}{x^1}.$$

From this, for any particular conditions, the cost for each thickness may be calculated, and a curve constructed showing the relation between cost and thickness, as shown in the diagram, Fig. 46.

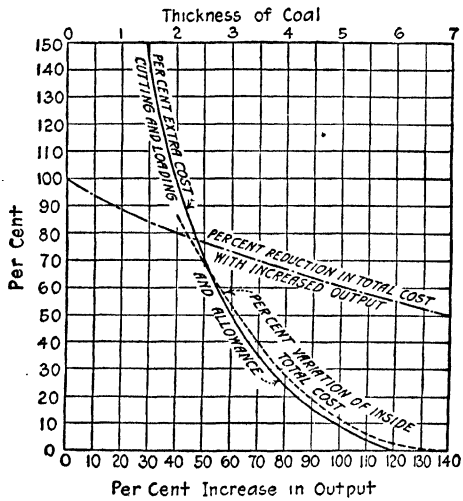


FIG. 46.—Relation of thickness of seam and output to cost of coal.

Unfortunately, all the costs which vary with the thickness of bed are not susceptible to calculation, but in general the inside costs increase rapidly with the mining of thinner coal, and the diagram showing the variation of cost with thickness is believed to represent fairly the average conditions in the anthracite fields. It was constructed by plotting a large number of actual costs and then drawing the average curve.

We find ourselves facing the dilemma whether we shall mine the coal which is profitable in itself or a mixture of profitable and unprofitable coal which, through the preponderance of the latter, will result in a profitable output. Our first thought would naturally be that only the coal which is actually profitable should be mined, but when we make a more complete inquiry we find another condition in the relation of output to cost. As but approximately one-third of the inside cost is expended in actual cutting and loading, and as the greater part of the outside cost is but slightly dependent upon output, the cost per ton will be found to vary greatly with the coal production, even with a fixed unit cost for cutting and loading. How great this variation may be is indicated on the diagram, and it is apparent that a large output from beds which show a relatively high mining cost may be actually more profitable than a much smaller output exclusively from the larger and cheaper beds.

Hence, it is apparent that, even from the standpoint of immediate profit, it may be advisable to mine the thinner beds with the thicker, and considering the ultimate yield of a property, there can be no question as to the advisability of such a course. The actual minimum minable thickness being dependent upon so many conditions is not susceptible to general determination and should be studied for each individual case.

Conveyor system.—The method of operation by conveyors herein described has been in use in a number of collieries working some comparatively thin measures in one of the coal fields in Scotland, and has proved its success and applicability through a period of at least 10 yrs. In some respects the method adopted was unusual in that while conveyors are in operation by themselves no coal-cutters are employed, undercutting being done by hand. Present-day practice always considers conveyor work an adjunct to machine mining; but here is a case of conveyor practice by itself. Another distinctive feature is that the opening and development of the mine for additional faces, as well as running the usual longwall faces with the conveyor, are being done with a conveyor wall.

The coal seam on the average is 3 ft. 9 in. thick, but owing to the presence of stone bands is rather broken up. This means that after removing 31 in. of coal there remains about 14 in. of stone to be disposed of. The thick stone parting near the bottom of the coal is a yellow-white sandstone that breaks in flat squares, eminently suited for building the road pillars in longwall working.

In working under the old system, "dooks," or "deeps," were driven direct to the dip, the inclination being 8 deg. These "dooks" were driven in the solid coal, with pillars turned off every 150 ft. on the dip and 60 ft. on the level course. Every 300 ft. levels were broken off right and left, and a longwall face commenced two pillar lengths from the center deep, in a fan-shaped fashion, which as it opened out gradually edged uphill until its upper corner worked along the waste of the level above, and the face stretched from one level to another.

In driving the "deeps" three roads were allowed—one for haulage and intake air current while the two on either side where needed for return air. In order to operate the longwall faces at low cost, slants were driven uphill from the lowest level, and from the slant several parallels to the main bottom level turned into the coal face, these being a distance of 40 ft. apart.

During the ordinary longwall methods of working the following men were employed in the section: Miners, 14; brushers, 7; trackmen, 2; timbermen, 3. The output was 45 tons.

The miners trammed their own coal to the main level. In the layout of the workings for the conveyor there was little actual difference in the direction of development. The faces still extended across the strike; but in place of the three parallel deeps a longwall face was laid out at an angle between the dip and strike, so that the left-hand end was the most advanced, which allowed the coal and water to gravitate to one point. The driving and formation of pillars were thus dispensed with, and the longwall system of extraction became a developing system as well.

The conveyor in use is of the shaking type, and has been adapted from continental practice. The height from the floor to the edge of the pan is only 9 in. This means that the workman is required to raise the coal only slightly over this height instead of a former 29 in., which accounts for more work with decreased effort. The width is only 18 in., and this allows of the distance from coal face to waste line of props being kept at a minimum.

The principal dimensions of the driving engine are as follows: Horsepower of engine, 12; air consumption at 60 strokes per minute and 60 lb. pressure, 18 cu.ft.; stroke of engine, 5 in.; diameter of cylinder, 7 in.; weight of engine, 572 lb.

The engine, driven by compressed air, is a simple, plain, air cylinder, with broad bed-plate, which may be bolted to planks, which are in turn wedged against the roof by timber. The total width is 18 in., and the length 24 in. Connection to the conveyer is through a lever action and rigid attachment, the cylinder being placed at right angles to the line of the conveyor.

The pack walls on the top side of the driving road are uniformly built in line, 2 ft. or so back from the edge, this space being utilized for the engine. Air is furnished from a power-driven air compressor that stands in a small recess in the side of the road. The principal dimensions are as follows: Horsepower, 15; r.p.m., 960; amperes, 16.5; voltage, 500; cycles, 50; air pipe, 1½ in.; cylinders, 4; strokes per minute, air cylinders, 480; pressure, 70 lb.; space occupied, 5 ft. 6 in. by 3 ft. 5 in. The air is passed through an 18-ft. hose to the air cylinder of the conveyor.

The rate of advance changed from 160 ft. in six months under the old system to 270 ft. over the same period under the conveyor system. This is not remarkable compared to machine working advances, but it represents a considerably increased rate of extraction for handwork. The operation of each face in the colliery is regular and at the same rate, the only determining factor being the number of men employed.

Shifting of the conveyor takes place every second night, so as to get under the fresh rock as soon as possible; but this is governed by the rate of cutting and loading. The air engine is moved at the same time as the conveyor, but the compressor only when the length of hose is reached. The operation of shifting is accomplished by a night force of eight, who also shift the compressor when necessary and set all timber required.

A comparison of the costs of operation and performance accomplished by the old and the new systems has worked out much as follows:

	Hand Operation	Conveyor Operation
Tons per man	3 2	4.6
Length of face	300 ft.	300 ft.
Tons, per section	45	55
Length of face, per man	43 ft.	40–50 ft.
Number of miners	14	12
Number of deadwork men	12	2
Number of conveyor men	......	1
Shifting conveyor	...(Average per night)	4
Number of roads to maintain	7	2
Tons per road	6.4 Main level Bottom "	49 00 6 00
Time stripping	9 hr.	9 hr.
Distance bottom level in advance	......	40 ft.

COSTS

Cutting, Shooting, Loading	0.72	0.72
Brushing	0.41	(Done by squad shifting conveyor)
Maintaining roads	0 08	0.08
Tramming	(Included with shooting and loading)	(Included for lower level in tonnage rate)
Shifting conveyor	......	0.15
Operating conveyor	......	0.03
	$1.21	$0.98

It will be seen that the saving in cost finally effected is entirely due to the elimination of roof troubles, which the conveyor system made possible. There are now installed 10 conveyor faces at this operation.

Mining machinery.—Man-power is about the most expensive energy purchasable. We pay a laborer, say $2 for 9 hr. work. This man is capable of exerting continuously about one-eighth of a horsepower. In other words, we have secured $1\frac{1}{8}$ hp.-hr. for \$2, or we pay about \$1.78 for man-power per horsepower-hour.

In marked contrast to this high cost of energy is the cost of current delivered to the motor of a mining machine which should not exceed 2 to $2\frac{1}{2}$c. per horsepower-hour.

It is the realization of this discrepancy between the cost of power developed by man and that developed by a steam engine, for instance, that is driving the coal industry to employ machinery wherever such employment is possible. Furthermore, it is frequently the case—as in undercutting, for example—that the machine does the work better; that is, it cuts deeper and affords less resultant fine coal than when mining is done by hand.

It is probable that most operations that may be performed by machinery require a greater expenditure of power than would the same operations performed by hand; nevertheless it has become almost axiomatic that it is economical to supplant manual power by machinery wherever possible. Consequently inventors are continually striving to perfect mining machines, and other power-driven devices that will do away with the employment of muscular energy.

Cutting machines.—Mining machines now produce about 65 per cent of the nation's coal output as compared with 35 per cent in 1907. The economies over hand mining may be summed up as follows: First, the actual cost of mining is lower, due to the fact that the greater cutting capacity of the machine makes possible a greater output with a given labor cost; second, the quality of the product is superior, due to the deeper and more uniform undercut of the machine, which increases the percentage of lump coal 10 to 30 per cent over hand mining methods; third, the mine may be more rapidly developed due to the much greater speed with which entries can be driven with machines insuring a rapid return on the capital invested; fourth, the ability to mine seams in which the height of the coal, or the character of the roof, has prevented mining by hand, on a commercial basis.

During the period from 1891 to 1914, the average tons of coal mined per mining machine, per year, in the United States was about 13,700. In 1913 there were 2208 shortwall mining machines in use and in 1914 there were 3024, an increase of 32 per cent in one year. In 1914 there were 6859 breast machines in use, which is an increase of 100 per cent over the year 1904.

One of the most noticeable increases in coal production mined by machines has been in Kentucky, where, in 1912, 66.4 per cent of the coal was mined by machines, while in 1914 the production mined by this method was 77.2 per cent.

In the accompanying table the unit of efficiency is given for the total production of bituminous coal in the United States for the years 1891 to 1915. There is also a column showing what percentage of the coal was machine-mined. By machine-mined coal is meant all coal won by the use of any of

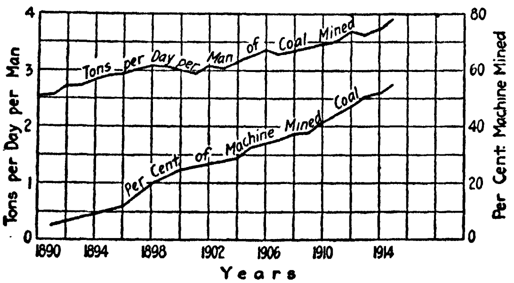


FIG. 47.—Curves showing per capita production and per cent of machine-mined coal.

the following types of machines: Punchers, radially mounted punchers and chain breast, shortwall and longwall cutters. The figures given in the table were taken from the reports of the Bureau of Mines on the yearly production of coal.

From these statistics it is at once apparent that the increase in the number of tons mined per day per man has corresponded closely with the increase in the percentage of machine-mined coal. In order to show this more clearly the two curves shown in Fig. 47 have been drawn. The upper curve shows the tons per day per man and the lower the percentage of machine-mined coal. From these two curves, unless some radical changes are made, it can be estimated that about the year 1928 all coal will be machine-mined and that the efficiency of the miners will have increased to about 4.9 tons per man per day.

Proportion of Machine Mined Coal in the United States and Production per Man

Year	Average No. of Men	No. Days Worked	Total Tonnage	Tons per Man per Day	Per Cent Machine-Mined
1891	205,803	223	117,901,238	2.57	5.26
1896	244,171	192	137,640,276	2.94	11.86
1897	247,817	196	147,617,519	3.04	15.35
1898	255,717	211	166,593,623	3.09	19.46
1899	271,027	234	193,323,187	3.05	22.74
1900	304,375	234	212,316,112	2.98	24.86
1901	340,235	225	225,829,149	2.94	25.61
1902	370,056	230	260,216,844	3.06	26.75
1903	415,777	225	282,749,348	3.02	27.58
1904	437,832	202	278,659,689	3.15	28.21
1905	460,629	211	315,062,785	3.24	32.82
1906	478,425	213	342,874,867	3.36	34.66
1907	513,258	234	394,759,112	3.29	35.11
1908	516,264	193	332,573,944	3.34	37.04
1909	....	....	379,744,257	....	37.52
1910	555,533	217	417,111,142	3.46	41.72
1911	549,779	211	405,907,059	3.50	43.89
1912	548,632	223	450,104,982	3.68	46.80
1913	571,882	232	478,435,297	3.61	50.07
1914	583,506	195	422,703,970	3.71	51.70
1915	557,456	203	442,624,426	3.91	55.00

Mining machines and the necessary equipment for successfully operating them at the average colliery cost a large amount of money. If this is injudiciously spent in equipping a plant for cutting coal with machines, overhead charges for interest, maintenance, depreciation and taxes will be correspondingly heavy. Conditions may justify the expenditure in order to properly recover certain coals and at the same time safeguard life and property; but such moneys should be carefully and wisely expended and then only after exhaustive analysis of conditions surrounding the proposed operation. The fact that one's neighbor mines his coal with machines is not sufficient reason for one to so equip his own property. Usually each mine, and especially each coal, has its peculiarities that deserve careful consideration. Many pointers and suggestions that are worthy of serious deliberations may be had from the man at the face. Such suggestions only cost when ignored or neglected.

It is fair to assume that all up-to-date companies maintain accounting systems that are a criterion by which they may determine approximately the relative costs of pick-mined and machine-mined coal. However, there are many angles that afford viewpoints not generally considered in this connection.

Interest on investment, maintenance, depreciation and taxes on all extra equipment over that necessary for the successful operation of the property as a pick mine should properly be charged to machine-mined coal. In this list should be included all extra housing, boilers, boiler settings, boiler fittings and accessories such as feed pumps, steam headers, etc., in addition to generating units, settings, switchboards and accessories, transmission lines and machines. Further, also, under the head of supplies should be included and charged to machine-mined coal all extra repairs, fuel, water, oil, tools and office supplies over and above that necessary for the successful operation of the property as a pick mine; and under the head of labor, should be included and charged to machine-mined coal all extra expenditures for electricians, wiremen, firemen, oilers, drivers, tracklayers, bit sharpeners or blacksmiths and clerical force over and above that necessary for the successful operation of the property as a pick mine.

Men are quite frequently required to timber after machines, clean slate and refuse at switches and turns on account of the additional space required for machines to turn; extra drivers are also frequently necessary due to the fact that they are required to get sharp bits to the machines and dull bits to the shop, must occasionally await the moving of a machine thereby losing time and in some mines they must drive further for their loads or past one extra place out of every three, due to the fact that three working places are allowed each two loaders. Extra track layers are frequently required for the same reason, viz., that they have more track to keep up and over a larger territory due to the fact that three places are allowed two loaders. Bit sharpener or extra blacksmith should properly be charged to machine-mined coal where the machine men are not charged for smithing.

Purchasing agents and bookkeepers spend considerable time in ordering machine supplies, checking freight bills and keeping track of supplies. Delays and shutdowns due to trouble with boilers or generating units should properly be charged to machine-mined coal where machines are responsible for such trouble. As an example, there are sometimes delays in both hoisting and haulage due to the generating plants being overloaded by reason of having been required to furnish power to the machines.

It is quite possible with the advice of the machine makers to buy equipment that will suit the underground conditions at the coal face, but the organizing of all the factors that make for successful operation of a mine to the new conditions created by the advent of the machine is a subject conveniently forgotten by the seller of the apparatus, and often not considered by the operator. To buy equipment without looking into this question is much like purchasing an automobile for which gasoline cannot be readily procured. Consequently, organization and reorganization of the mine are the most important factors in success. Consideration of the following table will serve to more clearly illustrate this point.

CONTRAST OF HAND- AND MACHINE-MINING CONDITIONS

	Under Hand Conditions	Under Machine Conditions
Tons	100	360
Number of roads to be kept open	18	18
Tons per road	6	20
Men in section	40	59
Timber to handle, single pieces	3000	12,108
Cars of coal	100	360
Rate of advance, inches	12	42

It is evident at once that the greatest feature is the traffic increase. Instead of 200 cars a day to deal with there are now 720, besides an additional number at night. Instead of 3000 pieces of timber to take in there are now 12,000 to supply. Other supplies have increased due to the use of the machine, and with the increase in traffic, ventilation of the mine workings has to be maintained at a higher state of efficiency.

The finest machines may be worthless if the system of back-ing them up fails. Ordinarily in all coal sections the amount of work done is limited or controlled by one single factor. This may be the amount of cars provided, the size of the section or the capacity of the outside haul. In machine mining only one thing should control the section, and that is the tonnage produced at the face each night. Everything should be subordinated and coördinated thereto.

Before the men leave their places at the face, the coal should all be squared up properly so that the machine can get to work promptly. If there is any coal left behind not cleared up, broken down but not loaded, or "noses" overhanging, a man should be sent round in advance of the machine to see that all these obstructions are cleared away. This extra cost is more than offset by the gain made in the time of the machine, as well as by the elimination of the risk of the possible loss next day.

The tracks by which the machine travels from place to place should be so arranged that the time lost in travel is reduced to a minimum. In a thick coal, where the machine cuts a relatively high tonnage in each place, there is more coal to set against this waste but with thin coal this lost time runs up alarmingly, because the amount of coal in each place is small.

It is obvious that all machines are doing their best work where the going is continuous. Idle time and idle men, or those employed on unproductive work, mean a loss. The traveling from place to place results in the loss of a certain amount of productive cutting time, and it should therefore be cut to the minimum. Tracks and curves should be easy and well placed, so as to facilitate traveling, and trouble in this connection should never occur twice running at the same spot. Nightmen should be at work on the bad piece of track that same evening.

Machine supplies should be kept handy to the face. If the section is a long one, supplies should be kept at several points. Tool chests with proper keys should be provided, otherwise the oil is often found to have disappeared, together with necessary hammers, keys and similar tools. Machinemen should be capable of making reasonable repairs themselves, and the materials for doing so should be kept on the ground.

There should be no hard and fast rule that only electricians are to repair machines, neither should every handy man be allowed to try his hand at machine troubles. It is a good thing to have spare machines. Each of the machines underground should be taken to the surface at regular intervals for overhauling. Hardly any class of machinery, unless it is the pumps, receives worse usage than does the coal cutter, and in the dark and poor light underground defects may develop that will never be noticed until it is too late to make the proper repair. All machines should be operated steadily on the surface for a number of hours and thoroughly oiled and tuned up before being returned within the mine.

Machinemen should be at their machines an hour or so before the time of starting work, in order that each cutter may be overhauled and oiled, the bits changed, and all such details given the proper attention. Spare bits should always be on hand and any that have been removed should be sharpened in the morning, to be sent down again at night. The proper place for all bits except those in stock in the warehouse is in the mine and not in the blacksmith shop. One smith should sharpen all these tools the first thing in the morning, regardless of any other work. It should never be necessary to stop the machine to hunt for cutters or telephone to the surface at two in the morning.

When electric current is used, the ratio of power given out by the cutter motor to the power required to drive the generator may be taken roughly as 70 per cent, while with compressed air the ratio of power given out by the machine to power required to drive the compressor may be taken as in the neighborhood of 35 per cent.

In other words the steam consumption of a compressed-air plant for coal-cutting is about double that of an electric plant. The figure 70 per cent taken for the electric cutter will not differ much whether the installation is well or badly designed, but in a poorly planned and badly maintained compressed-air plant, a considerably lower efficiency will be obtained than the 35 per cent taken as representative of a moderately good installation.

Installation and operating costs.—The cost of a 5-machine plant may be summarized as follows, figures as of 1905:

Power plant, including boilers, air compressor, air receiver, feed pump and feed-water heater......	$ 5,181.00
Mining machines, including all accessories, and freight..................................	4,125.00
Installation of power plant, including freight, compressor foundation, boiler settings, wooden boiler and engine house, water tank, fittings, piping, labor, etc..	2,260.00
Pipe lines above ground and in the mine, with all fittings, labor and freight....................	2,484.00
Total cost............................	$14,050.00

The expense of maintaining and operating this plant may be approximately estimated as shown below figures as of 1905:

Interest (6 per cent) on investment, depreciation (10 per cent), repairs and renewals on machines and power plant and extensions of pipe lines....	$ 2,250.00
Fuel, 6½ tons per day of one 8-hr. shift, at 50c. per ton, and oil and waste, 50c. per day; per year of 200 working days..........................	750.00
One engineer at $75 per month; one machine boss and pipeman at $75; one blacksmith to sharpen picks at $60..............................	2,520.00
Total maintenance....	$ 5,520.00

The above figures are considered the maximum, so that in actual practice, the cost of maintenance of the plant will probably be lower.

If we assume the pick rate at this mine to be 60c. per ton, and the differential rate for the machine runner to be one-fifth of the hand rate, then the machine rate will be 12c., and the loading rate 30c. Adding 3c. for blasting makes 45c., leaving a margin of 15c. for profit and payment on the plant. With an output of 700 tons per day and considering 200 days in the year the annual gross saving will be $21,000. If we now deduct the total cost of plant, including maintenance, from this saving, we have, $21,000 minus $19,570 which leaves a net profit of $1430 at the end of the first year.

Cost of machine mining.—It is practically impossible to compare the actual cost of mining with the various types of cutting machines. The machine that would show a considerable saving at one mine might prove inefficient at some operation. However, when it comes to comparing machine mining with hand mining, there is no difficulty. The most important point to consider is the size of the differential favoring machine mining over hand mining. In many districts this differential, or margin, amounts to about 15c. (in 1910) and it is out of this differential that the operator makes his profit and pays for the plant. At a mine producing 1000 tons per day and having a 15c. margin in favor of machine mining, the gross saving would be $150 a day, or $30,000 per year of 200 days. In such a case, the company can maintain its output with 20 per cent fewer men than are required when hand mining is employed.

Herewith is a statement showing the cost of machine mining with longwall machines at an operation where, owing to the tough and "woody" nature of the coal, which necessitated paying the miners excessive "allowances," the average cost of mining with picks was between 60c. and 63c. per gross ton, figures as of 1910:

Coal Cut by Electricity and Loaded by Day Laborers

	Hours	Rate	Amount	Cost per Ton
Cutting	530	30c.	$ 159.00	$0 0393
Scraping	531	20c.	106.20	0.0262
Trackmen	255	25c.	63.75	0.0158
Trackmen	260	20c.	52.00	0.0128
Shooting	222	25c.	55.50	0.0137
Slate	2716	20c.	543.20	0.1343
Loading	3067	20c.	613.40	0.1516
Foremen	270	30c.	81.00	0.0200
Total labor	......	......	$1674.05	$0.4137
Supplies			$136.01	$0.0336
Depreciation on machines			36.00	0.0090
Interest 6 per cent on $3400			17.00	0.0042
Repairs and maintenance (estimated)			5.00	0.0012
Power			24.04	0.0060
4045 tons, 19 cwt., at total cost of			$1892.10	$0.4677

The above cannot be taken as a typical or average statement, as the conditions at the operation referred to are much less favorable to pick mining than at most other operations in this field. The pick-mining rate along New River is 50c. per gross ton, and at most of the operations on Piney creek and Loup creek it is 40c. per gross ton, as compared with 69c. in central Pennsylvania, so that the economy by the use of machines is much less than would appear from the figures above quoted.

The following are detailed figures of the mining cost at two operations, one where all of the coal is cut by electric chain undercutters, and the other where all of the coal is cut by air-driven punching machines. The pick-mining rate for the year when these statements were compiled was 62c. per gross ton.

Electric Breast Machines, Output for One Year 321,808 Gross Tons

	Per Ton
Labor	$0.4134
Material	0.0259
Insurance and taxes	0.0009
Depreciation	0.0091
Interest charges	0.0025
	$0.4518

Compressed-air Punching Machines, Output for Year 99,207 Gross Tons

	Per Ton
Labor	$0.4810
Material	0.0211
Insurance and taxes	0.0095
Depreciation	0.0132
Interest charges	0.0036
	$0.5284

These two examples cannot be taken as a general average, because in the instance quoted where chain machines are used, the mining conditions are rather exceptionally favorable. In the other case, where punching machines are used, the conditions are about the same as the general average in that field.

The tonnage produced per machine of a given feed varies according to the thickness of the coal, its hardness, the width of the working places and the length of the transfers.

The West Kentucky Coal Co. operating nine mines in the western part of that state with seams ranging from 4 ft. 7 in. to 8 ft. 6 in. thick gets about 150 tons per machine-shift. This is an average of nine mines and includes machines used on development work in areas practically worked out where the output of the machines is naturally limited. In one seam ranging from 5 to 8½ ft. thick the production is 200 tons per machine-shift. The record cut for all the western Kentucky field up to 1917 was 300 lineal feet of face in 9 hr.

At the mines of the W. G. Duncan Coal Co. in this same field where five machines are in use the average production is nearly 250 tons per machine shift. At these mines the nature of the coal is such that they are able to use a 25-in. feed over a 6½-ft. cutter bar or a 21-in. feed over a 7½-ft. bar; they are also getting a good tonnage per bit sharpened, all of which factors make a large production per machine possible.

The maintenance cost is probably the most important item in connection with the mining machine. The successful and economical operation of a mining machine, like any other piece of machinery, depends largely on the human element. By using care in selecting hostlers who will later become runners, an efficient machine organization can be built up.. There is an instance where one man operated a machine continuously for four years without calling on the machine boss except occasionally for some small repair part to replace one actually worn out.

Unfortunately, men of this type are not numerous, and for the average runner some incentive is necessary to sufficiently interest him in getting the best results from his machine.

At the mines of the West Kentucky Coal Co. a bonus system is in effect as follows: A general supply stock is kept at each division and all supplies as purchased are charged to this supply account. One machine boss has charge of the machines and other electrical equipment in use at each division, with one or more helpers as conditions may require. All supplies are issued by either the machine boss or his helper and charged to the mining-machine account of the mine where used. The average maintenance cost for the year 1915 was taken as a standard.

The maintenance cost per ton mined for the year, including the month for which the bonus is to be figured, is subtracted from the standard described above. This difference, multiplied by the tonnage of the month, is divided equally between the company and the machine and repairmen.

It can readily be seen that in this way the men can make a gradually increasing bonus, and this they have succeeded in doing. At the same time it is impossible by skillful manipulation of supplies on the part of machine and repairmen to show a large premium in one month followed by a high maintenance cost and no premium the succeeding month, which could be arranged were each month figured separately.

As an example, to show how this premium system works out: A mine having a machine maintenance-cost standard of $2\frac{1}{4}$c. per ton, determined as above, produces in a given month 20,000 tons. The cost per ton for the year, including the month in question, is $1\frac{1}{2}$c., making a net saving on the tonnage of the month of $150, which is divided as follows: $75 to the company and $75 among five machine runners, five hostlers and one repairman, in proportion to their earnings for that month. For the most part the men engaged in the operation of the machines have taken unusual interest in reducing maintenance costs, and the men at the different mines rival one another in the attempt to establish the best record.

The cost of supplies for the year 1915 was 0.96c. per ton. This includes bits, bit boxes, cables and all supplies used in the operation of machines, but does not include depreciation.

In this connection it is worth notice that the first machine purchased has been in continuous service for 11 yr. and is still in as good operative condition as one just out of the factory.

From studying the results of several machine installations, particularly those of the United States Coal & Coke Co., and the Clinchfield Coal & Land Corporation, it was found that deep undercutting is a decided advantage, and where consistent with the mining conditions, nature of coal, etc., should be recommended since it reduces (1) first cost of machine installation, (2) powder consumption, (3) cutting cost per ton. Moreover, where a certain output is expected, deep undercutting should reduce the territory under development. This is an important feature when the maintenance of roads, ventilation, and supply of timber are taken into consideration. Furthermore, it should increase the percentage of lump and give the loader a more definite or dependable quantity of coal down, thus increasing his efficiency, and thereby raising that of the entire plant.

Comparative cost of alternating and direct current for machines.—In a mine where alternating-current machines are to be utilized, large tonnage should be developed in as small an area as possible in order to secure the most economical use of the machines. This is shown in the accompanying diagram of a typical room-and-pillar working, Fig. 48.

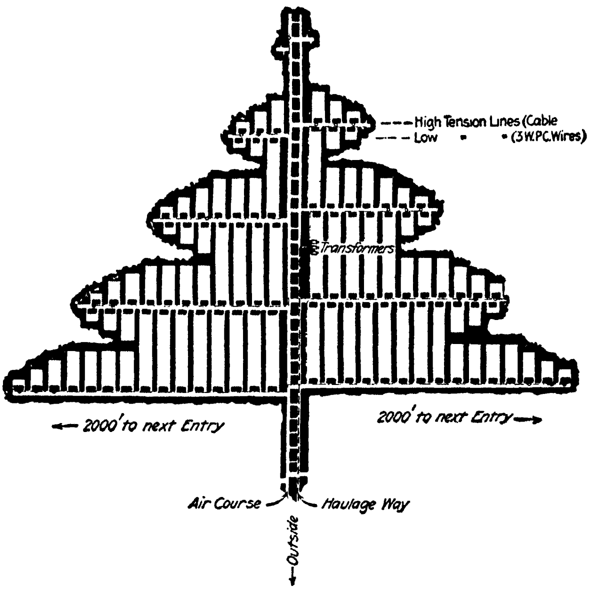


FIG. 48.—Typical room-and-pillar development using alternating-current machines.

In this mine the rooms available for the mining machines are possibly far in excess of what the mining machine is capable of cutting. In territory distributed as in this example one mining machine should cut at least 7 rooms in a working shift or 14 rooms on a double shift. This would give approximately 300 lin.ft. of coal per shift, depending, of course, upon circumstances, the above example being under average working conditions.

The power cost of operating a single alternating-current mining machine is comparatively low, as with the arrangement laid out in the diagram one mining machine would not use over 2000 to 3000 kw.-hr. per month working single shift. This small consumption of power is principally due to the mining machine having sufficient voltage at all times, and, therefore, working at its highest efficiency.

One mining machine in 4-ft. coal and with a 7½-ft. cutter-bar mining say, 300 tons to the shift, or 6000 tons a month, will have a power expenditure of less than ½ kw.-hr. to the ton. This would be less than 1c. per ton on the average rate of central-station contracts.

A comparison of maintenance between the alternating- and direct-current machines shows favorably for the former. With direct-current power in use, especially with a mine that is supplied from some isolated power house on the outside of the mine and a considerable distance from the workings, the voltage is usually low; consequently, much armature trouble is experienced in the motor on the cutting machine.

With an alternating-current mining machine served from central-station power there is an assurance of good voltage, as at no place need the machine be at a greater distance than 1500 ft. from the distributing transformers. This distance in any mine will give the mining machine more territory than it is possible for it to work.

Alternating-current power is probably more economical for general mining use, other than haulage. In fields where central-station power is available pumps, hoists, fans and tipple equipment are all run by alternating-current power and appear to give more satisfactory results than direct current.

In mines where alternating current is available and electric haulage is required it would be an added expense to install wire for alternating-current cutting machines, as the conductors that supply power for haulage can be used for direct-current cutting machines. This accounts for the general use of direct-current cutting-machines in the large mines where electric haulage is employed.

To the small operator with limited capital the alternating-current mining machine has shown the way to a greatly increased production without the large investment formerly required. It is possible to decrease mining costs and increase production at a comparatively small expenditure above the cost of the machine itself.

Arc wall cutters.—The Jeffrey-Drennen adjustable-turret coal cutter, was designed to make a cut at any elevation desired. This was installed at Jenkins, Ky., where the coal seam varies from 6 to 8 ft. in thickness. It is clear, bright and free from sulphur or other impurities with the exception of a band of shale located at a height of from 2 to 5 ft. from the bottom. This varies in thickness from nothing to 19 in.

With the customary methods of undercutting it would be impossible when shooting to prevent this shale from becoming mixed with the coal, but by the use of a machine adapted to cutting out or removing this parting before the coal is shot down this difficulty is overcome.

The machine is mounted on a turnable truck, which carries four heavy standards or uprights, on which the machine proper is raised or lowered or adjusted to the desired height at which to cut out the dirt seam. The cutter is designed for a minimum height of 2 ft. from the bottom, and can be adjusted to cut at any position between 2 and 5 ft. The raising or lowering of the machine is accomplished by power through a disk friction clutch, which enables the operator to absolutely control the elevation of the machine to a nicety, 3 ft. of a vertical movement being accomplished in about 25 seconds.

The cutting is done in the shale at the bottom of the band with the lower nose of the bits cutting into the coal about ¼ in., which causes the shale to fall down in the kerf, after which it is cleaned out, loaded in cars and hauled out of the mine. This insures an absolutely clean product.

A 15-ft. place can be cut in 11 min. from the time the machine enters the room until it is ready to leave. Twenty-five rooms have been cut in a shift of 10 hr.

The Utah Fuel Co. of Somerset, Colo., cut 258 lin. ft. of coal in 2 hr. 24 min. with this machine. The vein is 14 ft. thick.

Post punchers.—Compressed-air post mining machines of the radial type were installed at the Pacific Coast Coal Co.'s mines about 1909, and it was found that after a mining was put in with these machines the coal could be sent down the chutes with only a little pick work, and without any powder, except an occasional light shot at a corner or to shoot out a "nigger head." In fact the powder consumption was reduced over 95 per cent.

The lump coal was increased in this way from about 25 per cent to about 60 per cent, and the practical elimination of powder made the mine much safer.

The rooms are driven 45 to 50 ft. wide. The first cut is made from a post set 7 ft. from the left rib and about 18 in. from the face. A cut 8 ft. in depth is put in, using an extension bar 80 in. long. The chuck enters the cut, which accounts for the mining being deeper than the length of the extension. After the 80-in. extension has been swung, a 100-in. bar is used to square up the cut. One man operates the machine, swinging it by means of the worm-crank with one hand, and feeding the cylinder forward two or three turns with the other at each end of the swing.

The machine and the posts remain in the room at the face until the room is completed, for there is no shooting of the coal that can injure the machine nor any loading (as in a flat seam) that the machine would interfere with. Hence there is no waste of time due to moving, except from post to post, and little heavy lifting. Two men can set up the machine, with ease, as the heaviest parts (the machine and shell) weigh only 225 lb.

These machines will average about 300 sq. ft. in an 8-hr. shift, or from 45 to 55 tons per machine per shift, according to the height of the coal. While, as stated, the reason for the installation of these machines was solely to increase the proportion of lump coal, even if the cost of mining was increased, the results point strongly toward a material reduction in the cost of mining, after all interest, depreciation, power, pipe line, and maintenance charges have been made against the machines.

In shooting off the solid, the former method, a yardage system of payment was used, the rate being $9.50 for a 50-ft. room. Three men worked together, furnishing their own powder. The reason for using a yardage and not a tonnage system was because of the impurities in the seams, and the pitch.

Repair costs.—It pays well to have the machinery in shape to run a full day when the mine is working. By this means whatever men are in the mine are given a chance to produce some coal, and the cost of production is considerably cheapened because repair costs are reduced to a minimum. This is important, for the cost for repairs is too high in almost every mine.

Furthermore, by giving each machine the needed care there will be an increase in tonnage which will permit of a further saving in the cost of production. In order to lower the repair cost and increase the efficiency of the machinery, it is necessary to have a system simple but accurate, which will give an individual record of the performance and expense of each piece of machinery.

There should be a book kept at each mine by the electrician for the purpose of recording the working hours of each machine, and in this the hours idle should be marked down. With a little attention it will be possible to get the average number of tons or cars each machine is capable of producing, and thus it will be possible to find at the end of each month the number of tons lost through the inefficiency of any machine.

Every night the machineman should fill out a report showing the make and number of his machine and the hours it has been delayed, and he should state any defects he may have noticed in its operation. The electrician should then have these defects repaired and sign the report and forward it to the chief electrician. The mine electrician should also fill out a daily report on this order:

NAME OF COMPANY

Mine.................................. Date..........................

Hours of Labor	Wage Rate	Make and Number of Machine or Locomotive	Repair Parts or Material Used	Cost of Part	State if Broken or Worn Out and Cause
3	25	SS No. 1620 Mining	1 worm gear	$14.50	Broken teeth worn
2	40	machine	1 key $\frac{1}{2}\times\frac{1}{2}\times 6$ in.	0 06	too thin to stand strain

REMARKS.—Advise that this machine be changed to easier work as the section is full of rolls and the machine is too light for that work.—S. G. MILLER.

This report should be sent daily to the chief electrician, who would have each item charged against the machine on which it was used. Thus at the end of each month the exact cost of labor and material used on each machine could be easily found.

The wiremen and bondmen should report the section in which they have been working, the kind of work they did and the material used. This should be recorded, and it could then be seen at a glance how much copper or other material was used to work out a section.

The mine electrician should at the end of each month send a report to the chief electrician, showing the horsepower, make and number of each motor, machine or pump at the mine, the number of hours in use, the amount of time out of commission for repairs and the amount of oil used on it. This, with the electrician's daily report, gives the chief electrician a chance to get down to facts.

For example, if a certain mining machine gives much trouble, he can see at a glance just what part was at fault each time, and with these facts on hand he can proceed to investigate and remedy the trouble. Again, if a certain part wears out on each machine of a kind, he will know that it is a weak part in the construction of that machine, and he can take it up with the manufacturer who will probably be able to suggest some way to overcome it.

The mine electrician should order his supplies once a month, and they should be charged against him and not against the repair cost until they are actually used and charged on his report. At the end of each month he should take an inventory of all the supplies he has at the mine and should be credited with them.

For example, he has $500 worth of supplies on hand on the first of the month, and his requisition shows that he has received other supplies worth $500. At the end of the month, on taking his inventory he finds he has $400 worth left. This shows he has used $600 worth of material. On checking up his daily reports they should balance with this figure to show that everything had been charged in its proper place.

The mine foreman should mark the delays to the machinery on his report also, and this should check with the mine electrician's and efforts should be made to keep them correct.

Each piece of machinery should be treated as a workman. The time in use, the amount lost, tons of coal handled or cut, cost of labor and supplies should all be completely recorded. This will show which is the most efficient machinery to buy for future use and which to discard, what parts wear longest and what parts are most liable to give out first. It also shows the amount of material, such as wire and hangers, bond, etc., in each section of the mine.

In order to get results from this system it is necessary to have skilled and competent mechanics. One cannot reduce costs if the men fail to do their share. It is too frequently the case that companies using first-class machinery have inexperienced and underpaid mechanics tending their expensive equipment, while other companies having a poor grade of machinery have good mechanics.

Many coal companies will buy an expensive piece of machinery and then let an inexperienced man experiment with it. The result is the repair cost is large for the first few years, though it ought to be practically negligible. There is nothing gained by letting someone experiment with machinery, and it is a costly practice. Through ignorance, most of such high repair costs are blamed on the equipment, whereas, if the machinery had been used in the right place and treated as it should have been, there would have been no trouble.

The average cost of repairs per ton of coal mined is about 10c. while, with careful attention on the part of the chief electrician and the mine electricians, it would be an easy matter to bring it down to 5c. per ton, and if all did their best, and the machines are in first-class condition it should be brought to less than 3c. with no interruptions to service during working hours.

Machine bits.—The bit question is an important problem in connection with the operation of machines, for the bit is to the mining machine what the tooth is to the crosscut saw. When using chisel-point bits in connection with up and down pick points, the sharpening of these three classes of bits and their distribution to the different machines is quite difficult. The Sullivan Machinery Co. recognized this difficulty and following the old 5-position chain it later introduced the 9 position superdreadnought. Straight pick-point bits in this chain give a good clean kerf, making comparatively coarse machine cuttings, while placing on the machine only a moderate load.

Formerly all bits were sharpened by hand, but this is a slow and most expensive process. Small trip hammers will do more than any other one thing to ease bit troubles. One man, with a boy to heat the bits, can sharpen 2500 to 3000 bits per day, making on an average a better bit than a man will make without the hammer, because a blacksmith sharpening by hand will endeavor to get the bits as hot as possible so as to save hammering. In doing this he not only makes a badly tempered bit but burns away valuable bit material.

The die used in these hammers may be a home product that will probably be equal to anything on the market and that can be made for $20 per set. For tempering compounds a solution of cheap laundry soap and soft water gives good results at little expense. One cake of soap is used to 25 gal. of water, washing powder being added where the water is too hard to lather freely.

At the beginning of the shift this mixture is heated to the boiling point and the sharpened bits while at a cherry-red heat are thrown into the tempering tub, the hot bits keeping the mixture boiling, insuring a slowly cooled and evenly tempered bit. The object is to furnish each machine with plenty of bits and thus encourage the runner to change them before they become dull enough to load the machine. It is cheaper to sharpen bits frequently than to run the risk of burning out armatures and wearing out cutter chains.

In Nos. 9 and 11 seams the West Kentucky Coal Co. uses approximately 150 sharp bits to cut 100 tons of coal while in the No. 12 seam the tonnage per bit is more than double this amount. The number of bits required may seem high in this instance but it was frequently the case in these mines that 150 bits would be dulled in a single room where rock or heavy sulphur bands were encountered. In spite of these conditions the records over three years at these mines show an average production of 35,000 tons per 1000 bits purchased.

Loading machines.—Because of the decrease in the supply of labor throughout the country and the increasing wages more interest is being centered on ways and means for reducing the amount of muscular energy required in the mining and loading of coal underground. Undercutting and loading are probably the two jobs least sought for in American coal mines, and it is increasingly difficult to procure men who will perform this kind of work.

Many machines have been designed for the loading of coal. Some of them both mine and load the material, while others merely load it. In general, no difficulty has been encountered in devising an apparatus to place the coal upon the cars, the main disadvantage of such devices being that it is impossible to feed cars to the machines with sufficient rapidity to make mechanical loading practicable. As a result such machines stand idle a goodly portion of the working day waiting for the loaded cars to be taken away and their places filled by empties. Furthermore, these machines are, as a rule, cumbersome and difficult to move because of their great weight.

In hand shoveling the height through which the shovel must be raised determines the capacity of the loader. With the same expenditure of muscular energy to raise the coal, a man will load 29 tons into a wagon 52 in. above the rail, 44 tons into a wagon 32 in. above the rail, and 140 tons into a conveyor 8 in. above the floor, allowance being made for clearance in each case. Also it must be remembered that in raising the shovel, the foot pounds expended in raising the body is greater than the foot pounds exerted in lifting the coal. When loading into a conveyor in connection with a loading machine, the coal does not have to be lifted more than a few inches; some of it can be rolled or pushed on, and there is an important saving in muscular energy effected.

The Jeffrey loader weighs one ton. It has the motor and machinery located in the center of the conveyor over the supporting rail, and therefore one man is able to bear down on the back end and slew the machine around at will, and one man can readily push the loader along the supporting rail to any desirable position. The supporting rail is ordinarily placed on two wooden horses. The loading machine is made strong enough to admit of shooting the coal down on top of the front end of the conveyor.

It is provided with a self-propelling truck to move from one place to another.

For best results with the loading machine the mine should be laid out systematically with just enough loaders in each section to keep one under cutting machine busy. For instance, in Fig. 49 is shown a system of ten rooms. In rooms 1 to 5 are loading machines; in rooms 6 to 10 a shortwall machine.

Each room is loaded out in half a shift, therefore all ten rooms are undercut and loaded out once a day. One gathering locomotive will handle cars for the five loading machines, if fairly good size cars are used.

The Jeffrey pit car loader is a simple conveyor, driven by an electric motor, so located as to add to the stability of the machine.

Its novelty would appear to lie in its cost, which is about one-seventh that of a shoveling machine. It will doubtless appeal to those who contend that coal mining is attended with too many delays to tie up large amounts of capital in expensive machinery, and attract those who fear that the mine is no place for a machine designed to pick up the coal.

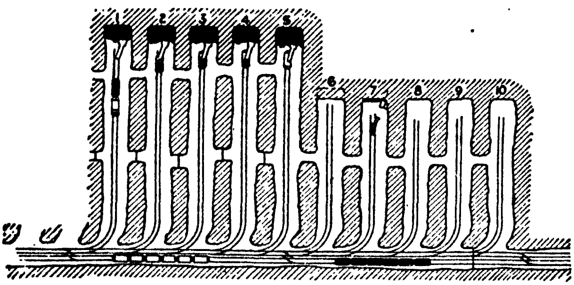


FIG. 49.—System of mining suggested for adoption with coal loading machines.

The claim of its maker that the output from a room can be doubled by the use of this machine appears to be conservative. The fact that it requires only two men to operate, whereas other machines require from four to eight, is vastly in its favor, for when the inevitable delays occur, it is not difficult for two men to find useful work in posting, extending track, etc. Quite a number of these machines are in operation at the mines of the Dominion Coal Co., Canada, some of them for a period of three years or more, and it has been proven that two men will average from 50 to 75 tons per shift.

Three men using this machine can load out the coal from a 16-ft. entry, 6-ft. undercut, in one hour, where the height of coal is 42 in. with a 2 in. to 6 in. parting, the dirt from which has to be picked out.

Three men have loaded 7 cars of slate in 35 min., the capacity of car being 3400 lb. of coal.

In 5 ft. coal with 4 in. binder, two rooms were drilled, shot and loaded out in 8 hr., making 39 cars of coal, each holding 3300 lb. About 6 in. of draw slate and binder were cleaned out and gobbed. The cost of this machine in December, 1921 was $1500.

The Westmoreland loader weighs 10 tons, is electrically operated and self-propelled. The truck is rigid, with 12-in. wheels, one front wheel loose axle, 4-ft. wheel base, and when in operation the truck is clamped rigidly to the rail. A 3-ft. geared turntable rests on the truck frame, and on this is mounted a solid steel case, 12 ft. long, 32 in. wide and 12½ in. high, which swings free above the wheels through an arc of 160 deg., 80 deg. right and 80 deg. left, the width of sweep being 16 ft.

The steel case contains a ram in the form of a chute, 2 ft. wide, 6 in. deep and above the floor, and with an adjustable shovel end at the front. The ram-chute hangs on rollers and is moved forward and backward by a rack and pinion at the top. This represents the unique feature of the machine. The ready horizontal movement of the ram, together with its sweep of 8 ft. on either side of the machine, enables it to keep close to the loose coal throughout the full 16 ft., or slightly more, of width and thus facilitates the work of the laborer.

The adjustable shovel end of the machine takes care of undulating bottom. It is rigid in operation for all sizes of coal, fine slack up to 200-lb. lumps being picked up at the rate of one ton per minute by the chain conveyor in the ram. The actual rated capacity of the loader under fair conditions is 20 tons per hour. Five men are required for its operation, including the driver. The machine will work in seams that are not less than 4½ ft. thick, and will pass over curves of as low as 12-ft. radius. The total power requirement is 16 hp., consisting of three reversible motors.

The Evans scraper loader is an adaptation of the old main-and-tail rope principle to the operation of a modified scraper as a means for conveying the coal from the face and out to the room neck, at which point it is dumped into the car.

The apparatus consists of a double drum hoist, which may be driven by either air or electric power, two wire ropes of suitable lengths, and a V-shaped bottomless scoop, or drag. In addition to this major equipment, each room is provided with deflectors, loading pans and aprons. All coal is loaded on cars in the entry, the plan being to set a trip of empty cars above the room neck so that no time will be lost in spotting another car when one is loaded.

It is figured that with rooms of 300 ft. length it is possible to load at least nine tons of coal per hour. The average haul in this case amounts to 150 ft., and the hoist being geared for a rope speed of 300 ft. per minute the traveling should be done in one minute actual running time. Assuming that a minute is lost at the face and another minute at the loading point, there would be 20 trips per hour, which on a basis of 900 lb. per scoop amounts to nine tons.

It requires five men to successfully operate the apparatus; one hoist man, one man on the entry to stop and trim the cars, two at the face and one timberman. The foremost saving is in the loading of cars.

A full crew for the Myers-Whaley machine consists of one runner and one car coupler, who load as much as 15 to 20 hand shovelers. With these machines the same working places can be loaded out twice a day, instead of once in two days, as is commonly the case. Hence a machine-equipped mine will produce a given tonnage with one fourth the development required where the loading is by hand. This concentration, with the consequent reduction of trackage, ventilation and mine car equipment, effects important economies.

The machine is operated by one man, will work on electricity or compressed air, and is self propelling forward or backward. The machine runs on either.

The machines operate at the rate of 13 to 18 strokes per minute, with a power consumption of .22 kw.-hr. per ton loaded. Three sizes are built for underground mining as follows:

Height	Length	Width	Net Weight	Mine Height Required
No. 2. 46 in.	19 to 21 ft.	4 ft. 9 in.	9,000	4½ ft.
No. 3. 47 in.	20 to 22 ft.	4 ft. 11 in.	11,000	5 ft.
No. 4. 54 in.	22 to 26 ft.	5 ft. 4½ in.	18,000	6 ft.

The machines are all capable of loading at over 1 ton per minute. The smallest handle 30 to 45 tons per hour, and the larger sizes from 50 to 60 tons per hour, actual shoveling time.

A test of one of the earlier makes of this machine was made at the mines of the United States Coal and Oil Co. at Holden, W. Va., about 1910, which is of interest. The coal at Holden

LOADER'S DAILY REPORT

Holden, W. Va., Aug. 31, 1910

Sixth day of official test run.

Time Started 7:30 a.m.	Time Started 12:33 p.m.	No. of Men Working:
Time Stopped 12:03 a.m.	Time Stopped 5:32 p.m.	a.m. 4
Total Time a.m. 4 hr. 33 min.	Total Time p.m. 4 hr. 59 min.	p.m. 4

Working Place	Time Loading and Shifting Cars	Time Changing Machine	Time Lost	Total Time Consumed	Cars Loaded
No. 6 room	1 h.	3 m.	1 h. 1 m.	2 h. 4 m.	24 tons
No. 7 room............	2 h. 29 m.	25 m.	23 m.	3 h. 17 m.	54 tons
No. 1 face room (neck).	1 h. 45 m.	24 m.	26 m.	2 h. 35 m.	33 tons
No. 1 room............	1 h. 25 m.	11 m.	None	1 h. 36 m.	39 tons
Totals..........	6 h. 39 m.	1 h. 3 m.	1 h. 50 m.	9 h. 32 m.	150 tons

REMARKS.—Time lost No. 6 room—loose setscrew on conveyor sprocket tightened between 7:30 and 8:15 = 45 min. Off track 7 min.; pulling down coal 9 min. Total, 1 hr. 1 min.

Time lost No. 7 room—waiting on driver, 1 min.; off track, 5 min.; pulling down coal, 17 min. Total, 23 min.

Time lost No. 1 face room neck—off track, 9 min.; pulling down coal, 17 min. Total, 26 min.

Time lost No. 1 room—none

J. B. HAILE, Machine Runner.

Report by W. Whaley.

is a peculiarly, hard, tough bituminous coal, known as No. 2 gaseous. Its coherent qualities make it somewhat difficult to shoot down and nearly as hard to shovel as large lumps of limestone.

During six days of its operation a record was kept of all features of the run, cars loaded, time required to load and shift, time to change the machine and time lost for any cause. During the six days the machine loaded 768 tons of coal, the time of loading and shifting cars being 36 hr., 6 min.; time changing the machine 4 hr. 21 min.; time lost, 14 hr. 33 min.; total time, 55 hr. The machine loaded out four rooms per day and part of the lost time was due to delay in shot firing, in waiting on smoke to clear out of the room and other items not chargeable to the machine. A copy of daily report of the last day of this test run is shown herewith.

In spite of the delays and lost time the machine averaged 128 tons per day. The lowest day's work (due to lack of coal) was 90 tons, and the highest 150 tons. The average time of loading and shifting a car was 8.4 min., or 21.3 tons per hour.

The territory in which the machine worked consisted of seven rooms recently turned off the airway in No 5 mine. Much of the work was done on curves. The rooms ranged in width from 21 ft. to 27 ft. and all but two had two tracks. The rooms were on the butt of the coal which greatly increased the difficulty of shooting.

The company had no difficulty whatever in keeping the machine supplied with cars, which were hauled to and from the side track on the entry by a mule.

The crew of the machine consisted of four men, as follows: One machine runner, one man in front, and two men to handle cars and pick slate.

Mining and loading machines.—The Ingersoll-Rand cutter and loader consists of a powerful air-driven puncher, mounted on a carriage over a conveyor. A lever is used to elevate or depress the puncher pick, a second lever moves it from rib to rib through the agency of a small air engine, while a third lever moves the whole puncher together with its truck bodily toward or away from the face, the conveyor remaining stationary within certain limits, or accompanying the forward or backward motion of the puncher as desired.

The puncher itself makes about 160 strokes per minute, and is controlled in the same manner as the ordinary hand-operated machine, going from rib to rib and making a cut extending the entire width of the entry. The conveyor is driven by a separate engine suitably controlled by a stop valve. The mine car is filled evenly by moving it about three times during the loading process.

Making the undercut is the longest part of the operation, requiring from 25 to 40 min., according to the hardness of the coal. The knocking-down process ordinarily takes place faster than cars can be supplied. When it is completed the conveyor is pulled back about 7 ft., a section of track is laid and the conveyor again moved forward into position.

Two set-ups during the shift constitute a 10-ft. advance of the entry. It is estimated that under suitable operating conditions, such as high coal of the quality found in the Pittsburgh seam, that the machine should average 250 ft. of advance per month of 25 days, working day shift only, the night shift being reserved for the laying of tracks, piping, etc. Where conditions have been suitable and the machine has had an adequate supply of cars, as much as 20 ft. of advance has been made in a single shift. The average advance in the entries of the Annabelle mine in West Virginia where several of these machines have been in operation since 1913, is somewhat over 10 ft. per shift. The cuts at this mine average 10½ to 11 ft. wide and 6½ to 7 ft. high.

The Jeffrey cutting and loading machine undercuts, shears, breaks down, and loads the coal into the mine cars. No explosives are used.

The machine stays in the entry until driven as far as desired. It is fed forward in a pan similar to a breast machine.

The feed forward is 7 ft. Average depth of cut 6 ft. 6 in., width 5 ft.

The time required to make the cut depends upon the conditions and nature of the coal. Ordinarily the sumping cut takes less than 30 min.; the open cut less than 20 min. Moving sideways to next cut, 3 to 4 min.

The Jeffrey cutting and loading machine has two vertical shearing chains between which is mounted a frame carrying a number of heavy punching picks. This frame is readily raised and lowered by the operator, who can cause the picks to strike at any height he desires. The coal falls onto a conveyor which is made thin enough to go into the kerf cut by the undercutting chain. This conveyor carries the coal to the rear end of the machine and dumps it into a second conveyor, which is mounted so that it can swing at any desired angle to the machine.

After a cut is made, the machine is moved sideways by means of a rope hitched to a jack at the opposite rib and then another cut is made. When slate is to be piled up at the side of the room, the rear conveyor is swung sideways, the slate rolled onto the front of the machine and gobbed by the two conveyors.

In a test run the machine required an average of 13½ min. to make a cut and about 3 min. to move the machine to the next cut, or about 16 or 17 min. time for each cut. The coal loaded averaged 21 tons per hour, although where roof conditions were good the machine had loaded 30 tons an hour in the same time. The height of the coal is 5 ft. 8 in. In another district an average of 60 tons per hour was made for the three months that the machine was in operation. In the driving of an 11-ft. entry it has averaged 20 ft. advance per shift of 8 hr. under rather unfavorable circumstances.

The crew for each machine consists of a machine runner, a helper and a driver. If the slate is heavy it may require an extra man for this handling.

At the Valier mine in Illinois these machines have been introduced to accomplish rapid development work, promote safety through elimination of explosives and prevent the shattering action of explosives on the ribs and roof of the entry. The cutting in this mine is unusually hard, but under ordinary conditions an advance of as much as 150 ft. per week was made with each machine, by working three shifts. On single shifts the machines make a general average of 300 ft. per month. With regard to the economy of the use of these machines, it is the opinion of Mr. Carl Scholz, general manager of the company, that the cost of installation and operation will be repaid many times by the saving in timbering because the coal is not affected by the use of explosives when these machines are used. The roof will stand much better than it does when so shattered and timbering will be unnecessary in most parts of the entries. These machines are being used on the main west entries where a possible distance of 3½ mi. can be driven. After an experience of about 1 yr. in entry driving in this mine, a difference between the standing qualities of these entries and those driven with explosives can be observed.

The O'Toole machine undercuts, breaks down the coal and loads it. It breaks up the coal completely and is thus designed specially for use in mines where the production of lump coal is of no advantage as for instance where the mine is producing solely for coke making purposes. The machine consists of a

OPERATION OF O'TOOLE MINING MACHINE, SHOWING WORK DONE AND COST

Month	Number of Days Worked	Feet Cut	Tons Mined	Labor Expense: Operation and Repairs	Bits	Pipe	Rash	Miscel-laneous	Total
June	15	384.77	872.8	$218.10	$16.10	$174.42	$60.90	$23.20	$492.72
July 1–15	7	264.23	613.2	80.58	4.16	53.84	106.00	10.55	255.13
July 16–31	11	402.07	924.8	102.00	9.59	91.35	120.55	38.53	362.02
Aug. 1–15	8	283.91	845.0	90.93	5.64	100.01	36.04	41.01	273.63
Aug. 16–31	11	518.03	1301.0	85.44	22.20	113.47	37.94	27.94	286.99
Sept. 1–15	13	394.67	1216.2	139.20	18.56	123.55	51.72	14.90	347.93
Sept. 16–30	13	216.54		97.51	8.99	88.89	48.56	21.47	265.42
Oct. 1–15	4	67.02	221.7	67.83	2.83	27.03	6.58	121.08	225.35
Oct. 16–31	7	135.13	217.9	45.17	7.90	12.93	9.56	9.05	84.61
Nov. 1–17	5	191.67	601.0	55.65	9.04	24.45	10.60	4.65	104.39
Totals	94	2858.04	6813.6	$982.41	$105.01	$809.94	$488.45	$312.38	$2698.19
Per ton mined	....	....	....	$0.1441	$0.0154	$0.1189	$0.0717	$0.0459	$0.3960
Per day worked	....	30.40	72.5						

motor-propelled truck, an oscillating chain-driven revolving head carrying the cutter bits and a scraper conveyor for loading the coal into mine cars at the rear of the machine.

A daily record of the machine, being the average of several days is as follows: Distance advanced 38 ft.; power consumption, 164 kw.; tons mined, 89· average time of operation, 6 hr. 40 min. These tests were conducted simultaneously with experiments on exhausting coal from the mine by means of an air blast, for which purpose a Root blower was connected to a large spiral-riveted pipe and operated exhausting. The pipe extended for several hundred feet into the mine and was connected with the mining machine. An attempt was thus made to draw the coal out of the mine and into a tank over the railroad track. This attempt was highly successful in so far as coal removal was concerned, but in its passage through the pipe the material was reduced to dust. Although this condition was not disadvantageous, as far as immediate coking was concerned, the coal obtained could not be shipped long distances in open-top cars.

Because readings were being taken at the time that the experiment was being conducted, the results obtained are not a true indication of what the machine can do. One instance of this kind is the record for one day in which 154 tons were mined in 9 hr. and 47 min., and the advance was 66 ft., the average consumption being 120 kw. The passage driven was in all cases approximately 10 ft. wide and 7 ft. high.

A table showing some of the work performed by this machine, the time consumed and the cost is presented herewith. These figures are as of 1915 and would have to be recalculated, using a new basic rate, in order to bring them up to date and render them comparable with present conditions and prices.

Blasting.—It has been found by experiment that when a hole is drilled in rock, loaded with powder, tamped, and discharged, the space in the rock after explosion has the shape of a cone. If, therefore, a hole is put in the face of an excavation and given no inclination the explosive will blow out the tamping or at best will break out a small cone near the mouth of the hole. The reason for this is that the gases in trying to escape follow the line of least resistance, which, in this case, is the drill hole, and the result is a blown-out shot. If a hole *a b* is put in at an angle of 60 deg. to the face *x y* in the plan, Fig. 50, the cone *a b c* would be broken out were there no reaction. The line of least resistance in this case is *b d*. Gas expands equally in all directions, and since it cannot escape except toward the free face, two forces come into action, one direct along the line *b d* and the reactive force along the line *b c*. The resultant of these two forces is represented by the lines *b e* and *b f*, consequently the cone is narrowed to a base *e f* instead of *a c*. The

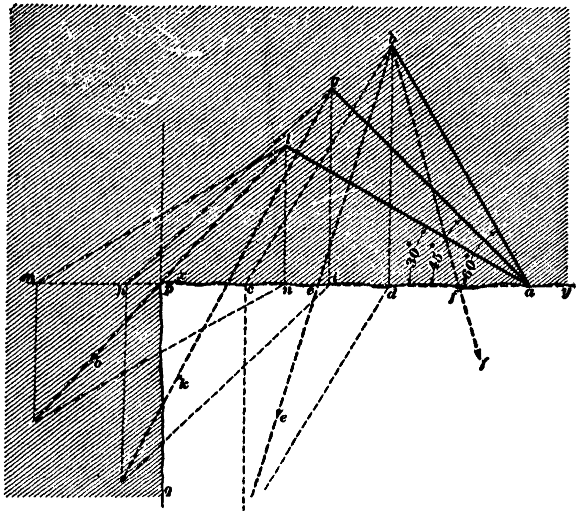


FIG. 50.—Sketch demonstrating theory of blasting.

greater the angle the narrower will be the cone base, and the more chances there will be for a blown-out shot.

If the hole is 6 ft. deep the length of the line of least resistance will be sin $60° \times 6$, or $0.86 \times 6 = 5.16$ ft. This is $6 - 5.16 = 0.84$ less in depth than when the holes is put in at right angles to the face and will in all probability be a blown-out shot where a safe charge of powder is used.

If the hole *a g* is 6 ft. long and put in at an inclination of 45 deg. to the face, the cone would have the dimensions *a g h* were there no resistance. In this case there is not so much resistance to the breaking of rock owing to the line of least resistance *g i* being less, which allows the reactive forces greater play. The resultant of the force to the left of the line *g i* is *g k*, and as this comes well inside the rib *q x* and the length of the line *g i* is sin 45° × 6 or 0.7 × 6 = 4.2 ft., the chances are that unless the explosive is properly proportioned and the charge is carefully tamped there will be a blown-out shot.

If a hole *a l* 6 ft. long is drilled at an angle of 30 deg. to the face, the cone in plan would have the shape *a l m*. In this case the line *l m* extends into solid rock beyond the free face *x y*, and therefore no rock would be broken past *p*. In this case the resultant *l o* comes to the corner of the rib at *p* and the rock broken would have the shape *a l p*. The length of the line of least resistance *l n* is sin 30° × 6 or 0.5 × 6 = 3 ft., or just half the length of the hole. Ordinarily with care a compromise can be effected between 30 deg. and 45 deg. where more work can be accomplished with the same quantity of powder, and this may be at 35 deg.

The best method of breaking down coal at any mine by the use of explosives is determined by experiments, which are to be carried on with intelligent observations.

The observations will include the thickness of the coal bed; the area of the face to be excavated; whether it is advisable to remove the whole bed or let some of it remain in place; whether advantage shall be taken of partings if any exist; the direction of the excavation with reference to the cleatage, and the tightness of the coal. When these matters have been definitely determined, then a few experimental shots will show the best positions for pointing the holes, their probable depth and the quantity of explosive needed for economically breaking down the coal.

The system of mining followed, whether it is shooting off the solid, shearing a loose end, or in the middle, or undercutting, will require the same careful observations, although the pointing of the drill holes will vary somewhat with each system.

Dynamite.—The meaning of the grade distinctions or "per cent strength" mark on dynamite is somewhat of a puzzle to many consumers, and often a source of misunderstanding between manufacturer and customer.

Originally a 40 per cent dynamite meant that the dynamite contained 40 per cent of actual nitroglycerin by weight, but as modern dynamites do not always contain this proportion as marked, a short description of the modern practice in grading is necessary. A slight knowledge of the history of the manufacture of high explosives may also help to explain the situation.

The dynamites in use at present were originally known as "active-base" dynamites in contra-distinction to the kieselguhr dynamites which had an inert base. Low grade dynamites were made a good many years ago, and are still a standard product, the composition of which is from 5 to 20 per cent of nitroglycerin absorbed in a combination of sodium nitrate, sulphur and coal dust, but as these are not a good absorbent, a mixture of wood meal and nitrate of soda is used. With these two ingredients, dynamites can be made with different proportions of absorbent to nitroglycerin, so that explosives containing as much as 75 per cent or as little as 15 per cent of nitroglycerin could be made, worked, packed and exploded.

It was also found that with an active base like wood meal and nitrate of soda, a dynamite having only 40 per cent nitroglycerin would develop as much power or more than a 75 per cent kieselguhr dynamite. A reasonably definite proportion of wood meal to nitrate of soda existed at which an explosive was not so wet that it would leak nor yet so dry that it could not be "punched" into the paper shells.

The proportions of wood meal and nitrate were changed to accord with any change in percentage of nitroglycerin. More wood meal and less nitrate were used when the absorbent was to retain a large percentage of nitroglycerin. More nitrate and less wood meal or wood meal of less capacity for absorption, like fine-grained sawdust, were used when making a dynamite with a lower percentage of nitroglycerin.

Using these three ingredients with minute proportions of other nonexplosive substances required to stabilize the dynamite, a type of high explosive known as "straight dynamite" is made which when thoroughly incorporated out of well dried and pulverized ingredients, constitutes the standard of strength against which all other dynamites are graded.

When other explosive substances are incorporated into dynamites they increase the power over the straight dynamite and it is then necessary to reduce the amount of nitroglycerin and otherwise modify the formula so that the new compound will develop the same power in actual work as the standard dynamite.

For instance, when guncotton is dissolved in nitroglycerin it makes a sticky jelly-like substance which when added to the wood meal and nitrate of soda makes an explosive, the cartridges of which are much more powerful than those of the same size in which nitroglycerin alone is used.

If such an explosive were graded according to its actual content of nitroglycerin, the cartridges would be so much more powerful than those of the standard grade of dynamite that it might not be safe to use in work where the blasters were accustomed to using that standard grade, as it would break the material too fine and throw it too far and perhaps do much damage.

When other active ingredients in the absorbent were employed, it was found necessary to reduce the amount of nitroglycerin until the mixture developed the same strength as the straight dynamite nitroglycerin by which it was graded.

There are now many explosives in the market which contain no nitroglycerin at all, some of them being equal to a 40 per cent straight nitroglycerin dynamite, and these are graded against the straight nitroglycerin dynamite.

The following table shows the total production of explosives in pounds in the United States, according to the United States Bureau of Mines, and the amount of the various kinds of explosives used in Pennsylvania to produce 91,626,964 tons of anthracite coal and 172,965,652 tons of bituminous coal in 1913:

	Black Powder	High Explosives	Permissibles
All mines in the United States	194,146,747	241,682,364	27,685,770
Pennsylvania anthracite region	44,001,660	16,093,035	3,323,645
Pennsylvania bituminous region	14,652,931	696,162	6,715,028

Thus it can be seen that the anthracite region used almost as much "permissible" explosive in proportion to coal produced as the bituminous region. The presence of gas in sufficient quantity to be detected on an ordinary safety lamp makes the use of permissible explosives advisable, if not absolutely necessary, to prevent explosions being caused by the flame of the black powder so commonly used for blasting coal. The mining of anthracite consumed over three times as much black powder and 23 times as much "high" explosive as the mining of bituminous coal, despite the fact that only half as much coal was produced.

The figures given are probably reasonably correct, though the operators only know what powder they sell to their employees and keep no track of what is purchased from other sources. However, the amount so purchased is probably not large. Of course the larger consumption of powder partly arises from the fact that the anthracite miner "lets powder do the work." If the bituminous operator invariably shot his coal off the solid, he could pay his pick miners a lower rate per ton and would nevertheless pay them more per day if he could only sell the product. The greater consumption of powder in the anthracite region is therefore not without its advantages in the production of cheap, though less marketable, coal.

Hydraulic cartridges.—A hydraulic cartridge was introduced about 1909 for breaking down the coal. The cartridge was operated by an improved form of pump. Though small it is extremely powerful, being capable of exerting a pressure of 7 or 8 tons to the square inch. It is of special design and is attached directly to the rigid pipe with which it is connected to the cartridge. No stand is required so that the appliance can be used either for breaking down or lifting up the coal without special connections and can be operated in holes drilled at any angle. The pump is fitted with a water tank, about 1½ pints being required for the operation, but most of this returns to the tank and can be used again. A pressure of 3 tons per square inch is usually required in seams up to 4 ft. thick, and this gives a total pressure of 60, 90 and 150 tons, respectively, on the three sizes of cartridges made.

After the coal has been undercut, a hole 3¼ in. in diameter is drilled into the coal, slightly less deep than depth of holing. This is done by means of ordinary machine and a special drill. The hole is put in parallel with the roof, and as nearly as possible along the parting to which the coal ordinarily comes off, or just below it.

The effect of the use of this machine upon the working cost is slight, while its general advantageous effect upon the selling price of the coal is quite striking. In a seam using five hydraulic cartridges, 450 tons of coal are produced per day, of which 75 per cent is large coal, and 25 per cent small. If in the same seam the coal is brought down by explosives, the percentage of large coal decreases to about 65 per cent, and that of small increases to about 35 per cent. The profit obtained by use of cartridges in the above seam on 450 tons is about $71 per day.

The advantages claimed for hydraulic cartridges are greater percentage of lump coal; better quality of coal because not shattered; better quality of slack; no waste coal; no dust; no flame; no smoke; no danger; the work is done in the daytime; hence, better supervision and no loss of work in waiting for night; the roof is not broken into or shaken; the same apparatus is used over and over again.

As many as 50,000 insertions of the mining cartridge per annum are made at the mines of the Hulton Colliery Co., in England, and the product is in better condition as it leaves the mine. In one seam alone, the Arley, 28,500 hydraulic "thrusts" are made per annum, by which it is estimated that 92,600 tons of coal are produced.

The coal face is well supported to begin with, by means of sprags, the holing being made to a depth of 3 ft. 6 in. or 4 ft., the drill holes being bored a few inches less deep, 3¼ in. in diameter, and placed at intervals of 6 ft. along the entire face. The boring of each of these holes occupies generally about 15 min. After boring the first hole, or whenever the collier is ready, the apparatus is inserted into the hole and the pump is fixed upon a movable and adjustable support. A small hand lever is first actuated until pressure is reached when an extension handle is attached. The pressure being fully on, the enormous power of the apparatus is soon apparent, for the coal is heard to be rumbling and cracking. This is allowed to continue until the back portion of the coal is broken off, after which the sprags are slightly slackened. By a continuance of the pumping, the pressure is brought to bear at the front of the face and continues to spread until the operation is completed. The sprags are then knocked out, whereupon the whole bank of coal falls down in large pieces.

Careful tests made, both with explosives and hydraulic cartridges, show that a great gain in lump coal is obtained when hydraulic cartridges are used. For two years the exact slack and round percentages were taken while explosives were used and the result showed:

(*a*) 51 per cent large lump; 17.3 per cent small lump; 31.7 per cent slack.

Hydraulic cartridges were then introduced, operating over an area covering one-half of the same mine, explosives being left in the other half For the first 12 months after using the cartridges the percentages were:

(*b*) 55 per cent lump coal; 18.5 per cent small lump; 26.5 per cent slack.

Slightly over 26 per cent slack was produced therefore, for the whole of the mine.

Three separate tests were then taken of cartridge coal, only, giving an average of:

(*c*) 64.37 per cent lump coal; 13.87 per cent small lump; 21.76 per cent slack.

The commercial value of the hydraulic cartridge as against explosives on the basis of the above percentages (*a*) and (*c*), and taking into account the cost of using the apparatus in England, 1907, was as follows:

HYDRAULIC CARTRIDGE

	£	s.	d.
Working Cost:			
Operator, 6 days at 6s. per day	1	16	0
Allow for charge on first cost (£30) and depreciation of cartridges, say		6	0
	2	2	0
Value of coal produced in a seam using 5 cartridges:			
30 "thrusts" each per day produce 450 tons of coal.			
450 tons at 78.24 per cent lump and small lumps = 352 tons at 10s.	176	0	0
450 tons at 21.76 per cent slack = 97.92 tons at 5s.	24	9	6
	200	9	6

EXPLOSIVES

	£	s.	d.
Working cost:			
On the same face a shot lighter would be required, but as he could probably fire the shots in less time this amount is deducted.			
Operator, 3 days at 6s. per day		18	0
Cost of explosives, 150 shots at 4d	2	10	0
	3	8	0
Value of coal produced:			
450 tons at 68.3 per cent lump and small lump = 307.35 tons at 10s	163	13	6
450 tons at 31.7 per cent slack = 142.65 tons at 5s	35	13	3
Hydraulic cartridge gain per week:	199	6	9
Less cost of working; £3 8s. 0d. less £2 2s. 0d	1	6	0
Increased value of coal £200 9s. 6d.			
189 6 9			
£ 11 2s. 9d. per day			
Per week of 5 days	55	13	9
Extra value of cartridges per week	56	19	9
Extra value of cartridges per year	2963	7	0

At this colliery 15 cartridges are in daily use with equal success, thus trebling the advantage.

The price of complete cartridge was £30 (f.o.b. Liverpool).

A new machine of this kind was introduced about 1920. Briefly stated, the principle employed in this machine is a duplication of a natural process found in every mine, in that the coal is broken down by developing and applying what might be termed an "artificial squeeze."

Rectangular incisions are cut in the body of the coal, one near each rib, and sometimes one in the center of the room. These are made parallel with and as near the roof as possible, those near the rib being cut parallel to it and close to the corner of the room. Where the coal has the usual cleavage planes and slips, the center incision is not necessary, as the coal in that case will break down readily from rib to rib. The machine used for cutting these incisions, or slots, is self-contained and self-propelled.

Sumping is the only work required of this bar—that is, it is simply inserted and withdrawn, cutting the incision in less than 3 min. after the machine is locked in position. The slots are cut to approximately the same depth as the undercut. The standard height of kerf is 4¾ in., and by simply changing the width of the cutter bar its width is varied to suit conditions. On the standard machine, cutter bars 18, 24 or 32 in. wide can be used.

This slotting machine is simply the combination of two old and highly perfected devices—the chain-track type of tractor, and the undercutting machine, simplified and modified, for cutting an incision near the roof instead of underneath the coal. Under normal conditions of operation in a 6-ft. bed three men will cut in eight hours on the average the required number of slots for breaking down 250 to 300 tons of coal.

This folding steel tubing is practically indestructible and solves one of the big problems in this process of mining, as it permits of water being conveyed at extreme pressure from the pump to the bar. A 24-ft. length of this tubing weighs approximately 42 lb. and will withstand a water pressure of 30,000 lb. per square inch. The normal pressure employed is 10,000 lb. per square inch.

The entire equipment, however, is designed for using water at 15,000 lb. per square inch, if such a pressure is required. All parts have an ample factor of safety. The pump is of standard design, suitable for delivering a constant volume of water, and the bars are of proper size for developing the necessary expansive forces.

The expanding bars are furnished in four sizes. The smallest develops a total expansive force of 1,000,000 lb. and the largest a force of 2,500,000. The pistons have large flat bearing surface to prevent indentation. The thickness of bed, character of coal and other local conditions determine the size of bar to be used. Under normal conditions the bar developing 1,500,000 lb. of expansive force unfailingly will break down coal 9 to 10 ft. in thickness.

The advantages resulting from the elimination of the use of explosives in coal mining is a subject on which volumes could be written. It effects economies in production and operation that are not anticipated.

The saving in life and property, the prevention of accidents in general and other humane and altruistic features are apparent. The roof fall is the greatest danger encountered in coal mining. Eliminate the shattering effect of explosives upon the roof strata and the accidents will be greatly reduced. Also remove the powder smoke and fumes and the working conditions become more healthful and pleasant.

Economically, the use of powder affects every item of production cost. Less timbering is required. What timber is placed is never blown out. Thus losses in output are avoided and the cost of cleaning up the resulting roof fall is eliminated. A saving in costs is effected through the elimination of shotfirers and other highly paid labor. No expense is incurred for installing and maintaining shotfiring apparatus and equipment.

Shot firing.—A system of shot firing by which all the shots can be exploded when every man is out of the mine, which is not expensive in installation and operation, and which will not heavily restrict the output of the mine was employed by the Utah Fuel Co. at its mines at Sunnyside and Castle Gate, Carbon County, Utah, about 1908. Here the coal veins vary from 5 ft. to 10 ft. in thickness. These mines are operated on room-and-pillar system and have a dip averaging 10 per cent (or 5 deg. and 45 min.). At Castle Gate the main haulage is in favor of the load, at Sunnyside No. 1, against the load, and at Sunnyside No. 2, the haulage is nearly level but slightly in favor of the load. All these mines are equipped with exhaust fans. At these mines all shots are fired from the surface and not then until it is known positively that every man is out of the mine. This method was a practical working success and has been for several years in operation in mines with a daily output of 900 to 1400 tons in an 8-hr. shift and can just as readily be adapted to a mine with an output of 3000 tons in an 8-hr. shift.

Two rubber-covered, or weatherproof (preferably the former) wires, size No. 6, are strung from the power plant to the mouth of the mine manway, or some opening through which they can be run without danger of disturbance from wrecking by haulage devices, etc. These wires act as feeders and this size should be carried into the mine through some opening which is well kept up and at the same time approximately divides the working places into halves in order to reach all these working places with the minimum expenditure of wire. From this feed wire, No. 12 rubber-covered wire is run, preferably along the haulage road because these passageways are kept in best condition. If some parallel gallery, however, is kept up as a manway, it should be used by all means. But if the haulageway is used the wire can be fastened as far as possible from actual haulage space. From this No. 12 wire, No. 14 rubber-covered wire is run into rooms, pillars, workings, etc., to the working faces, enough wire being left to comfortably reach every point in which a shot may be placed without being compelled to tighten the last 25 or 30 ft. of wire. These wires are fastened to plugs which have been driven into holes drilled into roof or rib, to props, to cap pieces, to legs or caps of timber sets, etc. They are attached to the above by either two-wire or three-wire porcelain cleats at intervals of about 25 ft. or sufficiently often to insure separation of the wires as the insulating covering decays and falls off in time.

Giant powder is used to bring down the coal, the per cent of nitroglycerin depending on the hardness and tenacity of the coal and on the use to which the coal is to be put. Sunnyside coal is almost wholly used for coking, and crushing being necessary, the more fines which can be produced in the mine the better after eliminating considerations of safety to roof, etc. As Castle Gate coal is sold commercially, only such strength of powder is used as will bring down the coal with a minimum of fines. "Reliable exploders" are used to discharge the shots, the exploder being merely laid upon or tied to the powder with a half hitch of the two wires, about 6 ft. of which are attached to each exploder. These wires are attached to the No. 14 rubber-covered wires, previously mentioned, in parallel and not in series. If attached in series, and several shots in the series, a defective exploder, or exploder wire, may cause several missed shots, but when placed parallel, a defective exploder will affect only the hole in which it is placed. In making wire connections all wires should be bared and twisting should be resorted to in order to insure good contact.

The power used at Sunnyside is derived from a continuous-current generator giving 500 volts, the cost of power for shot firing being practically nothing, as the current is used for but a few minutes daily, while for the remainder of the time it is used for underground hoists, electric locomotives, and lighting. If lack of electric power is the only consideration in the way of installing an electric-shooting system, a small plant can be installed and operated at a reasonable cost and the surplus power can be most profitably devoted to electric haulage, or to underground or surface-lighting systems.

The cost of installing the firing system with electric power already at hand is trivial. At Sunnyside Mine No. 1 the complete cost of installation from power house to the working faces, including new material for the entire line, was but $1250, of which $850 was for material. This material included 6000 ft. of No. 6 rubber-covered wire, 26,000 ft. of No. 12 rubber-covered wire, and 30,000 ft. of No. 14 rubber-covered wire; also 10 switches, about twenty 16-ft. poles with cross-arms, etc., for the surface line, insulators, porcelain cleats, etc. If a mine produces 1000 tons of coal per day for 250 days per year or a total of 250,000 tons per year, the total cost of the system if entirely paid for the first year would amount to but one-half cent per ton. In operation, a checkman at $75 per month, shot firer, or wireman, at $3.25 per day, and an inspector at $100 per month would amount to $2912.50 per year. On an output of 250,000 tons this would be but about 1c. per ton, leaving out material for repairs as of no consequence.

The system was installed at Sunnyside Mine No. 2 which is as extensive as No. 1 mine, for $640, the difference being due to the fact that the poles, etc., were utilized, which were already installed for the electric-haulage system.

Possible decrease of output due to missed shots, which prevent miners from working, may be advanced as a reason for condemning the system. This argument, however, is disposed of by the fact that out of an average of 300 shots fired daily, but 1 per cent missed except those occasionally due to the falling of roof on feed-wires. The shots of a whole level have been known to miss, but this occurrence has been very rare. At any rate no appreciable diminution of output was noticed when the change of shot-firing system was made at the Sunnyside mines.

To offset the cost of installation, operation, and of possible diminution of output, may be advanced the fact that in the

5 yr. the system has been in operation at Sunnyside where over 500 miners have been continually employed, not one accident, however trivial, has been caused by shot firing directly or indirectly. Under ordinary methods, it would be expected that several accidents due to shooting would have occurred in this length of time, any one of them resulting in damage suits involving $10,000 or more. The loss of one suit would amount to practically as much as the operating cost of the system for 4 yr.

Daymen.—The price paid the miners and loaders for placing the coal on the mine car at the working face is the largest single item in the costs of operating bituminous coal mines. The next largest is the day-labor expense. The former is practically fixed for each region and does not vary. Every ton of coal produced costs just so much, and regardless of the management it remains the same.

The superintendent's ability to handle and operate a mine successfully hinges principally on the one item—the day-labor cost of operation. This he can reduce or raise at will within certain limits. He can improve the underground conditions while better market prices rule or reduce the amount of labor done during dull times. The success or failure of the mine depends largely on this labor cost. It is subject to more variation than any other cost item on account of its being dependent almost entirely upon the individual ability of the management, and this quality has as many variations as there are individuals.

The question of what this cost should be is much disputed, and many estimates are made based on any one mine or group of mines. Each superintendent knows what he is doing himself, but there is more or less reticence in exchanging figures with neighboring mines.

An average value for this cost of labor is important, and the impracticability of obtaining average figures over large sections for periods of considerable time has led to much misunderstanding and criticism of the local management. The average cost is not what, in the opinion of a few men, the work should be done for, nor is it a few months' test run. It is a figure covering a long period of time, allowing for considerable mining difficulties and embodying some allowance for the labor necessary to the general maintenance and repairs of the mine and plant. It should include a wide variety of conditions, both favorable and adverse. Granting that this average can be established, the benefit to the operators is evident.

The accompanying diagram, Fig. 51, is compiled from published figures of 454 mines in Pennsylvania and West Virginia. The number of mines entering into the average is so great that any inaccuracy that may occur in the report from any one mine would be inappreciable in the result. Any temporary conditions such as an unusually high or low cost simply influence the result in making the figures represent average con-

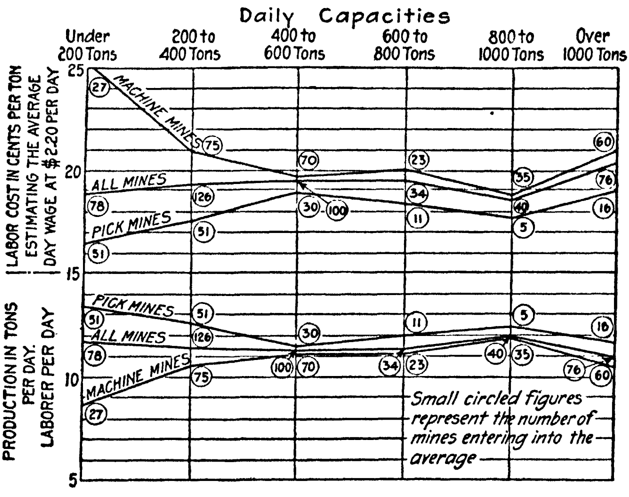


FIG. 51.—Diagram showing labor cost at 454 Pennsylvania and West Virginia coal mines.

ditions through long-time operation. Also mines were selected which work over 230 days in the year, so that the estimate may be taken as of mines producing their capacity and maintaining their equipment and mine conditions.

The foundation of the diagram is the mean force of day employees requisite at the mine for a period of one year; and the average daily shipments are based on the annual production. Some mines included in the average may be going through that period of reconstruction and betterment that is advisable periodically. Others may be using every means of economy. In fact, the figures may be taken as being very fair operating expenses covering a considerable period of time and variable conditions. They do not represent the cheapest mine nor the most expensive. Certain mines must necessarily call for higher labor costs than others, but these conditions are known, and any such allowance made to fit the individual case. Although the figures include an average amount of rock work they do not represent exceptions; and they represent the average haul over the large number of mines.

The basis of the diagram is the number of tons output per day-employee or day-laborer per working day. It does not take into consideration any extra-time track men, drivers and others who may be working while the mine is idle. It is customary for some day-men to work in idle time on something that may be then more easily done. Although this is a small percentage of the regular day-labor expenses it is something, and it would accordingly increase the cost given in the diagram. In explanation of the chart, it may be added that the force of miners or loaders necessary to produce the tonnage is not figured in the day-labor cost, the coal being loaded on the mine car at the working face by the requisite number of miners. The coal is hauled by drivers, track laid and repaired by trackmen, doors opened by trappers; these, with the motormen, brakemen, dumpers, trimmers, car shifters and all other day-labor necessary to move this tonnage from the working face to the loaded railroad car are represented on the diagram in the "Production in tons per day-laborer per day" and "Labor cost in cents per ton, estimating the average day wage at $2.20 per day."

Firebosses are included when employed, also shopmen and carpenters; but not the superintendent or mine foreman or the mine-office force. It is simply the bare day-labor cost. The cost per ton is derived from the tons per day-man per day. The greater the tonnage the less the cost. The averages given in the lower half of the diagram show that the production per day-man runs from 8.7 tons for the low-capacity machine mines up to 13.4 tons for the low-capacity pick mines. These are the extremes in the averages.

To reduce these figures to cents per ton, an average day wage throughout the industry must be assumed. This average varies in different regions, and the upper half of the diagram may have to be readjusted to a small extent to fit the prevailing wage scale. Curiously the average production per day-man per day remains practically unchanged throughout many thicknesses and characters of coal—for the central Pennsylvania thin steam coals, western Pennsylvania thick gas and coking coals, thick steam coals of southern West Virginia and the thick and thin gas coals of central and northern West Virginia.

If any variation from this is to be noted it is in a little greater efficiency in the regions where the higher wage rates are in effect. For this reason a moderate day rate was established; namely, \$2.20 per day—little higher than the existing average in the low-rate regions in 1915, and lower than those existing in western Pennsylvania. It is doubtful if the costs per ton shown on the diagram can be changed materially for a wide range of mining conditions and wage rates throughout the bituminous coal regions.

There is an almost inexplicable variation in the production per day-laborer per day when individual mines are considered, extreme cases running from 20 tons down to 5 tons. This indicates some confusion in this mine-labor item and the lack of any well-founded base for what this cost really should be.

It will be noticed that 51 pick mines with a capacity of less than 200 tons daily were operated with a labor cost of 16.6c. per ton—the cheapest group of mines. The interpretation is that these mines are so small and simple in their organization and so few men are employed that it is almost impossible to blunder in their operation. The only explanation advanced as to the rise in costs in pick mines up to 400 or 600 tons capacity is that these mines outgrow the crude management of the smaller mines up to that point; and from there up to the 1000-ton mark the more experienced management is in evidence.

Small capacity machine mines are a failure, as the cost of the power house and appliances are prohibitive for such small tonnages, and the operating costs of these mines decline generally up to 1000-ton capacity. There is little variation in this decline, as their equipment calls for experienced men in the start.

The reason for increase in cost for all mines over 1000 tons capacity can only be speculation, but the evidence points to their being too large for almost any human being to exercise the proper supervision. Large mines are almost always equipped with the most modern labor-saving appliances, and their labor costs should be correspondingly less.

A further study of these data gives the following results:

1. Averaged 42 tons per day for one year with two day employees, or 21 tons per day-man per day.
2. Averaged 365 tons per day for one year with 16 day employees, or 22.7 tons per day-man per day.
3. Averaged 429 tons per day for one year with 17 day employees, or 25 tons per day-man per day.
4. Averaged 218 tons per day for one year with eight day employees, or 27.2 tons per day man per day.
5. Averaged 131 tons per day for one year with five day employees, or 26.2 tons per day-man per day.

The average of these is 24.4 tons per day-man per day, and these mines were selected for their unusually efficient operation. There are very few of them in the list of 454 mines, and they are given to show that this grade of efficiency has been reached by the willingness of the employees. The aggregate production and the number of day-men employed at all of the 454 mines show their average efficiency to be 11.4 tons per day-man per day. These figures and the following tabulation include all the employees inside and outside the mine:

	Tons per Day-Man per Day	Cost per Ton (Labor Averaged at \$2.20 per Day)
Five mines	24.4	\$0.0902
454 mines	11.4	0.1930

In the 454 mines the labor work cost 10.28c. per ton more than it ought to have cost on a fair valuation.

The saving of 10.28c. per ton is a minimum, as many of the 454 mines are operated with a very efficient force of men, and they include the five mines.

Looking at the question from another point of view, we have:

	Tons per Day-Man per Day	Cost per Ton (Labor Averaged at $2.20 per Day)	Saving
64 efficient pick mines	16.76	$0.1313	
100 inefficient pick mines	9.66	0.2277	$0 0964
139 efficient machine mires	13.44	0.1637	
151 inefficient machine mines	8.29	0.2654	0.1017

One writer has suggested a "labor factor," to fix the ratio of the number of company men to the tonnage produced. This gives certain valuable comparisons for the immediate vicinity where the labor conditions may be all the same, but is subject to fluctuations in comparisons between all pick, puncher, short-wall and longwall mining, according to the division drawn between miners and others.

Efficiency of the daymen may be high, but the miners and loaders may be waiting for cars and not working to the best advantage. An extra employee here and there inside may not in himself be working to the best of his time, but he may be helping the miners to double their output. A better basis of comparison would appear to be one involving the output per man employed inside.

The accompanying table shows a comparison between the largest producers in the central Pennsylvania field operating on the thinner seams of coal, the Freeports and Kittannings, from data taken from the state mine inspector's report. These figures may be inaccurate in minor details, but are probably equally correct for all. The collieries are arranged according to the thickness of the seam, and the final tonnage is divided by the thickness of the seam in feet, so as to put all on an even basis.

It will be noticed that there is a wide variation in the tonnage handled by the men outside the mine. This is to be expected, as conditions vary more widely outside than inside, the matter of picking probably having more influence than any other. Some companies purchase power, but most of those on the list are so large they have found it economical to produce their own or are producing it because their plants were installed before it was convenient to purchase power. All employees engaged in the manufacture of coke have been eliminated. With a general knowledge of the conditions existing in each case, a fairly reliable comparison should be reached.

PRODUCTION EFFICIENCY OF VARIOUS COMPANIES IN CENTRAL PENNSYLVANIA

(Compiled from Mine Inspector's 1912 Report)

Thickness of Coal, Inches	Company	Tons per Man per Day			Foot-Tons	Standing
		Inside	Outside	Inside and Outside		
38	A	....	....	2.27	0.72	12
40	B	3.45	19.3	2.91	0.87	6
42	C	4.43	21.3	3.60	1.28	1
....	D	3.56	28.1	3.16	0.90	3
....	E	2.72	30.4	2.50	0.71	13
45	F	3.96	45.2	3.65	0.97	2
46	G	3.68	43.3	3.40	0.89	5
48	H	4.06	31.7	3.60	0.90	4
....	I	3.53	87.0	3.39	0.85	7
....	J	2.92	11.1	2.31	0.46	18
56	K	4.67	25.8	3.95	0.84	9
58	L	4.73	14.3	3.56	0.73	11
60	M	4.12	36.5	3.70	0.74	10
....	N	4.15	16.4	3.31	0.66	15
66	O	5.53	30.3	4.68	0.85	8
67	P	3.90	25.7	3.40	0.61	16
72	Q	4.77	34.4	4.12	0.68	14
....	R	4.27	20.7	3.52	0.58	17
Average....	....	....	30.1	....	0.79	

The United States Coal & Coke Co. in 1917 offered prizes to the men who would earn the most money in the loading of coal during the last half of June, the results of which are given in the accompanying table. These figures are of interest chiefly in showing the maximum labor effort that can be realized, some of the men having worked day and night and laying off for a full week after the test period was over.

Earnings of the Three Best Coal Loaders at Each Plant of the Gary Mines

Name	No. of Cars	Rate	Days	Rate	Slate	Rate	Total Earnings
No. 2 Works:							
K. Kolony	117	$1.02	3.0	$2.55	....	....	$126.99
G. Belone	103	1.11	....	....	13	$0.84	125.25
L. Sabo	95	1.04	2.0	2.55	10	.84	112.30
No. 3 Works:							
J. Heedo	114	.73	2.4	2.50	....	....	89.33
J. Seko	96	.80	18.0	....	18	.55	86.70
S. Rinko	47 27	.85 .73	8.0	2.50	3	.55	81.59
No. 4 Works:							
P Clinchook	152	1.11	1.0	3.00	3	.84	174.64
S. Miller	150	1.11	....	....	9	.84	174.06
J. Kromba	95	1.11	1.0	2.55	6	.84	113.04
No. 5 Works:							
F. Lasos	50	1.37	1.0	2.55	55	.84	117.25
P. Blink	64	1.37	1.0	2.55	29	.84	114.59
S. Borsus	51	1.37	1.0	2.55	50	.84	114.32
No. 6 Works:							
T. Hajeck	131	1.11	1.5	2.55	....	....	149.39
J. Sabinsky	102	1.11	1.0	2.55	....	....	115.77
A. Hajser	83	1.11	0.6	2.55	....	....	93.86
No. 7 Works:							
S. Kozlske	96	1.11	0.3	2.55	....	....	107.41
W. Karbovich	87	1.04	3.3	2.55	8	.36	101.01
F. Near	68 33	.97 1.04	....	....	....	....	100.28
No. 8 Works:							
W. Skarino	115	1.15	0.5	2.55	5	.84	137.87
B. Slovak	62	1.05	....	....	28	.84	91.17
Z. Rugan	64	1.11	4.0	2.55	6	.84	86.28
No. 9 Works:							
F. Markangelo	209	1.11	1.0	3.00	1	.55	235.54
D. Paron	194	1.11	1.6	2.55	....	....	219.59
S. Darboldi	205	1.11	2.4	2.55	....	....	233.78
No. 10 Works:							
N. Carper	168	1.11	2.5	2.55	1	.55	193.55
A. Dodaney	96	1.11	8.0	2.55	....	....	126.96
M. Simms	100	.85	1.0	2.55	30	.55	104.05
No. 11 Works:							
S. Coimoin	240	.95	13.0	2.50	1	.55	261.05
P. Miller	247	.95	2.0 1.0	4.10 2.50	1	.55	245.90
P. Pope	88 161	.90 .95	....	....	....	....	232.15
No. 12 Works:							
J. Drake	110	1.11	....	....	....	....	122.10
J. Scolgal	90	1.11	....	....	....	....	99.90
E. Lester	80	1.11	2.0	2.50	....	....	93.80

EARNINGS OF THE THREE BEST COKE PULLERS AT EACH PLANT OF THE GARY MINES

Name	Ovens	Rate	Ovens	Rate	Total Earnings
No. 2 Works:					
J. Hostin	41	$1.55	43	$1.25	$117.30
D. Hostin	13 98	1.55 .16½	7 17 Days at	1.25 2.75	91.82
J. Hostin	12 293	1.55 .16½	10 1.8 Days at	1.25 2.40	83.98
No. 3 Works:					
E. Young	50	1.25	17	1.55	88.85
W. Young	38	1.25	15	1.55	70.75
A. Jones	30	1.25	18	1.55	65.40
No. 4 Works:					
P. Kellam	20	1.55	56	1.25	101.00
N. Kellam	16	1.55	36	1.25	69.80
D. Kellam	13	1.55	32	1.25	60.15
No. 5 Works:					
S. Hodge	19	1.55	49	1.25	90.70
C. Wade	14	1.55	36	1.25	66.70
L. Nacne	13 ...	1.55	25 1.3 Days at	1.25 2.40	54.60
No. 6 Works:					
J. Mills	9 219	1.55 .16	20 3 Days at	1.25 2.40	81.19
T. Mills	13 ...	1.55	37 1.8 Days at	1.25 2.40	70.93
J. Moore	11 ...	1.55	19 1.4 Days at	1.25 2.40	44.27
No. 7 Works:					
W. Wilburn	17	1.55	37	1.25	77.60
J. Santos	11 ...	1.55	15 14.7 Days at	1.25 2.40	71.27
E. B. Rose	8 ...	1.55	21 0.6 Day at	1.25 2.40	40.25
No. 8 Works:					
W. Kellam	6 ...	1.00	51 28	1.25 1.55	113.15
G. Hall	34	1.25	12	1.55	61.10
J. Martin	25	1.25	14	1.55	53.60

The outside employees used in the anthracite regions would compare better with those required in the bituminous if we were to eliminate all the men engaged solely in the cleaning and sizing of the coal. Pennsylvania anthracite statistics supply information in which "slate pickers" are segregated and

TONNAGE PER EMPLOYEE IN PENNSYLVANIA, WEST VIRGINIA AND ILLINOIS

	Employees	Per Cent	Tons per Employee per Year	Tons per Employee per Working Day
Pennsylvania anthracite, 1913*:				
Miners	78,319	44.67	1136†	4 69
Other inside employees	50,348	28.72	1768†	7.31
Outside employees ‡	46,643	26.61	1964§	8.12
Total	175,310	100.00	523	2.16

* All mines reporting.

† Based on all tonnage except that from culm-banks. The whole tonnage was 91,626,964 in 1913, 2,619,785 being from mines and strippings, leaving 89,007,179 tons obtained from culm-banks.

‡ This includes 9121 slate pickers, men and boys, or 5.20 per cent, cleaning 10,046 tons per employee per year, or 41.51 tons per working day.

§ Based on the full tonnage; namely, 91,626,964 tons.

	Employees	Per Cent	Tons per Employee per Year	Tons per Employee per Working Day
Pennsylvania bituminous, 1913*:				
Miners	122,932	64.73	1407	5.61
Other inside employees	33,342	17.56	5188	20.67
Outside employees †	33,635	17.71	5142	20.49
Total	189,909	100.00	911‡	3.63

* All mines reporting.

† This includes 10,122 coke employees, or 5.33 per cent, coking 37,381,029 tons, or 3693 tons per employee. Without these men, who are really engaged in manufacturing, there would be only 23 513 outside employees, or 12.38 per cent, the production being 7356 tons for each such employee per year, or 29.31 tons per working day.

‡ Excluding coke-workers, this figure would be 962.

TONNAGE PER EMPLOYEE IN PENNSYLVANIA, WEST VIRGINIA AND ILLINOIS
Continued

	Employees	Per Cent	Tons per Employee per Year	Tons per Employee per Working Day
West Virginia report of 1912 *:				
Miners....................	43,581	62.60	1531	6.84
Other inside employees.......	13,277	19.07	5026	22.44
Outside employees †.........	12,758	18.33	5231	23.35
Total.....................	69,616	100.00	959	4.28

* All mines reporting.

† This includes 2297 coke employees, or 3.30 per cent, coking 3,310,250 tons, or 1441 tons per employee. Without these men there would be only 10,461 outside employees, or 15.03 per cent, the production being 6379 tons per year, or 28.48 tons per working day for each such employee.

	Employees	Per Cent	Tons per Employee per Year	Tons per Employee per Working Day
Illinois report of 1913 *:				
Miners.....................	53,588	69.73	1129	6.31
Other inside employees.......	16,718	21.75	3619	20.22
Outside employees...........	6,549	8.52	9240	51.62
Total......................	76,855	100.00	787	4.40

* Shipping mines only.

under this caption are included all employees inside the breaker, such as jigtenders, platemen, etc. The reports from the bituminous regions of Pennsylvania, West Virginia and Illinois do not give any enumeration of workmen engaged in cleaning and sizing, though there are many men so employed, some being actually on picking tables and others trimmers on cars.

Lest too much weight should be placed on the cost of breaker work, a quotient has been obtained by dividing the whole tonnage, 91,626,964 tons, by the number of men and boys employed on the outside, excluding those working within breakers and listed as slate pickers. The divisor is thus reduced from 46,643 employees to 37,522. The quotient to which we have referred is 2442 tons per year, or 10.09 tons per day, a figure so low that it seems hard to explain. And yet in comparing this figure with the larger figures of the bituminous region, it must be borne in mind that the latter have not been "doctored" by the omission of cleaners and sizers in making the estimation.

But there is another vitiating principle in making comparisons to which attention must be drawn. In the anthracite region there are many strippings, and the men engaged in this work would find their appropriate classification in our tables, not as outside men, but as miners. Unfortunately, the tonnage of strippings and the men employed at such open-cut work have not been segregated in the mine reports, except in Illinois, and there the stripping work is of almost negligible importance. In the year ending June 30, 1913, about 152 men at seven strippings in Illinois mined 137,448 tons.

Another difference between anthracite and bituminous conditions makes a comparison of the work of inside men somewhat unfair. In the anthracite region there is much labor expended in reworking old breasts, especially in the Lansford district. In some cases possibly there are advantages in this work, making it more profitable than first mining, but in nearly every case second mining of this description is far more expensive than work in undisturbed coal.

Perhaps there are no sections in the Union with more efficient outside handling systems than those found in western Pennsylvania and Illinois. In the former the economy largely arises from the largeness of the operations. In the latter it finds its source possibly in the fact that the mines are mostly shafts and that the preparation of the coal is limited to careful sizing. The fact that much tramming which is outside, or tipple, work in the case of a drift is inside, or caging, work in case of a shaft, also accounts for the lower productivity of Illinois inside hands. The gain secured on the surface is a loss below.

On the other hand in western and central Pennsylvania the coal instead of being tipped by self-dumping cages is dumped on a tipple, which takes more surface hands though less men are needed below. The long haulages common to drift mines in that section are also a cause of wastage of labor, though they probably more than pay for themselves by the saving in shaft sinking and other first cost. In many cases there are long gravity planes which need the services of many men. Moreover, picking tables are somewhat numerous in central Pennsylvania.

The necessity that shotfiring in Illinois should be performed by men specially employed for that purpose increases the number of men engaged in underground work. In many cases these men are supplemented by fire hunters whose business it is to see that the shots have not ignited the coal and to extinguish an incipient blaze if such be found.

It will be noted that, excluding coke workers, the production per outside employee in the bituminous regions of Pennsylvania is 29.31 tons per day; in Illinois, where the coke ovens are away from the mines and are therefore not considered in the state report of the mining industry, the production is 51.62 tons per day for each outside workman.

The showing for this latter state (Illinois) is truly remarkable and one of the largest operating concerns there states that it is not obtained by excluding men in official position from the enumeration, but represents a real economy in mine labor. Irregular operation may, and frequently does, help to increase the tonnage of coal mined per miner in any working day, but it does not give any aid to the management in increasing efficiency in other labor; for when places are vacated by the removal of workmen who become discontented at the frequent idlenesses, the transportation problems are made difficult and the outside men are often prevented from being supplied with the tonnage for which the operation was designed.

Some sections suffer from old-time regulations. In the Blossburg section of Pennsylvania, for instance, it was and perhaps still is customary to have boys at all switches, though in other regions the mule-drivers throw their own switch levers without any measurable waste of time. The American coal operator concentrates his attention on this question of tonnage per man employed as is indicated by a comparison with foreign practice. The following is a comparison of the tonnage per man employed in the various fields of the world:

Pennsylvania bituminous	1009	Pennsylvania anthracite	520
Virginia	964	Transvaal	504
West Virginia	954	Arkansas	480
Montana	893	Iowa	478
Wyoming	887	Texas	476
New Mexico	857	Oklahoma	461
Maryland	847	Missouri	414
U. S. bituminous	837	Michigan	373
Ohio	790	Orange Free State	327
Utah	783	German Empire	301
Illinois	775	Natal	293
North Dakota	773	Great Britain	273
Indiana	772	Austria	231
Colorado	770	France	224
United States	762	Sweden	185
Kentucky	745	Russia	173
Alabama	720	Belgium	173
Washington	669	Spain	162
Tennessee	613	Japan	133
Australia	607	India	124
New Zealand	563	Cape of Good Hope	81
Canada	529		

The figures for the tonnage per man in the United States are taken from the report of the U. S. Geological Survey for 1913; those for all other countries are for 1912, except in the case of Russia (1909), Spain and Japan (1911). These latter figures, of course, have been multiplied by 1.12 so as to obtain the equivalent in short tons.

Of course this order of precedence in economy of human labor varies from year to year according, largely, to the number of days worked. Thus, for instance, in 1912 West Virginia, being second, led the state from which it was originally dismembered, and Wyoming stood third in the list. In 1911 the output per man in Great Britain was 291 tons and that of the German Empire 278 tons, reversing the order of 1912. In many states and countries the tonnages are low because of the irregular call for coal where the main demand is for domestic use. In such communities, low output spells not inefficiency but diversity of occupation. The farmer has added mining to his ordinary vocation.

But the fact that every state in this Union leads every European country is significant. And the leadership is obtained despite the fact that the mines of the American continent do not work with, by any means, exemplary regularity. Still some allowance must be made for the deeper workings, more contorted measures, more perfect preparation of coal and more back-filling and flushing in the European mines. But these difficulties extenuate rather than explain. Smaller cars and haulage units, union restrictions, lower speeds and less use of mining machinery are the real reasons for the lowered human efficiency in the coal operations across the Atlantic.

In fine, it has been observed that the genius of the American machine is in capacity, that the merit of the English is in ruggedness and that the predominent quality of the German is ingenuity and delicacy. The American mechanism performs great service with little watching. The English machine frequently does less, but seems to live forever. The German machine is often complicated, but usually not needlessly so. The reason for the complexity is that the German has a delight in modeling those types of machinery which demand complication and refinement of detail.

The accompanying chart, Fig. 52, excerpted from the report of the Engineers Committee of the United States Fuel Administration gives a graphic comparison of the production per man in the United States and Great Britain from 1896 to 1913. The curves for both anthracite and bituminous are given for this country and the curves of the number of men employed both inside and outside are given for Great Britain.

Miners' wages.—It may be argued that the miners are making a day's pay where they are not lazy but this is not the case in many mines. The operator must bring pressure to bear upon the worker to get him to work regularly and do a full day's work. But this is not the principal difficulty, and the difference in earnings is small where opportunities to load coal are reasonably equal.

The rates paid for mining are high—almost too high. In the union districts in central Pennsylvania the price for pick coal (in 1915) is 72c. a ton. This is paid whether the mining is in rooms and the coal hard or whether it is in pillars which have been standing and the coal is crushed. In the latter case a man can dig and load a ton in 20 min. At that rate he ought to earn \$8.64 a day, allowing 50 per cent of the time for rest-

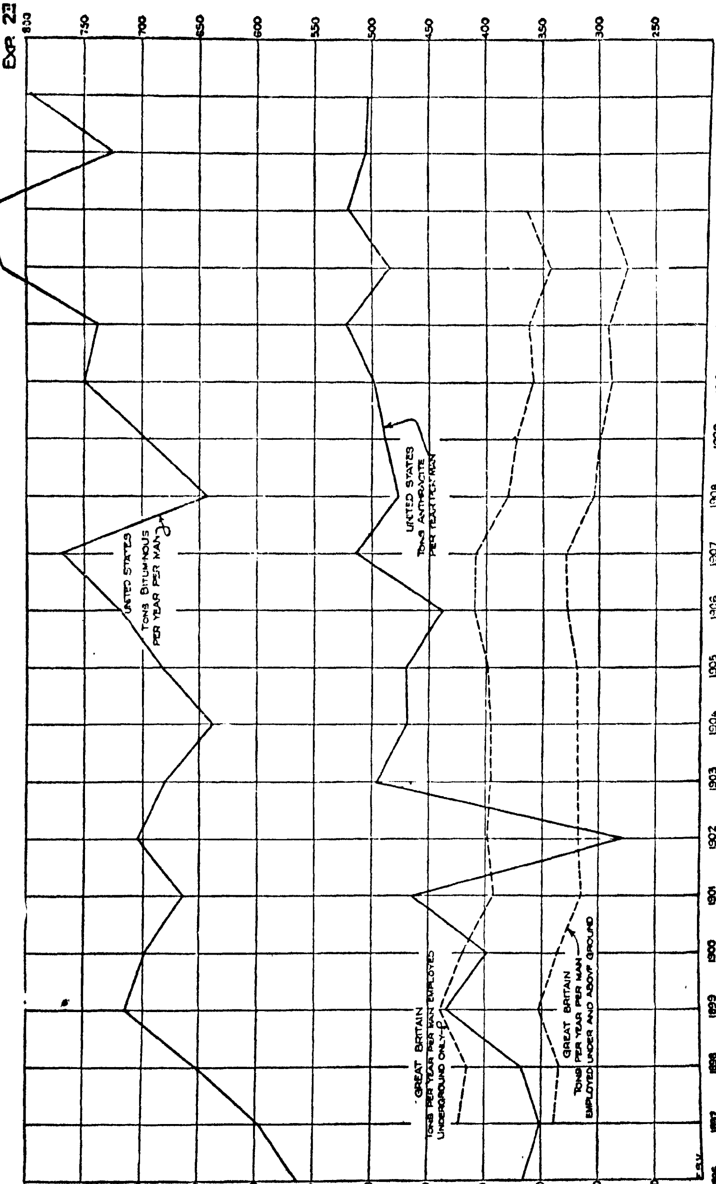


FIG. 52.—Comparison of production per employee at mines in the United States and Great Britain from 1896 to 1913.

ing. But seldom or never he gets enough cars to earn anywhere near that amount.

It has been stated that the men in central Pennsylvania earned more money when pick coal was 35c. a ton than they do now. But the past and present conditions have not been taken into account. The blame for the loss in earning power is often angrily put on the union, whereas it ought to be partly borne by operators for not maintaining correct conditions—men in proportion to the equipment and bosses in proportion to the mine.

There is an inflexibility in the rates paid for work which makes an inequality in the earning power of the men. Rates are generally the same for all parts of a mine or for all parts of a vein. In one anthracite mine $1.58 is paid for a 2-ton car of coal whether the vein is 8 ft., and free blowing or 4 ft. and hard coal. In the bituminous regions the same price is paid for pick coal whether it is in rooms or in crushed pillars. Business has been business in establishing the rates, and in any argument between employer and employee over what the rate should be, the attempt on both sides has been to charge all the traffic would bear. There has been no measurement of work by which attempts could be made scientifically to get an equitable basis for payment. Measurement of work is just as practical in mining as in manufacturing, where it is now being largely done.

Although the payment is made according to the ton or car loaded, responsibility for the amount of work done cannot be wholly cast aside by the management. Men are used in a mine entirely too freely and expected to be on their own resources. They cannot do it, for they do not work alone, but are dependent upon others for the opportunity to work.

The drivers, being paid by the day, are not anxious to deliver more cars than are necessary to keep their jobs and so neglect the miner. In consequence the miner does not earn as much as he might. One can hardly blame the driver so long as it is a fixed principle in life to get as much for as little as possible. Universally a mine is so undermanned by bosses that no work can be followed up, and the neglect of duty and failure to coöperate can become so common that the miners frequently suffer from needless delays.

It is hard to stir up trouble at a mine where the men are being properly treated and are earning a day's pay. It will be found that the strength of a union increases at a mine in proportion as there is more cause for discontent. But this is not to be regarded as an argument that the men should be paid whether they do any work or not.

Increased earnings of the individual can be reflected favorably on the cost sheet. A man who is working a piece of coal which has been formerly neglected and is so situated that he is given only a car or two a day and whose earnings in consequence are low is likely to be a source of trouble. When it is necessary to keep a man and a mule for only a few cars a day, the transportation charge will be high. The cost of gathering was taken in one mine and found to run from 2½c. to 25c. The total cost for haulage was high because on only a few roads were there enough men to give a driver a full day's work. When the work was apportioned so that there were enough men on each road, the cost fell.

It would be more advantageous to the workmen if the union leaders, instead of making demands for further increase in the rates of pay, asked for a better organization of the work so that the men were furnished an equal number of cars. Rates are high enough so that no man need overwork himself in earning good pay. In a boiler-room a fireman will shovel into the furnace from 18 to 20 tons a day, which is more than a miner will load into cars.

One well-known union leader has been quoted as saying that 15 tons was not too much for a man loading after machines, and the union representatives agreed to a task of 12 tons at one mine. When one gets into the complex changing conditions of an anthracite mine it might seem a little doubtful whether this could be done, but again it is a case of the arrangement of work rather than a crying need of an increase in rates.

Under the present general manner of payment the only cure is to promulgate the idea that each man should produce a large tonnage per day. An increase in their output will be of advantage to the miners, as they are paid by the ton. It will also be to the interest of the company, for an increased output per handling unit will result from the proper organization of work by which the larger product is secured. Though there is no one cure for all ills, the application of the principles of efficiency to mining will greatly help to alleviate the conditions which are now a frequent and natural source of much irritation.

The accompanying table gives the wage scale in effect in the Hocking District from 1898 to 1921, covering all classes of labor in and about the mines. This table will be found of use in computing the comparative cost of doing the various kinds of work cited throughout this book where the figures are not up to date.

The Hocking scale as it is generally known has been used for this purpose for the reason that it dates back the farthest of any of the important scales and also because it is used as a base for all wages in the Central competitive field and has influenced wage settlements in all union fields generally. The break in the table in 1914 was occasioned by the change from the screened coal basis of payment to the mine run.

Losses from idle time.—The yearly average of days of activity in the mines of the United States from 1908 to 1914 was 217. Thus the mining plants averaged only about 72 per cent of full time. In the year 1914 the number of working days, according to United States Geological Survey report, was only 207 for all coal mines throughout the country.

Unfortunately even this time is not by any means evenly distributed. Some mines are idle a large percentage of the year, while others work with regularity. A few mines in agricultural states work mostly in the winter, and in the summer the men must find farming jobs to keep them busy, though it is doubtful whether many of them do. A miner is not always willing to work under a hot sun. Arkansas and Oklahoma in 1913, for instance, only worked 174 days, or barely 55.5 per cent of the possible working time.

Perhaps under no circumstances will it be possible to keep coal mining at an even pace throughout the year, but it should be possible to do better than the 1914 season. Such depression as occurred in that year would be helped by a combination embracing all the mine operators, but the conditions of unemployment were nation-wide, and something broader than a combination of mine owners would be necessary as a stabilizer.

WAGE SCALES IN THE HOCKING DISTRICT, CENTRAL

	1892 to 1894	Feb. to April, 1894	1894 to 1895	June to Oct., 1895	1895 to 1896	March to Oct., 1896
Pick Mining						
Screened lump, per ton	$0.70	$0.50	$0.60	$0.51	$0.55	$0.61
Mine run, 5–7 lump price	.50	.35$\frac{5}{7}$	.42$\frac{6}{7}$	.36$\frac{3}{7}$	.39$\frac{3}{7}$	.43$\frac{4}{7}$
Entry, dry, per yard	1.75	1.25	1.50	1.27½	1.37½	1.52½
Entry—breakthroughs, yard	1.75	1.25	1 50	1.27½	1.37½	1.52½
Room—breakthroughs, yard	1.00	.75	1.00	.77½	.87½	1.02½
Room turning, per room	2.50	2.50	2.50	2.50	2 50	2.50
Track layers, per day	2.25	1.75	2.00	1.78½	1.87½	2.02½
Track layers' helpers, per day	....	......	......	......	......	......
Trappers, per day	.75	.75	.75	.75	.75	.75
Cagers, drivers, per day	2.00	1.50	1.75	1.52½	1.62½	1.77½
Trip (rope) riders, per day	......	......	......	......	......	......
Water haulers, per day	1.75	1.50	1.75	1.52½	1.62½	1.75
Machine haulers, x per day	*	*	.*	*	*	*
Timber men, per day	......	......	......	......	......	......
Pipe men, per day	......	......	......	......	......	......
Wire men, per day	......	......	......	......	......	......
Motor men, per day	......	......	......	......	......	......
Pumpers, per month	40.00	30.00	35.00	30.00	32.50	35.00
Other inside day labor	......	......	......	......	......	......
Outside						
First blacksmiths, per day	x	x	x	x	x	x
Second blacksmiths, per day	x	x	x	x	x	x
Blacksmiths' helpers, per day	x	x	x	x	x	x
Carpenters, per day	x	x	x	x	x	x
Dumpers-trimmers, per day	2.00	1.50	1.75	1.52½	1.62½	1.77½
Slack haulers, per day	x	x	x	x	x	x
Greasers, couplers, per day	x	x	x	x	x	x
Engineers, firemen, per day	x	x	x	x	x	x
Other outside day labor	1.75	1.25	1.50	1.25	1.37½	1.50
Machine						
Cutting, per ton—by						
Jeffrey styles—room	.08	.07	.08	.07	.07½	.08
Jeffrey styles—entry	.11	.10	.11	.10	.10	.11
Punch machines—room	.12½	.11½	.12½	.11½	.12	.12½
Punch machines—entry	.13½	.12½	.13½	.12½	.13	.13½
Loading, per ton—						
In rooms	.35	.25	.30	.25½	.27½	.30½
In rooms with hand drilling	.38	.28	.33	.28½	.30½	.33½
In entry	.43½	.31$\frac{1}{14}$	.36	.31½	.33½	.36½
In entry with hand drilling	.46½	.34$\frac{1}{44}$	.39	.34½	.36½	.39½
In breakthroughs in entry....Entry	Price	E. P.	E. P.	E. P.	E. P.	E. P.
In breakthroughs in room	.41	.29$\frac{2}{7}$	.35	.29½	.32$\frac{3}{14}$	.35$\frac{5}{7}$
In breakthroughs in rooms with hand drilling	.41	.32$\frac{2}{7}$	.38	.32½	.35$\frac{3}{14}$	.38$\frac{5}{7}$
Drilling, by hand, per ton	.03	.03	.03	.03	.03	.03
by machine, per ton	.02	.01½	.02	.01½	.01¾	.02
Machine by the day......Runner and helper jointly	......	....	......	......	4.50	4.50
Room turning to cutter and loader. Entry price	......	......	E. P.	E. P.	E. P.	E. P.

* Formerly rated with couplers, greasers, etc.

COMPETITIVE YIELD, FOR THE YEARS 1892 TO 1921

1896 to 1897	Jan. to July, 1898	1897 to 1898	1898 to 1900	1900 to 1903	1903 to 1904	1904 to 1906	1906 to 1908	1910 to 1912	1912 to 1914
$0.45	$0.51	$0 56	$0.66	$0.80	$0 90	$0.85	$0.90	$0.95	$1.00
.32 1/7	.36 3/7	.40	.47 1/7	.57 1/7	.64 2/7	.60 5/7	.64 2/7	.0785	0.71 3/7
1.12½	1.27½	1.40	1.65	2 00	2.25	2.12½	2.25	2.3748	2.4996
1.12½	1.27½	1.40	1.65	2.00	2.25	2.12½	2.25	2.3748	2.4996
.62½	.77½	.90	1.15	1.39	1.56	1.47½	1.56	1.6465	1.7330
2.50	2.50	2.50	2.50	3.03	3.41	3.22	3.41	3.5992	3.7884
1.65	1.78½	1.90	1.90	2.28	2.56	2.42	2.56	2.70	2.84
......	......	......	1.75	2 10	2.36	2.23	2.36	2.49	2.62
.75	.75	.75	.75	1.00	1.13	1.06½	1.13	1.25	1.32
1.40	1.52½	1.65	1.75	2.10	2 56	2.42	2.56	2.70	2.84
......	......	......	......	......	2.56	2.42	2.56	2.70	2.84
1.40	1.52½	1.65	1.75	2.10	2.56	2.42	2.56	2.70	2.84
*	*	*	*	2.10	2.56	2.42	2.56	2.70	2.84
......	......	......	1.90	2.28	2.56	2.42	2.56	2.70	2.84
......	......	......	1.85	2.22	2.50	2.36	2.50	2.63	2.78
......	......	......	......	......	2.56	2.42	2.56	2.70	2.84
......	......	......	......	......	2.56	2.42	2.56	2.70	2.84
30.00	30.00	33.00							
......	......	......	......	2.10	2.36	2.23	2.36	2.49	2.62
x	x	x	x	x	2.81	2.65½	2.81	2.96	3.12
x	x	x	x	x	2.53	2.39	2.53	2.67	2.81
x	x	x	x	x	2.36	2.23	2.36	2.49	2.62
x	x	x	x	x	2.53	2.39	2.53	2.67	2.81
1.40	1.52½	1.65	1.75	2.10	2.36	2.23	2.36	2.49	2.62
x	x	x	x	x	1.97	1.86	1.97	2.07	2.18
x	x	x	x	x	1.41	1.33	1.41	1.48	1.56
x	x	x	x	x	12½%	Reduced ½	Adv.	Adv.	Adv.
1.25	1.25	1.40	*	*	adv.	adv. of 1903	2.36	2.49	Adv.
.06	.07	.07½	.08	.09	.10	.09	.0965	.1030	.1050
.09	.10	.10½	.11	.12¼	.13½	.12¾	.1320	.14	14.35
.10½	.11½	.12	.12½	.13½	.14½	.14	.14½	.15	15.44
.11½	.12½	.13	.13½	.14¼	.16	.15⅛	.16	.16⅞	.1710
.22½	.25½	.28	.33	.41	.48	.45	.4835	.5170	.5550
.25½	.28½	.31	.36	.44	.51	.48	.5135	.5470	.5850
.28½	.31½	.34	.41½	.51½	.60	.56¼	.6030	.6435	.6885
.31½	.34½	.37	.44½	.54½	.63	.59½	.6330	.6735	.7185
E. P.	E. P.	E. P.	E. P.	E. P.	E. P.	E. P.	E. P.	E. P.	E. P.
.26½	.29½	.32½	139	.48¼	.56 15/100	.52 7/10	.5650	.6030	.6455
.29½	.32½	.35½	.42	.51¼	.59 15/100	.55 7/10	.5950	.6330	.6755
.03	.03	.03	.03	.03	.03	.03	.03	.03	.03
.01½	.01½	.01¾	.02	.02¼	.02½	.02⅜	.02½	.0263	.0276
4.20	4.50								
E. P.	E. P.	E. P.	E. P.	E. P.	E. P.	E. P.	E. P.	E. P.	E. P.

Renewed Effective April 1, 1908, to March 31, 1910

x Special prices according to nature of work.

WAGE SCALES IN THE HOCKING DISTRICT FOR THE YEARS 1892 TO 1921—*Continued*

	1914 to 1916	1916 to 1918	April to Oct., 1917	1917 to 1920	1920 to 1921
Pick Mining					
Run of mine, per ton	$0.676	$0.6764	$0.7764	$0.8764	$1.1164
Entries, dry, per yard	2.4996	2.6245	2.6245	3.0181	3.6217
Breakthroughs, in entries, per yard	2.4996	2.6245	2.6245	3.0181	3.6217
Breakthroughs, in rooms, per yard	1.7330	1.8196	1.8196	2.0925	2.5110
Room turning, per ton	3.7884	3.978	3.978	4.5747	5.4896
Inside Day Labor					
Tracklayers, per day	2.84	2.98	3.60	5.00	6.00
Tracklayer' helpers, per day	2.62	2.75	3.35	4.75	5.75
Trappers	1.32	1.40	1.90	2.65	3.18
(Where old men are employed)	1.50	1.59	2.19	3.59	4 59
Bottom cagers, drivers, trip riders, per day	2.84	2.98	3.60	5.00	6.00
Snappers on gathering locomotives, per day	2.84	2.98	3.60	5.00	6.00
Water haulers, machine haulers, per day	2.84	2.98	3.60	5.00	6.00
Timbermen, per day	2.84	2.98	3.60	5.00	6.00
Pipe men, for compressed air plants, per day	2.78	2.92	3.52	4.92	5.92
Wire men, per day	2.84	2.98	3.60	5.00	6.00
Motormen, per day	2.84	2.98	3.60	5.00	6.00
Motormen (minimum), per day	......	2.98	3.60	5.00	6.00
All other inside day labor, per day	2.62	2.75	3.35	4.75	5.75
Spike team drivers, 25 cents per day extra					
Machine Cutting					
By Jeffrey style of machines, in room, per ton	.07	.074	.0890	.1040	.14
By Jeffrey style of machines, in entry, per ton	.0970	.10235	.1173	.1365	.1790
By punching machines, in rooms, per ton	.1044	.1084	....	.1384	.1744
By punching machines, in entries, per ton	.1156	.12016	....	.1518	.1905
Loading					
In rooms, per ton	.38	.406	.4910	.5760	
In rooms, with hand drilling, per ton	.40	.426	.5110	.5960	.80
In entry, per ton	.4690	.49945	.5845	.6835	
In entry, with hand drilling, per ton	.4890	.51945	.6045	.7035	.9290
Breakthroughs in entry, per ton	E. P.	.51945	.6045	.7035	.9290
Breakthroughs in rooms	.46	.489	.5740	.6684	.9290
Breakthroughs in rooms	.44				
Drilling					
By hand, per ton	.02	.02	....	.02	
By machine, per ton	.0186	.0195	....	.0195	
Room turning, cutter and loader	E. P.				
Outside Day Wage Scale					
First blacksmith, per day	3.12	3.27	3.87	5.27	6.27
Second blacksmith, per day	2.81	2.95	3.55	4.95	5.95
Blacksmith helpers, per day	2.62	2.75	3.55	4.75	5.75
Carpenters, per day	2.81	2.95	3.55	4.95	5.95
Dumpers and trimmers, per day	2.62	2.75	3.35	4.75	5.75
Slack haulers, per day	2.18	2.28	2.88		
Greasers and couplers, per day	1.56	1.64	2.24	3.24	4.24
Where engineers and firemen are employed by the day, the minimum rate is $4.75 for 8 hrs. This does not apply to men employed at a monthly rate. This also applies to coal washers.			60c a day advance to all other labor.	All day and monthly men advanced $1.40 a day.	

The whole nation must get together to produce a stability in business which will make steady work in coal mines and in every other form of activity.

The case of the miner against irregular operation has already been forcibly set before the public. What is not so generally realized is that the case of the operator is just as damaging to him. His capital is idle and his mine equipment, instead of benefiting by a rest, is rapidly depreciating. Although the mine shuts down, his fixed charges run on—not only interest charges and salaries, but a host of maintenance charges as well. And in the end the coal consumer pays the bill for idleness of miner and mine.

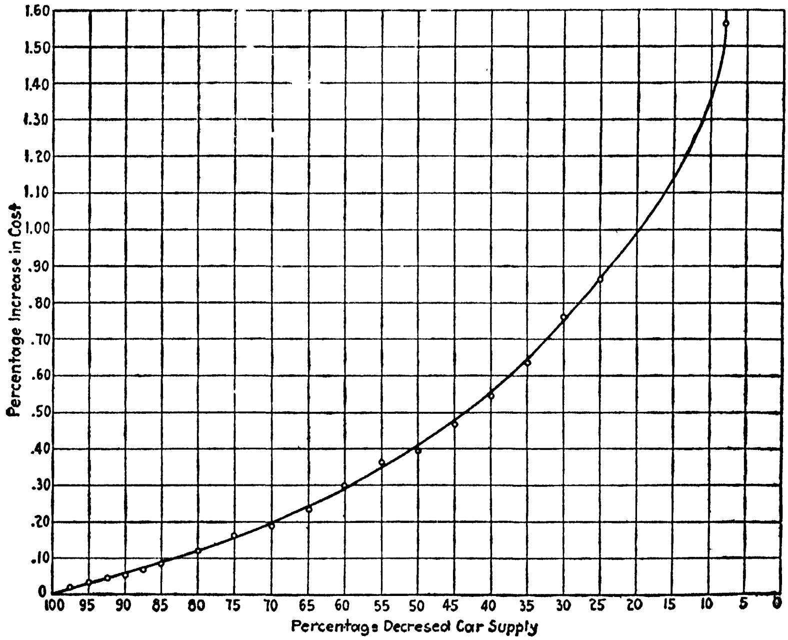


FIG. 53.—Percentage of increase cost due to irregular working time compiled from 830 observations in the New River field.

In this connection we may find instruction in an exceedingly valuable study made by Messrs. Garnsey, Allport, and Norris of the costs of production as effected by interruptions of working time. Fig. 53 is taken from their "Report of the Engineers' Committee of the United States Fuel Administration, 1918–19." It represents an analysis of the monthly records of seventy-three operators in the New River district of West Virginia. Each of these was carefully analyzed, and the percentage increase of cost for each of the 830 observations thus obtained was plotted; weighed averages were then taken at each 2.5 per cent from 70 to 100 per cent working time, and for each 5 per cent below 80 per cent. The result of this study is shown in Fig. 53, which has been checked by numerous observations from practically every field and has been found, within reasonable limits, to be correct. This diagram can and has been used in reducing to normal cost the reported cost of collieries shut down during parts of months. The reason for the increased cost per unit of output is, that the smaller the number of tons produced the larger is the share of fixed overhead expenses that must be borne by each ton. F. S. Peabody testifying before the Frelinghuysen committee in 1919 stated that: "the earning of the laborer and the cost of coal depend entirely on continuous work. Our costs will vary from month to month, dependent on the running time of our mines. There will be a variation of between 50 and 60c. a ton from month to month, depending on the number of hours the mines are idle."

The average number of productive days worked per annum in Illinois and Indiana is only about 175 out of a possible 300 or more. This idle time of the miners is not confined to one season or period during which they can find employment elsewhere. The men are always subject to call, for which reason they urge a greater daily wage so that that their annual income may be sufficient for their needs. This causes these operators to grant abnormal wage advances, which are directly reflected in coal cost.

Many industrial plants which produce standard or basic commodities find it possible to operate 24 hr. per day by using different shifts of men. They work also for 310 or more days a year, or a total of 7440 hr. per annum. Still other industries, on two 8- or 10-hr. shifts per 24 hr. work 300 to 310 days per year, thus operating 5000 to 6000 hr. every 12 months.

Even one 8-hr. shift in each 24-hr. period with 310 days per year gives 2480 working hours in every 12 months. Because of the unrestricted competition the mine operators of

Illinois and Indiana have built more plants than are needed and can only operate for 8-hr. out of every 24, and for 175 days per year, or 1400 hr.

It will be seen, therefore, that as against 100-per cent plant utilization (24 hr. for 310 days or 7440 hr. per annum) possible in some industries, and as against an average by all industries of 33 per cent to 45 per cent (one 8 or 10-hr. shift per 24-hr. period for 310 days), a coal plant is in actual productive use only about 18 per cent of the mine. This makes plant, interest and depreciation charges six times as heavy as for some other industries.

The 97,000 miners of Illinois and Indiana who are prevented from working 125 days per year might at the 1915 wage have earned an additional \$36,400,000, or \$371 per man per year, had their employers been able to give them work or had their efforts been expended in other directions.

Because the operators can give their miners work only a part of the time, these operators must pay higher daily wages than are warranted by the current selling prices. Their labor cost (in 1915) is 92.44c. per ton, whereas the selling price is but \$1.14 and \$1.11 respectively for the states of Illinois and Indiana.

Statistics of the coal-mining industry for 1912, give a total production of bituminous coal of 450,000,000 tons, with a total production cost per ton as follows:

Direct labor per ton	\$0.5425
Indirect labor (day work)	0.2383
Salaries	0.0575
Supplies	0.1305
Royalties	0.0320
Miscellaneous expenses	0.0530
Total per ton	\$1.0538

Assuming that as claimed by competent authorities, the output from the same workings might have been 600,000,000 tons, an increase of $33^1/_3$ per cent, with little advance in the expenditures, let us see how this increase in output would affect the production cost per ton.

Direct labor and royalties probably would remain about as before, but the other expense items would show a reduction due to the increased output about as follows:

Direct labor per ton	$0.5425
Indirect labor (day work)	0.1790
Salaries	0.0432
Supplies	0.0978
Royalties	0.0320
Miscellaneous expenses	0.0400
Total per ton	$0 9345

This shows a decrease in production cost per ton of 11.93c., or 11.4 per cent, due simply to increase of tonnage produced.

Economic aspects of conservation.—It has been customary to refer to the unlimited coal resources .of the United States, but while our country has been wonderfully blessed in this respect, the exhaustion of some of the choicest coal beds is already in sight, as for instance, the high-grade coking coals of the Connellsville region. Conservative estimators realize that the coal supply should now probably be measured by hundreds rather than by thousands of years, and that it behooves us to conserve our fuel resources. Large areas of high-grade fuel undoubtedly yet remain, and it is not too late to utilize these deposits much more fully than has been done with similar deposits in the past, both by more skillful mining and by a more thorough utilization of the coal after it is brought to the surface.

The Anthracite Coal Waste Commission in 1893 estimated that probably not over 30 to 35 per cent of the coal originally contained in the areas mined over has been saved, and that even by working over the old culm banks and reworking the area already mined, not over 10 per cent additional would be obtained, thus giving a loss of 50 to 60 per cent of the original coal. By means of the very close washing and sorting of the small sizes, and the better removal of the pillars through the crushing of culm and other methods, the amount of waste may be somewhat less than is estimated by the Coal Waste Commission, but still an enormous waste is going on, of a material which is not duplicated anywhere else in the country, and of which the supply is comparatively limited.

Dr. I. C. White, has estimated about 1909, that in mining bituminous coal in the United States not over 50 per cent of the coal in the ground has been obtained. This figure is excessive for present-day practice, still the amount of waste is entirely too great and should be decreased.

That great portions of our coal deposits have been skimmed over, leaving rich territory abandoned because of poor and inadequate methods, which aimed at the recovery of the easily accessible and abandoned the more difficult, is apparent to every observer. Much valuable coal property has thus been wasted or ruined.

It is in a sense a moral obligation on the operator to recover the pillar and top coal that the loss to the country may be lessened, but where this involves an additional expense, it cannot be undertaken. Still, for every two acres of coal land which the operations exhaust they leave one acre of coal unrecovered and unrecoverable in the ground. This means that in Illinois each year 12,000 acres of coal land are exhausted, whereas the exhaustion should be but 8000 acres. In Indiana the depletion is 3000 acres, whereas it should not be more than 2000. In the whole country 100,000 acres are exhausted, whereas not more than 65,000 or 70,000 acres should have been thus made of no mineral value.

A summary of the recovery effected in the different mining fields will give a basis for fixing a standard recovery to work towards. Conditions prevailing in most of the important mining fields in 1914 were described in a paper presented before the West Virginia Mining Institute in that year.

In order to get an idea of what is being accomplished in other fields, inquiries were sent out to different sections of the United States. The results are shown in a condensed form by the accompanying table.

It will be noted on this table, the wide variation of percentages given for different districts in various states. All are large producers of coal, with one or two exceptions, and employ what are presumed to be modern methods of mining. It is noticeable that the thin seams usually are overlaid with good roof and the percentage of recovery is high. Also, that but one operator expects the ultimate recovery to fall below his present percentage.

In the southern Colorado field, where the roof and bottom conditions are favorable for pillar drawing, no roof coal is left for protection and the recovery is given as 80 to 90 per cent, working on the room-and-pillar system. It is claimed that in the Cañon City district, where the longwall system is used, that 100 per cent of the seam is recovered. This is in the thinner seam, which measures about 3 ft.

Rooms in the southern Colorado district are driven 16 to 18 ft. wide on 45-ft. centers, while in the Walsenburg district, where the coal is harder and has less cover, rooms are driven 35 to 40 ft. wide, leaving the same thickness in pillars which are recovered by machine and pick work. Track is laid on each side of the room and frequently one or two cuts are taken off the side of the pillar with a machine before beginning pick work.

The bottom of the Colorado seams is usually slate of a soft character, which heaves when weight is thrown onto the pillars, making it necessary frequently to drive a skip along the pillar in order to reach the back end before beginning to draw it.

There are districts where both roof and bottom conditions are unfavorable and much difficulty is encountered in breaking the overlying strata. In these sections the recovery is estimated to be 60 to 65 per cent; 15 or 20 per cent is lost in roof coal because the strata next overlying the coal cannot be propped.

Another company, operating in practically the same field, states its recovery runs 75 to 80 per cent of the entire seam, while a higher ultimate recovery is expected. This firm is now driving room entries to the boundaries and the last rooms are worked first, thus making it possible to draw the pillars on the retreat.

In the Michigan field, especially in the Saginaw district, the coal is in pockets rather than a continuous seam. The basin lies for the most part in a low, flat country, and shafts about 200 ft. deep are necessary to reach the coal. The bed averages about 3 ft., and is of poorer grade than the Ohio and Pennsylvania fuels, so that its market is somewhat limited.

The top in these mines is usually black slate, while one mine has a fireclay roof, making it necessary to leave top coal. Yet rooms are driven 40 ft. wide with track along each rib. The length of the room is 150 ft., as the miner pushes his cars from the working place to the entry. With the conditions just given, the recovery claimed is between 80 and 90 per cent. The 65 per cent recovery given in the table (Item 4) represents the result of leaving pillars for surface protection within the city limits.

Going into central Illinois fields, where the No. 6 seam is operated extensively, there are adverse public feelings and unsettled industrial and labor conditions, which materially affect the percentage of recovery. Surface costing $100 to $250 per acre cost the operator two or three times these values in cases of subsidences, if the mining rights do not clearly cover the property. Besides these factors, the companies operating in the Glen Carbon, Mt. Olive and Divernon fields, state that owing to thick, soft clay under the coal or great overburden (300 ft to 400 ft.) that they do not recover more than 50 per cent. In the southern fields better results are claimed, since the cover is about 110 ft. thick and all soft.

The slightly inferior seams of coal above or below the Nos. 5 and 6 seams are now receiving considerable thought as to future values, and for this reason they are trying to prevent roof movements by leaving sufficient pillars.

In the Sherrard field of Illinois, the recovery is reported at 90 per cent. Here the top and bottom are good and conditions propitious for drawing pillars. The seam of coal is only 3 ft. 8 in. thick, with many clay veins running through it, which evidently must effect recovery to some extent.

An inquiry sent into the southwest section of Pennsylvania shows a recovery of 72.5 per cent. Here 10 in. of roof coal is allowed to remain on account of drawslate and the operations for the past three years have been under the plant and town.

The bottom in this mine is fireclay of rather soft character. The rooms are driven from both sides of entries on 60-ft. centers and widened to 21 ft., leaving 39-ft. pillars to be drawn by machine and pick work. By this method, the ultimate recovery is expected to show a material increase over that given.

The company reporting from Westmoreland County, Pennsylvania, where conditions seem favorable, both in the steam- and gas-coal fields, shows a recovery of from 82 to 86 per cent of the entire seam; and expects the ultimate recovery to fall below these figures.

In Somerset County, Pennsylvania, where coals of the Allegheny series are worked, the recovery is given as 94.75 per cent of the entire seam. Here, excellent roof and bottom conditions prevail, and most room headings are driven to the limit before any rooms are driven at all. Then, the rooms are started at the rear and pillars drawn as soon as these are finished.

Practically the same conditions exist in the George's Creek field in the Sewickley seam, only the recovery is reported as 97 per cent. In this same field, the results obtained in the

TABLE OF PRINCIPAL FACTORS GOVERNING

Item	Operating District and State	Per Cent of Recovery of Entire Seam	Ultimate Recovery Compared to Present	Period of Operation, Years	Average Height of Seam	Roof Coal Carried
1	Southern Colorado	80–90	Same	5 to 30	8′ 6″	0 to 24″
2	Colorado (other Districts)	60–65	Same	5 to 30	8′ 6″	18″ to 24″
3	Colorado (other Districts)	75 80	Better	10 to 35	3′ to 7′	Few places
4	Saginaw District, Michigan	65, 80–90	Better	15	3′	1 of 10
5	Central Illinois	50	Same	20	8′	None
6	Southern Illinois	65–70	Same	20	8′	Yes
7	Springfield District, Illinois	55–75	Increase[4]	20 to 25	6′ to 7′	Some places
8	Franklin, Willamson and Saline Counties, Ill.	55–75	Increase[4]	18	5½′	Some places
9	Sherrard Field, Ill	90	Same	20	3′ 8″	None
10	Extreme Southwest Section, Pennsylvania	72½	Better	3	7′ 6″	10″
11	Pennsylvania-Westmoreland Co	84	Below	25 to 35	6′ 8″	None
11	Pennsylvania-Somerset Field	95	Same	8	3′ 11″	None
13	Maryland-Georges Creek Field	97	Same	12	3′ 0″	None
14	Maryland-Georges Creek Field	88	Same	94	9′ 0″	18″
15	Ohio, Belmont County	60	Same	....	5′ 6″	None
16	Eastern Ohio, Harrison County	70–75	Same	....	3′ 8″ to 5′ 0″	None
17	West Virginia	90	Same	50	8′	Some places
18	Alabama	No reply				
19	Tennessee	No reply				
20	Kentucky	No reply				
21	Kansas	No reply				
22	Iowa	No reply				

[1] Length of room—150 feet.
[2] Advocates retreat mining.
[3] No. 6 seam.

"Big Vein" or Pittsburgh seam, show 88 per cent. Considerable propping is necessary, owing to the drawslate and the wild coal just above it. The systems of mining the coal in this field have changed from time to time until now headings are driven 9 ft. wide, rooms only 13 ft. wide and the distance between room centers maintained at 100 ft., thus providing against squeezes. Under this process, 90 per cent extraction is expected.

In Ohio, as well as some parts of West Virginia, no attempt is made to draw pillars at all. Rooms are driven 25 ft. wide

RECOVERY OF COAL IN DIFFERENT DISTRICTS

Nature of Top	Nature of Bottom	System of Mining	Are Pillars Drawn	Clay Veins Encountered
Slate	Soft slate	Room and pillar	Yes	Dikes
Very soft	Soft slate	Room and pillar	Yes	None
Sandstone, poor shale	Same as top	Room and pillar	Yes	None
Black slate	Fire clay	Room and pillar	Where allowed	None [1]
Slate, clod and limestone	Fire clay	Panel system	None	None [2]
Sandy shale	Fire clay	Panel system	To some extent	None [3]
Hard shale	Fire clay	Room and pillar	Where allowed	None
Hard shale	Fire clay	Room and pillar	Where allowed	None
Blue rock and cap rock	Slate and sand rock	Room and pillar	Yes	Yes
18" to 3' 0' draw slate	Soft fire clay	Room and pillar	Yes	None
Slate	Fire clay	Room and pillar	Yes	None
Hard black slate	Limestone	Room and pillar	Yes	Very few [5]
Sand rock	Sand rock	Room and pillar	Yes	Yes [6]
Gray shale, coal, dark shale	Hard gray shale	Room and pillar	Yes	None
Slate and shale	Fire clay	Room and pillar	None	None
10" firm slate	Fire clay	Room and pillar	None	None
Varies	Fire clay	Room and pillar	Yes	Yes

[4] Adjustment of labor situation.
[5] Projected work adhered to.
[6] Sewickley seam.

with 8 to 12-ft. pillars between. In one of the largest mines of Belmont County, rooms were driven from both sides of the headings and it was no infrequent occurrence to have a territory squeeze shut, leaving considerable blocks of coal between the ends of unfinished rooms. In this mine, 50 per cent would approximate the recovery.

In Harrison County, Ohio, it is nearly as bad. The recovery is reported as 70 to 75 per cent, but the same conditions exist in this section as in Belmont County, excepting perhaps the driving of rooms both ways from the same entry. The Ohio Mining Commission found in its investigations that 30, 40 and as high as 50 per cent of coal is being left underground as pillars in that state.

There are mines in West Virginia which show recovery from 85.6 per cent to 99.8 per cent, the highest percentage resulting from the fact that all the work was in the solid. The average result of the figures presented for 10 mines showed about 92.6 per cent.

The foregoing figures reveal what is possible, at the same time showing what is actually, presumably, being accomplished.

It would not be proper to accept an average of the percentages here given as a fair maximum, nor even an average of the same field, as it is unfair to compare ultimate recovery of mines now drawing to a close with those at the best of their production. No doubt, the systems under which they were inaugurated were considered modern, but they would not be considered so now.

From reports sent in, it is apparent that there are five factors limiting the possible recovery in these fields as follows:

1. Mining rights and public feeling.
2. Roof and bottom conditions.
3. Weight and character of overburden.
4. Labor conditions.
5. Market value of the coal.

1. Where the mining rights do not allow breaking of surface, the recovery naturally varies inversely in some ratio to the overlying weight.

2. Where roof and bottom conditions make it necessary to recover as quickly as possible, market conditions will affect recovery for pillar work of this kind will not wait.

3. Weight and character of overburden require systematic mining and competent supervision.

4. It is a matter of what is next best when unions insist on conditions which increase both cost of operating and loss of coal.

5. The market value of the coal dictates how far it is possible to go toward its recovery.

These points are mentioned because there is a tendency to compare straight figures of recovery without taking into consideration the conditions under which they are derived. The Ohio Mining Commission, for instance, uses the mines and operations at Gary for an example of what Ohio should follow. Conditions, however, are so different in these two localities, that to secure the same results in recovery would require several radical changes. Generally the roof in Ohio is poor, union scales require rooms entirely too wide for economic pillar drawing, the general labor situation is always more or less unsettled and the selling qualities of the coal are inferior to those of the Pocahontas seam at Gary.

Many of the difficulties incident to the adoption of adequate conservation measures were described in a paper presented before the West Virginia Mining Institute in 1908. There are four factors, any one of which is sufficient to cause a serious loss in the percentage of coal recovered and an increase in cost of production, reducing the ultimate earnings of the property:

First—Insufficient or incompetent engineering: Until very recently the engineer was regarded by many coal operators as a luxury and an unnecessary refinement. This unreasonable conservatism, or prejudice, still exists to some extent; but the rapid depletion of properties which have been regarded for many years as practically inexhaustible is finally bringing the operator to a realization of the necessity for the careful planning and scientific projection of his mine by a competent and sufficient engineering force.

Second—Incompetent management: There are some mine managers or superintendents who produce better results from a poorly designed mine than others can obtain from a first-class plant. Many managers have been entirely satisfied with a low cost-sheet, neglecting the conditions both inside and outside the mine, which ultimately resulted in an abnormal increase in cost of production or abandonment of valuable acreages of coal, in order to maintain lower cost. This method would be repeated until, finally, after millions of tons of coal had been ruthlessly buried, which could have been recovered by a prudent and careful manager without materially affecting his cost of production, he was brought to a sudden realization of an unnecessarily high cost and a diminished tonnage, resulting in loss of prestige and position for him and thousands of dollars to the owners. The necessity for a mine manager to be familiar with not only the outside, but also the inside, condition of the property, either personally or through tried and competent assistants, cannot be underestimated. Unless he is thoroughly familiar with these conditions, how can he know if the individual mine superintendent or foreman is giving him the desired results of low cost with maximum recovery, and the best conditions for a continuation of that low cost and recovery?

Third—Unfavorable labor conditions: The employment of unskilled miners renders it impossible to obtain a good recovery. Strikes and shut-downs have often interfered with the application of economic methods of extracting coal. Labor unions sometimes insist on conditions which, while operating for the convenience of the miner, increase the expense or induce an unnecessary loss of coal to the operator. For instance, in some districts of the country the track must be laid in the center of the room, with the gob on either side. Few or none of the pillars are recovered, and many acres of coal have been lost through squeezes and creeps because of wide rooms and small pillars. Nor is interference with the methods of mining the only manner in which the unions sometimes operate against the maximum recovery of coal. There have been men discharged for incompetency who were reinstated by the management on demand of the union. It is thus that discipline, which is such an important factor in the economic administration of a mine with a view to the best ultimate results, is destroyed.

Fourth—Impatience of owners for quick returns on investment: The deleterious effect on the economic development of a property by the demand of the investor for an immediate profit can hardly be overestimated. Many a rich and valuable property has been irreparably damaged by the insistence of owners for immediate and large profits before its proposed economic development has been fairly launched. This impatience and greed has at times resulted in the changing of slow, but good plans of development for bad ones, and the poor results thus obtained were further accentuated in later years by the demand for big tonnage, thereby causing the loss of millions of tons of coal and thousands of dollars to the investor. Indeed, this demand for large tonnage from a poorly developed mine has been the greatest factor in encouraging careless methods of recovering coal both from rooms and pillars, and it is no exaggeration to state, that the abandoning of many acres of good coal, which could have been recovered by more thorough and known methods, can be traced directly to this cause.

Use of the longwall system to effect conservation.—The adoption of the longwall system of mining, where possible, will be the ultimate solution to obtaining the maximum recovery of coal and the subject is, therefore, one for the serious consideration of the coal economist.

To obtain a comparison of the results of the different systems a 1000-acre tract of land with 6 ft. of coal lying at a depth of 400 ft. will be assumed. This depth is taken to make allowance for the possibilities of the longwall system on surface caving, one of its chief disadvantages.

A conservative estimate of the recovery to be obtained from such a tract under present systems of mining would be about 60 per cent. The total tonnage in this acreage would be 9,680,000 short tons, 60 per cent of which would be 5,800,000 tons which means a loss of nearly 4,000,000 tons. At a value of $1 per ton this would mean a loss in the national wealth of $4,000,000.

In removing the entire seam some damage has perhaps resulted to the farmer, but not much, certainly, at a depth of 400 ft., and nothing beyond easy repairs. Where a total extraction has been effected on a 5-ft. seam having 100 ft. cover in certain of the Pennsylvania fields, the break has extended to the grass roots. Where this extraction has been several acres in extent and the break has been general over the entire area at once, it cannot be said that any appreciable damage has resulted. In this instance there was no packing whatever, as would ordinarily be the case in longwall workings. The conditions were simply 100 ft. of easily breaking roof with a clean fall of 5 ft.

Taking our previous example again, of 6 ft. of coal at a depth of 400 ft. worked by the longwall system and packed carefully, the result would by no means be so serious. Furthermore, it should not be forgotten that there are approximately 1000 sq. mi. of coal in which the seams are 600 ft. or more below the surface.* At this latter depth it is doubtful if the strata would break to the surface, for it has been shown in the British mines that at a depth of 700 or 800 ft. work can be carried on successfully under the sea.

This proves that in average strata the highest fall reaches an apex well below that distance, and the miner who has had extensive experience in pillar drawing and is familiar with the quick oblique line that the ragged rock edges traverse toward a common juncture, will place the safety point much lower.

Turning again to our example of a 1000-acre tract, and assuming this to have been worked by the longwall system which has resulted in certain damage to the surface, a comparison of this surface damage, with the additional extraction obtained, may be made. Taking the mine on a royalty basis of 5c. a ton, the farmer has received $193,600 above what he would have received by the 60 per cent pillar-and-room method of working. This amounts to $193 per acre, or about twice as much as the average farm value of land. It should also be remembered that the expense of tipple erection, compressors and power plants and the general surface arrangement of a mine opened to develop a tract of 1000 acres, is the same for 60 per cent extraction as for 100.

It is said that in longwall working continuous operation is necessary, in order to properly control the roof, but this applies nearly, if not quite as well, to room-and-pillar work, particularly when the mine contains water. The statement of a certain large Western operator, a man who is both practical and theoretical, may be taken, *apropos* of this. He changed a number of his mines to the longwall system, and during a strike

*See Twenty-second Annual Report, U. S. Geological Survey, page 178.

lasting over a period of about five months and a half, all of his mines were shut down. On the resumption of operations he had careful records kept of the relative cost of opening the longwall and the room-and-pillar mines. These records showed the cost of opening the room-and-pillar mines to be nine times that of the longwall.

SECTION II

SHAFT SINKING

The investment involved in shaft sinking is heavier than that encountered with any other improvement. Mistakes can be neither rectified nor lived down. Time and first cost being the essence of the opening of a new property, important features are often sacrificed underground instead of on the surface, where future remodeling is practicable. It is, therefore, obvious that the preliminary engineering and estimating relative to a shaft operation deserve serious study.

In planning a shaft mine opening the following points come up for consideration: Avoidance of all unnecessary narrow-work at the bottom increasing the risk of loss from squeeze, compliance with all present and possible future legislation, minimum first and maintenance cost, the connecting up of the two shafts in the shortest time in order that a regular circulation of air may be obtained and the restrictions of the law limiting the number of men allowed in the mine before this is done complied with. The arrangement of the work should be such that during development cars may be placed with the utmost facility, that mining machines may be employed and steel timbering placed as the entries advance; that when operations begin, cars, supplies, waste, men, water, air, damaged equipment, etc., may be handled with safety, economy and speed; that protection is secured against coal-dust explosions, mine fires, flooding, freezing, etc.

Operations.—The universal method of shaft sinking in rock is to drill a number of holes in the bottom, charge them with dynamite and shoot them, and to load the broken rock by hand into buckets which are then hoisted out. When all the loose rock has been removed the process is repeated.

Shafts are drilled on the "center-cut" principle. Eight or ten holes are drilled on a slant, separated at the top but converging, thus forming a wedge known as the "sump." "Reliever," or bench, holes are drilled back of the sump holes, each row being more nearly vertical; the end or outside holes point slightly away from the vertical and toward the wall line of the shaft. The sump is first shot and the broken rock removed or "mucked" out, forming a cavity into which the bench rounds can be successively shot. All muck should be removed before each succeeding round is shot.

Two systems of drilling and mucking exist. In the first the holes for the entire cut—sump and benches—are drilled at one time, the sump is shot, and then the benches as required. In the second, the sump only is drilled and shot, and the benches are drilled while the sump is being mucked. The first plan is particularly applicable to small shafts and to circular shafts; a rectangular or elliptical shape is needed to give room for simultaneous drilling and mucking.

Fumeless, or gelatine, dynamite should in all cases be used for underground work. The fumes from ordinary glycerine dynamite make it impossible for the men to get back to work promptly after a shot. The strength of the dynamite used depends on the character of the rock, but 40-per-cent and 60-per cent gelatine are the most common strengths used.

The number and depth of the holes and the quantities of powder loaded vary so greatly with the size of the shaft and the nature of the rock that no general rules can be stated. The systems actually used at several shafts were as follows:

Shaft 13×26 ft., through Western Pennsylvania coal measures: Shale, slate, and limestone; horizontal stratification; 40-per-cent gelatine:

	Number	Depth, Feet	Inclination with Vertical, Degrees	Loaded with Pounds
Sump	8	10	45	4
Relievers	8	8	30	3
Benches	8	8	0	2½
End	8	8	10 back	2½
Total charge	....	....	....	96

Average gain per cut, 6 ft.

Average gain per week of 19 shifts, 24 ft. (no timber).

Mucking and drilling simultaneous; 2 drills used on 1 bar, double.

Shaft 14 × 48 ft., through anthracite measures: Red sandstone; stratification horizontal; 40-per-cent gelatine:

	Number	Depth, Feet	Inclination, Degrees	Loaded with, Pounds
Sump	8	10	45	5
Relievers	8	8	30	4
Benches	24	8	10	3
End	8	8	10 back	3
Total charge per round	....	....	....	168

Average gain per cut, 6 ft.
Average gain per week of 18 shifts, 16 ft.
Mucking and drilling simultaneous; 2 drills used on 1 bar.

Shaft 10 × 22 ft., through quartz conglomerate (Shawangunk grit); horizontal stratification, but very few bedding planes; 60-per-cent gelatine:

	Number	Depth, Feet	Inclination, Degrees	Loaded with, Pounds
Sump	8	10	45	3½
Sump	4	8	0	3½
Relievers	8	9	30	2½
Benches	8	8	0	2
End	8	8	10 back	2
Total charge per round	....	....	....	94

Average gain per cut, 5½ ft.
Average gain per week of 20 shifts, 22 ft.
Mucking and drilling simultaneous; 5 drills used on 2 bars.

The four additional sump holes shown were used on account of extra hardness of the rock.

Shaft elliptical, 19 ft. 4 in. × 33 ft., through West Virginia coal measures: Hard gray sandstones; 40-per-cent gelatine; horizontal stratification:

	Number	Depth, Feet	Inclination, Degrees	Loaded with Pounds
Sump	10	12	45	5
Relievers	8	10	30	4
Benches	14	10	10	4
End	6	10	10 back	3
Total charge per round	....	....	....	156

Average gain per cut, 8 ft.
Average gain per week of 20 shifts, 18 ft.
Mucking and drilling simultaneous; 3 drills used on 1 long bar, 1 short bar.

Shaft circular, 17 ft. diameter, through Hamilton and Marcellus shales: Rock distorted; stratification irregular; but about 45 deg.; 60-per-cent gelatine:

	Number	Depth, Feet	Inclination, Degrees	Loaded with Pounds
Sump	6	8	45	2½
Relievers	8	6	20	1½
Rib	16	6	10 back	1
Total charge per round	....	....	....	43

Average gain per cut, 5½ ft.
Average gain per week of 19 shifts, 33 ft.
All drilling on one shift, mucking on two shifts; 5 drills used on 5 tripods.

While the hand drilling has been displaced almost entirely by power driven drills, there are still occasions when a small job does not justify the installation of power equipment and it is more economical to resort to hand drilling. This is particularly so in foreign countries where labor costs are frequently quite low.

The customary procedure is the use of a 1-in. drill, turned by one man and struck by one or two others with 8-lb. hammers. Two strikers should always be used where practicable as they can obviously drill twice as fast as a single striker at three-fourths the cost. Three capable men can drill 1¼-in. holes in hard sandstone at the rate of 2 ft. per hour.

In soft material, churn drills 6 to 12 ft. long with a bit at each end are sometimes used with satisfactory results. These drills are sometimes weighted to give additional striking power and they are usually operated by two or three men.

Shaft sinking is usually carried on 24 hr. a day. The inside work is done by three shifts of men working 8 hr. each, the outside by three 8-hr. or two 12-hr shifts. The 12-hr. outside shift is customary in the coal fields; elsewhere, the 8-hr. shift for every one is prevalent. Shifts are usually changed at 7 a.m. and 3 and 11 p.m., sometimes an hour later. The men are given 20 min. for lunch in the middle of each shift.

Wages vary with the locality, but in general men are paid better for drilling and mucking in a shaft than in any other kind of rock excavation. On account of the high wages paid in America machine drilling is universal, and the shifts are limited to the number of men that can be worked to the best advantage. Speed is not attempted at the expense of efficiency. In South Africa, on the other hand, Kaffir labor is cheap, hand drilling is usual, and as many men are worked as the shafts will hold.

The great depth of the shafts on the Rand makes the highest possible speed desirable, even at an increased cost. In both countries speed is increased without an increase of cost by the payment of a bonus to the sinkers as a reward for additional progress.

The size of the shifts for any given shaft depends upon the number of drills required and upon the experience and ability of the sinkers obtainable. With first-class men, the men on each shift for a 13 × 26 ft. shaft would be as follows, wages as of 1909:

Inside men, 8 hr.: One shiftboss, at $3; two drillers, at $2.75; two helpers, at $2.50; six muckers, at $2.25.

Outside men, 12 hr.: One engineer; one head tender; three car-men on dump; one firemen; one compressor man.

General outside, 10 hr.: One foreman; one mechanic; two carpenters (on timber); one blacksmith and helper.

A 17-ft. circular shaft would require:

Drilling shift: One shiftboss; five drillers; five helpers; one extra man.

Mucking shift: One shiftboss; nine muckers.

Outside: Same as above.

In South African shafts, which are usually 9×26 ft., drilling is always done by hand and each shift consists one white shiftboss and 35 Kaffir laborers who drill or muck as may be required.

Thorough organization is essential to progress and economy. Each man must know his place and take it without losing time in getting started. Any system that prevents systematic work is fatal to economy.

Circular or Rectangular.—From a construction standpoint the circular or elliptical and rectangular types are equally feasible, and the choice depends upon the cost. In several cases a compromise has been effected by shaping the shaft as a quadrilateral with sides formed of circular arcs.

For a single compartment air-shaft the circular shape is in every way the most desirable, not only because the circular shaft is cheaper to sink than a square shaft of equal area, but also because a circular ring of plain concrete is the strongest lining possible with a given amount of material.

In the case of a shaft with two or more compartments, the selection of the most economical shape requires some calculation. At first sight it would seem that a simple rectangular shaft surrounded by a concrete wall only thick enough to be as strong as the usual timber lining, would be a satisfactory, as well as a cheap, shape, but this is not the case. A concrete lining, even when provided with weep holes, must resist some hydrostatic pressure; a timber lining has none to resist. Furthermore, permanent weep holes are most undesirable; the concrete should exclude the water entirely, and hence must be designed to bear very great pressure at considerable depth. Just what amount the theoretical pressure is reduced, by the adhesion of the concrete to the shaft walls and by the blocking of the fissures with grout, cannot be calculated. In solid rock, where the water enters in well-defined springs, the proper grouting of the springs will relieve the lining of all pressure. In very seamy rock, on the other hand, the lining may have to bear practically the full hydrostatic pressure.

In order to compare the costs of the different shapes, let us consider in detail three designs for a shaft with two 7×10 ft. hoistways and an airway with an area of 100 sq. ft. As the

QUANTITIES AND COSTS OF RECTANGULAR SHAFT

Depth in feet	20	50	100	150	200
Total thickness of lining in inches	14	21	28	34	39
Quantities per linear foot:					
Concrete to neat line in cubic yards	3.90	5.70	7.60	9.30	10.70
Concrete actual in cubic yards	5.80	7.70	9.70	11.50	13.00
Excavation to neat line in cubic yards	12.80	14.60	16.50	18.20	19.70
Excavation actual in cubic yards	14.70	16.60	18.60	20.40	22.00
Weight reinforcing steel in pounds	256	443	650	845	1030
Cost per linear foot:					
Forms	$25.00	$25.00	$25.00	$25.00	$25.00
Concrete at $5 cubic yard	29.00	38.50	48.50	57.50	65.00
Excavation (see note *)	49.60	53.20	57.00	60.40	63.40
Reinforcing steel at $0.02 pound	5.10	8.90	13.00	16.90	20.60
Total	$108.70	$125.60	$143.50	$159.80	$174.00

QUANTITIES AND COST OF ELLIPTICAL SHAFT

Depth in feet, 0 to	100	150	200	250	300	400
Thickness of lining in inches, ends	12	12	12	12	12	12
Thickness of lining in inches, sides	12	18	24	29	34	42
Quantities per linear foot:						
Concrete to neat line, cubic yards	2.60	3.40	4.30	5.00	5.70	6.80
Concrete actual in cubic yards	4.40	5.20	6.10	6.80	7.50	8.60
Excavation to neat line in cubic yards	$15.20	$16.00	$16.90	$17.60	$18.30	$19.40
Excavation actual in cubic yards	17.00	17.80	18.70	19.40	20.10	21.20
Costs per linear foot:						
Forms	15.00	15.00	15.00	15.00	15.00	15.00
Concrete at $5 cubic yard	22.00	26.00	30.50	34.00	37.00	43.00
Excavation *	54.40	56.00	57.80	59.20	60.60	62.80
Total	$91.40	$97.00	$103.30	$108.20	$113.10	$120.80

QUANTITIES AND COSTS OF QUADRILATERAL SHAFT

Depth in feet 0 to	100	150	200	250	300	400
Thickness of lining in inches	12	19	26	32	39	52
Quantities per linear foot:						
Concrete to neat line in cubic yards	2.70	4.40	6.20	7.90	9.90	13.90
Concrete actual in cubic yards	4.50	6.30	8.20	10.00	12.10	16.20
Excavation to neat line in cubic yards	14.90	16.60	18.40	20.10	22.10	26.10
Excavation actual in cubic yards	16.70	18.50	20.40	22.20	24.30	28.40
Costs per linear foot:						
Forms	$15.00	$15.00	$15.00	$15.00	$15.00	$15.00
Concrete at $5 cubic yard	22.50	31.50	41.00	50.00	60.50	81.00
Excavation (see note *)	53.80	57.20	60.80	64.20	68.20	76.20
Total	$91.30	$103.70	$116.80	$129.20	$143.70	$172.20

* Cost of excavation figured on basis of $4 per cubic yard for section containing 12 yards per linear foot; additional excavation at $2 per cubic yard. Thus cost of 16 cubic yard section = 12×$4+4×$2=$56.

whole area of a hoist shaft is ordinarily used for the passage of air, the size of the air compartment may be reduced if the rest of the shaft is enlarged; the airway must however be large enough to contain pipes and ladders and to provide in addition an ample passage for air if the hoistways are temporarily closed.

Let us assume a minimum thickness of 12 in. of concrete for a water-tight lining; also that in each case the lining carries the entire hydrostatic pressure; then the specifications for the three forms of shafts will be as follows:

Rectangular Shaft.—Fig. 1. Two hoistways 7 × 10 ft., one airway 10 × 10 ft. Ten-inch concrete dividing walls in place of buntons. Extreme inside dimensions 10 × 25 ft. 8 in. Area

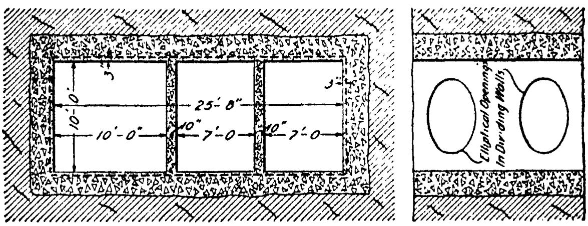


FIG. 1.—Rectangular concrete-lined shaft.

airway 100 sq. ft., total clear area 240 sq. ft. Thickness of lining at any point made equal to depth of simple beam of 10 ft. span required to sustain hydrostatic pressure at that point. Resisting moment and weight of reinforcement calculated by Johnson's formula, factor of safety 3. (Ultimate tensile strength of steel 65,000 lb. per square inch, compressive strength of concrete in beam 2500 per square inch.) Reinforcing steel set 3 in. inside of face of wall.

Cost of forms, given in the accompanying table, includes cost of forms for dividing walls, and is therefore greater than the cost in the elliptical shafts.

Excess of actual over theoretical quantity of excavation is estimated as 15 per cent for 28-ft. shaft. This excess increases with the length of the shaft only, as the ends are drilled to line.

Elliptical Shaft.—Fig. 2. Extreme inside dimensions 16 × 27 ft. Area of airway, 78 sq. ft. Total clear area, allowing for 10-in. buntons, 304 sq. ft.

Strength of lining calculated on the assumption that the stress in the elliptical cylinder at any point is equal to that caused in a circular cylinder with a radius equal to the radius of curvature of the ellipse at the given point, by the same hydrostatic pressure acting upon it. The lining is therefore made thicker at the sides than at the ends.

To prove this proposition assume the lining to be constructed of a number of small portions, each the arc of a circle. The stress in each portion caused by the hydrostatic pressure of the film of water between it and the rock is directly proportional to the radius, and the thickness of each section should therefore be made proportional to the radius. Considering any portion,

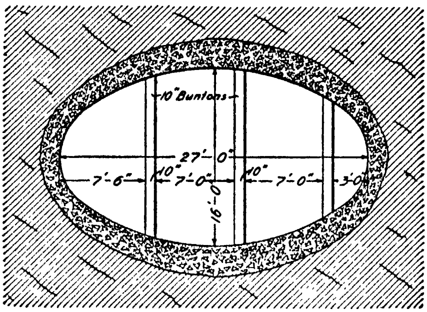


FIG. 2.—Elliptical concrete-lined shaft.

as *a-b*, Fig. 4, the skewback toward the side of the ellipse is formed entirely by the adjoining portion, while the skewback toward the end is formed partly by the adjoining portion and partly by the rock. If the number of circular portions is indefinitely increased, the unbalanced end thrust of each will be taken up by the irregularities of the rock.

Ultimate compressive strength of concrete, 3000 lb. per square inch; factor of safety, 3.

Excess of actual over theoretical excavation assumed as 12 per cent for smallest section. As the length of the shaft does not vary, this excess is constant.

Quadrilaterial Shaft.—Fig. 3. Inside dimensions, 16 × 24 ft. 8 in. Radius of ends and sides, 23 ft. Area of airway, 94 sq.ft. Total clear area, allowing for 10-in. buntons, 294 sq. ft.

For calculating stresses, sides and ends are considered as portions of a 46-ft. circular cylinder. Ultimate compressive strength of concrete 3000 lb. per square inch, factor of safety, 3.

Excess of actual over theoretical quantity of excavation assumed to be 12 per cent for minimum length and to increase with the length.

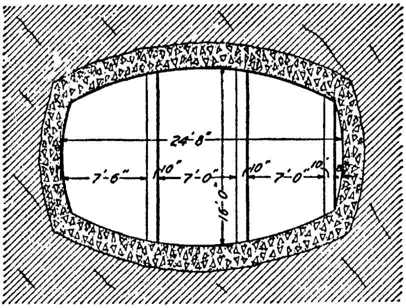


Fig. 3.—Quadrilateral concrete-lined shaft.

Germany is committed to the circular or elliptical form of shaft, the German engineers being of the opinion that the square or rectangular form is more expensive due to the extra work involved in excavating and keeping the corners squared up.

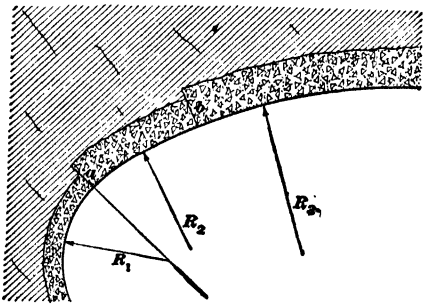


Fig. 4.—Design of the elliptical form of shaft.

It is evident, that assuming the same hoisting capacity in either form of shaft, the excess area, which makes ventilation possible, should be the same in either a circular or a rectangular shaft. A circular shaft of 20 ft. net diameter would be roughly equivalent to a rectangular shaft 12 × 20 ft. English mining engineers claim that the cost of lining is as 5 to 9 in favor of circular shafts, and it is generally conceded that where great pressure is encountered the circular form is the only safe one. A circular shaft, when once properly lined with iron or masonry, is a permanent affair, while timber lining under the best conditions cannot be expected to last more than 18 or 20 yr. and rarely more than 15 yr. It is also well known that for a given area, a circular shaft presents less rubbing surface, or resistance to the passage of the ventilating current, and the segments at the side of the cages furnish space for this air current without additional enlargement of the shaft.

The principal arguments advanced for rectangular shafts are that less material needs to be removed for a given cage space, and that in sinking, the permanent lining is at once put in place, as the work progresses.

It is probable that the matter of keeping a shaft in alinement during construction, by either method, is largely a matter of experience and character of labor employed.

While engineers have claimed cheapness of construction as an argument for both forms of shaft, the data at hand would indicate that the circular shaft may be excavated fully as cheaply as the rectangular, but costs more to line; while on the other hand the upkeep and repairs on a circular shaft properly constructed, are very much less than on a rectangular shaft of same capacity, and the danger, especially in deep mines or in quicksand, is very materially reduced, and water much more easily kept out, effecting a saving in pumping.

Equipment.—A two-stage compressor is the best for shaft sinking. With a steam consumption of 45 lb. per i.h.p. at half cut-off the simple compressor has for each indicated horsepower, a capacity of 5 cu. ft. of free air per minute compressed to 100 lb., while the two-stage machine will deliver 15 per cent more air with the same steam consumption. For 500 cu. ft. of free air per minute, the saving of the two-stage, over the simple type, will amount to 15 per cent × $\frac{1}{5}$ × 500 cu. ft. × 45 =675 lb. steam or 150 lb. of coal per hour. With the compressor operating to capacity 20 hr. a day, 6 days in the week, the saving in three months with coal at $4.50 per ton would thus be $525.

The cost of a plant for a single shaft, assuming a depth of about 500 ft. and a moderate inflow of water, say 30 or 40 gal. a min. was estimated in 1909 as follows:

Sinking engine	$1,000
Two 80-hp. boilers and setting	1,800
Pipe and auxiliaries	500
150-hp. heater	300
14-in. compressor	1,750
Three drills and steel	1,000
Shaft bar and clamps	100
Derrick	400
Head-frame	500
Two buckets	150
Rope	150
Buildings	500
Dump cars and rail	300
Electric plant, 10 kilowatts	750
Two pumps	500
Small tools	500
Total	$10,200

These figures are based on the cost of new machinery, and are large enough to include the necessary accessories. The cost of erecting and dismantling such a plant will be from $1000 to $2000, depending on location, labor conditions, etc.

Some of the most serious pumping problems in shaft sinking in this country have been encountered in the Pocahontas field along the Norfolk & Western Ry. Except in two instances, sinking has been very expensive in this territory owing, it is thought to the fact that the water-bearing rock is a very hard sandstone carrying a considerable volume of water. When a shaft is started beyond the toe of the mountain and in the valleys the sinking always encounters plenty of water, but in the two exceptions noted the location of the shaft in each case was back of the toe of the mountains and away from the valleys. The quantity of water in the wet workings, varies from 200 to 1100 gals. per min. The heavy charge of high explosive necessary to break the dense sandstone, coupled with the large volume of water, gives the worker constant pump trouble; moving the pumps during the blasting is out of the question, because the water would flow in so fast that the pumps could not be replaced for service. Having this constant pump trouble makes progress very slow, reducing it to 10 ft. per month in several instances. The following is a list of some of the shafts and amount of water encountered:

The Middlestate Coal Co. started an old shaft, abandoned 90 ft. down, which flowed 1100 gals. of water per minute, and they were about 18 months getting down the additional 90 ft.

The Pocahontas Collieries' 16 × 32 ft. shaft, at Boissvain, was 11 months in sinking 200 ft., hindered as it was by 500 gals. of water per minute.

The most expensive shaft proposition was the shaft for the Jed Coal & Coke Co. about 2 miles from Welch. This property's main shaft encountered 1100 gals. per min. while its air shaft met 500 gals. The coal expense for power has been as high as $1500 per month with coal at about $1.25 per ton.

Sinking costs.—The Nokomis Coal Co. sunk a shaft through the soft Illinois shales in 1913, the cost figures on which are of interest. The shaft was 631 ft. deep, timber lined and 17 ft. 5 in. × 11 ft. 5 in. inside the timbers, with an airshaft of the same dimensions 500 ft. distant. Eight hand-feed hammer drills, weighing 40 lb. each were used on the work and compressed air was furnished by a 9 × 10 × 12-in. compressor supplying 174 cu. ft. of free air per minute at a terminal pressure of 100 lb. per square inch.

After passing through the upper capping of hard rock, shale of various degrees of hardness was encountered with an occasional layer of limestone 10 or 12 ft. thick. The rock at times consisted of slaty bands, sandy shale or soft gray material, more like indurated clay than shale.

In sinking through limestone, from 28 to 32 holes constituted a round, the corner holes being bottomed 2 in. outside the line of curbing. As the shafts were timbered throughout, the break lines were 12 ft. 5 in. by 18 ft. 5 in. The holes were 4½ ft. deep and were connected and fired with an electric battery in the ordinary manner. Three 8-hr. shifts were worked, a round being drilled, blasted and mucked out on each shift. Four drillers, four muckers and a shift leader or boss constituted a sinking crew.

It was found that in trying to bore holes in the soft shale with these hammer drills, the tool cut so rapidly as to choke the passage in the bit with muck, stopping the flow of exhaust air and preventing proper cleaning of the drill holes. Hand drilling was temporarily substituted, but on adopting certain recommendations of the manufacturers, the hammer drills worked satisfactorily, and their use was resumed with a marked increase in speed over the hand work.

These changes gave the following results in soft shale, as compared with hand drilling. Hand drillers using $2\frac{1}{4}$-in. steel worked at the rate of $4\frac{1}{2}$ ft. per man per hour, or four men put in a round of 54 ft. in 3 hr. With the air drill, four men drilled 18.9 ft. per man per hour, or a round of 54 ft. in 45 min. thus accomplishing a saving of 2 hr. and 15 min. per 54-ft round.

Owing to the variation in the time required for the shooting and mucking, this increase in drilling speed meant an increase in depth per day of 1 ft. or $4\frac{1}{2}$ ft. per 24 hr., with hand drills. Sinkers, including drillers and muckers, were paid $3.39 per 8-hr. shift, the shiftboss receiving $4, making a total labor cost per day of roughly $93, or $26.50 per foot by hand drilling. The saving by using the hammer drills therefore, amounted to one foot in each shaft or $53 per 24 hr. for both shafts.

Interesting data on the comparative cost of sinking a vertical shaft and an inclined slope to accomplish the same purpose were disclosed in a request for tentative bids made to two leading contracting companies early in 1921. The conditions under which the work was to be done were as follows:

The shaft to be in Ohio, 5 miles from present railroad. Fuel, supplies, and equipment to be hauled in by the company. Company will furnish all timber. Strata shale and sandstones. Plenty of water available. Company will provide housing accommodations. Contractors to furnish all equipment except, ties, rails, and such supplies as can be used in permanent mine equipment. Waste disposal within radius of 500 ft. Assume an average amount of water in sinking.

First condition.—Shaft three compartments 10×26 ft. in clear 170 ft. deep to coal. Timbered with 8×10 in. sets on 5 ft. centers with 2 in. lagging.

Second condition.—Single track slope on 30 deg. pitch 5 ft. clear above top of rail by 7 ft. wide, 340 ft. long, timbered with 6 × 8 in. crossbars and posts with sets spaced 5 ft. centers and lagged with 2 in. planks for 170 ft.

Airshaft within 500 ft. of above slope mouth 10 × 10 ft. and 170 ft. deep timbered and lagged for 100 ft. with 8 × 10 in. sets on 5 ft. centers.

Third condition.—Same as first paragraph of second condition except duplicate slopes with 35 ft. horizontal pillar between. No airshaft.

Fourth condition.—Same as second condition except double track slope 12 ft. wide. Quote both with and without airshaft.

The first company submitted bids as follows:

The price of the shaft in the first condition will be $200 per vertical foot timbered.

Under the second condition, the price of the single track slope timbered will be in the neighborhood of $100 per lineal foot. An airshaft under this condition would cost in the neighborhood of $140 per vertical foot timbered and slopes under the third condition would be $100 each.

Under the fourth condition, the double track slope would cost approximately $125 per lineal foot sunk at the same time as an airshaft is being sunk. If no other opening is sunk at the same time, that is, with the same main plant and the same overhead, the slope would cost $140 per lineal foot.

The price for cement used as grout will run in the neighborhood of $12 per barrel.

The bids of the second company, which were gross, were as follows:

First condition.—Three compartment shaft, 10 × 26 ft. in the clear, 170 ft. deep, to cost $45,500.

Second Condition.—Single track slope, 5 ft. clear above top rail, 7 ft. wide, 340 ft. long and airshaft 10 × 10 ft., to cost $42,500.

Third condition.—Same as second condition, except duplicate slopes, no airshaft, to cost $42,400.

Fourth condition.—Same as second condition, except double track slope, 12 ft. wide with airshaft, $47,000; without airshaft $33,250.

There are certain items entering into the cost of such a short job as this which run the cost up considerably; for instance, the item of transportation in the first proposition, would make a cost of about $8 per foot of shaft.

Sinking costs given below were fairly representative for the different methods of work in 1909:

Shaft excavated 14 × 20½ ft. through 6 ft. of soil and 14 ft. of quicksand, not very wet. Sides supported by 2-in. oak sheeting driven by mauls and braced by five sets of 10 × 12 in. timber:

	Per Foot	Per Cubic Yard
Labor	$27.25	$2.57
Lumber, 6600 ft. B.M. at $30	9.90	.93
Erection of derrick, etc.	3.00	.29
Superintendence	3.00	.29
Sundry	2.00	.18
Coal and pumping	5.00	.47
Total	$50.15	$4.73

Shaft excavated 12 × 20 ft. 3 in. through 45 ft. of clay and gravel. Sides supported by sets of 10 × 10 in. pine timber spaced 4½ ft. centers and hung from top. 1½-in. lagging:

	Per Foot	Per Cubic Yard
Labor	$19.50	$2.17
Lumber, 240 ft. per foot at $25	6.00	.66
Bolts, 15 lb. per foot at $0.03	.45	.05
Erection of head-frame, etc.	2.00	.22
Superintendence	2.00	.22
Power	1.50	.17
Sundry	1.00	.11
Total	$32.45	$3.60

Shaft excavated 15 × 37 ft. through 21 ft. of dry sand. Sides supported by interlocking steel sheet piling driven with steam hammer and braced with sets of 8 × 10 in. timber:

Labor Costs Only	Per Foot	Per Cubic Yard
Driving sheeting	$6.55	$0.32
Removing sheeting	1.85	.09
Timbering	2.05	.10
Excavation	8.20	.40
Total	$18.65	$0.91

Caisson 26 ft. outside diameter, 21 ft. inside diameter, sunk through 56 ft. semiliquid mud and boulders:

	Per Foot	Per Cubic Yard Excavation
Concrete: Materials	$27.00	$1.35
Concrete: Labor	7.00	.35
Concrete: Forms and shoe	23.00	1.15
Sinking caisson	38.00	1.90
Plant erection	3.00	.15
Superintendence	5.00	.25
Sundry	5.00	.25
Coal and power	6.00	.30
Total	$114.00	$5.70

The H. C. Frick Coke Co. put down two shafts nearly 600 ft. deep near Brownsville, Pa., about 1909 that presented some interesting cost data. The main shaft is elliptical in shape and 13 × 28 ft. in the clear, the inside circumference being 69 ft. with a clear opening of 310 sq. ft. The airshaft is the same shape, 14 × 34 ft. on its main center lines, measures 81 ft. around the circumference and has a clear opening of 390 sq. ft.

All concrete for lining the shafts is composed of one part Portland cement; two parts clean, sharp, river sand, and five parts of stone crushed to pass through a 1½-in. ring. About 50 per cent of the stone used for concrete was obtained from the materials excavated from the shafts. About 30 per cent was shipped in, crushed ready for use, while about 20 per cent was obtained from a quarry on the grounds.

The average amount of concrete to each batch was ½ cu. yd. Through solid strata the proportion of the mixture is one part cement, two parts sand, and five parts crushed stone, while through softer strata the amount of cement was increased 50 per cent, the other ingredients remaining the same. The minimum thickness of the concrete lining wall is 12 in. In soft strata the concrete is as much as 33 in. in thickness, as no voids were left between the rock and lining, all such being carefully filled with concrete.

During the excavating of the shafts, blasting was done within 10 ft. of the concrete, but at no time did this blasting have any appreciable effect upon the concrete lining. Sixty-per-cent gelatine was used in the blasting, and the maximum charges were from 150 to 200 lb. per shot. The holes in the "cut," or sump, were usually drilled to a depth of 12 ft., while those in the side rounds were 10 ft.

The sinking and concreting were carried on separately. The work was pushed continuously from 12 o'clock Sunday nights until 12 o'clock the following Saturday nights. No Sunday work was done, except in rare cases, other than that necessary for pump operation. All men employed in the shafts worked 8-hr. shifts. No forms were removed under 72 hr., allowing ample time for the concrete to set. The average time for completing one 50-ft. section of the concrete lining wall occupied 6 days, during which operation the services of 9 men at the top and 4 men on the bottom platform were required. In sinking, 5 men were used on top and an average of 12 men on the bottom. An auxiliary hoisting engine was used in one compartment for the handling of the steel forms; the operation of placing and removing these forms required the services of 4 men at the top of the shaft and 6 men on the bottom.

A special feature of the work was the construction of a 4-in curtain wall in the airshaft. The reinforcing in this wall consisted of ½-in. round steel frames, 5 ft. high by 13 ft. 6 in. long, stiffened with two additional ½-in. round steel placed equal distance from the ends. Around the bars forming the frame, No. 10 gauge wire netting was laced, this netting having a 2-in. mesh. The concrete used in this curtain wall was made of one part of cement, and two parts of clean coarse river sand.

In the excavating of the shafts a record was kept of the muck taken out and of the materials entering into the work. From the main hoisting shaft 21,168 buckets of muck were taken, while the ventilating shaft gave 28,442 buckets, approximating 50,000 cu. yd. of loose excavation through earth, fireclay, shale, slate, sandstone, limestone, and coal. The increase of the actual muck excavated, after the blasting, over the calculated yardage in the solid shows 135.2 per cent for all the materials passed through, or 35.2 per cent more than double the amount in the solid.

There were used in the construction of the concrete lining in the shafts, 8217 barrels of cement, 4410 tons crushed stone and 2528 tons sand, made up into 12,951 batches of concrete, containing approximately 6500 cu. yd.

Exclusive of the archways at the bottom landings, 539 vertical feet of concrete lining wall was placed in the hoisting shaft. The time to complete the same covered a period of 46 weeks from the start of the work of excavating, showing a progress of 48 ft. per month.

Omitting archways at the bottom landing, 561 vertical feet of concrete lining was placed in the ventilating shaft. The time consumed in completing same covered a period of 62 weeks from the commencement of the work of excavating; a progress of 37.2 ft. per month, for the sinking, timbering, and placing of the concrete lining. Taking into consideration that this shaft has an area of 80 sq. ft. more than the hoisting shaft the progress of the work averaged about the same as at the other shaft.

The first cost of concrete-lined shafts over that of timber lined is about one-third more, which amount would probably be spent upon the first renewal of timbers and from the average run of timber now on the market this would likely be necessary in about 10 yr. In a concrete-lined shaft, the lining being indestructible, the only renewals required would be replacing of the buntons and guide rails from time to time.

COMPARATIVE QUANTITIES IN THE CONCRETE LINING FOR THE VERTICAL FOOT OF SHAFT

Calculated Yardage		Actual Yardage of Material Placed in Work	Increase of Actual Yardage over the Yardage Calculated
Thickness, Inches	Cubic Yards	Cubic Yards	Percentage
HOISTING SHAFT			
12	2.66	4.6	73
15	3.36	5.0	49
18	4.08	6.4	57
24	5.56	9.0	62
VENTILATING SHAFT			
12	3.10	5.40	74
15	3.92	5.62	43
18	4.74	5.84	23
24	6.45	9.50	47

The following summary gives the principal figures in regard to the work on both shafts:

HOIST SHAFT

Total depth, feet	565
Total number of weeks worked	46
Average thickness of concrete lining, inches	16
Average number of batches per 5-ft. form	56
Average cubic yards concrete per 5-ft. form	28
Average depth of lining placed per week, feet	12.3
Average depth of sinking and lining placed per month, feet	48

VENTILATING SHAFT

Total depth, feet	591.2
Total number of weeks worked	62
Average depth sunk per week, feet	9.5
Average depth of lining placed per week, feet	9.1
Average depth of sinking and lining placed per month, feet	37.2
Size of shaft, inside concrete lining wall	14 ft.×34 ft.
Area, square feet	390
Average thickness of concrete lining, inches	17
Average number of batches per 5-ft. panel	66
Average cubic yards of concrete per 5-ft. panel	33

Some interesting cost figures were obtained in the sinking of a 570-ft. inclined metal mining shaft in the Poverty Gulch region of Colorado about 1910. The shaft was 12 × 7.5 ft. in the clear and after it had been sunk a short distance two skips were added to the sinking equipment which weighed 550 lb. each, held 10 cu. ft. and cost $70 a piece, including a water valve in the bottom which cost $10. The sinking equipment consisted of the following:

Six Little Giant drills, 2¼-in. diameter	$900.00
Six columns and arms up to 8 ft	240.00
Six sets drill steel, each 15.4 lb., at $14.40	86.40
Buffalo exhaust fan, diameter outlet 24½ in., price with bed and countershaft	700.00
460 ft. of 24-in. pipe, at $54 per 100 ft	248.40
1200 ft. of 12-in. pipe, at $27 per 100 ft	324.00
Ten Leyner No. 5 stoppers, $135	1350.00
Drill steel, 10 sets, at $15	150.00
Sixteen ore cars, at $45	720.00
Compressor, motor and pipe, freight	2689.14
Freight on 19,553 lb., at $0.55 per 100	107.54
Total	$7515.48

In the development work two drills were required. The drills are 2¼ in. in diameter and use air at 90 lb. pressure per square inch at drill. According to the catalog specifications, each drill will need 67.2 cu. ft. of air per minute. The factor to determine a compressor capacity for four drills at 10,000 ft. altitude is 4.49; hence, 301.5 cu. ft. of air per minute will be required, but deducting 5 per cent for leakage and allowing the compressor a volumetric efficiency of 80 per cent, the total air required is nearly 400 cu. ft. per minute.

E. A. Rix allows 20 hp. for every 100 cu. ft. of cylinder displacement, to compress air to 90 or 95 lb. gauge pressure at sea level. Although 20 hp. is higher than the value given by Peele for the theoretical horsepower required, and figuring efficiency the figures would then be below 20; but, since compressors are usually purchased for excess power to supply possible additional uses, and the use of 20 hp. would only add a small percentage on the safe side, the power necessary for four drills is taken at 80 hp., and it was decided to purchase a two-stage air compressor 18×11 in. diameter, 12-in. stroke, 125 r.p.m., with a capacity of 440 cu. ft. at 10,000 ft. elevation.

This sized compressor gives a reserve of 21 per cent, and costs with freight from Denver to Cripple Creek $1903. The motor for the compressor is 80 hp., 900 r.p.m., 440 volts, and costs delivered $710.84.

The following calculations give the power consumed during development by two piston drills: The catalog multiplier is 2.39 and as each drill will need 67.2 cu. ft. of air, $67.2 \times 2.39 = 159$. Allowing for air loss in pipe line and efficiency of compressor, 210 cu. ft. are necessary, or 42 hp. per minute.

Each stope drill requires 25 cu. ft. of air per minute and the factor for seven drills is 7.55; therefore, $25 \times 7 = 189$ cu. ft., to which 63 cu. ft. is added to allow for loss and efficiency, and this is equivalent to 50.4 hp.

The power for two stope drills is 25×2.5 (multiplier) $= 62.5$ cu. ft. of air, and if to this be added 20.7 cu. ft. for pipe loss and efficiency, the power required is 16.68 hp. The diameter of the pipe needed for carrying air 800 ft. is 3 in. and will cost at Cripple Creek $75.30. The total cost of compressor, motor, and pipe is $2,689.14.

In sinking, 18, 4-ft. holes were used. Drilling was done at the rate of 39 ft. in 8 hr. per drill or 72 ft. in 7.4 hr. The rate of advance was 3 ft. per round which equals $3 \times 12 \times 7.5 = 270$ cu. ft. solid material which divided by 12.4 gives 21.8 tons per round and multiplying this by 21.5 gives 469 cu. ft. or 17.4 cu. yd. of loose material. Estimating the cost of mucking on the basis of 1.2 cu. yd. per man per hour it will take two men 7¼ hr. to clean up the rock after each round, so that allowing ¾ hr. for delays, changing buckets, etc. it will be seen that one round can be drilled, fired and mucked in 16 hr., which eliminating other delays would be equal to a progress of 90 ft. per month.

For a 4-ft. hole it was found that six sticks of 40 per cent dynamite were required each weighing 0.6 lb. or 3.6 lb. per hole so that for the 18 holes 64.8 lb. were required.

The shaft was sunk 570 ft. The time required to sink the shaft was:

$$\frac{570}{3 \text{ ft. per round}} = 190 \text{ rounds, or days.}$$

The detailed cost of sinking 570 ft. of a 90-sq. ft. inclined shaft is as follows:

Two machine men, 190 shifts, at $4.50		$1,710.00
Two muckers (also top men) 190 shifts at $3		1,140.00
Two hoistmen, 190 shifts at $4.50		1,710.00
One blacksmith, 190 shifts at $4.50		855.00
One blacksmith helper, 190 shifts at $4		760.00
One foreman, 190 shifts at $4.50		855.00
One superintendent, at $175 per month, 6⅓ months		1,108.35
One timberman, 190 shifts, at $3.50		665.00
Powder, 12,312 lb., at $1.27*		1,563.62
Fuse, 190 rounds, 7-ft. lengths, 23,940 ft., at $0.0035*		83.79
Caps, 3420, at $0.007*		23.94
Depreciation on steel		14.40
Operation compressor plant (power), 336 hp. hours per day:		
Installation charge	$20.50	
40,000 kw. hr., at $0.013	520.00	
7700 kw. hr., at $0.005	38.50	579.00
Timber, 95,440 ft., at $20 per thousand		1,908.80
Electric power, for hoist, average 6.23 hp. hr. per hour, 49.84 hp. hr. per day, 9470 kw. hr. (190 days) at $0.013		123.11
Coal for blacksmith, 28.75 tons, at $20.75		243.20
Candles, 950, at $0.0145		13.78
Rails (30-lb.), 570 ft., 5.08 tons, at $50		254.00
Cost for 570 ft		$14,207.55
Or cost per ft		$24.93

* These low costs of powder are those of the Portland Gold Mining Co. and include freight and unloading charges.

The following costs of sinking a mine shaft through andesite at the Esperanza Mine at El Oro, Mexico, are given by W. E. Hindry in the *Mining and Scientific Press* in 1910. The shaft was a three-compartment vertical shaft, having two 5 × 5 ft. hoisting compartments and a 5 × 7 ft. pump and ladderway. The timbering was 10 × 10 in. with 2-in. lagging; sills 5 ft. center to center, and 6 posts per set. The total depth of the shaft was 679 ft., of which 101 ft. were sunk by windlass and hand work, and 578 ft. by steam hoist and machine drills. The work was done in 1899 and the prices of materials and wages were as follows:

Materials	Prices
Timber per M ft. B.M	$13.58
Wood per cord	3.15
Coal per ton	7.27
Powder, 60 per cent, per pound	$0.14\frac{2}{3}$
Fuse per foot	0.0055
Caps, each	0.0058
Candles, each	0.0194
Labor	
Superintendent, per 24 hr	$4.850
Shaft men, foreign, per 8-hr. shift	3.220
Shaft men, native, per 8-hr. shift	0.528
Top men, per 8-hr. shift	0.422
Fireman, per 8-hr. shift	0.485
Hoistmen, per 8-hr. shift	0.970
Blacksmiths, per 8-hr. shift	1.455

The cost of excavation was as follows:

Labor	Per Linear Foot
Superintendence	$2.529
Shaftmen, foreign	3.510
Shaftmen, native	7.043
Top men	0.578
Blacksmiths	0.718
Firemen	0.317
Hoistmen	0.894
Miscellaneous	0.936
Total	$16.525
Materials	
Timber	$3.961
Wood, fuel	3.781
Coal	0.179
Powder	2.853
Fuse	0.014
Caps	0.294
Candles	0.223
Oil, grease, etc	0.025
Miscellaneous	0.051
Total	$11.381
Grand total	$27.906

The above costs are converted from Mexican money assuming the peso to have a value of 48½c.

An exhaustive study of shaft sinking costs in the Michigan region was prepared about 1910. The figures and computations were made upon the assumption that the shaft would be 6 × 16 ft. within timbers and reach a vertical depth of 1000 ft. For a vertical shaft, its total length would be 1000 ft.; if inclined at an angle of 45 deg. to follow the dip of the formation, its total length would be 1400 ft. to reach a total vertical depth of 1000 ft. A contract price of $40 per foot for sinking would apply only if the shaft was put down in the jasper formation. If diorite was encountered the contract price for sinking alone would be between $50 and $55 per foot, while all other items would remain the same.

A slight difference in cost of timber would appear in the amount of timber used in a vertical or inclined shaft, as the latter requires 10 × 10 in. stringers, while the former would take 6 × 8 in. skip runners, but this has not been taken into account.

The maximum flow of water to be handled (800 gal. per min.) is probably somewhat high.

Details of Sinking Contract, Shaft, etc.

	Maximum	Minimum
Contract price for sinking, $40 per foot, includes drilling, blasting, powder, caps, fuse, etc.		
For a vertical shaft, 1000 ft.	....	$40,000
For an inclined shaft, 1400 ft.	$56,000	
Computing all work for a shaft 6×16 ft. within timbers, three compartments, shaft sets, 5-ft. centers; and assuming 40 ft. per month as average sinking.		
1000 ft. would take 25 months, allow 26 months.		
1400 ft. would take 35 months, allow 36 months.		
Allowing 25 working days per month, 300 per year		
1000 ft. of sinking would take 650 days.		
1400 ft. of sinking would take 900 days.		
Sinking contract would be worked on three 8-hr. shifts. All other labor, one or two 10-hr. shifts.		
Cutting pump station 10×15×20 ft. at a depth of 500 ft.	350	350
A maximum flow of 800 gal. per min. when bottom of shaft is approached is used as the basis of pumping expenses.		
Total	$56,350	$40,350

DETAILS OF LABOR

	Maximum	Minimum
Blacksmith, at $2.25 per day; helper at $1.65 per day; $3.90 per day for 900 days	$3,510	
$3.90 per day for 650 days	....	$2,535
Two landers at $1.70 per day for 900 days	3,060	
Two landers at $1.70 per day for 650 days	....	2,210
Two timbermen at $1.75 per day. Assuming one set of timber can be cut and framed in 1 day by two men.		
1400 ft. of shaft, 280 sets, 280 days	980	
1000 ft. of shaft, 200 sets, 200 days	....	700
Two brakemen at $2.20 per day, for 900 days	3,960	
Two brakemen at $2.20 per day, for 650 days	....	2,860
Two firemen at $1.70 per day for 900 days	3,060	
Two firemen at $1.70 per day for 650 days	....	2,210
Allow one-fourth of mining captain's time, $25 per month, for 36 months	900	
Allow one-fourth of mining captain's time, $25 per month, for 26 months	....	650
Surveyor and helpers, allow $20 per month, for 36 months	720	
Surveyor and helpers, allow $20 per month, for 26 months	....	520
Total	$16,190	$11,685

DETAILS OF TIMBER

	Board Measure, Feet	Maximum	Minimum
2 Plates, 12×12 in.×18 ft. contain	432		
2 End pieces, 12×12 in.×6 ft. contain	144		
4 Corner posts, 12×12 in.×4 ft. contain	192		
2 Dividings, 10×12 in.×6 ft. 4 in. contain	126⅔		
4 Stringers, 10×10 in.×5 ft. contain	166⅔		
4 Center posts, 10×10 in.×4 ft. contain	133⅓		
Sheathing, 3 in.×5 ft.×44 ft. contain	660		
Boards, 1 in.×5 ft.×6 ft. 4 in. contain	31⅔		
Total amount of timber for 1 set	1,886⅓		
Total amount of timber in 280 sets	535,173		
Total amount of timber in 200 sets	377,267		
At $14 per thousand for hemlock timber:			
for 280 sets, $7,492.42, allow	....	$7,500	
for 200 sets, $5,181.74, allow	....	....	$5,200
Ladders, 17½c. per ft., 1400 ft.	....	245	
Ladders, 17½c. per ft., 1000 ft.	....	....	175
Total	....	$7,745	$5,375

DETAILS OF RAILS, PIPE, TIE-RODS, ETC.

	Maximum	Minimum
45-lb. rails for double skip road, allow 1500 ft., making 6000 linear ft. or 2000 yds., 45 tons at $25	$1,125	
200 pairs fish-plates, at 21c	42	
500 lb. rail spikes, at 3c	15	
1120 $1\frac{1}{8}$ in.×6 ft. 6 in. round iron tie-rods, at 2c, $487.46 allow	500	
2240 nuts and washers, $278.60, allow	300	
All of the above computed for 1400 ft. of shaft.		
For a vertical shaft of 1000 ft. depth, rails, fish-plates and spikes would not be used.		
Tie-rods, nuts, and washers for 1000 ft. of shaft, allow	.....	$550
1400 ft. of 10-in. water-column pipe	1,820	
1000 ft. of 10-in. water-column pipe	....	1,300
Drill steel	40	30
2-in. steam pipe; allow 250 ft. in excess of length of shaft. For 1650 ft	150	
For 1250 ft	....	110
4-in. air pipe; allow 250 ft. in excess of length of shaft. For 1650 ft	750	
For 1250 ft	....	575
Allow a maximum of 5 tons of coal per day for 26 months	....	2,280
For 36 months	3,285	
3000 lb. 60d. spikes at 10c per pound for 1400 ft	300	
200 lb. 10d. nails at 10c. per pound, for 1400 ft	20	
2160 lb. 60d. spikes at 10c. per pound, for 1000 ft	....	216
145 lb. 10d. nails at 10c. per pound, for 1000 ft	....	15
Total	$8,347	$5,081

DETAILS OF PLANT

	Maximum	Minimum
Shaft house and pockets	$14,800	$6,750
Engine house	9,625	5,000
Boiler house	4,200	3,500
Powder house	250	100
Coal trestle	3,000	2,500
Boilers (4)	11,660	10,500
Hoisting engine	15,000	12,000
Compressor	11,500	10,625
Skips (2)	1,000	300
Two No. 3 Rand drills, complete	375	375
Auxiliary pump at 500-foot depth	6,000	5,000
Sinking pump	1,000	900
Temporary equipment at start, small hoist, bucket, rope, tripod, etc., allow	1,500	1,000
Hoisting cable, $1\frac{3}{8}$-in. diameter	900	700
Incidentals—Teaming, pipe fittings, air hose, picks, shovels, hammers, wrenches, timber cutter's tools, axes, saws, oil, waste, candles, temporary bell signal system, etc	5,000	4,500
Total	$85,810	$63;750

RECAPITULATION

	Inclined Shaft, 1400 ft.	Vertical Shaft, 1000 ft.
Sinking contract	$56,350	$40,350
Blacksmithing	3,510	2,535
Landers	3,600	2,210
Timber cutters	980	700
Brakemén	3,960	2,860
Firemen	3,060	2,210
Captain and surveyors	1,620	1,170
Timber and ladders	7,745	5,375
Rails, fish-plates, spikes	1,182	
Air and steam pipes and water column	2,720	1,985
Tie-rods, nuts, and washers	800	555
Nails and spikes	320	231
Coal	3,285	2,280
Drill steel	40	30
Total	$88,632	$62,491

Total plant maximum	$85,810
Total plant, minimum	63,750
Inclined shaft with minimum plant	152,382
Inclined shaft with maximum plant	174,442
Vertical shaft with minimum plant	126,241
Vertical shaft with maximum plant	148,300

In spite of very difficult sinking problems, as compared with conditions in this country, shaft-sinking costs in Europe have been substantially less than in this country. Shafts in the Taff and Rhonddha valleys in England, which are circular and from 17 to 21 ft. in diameter were, about 1910, sunk at a total cost of $30 to $50 per foot including the lining. In the north of England the shafts are somewhat larger, varying from 20 to 24 ft. in diameter and are usually lined with steel tubbing. Some excellent speed records have been made at these shafts, at the Sherwood colliery for instance a shaft was sunk 858 ft. in 21 weeks, an average of 40.8 ft. per week.

In Belgium brick lining was used almost exclusively at one time, though the use of reinforced concrete is becoming more general. In 1910 sinking costs there were about $60 to $75 per meter and the lining $5 to $6 additional.

It is frequently necessary to put down small prospect shafts for depths up to 100 ft. and the approximate cost of such equipment as of 1907 was as follows:

One 25-hp. vertical boiler	$300
One 5×5 in. Bacon type of hoist	350
One $2\frac{3}{4}$-in. steam drill	180
One 7-ft. bucket	30
One 18-in. sheave and bearings	20
200 ft. of $\frac{1}{2}$-in. wire rope	13
Lumber for head-frame, hauling, and labor	107
Total	$1000

The items for blacksmith shop, bunk house, cook house, etc., must usually be added, but this amount will, of course, depend upon the size of the prospect and if they are needed. To provide a moderate equipment and to allow a certain amount of working capital another $1000 should probably be provided.

Shaft linings.—An interesting example of the costs of a concrete lined metal mining shaft is that of the Brier Hill shaft at Vulcan, Mich., sunk about 1909. This shaft is circular, 14 ft. in diameter and 850 ft. deep. Steel sets made up of 8-in. channels, 13¾ lb. per foot, placed on edge and spaced 10 ft. 8 in. on centers were used. Between the sets there are studdles of steel channels to which the wooden runners for the cage are bolted. The ladders are built of steel and the ladderway and skip compartment are lined with galvanized corrugated sheet steel.

The line of the shaft was through some old workings so that it was possible to make preliminary openings throughout the entire length from these different levels; this opening was made 6 × 8 ft. and a 20 × 20-ft. square shaft sunk from the surface to connect with this. Concreting was started at a depth of 79 ft. from the surface.

The concrete used was mixed in the proportion of one cement, three sand and six stone and the average thickness of the lining was 18 in. with a minimum of 6 in. Measurements of the actual excavation, taken every 3 ft., showed an average thickness of the lining of 19 in. which was equivalent to a little less than 3 cu. yd. of concrete to a vertical foot of shaft. Forms made of ⅛-in. sheet steel were used, there being two sets of forms, each 5 ft. 4 in. high and each set consisting of four segments.

After the excavation of the shaft to the proper size was carried downward for a distance sufficient to put in three or four sets of steel, a platform or "curb" is laid on the outside edge of the shaft all around near the bottom at the proper distance from the concrete above to allow for the number of sets proposed. One round of forms is then set upon this platform. Empty boxes are put in at the bottom to form "chute holes" for putting concrete into the form below at the proper time and a platform of 3-in. plank is laid over the top of the form. The concrete is then lowered in the kibble, dumped on the platform, slushed off and tamped in around the outside of the forms completely filling the space between the forms and the rock.

A set of steel is then lowered, laid in the hitches left in the concrete and cemented in. The round of forms is lowered upon this new set, expanded to the proper size and the second round of forms is set up and bolted to the first so that this time concrete is deposited for a height of 10 ft. 8 in. This can be done in an eight-hour shift, and 12 hr. are sufficient for the concrete to set so that it is possible to concrete 10 ft. 8 in. of shaft and put in the steel work in less than two days. As the space between the steel sets is 10 ft. 8 in. and in starting only half this amount of concrete is put in before a set is laid, the work is joined to the older concrete above by one round of forms of 5 ft. 4 in. This is filled through the spaces or "chute holes" left by the empty boxes previously mentioned.

No attempt was made to make a record of speed. The best work was done in September and October, 1909, which resulted in the excavating (enlarging the original opening) concreting and putting in the steel of $138^2/_3$ ft. of shaft or an average of $69^1/_3$ ft. per month. At that time the men were working two shifts a day or 11 shifts a week of 10 hr. each. Six men constituted the regular shift in the shaft either for excavating or constructing. In addition to the shaft work proper three stations have been cut out and concreted.

In keeping the record of the cost, the shaft has been divided into three parts: from surface to ledge, 62 ft.; from top of ledge to 7th level, 549.5 ft.; and from the 7th level down, ultimately, about 238.5 ft.

COST OF CONSTRUCTION

	Surface to Ledge, 62 ft.	Ledge to 7th Level, 549.5 ft
Preliminary excavation	$13.07	$18.46
Final excavation	18.19	15.10
Steel shaft frames	7 90	7.90
Steel forms	0.83	0.83
Temporary surface structures and equipment	10.18	10.18
Construction	56.29	25.26
Estimated charge for compressed air	1.00	1 00
Total per foot	$107.46	$78.73
Estimated salvage on shaft timbers	0 50	0.50
Estimated salvage on temporary surface structures and equipment	2.95	2.95
Net total per foot	$104.01	$76.28

The item "construction" includes the handling of the steel forms, depositing concrete, and setting and erecting the steel frames. These figures include estimates of power and every charge except for general management and engineering. The cost of the first 85 ft. below the 7th level was $87.19 per foot. The cost of this circular concrete-lined shaft is about the same as a rectangular shaft of the same capacity with steel framing. The small additional cost over a rectangular shaft with timber framing is abundantly justified by the increased safety and permanence.

One of the earliest concrete shaft linings in this country was put in by the River Coal Co. near Bridgeport, Pa., about 1905. The shaft lining measures 23 ft. on its major axis parallel to the railroad tracks and 15 ft. on the minor axis, inside measurements, the thickness of its concrete walls varying with the depth below the surface. The following table gives approximately the cost of the construction both per foot of depth and per cubic yard:

	Per Foot of Depth		Per Cubic Yard	
Stone		$5.90		$1.00
Sand		1.77		0.30
Cement		19.18		3.25
Labor:				
Mixing	$3.83		$0.65	
Placing	3.40		0.58	
Firemen and pumpmen	2.19		0.37	
	———	9.42	———	1.60
Forms:				
Lumber, $13 per thousand	$1.83		$0.31	
Making, $21 per thousand	2.95		0.50	
Placing	4.82		0.81	
	———	9.61	———	1.62
Platform for starting upper section		0.92		0.16
Superintendence		3.03		0.51
Plant		0.28		0.05
Oil		0.21		0.04
Sundry		1.06		0.18
Tools		0.24		0.04
		$51.62		$8.75

A novel shaft lining in the form of concrete blocks was used in a shaft in Belgium in 1912, the estimated cost of which was $8.65 per running foot of shaft. This lining was found to be equal in strength to a 32-in. masonry lining the cost of which would have been $13.50 per foot.

The shaft was 133 ft. deep and 13 ft. 4 in. in diameter, the excavation being about 17 ft. in diameter. The concrete blocks were 30 in. high with a minimum thickness of 3.2 in. and 14 were required to make the circumference of the shaft. They were set with joints staggered and the successive rings of blocks were joined by 12 × 0.6-in. dowels there being two of these to each block. Concrete with additional reinforcing was filled in between the back of the blocks and the excavation.

The increased cost of timber and inferior product being offered in recent years, has tended to cause the use of other materials for shaft linings, especially cement which has come into rapid favor because it has not increased in cost so much as the timber and because it gives a more permanent job and is fireproof and more or less watertight.

The average timber lining lasts from 12 to 15 yr. and in 6 to 8 yr. it becomes necessary to replace individual timbers and sections of lining, causing temporary shutdowns. In the life of a mine of any considerable size, say 30 yr., it will be necessary to re-timber the whole shaft at least once besides making many minor repairs. The cost of re-timbering a shaft (owing to the removal of the old lining, and the increasing price of timber and labor) will be much higher than the cost of the original lining; to this direct cost must be added the loss of income from the mine during the time of repairs.

The chief advantages of timber lining are its lower first cost, greater speed in placing it and the fact that timber is better adapted to the rectangular or square form of shaft. Timber lining can be placed in about one-third the time that concrete can which at the average rate of sinking would amount to a saving in time of about 13 days for each 100 ft. of shaft.

A comparison of a hoisting shaft and an air shaft of the usual design and with timber lining, with corresponding elliptical and circular concrete-lined shafts, will present an example which will closely approximate the conditions obtained in the mining region of western Pennsylvania; for other localities, the local conditions governing the cost can be substituted. The figures are as of 1905.

Two shafts of the United States Coal & Coke Co. (a subsidiary company of the United States Steel Corporation) at Tug river, West Virginia, were sunk through one seam of coal at 100 ft. in depth, and continued through a second seam at 175 ft. depth. The air shaft was 14 ft. 2 in. on the short axis, by 20 ft. on the long axis. The shaft was lined for a depth of 45 ft. in order to shut off the surface water. The concrete was 12 in. thick.

The main hoisting shaft was 17 ft. 4 in. on the short axis by 33 ft. on the long axis, the concrete being 12 in. thick at the sides and 18 in. thick at the ends. It was a four-compartment shaft, including a downcast airway, two hoisting-ways and a pipe-way. It was concreted throughout on account of the downcast air-way and the desire to shut off all the water

COSTS OF TWO SHAFTS FOR THE U. S. C. & C. CO. AT TUG RIVER, 1905

Main Shaft

	Elliptical	Rectangular	
		Timber-lined	Concrete-lined
Concrete, per foot of depth.	4½ cu. yd.	....	5.9 cu. yd.
Excavation, per foot of depth.	13.5 cu. yd.	12 cu. yd.	15 cu. yd.
Timber, per foot of depth.	90 ft. B.M.	500 ft. B.M.	80 ft. B.M.

Cost per Foot of Depth

Concrete, $9.00 per cu. yd.	$40.50	....	$53.10
Excavation, 5.50 per cu. yd.	74.25	$66.00	82.50
Timber, 60.00 per M....	5.40	30.00	4.80
Total cost of main shaft...	$120.15	$96.00	$140.40

Air Shaft

	Elliptical	Rectangular	
		Timber-lined	Concrete-lined
Concrete, per foot of depth.	3 cu. yd.	....	4.3 cu. yd.
Excavation, per foot of depth.	8 cu. yd.	8 cu. yd.	9.9 cu. yd.
Timber, per foot of depth.	70 ft. B.M.	450 ft. B.M.	70 ft. B.M.

Cost per Foot of Depth

Concrete, $10.00 per cu. yd.	$30.00	....	$43.00
Excavation, 6.00 per cu. yd.	48.00	$48.00	59.40
Timber, 61.00 per cu. yd.	4.20	27.00	4.20
Total cost of air shaft.....	$82.20	$75.00	$106.60

in the rocks; this was successfully done. The cross buntons were held by cast-iron boxes built into the concrete, but these boxes were probably unnecessary. In the hoisting shaft an average progress of 16 ft. per week was made, 20 ft. being the maximum. The total excavation in this case amounted to 21 cu. yd. per ft. of depth.

A paddle concrete-mixer was placed at the head of the shaft, and the concrete was lowered directly from the discharge spout of the mixer without further handling. While one bucket was lowering, the other was filling; thus no time was lost in delivering concrete to the placing gang. All form work was done at night, and the concreting on the day shift. The forms were built in 5-ft. vertical sections and were used repeatedly.

The cost of labor, mixing, placing forms, lumber, carpenters, hoisting engineers, oil, waste supplies, sundries and superintendence amounted to about $4 to $5 per yd. depending upon the size of the shaft and the thickness of the concrete. To this must be added the cost of materials.

The table, given herewith, illustrates the relative cost of elliptical concrete-lined shafts, and also the usual rectangular shaft lined with concrete and with timber. It illustrates the great economy of the more permanent concrete shaft.

An example of an air and main shaft, each 200 ft. in depth, would represent an outlay of $34,200 for a timber-lined rectangular shaft; the equivalent elliptical concrete-lined shaft would cost $40,440, both figures including all materials. The difference due to the increased cost of $6240 is more than offset by the fact that it would take over $15,000 to re-timber both shafts, not to mention the loss of time and repairs.

Rates of progress.—The best record in shaft sinking up to 1907 was made at the Dixon-Pocahontas Mine in West Virginia. This shaft at Olmstead, W. V., was completed in nine working weeks, although not in nine weeks' continuous work, as an explosion in which four men were killed disorganized the forces and interrupted the work. This work was practically free from water except the last 50 ft. when a No. 7 Cameron sinking pump was required, but after the sump was completed a No. 10 Cameron was installed for temporary use.

The shaft is 14 × 22 ft. and 180 ft. deep to coal and was

9 weeks in sinking. It is timbered with 8 × 10 in. wall plates and 6 × 10 in. buntons and lagged with 2-in. plank. The timbers have bearing sets about 30 ft. apart while the other sets are spaced on 5-ft. centers.

The following is the equipment installed: Two 50-hp. Erie City Economy and one 40-hp. Atlas internally fired boilers, a six-drill compressor, a 10 × 12 in. Exeter hoist having a 4-ft. drum and 7/8-in. plow-steel cable, a 70-in. fan coupled to a 10-hp. engine, and a blacksmith shop outfit.

The work was carried down with three 8-hr. shifts for shaft men and two 12-hr shifts for outside men.

The number of men and their wages was as follows: 1 shift foreman, $3; 3 drill runners, each $2.50; 3 helpers, each, $2.25; 8 muckers, each, $2; 1 blacksmith, $2.75; 1 hoister, $2; 1 compressor man, $2; 1 carpenter, $2.50; 2 helpers, each $1.75.

The coal for this 9 weeks' work at $1.25 per ton cost $460. The cost of installing the sinking plant was about $1000 while that of dynamite was $700. Hence, exclusive of machinery, the risk from accidents and the cost of moving the excavated material after dumping the buckets, we may get a close estimate of the cost of this work as follows:

Installing plant	$1000
Coal	460
Dynamite	700
Labor per day, $85 for 54 days	4590
Timber for shaft at $20 per M, 60 M	1200
Miscellaneous expenses	1000
Total	$8950

This is practically an expense of $9000 for 180 ft. of 14 × 22 ft. shaft, a net cost of $50 per foot.

The speed of sinking will be governed by the quality of the rock, size and shape of the shaft, amount of water present, class of labor available and the efficiency of the plant. The fastest sinking on record up to 1909 was on the Rand in Transvaal, South Africa. These shafts are sunk 9 × 26 ft. in the rock and the work is carried in three, 8-hr. shifts per day, seven days in the week. The labor is cheap and can be used under conditions that the white man will not work under so the shafts are filled up with every man that is possible to use. The records made at some of these shafts are given in

PROGRESS IN SINKING SOUTH AFRICAN SHAFTS

	Kind of Rock	Size	Depths between which Average Progress is Figured, Feet	Cost Per Foot	Average Progress per Month
Cinderella Deep	Quartzite or dike	9′ 8″×33′ 6″	0 to 3900	$104.45	90.0
Angelo Deep	Quartzite or dike	9′ 4″×24′ 4″	0 to 239	81.80	119.5
Simmer West	Quartzite or dike	9′ 4″×28′ 4″	0 to 120	99.70	120.0
Knights Central	Quartzite or dike	9′ 4″×29′ 4″	0 to 264	80.90	132.0
Catlin	Quartzitic sandstone	9′ 8″×29′ 8″	0 to 1839	....	145.0
Howard	Quartzitic sandstone	9′ 8″×29′ 8″	0 to 1767	77.86	146.0
Rudd	Quartzitic sandstone	9′ 8″×29′ 8″	0 to 1326	....	175.0
Wilmer	Quartzitic sandstone	9′ 8″×29′ 8″	0 to 1504	....	187.0
Nigel Deep (incline)	....	7′×14′	511 to 1206	....	173.7
New Kleinfontein Co	....	....	858 total	....	171.6

Best monthly records: Howard Deep, 203 feet; New Kleinfontein, 213.5.

PROGRESS IN SINKING AMERICAN SHAFTS

	Kind of Rock	Size	Depth, Feet	Cost per Foot	Average Progress per Month
Lincoln Gold Mine, Cal	Greenstone and slate	9′ 8″×18′ 8″	740	$37.92	61.0
Federal Lead Co., S. E. Missouri	Magnesian limestone	13′ 8″×23′ 8″	418	....	69.7
Tamarack, Mich	Trap	10′ 6″×31′ 0″	4580	99.36	70.5
United States Coal and Coke Co., Tug River, W. Va.	Sandstone (hard gray)	19′ 4″×33′ 0″ elliptical	170	75.00 (unlined)	69.3
Selinsgrove, W. Va	Sandstone and slate	14′ 9″×30′ 4″	202	....	86.5
Old Dominion Copper Mining and Smelting Co.	....	9′ 4″×28′ 4″	1025	87.00	41.0
M. R. C. C. and C. Co., Fayette City, Pa	Slate	10′×16′ 4″	207	54.00	102.0
Struthers Coal and Coke Co., New Salem, Pa.	Slate and limestone	12′ 6″×26′	529		70.0
New York City Aqueduct, No. 1 Rondout Siphon	Shale	17′ circular	166 to 593	....	107.0

Best monthly record: No. 1 Rondout, 138 feet.

the accompanying table, in which it will be noted that the average progress is about 135 ft. per month with a record sinking of 213 ft. in one month.

Progress in this country up to 1909 was under normal conditions at the rate of 60 to 80 ft. per month for the average timbered shaft, though the advent of improved drills since that time has much increased this rate. Faster records are made in the Middle West where the soft shales are easy to drill and shoot. Speed of 7 ft. per day was made at a shaft near Atchison, Kan., in 1902 which was regarded as rapid sinking at that time though it is not known how long this was maintained. In 1909 a record of 138 ft. was made in one month on a 17-ft. circular shaft through rock that was quite hard but broke readily on the New York Aqueduct which was fast sinking at that time, if not a record. The particulars of this shaft will be found in the accompanying table.

The accompanying tables give the dimensions of a number of shafts and the progress made in them. Wherever obtainable, the nature of the rock penetrated and the cost per foot is given. The figures were obtained from various articles in the technical papers, and from the proceedings of various mining institutes. Some of the South African data were taken from the "Deep Level Mines of the Rand," by G. A. Denny, 1902.

Most of the shafts sunk on the Witwatersrand, South Africa, have five compartments. Some are sunk on the dip of the ledge which varies from 70 to 35 deg. in different mines, the tendency being for the steeper dips to flatten to about 35 deg. as the mines get deeper. But most of the shafts are vertical. Some new vertical shafts that have been started for working the deep levels have seven compartments.

Most of the sinking is in quartzite and Leslie Simson, a graduate of the University of California, held the world's record in 1907, for sinking a vertical shaft 7 $\times$ 28 ft. in quartzite at the rate of 203 ft. in one month. The average rate of sinking eight shafts on the Consolidated Gold Fields mines, 7 $\times$ 28 ft., to depths varying from 1300 ft. to 3700 ft. was kept at about 100 ft. per month and the average cost $134 per foot. Two of these shafts, over 3000 ft. deep, were each sunk at the rate of 152 ft. per month for six consecutive months.

The time on the work was subdivided as follows:

	Hours
Hand drilling with Kaffirs	3.07
Winding rock	3.26
Winding men and tools	1.17
Blasting	0.50
	8 00

The number of hand-drilled holes per shift is 20 to 21; dynamite used per foot, 16 to 18 lb.; coal burned per shift, 2400 to 2800 lb.; buckets of rock hoisted, 34 to 36 (about 1 ton capacity).

Cost figures cover a wider range than progress figures and are harder to get. The cheapest shaft on record is the one near Atchison referred to above, the cost of which, as stated was \$7 per foot. This cost stands alone in its glory as the tabulated figures show. Mr. Henry Rawie published in *Mines and Minerals* an itemized statement of the costs of a shaft sunk in West Virginia, in 1906. These ran as follows:

HOIST SHAFT, 14×22 FT., 180 FT. DEEP

	Per Foot
Labor, sinking, and timbering	\$24.70
Plant	5.55
Superintendence	
Explosives	3.88
Coal	2.55
Timber	6.67
Miscellaneous	5.55
	\$48.90

The sinking costs of a pair of shafts sunk in Western Pennsylvania a year later were as follows:

HOIST SHAFT, 13×26 FT., 422 FT. DEEP

	Per Foot
Labor, sinking	\$51.00
Plant	2.40
Superintendence	4.35
Explosives	2.75
Coal	5.50
Oil	0.60
Freight	0.50
Miscellaneous	7.90
Total	**\$75.00**

Air-Shaft, 13×22 Ft., 383 Ft. Deep

	Per Foot
Labor, sinking	$57.50
Plant	2.40
Superintendence	4.90
Explosives	3.00
Coal	6.05
Oil	0.60
Freight	0.50
Miscellaneous	7.14
Total	$82.09

Water per minute: Hoist shaft, 50 gal.; air-shaft, 120 gal.

Costs have risen greatly in the last decade since no substantial improvements in methods or machinery have been made to offset the increase in wages. Contract prices are not generally obtainable, as most shafts are put down by private corporations, but prices, high enough to include a good profit to the contractor 8 or 10 yr. ago, would not cover his costs to-day.

Twenty-five shafts ranging in depth from 350 to over 1000 ft. for a portion of the New York Aqueduct were contracted for at prices ranging from $175 to $350 per foot.

Reports and contract forms.—Proper supervision of costs of supplies, labor and progress of work can only be obtained by keeping a detailed daily report of the sinking operations and the accompanying form, Figs. 5 and 6, show a very good method of doing this. These forms were used by the Cottonwood Coal Co., a subsidiary of the Great Northern R. R. on a 31 × 9-ft. five-compartment shaft at Lehigh, Mont. The forms are made out in triplicate and copies forwarded to the president and general manager and one kept on file at the plant. The report is self explanatory and can with modifications be adapted to almost any condition.

DAILY REPORT OF SHAFT DEVELOPMENT Lehigh MINE Thurs. DAY Feb. 5 1915

SHIFT BOSS / DATE	SHIFT # 1 Feet Sunk	Feet Drilled	Buckets Hoisted	Sets Placed	Material Used	SHIFT # 2 Feet Sunk	Feet Drilled	Buckets Hoisted	Sets Placed	Material Used	SHIFT # 3 Feet Sunk	Feet Drilled	Buckets Hoisted	Sets Placed	Material Used	TOTAL Feet Sunk	Feet Drilled	Buckets Hoisted	Sets Placed	Material Used
Sunday																				
Monday																				
Tuesday																				
Wednesday																				
Thursday	—	110	6	—	22 x 33 ft	—	—	—	1	—	1	0	56	—	—	1	110	62	1	22 x 33 ft
Friday																				
Saturday																				
Total for Week																				
Previous Weeks																				
Total for Month																				
Previous Months																				
TOTAL																				

REMARKS 1st Shift lowered part of Steel set and put 1 section of stairway on place
2nd Shift put down balance of Steel set and put it in placed, build up & lagged it.
3rd Shift took out staging and worked on bottom.

TOTAL DEPTH OF SHAFT BELOW COLLAR 126 ft TOTAL DEPTH TIMBERED 120 ft.

TEMPERATURE AT 7 A. M. 15° below WEATHER CONDITIONS Cold

REMARKS The weather being so cold it freezes up the muck in the pipes, also in the muck cars. I am using a steam jet with a great deal of success during the cold weather. The temperature last night was 30 deg. below zero. T. W. Powell, Supt.

FIG. 5.—Front of daily report form for shaft sinking.

DAILY LABOR STATEMENT Thur. DAY Feb. 5. 1914

LABOR CLASSIFICATION	SHIFT # 1 No.	Amount	SHIFT # 2 No.	Amount	SHIFT # 3 No.	Amount	TOTAL No.	Amount	RATE
Shift boss	1	$5 00	1	$5 00	1	$5 00	3	$15 00	$5 00
Shaftmen	6	24 00	1	28 00	8	32 00	21	84 00	4 00
Landers and dumpmen	X		1	3 15	1	3 15	2	6 30	3 15
Eng. Dump	1	3 15	1	3 15	1	3 15	3	9 45	3 15
Engineers	1	4 00	1	4 00	1	4 00	3	12 00	4 00
Firemen	1	3 35	1	3 35	1	3 35	3	10 05	3 35
Pumpmen	1	3 75	1	3 75	1	3 75	3	11 25	3 75
Pipemen	1	3 75					1	3 75	3 75
Master mechanic	1	4 25					1	4 25	4 25
Blacksmiths	1	4 00					1	4 00	4 00
Blacksmith helpers	1	3 00					1	3 00	3 00
Carpenter foreman	1	4 50					1	4 50	4 50
Carpenters	1	3 90					1	3 90	3 90
Surface foreman									
General surface laborers	7	20 30					7	20 30	2 90
Teamsters	1	3 15					1	3 15	3 15
Officemen									
TOTAL	25	$91 10	13	$80 40	14	$54 40	52	$194 90	
Accidents None									

FIG. 6.—Back of shaft sinking report form.

The following is a contract form used by one shaft company, revised to 1921:

SHAFT SINKING CO.

PITTSBURGH, PA......, 19..

....

....

....

We hereby propose to

in accordance with the following specifications:

I. SERVICES TO BE RENDERED

With respect to this work, we propose to.. ....

Upon your acceptance of this proposal, we will assemble at the mine site as promptly as possible a crew of experienced and efficient workmen, together with a mining captain, and such foremen and clerical help as are necessary. We will thereupon proceed with the prosecution of the work, carrying on the same thereafter with all reasonable diligence and in a skillful and workmanlike manner until its completion, in accordance with the plans and specifications adopted for the work, subject, however, to any delays which may be caused by weather conditions, labor strikes, accidents, and other causes beyond our control, or changes in the approved plans which may be required by unexpected conditions or by your instructions. In the prosecution of the work we will be guided in all respects by such specific instructions as you may give us from time to time.

II. EQUIPMENT AND MATERIALS

Adequate machinery, equipment, materials and supplies for the prosecution of the work are to be furnished and paid for by you; but at your request, or in case of your failure promptly to provide the same, and in order to prevent delay in the completion of the work, we will procure needful materials and equipment and charge the cost thereof to you, the amount of such cost to be paid to us by you at the next monthly settlement, as hereinafter provided.

III. COMPENSATION

As compensation for our services performed in connection with the said work, you are to pay us the sum of... any such compensation (as well as any expenditures made by us on account of machinery, equipment or materials, as provided in the preceding paragraph) to be paid to us as specified in Paragraph VI.

IV. COST OF WORK

It is understood that you are to pay all the costs of the work, including:

(a) The amount paid for all materials, machinery, tools, and equipment furnished by us, whether the same or any part thereof be purchased for the work or rented for use therein, the maintenance and insurance thereof, and the cost of replacement of any machinery, tools or equipment that may be worn out or destroyed; all such machinery tools and equipment, except that which may have been rented, to belong to you at the completion of the work.

(b) All amounts paid for labor and bonuses to workmen, provided that the scale of wages and amount of bonuses shall be approved by your engineers.

(c) The cost of all traveling and transportation expenses of men and of machinery, tools, equipment and materials supplied by us, including the cost of return transportation to........................ ....

(d) The cost of liability and other forms of insurance carried by us, and any expense incurred in connection with any accidents or damage to person or property.

(e) The cost at salary rate of our Mining Superintendent for the time actually spent by him on work under this contract, either in the field or at our office, and his expenses; and any other expenditures, not herein specified, made by us in the proper carrying out of the work, but not including any overhead expenses of our home office.

V. Insurance

All employers' liability or workmen's compensation insurance procured by us for our protection in carrying on the work under this contract shall be paid for by you, unless you shall elect to procure such insurance or to carry such risks yourselves, and shall enter into a satisfactory contract with us, assuming and agreeing to pay all damages and costs accruing through any accident or injury to workmen or others during the prosecution of the work, and to indemnify and hold us harmless from any liability or costs, including attorney's fees, on account of the same.

VI. Payments

As promptly as possible after the end of each month, we will render to you duplicate payrolls, with statements of all expenditures made by us during such month, the amounts shown by such payrolls and statements to be paid to us by you in New York exchange on or before the fifteenth day of the month in which the same are rendered, together with our compensation.

VII. Reports

We will render reports to you monthly showing the progress of the work, with any recommendations for changes in specifications adopted which it may seem advisable to make.

VIII. Termination of Contract

If, at any time, you shall become dissatisfied with the manner in which the work is being conducted by us, or shall wish, for any reason, to discontinue the work, you will be at liberty to terminate our employment hereunder upon giving us ten (10) days' notice in writing of your intention so to do, and at the expiration of said period of ten (10) days, we will surrender to you possession of the work; provided that, in such case, we shall be entitled to and you shall pay us, as our compensation for our services hereunder,

Acceptance and Approval

On acceptance of this proposal by you, and its approval by the President, Vice-President, or Manager of the Mining Department of this Corporation (unless the proposal shall be signed by one of them), this instrument shall constitute a binding agreement between us.

Shaft Sinking Co.

By..................

Accepte ...., 19...

.... Approved................., 19....

By.... ....

SECTION III

HAULAGE COSTS

With any method of handling coal underground the item of haulage is a rather small percentage of the total cost of production of a ton of coal. It represents, however, one of the items that can be varied by applying different methods, and a material reduction in costs can quite often be effected by a careful study of conditions and a proper application of equipment designed to do certain work.

The haulage system also has an important bearing on other costs that occur in producing coal and which are not directly a part of the haulage system. Because of the high speed and flexibility of its operation, it is at times possible to do intensive mining, thereby reducing the active area in the mine with a less ventilation cost, smaller trackage, easier supervision, and in many cases it permits of greater recovery.

The actual efficiency of a haulage system, when input power is compared with actual work in delivering coal from the face of the workings to the shaft bottom or tipple is extremely low, considering the difficulties to contend with. While it is well to consider the economical use of power at all times, it represents only a small percentage of the haulage costs and is not generally susceptible of material improvement.

The real reduction in haulage costs comes from so arranging the work that each piece of equipment is operated to its maximum capacity at all times. This is the really difficult problem to solve, as it is interlocked with methods of mining, drainage and ventilation, and the solution can only be worked out by men having an intimate acquaintance with the conditions surrounding any specific problem under consideration.

In considering the item of haulage the problem of obtaining an economic grade for each section of the development is primary in importance. There are localities, however, where the seam lies approximately level and entries or headings may be driven in any direction convenient for other reasons. The engineer must then plan the system of working to give the shortest haul with the minimum expense for construction and maintenance of roadways and also the least cost for entry driving.

In localities where union labor is employed a charge for yardage is made for driving the entries narrow. Narrow work

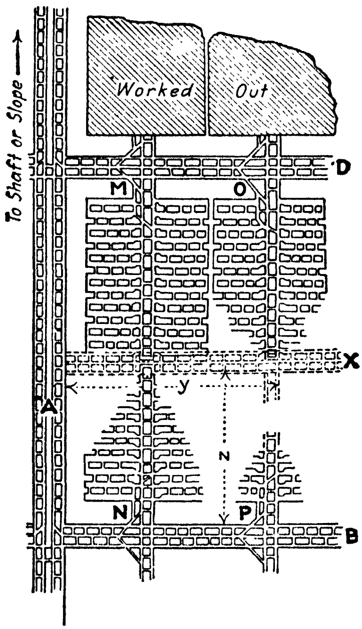


FIG. 1.—Plan for driving new entries to reduce length of haulage.

usually includes all work less than 18 ft. wide. The price per linear yard of entry varies somewhat in different fields, as it is fixed in the district contracts made between the operator and the labor organizations.

By increasing the amount of entry driven to a certain amount the shortest haul may be obtained. To effect this advantage the expense of constructing and maintaining the additional roadway and a reduction of efficiency through loss of ton mileage per foot of motor road operated must be considered.

The illustration, Fig. 1, shows a modified panel system of mining which is rapidly being adopted in many sections. From the main heading A secondary headings B and D are driven, and from the secondary headings the room or butt headings are driven on both sides. Partings are located on the butt headings just off the secondary heading.

In gathering from rooms on butt heading N the coal is hauled away from its ultimate destination; in gathering from rooms on heading M the coal is hauled toward its ultimate destination. In order to eliminate this back haul the secondary heading X, indicated in the sketch by dotted lines, could be driven and all coal from the territory of N and P taken out over a roadway located through this heading. By driving this additional heading the mine may develop sooner in that particular section, but the usual rush for coal at an immediate low cost too often induces a method of working which results in a sacrifice of future profits.

The cost for yardage in driving headlong X combined with many other items will be an additional expense to be prorated over the cost of producing the tonnage from the territory served. The items of expense for heading X alone may be expressed by formulæ.

Let y = length of heading X in feet;
c = yardage cost of driving 1 yd. of entry expressed in dollars;
n = number of entries on the heading;
C_1 = total cost of yardage for X;
C_2 = cost of construction of overcasts, brattices, doors, etc.;
C_3 = cost of construction of main haulage track, exclusive of switches, minus the salvage value of the material when released;
C_4 = cost of maintaining the roadway, stoppings, doors, overcasts, etc., during the period coal is being hauled;
E = the value of the loss of efficiency from a reduced ton mileage per foot of main motor road operated.

Then:

$$C_1 = \frac{cny}{3}, \quad \ldots \ldots \ldots \quad (1)$$

and

$$C = C_1 + C_2 + C_3 + C_4 + E, \quad \ldots \ldots \quad (2)$$

where C = total expense resulting from having driven the heading X.

The value of E is really a function of C although difficult of calculation. It is easy to see, however, that if heading X is omitted the traction on roadway B will be doubled and the combined tonnage (T) from butt headings N and P will have an average back haul of $Z/2$ ft.

Let V = cost of hauling 1 ton 1 mile underground, expressed in dollars;

K = total cost of back haul in dollars.

Then:

$$K = \frac{TVZ}{5280 \times 2} = \frac{TVZ}{10560}. \quad . \quad . \quad . \quad . \quad . \quad . \quad (3)$$

The advantages of driving the additional heading are now expressed by the value of K and the disadvantage expressed by the value of C. If other factors are not considered, the ratio of these values determines the ultimate plan to be pursued. The values for the assumed variables must be chosen only after accumulating and digesting all pertinent information obtainable in the same operating field or in a field supposedly similar. The value of the results will depend upon the competent judgment employed in the calculation, and the equations simply represent a plan of reasoning and investigation that should be undertaken before adopting a decisive system of operation.

Tractive effort, drawbar-pull and rating of mine motors.— A clear grasp of the relative economy and comparative cost of operation of the various types of under-ground haulage motors, necessitates a thorough understanding of the theories governing their operation. There are several distinctly separate factors concerned in locomotive haulage, which may briefly be described as follows:

The tractive effort of a locomotive is the force exerted by the motor at the circumference of the driving wheels. In a well-designed machine, the power of the motor is such as to equal the greatest adhesion of the wheels to the rails, which adhesion must of necessity limit the possible tractive effort of the machine. This tractive effort, as thus limited by the adhesion of the wheels to the rails, is therefore the force available to move the entire load, including the locomotive and the trip it hauls.

The drawbar pull is that portion of the tractive effort that is employed to move the trip attached to the locomotive. Thus the hauling capacity of a locomotive, as represented by the possible drawbar pull, is always less than the maximum tractive effort the locomotive can exert, by an amount equal to the force required to move the locomotive itself.

It should also be clear that the drawbar pull is always equal to the total resistance offered by the trip hauled, whether the locomotive is taxed to its full capacity or not. In other words, the drawbar pull in any event is limited by the resistance of the trip hauled.

The track resistance is the frictional resistance offered by the entire moving load and is estimated in pounds per ton of load.

The grade resistance is the gravity pull of a load resting on an inclined track and is equal to the weight of the load multiplied by the percentage of grade expressed decimally. Hence, grade resistance is always 20 lb. per ton for each per cent of grade, since 2000 lb. equals one ton and $0.01 \times 2000 = 20$ lb.

The actual drawbar pull, in any case, is equal to the sum of the track resistance and grade resistance of the load hauled. For purposes of estimate, it is common practice to assume the possible drawbar pull as varying from 20 to 30 per cent of the weight of the locomotive resting on the drivers, according to the condition of the rails and the kind of wheels used. But, the actual drawbar pull must be equal to the total resistance (track and grade resistances) of the cars, which is estimated in pounds per ton of load hauled.

Track resistance, in mining practice, will vary from 20 to 40 lb. per ton, while grade resistance is 20 lb. per ton for each per cent of grade. Assuming a level road and a track resistance of, say, 25 lb. per ton of load, the load that a 6-ton locomotive will haul on a level track, the drawbar pull being 2400 lb., is $2400 \div 25 = 96$ tons.

For the sake of illustration, let is be required to find the maximum load a 6-ton mine locomotive will haul up a 2½-per cent grade, assuming a track resistance of 30 lb. per ton. In this case, the grade resistance is $2\frac{1}{2} \times 20 = 50$ lb. per ton. The track resistance being 30 lb. per ton makes the total resistance of the load hauled $50 + 30 = 80$ lb. per ton. Then, taking the drawbar pull as approximately one-fifth of the weight of the locomotive, or $^1/_5\ (6 \times 2000) = 2400$ lb., the maximum load this locomotive will haul on such a grade is $2400 \div 80 = 30$ tons.

Generally, tractive effort is preferable to speed in a mine motor of any discription. As an example, a 15 hp. electric motor rated at 9 miles per hour requires 50 per cent more power to start a certain load than a 10 hp. motor rated at 6 miles per hour.

Probably the best speed to obtain the maximum economy in operation is about 6 miles per hour for motors up to 10 tons capacity and 7 to 8 miles per hour for those of larger capacity. The actual running speed of nearly all electric motors is from 30 to 40 per cent higher than the rated speed after the load is started.

Mine motors are rated at the end of the armature shaft. The following is a short and convenient formula for determining the horsepower output at drawbar of any motor:

$$\text{Horsepower} = \frac{\text{Tractive effort in lb.} \times \text{speed in mi. per hr.}}{375}$$

The tractive effort equals one-sixth the weight of the motor and the effective wattage per-mile-hour is double the tractive effort. The horsepower required to operate a $7\frac{1}{2}$ ton motor at 6 miles per hour when running at 90 per cent efficiency is:

$$\frac{6(2 \times 2500 \text{ tractive effort})}{0.90 \text{ efficiency}} = 33{,}333 \text{ watts} \div 746 = 45 \text{ h.p.}$$

The accompanying table gives the drawbar pull and haulage capacity of electric mine motors of from 3 to 30 tons weight working on grades up to 10 per cent, as compiled by the Jeffrey Manufacturing Co.

Where possible an actual test should be made in order to determine the average frictional resistance of the cars. This test can be made by pulling a car at a constant speed on a level track or on a track of a known grade and measuring the pull by means of a spring balance. Another method which will give the approximate friction is to find a grade down which a car will coast slowly at a constant speed without tend-

Haulage Capacity of Electric Mine Locomotives Equipped with Cast Chilled Wheels

This table gives the drawbar pull in pounds and the haulage capacity in tons for a given weight of locomotive on various grades when equipped with chilled cast-iron wheels. A coefficient of friction of 30 pounds per ton has been assumed on level track and 20 pounds per ton for each per cent of grade.

Grade	Level		1 Per Cent		2 Per Cent		3 Per Cent		4 Per Cent		5 Per Cent		6 Per Cent		7 Per Cent		8 Per Cent		9 Per Cent		10 Per Cent	
Locomotive Weight, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons
3*	1,200	40	1,140	23	1,080	15	1,020	11	960	9	900	7	840	6	780	5	720	4	640	3	600	2
4	1,600	53	1,520	31	1,440	20	1,360	15	1280	12	1200	9	1120	7	1040	6	960	5	880	4	800	3
5*	2,000	67	1,900	38	1,800	26	1,700	19	1600	15	1500	11	1400	9	1300	8	1200	6	1100	5	1000	4
6	2,400	80	2,280	45	2,160	31	2,040	23	1922	17	1800	14	1680	11	1560	9	1440	7	1320	6	1200	5
7*	2,800	93	2,660	53	2,520	36	2,300	25	2240	20	2100	16	1960	13	1820	11	1680	9	1540	7	1400	6
8	3,200	107	3,040	61	2,880	41	2,720	30	2560	23	2400	18	2240	15	2080	12	1920	10	1760	8	1600	7
10	4,000	133	3,800	76	3,600	51	3,400	38	3200	29	3000	23	2800	19	2600	15	2400	13	2200	10	2000	9
12*	4,800	160	4,560	91	4,320	62	4,080	45	3840	35	3600	28	3360	22	3120	18	2880	15	2640	13	2400	10
13	5,200	174	4,940	98	4,680	67	4,420	49	4160	38	3900	30	3640	24	3380	20	3120	16	2860	14	2600	11
15	6,000	200	5,700	114	5,400	77	5,100	57	4800	43	4500	35	4200	28	3900	23	3600	19	3300	16	3000	13
17*	6,800	226	6,460	129	6,120	87	5,780	64	5440	49	5100	39	4760	32	4420	26	4080	21	3740	18	3400	15
18*	7,200	240	6,840	136	6,480	92	6,120	68	5760	52	5400	41	5040	34	4680	28	4320	23	3960	19	3600	16
20	8,000	267	7,600	152	7,200	103	6,800	75	6400	58	6000	46	5600	37	5200	31	4800	27	4400	21	4000	17
25	10,000	330	9,500	190	9,000	118	8,500	95	8000	72	7500	58	7000	46	6500	38	6000	32	5500	26	5000	22
30	12,000	400	11,400	227	10,800	156	10,200	113	9600	87	9000	70	8400	56	7800	46	7200	38	6600	32	6000	26

*These are odd sizes, not in production according to the Electric Power Club standard.

Haulage Capacity of Electric Mine Locomotives Equipped with Steel-Tired Wheels

This table gives the drawbar pull in pounds and the haulage capacity in tons for a given weight of locomotive on various grades, when equipped with steel-tired wheels. A coefficient of friction of 30 pounds per ton has been assumed on level track, and 20 pounds per ton for each per cent of grade.

Grade	Level		1 Per Cent		2 Per Cent		3 Per Cent		4 Per Cent		5 Per Cent		6 Per Cent		7 Per Cent		8 Per Cent		9 Per Cent		10 Per Cent	
Locomotive Weight, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons	Drawbar Pull, Pounds	Haulage Capacity, Tons
3*	1,500	50	1,440	29	1,380	20	1,320	15	1,260	12	1,200	9	1,140	8	1,080	6	1,020	5	960	4	900	4
4	2,000	70	1,920	39	1,840	26	1,760	20	1,680	15	1,600	12	1,520	10	1,440	8	1,360	7	1280	6	1200	5
5*	2,500	84	2,400	48	2,300	33	2,200	24	2,100	19	2,000	15	1,900	13	1,800	10	1,700	9	1600	8	1500	6
6	3,000	100	2,880	58	2,760	39	2,640	29	2,520	23	2,400	18	2,280	15	2,160	13	2,040	11	1920	9	1800	8
7*	3,500	117	3,360	67	3,220	46	3,080	34	2,940	27	2,800	22	2,660	18	2,520	15	2,380	12	2240	11	2100	9
8	4,000	133	3,840	77	3,680	53	3,520	39	3,360	30	3,200	25	3,040	20	2,880	17	2,720	14	2560	12	2400	10
10	5,000	167	4,800	96	4,600	66	4,400	49	4,200	38	4,000	31	3,800	26	3,600	21	3,400	17	3200	15	3000	13
12*	6,000	200	5,760	115	5,520	79	5,280	59	5,040	46	4,800	38	4,560	30	4,320	25	4,080	21	3840	18	3600	16
13	6,500	216	6,240	126	5,980	85	5,720	63	5,460	50	5,200	40	4,940	33	4,680	27	4,420	23	4160	20	3900	17
15	7,500	250	7,200	144	6,900	99	6,600	73	6,300	57	6,000	46	5,700	38	5,400	32	5,100	27	4800	23	4500	19
17*	8,500	283	8,160	163	7,820	112	7,480	83	7,140	65	6,800	52	6,460	43	6,120	36	5,780	30	5420	26	5200	22
18*	9,000	300	8,640	173	8,280	118	7,920	88	7,560	68	7,200	55	6,840	45	6,500	38	6,120	32	5760	27	5 00	23
20	10,000	333	9,600	192	9,200	132	8,800	98	8,400	76	8,000	62	7,600	51	7,200	42	6,800	36	6400	30	6000	26
25	12,500	416	12,000	240	11,500	164	11 000	122	10.500	96	10,000	77	9,500	64	9,000	53	8,500	45	8000	38	7500	32
30	15,000	500	14,400	287	13,800	197	13,200	146	12,600	114	12,000	92	11,400	76	10,800	63	10,200	53	9600	46	9000	39

* These are odd sizes, not in production according to the Electric Power Club standard.

Courtesy Jeffrey Manufacturing Co.

ing to increase or decrease its velocity. If this grade is say 1 per cent then the frictional resistance of the car would be 1 per cent of 2000 or 20 lb. per ton.

When the frictional resistance of the cars is not given it should be assumed at 30 lb. per ton unless they are newly equipped with roller bearings of an approved make. Roller bearings when installed and looked after properly, will no doubt give frictional resistance ranging from 15 to 20 lb. per ton. However, a number of roller bearings which had been neglected and not properly lubricated were once tested and showed the average resistance of several cars was 36 lbs. per ton.

On account of the short wheel base of a mine car the friction may be considerably higher when pushed than when pulled. When a string of cars are pushed they are liable to wobble more or less, and considerable binding of the flanges against the rails may take place. When the cars are pulled they are stretched out straight and but little wobbling will take place. The frictional resistance of new cars will largely depend upon the type of bearing, while for the old cars it will be decidedly influenced by the manner in which the bearings are kept up.

The locomotive resistances will range from 12 to 20 lb. per ton. It is safe to take 15 lb. as an average since the friction of the locomotive is such a small percentage of the total tractive effort, and a change of several pounds in either direction will not affect the weight of the locomotive appreciably, and the effect on the capacity of the motors will be negligible.

The effect of the frictional resistance of the load will vary with the length and severity of the grades. If the track is practically level throughout, then a small change in the frictional resistance may have considerable effect on both weight and equipment. If, however, the grades are long and severe the effect will be small.

When a locomotive is operating at a constant speed on a straight, level track, the drawbar pull available for hauling a trailing load (provided there is sufficient motive power) is limited only by the adhesion that can be obtained between the driving wheels and the rails. When starting, the drawbar pull available is reduced, depending upon the rate of acceleration. As this rate is seldom more than 0.2 to 0.25 mi. per hour per second, the drawbar pull will be reduced from 19 to 24 lb. for each ton weight of locomotive.

If there are no grades the weight of the locomotive will be affected considerably by the rate of acceleration. With heavy grades, however, the acceleration will have little effect since the rate can be kept low if it becomes necessary to start on the heavy grade. Accordingly with the low rate of acceleration common to mine service this factor can be considered negligible as regards the weight of the locomotive, in view of the fact that a greater percentage of adhesion can be allowed for starting by the use of sand.

It has been found in practice that with cast-iron wheels a running drawbar pull equivalent to an adhesion of 20 per cent of the weight on the drivers can be obtained with clean dry rails on level track, without the use of sand. A steel-tired or rolled-steel wheel seems to obtain a better grip on the rails, and a drawbar pull equivalent to an adhesion of 25 per cent can be obtained under the same conditions. When starting heavy trips and when on steep grades it is permissible to use sand, in which case a drawbar pull equivalent to 25 to 30 per cent for cast-iron wheels and 30 to $33^1/_3$ per cent for steel wheels can be expected.

Where grades are short the higher rates of adhesion may be used, but for long grades it is not the best practice. Dynamometer tests have given adhesion values as high as 40 to 45 per cent by the use of sand. The average of the tests was, however, much lower so that it is not good practice to count on such high values. These high percentages require the liberal use of sand on both rails, a practice which should not be encouraged as the sand increases the frictional resistance of the locomotive and cars, and may work into the bearings and gears.

Where no grades exist the weight of the locomotive should, therefore, be five times the drawbar pull for cast-iron wheels and four times for steel wheels, unless the rate of acceleration is such that additional weight is required. When, however, a locomotive with a trailing load is ascending a grade the drawbar pull is necessarily greater than that required to overcome the friction of the trailing load as the weight of the load has to be lifted up the grade. For every 1 per cent grade 20 lb. per ton should be added to the drawbar pull required on straight level track since 20 is 1 per cent of 2000 lb.

The effect of grade on the locomotive as well as on the load must be considered. The heavier the grade the less will be the drawbar pull of the motor. This becomes evident when an abnormal grade is considered on which a motor will be barely able to propel itself, and if any trailing load is added the wheels will slip. The greater tendency for the wheels to slip on a grade is due to the increased tractive effort necessary to propel the motor itself and the weight transfer due to grade.

The weight transfer due to grade will depend on the wheel base and height of the center of gravity. With a short wheel base and a high center of gravity the weight transfer will be considerable. The modern mine motor, however, is constructed with a low center of gravity and a fairly long wheel base so that with the ordinary grades encountered the weight transfer is not serious.

The weight transfer due to height of drawbar will also effect the drawbar pull if the wheel base is short and the drawbar high. In Fig. 2 *a* represents the height of drawbar and *b* the

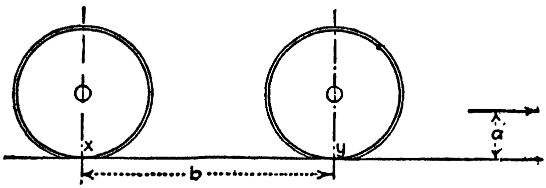


Fig. 2.—Influence of drawbar height on weight transfer of car.

wheel base. A horizontal force at the drawbar will act as a bell crank, one of whose arms is *a* and the other *b*. If the horizontal force represented by the drawbar pull is in the direction of the arrow, an upward force will be exerted at x and a downward force at y. If the drawbar pull is represented by D then the moment about y will be Da. This moment divided by b will give the lifting force at x. The adhesion of the wheel x will of course be lessened by the lifting force.

Assume a locomotive weighing 12 tons with a wheel base of

5 ft. and the height of drawbar 10 in. At 25 per cent adhesion Da will be $6000 \times \frac{10}{12} = 5000$ ft. lb. The upward pull at x will be $5000 \div 5 = 1000$ lb. The normal weight at x is 12,000 lb. therefore the weight when the drawbar pull is 6000 lb. will be 11,000 lb. so that the adhesion will be $^{3000}/_{11000} = 27.2$ per cent. It is thus to be seen that with the ordinary height of drawbar and wheel base the drawbar pull is not seriously affected. In metal mining a much higher drawbar is sometimes required so that the wheel base must be lengthened to lessen the tilting effect.

By tying the axles together by means of side rods, chains or gears the effect of weight transfer due to grade and height of drawbar can be eliminated, but these devices have not proven successful from an operating standpoint.

Ordinary mine cars will require from 30 to 40 lb. pull per ton to move them on a level track. The horizontal tractive resistance of modern cars in good condition may drop down as low as 20 lb. per ton, but it is not safe to figure much less than 30 to 35 lb. per ton on the level. For each 1 per cent grade against the load, 20 lb. per ton must be added. For example, if a car will move on 30 lb. per ton pull on a level, it will require 30 plus 20 or 50 lb. pull against a 1 per cent grade, or 30 plus 20 plus 20 or 70 lb. pull against a 2 per cent grade.

A. M. Wellington found that it required 5 lb. per ton to pull a loaded freight car and 7 lb. per ton to pull an empty freight car over a level railroad track at 10 mi. per hr.

R. Van A. Norris made 980 tests some years ago, which gave a resistance of 26 lb. per ton for 20 loaded mine cars and 42 lb. per ton for 20 empty mine cars of the same size over the same track, traveling 4½ mi. per hr. One reason for the wide difference between the two sets of experiments mentioned is found in the size of the car wheels.

The freight cars had 32-in. diameter wheels, while the mine cars had 16-in. diameter wheels, thus making it possible for the former to ride over the inequalities in the roadbed with more ease than the latter.

Mr. Norris's experiments brought out another important matter not usually mentioned in haulage articles; namely, that it requires more power per ton to haul short trains of mine cars than longer ones. This is of added importance to those mines which have adopted the two-unit locomotive in order to pull heavier loads.

One factor that probably reduces the ton resistance in longer trips is that the forward cars clean the track; and another is that the greater momentum of longer trips aids in keeping the cars moving.

Professor Baker made some experiments on clean and dirty tracks, the results of which briefly were as follows: On a perfectly clean track the resistance was 19 lb. per ton; on the same track with 1/8 in. of fine dust the resistance was 28 lb. per ton; while with 1/8 in. of powdered stone the resistance was 40 lb. per ton. This should comfort superintendents in the soft-coal mines who begrudge spending money on road cleaning, because when they clean the haulage entries, they not only lessen the dangers from explosions, but decrease the cost of haulage.

It is also a lesson for some anthracite superintendents, for fine anthracite is nearly equal to sand in offering resistance to traction effort.

A gasoline motor will exert a drawbar pull equal to one-fifth its weight in pounds when working on a level and on dry rail of proper weight for the motor, but from this drawbar pull exerted on a level we deduct 1 per cent of the weight of the motor in pounds for each 1 per cent grade. For example, a 5-ton motor will exert a tractive effort, or drawbar pull, on the level of one-fifth of 10,000 lb., or 2000 lb. Against a 1 per cent grade we would have 2000 lb. less 1 per cent of 10,000 lb., or 2000 less 100 lb., or 1900 lb., net; or 1800 lb., net against a 2 per cent grade.

It is not only advisable for the operator to prefer the locomotive with the higher tractive effort and low speed to the one with a lower tractive effort and higher speed having the same horsepower rating, but he should even be cautious in buying a locomotive with a high horsepower rating if such a rating is obtained on account of high speed.

The reason for this is that a high horsepower rating means increased power consumption without increasing the amount of work done by the locomotive, if the high rating is obtained through high speed. This may be made clearer by stating, for instance, that a 15-hp. locomotive with a rated speed of, say 9 miles per hour, will take 50 per cent more current for starting a certain train than a locomotive having a 10-hp. rating with a rated speed of 6 miles per hour.

It is evident from this how misleading a mere consideration of the horsepower rating of the locomotive may be. In the particular instance given above, the purchaser might think that he is getting a locomotive which will do 50 per cent more work when he buys a 15-hp. locomotive instead of a 10 hp., while actually he would not be able to pull any more with such a machine and would at the same time pay for his mistake in higher current consumption. As a matter of course,

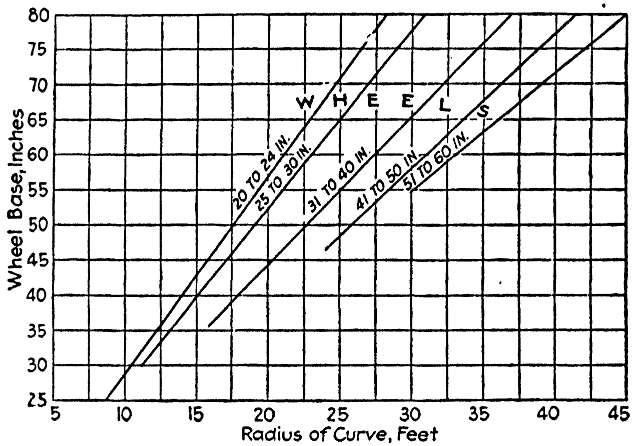


FIG. 3.—Curves showing the relation between wheel base and radius of curve.

there are certain limitations in speed, below which it would not be advisable to go, because if this is chosen below certain limits, the work done by the locomotive would be reduced.

In view of the numerous possibilities for being misled, it seems advisable for the buyer of a mining locomotive to request from the manufacturer the following information regarding the rating:

1. Weight of locomotive.

2. Will the motor be able to slip the wheels of the locomotive at all conditions of rail?

(These two questions will definitely determine the maximum tractive effort which the locomotive is able to exert.)

3. What is the one-hour rating of the locomotive according to the standardization rules of the A.I.E.E. bases on a temperature rise of 75 deg. C. on stand test of the motors?

4. What are the tractive effort and speed of the locomotive at the one-hour rating?

5. What is the continuous ampere rating of the motors on stand test at one-half and three-fourths of the rated voltage, and what tractive efforts correspond to these ratings; all bases on 75 deg. C. temperature rise on the stand according to the standardization rules of the A.I.E.E.?

By securing the above information and comparing same for the various locomotives in the market, the purchaser will be in a position to know what he is actually buying. Without it, he is liable to purchase almost anything without knowing just exactly what he is getting.

It may be advisable to cite here one of the many cases where purchasers have been misled by not taking a little time in getting the above information. At a certain mine, tests were made on three locomotives of different manufacture. One was a 15½-ton machine with motors rated at 185 hp. total, one was a 17-ton locomotive with motors rated at 210 hp. total. After an all day run, in which the number of car-miles hauled were practically the same, the temperature rises on the motors were as follows:

15½-ton locomotive, rated 185 hp., field 52 deg. C., armature 63 deg. C.

17-ton locomotive, rated 200 hp., field 56 deg. C., armature 65 deg. C.

16-ton locomotive, rated 210 hp., field 86 deg. C., armature 85 deg. C.

In other words, the locomotive with the smallest horsepower rating in this case proved to be the best of the three in actual service, while the machine with the highest horsepower rating not only showed up to be the poorest of the three, but even had temperature rises exceeding safe limits. This would mean a short life for the motor insulation and windings.

Number and size motors required.—In order to determine the proper equipment for a mine locomotive it is necessary to have the following information:

Plan and profile of the road.
Number of cars to be handled per trip.
Number of cars to be handled per hour.
Weight of empty cars.
Length of cars.
Weight of load.
Frictional resistance of cars.
Time of layover, including switching and making up trip.
Voltage of circuit.
Gage of track.
Weight of rail.
Radius and length of minimum curve.
Spread of track on minimum curve.
Limiting dimensions which locomotive can have.
Position and range of trolley wire.

It is seldom that all of the above information can be obtained, and in many cases it is necessary to make certain assumptions to supply the missing data. This can only be done by one having considerable experience in working out mining problems.

Motors for mine locomotives are rated on the one-hour basis with a 75-deg. C. rise in temperature. This rating does not indicate the capacity of the motor for all-day service, and is not used in determining its ability to meet with a certain set of conditions.

The capacity of a motor for all-day service depends upon the temperature which the windings will attain. This in turn depends upon the average heating value of the current. Since the heat generated by an electric current is proportional to the square of the current value, the average heating for all-day service must depend upon the square root of the mean square of the current.

Two motors may have the same one-hour rating, but one may have a much larger continuous capacity than the other, due to better design and the proper distribution of the losses. A poorly ventilated motor will in some cases have hot spots, which will lower the capacity of the machine. This is due to the fact that, in order to keep these spots within a safe temperature rise, the average temperature of the windings must be kept much lower than would be necessary if such spots were eliminated by proper design.

That the real capacity of a motor is its continuous capacity for all-day service and not the rating for one hour is apparently not generally appreciated among mine operators. The one-hour rating depends largely upon the thermal capacity of the motor, while the continuous rating depends on the ventilation, distribution of the losses and the capacity of the machine to radiate heat.

The one-hour capacity is not a fair rating of a motor for the foregoing reason and also because the speed of the motor is not taken into account. A fairer way would be to rate the machine on the pounds tractive effort at the wheels, irrespective of the speed, provided it is not considered essential for commercial reasons to capitalize the increased horsepower ratings due to increase in speed.

If the length of haul, the grade, curve, running time and time of layover are known, the current for each part of the run can be computed. In most main-haulage cases the locomotive will have a definite cycle to go through, this cycle being repeated throughout the working day. If the square root of the mean square current for one cycle can be found, this will, of course, determine the suitability of the motor selected for the all-day service as regards heating capacity.

To illustrate the working out of the above principles, the following conditions may be assumed to exist at a mine which desires to install electric haulage:

Locomotive required	1
Profile as follows:	
1300 ft. 2 per cent grade against load.	
1400 ft. 1 per cent grade against load.	
2200 ft. level.	
Number of cars to be handled per trip	20
Number of cars to be handled per hour	50
Weight of empty car	2000 lb.
Weight of load	4500 lb.
Total weight of loaded car	6500 lb.
Frictional resistance of cars	30 lb. per ton
Time of layover, including switching and making up trip	5 min. each end
Voltage of circuit	250
Gage of track	36 in.
Weight of rail	30 lb. per yd.
Radius of minimum curve	25 ft.
Length of minimum curve	20 ft.
Spread of gage on curve	¼ in.

Limiting dimensions of locomotive, 5 ft. wide, 4 ft. high.

Trolley wire 6 in. outside of rail; height above rail, 4 ft. 6 in. to 6 ft.

The total weight of the trip will be 65 tons.

The limiting condition in regard to weight is, of course, on the 2 per cent grade. The weight of the locomotive will be found as follows:

$$30 \times 65 + 20 \times 2 \times 65 + 20 \times 2 \times W = 400 \quad W$$

$W = 12.6$ tons if cast-iron wheels are used;
$W = 9.88$ tons if steel wheels are used.

It would, therefore, be necessary to use a 13-ton locomotive with cast-iron wheels or a 10-ton locomotive with steel wheels. Steel wheels should be used unless the customer specifies cast-iron. A locomotive to negotiate a 25-ft. curve should have a wheel base not more than 55 in. with 33-in. wheels or 65 in. with 30-in. wheels. With motors tandem hung no trouble will be experienced in keeping below 55 in. or 65 in. for a 10-ton locomotive. See Fig. 3.

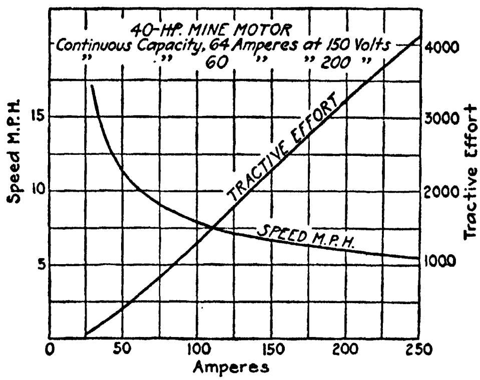


FIG. 4.—Characteristic curves for a 40 hp. mine locomotive motor.

The number of cars to be handled per hour being 50, the trips per hour will be $50 \div 20 = 2\frac{1}{2}$. The total time per trip, including layover at each end, will therefore be $60 \div 2\frac{1}{2} =$ 24 min. Allowing 5 min. layover at each end will make the actual running time 14 min.

For a locomotive of a given weight there are, as a rule, two or more motors to choose from. For a 10-ton machine these motors range from 40 to 50 hp. in capacity, although larger ones are sometimes required for special cases. A 40-hp. motor is selected for the first trial. The locomotive will have 30-in. wheels with a gear ratio of 4.78 to 1. The highest gear reduction is always selected unless a greater speed is required and can be obtained without overloading the motors. The characteristics of this motor are shown by the curves in Fig. 4.

This curve is made from an actual test, and the tractive effort given includes gear losses, so that to obtain the drawbar pull only the locomotive friction should be deducted. A table of data covering the case, such as shown in Table I or II, should be prepared, the values inserted being calculated from the motor curves and weights to be handled. Since the curves give values for one motor, the locomotive and trailing weight should be divided by two to give the weight each motor will be required to handle.

TABLE I

Speed, Miles per Hr.	Amp.	Total Tr. Eff.	Loco. Res.	Train Res.	Grade Res.	Distance, Feet	Time, Sec.	Amp.^{2}	Amp.$^2\times$ Time
8.5	87	1050	75	975	0	2200	177	7,569	1,340,000
6.5	165	2550	75	975	1500	1300	136	27,225	3,700,000
7.2	125	1800	75	975	750	1400	133	15,625	2,080,000
Returning with Empty Trip									
10	25	75	75	300	−300	1400	96	625	60,000
10	0	−225	75	300	−600	1300	89		
10	45	375	75	300	0	2200	150	2,025	303,000
Amp.$^2\times$time =									7,483,000
Plus 10 per cent for accelerating, switching, etc.								.. ....	748,300
Total amp.$^2\times$time									8,231,300
Total running time, sec.								781	
Total time at both ends, sec.								659	
Total time including layover, sec.								1,440	

8,231,300 ÷ 1440 = 5700 = mean squared current.
The square root of 5700 = 75.5 = square root of mean square current.
Capacity of 40-hp. motor is 60 amp.

For the above project the weight of locomotive is five tons per motor; the loaded trailing weight, 32½ tons per motor, and the light trailing weight, 10 tons per motor. Assume that the locomotive starts with a load on a level track and runs 2200 ft., when it encounters a 2 per cent grade. After ascending this grade for 1300 ft. the grade changes to 1 per cent for 1400 ft. The return trip will be with empty cars.

Starting with the loaded trip on the level, the locomotive resistance per motor will be $5 \times 15 = 75$ lb. This value is placed under "locomotive resistance" in the table. The train resistance will be $32.5 \times 30 = 975$ lb. The grade resistance will be zero. The total tractive effort will be 1050 lb.

Consulting the motor curves of Fig. 4 the current for a tractive effort of 1050 lb. is 87 amp. and the speed 8.5 miles per hour. The time to cover 2200 ft. at 8.5 miles per hour will be 177 sec. The amperes squared will be 7569 and the amperes squared multiplied by time will be 1,340,000. These values should be recorded in their proper place in the table.

When the 2 per cent grade is reached, the train and locomotive resistance will remain the same, while the grade resistance will be $40 \times (32.5 + 5) = 1500$ lb. The total tractive effort will be 2550 lb., which corresponds to a motor current of 165 amp. and a speed of 6.5 miles per hour. At this speed it will require 143 sec. to travel 1300 ft. By the same process the values for the 1 per cent grade are calculated and filled in the table.

On the return trip with the empty cars the locomotive resistance will be the same, the train resistance 300 lb. for 10 tons, and the down-grade resistance — 300 lb. for the 1 per cent and — 600 lb. for the 2 per cent grade. It will be noted that, running down the 2 per cent grade, the net tractive effort is — 225 lb., which means that the brakes must be applied in descending this grade.

On the 1 per cent grade and on the level the speeds shown on the curve of Fig. 4 are too high for most mines unless the track is in good shape. It is likely that the operator will not care to run faster than 10 miles per hour, which he can do by operating the motors in series on low notches, or by cutting off the power and coasting before the speed becomes too high.

The total running time is 781 sec., or 13 min. 1 sec. The actual running time will be close to 14 min., due to time taken to start and stop the trip and for possible slowing down at cross-overs. As the total time for a round trip is 24 min., a layover of five minutes is obtained at each end.

The product of the total current squared by the time is

7,483,000. To this, 10 per cent should be added to allow for acceleration and switching when making up trips, making a total of 8,231,300. The total time for making a round trip, including layover, is 1440 sec. Dividing 8,231,300 by 1440 = 5700 as the mean square of the current. The square root of 5700 is 75.5 amp., which is the square root of the mean square current for one trip or cycle.

The continuous capacity of the motor is 68 amp. at 150 volts and 64 amp. at 200 volts. The class of service is such that the average voltage applied to the motor will be near 200, so that the rating of the motor is about 65 amp., which shows that it is not of sufficient capacity for the service.

TABLE II

Speed, Miles per Hr.	Amp.	Total Tr. Eff.	Loco. Res.	Train Res.	Grade Res.	Distance, Feet	Time, Sec.	Amp.^{2}	Amp.$^2\times$ Time
10.1	100	1050	75	975	0	2200	149	10,000	1,490,000
7.6	187	2550	75	975	1500	1300	117	34,970	4,090,000
8.5	146	1800	75	975	750	1400	113	21,316	2,410,000
				Returning with Empty Trip					
10	35	75	75	300	−300	1400	96	1,225	117,500
10	0	−225	75	300	−600	1300	89		
10	56	375	75	300	0	2200	150	3,136	470,400
Amp.$^2\times$time =									8,577,900
Plus 10 per cent for accelerating, switching, etc.									857,790
Total amp.$^2\times$time									9,435,690
Total running time, sec								714	
Total time at both ends, sec								726	
Total time including layover, sec								1,440	

9,435,690 ÷ 1440 = 6550 = mean squared current.
The square root of 6550 = 81 = square root of mean square current.
Capacity of 50 hp. motor is 80 amp. at 150 v.

Care should be taken that a motor is not selected in which the commutating limit is exceeded when the wheels are slipped while using sand. The motor curves are generally stopped at the commutating limit, although with the modern commutating-pole motor it is rather difficult to find the commutating limit.

A larger motor should be selected, and Table II shows the results of the calculation using two 50-hp. motors. The curves of Fig. 5 show the characteristics of this motor. The total calculated running time is 714 sec., or 11 min. 54 sec. The square root of the mean square current is found to be 80 amp. The capacity of the motor at 200 volts is 78 amp. and 82 amp. at 150 volts, so that this motor will be of just about the proper capacity to meet the conditions. The actual running time will be about 12 to 14 min., allowing 5 to 6 min. at each end for

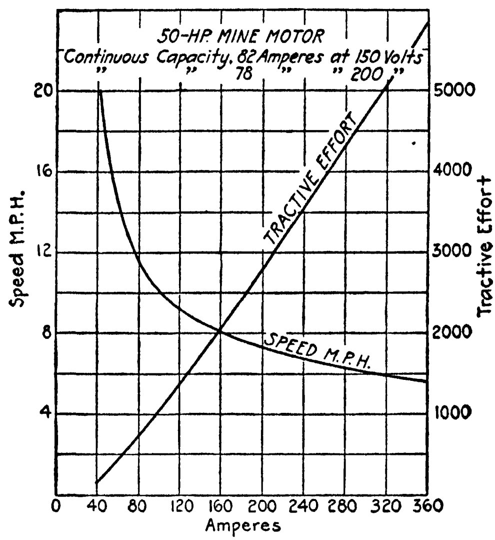


FIG. 5.—Characteristic curves for a 50-hp. locomotive motor.

switching and making-up trips. The higher root mean square current obtained for the 50-hp. motor is due to the fact that it is a higher-speed machine than the 40-hp. This shows the importance of having a low-speed motor.

It is not safe to figure on a short layover, as in many cases the average line voltage is much less than 500 or 250 volts, which means that the speed will be less than is figured on. A low line voltage signifies that a given current will be required for a much longer time than with the normal voltage, which in turn means additional heating. Where the voltage is likely to be poor, a margin should be allowed in the motor capacity, since the value of the square root of mean square current will be greater than that calculated.

The conditions outlined above in regard to profile are typical of what may be expected in mines. Sometimes it is possible to lay out a mine so that all of the main haulage will be with the grades; other mines will have mixed grades—that is, some inclinations in favor of the load and some against it; while too often a mine will be found with little or no level track and all grades against the load. If the profile instead of that given above were 2 per cent against load 300 ft., level track 2300 ft., 1 per cent in favor of load 2300 ft., the same weight of locomotive would be required, but the heating would be much less. Table III shows that the root mean square current is only 51.7 amp., which gives a large margin of safety when using the 40-hp. motor.

TABLE III

Speed, Miles per Hr.	Amp	Total Tr. Eff.	Loco. Res.	Train Res.	Grade Res.	Distance, Ft.	Time, Sec.	Amp.^{2}	Amp.$^2\times$ Time
6.5	165	2550	75	975	1500	300	31.5	27,225	870,000
10	40	300	75	975	−750	2300	157	1,600	251,200
8.5	87	1050	75	975	0	2300	184.5	7,569	1,395,000
Returning with Empty Trip									
10	45	375	75	300	0	2300	157	2,025	318,000
10	65	675	75	300	300	2300	157	4,225	664,000
10	0	−225	75	300	−600	300	20		
Amp.$^2\times$ time =									3,498,200
Plus 10 per cent for accelerating, switching, etc.								....	349,820
Total amp.$^2\times$ time								....	3,848,020
Total running time, sec								707	
Total time at both ends, sec								733	
Total time including layover, sec								1,440	

3,848,020 ÷ 1440 = 2670 = mean squared current.
The square root of 2670 = 51.7 amp. = square root of mean square current.

On the other hand, the grade may be 2 per cent for the entire distance against the load. In this case the root mean square current would be about 105 amp., corresponding to a motor having an hour rating of 75 to 80 hp. As it would not be practical to put two such large motors on a well designed

10-ton locomotive, it would be necessary to go to a 12- or 13-ton machine.

In another computation carried out along somewhat different lines, it will be assumed that it is desired to haul coal from a siding at the bottom of the slope, in trips of 30 cars. The empty cars each weigh 1 ton and have a capacity of 1.5 tons. This will make the total weight of the loaded trip $30\ (1 + 1.5) = 75$ tons. The accompanying profile of the road, Fig. 6, extending from the siding at the slope bottom in the mine to the tipple where the coal is loaded into the railroad cars, shows the length and grade of each portion of the track and the weight of rail in use.

It is desired to know the size of electric motor that will be required for this haul; also the weight of rail and style of rail bond that should be used, the size of trolley wire required for the transmission of the power from the generator to the mine, and the required horsepower of the generator and boiler.

The first step is to estimate the weight of locomotive required to haul a loaded trip of 75 tons up a 3-per cent grade, under ordinary mining conditions. It is not safe to estimate on the track resistance as being less than 50 lb. per ton. To this must be added 20 lb. per ton for each per cent of grade or, in this case, $3 \times 20 = 60$ lb. grade resistance, which makes the total resistance $50 + 60 = 110$ lb. per ton of total moving load including the locomotive.

The coefficient of adhesion of the wheels to the rails will be estimated at 0.16, which makes the tractive effort that the locomotive can exert to move itself and the loaded trip $0.16 \times 2000 = 320$ lb. per ton. Then, since this tractive effort of the locomotive must be equal to the total resistance of the entire moving load including the locomotive,

$$320W_m = 110(W_m + W_t) = 110W_m + 110W_t$$

and

$$W_m = \frac{110W_t}{320 - 110} = \frac{110 \times 75}{210} = say\ 40\ tons.$$

This shows conclusively that, under the adverse conditions common in coal mining, a 40-ton locomotive would be required to haul a loaded trip of 75 tons up a 3-per cent slope. This of course provides for the worst conditions that are liable to

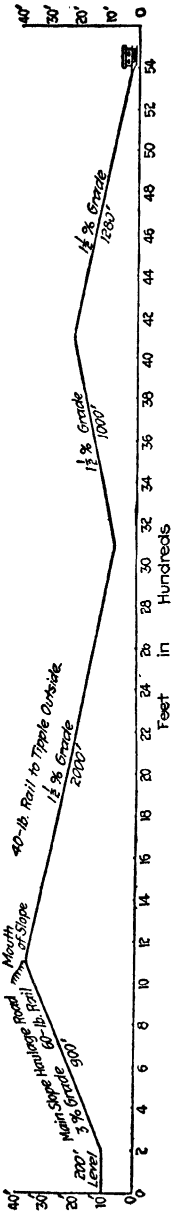


FIG. 6.—Profile of track showing a haulage condition from bottom of the slope to the tipple.

exist, in respect to both the track and rolling stock in the mine.

Before going further, it will be well to estimate the weight of locomotive that will be required to haul the same loaded trip on the $1\frac{1}{2}$-per cent grades outside of the mine. In this case, the track resistance, as before, is 50 lb. per ton, but the grade resistance is $1.5 \times 20 = 30$ lb. per ton, which makes the total resistance $50 + 30 = 80$ lb. per ton. This gives for the required weight of the locomotive:

$$W_m = \frac{80 \times 75}{320 - 80} = 25 \text{ } tons.$$

The same weight locomotive will haul practically two-thirds of the number of cars up the slope that it can handle on the outside grades, as shown by transposing the formula previously given and finding the load that a locomotive can be expected to haul regularly up grades of $1\frac{1}{2}$ and 3 per cent, respectively, under the conditions named, thus,

On a 3-per cent grade:

tractive effort of locomotive, 320 lb. per ton;
track and grade resistance, 80 lb. per ton.

Loaded trip, $$W_t = \frac{320 - 110}{110} W_m = 1.9 W_m.$$

On a $1\frac{1}{2}$-per cent grade:

tractive effort of locomotive, 320 lb. per ton;
track and grade resistance, 80 lb. per ton.

Loaded trip, $$W_t = \frac{320 - 80}{80} W_m = 3 W_m.$$

For this reason, it might be well to provide a siding at the top of the slope that will hold 20 or 40 cars as desired, and make three trips on the slope for every two trips to the tipple, using a 25-ton locomotive for the entire work.

So far, we have only considered relative weights of the locomotive and the load it can be expected to handle regularly and satisfactorily on different grades, under the worst conditions that are liable to exist or arise in mining practice. It is important to remember that having assumed a coefficient expressing the adhesion of the wheels to the rails it is the *weight* of the locomotive that determines the tractive effort the machine can exert, regardless of the power of the motors or engines with which it is equipped. It is a fatal mistake to equip a locomotive with more power than its weight will permit it to utilize.

Having decided on the weight of the locomotive necessary to haul the desired number of loaded cars over the given grades, the next step is to determine the power of the motors that will be required to produce a given speed of haul, say from 6 to 8 mi. per hr. As a basis of this calculation, it is important to observe that the effective power, in foot-pounds per minute, is equal to the tractive effort in pounds, multiplied by the speed in feet per minute, as expressed by the formula.

$$\underset{\text{(ft.-lb.p.m.)}}{Power} = \underset{\text{(lb.)}}{tractive\ effort} \times \underset{\text{(ft.p.m.)}}{speed.}$$

Now, estimating the horsepower required per ton on drivers, per mile-hour, assuming a tractive coefficient $c = 0.16$, we have:

$$Tractive\ effort\ (per\ ton) = 0.16 \times 2000 = 320\ lb.$$

$$Speed\ (per\ mi.\text{-}hr.) = 5280 \div 60 = 88\ ft.p.m.$$

$$Horse\text{-}power\ (per\ ton\text{-}mi.\text{-}hr.) \div \frac{320 \times 88}{33000} = 0.85\ h.p.$$

This is the effective horsepower per ton mile-hour, or the power that must be available to produce a speed of 1 mi. per hr., for each ton of weight resting on the drivers. Assuming an efficiency of 85 per cent, the input to the motors should be 1 hp. per ton-mi.-hr.

Using a 25-ton locomotive for this haulage and assuming that the entire weight of the locomotive is on the drivers, the actual horsepower of the motors required for a speed of 6 mi. per hr. will be $25 \times 6 = 150$ hp., which corresponds to a wattage of $150 + 746 = 111{,}900$ watts delivered to the motors.

It is interesting to note here that the effective wattage per mile-hour, in electric mine haulage, is practically twice the tractive effort of the locomotive, in pounds. This is shown by the following simple calculation, since a speed of 1 mi. per hr. is equal to 88 ft. per min. and 1 hp. is equivalent to 746 watts:

$$\textit{Effective watts (per mi.-hr.)}746 = \left(\frac{88 \times T.E.}{33000}\right) = \textit{say } 2\ T.E.$$

Under the assumed conditions, it was previously estimated that the total tractive effort when hauling up a 3-per cent grade is 110 lb. per ton of moving load. which makes the effective wattage required in that case $2 \times 110 = 220$ watts per ton-mi.-hr. In like manner, it was estimated that the total tractive effort when hauling up a 1½-per cent grade was 80 lb. per ton, which makes the effective wattage required to haul up that grade $2 \times 80 = 160$ watts per ton-mi.-hr.

In electric mine haulage, it is customary to estimate on hauling at a speed of, say from 6 to 8 mi. per hr. Taking the actual power of the motors at 1 hp. per ton-mi.-hr., to give an input of 111,900 watts on a 500-volt circuit will require a current of $111{,}900 \div 500 =$ say 224 amp.

The weight of rail in pounds per yard, in locomotive mine haulage, may be taken as twice the weight of the locomotive in tons; or, in this case, $2 \times 25 = 50$ lb. per yd. In estimating the size of wire required to transmit this current from the generator to the siding at the foot of the slope in the mine, we will assume a rail-return, using 50-lb. rails for the track, bonded with compressed-terminal bonds, 24 in. long. The length of track from the end of the siding at the foot of the slope to the tipple is about 5400 ft. The length of trolley wire extending to the power plant will be assumed as 5600 ft.

Referring now to the diagram shown in Fig. 7, first find the resistance of 5200 ft. of the two 50-lb. rails in the track-return, including the resistance of the bonds, assuming a rail to copper ratio of 1 : 11 and rails 30 ft. long, making $5400 \div 30 =$ say 180 bonds in a single length of rail. Following the horizontal line marked 50 on the left of the diagram, to its intersection with the curved line indicating a rail-to-copper ratio of 1 : 11; and, from that point, following the vertical line to the scale at the bottom of the diagram, we find a resistance of 18 microhms per foot of rail or $30 \times 18 = 540$ microhms per rail. In like manner, for a 24-in., 4-0 bond, follow the vertical line marked 24 on the top scale down to its intersection with the diagonal 4-0; and, from this point, follow the horizontal line to the right-hand scale, which shows a bond-resistance of 100 microhms.

The total resistance for 180 bonded rails is, therefore, 180 $(540 + 100) = 115{,}200$ microhms, or 0.1152 ohm. The resistance for a two-rail return is one-half this amount or 0.0576 ohm. The voltage absorbed in the rail-return is $E = CR = 224 \times 0.0576 = 12.9$ volts. The line drop for 5600 ft. of T. B., 4-0, copper wire, carrying a current of 224 amp., as taken from wire tables, is $5.6\ (224 \times 0.0489) = 61.3$ volts. The dif-

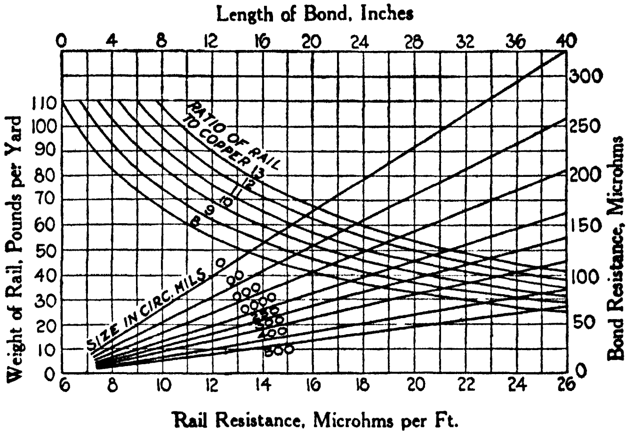


FIG. 7.—Resistance diagram for bonded rails.

ference of potential at the generator is therefore $61.3 + 12.9 + 500 = 574$ volts.

Finally, assuming an efficiency of 90 per cent, the power required to run the generator is:
Input to generator,

$$(574 \times 224) \div (0.90 \times 746) = \text{say } 190\ hp.$$

Boiler horsepower, efficiency of engine being 90 per cent.

$$190 \div 0.90 = \text{say } 210\ hp.$$

The narrow track gages now used for mine locomotives limit the axial length of the motor to a few inches. The desirability of obtaining the maximum tractive effort from a given machine by using the smallest possible pinion, limits the gear-center distance, and therefore the horizontal width of the motor, while restricted head room limits the height of the motor.

In Figs. 8 and 9 are shown two curves taken from a paper by G. M. Eaton, which demonstrates the fact that these con-

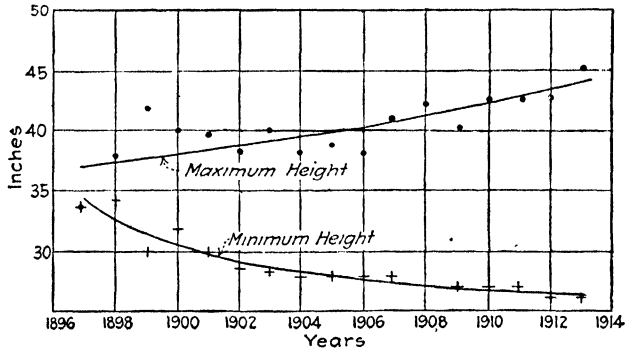


FIG. 8.—Maximum and minimum height of mining locomotives, 1896 to 1914.

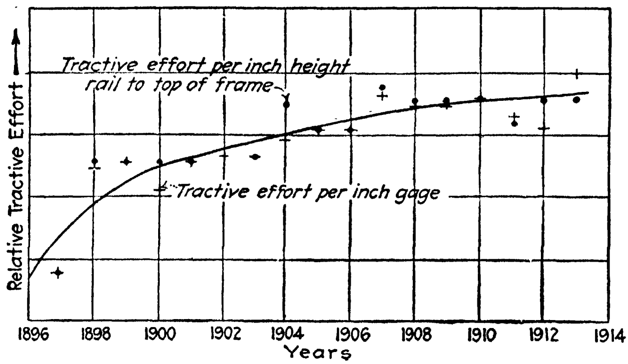


FIG. 9.—Tractive effort of mine locomotives per inch of height and of track gage.

ditions are becoming more and more exacting each year. One curve shows the minimum height allowed for mining locomotives over a period extending from 1896 to 1914, from which it will be seen that there has been a continuous decrease in minimum head room. The other curve shows the relative tractive efforts obtained per inch of height of locomotive as well as that per inch of gage. It will be seen from these curves that there exists a continuous demand for increasing the motor rating, crowded into a certain space.

The number and size motors used at different mines varies over a considerable range, the following being some typical examples:

The Peabody Kincaid Mine in Illinois uses 13-ton motors on the primary haulage and 6-ton on the secondary. The Madison Coal Corporation at its No. 6 Mine in the same state uses three 15-ton motors hauling 25 to 30-car trips and two 12-ton motors handling 20 to 25-cars trips. These motors are used exclusively on the main haulage, there being four hauls of 7000 ft. and five hauls of 5000 ft. Secondary haulage is done entirely with mules, there being 40 used for this purpose. A comparison of the number of tons handled per motor at the mines in Guernsey County, Ohio, showed this to vary from 250 to 500.

At a West Virginia mine, of small capacity, haulage costs per a single month (July, 1916) amounted to 5.09c. per ton. Room tracks at this mine were laid with wooden rails, mules were used for gathering, and a motor on the main haul, which was about one mile long. The capacity of the mine cars was 2500 lb.

Another West Virginia operation handled 35,000 tons in one month at the cost of less than 5c. per ton (September, 1916). This mine used a 10-ton motor on the main haul and a 6-ton on the secondary.

In the middle Western fields where large capacity cars predominate, it has been found that under average conditions a 5-ton gathering motor will handle from 80 to 125 cars per 8 hr. shift. These cars weigh one ton empty, have a capacity of 3½ tons, and the motor handles 15 loads on a level track or six to eight on a fairly stiff grade.

Experience in another field has shown that a 6-ton gathering motor working under average conditions will handle 80 to 90 cars of 2½ ton capacity in 10 hr.; where the average haul does not exceed 2000 ft. and under fairly good conditions, it will handle 100 to 125 cars. Computing on the basis of an actual working time of 8 hr., to make allowance for changing cars and unavoidable delays, and allowing 1½ tons for the weight of the empty car, this motor is performing 20 ton-miles of work an hour.

In general, small locomotives are more satisfactory than large ones. By small locomotives are meant those weighing from 5 to 10 tons. There are a number of 15-ton four-wheeled locomotives in fairly satisfactory service, and a few of this type weighing 20 tons are in use. Because of track conditions locomotives weighing 15 or 18 tons should have six wheels. When it is desirable to use machines underground weighing 20 tons or more the best results are obtained by using two four-wheeled units connected in tandem.

In a few special cases it is economical to use locomotives weighing around 30 tons, in which case it is advisable to employ two units connected in tandem and operated from a 250-volt trolley because of the difficulty of collecting the large amount of power required from a single wire. In general, when the size of the locomotive exceeds 15 tons, current at 500 volts should be used to secure good results.

The Berwind-White Coal Mining Co. in 1915 placed in operation at Windber, Penn., what was then the two largest and most powerful single-unit electric mine locomotives in the world. These machines weigh 30 tons each, and are of the three-motor, six-wheeled type, with equalizing levers to evenly distribute the weight upon the drivers. The side and end frames are constructed of what is known as "armor plate," solid rolled steel slabs, the sides being 5-in. thick, 40 in. high and 17 ft. long.

These members by the aid of angles of heavy steel casting are securely bolted together, forming a rigid unit.

The motor equipment of each locomotive consists of three 115-hp., 500-volt ball-bearing motors, capable of developing a tractive effort of 15,500 lb. at a speed of 8 miles per hour. Incorporated in the design of these motors are the essential mechanical and electrical features, which have, to a large extent, solved the troubles encountered in mining work. The application of ball bearings permits the use of gearing of 5½ in. face, and a commutator 5⅛ in. wide, within the space available on a 36-in. track gage, without a sacrifice in size of any part.

An electric mine locomotive that has remarkable record for length of service is at present operating in a mine of the Union Pacific Coal Co. at Rock Springs, Wyo. This locomotive was put into service in its present location 27 yr. ago. It has been giving continuous service ever since, records kept by the Union Pacific Coal Co. showing that it has hauled 3,712,500 tons of coal over an average distance of 1.5 miles, making a total of 5,568,750 ton-miles.

This locomotive is a terrapin back machine built for 500 volts and having a speed of 8 miles per hour. It has a drawbar pull of 3000 lb. and the wheels are 28 in. in diameter.

Gathering Locomotives.—When a locomotive is to be used for gathering it is difficult to determine the proper capacity by the preceding methods, since the service consists largely of starting and stopping with varying loads. From experience it has been found that if the horsepower per ton of weight of locomotive ranges from 6 to 10, the motors will have ample capacity.

This scheme is much of a makeshift and has little real reason behind its use. Locomotives have been in successful operation for years with but 6 hp. per ton and the motors were not overloaded. On the other hand, cases sometimes arise where 12 to 14 hp. per ton would be entirely inadequate. A high horsepower per ton is often obtained by using a high-speed motor and either allowing the locomotive to run fast or to use a pinion which is entirely too small for safe operation.

The placing of too large an equipment on a locomotive is a detriment instead of an advantage. It is somewhat similar to hitching a heavy dray horse to a light express wagon. The horse cannot work up to anything like his capacity and is likely to injure the wagon in attempting to perform his normal work.

Where the motor is too large the speeds will be high and the motor-man is often tempted to use an excess of sand and to hold the brakes on during acceleration to prevent slipping. This kind of operation results not only in a rise in power consumption but also a large increase in mechanical wear. The saving in electrical repairs may be more than balanced by the increase in mechanical repairs.

When reels are desired they can be furnished either of the traction or electrical type. The traction reel is used where steep grades are encountered and it is not desirable to have the locomotive enter the rooms. The capacity of the motor for a traction reel should be from 4 to 8 hp., depending upon the conditions. A small resistance type of controller should be used.

Of electrical reels two general types are used—the mechanically and the electrically driven. The mechanically driven reel is geared to one of the axles or to one of the main gears. This reel has the disadvantage that if the wheels are locked by the brake when coming out of a room the cable will be run over and cut in two.

The electrically driven reel consists of a horizontal or vertical axle reel equipped with a small motor, which acts as an electrical spring, always keeping tension on the cable. The motor is generally less than 1 hp. in capacity and is so wound that it can remain connected across the line continuously without danger of being burned out. The horizontal reel has the advantages that it does not interfere with access to the main motor and can be mounted low down in front without increasing the height of the locomotive.

Costs.—A leading manufacturer of electric mine motors quoted prices, f.o.b. mines, January, 1921, as follows: 4-ton, $3950; same with gathering reel, $4500; 6-ton, $4595; and 8-ton, $5540, both the latter being without gathering reels. The life of of mine motors may be estimated at 25 yr.

R. V. Norris, in the transaction of the A. I. M. M. E., Vol., 34, p. 976, gives interesting figures on the cost of the first electric underground haulage system installed in this country, and it is believed the second in the world, which was started July 23, 1887, at the Short Mountain Colliery of the Lykens Valley Coal Co.

The plant consists of two 5-ton motors of 32 hp. each, two 65 hp. generators driven by an old 18 × 24 in. plain slide-valve engine. The average voltage is 450, the amperage from 40 to 200, the average indicated horsepower of the engine, about 75, with a steam consumption of about 75 lb. per indicated horsepower. The cost of steam at the colliery was 8.11c. per 1000 lb., so that the steam cost per day was $4.56.

There were two main hauls at this colliery, one of 9500 ft. and the other of 10,400 ft. The average trip was made up of 15 cars, weighing one ton each and carrying 2.25 tons of coal or 3 tons of rock. The following is the work done with this system in the year 1901:

The total cost for labor and supplies in the year 1901 was $4510.42 and the approximate cost of power as noted above was $1286.83, making the total operating expense $5797.25. Computed on a ton-mile basis the net work costs were distributed as follows: Labor and repairs, 1.59c.; power 0.46c.; total 2.05c.

Gross work per ton-mile costs were: Labor and repairs, 0.86c.; power, 0.13c.; total 1.09c.

	9,500 ft. Haul	10,400 ft. Haul	Total
Cars of coal	48,120	14,681	62,801
Cars of rock	4,233	235	4,468
Tons of coal	108,270	33,032	141,302
Tons of rock	12,699	675	13,374
Total tons	120,969	33,707	154,676
Ton-mileage, effective load	217,744	66,600	284,344
Ton-mileage, dead load, cars and empty return trips	188,471	58,769	247,240
Gross ton-mileage	406,215	125,369	531,584

An excellent method of checking up the work of haulage motors, by means of charts and graphs, was described at the February, 1915 meeting of the A. I. M. M. E. a typical chart used by one of the larger companies in West Virginia being shown in Fig. 10. As will be noted the chart shows an adopted standard of 100 per cent efficiency for men and equipment, with the curves showing what is actually being accomplished plotted on. This particular chart shows the work being done by a gathering motor.

Motor Losses.—In any direct-current motor, the losses can be divided under three heads. The C^2R, or *copper loss*, is due to the resistance of the conductors on the armature and field windings. This loss varies with the square of the current, as will be seen from the formula $C^2R =$ watts lost.

The copper loss is composed principally of two parts, one due to the load or line current and the other due to the idle current that circulates in the short-circuited turns of the armature winding at the instant of commutation, due to the distortion of the field by the armature reaction.

The core loss can be separated into two parts. One is known as the eddy-current loss and is due to the current set up in the armature laminations by their being revolved in a magnetic field. This loss increases with the square of the number of magnetic lines per square inch, or the density of the field, and with the square of the speed.

The other part, known as the hysteresis loss, is due to the reversals of the magnetic lines of force in the armature iron, as

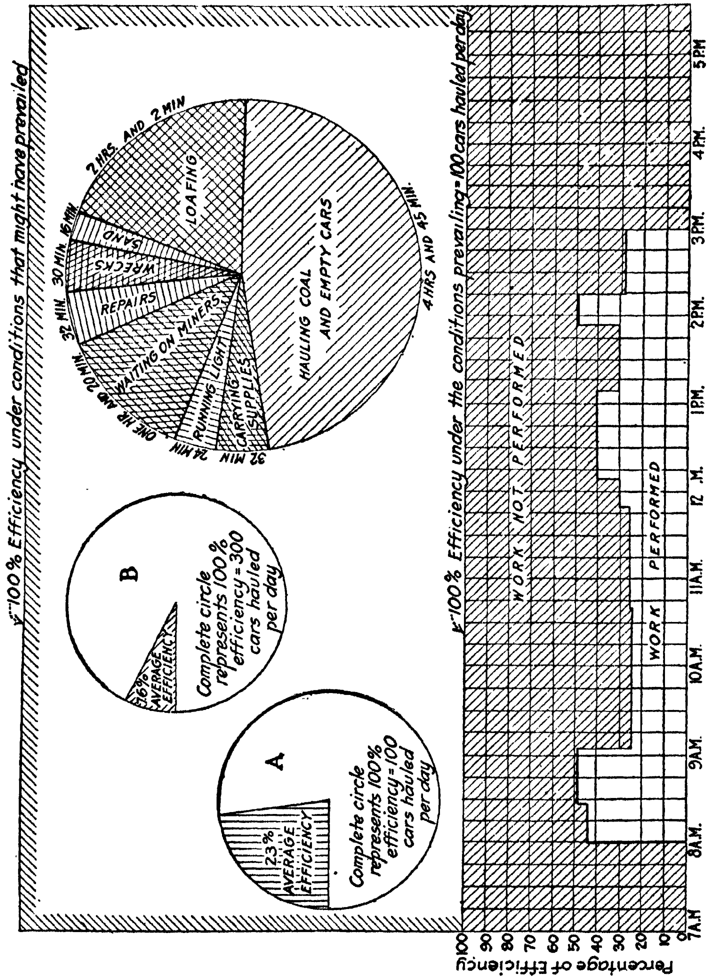


FIG. 10.—Efficiency chart of a gathering locomotive under conditions of actual operation.

it is being revolved in the magnetic field, which varies with the 1.6 power of the density and directly with the speed. These two losses combined are known as the core loss.

The third loss is the mechanical, or friction loss, and is due to the bearing friction, the brush friction on the commutator and the windage. This friction loss amounts to about 1 per cent at full load of the energy being supplied to the motor; and, as it is practically equal on all types of machines, it can be eliminated from the discussion.

The combined core and copper losses in a well designed motor will amount to about 12 to 14 per cent of the input at full load, and about 8 to 10 per cent of the input at one-half load. At full load, this loss is largely composed of the C^2R or copper loss, but at one-half load it is principally the core loss.

From this it will be seen that the copper loss is the greatest loss at full load, and the one largely responsible for the one-hour rating of the motor. On the other hand, it will be noted that for loads of one-half or less, the core loss is the principal loss and, therefore, largely responsible for the continuous rating of the motor.

The copper loss of the motor is well within the power of the engineer to control. He, therefore, can govern fairly well the hourly rating that the motor is to have. The core loss of a noncommutating-pole motor is not so well within the control of the engineer, because he must sacrifice low core loss in order to get good commutation, and in so doing, sacrifice the continuous rating of the machine.

The commutating pole minimizes the distortion of the field by the armature reaction, fixes the brush position for all loads with either direction of rotation, prevents all sparking and burning at the brushes, and reduces local current in the armature winding to a minimum.

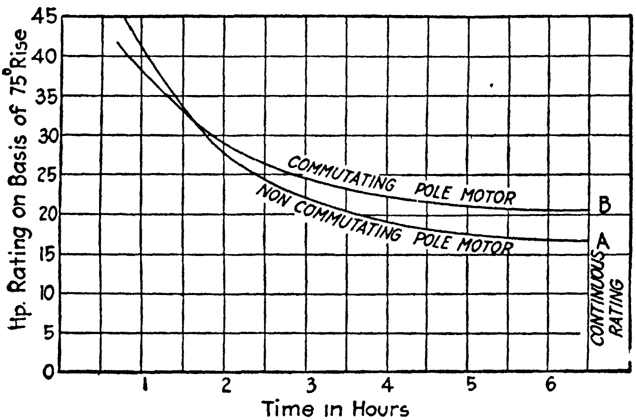
In order to prove these theories and substantiate the above statements, the horsepower time-curves and core loss-curves of two motors of practically the same one-hour rating are shown in Figs. 11 and 12. These curves are the result of actual tests conducted on these two types of motors. The mechanical features of the two machines are as follows:

	Noncommutating Pole Motor	Commutating Pole Motor
Weight	2130 lb.	1850 lb.
Speed	375 r.p.m.	395 r.p.m.
Mount on	28-in. wheels	28-in. wheels
Mount on	30-in. gage	28-in. gage
Builder's rating	40 hp.	37½ hp.

Figs. 11 and 12 show the horsepower time and core-loss curves of the two motors on the standard basis of the 75-deg. C. rise.

Curve *A* represents the noncommutating-pole motor, and curve *B* the commutating-pole machine. By referring to curve *A,* it will be seen that the noncommutating-pole motor has an hourly rating of 41 hp., and a continuous rating of 16½ hp., or 42 per cent of its one-hour rating for a continuous rating.

Referring to curve *B*, it will be seen that the commutating-pole motor has a one hour rating of 38 hp. and a continuous rating of 20.8 hp., or 54.8 per cent of its one-hour rating for a continuous rating.



IG. 11.—Horsepower curves for the two types of motors having the same 1 hr. rating.

If the noncommutating-pole motor had been designed with interpoles, along the lines of the commutating-pole machine, it would have 54.8 per cent of its one-hour rating, 22.6 hp. for its continuous rating instead of its present rating of 16½ hp., an increase of 6.1 hp., or 36.4 per cent in continuous rating.

Power Costs.—The cost of power to operate motors can be reduced by obtaining a high load factor. The maximum energy demand of the motors on the grade will be the same, since we presuppose that the largest number of cars will be pulled at all times, depending on the grade, but the average power consumption of the motor will be increased at all other points on the trip, thus affecting the load factor for the entire mine.

The results of many observations show that a definite relation exists between the cost of production of electrical energy and the load factor. With a small plant of one unit, as usually installed at a coal mine, the cost of power at no load is excessive, being approximately one-third to one-half that of full load. On account of the terms of a contract with the miners' union the labor cost is usually a constant, and the only difference in the

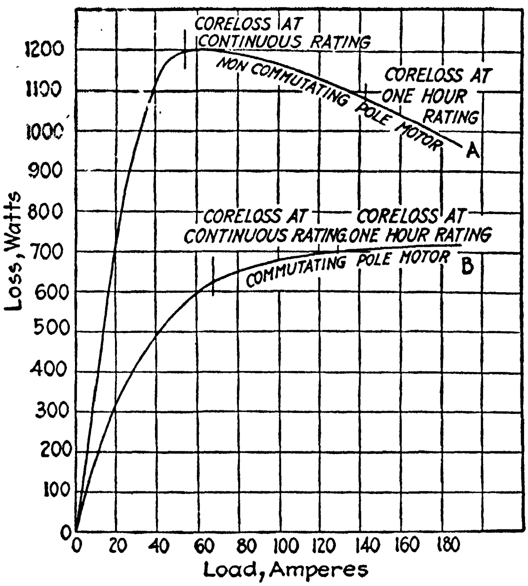


FIG. 12.—Core-loss curves of the two types of motors having the same 1 hr. rating.

cost between no load or part load and full load is in the amount of coal and supplies used. The advantage of keeping the plant load uniformly near the rated capacity without high peaks is plainly evident, and in case the power is purchased from a central station, the saving may be considerable by obtaining a sliding scale contract providing for this feature.

In order to express the total value of the power saved by a formula:

Let n = the number of trips made by the locomotive per shift;

g = gross weight of locomotive and trips, both loaded and empty, when the mine is producing t tons of coal;

G = gross weight of locomotive and trips, if mine produced T tons of coal through adding x cars to each trip;

$(T-t)=(G-g)n$ = increased tonnage hauled;

L = net tonnage of locomotive alone for all trips;

E_T = the efficiency or percentage of power used in hauling cars when the mine produces T tons;

E_t = the efficiency when the mine produces t tons;

K = cost of power per kilowatt-hour expressed in dollars;

R = per cent saved per kilowatt-hour due to reduction of load factor;

P_T = power used to produce T tons;

P_t = power used to produce t tons;

S_1 = saving from increased efficiency of power resulting from a reduced ton-mileage of the locomotive;

S_2 = saving per ton from reduction of the load factor;

S_3 = saving per ton due to lowered summits or of rise and fall, acceleration and braking power losses not being considered;

Sp = total saving per ton of coal produced, in power cost resulting from an increased tonnage produced by reducing the limiting grade.

Then:

$E_T=(nG-L)/Gn$ when producing T tons;

$E_t=(ng-L)/gn$ when producing t tons;

$$S_1=(E_t-E_T)KP_T/T;$$

$$S_2=RKP_T/T;$$

$$S_3=(P_t/t-P_T/T)K;$$

$$Sp=[(E_t-E_T)+R]KP_T/T-(P_t/t-P_T/T)K.$$

The following is a typical example of load allowances on a motor assuming a 4000 ft. haul with a stiff grade against the loads the full length: The haul up this grade at normal speed will require 10 to 12 min., under about three-fourths full power;

switching, coupling to empties, etc., will require 1 to 2 min.; the return trip with the empties will take 10 min. with an additional 1 or 2 min. to couple up to the load. The round trip thus requires from 22 to 26 min. During this interval the motor will be under three-quarter load for 12 min., little or no load for 3 or 4 min., and 10 min. on the return trip under about one-quarter load.

At the mines of the Copper Queen Mining Co., in Bisbee, the power used on trolley locomotives, measured at direct-current switchboard in power station, for the year 1912 amounted to 875 watt-hours per useful ton-mile. This amount, however, includes a few lights which are connected to the trolley circuit and gives too high a figure for the locomotives alone. It applies to cars with roller bearings, about one-half of the tonnage being carried in cars of two tons capacity and the other half in cars of one ton capacity. The conditions of the cars and tracks have quite an important bearing on power required per ton-mile.

For the year 1912 the cost of various items in cents per useful ton-mile at Bisbee for a total of 408,000 ton-miles was as follows:

	Cents
Locomotive maintenance	2.95
Car maintenance	1.64
Track maintenance	5.24
Trolley maintenance	3.60
Power	1.64

Locomotive maintenance includes all electrical and mechanical repairs and replacements on locomotives, as well as lubricating oil and supplies.

Car maintenance includes all repairs, oil and supplies on cars.

Track maintenance includes all track repairs and replacements, bonding, grading and realignment.

Trolley maintenance includes all trolley-wire repairs and replacements, and repairs to protective trough around trolley wire.

Track and trolley maintenance are very heavy, due to shifting ground.

The cost of power (1.1 kw-hr. per ton-mile) is taken at the high-tension switchboard and includes the loss in transforming the alternating current into direct current.

In comparing power used by storage-battery locomotives and trolley locomotives it would be fairer either to compare the actual input into battery with direct-current power used by trolley locomotives or use the alternating-current power input to rotary converter in both cases.

In the case of the storage battery the input was approximately 1.28 kw-hr. per ton-mile, as against 0.875 kw-hr. for the trolley locomotive.

The figures for power on storage battery are based on two days' test and therefore are not as reliable as those on the trolley locomotives, which cover a year's period.

The accompanying tables, the figures of which are as of 1917, may be used as the basis of power computation costs in connection with the computation of haulage costs. They are sufficiently close to give the approximate interest and depreciation charges on a deferred expenditure for equipment, made possible by reducing grades and thus lowering the maximum energy demands at peak loads, and for other purposes of an approximate nature:

Number of Kilowatts Required to Haul One Ton Gross Train Load (As Measured at the Trolley Wire)

The total resistance includes the tractive resistance of 20 and 30 lb. per ton on level track and grade resistances up to 4 per cent, inclusive.

Per Cent Grade	Velocity in Miles per Hour: 4 Tractive Resistance, Pounds per Ton. 20	30	5 Tractive Resistance, Pounds per Ton. 20	30	6 Tractive Resistance, Pounds per Ton. 20	30	8 Tractive Resistance, Pounds per Ton 20	30	10 Tractive Resistance, Pounds per Ton. 20	30
0	0.2	0.3	0.25	0.38	0.3	0.45	0.4	0.6	0.50	0.75
0.5	0.3	0.4	0.38	0.50	0.45	0.60	0.6	0.8	0.75	1.00
1.0	0.4	0.5	0.50	0.63	0.60	0.75	0.8	1.0	1.00	1.25
1.5	0.5	0.6	0.63	0.75	0.75	0.90	1.0	1.2	1.25	1.50
2.0	0.6	0.7	0.75	0.88	0.90	1.05	1.2	1.4	1.50	1.75
2.5	0.7	0.8	0.88	1.00	1.05	1.20	1.4	1.6	1.75	2.00
3.0	0.8	0.9	1.00	1.13	1.20	1.35	1.6	1.8	2.00	2.25
3.5	0.9	1.0	1.13	1.25	1.35	1.50	1.8	2.0	2.25	2.50
4.0	1.0	1.1	1.25	1.38	1.50	1.65	2.0	2.2	2.5	2.75

Formula:

$Kw = (Tr \times v \times 0.746)/(550 \times 0.8)$;

Kw = kilowatts used to haul one ton as measured at the trolley wire. (The losses in the line must be added to obtain the total power indicated at the switchboard.)

Tr = tractive resistance in pounds per ton;

v = velocity in feet per second.

NOTE.—The efficiency of the motor and gearing are taken as 80 per cent when running with the controller cut out. With the controller in circuit, the efficiency of the motor will be from 60 to 65 per cent.

AVERAGE COST PER KILOWATT PRODUCED FOR THE VARIOUS POWER-PLANT UNITS

Item	Minimum Cost per Kilowatt Capacity	Maximum Cost per Kilowatt
Boilers and settings	$10.75	$13.25
Stokers	1.30	2.20
Flues, dampers and regulators	.60	.90
Boiler-feed pumps	.40	.75
Feed-water heater	.20	.35
Piping, valves, etc	4.20	7.00
Economizers	1.30	2.25
Foundations for engines	2.00	3.00
Engines	22.00	30.00
Generators	16.60	22.80
Switchboards and wiring	3.20	4.20
Supervision	4.00	6.00
Miscellaneous	2.00	3.00
Total	$68.55	$95.70

A typical round-trip performance of a mine locomotive working actual mine conditions, is indicated by curves shown in Fig. 13. This is a 6-ton locomotive with two tandem motors, rated at 6 miles per hour, with an electrical input of 40 kw. on a drawbar pull of 2400 lb. or 400 lb. per ton.

The starting drawbar pull, or maximum tractive effort, is approximately 150 per cent of that of 3600 lb., with a current consumption of 500 v. of 115 amp. The ragged current and voltage curves are noted to occur during the whole of the round-trip.

The current serves also as a direct measure of the torque, which can be computed from the actual characteristic curves.

As the entry in this mine often runs on the strike of the coal, one side of the track is ballasted and the other rests on bed rock. This means a poor, uneven track, sometimes dirty from loose coal falling off cars or from rock falls. Then, too, wet spots and small increases of grades are often met with, requiring frequent sanding. Although the couplings are long, the grade

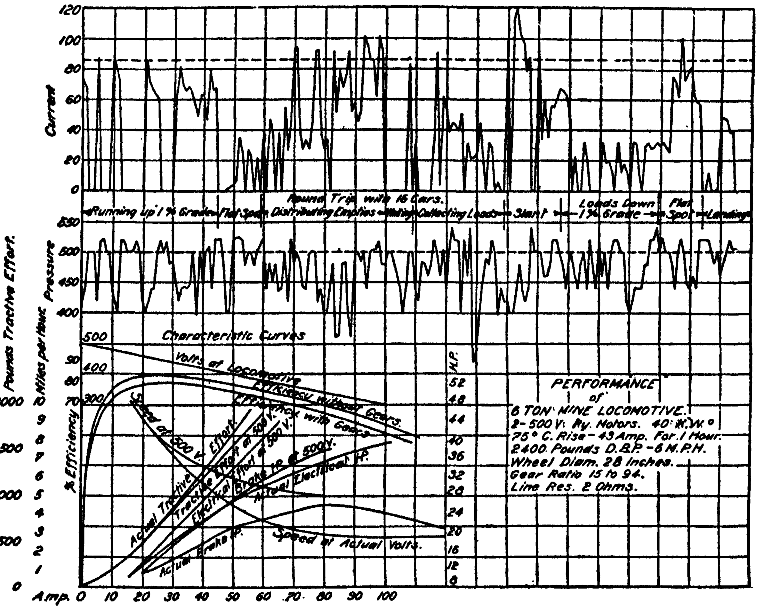


FIG. 13.—Typical round-trip performance of a mine locomotive working under actual mine conditions.

keeps them tight and requires a greater accelerative torque than when running on the level.

As this entry is one of four in a mine about 2 miles from the power station, and each of them has a locomotive, besides using four electric booster fans and an electric hoist, no further explanation is necessary of the apparently inconsistent variation of voltage. It is typical of mine service. The generators at the power house are not overcompounded, and there is, therefore, no compensation for line loss. This results in this particular case in quite a drop of pressure when the electric locomotive is pulling heavily; consequently, the characteristics obtained at 500 v. can not be very well used for correct calculation at 100 amp. consumption. The changes in such characteristics due to two ohms drop in the feeder line and return are sketched in roughly. The decrease in electrical and brake horsepower is very apparent, resulting in great loss of speed at heavy loads, but increasing the tractive effort at this lower speed.

It is interesting to note the enormous difference of performance between an ordinary day trip such as is shown here, and one with very cold weather outside. The empties coming from the outside run with great difficulty because of frozen boxings, and often require more than a 50-per cent increase in torque; by the time they are loaded, however, the mine temperature has thawed them sufficiently to make the outgoing trip a fairly normal one.

Where the inertia of the trip of cars at starting, or increased resistance due to a dirty track, causes the locomotive to pull heavily and run slowly, the effect of a long feed line causing an appreciable voltage drop at the locomotive is of interest. The drop, of course, is proportional to the current consumption. The motor torque is proportional to the current only, and is independent of the impressed electromotive force. With a heavy pull, the motor's speed need not be so great; as an increase in current flow means less speed for the necessary current electromotive force. As the applied electromotive force decreases (due to line loss) and the current electromotive force increases with the greater current flow, the torque, and therefore the current, does not increase as in the case of constant applied voltage. There is, therefore, less danger of overloading the motors. As the tractive effort also is higher at lower speeds, such line resistance allows of larger trips to be started by a certain weight locomotive without an excessive output at the power house. As the trip resistance is decreased after starting, the speed rises with the decreased current and increased applied electromotive force making up for time lost on the heavy pull when starting.

Another advantage of a long-feed line is decreased harmfulness of short-circuits. A short-circuit on this 2-mile line will blow the circuit breaker without causing a violent flashing of the generator brushes, or excessive arching at the contacts of the circuit breaker, since the current cannot exceed 300 amp.

Line costs and losses.—Line losses can best be shown by a practical example. For purposes of demonstration it will be assumed that the haulage road at a certain mine is 6000 ft. long and laid with 30-lb. rail, which are bonded with 00 wire, and bonding caps with cross bonds every 30 ft. There is a substation at the inside end, 250 volt current is used, the trolley wire is 0000 and a 10-ton motor using current at 150 amp., is used. Let it be desired to find:

What will be the drop in volts at 2000 ft., 4000 ft., and 6000 ft. from the substation.

What is the percentage of loss in each case.

What would be the saving, if any, in installing 0000 feeder wire, allowing say 6 per cent interest on the investment and computing the cost of power at 2c. per kw.-hr.

In estimating the drop at distances of 2000, 4000 and 6000 ft. from the substation, it is necessary to calculate the trolley-drop and rail-drop separately and add the results. Then, in order to estimate the possible advantage of installing a feeder-line of the same size as the trolley-wire (0000), it is necessary to calculate further the combined trolley and feeder-drop for the same distances.

The first step in the calculation is to take from an electric-wire table the circular mils of a 0000 copper wire, which is 211,600 circ.mils. The combined trolley and feeder wires, the circular mils is double this amount, or 423,200 circ.mils. Now, to find the copper equivalent of two 30-lb. rails, properly bonded and cross-bonded, assume a rail-to-copper ratio of 11:1; that is to say, take the rail resistance here as 11 times that of copper, for the same sectional area. Then find the combined sectional area of two 30-lb. rails as follows: Since the weight of wrought iron is 480 lb. per cu.ft., and two 30-lb. rails weigh 60 lb. per lineal yard (36 in.), the cubic contents of the two rails is $60/480 \times 1728 = 216$ cu.in. per yd. The corresponding sectional area for the two rails is therefore $216 \div 36 = 6$ sq.in.

Again, since 1 mil is 1/1000 in., and a circ. mil is the area of a circle whose diameter is 1 mil, 1 circ.mil $= 0.7854\ (1/1000)^2 = 0.0000007854$ sq.in. and 1 sq.in. $= 1/0.0000007854 = 1{,}273{,}200$ circ. mils. The sectional area of the two 30-lb. rails in circular mils is therefore $6 \times 1{,}273{,}200 = 7{,}639{,}200$ circ.mils. For a rail-to-copper ratio of 11 : 1, the copper equivalent is 7,639,200÷11= say, 694,500 circ.mils of copper.

It is now possible to calculate the drop of potential for the trolley line, feeder and rail return for each required distance by using the general formula:

$$Drop\ of\ potential = \frac{10.8\ (distance \times current)}{circ.mils}.$$

The drop of potential is expressed in volts and the current in amperes, while the distance is given in feet. Applying this formula to the present case gives for the drop at a distance of 2000 ft. from the substation the following:

$$Trolley\text{-}drop = \frac{10.8\ (2000 \times 150)}{211{,}600} = 15.3\ volts.$$

It is evident that the combined trolley- and feeder-drop would be one-half of this amount, or 7.65 volts, since there are two wires of equal size for the transmission of the current to the locomotive. The rail-drop is:

$$Rail\text{-}drop = \frac{10.8\ (2000 \times 150)}{694{,}500} = 4.6\ volts.$$

The above results give the drop of potential for a distance of 2000 ft. For distances of 4000 and 6000 ft. the drop will be two and three times the above amount respectively.

A comparison of these results gives the following:

Total drop without feeder,	4.6+15.3 =	19.9 volts
Total drop with feeder,	4.6+ 7.65=	12.25 volts
Voltage saved by feeder,	19.9−12.25=	7.65 volts
Wattage saved by feeder,	150× 7.65=	1147.5 watts

The percentage of drop in each case above is:

Without feeder (2000 ft.),	19.9×100÷250=8.0%
With feeder (2000 ft.),	12.25×100÷250=4.9%

Assuming 250 working-days of 8 hr. each (2000 working hours) in a year, the saving in that time at a cost for electricity of 2c. per kw.-hr. would be 2000×1.1475×0.02=$45.90.

Against this saving must be reckoned the interest or fixed charges on the investment for the erection of 2000 ft. of feeder wire. The weight of 0000 copper wire is practically 640 lb. per 1000 ft. making the costs for 2000 ft. of this wire, at 20c.

per lb., $2 \times 640 \times 0.20 = \$2.56$. The cost of erection, including poles, will be about $4 per 100 ft., or $80 for a distance of 2000 ft., which makes the total cost of this length of feeder wire erected, $336.

Estimating the fixed charges on this investment as, interest, 6 per cent; depreciation, 4 per cent, and taxes and insurance, 2 per cent, making a total of 12 per cent, gives $0.12 \times 336 = \$40.32$. This estimate makes the net saving per year, $\$45.90 - 40.32 = \$5.58$.

Since the drop in potential the saving in kilowatt-hours and the fixed charges are all proportional to the distance, the net saving per year will also be proportional to the distance, or $11.16 for a distance of 4000 ft., and $16.74 for a distance of 6000 ft.

In actual practice, however, the locomotive will not be running more than from one-half to three-quarters of the actual time, depending on the length of the haul and other conditions, which will reduce the saving in kilowatt-hours at the same rate, while the fixed charges must remain constant. Although the figures would then show an excess of fixed charges over the saving in the cost of electricity, it would still be good practice to add the feeder wire. The gain in the operation of the locomotive will exceed what the actual saving in power would indicate.

Grounding losses can sometimes be discovered on the ammeter when the mine is shut down or at any other time when all the power is supposed to be off. In one instance an investigation of a 75 amp. reading of the ammeter when the mine was supposed to be closed for the day disclosed the fact that 25 trolley wire hangers were grounded. When the trouble was remedied the ammeter read zero.

A simple calculation will show the loss from this leakage of 75 amp. under a pressure of 275 volts, being continuous for the time the switch on the supply current is open, which is 16 hr. a day. This loss in six months of 125 working-days of 16 hr. each, or say 2000 hr., would amount to $(75 \times 275 \times 2000) \div 1000 = 41{,}250$ kw.-hr. At a cost of 2c. per kw.-hr., the loss would be $41{,}250 \times 0.02 = \$825$ in six months. In the present case, the loss being due to the leakage of current by the grounding of 25 trolley-wire hangers, the loss is $825 \div 25 = \$33$ per hanger in six months, or $5.50 per hanger per month—an amount that is almost inconceivable. However, it shows the importance of carefully testing an electric circuit for grounds and locating and stopping the leak. Besides this direct loss of current, there is the danger from fire where the hangers are in the coal roof or insulator pins are used instead in the entry ribs.

The following is an approximate estimate of the cost of 1000 ft. of trolley construction in coal mines, including rail bonds as estimated in 1908:

36 single-bolt roof suspensions	$15
30 straight ears for grooved wire	6
6 curved ears for grooved wire	6
1000 ft. of 0000 grooved trolley wire (641 lb.)	94
210 ft. of 00 bond wire (85 lb.)	12
175 ft. channel pins, GE Catalog, No. 17,315	5
Labor and other material	50
Additional labor, if necessary, to drill rails for bonds with a hand drill	15
Total cost per 1000 ft	$197

Other sizes of trolley and bond wires may be readily substituted in the above estimate.

Bonding.—While a few operators are using modern methods of bonding, many are depending on channel pins and wire to carry the return circuit. The channel pin when first installed is more or less efficient, but as three-fourths of its contact is between steel pin and steel rail, it is impossible to obtain a union that will exclude air and moisture. Therefore it is only a short time until corrosion has started and a high resistance is introduced at the points of contact. The method of testing prevalent at most mines consists of examining the bond to see that the wire and pins are intact.

The return circuit of a mine that was bonded partly with channel pins and partly with compressed-terminal flexible-cable bonds was tested with a direct-reading bond tester which showed the resistance of each joint as equal to the resistance of a certain length of solid rail. Thirty-one per cent of the channel-pin bonds showed a resistance equal to, or greater, than that of a 30 ft. rail, or practically an open joint; the average resistance of the balance was equal to that of 13 ft. of rail. These channel-pins had been installed about 2½ yr. and on an exceptionally good roadbed.

The compressed terminal bonds had been installed 4 yr. on a roadbed that was in bad condition. The drainage was bad and the soft roadbed permitted a considerable rising and sinking of each joint when a car passed over, thus imposing a severe strain on the bond terminals; 16 per cent. of these bonds were found defective, and the balance showed an average resistance of 6.6 ft. Had these bonds been installed in track similar to that in which the channel-pins were used there is no doubt but that the depreciation would have been cut in half.

A point of special interest in this test was the fact that the majority of the compressed-terminal bonds were in good condition after 4 yr. of service and under rather unusually unfavorable conditions. Had this company tested these bonds at certain intervals, and replaced defective ones as found, they would have had, at a small expense for labor and material, a highly efficient return circuit at all times.

This operation had been suffering from an excessive drop in voltage, and the results of this test proved conclusively that it was caused by a defective transmission of the return current.

The relative resistances of 21 joints selected at random along the haulage system of this mine, bonded first with channel-pin bonds then with compressed-terminal bonds, are clearly shown in the accompanying curve, Fig. 14.

One mine manager reported that, with sufficient copper overhead, he found a drop of 100 volts, at less than 3000 ft. from the generator on 250-volt direct-current circuit. The channel-pin bonds in this mine were tested, and 90 per cent of them found to have a resistance greater than that of 30 ft. of rail. The channel-pins were replaced by compressed-terminal bonds, the line voltage went up to normal and the efficiency of the locomotive increased to a marked extent.

Computing losses in bonds.—It is essential, both in making tests of bond installations, and in estimating new work that the resistance of a bonded joint be known. The following tables and formula show just what resistance a well bonded joint will have.

To find the resistance of a rail joint bonded with compressed terminal bonds, the following formula is used:

$$\frac{L \times R + 2 \times CR}{RF} = JR$$

in which L = length of bond in inches;
R = resistance of one inch of cable or strands composing the bond;
CR = Contact resistance of bond terminals;
RF = Resistance of one foot of rail;
JR = resistance of bonded joint expressed in feet of rail.

As most bond-testing instruments show the resistance of the bonded joint, as compared with the equal resistance of a certain

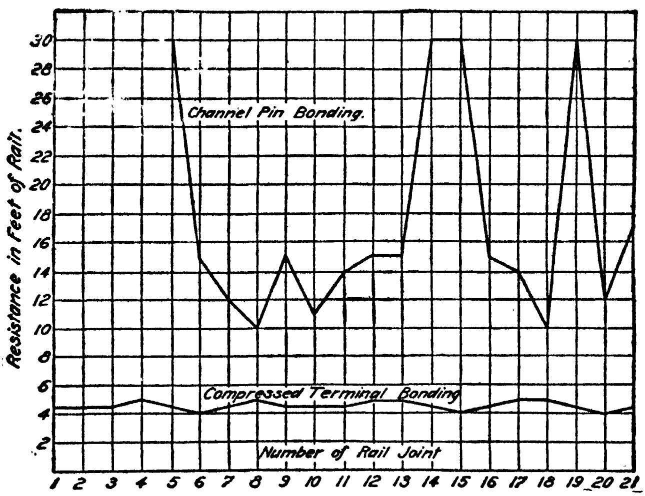


FIG. 14.—Results of 21 tests of channel-pin and compressed-terminal bonds.

number of feet of unbroken rail, it is more convenient to express this value in such terms than in ohms.

To illustrate the use of the above formula and tables, assume that a 40-lb. T-rail is to be bonded with compressed terminal flexible cable bonds, 4/0 capacity, ¾-in. terminals, 26 in. in length. The resistance of each joint when the bond has been installed is desired. By referring to the tables, the resistance of one inch of 4/0 cable is found to be 0.00000414 ohm, and the resistance of a 26-in. bond is 26 times 0.00000414, or 0.00010764. The resistance of a ¾-in. terminal is 0.00000053 ohm, and of the two terminals is 0.00000106 ohm; adding the cable resistance and the contact resistance the total ohmic resistance of the installed bond is 0.0001087 ohm. To express this in terms of

Resistance and Carrying Capacity of Rails

Figures Based on Rails Having a Ratio of 12 to 1, as Compared with Copper, and at 70° F.

Weight, Pound per Yard	Resistance, Ohms per Foot	Carrying Capacity in C.M.
16	0.0000622	169,764
20	0.00004923	212,206
25	0.00003935	265,257
30	0.00003321	318,309
35	0.00002844	371,360
40	0.00002489	424,412
45	0.00002212	477,463
50	0.000019355	530,515
60	0.0000166	636,618

Resistance of Solid Terminals

Figures Based on a Pressure of 15 Tons per Square Inch of Contact Surface

Diameters	Resistance, Ohms
½″	0.0000008
⅝″	0.00000064
¾″	0.00000053
⅞″	0.00000045
1″	0.0000004

Resistance of Bond Cables per Inch of Conductor at 75° F.

Size	Resistance, Ohms per Inch	Capacity in Amperes
1/0	0.00000829	210
2/0	0.00000657	265
3/0	0.00000521	335
4/0	0.00000414	425
250,000 C.M.	0.0000035	500
300,000 C.M.	0.00000275	600
350,000 C.M.	0.0000025	700
400,000 C.M.	0.00000219	800

equivalent rail length, divide by the resistance of 1 foot of 40-lb. rail, and the resistance of the bonded joint will be found to equal that of 4.2 ft. of unbroken 40-lb. rail.

Trials made with standard bond-testing instruments show that when bonds are installed with reasonable care, the resistance of the joint will very closely approximate this calculated resistance. Abutting rail ends and clean tight splice bars may slightly lower the resistance of the joint, but this is quite negligible and should be disregarded. Numerous tests made of complete haulage roads, installed under usual mining conditions, show that an average will vary but a fraction of a foot from the calculated resistance.

The use of the tables will not only be found of benefit in learning standards to test individual joints for their efficiency, but they can be used to great advantage in figuring voltage drop on proposed work. In calculating voltage drop on a circuit composed of a trolley and a rail return, a formula is generally used which is correct for the copper loss, but does not take into consideration the weight of the rail and the size and length of the bonds. By calculating the voltage drop on the trolley and feeders only, and then on the rail, when properly bonded, the exact drop for a given load is secured. This method has been found particularly advantageous where a potential of 250 volts is used and the current transmitted over long distances, as is common with bituminous mines.

For instance, taking the example mentioned above, assume that a road 3000 ft. long is to be bonded, and the actual drop on the return side of the circuit is desired. With 30-ft. rail lengths, there will be 100 joints on one rail, having a total resistance of 420 ft. About 10 per cent should be added to the joint resistance to take care of short rail lengths and bonding at switches.

Then the resistance of one rail will be equal to that of 3460 ft. of unbroken rail, or 0.00836394 ohm, or for the two rails in parallel 0.00418197 ohm. The voltage loss on the return side is the load times the resistance, then by calculating the drop on the trolley side, the exact drop on the circuit is easily ascertained.

In the same manner, these tables can be used to test the efficiency of the bonding of an entire haulage road on a certain section. With voltmeters at both ends of a section and an ammeter on a locomotive, the voltage drop at a certain load can be obtained. It is a simple matter to calculate the drop on the positive side of the circuit and on the return side, assuming the bonds are in good condition. Any difference found will be an increased joint resistance, and the individual bonds should be tested with a bond tester and the defective ones replaced.

A number of companies have adopted this method, as an entire mine can be gone over on an idle day or night, and the individual joints need only to be tested in those sections which show defective bonding. Mine operators who are desirous of obtaining efficient and economical results from electrical mining equipment will find it well worth their while to make such tests following them up with any necessary repairs.

One company recently tested in this manner a newly bonded road. The test showed that the joint resistance was 69 per cent of the total circuit resistance, where it should have been only 8 per cent. Upon making an examination of the road, it was found that the track men had neglected to install bonds at two of the switches.

As their load was heavy, this unnecessary resistance would have cost a considerable sum in a month's time for power, which in this instance was purchased.

The following is another example of computing bonding losses it being assumed that it is desired to know what is the difference in resistance between a 4-O round copper wire 3000 ft. long and the two 25-lb. steel rails, in a track, 2000 ft. long, followed by two 30-lb. steel rails, in 1000 ft. of track. The rails are bonded with pressed-terminal all-wire 2-O bonds.

Also determine how many kilowatt-hours will be available at the end of a 250-volt line, where a 30-hp. motor, 3000 ft. from the generator, is taking 100 amperes.

The resistance of 1000 ft. of a 4-O copper wire at, say 68 deg. F. (20 deg. C.), as taken from a table giving the resistances of copper wires for different gages and temperatures, in ohms per thousand feet, is 0.04893 ohm. The resistance for such a conductor 3000 ft. long is then, $3 \times 0.04893 = 0.14679$ ohm.

An approximate rule for calculating the resistance of copper wire per thousand feet, in ohms, is to divide 10,000 by the size of the wire in circular mils. Thus, for a 4-O wire (211,600 circ.mils), the resistance is, approximately,

$$R=\frac{10{,}000}{211{,}600}=0.04726\ \textit{ohm per}\ 1000\ \textit{ft.}$$

This rule should only be used in rough calculations. When accuracy is desired, the resistance for the wire should be taken from electrical tables, as above stated.

The resistance of steel rails, for the same cross-section and length, varies with the composition of the steel. The presence of sulphur and manganese, particularly the latter, greatly modifies the resistance of the steel. It has been found that this resistance will vary from about eight to thirteen times that of copper of the same sectional area and at the same temperature. This has given rise to what is termed the "ratio of rail to copper" or the "rail-to-copper ratio." The results of numerous experiments have made it possible to calculate the equivalent circular mils of copper corresponding to any given weight of rail in pounds per yard, for any rail-to-copper ratio. To do this, the weight of rail, in pounds per yard, is multiplied by the constant corresponding to this ratio, as determined by the composition of the steel. The value of this constant, for the several ratios, is as follows:

Rail-to-Copper Ratio	Constant	Rail-to-Copper Ratio	Constant
8	15,550	11	11,360
9	13,820	12	10,360
10	12,500	13	9,590

Applying this method and assuming a rail-to-copper ratio of 10, the constant for this ratio, as taken from the above table, is 12,500. Then, for a 25-lb. rail, the equivalent circular mils of copper is $25 \times 12{,}500 = 312{,}500$. Electrical tables are not generally extended to include as large a wire as this area indicates. The resistance in ohms per thousand feet, however, can be calculated, approximately, by the rule previously given. Thus, $10{,}000 \div 312{,}500 = 0.032$ ohm. The resistance of the two rails, in the first 2000 ft. of this track, is the same as the resistance of a single 25-lb. rail 1000 ft. long, or 0.032 ohm.

Again, for a 30-lb. rail of the same composition, the equivalent circular mils of copper is $30 \times 12{,}500 = 375{,}000$. The corresponding resistance is, therefore, approximately, $10{,}000 \div 375{,}000 = 0.0267$ ohm. The resistance of the two rails, for 1000 ft. of track, is one-half of this amount, or 0.01335 ohm. The total rail resistance in this track is, therefore, $0.032 + 0.01335 = 0.04535$ ohm; and the difference, in favor of the iron rails, is $0.14679 - 0.04535 = 0.10144$ ohm.

The diagram, Fig. 7, taken from the Ohio Brass Co.'s catalog, shows graphically the circular mils of copper, of equal electrical resistance to steel rails of different weights and "rail-to-copper ratios." The curved lines show the resistance, in microhms, of steel rails of different weights and ratios.

The diameter, in inches, of a copper wire that is the electrical equivalent of a steel rail of a given weight (lb. per yd.) may be calculated by multiplying the respective constant taken from the above table, by the weight of the rail, extracting the square root of the product and dividing that result by 1000 Thus, for a rail-to-copper ratio of 10, the constant is 12,500. Then, the diameter of the copper-wire equivalent is:

$$d = \frac{\sqrt{25 \times 12{,}500}}{1000} = 0.559\ in.$$

As regards the second problem, a 30-hp. motor consumes $30 \times 746 = 22{,}380$ watts or 22.38 kw. If this motor is taking, as stated, 100 amp., the voltage, at the full capacity of the motor, is $22{,}380 \div 100 = 223.8$ volts. The drop in voltage for this line is, therefore, $250 - 223.8 = 26.2$ volts. The work performed by this motor, in each hour, when working at its rated capacity, is 22.38 kw.-hr.

Tests made at a colliery in Pennsylvania showed that the bonding was so bad that the return current was leaving the rails and finding its way to the generating plant by way of the ditches and water pipes, which, of course, had a high resistance.

The locomotives with a rated speed of 6.3 miles per hour were found to be traveling at about 2.5 miles and making an average of 12 trips per day. The actual load on the generator was 30 per cent over the rated load on the line. The main haulage roads were bonded with compressed terminal bonds of sufficient capacity to equal the size of trolley and feeders.

The first month following this installation the production was the largest in the history of the mine. The locomotives instead of making 12 trips per day were averaging 18 trips. Later another locomotive was added and the whole load on the generator was less than that carried before the bonding was changed.

At another colliery the bonding resistance of the main haulage road was reduced 80 per cent by the use of compressed terminal bonds. During the last six months of the channel-pin installation the track bonder had averaged three days per week on this road replacing broken and defective bonds. During the first seven months of the new installation the bonder spent two hours on the road replacing some bonds broken by a wrecked trip.

Investigation of bad haulage conditions at another mine disclosed a reading on the voltmeter of 180 volts on a 250-volt circuit at a point 6000 ft. from the mine mouth. A 10-ton motor was used on the main haulage in to this point, beyond which there were three motors engaged on secondary haulage. When the main haulage motor started out with a trip the current would drop to 80 volts and remain at this for 8 to 10 min. during which time the gathering motors could not move. It was estimated that the haulage charges at this mine were increased about 10c. per car or 4c. per ton from these causes which on the basis of a working schedule of 20 days per month would entail a loss of $800 per month.

Table Showing Cost of Energy-Loss in Return Circuit per 100 Amperes Load, per Year of 240 9-Hour Days, Power Costing One Cent per Kilowatt-Hour (1913)

One Mile. Two 42-lb. rails in parallel, assuming continuous joints	$15.78
One Mile. Two 42-lb. rails in parallel, bonded with channel-pin bonds, resistance found by actual test	37.80
One Mile. Two 42-lb. rails in parallel, bonded with compressed terminal bonds, resistance found by actual test	17.99
Fixed cost of rail resistance	15.78
Increased cost with compressed bonds	2.21
Increased cost with channel-pin bonds	22.02
Saving per year with compressed bonds for power alone per 100 amperes	19.81

A method to overcome bonding troubles and to reduce the bad effects to a minimum is to put in ground wires along the track and to tap these, at suitable distances, to the rails as shown in Fig. 15; this insures the bonding and provides an uninterrupted metallic circuit. The high price of copper practically prohibits its use for this purpose.

Wornout wire hoisting ropes, in lengths of 500 ft. and over, which make few joints necessary, having a scrap value of approximately $8 per ton (as of 1912) and a specific resistance of 8 to 1 as compared with copper, may be utilized for this purpose. It is a better conductor than rails and a rope of,

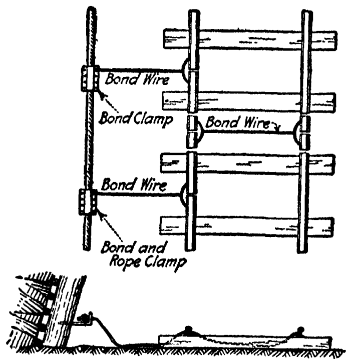


FIG. 15.—Method of improving the bonding by the use of old wire cable.

say, 1½ in. diameter has about the same current-carrying capacity as a No. 000 B. & S. gage, or about a ⅜-in. copper wire. Smaller ropes may also be used, the size required depending upon the conditions.

The wire rope may also be used where ground wires are required, as between substations and the mines. For this work, copper cables 1 in. in diameter are quite frequently used. An installation requiring 1000 ft. of such copper cable was estimated at approximately $550 in 1912, and the same result would be obtained with wire rope at a cost approximating $100. For such outside work 1¾-in. and 2-in. ropes, which cannot be handled very well inside, may be utilized. The installation should be made in the following manner:

From the substation to the entrance of the mine, where no rails are used, 1¾-in. or 2-in. rope, either single or in multiple, according to the load on the circuit and the distance, should be used. Along the track in the main gangway $1\frac{1}{2}$-in. rope, and for extensions $1\frac{1}{4}$ and 1-in. single ropes are suitable. The rope is stretched out by attaching it to a mine motor or locomotive and in this manner several thousand feet may be laid in a few hours. As the ropes are seldom shorter than 500 ft., only a small number of joints, which should be made with substantial 10-in. clamps, are necessary. Sheet-lead lining should be used between the clamps and rope, thus providing a reliable contact. One of the bolts used with the clamps serves as a bond terminal. Where the rail taps make connections with the ropes, smaller clamps are used at distances of about 200 to 300 ft. It is advisable, in order to increase the durability, to put the rope on the high side of the track to keep it out of the water as much as possible.

The efficiency of the return track, either single or double bonded, and provided with such a ground wire, will be very little affected by a few defective bonds, and the practical results will be far superior than with the regular bonding only. On account of the stable conditions obtained with the ground wire, in many cases single bonding with some cross taps between the rails will be sufficient; where distances are short and the load light, satisfactory results will be attained by the ground tapped to the rails at suitable distances without the regular bonding. The inspections of the track in regard to bonding will also be reduced to a minimum.

At one operation where three $8\frac{1}{2}$-ton electric mine locomotives were working there was considerable complaint due to the inefficiency of these motors and in fact a request was made for an additional motor. An examination of the place showed that there was lost time for power, a large amount of sand was being used on the track and trolley wheels did not last longer than a day or two. The track at this operation was laid with 40-lb. rail at the bottom of the shaft and 30-lb. rail in the gangways. The bonding was about equal to the average condition of bonding in mines.

As promptly as possible, 7500 ft. of wire rope was installed, 2000 of which was $1\frac{1}{2}$-in., 4000 $1\frac{1}{4}$-in. and the balance 1 in. in diameter. The results were entirely satisfactory. The motors could then do good work, the amount of sand used on the track has been reduced 65 per cent, which also reduces to a minimum the wear on the motor wheels. There is no further complaint regarding power or poor trolley wheels.

Track costs.—The determining factors in fixing grades in the mines are drainage, haulage and sometimes loading. The grade sought after in the mines of this country is 4 to 6 in. per 100 ft. There is a theoretic grade at which the inclination of the haulageway will compensate the added resistance due to loading the cars and thus equalize the load on the motor for both empty and loaded trips. When this condition has been realized the motor is able to handle the same number of loads and empties going in their respective directions.

The greater the difference in weight between the loaded and empty car, the steeper should be the grade in favor of the load, to overcome the extra resistance. Consequently whenever a larger capacity mine car is adopted, a corresponding change in the standard grade should be made for all haulageways not influenced by other considerations. This, however, is seldom done, the original grade being maintained. On haulageways where the 0.35 per cent grade is used with cars of 3 tons capacity, the number of empties it is possible for the motor to haul invariably exceeds the number of loads.

An all-steel car (115 cu. ft. capacity at water level full) empty weighs 5080 lb. and loaded with coal weighs 12,230 lb., while loaded with rock it would weigh 17,000 lb. Assuming the sum of the resistance due to friction and track to be 30 lb. per ton for an empty car and 25 lb. per ton for a loaded car, the coefficient of friction per short ton for loaded and empty cars will be 0.015 and 0.0125 respectively.

With an 8-ton motor having a rated drawbar pull of 3000 lb. on the level, letting $N=$ the number of cars and $g=$the theoretic grade for equalizing the drawbar pull, then when the motor is handling coal-loaded and empty cars:

$$\frac{3{,}000+16{,}000g}{(12{,}230\times 0.0125)-12{,}230g}=\frac{3{,}000-16{,}000g}{(5080\times 0.015)+5080g}$$

$$g=5 \text{ in. per } 100 \text{ ft.}$$

With the same motor handling empty and loaded rock cars:

$$\frac{3000+16{,}000g}{17{,}000\times 0.0125-17{,}000g}=\frac{3000=16{,}000g}{5080\times 0.015+5080g}$$

$$g=7 \text{ in. (almost) per } 100 \text{ ft.}$$

The foregoing is a somewhat involved quadratic equation not always easy of solution. A method which is perhaps simpler is as follows:

Let θ = desired angle of grade to give equal drawbar pull in both directions, then the percentage of grade, or the rise in unit length of track, is practically equal to sin θ.

If W = weight of empty car;
W' = weight of loaded car;
C = coefficient of friction for empty car;
C' = coefficient of friction for loaded car;

then $$WC+W \sin\theta = W'C' - W' \sin\theta.$$

Substituting values in the above case

$$(5080\times0.015)+5080 \sin\theta = (12{,}230\times0.0125)-12{,}230 \sin\theta$$

$$76.2+5{,}080 \sin\theta = 152.875-12{,}230 \sin\theta \quad 17{,}310 \sin\theta = 76.675$$

$$\sin\theta = \frac{76.675}{17{,}310} = 0.00443$$

Multiplying by 100 to secure the rise in 100 ft., we have 0.443 ft., or about 5¼ in.

With straight track, then, the 8-ton motor on a grade of 7 in. per 100 ft. could pull in 28 empty cars and return with the same number loaded with rock; but as the resistance of the curves, of which a major portion of every gangway consists, is a corollary of the weight, and determined grade should be augmented somewhat in favor of the loaded cars to partly compensate for curve resistance.

The value of the track resistance per ton on straight track will have to be determined experimentally for any type of car. Equipping the cars with patent bearings will have salutary effects, should this resistance prove too high.

On a recent test of nine cars, three each of the three various types ordinarily employed, on fairly clean straight track, on the surface, with cars weighing 12,230 lb. each; loaded with coal, capacity 115 cu. ft. loaded water level; wheel base, 3 ft. 6 in.; gage of track 3 ft. 6 in.; 40-lb. rail; both wheels tight on the axle; center of drawbar pull to top of rail, 18½ in., the tractive effort required was as follows:

21.2 lb. per ton for cars with 160 deg. iron case, babbitt lined; 10.5 lb. per ton for cars with 160 deg. brass lining; 6.5 lb. per ton for cars with Hyatt bearings.

The cars were of all-steel construction identical in every respect, except the bearings. The brass-bearing cars were new and in first-class condition, while the Hyatt and babbitt-bearing cars had been in use almost 2 yr. The results, if secured on tracks underground, in their usual dirty condition and with inferior ballast, would no doubt have been higher.

A contingency not always considered in planning a haulage is that, since the abandonment of mule haulage, the loaders are compelled to move the cars at the loading chutes by their own exertions. Four cars per trip are not infrequently loaded from one chute. On the prevalent 0.35 per cent grade to move the cars the distance necessary to accomplish this would require an additional man. To dispense with this, the foreman resorts to increasing the grade immediately beneath the chutes with an equalizing diminution between them. This expedient permits loading the cars, but makes the roadbed a series of undulations with pools of water in the "dead spots." It raises the haulage cost, destroys the rolling stock and sets the cars bumping and jerking over the entire working section. It is needless to mention that a heavier grade would abate, if not wholly remove, this condition.

To be in congruity with the preceding, a grade of 7 to 8 in. per 100 ft. for the haulage and up to 8½ in. for the primary gangways would seem to have more to recommend it than the lesser grade. To be sure, the first installation of the heavier grade would reduce the available lift as the gangway advanced, but this objection would remedy itself in all subsequent levels.

Again, a ditch averaging 18 in. wide edge to edge and 6 in. deep in the center, on a 0.35 per cent grade, would give 38.4 ft. velocity and almost 108 gal. capacity per minute, against 54.4 ft. and 153 gal. for an 8½-in. grade.

If an analysis of any haulage problem indicates that a saving is possible by cutting down grades, it becomes at once necessary to determine the approximate amount that can be profitably spent on the proposed improvement.

The advantages accruing from the improvement will be: (1) Reduction in general labor costs; (2) reduction of power cost per ton hauled; (3) reduction in general expense, including the saving made possible by the postponement, either temporarily or permanently, of expenditures for additional equipment such as motors, generating units, engines and boilers.

If a haulage motor pulls a certain number of extra cars per trip as a result of a grade reduction, the total number of tons produced at the mine will be increased without a proportionate increase in the expense. This saving is expressed by a formula in which let:

t = tons produced per shift before increasing the tonnage;
T = tons produced per shift after increasing the tonnage by pulling extra cars per trip;
c = labor cost to produce t tons, in dollars;
C = labor cost to produce T tons, in dollars;
S = saving per ton in dollars resulting from the increased tonnage.

We then have the formula:

$$S=\frac{c}{t}-\frac{C}{T}.$$

For example, a mine with one main-line haulage motor making 16 trips in 8 hr. from two partings produces 950 tons per day at a cost of \$145.38 for inside labor and \$52.69 for outside, making a total of \$198.07.

Investigation of the haulage profile shows that by reducing the grade on one of the runs the motor will be able to handle 1000 tons per day. To handle this increased tonnage an extra driver inside will be required and a track layer, one hour per shift, bringing the total labor cost including full allowance for feed, car and depreciation of the mule up to \$202.45. Subtituting these values in the above formula, we have:

$$\frac{198.07}{950}-\frac{202.45}{1000}=0.6c.=\text{the saving per ton.}$$

When the summit in a grade is lowered, the power required to overcome grade resistance is less and by increasing the number of cars per trip less power is required per ton of coal hauled since the proportionate ton-mileage of the motor itself is reduced. The weight of the motor inbound may be about one-third of the total weight of the trip, and outbound about one-fifth, so that it is evident that considerable power is consumed in moving it alone.

A heavy pull exerted on a stiff grade will tend to increase maintenance and repair charges for both rolling stock and track. On the other hand a reduction in the grade may result in an increase in the car repair bill, due to the greater number of cars handled, but the charges per car-mile and the maintenance per ton hauled will remain the same and will not effect the unit cost per ton of coal produced.

The elimination of bad grades also reduces the hazard to operations and though the danger to life is not directly calculable, it is of vital importance and must not be overlooked in considering the possibility of any proposed improvement; in other words it is simply applied safety and could properly be included under the charge for insurance.

If the advantages resulting from a certain grade elimination can be reduced to cents per ton handled over the section of track improved and this is multiplied by the number of tons so handled, the result will be the amount that may be expended on the proposed work. Or expressed in a formula, let:

y = the estimated number of tons available;
C = cost of grade reduction, including interest on money invested;
S = summation of all savings, in cents per ton;
V = value of safe operation.

Then $(yS + V)$ should be equal to, or greater than, the value of C. If the value of yS (total saving in dollars) is less than the cost of the improvement, and the value of safe operation does not, in the opinion of the management overcome the difference, then the project should be abandoned. As a matter of fact the judgment of the financier must be relied upon throughout the whole study of the question. False assumptions may be made in some cases, leading to erroneous conclusions, but by following carefully the steps indicated, it is possible to make a fairly accurate estimate of the results to be obtained in any contemplated work of this description.

Haulage grading estimates.—When all the data for a prospective change of grades has been assembled, the estimated cost of the various plans, routings and schemes should be made for purposes of comparison. An intelligent estimate of cost must consider detail and be based on accurate knowledge of the proposed requirements, together with the application of the unit costs of similar work formerly completed. A con-

FORM No 1

ABC COAL COMPANY

General Estimate ______________ Proposed Extension of Motor Road

______________ Track and Construction of ______________ Ft. Parting ______________

MINE No. ______

ITEM	Material Req'd	On hand or credited	Mat'l to be bought	Unit Price	Material Cost	Labor Cost
Tons Rails						
Laid						
Taken up						
Tons Rails						
Laid						
Taken up						
Kegs Small Spikes						
Size						
Kegs Motor Spikes						
Size						
Motor Ties						
Size						
Small Ties						
Prs Fish Plates						
No Rails						
No Rails						
Kegs Track Bolts						
No. Rails						
No. Rails						
Bonds						
Bonding Caps						
Bonding Sleeves						
Frogs and Switches						
No. Rails						
No Rails						
2/0 Trolley Wire						
4/0 Trolley Wire						
Hangers and Clamps						
Wire Splicing Sleeves						
Trolley Frogs						
Automatic Cut-out Switches						
Trolley Wire Guards						
Insulated Telephone Wire						
Porcelain Insulators and Pins						
Material to be Delivered						
Cleaning Gob, Etc.						
Grading.. ..... .Yds. Top						
Grading. .. ..Yds. Bottom.						
Setting Props						
Setting Timber Sets and Cross Bars						
Misc. Expense						
Total Estimated Cost						

Credits of Material not applied to estimate.

Tons ________ Rails ______________

Small Ties ______________

Total Value Credits to be Deducted

Grand Total Estimated Cost ______________

The extension of this Motor Road will ______________

Estimated by ______________

Approved by ______________ Date ______________

FIG. 16.—Form for assembling estimate of cost for grade revision in a motor road.

venient form for assembling an estimate of cost of a motor road extension or revision is shown in the form Fig. 16. The detail required is not exacting, yet a fairly complete record is indicated. When it is finally decided to carry through the work a final estimate is made and a copy bearing the official signature of approval sent forward to all departments concerned. ·The record is thus made complete.

Economy in the execution of the work will depend on the evolution of a systematized method and a strict adherence to two fundamental principles of good management: First the

FORM No. 2

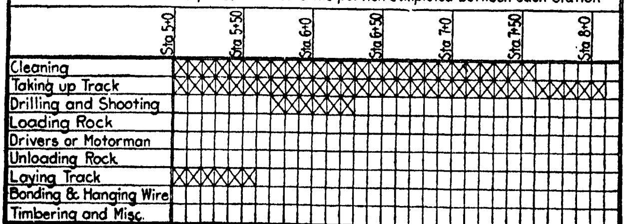
DAILY TIME AND PROGRESS CHART
MOTOR ROAD CONSTRUCTION

Mine No. 3 Estimate No. 410 Date 4-6-15

Check No.	Name	RATE	Cleaning	Taking up Track	Drilling & Shooting	Loading Rock or Mucking	Drivers or Motorman	Unloading Rock	Laying Track	Bonding & Hanging Wire	Timbering & Misc	MATERIAL RECEIVED No.	Item
			Hours	Hours	Hours	Hours	Hours	Hours	Hours	Hours	Hours	No.	Item
112	Tom Smith	3.00		8								2	Kegs Motor Spikes
158	Jack Leid	2.75	8									100	Pairs Fish Plates
143	Wm Ruth	2.75	8									100	30# Rails
146	Mike Shosky	3.00	8									375	Motor Ties
79	Tony Rolito	3.00			8								
27	Harry Thomas	3.00			8								
63	Joe Woods	2.75	8										
102	Jake Myers	2.75	8										
28	Frank Harris	2.75	8										
96	James McDonald	2.75	8										
173	Tim Davis	2.75	8										
174	R.L. Davis	2.75	8										
189	John Tulo	2.75	8										
201	Sam Mills	3.00		8									
48	J. Skolinsky	3.00		8									
51	Tom Tulo	3.00							8				
52	R. Tulo	3.00							8				
41	John Small	3.00		8									
42	E. Stewart	3.00					8						

PROGRESS CHART

Note: Place a cross in the squares to indicate the portion completed between each station



Correct Tom Smith Foreman.
Checked Sam Evans Supt.

FIG. 17.—Progress chart for use in analyzing grade revision costs.

number of men employed in any one section in any period shall be adjusted to the amount and classification of the work to be done and, secondly, only experienced men should be employed.

A careful study of the progress made from day to day will plainly show any weak points in the organization and will often indicate a probable remedy. Progress charts, coupled with the report of the daily time, are useful in judging the comparative efficiency of the organization as a whole or in part and also gives a record of the unit costs for making future estimates and comparisons. The accompanying form, Fig. 17, shows a good method of accomplishing this. The progress for any particular section can be indicated in the daily report by filling in the squares of the progress report at the bottom of the form with crosses to express the estimated amount of work completed in that section. Thus a daily chart can be sent to the administrative department as a record of both the work done and the efficiency of the organization.

Haulage track curves.—Viewed solely from the haulage standpoint, the determining factors of the curve radius can be covered by two heads:

1. The cost of resistance due to curvature on the total estimated number of cars that can be hauled.

2. The probable number of cars to be hauled in each trip and the speed of haulage.

The amount of resistance due to curvature varies with each type of car, and to a lesser degree with each car of a certain type. This resistance expressed in terms of grade, with curves of from 30 to 100 ft. radius, will run 0.015 ft. to 0.025 ft. per 100 ft. of track for each degree of curvature. That is, with a 50-ft. radius, or 115-deg. curve, moderately clean track, fair running cars with both wheels keyed on the axle, approximately a 1.8 per cent down grade would be necessary to secure the same drawbar pull as on a tangent. The value of a curve expressed in degrees can be obtained by dividing 5730 by the radius in feet. This formula will have to be employed especially in small radius curves; the actual arc is used to find the degree, rather than the central angle subtending the 100-ft. cord, the practice on standard-gage roads. By using the actual arc, a 50-ft. radius $=\frac{5730}{50}=115$ deg. curve; by using a 100-ft. cord, a 50-ft. radius $=\frac{50}{sin\ \frac{1}{2}d}=180$ deg. curve, showing a disparity of 65 deg.

Assuming a curve with a central angle of 90 deg. a ruling grade of 0.5 per cent and allowing the same rate of resistance per degree on a 25-ft. and 50-ft. radius curve, the motor in traveling over the two would have to work equivalent to mounting a 4.5 per cent grade for 39 ft. and a 2.5 per cent grade for 78 ft. respectively. From the beginning of the 50-ft. radius to the point of tangent there would be a total of 1.96 ft. vertical, while to travel between the same points by way of the 25-ft. radius curve, including the 25 ft. of tangent on each end of the curve, there would be a total of 2.02 ft. vertical, or essentially the same vertical rise in either case.

While actually with the smaller radius curve there would be a lower rate of resistance per degree, this would be more than balanced by the increased resistance due to the slower speed compelled by the sharper curve. If the resistance due to grade and curvature between the similarly located points is accepted as equal, then there remain in favor of the 50-ft. radius the greater speed at which the trip can travel, the reduced danger from cars jumping the track, the better adherence of trolley to wire, and 11 ft. shorter haul, and under some conditions 11 ft. less of track. With a gangway producing six trips per day of twelve 5-ton cars each, this 11 ft. twice per trip would consume enough power to draw 1 ton 7920 ft. each day, or 375 mi. per year. A self-recording dynamometer will reveal the frictional resistance of any type of car, and the monetary value per unit of haulage can be readily ascertained.

In estimating the number of cars per trip, the future output, as well as the length of haul, must be considered. A haulage over which 100 cars travel per day may be increased threefold by a tunnel to the veins. This will mean the installation of a larger motor if the haul is long, or possibly the use of two motors. The maximum speed of haulage underground being fixed by law (6 mi. per hour), nothing can be expected from faster transportation. A 15-ton locomotive will not traverse curves possible to an 8-ton machine, and this fact will demand an extra motor, with its attendant expense.

For obvious reasons no compensation is allowed for curvature underground; and if a motor is required to work at its capacity, the additional resistance to be overcome, because of curvature, will be the factor limiting the length of the trip. With the large curve a locomotive may pull through on its potential velocity, but on a curve of 25 or 30 ft. radius, the velocity will have to be reduced before reaching the curve.

Rails.—Much extra expense can be incurred by not having the weight of rail used in the track properly proportioned to the weight of the motor operating on it. Where the rail is too heavy, there is an unnecessary expenditure in first cost and where too light, as is more frequently the case, costs of track maintenance, together with extra wear and tear on motors, due to poor track conditions, will mount up rapidly, though perhaps not be so evident. Working under average conditions, the Baldwin-Westinghouse Co. recommend the following minimum weight rails for general mine service with motor haulage:

Weight of Motor in Tons	Weight of Rail in Pounds per Yard
4 to 6	16
6 to 8	20
8 to 10	25
10 to 13	30
13 to 15	40
15 to 20	50

Mines having average size mine cars customarily use 40 to 60 lb. rail on the main haulage, 20 to 40 lb. on secondary haulage and 16 to 25 lb. in the rooms.

An approximate rule sometimes used for this purpose is to have a rail that will weigh at least 4 lb. per yard for each ton of weight in the locomotive. For example, a 4-ton motor should run on a 16-lb. rail; a 5-ton on a 20-lb. rail, etc. This rule gives somewhat excessive rail weights when applied to the heavier types of motors and the above table is preferable if available.

In purchasing rail for mine use the buyer will require: (1) Stiffness, (2) strength and (3) durability rather than tons of steel. If the strength of various sections is compared, it will be found that these requisites can be purchased at a lower unit rate in the larger sections. In "stiffness" we have that property which allows the rail to span the ties and support the load without bending, affording thereby a smooth running surface for the cars; in "strength" we have that quality which bears the load without yielding or breaking, while in "durability" we have the ability to resist wear over extended periods of time.

The stiffness varies as the square of the weight, and the strength as the 3/2 power, while the price per ton is nearly constant. If the unit weight is assumed as being 30 lb. per yd., then the stiffness will increase as follows:

THIRTY POUNDS PER YARD—STIFFNESS = 1

$16\frac{2}{3}$ per cent increase in weight 35 lb. per yard stiffness = 1.36 or a 36 per cent increase.
$33\frac{1}{3}$ per cent increase in weight 40 lb. per yard stiffness = 1.78 or a 78 per cent increase.
50 per cent increase in weight 45 lb. per yard stiffness = 2.25 or a 125 per cent increase.
$66\frac{2}{3}$ per cent increase in weight 50 lb. per yard stiffness = 2.79 or a 179 per cent increase.
100 per cent increase in weight 60 lb. per yard stiffness = 4.00 or a 300 per cent increase.

The ultimate strength will increase as follows:

THIRTY POUNDS PER YARD—ULTIMATE STRENGTH = 1

$16\frac{2}{3}$ per cent increase in weight 35 lb. per yard ultimate strength = 1.26 or a 26 per cent increase.
$33\frac{1}{3}$ per cent increase in weight 40 lb. per yard ultimate strength = 1.54 or a 54 per cent increase.
50 per cent increase in weight 45 lb. per yard ultimate strength = 1.84 or a 84 per cent increase.
$66\frac{2}{3}$ per cent increase in weight 50 lb. per yard ultimate strength = 2.15 or a 115 per cent increase.
100 per cent increase in weight 60 lb. per yard ultimate strength = 2.83 or a 183 per cent increase.

The advantages of the heavy section over the light, as regards stiffness and strength, would show a higher comparison as the rail wears or wastes away from any cause whatsoever.

In determining the durability of rail, it is obvious that a great amount of wear cannot be expected if the weight selected conforms closely to the immediate duty it has to withstand.

We can assume for practical purposes that half the total weight is in the head, and that about half of this weight, or one-quarter the weight of the rail, can be worn away before the rail is discarded, if a sufficient margin of metal has been allowed; otherwise, the rail will fail before it has attained much more than a high polish.

In mining work, particularly underground, with the trackmen in absolute charge, trackwork, derailments, rail breakage, etc., are taken as part of the day's routine and pass unnoticed, except that part which appears indirectly in the high maintenance charges.

If we assume that a wear of $^1/_5$ the weight of the head was allowed as a safety factor in the lighter rail, then the durability of light and heavy sections will compare as follows:

Available for Wear				Left in Head After Minimum Wear	Spare Metal in Next Heaviest Rail before Head Becomes as Light	Times Increase of Wear by Adding 5 Lbs. to Section	Increase in Weight by Adding 5 Lbs. to Section
Weight in Pounds per Yard	Weight in Head Only	Maximum Half Head	Minimum One-fifth Head				
30	15 0	7.5	3.0	12	5.5	1.830	1/6
35	17.5	8.75	3.5	14	6.0	1.710	1/7
40	20.0	10.00	4.0	16	6.5	1 625	1/8
45	22.5	11.25	4.5	18	7.0	1.550	1/9
50	25.0	12.50	5.0	20	7.5	1.500	1/10
55	27.5	13.25	5.5	22	8.0	1.454	1/11
60	30.0	15.00	6.0	24	8.5	1.420	1/12

Or, using 30-lb. rail as a unit, the metal available for wear would compare as follows:

Weight in Pound per Yard	Weight in Head Only	Available for Wear Before Head Would Become as Light		Increase in Weight per Yard Per Cent
		Maximum Per Cent	Minimum Per Cent	
30	15	7.5 or 100	3.0 or 100	
35	17½	10.0 or 133⅓	5.5 or 183⅓	16⅔
40	20	12.5 or 166⅔	8.0 or 266⅔	33⅓
45	22½	15.0 or 200	10.5 or 350	50
50	25	17.5 or 233⅓	13.0 or 433⅓	66⅔
55	27½	20.0 or 266⅔	15.5 or 516⅔	83⅓
60	30	22.5 or 300	18.0 or 600	100

Briefly, if we were about to build a permanent (so-called) narrow-gage road for mine traffic, for which 30-lb. steel would ordinarily be used, we would gain, by using a 60-lb. section, the economy in maintenance, a more easily operated road with its attendant benefits, fewer ties, fewer derailments and a larger scrap value when the rail was reclaimed. Furthermore, we would have a stiffness four times, an ultimate strength 2.83 times and a durability three to six times as great, for a rail expenditure but double that for 30-lb. steel.

Some concerns, by purchasing "second" rail from the railroad companies, obtain the heavier rail for the same price per lineal foot as for new sections one-half to two-thirds their weight. This quality of rail for most mining purposes will serve as well as new sections.

In localities where acid water abounds the corroding of the steel is frequently the limiting factor in the life of the rail. It would be futile to lay heavy section rail in locations where the water would soon destroy it. As the web and edges of the flange are the portions destroyed first, an inspection of the standard dimensions will evidence that by increasing the weight we do not secure a proportionate increase in the acid-resisting properties of the rail. Rail weighing 25 lb. per yd. has been taken as the basis of unity.

Weight of Rail	Increase in Weight, Per Cent	Thickness of Web	Increase in Thickness, Per Cent	Thickness Ends of Flange	Increase in Thickness, Per Cent
25	...	$\frac{19}{64}$	...	$\frac{11}{64}$	
30	20	$\frac{21}{64}$	11	$\frac{11}{64}$	
35	40	$\frac{23}{64}$	21	$\frac{12}{64}$	9
40	60	$\frac{25}{64}$	32	$\frac{14}{64}$	27
45	80	$\frac{27}{64}$	42	$\frac{15}{64}$	35
50	100	$\frac{29}{64}$	47	$\frac{15}{64}$	36
60	140	$\frac{31}{64}$	63	$\frac{18}{64}$	64

In the standard tee rail, adopted by the American Society of Civil Engineers, 42 per cent of the metal is in the head, 21 per cent in the web and 37 per cent in the flange. The top corners are curved to a $^5/_{16}$-in. radius, and the car wheels are designed to give on this as little friction as possible; as the rail more nearly wears to the shape of the flange the friction is augmented. The height of the rail is identical with the width of the flange, so if this dimension is measured the weight can be determined.

The table shows the weight of rail per yard corresponding to the height of flange width.

Track frogs.—Standardization of switches and frogs at mines to a limited number of sizes to meet requirements will substantially lower the cost of making these. The accompanying illustration, Fig. 18, shows a standard frog and switch used by the O'Gara Coal Co. in 1916.

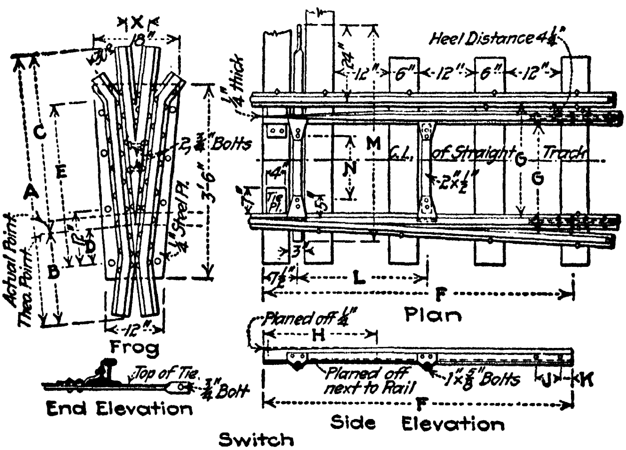


FIG. 18.—Standard frog and switch used by the O'Gara Coal Co.

The designs were made after considering both simplicity and economy of construction. Any ordinary blacksmith or ironworker will make these parts without difficulty. The cost of making a No. 5 frog and two 6-ft. switch points at a well-equipped mine shop was about 1916: Material, $6.37; labor of blacksmith and machinists, $6.58; total, $12.95.

A cast-steel frog supplied by manufacturers at a cost of about $6.50 is inherently more rigid than the riveted frog, but it is difficult to fasten it securely to the ties. In order to stiffen the riveted structure, cast-iron fillers may be added which also support the flange of the wheels in passing over the throat of the frog, thus relieving the jar to the rolling stock.

Dimensions of Standard Frogs and Switches for Narrow-Gage Industrial and Mine Tracks

Standard Frog for Motor Turnout (Right or Left)

Frog No.	Frog Angle, X		Rail, per Yard	Length of Frog, A		Wing Rail, B	Heel Distance, C		Length of Throat, D	Straight Rail, E	
	Deg.	Min.	Pound	Ft.	In.	In.	Ft.	In.	In.	Ft.	In.
3	18	55	30	4	0	16	2	8	$3\frac{15}{16}$	2	3
4	14	15	30	4	8	20	3	0	$5\frac{3}{4}$	2	9
4	14	15	40	4	8	20	3	0	$6\frac{1}{2}$	3	0
5	11	25	30	4	10	20	3	2	$7\frac{3}{16}$	3	0
5	11	25	40	5	0	20	3	4	$8\frac{1}{8}$	3	0

Standard Switch for Motor Turnout (Right or Left)*

Weight of Rail, Pound per Yard	Length of Point, F		Distance between Bridle Rods L		Length of Rail Planed, H		Rail Punching,		Gage of Track, G	Length of Bridle Rod, M		Rod Punching, N
	Ft.	In.	Ft.	In.	Ft.	In.	J Ft.	K In.	In.	Ft.	In.	In.
20	4	0	(1 rod)		1	4	4	2	36	5	$8\frac{1}{4}$	25
25	4	0	(1 rod)		1	6	4	2	40	6	$0\frac{1}{4}$	29
30	4	0	(1 rod)		1	9	4	2	42	6	$2\frac{1}{4}$	31
30	6	0	2	3	2	6	4	2	44	6	$4\frac{1}{4}$	33
30	7	6	3	0	3	0	4	2	48	6	$8\frac{1}{4}$	37
40	6	0	2	3	2	9	5	$2\frac{1}{2}$				
40	7	6	3	0	3	6	5	$2\frac{1}{2}$				

*The throw of switch point is $3\frac{1}{2}$ in. all for cases.

In designing various parts of the turnouts it was kept in mind that all such turnouts may be of only temporary usefulness in one particular location and that the constituent parts may be used many times before being cast aside as useless. The standard frog is somewhat shorter than one designed to the specifications of the American Railway Engineering Association, but the saving in weight and bulk, with the consequent saving in making the several installations, will more than offset any loss due to instability.

The cost of laying and ballasting a No. 5 turnout complete, as shown in Fig. 19, was about 1916 as follows:

One 30-lb. No. 5 frog and two 6-ft. points	\$12.95
40 ties, 5×6 in., at 20 cts	8.00
Spikes, bolts, tie-plates, etc	.75
1 low switch stand and rods	2.25
2 headblocks, 5×6 in., 8 ft. long	1.00
Total material	\$24.95
Laying, 16 hr. at 35½ cts	5.68
Ballasting and surfacing, 8 hr. at 35½ cts	2.84
Total labor	\$8.52
Total cost of material and labor	\$33.47

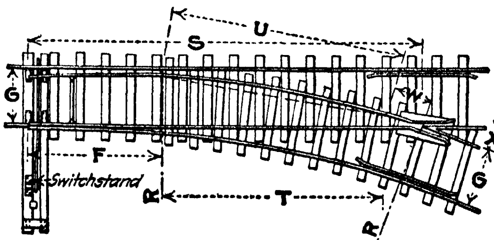


FIG. 19.—Standard turnout used by the O'Gara Coal Co.

The dimensions of the standard frogs, switches and turnouts are given in the accompanying tables. The formulas used for the turnouts are as follows:

$$\textit{Chord length } U = \frac{G - B \sin X - F \sin Y}{\sin \frac{1}{2}(X+Y)};$$

$$\textit{Radius } R = \frac{G - B \sin X - F \sin Y}{\cos Y - \cos X} - \tfrac{1}{2}G;$$

$$\textit{Lead } S = (R + \tfrac{1}{2}G)(\sin X - \sin Y) + B \cos X + F + O \ ;$$

in which

X = frog angle;
Y = angle of point rail;
B = length of wing rail;
F = length of switch rail;
O = distance from actual to theoretical frog point;
G = gage of track;
R = radius of turnout.

The dimension of O was taken as 2 in. and the heel distance of switch points as $4\frac{1}{4}$ in.

The spacing of the ties depends on the size of the tie and the style of the turnouts. If the regular set of switch ties is used as in standard-gage trackwork, 5 × 6-in. ties spaced 18 in. center to center will give good results for track laid with rails of up to 40 lb. in weight. If the turnout is laid with ties of even length staggered in as shown in Fig. 19, a spacing of 16 to 18 in. for each branch has proved satisfactory. This style of construction is specially well adapted to underground turnouts, where headroom is limited and flat ties 3 × 5 in. or 3 × 6 in. are used. All switch ties should be of hardwood and treated if possible, as decay will set in before mechanical wear destroys their usefulness.

Dimensions for Turnouts in Narrow-Gage Tracks

Gage of Track, 36 In.

Frog No.	Length of Switch Points, F	Radius of Turnout, R		Length of Lead, S		Length of Straight Rail, T		Chord Curved Rail, U		Mid-ordinate Curved Rail, V
	Ft.	Ft.	In.	Ft.	In.	Ft.	In.	Ft.	In.	In.
3	4	42	$7\frac{3}{16}$	16	$3\frac{11}{16}$	10	$9\frac{11}{16}$	10	$10\frac{9}{16}$	$4\frac{3}{64}$
4	4	81	$1\frac{1}{2}$	19	3	13	5	13	8	$3\frac{3}{8}$
4	6	75	$8\frac{1}{4}$	22	6	14	8	14	$10\frac{5}{8}$	$4\frac{5}{16}$
5	4	141	$7\frac{1}{4}$	22	$2\frac{7}{8}$	16	$4\frac{7}{8}$	16	$7\frac{5}{16}$	$2\frac{7}{8}$
5	6	126	7	26	$0\frac{1}{2}$	18	$2\frac{1}{2}$	18	$4\frac{3}{4}$	$3\frac{15}{16}$
5	$7\frac{1}{2}$	122	$9\frac{7}{8}$	28	$4\frac{1}{2}$	19	$0\frac{1}{2}$	19	$2\frac{5}{8}$	$4\frac{7}{16}$
6	$7\frac{1}{2}$	184	$9\frac{3}{8}$	30	$9\frac{5}{8}$	21	$5\frac{5}{8}$	22	9	$4\frac{3}{16}$

Gage of Track, 42 In.

Frog No.	F Ft.	R Ft.	R In.	S Ft.	S In.	T Ft.	T In.	U Ft.	U In.	V In.
3	4	52	$1\frac{13}{16}$	18	$9\frac{1}{4}$	12	$11\frac{1}{4}$	13	$3\frac{3}{4}$	$4\frac{15}{16}$
4	4	99	$2\frac{1}{8}$	22	3	16	5	16	$8\frac{1}{4}$	$4\frac{1}{8}$
4	6	92	$6\frac{3}{8}$	25	9	17	11	18	$2\frac{1}{8}$	$5\frac{1}{4}$
5	4	172	$0\frac{1}{2}$	25	$9\frac{1}{8}$	19	$11\frac{1}{8}$	20	2	$3\frac{1}{2}$
5	6	153	$8\frac{3}{8}$	29	$11\frac{1}{4}$	22	$1\frac{1}{4}$	22	$3\frac{7}{8}$	$4\frac{13}{16}$
5	$7\frac{1}{2}$	149	$1\frac{7}{8}$	32	$5\frac{1}{2}$	23	$1\frac{1}{2}$	23	4	$5\frac{7}{16}$
6	$7\frac{1}{2}$	223	$5\frac{3}{4}$	36	$7\frac{1}{2}$	27	$3\frac{1}{2}$	27	6	$5\frac{1}{16}$

Track.—The following is an interesting example of estimating the cost of laying 5000 ft. of track where the grade is 1 per cent in favor of the loads; a 12-ton motor is used and 3½ ton cars assuming that the rails cost \$26 per ton, ties 10c. each, spikes \$3.75 per keg of 200 lb., labor for trackmen \$2.50 per day, and helpers \$1.75 per day, these figures being as of 1911.

The rails for a 12-ton motor haulage should not be lighter than 40 lb. per yard which would require $(2 \times 5000 \times 40) \div (3 \times 2240) = 59.5$ tons at a cost of $26 \times 59.5 = \$1547$. For 40 lb. rails, use $3\frac{1}{2} \times \frac{7}{16}$ in. spikes, 12 kegs, at \$3.75 per keg = \$45; and 4 × 6 in cross-ties, spaced 2 ft. center to center, 2500 at 10c. each = \$250. There will be required also, using 24-ft. rails, 832 angle or fish-plates, 6240 lb., at 1½c. per pound = \$93.60, and 8 kegs bolts, nuts, and washers at \$5 per keg = \$40; making the total track material \$1975.60. The laying and surfacing of 5000 ft. of track in mine entries, under ordinary conditions, including the handling of the material in the shaft and its distribution in the entry will require, approximately 120 days' labor for helpers at \$1.75 per day = \$210; 50 days, trackmen, at \$2.50 per day = \$125; and 6 days, drivers, at \$2 per day = \$12; total for labor \$347. The total cost of the track laid is therefore \$2322.60, making no allowance for special grading which might be required at some points in the entry.

A number of interesting figures on the comparative cost of track laid with steel and wood mine ties under varying conditions were given in a paper presented before the West Virginia Mining Institute, in 1913 from which the following have been excerpted:

The Peyton Block Coal Co. used four steel mine ties per rail length which at a cost of 32c. each made a total of \$1.28 per each pair of rails. Under the same conditions, 11 wood ties would be required, which at a cost of 5c. and allowing 10c. for spikes brought the total cost per pair of rails to 65c. Offsetting this difference, however, it was found that the miners would lay track with steel ties in their working places themselves while they always insisted on the regular mine track layer performing the work when wood ties were used because of the special tools and labor required. It was believed that this saving in the time of the track layer compensated for the difference in the cost of the material involved, so that the extra life of the steel tie could be regarded as clear profit.

The Allegheny River Mining Co. advances the opinion that the life of the steel mine tie is about six times that of the wood tie and since only one-half as many are required per foot of track, the ratio of comparative utility was 1 to 12. On the basis of 32c. for the steel ties and 8c. for wood, which was the delivered cost to this company, and disregarding cost of the spikes required for wooden ties, the cost ratio is 4 to 1.

Track costs at the Dartmore Mine of the Davis Coal & Coke Co., when laid with steel ties were found to be $1.28 per 30-ft. length for material. When using wooden ties, 10 of these were required per 30-ft. length of track which, at a cost of 10c. each and allowing 12c. for spikes, brought the cost of material for the track with wooden ties up to $1.28 per 30 ft. The difference in the cost of material for wooden and steel ties at this mine amounted to ½c. per lineal foot of track, disregarding economies in laying, salvage, etc. as indicated above.

At mines Nos. 14 and 20 of the same company, it was found that the steel tie saved sufficient head room to eliminate the necessity of brushing the top and effecting a computed saving of 63c. per yard of track.

The Hutchinson Coal Co. compiled an interesting study of the comparative cost of steel and wooden ties at its Kirkwood Mine, near Bridgeport, Ohio, during the years 1907 to 1909 inclusive when it was changing from the wood to the steel ties. In the 1907 period all wood ties were being used; in the 1908 period 75 per cent steel ties were in use and in 1909 all steel ties were in use. The comparative figures are as follows:

	1907		1908		1909	
	Output	Cost in Cents per Ton	Output	Cost in Cents per Ton	Output	Cost in Cents per Ton
September	16,083	4.80	12,620	3.98	21,556	2.84
October	26,216	4.02	12,620	3.36	22,540	2.81
November	22,617	3.70	16,281	3.21	25,191	2.82
Average cost per ton	......	4.10	......	3.48	......	2.82

The saving per ton with all-steel track thus appears to be 1.28c., which in the three months period of 1909 amounted to $886.85.

This same company also compiled an interesting comparative estimate of the cost of track in a room, computed on the basis of a width of 24 ft., length of 200 ft. and a thickness of coal of 5 ft. 4 in. which worked out as follows:

WITH WOOD TIES

Ties, 2½ ft. apart, 80 ties at 12 cts	$9.60
320 spikes equals 40 lb	.90
Laying and removing track, labor	7.50
Depreciation of ties and spikes	3.50
Total	$21.50
Salvage	7.00
Net cost	$14.50

WITH STEEL TIES

35 ties at 33 cts	$11.55
Labor, removing (track laid by the miners)	1.50
Depreciation	1.00
Total	$14.05
Salvage	10.55
Net cost	$3.50

Estimating the output of coal from this room at 1000 tons on which a saving of $11 will be effected, this amounts to 1.1c. per ton.

After an exhaustive series of tests the Carnegie Steel Co. prepared the accompanying comparative cost of track laid in rooms with steel and wooden ties. The table is based on rooms 280 ft. long with steel ties spaced at 4 ft. center to center and wood ties 2 ft. center to center. This estimate contemplates using each wood tie in two consecutive rooms and after the second year renewing annually 15 per cent of steel ties, or 10 new ties per year, per room.

Number of Ties in One Room	Steel, 70			Wood, 140	
Cost of ties f.o.b. mine in carload lots	@ 0.29	.....	$20.30	@ 0.06	$8.40
560 spikes	....	.....		@ 0.00½	2.80
Laying in first room	@ 0.02	$1.40		@ 0.04	5.60
Taking up when room worked out	@ 0.01	.70		@ 0.02	2.80
Maintenance cost for first room steel ties	....	$2.10	2.10		
Total cost at end of life of first room	....	.....	$22.40		$19.60
Laying in second room	@ 0.02	$1.40		@ 0.04	5.60
560 spikes	....	.....		@ 0.00½	2.80
Taking up when room worked out	@ 0.01	.70			
Maintenance cost for second room steel ties	....	$2.10	2.10		
Total cost at end of life of second room	....	.....	$24.50		$28.00
Saving per room in favor of steel ties ... $3.50					
10 new steel ties	@ 0.29	$2.90			
10 old ties, scrap credit	@ 0.05	.50			
		$2.40			
140 new wood ties	....	.....		@ 0.06	$8.40
560 spikes	....	.....		@ 0.00½	2.80
Laying in third room	@ 0.02	$1.40		@ 0.04	5.60
Taking up when room worked out	@ 0.01	.70		@ 0.02	2.80
Maintenance cost for third room steel ties	....	$4.50	$4.50		
Total cost at end of life of third room	....	.....	$29.00		$47.60
Saving per room in favor of steel ties ... $18.60					
10 new steel ties less scrap credit	....	$2.40			
560 spikes	....	.....		@ 0.00½	2.80
Laying in fourth room	@ 0.02	$1.40		@ 0.04	5.60
Taking up when room worked out	@ 0.01	.70			
Maintenance cost for fourth room steel ties	....	$4.50	$4.50		
Total cost at end of life of fourth room	....	.....	$33.50		$56.00
Saving per room in favor of steel ties ... $22.50					

The matter of track friction is important, and most mining men realize that there are material advantages in a good track. Few, however, really comprehend the reduction in power requirements that can be effected on much-used roads by making a strictly firstclass track in every respect. When we consider that it is possible to have a track friction as low as 12 lb. of drawbar pull per ton, while as many roads show as much as 40 lb. per ton, it is apparent that this is an extremely important item. The use of heavy rails is an essential feature, but it is even more necessary that the track be kept clean.

Mine cars.—In 1911 a wooden car of 49 cu. ft. capacity, without brakes, cost at a certain mine in the neighborhood of $45 each. The same company was offered steel cars of the same outside dimensions, but having greater capacity, as follows:

Capacity	Cost, Delivered	Increase, Per Cent
7 cu. ft	$65.00	45
1 cu. ft.	62.50	38
1 cu. ft.	75.00	66
5 cu. ft.	68.50	52

The cost of an all-steel car thus ran from 38 to 66 per cent more than a wooden one, in some cases more, possibly, especially in case of the addition of improvements in the way of draft gear, running gear or wheels. The manufacturers themselves state the increase of cost to be from 50 to 100 per cent.

Steel cars have a somewhat greater capacity than the wooden car, varying somewhat with the design of the car, but ranging generally between 10 and 20 per cent. The accompanying table gives the comparative capacities and weights of some steel and wooden cars of the same dimensions, taken from actual practice:

Capacity, Cubic Feet			Weight in Pounds			
Wood	Steel	Increase, Per Cent	Wood	Steel	Per Cent Gain in Capacity	Saving in Weight, Per Cent
49	58	18	2010	1800	18.0	11.5
94	105	11¾				
53	60	13	1920	1985	13.2	3.0*
14	19	35	1085	855	35.0	26.0

* Increase

Weights of both wooden and steel cars vary widely and it is difficult to make any accurate comparison. Steel cars are sometimes heavier than the wooden ones of the same capacity, but generally they appear to run from 10 to 20 per cent less in weight for the same capacity.

The chief advantages of the steel car are:

That although the first cost of the steel car is greater, the increased life and decreased cost of maintenance, together with increased capacity, thereby necessitating a fewer number of cars to handle a given output, more than make up for the difference.

That the advantage of the steel car having a greater capacity with the same outside dimensions, or the same capacity with smaller dimensions, is of great value, especially in low veins.

The increased capacity of the steel car should materially reduce the cost of haulage, and incidentally tend to increase the output of the miner.

The saving it is possible to effect in the tare weight of the car itself would also be a factor in the reduction of costs by reducing the proportion of dead to live load.

The steel cars will not warp, shrink or split, which are advantages that are apparent to all, besides preventing the leaking of dust coal on haulways—not only a nuisance, but a constant source of danger and expense.

On a special type of steel car, repair charges have been estimated at 1c. per 50 ton-miles. The total ton-miles on which this was computed amounted to 120,000 on which the repair charges were $22.50.

The manufacturers claim that the cost of maintenance of steel cars is only about 20 to 25 per cent of that for wooden cars. From the experience of the railroad companies with their steel-car equipment, it would seem this cannot be regarded as an underestimation. Four different manufacturers estimate the life of the steel car to be two to four times the life of the wooden cars.

The relation of the tare, or weight of the car, to its capacity is an important consideration in the economics of haulage, the tare of course representing dead load haulage. The accompanying table, compiled from bulletins of the Illinois Coal

Mining Institute, and data received from car manufacturers give average ratio of capacity to tare for the normal mine car.

Light Cars

Tare	Capacity of Cars in Pounds	Ratio of Capacity to Tare
800	2000	72–28
900	3000	77–23
1000	4200	80–20
1100	2600	70–30
1200	2600	68–32
1300	4200	76–24
1400	3000	68–32
1500	3500	70–30
1600	3000	65–35
1700	5000	75–25
1750	5600	76–24
1900	4000	68–32
2000	5000	72–28
Average ratio.......		72–28

Medium Cars

Tare	Capacity of Cars in Pounds	Ratio of Capacity to Tare
2200	6000	73–27
2400	6000	71–29
2525	4000	61–39
2665	4800	64–36
2850	4800	63–37
Average ratio.......		66–34

Heavy Cars

Tare	Capacity of Cars in Pounds	Ratio of Capacity to Tare
3240	6500	67–33
3330	6000	65–35
3500	6500	65–35
3700	8000	68–32
3780	6750	64–36
Average ratio.......		66–34

At mines having cars already equipped with plain bearings which it is desired to change to roller bearings the cost under average conditions will be about $50 per car, this figure being as of 1917. Computing on the basis of 500 cars this would involve an expenditure of $25,000 less the salvage value of the old axles, journal boxes, and wheels which would be about as follows (figures as of 1917):

1000 axles, 100 lb. each, @ $20 per ton..............	$1000
2000 boxes, 25 lb. each, @ $15 per ton..............	375
2000 wheels, 135 lb. each, @ $15 per ton..............	2025
	$3400

The actual first cost of the roller-bearing installation would then be $25,000 – $3400 = $21,600. The interest on this investment is $21,600 × 6 per cent = $1296 per year.

Let it now be assumed that we can wipe out the initial investment by creating a sinking fund. To provide a sinking fund for \$1000, for example, over a period of 10 years, which is the conservative life of the bearings if properly used, we must lay aside \$75.87 each year. This is based on an interest rate of 6 per cent. For \$21,600, we must lay aside \$1638.79 each year. The interest on \$21,600 and the sinking fund amounts yearly to \$1638.79 + \$1296 = \$2934.79.

Let us calculate what this saving amounts to in the case of 500 cars.

Power costs, say 1c. per ton of coal hauled. Drawbar pull per ton (plain bearings) equals 32 lb. + 20 lb. for every 1 per cent of grade. Drawbar pull per ton with roller bearings equals 13 lb. + 20 lb. for every 1 per cent of grade. These figures are from dynamometer tests made by P. B. Liebermann, chief engineer of the Hyatt Roller Bearing Co.

The saving in drawbar pull equals 52 lb. — 33 lb. = 19 lb. per ton hauled up a 1 per cent grade. The saving in power $= \frac{19}{52} \times$ 1c. = 0.4 mill per ton hauled.

Each car, let us assume, hauls 6 tons per day. The saving in power per year with 500 roller bearing cars = 0.4 mill × 6 tons × 300 days × 500 cars = \$3600.

Plain-bearing cars require oil once a day. It takes two men at least to attend to the oiling. The cost of oil and waste for 500 plain-bearing cars per year equals \$525. Cost of two men per year equals \$1200. Total, \$1725. The cost of oil and labor for 500 roller-bearing cars is \$290. The saving is thus \$1435.

These figures are the results of carefully made tests. With plain-bearing cars it is necessary to use two men at the tipple, in order to push the cars. Since roller-bearing cars push with one-half the effort required for plain-bearing cars, it takes but one man to handle the roller-bearing cars at the tipple. For the same reason, the services of one cager can be dispensed with at the foot of the shaft.

The saving on these two men equals \$1875 per year, figuring that the man at the bottom costs \$2.75 per day and the one at the top costs \$3.50 per day.

The total yearly saving on 500 roller-bearing cars is thus estimated at: Power, \$3600; lubrication, \$1435; labor, \$1875. Less a yearly cost of \$2944.79, or \$3965.21.

The Hyatt Roller Bearing Co. conducted experiments with a dynamometer car to determine accurately the actual train resistance of cars equipped with their type of roller bearing and those equipped with plain bearings.

One of these tests conducted at Greensburg, Pa., on a track with an average grade showed an average drawbar-pull of 12.8 lb. at a speed of 5.98 miles per hour for the roller bearing and 24.3 lb. at 5.59 miles per hour for the plain bearing. The advantage in favor of the roller bearings works out at 47.25 per cent.

A second test at Carbondale, Pa., on an average grade of 0.45 per cent showed an average drawbar-pull of 13 lb. at 7.8 miles per hour for the roller bearing as compared with 32 lb. at 8 miles per hour for the plain bearing. The saving in drawbar-pull in this case amounts to 59.3 per cent.

The diameter of the car wheel has an important influence on the power required to move the car. The smaller the wheel the more difficult it is to move. Cars move with less power on the narrower track gages as well, other conditions being equal. Wheel bases on mine cars rarely exceed 42 in. and the shorter this is the sharper the curve the car will negotiate.

On a good clean, level track a man can push a car that weighs 1½ tons and carrying a load of 2½ tons, making a total load of 4 tons, though the car will be difficult to start and stop. It is probably inadvisable to have cars that have to be handled by men weigh when loaded over 3 tons and then the track should be in good condition.

Many progressive operators are now providing a stretcher car for emergency purposes, a move that will be commended by all who have ever had occasion to assist in bringing a badly injured man from the mine. Fig. 20, shows an excellent type of car for this purpose, the cost of which was \$80, including material and labor, in 1916. The design is quite simple, being an ordinary mine truck with the stretcher box supported on carriage springs. A box is slung from the truck portion of the car. In this will be kept bandages, salves and stimulants and a lungmotor. The car will be kept in a dry place specially

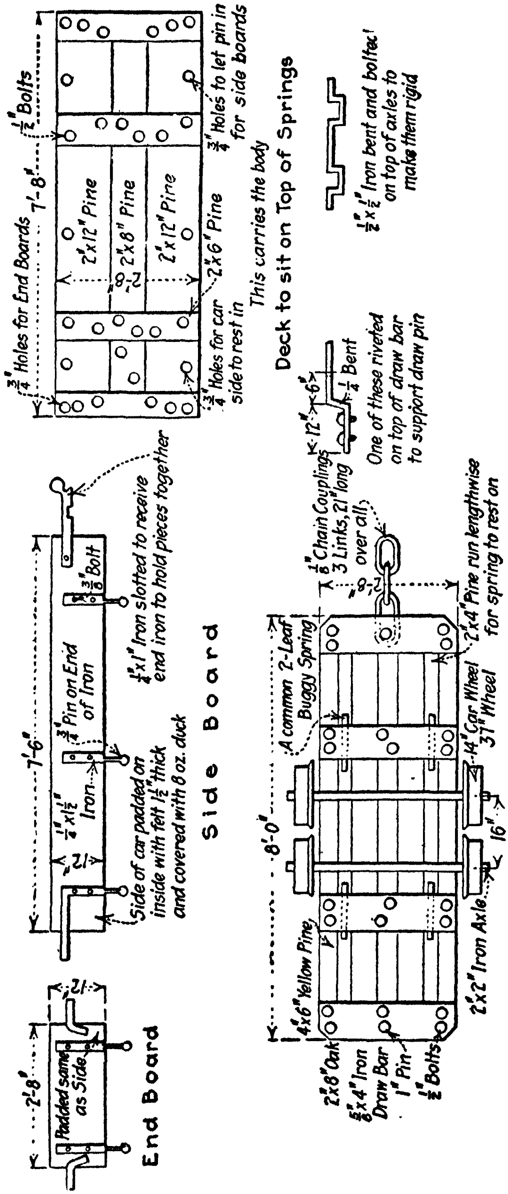


FIG. 20.—Detail of side board, end board, bottom and stretcher deck bottom of car.

provided for the purpose. This place will be at some point near the face of the workings.

The length over all is 8 ft., the width 2 ft. 8 in., and the height of the box 12 in. The bill of material is as follows:

Lumber:

2 pieces 4×6 in. by 8 ft. long, yellow pine.
2 pieces 2×12 in. by 7 ft. 8 in. long, yellow pine.
1 piece 2×10 in. by 7 ft. 8 in. long, yellow pine.
4 pieces 2×6 in. by 2 ft. 8 in. long, yellow pine.
4 pieces 2×8 in. by 2 ft. 8 in. long, oak.
2 pieces 2×4 in. by 8 ft. long, yellow pine.
4 pieces 1×12 in. by 7 ft. 8 in. long, yellow pine.
2 pieces 1×12 in. by 2 ft. 8 in. long, yellow pine.
1 piece 1×10 in. by 7 ft. 8 in. long, yellow pine.

Iron and bolts:

60 $3/8 \times 1\frac{3}{4}$-in. carriage bolts.
40 $1/2 \times 4\frac{1}{2}$-in. carriage bolts.
35 $1/2 \times 9$-in. carriage bolts.
20 $1/2 \times 3$-in. carriage bolts.
10 $3/8 \times 3$-in. carriage bolts.
1 iron $5/8 \times 4$ in. $\times$ 8 ft.
2 irons $5/8 \times 4$ in. $\times$ 20 ft.
10 irons $1/4 \times 1\frac{1}{2}$ in. $\times$ 14 ft.
1 $7/8$-in. chain, 21 in. long.
2 irons $1/2 \times 2$-in. $\times$ 30 ft.
10 irons $1/4 \times 1\frac{1}{2}$-in. $\times$ 14 ft.
2 irons $1/2 \times 2$-in. $\times$ 30 ft.
5 $3/8$-in. cut washers.
10 $1/2$-in. cut washers.
1 1-in. pin., 8 in. long, with 12 in. of $1/4$-in. chain attached, to couple to 4 common two-leaf buggy springs made of $1/4 \times 1\frac{1}{2}$-in. spring steel; length over all 3 ft. with 8 in. between springs.
1 set 14-in. car wheels.
2 2-in. standard axles, 3 ft. 1 in. gage.

Rope haulage.—The Chicosa Fuel Co. in Colorado installed a rope haulage system about 1910, operated by a number of small electric hoists. Single-drum hoists are used on the cross-entries and double-drum tail-rope engines on the levels.

The pitch on the cross-entries averages $8^1/_3$ per cent, the highest is $13\frac{1}{2}$ per cent. No trouble has been experienced in hauling 6 cars per trip, and at times 8 cars per trip. The car and coal combined weigh about 4900 lb. The speed of the single-drum hoists working on the cross-entries is 300 ft. per min. on empty drum. This is what the manufacturer calls the starting speed. The 300 ft. per min. was considered the best speed for switching purposes, after several speeds had been tried. On the double-drum hoists, from 12 to 24 cars are hauled with the rope speed of 400 ft. per min.

One engineer, one rope rider and one hoist can move as much coal as 8 or 10 drivers and mules, besides eliminating risks of killing both mules and drivers. It is a well-known fact that the depreciation on machinery is much less than that on mules. What it takes to feed three mules will pay one engineer, and what it costs to keep up two sets of mine harness will keep in repair an electric hoist.

There are at present five 50-hp., single-drum, electric hoists and two 25-hp., double drum, tail-rope hoists in operation. The five single-drum hoists work on cross-entries, and the two double-drum hoists haul the coal from the crosses to the main-slope partings. The entire output of the mines which amounts to 1100 tons of coal per 10 hr. is moved with this machinery.

None of the hoists are handling at present more than 50 per cent of their capacity, as the equipment inside the mine could easily handle 2200 tons of coal per 10 hr. With the present output the cost of hauling coal inside the mine is 35 per cent cheaper than if hauled by mules, and 15 per cent cheaper than if hauled by mine locomotives. On the present basis of the output the cost per ton inside the mine for hauling from the rooms to the main slope partings is 3c. per ton; by increasing the output to 2200 tons the cost would $1\frac{5}{8}$c. per ton. For each 1000 ft. the level entries are driven in, the cost will only increase 3 per cent. These figures are based on actual tests which are made daily, and all expenses, labor, oil, power, interest on investment, depreciation, maintenance of hoists, power lines, ropes, and bell wire system, are taken into account.

The electric-haulage system which is now being used is far ahead of the electric locomotive. Even where a room-gathering locomotive is used, the expense of keeping up trolley wires for running locomotives is 70 per cent higher than keeping up power wires in back entries. The danger to both men and mine from trolley wires is 100 per cent greater than from power wires for electric hoists. The troubles, expenses, and power losses in rail bonding for electric locomotives are entirely done away with in the electric hoist, and the cost of keeping up the electric locomotive will be more than keeping up the electric hoist and rope.

With electric locomotives, heavy steel rails are necessary in the tracks, while ordinary 20-lb. steel rails answer the purpose where the electric hoist is used. This item makes a difference of 50 per cent in track cost in favor of the electric hoist.

The electric locomotives cannot be used in gassy mines, while the electric hoist can be used with perfect safety.

Electric locomotives cannot be used to an advantage on grades exceeding 5 per cent, while with the hoist the grade makes no difference.

Gravity planes.—Some interesting cost figures on gravity plane haulage were contained in a paper presented before the West Virginia Mining Institute in 1914. The figures applied to an installation at the mines of the LaFollette Coal, Iron & Ry. Co. in Tennessee.

The plane operates through a vertical head of 613 ft. and a horizontal distance of 3640 ft. and the average inclination is 16.8 per cent. It is operated by seven men, two men and the drumman at the head house to attach the trips and two at the tipple to attach the empties with two others on the plane to test the grips, oil rollers, etc.

The rope for the plane lasted two years and cost $2000. In 1913 the plane handled 140,497 tons in 268 days of 9 hr. or an average of 524 tons per shift. The maximum tonnage handled in one day was 885 tons in 8½ hr. which is at the rate of 104 tons per hour.

It will be seen from the above that the depreciation on the rope amounts to 0.7c. per ton and the labor of the seven men, which includes oiling the cars, amounts to $11.83 per shift or 2¼c. per ton. Maintenance costs for new ties, sheaves, grips, etc., was found to average $25 per month or 0.2c per ton of coal handled; this includes cleaning up wrecks, replacing derailed cars, etc., there being an average of one wreck per month which took about two hours to clean up. The total cost of operating the plane in cents per ton, is: Depreciation on rope, 0.70c.; labor, 2.25c.; maintenance, 0.20c.; total, 3.15c.

Endless rope haulage.—The expression face haulage, indicates a means of disposing of the output from the working face by a haulage system sufficiently flexible to be capable of rapid extensions. A system conforming to these requirements, upon which some valuable cost data are available, and which has been in use for a number of years, consists briefly of the following:

The method is applicable to either room-and-pillar or long-wall mining. At the special mine under consideration, the output was 600 tons per day and endless rope haulage was used the empty cars entering the section at one end and the loads passing out at the other, the empty cars being taken off along the rope and the loaded ones attached. The cars can pass around curves with a relatively small radius and they are automatically detached at any point that may be desired. The haulage is a side rope system, the rope being on the side farthest away from the working places.

When first started the miners were required to push their cars to the rope and attach them, but it was found that this led to an unequal distribution of the cars, the men at the beginning of the section taking the most of the cars and finishing their day's work first. To obviate this six boys were put on pushing the cars whose duty it was to see that the cars were equally distributed.

The following are the particulars of a typical installation:

Number of miners in section	98
Number of tons per man	6
Number of men per place	2
Cars per day	860
Tons per day	600
Weight per car	14 to 13 cwt.
Tare of car	4½ cwt.
Size of car	4×2×3 ft.
Rails handled in 12-ft. lengths	**weight 24 lb. per yd.**

Gauge of road.. 24 in.
Rope, plow steel.... weight ¾ lb. per ft. and 2½ in. in circumference
Height of coal.. 72 in.
Nature of roof............................... Good, sandstone
Nature of section, flat, little water, good roof and floor, coal easily mined, roof weight helping considerably.
Grade, practically horizontal, about one-half of 1 per cent.

Labor in Section	Wages per Day
Deputy, at $3, for one-third time........................	$1.00
Haulage man, at $2.50..............................	2.50
Six drawers, at $1.75..............................	10.50
Two roadmen, at $2, for one-half time..................	2.00
One boy, at $1.50..................................	1.50
One night roadman, at $2...........................	2.00
Shifting and haulage cost, about $210 a month, per day....	3.50
Total..	$23.00

Of these only part time of the roadmen and very little (about one-third) of the deputy's time are charged against the haulage, which gives a cost on the tonnage named of 3.83c. per ton. The distance traveled is 2.08 miles, which works out at a rate of 1.84c. per ton-mile, which for a face haulage compares favorably with the larger and more permanent of endless-rope haulages.

Cost of wire rope.—List prices of crucible cast steel rope of either standard or lang lay were quoted in 1920 as follows:

Price per Foot	Diameter in Inches	Approximate Weight per Foot	Approximate Strength in Tons	Working Load in Tons	Diameter of Drum in Feet
$0.60	1½	3.55	63	12.6	11
0.51	1⅜	3	53	10.6	10
0.43	1¼	2.45	46	9.2	9
0.36	1⅛	2	37	7.4	8
0.29	1	1.58	31	6.2	7
.22½	⅞	1.20	24	4.8	6
.17	¾	.89	18.6	3.7	5
.14½	$\frac{11}{16}$	.75	15.4	3.1	4¾
.12	⅝	.62	13	2.6	4½
.10	$\frac{9}{16}$	.50	10	2	4
.08	½	.39	7.7	1.5	3½
.06½	$\frac{7}{16}$	.30	5.5	1.1	3
.05½	⅜	.22	4.6	.92	2¾
.04½	$\frac{5}{16}$	.15	3.5	.70	2¼
.04	$\frac{9}{32}$	.125	2.5	.50	1¾

List prices of plow steel scale lay rope were quoted as follows:

Price per Foot	Diameter in Inches	Approximate Weight per Foot	Approximate Strength in Tons	Working Load in Tons	Diameter of Drum in Feet
$1.30	1¾	4.85	112	22	7
1.08	1⅝	4.15	94	19	6.5
.93	1½	3.55	82	16	6
.79	1⅜	3	72	14	5.5
.65	1¼	2.45	58	12	5
.54	1⅛	2	47	9.4	4.5
.43	1	1.58	38	7.6	4
.34	⅞	1.20	29	5.8	3.5
.26	¾	.89	23	4.6	3
.19	⅝	.62	15.5	3.1	2.5
.16	9/16	.50	12.3	2.4	2.25
.14	½	.39	10	2	2
.13	7/16	.30	8	1.6	1.75
.12½	⅜	.22	5.75	1.15	1.50
.12¼	5/16	.15	3.8	.76	1.25
.12	¼	.10	2.65	.53	1

Wire rope lubrication.—Wire rope deteriorates with use, but not with age when properly cared for; but the rate of deterioration depends in large measure upon the character of the metal used, the construction of the rope, the diameter of the drums, sheaves and pulleys over which it operates, and to a still greater degree upon how it is lubricated. A rope may be made with great accuracy and meet every specification that human ingenuity can devise, but if not properly protected from the elements which may attack its constituent parts, value and the desired economy cannot be secured. Consequently, the protection and lubrication of wire ropes are of much importance. This applies to cables lying idle as well as to those in service, since rust destroys as effectively as hard work.

The question of efficient lubrication has recently assumed a position of considerable importance. The manufacturer may use the best technical knowledge at his disposal and be most careful in selecting the material which goes into his product, but he cannot be expected to estimate beforehand the rapid and varying degrees of deterioration of the rope when it is in the hands of the user, who it often happens does not fully appreciate that conservation of the life of the cable is entirely under his direction. Even the most perfect rope can be used under such severe conditions and with such lack of attention that it will have a short life, while one of much inferior quality, used under the same conditions of service but carefully taken care of in the way of lubrication, will outlive the higher grade rope.

A careful investigation of all steel cables used in mine service has shown that no two manufacturers are using the same grade of lubricant, and an analysis of the materials most commonly employed and recommended for this class of work shows that tar, graphite and other fillers are used in large proportions, this no doubt for the purpose of developing heavy, adhesive mixtures, which are commonly sold under the names of "rope shield," "rope dressing" and "protectors." Such materials have the effect of only partially protecting the external parts of the rope, and at low temperatures will crack and peel. This may be noticeable only in spots, but it has the effect of permitting the deteriorating elements to attack the internal portions of the cable, and instances are not infrequent of ropes suddenly breaking while the visible wires show no signs of deterioration. It is invariably found upon examination in such cases that the internal wires have perished by corrosion.

In some instances ordinary black oil, or what is commonly known as "waste oil," is used. Both of these materials are worthless as a rope lubricant, as neither will cling to the outer surfaces, penetrate to the core, nor resist the effects of moisture or other damaging elements. Where such materials are used, frequent applications are necessary; and owing to their characteristics, they afford no lubrication to sheaves, drums and pulleys, and are either thrown off or evaporated within a few hours after being applied, thus leaving the rope at the mercy of the elements and of frictional wear.

Many high-grade and expensive greases have been used which perhaps by laboratory analysis are shown to possess the qualifications necessary for resisting the effects of moisture and chemicals, but which on account of the high speed at which ropes are often run and the consequent stress and vibration to which they are subjected are readily thrown off and at no period after application afford more than a temporary protection to the external surfaces of the rope.

It rarely occurs nowadays that the full efficiency of a rope is developed, owing to service conditions requiring it to come into contact with water containing deteriorating elements which have the effect of quickly producing corrosion and consequent brittleness of the wires. Furthermore, the higher the carbon content of the metal the more susceptible are the wires to this corrosion.

A series of practical tests conducted by a number of high-grade technical men has demonstrated that the use of a poor or unsuitable lubricant will often do more to lessen the durability of a rope than using no lubricant at all and that the cost of a lubricant that will meet all operating conditions is trifling as compared with the saving which its use makes possible.

An efficient wire rope lubricant must be free from any material that will attack the constituent parts of the rope, must remain soft and pliable under all atmospheric conditions and must not be subject to evaporation. It must be insoluble in water, so as to prevent water from coming into direct contact with the surfaces of the rope, and must be unaffected by water heavily charged with the chemicals that are encountered in mine operation. It must be of such a nature that it will penetrate between the wires in the strands and between the strands to the core of the rope preserving the latter as well as the metal which surrounds it. It must not be subject to decomposition under the severest conditions of wear and must possess great adhesiveness so that it cannot be thrown off by any force. It must be a material that will not harden or peel either through too frequent application as is often the case or under low temperatures, and must be of such a nature as to permit of it being liquefied to a consistency to permit of application being made while quite thin.

In addition to meeting these requirements, the lubricant used for wire ropes should possess characteristics which permit of its showing equal efficiency as a lubricant for sheave wheels, drums, pulleys or other machine elements over which ropes are liable to pass. And to secure the greatest efficiency this same material should be used to lubricate each wire in the strand

and to saturate the core during the process of manufacture, as it often occurs that with the use of one material in the manufacture and another after the rope has been put in service, the two being of decidedly different nature, no lubrication can be effected on account of one material preventing the other from adhering or penetrating to the interior of the rope.

Too little attention has been given to the initial lubrication of wire rope, particularly as regards proper saturation of the hemp core. It has been the general practice to pass the hemp center through the lubricant used at the time the rope is laid up. When applied in this manner, the greater part of the lubricant is forced out between the strands and, in the case of some grades of lubricant, drips entirely away from the external surfaces. The object of inserting the hemp center is to increase the flexibility of the rope, and the deterioration of this element has the same effect as the deterioration of the material surrounding it. If the hemp center is thoroughly lubricated before being inserted into the rope, it will act as a container of the lubricant and will assist in distributing it to all parts of the cable; also if the proper lubricant is periodically applied it will penetrate and maintain the hemp center as a continuous lubricator.

Another important feature in connection with wire rope lubrication which does not generally receive the proper attention is the method of application. In some instances the lubricant is poured onto the rope either at the sheave wheel or at the drum; in other instances it is applied with a brush. Both of these methods are crude and wasteful.

The proper way to apply the lubricant is to use a split box large enough to hold about 25 lb. of the lubricant to be used. This should be constructed with a hole in the center large enough for the passage of the largest rope in the mine, and when coating smaller cables an old rubber pump valve or a piece of ordinary burlap wrapped around the rope may be employed to act as a wiper and regulate the thickness of the application. These boxes can be constructed so as to be used on either horizontal or vertical ropes.

The lubricant should be thoroughly liquefied in a metal container, and after it is poured into the split box the rope should be permitted to pass slowly through it. In this manner a uniform and economical application can always be made.

When possible the external surface of the rope should be dry when the application is made.

Comparative costs of different systems of haulage.—Many of the opinions expressed in regard to relative economy of the different systems of haulage are founded largely on prejudice, with little or no basis for accurate comparison. The relative costs of haulage by all systems depend upon their intelligent installation and handling, and with equal energy and experience the difference in the cost of haulage by any system will often be only a fraction of a cent. If the animal haul can be kept short and the grades not too steep, the mule or horse in gathering service is a close competitor with the locomotive except where the three- or four-ton mine car enables the locomotive to get more coal each time it makes a trip to the room.

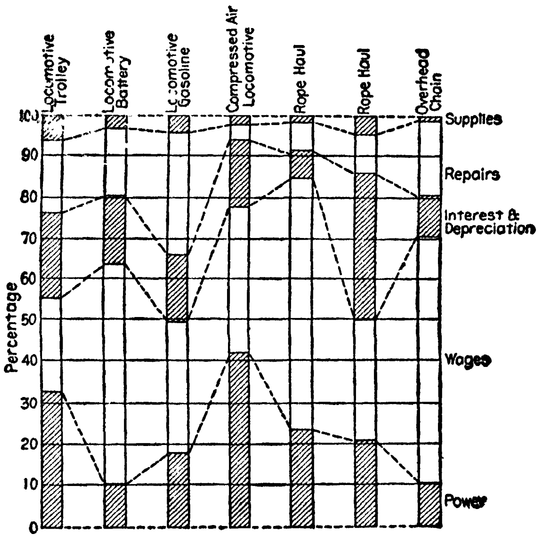
An important factor in securing maximum economy in haulage lies in correctly delimiting the line of secondary haulage, or as it is more commonly known, the gathering. This will vary somewhat according to local conditions at the different mines but a good maximum to set between the working face and the main haulage switch is 1000 ft.

Where long distances have to be covered on secondary haulage motors are substantially the most economical. One instance is on record of a 6-ton motor on secondary haulage work handling 60 cars per shift over grades ranging from 5 to 17 per cent and distributing to 14 different entries, none of which were in, over 500 ft.

Where approximate values are required for general preliminary estimates, the following figures were those commonly used by the German engineers about 1912: Working costs for continuous current locomotives with overhead contact lines, 0.9 to 1.2c. per ton-mile; for single-phase locomotives with overhead contact line, 0.9 to 1.2c. per ton-mile; for accumulator locomotives, 1.8 to 2.1c. per ton-mile.

The working costs are here based upon the output in ton-miles; i.e., the costs stated are those incurred in hauling one ton of material over a track one mile in length. The above figures, which can be reduced under favorable conditions, enable any expert acquainted with his own working costs to ascertain by comparison whether he would be able to effect economy in his own installation by the introduction of electric haulage.

Comparison of all systems.—One of the most exhaustive comparative cost studies of all types of haulage (except animal) that has come to the attention of the author was that appearing in a German technical journal of a number of years ago, the results of which are given herewith together with a chart, Fig. 21, showing graphically the percentage of power, wages, interest and depreciation, repairs and supplies for each of the different systems.



GRAPHIC COST CHART

FIG. 21.—Distribution of costs for all kinds of haulage except animal.

ELECTRIC LOCOMOTIVE

Conditions—Tonnage, 2530 per day of 16 hr.; distance 3¾ mi.; speed, 7½ mi. per hr. Material said to be ore.

Cost of Plant—Four locomotives, including transformer apparatus and trolley wire, $54,748.

Working Expenses—$25,911, distributed as follows:

		Approximate Per Cent of Total
Interest and depreciation, 10 per cent......	$5474	21
Upkeep of locomotives....................	2623	10
Upkeep of trolley wires, etc...............	1810	7
Wages....................................	5975	23
Power....................................	8399	33
Oil and Waste............................	1630	6

The operation cost worked out at ¾c. per ton-mile.

Storage Battery

Conditions—Tonnage, 2400 per day of 16 hr.; distance, 1200 yd.; speed, 6½ mi. per hr.

Cost of Plant—Cost of five locomotives (20-hp.), weighing 6½ tons, also of transformers, switchboard and reserve $14,600
Accumulators 10,219
Cable, rooms for transformers and charging 7,542

Total $32,361

Working Expenses—Annual amount, $14,133.

		Approximate Per Cent of Total
Depreciation	$1460	10
Parts for batteries	1469	10½
Interest on total first cost	1095	8
Wages of drivers	3581	25½
Brakemen's wages	992	7
Attendance on transformers and batteries	2671	19
Upkeep and cleaning	569	4
Oil and waste	141	1
Power, 186,349 kw.-hr	1790	12½
Acid, distilled water, etc	355	2½

The operation costs were 3c. per ton-mile, but it is considered that the conditions were not favorable.

Benzol (Gasoline) Locomotive Plant

Conditions—Tonnage, 1250 per day of 16 hr.; distance, 1000 yd., sloping toward shaft at 4 per cent grade; speed of 3¾ to 5½ mi. per hr.

Cost of plant $8930
Four locomotives, 8-hp $6813
Filling apparatus 73
Engine shed for six machines 2044

Working expenses—for six months, $587.

		Approximate Per Cent of Total
Interest and depreciation	$87	15
Upkeep of locomotives	180	31
Wages of engine drivers	136	23
Wages of brakemen, etc	68	11
Benzol	97	17
Oil and waste	19	3

This operation works out at 3c. per ton-mile.

Compressed-Air Haulage

Conditions—Tonnage, 1242 tons in 16 hr.; distance 1½ mi.

Cost of Plant—Four 12-hp. locomotives weighing 5½ tons, hauling 40 to 50 cars of 17 cwt. gross; compressor, plant and accessories, $14,500.

Working Expenses—Annual amount, $9391.

		Approximate Per Cent of Total
Interest and depreciation at 10 per cent....	$1460	16
Repairs, upkeep, oil and waste............	608	6
Wages, brakemen and switchmen..........	851	9
Wages of drivers (5)....................	1971	21
Cost of attendance on plant..............	365	4
Consumption of power (170 hp., 16 hr. per day, 300 days)......................	4136	44

This works out at 2¼c. per ton-mile.

Rope Haulage

Conditions—Tonnage, 1690 in 18 hr.; distance, 1400 yd., main road served by a number of branches. Gradient of 5 per cent for 700 yd., including right-angled turn.

Cost of Plant—$5092.

Total cost of installing..	$2929
Rope..	973
Share of outlay on haulage engine and buildings..................	2190

Working Expenses—Annual amount, $7526.

		Approximate Per Cent of Total
Depreciation and interest.................	$496	6½
Wear and tear on rope..................	486	6½
Wages of hookers-on....................	3504	46
Two engine drivers..	734	10
Roadman............................	365	5
Upkeep, repairs, oil, cleaning, waste......	116	2
Cost of power........................	1825	24

This cost works out at 1¼c. per ton-mile.

Second Rope Haulage

Conditions—Tonnage, 2100 in 10 hr.; distance, 3 mi. on a level track.
Cost of Plant—Including engine, 250-hp., and accessories, $48,666.
Working Expenses—Annual amount, $15,069.

		Approximate Per Cent of Total
Interest and depreciation at 10 per cent....	$4866	32
Wear and tear of rope....................	851	5½
Upkeep, repairs, cleaning, oil waste........	618	4
New parts for rope haulage and rope clips..	837	5½
Wages of hookers-on.....................	4088	27
Engine driver..........................	296	2
Cost of power..........................	3513	23

This works out to ¾c. per ton-mile.

Overhead Chain

Conditions—Tonnage, 1750 per day of 16 hr.; level track 1¼ mi. long.
Cost of Plant—Machinery, including 65-hp. engine, $6082; chain, $2929; total, $9011.
Working Expenses—Annual amount, $13,828.

		Approximate Per Cent of Total
Interest and depreciation................	$876	6
Interest and depreciation of chain.........	525	4
Wages of hookers-on.....................	7878	57
Repairs and upkeep of plant..............	111	1
Oil and waste..........................	141	1
Cost of power..........................	1786	13
Upkeep of haulage road..................	2501	18

This is almost exactly 2c. per ton-mile.

Animal, compressed-air and electric haulage costs.—A most interesting and valuable comparison of the cost of animal, compressed air and electric haulage at a mine in Western Pennsylvania, producing 35,000 tons a month was worked up in 1912. The grades in this mine are variable, some of them in favor of, and some of them against the loads, and are in places as steep as 6 per cent. The capacity of the mine cars is two tons, and their empty weight 2700 lb. In the case of the electrically operated mine, the car capacity was not definitely stated, but presumably it was about 3000 lb., and the weight of the car itself about 1300 lb. A comparison of figures for the two cases is given in Tables I and II.

TABLE I

ACTUAL COSTS ON THE BASIS OF 553 LOCOMOTIVE HOURS PER MONTH, OR TWO SHIFTS PER DAY, FOR A 4-TON ELECTRIC LOCOMOTIVE AND A 6-TON COMPRESSED-AIR LOCOMOTIVE IN GATHERING SERVICE

	Hours		Hourly Rates		Amount, Dollars		Tons		Cost per Ton	
	Electric	Air	Electric	Air	Electric	Air	Electric	Air	Electric	Air
Locomotive runners	553	553	$22\frac{1}{2}$	$30\frac{5}{9}$	124.43	168.97	4045.19	10,404.6	0.0307	0.0162
Brakemen	692	553	20	$28\frac{8}{9}$	138.40	159.76	4045.19	10,404.6	0.0342	0.0153
Total labor					262.83	328.73	4045.19	10,404.6	0.0649	0.0315
Supplies					18.42	78.00	4045.19	10,404.6	0.0045	0.0075
Repairs and maintenance					18.00		4045.19		0.0045	
Power					20.00	117.56	4045.19	10,404.6	0.0050	0.0113
Interest 6% on $10,000 for one month					50.00	......	4045.19	......	0.0123	
Interest 6% on $5900 for one month					......	29.50	......	10,404.6	......	0.0028
Depreciation eight years					35.42	......	4045.19	......	0.0088	
Depreciation 10%					......	49.17	......	10,404.6	......	0.0047
					404.67	602.96	4045.19	10,404.6	0.1000	0.0578

TABLE II

RELATIVE COST PER MILE AND DATA FOR ESTIMATE

Average haul with electric locomotive	3600 ft.
Average haul with compressed air locomotive	2100 ft.
Total ton-miles, electric locomotive	2758
Total ton-miles, compressed air locomotive	4138
Total cost per ton-mile, electric locomotive	14.68c.
Total cost per ton-mile, compressed-air locomotive	14.57c.

Working two shifts is a decided advantage to either electric locomotives or compressed-air locomotives as compared with mules, because mules cannot be worked two shifts, whereas locomotives can, thereby distributing the increased interest and depreciation charges over double the number of hours. As a matter of fact, the compressed-air locomotives at this mine only worked one shift, but the costs per hour and per ton have been held exactly the same as they actually were with the exception that the interest and depreciation are distributed over 553 hours per month and over the amount of coal that would have been gathered in 553 hours had the locomotives continued to work at the same rate which they maintained for nine hours per day and six days in the week. This was done in order to make the costs more truly comparable.

It should be observed in connection with the above figures for cost per ton and ton-mile that neither of these costs is in general an accurate basis for testing the comparative merits of the two types of haulage, particularly as regards gathering service, although the same considerations apply to a more limited extent in connection with main haulage. When gathering, the locomotive necessarily spends most of its time in assembling the trip, while the ton mileage is run up by the long haul between the point where the coal is gathered and the point where it is delivered. The expense of haulage does not increase directly in proportion to the length of the haul because the time spent in gathering remains a constant regardless of the length of the direct run. Usually if an analysis is made of the power expended and time spent in gathering and in hauling the gathered cars to the terminus, it will be found that the strictly gathering service is the most expensive part of the work.

Changing those items in the foregoing table which are either manifestly not chargeable to the gathering type of locomotive or else need correction, Table IV is derived. The figures are for the same locomotives working under the same conditions as before, but with the engineers and brakemen receiving the same rate of pay, with the charge for power increased from \$20 to \$50 per month, and the item depreciation 8 yr., \$10,000 raised to \$84.20 per month. The reason for changing the rate of \$20 per month for power is explained later.

Based on the corrected and equalizing figures of Table IV the cost per net ton-mile would be 17.55c. for electric locomotive and 12.32c. for the compressed-air locomotive.

In the same mine where the compressed-air gathering locomotives operate, horses are used for gathering coal in another part of the mine. The coal gathered by both the small air locomotives and the horses is hauled to the foot of the shaft by large compressed-air locomotives. The cost of gathering by horses in this mine is as follows:

TABLE III

COSTS OF GATHERING BY HORSES

	Day	Week	Cost per Ton
Nine drivers at \$2.60 per day	\$23.40	\$140.40	
One driver at \$2.70 per day	2.70	16 20	
Total labor	....	\$156.60	\$0.0318
Stable expense	....	83.60	0.0170
Thirteen horses, \$3250 depreciation 5 years	....	12.50	0.0025
Interest, 6% on \$3250	....	3.75	0.0007
		\$256.45	\$0.0520

With the ten drivers, eleven work horses and two spares, 4918 tons of coal were gathered per week and hauled an average distance of 900 ft., giving the cost per ton as stated above in detail and a total cost per ton-mile of 30.6c.

The cost per ton for gathering with mules or horses in the electrically operated mine was as follows:

TABLE IV

ACTUAL COSTS CORRECTED FOR COMPARISON—COMPRESSED-AIR LOCOMOTIVE AND ELECTRIC LOCOMOTIVE IN GATHERING SERVICE

	Hours		Hourly Rates		Amount, Dollars		Tons		Cost per Ton	
	Electric	Air	Electric	Air	Electric	Air	Electric	Air	Electric	Air
Locomotive runners	553	553	22½	22½	124.43	124.43	4045.19	10,404.6	0.0307	0.0120
Brakemen	692	553	20	20	138.40	110.60	4045.19	10,404.6	0.0342	0.0106
Total labor	....	....	....	....	262.83	235.03	4045.19	10,404.6	0.0649	0.0226
Supplies	....	....	....	....	18.42	78.00	4045.19	10,404.6	0.0045	0.0075
Repairs and maintenance	....	....	....	....	18.00		4045.19		0.0045	
Power	....	....	....	....	50.00	117.56	4045.19	10,404.6	0.0123	0.0113
Interest 6% on $10,000 for one month	....	....	....	....	50.00	....	4045.19	....	0.0123	
Interest 6% on $5900 for one month	....	....	....	....	......	29.50	....	10,404.6	......	0.0028
Depreciation eight years on $10,000	....	....	....	....	84.20	....	4045.19	....	0.0208	
Depreciation eight years on $5900	....	....	....	....	......	49.67	....	10,404.6	......	0.0047
					483.45	509.75	4045.19	10,404.6	0.1193	0.0489

TABLE V

Costs of Mule Haulage

Average distance of mule haul			1200 ft.
Drivers, 250 hr. at 30c	$75.00		
Drivers, 398 hr. at 18c	71.64		
Drivers, 294 hr. at 19c	55.86		Cost per
Drivers, 310 hr. at 20c	62.00		Ton
		$264.50	$0.0646
Stable boss		35.00	0.0086
Blacksmith, shoeing		30.00	0.0074
Feed		79.20	0.0193
Miscellaneous supplies		18.52	0.0043
Depreciation, 5 years		25.00	0.0062
Interest, 6% on $1500		7.50	0.0018
Total		$459.72	$0.1122
Total cost per ton mile			49.4c.

The coal gathered by the mules in this mine was hauled to the pit mouth, an average distance of 2500 ft., by an electric locomotive.

The comparative figures for main haulage in the two mines under consideration, one using electricity and the other compressed air are given in Table VI.

The electric locomotive moved the coal an average distance of 2500 ft. and the compressed-air locomotive hauled it for an average distance of 3400 ft., so the costs per ton-mile, are as follows:

For electricity:
$$\frac{2500 \times 4095}{5280} = 1938 \text{ ton-miles,}$$

$$\frac{209.83}{1939} = 10.83\text{c. per ton mile.}$$

For air:
$$\frac{3400 \times 19{,}688}{5280} = 12{,}677 \text{ ton-miles,}$$

$$\frac{386.31}{12{,}677} = 3.05\text{c. per ton mile.}$$

In explanation of the correction of the figure of $20 per month for power as given in Table I and increased to $50 in Table IV, it should be noted that the costs of power per ton-mile

TABLE VI

MAIN-HAULAGE COSTS—ELECTRIC LOCOMOTIVES AND COMPRESSED-AIR LOCOMOTIVES

	Hours		Hourly Rates		Amount, Dollars		Tons		Cost per Ton	
	Electric	Air	Electric	Air	Electric	Air	Electric	Air	Electric	Air
Locomotive runners	277	277	$22\frac{1}{2}$	$30\frac{5}{9}$	62.32	84.64	4095.10	19,687.7	0.0153	0.0043
Brakemen	346	277	20	$28\frac{8}{9}$	69.20	80.02	4095.10	19,687.7	0.0169	0.0041
Total labor					131.52	164.66	4095.10	19,687.7	0.0322	0.0084
Supplies					2.65		4095.10	....	0.0006	
Repairs and maintenance					7.24	25.39	4095.10	19,687.7	0.0018	0.0013
Other repairs and maintenance					5.71		4095.10	....	0.0014	
Power					20.00	80.18	4095.10	19,687.7	0.0049	0.0041
Interest on investment, 6% on $5000					25.00	....	4095.10	....	0.0061	
Interest on investment, 6% on $8650					......	43.25	....	19,687.7	......	0.0022
Depreciation eight years					17.71	....	4095.10	....	0.0042	
Depreciation eight years on $8650					......	72.83	....	19,687.7	......	0.0037
					209.83	386.31	4095.10	19,687.7	0.0512	0.0197

for gathering and main haulage as previously considered were as follows:

Cost for power in gathering........................	\$20.00
Ton mileage as calculated...	2758

$$\frac{\$20.00}{2758} = \$0.00726 \text{ per ton-mile.}$$

And for main haulage:

$$\frac{\$20.00}{1939} = \$0.01032 \text{ per ton-mile.}$$

It will be seen from these figures that the cost per ton-mile is 42 per cent greater for the main haulage than for the gathering service.

The costs of power per ton-mile for the same two classes of service at the compressed-air operated mine were:

For gathering: $\frac{\$117.56}{4138} = \$0.0284$ per ton-mile.

For main haulage: $\frac{\$80.18}{12677} = \$0.0063$ per ton-mile.

These figures show the cost for gathering with compressed-air locomotives to be 4.48 times as great as for main haulage. There are many reasons why the cost for power per ton-mile for gathering should be much more than for main haulage, but none why it should be less, unless under such a condition as where all the main haulage grades are against the loads and all the gathering is down hill.

In all the above figures it is the ton of coal moved one mile in the desired direction that is considered. In actual service, the cars and the locomotive must be moved as well as the coal, and because the car and the locomotive must move approximately twice as far as the coal in order to get the empty cars back to the loading place, the gross ton-mileage is nearly twice the net ton mileage for main haulage. For gathering work the gross ton-mileage will be from four to six times the net ton-mileage because in gathering cars from the rooms the locomotive must make many movements with only one car or without any cars. While doing this work the percentage of net to gross tonnage is exceedingly low because the locomotive itself forms a large part of the total weight of the moving train.

Any remaining inconsistency in the relative costs of main haulage and gathering service with the compressed-air locomotives is readily accounted for by considering the following adverse conditions of gathering as compared with main haulage: Ordinarily the gathering locomotives has poorer track than the main-haulage locomotive, a greater percentage of curves, more frequent stops and starts, and a somewhat lower efficiency, because of handling only one car at a time instead of being loaded approximately up to its capacity.

From the foregoing considerations it seems certain that the figure of $20 as the cost of power for the electric gathering locomotive needs correction; more especially as this locomotive operated two shifts per day, while the main-haulage locomotive operated only one shift.

The conclusion that, as it cost only 10c. per ton to gather and haul 4045 tons of coal per month with one locomotive working two shifts, and as it cost 16.34c. per ton to gather 4095 tons of coal with mules and haul it 2500 ft. to the pit mouth, there is, therefore, an advantage of not less than 4c. per ton in gathering by electric locomotives, does not seem to have sufficient support to make it generally true.

In the mine using compressed-air haulage, coal was gathered and hauled by mules an average distance of 900 ft. at a cost of 5.2c. per ton and was afterward hauled an average distance of 3400 ft. by main-haulage compressed-air locomotives at a cost of 1.97c. per ton, giving a total cost for gathering and hauling of 7.17c. per ton, a lower figure than that in the electric haulage mine. This figure is also slightly lower than the results achieved with compressed-air gathering and main-haulage locomotives in sections of the same mine where the gathering locomotives worked but one shift, as is shown by the following costs:

Gathering by compressed-air locomotives........	6.72c. a ton
Main haulage................................	1.97c. a ton
Total cost..................................	8.69c. a ton

Thus it is seen that the coal was delivered to the shaft bottom more cheaply by mules and main-haulage locomotives than by gathering locomotives and main-haulage locomotives; but in order to achieve these results with the mules it was necessary to keep the mule haul down to 900 ft. or less in order to enable the animals to gather, as they did, 41.6 two-ton cars per mule. If the mules had to haul the coal an average distance of 1200 ft., as did the gathering locomotives, and had been forced to encounter the adverse grades that the locomotives did, the cost for mule haulage would have been 50 per cent greater, throwing the balance again in favor of the gathering locomotive. In other words, it was possible to obtain the good results that were obtained with mules in this mine only by working them in selected places under the most favorable conditions, all of which goes to show that it is extremely dangerous to draw general conclusions in regard to the relative economy of the various types of haulage unless the conditions are strictly comparable, or accurate corrections are made to compensate for differing conditions.

Single- and two-stage air motors.—The two-stage compressed-air motor will do from 40 to 60 per cent more work with the same amount of air than the single-stage machine. The cost of power for this type motor will thus be about 30 per cent less than with the single-stage, the compressor and boiler capacity will be reduced by about the same amount and the first cost of the installation will be about 15 per cent less, while the motors will travel substantially further on one charge.

A practical test of the difference between single-expansion and two-stage compressed-air locomotives was once made as follows: The same train was hauled by the single-expansion and two-stage locomotives the same distance over the same piece of track, with the same operator, giving the locomotives as nearly as possible the same work to do under the same conditions. In all cases the trains were started from a given point and were allowed to come to rest as near as possible to another given point without the use of brakes. Air consumed was determined by the difference in pressure recorded at the beginning and end of the trip by the gauge on the main reservoir. Running conditions were the same as would obtain in the regular hauling of coal in the mine.

Trial 1 was conducted at the Susquehanna Coal Co.'s No.

7 colliery, Nanticoke, Pa., in the presence of Mr. McMahon, Chief Engineer, Susquehanna Coal Co., and Mr. C. B. Hodges, of the H. K. Porter Co.

TRIAL 1

LOCOMOTIVE DATA

	Single-Expansion	Two-Stage
Weight	10,000 lb.	10,600 lb.
Cylinders, high-pressure, diameter	6 in.	5 in.
Cylinders, low-pressure, diameter	....	10 in.
Cylinders, stroke	10 in.	10 in.
Driving wheels, number and diameter	4–23 in.	4–23 in.
Working pressure, high-pressure cylinder	150 lb.	250 lb.
Maximum charging pressure	900 lb.	900 lb.
Capacity of main reservoir	41 cu. ft.	41 cu. ft.
Age of locomotive	8 months	1 month

TRAIN AND ROAD DATA

Length of trial run, feet	2200
Average grade, per cent	0.96
Maximum grade, per cent	2.00
Track gage, inches	42

Train consisted of loaded coal cars weighing about 9500 lb. each.

The excessively high efficiency indicated on lines Nos. 4 and 6 of the table of log of runs, hauling four and five cars down grade, may be due to the fact that considerably more air is wasted in the single-expansion locomotive when giving the train just a little assistance when the grade is nearly steep enough to cause it to run down by itself.

Lines Nos. 7 and 8 show additional trips up grade with the two-stage with five-car trips. The trip with the two-stage (line 5) was made with the reverse lever "in the corner," using the air full stroke all the way. The superior efficiency achieved on trips shown on lines Nos. 7 and 8 was the result of using the air as expansively as possible.

Taking the total pressure reduction of the three round trips with the single-expansion locomotive and two-stage locomotive (lines 1, 2, 3, 4, 5, and 6), we find that the two-stage locomotive used $\frac{1245}{2225}=56$ per cent of the air used by the single-expansion engine under exactly the same conditions of service, this average per cent for the entire test showing a saving of 44 per cent of the air used by the single-expansion machine.

In making the test every care was used to obtain reliable results. The pressure gauge on the two-stage locomotive was removed and placed on the single-expansion locomotive during the test of this locomotive, and then shifted back to the two-stage for testing it. This gauge was a comparatively new one, and presumably correct, and if any error did exist, it would have been the same for both locomotives. The tanks were exactly of the same capacity. The same engineer operated both locomotives alternately during the trials.

Trial 2 was conducted at Orient Mine of the Orient Coke Co., Orient, Fayette County, Pennsylvania, in the presence of Mr. Chas. Opperman, of the Orient Coke Co.; Mr. G. E. Huttelmaier, of the H. C. Frick Coke Co.; Mr. C. B. Hodges, of the H. K. Porter Co.

TRIAL 2

LOCOMOTIVE DATA

	Single-Expansion	Two-Stage
Weight	9600 lb.	10,500 lb.
Cylinders, high-pressure, diameter	6 in.	$5\frac{1}{2}$ in.
Cylinders, low-pressure, diameter	....	11 in.
Cylinders, stroke	10 in.	10 in.
Driving wheels, number and diameter	4–23 in.	4–23 in.
Working pressure, high-pressure cylinder	150 lb.	250 lb.
Maximum charging pressure	800 lb.	800 lb.
Storage tanks	1	1
Capacity of main reservoir	40.26 cu. ft.	40.26 cu. ft.
Age of locomotive	6 months	2 months

TRAIN AND ROAD DATA

Length of trial run, feet	2500
Average grade, per cent	.52
Track gage, inches	44

In this run there was a reverse curve in a chute leading from one heading to a parallel heading. The train consisted of four loaded wagons, each about 7000 lb.; and six empty wagons, each about 2200 lb.

Log of Trial Runs

No. of Run	Tank Pressures: Type of Locomotive	At start	At finish	Amount of Drop	Time (P.M.): Start	Finish	Elapsed
1	Single-expansion.	705	265	440	7:23½	7:27½	.04
2	Two-stage......	740	420	320	8:06½	8:12	.05½
3	Two-stage......	685	385	300	8:48	8:52¾	.04¾

No. 1 run: Very satisfactory.

No. 2 run: Very irregular; operator not so familiar with two-stage machine, hence decided to rerun.

No. 3 run: Much better and smoother than second.

Deductions

Calculated by Mr. G. E. Huttelmaier, H. C. Frick Coke Co.

	Trial No. 1	Trial No. 2
Free air consumed, cubic feet............	1,206.92	877.76
Drawbar effort { Loco. 97.92, Trip 832.24 }	930.16	{ 107.10, 832.24 } 939.34
Total work performed in foot-pounds......	2,325,400.00	2,348,350.00
Foot-pounds performed per minute.......	581,350.00	426,973.00
Average speed { Feet per minute	625.00	454.50
Miles per hour }	7.10	5.17
Average horsepower developed..........	17.61	12.94
Foot-pounds work per cubic foot free air....	1,926.00	2,676.00
Free air consumed per minute............	301.70	159.59
Air per minute per horsepower..........	17.13	12.33
Percentage of air consumed as compared with Trial No. 1......................	100 per cent	72 per cent
Amount of work per unit of air compared to Trial No. 1.......................	100 per cent	1.38 per cent
Saving of air effected over Trial No. 1....	....	28 per cent

LOG OF RUNS—WITH PRESSURE REDUCTIONS AND PERCENTAGE USED BY TWO-STAGE LOCOMOTIVE

Line	Single-Expansion					Two-Stage					Percentage Used by Two-Stage	
	No. Cars	Gage Pressure Start	Gage Pressure Stop	Pressure Reduction	Time	No. Cars	Gage Pressure Start	Gage Pressure Stop	Pressure Reduction	Time		
1	3	690	280	410	3:15	3	680	400	280	.....	68.3	Up grade
2	3	280	140	[illegible]	.....	3	400	320	80	.....	57.1	Down grade
3	4	770	180	590	3:30	4	570	250	320	2:40	54.2	Up grade
4	4	760	540	220	.....	4	250	175	75	.....	34.1	Down grade
5	5	805	200	605	2:55	5	710	290	420	2:50	69.5	Up grade
6	5	730	470	260	.....	5	290	220	70	.....	26.9	Down grade
				2252					1245			
	Total pressure reduction 2225 lb.—3 round trips					Total pressure reduction 1245 lb.—3 round trips						
7	...	.....	.....	....	.....	5	610	220	390	.....	....	Up grade
8	....	.....	.....	....	.....	5	735	360	375	3:15	....	Up grade

Compressed-air and animal haulage costs.—The Consolidation Coal Co., in 1902, conducted an investigation into the relative cost of mule and compressed-air haulage, the results of which were described in the transactions of the A.I.M.M.E., Vol. 34, p. 144.

Five of these machines were placed in the company's Ocean No. 3 mine (Hoffman), displacing a number of mules, but leaving 19 still working. This opportunity was embraced to make a close comparison between the two methods of gathering.

The mules working in the North Heading and the South Heading deliver their cars directly to the rope on the slope The other mule routes deliver to the heavy motors, as do all the motor routes. The mules used weigh from 1200 to 1400 lb., and pull an average of 2.4 long tons.

The following table shows the work performed by the mules during a period of 18¾ working days in the month of December, 1902:

Route	Cars Moved	Average Haul, Feet	Constant	Tons Moved 1000 Feet
South heading	1119	2900	$\frac{2.4}{1000}$	7788.24
North heading	268	1300	do.	836.16
First cross	1629	2100	do	8210.16
Second cross	3042	1100	do.	8030.88
Third cross	747	400	....	717.12

This represents a total of 339 days' work for one mule. The company's accounts show a cost of $1.15 per day for each day worked by a mule, including expense of replacing worn-out animals. Drivers are paid $1.98, and there is one with each mule. This makes a cost of $3.13 per day for each day worked by a mule. The cost per ton hauled 1000 ft. would therefore, be:

$$\frac{339 \times \$3.13}{25{,}582.56} = 4.15\text{c.}$$

For the work of the motors during the same time we have.

Route	Cars Moved	Average Haul, Feet	Constant	Tons Moved 1000 Feet.
Tippens	1122	2300	$\frac{2.4}{1000}$	6,193.44
Scobies	1073	2050	do.	5,279.16
First Klondyke	1147	1835	do.	5,046.80
Second Klondyke	1032	1800	do.	4,458.24
Third Klondyke	1147	1865	do.	5,138.56
Fourth Klondyke	114	1992	do.	544.92
Total	....	....	....	26,661.12

This work was done by the five small motors operated by compressed air, working a total of 94 days. This plant is supplied with steam by a battery of boilers, which also supplies steam to the large pumps. The plant consists of the following items, with their approximate first cost:

One straight-line Norwalk air-compressor, 18 and 28 compound steam, 18½, 13½, and 6½ three-stage air, 30-in. stroke	$5,300
5600 ft. of 5-in. pipe	5,600
3100 ft. of 2½-in. pipe	1,700
1000 ft. of 1½-in. pipe	300
2 motors, 30,000 lb. each	6,000
5 motors, 8,000 lb. each	10,000
Estimated proportion of boilers	1,000
Installation	4,000
	$33,900

Allowing $3000 per year for interest and depreciation, to be earned in 300 working-days, would justify a charge of $10 per day from this source against the entire plant.

This same compressor also drives the large motors mentioned above, which weigh 30,000 lb. each (60,000 lb. for the two); the five small machines weigh 8000 lb. each (40,000 lb. for the five). Dividing the general expenses according to the weight would result in four-tenths being charged against the small motors.

These general expenses may be summed up as follows per day:

Coal, 4 tons at $1	$4.00
Fireman	2.00
Mechanic in charge of compressor	2.50
Interest and depreciation	10.00
	$18.50

The cost of operation of the five small motors would then be:

5 motormen @ $2.67	$13.35
5 brakemen @ $2.03	10.15
General expenses $0.4\times\$18.50$	7.40
Repairs and oil	3.00
	$33.90

Dividing this among the five machines would give $6.78 per day for each machine and the cost per ton moved 1000 ft. would be $\frac{6.78\times 94}{26,661.12}=2.44$c.

In the matter of community of service the motors show to great advantage. A broken down motor can usually be repaired over night, while an injured mule can only be replaced by a new one that must usually be broken in and inured to the work before he is thoroughly efficient, entailing loss of time and output in each case.

Electric motor and animal haulage costs.—An interesting and valuable comparison of the cost of electric and mule haulage at one of the mines of the Peabody Coal Co. was worked up in 1907. The introduction of the electric haulage not only resulted in reducing the cost of production, but also made practicable the development of more extended operations and increased the output from 1400 tons to a daily average of 2000 tons. The motors have pulled as much as 2570 tons in 8 hr.

Prior to installing electric haulage there were 16 gathering mules and 17 mules working in spike teams, pulling to the bottom, producing 1400 tons of coal per day. Owing to the size of the cars, grade and average haul of 1800 ft. to the bottom of the shaft, the output had reached its limit with mule haulage, and it was decided to install electrical haulage. Two 15-ton traction locomotives with double-end control were installed. The locomotives have pulled 17 loaded cars up a 2½ per cent grade 1200 ft. long. These cars weigh when empty 1950 lb. and hold on an average 6600 lb. coal, so the weight of the loaded trip would be over 72 tons.

The power for operating the motors in the mine is supplied by a 175-kw. generator belted to a 200-hp. high-speed engine, 18 × 18 in. The generator also furnishes light for the underground haulage-ways.

Steam to run the electric plant is furnished by a battery of four 150-hp. tubular boilers, which also furnishes steam for the large hoisting engines; but in order to make the proper comparisons between mule and electric haulage, the cost of two complete power units has been added to the electrical equipment. The machinery making up the electrical installation is as follows:

Cost of Electric Installation

Two 15-ton locomotives @ $2300	$4,600.00
One 175-kw. generator and switch-board	2,400.00
One McEwen engine, 18×18 in., 200 hp	2,000.00
Foundations and placing engine and generator	300.00
Two 72 in.×18 ft. tub. boilers, 150 hp., complete	2,800.00
9000 ft. trolley wire	1,019.90
200 ft. 400,000 cm. lead cable @ 55c	110.00
635 trolley hangers @ 65c	432.25
768 bonds @ 35c	268.80
75 crossbonds @ 35c	26.25
18 interchangeable trolley frogs @ $2.75	49.50
1 extra 250-volt armature	375.00
2 motor jacks @ $12.80	25.60
Extra fittings for motors	86.24
116½ tons 40-lb. rail @ $28.25, $3,291.13. Cr. for 25-lb. rails, $2,056.75	1,234.38
6055 white-oak ties @ 10c	605.50
65 kegs 4½×½-in. spikes @ $3.75	244.50
22 split switches, material and labor @ $17.00	374.00
Fish plates and bolts	280.00
Lumber for trolley supports	76.11
Sundries	3,810.21
Entire labor cost	3,810.21
Total of complete installation	$21,172.79

Cost of Mule Haulage

Mules, average cost	$225.00
Mules, depreciation	20 per cent
Mules, interest	6 per cent

Cost per Mule, Divided for 275 Work Days, per Work Day

Depreciation	$0.163
Interest	0.049
Feed	0.20
Shoeing and stableman	0.158
Total per day, 275-day basis	$0.57

Team Haulage, No. 3 Mine Daily Average Tonnage, 1400 Tons

17 mules @ $0.57	$9.69
9 drivers @ 2.56	24.24
Total	$33.93

Team drivers, 15c. extra, or $1.20 extra for 8.
Cost per ton outside mule haulage, 2.4c.

Cost of Operating Electrically, 275 Work Days

2 locomotive runners @ $3.20	$6.40 per day	
2 trip riders @ $2.56	5.12 per day	
⅓ electrician @ $75 per month	1.08 per day	
⅓ fireman @ $2.02	0.67 per day	
Fuel, 5 tons @ 75c	3.75 per day	
Total fixed labor, etc	$17.02 per day	
Interest on investment, $21,172.79 @ 6 per cent		$4.62
Depreciation and repairs @ 8 per cent		6.16
Oil and waste		0.30
Taxes		0.50
Total others		$11.58
Total daily operating cost electrically		$28.60
Cost per ton based on 2000 tons		1.4c

It will be seen from the foregoing that the plant, besides increasing the output and saving 1c. per ton, which means $20 per day, practically pays for itself in four years.

The following figures show a comparison of cost previous to 1910, between mules and an electric locomotive at a mine where 14 mules were replaced by one locomotive. The output of the mine averages 1500 tons per day for 245 working days per year. The cars weigh 2400 lb. empty and hold 3600 lb., making a total weight of 6000 lb.

Mule Haulage

14 mules @ $180 each	$2,520.00
14 sets harness @ $25 each	350.00
	$2,870.00

Interest and Depreciation

20 per cent depreciation on $2870	$574.00
6 per cent interest on $2870	172.20
	$746.20

Working Expenses for 245 Days

14 mules feeding, shoeing, repairing harness and care @ 50c. per mule, per day	$1,715.00
6 drivers @ $2.80 per day	4,116.00
	$5,831.00
	$5,831.00
	746.20
	$6,577.20

Fifteen hundred tons per day for 245 days equals 367,500 tons per year at a haulage cost of 1.8c. per ton.

Electric Haulage

Engine, locomotive, boiler and generator	$9,000.00
Switches, insulators and wire	1,200.00
Cost of erecting, etc	1,000.00
	$11,200.00

Interest, Depreciation, etc.

Interest at 6 per cent on $11,200	$672.00
Depreciation on boiler, engine, etc. @ 9 per cent	810.00
Repairs on boiler, engine, etc., @ 9 per cent	810.00
Depreciation on switches, wire, etc., @ 5 per cent	110.00
Repairs on switches, wire, etc., @ 5 per cent	110.00
	$2,512.00
Engineer power house @ $75 per month	$900.00
Motorman @ $2.80 per day	686.00
Oil and waste	100.00
Nipper on motor @ $1.50 per day	367.50
Sand	50.00
	$2,103.50
From above	$2,512.00
Total	$4,615.50

Fifteen hundred tons per day for 245 days equals 365,700 tons per year. This makes the haulage on each ton of coal, where electric locomotives are used, cost 1.27c. per ton. These estimates, taken from an actual case, show a considerable difference in favor of electric haulage. The cost of installing mechanical haulage is greater than when a mine is supplied with mules; however, when we consider the cost of erecting a stable and the great loss due to mules killed in accidents. the initial expenditure is not so favorable to the use of mules.

Cost and care of mules.—Frank Amos, of the Fairmont Coal Co., in 1911, made the statement "that the average life of a mine mule was 3½ yr., and unless conditions were changed to prolong life, the use at the present cost of the animal was unprofitable."

When an animal of this kind lives indefinitely on the farm it seems incredible that his life should be shortened to 3½ yr. in the mine. There are mules to-day in anthracite mines that have been working 20 yr., yet it is probable that the average life of such animals in all anthracite mines does not exceed 5 yr.

In purchasing stock for underground haulage, activity, eyesight, feet, temperament, strength, and wind are considerations, but if the animal lacks intelligence he has no place in the mine. In most instances it is better to deal with those who make a business of furnishing mules to mining companies, and if it is possible, to go to the stock yards and pick out the animals rather than trust to the dealer's judgment. This suggestion is made because the dealer scarcely knows more about the animals than the purchaser, and does not know the conditions under which the animals must work.

Many animals which act and look right on the surface are most unsatisfactory underground, for which reason the suggestion is made that after the animals are picked out an agreement should be made with the dealer that in case any of them do not act rightly in the mine they may be exchanged.

While a mule's heels are not to be trusted, nevertheless the humane driver and his mule become good chums. If, after a 3-day training period the mule does not appear active on his feet and to use judgment, he should be taken from the mine, as he is unsuited to the work and cannot be depended upon to look out for himself when occasion demands.

According to two authorities the animals should be fed as follows: First, hay, next water, and then grain. Animals should eat hay at least half an hour before being given grain. If the water is given last it washes the food into the intestines before it is acted upon by the gastric juices. If the hay is given after the grain it carries the grain with it, for the hay is principally digested in the intestines, while the grain is acted upon by the stomach for the most part.

Corn is richer in fat than oats; therefore, for strength, feed corn, and for speed, feed oats. For an illustration, race horses are fed oats, and the experienced teamster will favor the feeding of corn.

Dr. I. C. Newhard, Chief Veterinarian of the Philadelphia & Reading Coal and Iron Co., in 1911, experimented with various feeds and found that two-thirds crushed oats and one-third cracked corn the most reliable. "A handful of coarse ground pure salt should be fed to each mule twice a week." Dr. Frank Amos, who is in West Virginia, suggests a coarse-crushed feed, about two-thirds corn and one-third oats.

Mine stock will consume about 12 lb. per head per day of this feed and about 15 lb. of hay. If a horse or a mule has not cleaned up its former feed the troughs should be cleaned and less put in the next time, until it is ascertained just how much it takes to keep them. The animal should have about all it will eat, but it is better to give not quite enough than too much. Too much grain will cause acute indigestion, paralyze the walls of the stomach, and usually results in death. A stable boss will make a great mistake by feeding too much and allowing food to stand before the animals all the time. While this method will increase flesh for a short period the animals eventually break down through their digestive organs being destroyed.

Grain should not be placed in the animals' troughs ready for them when they come in from work, and it is better for them to be without grain at noontime than to be without water, but by all means give them three feeds a day. Plenty of water will keep the digestive organs in good condition, while large quantities of grain and no water will destroy them.

To give a feed of good bran once a week will aid the conditioning of stock, keep the bowels open, and reduce fever, which is caused by strong grain.

The Fairmont Coal Co.'s records for 1905 show that 26 per cent of their stock either died, was killed, or had to be disposed of at practically nothing, on account of being crippled and worn out. There is probably no part of the company's business in which the loss is so great, and one-half of this is brought about by carelessness and neglect. There is probably no other business that requires the use of stock in which the loss is so great, and this in face of the fact that the facilities furnished, with the exception possibly of good roads inside the mines, are the best that can be had.

The maintaining of live stock is no little item, and in cases there is an average of 5 per cent of the total number of animals standing in the barns all the time unfit for service on account of having been crippled. The feed for this stock, besides other expenses and the loss of their work, cost one company in 1911, over $6000 per year.

Where mine stock is given good attention, the upkeep is reduced to a minimum, more work is obtained, and the animals are more valuable.

Professor Ihlseng estimates the capacity for work of the mine mule on a level track as between 30 and 80 gross ton-miles in 8 hr. and the work of the average mule for the same time as between 40 and 50 gross ton-miles (5 to 6 ton-miles per hour) with a limiting grade of 3 per cent. Hughes gives examples varying all the way from 3 to 16 ton-miles per hour on level track. Other conditions being equal the condition that really determines the work that a mine mule can do is the length of the haul, since a large part of the time is consumed in changing and waiting for trips. The average of a number of actual working conditions shows that 4 to 6 ton-miles per hour is a good average for the work of an ordinary mine mule, in a flat seam, under usual mine conditions, though this will increase slightly with the length of haul. With a normal load the limiting grade under which a mule can be worked to an advantage is about 3 or 4 per cent. As loaded mine cars will slide on iron rails, with 4 sprags, on grades varying from 6 to 8 per cent the safe down grade for a mule to work on is limited to this.

Figures gathered from a number of years experience in a level seam $5\frac{1}{2}$ ft. thick show that a good 2-mule team, with an efficient driver and helper, will handle from 50 to 60 cars of $2\frac{1}{2}$ tons capacity in a 10-hr. shift when the haul does not exceed 2000 ft. Allowing two hours lost time and assuming the weight of the empty car to be $1\frac{1}{2}$ tons each mule has accomplished 7 ton-miles of work per hour.

An additional expense that must be allowed for in animal haulage in low seams is the cost of brushing to obtain sufficient headroom. Haulage expenses have been increased as much as 3c. per ton (about 1912) from this cause and the decreased efficiency of the animals.

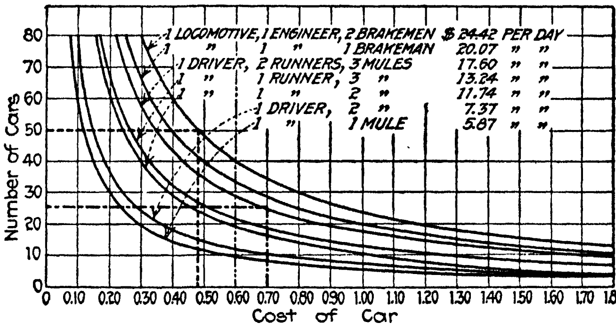


FIG. 22.—Chart showing relative economy of mule and motor haulage.

Additional ventilation must also be provided where animal haulage is used, most state mining laws providing that 500 cu. ft. per min. be allowed for each animal in the mine. In a mine using 50 mules this would mean that an additional 25,000 cu. ft. of air per minute be provided.

The working life of the average mule is seven to ten years. Because of the hard working conditions and generally frequent accidents it is necessary to keep a good reserve supply of animals available which increases the investment, as well as costs of maintenance.

The chart shown in Fig. 22, will give a quick approximate solution of the relative economy of mule and motor haulage where the conditions are known. Other factors, such as availability of power and the possibility of securing locomotives and equipment are matters for consideration after or before the relative costs of the different organizations are determined. The curves are made up from figures similar to those appearing in Table I:

TABLE I

COST PER CAR FOR ONE DRIVER AND ONE MULE

Cars	Cost	Cars	Cost	Cars	Cost	Cars	Cost
4	$1.47	8	$0.73	16	$0.37	30	$0.195
5	1.17	12	.49	18	.33	40	.147
6	.98	14	.42	21	.28	60	.098

These when plotted give the basis for the regular curve for an organization or transportation unit of one driver and one mule. The other curves shown are plotted in a like manner. In the case of locomotive haulage, the power, installation and repair costs given or estimated for any single locality will not be correct for every installation. This is one weak point in the tabulation. However, because unforeseen conditions usually develop in mine electric installations, it is best to be on the safe side and figure locomotive maintenance high.

A close inspection of these curves will also show that the handling of four or five more cars makes much more difference in the cost per car than does a few cents in the figures representing power, repairs, etc. For instance, in the case of a crab locomotive with one engineer and two brakemen working in pitching rooms, let us estimate that this machine will handle 40 cars per day. This, from Fig. 22, would cost 60c. per car. If another place can be added so that the machine will handle 44 cars, it will do so at a cost of 56c. per car—a saving of four cents per car, or $1.76 per day. We cannot always estimate mine haulage possibilities closer than four cars, but our cost figures certainly will not be in error to the amount of $1.76.

Two representative problems are given below to illustrate the use of the curves:

Problem 1—Assumption by colliery superintendent that his transportation gathering cost shall not be over 60c. per car.

How many cars must each unit handle in order to give him this cost?

Follow the dash line on Fig. 22: It crosses one driver and one mule at 10 cars; one driver and two mules at 12 cars; one driver, one runner and two mules at 19 cars; one driver, one runner and three mules at 22 cars; one driver, two runners and three mules at 29 cars; one locomotive, one engineer and one brakeman at 34 cars; one locomotive, one engineer and two brakemen at 40 cars.

		Per Day
Locomotive Data:		
Power, $90 per month..............................		$3.60
Installation, $2000, 5 per cent of $2000	$2200 pr. yr.	
Depreciation –Locomotive, 15% of $8000		
Repairs—Locomotive lines, etc., $900 per year		7.33
Labor—Engineer at $0.28, plus $0.25 allowance for 9 hrs.		4.77
Brakeman at $0.2351, plus $0.25 allowance for 9 hours..............................		4.36
Mule Data:		
Mule value, $250; 20% depreciation; 5% interest....		0.23
Feed and care..............................		1.25
Labor—Driver at $0.2351, plus $0.25 allowance.		
Runner at $0.2351, plus $0.25 allowance.		

One driver, two runners and three mules in a certain section where there are pitching rooms will handle 25 cars per day. This section due to its development can produce 50 cars per day with two-mule organizations. Which relatively is the cheaper plan, to use two-mule organizations or one crab locomotive to handle 50 cars?

By following the two dotted lines in Fig. 22 it will be seen that one driver, two runners and three mules will handle 25 cars at a cost of 70c. per car; one locomotive, one engineer and two brakemen will handle 50 cars at a cost of 48c. per car. In this case, of course, the locomotive is the better installation.

Close inspection of the curves in Fig. 22 will also show that there is a certain fairly definite limit to the number of cars handled, below which the cost per car increases rapidly. This limit can be taken from the curves by noting the point at which the curve from right to left deviates from practically a straight line. Thus, for instance, in the case of the one driver and one mule curve, the line is practically straight from the right hand side of the figure to the 10 or 12 car point, and from here turns rapidly into a curve. This possibly is better illustrated by Table II.

Taking the limit for the differences or decrease in cost per car at 4c., we see that 12 cars must be handled by one driver and one mule, 18 cars by one driver, one runner and three mules, etc., before this limit is reached. Taking 2c. as the limit 16 cars must be handled by one driver and one mule, and a corresponding number by other units before reaching this limiting figure.

TABLE II

Cost per Car for Different Units for Increasing Number of Cars

Cars	1 Driver, 1 Mule	Decrease	1 Driver, 2 Mules	Decrease	1 Driver, 1 Runner, 2 Mules	Decrease	1 Driver, 1 Runner, 3 Mules	Decrease	1 Driver, 2 Runners, 3 Mules	Decrease	1 Engineer, 1 Brakeman, 1 Locomotive	Decrease	1 Engineer, 2 Brakemen, 1 Locomotive	Decrease
1	5.87		7.37		11.74		13.24		17.60		20.07		24.42	
2	2.93	2.94	3.68	3.69	5.87	5.87	6.62	6.62	8.80	8.80	10.03	10.04	12.21	12.21
3	1.96	.97	2.46	1.22	3.91	1.96	4.41	2.21	5.87	2.93	6.69	4.34	8.14	4.07
4	1.47	.49	1.84	.62	2.94	.97	3.32	1.09	4.40	1.47	5.02	1.67	6.11	2.03
5	1.17	.30	1.47	.37	2.35	.59	2.65	.67	3.52	.88	4.01	1.01	4.89	1.22
6	.98	.19	1.22	.25	1.96	.39	2.21	.44	2.94	.58	3.35	.66	4.07	.82
7	.84	.14	1.05	.17	1.68	.28	1.89	.32	2.52	.42	2.87	.48	3.49	.58
8	.74	.10	.92	.13	1.47	.21	1.65	.29	2.20	.32	2.51	.36	3.06	.43
9	.65	.09	.82	.10	1.31	.16	1.47	.18	1.96	.24	2.23	.28	2.72	.34
10	.59	.06	.74	.08	1.17	.14	1.32	.15	1.76	.20	2.01	.22	2.44	.28
11	.53	.06	.67	.07	1.07	.10	1.20	.12	1.60	.16	1.83	.18	2.22	.22
12	.49	.04	.61	.06	.98	.09	1.10	.10	1.46	.14	1.67	.16	1.04	.18
13	.45	.04	.57	.04	.91	.07	1.02	.08	1.35	.11	1.54	.13	1.88	.16
14	.42	.03	.53	.04	.84	.07	.95	.07	1.26	.09	1.44	.10	1.74	.14
15	.39	.03	.49	.04	.78	.06	.88	.07	1.17	.09	1.34	.10	1.63	.11
16	.37	.02	.46	.03	.73	.05	.83	.05	1.10	.07	1.25	.09	1.52	.11
17	.35	.02	.43	.03	.69	.04	.78	.05	1.03	.07	1.18	.07	1.44	.08
18	.33	.02	.41	.02	.65	.04	.74	.04	.98	.05	1.11	.07	1.36	.08
19	.31	.02	.39	.02	.62	.03	.70	.04	.93	.05	1.05	.06	1.28	.08
20	.29	.02	.37	.02	.59	.03	.66	.04	.88	.05	1.00	.05	1.22	.06
21	.28	.01	.35	.02	.56	.03	.63	.03	.84	.04	.96	.04	1.16	.06
22	.27	.01	.33	.02	.53	.03	.60	.03	.80	.04	.92	.04	1.11	.05
23	.26	.01	.32	.01	.51	.02	.57	.03	.76	.04	.88	.04	1.06	.05
24					.49	.02	.55	.02	.73	.03	.84	.04	1.02	.04
25					.47	.02	.53	.02	.70	.03	.80	.04	.98	.04
26					.45	.02	.51	.02	.67	.03	.77	.03	.94	.04
27					.43	.02	.49	.02	.65	.02	.74	.03	.90	.04
28					.42	.01	.47	02	.63	.02	.72	.03	.87	.03

The following are the results obtained through the installation of five storage-battery locomotives in the Red Ash vein at Exeter Colliery of the Lehigh Valley Coal Co.*

In the fifth vein, one locomotive, operating from inside slope to chambers, gangways and airways on roads Nos. 1002 and 1006, with chambers pitching both ways and grades in some places as high as 9 per cent, handles fifty cars with a maximum run of about 1800 ft. When replacing the empty cars this locomotive is assisted by a second locomotive. The latter also covers chambers, airways and gangways on roads Nos. 1001 and 1006 and handles sixteen cars with a run of 1300 ft. in each gangway with grades ranging as high as 7 per cent. To replace these two motors with mule power would take fifteen mules, five drivers and five runners.

In the Babylon vein, one motor hauls from the various working faces on roads Nos. 144 and 147 and handles twenty-five cars daily, having a maximum run of 2100 ft., and delivers coal to the head of No. 9 plane. It would take nine mules, three drivers and three runners to replace this motor.

The fourth locomotive collects thirty-two cars from roads Nos. 39, 46 and 50 and delivers coal to a big turnout, having a maximum run in each road of 3500 ft., 2700 ft., and 3000 ft. respectively. Nine mules, three drivers and three runners would be required to replace this motor.

The fifth locomotive, operating as a collecting locomotive on road No. 5 and in working faces between chambers Nos. 33 and 47, and assisting in concentrating coal from roads Nos. 65 and 4, handles thirty cars daily over a run of 1000 ft. having grades up to 5 per cent against loaded trips. It would require six mules, two drivers and two runners to replace this motor.

As compared with mule haulage there is case on record in the Ohio fields where one storage battery 6-ton motor replaced 12 mules and four drivers, handling 85 to 100 cars on grades up to as high as 17 per cent it being necessary to sand the track on the steeper grades of course. The economy effected in this case, after generous allowances for depreciation, is obvious.

* Extract from "The Storage Battery Locomotive for Gathering Purposes" in *Employees Magazine* of the Lehigh Valley Coal Co.

Gasoline motor vs. animal haulage.—The Long Branch Coal Co. installed a gasoline-motor haulage system at its mine in West Virginia, that materially reduced haulage costs and at the same time increased the output. The supplanting of the old system of mule haulage was done in 1913 with the idea of cutting down operating costs. The company purchased a 7-ton gasoline locomotive which was put into service in the summer of 1913. At the end of the month of August, a comparison was made with the month of February, the most productive month during the régime of mule haulage. A summary of this comparison is given below:

COMPARATIVE COSTS OF THE TWO SYSTEMS OF HAULAGE

Total cost of mule haulage, per month	$810.00
Total cost of gasoline haulage, per month	529.63
Decrease in haulage cost, per month	$280.37
Total coal tonnage by gasoline locomotive, tons	11,601
Total coal tonnage by mules, tons	7,848
Increase in coal output, tons	3,753

The analysis of the above summary is shown by the following detail comparison between the cost of haulage by mules and by a combination of gasoline locomotive and mules for gathering.

Mule Haulage—Month of February, 1913

Length of haul one way, feet	2,000
Maximum grade against loads, per cent	5.625
Average grade against loads, per cent	3.0
Total tonnage per month of 24 days	7848
15 mules—feed and upkeep per day @ 60c	$ 9.00
11 drivers—wages per day @ $2.25	24.75
	$33.75
24 working days @ $33.75—cost per month	$810.00
Total haulage cost per ton of coal	0.103
Total haulage cost per ton-mile	0.272

Gasoline Locomotive Haulage in Connection with Gathering by Mules—Month of August, 1913

Length of haul one way, feet	3000
Maximum grade against loads, per cent	5.625
Average grade against loads, per cent	3.0
Total tonnage per month of 25 days	11,601
Expense of mules and drivers for gathering, 25 working days	$313.20

Cost of Operating Locomotive, 25 Days

1 motorman @ $3 per day	$75.00
1 trip rider @ $2 per day	50.00
300 gal. gasoline @ 20c. per gal	60.00
38 gal. engine oil @ 40c. per gal	15.20
3 gal black oil @ 16c. per gal	0.48
Cup grease	0.50
Waste	0.25
Repairs on motor	15.00
Total operating expense of locomotive	$216.43
Total haulage cost per month	$529.63
Total haulage cost per ton of coal	0.0456
Total haulage cost per ton-mile	0.0803
Cost per ton of coal—mule haulage	0.103
Cost per ton of coal—locomotive haulage	0.0456
Saving per ton of coal	$0.0574
Cost per ton-mile—mule haulage	0.272
Cost per ton-mile—locomotive haulage	0.0803
Saving per ton mile	$0.1917
On a basis of 12 months the cost by mule haulage for one year ($810×12)	$9720.00
By locomotive for one year ($529×12)	6348.00
Yearly saving	$3372.00

With the gasoline haulage system the tonnage of the mine has been increased on an average of 25 per cent per month, and the company has dispensed with 6 double teams or 12 mules and 6 drivers, a total monthly saving in expense of $496.80.

The management of the Midvalley Coal Co. in Pennsylvania, in 1911, effected a saving of 32.2 per cent on coal hauled by substituting the gasoline locomotive for mules.

This comparison seems too conservative, because the locomotive was in a position where its full capacity could not be demonstrated, in fact was idle a large part of the time, making only 24 miles per day when it is capable of doing more than twice as much. The management estimates that a second locomotive that was to be installed would save practically 50 per cent over the present system of haulage as it was to be placed on a level and have sufficient work to keep it moving, displacing 15 mules.

By comparing the cost of haulage with the gasoline locomotive at Midvalley and the average cost of electric locomotive haulage as furnished in the Coal and Metal Miners' Pocket-book, it was found that there was a lessened cost of 27.9 per cent in favor of gasoline. The locomotive uses naphtha for fuel, it being less dangerous and better to handle than gasoline. The consumption of naphtha is about 15 gal. per day ($1.50 per day), where if gasoline were used the consumption would be about 12 gal. for the same work while the cost would be 30c. more.

The Midvalley locomotive, rated as a 9-ton locomotive, has about the following dimensions: Length, 150 in.; width, 59 in.; height, 60 in.; wheel base, 48 in.; and diameter of driving wheels, 24 in. The following table gives the detail of the average work performed daily by the first locomotive in 6 months, during which period approximately 2 hr. out of a 9-hr. day were devoted to switching, a feature which fails to show on the cost sheet:

Average tonnage of loaded cars per day	550 tons
Average tonnage of empty cars per day	250 tons
Average mileage of loaded cars per day	12 miles
Average mileage of empty cars per day	12 miles
Weight of one loaded car	5½ tons
Weight of one empty car	2½ tons
Average number of cars per train	8 cars

Cost of Operation

Wages of operator and helper per day	$3.35
Cost of fuel per day	1.50
Cost of lubricating oils per day	.12
Consumption of fuel in gallons daily	15

Maintenance

Cost of maintenance of locomotive for six months, including repairs and labor	$65.14

The substitution of gasoline-motor haulage for mules at some mines near Rockwood, Tenn., in 1911, presented some interesting figures on the comparative costs of these two methods of haulage. The coal in this mine is collected on sidetracks on the main entries by mules or by rope from cross-entries, the nearest parting being 1½ miles from the slope. Mule haulage had been used on this long haul and this had been found so expensive that it was decided to install three gasoline motors. The total output of the mine was between 600 and 700 tons a day all of which passed over this long main-haul distances varying from 1½ to 2 miles.

All the extra work in the mines, necessary for the installation of these motors, was some slight trimming of the rib and top in places, so as to give ample clearance for the motors and going over the track to replace with 20-lb rail, the places on the entry where a lighter rail had heretofore been used.

There was no difficulty found by reason of the many curves. as the motors have a 4-ft. wheel base, and can take a curve of 25-ft. radius. The locomotives are 6 tons each, and were built for the mine gage of 33 in. They are designed with 4-cylinder engines, of ample power to slip the wheels, and all parts are well protected, as is necessary for mine use.

The mine cars used are about 1400 lb. in weight, and carry $1^1/_5$ tons of coal. As the grade is in favor of the loads, the empty cars up the entry make the load for the motor. The regular 20-car trips are handled without difficulty, and on trial trips 40 cars have been taken up the entry.

These three motors replaced 23 mules. The comparative estimate of mule and motor haulage on one entry was as follows:

10 twenty car trips equals		224 tons
By mules:		
4 drivers, at $1.65	$6.60	
9 mules, at 50c	4.50	$11.10
By motor:		
1 motorman, per day	$2.05	
1 coupler, per day	1.65	
13 gal. gasoline, at 11½c	1 50	
2 lb. carbide, at 4c	.08	
½ gal. gasoline engine oil, at 23c	.12	
1 gal. transmission case oil	.24	$ 5.64
Saving by motor		$ 5.46
Or. 49.1 per cent		

These motors use 12 to 13 gal. of gasoline each, per shift. The Connellsville Central Coke Co. converted from horse to gasoline motor haulage in 1915 and comparison of the results obtained are of value. Twenty-nine horses had formerly hauled the output of 700 cars, to the haulage rope or to the shaft bottom as the case might be. The coal from some of the flats was handled independently of the haulage rope. This represented a net tonnage of only 41.5 tons per horse, the tonnage per unit being low, not only because of the excessive length of haul, but because of the heavy grades, which averaged 6.5 per cent in the butt headings. The situation evidently needed corrective treatment.

Four hundred mine cars are used in hauling the coal. These are 44-in. track gage and have a capacity of approximately 4000 lb. per car, which weigh when empty about 2000 lb. As the daily output is 1700 tons, it is about twice the capacity of the mine cars.

On all main haulageways 40-lb. rail is used, and on the flats and subsidiary butts the rails weigh 25 lb. per yard. The joints are all fishplated, and the track is well ballasted and carefully graded.

A 5-ton gasoline locomotive was selected and put in operation in September, 1914. It pulls 15-car trips in No. 5 flat right and 20-car trips in F flat on the left, the latter being a one-way haulage of approximately 2000 ft. and the other being roughly half that length. The cars are delivered to the rope at the main butt entry.

The grades on both headings are partly in favor and partly against the load. Thus in the No. 5 flat there is a grade which averages 1¾ per cent against the load extending for the whole distance between two of the butt entries, and in F flat there is a grade averaging about 1.2 per cent against the load and nearly 1200 ft. long. It is easy to see that conditions more favorable might have been chosen. The operating expenses are as follows:

Locomotive runner per 9-hour day	$2.75
Trip rider per 9-hour day	2 60
11 gal. of gasoline at 12c. per gal	1.32
¼ gal. lubricating oil at 22c. per gal	.06
Cup grease, waste, oil, etc	.05
Total	$6.78

The results obtained from this locomotive were so satisfactory that a duplicate was purchased and shipped to the mine in January, 1915. This locomotive is working in C flat on a haul which measures 2700 ft. one way and in D flat where the haul is 3000 ft. one way. Both flats have grades favorable to the load. However, in D flat there is a grade about 300 ft. long which runs about 1.1 per cent against the load. At present each of these locomotives is hauling about 250 cars per day, and handling about 500 tons. It is expected that when certain grades are made more even and when delays are eliminated 300 cars loaded with 600 tons will be handled by each unit.

For the months of April and May, 1915, the locomotive in F flat and No. 5 right averaged 260 cars daily; on this basis the comparative haulage costs per day are as follows:

Horse haulage:		
11 drivers at $2 60		$28.60
Feeding 11 horses at 50c. each		5.50
		$34.10
Motor haulage:		
6 drivers at $2 60	$15.60	
Feeding 6 horses at 50c. each	3.00	
1 motorman	2.75	
1 snapper	2.60	
Gasoline, oil, grease, etc	1.51	
		25.46
Savings per day accomplished by use of gasoline motor		$ 8.64

The locomotives when fully equipped weigh 10,500 lb. Over all they are 144 in. long, 55 in. wide and 46 in. high and their wheels are 18 in. in diameter. They are each equipped with 5 × 6-in. four-cylinder four-cycle engines of the vertical type, specially designed for mine-locomotive service and capable of delivering 25 hp. to the wheels at 800 r.p.m.

The volume of tonnage is the principal determining factor in deciding when to substitute motor for animal haulage. Care must be exercised not to make a change before the motor becomes the most economical method. It is not possible to prescribe exact limitations for animal haulage but as a general rule where the haul exceeds one-half mile or where the cost of hauling 300 tons on the main haulageway amounts to $1800 a year (these figures as of 1910) it is usually economy to install some form of motor haulage.

A comparison between gasoline motor and mule haulage was compiled at the mines of the Shade Coal Mining Co. in Pennsylvania, the results of which are given herewith. The coal is handled in 185 mine cars of 2500 lb. capacity and weighing 1000 lb. each. The outside and main entry haulage track is laid with 30 lb rail to the first gathering point in the mine,

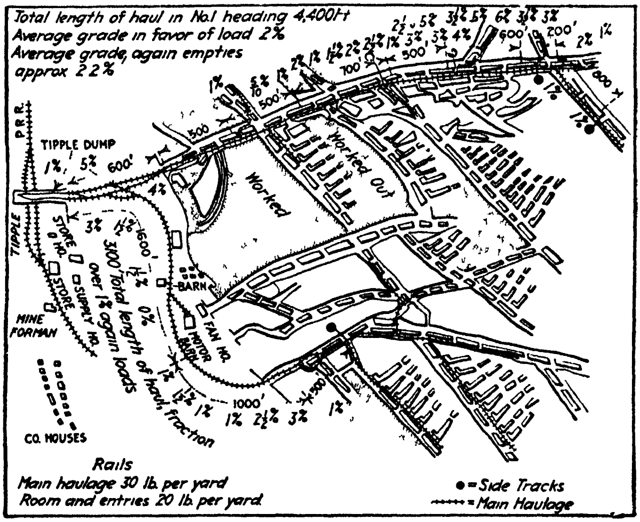


FIG 23 —Haulage layout at Shade Coal Co 's Mine in Pennsylvania

beyond which 20 lb. rail are used. A map of the haulage arrangement is shown in the accompanying drawing, Fig. 23.

The first motor was installed in October, 1911, and was a 7-ton machine, with a 4-cylinder, 4-cycle engine, having a 6 in. bore and stroke and developing 35-hp. at 800 r.p.m. The second motor was a duplicate of the first and was put in service in March, 1913.

The coal at this mine is hauled in one hundred and eighty-five 1000-lb. mine cars, having a capacity of 2500 lb., all of these cars being equipped with plain bearings. The outside and main-entry track consists of 30-lb. rail, and back of the first gathering point in the mine, the rail is 20 lb. The minimum radius of the curves is approximately 30 ft.

The first gasoline machine was put in operation, October, 1911, and was 7-ton, 36-in. gage locomotive. It has a vertical, 4-cylinder, 4-cycle engine, 6-in. bore and 6-in. stroke, which develops 35 hp. at 800 r.p.m.

Mule haulage was used before the gasoline locomotives were introduced and at that time the following conditions prevailed:

Operating Conditions

Length of haul one way, 2640 ft.
Maximum grade in favor of loads, 2½ per cent
Maximum grade against empties, 5 per cent
Tonnage per month of 24 working days, 9600 tons

Cost of Mule Haulage

10 mules to handle tonnage @ 60c. per day. Above day rate includes feed, harness and shoeing expenses	$ 6.00
Eight drivers @ $2.25 per day	18.00
Investment in 10 mules @ $200 is $2000. Assuming the average life of a mule is 5 years, this gives a 20 per cent depreciation per year, 24 working days per month, the cost for depreciation per day will be	1.38
Interest on investment at 6 per cent per annum per day	0.41
Mule haulage cost per day	$25 79
Total mule haulage cost per month of 24 working days	618.96
Mule haulage cost, per ton	0 064
Mule haulage cost per ton-mile traveled by loads	0.128

Gathering Cost Using Mules at Mine No. 1

Tonnage per month of 24 working days, 9000 tons	
3 mules for gathering @ 60c. per day	$1.80
Three drivers @ $2.50 per day	7 50
Mule depreciation	0.417
Interest on investment at 6 per cent per annum, per day	0.125
Total gathering expense per day	$9.842
Total gathering expense per month	$236.208
Gathering expense per ton	0.0262

Main Entry Gasoline Haulage at Mine No. 1

Length of haul one way, 4400 ft.
Maximum grade in favor of loads, 6 per cent
Average grade in favor of loads, 2 per cent
Maximum grade against empties, 5 per cent.
Average grade against empties, 2.2 per cent
Tonnage per month of 24 working days, 9000 tons

Motorman for 24 days @ $2.75	$66.00	
Trip rider for 24 days @ $2.75	66.00	
Total labor		$132.00
384 gal. of gasoline per month @ 15½c.	$59.52	
24 gal. engine oil per month @ 30c.	7.20	
6 gal. black oil per month @ 15c.	0.90	
6 lb. cup grease per month @ 15c.	0.90	
Waste per month	1.00	
Total supplies		69.52
Repairs:		
Material	$28.95	
Labor	6.86	
Total repairs		35.81
Depreciation on locomotive at 10 per cent per annum, per month		29.16
Interest on locomotive investment at 6 per cent per annum, per month		17.50
Total operating cost per month		$283.99
Operating cost per day	$11.83	
Operating cost per ton	0.0315	
Operating cost per ton-mile traveled by loads	0.0379	

Gathering Cost Using Mules at Mine No. 3

Tonnage per month of 24 working days, 3,500 tons

1 mule required for gathering at 60c. per day		$0.60
1 driver required per day		2.40
Mule depreciation per day		0.138
Interest on investment at 6 per cent per annum, per day		0.0416
Total gathering expense per day		$3.1796
Gathering expense per month	$76.308	
Gathering expense per ton	0.0218	

MAIN ENTRY GASOLINE HAULAGE AT MINE NO. 3

Length of haul one way, 3100 ft.		
Motorman for 24 days at $3 per day		$72.00
Trip rider for 24 days at $2.75 per day		66.00
Labor		$138.00
120 gal. gasoline per month @ 15½c	$18.60	
4 gal. engine oil per month @ 30c	1.20	
2 gal. black oil per month @ 15c	0.30	
2 lb. cup grease per month @ 15c	0.30	
2 lb. waste per month @ 15c	0.30	
Total supplies		20.70
Repairs:		
Material	$28.95	
Labor	6.86	
Total repairs		35.81
Depreciation on locomotive @ 10 per cent per annum, per month		29.16
Interest on locomotive investment @ 6 per cent per annum, per month		17.50
Total operating cost per month		$241.17
Operating cost per day	$10.04	
Operating cost per ton	0.0691	
Operating cost per ton-mile	0.1173	

Mine No. 3 is under development, and the motor, as well as the gathering mule and driver, are not working to exceed 40 per cent of the time. The gasoline consumption, and the amount of sand used, varies with the weather. The gasoline consumption runs from 4 to 7 gal. per day, bad weather causing more wheel slippage and a higher engine speed, and, thereby, increasing gasoline fuel consumption.

SUMMARY BASED ON EXPERIENCE IN MINE NO. 1

Mule haulage cost per ton	$0.064
Gasoline haulage per ton	0.0315
Saving per ton	0.0325
Mule haulage cost per ton-mile	0.128
Gasoline haulage per ton-mile	0.0379
Saving per ton-mile	0.0901

The following is a typical example of gasoline consumption on a 7-ton motor, developing 50 hp. at a speed of 500 r.p.m. On a break test this motor would consume about a pint of gasoline per horsepower-hour, equal to 50 pints or 6¼ gal. per hour. Theoretically, therefore, this motor would consume 50 gal. of gasoline per 8-hr. shift, while in actual practice the consumption varies from 15 to 18 gal. indicating that the average horsepower developed varies between the same figures.

The following are some of the results obtained with an Otto internal combustion mine locomotive, working at the Barton mines in Nottingham, England, about 1910.

The length of haul inside the mine is about 700 yd. and on the surface about 1½ miles. With the previous horse traction one round trip on the surface line occupied over two hours, including shunting at the tipple, hauling a train of 10 loaded wagons of about 20 tons total gross load. The locomotive makes one trip with the same number of wagons in three-fourths of an hour regularly, and in some cases in 35 min. The line is for the greater part level but partly in favor of the loads. The heaviest grades are 0.77 per cent against loads and 4 per cent against empties.

The gasoline consumption during 27 working days was 38 gal. which equals 1.4 gal. per day from 7 a.m., to 5 p.m. During this time 1274 net tons of stone were hauled. This shows that with one gallon of gasoline, 33½ net tons were covered; at a tare of 10 cwt. per wagon this represents a total gross load of 47.6 tons, this being over a line of 1½ miles, so that 71.4 ton-miles were covered with one gallon of gasoline. The locomotive is fitted with two speed-gears in either direction, i.e., for 3¾ and 7½ miles per hour.

This locomotive has replaced six horses which cost $14.40 per week in fodder alone, while also requiring four boys. The locomotive only requires one driver and one boy for shutting the gates, when crossing the roads. The gasoline consumption per week of about 8½ gal. at 16c. exclusive of rebate, amounts to about $1.34 per week.

The Germans and Austrians were the pioneers in the use of gasoline motors for mine use. In 1910 it was estimated that there were about 300 of these in use in various parts of the world.

Gasoline motor haulage costs were estimated in 1910 on underground haulage work where the tracks were inferior and curves sharp at 2.4 to 2.6c. per ton-mile. Actual working costs taken over a sufficiently long time to give reliable results were found on the surface track to be as low as 1.2c. per ton-mile after deducting 20 per cent for amortization.

Fuel consumption on one type of gasoline motor was found to be slightly less than 0.1 gal. per hp.-hr. when working under full load. As, however, the motor is never run at full load except when starting and running up grade, it is found that 0.05 gal per hp.-h . is the normal consumption.

Storage battery and trolley motor haulage costs.—Some excellent figures on repairs and maintenance costs of storage battery motors on metal mine work at the Bunker Hill and Sullivan Mine are given on p. 229, Vol. 51 A.I.M.M.E. It was found there that low voltage, as compared to the 500-volt d.c. used on the trolley-type locomotive, practically eliminates brush and commutator troubles, which always have been a source of heavy expense. About the only charge against the batteries is the time of one man for a few minutes each morning, giving them the daily inspection and refilling the cells with distilled water to replace that evaporated during the previous day—the amount of distilled water required for three batteries being about 20 gal. per week. In addition, there is a monthly charge, not exceeding $10 per battery, for cleaning and overhauling. The principal source of repair expense on the locomotives was for new wheels.

The figures given below are the total average monthly cost of repair and upkeep from the date of installation to November 1, 1914, and also include the cost of installation, which was quite large, because the battery boxes had to be altered and partly rebuilt, to adapt them to the company's charging system, and to protect them from the water issuing from the chutes under which they pass.

	Average Monthly Repair Cost.	Average Monthly Tonnage Hauled.	Repair Cost in Cents.
2½-ton Jeffrey, No. 11 level	$48 513	13,501	0.359
4-ton Westinghouse, No. 12 level	55.456	12,755	0 434
4-ton Gen. Electric, No. 13 level..... ..	28.612	2,406	1.189

The last figure is high because the costs are figured only on the tonnage of ore hauled, and most of the material handled by this motor was waste.

A comparison of these costs with the trolley motors which they replaced is interesting. They cover a period of two months in the first case and four months in the second, so that the figures must not be taken to represent an average cost over a long period. Separate repair costs were kept for all motors until January, 1913, which accounts for the short period taken for the above motors, when it is remembered that they were replaced by storage-battery motors in March and June respectively, in the same year. The 2½-ton Jeffrey trolley motor on the No. 11 level had an average monthly repair cost of $39.88 as compared with $93.912 for the 4½-ton General Electric working on the No. 12 level; the average monthly tonnage handled by the two machines was 9154 for the 2½-ton and 14,645 for the 4½-ton motors; and the repair costs per ton were 4.35c. and 6.41c. respectively.

These figures do not include the initial cost of upkeep and the trolley wires and track bonding, which kept two men busy practically all the time, and which was consequently a heavy expense. No separate costs were kept for the two levels, however, so they may be omitted in this connection. It has been estimated by the company's electrical engineers that, with a few minor improvements in the charging system, and a better understanding of, and more careful attention given to the operation of these motors by the motormen, the cost of repairs and operation would be 75 per cent less than the trolley motors doing the same work.

Repair records on a 4-ton Westinghouse storage-battery motor at the Big Five Tunnel in Colorado, covering a 6-months period, showed a cost of $12.60 per month, or 0.8c. per ton-mile. The motor was doing very light work during three months of this time so that the repair costs may be relatively high; for two months during which time it was working under a more nearly full load, the repair cost per ton-mile was 0.65c.

One of the chief objections advanced to the storage battery motor is the cost of the batteries and their comparatively short life which ranges from two to four years. It is doubtful, however, if this extra charge against the motor will exceed the cost of copper wire, bonding and twin cables and cost of upkeep for the trolley type motor. It must also be remembered that the storage battery motor uses less than half the power required to operate the cable and reel motor which it usually replaces on secondary haulage.

Constant fluctuations in output in different sections of the mine makes it difficult to operate trips on any regular timetable as in the case of railroads. This is due largely to the fact that there are so many elements in the movement of the cars, a delay or accident in any one of which would derange the whole schedule. The system here described was in use in a mine of the Sterling Coal Co. in Ohio.

The coal in this mine is handled over one main entry off which there are 11 butt entries having from 12 to 15 working rooms each. The motive power consists of one, 8-ton trolley motor on the main haul, two 3½-, one 4-, and one 4½-ton motors for the butt entry hauls and eleven, 2½-ton storage battery motors for gathering. The 8-ton motor on the main haul handles 36-car trips taking all the loads from, and placing all the empties at a sidetrack along the main haul.

The butt-entry locomotives take the empties from the sidetrack in trains of 12 cars and deliver them to one of the 2½-ton storage-battery locomotives. Each butt-entry locomotive takes care of three butt entries.

When the butt-entry locomotive brings in a trip of 12 empties, this trip together with the 2½-ton storage-battery locomotive is pushed into a room. In the meantime the butt-entry locomotive makes into a train the 12 loaded cars which have been placed on the entry by the storage-battery locomotive. After the loaded trip is coupled up it is pulled down to the room where the empty trip and storage-battery locomotive are waiting and coupled onto them. The empty trip is then pulled with its battery locomotive out on the entry, after which the empty trip is uncoupled, the loaded trip taken to the parting by the butt-entry locomotive, and the storage-battery locomotive then proceeds to distribute the empty cars and push them to the face of the rooms.

The reason for pushing the empty trip into a room and pulling it out again with the trolley locomotive is to save the storage battery from handling the trip of empties while the trolley locomotive is on the entry. The twelve loads together with two other similar trips taken from two other butt entries are taken to the siding to make up a trip of 36 loads for the main-haulage locomotive.

With this system of haulage, a schedule is maintained approximately as follows: One entry produces 12 cars per hour. This means that the 2½-ton storage battery locomotive in each hour places 12 empties at the face and takes 12 loads away to the entry leaving them just outside the room neck. When these loads have been placed on the entry, the butt-entry trolley locomotive comes along, leaves 12 empties and picks up the trip of 12 loads, taking it to the sidetrack.

This trolley locomotive comes into each butt entry once every hour, and inasmuch as it has three entries to take care of, there are 20 min. available for taking care of each entry. The main-haulage locomotive, handling 36 cars per trip, must make four round trips per hour, or a round trip every 15 min.

The average weight of coal loaded into each car is 2700 lb. and at a rate of production of 12 cars per hour per butt entry for 11 butt entries, the capacity of this haulage system is approximately 178 tons per hour. On this basis, the haulage capacity per day of 8 hr., is 1424 tons.

Prior to the installation of these locomotives the cars were handled in rooms by pushers, three being required on each butt entry. Now the motorman on the locomotive is the only man required to handle these cars. It is estimated that the saving thus effected will amount to approximately $15,000 per year.

A comparison between this three-element and a two-element haulage system which might be used in its place, may be of interest. The capacity of each storage-battery locomotive in an 8-hr. day under the present system is 96 cars. If these locomotives were eliminated and the butt-entry locomotives provided with cable reels, to enable them to go up into the rooms to place the empties and pull the loads, a cable-reel locomotive would be required on each entry. The cost of a storage-battery locomotive, such as is used here, and that of a cable-reel locomotive, such as would be required, are practically the same.

With the two-element system, then, only 11 locomotives would be required as compared with 15 under the present system, saving the installation of four locomotives representing an investment of approximately $7200. Assuming depreciation and interest at 25 per cent, this investment costs approximately $1800 per year. On the other hand, with cable-reel locomotives, two men would be required on each locomotive, or a total of 22 men for 11 locomotives.

With the present system, only one man is required on each storage-battery locomotive and two men on each butt-entry locomotive, making a total of 19 men. At $60 per month per man, the present system thus effects a saving of $2160 per year, which more than offsets the interest and depreciation on the added investment. To this should be added the freedom from cable trouble and expense, freedom from the expense of more carefully laid and bonded track in rooms and the possibility of operating on a smaller generator equipment. Thus the economic advantages of this three-element system become evident. No doubt there are many operations where this plan, which has proven its advantages and economy at this mine, can be applied with equal success.

While the storage-battery locomotive is of course not absolutely safe in the presence of gas, the sparks it makes are not near the roof, so the danger is lessened, for the gas must be in large quantity if it is to settle so low as to be ignited.

Another increase of safety with the battery locomotive results from its shortened electric circuit. The current does not pass from the power house to the motor and back through the rails to the generator, but the circuit is contained within the locomotive—not even the wheels are included in its range.

Thus if it is true that there is a risk of stray currents prematurely igniting shots, the current of the storage-battery locomotive can meet the accusation with a perfect alibi. And, of course, as there is no conducting wire, there can be no short circuits to ignite gas, coal dust or wooden structures.

The travel of the electric current along the drawbars and couplings of a train of mine cars has always been an objection to the trolley locomotive, as it has occasionally shocked men and ignited powder. For this reason operators have sought to improve the grounding of the traction rigging and to make powder cars relatively nonconducting throughout. In other cases the powder car has been hauled around by a mule.

The battery locomotive, however, having its current self-contained, does not offer any such risk. It seems that it would serve admirably for hauling men and transporting material, explosive and otherwise, into and out of the mines. By switching off the electric current on the main haulage road, the loading of men on the man trip and their passage along the road at the beginning and end of the day would be robbed of its dangers, some of which, though unnecessary, are unavoidable so long as careless and ignorant men are employed.

The use of a section insulator at the point of embarkation or disembarking has been occasionally adopted and has probably saved many lives.

In mines where all or part of the night load is quite light, it is customary to delay pumping till the mines are closed down. If undercutting is done at night and the coal is loaded and hauled out by day, there is a third shift during which the storage-battery locomotives can be charged. This tends to keep the load curve even and to save in expense. Additional boosting of the locomotive batteries can be performed during the lunch hours and when shifts are being changed, should the batteries need it and should the men walk to their work.

It has been generally thought that the storage-battery repairs would be excessive, but offsetting this there are no trolley harps, wheels and supports to be maintained in condition. The chance of the armature bearings and poles heating or rubbing is about the same in both battery and trolley locomotives. The battery renewals might, however, cost so much that the cost of harps, wheels and supports would appear only a trifling drawback to electric locomotives.

Compressed-air and electric haulage costs.—The general superiority of electricity for underground haulage has been too well established by its widespread popularity during the last two decades to admit of any serious controversy as to the relative economy of it and the compressed-air haulage methods. Certainly in any event, the economic possibilities of the air-motor are limited to specific and unusual conditions such in gaseous mines where there would be danger in the use of the electric motor and even here the motor would find its greatest application for secondary haulage or gathering purposes. So long as it is still used, however, a few examples of its costs of installation and operation will be of value.

The accompanying table excerpted from Vol. 34, A.I.M.M.E., gives the comparative cost of electric and compressed-air haulage as worked up by the H. K. Porter Co., covering results with the fourth and fifth motors built by that concern. They represent the results of operations at the Glen Lyon plant in 1898. The table gives the cost per ton in preference to the ton-mile basis, because the delays at terminals forms so large a part of the time lost that it remains a fixed quantity, regardless of the length of haul so that the best showing on a ton-mile basis is only obtained on long hauls.

	Compressed Air	Electricity	
		Estimated	Actual
Number of working days during year	160	200	141¼
Output per day in tons	*2362½	989	989
Cost per day:			
Engineer, powerhouse	$1.16	$1.20	$2.84
Motorman	4 20	4 23	9.31
Helpers (brakeman)	3.20	3.20	3.61
Electrician	.....	1.67	3.68
Repairs to motors	0 74	5.95	8.42
Repairs to line	.....	.....	0.46
Repairs to generator	0.57	.....	0.61
Fireman	.....	.....	2.50
Depreciation	†4.74	‡5.20	‡8.17
Interest	4.73	.....	4.41
Interest, repairs and depreciation, 174 hp. boiler	1.63		
Oil and waste for motor	0.25	0.22	0.35
Oil and waste for generator	0.47	.....	0.74
Steam (fuel and firing)	2.32		
Totals	$24.01	$21.67	$45.10
Cost per ton	0.01015	0 02192	0.0456

* Tons of coal hauled.
† At 5 per cent.
‡ At 3 per cent.

In this table interest on the compressed-air motors is calculated at 5 per cent and at 3 per cent on the electric motors. The column headed "Actual" gives the results accomplished at the electric-haulage plant of the No. 2 Mine of the Hillside Coal & Iron Co.; under the column "Estimated" those results are given as re-calculated on the assumption of 200 working days in the year instead of 141.25.

About 1900 one of the larger bituminous coal companies installed a compressed-air haulage system consisting of:

1 compound-steam three-stage air compressor, 800 lb. pressure.

5300 ft. 5-in., triple strength, pipe-line.

3600 ft. 2 in., triple strength, pipe-line.

Three 14-ton motors.

The cost of this installation, exclusive of boiler plant was approximately $37,000. The time consumed for each trip was 45 min. There were two charging stations and the compressor was intended for the operation of all three motors but under actual operating conditions there was very little margin when one motor was in operation. The cost of maintenance of the high-pressure plant and the motors was $8 and $7.50, respectively, per day.

An electric-haulage system was later installed at this mine, consisting of two, 150-kw. generators, directly connected to two 22 × 20 in. simple engines and six, 13-ton electric mine motors, feeder and trolley lines, etc., the entire cost of which was about $42,000, exclusive of the boiler plant. One of these motors performs all the work of the above described compressed-air plant, making the round trip in 30 min. and in addition is used on other entries for two or three hours per day. That part of the cost of the electric plant properly chargeable, for the purpose of comparison, against the old compressed-air haulage plant would be about $7000. Also a few months before the compressed-air plant was abandoned, one of the motors was overhauled at an expense of $2000.

SECTION IV

TIMBERING COSTS

The enormous quantity of timber, poles, lagging, mine ties, plank and lumber in general used in mine operations, makes this subject of great importance, and upon the intelligent handling of this material in the future will depend in a great measure the cost per ton output of coal.

Although wood has been in universal use since creation, there is a remarkable lack of knowledge as to its structure and behavior in its various uses, by those who might be expected to know its properties, thereby using species totally unfit for certain purposes, and consequently expensive to the company. In the past, when timber was plentiful and cheap, it mattered little in many cases how long it lasted, as the service it gave was ample return on the cost; smaller quantities being consumed then, few thought it necessary to study the structure and behavior of wood in order to lengthen its service.

How to buy timber.—The cost of wood has increased enormously in the past decade, while the quality is steadily decreasing. The greater care in its selection and use is therefore self-evident, in order to lengthen its life by elimination of infected, or poor quality. The same care should be used in the selection of the proper species for the various uses.

It is necessary, therefore, to use improved methods in selecting prop timber, mine plank and various other wood, first, by determining the species meeting the most important requirements, or several qualities in combination as shown by actual experience, and tests. Second, the methods of procuring the supply. Third, the handling and preparation, and finally the placing in the mines.

Conceding that the present prop material in healthy condition (such species as Southern short leaf, loblolly pine and spruce pine) is probably as good as can be secured at a reasonable first cost, the next question is, when, how and where to purchase same. Timber cut between September and March is preferable to timber cut during the spring and summer months. Timber that is cut during hot weather is subjected to attack through a period when all insect life, such as worms, wood borers, beetles and other low plant life (fungus parasites) are most active in their work of wood destruction, and the timber during this period is in its most favorable condition, with fresh green sap, inviting attack of fungi spores and borers of all kinds, thereby causing the first signs of decay. Experience has shown that summer-cut timber does not give the satisfaction, nor will it give as long service as winter-cut stock.

Black oak and red oak are approximately of equal value with short-leaf, loblolly and spruce pine, all of these being an easy prey of fungi, when in contact with soil, while white oak is of greater strength and durability. A white-oak plank, one inch full thickness, is equal to an inch and a half pine plank of equal width at an approximate decrease of one-third in cost per surface foot.

Because of the use of heavy electric motors instead of mule haulage in mines, it is essential that improved provision be made to take care of mine tracks, by the purchase of mine ties, manufactured to a rigid specification, from live winter-cut white oak, chestnut oak and young chestnut properly seasoned before using, thereby reducing purchases and track repairs.

Sound pine trees cut down in the winter season and cut into log lengths, stripped of their bark and piled in layers with sticks between each layer so that a free circulation of air can pass through the pile, will harden the exterior juices. This will form a coating which will, to a great extent, furnish protection from exterior checking and materially resist the attack of the fungus, spores and wood-destroying insects, which protection it does not have if cut during the summer season.

Green and unpeeled pine timber placed in mines for gangway use is sure to give short service and minimum strength. Consequently such timber is the most expensive for the service it renders. In such condition crushing is most liable and decay sets in quickly.

Tests should be made to determine the most efficient species for each particular use, bearing in mind cost and service and specifications covering such needs should be prepared and purchases and inspection made accordingly.

Timber used and costs for United States.—The following statistics on the timber used in the mines of the United States in 1905 are based upon data gathered by the Forest Service in cooperation with the United States Geological Survey. Nearly 14,000 mines were selected in which the use of timber seemed certain or possible, and from more than 5000 of these, reports were received showing that timber had been used.

It will be noted that 2940 bituminous coal mines used nearly $6,400,000 worth of timber, while 216 anthracite mines used over $4,400,000 worth, the average cost of timber per mine being $2170 for the bituminous and $20,524 for the anthracite mines.

It will also be noted that the timber used in the 216 anthracite mines was of slightly greater value than that used in 1718 mines for precious metals. The much higher cost of the timbering required for the anthracite mines is due to several causes. In the first place, many of the anthracite workings lie at great depths, and some of the larger properties have many miles of gangways which have to be carefully maintained. They are below water level, and, as a result of the combined action of air and mine water, the timbers decay rapidly. Some of the beds are of enormous thickness, and require vast quantities of timber in the construction of "square sets" to support the roof and preserve the workings in overlying coal. Moreover, since the hills in the immediate vicinity of the anthracite mines have been largely denuded of timber suitable for mine supports, operators are obliged to obtain their supplies from considerable distances.

Pennsylvania, with 524 mines, used 37,826,000 cu. ft. of round timber and 55,716,000 board feet of sawed timber, costing altogether $2,290,053. The average cost of the round timber was 3.5c. per cubic foot and that of the sawed timber $17.39 per thousand board feet. In Illinois 400 mines used 10,342,300 cubic feet of round timber, costing 6c. per cubic foot, and 7,025,000 board feet of sawed timber, costing $22.04 per thousand board feet, the total cost being $778,186. The 325 mines in West Virginia used 6,716,000 cu. ft. of round timber and 19,645,000 board feet of sawed timber. The total cost was $561,061; that of the round timber being 4.6c. per cubic foot, and that of the sawed timber $12.76 per thousand board feet. Next in order of total outlay for timber is Ohio, with $471,730; Iowa with $232,148; Indiana with $220,209; and Alabama with $216,221. None of the other states used over $200,000 worth of timber.

Timber is more expensive in Colorado than in any other state. The average cost of the round timber in that state was 11.6c. per cubic foot and that of the sawed timber $33.76 per thousand board feet. The large amount of mining has made a heavy demand for timber, and, although most of the round timber is obtained locally, much of the sawed timber must be shipped in from considerable distances at high freight rates. Round timber in Wyoming and New Mexico cost 10.4 and 10.5c. per cubic foot, respectively, or nearly as much as in Colorado; but the fact that sawed timber was obtained locally kept its price down to $16.93 per thousand in Wyoming and $12.18 in New Mexico. The lowest average price reported for round timber was 3.3c. per cubic foot in Indiana, and for sawed timber $5.58 per thousand in Washington. It must be borne in mind, however, that in many cases the timber used was cut from land belonging to the mine operators, and the cost includes only cutting and hauling.

Reports were received from 216 collieries in the anthracite regions producing approximately 83 per cent of the total anthracite tonnage of the United States. Figures for the remaining 17 per cent were computed, using as a basis the reports actually received, assuming that conditions and requirements were uniform throughout the state. The results of the tabulation show that 121,565,000 ft. board measure of sawed timber (equivalent to 10,130,000 cu. ft.) and 52,440,000 cu. ft. of round timber were used during 1905.

The total value of the sawed timber was $1,842,000, or $15 per thousand feet board measure. The total value of the round timber was nearly double that of the sawed timber, being $3,468,000, or $6.60 per 100 solid cubic feet—the approximate equivalent of the average standard cord of 128 cu. ft. The total value of the round and sawed timber combined was $5,310,000, or about 8½c. per long ton of coal, using as a basis for the calculation the production in 1905—in round numbers 61,000,000 long tons.

So far as reported, the kinds of wood have been tabulated separately, but in many cases the operators were unable to furnish information in regard to the quantity of each species used, and it has therefore been necessary to classify a large amount as "mixed" or "miscellaneous."

Round Timber		Sawed Timber	
Kind	Cubic Feet	Kind	Board Feet
Yellow pine	9,250,000	Hemlock	63,600,000
Oak	6,220,000	Yellow pine	14,200,000
Hemlock	1,180,000	Oak	2,860,000
Pitch pine	590,000	Maple	1,740,000
Chestnut	444,000	Spruce	371,000
Beech	236,000	White pine	328,000
Jack pine	165,000	Pitch pine	84,000
Spruce	115,000	Mixed hardwoods	28,642,000
Mixed hardwoods	10,263,000	Mixed softwoods	1,370,000
Mixed softwoods	477,000	Miscellaneous	8,370,000
Miscellaneous	23,500,000		
Total	52,440,000	Total	121,565,000

Of the species used for round timber, yellow pine, of which a large amount is loblolly pine from the South, furnishes one-half. Oak ranks next, but furnishes a much smaller proportion, according to the reports. The proportion of oak would unquestionably be increased if the large items reported as "mixed hardwoods" and "miscellaneous" could be separated into species, and it is not improbable that oak would then displace yellow pine in rank.

For sawed timber hemlock holds first place in quantity, while yellow pine ranks next. The amount of oak reported is doubtless too small, but an explanation is found in the classification for "mixed hardwoods" and "miscellaneous," which contains over 37,000,000 ft. board measure, of which probably a large amount is oak.

Computing size of timber.—There seems to be no definite rules by which the size of mine-gangway timbers may be calculated. Experience largely governs their use and, as both experience and judgment differ widely, in many cases the timbers are either too small or too large for the work required of them. The compressive stress of timbers—wood or steel—is often neglected, resulting in economic waste.

Owing to the adjustment of stresses underground, it is impossible to calculate the loading of the timbers with mathematical accuracy; however, rules may be formulated that will give approximate results. The experienced miner knows the size of collar best adapted for certain conditions in the gangway, so that by taking an average of the strength of collars used in several different gangways we arrive at a value that may be used in calculating other gangway timbers.

Where the strata are horizontal or the dip is light the loads are applied normal to the collar, and produce therein bending stresses, while compressive stresses result in the legs, each leg taking one-half of the total load on the collar.

Where the dip of the strata is great, however, legs and collars may be subject to both bending and compressive stresses at the same time. The compressive stresses in the collars may safely be ignored, as they are taken care of in the average bending load that a collar will support, but the legs must be calculated for both bending and compression. This bending load per foot of length will be assumed as being equal to that applied to the collar.

When the diameter and length of an existing collar are known the safe load per foot of length that it will support may be calculated by the formula:

$$w = 0.065 f \frac{d^3}{l^2}, \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (1)$$

in which

w = safe load per foot of length of span (lb.);
d = least diameter of collar (in.);
l = length of clear span (ft.);
f = safe unit fiber stress (1200 lb. per sq. in. of section, for long-leaf yellow pine and white oak, and 900 lb. per sq. in., for short-leaf yellow pine).

In calculating legs for compressive stresses only, the formula

$$f_c = \frac{L\,(700+15c+c^2)}{a(700+15c)}, \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (2)$$

should be used, in which

L = total load on leg (lb.);

a = area of cross-section of leg (sq. in.);

c = length of leg, in inches, divided by its least diameter in inches.

The diameter of the leg must be assumed for trial, the most economical section being that in which the safe unit fiber stress is not exceeded.

The following practical examples will show the correct method of using these formulas:

Example 1.—What size legs would be required for a long-leaf yellow-pine timber set in a gangway 12 ft. wide, with clear head-room of 8 ft.; the strata being horizontal? Assume 2000 lb. per lineal foot for load on collar.

Solution.—$2000 \times 12 = 24{,}000$ lb., total load on collar, and $24{,}000 \div 2 = 12{,}000$ lb., total load on each leg.

Assuming the least diameter of the leg to be $4\frac{3}{8}$ in., the area of section is $0.7854\,(4\frac{3}{8})^2 = 15.03$ sq. in.; and $c = 8 \times 12 \div 4\frac{3}{8} = 21.94$, which substituted in Formula 2 gives for the compressive stress in the leg,

$$f_c = \frac{12{,}000\,(700+15\times 21.94+21.94^2)}{15.03\,(700+15\times 21.94)} = 1172 \textit{ lb. per sq. in.}$$

Since 1172 lb. is less than the unit stress for long-leaf yellow pine (1200 lb.), a $4\frac{3}{8}$-in. stick will support the load; however, in this case, it would be advisable to use a larger stick, say 8-in. diameter, to allow for defects in the wood and unforeseen bending stresses.

Example 2.—Design a long-leaf yellow-pine timber set for a gangway 12 ft. wide, with a clear headroom of 8 ft., the strata having a dip of 45 deg. It has been found that 2000 lb. per lineal foot of collar is the average load in districts where the dip of the strata is very heavy; so we will assume this value in solving the problem.

Solution.—Reversing Formula 1, so as to give the value of d

and substituting the values given for the load per lineal foot of collar, unit fiber stress and length of span, we have

$$d=\sqrt[3]{\frac{wl^2}{0.065\,f}}=\sqrt[3]{\frac{2000\times12^2}{0.065\times1200}}=15.45,\ \textit{say}\ 15\tfrac{1}{2}\ \textit{in.}$$

The load producing compression in each leg is $24{,}000\div2=12{,}000$ lb. Assuming as before the same bending load for the leg, as for the collar $2000\times8=16{,}000$ lb. bending load on each leg. Then, assuming $12\tfrac{1}{2}$ in. as the least diameter of leg and solving for the allowable unit stress due to compression, we have by Formula 2, since $c=8\times12\div12\tfrac{1}{2}=7.68$.

$$\frac{L}{a}=\frac{f\,(700+15c)}{700+15c+c^2}=\frac{1200\,(700+15\times7.68)}{700+15\times7.68+7.68^2}=1118.4\ \text{lb.}\ \textit{per sq. in.}$$

The unit stress due to bending, for a total load of 16,000 lb., as found above, is calculated by solving Formula 1 for f; thus,

$$f=\frac{wl^2}{0.065\,d^3}=\frac{2000\times8^2}{0.065\times12.5^3}=1008.2\ \textit{lb. per sq. in.}$$

The unit stress due to compression is

$$f_c=\frac{L}{0.7854\,d^2}=\frac{12{,}000}{0.7854\times12.5^2}=97.8\ \textit{lb.}$$

The total stress in the leg due to bending and compression is therefore

$$1008.2+97.8=1106\ \textit{lb. per sq. in.}$$

As the unit stress produced by the loads is thus shown to be nearly equal to but less than the allowable unit stress, a $12\tfrac{1}{2}$-in. stick is the most economical section to use.

When calculating beams of special cross-section, as steel I-beams, it is customary to employ an expression called the "section modulus" of the beam. The section modulus (S) of a beam is a value that multiplied by the unit fiber stress (f) of the material gives the bending moment (M) of which such beam is capable of supporting, as expressed by the formula:

$$M=fS. \qquad \text{(3)}$$

But, for a beam uniformly loaded and supported at each end, the bending moment (M), in inch-pounds, is

$$M=\frac{12wl^2}{8}=1.5\,wl^2. \qquad \text{(4)}$$

Combining equations 3 and 4 gives for the section modulus

$$S=1.5\frac{wl^2}{f}. \qquad \ldots\ldots\ldots \qquad (5)$$

Example 3.—Design a steel timber set for a gangway 12 ft. wide, with a clear headroom of 8 ft., the strata having a heavy dip, assuming a fiber stress for steel, $f=16{,}000$ lb. per sq. in.

Solution.—The average load per lineal foot on the collar being assumed, as before, $w=2000$ lb. and the span being $l=12$ ft., the required section modulus, in this case, is

$$S=1.5\frac{wl^2}{f}=\frac{1.5\times2000\times12^2}{16{,}000}=27.$$

Any one of the following sections, taken from steel manufacturers' handbooks, will be satisfactory: 8-in.-32.5-lb. Bethlehem girder beam, section modulus, 28.6; 8-in.-32-lb. Bethlehem H-column, section modulus 26.9; 10-in.-30-lb. standard I-beam, section modulus, 26.8. A 10-in.-30-lb. I-beam is the most economical section, but an H-beam, or girder beam, is to be preferred as it has a broader bearing surface.

Assuming the leg to be a 10-in.-25-lb. I-beam, which has an area $a=7.37$ sq. in.; radius of gyration, $r=0.97$; and section modulus, $S=24.4$; using Gordon's formula for medium steel, fixed-end column and solving for ultimate strength (P), we have

$$P=\frac{50{,}000}{1+\frac{(12\,L)^2}{36{,}000\,r^2}}=\frac{50{,}000}{1+\frac{(12\times8)^2}{36{,}000\times0.97^2}}=39{,}308\ lb.$$

With a safety factor of 4, the safe unit stress is $\frac{39{,}308}{4}=9827$ lb.

Again, assuming the same unit bending load for the leg as for the collar, $w=2000$ lb. per lineal foot, and the length of the leg being $l=8$ ft. the bending moment, applying Formula 4, is

$$M=1.5\,wl^2=1.5\times2000\times8^2=192{,}000\ in.\text{-}lb.$$

This makes the unit fiber stress due to bending

$$f=\frac{M}{S}=\frac{192{,}000}{24.4}=7869\ lb.\ per\ sq.\ in.$$

The unit stress due to compression, for a 12-ft. span, the sectional area of the leg being $a=7.37$ sq. in. is

$$f_c=\frac{L}{a}=\frac{2000\times12}{2\times7.37}=1628\ lb.\ per\ sq.\ in.,$$

which makes the total unit stress due to bending and compression

$$7869+1628=9497 \text{ lb. per sq. in.}$$

This actual stress in the leg is less than the safe stress, therefore a 10-in.-25-lb. I-beam will satisfy the conditions assumed in this case.

The accompanying table sums up the results of a series of tests made by the U. S. Forest Service, to determine the effect of knots of different classifications on the crushing strength of certain varieties of timber. It will be noticed that in some cases the presence of knots seems actually to increase the strength.

RATIO OF RESULTS OF STRENGTH TESTS ON KNOTTY TIMBER TO RESULTS ON CLEAR TIMBER, STRENGTH OF CLEAR TIMBER TAKEN AS UNITY

	Compressive Strength at Elastic Limit per Square Inch	Crushing Strength at Maximum Load per Square Inch	Modulus of Elasticity per Square Inch
Douglas fir:			
Pin knots	0 95	0.94	1.06
Standard knots	0 87	0 86	0.90
Large knots	0 78	0.78	0.71
Western larch:			
Pin knots	1 12	1 04	1.19
Standard knots	0 98	0 89	1.00
Large knots	0 98	0 85	
Western hemlock:			
Pin knots	0.96	0.97	1.00
Standard knots	0.94	0.91	0.97
Large knots	0.86	0.83	0.81

Pin knots are defined as sound knots ½ in. or less in diameter. Standard knots are defined as sound knots ranging from ½ to 1½ in. in diameter. Large knots are also sound knots from 1½ in. in diameter, up.

The accompanying table shows what different sizes of steel rails will support when uniformly loaded and from these loads, the sizes of equivalent standard I-beams and the different

Table Comparing Strength of Steel and Wood for Supporting Mine Roofs

Ten-foot Span

Uniform Load in Lb.	Required Std. T-rail	Standard I-beam In.	Lb.	White Oak Sawed	White Oak Round	Chestnut Sawed	Chestnut Round	White Pine Sawed	White Pine Round
1,015	16	3	16 5	5×3	5	6×4	5	6×4	5
1,385	20	3	16 5	5×4	5½	6×5	6½	6×5	6½
1,920	25	3	19 5	6×4	6½	7×5	7½	7×5	7½
2,450	30	4	22 5	6×5	7½	7×7	8	7×7	8
3,090	35	4	22 5	7×5	7½	8×7	8½	8×7	8½
3,840	40	5	29 3	7×6	8	8×8	9	8×8	9
4,475	45	5	29 3	7×7	8½	10×6	9½	10×6	9½
5,225	50	5	36 8	8×6	9	10×7	10	10×7	10
6,290	55	6	36 8	9×6	9½	10×8	10½	10×8	10½
7,140	60	6	36 8	9×7	10	10×10	11	10×10	11
7,890	65	6	44 5	10×6	10	12×7	11½	12×7	11½
8,955	70	7	45 0	10×7	10½	12×8	12	12×8	12
9,700	75	7	45 0	9×9	11	12×9	12½	12×9	12½
10,765	80	7	45 0	10×8	11	12×10	13	12×10	13
11,940	85	7	52 5	10×9	11½	12×11	13	12×11	13
13,110	90	8	54 0	10×10	12	12×12	13½	12×12	13½
14,180	95	8	54 0	12×8	12	12×13	14	12×13	14
15,670	100	8	60 8	12×9	12½	12×14	14½	12×14	14½

Twelve-foot Span

Uniform Load in Lb.	Required Std. T-rail	Standard I-beam In.	Lb.	White Oak Sawed	White Oak Round	Chestnut Sawed	Chestnut Round	White Pine Sawed	White Pine Round
845	16	3	16 5	5×3	5	6×4	6	6×4	6
1,155	20	3	16 5	5×5	5½	6×5	6½	6×5	6½
1,600	25	4	22 5	6×4	6½	7×5	7½	7×5	7½
2,045	30	4	22.5	6×5	7	7×7	8	7×7	8
2,580	35	5	29 3	7×5	7½	8×6	8½	8×6	8½
3,200	40	5	29 3	7×6	8	8×8	9	8×8	9
3,735	45	6	36.8	7×7	8½	10×6	9½	10×6	9½
4,355	50	6	36.8	8×6	9	10×7	10	10×7	10
5,245	55	6	36 8	8×8	9½	10×9	11	10×9	11
5,955	60	7	45 0	9×7	10	11×8	11	11×8	11
6,580	65	7	45.0	10×6	10	10×10	11½	10×10	11½
7,470	70	7	45.0	10×7	10½	11×10	12	11×10	12
8,090	75	7	52.5	10×8	11	12×9	12½	12×9	12½
8,980	80	8	54.0	11×7	11	12×10	13	12×10	13
9,955	85	8	54.0	10×9	11½	12×11	13½	12×11	13½
10,935	90	8	60.8	10×10	12	12×12	14	12×12	14
11,825	95	9	63.0	12×8	12½	13×11	14	13×11	14
13,070	100	9	63.0	11×10	12½	13×12	14½	13×12	14½

Note.—Loads given in table are the safe uniform loads that T-rails will carry: Othe members show sizes necessary for these loads. Timber presumed as seasoned. For green timber use ¾ loads. Factor of safety 6 (about). Fiber stress white oak, 1200 lb.; white pine and chestnut, 800 lb. Timber to be placed narrow side against roof.

wood beams have been calculated. Thus, taking a 20-lb. mine rail it is seen that the safe uniform load it will carry on a 10-ft. span is 1385 lb., then following this line to the right, under the same conditions it is seen that the 3-in. 16.5-lb. I-beam will do the same and save 3.5 lb. per yard, or about 14 lb. to a beam of 12-ft. length. Under white oak is found a 5×4-in. sawed or $5\frac{1}{2}$-in. round timber for this load, while chestnut and white pine require a 6×5 in. sawed or $6\frac{1}{2}$-in. round timber.

Quite frequently a requisition will call for a certain sized timber, say, an 8-in. round timber, to be 14 ft. long, to be used for a 12-ft. span. It is seen by the table that a 4-in. 7.5-lb. I-beam will carry the same load. The importance of the place and the length of time it is to be maintained should govern which should be used. Timber in a mine, if it carries approximately the safe load, will seldom last much longer than 18 months, and the replacement plus the first installation will be more than the cost of placing the steel.

This table is presented with the idea that it will be used by mine officials in the proper selection of material when replacing or retimbering, and to give them some idea what their selection of sizes will carry in the weight of roof supported.

The Scranton Mine Cave Commission conducted a series of interesting experiments to determine the comparative strength of various materials used for supporting mine roofs, a summary of the results of which are given in the accompanying table.

Timber framing equipment.—Where a large amount of timber framing is necessary, say sufficient to require the services of four to five framers continuously, the introduction of machinery to replace these men should be considered. This will be something to eliminate the use of the two-man crosscut saw, hewing, gaging and framing the ends as required.

A regular timber framing machine for this work will cost from \$2500 to \$3000 (in 1914) and weigh about 9000 lb. It will require about a 50-hp. engine and boiler to drive it and will then only cut the framing gages at the ends. The total cost in place will be about \$7000 allowing for a suitable building to house it.

A good and economical substitute for this is a slabber and swinging cut-off saw which can be purchased complete for

$420 (figures as of 1914). This equipment can be erected as shown in the accompanying illustration, Fig. 1, in which it will be noted that the cut-off saw is set horizontal instead of vertical and another saw introduced, though these do not both operate at the same time. This particular outfit is driven by an old 7 × 10-in. engine, connected up as shown.

Description of Test	Maximum Load		Maximum Settlement	
	Total Pounds	Pounds per Square Foot	Inches	Per Cent
Rectangular pillar of mine rock. .	489,150	42,000	5.26	29
Timber crib filled with mine rock. . .	900,000	63,300	7.08	30
Circular pillar of mine rock.	361,600	85,000	4.51	31
Pile of broken sandstone, small sizes	581,000	209,000*	4.36	46
Pile of broken sandstone, large and small sizes.	417,000	150,000*	4.61	41
Pile of river sand.	600,000	216,000*	5.00	63
Pile of small broken sandstone and sand.	800,000	228,000*	4.69	45
Wet culm in cylinder.	200,000	886,000	2.73	30
Broken sandstone in cylinder.	300,000	1,330,000	3.66	35
Broken sandstone and sand in cylinder.	300,000	1,330,000	2.42	23
Cinders in cylinder.	300,000	1,330,000	5 33	·51
Wet culm in cylinder.	300,000	1,330,000	3 00	33
River sand in cylinder.	300,000	1,330,000	3 35	32
Pillar of mine gob.	600,000	. . . . . .	2 43	27

* Pressure under 20×20 in. bearing plate

The posts are first sawed on the slabber, and then squared on the ends by running through the machine, say one hundred of them, and the saw is then raised up so as to cut just 2 in. deep. The horizontal saw remains stationary, the stick is shoved through and the cut made on top; the carriage is then shoved on the horizontal saw and a slice taken out of the bottom; it is then pulled back and rolled over one quarter and the operation repeated; it requires four cuts to finish the end.

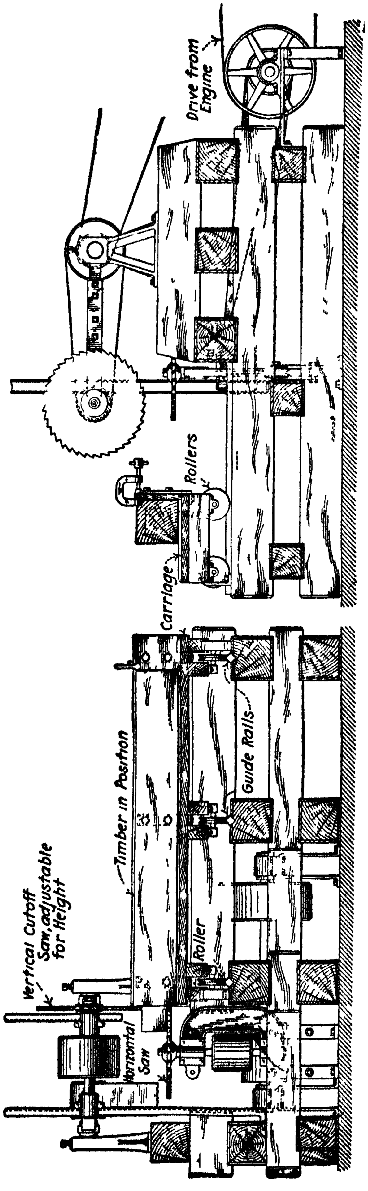


FIG 1 —A homemade timber framer, showing bed, carriage and jaws

The complete operation averages $3\frac{1}{2}$ min. to frame both ends of an 8×8-in. post. The slabbing saw takes logs up to 8 ft. in length and the company made a set of dogs and some perforated plates to hold pins, and bought an inserted tooth saw to replace the thin saw which came with it.

The machine can also be used for sawing straight lumber of any dimensions up to 8 ft. in length, this particular installation reducing lumber costs at the mine about one-half. A small wedge saw was added for cutting scraps into wedges which were made at a cost of $\frac{3}{8}$c. each as compared with a cost of 6c. when made by hand. Some of the other advantages of the machine are that the slabs made can be generally used for lagging in the less critical places and the machine framing has been found more accurate and giving better fits in the mine.

Timber preservatives.—In 1906 the United States Forest Service, in cooperation with the Philadelphia & Reading Coal & Iron Co. carried on a series of experiments to determine the best method of prolonging the life of mine timber. It was found as a result of these studies that 45 per cent of the mine timber is destroyed by decay, while breakage, wear and insects accounted for the balance. Germs or spores which produce decay may gain access to the timber at any time before or after it is cut, though for the most part the disease is contracted in the mines from decaying timber near by. In untreated timber, rough surfaces of bark and wood furnish a foothold for the spores, which subsequently germinate and attack the wood tissues. Spores may also enter timber only superficially treated through checks, cracks, or nail wounds.

For a fungus to exist it must have a definite amount of air and water, food, and heat. If mining conditions were such that the timber would be kept always wet or always dry, it would never decay. It is the alternating wet and dry conditions or continuous dampness which produce rot. Ventilation is a very large factor in the life of mine timber. Poorly ventilated gangways and air passages, with a fair degree of moisture and a fairly high temperature, are favorable to fungous growth, and hence to rapid decay. It is probably impossible to exterminate disease, sufficiently to wholly prevent decay in mine timber. Right preservative treatment, together with careful handling of the timber will, however, reduce both to a minimum.

The important part which insects play in the destruction of mine timber is seldom realized. They are for the most part brought into the mines with the timber. Regular and thorough inspection and the rigid condemnation of insect-infested timber would therefore greatly reduce the loss from this source.

Insects bore into the sound wood and greatly weaken it and, moreover, leave holes which encourage the entrance of wood-destroying fungi. A good preservative treatment will protect the timber from insect attack, as well as prevent decay. If the bark is removed from the timber soon after it is cut, it will not be attacked by wood-destroying insects until the wood becomes old and dry, after which it may be attacked by "powder post" and other borers.

Sets of round gangway timber averaging 13 in. in diameter were chosen as the basis for the experimental treating work. These sets in the anthracite regions consist of two legs, usually 9 to 10 ft. long and a collar 6 to 7 ft. long, and they are usually placed on 5-ft. centers. The sets contain 26 cu. ft. of timber and the average life in the anthracite mines is two years.

Experiments have shown that peeled timber is superior in durability to unpeeled timber. The space between the bark and the wood especially favors the development of wood-destroying fungi, and is a breeding place for many forms of insect life. When, after placement in the mines, the bark begins to flake off, the timber has already begun to decay. The cost of peeling timber before it goes into the mine ranges from 20c. to 50c. per ton of wood (figures as of 1905), according to local conditions and the kind of timber.

Seasoning or drying gives mining timber greater strength and durability. A stick of wet timber has only one-half the strength it has when thoroughly dry. Though it is not practicable for mining companies to hold their timber until it is absolutely air dry, peeled timber will dry out sufficiently in a few months to gain in both strength and durability. From two to four months is necessary for proper seasoning.

To determine the possible loss in weight in round timber, due to peeling and seasoning, a test was conducted at one of the collieries of the company. Representative sticks of Southern loblolly pine, averaging 11 to 13 in. in diameter and from 9 to 10 ft. in length were chosen. This timber was weighed immediately before and after peeling, to determine the weight of the bark. It was then weighed every two weeks until seasoned, to learn the weight of the water evaporated. The time of the year greatly favored rapidly seasoning. The short lengths into which the timber was sawed gave a large drying surface in proportion to volume and longer sticks would season more slowly. The accompanying diagram, Fig. 2, shows the

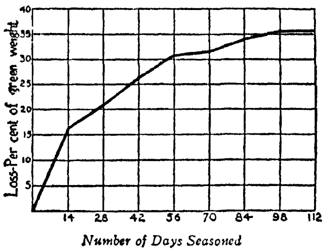


FIG. 2 —Percentage of loss of green weight in seasoning.

average percentage of loss from the green weight in seasoning, a synopsis of the results of the tests being as follows:

PEELING AND SEASONING TEST, SOUTHERN LOBLOLLY PINE ROUND TIMBER, APRIL 17 TO JULY 24, 1906

	Per Cent
Total loss of green weight by peeling	8 1
Total loss of green weight by seasoning..................	35 1
Peeling and seasoning	43.2

RATE OF SEASONING

Number of Days Seasoned	Percentage of Green Weight Lost	Number of Days Seasoned	Percentage of Green Weight Lost
14	16 2	70	31 4
28	20 5	84	33.7
42	26 2	98	35.1
56	30.3		

If a mining company handles its own timber from the woods to the mines, the saving in freight made possible by peeling and seasoning can readily be estimated. Labor is the principal factor in the cost of peeling, while the cost of seasoning must be represented by the loss of interest on the capital invested in the timber during the seasoning period. However, these additional items of expense are more than offset by a maximum reduction in freight of from 30 to 40 per cent and by the far better condition of the timber with regard to both its life at the mines and the readiness with which it will take preservative treatment. The peeling of timber at the mines has been unsatisfactory and expensive, because of the limited amount of yard room and the accumulation of bark. The following considerations favor peeling in the woods: (1) The saving in the cost of freight due to peeling and seasoning; (2) the saving of yard room at the mines; and (3) the prevention of fungous disease and insect attack by early peeling.

Peeling and seasoning mine timber unquestionably increase its durability. However, in order to prolong its life to the fullest extent, a preservative treatment is necessary. The increased life necessary to justify the cost of applying a preservative by the several methods in vogue is indicated by the accompanying diagram, Fig. 3.

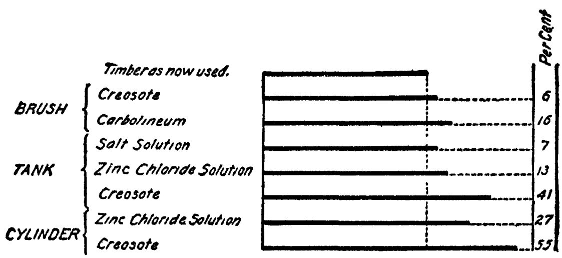


FIG. 3.—Increased life necessary to pay cost of preservation treatment.

Impregnated wood resists decay because the preservative is antiseptic and excludes the moisture necessary for fungous growth. Timber used in mines was treated with a variety of preservatives under several methods of application. Both green and seasoned timbers were treated to determine both the relative value of the treatments and the best method of handling preparatory to treatment. If treated at all the timber must be peeled. The accompanying table shows the method of treatment, the preservative applied, the cost of same and the cost of the treatment for an average gangway set and per cubic foot:

Method of Treatment	Preservative Applied	Cost of Preservative	Cost of Treatment		Absorption per Cubic Foot,
			Per Set of Gangway Timber (25.8 Cu. Ft.)	Per Cubic Foot	
		Per gal.			Pounds
Brush	Creosote (dead oil of coal tar)	$0.09	$0 40	$0.015	
	Carbolineum	.70	1.15	.045	
Open tank without pressure	Salt solution, magnesium, chloride 15 per cent	.01	.50	.020	
	Zinc chloride solution, 6 per cent	.04	.90	.035	
	Creosote	.09	2.85	.110	10
Cylinder with pressure	Zinc chloride solution, 6 per cent	.04	1.90	.075	
	Creosote	.09	3.85	.15	10

Brush treatments with both creosote and carbolineum were applied in two coats to the Pennsylvania and Southern pines. A large flat brush and kettle of the hot preservative are all that is required for this treatment. A very small amount of the preserving fluid suffices, but the cost of application in proportion to the results obtained is considerable. For small individual operators who cannot afford the cost of a large plant, brush treatments are feasible and economical.

The disadvantages of brush treatments are:

(1) The difficulty of completely covering the timber and filling all checks and cracks.

(2) The very slight penetration secured. The subsequent checking or opening of the timber may often allow disease to pass through the shallow exterior band into the untreated interior wood.

Pitch pine and loblolly pine have been most successfully treated with both creosote (dead oil of coal tar) and a 6-per cent solution of zinc chloride by the open-tank process.

The open-tank treatment as given in this experiment was briefly as follows: Green, partially seasoned, and thoroughly seasoned timber was lowered into the tank and immersed in creosote, or in a zinc chloride or salt solution, at a temperature of from 90 deg. to 120 deg. F. The temperature of the creosote was raised by the coils to from 212 deg. to 220 deg. F., and that of the zinc chloride or the salt solution to about 212 deg. F. In no case, however, was the temperature allowed to go above 240 deg. F. for fear of injuring the fiber of the timber and so decreasing its strength. When this hot bath was over the steam was turned off, and the timber was allowed to stand until the liquid cooled to a temperature of from 170 deg. to 100 deg. F. The periods of heat and of cooling were varied for each kind of timber and for each stage of its seasoning. The time required for the cooling operation, which depended largely upon the temperature of the atmosphere, was usually from 3 to 12 hr. For the whole treatment the time varied from 6 to 20 hr.

Loblolly and pitch pine can be successfully and economically treated by simple immersion in successive hot and cold baths in an open tank, at a cost of about 11c. per cubic foot. Green timber is treated with far more difficulty than seasoned timber.

AVERAGE AND REPRESENTATIVE TREATMENTS OF LOBLOLLY AND PITCH PINE BY THE OPEN-TANK PROCESS

CREOSOTE

AVERAGE ABSORPTION AND PENETRATION, LOBLOLLY PINE (PINUS TÆDA)

Condition of Timber	Absorption per Cubic Foot, Pounds	Depth of Penetration, Inches
Green	5–7	¾–1½
Seasoned (1 to 2 months)	12–15	2–4
Seasoned (3 to 4 months)	20–25	5–complete

Representative Individual Runs, Seasoned Loblolly Pine (Nearly all Sap Wood)

Time Seasoned, Months	Total Length of Treatment, Hours	Duration of Hot Bath, Hours	Temperature: Average, ° F	Temperature: Maximum, ° F	Absorption per Cubic Foot, Pounds	Penetration, Inches
3	24	7	230	240	22.0	4–5
3	24	4½	225	235	21.5	4–5
3	6½	1¾	178	220	10.7	2–3
3	6	2	173	210	10.7	2–3
3	4¼	1½	174	198	10.2	2–3

Representative Individual Runs, Seasoned Pitch Pine (Heart Wood and Sap Wood)

Time Seasoned, Months	Total Length of Treatment, Hours	Duration of Hot Bath, Hours	Temperature: Average, ° F.	Temperature: Maximum, ° F.	Absorption, Pounds per Cubic Foot	Penetration, Inches	Width of Sap Wood, Inches
4	22	7½	215	240	5½	¾	¾
4	22	7½	218	240	12	1¾	1½
4	22	7½	209	232	21½	3	2¾

Solution of Zinc Chloride (6–8 Per Cent)

Thoroughly Seasoned Loblolly Pine (Nearly all Sap Wood)

Total Length of Treatment, Hours	Length of Period in Hot Solution, Hours	Temperature: Average, ° F.	Temperature: Maximum, ° F.	Absorption, Pounds per Cubic Foot	Penetration, Inches
20	6	200	210	20	3–5
20	6	200	210	35	4–6

The difference in weight of green timber before and after treatment is by no means indicative of the amount of the preservative absorbed. The simple application of the hot liquid to green timber slightly reduces its weight and yields no penetration. The same application to seasoned timber slightly increases its weight and gives a slight penetration. Green timber after treatment may show a penetration of 1 inch without an increase in weight.

Heart wood of both loblolly pine and pitch pipe is penetrated with far more difficulty than is the sap wood of the same species. This is especially the case with pitch pine which clearly shows after treatment a distinct division between the treated sap wood and the untreated heart wood.

Experiments indicate that for pine timbers of the same degree of dryness, or containing equal proportions of heart wood and sap wood, impregnation can be regulated by increasing or decreasing the duration of the cooling bath.

During the year 1906–7 gangway timber of various species, peeled and unpeeled green and seasoned, and treated and untreated was placed in gangways in the collieries of the company. Each and every kind and condition of gangway timber has been compared with the timber in most general use in the southern anthracite region, namely, green, unpeeled loblolly, and pitch pines. The object of this comparison is to prove exactly to what extent the experimental timber is superior to that at present used. In the course of the experimental work the following comparisons have been made:

Species Compared	Treatments Compared	
Loblolly pine (*Pinus Tæda*)	Brush	Creosote
Pitch pine (*Pinus rigida*)		Carbolineum
Longleaf pine (*Pinus palustris*)	Open tank	Creosote
Chestnut (*Castanea dentata*)		Solution of zinc chloride
Red oak (*Quercus rubra*)		Solution of sodium chloride and magnesium chloride
	Cylinder	Creosote
		Solution of zinc chloride

The history of each set of gangway timber and each part of each set has been recorded in writing and on maps. These records include: (1) The date of setting; (2) the colliery;

SUMMARY OF EXPERIMENTAL SETS OF TIMBERS

	Colliery			Total
	Silver Creek	Eagle Hill	Wadesville	
Untreated.				
Seasoned peeled	11 loblolly	26 loblolly		37
Green peeled....	44 loblolly 16 longleaf	16 loblolly 8 longleaf	14 pitch pine	98
Green unpeeled....	112 loblolly 31 pitch pine 8 black oak 1 white oak	36 loblolly 7 pitch pine	26 pitch pine	221
Total untreated.........				356
Brush treatment:				
Green—				
Carbolineum..........	14 loblolly	9 loblolly	4 pitch pine	27
Creosote	18 loblolly 5 chestnut	9 loblolly	6 pitch pine	38
Seasoned—				
Carbolineum...........	7 loblolly	9 loblolly	6 pitch pine	22
Creosote	9 loblolly	28 loblolly	5 loblolly	42
Total brush treatment..				129
Tank treatment:				
Green—creosote..........	104 loblolly 7 chestnut 5 pitch pine 2 black oak	6 loblolly		124
Seasoned—				
Creosote...............	20 loblolly 11 pitch pine			31
Salt....................			17 pitch pine 14 loblolly	31
Zinc chloride...........	6 loblolly 5 longleaf			11
Total tank treatment...				97
Cylinder treatment, seasoned:				
Creosote............	23 loblolly			23
Zinc chloride...........	50 loblolly			50
Total cylinder treatment..				73
Grand total...... ...				755

(3) the gangway; (4) the position in the gangway relative to the nearest chute and adjacent set of timber.

The accompanying table gives a summary of the experimental sets of timber placed in the mines of the Philadelphia & Reading Coal and Iron Co. in 1906.

PRESERVATIVE TREATMENT APPLIED

Method of Application	Preservative Used	Approximate Cost of Preservative	Approximate Cost per Set of Gangway Timber of 26 Cu Ft.	Cost per Cubic Foot
The preservative heated to 180° F and applied in two coats with a brush	Creosote (dead oil of coal tar)......	$0.09 per gal.	$0 40	$0 01½
	Avernarius carbolineum	0.70 per gal.	1.15	0 04½
Immersion in an open tank without pressure—successive baths of hot and cold fluid Plant of simple construction	Solution of common salt (15 per cent)	$0.009 per lb.	$0.50	$0.02
	Solution of zinc chloride (6 per cent)..........	0.04½ per lb.	0.90	0.03½
	Creosote (dead oil of coal tar)....	0.09 per gal.	2 85	0.11
In a closed cylinder under vacuum and pressure Plant of complex construction	Solution of zinc chloride (6 per cent)......	$0.04½ per lb.	$1.90	$0.07
	Creosote (dead oil of coal tar) . . .	0.09 per gal.	3 85	0 15

Steel timbering.—The following are the comparative costs of steel and frame timbering as found under actual working conditions:

In 1908 at their Maxwell colliery the Lehigh & Wilkes-Barre Coal Co. timbered a double-track gangway with 20 in. 65-lb. I-beam collars 17 ft. long between supports, and 8 in. H-beam legs 10 ft. 6 in. high in the clear, weighing, with base plates, 1720 lb. per set. These took the place of wooden sets made of 24 in. round yellow pine timbers, the cost of which erected was $15 per set, weight 5040 lb. and the life of which was two and one-half years. In view of their probable durability, the steel sets were erected on concrete bases and this added to the cost of installation, which reached a total of $40 per set.

Capitalized at 6 per cent interest, the value of the steel sets at the end of 15 yr. will be $95.86 each, while the capitalized value of the six wooden sets needed in that time will be

\$153.56. At the end of the 15 yr., the steel will have a scrap value per set of \$12.03, while the wood will be worth nothing, a saving by the use of steel of \$69.73 per set or \$4.65 per year.

The pump house at the Dodson colliery, of the Plymouth Coal Co, is 100 ft. long, 8 ft. high in the clear and 18 to 22 ft. wide. The roof is exceedingly tender and has caused all kinds of trouble in the pump house, especially in connection with the pipes Before retimbering with steel, 18 to 22 in. round timbers were used, on 2-ft. centers, practically skin to skin. It is estimated that the pump room was retimbered once a year. Beginning with April, 1910, the 70 sets of wood timbers were replaced by 48 sets of steel. The last steel set was placed December 15, 1910. According to figures furnished by John C. Haddock, president of the company, the relative costs were:

1. Wood—70 sets; weight per set 4150 lb.; cost per set f.o.b. cars at mine, \$12; cost, erected in place, \$34.50; total cost, \$2415; life, one year.

2. Steel—48 sets; weight per set 1483 lb.; cost per set f.o.b. cars at mine, \$31.47; cost erected in place, including concrete footings, \$61.47 per set; total cost \$2889.09, or not quite 20 per cent more than wood.

Based on its life, the cost per month of a wood set without interest was \$201.25. The cost of the steel sets at the end of 16 months without interest was \$180.57 per month. At the end of that time they had shown no signs of failure.

At the No. 8 mine of the West Kentucky Coal Co., steel timbers are used in a slope, both for the main entry and air course. The sets are composed of a 10 in. 25-lb. I-beam collar and 4 in. H-beam legs. They are spaced 3 ft. centers and lagged with oak plank 3 in. thick on top, and 2 in. thick on the sides. Between the sets, concrete is placed up to a height of 4 ft. This makes a solid reinforced concrete slope from the entrance to the point where the ribs are hard and top good. According to figures furnished by W. H. Cunningham, general manager of the company, the comparative costs of wood and steel for his mine were (about 1912):

Wood—Yellow pine creosoted; size 12 $\times$ 12 in., 264 ft. b.m.; cost at Sturgis, \$10.56 per set; cost in place, \$15.70; weight 1575 lb.

Wood—Native white oak; size 12×12 in., 264 ft. b.m.; cost at Sturgis, \$7.92; cost in place, \$13.06 per set; weight 1340 lb.

Steel—Cost of steel at Sturgis, \$9.75 per set; cost of placing \$1; cost of concrete per panel \$5.16; total cost in place per set, steel alone \$10.75, steel concreted \$15.91; weight of steel sets 425 lb.

Saving in the use of steel without concrete, over native white oak, \$2.31 per set, over yellow pine \$4.95. Excess cost of steel with concrete, over white oak \$2.85 per set, over yellow pine 21c. This favorable comparison is due to the high unit cost of the wood and to the elimination of waste. The safe uniformly distributed load on the wood collar is 1200 lb., on the steel collar 26,000 lb. The safe compressive strength of the steel leg is 43,200 lb., while that of the wooden leg is 105,100 lb.; in the one case more than ample, in the other case out of all proportion.

In some cases it has even been found, where transportation costs are not excessive, that the first cost of the steel timbering erected in place is equal to or but slightly more than the cost of a wooden installation of similar strength. It has also been found that it is possible to fit the steel exactly to the structural necessities, whereas it is impracticable in all cases to do so with wood. This circumstance eliminates an economic waste due to the use, for practical considerations, of larger sized sticks of wood than are really necessary.

The accompanying table was prepared by a mining company in western Illinois and compares the relative cost of steel beam collars, square-sawed white oak beams and square-sawed yellow-pine beams delivered underground. It also estimates the comparative maintenance costs for a period of 20 yr., based on the most favorable and the least favorable probable conditions. The safe working loads are based on the 1915 values for the materials under bending stress.

The smaller table gives the first cost of such beams and their cost at the end of the 20-yr. period under the conditions assumed. It, therefore, represents what might be called a reasonable expectation of relative service in a particular district where the conditions as to transportation, cost of steel and the relative availability of wood are normal.

Relative Cost of Steel and Sawed Oak and Pine Beams Delivered Underground and Estimated Expense of Maintenance for 20 Years

STEEL BEAMS

Span in Clear, Ft.	Length of Beam, Ft.	Depth in Inches and Shape of Section	Weight in Lb. per Ft.	Weight of Beam, Lb.	Size of Bearing Plates, In.	Weight of Bearing Plates, Lb.	Total Weight of Steel, Lb.	Safe Uniform Load, Lb.	Cost Delivered Underground @ $2 per Cwt.	Estimated Maintenance for 20 Years: Favorable Conditions	Estimated Maintenance for 20 Years: Unfavorable Conditions
16	18	10—I	25.0	450	5×½×12	17.0	467.0	16,280	$9.34	Paint 5 times @ $1.00 = $5 00	Paint 20 times @ $1.25 = $25
12	14	8—I	18.0	252	4×½×12	13.6	265.6	12,640	5.31	Paint 5 times @ .90 = 4 50	Paint 20 times @ 1.15 = 23
8	10	6—I	12.25	122.5	4×½×12	13.6	136.1	9,680	2.72	Paint 5 times @ .80 = 4.00	Paint 20 times @ 1.05 = 21
14	16	6—H	23.8	380.8	6×½×12	20.4	401.2	11,460	8.02	Paint 5 times @ .90 = 4.50	Paint 20 times @ 1.15 = 23
12	14	6—H	23.8	333.2	6×½×12	20.4	353.6	13,370	7.07	Paint 5 times @ .80 = 4 00	Paint 20 times @ 1.05 = 21
14	16	4—H	13.6	217.6	4×½×12	13.6	231.2	4,070	4.62	Paint 5 times @ .85 = 4.25	Paint 20 times @ 1.10 = 22
12	14	4—H	13.6	190.4	4×½×12	13.6	204.0	4,750	4.08	Paint 5 times @ .75 = 3.75	Paint 20 times @ 1.00 = 20

SQUARE SAWED WHITE OAK BEAMS

Span in Clear, Ft.	Length of Beam, Ft.	Size of Beam, In.	Weight of Beam, Lb.	Feet, Board Measure	Safe Uniform Load, Lb.	Cost Delivered Underground @ $40 per M.	Estimated Maintenance for 20 Years: Favorable Conditions		Estimated Maintenance for 20 Years: Unfavorable Conditions	
16 12	18 14	11×14 9×12	885 483	231 126	16,470 13,200	$9.24 5.04	Renew 2 times $18.48 Labor $5.00 10 00	$28.48	Renew 6 times $55 44 Labor $5.00 30 00	$85.44
							Renew 2 times 10.08 Labor 4.50 9.00	19.08	Renew 6 times 30 24 Labor 4.50 27.00	57.24
8	10	6×10	192	50	9,170	2.00	Renew 2 times 4.00 Labor 4.00 8.00	12.00	Renew 6 times 12.00 Labor 4.00 24 00	36.00
14	16	9×12	552	144	11,310	5.76	Renew 2 times 11.52 Labor 4.50 9.00	20.52	Renew 6 times 34 56 Labor 4 50 27 00	61.56
12	14	9×12	483	126	13,200	5.04	Renew 2 times 10.08 Labor 4.50 9.00	19.08	Renew 6 times 30 24 Labor 5.50 27 00	57.24
14	16	8× 8	327	85	4,470	3.40	Renew 2 times 6.80 Labor 4.00 8 00	14.80	Renew 6 times 20 40 Labor 4.00 24 00	44.40
12	14	8× 8	286	75	5,210	3.00	Renew 2 times 6.00 Labor 4 00 8.00	14.00	Renew 6 times 18.00 Labor 4.00 24.00	42.00

SQUARE SAWED BEAMS. LONGLEAF YELLOW PINE

Span in Clear, Ft.	Length of Beam, Ft.	Size of Beam, In.	Weight of Beam, Lb.	Feet, Board Measure	Safe Uniform Load, Lb.	Cost Delivered Underground @ $40 per M.	Estimated Maintenance for 20 Years: Favorable Conditions			Unfavorable Conditions		
16	18	9×14	693	189	15,920	$.756	Renew 2 times Labor $5.00	$15.12 10.00	$25.12	Renew 6 times Labor $5.00	$45.36 30.00	$75.36
12	14	10×10	428	117	12,040	4.68	Renew 2 times Labor 4.50	9.36 9.00	18.36	Renew 6 times Labor 4.50	28.08 27.00	55.08
8	10	8× 8	195	53	9,240	2.21	Renew 2 times Labor 4.00	4.24 8.00	12.24	Renew 6 times Labor 4.00	12.72 24.00	36.72
14	16	8×12	469	128	11,890	5.12	Renew 2 times Labor 4.50	10.24 9.00	19.24	Renew 6 times Labor 4.50	30.72 27.00	57.72
12	14	8×12	411	112	13,860	4.48	Renew 2 times Labor 4.50	8.96 9.00	17.96	Renew 6 times Labor 4.50	26.88 27.00	53.88
14	16	6× 8	235	64	3,960	2.56	Renew 2 times Labor 4.00	5.12 8.00	13.12	Renew 6 times Labor 4.00	15.36 24.00	39.36
12	14	6× 8	205	56	4,620	2.24	Renew 2 times Labor 4.00	4.48 8.00	12.48	Renew 6 times Labor 4.00	13.44 24.00	37.44

Figures based on bending stresses of 1100 lb. per sq. in. for white oak and 1300 lb. per sq. in. for longleaf yellow pine and 16,000 lb. per sq. in. for steel. Loads to be uniformly distributed over length of span.

FIRST COST AND COST AT END OF 20 YEARS OF STEEL AND SAWED OAK AND PINE BEAMS

Span in Clear, Ft.	Steel			White Oak			Longleaf Yellow Pine		
	First Cost	20-Year Cost: Favorable Conditions	20-Year Cost: Unfavorable Conditions	First Cost	20-Year Cost: Favorable Conditions	20-Year Cost: Unfavorable Conditions	First Cost	20-Year Cost: Favorable Conditions	20-Year Cost: Unfavorable Conditions
16	$9.34	$14.34	$34.34	$9.24	$37.72	$94.68	$7.56	$32.68	$82.92
12	5.31	9.81	28.31	5.04	24.12	62.28	4.68	23.04	59.76
8	2.72	6.72	23.72	2.00	14.00	38.00	2.12	14.36	38.84
14	8.02	12.52	31.02	5.76	26.28	67.32	5.12	24.36	62.84
12	7.07	11.07	28.07	5.04	24.12	62.28	4.48	22.44	58.36
14	4.62	8.87	26.62	3.40	18.20	47.80	2.56	15.68	41.92
12	4.08	7.83	24.08	3.00	17.00	45.00	2.24	14.72	39.68

Examination of this table will indicate that in the case of the 16-ft. spans there is but little difference in the first cost between steel and white oak, and that at the end of 20 yr. under any circumstance steel is the most economical.

This economical advantage of steel as compared with wood would be somewhat enhanced if consideration were had to the matter of interest on the investment, which however, the mining company which prepared the table did not consider. It can also be said that repainting of steel once in 4 yr. is about the limit of probability. There are installations that have been in place two and three times as long as that without repainting, and it must not be forgotten that, except in special locations, conditions underground as regards paint are much better than they are above ground.

Steel beams for roof supports may be used just as they come, cut to length, from the rolling mill; all the other types of mine timbering need to be fabricated, that is, framed and fitted, before they are ready for use in the mines. A most important consideration is to reduce the cost of this fabrication to the lowest possible amount by simplification of the details. Blacksmith work always should be avoided; and that form of framing is the cheapest in which the fitting shop work is the simplest. Intelligent skill in structural design here means great saving.

Eight different types of framing for steel gangway supports have come into serious consideration. With plain material at 1.25c. per pound f.o.b. cars Pittsburgh, and the usual extras for workmanship, the comparative costs of these styles are shown in the accompanying tables and from which can be seen at a glance how great a part attention to details may play in the economic use of materials and the avoidance of unnecessary work in fabrication. The eight sets of each table are all of equivalent theoretical strength.

The figures in these two tables do not include painting. The cost for this varies considerably with the kind of paint used but may be estimated at $2 per ton (1912).

Concrete timber.—Many modern mines are using steel for supporting gangways, and satisfaction has attended these examples. Even where loads are not great and the fire hazard low, the use of metal in place of wood has been found a profit-

Steel Gangway Supports for a Double Track Gangway

Collar 17 ft. long between legs; legs 10 ft. 6 in. high in the clear; equivalent in strength to 24 in. round longleaf yellow pine timbers.

Style	Size of Collar	Size of Pegs	Weight per Set		Cost per Set	
			Without Base, Lb.	With Base, Lb.	Without Base, Dollars per Set	With Base Plates, Dollars per Set
B	20″ 65-lb. beam	2–7″ C.—14.75 lb.	1930	2030	36.82	39.48
D	20″ 65-lb. beam	2–7″ C.—14.75 lb.	1930	2100	36.82	43.95
C	20″ 65-lb. beam	1–8″ H.—34.6 lb.	1710	1800	25.39	27.80
A	20″ 65-lb. beam	2–7″ C.—14.75 lb.	1930	2500	36.82	58.50
F	20″ 65-lb. beam	1–8″ H.—34.6 lb.	1690	1730	25.81	26.88
I	20″ 65-lb. beam	1–8″ H.—34.6 lb.	1730	1780	25.22	27.12
E	20″ 65-lb. beam	2–7″ C.—14.75 lb.	1670	1770	25.94	28.60
G	20″ 65-lb. beam	1–8″ H.—34.6 lb.	1690	1730	25.81	26.88

Steel Gangway Supports for a Single Track Gangway

Collar 10 ft. long between legs; legs 8 ft. high in the clear; equivalent in strength to 15 in. round longleaf yellow pine timbers.

Style	Size of Collar	Size of Legs	Weight per Set		Cost per Set	
			Without Base, Lb.	With Base, Lb.	Without Base Dollars, per Set	With Base Plates, Dollars per Set
A	10″ 25-lb. beam	2–6″ C.—10.5 lb.	765	945	14.81	23.17
D	10″ 25-lb. beam	2–6″ C.—10.5 lb.	765	800	14.81	15.85
B	10″ 25-lb. beam	2–6″ C.—10.5 lb.	765	810	14.81	16.12
E	10″ 25-lb. beam	2–6″ C.—10.5 lb.	605	660	9.39	10.87
F	10″ 25-lb. beam	1–5″ H.—18.7 lb.	569	590	8.84	9.43
G	10″ 25-lb. beam	1–5″ H.—18.7 lb.	566	587	8.80	9.39
C	10″ 25-lb. beam	1–5″ H.—18.7 lb.	565	605	7.91	9.04
I	10″ 25-lb. beam	1–5″ H.—18.7 lb.	600	360	9.80	11.11

able investment on the basis of the ultimate economy in the expenditure, which results from low maintenance charges and infrequent renewals.

Inasmuch as steel, however, is considerably more expensive than reinforced concrete and as the latter possesses all the advantages of the former, it would appear that reinforced concrete would be well adapted to mine timbering. Of course it might be contended that on account of its low tensile strength and relatively high resistance to compression and crushing, that concrete would not be as suitable as steel, which has a high tensile strength, and is therefore fitted to carry large loads over wide spans with a minimum ratio of dead weight to external loading.

In most mines, however, the gangways are so driven that it is unnecessary to cover wide spans. If it were found necessary to meet such conditions, a more complicated system might have to be considered, and a few props could be so placed as to reduce the span while leaving the passages as unobstructed as possible.

Proper design for reinforced concrete used in mine timbering should be based on correct principles of engineering. Props could be planned according to the principles of columns used in the erection of buildings, as shown in Fig. 4, consisting of rods embedded in the concrete near the periphery. These are connected by means of ties, or flats, hoop iron or wire. Thus the radius of gyration is increased and the rods take care of the tensile stresses which occur from eccentric loading or from deflection of columns. The horizontal ties prevent the buckling of the rods and increase the strength of the concrete. They form a hooped column.

The size and dimensions of these props or columns must be determined by the nature of the roofs of the mine they have to support and on the height of the slope or entry wherein they will be used. Above ground it is possible to calculate the loading and the stresses involved with mathematical correctness, but underground there are no well defined rules by which for example the strength of square timber sets or props may be calculated other than those given in this chapter.

Regarding the construction of these reinforced-concrete members, inasmuch as suitable sand and gravel can be found in almost any locality, it would only be necessary to ship to the mine the steel and cement. The props and collars could be made in the quantities desired on the surface in close proximity to the mine.

The method of procedure followed would be along the lines of that pursued in the manufacture of concrete telegraph poles, and the form work would be similar in construction. The lumber used for forms should be of either 1-, 1½- or 2-in. boards, securely braced by 3 × 3- or 4 × 4-in. braces, and bolted in order to prevent bulging. The greatest economy is secured

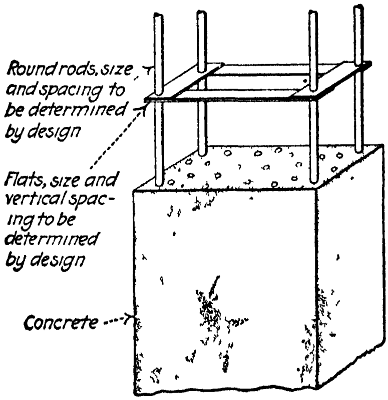


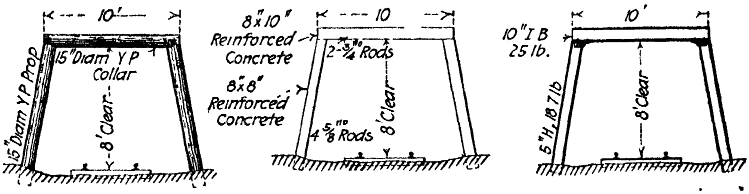
FIG 4.—A prop designed like a building column

by constructing the forms so that they can be used over and over again. Economy may also be gained by fastening the form work together with the minimum amount of nailing.

The relative proportions of strength of reinforced concrete, steel and timber are based on the ordinary method of calculations, as shown in Figs. 5 to 7, and the use of their equivalents produces much stronger and stiffer mine sets than the comparison would seem to indicate. Stiffness is as important as strength, and the spacing should be such as to compel the different sets to act together as a unit under any sudden stress or shock. Light designs with a close spacing will therefore be preferable to heavy ones with wide spacing, the roof itself serving as a beam to distribute the load over two or more sets, whereas on wide spacing there is more danger of the roof falling in between sets. The closer spacing also permits of much lighter lagging.

Conditions in the mine as to sizes of timbers used, the nearness to a supply and the cost of lumber vary widely, and an infinite number of comparisons as to the relative costs of reinforced concrete, steel and wood could be instituted, results of which might not be of exact value. Under market conditions as of 1914 the cost of steel in Pennsylvania is nearly three times the cost of wood used in the square timber sets, and the cost of reinforced concrete may be taken at about one and one-third times the cost of wood. In the first cost of the installation therefore the advantage will lie with wood.

Where, however, a gangway has to be maintained over a number of years and the workings are in any way permanent,



Timber Set | Reinforced Concrete Set | Steel Set

FIG 5. | FIG 6. | FIG 7.

Comparative size of timber, steel and concrete sets of equal strength.

consideration should be given to the capitalized value of the material as compared with the first cost of the installation; and reinforced concrete will be found economical in most cases on the basis of ultimate cost by reason of its long life and endurance.

Taking as a basis of estimate the price of structural steel at $1.405 per 100 lb. f.o.b. cars Pittsburgh, with the usual extras for workmanship, etc., a comparison of the relative costs of the form of gangway supports shown in Figs. 5 to 7, is as follows:

The cost of the timber sets, as shown in Fig. 5, was $7.50, erected. The cost of the steel sets, as shown in Fig. 7, would be $22, erected. If painted, the cost would be about $2 per ton additional. The cost of the reinforced-concrete set would be $10, erected.

Reckoning 6 per cent compound interest, on the low assumption of 15 yr. life, the steel set at the end of this period will represent an investment of $52.72, and the reinforced concrete set $23.97. During the same length of time, based on past experience and on the present cost of timber, six wooden sets would be required at a capitalized value of about $100.

On this basis, the saving on the steel timbers can be set down as $47.28 per set, which means a saving per year of $3.15; on the reinforced concrete, the saving per set is $76.03, which amounts to a yearly saving of $5.07 over lumber, and over steel of $1.92 per year per set.

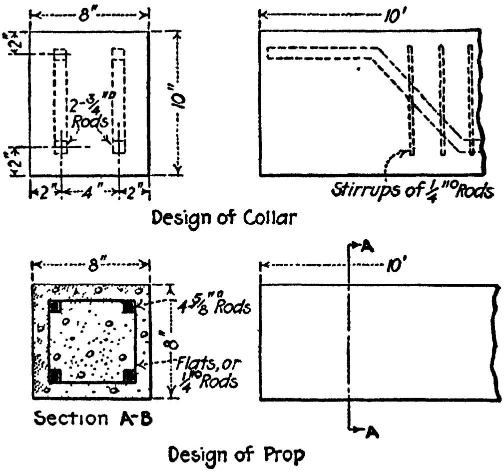


FIG. 8.—Design for a reinforced-concrete timber set.

A reinforced-concrete design is shown in Fig. 8, and the above cost is figured as follows:

Steel rods, including stirrups, labor and material	$2.25
Aggregates, sand, cement and stone, hauling, mixing concrete and placing, including foreman's wages	3.28
*Forms, carpenter work, labor and material (assume forms used three times, equals one-third of $6.25, equals)	2.25
	$7.78
Transporting to mine and erecting (30% of $7.78)	2.22
Total	$10.00

* This item includes removal and resetting of forms.

Regarding the handling, the following is a comparison of the weights per set: Timber 2200 lb., steel 650 lb., reinforced concrete 1800 lb. From these figures it will be seen that the advantage as regards weight lies with steel, but this handling would only figure in the cost of transportation from the surface in proximity to the drift, slope, or shaft mouth to the destination in the mine, and would not in any way, enter into freight charges, the transportation costs on the plain concrete material being considerably less than on the steel set. It will also be seen that concrete compares favorably in weight with lumber.

At shafts No. 3 and No. 4 of the copper mine of the Ahmeek Mining Co., at Ahmeek, Mich., about 1912 the use of concrete timbers was tried, the results and costs of which are given herewith:

After some experimenting, the concrete as finally used was composed of portland cement, conglomerate sand and trap rock trommeled over ¾-in. through 1½-in. screens. The proportions used were 1:3:5 in wall plates, end plates and dividings, and 1:2:4 in the studdles (or struts). The reinforcement in wall and end plates consisted of three, ¾-in. monolith-steel bars with ¼-in. webs crimped onto them, together with two straight ¾-in. bars. The dividings were reinforced by four ½-in. bars wound with ¼-in. steel wire, the whole presenting a column with square cross-section. Studdles were reinforced with two pieces of old 1¼-in. wire rope. Offsets were molded in all plates 5 in. from the inside face to accommodate lining slabs. The design of the different members is clearly shown in Fig. 9.

Reinforced-concrete slabs were molded for the shaft lining, the material used being fines of trap rock (under ¾ in.), and conglomerate sand, with Kahn expanded metal as reinforcement. The mixture used for slabs was 1:2:4. By way of experiment a piece of No. 1 hemlock plank of the same length, width and thickness as a concrete slab which had seasoned for a year was supported at either end, and placed side by side with a concrete slab, and submitted to an equal pressure applied across the center of each.

Three failure cracks appeared in the concrete slab just previous to the breaking of the hemlock plank, although total collapse of the concrete slab did not occur until the pressure was considerably increased. While the method of the test employed was crude, it proved that the concrete slab was much superior in strength.

A concrete mixer of the drum type was employed in preparing the charge for the forms. The amount of water used

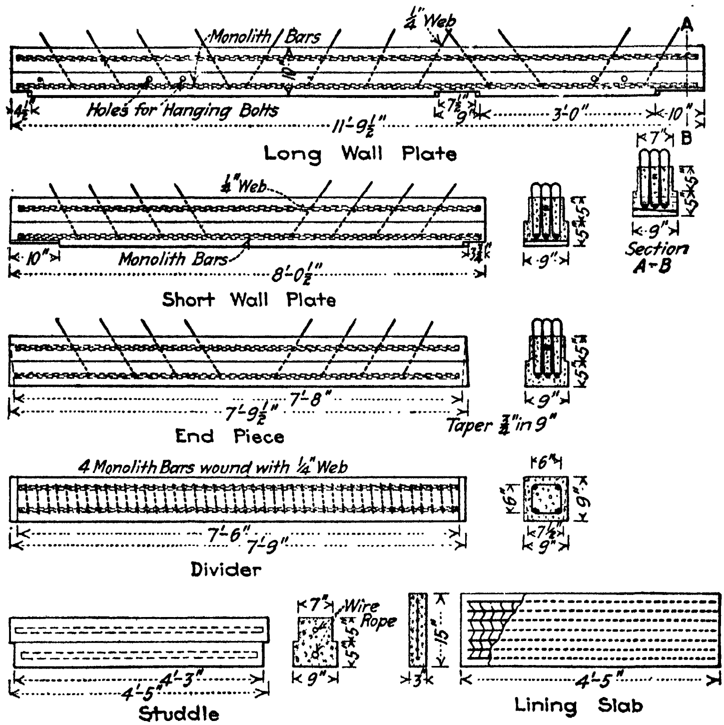


FIG. 9.—Reinforced-concrete sets or timbers for shaft lining.

in the mix was such that when the batch was piled, it settled rapidly without agitation. A drier mix was attempted by way of experiment, but owing to the amount of reinforcement employed, it was found impossible to ram this concrete into place.

The labor force required was six men, as follows: Two carpenters, setting up forms and keeping them in repair; one man wheeling forms onto skidways ready for filling, returning used forms to shop and cleaning the forms; one man feeding the mixer from stick piles of rock, sand and cement; one man delivering mix to forms and shoveling material into place; and one mason ramming charge into final position. With this combination of men as many as four complete sets, consisting of 64 separate pieces, have been molded in one day of nine hours.

The shafts lined in this way are of the three-compartment type (with two skipways and one manway), dipping at an angle of 80 deg. The compartments are 7 ft. 6 in. high inside, with a width of 6 ft. 10 in. for the skipways and 3 ft. for the manways, with the end plates and dividings making the greatest span 7 ft. 6 in.

The weights of the different pieces comprising the set are as follows:

	Lb.
Long section of wall plate	1035
Short section of wall plate	700
End plate	600
Dividers	645
39-in studdles	268
Complete set, 16 pieces	8104

Taking the weight of No. 1 Western fir, which has been exposed to the weather in stock piles, as 33 lb. per cu.ft., the above concrete set weighs almost three times that of a 12×12-in. timber set which the concrete set is intended to replace. Because of this additional weight of the concrete set, it was found necessary to increase the erecting gang from the usual 5 or 6 men on the timber sets to 7 men for the concrete sets. In a vertical shaft to which the concrete sets are especially adapted, the number of men per gang might be reduced.

The comparative cost of the concrete set and timber set, delivered at the shaft collar is striking. The concrete set was delivered for $22.50, the timber set for $37.50. These figures are based on the following prices:

Western fir	**$28.00 per M., f.o.b. car**
Crushed rock	**35c per yd., f.o.b. shaft**
Conglomerate sand	**60c per yd., f.o.b. shaft**
Portland cement	**$1.15 per bbl., f.o.b. works**
Reinforcement	**$12.00 per set, f.o.b. factory**

Cement gun.—The cement gun has come into favor for certain uses in connection with mine timbering and the following are some representative costs of this work as of 1920:

In the examples given, details of cost such as are available have been presented, and are incomplete, in that power and replacements are usually omitted. In the cement gun itself the gaskets for the cone valves require replacement from time to time, as does the outlet-valve body liner. Liners are used for the nozzles. Compressed air and water hose are subject to the wear which comes from frequent handling, and would require replacement no more frequently than drill hose. The material hose is subjected to considerable wear. Estimates of its life range from four to six months with continuous use. Nozzle liners last eight days and cost 80c. for renewal. The upkeep cost on the work in the Anaconda properties at Butte amount to 50c. per eight-hour shift per machine operated.

The elements of cost of a single job in summary are:

1. Assembly of machine, compressed air and water piping, materials, mixing apparatus.
2. Preparatory cleaning of surfaces, placing wire reinforcing, staging.
3. "Guniting": Labor, power, water, cement, sand, lubricants.
4. Wear and replacement of liners, gaskets, hose, gun parts.
5. Tearing down, cleaning up and removal of apparatus.

H. V. Croll has given the cost data shown in the table. In this example "gunite" was used to prevent the walls of a mine "tunnel" from slaking.

M. S. Sloman described the coating of a coal mine slope with gunite in 1918. The surface was first cleaned and scaled. No reinforcement was used and the coating averaged ½ in. in thickness, 1:3 mixture. Timbers were covered with ½-in. wire mesh. The slope was 12 × 12 × 6250 ft. The total cost was $7488.58, or 30c. per sq. yd. (3.3c. per sq. ft.). A 900-ft. section of this slope required 13½ 8-hr. shifts for a working crew of eight men; 2376 sq. ft., or approximately 100 cu.ft., was averaged per shift. Materials required were 540 sacks cement at 60c. and 1620 sacks sand at 12c. per sack. The working crew comprised one mechanic, one engineer on hoist, one operator on cement gun, two mixers, one nozzle man, one man drying sand, and the part time of one man hauling sand. No mention is made of power cost.

COST DATA OF PLACING "GUNITE" IN TUNNEL AT UNITED VERDE EXTENSION MINING CO., JEROME, ARIZ.

1 Gun man, also motorman	$ 7 00	
1 Nozzleman	5 60	
1 Man holding lights	5 60	
1 Man loading gun	5 60	
1 Man cleaning roof	5 60	
3 Men mixing at $5 60	16 80	
Total labor		$ 46 20
Cement, 77 bags at $1		77 00
Sand 9 cu yd at $1		9 00
Air and supplies		10 00
Superintendence		5 00
Total		$147 20

3750 sq ft at $147 20 = 4c. per sq. ft.

Above crew placed 125 running feet of tunnel in one 8-hour shift, equivalent to 3750 sq ft; average thickness ½ in.

Stephen Royce described the use of gunite at the Cary "A" shaft at Hurley, Wis. This is a steel, five-compartment shaft with the steel sets blocked in place with wooden blocking and lagging of 3-in. tamarack planks wedged into the flanges of the I-beams. "Gunite" was applied to fireproof the lathing and wooden blocking and to protect the lagging from decay by keeping it from contact with air; also to prevent corrosion of shaft sets and water-proof the shaft. The surface to be covered was cleaned thoroughly. This was done partly with water under heavy compressed air pressure, partly by sand blasting, and partly by chipping the rust and accumulated coating from the steel. Reinforcement consisted of No. 7, A. S. & W. Co.'s triangular-mesh reinforcing wire for the side walls. This was stapled directly to the I-beams and to the lagging at intervals. To keep the wire mesh about one-eighth of an inch away from the surface to be covered was accomplished by stapling the reinforcing wire on with nails underneath it. The I-beams, before the cement was applied, were covered with 1½-in. mesh chicken wire, clamped on with wire clamps. The total area of wall surface was 14,260.9

sq.ft.; of steel covered 3749.96 sq.ft., or a combined total of 18,010 sq.ft. The materials required were sand, 102.5 cu.yd.; cement, 173 bbl.; reinforcing, 14,260.9 sq.ft., and chicken wire, 3750 sq.ft. Linear feet of shaft was 263.13. The work required one foreman and six men for thirty-two working days. The cost was given as $9.30 per linear foot of shaft. As the area per linear foot equaled 68.4 sq.ft., the per-square-foot cost is found by calculation to be 13.6c. The thickness of the coating was given as 1½ in. The "gunite" was applied in from one to three coats.

Reclaiming timbers.—Although mine props and sets of timber are often broken a short time after being set, the broken ends are valuable as they can still be utilized for the purpose of cap-pieces, wedges, track ties, or for the building of "cogs" or *"chocks." Also, post timber broken in a thick seam can often be used again, in a thinner seam at the same colliery.*

In some mines there is a considerable loss of timber, through the carelessness of miners who will let them lie in the waste where they are finally buried. By keeping a careful watch in their daily rounds through the mine, the mine officials can do much toward reducing this loss or waste of timber.

It is well to emphasize the importance of drawing all kinds of timber, as the work proceeds, using, if necessary, some suitable appliance for this purpose. Timbers left standing in the waste often cause a loss greater than their own value, by preventing the roof from caving and frequently making it necessary to build extra packwalls or timber cogs to keep the roads open. The material for these packwalls often has to be transported a considerable distance; whereas, if the timber was drawn and the roof allowed to fall, there would be plenty of material for the building of all necessary packwalls in most cases.

Again, under many conditions, when the roof does not fall but a large standing area is kept open a great weight is thrown on the timbers standing next to the face of the coal, with the result that these timbers are broken more quickly, or they kick out and the roof is ruptured at the face. When this occurs, the condition is bad, as the influence of the roof in breaking the coal after the latter is undermined, is destroyed. When the roof is of such a nature that it breaks readily, it is a very good policy to set a line of large breaking posts, with good sized cap-pieces, on one side of the track, which should be carried along the straight rib of the room.

As the face of the room advances, up to the last crosscut, there is not only a saving of timber, but the caving of the roof prevents the crushing of the pillar coal when the roof "weights" and cannot fall. In heavy pitching seams, the recovery of timbers is much more difficult and dangerous than in flat seams; because the worked-out portion, from which the timbers are drawn, is located up the pitch, and any loose pieces of rock that fall when the post is drawn are liable to roll or slide down upon the men engaged in drawing the timber who are unprotected.

The danger may be avoided, in part or wholly, by using a long ½-in. steel cable or chain that will reach from the timbers to the first crosscut, in which the drawing machine should be placed. This will not only afford the necessary protection for the men, but will enable them to recover a larger percentage of timbers. The cost of timber in pitching seams is much greater than in flat seams, owing to the labor required in handling the timber on steep pitches. In a seam pitching 35 deg. the cost of timber frequently amounts to an average of about 5½c. per ton of coal mined. This was in a mine where the roof conditions were fairly good.

V

MISCELLANEOUS INSIDE COSTS

TUNNELING COSTS

American and foreign tunneling records compared.—The accompany table gives some of the most creditable American tunnel records made up to 1909. The ranking order of some of the tunnels is probably open to discussion. That given, based on the 31-day record, is by no means necessarily the order of merit. There is a tendency among mining men to mention the highest one month's record, however, rather than to recall record figures extending over longer periods.

A comparison of foreign records with our own is shown in another table. On the face of this comparison, American records appear to rank ahead of the Continental records. Thus the best month's record on the Gunnison tunnel actually exceeds the best Simplon record by 87 ft. While we have no explicit data concerning the kind of rock in which the record progress for the Simplon was made, we know that the Gunnison record was made using air-driven augers in the soft shale of the east heading, in which 7500 ft. of progress was made for the first year, making the high average of 625 ft. per month. In the granite of the west heading, the best Gunnison record 449 ft. per month so that this is the figure that should in all fairness be compared with the Simplon record of 755 ft. While no long-time average figure is available for the Gunnison progress in granite, we can hardly anticipate a serious rival to that remarkable record in the Simplon of 426 ft. per month for 76 months. The top notch American record in granite, however, was that made in October, 1908, by the Elizabeth Lake tunnel at Los Angeles, Calif., where a progress of 466 ft. was made in the 31 days ending Oct. 31.

AMERICAN TUNNEL DRIVING RECORDS

Name of Tunnel	Location	Length, Feet	Size Heading, Ft	Begun	Rocks Penetrated	Best Month s Record	Best Long Time Average	Flows of Water Encountered	Dist Ti ber
Elizabeth Lake	Los Angeles, Cal	26,860	12×12	Oct 15, 1907	Granite	466 ft, Oct, 1908			
Gunnison	Montrose, Colo	30,582	8×12	Jan, 1905	Soft shale, Hard granite	842 ft, 449 ft in Jan, 1908	625 ft per month for 12 months in shale Driver by augers	2 8 sec feet at 85° F	15,000 ft.
Roosevelt	Cripple Creek, Colo	15,550	7×10	May, 1907	Pikes Peak granite	435 ft, Jan, 1909	292 ft per month fo 12 months ending Feb 1, 1909	800 gal per min	None
Cowenhoven	Aspen, Colo	8,319	10×10	July 29, 1889	Hard dolomite	421 5 ft, May, 1894	198 ft per month for 42 months	Ordinary flow 1200–1500 gal Occasionally 4000 gal per min.	Several thousand feet
Ophelia	Cripple Creek, Colo	8,500	9×9		Granite Phonolite Basalt	395 ft 8 in	350 (?)		
Newhouse	Idaho Springs, Colo	22,000+	12×12 6×9	1893, worked intermittently	Granite and gneiss	290 ft (6×9)	244 ft per month for 12 months		Little
Joker	Red Mt, Colo	5,055	11×12	July, 1904	Andesite breccia	282 ft	220 ft per month for 12 months		Entirely
Busk	Colorado Midland R R			1890		202½ ft			
Cascade	Great Northern R R, Wash			1897		301 ft			
Kellogg	Kellogg, Idaho					354 ft in 1898			
Aspen	N P R R					306 ft in 1901			
Raton	A T & S F R R					412 ft in 1907, driven by augers			

COMPARISON OF AMERICAN AND FOREIGN RECORDS

	Name	Continental	American
		Simplon Tunnel	Gunnison Tunnel
1	Best month's record Best long-time record	755 ft 426 ft per month for 76 months	842 ft shale, 449 ft granite 625 ft for 12 months (shale)
	Name	St Gothard Tunnel	Roosevelt Tunnel
2	Best month's record Best-long time record	563 ft 263 ft per month for 93 months	435 ft 292 ft for 12 months

A mere comparison of records, however, is more favorable to Americans than will appear because of the difficulties of driving the Continental tunnels due to great length and depth beneath the surface, resulting in excessive rock pressure and high temperature, not to mention copious flows of hot water. In none of our long American tunnels, unless it was the Comstock at Virginia City, Nev., were conditions of high temperature and flows of hot water encountered, comparable to those of the Simplon and St. Gothard tunnels. For the Sutro, by the way, the best month's progress is recorded as 417 ft.

In case it may be argued that it is unfair to compare railway tunnels with mine tunnels, it is only necessary to point out that all data given relate to the speed of driving the headings only, whose size does not greatly differ in a railway tunnel from a mine adit.

The argument often advanced in excuse of American backwardness is that the breaking qualities of the rocks penetrated by our tunnels are much inferior to those of the Continental tununels. In the accompanying tables the seventh column gives the type of rocks encountered. This will not help the comparison much except in a general way, for every mining man knows that what may be a soft rock in one locality is hard and tough in another. Hardness of rock is hardly a fair basis of comparison, nor is, indeed, specific gravity. It is a well-known fact that in many of our American tunnels, some of the hardest rocks to drill actually blast the best, while the best Simplon tunnel records are said to have been made in the hard Antigorio gneiss.

FOREIGN TUNNEL DRIVING RECORDS

Name of Tunnel	Location	Length, Feet	Size Headings, Feet	Begun	Completed	Rocks Penetrated	Best Month's Record	Best Long-Time Average	Flows of Water Encountered	Rock Temperature
Simplon	Brigue, Switzerland	64,818	6.6×9.8	Nov. 13, 1898	Feb. 24, 1905	Schist, hard Antigorio gneiss	755 ft.	426 ft. per month for 73 months	8000 to 13000 gal. per minute, maximum temperature of 46.1° C.	12.4–31.7° C.
Arlberg	Alps Mountains	33,696		1880	1883		641 ft., July, 1883	400+ ft. for 40+ months (?)		
Albula	Alps Mountains			1900	1902	Antigorio gneiss	607 ft., Dec., 1901			
Loetschberg	Switzerland			1906			574 ft., Sept., 1907	555 ft. average per month of 5 selected months, 1907–8		
St. Gothard	Airolo, Switzerland	48,887	(?)	Aug. 7, 1872	Apr. 30, 1880	Gneiss, schist and limestone	563.3 ft.	263 ft. per month for 93 months and 343 ft. per month for 12 mos.	One flow 6000 gal. per minute	18.3–31.1° C.
Karawanken	Austrian Alps			1900	1905		553 ft., July, 1904			
Tauern	Austria			1902	1907		548 ft., Oct., 1904	528 ft. per month for 12 months of 1905		
Bosruck	Austrian Alps			1902	1905		546 ft., Feb., 1905			
Ricken	Switzerland			1903	Work suspended 1907 by gas		461 ft., June, 1905	421 ft. average of 4 selected months, hand drilling alone		
Mont Cenis	Alps Mountains	42,157	8.2×9.8	Aug. 18, 1857	Dec. 28, 1870	Metamorphosed slates, sandstones, limestones, and schists	297 ft., 299.5 ft.	131 ft. per month average for 160 months	Temperature one flow 28.7° C.	15.6–29.5° C.

In America, the heavy Sullivan drills made the first Ophelia tunnel record at Cripple Creek, and then beat it by 54 ft. on the west heading of the Gunnison. The Leyner No. 9 hammer drill beat the Ophelia record at the Roosevelt tunnel work, and finally captured the American record previous to 1910 by its run of 466 ft. at the Elizabeth Lake tunnel, Los Angeles, Calif., October, 1908.

In Europe the Ferroux percussion drill and the Brandt rotary core drill, both of foreign make, have established a long series of records that threw the performance of American drills far in the rear until the appearance of the Ingersoll-Rand drill at the Loetschberg tunnel in 1906. Since then, this American drill has not only eclipsed all American records by its September, 1907, record progress of 574 ft., but has exceeded all the previously established foreign records except three, namely, the Simplon (755 ft.), the Arlberg (641 ft.), and the Albula (607 ft.).

But, if any one individual machine is to have the credit for the world record performance of tunnel driving up to 1909, that credit belongs to the Jeffrey A-2, air-driven auger with which 842 ft. of soft shale were removed in one month's time from the east heading of the Gunnison tunnel.

When the Brandt drill accomplished the wonderful records of the Simplon, some engineers went so far as to say that its success spelled the finish of air-driven percussion drills. Others, however, pointed to the fact that in driving the Arlberg tunnel with Ferroux percussion drills in one heading and with the Brandt rotary drills in the other, the best month's record of the two differed by less than 1 per cent. As a matter of fact, an average of the four best months' records of the Ferroux drills was 613 ft. as against 576 ft. of the Brandt drills. The top-notch record of each was 637 ft. for the Ferroux and 641 ft. for the Brandt.

The secret of the great speed made in the Alpine tunnels appears to have been in the very careful study made of the various causes of delay in the successive operations of drilling, blasting, mucking out, and setting up the drills again. As a result, a radically different method of mounting the drills in the heading has been employed from that practiced in

America. In the mounting of the drills, the return to the old carriage drill is seen, but there is no such cumbersome affair as the drill carriage used in the early tunneling operations in this country.

The drill carriage at the Loetschberg consists simply of a small truck whose wheel base is about 4 ft., running on the regular heading track. Mounted on this truck in the longitudinal axis of the tunnel, and hinged to swing vertically, is an I-beam set with its web vertical and reinforced to give it lateral stiffness. On the forward end of the I-beam there is mounted what would in America, be called a shaft bar, set transversely while the beam is pivoted so as to swing the bar horizontally. On the opposite end of the I-beam is mounted a counterweight. Four drills are mounted on the shaft bar, the compressed-air connections from these drills running back to one hose connection in the rear of the truck. When not in use, the shaft bar is swung so that it lies directly over the I-beam and the carriage can then be run anywhere over the heading tracks occupying no more room in the heading than two muck cars. This carriage is practically the same as that used by Brandt on the Simplon tunnel, the main difference between the two systems of work being found in the drills.

Before blasting, a $\frac{3}{8}$-inch plate of steel about $6\frac{1}{2} \times 3\frac{1}{2}$ ft. is laid down just ahead of the track end. After blasting, the gases are sucked out of the heading through a pipe of 24 in. diameter, the fan being capable of running either as an exhauster or blower. A cut is then mucked through the center of the muck pile down to the steel plate sufficiently wide to allow the arm of the drill carriage to introduce the bar carrying the drills into the top of the heading, the drill carriage, of course, running on the steel plate laid down ahead of the track. The shaft bar is then jacked firmly against the sides of the heading and drilling immediately begins on the top holes. Mucking out then continues while drilling is in progress. On account of the drills remaining on the carriage all the time with the air connections at the drills undisturbed, there is little chance for grit to get into the working parts and so impair their efficiency. The following table gives the approximate time required for the various operations in the heading.

	Minutes
Setting up drill carriage in the heading	20
Drilling 12–14 holes, 4 ft. deep, 2 in. diameter, at bottom	60
Removing drill carriage from heading	20
Loading and firing holes	30
Clearing out smoke	20
Cutting center of muck pile to admit carriage	90
	240 =4 hours

As before mentioned, work is carried on in three 8-hr. shifts, each shift being expected to drill two rounds and shoot twice making 7 ft. per shift advance or 21 ft. per day.

Here we have an American drill adapted to a European system of work, beating all American records and threatening to rival those of the Simplon before construction is completed. It is apparent that system rather than the various European makes of drills is to be credited with the Continental records.

It is interesting to compare these time items with those of the Simplon tunnel which were described in a paper on tunneling, read before the Royal Institute of Great Britain, May 5, 1900:

	Minutes	Minutes
Bringing up and adjusting the drill	20	25
Drilling	105	150
Charging and firing	15	15
Cleaning up rock debris	120	120
Total	260 =4 hours, 20 minutes	305 =5 hours 5 minutes

The average advance made by this method is given as 3 ft. 9 in., or about 7½ ft. per 8–10-hr shift.

Perhaps the first American engineer to successfully apply a bonus system of payment for tunnel driving in addition to the usual wages was D. W. Brunton, of Denver. The following were the rates of payment used by him in driving the Cowenhoven tunnel at Aspen, Colo., in 1889, when the progress made per month exceeded 150 ft.

Progress, Feet	Bonus for every Foot
150–200	$1 00
200–250	1 50
250–300	2 00
300–350	2 50
Over 350	3 00

With these payments drillmen often made $120; helpers, $112; and laborers $95 per month in addition to their regular wages.

Wherever tried in America, the bonus system has generally proved a success. This is for the simple reason that an unruly giant will not especially exert himself to earn a $3 or $4 per shift. But make him a bonus-system proposition whereby exertion of wits as well as muscles may bring him from $6 to $8 per shift, and we have a transformation from halfway indifference to keen interest in his work. Stimulating the spirit of competition between the various crews has in some cases given good results in increased work but in no case has the glory of beating the other fellow proved so satisfying as that extra $2 or $3 per shift.

Cost of rock tunnel at a coal operation.—As a general rule the only literature on driving rock tunnels is that descriptive of work in metal mines, or for irrigation, water supply and railroads. Such tunnels are usually driven by organizations specializing in this work and much effort is expended toward making records.

In coal-mining operations and the opening of new coal fields it is often necessary to drive rock tunnels but only small mentioin is made of them and the special methods used and equipment available.

The tunnels here described were used to open a new development adjacent to the Utah Fuel Co.'s Clear Creek mine No. 1, about 1914, and were driven as haulage and air-course entries.

In order to systematize and expedite the work as much as possible a separate organization was created—made up of expert hard-rock men. Although kept distinct from the regular operating organization it was necessary to coördinate their work with that of the mine in order not to interfere with the production of coal.

The main haulage rock tunnel was driven with a rectangular cross-section having a minimum height of 8 ft. and a minimum width of 10 ft., while the air course, of a similar cross-section, had minimum dimensions of 8 ft. in height and 12 ft. in width. Overbreakage and trimming have increased these dimensions on an average of one foot each way. Two tunnels were driven in order to provide for adequate ventilation of the new mine as it was impracticable to sink a shaft or drive an adit from the outside for air.

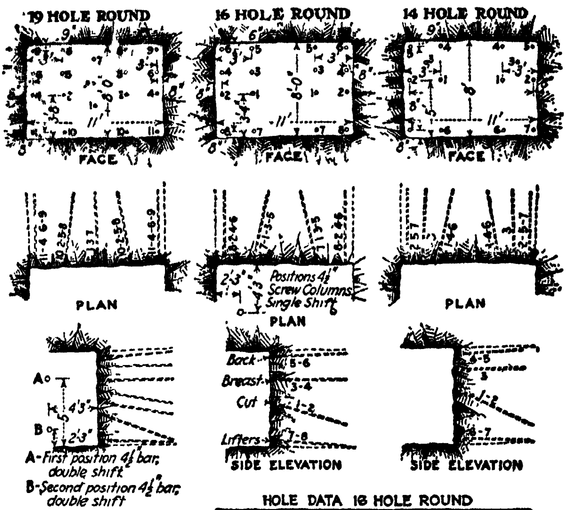
The rock strata penetrated were sandstones and shales of medium hardness. The shales were the hardest to drill as the gumming of the cuttings made it difficult to keep the holes clean. The first halves of the tunnels, driven on a ½ per cent grade, were parallel with the bedding planes of the strata and thus much harder to break. The last halves, driven on a 7½ per cent grade, took the tunnels off the bedding planes somewhat but increased difficulties were encountered in the numerous small faults which were cut through as a main faulting plane was approached.

On the average the ground was fair drilling but in some places it was necessary to use water under pressure to clean out holes so as to render it possible to drill them at all. In such cases the water, at 100 lb. pressure, was forced into the holes through ⅛-in. pipe 16 in. long used as a nozzle. The time necessary to drill the rounds varied from one hour and 45 min. to the full 8-hr. shift. The outside back holes were drilled first, then the outside breast holes, then the inside breast holes, and so on as shown in the accompanying sketches, Fig. 1, the numbers designating the order of drilling.

The source of power was the power plant of the mine, consisting of six 125-hp. 72-in. by 16-ft. horizontal return-tubular boilers hand fired with slack or the poorest grade of run-of-mine coal coming from the mine, and two Thompson-Ryan generators of 175-kw. capacity, 320 amp. 500 volts, driven by 19 $\times$ 18-in. McEwan engines of 260 hp. each at 200 r.p.m. Power was charged against the work at the rate of 1.24c. per kilowatt-hour.

Current from the generators was conducted about 200 ft. to a 24 $\times$ 64 ft. combination compressor and blacksmith shop, 40 longitudinal feet of which was used for the compressors.

Two Ingersoll-Rand compressors of the Imperial type No. 10, were used. These compressors had a rating of 427 cu.ft. of free air per minute at 175 r.p.m. at sea level. Each machine was driven through a belt connection from a 500-volt 116-amp. 75-hp. General Electric continuous-current motor operating at 850 r.p.m.



HOLE DATA 16 HOLE ROUND

KIND OF HOLE	LENGTH INSIDE RD	LENGTH OUTSIDE RD	STICKS OF 60% POWDER 8x1¼
Back	6'-4"	5-6"	8 to 10
Breast	6'-8"	5'-10"	9 to 11
Cut	7'-2"	6'-4"	12 to 15
Lifters	6'-4"	5'-6"	12 to 15

STEEL DATA

KIND	MATERIAL	LENGTH	SIZE	GAGE
Starters	Cruciform Steel	2'-6"	2"	3"
Seconds	Cruciform Steel	4'-6"	1¾"	2¾"
Thirds	Cruciform Steel	6'-6"	1½"	2½"
Fourths	Cruciform Steel	7'-6"-8'6	1½"	2¼"

FIG. 1.—Method of drilling rock tunnel at the Sunnyside Mine.

Air was conducted to two receivers in the compressor house, one being 3 ft. in diameter by 8 ft. 3 in. long and the other 3 ft. 6 in. by 8 ft., then into the mine through a 4-in. pipe 3725 ft. long to a 3-ft. 6-in. by 8-ft. receiver, thence 350 ft. through a 3-in. pipe line to the starting points of the tunnels and through 3-in. branch lines to within 50 ft. of the faces, 1-in. air hose being used from thence to the drills.

Drill steel was sharpened with a Numa rock-drill sharpener. For heavy blacksmith work the smith set up an old Sullivan coal puncher for a hammer. This required only the fitting on of a hammer-block $4 \times 6 \times 6$ in. in place of the bit of the puncher, setting the machine on a suitable frame and making a foot control. The expense of this makeshift was slight but the saving effected in the cost and in the grade of work done was quite noticeable. For forge fires a combination of coke breeze and slack coal from the Sunnyside mines of the company gave good results.

Chicago giant rock drills, size, 3¼ in. were used in driving, while jackhamers and Sullivan stoping drills were used for trimming and widening. Two drills were used at each face and these were mounted on 7-ft. double screw columns. On double shift work, the muck was always in the way during the first drilling so it was necessary to mount the drills on 10½-ft. single screw bars, having a single screw brace to the face to take up the vibration in the bar. In this way the upper rows of holes were drilled first with the bar set near the roof and by the time these were completed the muck was cleared away from the face and the bar could be set up in a lower position and the remainder of the holes drilled. A 3-in. water main having ¾-in. hose connections was used along the back entry to within 50 ft. of the face. From these connections to the faces, ¾-in. pipe was used with $\frac{1}{8} \times 16$-in. special reducing nozzles.

The explosive used was mainly 60 per cent dynamite with sticks 8 in. long and 1¼ in. in diameter.

On single shift the machine men worked during the day, that is from 7:30 a.m. to 4 p.m., and the muckers and drivers during the night shift beginning at 8 p.m. and ending when the muck was all cleaned up. The shift for the machine men, muckers and drivers, while nominally 8 hr., was considered finished whenever their work was completed. A bonus was given of 10 per cent of the day's wage for each foot over 3 ft. of tunnel driven that they averaged per shift. The bonus was calculated at the end of the month instead of each day, except in the case of men quitting before the end of the month. The average wages earned by this system by the machine men was $4.17 per shift. These men were required to do the drilling, extend both the air and water lines, make up primers, load and shoot holes.

Timbermen were paid the same wages as machine men including the bonus. There was comparatively little timbering to be done so that machine men were used for this work and it was thought fair to allow them the same wages as they would have earned if running machines.

There were four muckers to each heading who were paid \$3.25 per 8-hr. shift and a bonus of 10 per cent of the day's wage for each foot over three feet of tunnel driven per shift, just as paid the machine men. By this system the average wage earned by muckers was \$3.84 per shift. The muckers laid all track, not including switches, put down the mucking sheets, loaded and unloaded drill steel and kept track clean for 100 ft. from the faces.

Drivers were paid \$3.15 per 8-hr. shift without any bonus. They were required to take muck away from the faces to the parting, bring in empties and haul steel, powder, caps, fuse, rails, ties, etc., from parting to faces.

The dumpers outside were paid \$3 and \$3.15 per 8-hr. shift without any bonus. They were required to handle the rock but the extending of the tipples was done by the regular mine carpenters paid \$3.40 and \$3.15 per 8-hr. shift.

The blacksmith was paid \$4 per 8-hr. shift and straight time for overtime. His duties were the sharpening of drill steel, general repairs and other blacksmith work. The compressor attendants were paid \$3.50 per 8-hr. shift and were required to run the compressors and assist the blacksmith. The head foreman was paid \$175 per month and his assistant \$4.50 per 8-hr. shift.

The work of widening the main haulage tunnel for the pass-by and parting was done after the tunnels were driven to the coal. For this work Ingersoll-Rand Jackhamer drills with 7/8-in. hollow steel were used. The men were paid for straight time at the same rate as for tunnel driving, no bonus being given. The shots were detonated by electricity, using 6X detonators. A total lineal distance of 700 ft. 8 ft. high was widened from 3 to 5 ft., the unit cost for the work being given on p. 418.

If the cost of the widening operations is added to that of driving the tunnels the total expense per foot would be about \$17, but if salvage is allowed on pipe and material left over

UNIT COSTS OF STRAIGHT TUNNEL

Engineering		$0.0503
Superintendence		0.9885
Drilling	Machine men Machine men bonus	3.8221
Handling rock	Muckers Muckers' bonus Dumpers Extending dump	4.0331
Hauling	Drivers Motor men Shafts Stable expense Horse killed Ditch and track Tracks and switches	1.3193
Blacksmithing		0.3844
Power	P.H. charge Running compressor Repairs, compressor Pipe lines Repairs, drills	1.8604
Timbering		0.4389
Blasting material		2.7707
Machinery	Housing Setting Depreciation Miscellaneous	0.6655
Total cost per foot of tunnels		$16.3332

COST PER LINEAL FOOT OF WIDENING OPERATIONS

Superintendence		$0.4675
Drilling		0.3656
Handling rock	Muckers Dumpers Extend dump	1.3243
Hauling	Drivers Stable expense Motor men	0.3171
Blacksmithing		0.1712
Power	Compressor men Pipe lines	0.1681
Blasting material		0.5292
Miscellaneous		0.0486
Total cost per linear foot		$3.3916

and subsequently used for coal-mining operations the cost per foot is brought down to $15, which would be the net cost.

The average progress per shift for both single- and double-shift work during the whole time of driving the tunnels was 5.11 ft. The highest average per shift per month was 5.56 ft. The greatest distance driven in a tunnel in a single month was 323 ft. in 60 shifts.

Single-shift work was found to be less efficient than double-shift work, although to a certain extent this was due to the forming of a working organization and to the necessary experimenting with the drilling and shooting of the ground during the earlier stages of the undertaking.

Some examples of tunnel costs.—A tunnel located at Idaho Springs, 37 miles west of Denver, in the lower Clear Creek mining district of Colorado started in 1893 and extended at intervals over a 10-yr. period, presents some interesting cost data. The original purpose of this tunnel was to cut the well-known veins on the line of the tunnel so as to receive royalties from the property owners for drainage and for transportation of ore.

The yearly progress made is as follows: 1893, 80 ft.; 1894, 1405 ft.; 1895, 1992 ft.; 1896, 2061 ft.; 1897, 1080.5 ft.; 1898, 99 ft.; 1899, 455 ft.; 1900, 2285.3 ft.; 1901, 2923.5 ft.; 1902, 761.1 ft.; 1904, 1389.3 ft.; 1905, 672.5 ft. A summing up of the above footages gives a total length of 15,154 7 ft. from the original starting point.

The first 80 ft. of the tunnel were driven by hand, but as there were no available data concerning this hand work, very little can be said about it. This method necessarily made the progress of the work very slow, and it was deemed advisable to install a compressed-air plant to supply power drills. The equipment was in duplicate, consisting of two Norwalk compound 14 $\times$ 16 in. high-altitude compressors and two 80-hp. boilers. These compressors supplied air to the drills, which were of the 3-in Leyner percussion type. From 100 lb. to 175 lb. of dynamite were used to the round, and the greatest footage made was 160 ft. per month by a working force of 26 men.

In October, 1899, there was a change in the management, and the work was done in a more systematic manner. The equipment of the power plant was increased by the addition of another 80-hp. boiler and a 22 $\times$ 24 in. compound Norwalk compressor.

The holes were placed in the breast according to the American center-cut system with an extra plunger hole at the upper center of the cut. The side and cut holes were drilled by two 3-in. Leyner sluggers mounted directly on separate columns placed on each side of the tunnel, while one model 5 Water Leyner, mounted on an arm, drilled the back and plunger holes. One Slugger was started at the bottom and worked up, while the Water Leyner put in the back holes on that side. The other Sluggers started at the top and worked down so that it was out of the way when the Water Leyner machine was ready to shift to the other column.

This system, together with the use of high pressure air, 160 lb., resulted in deeper holes in less time, and therefore greater progress. In blasting, the cut holes were electrically fired first, then others until the whole cut was taken out clean, using about 100 lb. of 60-per-cent gelatin powder. The side and back holes were then fired, using from 50 to 70 lb. of 40-per cent gelatine powder. In no case were the holes tamped.

The greatest footage made in any one month under this system and management was 267 6 ft., and a total of 2760 was made from September, 1899, to August, 1900, at an average cost of $28 per foot, or a total of $79,470.

The premium system of wages for employes was used. This consisted of paying $6 for every foot driven over 160 ft. per month. This $6 was divided proportionately between the drill gang, powder gang, and muckers, according to their wages.

The success of the rapid progress of the tunnel under this management can be attributed to the following points:

1. The use of high-pressure air, 160 lb., being the minimum pressure kept at the breast.

2. The use of the Water Leyner machine for the back holes; for by putting in these holes to a greater depth, it was possible to put in a much deeper round throughout.

3. The premium system of paying, which offered an incentive to employes to do their best work.

Below is given a typical monthly footage expense during the year above mentioned taken from the 1900 annual report:

Drill-crew men and foreman	\$3.11	
Trammers and drillers	3.88	
Blacksmith shop	1.13	
Engineers	1.08	
		\$9.20
Ammunition		4.09
Oil and waste		.18
Coal for power		3.78
Feed and shoes for mules		.23
Drill repairs		.81
Premiums on footage		.61
Track equipment and repairs		.78
Mine timber		.31
Labor		1.92
Material		1.10
Labor, repairs along tunnel		1.29
Surveying		.21
Legal expense		.19
Insurance		.07
Salary and office expense		2.39
Minor expenses		.58
Total		\$27.74

Some interesting cost data were obtained in driving a rock tunnel at the property of the Iron Mountain Tunnel Co. at Superior, Mont., in 1906 and 1907.

The new tunnel is 7 × 6 ft., on a grade of 4 in. per 100 ft., and has a drainage flume 12 × 12 in. laid in the flooring. The work of driving the tunnel which is to be 5602 ft. long was begun February 10, 1906. In October, 1906, it had been driven 1500 ft., and 4345 ft. had been excavated by September, 1907, leaving uncompleted a distance of 1255 ft.

Excepting four stretches of ground, aggregating in all about 800 ft., the rock encountered was very hard, requiring a heavy expenditure of explosives. In the distance the tunnel has thus far been driven, it has been necessary to timber only 535 ft., the formation requiring support being encountered in four different places, one of them 325 ft. long.

The accompanying table gives the monthly costs of the work up to September, 1907:

Time of Driving	Feet	Cost per Foot			Feet Driven	
		Gross	Driving and Equipping	Driving	Side-track	Cross-cuts
Prior to June, 1906	559					
1906						
June	231	$20.11	$18.70	$15.58		
July	215	14.81	11.81	11.01	40	
August	229	14.96	14.85	14.15	....	10
September	227	13.29	12.31	11.59	80	
October	285	12.95	12 59	11.80		
November	264	12.66	12.30	11.00	85	
December	260	15.77	15.36	13.39	....	9
1907						
January	288	12.08	11.79	10.64	100	4
February	267	12.09	11.76	10.69	60	
March	244	13.81	13.44	12.75	50	
April	251	13.03	12.68	11.53	70	
May	254	14.35	13.99	13.03	43	
June	251	13.37	13.00	11.88	50	
July	206	14.65	14.22	13 02	64	
August	228	18.11	17.67	16.27		
September	170	15.95	15.55	14.45	100	
Totals	4429	$231.99	$222.02	$202.78	742	23
Averages	233.1	14.50	13.88	12.67	$46\frac{3}{8}$	$1\frac{7}{8}$

NOTE.—No report was made of costs per foot of the first three months' operation.

The city of Los Angeles, in California, started constructing an aqueduct about 217 miles long in 1909, including 105 tunnels whose aggregate length is 28 miles. The Elizabeth tunnel, which is 26,860 ft. long, was driven from both ends, termed the north and south portals. This tunnel has a cross-section 12 ft. 4 in. $\times$ 12 ft. 9 in., a grade of 1 ft. in 1000 ft., and a water capacity of 1000 sec.-ft

The general equipment at this tunnel consists of the following:

Two compressors, 520 ft. capacity, belt driven from electric motors; two motor-generator sets, 150 hp.; one 50-hp. electric locomotive; one 30-hp. electric locomotive; nine water Leyner drills; 38 rocker dump cars, 32 cu.ft. capacity; one drill sharpener.

In addition, the machine-shop equipment included a lathe, drill press, saws, blowers, motors, and the necessary tools for such work Most of the destructible equipment was supplied in duplicate, and extra machine drills were furnished. Each shift was supplied with a tool box and all the tools necessary for its members' work. These tool boxes were locked and one man on each shift was held responsible. A station was cut in the tunnel where all repairs to machines, hose, tools, etc., were made. Wherever possible each individual was held responsible for the tools he used.

Each shift was required to drill and blast, the length of round being regulated by the nature of the ground encountered in the first hole drilled. Discipline and strict attention to business while on shift were required of everyone, while at the same time a spirit of friendliness was fostered and every man made to feel that in a large measure the success of the project was due to his own efforts and the interest he took in the work.

A bonus of 40c. per foot per man for every foot driven over $2^2/_3$ ft. per shift was paid, bringing the wages up to a good figure above the scales usually paid in other mining camps and consequently bringing a better grade of men than could have otherwise been obtained.

The rock encountered was mainly granite, which in places shades into both gneiss and schist. The granite is composed mainly of feldspars and biotite with only a small amount of quartz usually present. Its texture varies at different points along the tunnel, the finely crystalline rock usually being without joints or seams and the coarsely crystalline rock usually being full of seams. The finer crystalline rock, therefore, allows much the slower progress in tunnel driving, while the blocky ground gives about as near ideal conditions for record-breaking drives as one is apt to find.

The rock in the south heading is hard, unaltered, and in general blocky enough to afford fine breaking qualities. It requires no timbering except one or two sets at long intervals

PROGRESS OF ELIZABETH TUNNEL

	NORTH PORTAL								SOUTH PORTAL		TOTAL	
	Portal Heading		North Shaft Heading		South Shaft Heading		Total					
	Excavation	Timbering	Excavation	Timbering	Excavation	Timbering	Excavation	Timbering	Excavation	Timbering	Excavation	Timbering
									72	72	72	72
November, 1907	165	158					165	158	117	102	282	260
December, 1907	142	130					142	130	91	75	233	205
January, 1908	115	98					115	98	85	70	200	168
February, 1908	162	71					162	71	115		277	71
March, 1908	135	140					135	140	110	33	245	173
April, 1908	42	79					42	79	156		198	79
May, 1908	100	98					100	98	244		344	98
June, 1908	132	99					132	99	237		369	99
July, 1908	41	128					41	128	285		326	128
August, 1908	83											
8′×8′	18	100					101	100	288		389	100
September, 1908		143						143	242		242	143
October, 1908	59											
8′×8′	122	30			86		267	30	*466	228	733	258
November, 1908	164	159	41	21	62	87	267	267	395	6	662	273
December, 1908	138	95	96	78	166	171	400	344	206	93	606	437
January, 1909	121	101	141	106	206	186	468	393	383	45	851	438
			5									
February, 1909	226	151	90	78	271	150	592	379	319	91	911	470
March, 1909	60											
8′×8′	185	115	230		246	313	721	428	†474	151	1,195	579
April, 1909	85	232	105	153	287	228	477	613	453	111	930	724
May, 1909	332	83		77		232	332	392	349	117	681	509
June, 1909	229	590					229	590	251	134	480	724
July, 1909	363	210					363	210	339	59	702	269
August, 1909	404	460					404	460	282		686	460
September, 1909							‡488	440	355		843	440
October, 1909							547	520	413	159	960	679
November, 1909							471	545	378	85	849	630
December, 1909							476	460	480	50	956	510
January, 1910							481	445	515	104	996	549
February, 1910							529	527	516	188	1,045	715
March, 1910							546	603	518	12	1,064	615
April, 1910							¶561	560	‖604	.	1,165	560
Totals	3623	3390	708	513	1324	1367	9754	9450	9738	1985	19,492	11,435

*, †, ‡, §, ¶, ‖, records broken.

PROGRESS AT NORTH PORTAL OF ELIZABETH TUNNEL, APRIL, 1910

Classification	Labor	Live Stock	Material and Supplies	Transfers		Freight and Handling	Total	Unit Costs
				Labor	Stock			Linear Foot
Month Cost and Progress								
(A) Engineer and superintendent	$ 150.00		0.14	$132.50	$ 27.00	$ 0.03	$ 309.67	$ 0.55
(B) Excavation	3,123.40	$37.80	1,798.17			359.63	5,319.00	9.48
(C) Mucking	4,572.97	32.40	110.73			22.14	4,738.24	8.44
(E) Drainage	55.00		25.95			5.19	86.14	0.15
(F) Ventilation	52.50						52.50	0.09
(G) Light and power	596.92		308.30	Energy	1380.51	61.66	2,347. 9	4.17
								22.88
(D) Timbering	2,766.88	27.90	812.22			162.44	3,769.44	6.73
Totals	$11,317.67	$98.10	$3,055.51	$132.50	$1407.51	$611.09	$16,622.38	$29.61
Total Cost and Progress								
(A) Engineer and superintendent	$12,296.00	$ 713.50	459.91			74.51	$13,543.92	$ 1.38
(B) Excavation	85.531.20	775.90	26,969 03			5,117.02	118,393.15	12.13
(C) Mucking	113,492.20	490.17	2,011.47			327.00	116,321.06	11.93
(D) Timbering	41,999.31	238.30	16,520.31			2,983.54	61,741.46	6.53
(E) Drainage	3,668.42		502.64			86.00	4,257.06	0.43
(F) Ventilation	1,000.77		1,284.75			194.89	2,480.41	0.25
(G) Light and power	14,128.23		33,003.25			1,756.94	48,888.42	5.01
(H) Concreting	1,105.75	343.80	30.71			3.84	1,484.10	
Backfill	539.21						539.21	
(I) Portal cut	1,677.61	84.30	24.06			2.64	1,788.61	
(J) North shaft	8,345.08		3,468.50			629.62	12,443.20	
(E) Equipment	11,750.36	32.40	69,978.24			11,831.13	93,592.13	
Totals	$295,534.36	$2,678.37	$154,252.87			$23,007.13	$475,472.73	$37.66

Powder 16.8 lb. per ft. Estimated cost $763,420. Cost for 9754 ft., $475,472.—E. J. Sharpe, Superintendent.

where the roof is heavy, and there is comparatively little water encountered.

In the north heading, the rock has been shattered by scores of fissures, and in general is so kaolinized by percolating waters that it is for the most part soft, friable and frequently pasty Considerable water enters the tunnel, and the ground is so heavy it requires timbering, except at one or two places in the tunnel of small extent where hard unaltered rock was penetrated. Long extents of heavy, treacherous ground are continually encountered, so that the timbering must be kept well up to the face.

As a result of these widely varying conditions neither the costs, nor the record performances of work at one heading may be fairly taken as a criterion of the work at the other heading of this same tunnel.

In spite of this diversity of work, however, the progress made at each heading was surprisingly uniform as shown in the accompanying table, until at the end of April, 1910, 30 months after starting work, the north heading had penetrated 9754 ft. and the south heading 9738 ft , totaling 19,492 ft., or nearly 72 per cent of the entire length.

To those interested in the rivalry between the crews of the north and south Elizabeth tunnel headings, the accompanying tables will be replete with interest as well as with valuable data. These give the official costs for both headings during the month of April, 1910, when the south portal advanced the American record to 604 ft., while the north portal progressed 561 ft. in spite of exceptionally heavy ground. It will be noted that, as might be expected, the cost of explosives was the heavier for the south heading, while the cost of timbering is mainly responsible for increasing the cost per foot of the north heading to $4.36 more than the $25.25 of the south heading. It is also significant that the cost of mucking in the two headings is very nearly the same.

The appended study covers unit costs of driving a timbered tunnel (North Portal, Tunnel No. 7) in the little Lake subdivision of the Grapevine division of the Los Angeles aqueduct.

Ninety feet were driven in 15 eight-hour shifts, the period covered by detailed cost keeping.

Progress Sheet of South Portal, Elizabeth Tunnel, April, 1910

Month Cost and Progress								Unit Costs
Classification	Labor	Live Stock	Material and Supplies	Transfer	Energy	Freight and Handling	Total	Linear Foot
(*A*) Engineer and superintendent	$ 70.00		$ 2.83	$89.50		$ 0.56	$ 162.89	$ 0.27
(*B*) Excavation	3,729.68		2,861.35			572.27	7,163.30	11.86
(*C*) Mucking	5,136.49	$ 81.00	48.58			9.71	5,275.78	8.73
(*E*) Drainage	44.49						44.49	0.07
(*F*) Ventilation	85.10						85.10	0.14
(*G*) Light and power	452.04		172.91		$1,450.02	34.58	2,109.55	3.49
	9,517.80		3,085.67			617.12	14,841.11	24.56
(*K*) Back trimming	416.50						416.50	0.69
Timbering	12.00		43.83			34.83	90.66	
Totals	$9,946.30	$ 81.00	$3,129.50	$89.50	$1,450.02	$651.95	$153,48.27	$25.25

Total Cost and Progress

					Freight			
(*A*) Engineer and superintendent	$ 9,600.96	$ 149.90	$ 327.84		$ 42.69		$10,121.39	$ 1.04
(*B*) Excavation	94,468.43		50,841.14		9,766.20		155,075.77	15.92
(*C*) Mucking	105,744.49	1421.30	3,345.07		589 65		111,100.51	11.41
(*D*) Timbering	6,017.48		4,209 87		1,085.93		11,313.28	5 70
(*E*) Drainage	1,393.03		13.03		1.56		1,407.62	0 14
(*F*) Ventilation	1,481.30		3,097.86		530.27		5,109 43	0.52
(*G*) Light and power	13,495.78		32,669.59		2,100.22		48,265 59	4.95
(*I*) Portal cut	720.49						720.49	
Totals	$232,921.96	$1571.20	$94,504.40		$14,116.52		$343,114.08	$39.68

Powder 19.15 lb. per ft. Feet to be driven, 13,430. Estimated cost, $750,570. Feet so far driven, 11,693. Actual cost so far, $343,114.—W. C. Aston, Superintendent.

South Portal Elizabeth Tunnel. Details of Annual Summary of Tunnel Reports for 1909

	Totals	Units	Total Cost	Unit Cost
Footage to date	7,585 ft.			
Footage during 1909	4,476 ft.			
Daily footage (365 days)	....	12.26		
Footage of untimbered section	3,274 ft.			
Actual cost of untimbered section	....	....	121,787.45	37.198
Footage of timbered section	1,202 ft.			
Actual cost of timbered section	....	....	51,941.11	43.20
Average cost of timbering per foot progress	....	....	....	1.16
Total bonus footage	1,563 ft.			
Total bonus pay roll	....	....	23,440.23	
Average cost of bonus per foot progress	....	....	....	.523
Actual expenditure	....	....	173,728.56	
Number of shifts worked	1,055			
Number of shifts lost	40			
Number of men days	21,452			
Men per foot progress	....	4.79		
Average progress per shift	....	4.24		
Number of holes drilled	21,066			
Total feet drilled	135,896			
Feet drilled per foot progress	....	30.36		
Total time drilling (in hours)	3,656.25			
Average time drilling per foot progress (in minutes)	....	49.01		
Average time drilling one hole 1 foot deep	....	1.61		
Pounds of powder used (including trimming)	143,659			
Average pounds per foot	....	32.09		
Number of cars mucked (32 cubic feet)	28,561			
Cars mucked per foot	....	6.40		
Leyner No. 9, drill repairs (3 machines per shift)	....	....	4,116.99	
Average cost of repairs per machine per foot	....	....	....	.30
Leyner No. 2 drill sharpener repairs (for 3300 feet)	....	....	387.13	.117
Energy used (kilowatt hours)	608,762	135.00		

South Portal Elizabeth Tunnel. Summary of Total Expenditure During 1909

	Totals	Unit Cost
A. Engineer and superintendent	$3,851.24	$0.86
B. Excavation	79,607.15	17.78
C. Mucking	55,556.33	12.41
E. Drainage	421.01	0.09
F. Ventilation	2,313.44	0.52
G. Light and power	24,750.71	5.53
Total cost untimbered	$166,499.88	$37.19
D. Timbering	7,228.68	6.01
Total cost of timbered tunnel	$173,728.56	$43.20

South Portal Elizabeth Tunnel. Details of Tunnel Reports, April, 1910

	Totals	Units	Total Cost	Unit Cost
Total footage to date	9,738 ft.			
Footage during April, 1910	604 ft.			
Daily footage (30 days)	....	20.13		
Footage of untimbered section	604 ft.			
Actual cost of untimbered section	....	......	15,257.61	25.25
Footage of timbered section	000 ft.			
Bonus footage	364 ft.			
Bonus pay roll	....	......	2,377.48	
Cost of bonus per foot	....	......	....	3.93
Energy used—kilowatt hours	78,379	129.76		
Cost of energy, at 1.85 c. per kilowatt hour	....	......	1,450.02	2.40
*Estimated expenditure	....	......	24,238.52	
Actual expenditure	....	......	15,257.61	
*Amount saved over estimated expenditure	....	......	8,980.91	
Number of underground men days:				
Foreman and heading crew	1,860	3.079		
Timber, pipe, track, car, and machine repair men	458	.758		
Mechanics, electrician and helpers	154	.255		

South Portal Elizabeth Tunnel. Details of Tunnel Reports, April, 1910—*Continued*

	Totals	Units	Total Cost	Unit Cost
Number of mule days	90	......	81.00	0.134
Number of shifts worked	90			
Average progress per shift	....	6.711		
Number of holes drilled	1,924			
Total feet drilled	16,079			
Feet drilled foot progress	....	26.62		
Total time drilling heading (hours)	324			
Average time drilling heading per foot (minutes)	.....	32.18		
Average time drilling one hole one foot (minutes)	......	1.209		
Pounds of powder used (including trimming)	16,100	26.65		
Number of cars mucked (32 cu. ft.) heading and ditch	3,216	5.324		
Drill repairs (three No. 9 Leyners)	....	......	248.77	
Average cost of repairs per foot (three machines)	....	....	....	0.411
Average cost of repairs per foot (one machine)	....	....	....	0.137
Drill sharpener repairs (Leyner No. 2)	....	.....	61.70	0.102
Drill steel broken	380	.629		
Drill steel sharpened	6,865	11.36		
Drill steel welded	445	.736		
Cars repaired	38	.062		
Cars repairs	....	......	48.52	0.8
Car equipment (changing from 12-inch to 14-inch wheels)	....	......	260.68	0.431

Note.—New American hard rock tunnel record established April, 1910, 604 ft.

Summary of General Expenditures During 1909

	Total Cost
Miscellaneous structures	$165.79
Tunel construction	15,348.27
Miscellaneous construction equipment	5,168.85
Miscellaneous camp equipment	19.22
M. & O. live stock	183.75
M. & O. local telephone lines	31.73
M. & O. water supply	2.34
Division administration	740.86
M. & O. roads and trails	116.05
Total expenditure	$21,776.86
Pay roll	9,419.76
Bonus roll	2,377.48
Material issues	6,915.83
Material receipts	2,841.54

Summary of Tunnel Expenditure During April, 1910

	Totals	Units
E. W. O. 33—A. Engineering and superintendent	$162.89	.27
B. Excavation	7,163.30	11.86
C. Mucking	5,275.78	8.73
E. Drainage	44.49	.07
F. Ventilation	85.10	.14
G. Light and power	2,109.55	3.49
	14,841.11	24.56
D. Timbering	90.66	
K. Back trimming	416.50	.69
Total expenditure	$15,348.27	25.25

The tunnel is approximately 10 × 10 ft. in section; 3½ cu.yd. in place per linear foot to pay line; overbreakage about 17 per cent, making a total of 6½ cu.yd. of broken material per foot of tunnel.

The heading was in 800 ft., lighted by electricity at 110 volts, ventilated by a No. 3 Champion blower through 12-in. pipe, the heading being cleared in 15 min. after shooting.

Drilling is done by one No. 7 Leyner drill, water being forced through hollow steel; drill uses approximately 66 cu.ft. free air per minute at 83 lb. pressure per square inch, drilling holes to 10 ft. in depth.

Mucking is accomplished by use of steel sheets laid down before shooting; No. 3 D-handle, square-point, shovels, and 32-cu.ft. rocker dump cars pulled by a 3½-ton locomotive, running on a 24-in. gauge single track laid with 25-lb. steel.

The rock is a close-grained, hard, gray granite with numerous seams, causing the drill to run from alignment, but breaks well. The seams and water combined make it necessary to timber all this ground. The ground carries enough water to make disagreeable mucking, and has to be pumped out.

Timbers are of 6 × 8 in. Oregon pine spaced 5 to 8 ft. apart, as ground permits, lagged with 2 × 6 in. plank. Sets of timbers consist of two vertical posts and a four-segment arch.

The crew consisted of 1 shift boss at $3.50 per day; 4 miners at $3.50; 5 muckers at $2.50, and 1 trammer at $2.50. The blacksmith doing repair work was paid $4 per day.

The four miners worked on day shift drilling the ground, timbering and shooting, the muckers following on night shift, resulting in a clean heading for the drill crews, and nothing interfering with the mucking crew.

In February, 1908, the Chamber of Mines and the Transvaal Government jointly offered a prize amounting to £7500 for the best drilling machine that could be produced for mine work on the Witwatersrand. There were 23 entries for this prize, 19 of which started in the competition.

In the elimination trials, which were carried out on the surface at the Transvaal University College and underground at the Ferreira Deep, Ltd., nine drills were eliminated and 10 entered for the competition of 300 shifts. These were the Holman 2⅛ in., Holman 2¾ in., Siskol, Climax Imperial, New Century 00, Konomax, Chersen, Waugh, Murphy, Westfalia.

In the surface elimination test the most rapid rate of drilling was accomplished by the Westfalia machines; namely, 4.996 in. per minute of actual drilling time. The largest air consumption recorded was that of one of the Konomax machines; namely, 125.9 cu.ft. of free air per foot drilled.

LABOR COSTS

Class of Work	Total Hours Labor	Total Labor Costs	Cost per Foot of Tunnel
Inside Labor:			
Squaring heading	23.50	$ 9.03	$0.100
Setting up and tearing down machine	36.00	16.59	0.184
Drilling, one No. 7 Leyner; including shift boss' time	55.33	43.21	0.480
Number of holes drilled 150			
Total footage of holes 1,202.30			
Feet drilled per hour, including lost time 15.84			
Feet drilled per hour, actual drilling time 21.74			
Average depth of holes, feet.. 8.00			
Cost per foot of hole, cents... 3.60			
Fastest hole 9 ft. 6 in. in 10 min			
Slowest hole 8 ft. 6 in. in 1 hour 18 min.			
Average hole 8 ft. in 22 min			
Blowing out holes	5.75	4.15	0.046
Loading and shooting	56.25	22.31	0.248
Mucking, 412 cars	835.00	268.44	2.980
Trimming, stulling, caves, etc	102.00	39.24	0.436
Timbering (cost per M. ft. = $11.32)	107.25	40.82	0.453
Lost time	40.75	15.66	0.174
Bonus, 30 c. per man per foot in excess of 2.3 ft. per shift		112.08	1.240
Repairs, to trolley, pump, etc	3.25	1.20	0.013
Totals, inside labor	1,265.08	572.73	6.354
Outside Labor:			
Sharpening steel, with Leyner No. 2 machine	44.00	17.83	0.198
Repairing drill	7.50	2.88	0.032
Framing timbers, at shop, per M. ft. .... = 2.42		8.75	0.097
Light and power	90.00	33.75	0.375
Totals, outside labor	141.50	63.21	0.702
Auxiliary Labor:			
Laying track, 90 ft	31.50	12.75	0.141
Drainage line, 90 ft	43.50	14.33	0.160
Ventilation line, 72 ft	11.00	3.44	0.048
Trolley line, 95 ft	18.00	6.35	0.067
Air line, 80 ft	2.50	0.80	0.010
Water line, 80 ft	2.50	0.79	0.010
Lights line, 90 ft	8.00	2.86	0.032
Totals, auxiliary labor	117.00	41.32	0.468
Local Administration and Engineering:			
Proportion of division engineer and assistant's time		50.40	0.560
Total labor costs	1,523.58	$727.66	$8.094

COST OF MATERIALS AND SUPPLIES

	Total Material Costs	Cost Per Foot of Tunnel
Construction Materials and Supplies:		
Drill repairs, 2 side rods, $1.40; 1 chuck, $15; 2 rings, $2.02; 1 oil can, .15; 1 belt, .16; 20 per cent. freight, $3.74	$ 22.47	$0.250
Cost per foot of hole = .018		
Drill supplies, machine oil, .58; drill steel, 45 ins., $4.12; 412 lb. blacksmith coal, $3.14; 20 per cent freight, $1.57	9.41	0.104
Cost per foot of hole = .008.		
Mucking supplies, car oil, .20; pick handle, .26; hammer handle, .15; 20 per cent freight, .12	.73	0.008
Power, machine, 2052 kw. h.; blower and lights, 355 kw. h.; locomotive, 1800 kw. h. = 4207 kw. hours, at 1.7 c.	71.52	0.795
Explosives, tamping stick, .40; 2700 ft. fuse, $11.54; 306 15 gr. Lion caps, $2.08; 650 lb. 1⅛ inch, 40-per-cent gelatin powder, $69.88; 250 lb. 1 in. 40-per-cent gelatin powder, $26.88; 150 lb. 1 in., 60-per-cent gelatin powder, $20.63; 20 per cent freight, $26.28	157.69	1 752
1050 lb. powder = 11.66 lb. per foot of tunnel.		
1050 lb. powder = 3.3 lb. per cubic yard in place.		
Explosive cost = $.050 per cubic yard in place.		
Explosive cost = $.27 per cubic yard broken.		
Timbers, 3597 feet B. M. lumber, $59.24; freight on same $55.01; 775 wedges, $5.43; 50 dowel pins, .42; nails, .73; freight, $1.33	122.16	1.360
Timber per foot of tunnel, B. M. = 40.		
Lighting, candles, $7.15; 14, 16, and 32 candle-power globes, $2.58; 20 per cent freight, $1.94	11.67	0.130
Totals for construction materials	395.65	4.399
Auxiliary Material:		
Trackage, 180 feet, 25 lb. rail, $15; splices and bolts, .83; spikes, .18; ties, $1.42; 20 per cent freight, $3.68	21.11	0.235
Drainage 90 ft., 250 ft., wire, $2.13; knobs, .07; 2-inch pipe, $4.36; 20 per cent freight, $1.31	7.87	0.087
Ventilation 72 ft., 12-inch pipe, $24.34; 20 per cent freight, $4.87	29.21	0.405
Trolley 95 ft., wire, $6.32; lumber, .37; fittings, $1.42; 20 per cent freight, $1.62	9.73	0.102
Air line 80 ft., pipe, $5.28; fittings, .76; 20 per cent freight, $1.21	7.25	0.090
Water line, 80 ft., pipe, $5.28; fittings, .76; 20 per cent freight, $1.21	7.25	0.090
Light line, 90 ft., wire, $1.48; fittings, .37; 20 per cent freight, 37	2.22	0.025
Total auxiliary materials	84.64	1.034
Total material costs	480.29	5.433
Live Stock:		
Mule 15 days at 90 c. $13.50		0.150
Total direct and auxiliary field charges $1,523.58 $727.61	$480.29	$13.667

Recapitulation. Costs Per Foot of Tunnel

Labor, direct charge		$ 7.056
Material and supplies, direct charge		4.399
Local administration and engineering		0.560
Stock service		0.150
		12.165
Labor on tracks, etc.	.468	
Material for tracks, etc.	1.034	
	1.502	
As this work will salvage at about 66 per cent, we deduct	.690	
Net charge for auxiliary work		0.812
Estimated proportion of charge for roads and trails on division		1.500
Estimated proportion of charge for buildings on division		0.200
Estimated proportion of charge for water supply on division		0.220
Estimated proportion of charge for machinery and tools		1.060
Total field charges		15.957
Add 3 per cent to cost for executive office administration		0.475
Total cost of tunnel ready for lining		$16.432

In the underground elimination air trials the quickest drilling speeds were attained by the Chersen and Holman 2¾-in. machines. The former drilled 1.81 in. per minute over drilling and changing time and 1.56 in. per minute over total time, which consisted of three 8-hr. shifts. The figures recorded for the Holman 2¾-in. machine on this trial were 1.94 and 1.47 in. per minute, respectively.

As the competition proceeded the following machines withdrew: Climax Imperial, Konomax, Murphy, Waugh, and Westfalia. It was the intention to run 300 shifts, but owing to lack of air pressure, 215 shifts were run. The stopes drilled in were 24 to 45 in. wide. The number of feet drilled by each pair of the four leading competing machines were as follows: Holman 2⅛-in., 12,779 ft.; Siskol, 14,083 ft.; Holman 2¾-in., 11,744 ft.; Chersen, 11,781 ft. This was drilled in 215 shifts of 8 hr. each.

The drilling speeds attained over total times by the four machines are as follows:

Holman, 2⅛ inches	.742 in. per min.
Siskol	.818 in. per min.
Holman, 2¾ inches	.682 in. per min.
Chersen	.684 in. per min.

At the surface elimination trials the average rate of drilling of each pair of machines was as follows:

Machine	Total Time Inches Per Minute	Actual Drilling Time Inches Per Minute
Holman 2⅛ in.	1.566	2.393
Siskol	2.058	4.337
Holman 2¾ in.	1.988	3.110
Chersen	2.515	4.110

The cost of drilling per foot with these four leading machines amounted to: Holman 2⅛-in., 9.77d.; Siskol, 9.90d.; Holman 2¾-in., 10.91d.; Chersen, 11.94d. These figures compare favorably with the cost of hand drilling on the Rand.

In the competition, two sets of machines, the Holman 2⅛-in. and the Siskol have cost approximately 8.3d. per foot drilled plus the cost of steel and drill sharpening, which would come to, at the most, 1.5d. per foot, making 9.8d. per foot. But it has taken 1.282 mine shifts to make one 8-hr. shift, therefore the wage cost must be increased in that ratio. Further, instead of assuming 10s. per shift for two machines as the white wage, 25s. per shift were taken for four machines. This alteration means an increase of ½d. per foot drilled, making the total cost per foot of these two sets of machines practically 11.8d. per foot. This is as near as one can get to the actual cost per foot for 28,528 ft. drilled by these machines. This cost of 11.8d. per foot compares favorably with the cost per foot by native labor, for which 1s. 1d. per foot would be an exceedingly low figure. The tonnage broken would be as 6 to 5 in favor of the machines. The cost of explosives would be as 6 to 5 in favor of native labor. The tendency is for native wages and cost to increase and this is the heavy item, 10d. per foot, in hammer work, but a much smaller item, only 3d. per foot in machinery. With a proper size of bit a hole 4 ft. 8 in.

or 5 ft. deep is much better as regards tonnage than the ordinary hand-drill hole, for it will carry more explosives and so a greater burden.

Name of Drill	Weight of Drill in Pounds	Inches Drilled Per Pound. Weight of Machine Over Drilling Period of 4 Hours		Depth of Holes Drilled				Percentage Increase in Depth Using 60 Pounds Air Pressure
		Air Pressure 50 Pounds Per Square Inch	Air Pressure 60 Pounds Per Square Inch	Air Pressure 50 Pounds Per Square Inch		Air Pressure 60 Pounds Per Square Inch		
				Ft.	In.	Ft.	In.	
Flottman.. .	52.250	4 33	5.83	18	10⅜	25	4⅝	34.4
Gordon.....	72.625	4 69	6.08	25	4⅝	36	9¼	29.5
Holman.....	97.500	1.77	2.09	14	4¼	17	0¼	18.6
Kimber.....	100.000	1.55	2.03	12	11⅜	16	11	30.6
Little Kid....	102.500	1.85	2.63	15	9⅜	22	6⅛	42.6
Chersen......	113.625	2.79	3.50	26	5¾	33	2	26.0
Little Wonder	118.500	1.68	2.02	16	7⅞	19	11⅜	19.7
Baby Ingersoll. .	129 500	2.34	2.74	25	3	29	6½	17.0

Drill steel.—There is perhaps no single item in rock drilling that will affect the cost of the work so quickly as the efficiency of the drill steels. The essential qualities of a drill steel as laid down in a paper presented at the February, 1921, meeting of the Am. Inst. of Min. and Met. Engr's., are: First, it must be easily forged; second, the forged bit end must be such that it can be easily heat treated to obtain hardness to resist chipping; third, the bar or body must be stiff to resist bending or twisting and yet tough to resist shock and vibration, with resulting breakage, and fourth, the forged shank end must be such that it can be easily heat treated to obtain some hardness with great toughness.

Drill steel must be properly forged either by hand or machine, and this operation requires pyrometric control. Two causes of trouble can be eliminated at this point, as there is no doubt that drill steels as a rule are forged at too high temperatures, and the forging operation is continued after the temperature has dropped below the critical. Furthermore, little if any annealing is done on drill steel after forging; hence the forging operation must be conducted with great care.

Assuming the forging temperature is correct, the other minor requirements are that the bit and the shank be in alignment with the body; that the shank shall be of the proper shape and length and the shank collar or lugs be of the proper diameter and length; that the hole, if any, be free from obstruction; that the striking end of the shank be flat and square; that the bit be of the proper shape, with the cutting and reaming edges formed full and to the required size; that the reaming edges are concentric with the axis of the steel, and that there are no sharp corners at the shoulder where the bit blends into the body.

It is obvious that the above requirements can best be obtained day in and day out by means of a drill steel sharpening machine. The efficiency of such a machine will accordingly depend, first, on the initial forging temperature required, for the lower the initial forging temperature the better the steel structure; second, on the accuracy of the forged bit and shank; third, on the speed of operation; fourth, on restriction of the hole in the shank and bit when using hollow drill steel; fifth, on the number of heats or times required to heat the steel before securing the finished bit or shank, and sixth, on the air consumption of power required. The means of heating for forging will be considered later.

The drill steel must have an ideal bit and shank. The essential qualities of such a bit, regardless of the conditions under which it is operated, are that its shape be such that maximum cutting speed can be maintained for as great a distance as possible before wear of the gauge and cutting edge reduces the speed of penetration to a point where a change of steel is made necessary, and that the size or diameter of the drill hole corresponding to the gauge of the bit can be maintained with the least possible reduction as the depth of the hole increases, and also that the shape of the bit is such that it can be correctly and readily formed and heat treated.

The following features of bit design require attention: Shape, total length, and angle of cutting edge; length and area of the reaming edges or surfaces; size and shape of clearance grooves for ejection of cutting, and lengths and angle of the wings and the manner in which they are blended into the body of the steel.

The combined length of the cutting edge and the manner in which it is applied is a big factor in the drilling operation regarding the speed and the life of the drill steel and the drill. The longer the cutting edge the greater the amount of rock cuttings per blow, assuming that the cutting edge is and remains sharp or sharp enough for the conditions. Furthermore, the drilled or blunt edge, besides decreasing the penetration per blow, lessens the cushioning effect, and this causes the drill steel to rebound from the rock, which may cause breakage of the drill steel or parts of the drill. In a radial cutting-edge bit the work done is greatest at the extreme cutting and reaming edges, which accounts for the unequal wear along the cutting edge. Therefore it is apparent that the only way to improve this condition is to so shape the cutting edge that the work is evenly distributed throughout its length, and so that the extremities of the cutting edge have suitable reaming surfaces properly tapered back to improve the wearing qualities of the gauge.

A good deal has been written about the angle and shape of the cutting edge, and many kinds of bits have been brought out, such as the bull bit, rose bit, double cross bit, chisel and double chisel bit, and other types; but the bit that approaches the ideal design is the double-arc, double-taper bit. The accompanying illustration, Fig. 2 shows the characteristic dimensions of this bit. As to the ideal shank, there is no doubt that the so-called shankless steel approaches the ideal, and it is regrettable that this type is not suitable for all conditions. Its advantages and disadvantages are evident.

The third and last requirement of the drill steel which has ideal qualities is that it must be properly heat treated. This without doubt is the most essential operation in securing best results, and yet it is safe to say it receives the least attention.

The good results which are expected from all previous operations can be obtained only through proper heat treatment. Theoretically, this operation of heat treating should be simple; practically it is not, for all too much depends on the equipment.

It is interesting to note that answers to a recent questionnaire brought out the following facts and are representative of all fields and conditions of mining.

To the question, what type of bit was giving the best service, returns showed that the double-arc, double-taper bit and cross bit were far in the lead.

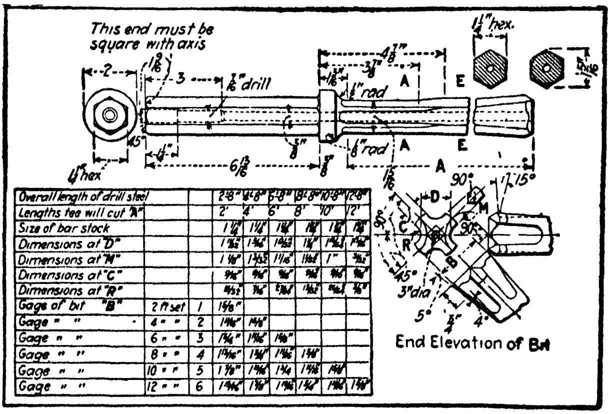


FIG. 2.—Standard design for double-arc, double-taper bit.

To the question as to what necessitates the resharpening of the drill steels most frequently, wear of gage was unanimously first choice; chipping of bit was unanimously, with one exception, second choice; breakage of shank was unanimously, with one exception, third choice; upsetting of bit, upsetting of shank and breakage of body were about on a par for fourth choice.

To the question as to what was the direct cause of the necessity of resharpening, poor heat treatment was unanimously first choice, and faulty steel and severe rock conditions were about on a par for second choice.

Explosives.—The selection of explosives for tunnel blasting, probably requires a more careful study of conditions than for any other kind of excavating. Maximum speed and economy in driving cannot be attained unless the explosive best adapted to the work is used. When starting a tunnel or drift, it is a good plan to thoroughly try out several explosives, which are distinctly different in action, before finally adopting any one of them. The results, however, from this preliminary trial will be of little or no value, unless each different explosive is used under exactly the same conditions. Care must be taken to see that no change occurs in the character of the rock, number and direction of the bore holes, strength of the detonator, kind and quantity of tamping, amount of water encountered, method of connecting up the bore holes for firing, and that the explosive is always thoroughly thawed. If a material change in any of these conditions occurs as the work progresses, further tests should be made to determine whether a quicker or slower, a stronger or weaker, explosive might not break the ground, or bottom the bore holes better, or make it possible to bring out the cut with fewer holes or deeper ones. The speed at which rock can be drilled does not indicate how it will break, and not infrequently that which can be easily drilled is very difficult to blast.

High explosives suitable for tunnel blasting should not give off objectional fumes on detonation, and accordingly gelatin dynamite, blasting gelatin, or ammonia dynamite should always be selected.

Gelatin dynamite is made in various grades of strength, from 25 to 80 per cent, inclusive. It is comparatively slow in action, the higher grades being little, if any, quicker than the lower ones.

Blasting gelatin is manufactured in only one strength, which for comparative purposes may be said to be 100 per cent. It is more powerful and quicker acting than any other blasting explosive. It should be used sparingly, therefore, until the maximum safe charge has been learned from experience. Good results will often be had in hard ground, if a few cartridges of blasting gelatin are used in the point of the bore hole, with gelatin dynamite on top. When this is done, it is best to put detonator in one of the cartridges of blasting gelatin.

Ammonia dynamite is made from 25 per cent to 75 per cent strength. All grades are quicker than gelatin dynamites, and generally speaking the quickness increases with the strength. That is, the stronger grades are quicker, and the lower grades slower, in action.

The various grades of these three high explosives, offer a wide range in strength and quickness to select from, and it is always possible after a few trials to find an explosive exactly suited to the conditions.

Railroad tunnels, mine tunnels and drifts, highway tunnels, and irrigation tunnels, are being driven daily through various kinds of "ground." Often it is a matter of first importance to finish them quickly, and consequently details in regard to methods and equipment are matters of general interest.

In *Engineering Contracting* of October 20, 1909, Mr. J. B. Lippincott, assistant chief engineer of the Los Angeles aqueduct, gave an interesting account of the driving of the Red Rock tunnel of the Los Angeles aqueduct system. In August, 1909, this tunnel, which is 9-ft. 10 in. $\times$ 10 ft. $8\frac{1}{2}$ in. in section, was advanced 1061.6 ft. Mr. Lippincott states that the explosives used were Du Pont 40-per-cent ammonia dynamite and blasting powder.

In the *Engineering News* of November 18, 1909, the Red Rock tunnel is again referred to, and details are also given by Mr. C. H. Richards, division engineer, in regard to a tunnel on the Little Lake Division of the Los Angeles acqueduct. The explosives used in this tunnel were Hercules 40-per-cent and 60-per-cent gelatin dynamite, the average weight of explosives per cubic yard of rock, place measurement, having been only 3.3 lb., or about 35 lb. per linear yard of tunnel almost 10 ft. $\times$ 10 ft. in section.

A short time before, accounts were given in several engineering magazines, of a record driving speed made in the Roosevelt drainage tunnel, at Cripple Creek, Col. The explosives used in this tunnel were 40-, 50-, and 60-per-cent Repauno gelatin dynamite and Du Pont blasting gelatin.

A very interesting description of the Rondout pressure tunnel of the Catskill aqueduct, written by John P. Hogan, assistant engineer of the New York City Board of Water Supply, was published in the January 1, 1910, number of the *Engineering Record.* Very rapid progress was made in this tunnel, and also in the Moodna pressure tunnel of the same system, described in an article in the *Engineering Record* of June 4, 1910. The explosive which gave best results, and which was used exclusively in both of these tunnels was 60-per-cent Forcite—a gelatin dynamite.

Reference to a paper by B. H. M. Hewett and W. L. Brown on the land section of the Pennsylvania Railroad North River tunnels, published in Vol. XXXVI of the Proceedings of the American Society of Civil Engineers, and reprinted in part in *Engineering Contracting* of May 11, 1910, shows that 40-per-cent. Forcite was used in blasting on the Manhattan section, and 60-per-cent Forcite on the Weehawken section.

The records of many other tunnels recently constructed, further illustrate how many kinds and strengths of explosives are used for blasting under the different conditions encountered in one class of work.

The specific cases referred to above, were all connected with large and important contracts, where equipment and methods were at the best, and several of these tunnels were driven at record speed. The fact that so many different explosives were used in the several tunnels, goes to show that care was taken to use the explosive which was best adapted to the conditions, and it is not unlikely that the speed of driving these tunnels, was largely due to the attention given to the selection of the explosives.

VENTILATING COSTS

There are three forms of efficiency that must be considered in all fans, and the value of the fan depends largely upon its ability to give that efficiency, which is most valuable for the particular mine on which the fan is to operate.

The mechanical efficiency is to be considered first. Mechanical efficiency is the relation the useful work performed by the fan, bears to the power required to propel the fan. The manometric efficiency is of second importance Manometric efficiency is the relation which the depression caused by the fan, is to the theoretic depression which the fan would make if it were a perfect machine, and working against a closed airway. The volumetric efficiency is the third to be considered.

This is the relation which the volume of air discharged by the fan, bears to the volume or cubic contents of the fan, taken the number of times of rotation during the given interval of measurement.

A fan may be high in one of these efficiencies and low in the others For instance, a fan may produce a large volume of air at a very low pressure, and be excellent in volumetric efficiency, and yet be low in manometric efficiency and mechanical efficiency. In fact, fans high in mechanical efficiency are usually high in manometric efficiency, and almost all fans high in mechanical efficiency, when working under favorable conditions, are low in volumetric efficiency.

A fan may be high in volumetric efficiency and low in mechanical efficiency, and also low in manometric efficiency; these conditions all depend on the construction of the fan, and the conditions under which it works. The mechanical efficiency of the fan may be determined if the volume of air passing through the mine is known, and the pressure (water-gage) reading is taken at the same time the air reading is taken, together with the indicated power of the motor (either steam engine or electric motor) when resolved into foot-pounds.

The manometric efficiency may be determined if the tangential speed of the fan is known, and the actual pressure as read by the water-gage. The volumetric efficiency may be determined if the rotations of the fan are known, and the volume of air discharged is known for the same interval. A fan may have high manometric efficiency, and not be useful for mine ventilation; for instance, a cupola fan can produce the required pressure for a mine and not give any volume worth consideration. A fan may have a large volumetric efficiency under favorable conditions, and yet under unfavorable conditions be unable to produce pressure, and hence be a poor fan for mine ventilation.

Testing a fan is a proceeding requiring a considerable degree of skill and care. It is customary in making an anemometer test to organize the force so that each main airway shall have two men allotted, one to take the anemometer readings, and one to hold the watch and light and call the time. The time of readings should be one minute, two minutes, three minutes, or any number of minutes that may be agreed upon.

It is difficult, however, to hold the anemometer in a swift air current longer than three minutes, and for this reason this is usually the time limit. The readings should be taken about 100 ft. away from the fan in a drift, or about 100 ft. from the bottom of the shaft in a shaft mine, and should be taken at a place where the section of the airway is nearly uniform and as smooth as possible, and not close to any turns, where the air may be deflected into eddies.

It is well to adopt a schedule of readings at intervals of fifteen or twenty minutes so that the velocity of air, the water-gage, the speed of motor, and temperature and barometer may be taken at exactly the same time as the motive power applied is indicated. With large fans it is customary to use steam engines, and the indicator is used to ascertain the motive power applied. In the usual tests at coal mines, the barometer and temperature are neglected, as not having sufficient bearing to warrant the calculations necessary to apply them to the test. It is essential that a certain time be set for each and every reading, that they may be all taken simultaneously, and at least three readings should be taken at varying speeds of the fan. It is preferable to test under such speeds as may be employed under the conditions which the mine may require. To get proper readings of the water-gage, it should be so placed that it will record the pressure of the air immediately in front of the fan, and in the full current where there is no possibility of an eddy in the current.

To obtain the proper position it is necessary to pipe from the water-gage in the engine room to a place in front of the fan where the air current is in one body. The end of this pipe should be from 10 to 25 ft. from the periphery of the fan and should be bent so the air current will flow directly into it. If it is hung down a shaft, the pipe should be curved returning upward, and have a small hole in the lower side of the curve to drain the moisture that would collect and form a water seal, if there were no drainage. This water seal will destroy the true reading if the pipe is not drained.

The indicator should be operated by a competent engineer who understands the theory of steam consumption of a steam engine as well as the mere knowledge of the operating of a steam-engine indicator, as there may be defects in the valve gear that the indicator will show to the practiced eye, which might be overlooked by the novice. It should be borne in mind that the engine or motor is having its efficiency tested along with the fan, and any loss of efficiency in the engine or motor will reflect on the fan. In taking the readings it is well to begin with the even hour, and allow five minutes to each party detailed to take the various readings to measure the air velocity, the water-gage and to take the indicator diagrams. The fan should be maintained at a constant speed during this interval. At the end of the first five minutes, the fan should be accelerated to a desired speed during the succeeding ten minutes, and beginning with the even quarter hour, be maintained at a constant speed during the succeeding five minutes, so the various readings may be taken at the desired speed of the fan. At the end of this five minutes, the fan may be accelerated again, and other readings made at intervals of one quarter hour, and as long as may be desired.

The following table gives a test taken by the Westmoreland Coal Company, of Irwin, Penn., which is a fair sample of the manner of taking the various readings on a 20-ft. centrifugal fan driven by a steam engine:

FAN TEST AT WESTMORELAND SHAFT, IRWIN, PA.

Time P M	Rev per Min	Split No 1		Split No 2		Split No 3		Volume	W G	H P in Air	H P Ind	Mech Eff
		Area	Vel	Area	Vel	Area	Vel					
4 00	100	50	1428	58	1593	44	500	195,134	2 40	73 8	114	65
4 : 15	132	50	2245	58	2200	44	740	286,791	4 20	190	265	71 7
4 . 30	150	50	2853	58	2383	44	822	335,914	5 10	272	344	79 0
4 · 45	150	50	2800	58	·2463	44	833	337,171	5 20	276	360	77 7

The table shows the general method of a test for practical purposes, where boiler test is omitted, and also the variation of the barometer. For all practical purposes, this test is sufficient, and shows the mine engineer where he can practice economy on his ventilation, through the mechanical efficiency of his fan.

Power required.—The power required to drive air through a given mine increases as the cube of the volume, provided no change is made in the air courses: Let V represent volume of air, in cubic feet of air per minute; let P represent pressure of air as represented by inches in water-gage; Let W represent pressure in foot-pound performed on the air; then $W = V \times P \times 5.2$, as 5.21 lb. is the pressure of air per sq.ft. as represented by 1 in. of water gage. When reduced to horsepower, the formula then becomes

$$h.p. = \frac{V \times P \times 5.2}{33{,}000}.$$

To illustrate this formula, suppose a mine passing 100,000 cu.ft. of air per min. against a 2.3-in. water-gage. The work performed in passing the quantity of air at the given pressure, will be in terms of

$$hp. = \frac{100{,}000 \times 2.3 \times 5.2}{33{,}000},$$

which is equal to 36.24 hp. Now the pressure of air increases as the square of the volume, so that if we desire to increase the volume of air from 100,000 cu.ft. of air per min. in the above instance to 200,000 cu.ft. of air per min., we will require eight times as much power as to produce the 100,000 cu.ft. of air per min., as in the first case, for having doubled the velocity, the water-gage will have increased from 2.3 in. to four times 2.3, or 9.2 in. Now substituting in our second formula, we have

$$hp. = \frac{200{,}000 \times 9.2 \times 5.2}{33{,}000},$$

which equals 289.92 hp., which is eight times 36.24 hp., the amount of power required in the first example. It will be seen that to get the increased 100,000 cu.ft. of air, it will require 252.68 hp., whereas in the first example only 36.24 hp. was required, so that the second 100,000 cu.ft. of air required seven times as much power to pass it through the mine as the first 100,000 cu.ft.

When we consider that the installation of a ventilator, capable of producing 289 hp. in the air, would cost eight times as much as a ventilator producing 36 hp. in the air, we can understand why the management hesitates to purchase such expensive apparatus. Furthermore, if a horse-power in the most favored coal regions at the mine costs about $50 per year, then to increase the ventilation as above cited from 100,000 cu.ft. of air per min. to 200,000 cu.ft. of air per min., would cost for power alone, about $12,634 per year. This annual cost at many mines would be a serious consideration. What, then, would be required to get the necessary amount of air without making such an outlay of money in running expense? In most mines the quantity of air could be doubled without increasing the power eight times, by cleaning up, enlarging and increasing the number of air courses, thereby reducing the velocity of air and consequently the pressure. The pressure is the great absorber of power, and it is well to bear in mind that for a given amount of air, a certain pressure will be necessary to propel the air through the mine, and regardless of what form of ventilator is used, this pressure will be the same.

It is a common but mistaken belief among mining men, that one favored form of ventilator will propel the current through a mine at a less pressure than another ventilator. This fallacy should not be entertained by any mine manager.

Specification of fans.—The engineer is sometimes called upon to prepare specifications for a fan to be used at a particular mine and to obtain bids for the erection of same. Certain requirements will be laid down, say that the fan will furnish 250,000 cu.ft. per minute against a 6-in. water gage. A manufacturer in presenting his bid will sometimes state that he can do better than what the specifications ask, often claiming that the dimensions given are too great and the fan unnecessarily large. Where the manufacturer is left to make his own estimate of the size of fan required, he will often make no inquiries in reference to the size of the mine airways or the present circulation in the mine; but will take chances in reference to these important data. In nine cases out of ten the new fan is installed at a new mine, where, owing to the short airways of ample size, it continues to do good work for a few years. In the later development of the mine, however, there is experienced a scarcity of air, and, referring to the guaranteed capacity of the fan, the superintendent orders the mine foreman to speed it up, claiming that it is not run at its full capacity. The results are still far from satisfactory, but the fault is not with the fan, which is now forced to operate under conditions for which it was not designed.

A short familiar calculation of the pressure required to pass a given quantity of air through an airway of given dimensions, using the Fairley coefficient ($k = 0.00000001$), shows that a water gage of 1.9 in. is required to pass 100,000 cu.ft. of air per minute through an airway 7×10 ft. in section, for each 1000 ft. in length. Suppose the main airway in the mine is 3000 ft. long to the first point of split. The distance the main air current must traverse, including the return, is then 6000 ft., and the water gage required, for the passage of the main airways only, is $6 \times 1.9 = 11.4$ in. It is clear at once to any mining engineer or foreman that it would be impracticable to attempt to pass 100,000 cu.ft. of air through a single airway of this size for a distance of 6000 ft., including the return. By providing two air-ways of this size for the intake and two airways for the return, both the water gage and the power on the air would be reduced to one-fourth of the previous amount.

By this means, a main air current of 100,000 cu.ft. per min. can be conducted a distance of 3000 ft. to the first point of split and returned to the upcast, under a water gage of 2.85 in., requiring 45 hp. This is the horsepower on the air consumed in the main airway. These are the more important calculations in estimating the requirements of the mine in respect to ventilation. To these amounts, however, must be added the water gage and power consumed in the splits.

Finally, to determine the power of the engine driving the ventilating fan, it is customary to assume an efficiency of 60 per cent. For example, if the entire horsepower of the air, for the main airways and splits, is, say 60 hp., the required horsepower of the engine would be $60 \div 0.60 = 100$ hp. A similar calculation shows that a water gage of 5.08 in. and 160 hp. on the air are required to pass 200,000 cu.ft. per min. in three airways, a distance of 6000 ft., but, using four airways, the same air volume, circulated the same distance, will only require a water gage of 2.85 in. and 90 hp. on the air.

Assuming a consumption of 5 lb. of coal per hp.-hr., the saving of $160 - 90 = 70$ hp., by employing four instead of three main airways, would correspond to a saving in fuel of

$5 \times 70 = 350$ lb. of coal per hour, or 1¾ tons in 10 hr., which figures are a fair approximation to fact, in actual practice. Assuming, then a saving of 3 tons of coal per day of 24 hr. (less fuel being consumed during the night, when the mine is not working), at a cost of $1 per ton for coal at the mines, the total saving per month would be practically $3 \times 30 = 90$ tons or $90 per month, at the least computation.

Experience suggests that economical ventilation requires that the water gage be kept down to at least 1 in. per 100,000 cu.ft. of air in circulation. As has often been suggested before, this can be done by properly splitting the air current in the mines. Experience indicates further, that for the circulation of from 80,000 to 100,000 cu.ft. of air per minute, two main intakes and two return airways should be provided; or three main airways for a circulation up to 150,000 cu.ft. per min.; and four main airways for a circulation up to 200,000 cu.ft. per min., the sectional area being 70 sq.ft. If this plan is followed, a great saving in coal will be found to result. It is important, in overcasting the air, to see that all air bridges on main airways are substantially built so as to prevent the leakage of air. Such overcasts and all stoppings on main airways should be built of concrete, for the same reason.

It has been the policy of the Consolidation Coal Co., when opening a new mine, to make the projections on the maps of that mine, with enough intakes, returns and overcasts, to ventilate the mine under a low water gage. Where old mines were contending with a high water gage, air shafts were sunk, in order to shorten the airways and lower the water gage. At Mine No. 26, an air shaft was sunk 11,700 ft. from the mouth of the mine, and a new 6×20-ft. fan and a boiler plant installed. By installing the fan at this last location, it shortened the airway one-half, besides permitting the two return airways to be used as outlets or intakes, as required, thus providing four inlets in place of two, as before. Previous to the change, this mine had but two inlets and two returns. By moving the fan, the water gage was reduced from 3 in. for 100,000 cu.ft., to 1.36 in. for the same air volume, a splendid demonstration of the practical value of shortening airways. It will be noticed that 1.36 in. water gage is too high for a circulation of 100,000 cu.ft. of air, as, in case 200,000 cu.ft. were required the necessary gage would be 5.44 in., which is far too high for everyday practice. It may be possible that this is the best that can be done with an old mine sometimes.

In estimating the saving of coal effected by moving this fan, the calculation is based on the difference in actual total horsepower required of each engine, which is 123 hp. for 150,000 cu.ft. of air. Five pounds of coal per hp. per hr. then means 615 lb. of coal per hr., or 14,760 lb., or, say 7.4 net tons per day of 24 hr. Coal at $1 per ton then means $7.40 per day. or $222 per month and $2664 per year. The water gage on the new fan will be 3.06 in. for 150,000 cu.ft. of air; while on the old fan, it would require a 6.75 in. water gage for the same air volume.

The Consolidation Coal Co. has adopted a different method for the ventilation of its new coal field in the Elkhorn Division, Kentucky. The mines are all projected on the map of the coal field, showing the mines about as they will be, and giving the amount of coal they expect each mine to produce and the amount of air they expect to put into each mine. Then the proper number of airways are projected on the map, together with all overcasts, so as to handle this amount of air with a given water gage. The large mines from which it is intended to produce not less than 2000 tons of coal per day, are expected to require about 200,000 cu.ft. of air per minute when the mine becomes fully developed.

In these large mines, there will be four return airways and four intakes, each of which will average 6½ ft. high by 10 ft. wide, making an area of 65 sq.ft. each. From the fan to the first split is generally about 1200 ft., and here two double-face headings are turned off—two to the right at about 1200 ft. and two to the left at about 1320 ft.—so the switches will not interfere, as the face headings are turned with a 112-ft. radius curve. Each of these two face headings will have about four room headings, making eight room headings for each 25,000 cu.ft. of air. The four room headings will be ventilated with one split of air, say about 12,500 cu.ft. to each four room headings, which will be about 1000 ft. long. Face headings will be turned every 2100 ft., center to center, and room headings are then to be turned both ways from the face heading.

The water gage that will be required in passing 200,000 cu.ft. of air to the first, second, third and fourth splits, there being four return airways and four intake airways, is estimated as follows: The distance to the first split of air is, say 1260 ft., an average between the two face headings where 50,000 cu.ft. of air goes to the four face headings. Using Fairley's coefficient of friction ($k = 0.00000001$), the water gage for 200,000 cu.ft. of air up to this first split will be

$$w.g = \frac{0.00000001 \times 1260 \times 2(6.5+10) \times 200{,}000^2}{5.2\ (6.5 \times 10)^3} = 0.72\ in.$$

To the second split is a distance of 2100 ft., the air volume being 150,000 cu.ft., the water gage for this section is found in the same manner, and is 0.68 in. Likewise, to pass 100,000 cu. ft. of air to the third split, a distance of 2100 ft. again, will require a water gage of 0.30 in., and finally, to pass the remaining 50,000 cu. ft. the same distance to the last face headings will require a gage of 0.075 in. The sum of these four gages must then be doubled to provide for the return, in each case, which gives for the total water gage absorbed in the main headings $1.775 \times 2 = 3.55$ in.

It is well to estimate on a loss of at least 10 per cent of the air, which means a loss in water gage of 19 per cent, which taken from 3.55 in. leaves only 2.87 in. water gage for 200,000 cu.ft. of air on the whole mine, to this point. The splits in the face headings show a water gage of only 0.075 in. for 2100 ft. with 12,500 cu.ft. of air.

All regulators on face headings should be placed near the main airway and must not be set beyond the first room heading, under any consideration. In reference to the percentage of water gage spoken of as being lost (19 per cent), it has been found in practice this loss reaches as high as 25 per cent, so that it is safe to call it 19 per cent, as 10 per cent loss in air means 19 per cent loss in gage. This is not all due to loss of air by leakage, but from air passing crosscuts and other wide places, which reduce the velocity.

Most of the old style paddle-wheel and screw-propeller fans are working with less than 20 per cent mechanical efficiency, while the latest high-grade speed fans are working above 60 per cent mechanical efficiency, and in the great majority of cases, are working with a mechanical efficiency of between 70 and 80 per cent. Assuming that the old-style fan is working at 20 per cent mechanical efficiency, and the new high-speed fan at 60 per cent efficiency, also that 100,000 cu.ft. of air per min. at a 2-in. water-gage is required, the horse-power in air

$$\frac{100{,}000\times 2\times 5.2}{33{,}000}=31.51\ hp.$$

Now 31.51 hp is 20 per cent of 157 hp.; 31.51 hp. is 60 per cent of 52.51 hp It will be seen that 157.55 hp. will be required to drive the old-style fan to produce the required ventilation, while only 52.51 hp will be required to drive the new high-speed fan. Thus the new high-speed fan will save the difference between 157.55 hp. and 52.51 hp., or 105.04 hp. Now, as one horse-power is produced at a cost of $50 per year, it follows that the saving effected with the new high-speed fan is 105.04 × 50 = $5252 per year. This amount of money will install a fan to produce the required 100,000 cu.ft. of air per min. at a 2-in. water-gage, under the worst conditions likely to be found, and would therefore return to the owner the entire value of the fan each year.

Change in volume of air required.—Ventilating conditions at all mines change from week to week. As the workings extend underground, the amount of air required is increased, the resistance runs up, the number of stoppings multiply, the current of air may be lengthened or shortened. All of these things take place, resulting in a change in the amount of work required of the fan. If, by making repairs, extensions and changes, costing, for example, $100, the fan is relieved of 10 per cent of its work, we have something to show on both sides of the account, and can decide whether or not the money has been well spent, if we know the amount of power saved and its cost per unit.

There are few cases where the cost of power per horse-power-hour is known in installations where a steam engine is used for driving the fan. The practice usually is to operate at a certain number of revolutions per minute, this speed being maintained by opening and closing the throttle valve. This critical speed is governed by the needs of the mine. If there is "bad air" on "Third Right," the foreman may "speed 'er up" a few revolutions. Or, again, the drivers are "not able to keep a light," in another part of the mine. In this case the fan may be "slowed down a little."

Under such conditions, and they exist more generally than one would imagine, the cost of power used cannot always be figured accurately. In many instances it is not known at all. The mine, however, may be amply ventilated at some one of the different speeds at which the fan is run. This particular speed, whatever it is, is the proper one at which the fan should be driven during the time when the underground operations are in full activity.

Where a mine is equipped with a fan of ample capacity, and has airways as well as underground structures in fair condition, a certain number of revolutions of the fan will furnish a sufficient amount of air to comply with the mining law and to fully ventilate the mine when it is producing its maximum tonnage. Such ventilation will require the maximum horsepower applied to the shaft of the fan.

During the hours when the mine is not producing coal, say at night, on Sundays and on holidays, only a fraction of this ventilation is needed. In mines where no inflammable gas is found, tests have shown that at night and on idle days, about half the full ventilating current is sufficient for all needs and all the demands for safety.

Under conditions such as we have outlined, a steam-driven fan can be slowed down to, say, half the speed used in the daytime, and the volume of air will be reduced in almost the same proportion. The steam, however, will vary in pressure, and this, too, will further vary the speed of the fan as well as the volume of air passing.

In nongaseous mines and others where only half the normal ventilation is required when the mine is idle, it is an inexcusable waste to use more. The responsible officials at the mine should decide what amount of air is necessary when the mine is not working, making due allowance for any men who may be required in the mine at such times and this should be rigidly adhered to.

If a speed-regulating device were supplied as in the case of a steam engine, for example, the control of the power consumption, and therefore to a certain degree the cost of ventilation, would be in the hands of a comparatively irresponsible attendant who cannot be depended on to do one thing the same way every time. A motor capable of running at full speed, whatever that may be, and at half speed fills the requirements of fan drive fully.

Once it is installed it assumes all responsibility as to the speed of the ventilating apparatus It operates at either one or the other of its two possible speeds, and the fan, as a result, is either handling its full prescribed capacity or approximately half of it.

Occasions arise which demand that the maximum rotation of the fan be changed, either increased or decreased. Such changes can be well and cheaply made by substituting a different diameter of pulley on the motor shaft, when a belted installation is being considered.

The power saved by making the high-speed rotation exactly what is needed to fully ventilate the mine, and no more, when underground conditions are reasonably good, soon amounts to enough to pay for an extra or different driving pulley, or even several of them.

Money can frequently be saved and better mine ventilation assured by the use of canvas piping. Wherever blind entries, such as water courses, air courses, tunnels and shafts are contemplated or wherever gases will not clear, this system can be used to an advantage. Every mine should carry at least one of these outfits on hand all the time.

Briefly, the system consists of a flexible, pliable, treated canvas tubing and a fan directly mounted on the armature shaft of a motor. Protection for the tubing is afforded by suspending it from a wire attached to pegs driven in the roof at 15-ft. intervals, or to a wire fastened to upright posts. Additional sections are coupled within 15 seconds with a special coupling furnished as an integral part of each section.

The outfit provides a compact, portable, practical, "foolproof" blowing unit, to be used as an auxiliary to the large mine fan. In nearly all mines, places occur where the air has been short-circuited to such an extent it does not provide sufficient ventilation to carry off gases and smoke in blind headings. By installing a booster outfit at this point, the air can be carried for distances up to 1000 ft. in blind entries in sufficient volume to work up to eight men. This added circulation of air strengthens the main air system to permit work of any class without the slightest handicap or loss of time.

It is often possible to drive headings 25 per cent faster by eliminating cross-cuts, and save $50 to $100 through the elimination of each stopping. These units are capable of giving volumes of air varying from 1900 to 1200 cu.ft. per minute for distances from 100 to 1000 ft. and are especially designed for driving single entries in coal mines.

Spiral riveted galvanized pipe is also often used in ventilating tunnels and gangways in which there are no return airways. A system in general use is to place an electric booster fan to drive air through an 18-in. pipe to the face of the tunnel or gangway, and allow the air to return out the tunnel or gangway, until a hole is driven to the surface or the upper level, when the fan is moved nearer the face and pipe used again to advance the tunnel or gangway. Gangways and tunnels four miles long have been driven by this method; they are usually ventilated a distance of 2000 ft., when fan must be advanced.

Mine lighting.—To measure accurately the illuminating power of a lamp, we must consider not only the intensity of light (or candlepower), but also the solid angle over which the intensity is maintained. A lamp which gives an intensity of light of one candlepower all around gives twice as much light as one which gives a light of equal intensity half way round it. The term "flux" is used by illuminating engineers to designate the product of intensity and the angle over which it is exhibited, since this product most represents the light which flows from the lamp. The unit of flux is called a lumen and is about $^{8}/_{100}$ of the total flux of light produced by a source of one spherical candlepower.

The term candlepower used without qualification is not only confusing but really meaningless. If all sources of light distributed light equally in all directions, then a single measurement of their candlepower would suffice to compare them. Practically, however, sources of light differ a great deal in the way they distribute light, and this is especially true if reflectors are used.

Therefore, if a lamp is stated to give two candlepower, the statement should also explain whether "head-on" candlepower is meant, or average candlepower over the stream of light, or average candlepower in a given plane—such as, for instance, the horizontal. A lamp that uses a reflector may have a "head-on" candlepower 3 to 10 times the average candlepower over its entire stream of light. Generally it is best to state the average candlepower of a lamp instead of the candlepower at a single point or group of points.

A statement of the candlepower of a lamp does not sufficiently define its light-giving capacity. A 100-cp. lamp is seemingly 33 times as desirable as a 3-cp. lamp and yet a 100-cp. lamp shining through a hole ½ in. in diameter gives less actual light and much less useful light than a 3-cp. lamp shining through a hole 3 in. in diameter. Therefore, in order to define properly the light-giving capacity of a lamp, a statement must be made regarding both the candlepower and the total flux of light (or lumens) produced by the lamp.

The selection of proper lower limits for intensity of light and its flux is, aside from safety, the most important consideration in selecting portable electric lamps. Without these standards of reference accurate and intelligent comparison of lamps is not possible. In an attempt to establish such lower limits the Bureau of Mines searched for some time for standards which should be fair, not too low in value, not arbitrarily selected, and which should bear an easily recognized relation to something already in use.

It was finally decided to prepare a standard Wolf safety lamp to give its best performance and, after adjusting the flame height to 1 in., measure the average intensity of the stream of light and also the total flux of light in the stream. This was accordingly done at two different times, using different lamps, prepared by different men, and tested with different instruments of different types. The first measurements were made by Dr. L. O. Grondahl, of the Carnegie Institute of Technology, and the second measurements at the Bureau of Mines. The results of the two tests checked within a very few per cent.

The lamp used was a Wolf miner's safety lamp, 1907 model, round burner, burning 70–72 deg. naphtha, and prepared and trimmed in accordance with the standard practice of the Bureau of Mines. The average intensity of light stream, as determined by these tests, was a trifle under 0.4 cp. and the total flux of light was found to be not quite 3 lumens.

The bureau therefore concluded that a satisfactory lower limit of flux of light for hand lamps would be 3.0 lumens and a satisfactory lower limit of average intensity would be 0.4 candlepower.

The bureau suggests that lamps designed to be worn upon the cap should give the same intensity of light as that required for hand lamps, but that the minimum flux of light required from cap lamps should be not more than half the minimum demanded from hand lamps, because when a lamp is worn upon the head any light that is thrown to the rear is wasted. If the equivalent of a safety lamp were mounted upon a man's head, one-half of its light would fall behind the man and thus could not be used. Therefore the bureau concluded that 1.5 lumens would be a satisfactory lower limit for the flux of light produced by a cap lamp.

Twelve hours was selected by the bureau as a reasonable time of burning. This length of time was chosen after consultation with several people outside of the bureau, who were competent to express an opinion in regard to the subject.

The folowing table prepared by E. M. Chance, gives comparative candlepower of various types of lamps. These data were accumulated during about eight years and are general averages. The photometric determinations were made upon a United Gas Improvement Co. 60-in. bar photometer. The photometric standards used were 10-volt tungsten lamps, pre-

Candlepower of Various Types of Portable Miners' Lamps

	Candle-power	Cost per Shift, Cents
Miners' open oil cap lamp	1.50	2.4
Miners' open acetylene cap lamp	5.00	1.5
Electric cap lamp	1.10	
Davy safety lamp	0.12	
Clanny safety lamp	0.35	
Wolf-type safety lamp	0.65	
Akroyd and Best safety lamp	1.10	
T. M. Chance acetylene safety lamp	3.80	

Note.—The above candlepowers are in no sense maximum, but are the average values over the field illuminated by the lamp in question and have been obtained from many determinations. These are the value that may be expected to be realized in practice under working conditions pared and calibrated by the National Lamp Works, and standard sperm candles.

No estimate of the cost per day of electric cap or flame safety lamps is given in the table. The labor charge on both the electric cap and flame safety lamps is so large and varies so much with the size of the installation that such figures as could be given would have but little meaning.

Portable electric lamps.—The qualifications of portable electric lamps can be grouped under three main heads as follows: Weight, cost and capacity.

The weight of a lamp can be easily ascertained and each prospective user of a lamp must decide for himself whether or not its weight is excessive. Under the head of cost would be included the first cost of the equipment, as well as all proper charges for operating and maintaining the lamp. Some of these charges will vary with each installation and whether or not the cost is excessive will depend somewhat upon the conditions which surround each case.

The capacity of the lamp is taken to mean its ability to produce a certain amount of light for a definite number of hours per day, every day in the year if need be. A lamp that can do this with the fewest interruptions has the greatest capacity for performing the duty for which it is intended. The capacity of a lamp as thus defined takes into consideration not only the ampere-hour capacity of the battery and the efficiency of the lamp bulb, but also the life of battery plates, the mechanical strength of parts and the resistance to wear and tear.

We need to define (1) what is the proper amount of light for a lamp to give; (2) the proper time it should burn each day, and (3) what are reasonable interruptions of service and how often they may occur.

Proper care of the lamps has considerable effect on the reliability of the service. One of the large German mines, having several thousand electric lamps in daily use, reports that at first about 5 per cent of all lamps taken into the mine at the beginning of the shift were returned at the end of the same shift, either burning poorly or not at all. By a careful study of all details in the lamp house and by putting a skilled man in charge of the lamp-house work, this percentage was reduced to less than 1.5, with the expectation that it would eventually drop below 1 per cent.

While the first cost of electric lamps is undoubtedly higher than that of those burning benzine, the cost of operation, including maintenance, is claimed to average from 10 to 15 per cent less. The cost of the electrical energy is small and the cost of maintenance consists about one-third of labor and two-thirds of renewal of parts, and depreciation.

Of especial importance is the cost of renewing the electrodes of the storage batteries, replacement of complete lamps, which are broken on account of rough handling and accidents, and renewing the incandescent bulbs.

The life of the electrodes for lead cells ranges from about 100 to 400 shifts, depending entirely upon the treatment which they receive.

The replacement of complete lamps which are broken on account of rough handling and accidents undoubtedly varies more or less in accordance with the character of the work performed in the mine. European practice shows that about 0.1 per cent of all lamps per shift are lost in this manner.

The incandescent-lamp renewal already has been expressed in figures, in connection with the reliability of service. Excellent results have been obtained, the average life of the lamps being approximately 1000 hours.

The Manlite lamp complete, including the battery, cable and head piece, weigh 3½ lb.; of this the head piece weighs 4 oz. The light of this lamp is brilliant, but soft, and at the same time absolutely steady and unflickering for at least 12 hr. per single charge of battery. The curves of discharge of the latter shown in Fig. 3, show how it improves in service.

On the first discharge the voltage falls to 1.8 in a little more than 8 hr. The second discharge reaches the same point in a little over half as long again. The progress is thereafter not so rapid, but by the twelfth discharge the voltage is maintained at over 1.8 for more than 15½ hr. By the thirty-seventh discharge that period is extended over the sixteenth hour.

But these curves are based on a discharge of 1 amp., whereas the lamps only use 0.78 to 0.83 amp., and consequently the voltage is maintained for much longer periods than those mentioned. The curve of charge with 2 amp. shows a slow increase in voltage till the fifth or sixth hour, when the rise becomes quite rapid till the seventh hour.

The batteries will supply light after 10 charges and discharges for 20 hr. at a stretch. This has been shown by tests duly authenticated by mining companies which have used the outfits.

The bulb employed, and approved by the Bureau of Mines, has a guaranteed life averaging 600 burning hours. It is held in the burning position by a perfectly ground crystal lens carried securely in place by the lens holder, the whole being sealed so that the miners cannot tamper with it.

Accurate figures of the actual cost of upkeep of the lamp in continuous practical service for periods ranging from six months to over a year at various large mines have been col-

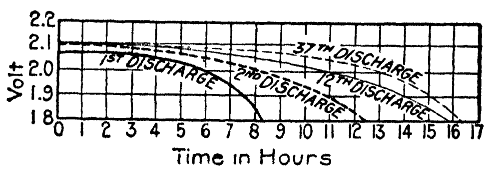


FIG. 3.—Curves showing how battery maintains its voltage for a lengthened period after repeated use.

lected which show that the cost of material per lamp-shift does not exceed 1½c. In a lamphouse operating 320 of these lamps during five months no battery repairs or renewals were required aside from a small quantity of electrolyte. The cost of other mechanical repair parts for the 320 lamps amounted to only $15.25.

The equipment is sold at a reasonable figure, and the cost of renewal and repair parts is equally low in price; for instance, the cost of a positive plate approximates 50c., while a set of negative plates costs $1.

At the Merchants Coal Co., Orenda No. 2 mine, Boswell, Penn., 250 Edison lamps were installed on February 15, 1916, at a total cost of approximately $3200. These lamps were in continuous service for about 5 months during which time they were used the equivalent of 30,450 lamp-shifts. During this time the total cost for bulbs, cords, lenses and all other repair parts was $126.49, or a trifle less than 15c. per lamp per month. On the basis of 30,450 lamp-shifts, the cost per lamp-shift was 0.4c. Add to this the cost of lamp tender, current, approximate depreciation and interest on original investment, and the total is 2c. per lamp per shift.

This installation shows the approximate range of cost, fixed charges, depreciation, interest on original investment, service and current.

The figures given herewith were taken from the invoices covering material shipped to the mine during the period mentioned, and no account has been taken of the material on hand on June 1, 1916, in either case; so that there is a satisfactory margin, and the costs shown are slightly in excess of the actual replacements during the period outlined. No expense has been incurred for repairs or replacements of the batteries themselves.

A 12-months test of electric lamps was made at the Vulcan mine, Newcastle, Colo., of the Rocky Mountain Fuel Co., about 1915. It is important to note that the installation was not large, and so the figures well represent what might be readily duplicated in a similar small station. Moreover it may be noted that the lamps were tended by the regular lampman of the mining company and not by an experienced and specially trained man.

The Vulcan mine dips at an angle of over 45 deg., and the lamps are operated under the most unfavorable conditions, being subjected to the roughest usage. On December 23, 1914, 27 lamps were put into service, and monthly statements were made out, showing not only the number of delivered lamp shifts and burning hours of the different lamps, but every repair necessary and every spare part consumed.

The total number of delivered lamp-shifts during the 12 months was 6350, making a total of burning hours of 61,700. The number of lamp-months was 324, and the cost of upkeep excluding labor was, as shown, $65.52 net or 20.22c. per lamp per month. This figure included all material needed to keep the lamps in good working condition. Even lye solution, vaseline and acid received due consideration.

The average number of lamp-shifts was 19.6 per month. Thus the upkeep per shift per lamp was only 1.03c. Including the 27 bulbs furnished with the lamps, 113 bulbs were rendered valueless during the year. Of these 34 were destroyed by the miners. These may be figured as being half-consumed before they were destroyed, and the same assumption is made about the 27 bulbs in use at the end of the period.

COST OF MATERIAL AT SELLING PRICES USED AT VULCAN MINE DURING 12 MONTHS ABOUT 1915

Material Consumed and Destroyed by Carelessness:		
86 bulbs, 45c. each	$38.70	
49 cable lengths, 4 ft. each, 20c. each	9.80	
20 spring terminal sockets, 12c. each	2.40	
8 connection pieces 20c. each	1.60	
2 lens holders, 20c. each	.40	
100 lead seals for lamps, 60c. per 100	.60	
15 rubber corks, 5c. each	.75	
10 lead check nuts with washer, 13c. each	1.30	
17 lamp holders, 15c. each	2.55	
2 lenses, 20c. each	.60	
3 lamp bodies, 70c. each	2.10	
8 celluloid battery casings, $1.40 each	11.20	
43 celluloid battery tops, 30c. each	12.90	
9 oz. celluloid strips, 25c. per oz	2.25	
5½ pt. celluloid paste, $1.75 per pt	9.62	
Gross total of material	$96.77	
Less 20 per cent trade discount	19.35	
Net total of material		$77.42
Material for Care of Batteries:		
10 cans of lye solution, 10c. each	$1.00	
8 lb. of vaseline, 8c. lb	.64	
19 gal. of acid, 18c. gal	3.42	
		5.06
Net total		$82.48
Material Destroyed by Carelessness of Miners:		
34 bulbs, 45c. each	$15.75	
17 lamp holders, 15c. each	2.55	
3 lenses, 20c. each	.60	
3 lamp bodies, 70c. each	2.10	
1 lens holder, 20c. each	.20	
Gross total of material	$21.20	
Less 20 per cent trade discount	4.24	
		16.96
Material consumed by use during 12 months		$65.52

Thus 61 bulbs may be figured as half-consumed, which is equivalent to about 31 bulbs wholly burned out. Thus it is fair to assume that the requirements of the year's running were 82 bulbs. These delivered, as stated, 61,700 burning hours, or 753 hr. per bulb, which is a good result.

It is interesting to note that no battery plates had to be renewed during the whole year, and the mine reports that all the electrodes were still in first-class condition.

It will be noted that 82 bulbs were sufficient for 27 lamps, so that less than three bulbs served for each lamp for one whole year.

Forty-nine cable lengths had to be renewed during the trial year. Let it be assumed that the 27 cables were half worn out when the year was concluded. This will make the equivalent number of cables a trifle under 63. In other words, as the total number of delivered lamp shifts was 6350, a cable lasted for over 100 shifts, or as 19.6 shifts were delivered per month, it lasted for over five months, a long life for a part under such stress.

This estimate does not cover the maintenance charges though such a small installation does not give representative results in this respect. On the other hand a new fastening has been adopted since this lamp was put in service which will greatly lengthen the life of these.

It has been found that 250 lamps can easily be tended by one lampman at $75 per month and one assistant at $45 per month or a total of $120 per month. Assuming that each lamp is operated for 19.6 shifts, the 250 lamps deliver 4900 shifts for a labor cost of $120 or 2.45c. per lamp-shift. Adding the cost of upkeep to this the total cost including all charges will be 3.48c. or roughly 3½c.

At the Keystone Coal and Coke Co.'s Salem mine, New Alexandria, Penn., 200 Edison electric safety mine lamps are in service. One hundred of these were installed September 17, 1915, 50 were added November 18 of the same year and another 50 on January 6, 1916.

Including the erection of charging racks, switchboards, etc., the cost of installing these 200 lamps was approximately $2000. From September 17, 1915, when the first lot was installed, until June 1, 1916, the only expense for maintenance and upkeep, all necessary supplies and repair parts, such as cords, bulbs and lenses, was $383.87, which is only a trifle over 24c. per lamp-month.

During the period from September 17, 1915, to June 1, 1916, there was a total of 34,250 lamp-shifts, so that the actual cost per lamp-shift for maintenance and upkeep during that time was 1⅛c. The total cost for service, including salary of lamp tender, current, interest on investment and depreciation, was 2¾c. per lamp-shift.

As these lamps were in continuous service for a period of 6 mo., and some for more than 8 mo., these figures are interesting, in view of the fact that they represent the maximum cost at any time during the operation of this type of lamp, since from its very construction, there can be no further depreciation or necessary repairs except those included in the amount given.

This same company also had 250 lamps in operation at its Crow's Nest mine in 1915, the operating costs of which are of interest.

In a six-months' period the lamps were burned for 34,419 lamp-shifts. The list cost of the spare parts totals $521.83. A discount of 20 per cent may be figured on these prices, leaving the net cost of parts and acid consumed $417.45.

But of this some material was broken while in the care of the miners:

6 lamp bodies, at 70c	$ 4.20
20 lenses, at 20c	4.00
218 bulbs, at 45c	98.10
3 safety devices, at 25c	.75
2 lamp holders, at 15c	.30
	$107 35
Less 20 per cent discount	$21 47
Net cost of material	$85.88

This leaves $331.57 of the cost chargeable against the lamp-station expense. and dividing the sum thus obtained by the number of lamp shifts 34,419, the cost of material obtained is 0.963c., or less than 1c. per lamp-shift.

It will be noted that the figures given include not merely what are known as spare parts and repairs, but sulphuric acid and celluloid paste also. Thus all possible material charges are included, though it will be observed that the cost of current, amortization, interest and labor of lampman are not figured.

It is noticeable that the cost of electric bulbs and cable are the two principal items in the list. Schedule 6A of the United States Bureau of Mines requires that the average life of lamp bulbs shall be not less than 300 hr. for primary and acid storage batteries and not less than 200 hr. for storage batteries using an alkaline solution. Not more than 5 per cent of the bulbs examined shall give less than 250 hr. life with acid batteries, nor less than 170 hr. life with batteries having an alkaline electrolyte.

If the number of working days in the month is taken at 25 and the number of shift-hours at 12, there would be 300 hr. during which the lamps would be in use in any month. Under such circumstances a bulb of 300-hr. capacity would have to be renewed every month, or 12 times in a year. If the price of each bulb is 36c., the cost per annum would be $4.32 per lamp per year. This maintenance would be far too heavy, as it covers the cost of bulbs only.

At Crow's Nest the total number of approved bulbs received up to September 18, 1915, was 995. Some of these of course were mounted in new lamps and others were placed in stock. On September 18 there were 157 bulbs in reserve. Thus there were 838 bulbs broken, consumed or in use. The miners had broken 218 bulbs in service. It seems conservative to rate half of these as burned-out bulbs. The bulbs were destroyed by the mishandling of the miners, but not being new they would not have burned as long as bulbs from stock, even if they had not been injured. The loss from negligence can therefore be calculated as being equivalent to 109 bulbs.

This can be deducted from the loss as previously obtained, leaving 729 bulbs destroyed by burning out. The same assumption, that the bulbs when unbroken have still half their life unexpired, may be applied to the bulbs in the lamps. There are 250 of these. It will be assumed that their life is equivalent to that of 125 bulbs fresh from stock. These bulbs can be deducted from the number already obtained and the bulbs actually consumed will be 604.

The number of lamp-shifts of these 604 bulbs is 34,419.

Each shift is taken at 11 hr., giving 378,609 lamp hours, or 627 burning hours per bulb, or 57 shifts of 11 hr., instead of the minimum average as required by the bureau for an acid battery, namely, 300 hr.

Assuming the number of shifts in a year to be 300, then 5.26 bulbs will be needed per year, which at 36c. would entail an expenditure of $1.89. Where the Mannesmann Light Co. maintains its lamps at so much per shift it stipulates that the operating corporation shall charge its miners with the net amount of all material destroyed through the carelessness of these employees. It is fully justified in assuming therefore that the estimates of lamps broken by miners is not in any way excessive.

The cable used is marked as 5 ft. long. As a matter of fact it need only be 4½ ft. The additional 6 in. is added for conservative estimation. The cable is the one part of the lamp still furnishing a small problem to the manufacturers, and efforts are being made to solve it in a more satisfactory degree.

MATERIAL DESTROYED BY MINERS AT CROW'S NEST MINE FROM MARCH 22 TO SEPTEMBER 18, 1915

Period	Lamp Bodies	Lenses	Bulbs	Safety Devices	Lamp Holders
March 22 to April 30	1	5	50	1	1
During May	3	4	52	1	
During June	.	3	45	..	.
During July	..	1	34	.	..
During August	1	6	23	1	..
September 1 to September 18	1	1	14	.	1
	6	20	218	3	2

According to observations made, a lamp cable is bent on the miner's back about 7000 times during the length of a single shift. It is easy to understand, therefore, to what a severe service it is subjected. It is not so much the kind and quality of the cable which is in question. The important matter is the manner in which the cable is attached to the lamp and battery casing. The tests mentioned are being made from this point of view.

MATERIAL USED IN MAINTAINING AN INSTALLATION OF 250 LAMPS AT CROW'S NEST MINE OF THE KEYSTONE COAL AND COKE CO., GREENSBURG, PENN., FROM MARCH 22 TO SEPTEMBER 18, 1915

Spare Parts Used	Mar. 22 to Apr. 30			During May			During June			During July			During August			Sept. 1 to Sept. 18		
	Pieces	List Price	Total	Pieces	List Price	Total	Pieces	List Price	Total	Pieces	List Price	Total	Pieces	List Price	Total	Pieces	List Price	Total
1. Lamp bodies	3	$0.70	$2.10	5	$0.70	$3.50	2	$0.70	$ 1.40				7	$0.70	$ 4.90	2	$0.70	$1.40
2. Lenses	8	.20	$.60	3	.20	.60		.20	.20	2	$0.20	$ 0.40	8	.20	1.60	2	.20	.40
3. Rubber rings for lenses	1	.10	.10	1	.10	.10							105	.10	10.50			
4. Bulbs	97	.45	43.55	105	.45	47.25	99	.45	44.55	83	.45	37.35	45	.45	20.25	16	.45	7.20
5. Bulb springs	1	.05	.05	1	.05	.05												
6. Safety devices	6	.25	1.50	5	.25	1.25										1	.25	.25
7. Connection pieces	5	.20	1.00															
8. Rubber washers				1	.02	.02												
9. Lamp holders	1	.15	.15	2	.15	.30							5	.15	.75	3	.15	.45
10. 5-ft. cable lengths	7	.25	1.75	221	.25	55.25	233	.25	58.25	309	.25	77.25	169	.25	42.25	84	.25	21.00
11. Leather sleeves							4	.35	1.40									
12. Aluminun shields	2	.10	.20	2	.10	.20				4	.10	.40						
13. Casings with belt hook													15	.51	7.75			
14. Hooks for locks										2	.25	.50	3	.25	.75			
15. Rubber corks with pipe	3	.07	.21															
16. Rubber corks without pipe	4	.05	.20															
17. Celluloid covers of casings										3	.30	.90						
18. Celluloid covers for gas chamber							1	.05	.05									
19. Celluloid vessels										4	1.05	4.20						
20. Pints of celluloid paste										1	1.75	1.75				1	1.75	1.75
21. Carboys of acid	1	2.20	2 20				1	2.20	2.20	1	2.20	2.20	1	2.20	2.20	1	2.20	2.20
Total			$54 71			$108.52			$108.05			$124.95			$90.95			$34.65
Delivered lamp shifts			6,434			5,813			6,561			6,047			5,905			3,659

It is necessary to consider not only cost of material but the labor and amortization charges. The experience of the Mannesmann Light Co. has been that one man can handle at least 150 electric safety cap lamps and give perfect satisfaction. Thus 300 lamps could be tended by one lampman and his assistant. If these men are paid $3 and $2 a day respectively, and 25 days are figured to a month, the monthly payroll will be $125. If they work to capacity—that is, charge and clean 300 lamps for each of the 25 shifts—each lamp-shift labor charge will be 1.66c., or about 1¾c.

In figuring the battery maintenance charges no consideration has been given to battery renewals, because in the six months during which the lamps were installed at the Crow's Nest mine the batteries did not need any repairs. But the time will come when they will, so it is well to be safe and to double the repair bill arbitrarily, making that item 2c. Adding the labor charge to this, the cost of operating is 3.7c. per lamp-shift.

The custom in most American coal mines is to charge the miner at least 5c. for the use of a safety lamp during one shift. Taking 5c. as the charge, the balance left for the operator would be 1.3c., and this would have to cover amortization and interest on invested capital. A small lamp plant of 300 lamps running 250 shifts per year would give a total of 90,000 lamp-shifts, which at 1.3c. would provide $975 for amortization and interest.

Such a plant for the Manlite lamps, including the lights themselves, would cost about $2750. An amortization charge of 30 per cent and 6 per cent interest would amount to $962.50. So it will readily be seen that the electric cap lamp not only gives a good light and assures safety from gas and other causes, but also is at 5c. per lamp-shift by no means an unsatisfactory investment.

At the mine of the Federal Coal and Coke Co., Grant Town, W. Va., an installation was started in October, 1914, with 80 electric hand lamps and this number was increased to 560 lamps in April, 1915. A record is therefore available for this considerable number of lamps, for an entire year.

The total cost of material shipped to this particular mine during the 12 mo. in question amounted to $1769.51. During this period, the sum charged to the miners on the pay roll, for material intentionally or carelessly destroyed, amounted to $196.88 so that the total cost of material to the company, ranging over a period of a full year, was $1572.63. In this amount no consideration was taken of the materials and repair parts still on hand in the stock-room of the mine, at the expiration of this period. The value of these materials and parts was quite appreciable.

During this period the number of lamp-shifts furnished to the mine amounted to 141,316, so that the cost per lamp shift for maintenance and upkeep would amount to 1.11c.

The above figures do not include labor. The lamps at the above-named mine were in charge of an expert lamp man with from 2 to 3 assistants, according to the conditions prevailing at the mine, but the total wages paid out at the lamp house at no time amounted to more than about $200 per month. This would amount to approximately 1.4c. per lamp-shift.

The total cost of operation for material and labor therefore amounted to approximately 2½c. per lamp-shift.

These lamps have been used daily without interruption since they were first installed. It should also be noted that only a small percentage of the material actually used was charged to the miners; viz., about 11 per cent.

Oil and acetylene lamps.—The Union Pacific Coal Co., prepared the following figures as to the cost of burning acetylene and oil lamps in 1914. No attempt has been made to compare the candlepower-hours. Such a comparison would be more favorable to the acetylene lamp than is here given. The period on which the estimates are based is a day of 8 hr.

COST OF ACETYLENE AND OIL LIGHTING

Name of Mine	Carbide Lamp	Lard-oil Lamp
Reliance	$0.015	$0.048
Rock Springs	0.055	0.075
Cumberland	0.035	0.045
Superior	0.034	0.075
Hanna	0.034	0.070
Average	0.0346	0.0626

The estimates are based on a charge of $8^1/_3$c. per lb. for carbide and 65c. to 75c. per gal. for lard oil.

The oil lamp is rapidly disappearing from the mines in this country but for the benefit of those mines still using the oil lamp the following particulars are given.

Oil lamps.—The viscosity of an oil is the property of the different particles to hold together and at the same time adhere to other bodies. It may be called the "fluidity" or "thickness" of an oil. Oil of high viscosity does not flow so freely as an oil of low viscosity. Viscosity has been the cause of complaint with miners' oil, otherwise satisfactory, as for instance, an oil which gave the following results upon test: Viscosity, 16.2; density, 22 deg. Bé.; grams oil burned per minute, .2; height of flame, 4 inches. This oil was the cause of dissatisfaction at several of the mines the complaint being that it did not keep lighted on headings and air-courses where strong air-currents were met with, that it was hard to relight, and that the oil did not feed up the wick fast enough. While the flash point of this oil was somewhat to blame for the trouble, it was principally due to the high viscosity.

The congealing point of oil is an important factor in the winter months. It is the temperature at which the oil will solidify, and if this point is too high it is the cause of considerable trouble both to the miner and storekeeper, which results in a loss of time and waste of oil. Although the temperature inside the mines varies but little, there are many cases where roadmen, drivers, motormen, and others, come in contact with air that is almost as cold as that outside, consequently, when buying winter oil there should be a guarantee that it will not congeal at a temperature above 32 deg. F. and it is advantageous to have this point even lower. The winter oil used by the Fairmont Coal Co. shows a congealing point of 21 deg. F. on an average of 22 samples tested. This allows it to be stored and handled in cold places without freezing.

The flash point is the temperature in degrees Fahrenheit at which the oil will flash or ignite. For instance, an oil with a flash point of 300 deg. is one that will ignite when heated to that temperature, if a flame be applied to it. The flash point decreases as the density becomes less; the lighter oils flashing at lower temperature. Some companies have 300 deg. to 425 deg. F. established as the limit in their specifications for miners' oil, that is, the flash point must not drop below 300 deg. nor exceed 425 deg. F.; the lower limit being applied as a measure of safety; the higher one as a limit at which the oil will readily burn and relight quickly.

The quantity of oil burned for a given length of time is important from the miner's point of view (although it is hard to make him realize it). In trying to comply with the state law, and also furnish an oil that will give the miner the best results per unit of cost, it is necessary to take into consideration its efficiency. The grams of oil burned per minute depend on the size of the lamp used, the size of the wick, and the length of the wick beyond the spout, as well as the quality of oil. With the first-mentioned conditions remaining constant, the variation in the grams burned per minute depends to some extent upon the density. With a cottonseed oil of high specific gravity the oil burned per minute in a standard No. 2 mine lamp is about 0.2 gram, while in an oil highly adulterated with kerosene, or a light, highly volatile oil, and of a low specific gravity, the figure is about 0.5 gram.

The candlepower of oil naturally bears a close relation to the quantity of oil burned, and to get a comparative figure on different oils as to this property, the candlepower is calculated to candlepower per gram per minute. This takes into consideration the effect of different rates of burning. In making the test of the candlepower, standard photometric candles (12 to a pound) are used, in an ordinary photometer, and the results check very closely. The present specifications call for an equivalent candlepower per gram per minute at 8.

The height of flame is also taken into consideration and is specified in purchasing oil to be between 4 and 5.5 in. with the wick extending 1/4 in. beyond the end of the lamp spout, with 12-strand wick. This height has been found to produce the best candlepower with the least amount of smoke, and if the height of flame is limited, the evolution of smoke is to some extent controlled.

In testing miners' oil there is probably nothing of greater importance than the quantity of smoke it gives off while burning. This smoke is, in a measure, indicative of the quality of the oil, as an inferior oil always makes denser smoke and a greater quantity of it, and it has often been noted that the smoke increases as the oil deteriorates in quality. It is this point which guides the mine inspector in his inspection of the oil used at different mines. If the oil gives off an excessive amount of smoke, he becomes suspicious and forwards a sample to the head of the department for test, and it usually develops that the oil is adulterated and not up to the standard required. Aside from the dust and the smoke of the powder, smoke from open lights is the principal cause of air vitiation in the working places of a mine. This is especially noticeable in places where the velocity of the current is very low, and with oil of low grade. An oil to which kerosene has been added (which is common practice in most regions) is the greatest offender in this direction.

Although it is difficult to measure the quantity of smoke given off by any particular lamp, the following method, which is similar to the one used by the state mine inspector, gives comparative results and is sufficient for testing miners' oil. This test is made by observation, using pure cottonseed oil as the standard of comparison. The size of lamp, size and length of wick, and the quantity of oil are the same in each case, and length of the smoke pencil on the standard oil is taken as one, and the length of the smoke pencil on the oil tested is relatively greater or less. The lamps are placed in a box with a glass front, and a white cloth, graduated in inches, serves as a back-ground. The limit placed on the smoke is a pencil about 1.5 greater than that of the standard cottonseed oil. This test, however, is unnecessary quite often when other requirements above mentioned are complied with, as they govern to a very great extent, the evolution of smoke.

Apparently the most difficult requirement to meet is the density, which must not exceed 24 deg. Bé. scale, which corresponds to .913 specific gravity. Of many samples of miners' oil of different kinds submitted by many companies, there has been only a small percentage to come within the limit. Its adoption as part of the specifications of some companies has been due solely to the fact that the State Oil Regulations require it. Miners' oil of excellent quality can be obtained at very reasonable prices, only a degree over the state law limit, but is not purchased because it exceeds 24 deg. Bé. This test is made at 60 deg. F., and while certain properties do bear some relation to the density, there is no room, apparently, why this limit should have been placed as low as 24 deg. Bé. Very seldom has a miners' oil sample been received in the laboratory that does not exceed this point. We have found oils up to 30 deg. Bé. that were highly satisfactory for use in miners' lamps, being pure unadulterated oils "as free from the evolution of smoke as a standard cottonseed oil."

A trial test was made extending over several months on standard cottonseed oil, and the reports from 41 mines using it were, in effect, that it was of such high density (low Baumé) and high flash point as to be very unsatisfactory. The flame would not hold to the wick, the lamp once out was hard to relight, the light furnished was of low candlepower, and the cost was excessive.

The specifications for miners' oil as adopted by the Fairmont Coal Co. in 1910, were as follows: Density, 24 deg. Bé. or under; congealing point 24 deg F. or under; flash point, 300 deg. to 425 deg. F.; flame height, between 4 and 5½ in. in a No. 2 miners' Star pattern lamp with ¼-in. wick projection and a 12-strand cotton wick; equivalent candlepower per gram per minute, 8 or over; smoke, to be light in quantity.

When a new oil is to be purchased, samples of it are received and tested and unless it comes up to these specifications it is not considered by the purchasing agent. If a purchase is made of an oil that has been found to be all right, each consignment is tested as it is received at the supply department. If it should here fail to come up to the requirements, as stated, it is rejected. It is a rule to test the oil at every mine at regular intervals as well as each shipment received by the supply department.

An example of the manner in which a law, that was honestly intended to be beneficial to the miner, failed of its purpose is found in the Pennsylvania Bituminous Mining Law of 1911. This law stipulated that oils sold for use in miners' lamps should not yield more than 0.11 per cent of soot when burned in a miners' lamp under standard conditions. One of these conditions was that the flame of the lamp should be 1½ in. long. Now, low-grade oils when burned under these conditions yield as much as 1 per cent of soot, while high-grade oils will give as little as 0.03 per cent. Thus it would seem, at first glance, that this law would considerably better conditions in the mines.

Such is not the case, however. Oils to pass this test must be very largely composed of costly fatty oils and this so greatly increased the cost to the miner that he was obliged to look for some cheaper illuminant. Moreover, instead of a flame $1\frac{1}{2}$ in. long, the miner burns one of a maximum length because he wants as much light as he can get. It has been found that while costly oils, containing high percentages of fatty ingredients, will produce much less soot than oils of medium price, and less fatty material, when burned under legal test conditions, these differences very largely disappear when these oils are burned under the conditions that obtain in the mines. With very long flames the high-priced oils still show a superiority to the medium grade, but the differential is so slight as to be of little real moment.

Indeed, the soot-forming propensities of both these oils under the conditions of use are so great that it is idle to attempt to classify one as better than the other. They are both very bad. Thus with a legal requirement of 0.11 per cent soot or lower, we find the oil passing this test will give about 8 per cent of soot when burned as it would be in the mines—that is, with a flame 5 to 6 in. long—while the oil that will not pass the legal requirement, giving under test conditions, let us assume, 0.5 per cent soot, will make under actual working conditions about 9 to 10 per cent soot.

Cost of underground stables.—The underground stable shown in the accompanying illustration, Fig. 4, is designed to meet the needs of mine owners who wish to construct a stable which is thoroughly modern, conforming to all the requirements of the mine law, and one which at the same time will not be too expensive to permit of its being built.

The drainage from each stall is obtained by a grade of 2 in. in 10 ft. from the manger to a gutter running along the back of the stalls. (Dr. Charles A. Leuder, professor of veterinary science at West Virginia University, states as his opinion that a slope of from 3 to 4 in. in 10 ft. will not be harmful to a horse or mule standing in a stall over night.) The gutter should have a slope of at least $1\frac{1}{2}$ per cent. This

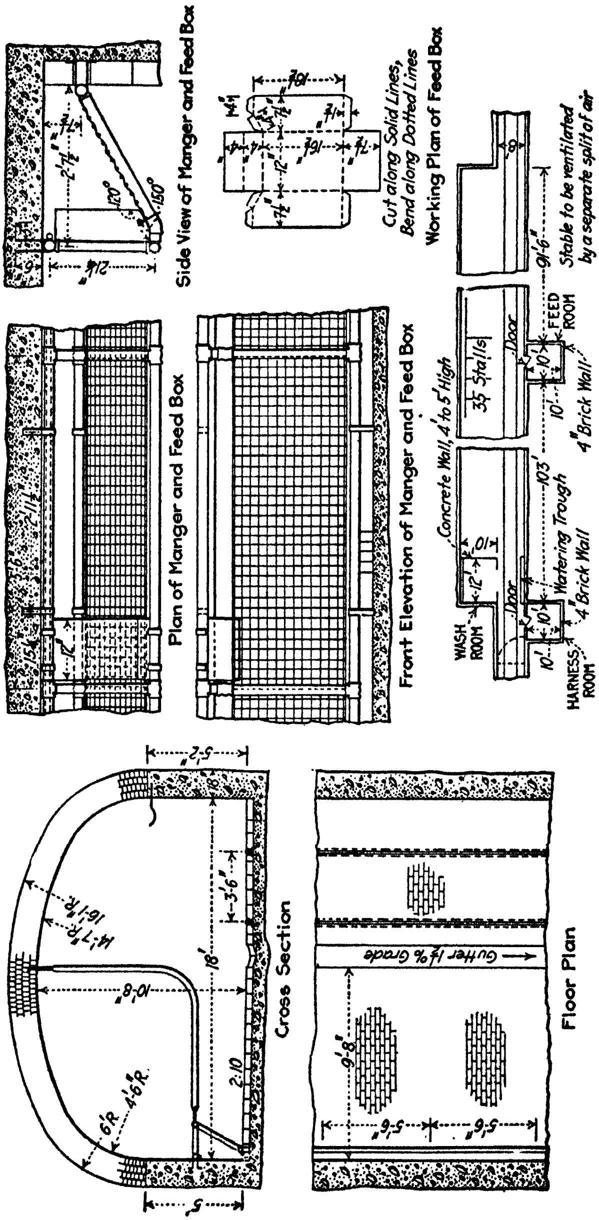


FIG. 4.—Plan, elevation and section of an economical type of concrete, brick and steel mine stable.

can usually be secured naturally by the location of the stable. If this grade cannot be obtained easily, the gutter may be made deep enough at one end to afford the proper slope. In this case all or a part of it should be covered. The gutter leads to a sump from which the water is pumped or drained naturally to the main sump.

At the entrance to the stable is a washroom with concrete floor and walls, the latter being from 4 to 5 ft. high. This may be made to drain either to the center or to one corner and from thence to the sump. The horses or mules are thoroughly washed here by means of a hose while they are being watered. Plenty of clean water for this purpose is usually available from water rings around the shaft. If it is deemed expedient, a small reservoir in connection with the water rings may be constructed at a point sufficiently high to secure plenty of pressure. A water pipe runs the full length of the stable, with taps for connecting a hose at points convenient for washing out the stalls. An additional watering trough is placed opposite the washroom for convenience in watering the animals as they are taken out in the morning.

The stalls are 5 ft. 6 in. wide and 7 ft. 8 in. long exclusive of the mangers. This gives space ample for the comfort of even the largest size of horse used in the mines. From the back of the stalls to the track the distance is 2 ft. 3 in. Between the track and the wall there is 2 ft. 7 in. of clearance. This allows plenty of space for a large feed truck.

The side walls are of concrete and are 18 in. thick. This gives a good substantial wall which is easily built. The roof is an elliptical four-ring brick arch, this type being recommended because it gives the maximum of useful space in proportion to the excavation required. Brick is used because it is believed that an arch may be more readily constructed underground of this material than of concrete.

The floor is of paving brick carefully laid with cement mortar on a 6-in. concrete base. This is used instead of concrete because it is believed that it wears better and gives a better surface for the animals to stand on. Dr. Charles A. Leuder, who has been referred to before, states that the hardness of a brick or concrete floor will not be harmful to a horse or mule. One of the best-equipped shaft mines in the Connelsville field in Pennsylvania has an underground stable floored with brick. Another up-to-date plant in the Fairmont region in West Virginia has a stable with a concrete floor. If concrete is desired, it will be a simple matter in this design to leave out the brick and put the proper surface on the concrete.

The mangers and feed boxes are simply and substantially constructed of 1½-in. gas pipe 27 × 32-in. sheet-iron plates and 2-in. mesh wire screen, securely held in place by means of straps embedded in the concrete wall and sub-floor. Standard pipe fittings are used, and no difficulty should be encountered in constructing these mangers.

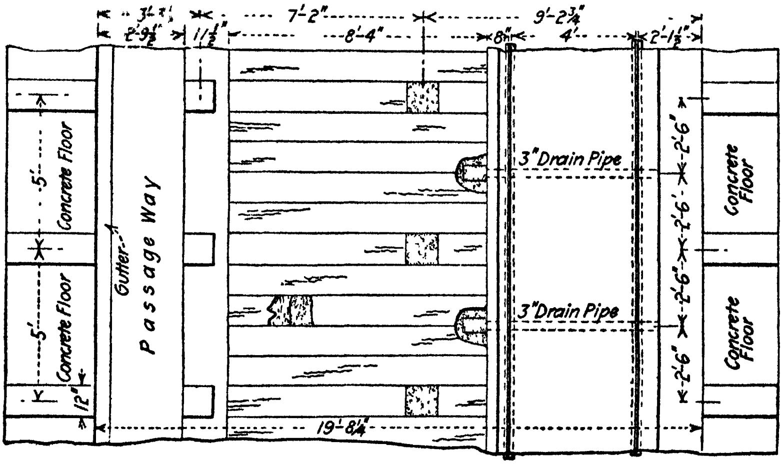
Ventilation is secured by means of a box regulator placed at the back of the stable. This should be so arranged that it can be conveniently closed when the animals are first brought into the stable in the evening, in order that drafts may be kept away from them while they are cooling off. This should be reopened in about half an hour and allowed to remain open all night. The stable is located so as to secure a separate split of air from the intake. This can usually be arranged without difficulty. Owing to the fact that the partitions between the stalls consist of gas pipes suspended from the roof and mangers, the entire stable is open to a free sweep of air, which insures fresh air for the animals even when lying down.

This form of partition is in use at the Continental No. 1 mine of the H. C. Frick Coke Co. The stable boss at this mine states that there has never been any trouble owing to the horses kicking each other and that the partitions are entirely satisfactory. If any difficulty is anticipated, it is suggested that the partitions be made more secure by suspending from the gas pipe a wire screen similar to that used in the construction of the mangers. This, when hung, will resemble somewhat the danger signals commonly used by the railroads to warn trainmen of the nearness of an overhead bridge or tunnel.

Owing to the wide variation in different localities of the cost of labor and materials, no attempt has been made to estimate accurately the cost of the stable. An approximate estimate of the cost of such a stable with 35 stalls, computed on what is taken as an average of labor conditions in West Virginia in 1916, is $3800. This, of course, does not include excavation, as it would be impossible to estimate this without a knowledge of the thickness of the seam and other conditions.

The stable drawings shown in Figs. 5 to 10 were adopted as standard by one bituminous and two anthracite coal-mining companies about 1912. The design is such as to meet the requirements of the mine inspector, mine manager and veterinary surgeon. The stable is of fireproof material throughout except the top floor of the stalls, which for the comfort of the mules is made of plank.

Fig. 5 is a ground plan of the stable. It is designed on the unit plan, so that the drawing is available for a stable



Plan

FIG. 5—Plan of two stalls of another type of underground stable.

to contain one or any number of mules. Ample room is provided in front and behind the stalls. It will be noticed that a good concrete floor is first laid, in which are imbedded the tracks for the feed cars. Facing and also at the back of the stalls are concrete gutters for carrying out the water daily used to cleanse the stable.

Each stall is 8 ft. 4 in. long and 5 ft. wide, and so is commodious enough for the largest of mine mules. The concrete piers for the center posts are clearly shown in the drawing.

Fig. 6 is a cross-section of the stable. The thickness of the concrete floor, position of drain pipe, slight pitch of the plank floor, car tracks, passage ways, gutters, manger, feed box, center posts, end arches, and the 2-in. gas pipe which alone forms the partition between the stalls, are all clearly outlined in the drawing.

The single gas pipe is an improvement over the old style high board partition, as it offers no resistance to the free circulation of air, and the building is always free from obnoxious odors. When it is necessary to stable a fractious mule, he is usually placed in one of the end stalls, and the

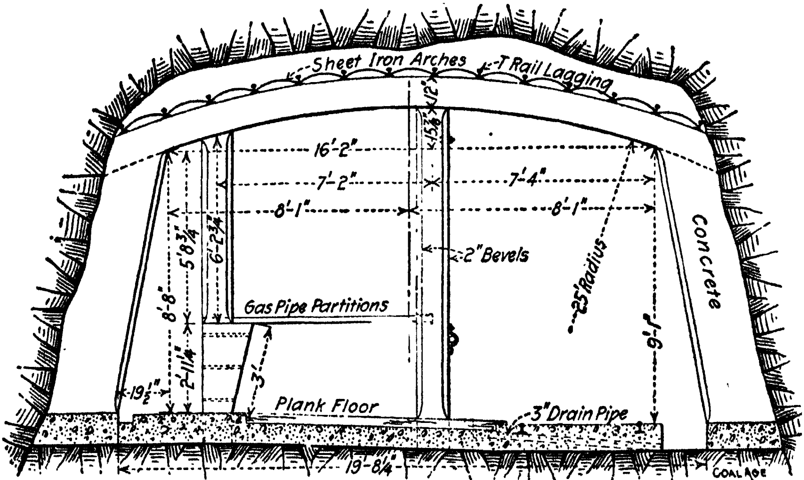


Fig. 6.—Cross section of stable shown in Fig. 5 showing manger and drainage system.

gas-pipe partition is reinforced by a strap-iron lattice-work, which effectually prevents him from annoying his neighbors.

Fig. 7 is a detail drawing showing side view and plan of the center post. This exemplifies a method of supporting a stable roof which is both effective and inexpensive, and in the eight stables where it has been tried, no failures have been recorded. It may here be noted that the end arches extend throughout the entire length of the stable, serving the double purpose of sealing any coal measures and supporting the mine roof.

Fig. 8 gives the construction details in a longitudinal section through the center posts. The method of supporting the

T-rail laggings, and the position of the mangers are clearly shown. Fig. 9 is a plan of the arches with the T-rails in position.

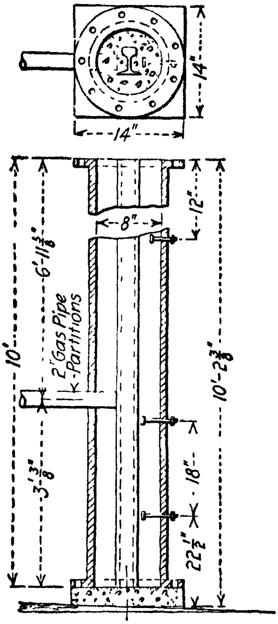


Fig 7.—Cast-iron center post used in stables shown in Figs 5 and 6.

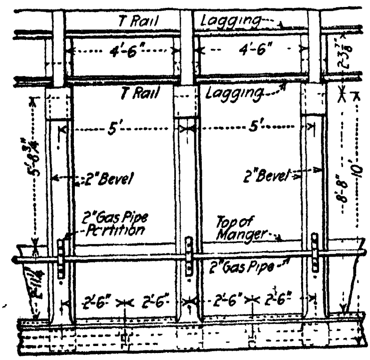


Fig. 8.—End elevation of arches for stable shown in Figs. 5 and 6.

Fig. 10 supplies all needed details of construction of the manger with its feed box, which have been designed with a view to sanitation as a leading desideratum. They are usually constructed in the shops and sent to the mines with all parts

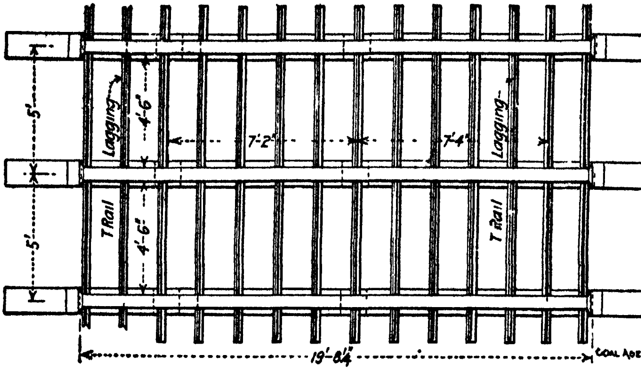


FIG. 9.—Plan of arches in Fig 8 showing T-rail reinforcing.

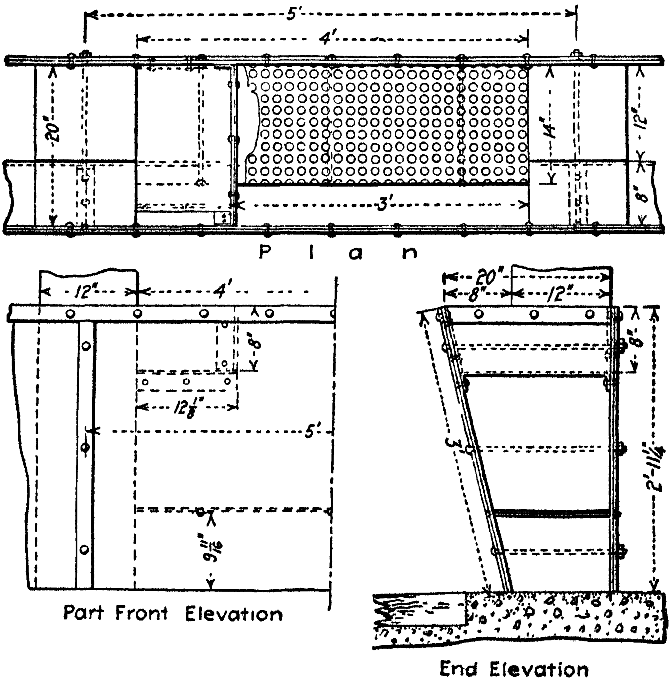


FIG. 10.—Detail of manger and feed box for stable shown in Figs. 5 and 6.

marked to facilitate in erecting. The work of installing them is thus simplified, and can be done by the average mine timberman. Particular attention is called to the hole provided in the bottom, affording a convenience for cleaning same.

When construction of the building is completed, the sheet iron which forms the temporary support for the concrete arches is not withdrawn, but is left in position and when whitewashed serves as an efficient reflector for the electric lights which are provided for each stall. A 32-cp. lamp with a heavy guard is provided with a No. 14 (BX) extension cord 10 ft. long. With this the stableman can make a careful inspection of the mule as it returns from its day's toil.

The following is the bill of material for one stall only, and an equal increase should be made for each additional mule for which accommodation is required.

BILL OF MATERIAL FOR ONE STALL

40 bags of cement. 5 tons of sand.
14—T-rail laggings—4 ft. 6 in. long (40-lb. rail)
70 ft.—odd lengths old T-rail (for reinforcing).
1—10 ft. length, 8 in. dia., cast-iron column pipe
13 pieces of sheet iron—4 ft. 6 in. long × 1 ft 6 in. wide (for arches)
2 pieces of No. 8 sheet iron 5 ft. long × 3 ft. wide.
1 piece of No. 10 sheet iron 2 ft. × 2 ft. for feed box.
5 floor planks 8 ft. 4 in. × 1 ft. × 4 in. thick.
1 bolt—1 ft. $8\frac{5}{16}$ in. long × $\frac{5}{8}$ in. diam.
1 bolt—1 ft. $5\frac{5}{16}$ in. × $\frac{5}{8}$ in. diam.
1 bolt—1 ft. $2\frac{9}{16}$ in. long × $\frac{5}{8}$ in. diam.
3 bolts—1 ft. $3\frac{3}{4}$ in. long × $\frac{5}{8}$ in. diam.
3 bolts—4 in. long × $\frac{5}{8}$ in. diam.
2 bolts—3 in. long × $\frac{5}{8}$ in. diam.
28 rivets 1 in. × $\frac{1}{2}$ in. diam.
1 piece 2 in. diam. gas pipe, 7 ft. 8 in. long (for partition between stalls).
1 piece 3 in. diam. gas pipe, 5 ft. $9\frac{1}{2}$ in. long. (for drain).
2 iron straps, 2 ft. 10 in. long × 2 in. wide × $\frac{1}{2}$ in. thick.
2 iron straps, 2 ft. $9\frac{1}{2}$ in. long × 2 in. wide × $\frac{1}{2}$ in. thick.
10 lin. ft. of strap iron 2 in. wide × $\frac{1}{2}$ in. thick.
1 piece 1-in. mesh segment for tray 4 ft. long × 1 ft. 2 in. wide.
2 brackets.

Overcasts.—The Lehigh Valley Coal Co. in 1907, started substituting for the wood overcast one of concrete and steel. These overcasts will be permanent and substantial, their destruction only being accomplished by a squeeze, in which event all other construction would be destroyed as well.

The construction of the overcasts is shown in Fig. 11. Being located in an old portion of the mine, much gob and other

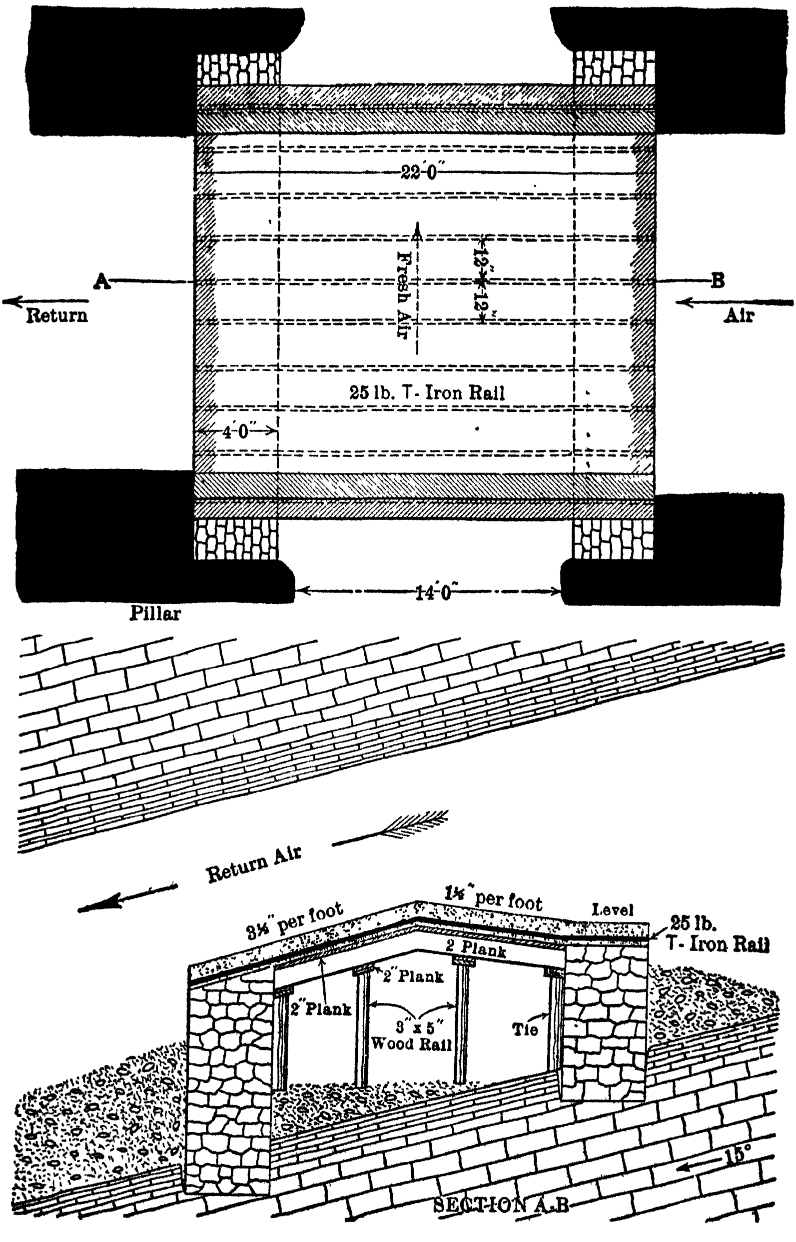


Fig. 11.—Plan View.

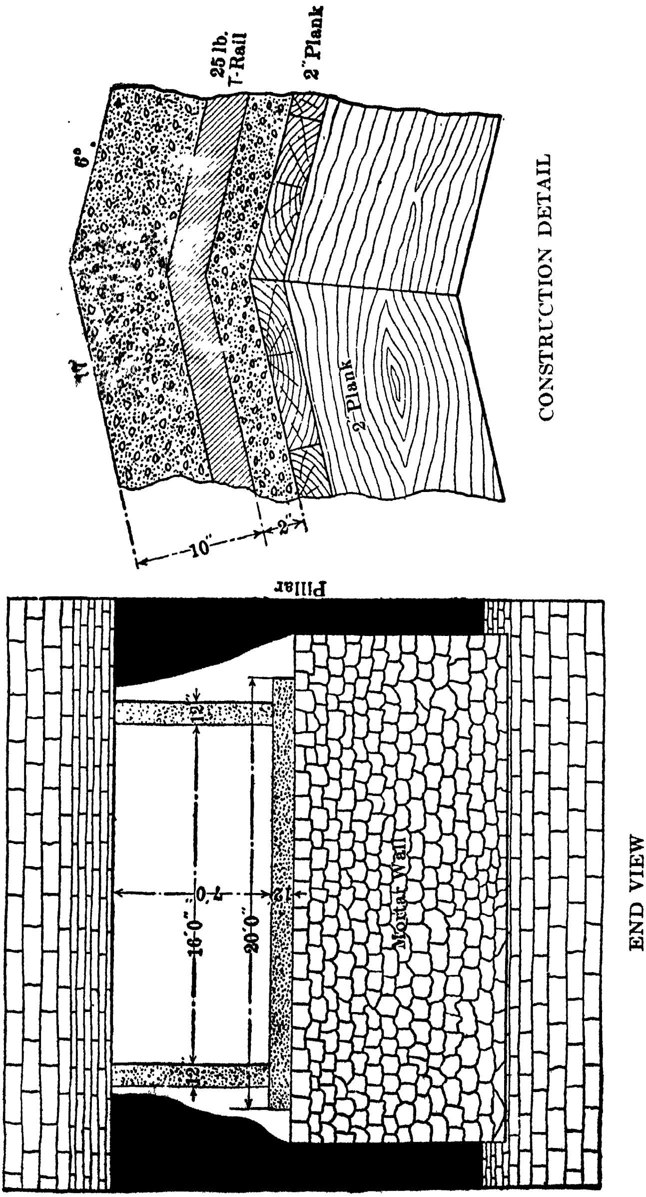


FIG. 11.—Concrete overcast used by the Lehigh Valley Coal Co. about 1907.

refuse is found on both sides of the road and must be cleared away at the immediate location of the bridges. After this trenches for the masonry walls to carry the bridge are dug and carried to sufficient depth below the bottom slate of the vein to obtain a solid foundation. The walls are then built 4 ft. thick, consisting of rock and bone of sufficient size selected by the mason from the gobbed refuse made and packed in the chambers during mining. Lime mortar is used and the faces of the wall are almost entirely surfaced off with a coat of mortar after completion.

Next in order, timbers for carrying the concrete are placed by the timberman, and consist of second-hand 3 × 5-in. wooden rails taken from old chambers and of second-hand mine ties and props for uprights, on which are laid 2-in. planks double thick, which in turn carry 2 in. planks on edge. To these planks are nailed the 2-in. planks which carry the concrete.

The time consumed is somewhat greater than one might consider and an explanation is necessary. The clearing away of the refuse accumulated is very laborious and progress is slow, and in excavating for the trenches considerable heavy rock and bone must be removed. The work being done in the return current makes it warm for men to work, and less is accomplished than if the same were located in a fresh-air current. Again, the rock and bone constituting the material for the masonry wall are all collected in the old chambers in the vicinity and taken to the location for the wall. The broken stone used in the concreting is obtained from a chamber some distance away, the stone having been dumped there during the driving of the rock slope. The mason selects what material is fit for use. At the same time sand suitable for concreting is gathered from the same place, having been made during the blasting in the slope and loaded out with the rock.

As will be seen by the end view, Fig. 11, two concrete walls form the sides of the overcast and become necessary where the entire vein has been mined out. The Baltimore vein, which in this particular location is one seam, more frequently splits into two distinct seams, the top split known as the Cooper or Upper Baltimore, and the bottom split known as the Bennett or Lower Baltimore vein. At the location in question the dividing slate is only about 1 ft. thick and in most instances both splits have been removed. At the location of No. 4 overcast only the bottom split was mined and it was necessary to take down the top vein to get sufficient height for the roof of the overcast. In taking down this top coal, or split, the ribs were neatly dressed and the concrete floor of the overcast was carried high enough to dispense with the walls. At the location for Nos. 1, 2 and 3 overcasts the total vein has been mined and concrete side walls will become necessary.

The two masonry walls are built 4 ft. thick, the walls on the up-pitch side being on an average 7 ft. high, the one on the down-pitch side averaging 10 ft. high.

The concrete floor is 12 in. in thickness and 22 ft. long by 20 ft. wide, consisting of a 1-2-3 mixture. The breadth and length will of course be greater or less depending on the dimensions of the openings through the coal at the different locations.

The T-iron rails are spaced 12 in. center to center, the bottom or base of the rail being embedded 2 in. in the concrete. During concreting the rails are supported by blocking under the head.

For the side walls a 1-2-5 mixture will be used, the walls being 12 in. thick and cement worked well into the crevices of the coal and top rock.

The details of cost of the No. 4 overcast are given herewith, the figures being as of 1907. The day's work consists of nine hours and the hourly rate includes the strike percentages and sliding scale. The proportions for the concrete mixture were 1-2-3. Sand and stone for the concrete and walls do not appear in cost for material, the only cost on these items being included in the labor.

The total cost per cubic yard for the masonry wall is therefore $1.63. The cost for labor and material is about equal for the overcast or $5.25 per cu.yd. for each, a total of $10.50 per cu.yd. The two walls needed for Nos. 1, 2 and 3 overcasts make an additional 11.5 cu.yd. at a somewhat smaller rate per cu.yd., say $8, so that the entire cost of the overcast, including two supporting masonry walls and two concrete side walls, in a vein of this thickness will be about $365, and about $270 without the side walls.

MASONRY WALLS, 60.5 CU. YD.

LABOR

Clearing away refuse, digging trenches, getting stone, mixing mortar and building two walls, 2 masons, 20 days each, or 360 hr. at 25.4c	$91.44

MATERIAL

25 bu. of lime at 30c	7.50
	$98.94

CONCRETE OVERCAST, 16.5 CU. YD

LABOR

Getting lumber for supporting concrete work and placing same, 3 men at 2 days each, or 54 hr. at 25.2c	$13.61
Placing 2-in. plank for concreting, 1 man 2 days or 18 hr. at 25.2c	4.54
Getting broken stone, mixing and placing concrete, 2 masons at 14 days each or 252 hr. at 25.2c	63.50
Bending and transporting rails approximate	5.00
	$86.65

MATERIAL

120 bags portland cement at 45c	$54.00
1000 ft. 2-in. hemlock plank at 22c	22.00
20 lb. 20d. nails at $2\frac{1}{2}$c	.50
1850 lb. ($\frac{8}{10}$ ton) second-hand 25-lb. T-iron rails at $12.50 a ton	10.00
	$86.50

Tile stoppings and overcasts.—Terra-cotta blocks have proven their usefulness in a mine, as well as for the purposes to which they are put outside. Not only have they been used economically for stoppings along the main air courses, but they are now being adopted successfully for the walls of overcasts. It appears also that scrap iron is to be displaced by a reinforcement which gives a concrete structure not only more artistic but more economical and more quickly erected.

The Consolidation Coal Co., as a preventative of fire at the same time as a safety-first measure, decided about 1914 to eliminate wood stoppings and wood overcasts where practicable and to allow no more new overcasts anywhere to be constructed of combustible material, the same to apply to permanent stoppings along main entries.

The most natural step was to use the cinders from the boiler house at the mine and to build these structures of concrete. However, investigation proved that too much money was being spent for these improvements. The stoppings were reduced to a thickness of 4 in., but the cost still appeared too large.

The terra-cotta blocks were first used for stoppings and their success was readily apparent. As the 4-in. cinder-concrete stopping had proven stable, a hollow block of 4-in. width was at first advocated; but the $5 \times 8 \times 12$-in. block was finally adopted.

On main entries where the stoppings are intended to remain for a considerable length of time they are bonded with a mortar composed of one part cement to two parts sand. On room headings where these stoppings have been built, lime was used in place of the cement, this making it possible to tear down the stopping when desired, without destroying the tile.

Sufficient cost data have been obtained on the construction of this type of stopping at the mines using this material to show a saving of from 40 to 50 per cent in every instance as against the use of concrete. The cost compared to wood stoppings for room headings shows about the same difference in favor of the wood. It was thought that by the recovery of the blocks and their reuse this difference could be more than offset; but accurate costs kept on this work shows that it would be necessary to recover these blocks several times over to do this. Therefore, it is not advisable to use the blocks for temporary structures unless for other reasons than economy.

The crosscuts in the mines of the Fairmont region run about 10 to 12 ft. wide and 7 to 8 ft. high, and average about 80 sq.ft. in area. It has been found that two men will construct three or four stoppings per day, providing the material is placed ready for their use.

It might be possible to reduce the cost of stoppings further by the use of "Self-sentering," "Hy Rib," or "Rib-Lath," plastering these with a cement mortar an inch or two thick. A stopping made with any of these materials properly placed, should carry as much as, or more pressure than, the ordinary wood stopping. The plan of erection would be simple, consisting of trenching the ribs, roof and bottom an inch or two, slipping the adopted metal in place and plastering with cement or lime mortar, the material depending upon the desired life of the stopping. Price and the strength which could be secured should govern the selection of the reinforcement. A heaving of the bottom would in all probability be cut by the stopping, since heaving is caused by the swelling of the fireclay, this material at the same time softening markedly. If the pressure comes from the top or roof there might be much deflection in the stopping without doing it any material injury, besides it could be as easily repaired as the wood stopping in case of sufficient pressure to cause cracks in the plaster.

All well ventilated mines must contain overcasts, and the costs of these structures soon run into considerable amounts, if they are not carefully planned and constructed. There are few mine foremen or even superintendents who take the trouble to ascertain accurately just what it costs to erect an overcast so that he can tell you how much was spent per cubic yard for concrete or what the walls and roof cost separately.

Fig. 12 shows the type of overcast now being constructed in the mines of The Consolidation Coal Co. in the Fairmont Region. Blank spaces are left in the blueprints, and dimensions to suit the location are supplied. Since the sheets of "Self-sentering" are sold in even foot lengths, the distances between walls are made to correspond. Thus for an 11-ft. sheet 2 in. are deducted for the 12-in. rise and two or more inches on each side for bearing spaces, leaving the distance between walls 10 ft. 6 in.

This style of overcast makes use of the terra-cotta blocks, the $5 \times 8 \times 12$-in. block being laid to give 8-in. walls. On top of each wall is placed a layer of concrete on which is set a small steel rail 2 in. from the edge of the wall. Then the reinforcing sheets of "Self-sentering," which come already shaped, are placed abutting on the flanges of the rails and the center line or middle of the reinforcement supported by a 3×5-in. wood stringer; then the concrete is spread on the top as required on the plan. After the concrete sets, the wood stringer is knocked down and a coat of cement mortar plastered underneath. Not a single wood form is required on the whole structure, except around the iron rails on top of the walls, and even this can be eliminated by standing a row of the terra-cotta blocks on the 5-in. side just outside of the rails, thus making them serve as forms.

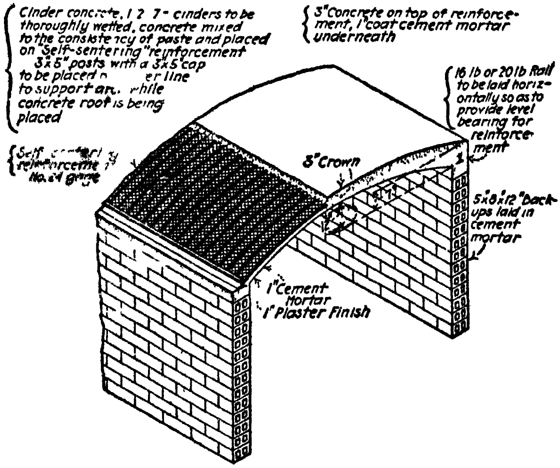


FIG 12.—Reinforced concrete overcast supported on terra-cotta blocks used by the Consolidation Coal Co

There are times when it is necessary to deviate from the arched-roof type on account of the skew of the air bridge, in which case it is more economical to use the flat roof. Fig. 13

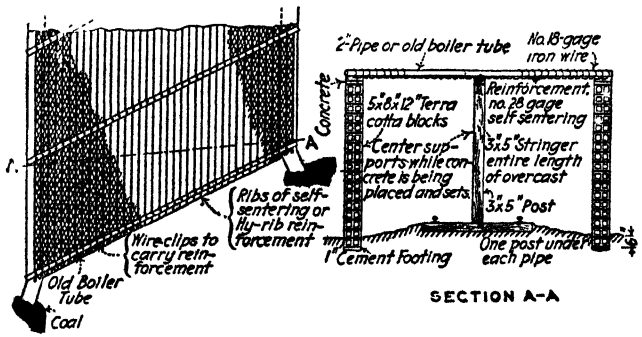


FIG. 13.—The Consolidation Coal Co.'s reinforced concrete, hollow tile overcast with flat roof.

presents a design of this kind where the roof is supported by three or four old boiler tubes, the reinforcement being suspended from these tubes or pipes by means of wire of sufficient strength to carry the weight. This type of overcast also needs only temporary supports along the middle while the concrete is being placed, thus saving considerable labor and material in the construction and emplacement of forms.

The Utah Fuel Co. constructed six concrete stoppings and one overcast in some work they did about 1914. On account of the height of the stoppings they were built of reinforced concrete with reinforcing wings as shown in the accompanying drawings, Figs. 14 and 15, old rails and wire rope being used for the reinforcing material. The reinforcement of the overcast was made with the same class of material. The cost of these is given in the figures.

Comparison of doors and overcasts.—A mine door is a costly as well as dangerous item of equipment, yet this seldom receives the thought that it deserves. When it is necessary to install a permanent door the conditions should be carefully considered to see if it will not be cheaper and safer to put in an overcast.

A permanent door involves much expenditure in addition to that for lumber. The wages of two men must be paid for its construction and erection. A trapper will have to be constantly employed to open and close it. A shelter hole must be constructed at the door and another hole also must be provided for a barrel of water, for all permanent doors should be protected against a possible fire. The continual breaking and smashing of the door by trips involves a further expenditure for lumber and for the workmen who replace it.

A light should be provided at all doors. This involves the wiring of the light, the replacing of globes, and the employment of wire men who must spend a day, more or less, in installing it. Trolley wires must be guarded on both sides of the door, which means the use of more lumber and the employment of more labor. These boards often are knocked down by trolley poles and again the service of a high-class or highly-paid man is needed.

Where permanent doors are used an extra door should be placed that will have the same effect on the ventilation. This door, of course, should be a full trip length away. Heavy blasting or heavy caving often damages doors. The weighting of the roof, the movement of the sides or the heaving of the bottom will affect them also.

Looking at the matter from a safety viewpoint we often find the door frame cuts the clearance, frequently to as little

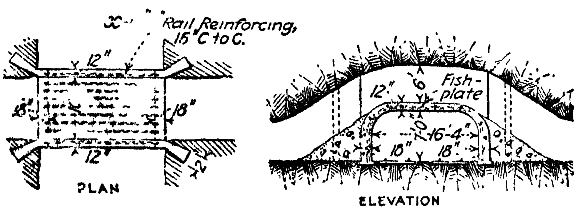
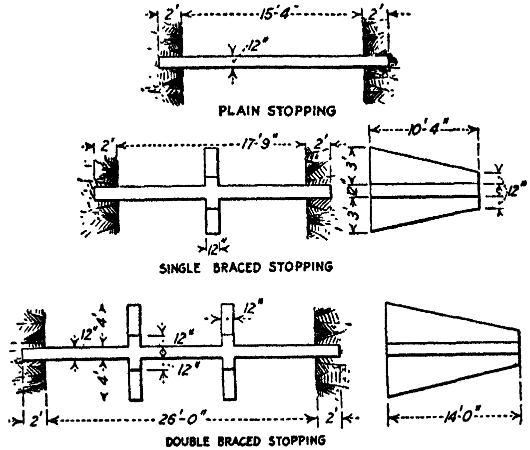


FIG. 14.—P[illegible] and elevation of the Utah Fuel Co.'s concrete overcast.



DETAIL OF COST PER CU. YARD

6 Stoppings and 1 overcast. Total 119 0 Cubic Yards
All hand work Labor $3 15 per 8 Hrs. Carpenter's $3.40 per 8 Hrs.

ITEM	LABOR	MATERIAL	TOTAL
Reinforcing Material (Scrap Iron)	$0 278		$0.278
Gathering & Distributing Gravel in Mine	2.683		2 683
Cement	0 076	$2.659	2.735
Forms	1.936	0.554	2.490
Redistribute Gravel, Mix & place Concrete	3.835 *		3.835
Totals	$8.808	$3.213	$12.021

Gravel hauled in winter. Difficult to redistribute gravel and place concrete on account of old workings

* INCLUDES COST OF MAKING OLD WORKINGS SAFE

FIG. 15.—Plan and cost of reinforced-concrete stoppings at the Utah Fuel Co.'s mines.

as 12 in. A brakeman or driver, knowing he has 30 in. clearance, forgets the door frame is in his way and accordingly at this point a man may be squeezed to death. The required shelter hole is not always provided, and an instance is recorded where a trapper stood back of his door and the heavy iron guard receiving the weight of the cars crushed him so severely that he died.

Should the boy neglect his work the motorman is in great risk of losing his life by running into the closed door. The cross bar over the door often is far below the uniform height of the roof and may easily dash a man's brains out. Doors have been known to take fire and cause serious damage. One at Delagua, Col is thought to have been set on fire by the trapper and been the cause of a destructive fire in which many lives were lost.

Trappers, drivers and workmen leave doors fastened or thrown back and as a result that portion of the mine where the air is short-circuited may be endangered by an accumulation of gas, with a result that is not to be reckoned in dollars and cents but in humanity. It is not advisable to place doors at the foot of steep grades. Even automatic doors are dangerous where a man cannot control his trip, as these doors are not always positive in action.

The penalty under the compensation schedule for neglect to dispense with the door certainly makes it a costly and dubious economy. Doors must be so hung that they will close themselves, must be strongly built, tightly sealed into the roof, and a second door that has the same effect on the ventilation is required.

For neglecting to comply with these specifications a charge of 1c. per $100 of payroll is the penalty provided. Under the rules of the compensation schedule shelter holes must be maintained at all permanent doors. A penalty of 6c. can be charged the operator not complying with this requirement. Doors must be whitened or enclosed lights maintained. A 6c. penalty is provided for non-compliance with the requirement that the trolley wire at doors be guarded on both sides.

A clearance of 30 in. must be provided between the widest part of the motor or cars and the frame of the door or a 12c charge can be made. When these charges are totaled we find that one door can create a maximum of 31c. per $100 payroll. When the total payroll of the mine is computed and this proportion deducted it will give some idea why a permanent door should not be placed, but even when conditions seem to warrant paying the penalty to save the first cost of an overcast it is generally more advisable to erect it.

The initial cost is greater, of course, but in the end the overcast will be found to have paid for itself in saving the expenditures enumerated. With the overcast the clearance can be made ample.

Cost of stone and wood brattices.—An ordinary wooden brattice, single thickness, may be figured as follows, figures as of 1907:

1-in. plank, ½-in. strips and waste, 100 ft. B.M. @ $20 per thousand	$2.00
Labor $1 [illegible], 3 props and caps, 5c.	1.90
Daubing 35c, nails, etc., 25c	0.60
	$4.50

The life of this brattice will ordinarily be from 4 to 5 yr., so that $1 per year per brattice may be taken as the cost of maintenance. Where top is shot, or where there is much draw slate, it will usually be found, particularly in low coal, that there is considerable cleaning out to do to get at the old brattices; which work may readily cost as much as the brattice itself thus making the cost of maintenance $2 per year, instead of $1.

Doors cost in 1907:

Lumber, say 120 ft. B.M. @ $20, $2.40, nails $0.10	$2.50
Labor $3.75, hinges $0.75	4.50
	$7.00

There are in most mines, at least one or two places where a door boy at 75c. per day, or say $200 per year, could be replaced by an overcast at $50 first cost, good for 5 yr., and thus costing only $10 per year, making a net saving of $190 per year for each door boy replaced.

Where the old brattices have to be dug out before they can be replaced, this work may readily amount to $1 per year for each old brattice, or $90 additional. Where the mine workings are so arranged that the brattices on the cross entries will not be in service longer than the life of the first brattices put in, the $160 for maintenance may be dropped. It often occurs that shorter new lines are driven, but this is objectionable, as it leaves no suitable air current along the haulways to carry off the dust from haulage and this is the class of dust most to be dreaded.

Where the brattices have to be dug out the total cost may be $2250, or 1.25c. per ton, for a year's output.

Stone brattices are preferably from 1½ to 2½ ft. thick, depending upon thickness of coal, top, etc., and should be well mined or cut into the rib, particularly in the case of soft coals that spall off readily. Where the coal is low and hard, with good roof and bottom, a 1-ft. wall would seem ample. The following table shows the contents in cubic yards of different sizes of brattices:

FOUR-FOOT COAL .

Width Crosscuts or Breakthroughs in Feet	1 Foot Thick, Cubic Yards	1.5 Foot Thick, Cubic Yards	2 Feet Thick, Cubic Yards
10.0	1.48	2 22	2 96
12.0	1.78	2.67	3.56
14.0	2.08	3.11	4 15
16 0	2.37	3 56	4.74
18.0	2.67	4.00	5.33
20.0	2.96	4.44	5.92
22.0	4.89	6.52	8 15
24.0	3.56	5.33	7.11

SIX-FOOT COAL

Width Crosscuts in Feet	1 Foot Thick, Cubic Yards	1.5 Feet Thick, Cubic Yards	2 Feet Thick, Cubic Yards
10.0	2.22	3.33	4.44
12.0	2.67	4.00	5.33
14.0	3.11	4.67	6.22
16.0	3.56	5.33	7.11
18.0	4.00	6.00	8.00
20.0	4.44	6.67	8.89
22.0	4.89	7.33	9.78
24.0	5.33	8.00	10.67

Where the draw slate forms a durable building stone, double-stone brattices, filled with muck, may be built for $10 to $15 (figures as of 1907), part of the actual cost being properly chargeable to "slate," or entry cleaning. Where stone must be quarried and brought in from the outside, a good brattice, single thickness, may cost anywhere from $15 to $35. Where coke cinder is available, cinder concrete may be used for brattices and overcasts.

Disregarding the cross entries, which may be assumed to be the same in either case, and taking $25 as the cost of a stone brattice, which, particularly for low coal, may be considered a liberal estimate, the costs of wooden and stone brattices may be figured as follows:

Stone brattices:		
Main, 10 brattices @ $25.00 per year............		$250.00
Wood brattices:		
Main, 90 old brattices, maintenance @ $1.00	$90.00	
Main, 10 new brattices @ $4.50..........	45 00	
		135.00
		$115.00

The balance of $115 in favor of wood brattices is equivalent to 0.064c. per ton, for 180,000 tons.

Figuring the cost of maintenance of wooden brattices at $2, instead of $1, the balance in favor of wood brattices is only $25, or equivalent to 0.014c. per ton for 180,000 tons.

Figuring stone brattices at $25, and the maintenance of wooden at $1, a stone brattice will need to be in service 25 yr. to be as cheap as a wood brattice; but figuring the maintenance at $2, will only need to be in service 12½ yr.

Double wood brattices filled with muck may be figured at $10 to $11 each, with say $2 to $2.25 per year, for maintenance. Compared with this a stone brattice will only need to be in service 12½ yr., or allowing $1 per year for digging, say 8 yr., to be as cheap as a wooden brattice. Stone brattices cost somewhat more than the double air-course arrangement, where the latter does not involve shooting considerable top, or equivalent yardage for narrow work, but cost less where yardage must be paid.

Refuge Chambers.—Subsequent to the Cherry (Illinois) mine fire there was a general feeling among the engineers of the country that underground refuge chambers should be established at all mines to prevent a repetition of this insofar as was humanely possible. A paper was presented before the West Virginia Mining Institute in 1910, dealing with this question some excerpts from which are given herewith.

The maximum size of a district to be supplied by a refuge chamber depends somewhat on the geological and other physical conditions presented by the seam and the system of working same. It would seem desirable to have it bear some relation to the maximum number of men employed in a district ventilated by a separate split of air. We will assume that the maximum number of men is one hundred, a not uncommon maximum allowed for a single split of air. As there will be new districts or panels forming while others are being worked out, the average number of men we will figure at 50. A medium-sized mine has about 200 men employed on the day shift, and a large mine about 500. Accepting the average of 50 men in a district, there would be from 4 to 10 live districts in a medium- to large-sized mine, and as many refuge chambers under the system proposed.

Cost of Refuge Chamber

500 ft. 2-in. common pipe casing, in place, say	$50
50 ft. of excess room neck yardage and special entrance, say	50
5 room crosscuts, say, 100 ft. of yardage	50
5 masonry stoppings, at $10	50
6 masonry door frames, at $5	30
6 doors and frames, at $6	36
Sanitary closet and fixtures	15
Wall cases with glass fronts	20
Casks, pails and miscellaneous fittings	10
Food in tins and cans, say	25
6 dry cell electric lights, say $5 each	30
2 safety lamps, at $5	10
1 oxygen resuscitating box, with two cylinders	45
First aid box, medicines and disinfectants	25
Miscellaneous, say	54
Total	$500

To establish these refuge chambers may appear to be a serious task, but if they are planned for in laying out the mine, the cost per ton would be insignificant. Nearly all modern coal developments, as a matter of good engineering, are, or should be preceded by thorough prospecting, both to know the continuity of the seams and to properly plan the mine.

In the following estimate, the room is not considered an added expense, except for the extra length of room neck. The cost of drilling the hole is considered part of the cost of prospecting; the cost of its casing for an assumed depth of 500 ft. is alone considered. The telephone is not regarded as an extra cost.

The foregoing provides for a good equipment; other apparatus mentioned previously should be considered as part of the mine equipment.

If a mine had 6 such stations, the cost underground would be $3000. On the surface the special equipment would vary widely with the physical conditions and regular equipment. If a mine used compressed air, the only additional cost for the stations would be the outside pipe lines. These pipe lines need not be large, as economy of operation would not enter into the calculations. It is probable that all such lines to drill holes of six refuge chambers could be supplied at from $1500 to $2000, under ordinary conditions.

When the mine has an electric plant but not a compressor plant, the additional surface equipment would be the cost of the power lines to the various drill holes and the cost of the small motor-driven fans or compressors. Each drill hole surface instalment could probably be put in at a cost not exceeding $500.

When a mine had neither compressed-air nor electric plant, the cost of instalment would, of course, be much greater, as it would involve a small central plant. However, it may be pointed out that such a plant would be extremely useful, and no doubt pay for instalment on other grounds.

Let us assume that the average total cost of instalment of district refuge chambers figures as much as $10,000, or let us say 5 per cent of the total cost of the mine investment, the possibility of saving a considerable number of lives, if disaster comes, makes it seem a good investment.

Mine sprinkling costs.—At the Sunnyside, No. 2 Mine in Carbon County, Utah, quite an elaborate sprinkling system was installed about 1908. In this system the smallest pipe used on the haulageways was 1½ in. and in the rooms, ¾ or 1 in. pipe is used equipped with a brass hose bibb and kept within 200 ft. of the working face.

In operation, two men are continually employed and they are required to attend to all extensions and repairs, as well as keep the mine sprinkled. In an ordinary mine the necessary work on pipe lines will not occupy more than an average of 2 or 3 hr. of their time per day, the work including extensions in rooms and entries where work is advancing, taking up pipe where work is retreating and roof expected to cave, repairing broken pipe, packing leaky valves, etc.

Each water man will carry with him 150 to 200 ft. of ¾-in. rubber hose, attaching one end to the hose bibbs which are opened by the key or lever, which he carries. He will thoroughly wet down the roof, floor, and sides, or ribs, of all openings accessible, paying especial attention to the vicinity of working faces, brattice and timbers and from time to time wetting down abandoned rooms, etc., which are still open.

The water men are instructed to keep all parts of the mine sufficiently damp so that upon taking a handful of debris from the floor, and subjecting it to pressure of the hand, it will cake and retain its shape after removal of pressure. All brattice must also be kept damp, this requirement alone demanding the presence of water men at least every other day. If fine dust is dampened and then allowed to dry, it is very difficult to penetrate it with water, due to the tendency of this fine dust to form an almost impervious film of dust around globules of water, while if this dust is kept damp, there is no dry fine dust present to imprison and waste the water. Pouring water in quantity, as from pail or barrel, on dry fine dust, is wasteful of water and absolutely ineffective, due to the tendency of the fine dust to form the film above mentioned, while the use at frequent stated intervals of a stream with good pressure finely divided by hose or nozzle moves the dust and permits the water to penetrate the dust particles, not superficially, but through to the floor.

At Sunnyside No. 2 Mine, Carbon County, Utah, the coal vein is 6 to 10 ft. thick, and pitches about 10 per cent. Here the coal is absolutely dry and dust very inflammable, making the wetting of dust absolutely necessary. The above-described sprinkling system is here an unqualified success. The cost of the system was about as follows in 1907:

	Labor	Materials	Total
50,000 gal. redwood tank in place.......	$600	$ 900	$1500
Pressure pump in powerhouse..........	175	825	1000
3000 ft. 3-in. pipe...................	300	750	1050
1000 ft. 2-in. pipe	30	120	150
20,000 ft. 1½-in. pipe................	300	1,600	1900
17,500 ft 1-in. and ¾-in. pipe.........	130	770	900
Hose, hose bibbs, valves, elbows, etc. ..	......	......	500
Total.	......	......	$ 7000

The cost of this plant is somewhat higher than would be necessary elsewhere, due to the fact that very seldom would both tank and pump be required. The above figures, moreover, are for a well-developed mine with distances somewhat well advanced from the surface. Here also, the following of a systematic plan from the start for laying pipe, instead of proceeding hit or miss, would have diminished the amount of pipe required considerably. Even installed at the above cost and on a tonnage of about 300,000 tons per year, the entire plant could be paid for in one year at cost of $2^1/_3$c. per ton and if the cost were distributed over a period of five years, less than ½c. per ton would suffice, which is certainly not prohibitive.

The operating cost in 1908 was about as follows:

One pipe man, 275 days at $2.75 (he also sprinkles)..	$ 756.25
One water man, 275 days at $2.75................	756.25
20,000 gal. of water per day for 275 days at 12c. per 1000 gal................................	660.00
Powerhouse expense including labor running pump, coal for steam, etc.........................	275.00
6 per cent interest on investment.................	420.00
Depreciation 10 per cent........................	700.00
Extensions, repairs, etc. (material)................	500.00
Total................................	$4,067.50

The sum of $4067.50 per year on 300,000 tons amounts to $1^1/_3$c. per ton, a mere trifle compared to benefits derived as will be quickly admitted by any company which installs the system. In the above operating cost the 12c. per 1000 gal. is variable even for this mine, and covers the cost of bringing the water to the storage tank. This cost has at times amounted to $1.50 per 1000 gal., when water was hauled 40 miles in railroad cars and even at this cost sprinkling was kept up, though its extent was somewhat restricted. Depreciation at 10 per cent is liberal as there is little wear on tank or pressure pump, and the pipe will, under ordinary treatment, last 6 or 8 yr., even where laid on coal debris and subject to the corrosive action of acids produced by leaching of coal. Material for extensions, repairs, etc., will amount to more than $500 per year when a mine is new and ground being opened up fast, but during the period of retreating there will not only be no new material needed, but pipe fittings, etc., will accumulate, hence $500 is placed as an average. Power-house expense is the cost of running the pump at the power house to pump water from the tank to the mine, and includes attendance of engineer, who also attends air compressor, dynamos, etc., as well as cost of coal, water, labor, etc., used in generating steam to run pump.

INDEX