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ROCK QUARRYING FOR CEMENT MANUFACTURE

BY

OLIVER BOWLES

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PREFACE.

Under a cooperative agreement between the Bureau of Mines, the United States Geological Survey, and the United States Bureau of Standards, a study of the stone-quarrying industry of the country was begun in 1914. As a result of the investigations conducted, a series of reports dealing with various phases of stone quarrying are being published by the Bureau of Mines. The first publication, "Safety in Stone Quarrying," Technical Paper 111, appeared in 1915, and was followed by "The Technology of Marble Quarrying," Bulletin 106, in 1916, and by "Sandstone Quarrying in the United States," Bulletin 124, in 1917. Because of the wide demand for the reports already published and for others of similar type, additions to this series will be made as rapidly as the necessary data can be prepared. The present report deals with the quarrying of rock for cement manufacture.

The Portland cement industry has attained such magnitude in the United States during recent years that a report covering the quarrying of the raw materials of cement will, it is believed, be of value to the industry. The enormous increase in the use of concrete for buildings, roads, bridges, dams, retaining walls, and many other purposes has resulted in a vast production of cement. About the years 1907 and 1908 such heavy demands were made upon quarries producing the raw materials that the introduction of large and powerful quarry equipment became imperative. Since then the wide use of such equipment has revolutionized quarry methods. Consequently this is an opportune time to investigate efficiency and safety under modern conditions of operation.

Recent rapid advances in the construction of concrete ships undoubtedly gives added impetus to the cement industry, and tends to increase further the demand for greater efficiency in quarry operation.

Mr. Bowles personally visited 64 of the largest cement plants in the United States, from Oklahoma eastward. The results of his investigations comprise the major part of this bulletin.

In the Bureau of Mines bulletins dealing with the quarrying of marble and sandstone, the necessity for careful blasting is strongly emphasized, as the production of sound blocks is desired. In quarrying rock for cement manufacture, however, the rock must be broken into fragments that can be easily handled, hence it is important to

obtain a maximum of fracturing with a minimum of explosive. Effective blasting is, therefore, one of the most important subjects dealt with in this publication. It involves the diameter, distribution, and depth of drill holes, size and nature of the charge, method of firing, quarry conditions that influence blasting methods, and various other factors. Mr. Bowles presents many examples of actual blasting results and their controlling conditions.

During recent years high wages and the scarcity of labor with a greatly increased demand for cement have resulted in important modifications in general quarry methods. The modern tendency is to use labor-saving machinery and methods wherever possible. As a result the method of drilling small holes close together, shooting down the rock in successive benches, and loading by hand has in most quarries given way to steam-shovel loading of rock thrown down in great quantities by large blasts in churn-drill holes.

Certain operators who still retain tripod-drilling and hand-loading methods claim that changing to churn drills and steam shovels would require too great an outlay of capital. The facts brought to light in this investigation indicate that as a general proposition, if the necessary capital is available, it is more economical to scrap the old equipment and purchase modern machinery than to attempt by antiquated methods to compete with modern plants.

Other cement-plant operators having quarries of variable rock claim that the maintenance of a proper mixture requires tripod drilling on low benches and selective hand loading of the rock. By considering every phase of this problem Mr. Bowles makes it clear that in most instances this claim is invalid. Through the application of methods now in successful operation, or a combination of these methods, a uniform mixture may be maintained and the most modern quarry methods be employed.

Methods of transportation within quarries, and from quarries to crushing plants, are discussed in detail, and many improvements are suggested. The chief features reflecting inefficiency are: Needless complication of trackage, especially in quarries with multiple benches; haulage on excessively heavy grades; the combination of too many different systems of transportation in a single quarry; and excessive rehandling of rock.

No attempt is made to discuss problems connected with manufacturing processes except insofar as they are related to quarry methods. The method of mixing the raw materials in order to obtain the proper proportions of the constituents used has, in many instances, a marked effect on the quarry method, and consequently this particular phase of cement manufacture is discussed at considerable length.

Many cost figures, obtained through the kindness of quarry operators, are given, with the principal controlling conditions, to aid quarrymen in judging the efficiency of the methods they use as compared with results obtained by others.

In the type of quarries considered in this publication, quarrymen are subject to many dangers. The discussion of safety problems is not confined to one separate chapter, but wherever machines or methods are discussed the dangers involved are considered and the best means of minimizing such dangers are pointed out.

CHARLES L. PARSONS,
Chief, Division of Mineral Technology.

ROCK QUARRYING FOR CEMENT MANUFACTURE.

By Oliver Bowles.

INTRODUCTION.

As the preface states, this bulletin is the fourth of a series of reports by the Bureau of Mines on different phases of quarrying in the United States. The first part of the bulletin describes the chief types of cements, the growth of the cement industry in this country, and the character of the raw materials used. The bulk of the report deals with quarrying methods and equipment, and gives especial attention to drilling and blasting; a chapter on rock mining and one on prospecting are included. Methods of manufacture are mentioned briefly, as that subject, except as related to quarrying, lies outside the scope of this report; but transportation methods and the effects of the manner of mixing the materials at the quarry or crushing plant on quarrying methods are discussed.

ACKNOWLEDGMENT.

Acknowledgment is hereby given of the cordial cooperation of quarrymen during the field investigations preliminary to the preparation of this bulletin. Special mention should be made of the valuable assistance rendered by Mr. F. J. Leonard, of the Pennsylvania Trojan Powder Co., of Allentown, Pa., and by Mr. S. R. Russell, of E. I. du Pont de Nemours & Co., of Wilmington, Del. Others who rendered noteworthy service are S. S. Smith, of Ironton, Ohio, and R. H. MacFetridge, of Fordwick, Va.

HYDRAULIC LIMES AND CEMENTS.

The presence of lime mortar between the stones forming the ruins of an ancient temple on the island of Cyprus indicates that even primitive man recognized the advantage of using some type of binding or cementing material in building stone structures. Lime mortar, as far as is known, was the first material to be used for this purpose. One by one other types of mortars and cements were invented.

The chief types of cements now used are hydraulic lime, natural cement, puzzolan cement, and Portland cement.

HYDRAULIC LIME.

Hydraulic lime is the product of burning an impure limestone that carries so much lime carbonate in proportion to its silica and alumina content that the burned product will contain considerable free lime in addition to the silicates and aluminates formed. Hydraulic limes, as a rule, contain 75 to 80 per cent calcium carbonate and 20 to 25 per cent clay. The rock is calcined at a temperature slightly above that of decarbonization, and under these conditions silicates, aluminates, and ferrites of lime are formed. The burned product must contain enough calcium silicate to set under water, and also enough free lime to slake when water is added. Hydraulic limes are thus intermediate between natural cements and true limes. Little hydraulic lime is manufactured in the United States, but it is still much used in Europe.

NATURAL CEMENT.

Natural cement is formed by burning an impure limestone containing 15 to 40 per cent of silica, alumina, and iron oxides, at a temperature a little higher than that used in lime burning. Natural cement differs from hydraulic lime in that it does not contain sufficient free lime to cause slaking when water is added. During burning the carbon dioxide is driven off, and most of the lime combines with other constituents, forming silicates, aluminates, and ferrites of lime. As the burned mass will not slake, grinding is necessary. The ground material when wetted hardens or sets rapidly, both in air and under water. Unlike Portland cement, natural cement may vary greatly in composition. Also, it is burned at a much lower temperature than Portland cement, and its specific gravity is somewhat lower. The hydraulic property of natural cement does not depend on its magnesium content, but the presence of magnesium does not impair its value. Twenty-five years ago the cement produced in this country was chiefly natural cement, but for some years this type has formed only about 10 per cent of the total output.

PUZZOLAN CEMENT.

Puzzolan cement is made by mixing slaked lime with volcanic ash or blast-furnace slag. The mixture is finely ground but is not burned after mixing. When the powder is wetted it hardens like natural or Portland cement. It will set under water, and is claimed to give better service in water than in air.

PORTLAND CEMENT.

Portland cement is obtained by burning to the point of incipient fusion an intimate mixture of pulverized materials that contain

calcium carbonate, silica, alumina, and iron oxide in proportions varying within certain narrow limits, and by pulverizing finely the clinker that results. The ground clinker sets under water, and is the strongest and most durable form of hydraulic cement.

HISTORICAL REVIEW OF CEMENT INDUSTRY.

ROMAN PUZZOLAN CEMENT.

Puzzolan cement is the most ancient form of hydraulic cement of which there is any record. The Romans manufactured it in the vicinity of Naples under the name of "Pozzuolana" and employed it in many of the early engineering works in Rome. The cement consisted of volcanic ash powdered and mixed with lime and possessed distinct hydraulic properties. In modern puzzolan cements blast-furnace slag is substituted for volcanic ash. Throughout the Middle Ages even the use of the primitive puzzolan cements seems to have been discontinued, lime mortar being used.

EUROPEAN NATURAL CEMENT.

The natural-cement industry had its beginning in an early discovery that certain limes would set under water. At first this property was thought to be due to the purity of the limestone from which the lime was made. About 1756 Smeaton, an English engineer, in investigating material suitable for constructing the Eddystone Lighthouse, discovered that the reverse was true, for the impure clayey limestones produced limes that would harden under water. The record of Smeaton's experiments[a] was not published until about 1791. His experiments were not carried to the point of making a true cement, but his conclusions regarding the effect of clay in limestone opened the way for further investigation.

In 1796, Parker[b] took out an English patent for a product which he termed "Roman cement," although it was much different from the cement used by the Romans. Parker's cement was made from concretions of clayey and limey material, burned at a temperature higher than that employed in burning lime. The burned product would not slake, but when ground to powder and wetted it would harden, in air and under water. A process for making natural cement of similar type was developed in France at about the same time as in England.

[a] Eckel, E. C., and others, Portland cement materials and industry in the United States: U. S. Geol. Survey Bull. 522, 1913, p. 19.

[b] Eckel, E. C., and others, work cited, p. 19.

EUROPEAN PORTLAND CEMENT.

In 1824, Joseph Aspdin,[a] of Leeds, England, received a British patent for a product which he named "Portland cement," because the set cement resembled the oolitic limestone of Portland, England. The specifications for the new cement were somewhat vague; a very pure limestone was to be burned to lime, the lime mixed with a definite quantity of clay, and the mixture pulverized wet. The wet mixture was to be dried and crushed and then calcined in a vertical kiln and finally the calcine was to be powdered. The patent does not state what proportions of lime and clay should be used, nor at what temperature the mixture should be burned.

The manufacture of Portland cement began in England and on the Continent shortly after the patent was issued. However, at that time natural cement was widely used, and as it could be manufactured more cheaply than the Portland cement, the growth of the latter industry was very slow.

THE CEMENT INDUSTRY IN THE UNITED STATES.

DEVELOPMENT AND USE OF NATURAL CEMENT.

In early days in the United States the greatest demand for cement was in canal construction, and consequently cement enterprises were developed along the courses of canals.

Natural cement was first used in America during the years 1818 and 1819 for the locks and walls of the middle section of the Erie Canal. Its use resulted from investigation of a peculiar type of lime, manufactured for this canal, that would not slake. One of the engineers, Canvass White, who had visited England and there become acquainted with natural cement, tested the material and found that it had hydraulic properties when ground and mixed with water. The material, which was really natural cement, was used in this canal and was much better than the lime that had been used theretofore. White took out a patent on this cement. According to Eckel,[b] the patent right for the State of New York was in 1825 purchased by that Commonwealth for $10,000, and the process rendered free to the citizens of the State.

This discovery was rapidly followed by the discovery of deposits of natural cement rock along the lines of other canals. Meade[c] states that in 1825 cement rock was discovered in Ulster County, N. Y., along the line of the Delaware & Hudson Canal, and in 1826 a mill

[a] Aspdin, Joseph, British patent 5022, issued Oct. 21, 1824.

[b] Eckel, E. C., and others, Portland cement materials and industry in the United States: U. S. Geol. Survey Bull. 522, 1913, p. 21.

[c] Meade, R. K., Portland cement, 2d ed., 1911, pp. 7–8.

was started at High Falls. In 1828 a mill was built in Ulster County, at Rosendale, which became a center of some importance. The cement made here was called "Rosendale cement," and this name is still applied to American natural cement. In 1829 natural cement was manufactured at Shippingport, near Louisville, Ky. In 1836, during the construction of the Chesapeake & Ohio Canal, natural cement was manufactured at Round Top, near Hancock, Md. Natural cement was manufactured for other canals as follows: The Illinois & Michigan Canal, at Utica, in 1838; the James River Canal, at Balcony Falls, Va., in 1848; the Lehigh Coal & Navigation Co. Canal, at Siegfried, Pa., in 1850.

THE PORTLAND CEMENT INDUSTRY.

According to Meade,[a] the first Portland cement made in the Lehigh district in the United States resulted from experiments begun by David O. Saylor in 1866 at Coplay, Pa., where natural cement had been manufactured for some years. After experimenting for some time with mixtures of ordinary cement rock and high-calcium cement rock, a mixture was found which, after being burned and ground, gave results equal to those obtained with imported English and German Portland cements. A feature of interest in connection with this achievement is the fact that the raw materials were very different from those employed in any European cement.

However, introducing the domestic product was difficult because European cements had an established record and American cement had no reputation. Eventually, through liberal advertising and guaranty, a market was established.

In 1872 a plant for making Portland cement from marl and clay was erected near Kalamazoo, Mich. In 1875 Portland cement made from limestone and clay was being manufactured at Wampum, Pa. Up to 1881 six plants were operated for the manufacture of Portland cement, three of which were failures and involved their promoters in heavy loss.

Before the Portland cement industry made any marked progress in the United States methods different from those employed in Europe were invented. Such modifications were necessary on account of the higher wages and cheaper fuel in the United States as contrasted with European countries. To meet these conditions it was necessary to cut down the labor cost for both burning and grinding by means of mechanical improvements. The stationary vertical kiln was, therefore, replaced by the rotary kiln, and the millstones formerly employed for grinding were replaced by modern grinding machinery, such as gyratory crushers and ball and tube mills. Soon

[a] Meade, R. K., work cited, p. 10.

after the rotary kilns were introduced it was found that the materials could be mixed and ground dry and fed directly to the kilns without wetting. This was the beginning of the dry process. In 1891, at Montezuma, N. Y., naturally wet raw materials (marl and clay) were charged into the kilns without preliminary drying. This was essentially the wet process as employed at present. Most of the cement manufactured in the United States is made by the dry process.

GROWTH OF THE INDUSTRY.

The Portland cement industry grew steadily from its beginning until 1908, and since that date its growth has been phenomenal. Few industries in the country can record such wonderful development as is indicated by an increase in production from approximately 3,500,000 barrels in 1898 to over 92,000,000 barrels in 1913. In the year 1900 the production of Portland cement in this country for the first time exceeded that of natural cement, and since then there has been a gradual decline in natural-cement manufacture. Chiefly because of overproduction and the immense stocks of Portland cement on hand, the production in 1914 and 1915 fell a little below that of 1913.[a]

The production of hydraulic cements in the United States from 1818 to 1915, as compiled by Burchard,[b] is shown in Table 1. A graphic illustration of the production of Portland and natural cements and the value of Portland cement for the period 1890 to 1915, prepared by Burchard,[c] is shown in figure 1.

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[a] Burchard, E. F., Cement: Mineral Resources of U. S. for 1915, U. S. Geol. Survey, 1916, pt. 2, p. 192.

[b] Burchard, E. F., work cited, p. 191.

[c] Preliminary estimates by the United States Geological Survey show that the output of Portland cement in 1917 amounted to 93,500,000 barrels, or about 1,100,000 barrels more than in 1916.

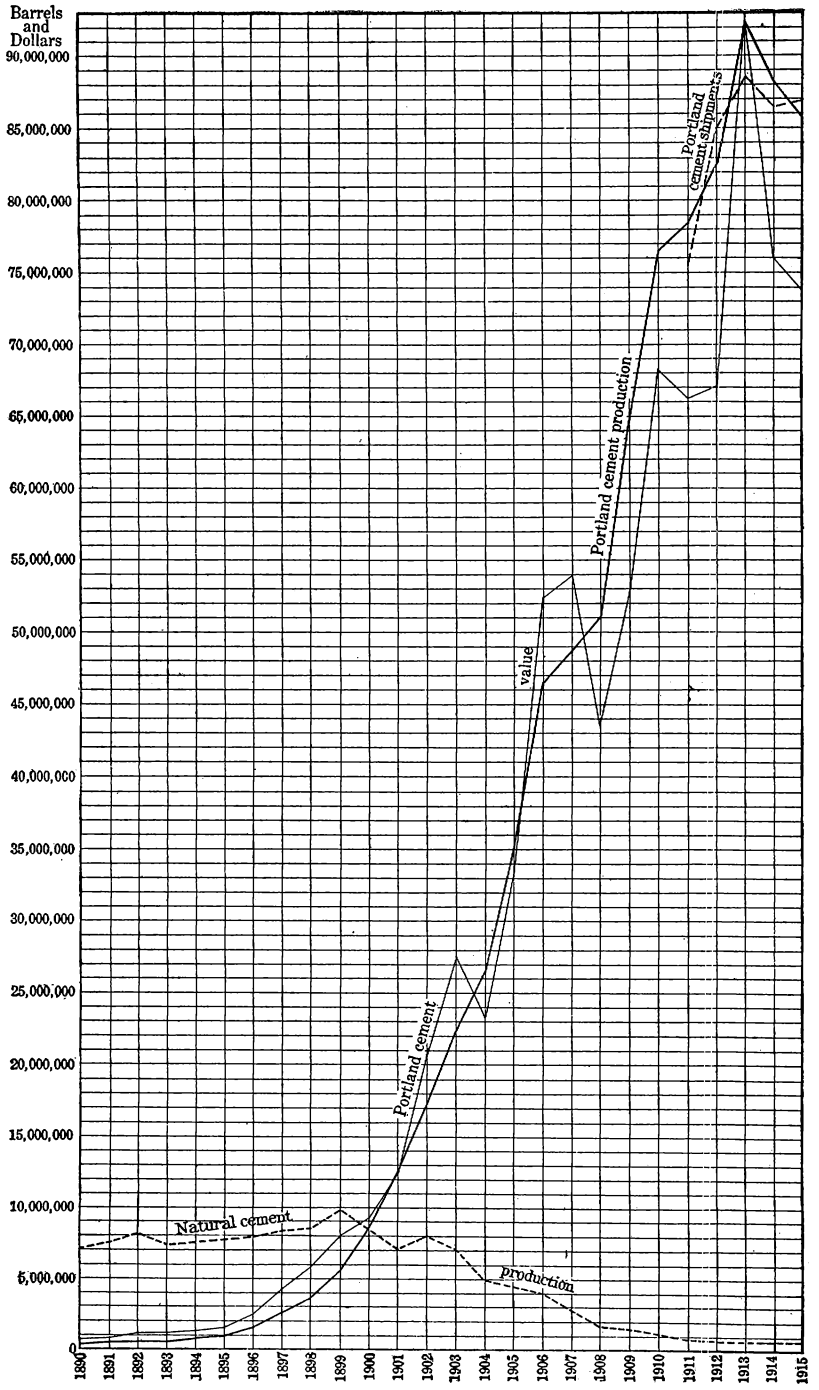


FIGURE 1.—Production of Portland and natural cements and value of Portland cement, 1890–1915, and shipments of Portland cement, 1911–1915. After Burchard.

TABLE 1.—*Total recorded production of principal hydraulic cements in the United States, 1818 to 1915, inclusive, in barrels.*

Year.	Natural cement.		Portland cement.		Puzzolan cement.		Total.	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
1818–1829	300,000	$246,000					300,000	$246,000
1830–1839	1,000,000	850,000					1,000,000	850,000
1840–1849	4,250,000	3,612,500					4,250,000	3,612,500
1850–1859	11,000,000	9,350,000					11,000,000	9,350,000
1860–1869	16,420,000	13,957,000					16,420,000	13,957,000
1870–1879	22,000,000	18,700,000	82,000	$246,000			22,082,000	18,946,000
1880	2,030,943	1,726,707	42,000	126,000			2,072,943	1,852,707
1881	2,440,000	2,379,000	60,000	150,000			2,500,000	2,529,000
1882	3,165,000	3,481,500	85,000	191,250			3,250,000	3,672,750
1883	4,000,000	4,100,000	90,000	193,500			4,090,000	4,293,500
1884	3,900,000	3,510,000	100,000	210,000			4,000,000	3,720,000
1885	4,000,000	3,200,000	150,000	292,500			4,150,000	3,492,500
1886	4,350,000	3,697,500	150,000	292,500			4,500,000	3,990,000
1887	6,692,744	5,186,877	250,000	487,500			6,942,744	5,674,377
1888	6,253,395	4,533,639	250,000	487,500			6,503,395	5,021,139
1889	6,531,876	4,702,951	300,000	500,000			6,831,876	5,202,951
1890	7,441,116	3,822,501	335,500	704,050			7,776,616	4,526,551
1891[a]	7,767,979	3,671,147	454,813	967,467			8,222,792	4,638,614
1892	8,211,181	3,991,455	547,440	1,152,600			8,758,621	5,144,055
1893	7,411,815	3,251,757	590,652	1,158,138			8,002,467	4,409,895
1894	7,563,488	3,635,731	798,757	1,383,473			8,362,245	5,019,204
1895	7,741,077	3,895,424	990,324	1,586,830			8,731,401	5,482,254
1896	7,790,450	4,049,202	1,543,023	2,424,011	12,265	$12,250	9,345,738	6,485,463
1897	8,311,688	3,862,392	2,677,775	4,315,891	48,329	48,500	11,037,792	8,226,783
1898	8,418,924	3,888,728	3,692,284	5,970,773	233,000	198,050	12,344,208	10,057,551
1899	9,868,179	4,814,771	5,652,266	8,074,371	335,000	268,000	15,855,445	13,157,142
1900	8,383,519	3,728,848	8,482,020	9,280,525	365,611	274,208	17,231,150	13,283,581
1901	7,084,823	3,056,278	12,711,225	12,532,360	272,689	198,151	20,068,737	15,786,789
1902	8,044,305	4,076,630	17,230,644	20,864,078	478,555	425,672	25,753,504	25,366,380
1903	7,030,271	3,675,520	22,342,973	27,713,319	525,896	542,502	29,899,140	31,931,341
1904	4,866,331	2,450,150	26,505,881	23,355,119	303,045	226,651	31,675,257	26,031,920
1905	4,473,049	2,413,052	35,246,812	33,245,867	382,447	272,614	40,102,308	35,931,533
1906	4,055,797	2,423,170	46,463,424	52,466,186	481,224	412,921	51,000,445	55,302,277
1907	2,887,700	1,467,302	48,785,390	53,992,551	557,252	443,998	52,230,342	55,903,851
1908	1,686,862	834,509	51,072,612	43,547,679	151,451	95,468	52,910,925	44,477,656
1909	1,537,638	652,736	64,991,431	52,858,354	160,646	99,453	66,689,715	53,610,543
1910	1,139,239	483,006	76,549,951	68,205,800	95,951	63,286	77,785,141	68,752,092
1911	926,091	378,533	78,528,637	66,248,817	93,230	77,786	79,547,958	66,705,136
1912	821,231	367,222	82,438,096	67,016,928	91,864	77,363	83,351,191	67,461,513
1913	744,658	345,889	92,097,131	92,557,617	107,313	97,663	92,949,102	93,001,169
1914	751,285	351,370	88,230,170	81,789,368	68,311	63,358	89,049,766	82,204,096
1915	750,863	358,627	85,914,907	73,886,820	42,678	39,801	86,708,448	74,285,248
Total	234,323,417	149,179,644	856,433,138	810,475,954	4,806,757	3,937,695	1,095,563,312	963,593,293

[a] The figures for 1890 and previous years are estimates made at the close of each year and are believed to be substantially correct. Since 1890 the official figures are based on practically complete returns from all producers.

COMPOSITION OF PORTLAND CEMENT.

VARIATIONS IN COMPOSITION.

Portland cement contains approximately 60 to 70 per cent lime, 20 to 25 per cent silica, and 5 to 12 per cent alumina and iron oxide. The manner in which these various elements combine during the burning process has for many years been somewhat in doubt. Various investigators have studied the problem, and their conclusions do not agree. The chemical analysis of a Portland cement is somewhat difficult, but many reliable analyses have been made. Comparison of the analyses of various cements, all of good quality, shows considerable variation in composition. The major constituents may therefore vary within certain narrow limits without impairing the quality of the product. Basing his opinion on the results of many analyses, Meade[a] has pointed out that cements may vary in composition within the following limits and still conform with standard specifications:

Permissible limits of variation in composition of Portland cement.

Constituent.	Per cent.
Silica	19—25
Alumina	5— 9
Iron oxide	2— 4
Lime	60—64
Magnesia	1— 4
Sulphur trioxide	1— 1.75

According to specifications recently established,[b] the maximum permissible percentage of magnesia has been increased from 4 to 5, and that of sulphur trioxide from 1.75 to 2.

The fact that cements from different plants may vary several per cent in their contents of silica, lime, and alumina and yet give equally satisfactory results is difficult to explain. As the raw materials vary considerably in different localities, evidently the proportions of the various constituents are interdependent to a certain extent. The difficulty consists in determining what Meade terms the "proximate" composition; that is, the manner in which the various elements are combined, and their relation to each other.

INVESTIGATIONS OF "PROXIMATE" COMPOSITION.

As a result of investigations conducted by S. B. and W. B. Newberry[c] they concluded that Portland cement consisted of tricalcic

[a] Meade, R. K., Portland cement, 2d ed., 1911, p. 28.

[b] U. S. Government specifications for Portland cement: Bureau of Standards Circular 33, 3d ed., 1917, p. 27.

[c] Newberry, S. B., and Newberry, W. B., The constitution of hydraulic cements, 1897, p. 7.

silicate and dicalcic aluminate. They found that lime may be combined with silica in the proportion of 3 molecules to 1 and the product still have practically constant volume and good hardening properties, though hardening very slowly. With $3\frac{1}{2}$ molecules of lime to 1 of silica the product is not sound and cracks in water. Also, they found that lime may be combined with alumina in the proportion of 2 molecules to 1, giving a product which sets quickly and shows constant volume and good hardening properties. With $2\frac{1}{2}$ molecules of lime to 1 of alumina the product is not sound. The formula for tricalcic silicate, $3CaO.SiO_2$, corresponds to 2.8 parts of lime by weight to 1 part of silica. The formula for the dicalcic aluminate, $2CaO.Al_2O_3$, corresponds to 1.1 parts of lime by weight to 1 part of alumina. The permissible maximum percentage of lime is, then, the sum of the percentage of silica multiplied by 2.8 plus the percentage of alumina multiplied by 1.1.

The Newberrys deduced for Portland cement the general formula $X(3CaO.SiO_2)+Y(2CaO.Al_2O_3)$, in which X and Y are variable, depending on the relative proportions of silica and alumina present in the clay employed.

However, more recent investigations indicate that these conclusions were incorrect. Rankin and Wright,[a] after a long series of careful determinations, decided that Portland cement clinker made from the pure oxides consisted essentially of a mixture of tricalcic silicate, $3CaO.SiO_2$, dicalcic silicate, $2CaO.SiO_2$, and tricalcic aluminate, $3CaO.Al_2O_3$, with some of the substance $5CaO.3Al_2O_3$, and possibly a small amount of free lime. In a later publication Rankin[b] points out that the lime, and the substance $5CaO.Al_2O_3$, result from imperfect burning. In most cements they occur in small amounts.

A recent publication by the same author[c] describes a series of investigations conducted at the geophysical laboratory of the Carnegie Institute and the results of these studies, with a review of the various steps made by previous investigators leading toward a solution of the problem of the constitution of Portland cement. Rankin emphasizes the fact that the compound tricalcic silicate is the constituent of Portland cement that develops the greatest strength upon hardening. He concludes, therefore, that the essential process in the manufacture of cement is the formation of this compound, and that increase in the strength of the cement may be best brought about by increasing the content of tricalcic silicate. At present the average

[a] Rankin, G. A., The ternary system CaO-Al_2O_3-SiO_2, with optical study by F. E. Wright: Am. Jour. Sci., vol. 39, January, 1915, p. 71.

[b] Rankin, G. A., The constituents of Portland cement clinker: Jour. Ind. and Eng. Chem., vol. 7, June, 1915, p. 473.

[c] Rankin, G. A., Portland cement: Jour. Franklin Inst., June, 1916, pp. 747–784.

normal Portland cement contains only 30 to 35 per cent of this constituent. In order to work out a method of increasing the percentage of tricalcic silicate, further experimental work is needed. The rate of formation of this compound in a series of mixtures must be determined, and also its equilibrium relations at high temperatures.

In the past the manufacture of Portland cement received little scientific investigation, and the general influence of chemical composition on physical properties has been determined almost exclusively by practical experience. As noteworthy contributions to the problem of the constitution of Portland cement are now available, it seems likely that scientific research will play a more important part in the solution of practical cement problems. In fact, the way is now paved for laboratory investigations of practical methods for increasing the strength and the general quality of Portland cement.

OTHER INVESTIGATIONS.

Another field of investigation in which much remains to be done is a determination of the necessary modifications in the composition of cement to render it most suitable for certain climatic or other conditions affecting the structures in which it is employed.

The following examples illustrate problems of this nature. Cox[a] claims that a Portland cement best adapted for use in a tropical climate should have a high silica-alumina ratio, at least 3 parts of silicia to 1 part of alumina. Meade[b] refers to the claim made by numerous authorities that Portland cement containing high percentages of ferric oxide are best adapted for use in sea water, as such cements offer great resistance to the disintegrating action of the salt water. Blatchley[c] claims that the presence of sulphur hastens disintegration of Portland cement exposed to sea water.

RAW MATERIALS OF PORTLAND CEMENT.

ESSENTIAL ELEMENTS.

As pointed out on a previous page, Portland cement contains approximately 60 to 70 per cent lime, 20 to 25 per cent silica, and 5 to 12 per cent alumina and iron oxides. The essential elements are, therefore, calcium, silicon, aluminum, and iron, each of which is in nature usually combined in some way with oxygen. As these are the five chief elements of the earth's crust, it is evident that raw

[a] Cox, A. J., Philippine raw cement materials: Phil. Jour. Sci., vol. 4, May, 1909, p. 217.

[b] Meade, R. K., Portland cement, 2d ed., 1911, p. 36.

[c] Blatchley, W. S., Portland cement: 25th Ann. Rept. Dept. Geol. and Nat. Res. Indiana, 1900, p. 15.

materials for cement manufacture are abundant and may be combined in innumerable ways to give the desired ratio.

LIMITATIONS IN CHOICE OF RAW MATERIALS.

Although the raw materials for making cement are abundant and widely distributed, the necessity for maintaining a low manufacturing cost eliminates many sources of supply. Thus, silica is abundant in nature as the mineral quartz and alumina is found as corundum or emery, but these substances are so extremely hard and difficult to pulverize that they could not be used to advantage in making cement. Many silicates contain lime, iron, silica, or alumina, or combinations of these substances, but most of these also are difficult to quarry and costly to pulverize. Materials that may be employed for commercial manufacture with current prices of cement must, therefore, possess certain characteristics. The chemical composition must be such that a proper mixture may be readily procured and maintained. The physical characteristics must be such that quarrying, crushing, and pulverizing may be done at low cost, and the fusion points must be sufficiently low that clinker may be made without using an excessive amount of fuel.

SUITABLE MATERIALS.

Calcium in its most available state occurs as calcium carbonate in the form of limestone, marble, chalk, or marl. Vast quantities of alumina and silica occur in combination in the form of clay or related materials such as shale or slate. Limestone and clay are frequently mixed and grade into each other. When the clay content is 18 per cent or more the rock is termed argillaceous limestone or cement rock.

In addition to the rocks mentioned certain artificial by-products such as blast-furnace slag, ashes, oyster shells, or precipitated calcium carbonate from alkali works may be used as ingredients of Portland cement.

CLASSIFICATION OF MATERIALS.

The raw materials enumerated may be classified in two groups—the calcareous, supplying the major part of the lime, and the argillaceous, supplying the alumina and silica. Iron oxide may be present in any of the rocks mentioned and also in blast-furnace slag. Classification of the raw materials on this basis is as follows:

Classification of raw materials of Portland cement.

Calcareous.	Argillaceous.
Natural:	Natural:
Limestone.	Clay.
Marble.	Shale.
Chalk.	Slate.
Marl.	Cement rock.
Cement rock.	
Artificial:	Artificial:
Alkali waste.	Ashes.
Oyster shells.	Blast-furnace slag.

Cement rock consisting of both calcareous and argillaceous materials may fall in either group, its classification depending upon the predominating constituent. Most of the cement rock used for Portland cement in this country is argillaceous, requiring the addition of a certain amount of high-calcium rock. However, in a few localities the rock used is rather high in calcium and requires the addition of a small amount of clay or shale.

In ordinary cement mixtures, consisting of limestone or marl with clay or shale, the calcareous materials constitute about three-fourths of the mass, and argillaceous materials one-fourth. In cement rock the proportion may differ widely from this ratio.

COMMON IMPURITIES IN RAW MATERIALS.

SILICA.

In clay or shale silica is one of the essential constituents and can not be termed an impurity, but it may also occur in the limestone.

Silica when present in limestone may occur in three forms—(1) as free silica, in the form of vein quartz, sand, gravel, flint, or chert; (2) in combined form as clay, mica, hornblende, or other silicate mineral; (3) as layers or strata of hydrated silica, the result of chemical precipitation or organic action. The first form, free silica, is to be avoided in limestone. Quartz, flint, or chert are difficult to grind, and unless finely pulverized raise the fusion point of the mixture decidedly. The silicate form, clay, is usually desirable. If clay is present in proper amount for a cement, the mixture constitutes the ideal raw material. Other silicate minerals are to be avoided, some on the basis of undesirable constituents and others because of hardness or high fusibility. The third form, hydrated silica, is of rare occurrence.

IRON.

Iron may be present in oxide, sulphide, carbonate, or silicate forms. This metal, when present as oxide, carbonate, or silicate, is a useful flux, aiding in the combination of the lime and silica in the kiln. The iron sulphide, pyrite, in quantities exceeding 2 or 3 per cent is to be avoided. In any event the ferric oxide content of the finished cement should not exceed 4 per cent.

MAGNESIUM.

Magnesium is usually present in the limestone more than in the clay or shale and can scarcely be termed an impurity in limestone, being rather a constituent that replaces an equivalent amount of calcium. It has been claimed that any considerable proportion of magnesium causes Portland cement to expand and crack after a time. The effect of small proportions of magnesium has been the subject of many investigations, and of late years the opinion has been generally held that magnesium is less injurious than was formerly supposed. This opinion is reflected in the latest standard specifications for Portland cement[a] as prepared at a joint conference of representatives from the American Society of Civil Engineers, American Society of Testing Materials, and a United States Government departmental committee. According to these specifications the permissible percentage of magnesia is increased from 4 to 5.

SULPHUR.

Sulphur may be present as the mineral gypsum or calcium sulphate in the marl or clay, and as pyrite or iron sulphide in the limestone or in the coal used as fuel.

Blatchley[b] claims that sulphur is harmful in cement, especially when the concrete is exposed to sea water, as sulphur in any quantity hastens disintegration. Meade[c] asserts, however, that most of the sulphur in the raw material is burned in the kiln and therefore is not present in the finished cement. On this account it is probable that at least 5 or 6 per cent sulphur trioxide may be present in the raw material without impairing the quality of the cement.

ALKALIES.

According to Blatchley,[d] the Newberrys claimed that the alkalies, soda and potash, were probably of no value in promoting the com-

[a] U. S. Government specifications for Portland cement, Bureau of Standards Circular 33, 3d ed., 1917, p. 35.

[b] Blatchley, W. S., Portland cement: 25th Ann. Rept. Dept. Geol. and Nat. Res. Indiana, 1900, p. 15.

[c] Meade, R. K., work cited, p. 52.

[d] Blatchley, W. S., work cited, p. 15.

bination of lime and silica, whereas the German chemist Schoch, on the other hand, claimed that they act as a flux and are of great benefit in the hardening of Portland cement.

Meade[a] has shown by experiment that alkalies are useful fluxes.

The curtailment by the European war of the foreign supply of potash salts, and the consequent large advances in price, has led a number of cement manufacturers to add potash-recovery equipment to their plants. A brief and concise treatment of this subject is given by Porter.[b] Potash may, therefore, be regarded not only as a useful flux but also as an actual or potential source of a valuable by-product.

CALCAREOUS MATERIALS.

GENERAL SPECIFICATIONS FOR CALCAREOUS MATERIALS.

Calcareous materials to be suitable for cement manufacture should conform with the following conditions:

1. The rock should be free of concretions of iron minerals, should contain little free silica in the form of chert, flint, or quartz veins, and should be free of silicate minerals such as tremolite and diopside.
2. The silica and alumina contents should be sufficiently low and in such ratio that they will not interfere with the desired silica-alumina ratio in the finished product.
3. The rock should be sufficiently low in magnesium that the finished product will not contain more than 5 per cent magnesia.
4. The content of iron should be sufficiently low that the ferric oxide content in the finished cement does not exceed 4 per cent.
5. The sulphur content should be low.

LIMESTONE.

ORIGIN OF LIMESTONE.

Calcium is a common constituent of the rocks of the earth's crust. Calcium is always found in combination with other elements, as carbonate, sulphate, silicate, phosphate, or other forms. By the action of water and of acids at the surface of the earth or within the rocks, calcium is dissolved. Analyses of river waters show that they contain a considerable amount of carbonic acid. This aids in keeping the calcium in solution and it is carried by the rivers to the sea. Marine organisms of various types, such as foraminifera, corals, and mollusks, secrete shells consisting principally of calcium carbonate. As myriads of these organisms, in succeeding

[a] Meade, R. K., work cited, p. 39.

[b] Porter, J. J.: Recovery of potash as by-product in manufacture of cement. Concrete, October, 1917, Cement Mill section, p. 27.

generations, live and die, their shells form vast accumulations at the bottom of the sea. Some of the limestones still show the fossils of which they were formed, but in others all trace of organic origin has been destroyed by the fine grinding to which the shells were subjected before their final consolidation. In addition to this mechanically precipitated material, calcium carbonate may be precipitated chemically from the sea water. Such mechanical and chemical precipitates, when consolidated, may form beds of limestone of great thickness. These may at later periods be elevated above the surface of the water by earth movements, and thus be rendered available for use.

COMPOSITION OF LIMESTONE.

Limestone consists essentially of calcium carbonate, and when pure forms the mineral calcite. However, nearly all limestone contains certain other materials that may be classed as impurities. For a rock to be termed a limestone it must contain at least 50 per cent of calcium carbonate. The principal foreign elements in limestone are silica, iron oxide, alumina, magnesium carbonate, and the alkalies potash and soda. When 30 per cent or more of magnesium carbonate is present, the rock is termed a dolomite. When little silica, alumina, and iron oxide are present the relative proportion of each is of small moment to the cement manufacturer, but if the limestone contains a considerable proportion of these constituents, the relative amount of each should be such that when the shale or clay is added the resulting mixture shall contain silica and alumina in the proper ratio.

In determining the suitability of a limestone for cement making the composition of the shale or clay to be mixed with it must be considered, as the composition of the final mixture depends on the compositions of the original ingredients.

In general, pure limestones are much harder than argillaceous limestones, and consequently are less desirable for making cement. In South Dakota, Alabama, Mississippi, northern Texas, and southeastern Arkansas are certain rotton limestones or chalks that grind easily and are therefore desirable materials for making cement.

VARIETIES OF LIMESTONES.

Special names are applied to certain types of calcareous rocks differing from ordinary limestone in origin, texture, or composition.

Marble is limestone that, through the action of heat and pressure, has become more or less distinctly crystalline, although the term is also used in a commercial sense for any limestone that will take a

good polish. Marl is a term applied to a loosely cemented mass of lime carbonate formed in a lake basin. Calcareous tufa and travertine are more or less compact limestones deposited around springs or along the courses of resulting streams. Oolitic limestone, so called because of its resemblance to fish roe, is made up of small rounded grains of lime carbonate having a concentrically laminated structure. Chalk is fine-grained limestone composed of particles of finely ground shells loosely cemented together. A siliceous or cherty limestone is one that contains considerable silica, an argillaceous limestone is one containing clay, and a carbonaceous or bituminous limestone is one containing carbonaceous matter.

PHYSICAL CHARACTERISTICS OF LIMESTONES.

Limestones vary greatly in such physical properties as hardness, color, weight, porosity, and texture. In color limestones range from pure white to black, depending on differences in chemical composition; in texture they may be amorphous, semicrystalline, or crystalline; and in compactness they vary from the loosely consolidated marls through the chalks to the compact, normal limestones and the still harder marbles. Porous limestone may weigh as low as 110 pounds per cubic foot, whereas the more compact varieties weigh 150 to 185 pounds to the foot.

From the standpoint of the cement manufacturer, porosity and hardness are the more important physical properties of a limestone. Porous limestones are capable of carrying a large content of water, the removal of which requires considerable fuel. The hardness has a direct bearing on the cost of quarrying, crushing, and grinding. Harder rocks pulverize with more difficulty and wear the grinding machinery much more rapidly than softer ones. These two properties, hardness and porosity, counteract each other to a great extent, for the harder limestones are usually of low porosity and vice versa.

The attitude of the beds and the presence or absence of open bedding planes or joints are structural features that have a direct bearing on the system of quarrying. As these chiefly affect drilling and blasting methods, they are considered in detail under "Drilling" and "Blasting" on pages 42 to 51.

CALCAREOUS MATERIALS USED IN CEMENT MANUFACTURE.

COMPACT LIMESTONE.

In the early days of Portland cement manufacture the amount of hard limestone used was small, argillaceous limestone, the so-called cement rock, being chiefly employed. When the process of making cement from hard limestone and shale or clay was placed on a profitable basis, this branch of the industry developed rapidly. Hard

limestone is the principal ingredient in practically 57 per cent of all the Portland cement manufactured in this country, and (see Table 2, p. 26) is by far the most widely used calcareous material in making cement. Hard limestone is usually of reasonably low porosity, but some types are difficult to pulverize.

CEMENT ROCK.

In certain parts of the United States, notably in the Lehigh River Valley of eastern Pennsylvania and western New Jersey, in the Shenandoah Valley of Virginia, and near Glens Falls, N. Y., are limestones having a high clay content. As such rocks contain all the chief constituents of Portland cement, they have been termed "cement rock." Cement rock was the chief source of raw material in the early days of the Portland cement industry in the United States and still constitutes an important source.

The cement rock quarried in the Lehigh district is mostly derived from the middle beds of the Trenton formation. In most places the beds dip 15° to 25°, usually northwest, but at a few quarries, particularly in New Jersey, the dip is much steeper. Most of the cement rock runs 60 to 70 per cent lime carbonate, hence the addition of a small amount of high-calcium limestone is necessary. The limestone may be obtained from beds in the lower part of the Trenton formation or from certain low-magnesium beds in the underlying Kittatinny formation. High-calcium limestone is shipped from outside points to a number of plants. Near Bath and Nazareth, Pa., the cement rock may carry 70 to 80 per cent of lime carbonate and the addition of limestone be unnecessary. In some places, on the other hand, the rock is so high in lime that the addition of a small amount of clay or shale is necessary. A high-calcium cement rock, requiring a small addition of clay, is usually more desirable than one requiring the addition of limestone, because the cement rock is usually overlain with clay, which is readily available, whereas the limestone may have to be shipped in.

In a few places the cement rock approximates so nearly the desired proportions for a cement mixture that by careful and judicious quarrying no limestone or shale is necessary. Such a rock, properly proportioned in nature, constitutes the ideal raw material for cement.

Cement rock is commonly intersected by close bedding seams and joint planes, consequently when great masses are hurled down in blasting they readily break into small pieces. Not only does cement rock quarry easily, as a rule, but it offers certain advantages in milling. As the particles of clay and limestone are intimately mixed by nature, the rock does not need to be so finely ground as lime-

stone must be for intimate mixing with clay or shale. Moreover, cement rock is, as a rule, much easier to grind and pulverize than hard limestone. Such rock may, therefore, be regarded as one of the most desirable raw materials for making cement.

MARBLE.

Marble is an extremely compact or crystalline form of limestone. It is usually much harder than ordinary limestone, and consequently is somewhat difficult to quarry and pulverize. Marble is not commonly used as a cement material. It is employed by a company operating near Union Bridge, Md.

CHALK.

Chalk, being much softer than limestone and easily excavated and pulverized, is a desirable raw material for cement manufacture. The chief disadvantage of chalk is its high porosity, which permits excessive absorption of water during rainy seasons. Where chalk is employed large covered bins in which a supply can be stored for use during wet seasons is desirable.

MARL.

Marl is soft, unconsolidated lime carbonate that has been deposited in the bottoms of existing or extinct lakes. The lime carbonate is precipitated from solution in the lake water by the agency of certain algæ or water plants. The chief impurities in marl are clay, organic matter, and magnesium carbonate. The clay content, if of suitable composition and limited in amount, is not regarded as undesirable. The organic matter is burned out in the kilns and has little or no effect on the finished product.

As most marl deposits are partly or completely submerged, the marl is won by dredging. The wet material may be hauled to the plant in cars, or may be pumped as a slurry. The process of winning marl is relatively cheap, but this advantage is largely counteracted by the cost of fuel for driving off the water.

ALKALI WASTE.

Limestone is employed in the manufacture of caustic soda from common salt. The calcium is precipitated in carbonate form and removed as waste. Where the Leblanc process is employed the waste contains too large a proportion of sulphides to be used as an ingredient of cement. The ammonia process, however, yields lime carbonate relatively free from sulphur.

The adaptability of the precipitated carbonate for use in cement manufacture depends largely on the nature of the limestone. If dolomitic limestone is used the resulting waste will be high in magnesium, but if high-calcium limestone is used the waste is of suitable character.

Alkali waste is employed to some extent as a Portland cement constituent, notably at Wyandotte, Mich.

OYSTER SHELLS.

It has been determined by analyses that ordinary commercial oyster shells have about the same composition as the purest limestones ordinarily quarried. The available supply of shells in certain coast and Gulf regions is sufficient to justify their use for cement manufacture. They are now employed for this purpose near Norfolk, Va., and Houston, Tex.

ARGILLACEOUS MATERIALS.

GENERAL SPECIFICATIONS FOR ARGILLACEOUS MATERIALS.

Argillaceous material for making cement should conform with the following specifications:

1. The material should be free from iron concretions and from silica in the form of flint and quartz.
2. The silica content should form 60 to 70 per cent of the mass.
3. The magnesium content should be sufficiently low to conform with standard specifications.
4. The presence of iron oxide is desirable as a flux, but the content of iron oxide in the finished product should not exceed 4 per cent.
5. For the manufacture of white cement, the ferric oxide content should be less than 0.5 per cent.
6. The sulphur content should be low.

CLAY.

DEFINITION OF CLAY.

Clay may be defined as an earthy substance consisting of a base of kaolin mixed with a greater or less proportion of mineral or organic impurities. Kaolin is a hydrous silicate of alumina, its composition being represented by the formula $Al_2O_3.2SiO_2.2H_2O$. Kaolin rarely occurs pure in nature, the more common impurities being silica sand, lime carbonate, iron oxide, and carbonaceous material.

ORIGIN OF CLAY.

Clays are all of sedimentary origin, and are derived from the disintegration and alteration of igneous, metamorphic, or sedimentary

rocks. Feldspar, pyroxene, and hornblende, which consist largely of silica and alumina, are the chief sources of kaolin that originates from igneous or metamorphic rocks. The more soluble elements are gradually leached out, leaving behind the less soluble silica and alumina.

In various places vast beds of clay have been formed by the disintegration of argillaceous limestones. Water containing carbon dioxide slowly dissolves and removes the calcium and magnesium carbonates, while the insoluble clay remains and accumulates.

VARIETIES OF CLAY.

RESIDUAL CLAY.

Residual clay is formed through the disintegration of rocks in place, and is found replacing the parent rock and resting on unaltered masses of it. Residual clay results from the alteration of certain components of the rock mass, and those parts that have not altered to clay are usually present as impurities. Thus the residual clay from the disintegration of a granite may contain grains of unaltered quartz.

TRANSPORTED CLAY.

Most clays are not deposited in the localities where they originated, but have been transported by the agency of water or of ice. Alluvial clay is that which has been transported by streams, and deposited along their courses, on their flood planes or in the basins into which they flow.

Another type of transported clay is glacial clay. During the glacial period most of the northern part of America was covered with ice which in passing over the surface of the rocks eroded them. The loose sand, gravel, and clay were carried along by the ice and deposited at various distances from their original sources.

IMPURITIES IN CLAY.

As previously stated, pure kaolin is rare. Residual clays usually contain unaltered or partly altered grains of the parent rock. Transported clay may be freed of original impurities, but, on the other hand, impurities may be added in transit.

As clays are commonly formed through the disintegration of granites and related rocks, feldspar, mica, and quartz are common impurities. Feldspar acts as a flux and thus lowers the fusion point of a clay. Mica also acts as a flux, but fuses at a slightly higher temperature than feldspar. Quartz in any considerable amount is undesirable in clay to be employed for making cement.

Most clay contains iron. The metal may be present as the oxide in such minerals as magnetite, hematite, and limonite, or as the sulphide in the mineral pyrite.

Calcium and magnesium carbonates are common constituents of clay. As pointed out on a previous page, a high proportion of calcium carbonate may be present, forming a cement rock. Magnesium carbonate is undesirable except in small amounts.

Gypsum or calcium sulphate is present in many clays. Its presence may increase the amount of sulphur in the raw mix beyond the permissible percentage.

The alkalies, potash and soda, are commonly present. They act as fluxes, tending to lower the fusion point of the clay.

COMPOSITION OF CLAY FOR CEMENT MANUFACTURE.

In judging the suitability of a clay for making cement, the condition of the free silica should be noted with care. Practically all clay contains some uncombined silica in the form of quartz sand or pebbles. The proportion of free silica should be low, and it is desirable that the sand, if present, should be in a finely divided condition. Meade[a] claims that clay containing more than 5 per cent of sand in grains that will not pass through a 100-mesh sieve, is unsuitable for cement manufacture.

The ratio between the silica, alumina, and iron contents is important. The ratio of silica to alumina should range between 3 to 1 and 4 to 1. The clay should not contain more iron oxide than alumina, the best proportion being about 1 of iron oxide to 3 of alumina. The sum of the alumina and the iron oxide contents should not be more than one-half the silica content, and the nearer it approaches one-third, the better. If the percentage of alumina is too high, the cement sets too fast, and also is weak. Plants have been known to suffer heavy financial loss through the rejection of inferior cement on account of the clay being low in silica. This difficulty could be easily overcome by adding free silica in some form to the mixture.

The sum of the silica and alumina percentages in hydrous clay should not exceed 87 per cent, thus leaving at least 13 per cent for combined water and fluxes like ferric oxide, potash, and soda. The presence of these fluxes, especially ferric oxide, promotes the formation of an easily fusible magma. The lower the fusing point of the magma the more readily will the cement mixture sinter or clinker, and the lower will be the cost of burning. However, the content of alkalies should not be more than 3 per cent, as an excess of these is likely to cause unsoundness and quick setting.

[a] Meade, R. K., Portland cement, 2d ed., 1911, p. 54.

SHALE AND SLATE.

ORIGIN.

Shale consists of mud or clay that has been deposited in water and subsequently through pressure been consolidated into rock. The pressure may be due either to the weight of overlying rock formed by subsequent deposition, to forces to which the rocks were subjected during mountain-forming periods, or to both of these causes combined. Under further pressure and increased temperature the shale may be changed in character and structure, chemical changes may take place, recrystallization be brought about, and a perfect cleavage or fissility developed. The resulting rock is termed slate.

COMPOSITION.

As shale is merely solidified clay, shale to be used for cement manufacture should conform with all the conditions prescribed for clay. Slate has usually undergone some recrystallization, but its chemical composition is essentially that of the original clay from which it was formed. Hence, all that has been stated relative to impurities and relative proportions of clay constituents is equally applicable to shale and slate.

PHYSICAL CHARACTERISTICS.

Clay is usually earthy, but shale and slate are more firmly consolidated and more closely resemble hard rocks such as limestone or cement rock. This difference in physical character is of some importance in the selection of suitable material for cement. Clay is the most suitable argillaceous material to mix with marl, as the two materials are of similar physical character. If clay is mixed with coarsely ground limestone the two materials tend to segregate; that is, if the mixture falls in a stream onto a conical heap below, as in filling a storage bin through a spout, the finely divided clay tends to remain in the center of the pile and the limestone fragments to roll to the sides. This separation of materials tends to cause lack of uniformity when the mixture is drawn from the bin. If shale or slate is mixed with limestone the constituents are more nearly alike in character and such segregation is not so likely to occur. It is better, therefore, to mix substances of like physical characteristics—shale or slate with limestone or cement rock, and clay with marl.

Where limestone deposits are close to slate quarries, waste slate might be utilized as a cement constituent. The utilization of the waste would thus be beneficial to both industries.

BLAST-FURNACE SLAG.

True Portland cement may be made from a mixture of ground blast-furnace slag and limestone burned together. Slag consists essentially of lime, silica, and alumina, with small contents of iron oxide, magnesium, and sulphur. As an ingredient for cement slag may, therefore, be regarded as very impure limestone, or as very calcareous clay from which the carbon dioxide has been driven off.

The suitability of a blast-furnace slag as a cement constituent depends largely on the nature of the limestone used for flux in the blast furnace. A magnesium limestone or dolomite naturally produces a slag high in magnesium. Hence only slags from furnaces that employ high-calcium limestone for flux can be used for making Portland cement.

ASHES.

Coal ashes are used to some extent to replace or supplement clay or shale in cement mixtures. It is claimed by one company that ashes tend to decrease the rate of setting in the finished cement.

COMBINATIONS OF RAW MATERIALS.

In a few localities cement rock has been found that contains lime, silica, alumina, and iron oxide in almost the proper proportions for a cement mixture. However, such rock is rare, and as a rule a mixture of calcareous and argillaceous materials is necessary. The various combinations of raw materials employed in cement plants in the United States are outlined in the following paragraphs.

CEMENT ROCK AND LIMESTONE.

Most of the cement rock now used in the Lehigh district is so low in calcium as to require the addition of a small amount of high-calcium limestone. In some quarries suitable limestone is found underlying the cement rock, whereas for others it must be shipped in from outside points. Cement rock and limestone are also available in Virginia and in some of the Pacific Coast States, but are not used extensively.

CEMENT ROCK AND SHALE.

In a few quarries in the Lehigh district and in one quarry near Glens Falls, N. Y., the cement rock contains an excess of calcium, and the addition of a small amount of shale is necessary.

HARD LIMESTONE AND CLAY OR SHALE.

As stated on page 18, hard limestone is the chief calcareous constituent of Portland cement made in this country. The combination of hard limestone with clay or shale is practiced by cement plants in many States and in widely separated localities. The substitution of ashes for part of the shale has been mentioned.

CHALKY LIMESTONE AND CLAY.

Chalky limestone is easier to quarry than hard limestone, and carries less water than marl. It is available for cement manufacture in three localities: (1) Parts of Texas, Oklahoma, and Arkansas; (2) parts of Alabama and Mississippi; and (3) parts of North Dakota and South Dakota.

MARL AND CLAY.

Marl and clay are used for making cement in Ohio, Michigan, Indiana, and central New York.

ALKALI WASTE AND CLAY.

Precipitated calcium carbonate from alkali works is a limited source of supply as a cement constituent and is now utilized only at Wyandotte, Mich.

SLAG AND LIMESTONE.

Blast-furnace slag and limestone are used for the manufacture of cement in Illinois, Ohio, and Pennsylvania.

OYSTER SHELLS AND CLAY.

Oyster shells, which have about the same composition as limestone, are (see p. 20) employed in a few localities where the supply of shells is adequate.

RELATIVE IMPORTANCE OF MATERIALS USED.

Table 2, compiled by Burchard,[a] indicates the relative importance of the chief sources of raw materials for the period 1898 to 1914, inclusive.

[a] Burchard, E. F., Cement: Mineral Resources of U. S. for 1914, pt. 2; U. S. Geol. Survey, 1915, p. 232.

TABLE 2.—*Production, in barrels, and percentage of total output of Portland cement in the United States according to type of material used, 1898–1914.*

Year.	Type 1. Cement rock and pure limestone.		Type 2. Limestone and clay or shale.		Type 3. Marl and clay.		Type 4. Blast-furnace slag and limestone.	
	Quantity.	Per-centage.	Quantity.	Per-centage.	Quantity.	Per-centage.	Quantity.	Per-centage.
1898	2,764,694	74.9	365,408	9.9	562,092	15.2	.....	.....
1899	4,010,132	70.9	546,200	9.7	1,095,934	19.4	.....	.....
1900	5,960,739	70.3	1,034,041	12.2	1,454,797	17.1	32,443	0.4
1901	8,503,500	66.9	2,042,209	16.1	2,001,200	15.7	164,316	1.3
1902	10,953,178	63.6	3,738,303	21.7	2,220,453	12.9	318,710	1.8
1903	12,493,694	55.9	6,333,403	28.3	3,052,946	13.7	462,930	2.1
1904	15,173,391	57.2	7,526,323	28.4	3,332,873	12.6	473,294	1.8
1905	18,454,902	52.4	11,172,389	31.7	3,884,178	11.0	1,735,343	4.9
1906	23,896,951	51.4	16,532,212	35.6	3,958,201	8.5	2,076,000	4.5
1907	25,859,095	53.0	17,190,697	35.2	3,606,598	7.4	2,129,000	4.4
1908	20,678,693	40.6	23,047,707	45.0	2,811,212	5.5	4,535,300	8.9
1909	24,274,047	37.3	32,219,365	49.6	2,711,219	4.2	5,786,800	8.9
1910	26,520,911	34.6	39,720,320	51.9	3,307,220	4.3	7,001,500	9.2
1911	26,812,129	34.1	40,665,332	51.8	3,314,176	4.2	7,737,000	9.9
1912	24,712,780	30.0	44,607,776	54.1	2,467,368	3.0	10,650,172	12.9
1913	29,333,490	31.8	47,831,863	51.9	3,734,778	4.1	11,197,000	12.2
1914	24,907,047	28.2	50,168,813	56.9	4,038,310	4.6	9,116,000	10.3

PROPORTIONING THE MIXTURE.

LIMITATIONS OF DISCUSSION.

Meade [a] has discussed the proportioning of raw materials at length, and readers particularly interested in the subject may consult the publication cited, or others of similar character. In a publication like the present one, dealing primarily with the process of winning the raw materials from the earth, the principles and the method of obtaining a suitable proportion of raw materials can be dealt with only in a general way.

NEED OF DEFINITE PROPORTIONS.

In discussing the composition of Portland cement it has been pointed out that the properties of cements may be materially changed by very small variations in the proportions of the essential elements. For example, cements high in lime or silica are slow-setting, whereas those high in alumina are quick-setting. Variations beyond certain narrow limits may cause the cement to be of poor quality. The cement manufacturer must, therefore, control his mixture by certain definite rules.

FORMULAS FOR CALCULATING MIXTURE.

Certain general formulas are applicable to most cement materials, though peculiarities in composition may require minor modifications.

[a] Meade, R. K., Portland cement, 2d ed., 1911, pp. 69–93.

The Newberrys [a] based their formula on the presumption that Portland cement consisted of tricalcic silicate and dicalcic aluminate. Tricalcic silicate ($3CaO.SiO_2$) consists of 2.8 parts of lime by weight to 1 part of silica; and dicalcic aluminate ($2CaO.SiO_2$), 1.1 parts of lime by weight to 1 part of alumina. The maximum permissible percentage of lime, then, is equal to the sum of the percentage of silica multiplied by 2.8 plus the percentage of alumina multiplied by 1.1.

The following rule was deduced: Multiply the percentage of silica by 2.8, and the percentage of alumina by 1.1, add the products, and the sum will be the number of parts of lime required for 100 parts of clay.

As 2.8 parts of lime correspond to 5 parts of lime carbonate, and 1.1 parts lime correspond to 2 parts of lime carbonate, it follows that five times the percentage of silica plus twice the percentage of alumina equals the number of parts of carbonate of lime required for 100 parts of clay.

It has been established that the ratio between the percentage of lime and the combined percentages of silica, alumina, and iron should be within the limits 1 to 1.8 and 1 to 2.2. Many chemists assume a fixed ratio of 2, though others obtain better results by slightly increasing or decreasing this ratio.

After the raw clay and limestone have been analyzed the proportions of each necessary for any given ratio may be determined as follows:

Let—

M =the ratio of lime to silica, alumina, and iron;
S =the oxides of silica, alumina, and iron in the clay;
S_1=the oxides of silica, alumina, and iron in the limestone;
C =the calcium oxide in the limestone;
C_1=the calcium oxide in the clay.

Then—

$$\frac{\text{Limestone}}{\text{Clay}}=\frac{MS-C_1}{C-MS_1}.$$

The chief defect of this formula is that the relative proportions of silica, alumina, and iron oxide are not taken into consideration, but for dealing with well-balanced raw materials it is satisfactory. More accurate methods for calculating these proportions are given by various authors.

CALCULATION ON BASIS OF FIXED LIME STANDARD.

A formula such as the one given is used to determine the best mixture of any given raw materials and to check the mixture from

[a] Newberry, S. B., and Newberry, W. B., The constitution of hydraulic cements, 1897, p. 7.

time to time. However, the "fixed lime standard" is found to be more practical in actual millwork. After experience has shown that a certain percentage of lime carbonate gives satisfactory results, the operator maintains this lime standard as nearly as possible, on the assumption that the ratio of other constituents is approximately self-adjusting. The method is satisfactory where the raw materials are of fairly constant composition, but if marked fluctuations in composition are likely to occur the mixture should be checked frequently by the application of a more accurate formula.

In the Lehigh district the percentage of carbonate of lime in the raw mixture varies from 74.5 to 75.5, and as the limestone and cement rock are of fairly constant composition, the fixed lime standard is in common use.

Meade[a] gives the following simple formulas for determining the proper mixture on this basis. The first formula is for determining the percentage of limestone to be added to a given cement rock or clay to make a given mixture.

Let—

X=percentage of limestone necessary,
L=percentage of $CaCO_3$ in limestone,
R=percentage of $CaCO_3$ in cement rock or clay,
M=percentage of $CaCO_3$ desired in mixture,

Then—

$$X=\frac{M-R}{L-M}\times 100$$

The second formula is for determining the percentage of shale or clay to be added to a given marl or limestone to make a given mixture, and is as follows:

Let—

X=percentage of clay or shale necessary,
C=percentage of CaO in clay or shale,
L=percentage of CaO in marl or limestone,
M=percentage of CaO in desired mixture.

Then—

$$X=\frac{L-M}{M-C}\times 100$$

FLUXES.

The addition of certain substances to promote combination of silica and lime in the kiln is necessary in certain plants working on poor raw materials, such as cherty limestone or high-silica and low-alumina clay. The common fluxes used in cement making are iron oxide, fluorspar, cryolite, and certain alkaline compounds. Where

[a] Meade, R. K., work cited, p. 80.

fairly good raw materials are available, the use of fluxes usually should be avoided, as the disadvantages may more than outweigh the advantages.

PROCESSES OF MANUFACTURE.

NATURAL CEMENT.

The cement rock used for the manufacture of natural cement requires no pulverizing or mixing prior to burning. It is burned in vertical kilns similar to those employed in lime burning, though usually larger. The kilns are operated continuously, the rock and fuel being fed in at the top, mixed together or in alternating layers. The stone is burned at a temperature a little higher than that employed in lime burning. The burned stone, or clinker, is drawn from the bottoms of the kilns, cooled, and ground to powder.

PORTLAND CEMENT.

During the early period of the Portland cement industry all European and American plants used the wet process only. As stationary kilns were used the material had to be wetted and molded into bricks to give a free draft in the kiln.

With the introduction of the rotary kiln the dry process was made practicable. The dry process involves, first, the crushing and drying of the limestone or cement rock and the addition of the necessary amount of clay, shale, or limestone, whichever is used, these being also crushed and dried. The mixture is then finely pulverized. With cement rock satisfactory results are obtained if the material is sufficiently pulverized that 85 per cent passes through a 100-mesh sieve. If limestone and shale are used, finer grinding is necessary; for the best results, 95 to 98 per cent should pass through a 100-mesh sieve. The properly proportioned and finely pulverized mixture is then fed into the kilns.

In the wet process the materials are mixed and usually pulverized in a wet condition. The mixture known as "slurry," having the consistency of thin mud, is fed by pumps or by screw conveyors into tanks, where it is throughly mixed either by mechanical or air agitation. The mixture enters the kilns as slurry, the water being driven off by evaporation in the upper part of the kilns.

Modern rotary kilns are 5 to 9 feet in diameter and 60 to 200 feet long, the longer kilns representing a more modern development. They are cylindrical in form and are made of sheet steel lined with fire brick. They slope gently downward toward the firing end. Fuel consisting of powdered coal, gas, or oil is blown in through a small pipe at the lower end of the kiln. The cement mixture is fed in at the

upper end, and as the kiln rotates slowly the mixture works its way toward the lower end, where it is discharged as clinker burned to the point of incipient vitrification.

The clinker is cooled and seasoned for some time before grinding. Two to 3 per cent of retarder, usually gypsum, is added to the clinker to prevent too rapid setting of the cement, and the mixture is pulverized and conveyed to the stock house. To prepare for shipment, the cement is packed in barrels of 380 pounds, or bags of 95 pounds, each. Automatic packing devices are in common use.

PROSPECTING.

NEED OF THOROUGH PROSPECTING.

It is extremely unwise to begin quarrying rock for cement manufacture without first driving enough prospect holes to determine whether a sufficient supply of material is available. The opening of a new quarry involves considerable expense for equipment and labor, and should, therefore, not be undertaken without definite assurance of practical operation.

CHEMICAL COMPOSITION OF RAW MATERIALS.

The matter of supreme importance to the cement manufacturer is the chemical composition of the rock. In this respect there is a wide difference between prospecting for rock to be used in making cement and for rock to be used for structural purposes. When rock is to be quarried for building purposes, the soundness, color, texture, and condition of cementation are of more consequence than the chemical composition. On the other hand, the physical properties of rock quarried for cement manufacture, although they may have considerable influence on the method of blasting, are secondary in importance to chemical composition.

In prospecting to determine the suitability of a deposit of clay, limestone, or cement rock, for cement manufacture the deposit should be sampled carefully and systematically in order that the samples may fairly represent the entire deposit. The less homogeneous the deposit is, the more samples should be taken and the more carefully should the work be done. Neglect to make a proper preliminary examination of a deposit of either clay or limestone before beginning quarrying may result in the loss of thousands of dollars. The substances that must be most carefully avoided are magnesium and free silica, although small percentages of either may usually be safely allowed.

INTERRELATION OF VARIOUS CONSTITUENTS.

In determining the suitability of raw cement materials the composition of the limestone or marl and that of the clay or shale must not be considered alone, for each must be considered in its relation to the other. Raw cement material usually consists of a mixture of two substances, and a deficiency in one may be compensated by a corresponding excess in the other. Thus a low silica shale may be satisfactory when mixed with a siliceous limestone.

In limestone the proportion of silica and alumina to calcium may vary widely and still the rock may be satisfactory. Thus limestone having as high as 98 or 99 per cent calcium carbonate may be satisfactory if the necessary shale or clay to mix with it is readily available. On the other hand, the rock may be very low in calcium—it may even contain less calcium than is necessary for a cement mixture—and still be satisfactory if high-calcium limestone to mix with it is available. Hence considerable latitude may be permitted in the composition of the main rock mass, the limits of variation being governed by the availability of the raw material that must be added in order to give the proper mix.

METHODS OF PROSPECTING FOR CLAY.

Bleininger[a] suggests that in sampling clay, if samples taken from an outcrop show a satisfactory composition, the surface should be removed and samples taken from within the bank. If further outcrops of the same deposit can be found they should all be sampled. The area comprising the clay deposit should then be surveyed and divided into squares whose sides should range between 50 and 100 feet in length, depending on the thickness of the deposit. If the deposit is thin and covers a wide area the squares should be larger than where it is thick. At the center of every square a test hole should be drilled and the material brought up by the drill put aside for analysis.

If the clay is reasonably soft and free from stones, the drilling may be done with a soil auger. For harder material a simple type of churn drill, operated either by hand or by power, may be employed.

EXAMPLES OF INSUFFICIENT PROSPECTING.

As an example of loss due to insufficient prospecting, attention may be directed to a certain Pennsylvania quarry where a depth of 10 to 15 feet of overburden was removed from a considerable area of rock, and it was subsequently found that the rock was useless for the manu-

[a] Bleininger, A. V., The manufacture of hydraulic cements: Geol. Survey of Ohio, ser. 4, Bull. 3, 1904, p. 103.

facture of cement. The drilling of a single hole would have revealed the inferior nature of the rock and thus saved the expense of useless stripping.

AVAILABILITY OF ROCK LEDGE.

Next in importance to the composition of the rock is its availability for quarrying. The thickness and character of the overburden, and the effect of continued development on the depth of overburden, should be carefully ascertained. The relation of the thickness of available stone to depth of overburden is important, for thin beds would necessitate wide lateral development and, therefore, the removal of relatively much more stripping than where deep quarrying may be employed.

If limestone and shale quarries are not close together, it is desirable to build the mill near the limestone quarry, as about three or four parts of limestone are used to one part of shale.

UTILIZATION OF GRAVITY SYSTEMS.

The quarry and the cement plant should, if possible, be so situated that the raw materials can be conveyed to the plant and pass through each process to the final bagging and packing by means of a gravity system. If the materials must be raised repeatedly to higher levels, much power is required, whereas if the materials can be moved by gravity the cost of production may be greatly reduced.

Quarries should, wherever possible, be so located that their floors lie above drainage level.

QUARRY METHODS AND EQUIPMENT.

GENERAL METHOD.

The raw materials for the manufacture of cement are excavated by three different methods, quarrying, mining, and dredging.

Open-pit quarrying is the method most commonly employed. It is claimed that about 85 per cent of all the raw material used in the manufacture of cement is quarried. Mining of cement material is a modification used where the overburden is so thick that stripping would not be practicable. Dredging is employed in the removal of marl from lake bottoms.

FACTORS GOVERNING PLAN OF QUARRYING.

The plan of quarry development that will give the best results is governed by the depth of stripping, the thickness and attitude of the beds, the structure of the rock, and the uniformity of its chemical composition in the same and in successive beds.

If the overburden is thick, deep quarrying should be pursued if conditions will permit, for thus the maximum quantity of rock is obtained for a minimum of stripped surface. If the good rock is not thick enough to make deep excavation possible, cheap methods of stripping must be employed to make quarrying profitable.

Where the overburden is thin and a great thickness of rock is available, the depth to which the quarry may be worked is a matter of choice. The quarryman should seek to develop a quarry face of the height that can be worked the most cheaply. Usually a face 30 feet high or higher is more cheaply quarried than one less than 30 feet in height.

If the beds are flat the direction of the quarry walls is influenced only by surface contour, by structural features such as joint planes, or by the most convenient situation for transportation lines. However, for steeply inclined beds it is usually better to make the quarry face at right angles to the strike of the beds and thus cut across successive strata. When the face is in this direction the rock is more easily blasted and a more uniform mixture is obtained.

Every quarry plan is influenced more or less by drainage conditions. If a large flow of water is encountered, automatic drainage at a high level, leaving sufficient rock face above water level, would discourage deep quarrying which involved excessive pumping.

In beds dipping at a low angle it is usually wise to quarry in such a manner that the face is moved up the dip. By this method drainage from the face is insured, the rock is thrown down and loaded more easily than when worked in the opposite direction, and the loaded cars may be partly or entirely moved by gravity.

In choosing a suitable level for a bench or quarry floor in a rock deposit where the beds are approximately horizontal, the operator should be governed chiefly by the position of open bedding planes. Such planes promote effective blasting and also make a smooth floor, which greatly simplifies the laying of tracks and the loading of rock.

Where the strata are extremely folded and contorted with synclines, anticlines, and faults the process of quarrying may be more complicated. Especially is this the case where dolomite and limestone are interbedded as in a certain Georgia quarry. In that quarry the beds are folded, have irregular axes, and the structure is complicated by faults. The dolomite, in general, forms the axes of the anticlines and is interbedded with the limestone at various levels. It is a difficult matter to quarry this rock in such a way as to avoid the dolomite. The method used is, in general, to maintain a quarry face at right angles to the axes of the folds and to work out the synclines. The dolomite, however, occurs in many unexpected places, and the problem is therefore beset with many uncertainties. The writer is confident that working out of the structural relations of the dolomite and the limestone by a competent geologist, and careful mapping of the quarry area would be of immense value to the company. Cross sections showing the relation of the dolomite to the limestone and the position and throw of faults would be of especial value. At many quarries geologic study and the careful preparation of maps would probably be of great help in laying out and developing the quarry. In such investigations emphasis should be placed on the relation of good rock to impure rock. As far as practicable the position of each in the deposit should be ascertained in order that a plan of development may be worked out by which the good rock may be obtained and the defective material avoided.

POWER.

Quarry drills are commonly operated by compressed air, steam, or electricity obtained from the central power house of the cement plant. Where the quarry is situated at a considerable distance from the cement plant a separate power plant may be necessary. Shovels for loading rock or overburden may be operated by electricity, compressed air, or steam. The steam shovel is the more common type, and coal is almost invariably used for fuel. The power cost depends largely on the availability of a coal supply. In one of the Ohio

quarries visited, a thin bed of coal beneath the limestone supplied fuel for the steam shovels.

In quarries of wide extent, situated in localities where the winters are severe, the loss of power in long steam lines may be considerable. If it is found advisable to use steam for drills in such quarries, power losses may be minimized by establishing several small boiler houses near the drilling operations rather than conveying the steam long distances from a central generating plant.

A detailed discussion of power plants is not within the scope of this bulletin. Much useful information may be obtained by consulting the work of Brunton and Davis.[a]

STRIPPING.

NATURE AND EXTENT OF OVERBURDEN.

Stripping or removal of overburden is a very variable factor in quarrying. Some deposits are exposed and no stripping is necessary. In other places the removal of 40 to 50 feet of cover may be necessary, whereas a depth in excess of 50 feet is common. As a rule no attempt is made to remove such excessive overburden. The rock deposit is either classed as unavailable or tunnel methods of quarrying may be employed.

The overburden may consist of clay, sand, gravel, or a mixture of these substances, and may contain bowlders of various sizes. It may be soft and easily excavated by hand or with a steam shovel or it may be so hard that blasting is necessary. All these factors must be considered in determining the best means of removal.

OVERBURDEN QUARRIED WITH ROCK.

The chemical composition of the overburden is of considerable importance. If the underlying rock is a high-calcium limestone requiring the addition of clay or shale, it is possible that the overburden may be used as a constituent of the raw mix. Under such conditions stripping may not be necessary. Instances have been observed where a depth of 2 to 4 feet of clay or weathered shale is not removed from the rock surface, but is shot down and loaded with the rock. There are two objections to this process which condemn it in the minds of some operators. First, in wet weather the clay makes the rock muddy and difficult to handle and crush, particularly where gyratory crushers are employed. In the second place, the presence of large quantities of clay, which are often imperfectly mixed with the rock, makes it difficult for the chemist to maintain a uniform mixture.

[a] Brunton, D. W., and Davis, J. A., Safety and efficiency in mine tunneling: Bull. 57, Bureau of Mines, 1914, pp. 46–78.

It is doubtful whether shooting down a thickness of 2 to 4 feet of clay with the rock is advisable. However, the fact that clay is one of the constituents commonly added to limestone to make cement obviates the necessity for clean stripping in many places, and thus relieves the quarryman of an operation which is usually tedious and costly.

Where the depth of overburden varies in different parts of the quarry, the feasibility of shooting down the overburden with the rock depends to some extent on the method of quarrying. In one Kansas quarry the overburden varies in thickness in different parts of the quarry from a few inches to 10 feet. At one time the overburden was shot down with the rock. Hand-loading methods were then employed, and while some cars were loaded with an excessive amount of earth, others from that part of the quarry where the overburden was thin were loaded almost entirely with rock. By sending the cars to the crusher in proper order a uniform mix was maintained. Later a steam shovel was introduced for loading rock in the quarry, hence the rock could be taken only from one part of the quarry at one time. If the overburden at that place was thick the proportion of clay loaded with the rock would be altogether too high. Therefore it was found necessary to put another steam shovel on the surface and strip the rock mass. Rock quarried from all parts of the face was then relatively free from clay, the necessary amount of clay or shale being added at the cement plant.

Where it is desired to use the rock without admixture with stripping, the surface should be stripped back far enough from the face to prevent masses slipping into the pit. An instance was observed where soil had slipped into the quarry and was later removed by the slow and tedious hand-shovel and dump-cart method. It is much better to employ the most efficient stripping means possible, and to strip a fairly wide surface ahead of rock-quarrying operations.

In several quarries, notably in Illinois and Kentucky, a shale bed underlies good limestone, and when the limestone is removed in quarrying, the shale is rendered available.

CONDITIONS GOVERNING METHOD OF REMOVAL.

In choosing the cheapest method for stripping a number of factors must be considered. If a great quantity of material must be removed, rapid and efficient means, such as a steam shovel, should be employed, whereas a thin overburden may justify the use of more inexpensive equipment, although the cost per ton of stripping might be larger.

Much depends on the nature of the rock surface. If it has channels and pockets filled with clay or other material, removal of this

material with machinery may be difficult, and the use of slower and less efficient hand methods may be necessary. Conditions become more complicated if the pockets contain masses of rocks.

The firmness or solidity of the overburden also influences the method. In some quarries the material may be readily removed with shovel, scraper, or clam-shell bucket without previous loosening, whereas in other places blasting may be necessary. The availability to a suitable dumping place governs to a considerable extent both the method of excavation and of transportation. Water supply and drainage are factors of extreme importance where hydraulic methods are contemplated.

METHODS EMPLOYED.

HAND LOADING INTO DUMP CARTS.

Hand shovels and dump carts are employed at some quarries where the amount of material to be removed is limited, and also at some quarries where the overburden is thick, but where the irregular rock surface discourages the use of steam shovels or other mechanical devices. In many instances clay could probably be removed more cheaply from large pockets with a clam-shell bucket than by hand loading. Hand loading is expensive, a cost of 30 cents or more a yard being common, hence this method should be avoided wherever possible.

TEAMS AND SCRAPERS.

Where a thin overburden must be removed from a wide surface and the dumping place is convenient, teams and scrapers may be used. Where removal to a considerable distance is necessary, the material is sometimes rehandled. At one Iowa quarry the soil was taken in scrapers to an elevated platform, dumped through an opening into dump carts, and hauled away with horses. By using wheel scrapers rehandling could have been avoided.

DRAG-LINE SCRAPERS.

A drag-line scraper equipment used for stripping in a Kentucky sandstone quarry is described in Bulletin 124.[a] An interesting modification of the method was observed in a Pennsylvania quarry. One of the conditions usually considered necessary for successful operation of a drag-line scraper is a near-by place of disposal, preferably at a lower level than the area being stripped. At this quarry the stripped material had to be hauled in cars to the dumping ground,

[a] Bowles, Oliver, Sandstone quarrying in the United States: Bull. 124, Bureau of Mines, 1917, pp. 29, 30.

which was some distance away. A single scraper of $\frac{1}{8}$-yard capacity, working on a drag line, stripped the soil back from the face. The loaded scraper was dragged up a plank incline onto a trap door in the platform at the upper end of the incline. The trap door dropped under the weight of the load and dumped the soil into a car on a track below. The incline and the platform could be shifted laterally as required. Two men were employed to guide the scraper during loading, and part of the time of another man was required to control the cable and place the scraper in position. The entire operation of loading required only four men, and part of the time of a fifth. Except for the engine runner, the men were all unskilled laborers. The material was handled rapidly and the cost of loading was said to be about 3 cents a cubic yard. The loaded cars were hauled by cable to the dumping ground. The method seemed to be cheap and efficient.

In some quarries a scraper or bucket is lowered from the arm of a derrick and is loaded by hauling it toward the derrick with a horizontal cable wound on a drum. When full it is hoisted, swung over a car by the derrick arm, and dumped. No figures have been obtained indicating the efficiency of the method.

STEAM SHOVEL.

For removing large masses of material, a steam shovel is probably the most practical and efficient means. As the first cost of a steam shovel is high, to purchase one is unwise except where stripping is to be on a reasonably extensive scale. The nature of the rock surface also has a marked influence on the success of steam-shovel operations. In many limestone deposits the surface is eroded into deep channels and cavities which have been filled with soil. It is difficult, or impossible, to remove the material from such pockets with a steam shovel, also great difficulty may be encountered in laying tracks on the uneven surface. If the surface is irregular and the overburden is thin, a steam shovel should not be employed. On the other hand, if the rock has a smooth and uniform surface, a steam shovel may be operated efficiently, even if the overburden is only 2 or 3 feet thick.

The usual method of stripping with a steam shovel is to load the material into side-dump cars and haul them with a locomotive in trains of 6 to 20 cars to a near-by dumping ground. Loading into dump wagons has been noted, but a large number are needed to keep the shovel working. The use of wagons may be justified for a limited period of time if it is desired that the stripped material should be placed where laying a track is not feasible.

Waiting for empty cars is one of the chief causes of poor steam-shovel efficiency. This may be due to lack of cars or locomotives, or to inadequate trackage facilities. At one quarry visited, in order to avoid unnecessary loss of time, a train of empties stood on a siding ready to be shifted into place as soon as the loaded cars were removed.

Periods when cars are not available may be usefully employed in moving the shovel. In one New York quarry sufficient material for one train load could be reached from one position of the shovel. While the loaded cars were being hauled away, the shovel was moved to a new position and was ready for loading another train as soon as the empties returned.

Where the stripping is thin, much time may be lost in frequent moving of the shovel. When a full train load can not be obtained from one position of the shovel, poor efficiency must result if locomotives and train crews are thus kept in idleness. Under such conditions it may be wise to let the train crews work elsewhere while the steam shovel "casts over" a cut the entire length of the face. "Casting over" is a quarryman's term for making a cut with a steam shovel and depositing the material on the bank, thus forming a long ridge. When the next cut is made, a double amount of material is available from each position, and train crews are not kept idle while the shovel is being moved. Where the overburden is very thin, "casting over" may be repeated two or three times.

Usually the material overlying a rock deposit is soft enough to be handled readily with the steam shovel without blasting. In some localities, however, blasting is necessary. If large bowlders that require blasting are encountered, the rate of progress may be greatly reduced.

The cost of stripping with a steam shovel depends on the depth and nature of the materials to be removed, capacity of the shovel, condition of the rock surface, and facilities for removal of cars. Four cents a ton, actual working cost for loading only, is the minimum figure obtained. At one quarry having 1 to 10 feet of overburden, the total operating cost of stripping, including transportation, was 11 to 13 cents a ton. At an Illinois quarry having 20 to 30 feet of soil overburden, the operating cost of stripping, including transportation to a near-by dump, averaged 16 cents per cubic yard. Where the overburden is thin, the cost per ton will be relatively high, but the cost of stripping, per ton of rock obtained, will be relatively low.

A stripping conveyor operated in connection with a steam shovel by one Ohio company is noteworthy. The soil to be removed is 30 to 45 feet thick, and the bed of rock beneath is only 8 feet thick.

With the low efficiency of stripping observed in some localities, profitable quarrying under such conditions would be almost impossible. As the rock is removed, the area worked out is available for disposal of overburden. The equipment, as shown in Plate I, consists of an inclined bridgelike structure of steel mounted on wheels. On this incline, two independent cable cars on separate tracks are operated with an electric hoist. The cars are loaded with the steam shovel, hauled up the incline, and dumped. The total length of the conveyor is 175 feet, and it places the material 150 feet from the steam shovel. The shovel has a 2½-yard dipper, and each car is of 5-yard capacity. The whole machine may be moved forward as the shovel is advanced. The company claims that stripping can be done by this method at an actual working cost of 2 cents a cubic yard, not including overhead charges, depreciation, or interest on the investment, and that with a gang of 9 men 2,000 yards can be handled per day. The equipment is well adapted for stripping back into pits from which all serviceable stone has been removed.

HYDRAULIC STRIPPING.

Hydraulic stripping is practicable only where there is an abundant water supply and good drainage. If the conditions are favorable it is one of the cheapest methods that can be used.

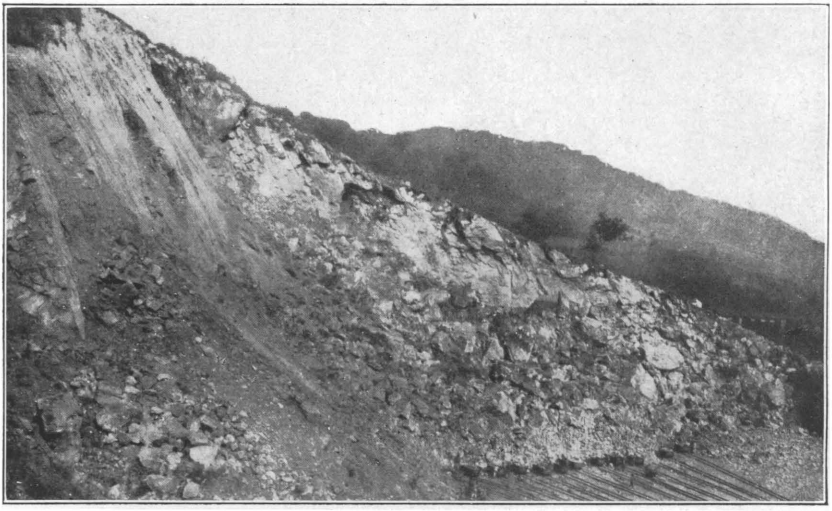
The usual method of hydraulic stripping is to pump the water through pipes and force it at high pressure through a nozzle directed against the bank. If the water has to be pumped some distance, it has been found advisable, in order to reduce the loss of power by friction, to use pipe of 4 or 5 inch diameter, close to the pumping station, and to reduce the size at successive intervals. The jet from the nozzle is usually so directed as to undermine the material and let it fall in a loose mass. It is then easily carried away in the stream. It is obvious that bowlders too large to be washed away retard the process, and if the material contains many bowlders the use of a steam shovel may be preferable.

In a quarry in northeastern Missouri where this method is employed a ravine affords a means of drainage back from the quarry face. As the material is rather hard holes 8 to 10 feet deep are bored with a soil auger, and the ground is loosened with small charges of explosive. The cost of stripping is said to be about 7 cents a cubic yard.

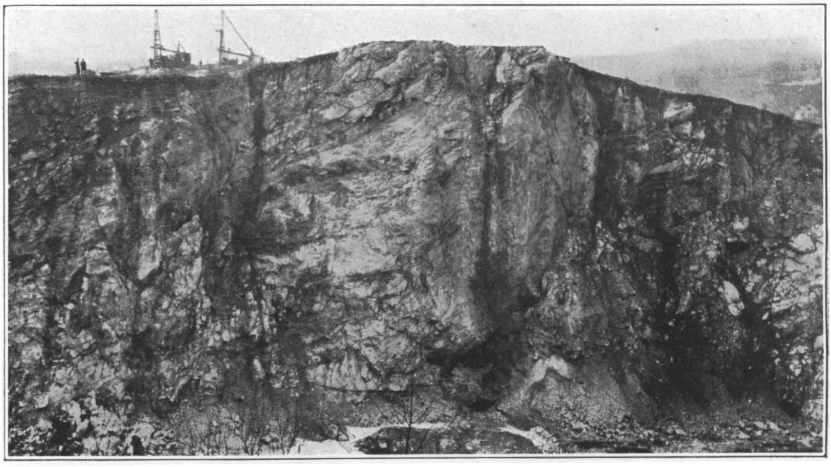
In an Indiana quarry stripping with steam shovels is supplemented with hydraulic stripping during seasons when the water supply is adequate. The material is washed down a channel, which is nearly parallel with the quarry face, and into a creek. A settling basin has been provided, but complaints are sometimes made of the turbidity

CONVEYOR AND STEAM SHOVEL USED FOR STRIPPING IN AN OHIO QUARRY.

A. ROUGH ROCK SURFACE EFFECTIVELY STRIPPED BY HYDRAULIC METHOD.



B. QUARRY FACE PARALLEL WITH STEEPLY INCLINED OPEN BEDS.



C. OPENING OF GOPHER HOLE AT FOOT OF QUARRY WALL.

of the stream below the basin. The effects of hydraulic stripping may be far reaching, and the operator must carefully consider a possible conflict with other interests.

In one quarry in eastern Missouri the conditions are unusual. The quarry is situated on a bluff facing the river. The water supply is therefore abundant, but the drainage is toward the quarry face, whereas in most localities where hydraulic stripping is practiced the water may be drained away from the quarry pit. Owing to this condition the soil-laden water must be conveyed through an aqueduct from the quarry face to the river bank. However, the operators claim that the cost of hydraulic stripping, including the construction and maintenance of the aqueduct, is considerably lower than with steam shovels.

At one Virginia plant, where the hydraulic method is employed successfully, the quarry faces a river. The overburden consists of red clay and sand, varying in thickness from 2 to 15 feet. Clay seams and clay pockets are abundant. Stripping is conducted during the winter months, when the resulting turbidity of the stream causes the least inconvenience to other industries. Work on rock quarrying is suspended during the stripping. The rock surface dips toward the river, and the overlying material is washed down onto the quarry floor. A row of barrels filled with earth placed near the river bank serves as a dam or weir to hold back the bulk of the stripped material, which would otherwise obstruct the stream. This material, after it becomes comparatively dry, is loaded into cars with a steam shovel and removed to a dump.

To those unfamiliar with the conditions in this quarry, it would seem more reasonable to remove the overburden with a steam shovel in the first place. The rock surface is, however, so deeply eroded that steam-shovel work would be impossible. Plate II, *A*, illustrates the extreme irregularity of the stripped surface. This view is an excellent example of the effectiveness of the hydraulic method in removing soil from erosion cavities.

ROCK WASHING.

In some localities quarrying is made difficult by open seams or pockets that extend many feet down into the rock. The material with which these erosion cavities are filled may be undesirable as a cement constituent. The hydraulic method of cleaning out such cavities is very successful where it can be employed. The usual method, however, is to clean the seams as well as possible by slow and laborious hand methods. The material that can not be thus removed is shot down with the rock. The subsequent sorting which is necessary makes it undesirable to use a steam shovel, and hand-loading methods are

employed. When the good rock is removed the fine débris is loaded by hand into cars and removed to a dumping ground. The high cost of loading both rock and waste by hand would seem to justify the introduction of a more economical method.

The writer is of the opinion that in many quarries where such conditions are found a much better method would be to load with a steam shovel, taking rock and soil indiscriminately, and after crushing the material, pass it through washing screens. A sufficient water supply, of course, would be essential. The cost of installing a washing equipment would not be high. With sandy or other friable material the success of such a method would seem to be assured. It would not, however, work as successfully with sticky clay, which would be difficult to wash out. Also the crushing of rock mixed with clay may involve great loss of time in wet weather.

UTILIZATION OF STRIPPING.

As a rule the overburden of limestone deposits is of clay and can be used as one of the necessary constituents of Portland cement. Therefore in many localities part or all of the overburden is utilized for this purpose.

The material removed may also be used to fill in swamps, ravines, or other low places and thus render such areas available for agriculture or building. In one instance the reclaiming of swamp land for city lots brought in a return to the quarry company more than sufficient to pay the cost of stripping. The quarry operator may also utilize stripped material for making dams, roadways, or railway embankments on his property.

DRILLING.

RELATION OF DRILLING TO BLASTING.

Only certain phases of the subject of drilling can be considered independently of blasting, for the depth, size, spacing, and arrangement of drill holes are all related more or less closely to the character of the explosive and the method of blasting employed. In this chapter, therefore, the types of drills used and facts relating to their actual operation will be considered, other matters being reserved for subsequent consideration under "Blasting" (pp. 46 to 97).

PRIMARY AND SECONDARY DRILLING.

Two distinct processes of drilling are employed in practically all quarries where rock is obtained for cement manufacture. One, two, or more rows of holes are first drilled in the undisturbed bedrock,

in order that heavy charges of explosive may be fired at various points within the rock mass to break it up for removal. The drilling of these holes is termed "primary drilling."

In most instances the primary shots do not break all the rock loosened into sizes suitable for handling, even with large steam shovels. The larger masses must, therefore, be broken up with subsequent shots. Whatever drilling is deemed necessary for these subsequent shots is termed "secondary drilling." The holes thus drilled are sometimes termed "pop holes."

PRIMARY DRILLING.

TYPES OF DRILLS EMPLOYED.

Primary drilling may be done with churn, wagon, tripod, electric, or hammer drills. The choice of drill depends on the nature and thickness of the rock ledge and on the method of blasting employed.

Churn drills.—The type of drill most commonly used in quarrying rock for cement manufacture is the ordinary well drill, also known as the cable or churn drill. Various types of churn drills are now on the market, the details of which may be easily obtained from trade catalogues. The diameter of drill hole used in cement practice varies from 4 to 6 inches. The rate of drilling varies according to the size of the hole and the hardness of the rock. In limestone, rates of 30 to 75 feet a day for 6-inch holes and 50 to 115 feet a day for 5-inch holes have been noted. In cement rock the rate is somewhat higher. The cost of churn drilling varies from 20 to 50 cents a foot. A fair average in moderately hard limestone is 30 cents a foot for $5\frac{5}{8}$-inch holes. A cement company near Glens Falls, N. Y., claims that the cost of drilling 6-inch holes with an electric-driven churn drill, which required only one man to run it, was less than 15 cents a foot, including repairs.

Wagon drills.—The so-called wagon drill is of the reciprocating type, operated by steam or compressed air. It makes a hole 3 to 4 inches in diameter. The machine may be so rotated as to drill at either side or in front of the wagon upon which it is mounted. The common method is to drill a hole at each side of the wagon from each position, and thus carry two rows of holes simultaneously. The drill has a long vertical range; hence the steel does not need to be changed so frequently as in tripod drilling.

Tripod drills.—Tripod drills, with which every quarryman is familiar, are used to some extent in cement practice, although during recent years they have largely given place to churn drills. They are used most commonly in unsound rock, in thin ledges, or for drilling horizontal or so-called "snake" holes (see p. 57) at the base of the quarry ledge. Common sizes of tripod drill holes are $1\frac{3}{4}$ to $2\frac{3}{4}$ inches. The "electric air drill" is used successfully in some localities. Air

is supplied to it through flexible hose from a portable motor-driven compressor, which may easily be moved whenever the drill is moved to a new position. An average cost of tripod drilling in limestone is 15 cents a foot.

Electric drills.—Motor-driven electric drills give satisfactory service, especially in rock mining.

Hammer drills.—The larger hammer drills in common use are non-reciprocating drills, operated without a tripod, and are provided with an automatic rotating device. They use hollow steel, through which the exhaust air passes and blows the cuttings from the drill hole. They are employed in primary drilling in some quarries with low benches not exceeding 8 or 10 feet in height. They are also used for making shallow holes to supplement churn-drill holes where an unusually hard bed of rock occurs at a short distance below the surface. In the opinion of a number of quarrymen they drill more rapidly than tripod drills, and as one man only is required for each drill they are less expensive to operate than the machines requiring both a driller and a helper. A drilling rate of 180 feet per day in limestone has been noted. The hammer drill works most successfully when run dry, as the presence of a small amount of water in the hole may make the cuttings muddy so they can not be blown out with the exhaust.

EFFECT OF ROCK STRUCTURES ON DRILLING.

The presence of numerous open joints in a rock ledge, particularly if the joints are irregular, complicates drilling. The drill may be deflected by running into a fissure, thus making drilling slow and difficult. However, the chief objection is that a drill hole penetrating an open seam or cavity is usually useless, as the gases generated by the blast will escape into the open spaces and the shot will be relatively ineffective. Usually such a hole is abandoned and a new one made. Many of the holes drilled in unsound rock are for this reason never used.

ADVANTAGE OF PLOTTING JOINT SYSTEMS.

Where the joints are straight and regular and can be observed on the rock surface, the driller can usually avoid them by using good judgment. If the upper strata are extremely shattered, it may be impossible to detect the joints on the surface, and the driller must judge their position by observing the vertical face. If their trend is fairly constant, their position at a distance of 20 or 30 feet back from the face may be estimated with a reasonable degree of accuracy. But if the joints are irregular or curving, or tend to change abruptly in direction, the placing of holes may become mere guesswork, and many useless holes will be drilled. In one quarry with a 50-foot face, where tracing the probable curves of open joints was a difficult

matter, the writer was informed that one driller had in three weeks drilled only two serviceable holes, all others having been abandoned when partly completed on account of their penetrating clay pockets.

A detailed description of the origin and nature of joints is given in Bulletin 106.[a] It is pointed out that in general joint planes tend to occur in parallel systems, and the planes of each system tend to be more or less constant in direction. By making a careful and systematic measurement of the joints with compass and clinometer and plotting them carefully on paper, it will usually be found that they have a systematic arrangement and are less irregular and haphazard than they appear to be from casual observation. A chart of all the joints on which definite observations can be made in a given deposit would, without doubt, be of great assistance to the driller in choosing a proper location for drill holes, and much useless drilling would thus be avoided.

KEEPING DRILL HOLES CLEAN.

On account of the modern method of firing a large number of drill holes at a single blast, several months may elapse from the time a hole is drilled until it is loaded. If proper precautions are not observed, the hole may become partly filled with débris, which must be cleaned out. A good plan, therefore, is to drive a wooden plug into the hole as soon as it is completed. Where the upper part of the ledge is thin-bedded, loose rock, it may not be possible to plug the holes effectively without casing them for a depth of several feet. Sheet-metal casings may be made at low cost and are highly recommended for use in deposits of this type.

CLEANING DRILL HOLES.

The method used by one Oklahoma company for removing débris from holes before loading is to place a gas-pipe tripod over the hole, and by means of a rope and pulley lower a section of gas pipe into the hole. By repeatedly raising the pipe several feet and allowing it to drop, it gradually sinks into the débris. Compressed air is then forced through the pipe and blows the refuse out. If the material is too coarse to be blown out, it is first cut up with a steel bar. As the cost of cleaning some drill holes has exceeded the original cost of drilling, all holes recently drilled by this company are cased at the top and carefully plugged.

SECONDARY DRILLING.

For secondary drilling, that is, the drilling of blocks that were insufficiently broken by the primary shot, hammer drills are most

[a] Bowles, Oliver, The technology of marble quarrying: Bull. 106, Bureau of Mines, 1916, pp. 22–29.

commonly used. The lines for supplying compressed air to the primary drills are usually laid on the quarry bank adjacent to the face. Air for secondary drilling is usually obtained from these lines by means of flexible hose. In one Iowa quarry the air is supplied by a small compressor mounted on the steam shovel. In a New York quarry a motor-driven compressor is mounted in a box car and hence can be readily moved to near the point where the hole is to be drilled.

BLASTING.

IMPORTANCE OF BLASTING.

In the blasting of rock for use in making cement, breaking into fragments that can be easily loaded into cars and can be handled by the crushers at the cement plant is essential. Because of the importance of blasting and the cost of explosive, no phase of the quarry industry under consideration is receiving more attention by investigators at present. As blasting represents such an important item in the cost of quarrying, evidently the introduction of efficient blasting methods at any quarry may materially reduce the total quarry costs. The wide variations in the character of rock deposits in different localities make it impossible to formulate rules for successful blasting that will uniformly apply. However, an attempt will be made to point out the methods followed in various localities and the conditions under which the various methods may give the best results.

PRIMARY AND SECONDARY BLASTING.

Blasting, like rock drilling, may be divided into primary and secondary processes. The term "primary blasting" refers to the shots by means of which the quarry ledge is broken into fragments of various sizes. "Secondary blasting" includes whatever shots may be necessary to break up the larger masses resulting from the primary shots into fragments small enough for loading.

PRIMARY BLASTING.

COMPLEXITY OF PROBLEM.

Primary blasting is one of the most complex problems with which the quarryman has to deal, owing to the many variable factors and the multitude of ways in which these factors may be combined. The chief variable factors are: (1) Height of face; (2) hardness and uniformity of rock; (3) attitude of beds; (4) prevalence of open bedding seams and joints; (5) attitude of quarry floor; (6) size and depth of drill holes; (7) arrangement and spacing of drill holes;

(8) number of holes shot at one time; (9) size of charge; (10) position of charges in drill holes; (11) type of explosive used; (12) method of firing shots; (13) method of loading rock; (14) size of crusher.

MODERN TENDENCIES IN BLASTING.

During recent years the tendency among quarrymen has been to throw down a large mass of rock at a single shot. Formerly the method of quarrying a ledge in successive benches or steps was widely employed. Of recent years, however, this method, which may be termed "multiple bench quarrying," has been generally abandoned, and most quarries are worked as single benches, even to depths upward of 150 feet. As a result, churn drills have been substituted for tripod drills in most quarries. This tendency has been increased by the wide use of steam shovels for loading. With a steam shovel rock may be loaded rapidly, and much larger fragments can be handled than was possible under hand-loading methods.

ADVANTAGES OF SINGLE BENCH.

One of the chief reasons for single rather than multiple bench quarrying is the increased efficiency attained by throwing down larger masses of rock at a single shot. Also, when several benches are worked it is necessary to clean the upper benches after shooting, and this cleaning requires a great deal of time. In quarrying with a single bench no bench cleaning is necessary. Another advantage is the increased safety which results. If quarrying is conducted on several narrow benches rock may fall from the upper benches and injure workmen on the lower levels. However, where benches are wide this danger may not exist.

DANGERS OF SINGLE BENCH.

In some quarries single-bench quarrying with churn drills has resulted in the development of vertical faces 80 to 150 feet high. The danger from loose rock falling many feet is obvious. On account of this danger, in some European countries the height of the face is restricted by law. Where high faces are maintained great care should be exercised in cleaning down all loose material from the face and from the rock surface close to the edge of the pit. In quarries where the rock is unsound men may be suspended with ropes at the face after each shot, to bar or blast down all loose material.

THE USE OF CHURN DRILLS.

In operating quarries as single benches many feet in height churn drills are widely used. In judging the efficiency of drills one must

keep in mind the prime object of drilling. Drill holes are made for the purpose of placing in the proper position enough explosive to break the rock efficiently. It is regarded as a necessary condition for successful blasting that a considerable part of the charge shall be near the bottom of the mass to be moved. For making deep holes churn drills are found to be more efficient than tripod drills. In tripod drilling the steel has to be changed frequently and long drills are difficult to handle. Moreover, tripod drill holes are rarely more than $2\frac{3}{4}$ inches in diameter at the top, and the decrease in gage as successive drills are used makes the deep holes much smaller in diameter at the bottom than at the top. As more explosive is required at the bottom of the hole than near the top, for practical purposes the hole is upside down. Hence, to shatter the rock efficiently in tripod drilling the holes must be drilled close together or else the holes must be "sprung" or "chambered"; that is, enlarged at the lower end by discharging small amounts of explosive in them. In springing, a small charge is used at first, and for each successive charge a larger quantity of explosive is employed. Commonly, about one-half pound of "40 per cent" dynamite is used for the first charge, the amount being doubled for each succeeding charge. The process is slow, requires a good deal of explosive, and is dangerous.

Churn drills, on the other hand, make holes 4 to 6 inches in diameter, and there is no loss of gage. Thus, the cost per foot of hole drilled may be much higher than that of tripod drilling, but the holes may be spaced much farther apart and still admit sufficient explosive to do the necessary work without springing.

In this connection the driller must keep in mind the proper relationship between volume and diameter of drill holes. Many drill men are under the impression that as a 6-inch hole has twice the diameter of a 3-inch hole, the former will hold twice as much explosive per foot as the latter. The volume, however, varies as the square of the diameter, hence a 6-inch hole holds four times as much as a 3-inch hole, a 5-inch hole holds $2\frac{7}{9}$ times as much as a 3-inch hole, etc. When the driller realizes, as most drillers do and as all drillers should, that doubling the diameter of the drill hole increases its capacity for explosive four times, he will better appreciate the advantage of a churn drill for deep-hole blasting.

Springing of churn-drill holes to increase their capacity for explosive has been employed in some mining operations where a heavy toe is to be removed, but is not commonly employed in ordinary quarry practice.

The efficiency of large holes may be shown in another way. Let us assume that it has been found by experience that blasting in tripod-drill holes is effective if the holes are 10 feet apart and have 10 feet of burden, that is, are 10 feet back from the face; whereas churn-

drill holes would give equally good results when spaced 20 feet apart and with 20 feet of burden. In order to shatter a mass 200 feet long and 20 feet wide, 11 churn-drill holes would be sufficient, whereas 42 tripod-drill holes would be necessary, as shown in figure 2. If the face is 30 feet high, the total mass of rock moved would be 200×20×30, or 120,000 cubic feet; that is, about 10,900 cubic feet for each churn-drill hole, and only 2,850 cubic feet for each tripod-drill hole.

In general, the churn drill is not satisfactory for quarrying thin ledges of rock. Thin ledges require small charges. When a small charge is placed in a large hole, too great a proportion of the explosive is concentrated at the bottom of the hole. Moreover, it is obviously a waste of time and power to drill a larger space than is

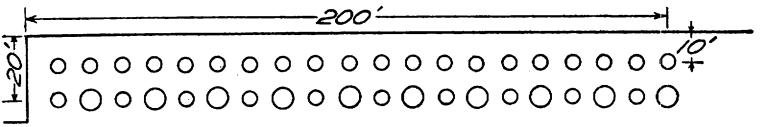


FIGURE 2.—Diagram showing relative efficiency of churn-drill holes and tripod-drill holes. Large circles represent churn-drill holes; small circles, tripod-drill holes.

necessary to hold the explosive. In general the use of churn drills for benches less than 25 or 30 feet high is not considered wise.

THE USE OF WAGON DRILLS.

The wagon drill is capable of drilling to a maximum depth of about 40 feet. It drills smaller holes than are usually made with the churn drill, but drills much more rapidly. Experience has shown that in some quarries the rock breaks better for a given quantity of explosive if the holes are smaller and spaced more closely than churn-drill holes are usually placed. Therefore, wagon drills are well adapted for drilling in such rock.

The wagon drill may also be used with success on thin ledges where, as pointed out in a previous paragraph, churn drills can not be used to advantage.

THE USE OF TRIPOD DRILLS.

Tripod drills are commonly employed in quarries where the rock is worked in one or more low benches. Where such drills are used, benches are rarely more than 16 feet high. The multiple-bench method, which has recently given place largely to single-bench quarrying, is still practiced in some localities because the advantages of using the churn-drill are not realized. In other places, lack of capital prevents the purchase of the new equipment necessary to make the change, although the advantages of the more modern methods are admitted.

In some quarries, however, it is claimed that the conditions justify the use of tripod drills. These conditions are discussed in the following paragraphs.

ADVANTAGE OF USING TRIPOD DRILLS FOR UNSOUND ROCK.

As pointed out on page 44, the presence of irregular seams or pockets filled with clay greatly complicates drilling, because these are often difficult to avoid in placing the holes. Mere avoidance of seams is, however, only part of the problem. The holes may all be in sound rock and yet be so arranged that the blast will be ineffective. In general, it may be stated that in seamy rock each individual mass bounded by open seams should have at least one hole drilled in it, and, if the mass is small, a small hole is sufficient. The reason for this statement is given in the following paragraph.

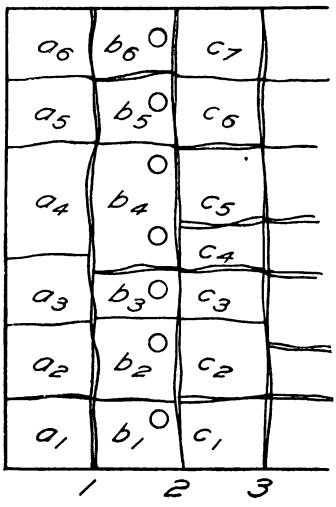


FIGURE 3.—Arrangement of drill holes in seamy rock.

Suppose that a series of open seams parallel the quarry face and are spaced 10, 10, and 30 feet, respectively, from the face, as shown in figure 3. An average burden for churn-drill holes is 18 to 25 feet. If, in this instance, churn-drill holes are placed 18 feet from the face, what will be the effect of the shot on the masses designated a_1 to a_6, between the face and seam *1*? It is a well-known fact that an open seam acts as a cushion and almost or entirely checks the shattering effect of an explosion. The masses b_1 to b_6, between seams *1* and *2* would without doubt be shattered normally, as the drill holes are situated in these masses. However, the masses a_1 to a_6, protected by the open seam *1*, would not be shattered much and would merely be pushed out. If the drill holes were placed 25 feet from the face, they would be in the masses c_1 to c_7, and obviously the masses designated a_1 to a_6 and b_1 to b_6 would receive little of the shattering effect. A blast of this type, which pushes out great blocks weighing many tons each, would be very inefficient, as an excessive amount of secondary blasting would be necessary.

Therefore, in order to effectively shatter rock intersected by open joints, such as are represented in figure 3, at least one drill hole in each individual mass seems to be necessary. Moreover, the hole should be so placed that the shot is balanced, thus promoting equal effectiveness in all directions from the hole. Holes drilled in the masses

designated a_1 to a_6 should be nearer to seam *1* than to the face, for the rock near the open face offers less resistance to the explosive force than that near the earth-filled seams at the back. The proper position of the hole can not be stated arbitrarily; it depends on the character of the rock, and can be determined only by experiment. However, on the assumption that the proper position is 7 feet from the face and 3 feet from the seam, it is obvious that a churn-drill hole 7 feet from the face would be too large. The small amount of explosive necessary to break a mass 10 by 12 feet or 10 by 15 feet in area could be placed in a hole much smaller than that ordinarily made with a churn drill. Consequently, drilling a large hole in such a position would be a waste of time and labor.

For seamy rock one of the smaller types of drills is to be preferred, and the tripod is commonly employed. Thus multiple-bench quarrying may be the best method to use in unsound rock. When churn drills are used in rock having steeply inclined open seams, the drill bit tends to follow the seam and thus make further drilling difficult or impossible. This involves long delays, and may necessitate the abandonment of partly completed holes. Instances have been noted where many feet of drilling are wasted in this manner every month. Some quarrymen claim that this difficulty is less pronounced with tripod drills. For those skilled in the use of churn drills, however, the difficulty is no greater than where tripods are used. A small charge of explosive is sometimes used in straightening a drill hole that crosses an inclined open seam. Some operators drop pieces of scrap iron in the hole to delay drilling until the undesirable shoulder is cut away.

USE OF TRIPOD DRILLS IN OBTAINING A UNIFORM MIX.

It has been stated that the chemical composition of rock quarried for cement manufacture must be carefully taken into account. The rock is continually being sampled and analyzed as it enters the plant and proportioned in the way that will give the best product. If the rock in the quarry ledge is uniform in composition, it is a simple matter to obtain a mixture of the desired composition, and the method of quarrying has little or no effect on the mixing. However, if the rock varies in successive beds the method of quarrying has a marked influence on the task of maintaining a uniform mixture. This is particularly true of certain cement-rock quarries of the Lehigh district, where, by advantageous handling of rock, a proper mixture can be obtained from one ledge without the addition of either limestone or shale from any other locality.

Let us assume that the beds are flat-lying and that they vary considerably in their calcium content. If the upper beds have a low and the lower beds a high calcium content, quarrying from the upper

beds alone would shortly give a lean mixture, and quarrying of the lower beds alone would give one too high in calcium. A method employed by one company having such a quarry is to work it in two benches, the upper bench including the low-calcium stone and the lower bench the stone higher in calcium. As these benches are low, tripod drills are employed and the rock is loaded into cars by hand. As the rock is hauled to the crusher the number of carloads from each of the benches is proportioned according to their relative composition so as to give a proper mixture.

A method of shooting down the entire quarry face, using churn-drill holes, has been suggested. Some quarrymen state that by properly balancing the charges in the holes the material from the upper and that from the lower beds may be mixed fairly well as the rock is shot down, and that the rock then may be handled much more efficiently with steam shovels than by hand methods.

The strata may not, however, be flat-lying, but may be steeply inclined, as in many limestone and some cement-rock quarries. In several quarries having steeply inclined beds the rock is drilled with tripod drills, in benches 8 to 16 feet high, and after being shot down is loaded by hand. The quarry face is perpendicular to the strike and intersects many beds. Rock is loaded from various points along the face at the same time, and a constant mixture is thus maintained. A mixture is sought which, with the addition of sufficient shale to give the proper ratio of calcium to aluminum, will have the proper proportion of silica and a minimum of magnesium.

The objection raised to the use of a steam shovel in such quarries is that it loads for a considerable time from one position and thus prevents proper mixing of the various types of rock. For example, the shovel might work for a period in rock having only 65 per cent of $CaCO_3$, which would be too low in calcium for a cement mixture, and during another period in rock having 85 per cent of $CaCO_3$, which would be too high. For this reason, a number of quarrymen recommend hand loading for variable rock.

It is a generally recognized fact that blasting rock in low benches and loading it by hand is a costly method of quarrying, hence the devising of means for overcoming any difficulties that may stand in the way of churn-drill and steam-shovel methods is desirable. Consequently the following suggestions are made regarding improved methods of quarrying steeply inclined beds that vary in composition.

In the methods described in preceding paragraphs quarrying is so conducted that the rock may be selected proportionally from each of the variable beds. In steeply inclined beds, selection is made possible by cutting across the beds horizontally. If a high face is developed instead of long, low benches, approximately the same mixture may be obtained vertically as is now obtained horizontally. This is illus-

trated in figure 4. The beds that differ in composition are designated *a*, *b*, and *c*. The quarry face is bounded by the lines *1* and *2*. If deep quarrying and steam-shovel loading were pursued the source of the material available from one position of the shovel may be represented as bounded by the vertical lines *3* and *4*. As may be seen from the figure, the same beds would be intersected as by the other method. From other positions to right and left of the one represented variations in thickness may occur, giving more material from certain beds and less from others, but in general a fairly representative mixture can be obtained.

For both flat-lying and inclined beds the objection may be raised that the use of churn drills and steam shovels will not permit sufficient latitude in varying the proportion of stone from each bed, for the steam shovel must take all material shot down, irrespective of its composition. If the various beds differ markedly in composition, this objection may be valid. It is suggested, however, that difficulties caused by local variations in composition may be largely overcome by using a large storage bin, where the rock may be distributed in such a manner as to equalize all variations. This matter is discussed at greater length in subsequent pages where the preparation of the raw mix is considered.

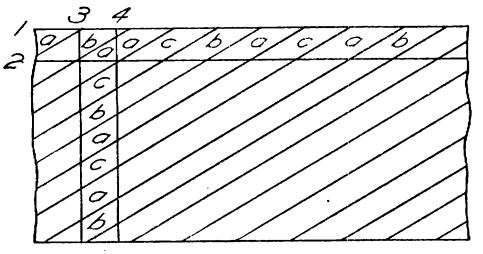


FIGURE 4.—Method of working steeply inclined beds to obtain uniform rock mixture; *a*, *b*, and *c* are beds of rock differing in composition.

USE OF TRIPOD DRILLS WHERE THE ROCK CONTAINS POCKETS OF CLAY.

In certain quarries, clay pockets are prevalent in the upper beds and separation of the clay from the rock may be desirable. If the entire face were shot down by means of blasts in churn-drill holes, the clay would be mixed with the rock, which would make the removal of both clay and rock difficult, especially in wet weather. Therefore, in some quarries it is regarded as a better plan to drill the upper bench with tripod drills, and shoot it down. When this material is removed the lower solid ledge may be shot down.

Usually, however, the clay may be used as a constituent of the cement, then, if it can be done successfully, quarrying rock and clay together is much better than to separate them and later mix them together again. One objection to handling them together is that in wet weather the clay makes the rock muddy and difficult to crush. With roll crushers, less difficulty is encountered than with gyratory

crushers. Another objection is that the steam shovel may at one time be loading material that is largely clay and at another time almost pure rock, and thus complicate the problem of maintaining a uniform mix. This difficulty may be minimized by having a large storage bin.

Another method of obtaining a uniform mixture of clay and rock is well illustrated in an eastern Missouri quarry. In this quarry the upper beds contain many clay seams, and a face 50 feet high is shot down with blasts in churn-drill holes. The charge is so adjusted that the clay is thrown out evenly over the whole mass. When the material is loaded with a steam shovel, the rock and clay are found to be proportioned with a fair degree of uniformity.

THE USE OF HAMMER DRILLS.

Hammer drills may be substituted for tripod drills wherever the use of the latter seems to be justified, for drilling holes ranging up to 12 or 15 feet deep, and their successful use for even greater depths is claimed. They work most successfully where the rock is free of water. If resort to bench quarrying is deemed advisable, hammer drills should be substituted for tripods wherever possible. A drill of this type may be operated by one man, and 200 to 250 feet of hole per day has been attained. To obtain the highest efficiency in the use of hammer drills, the air pressure should be not less than 90 pounds. Hammer drills are used widely for secondary drilling.

ARRANGEMENT OF CHURN-DRILL HOLES FOR MULTIPLE SHOTS.

The arrangement and spacing of drill holes is an extremely variable factor. In most primary blasting, a series of holes are fired simultaneously by electricity. Such a series is termed a "battery of holes" or a "multiple shot" to distinguish it from a "single shot," where only one hole is discharged at a time. For multiple shots the holes are drilled in one or more rows. The distance of the row of holes from the face is termed the "burden," and the distance apart of the holes in the row is termed the "spacing." Where several rows of holes are shot at one time, the burden of the second and following rows is the distance between that row and the next succeeding row toward the face. In some instances the holes of the second and following rows are immediately behind those in the first row, as illustrated at *A*, figure 5. Usually, however, the holes are staggered, as illustrated at *B*, figure 5. Generally better results may be obtained by staggering the holes than by placing them in straight rows.

Difficulty is sometimes experienced in properly measuring the burden from an uneven face. Usually when the burden is heavy the effect of the blast is to shatter and blow out the surface rock for some

distance back of the line of drill holes. Such "break-backs" occur most commonly where the upper beds are unsound or where the charge is placed too high in the drill hole. In placing the next line of holes the burden should not be measured from the edge of this slanting face, because it would obviously be too heavy for the lower part of the face, but should be measured from the position of the previous line of holes. In some quarries where holes are shot in single rows, when one row of holes is completed the burden of the second row is measured and the positions for the holes marked before the first row is fired. Thus the difficulty of measuring from an uneven face is eliminated.

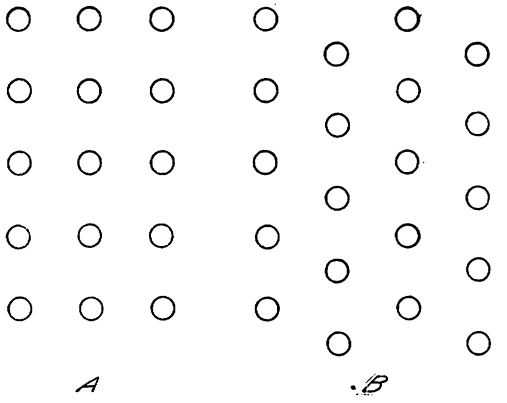


FIGURE 5.—Methods of placing drill holes for multiple shots: *A*, Holes in parallel; *B*, holes staggered.

The burden and spacing of drill holes varies according to the nature of the rock. As a rule, the tougher the rock the closer the spacing should be. If the toughness varies from point to point along a rock ledge the spacing of holes for a multiple shot should be varied accordingly.

In order to determine the proper spacing for drill holes in any particular quarry, one blasting engineer recommends beginning with a close spacing and then gradually increasing the distance until the maximum spacing that will properly shatter the rock is attained. In a 35-foot ledge of moderately hard limestone, drill holes should have a spacing of about 12 feet and a burden of 15 feet. The spacing and burden should increase gradually with increasing depth of holes. Where holes are 100 to 150 feet deep the spacing should rarely exceed 20 feet or the burden 25 feet.

The arrangement of drill holes for typical multiple shots in a number of quarries throughout central and eastern United States is given in Table 3.

TABLE 3.—*Kind of rock, number, depth, and arrangement of churn-drill holes for typical multiple shots.*

Kind of rock.	Average depth of holes.	Number of holes.	Number of rows.	Opposite or staggered.	Burden.	Spacing.
	Feet.				*Feet.*	*Feet.*
Cement rock	145	10 to 22	1	.....	20	18
Do	80	3 to 6	1	.....	20	20
Do	85	20	2	Staggered..	18 and 18	18
Limestone	50	50 to 60	1	.....	15	15
Do	30	20 to 30	2	Opposite...	12 and 12	12
Do	22	52	2	Staggered..	10 and 10	12
Do	40	26	1	.....	18	18
Do	55	50	2	Staggered..	15 and 15	15
Do	75	30 to 50	2	.....do.....	15 and 14	18
Do	19	41	2	.....do.....	8 and 7	12
Do	85	18 to 25	2	.....do.....	20 and 5	20
Do	40	10 to 15	1	.....	20	12 to 15
Do	15 to 20	40 to 60	3	Staggered..	8, 5 and 5	7 to 8

Of the examples in Table 3, wherever more than one row of holes is employed, the holes are staggered in every instance except one. Some have the holes arranged in three, four, five, or even six rows, in all these the adjacent rows are staggered. Where two rows of holes are used the burden of the second is in some examples equal to that of the first, whereas in others it is less than the first. It is regarded better practice to make the burden for the second and subsequent rows one-half to two-thirds that of the first row.

No attempt is made to judge the relative efficiency of the various arrangements recorded. In fact, to do so would be almost impossible, because the rock varies so widely in different localities that no one method is applicable to all types. Possibly the arrangements shown are not the most efficient that could be used. However, they represent practice at quarries that have been in operation for many years and where various methods have been tried, consequently it may be assumed that a fair degree of efficiency has been attained.

ARRANGEMENT OF SMALL DRILL HOLES FOR MULTIPLE SHOTS.

Wagon-drill holes may be fired in a single row or in several rows. They are spaced much more closely than churn-drill holes. Holes made with a tripod or hammer drill may be fired in one or in several rows, and the burden and spacing are usually not more than 6 feet.

ATTITUDE OF BEDS IN RELATION TO BLASTING.

ADVANTAGE OF FLAT-LYING BEDS.

In blasting rock a certain proportion of the explosive force is expended in separating the mass from the quarry floor, the proportion required depending on the ease of separation. Hence an open bedding plane at the floor greatly facilitates separation and promotes efficiency

in blasting. Where the rock lies approximately flat and where open bedding planes occur at intervals, the quarryman should endeavor to make one of the open seams the quarry floor.

DIFFICULTIES ENCOUNTERED IN STEEPLY INCLINED BEDS.

If the beds are steeply inclined, the quarry floor must of necessity cut them, consequently open bedding seams can not be utilized to make a smooth quarry floor. In such quarries, breaking the rock free from the quarry floor requires considerable force. The burden of material between the bottom of the bore hole and the face is termed the toe. It is always desirable to obtain a "clean toe," by which is meant sufficient shattering of the material that constitutes the toe to make its entire removal possible without excessive secondary blasting. When there is no plane of separation at the quarry floor it is customary to drill the holes 2 to 5 feet below the floor level, in order that enough of the explosive may be concentrated at the base to give a clean toe.

If insufficient explosive is used or drill holes are improperly arranged, masses of solid rock may remain projecting into the quarry at the toe and cause great difficulty in obtaining a clean toe for the next shot. Under such conditions the churn-drill holes are as a rule supplemented with "snake holes." The term "snake holes" is applied to holes drilled in an approximately horizontal direction and usually near the quarry floor.

DIRECTION OF QUARRY FACE FOR STEEPLY INCLINED OPEN BEDS.

Where rock is quarried from steeply inclined beds with open bedding planes at intervals, the main quarry face may be parallel or be perpendicular to the strike. The plan that tends to develop the quarry face in the direction that results in the most efficient blasting should be followed.

When the quarry face is parallel with the strike and the beds dip toward the face, the rock tends to separate along the bedding planes and slide down, resulting in a steeply inclined rather than a vertical face, as illustrated in Plate II, *B* (p. 40). An unfavorable feature of blasting a sloping face is that the burden is heavier at the toe and the churn-drill holes have to be supplemented with "snake holes." Plate III, *A*, illustrates the drilling of snake holes in a steeply inclined face. This supplementary process adds to the cost of both drilling and loading holes. Under normal conditions of quarrying with a vertical face, snake holes are rarely required. Another objection to a slanting face is the danger to which quarrymen are exposed from the rock sliding down the smooth, steep slope. If the face is maintained more nearly vertical than the dip of the beds, the danger is increased.

Also, where the beds dip toward the face the rock tends to slide down in large masses not sufficiently shattered, especially when low-grade explosives are employed. Under such circumstances it is best to use a high-grade rapid-detonating explosive, which will act more quickly and effectually shatter the mass before it begins to slide.

If the quarry face parallels the strike and the beds dip away from the face, a still more unfavorable condition prevails. This condition is illustrated in figure 6. As the face is continually advanced in the direction of the dip, the rock must be forced up the dip, an operation that requires an excessive amount of explosive. Furthermore, "snake-hole" blasting, if employed, tends to undermine the face and render it unsafe. Also, there is usually great difficulty in obtaining a clean toe. This method of quarrying has been observed by the writer in three localities, and in each instance the costs were excessively high.

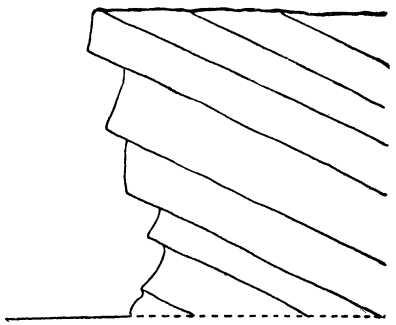


FIGURE 6.—Diagram showing conditions where rock beds dip away from quarry face.

When the quarry face is perpendicular to the strike, as illustrated in Plate III, *B*, a vertical face may be maintained, and the rock is thrown down with comparative ease. If the rock has open bedding seams, part of the force of the explosive may be lost. However, this loss will occur to some extent whatever the direction of the face. If all the factors are taken into consideration it is evident that a quarry can be developed to best advantage with the face crossing the strike.

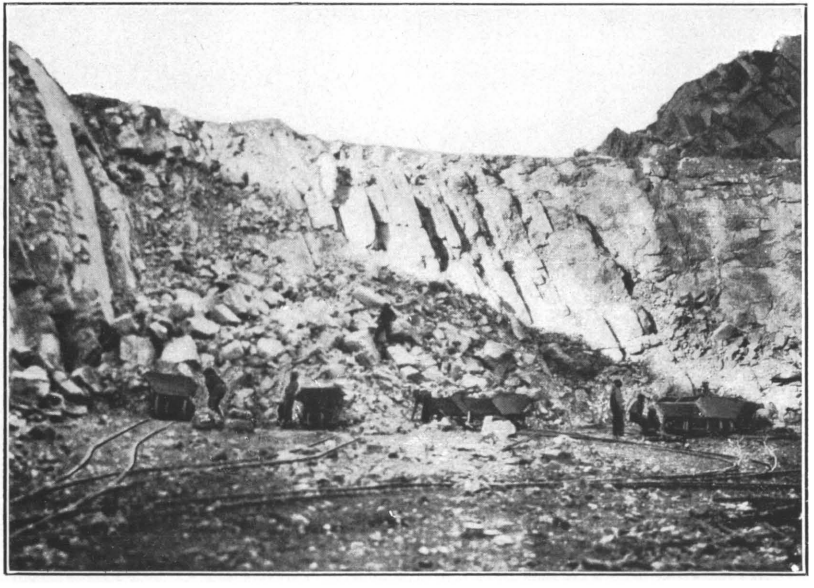
BLASTING IN CHAMBERED "SNAKE HOLES."

Under certain conditions blasts in chambered "snake holes" are found to give the best results. This method is employed in the steeply tilted irregular limestones of eastern Pennsylvania. The method could probably be used to advantage in some ledges consisting of steeply inclined beds with open seams, as, for example, those near Security, Md., and Spear's Ferry, Va. If the face exceeds 60 feet in height the snake holes should be supplemented by vertical holes, or the rock should be removed in two benches with snake holes.

The snake holes are drilled with tripod drills and are 24 to 30 feet deep. It is most convenient to start the hole about 2 feet above the quarry floor and to slant it enough to reach a little below floor level at its inner end. As recommended by S. R. Russell,[a] blasting en-

[a] Russell, S. R., Modern quarrying: Du Pont Magazine, June, 1916, p. 9.

A. DRILLING "SNAKE" HOLES IN A WEST VIRGINIA QUARRY.



B, QUARRY FACE MAINTAINED PERPENDICULAR TO STRIKE OF STEEPLY INCLINED OPEN BEDS.

gineer of E. I. Du Pont de Nemours & Co., the holes should be sprung about five times to give chambers of sufficient size. The first springing charge should consist of two $1\frac{1}{2}$ by 8 inch cartridges of dynamite, the second six cartridges, the third 12 to 15 cartridges, the fourth 20 to 30 cartridges, and the fifth 40 to 50 cartridges. A mark is placed on the tamping stick, and after each springing shot the chamber is filled to the mark on the stick. As a safety precaution a period of at least two hours should elapse between springing shots in order to allow the rock to cool. Holes are usually so sprung that they can each be loaded with 100 to 300 pounds of explosive. The charge should not extend more than 4 or 5 feet from the chambered part into the shank of the hole. "Forty per cent" dynamite gives satisfactory results in limestone. A convenient device for loading snake holes is a tin or brass pipe, which can be pulled out gradually as the explosive rises in the chamber. This method of blasting is cheap and efficient where conditions are favorable, although considerable care and judgment are required of the blaster.

"GOPHER-HOLE" OR "TUNNEL" METHOD OF BLASTING.

Another method, known in the East as "tunnel" blasting, in the Middle West as "gopher-hole" blasting, and on the Pacific coast as "coyote-hole" blasting, consists in firing large charges in small tunnels driven into the quarry face at the floor level. The method is simply snake-hole blasting on a large scale. The tunnel or drift is made as small as can be conveniently driven, being usually 3 to 4 feet wide and 4 feet high. The tunnel driving is usually done by contract at a price ranging from $1.50 to $2 per foot. Plate II, *C* (p. 40), shows the opening of a "gopher hole" at the base of a ledge. As a rule, the tunnel is driven 40 or 50 feet and then right and left cross headings are driven at right angles to the main leg, thus making a T-shaped opening. The dynamite is all placed in the cross headings and none in the main leg. It can be packed more closely if the cartridges are removed from the boxes. The charge should be divided into several parts having intervals of 15 to 20 feet between, filled with muck. Some quarrymen fill the intersection of the legs and at least half of the main leg with lean concrete, which is allowed to stand for at least 48 hours. Owing to the danger of premature discharge or damage by water seepage where a charge is left standing for 48 hours, a better practice is to build rough arches of small bowlders facing the charge in each branch and in the main leg. The charges should be wired in parallel and fired with a power current if possible. There should be at least two detonators in each explosive chamber, and to insure against misfires the chambers should be connected with trinitrotoluene detonating fuse.

The method is best adapted for quarrying a high face where the strata are irregular or conditions make it difficult to operate cable drills, and is not suited for working a face less than 70 or 80 feet high. The maximum height permissible is 150 to 175 feet. If the face is more than about 100 feet, it is wise to sink a few supplementary churn-drill holes from the surface between the back line of the tunnel and the face. These holes should be about one-third the depth of the face. When loaded with light charges and fired simultaneously with the main charge they assist greatly in breaking the upper rock.

EXAMPLES OF "GOPHER-HOLE" BLASTS.

The charge used in a New York quarry for a "gopher-hole" blast supplemented by blasts in churn-drill holes consisted of 13,700 pounds of 60 per cent gelatin dynamite and 15,350 pounds of granulated nitroglycerin blasting powder. The blast threw down about 72,000 cubic yards of stone, or approximately 5.6 tons per pound of explosive.

The charge for another blast of the same type consisted of 23,575 pounds of 60 per cent gelatin dynamite and 26,550 pounds of granulated nitroglycerin blasting powder. About 210,000 cubic yards of rock were moved, or about 9.5 tons per pound of explosive.

TYPES OF EXPLOSIVES USED.

The types of explosives commonly used in quarrying rock for cement manufacture are "straight" nitroglycerin dynamite, ammonia dynamite, gelatin dynamite, low-freezing dynamite, and nitrostarch blasting powder. A detailed discussion of the composition and properties of these explosives is given in Bulletin 48.[a]

Straight nitroglycerin dynamite was in past years commonly employed for primary shots, but recently has given place largely to ammonia dynamite and nitrostarch blasting powder. The compositions of typical explosives of the former type are shown in Table 4.

TABLE 4.—*Compositions of 40 per cent and 60 per cent "straight" nitroglycerin dynamites.*[b]

Ingredients.	40 per cent strength.	60 per cent strength.
Nitroglycerin	40	60
Wood pulp	15	16
Sodium nitrate	44	23
Calcium or magnesium carbonates	1	1
	100	100

[a] Hall, Clarence, and Howell, S. P., The selection of explosives used in engineering and mining operations: Bull. 8, Bureau of Mines, 1913, 50 pp.
[b] Hall, Clarence, and Howell, S. P., work cited, p. 7.

Ammonia dynamite is now commonly employed. Its tendency to absorb moisture condemns its use in wet quarries. The compositions of typical ammonia dynamites sold in the United States are shown in Table 5.

TABLE 5.—*Compositions of typical 40 per cent and 60 per cent ammonia dynamites.*[a]

Ingredients.	40 per cent strength.	60 per cent strength.
Nitroglycerin	22	35
Ammonium nitrate	20	30
Sodium nitrate	42	24
Combustible material [b]	15	10
Calcium carbonate or zinc oxide	1	1
	100	100

[a] Hall, Clarence, and Howell, S. P., work cited, p. 8.
[b] Consisting of wood pulp, flour, and sulphur.

In quarries where much water is encountered gelatin dynamite has been used with success. Gelatin dynamite contains a small percentage of nitrocellulose, which gelatinizes the nitroglycerin and renders it impervious to water. The compositions of typical gelatin dynamites as sold in the United States are shown in Table 6.

TABLE 6.—*Compositions of typical 40 per cent and 60 per cent gelatin dynamites.*[a]

Ingredients.	40 per cent strength.	60 per cent strength.
Nitroglycerin	33.0	50.0
Nitrocellulose	1.0	1.9
Sodium nitrate	52.0	38.1
Combustible material [b]	13.0	9.0
Calcium carbonate	1.0	1.0
	100.0	100.0

[a] Hall, Clarence, and Howell, S. P., work cited, p. 8.
[b] Wood pulp is used in 60 per cent strength gelatin dynamite. Sulphur, flour, wood pulp, and sometimes resin are used in other grades.

Low-freezing dynamites are recommended for blasting in winter. Their freezing point is about 35° Fahrenheit. Other types of explosives may be kept from freezing, or if frozen may be safely thawed by using proper apparatus,[a] but there is still danger that the explosive may become chilled from the air or from the cold rock while being loaded. Compositions of typical low-freezing dynamites are given in Table 7.

[a] See Hall, Clarence, and Howell, S. P., Magazines and thaw houses for explosives: Tech Paper 18, Bureau of Mines, 1912, 34 pp.

TABLE 7.—*Compositions of typical 40 per cent and 60 per cent strength low-freezing dynamites.*[a]

Ingredients.	40 per cent strength.	60 per cent strength.
Nitroglycerin	30	45
Nitrosubstitution compounds	10	15
Combustible material [b]	15	16
Sodium nitrate	44	23
Calcium or magnesium carbonate	1	1
	100	100

[a] Hall, Clarence, and Howell, S. P., The selection of explosives used in engineering operations: Bull. 48, Bureau of Mines, 1914, p. 7.
[b] Composition similar to that of the "straight" nitroglycerin dynamites.

Nitrostarch blasting powders are used in a number of quarries. They are relatively insensitive to shock and are consequently safer to handle than more sensitive types. The compositions of typical nitrostarch blasting powders used in the United States are given in Table 8.

TABLE 8.—*Composition of typical 40 per cent and 60 per cent strength nitrostarch blasting powders.*[a]

Ingredients.	40 per cent strength.	60 per cent strength.
Nitrostarch	40.0	46.0
Sodium nitrate	52.5	38.5
Ammonium nitrate	3.0	14.0
Sulphur	3.0	.....
Oil	0.5	0.5
Chalk	1.0	1.0
	100.0	100.0

[a] Table supplied by S. R. Russell, of E. I. du Pont de Nemours & Co., Wilmington, Del.

Much useful information on the use of explosives in quarrying may be found in Bulletin 80 of the Bureau of Mines.[a]

RATE OF DETONATION.

An important feature of explosives, from the quarryman's viewpoint, is the rate of detonation. Explosives that have a relatively slow rate of detonation, such as the lower grades of nitroglycerin dynamite, tend to push out masses of rock rather than to shatter them, or, in other words, the effect is propulsive rather than disruptive. On the other hand, explosives that have a rapid rate of detonation, such as the higher grades of nitrogylcerin dynamite, tend to shatter the rock; that is, their effect is primarily disruptive.

[a] Munroe, C. E., and Hall, Clarence, A primer on explosives for metal miners and quarrymen: Bull. 80, Bureau of Mines, 1915, 125 pp.

In blasting rock for the manufacture of cement, the aim of the quarryman is to break the rock into small pieces, hence the explosive used should have great disruptive force. "Straight" nitroglycerin and gelatin dynamites develop the strongest disruptive force of any of the ordinary explosives available for quarry work.

GASES EVOLVED BY EXPLOSIVES.

Carbon monoxide and hydrogen sulphide are the two most common poisonous gases formed by the detonation of explosives. Hydrogen sulphide can be easily recognized by its strong characteristic odor. Carbon monoxide is invisible and odorless and its effects are swift and deadly. It is, therefore, the most to be feared of all the gases evolved. Carbon monoxide is produced chiefly by the low-freezing and "straight" nitroglycerin dynamites, and hydrogen sulphide by the gelatin dynamites and black blasting powder.

Considerable quantities of the oxides of nitrogen may be formed when the explosive burns rather than detonates. This may occur from the side-spitting of a fuse or through the use of weak detonators. After careful determinations, Mann[a] found that where complete detonation occurs, very small amounts of the oxides of nitrogen are formed.

The fact should be emphasized that when detonation is incomplete, poisonous gases of various kinds are produced in excessive amount. Therefore, greater safety, as well as increased efficiency, is brought about by strong and complete detonation of the charge.

As most quarrying is of the open-pit type, the gases from small blasts can usually escape readily. Where large blasts are fired, however, especially in deep quarries, sometimes too little attention is given to the dangers from the gases evolved. An instance has been recorded[b] of the poisonous gases resulting from a heavy blast causing the death of 7 men and rendering 40 others unconscious. They entered the quarry about one-half hour after the blast had been fired. Obviously, after a large blast, sufficient time should be allowed for the gases to escape before anyone is permitted to enter the quarry. In any event enough time should be allowed to render the men safe from possible delayed explosions.

In a number of cement-manufacturing localities the rock is mined, and as the blasting is in tunnels the presence of poisonous gases may constitute a grave danger to the workmen. For underground work those explosives should be selected that will break the rock efficiently and at the same time evolve a minimum of poisonous gases. Through

[a] Mann, E. A., Report on investigations into the compositions of the gases caused by blasting in mines: Perth, Australia, 1911, pp. 14–15.

[b] See Munroe, C. E., and Hall, Clarence, A primer on explosives for metal mines and quarrymen: Bull. 80, Bureau of Mines, 1915, p. 16.

experiments conducted by the Bureau of Mines,[a] it was found that the amount of poisonous gases evolved could be decreased by reducing the proportion of combustible material in the explosive and in the paper wrapper containing it. Such a reduction has the effect of increasing the amount of oxygen available for combustion. The tests showed that the proportion of poisonous gases was lowest with gelatin dynamite and may be further reduced by diminishing the content of combustible material, and also by substituting flour for sulphur, wood pulp, or resin in the combustible portion of the explosive. It is recommended, therefore, that for underground blasting a "40 per cent" gelatin dynamite of the following composition be used:

Special formula for 40 per cent strength gelatin dynamite.[b]

Nitroglycerin	33
Nitrocellulose	1
Sodium nitrate	54
Combustible material[c]	11
Calcium carbonate	1
	100

Where blasting is conducted in confined drifts, the use of a jet of compressed air at the face assists in driving the poisonous gases away. Where work is carried on in several places simultaneously, it is wise for the men, after a blast has been fired, to work for a few hours at another breast and thus allow sufficient time for the gases to escape.

AMOUNT OF EXPLOSIVE FOR PRIMARY SHOTS.

In proportioning a charge, good judgment should be exercised. The object to be kept in view in blasting is to shatter the rock into reasonably small pieces and at the same time move it as little as possible.

There are several disadvantages in blasting in such a manner that the rock is thrown too far. The most obvious loss in overcharging is the waste of explosive involved. Also, if the rock is scattered far from the face, the cost of loading the scattered fragments is excessive where steam shovels are employed. The disadvantage of overcharging is, in this respect, most pronounced where the face is high.

Moreover, the tracks for cars or steam shovels are usually close to the quarry face. If the tracks are where rock may be thrown on them by the blast, considerable time and labor are required to move them, or, if left in position, to remove obstructions and repair damage

[a] See Hall, Clarence, and Howell, S. P., work cited, p. 10.
[b] Hall, Clarence, and Howell, S. P., work cited, p. 11.
[c] Flour.

caused by flying rock. Furthermore, the danger to the workmen is great where rock is likely to fly long distances.

GOOD JUDGMENT AND EXPERIENCE REQUIRED.

Lack of good judgment on the part of a blasting foreman may cause much trouble and annoyance to the men handling the rock. An instance was noted where, through poor judgment, the blasting foreman advised removal of the tracks from the vicinity of the quarry face. When the shot was fired the rock was moved so little that no damage or obstruction would have been caused to the tracks. For a subsequent shot the foreman expressed the opinion that the tracks were in no danger. When the shot was fired they were buried with rock.

Good judgment in determining the amount of explosive necessary is manifested also in the effect produced on the rock mass. When drill holes in a low bench are loaded too heavy, the blast tends to throw out a trench along the line of holes. When this takes place the rock near the holes is thrown too far and, at the same time, the blast loses much of its effectiveness for shattering the rock farther from the holes. The throwing out of a trench in this manner is known locally as a "chimney" shot. At one quarry visited where an overcharged blast was fired the charge was sufficient to blow out a trench nearly to the bottom of the quarry face, which at that point was 22 feet high. On the other hand, if the holes are loaded too light the rock is insufficiently shattered and much secondary blasting is required.

On a bench less than 30 or 35 feet high the ideal shot thoroughly shatters the mass and moves it but little from its original position. This result is best accomplished by employing the "buffer" method, as described in a subsequent paragraph. As a result of such a shot, the mass of shattered rock may, on account of the air spaces formed within it, stand up several feet higher than the face, and is in excellent condition for loading with steam shovels.

Success in loading drill holes depends on good judgment and experience. Experience gained in one locality may not serve for another locality on account of variations in local conditions. Each rock deposit has its own peculiar features, and it is only by long experience that one can become sufficiently familiar with the conditions to attain a high degree of efficiency in blasting.

CALCULATION OF CHARGE.

Mere guesswork in proportioning the charge can not be too strongly condemned. Many failures of shots are due to the crude methods of determining the amount of explosive used. When churn-drill

methods are first tried, a common mistake is to make the burden too heavy. It is then impossible to place enough explosive in the drill holes to effectually shatter the mass of rock blocked out. Blasting in churn-drill holes, as practiced in many quarries at the present time, involves large costs for drilling and explosives for a single shot. Where the cost of one shot may reach several thousands of dollars, failure of the shot results in considerable loss. Success in blasting depends largely on the proper proportioning of the charge, consequently a careful and systematic method of determining the charge should be followed.

The best method is to calculate the tonnage of rock to be moved and to proportion the charge accordingly. The tonnage in short tons may be easily calculated by multiplying together the length, breadth, and height, in feet, of the mass to be moved, multiplying this product by the weight of a cubic foot of the rock, and dividing by 2,000. An average weight per cubic foot of solid limestone is 168 pounds. If a reliable determination has been made of the weight of the rock in any quarry, the figure obtained may be used in place of 168.

If, for example, the length of the mass of rock to be shot down is 200 feet, the width 25 feet, and the height 80 feet, the approximate tonnage would be $\frac{200 \times 25 \times 80 \times 168}{2000} = 33{,}600$ tons.

The amount of explosive required for a given tonnage depends on the physical properties of the rock, such as hardness, soundness, the attitude of the beds, and also on the type of explosive employed. As a rule, 1 pound of the explosive ordinarily used in churn-drill holes is sufficient for breaking 4 to 5 tons of rock. An exact figure can be determined only by experience. If the charge is calculated on a basis of 1 pound of 40 per cent "straight" nitroglycerin dynamite or nitrostarch blasting powder for each 5 tons of rock, and the rock is not sufficiently shattered, either the proportion of explosive should be increased or a higher grade should be employed. If, on the other hand, a V-shaped mass is blown out and the rock is hurled to a great distance, subsequent charges may be reduced in size or a weaker explosive may be substituted.

The amount of explosive as determined on some such basis may be proportioned among the various drill holes. Even when all the holes are of equal depth, it may be wise to vary the charges. In blasting a heavy toe, or a corner, heavier charges may be required for certain holes than for others.

Usually the drill holes are not of equal depth because the surface is curved, sloping, or irregular. Under such conditions it is necessary to calculate the charge for each drill hole. To simplify such de-

terminations, Table 9 has been compiled, showing the number of tons of rock for each foot of drill hole for various spacings and burdens. The figures in the horizontal line at the top represent the number of feet of spacing, and in the column at the left, the number of feet of

TABLE 9.—*Number of tons [a] of limestone per foot of drill hole for various spacings and burdens.*

Burden of drill holes.	Spacing of drill holes, feet.																			
	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Feet.	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*
5	2.08	2.52	2.92	3.36	3.78	4.20	5.54	7.05	8.73	10.58	12.60	14.78	17.13	19.65	22.34	25.20
6	2.52	3.02	3.52	4.03	4.53	5.04	6.46	8.06	9.82	11.76	13.86	16.13	18.56	21.16	23.94	26.88
7	2.94	3.52	4.11	4.70	5.29	5.88	7.39	9.07	10.92	13.10	15.12	17.47	19.99	22.68	25.53	28.56
8	3.36	4.03	4.70	5.37	6.04	6.72	8.31	10.08	12.01	14.11	16.38	18.81	21.42	24.19	27.13	30.24
9	3.78	4.53	5.29	6.04	6.80	7.56	9.24	11.08	13.10	15.28	17.64	20.16	22.84	25.70	28.72	31.92
10	4.20	5.04	5.88	6.72	7.56	8.40	10.16	12.09	14.19	16.46	18.90	21.50	24.27	27.21	30.32	33.60
11	.....	.....	.....	7.39	8.31	9.24	11.08	13.11	15.28	17.64	20.16	22.84	25.70	28.72	31.92	35.28
12	.....	.....	.....	8.06	9.07	10.08	12.01	14.11	16.38	18.81	21.42	24.19	27.13	30.24	33.51	36.96
13	.....	.....	.....	.....	9.82	10.92	12.93	15.12	17.47	20.16	22.84	25.70	28.72	31.92	35.11	38.64
14	.....	.....	.....	.....	.....	11.76	13.86	16.13	18.56	21.16	23.94	26.88	29.98	33.24	36.70	40.32
15	.....	.....	.....	.....	.....	.....	.....	.....	19.65	22.34	25.20	28.22	31.41	34.77	38.30	42.00
16	.....	.....	.....	.....	.....	.....	.....	.....	20.74	23.52	26.46	29.56	32.84	36.28	39.90	43.68
17	.....	.....	.....	.....	.....	.....	.....	.....	.....	24.69	27.72	30.91	34.27	37.80	41.49	45.36
18	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	28.98	32.25	35.70	39.31	43.09	47.04
19	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	33.60	37.12	40.82	44.68	48.72
20	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	38.55	42.33	46.28	50.40
21	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	43.84	47.88	.....
22	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
23	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
24	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
25	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
26	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
27	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
28	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
29	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
30	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

[a] Rock assumed to weigh 168 pounds per cubic foot.

burden. Thus, by following to the right the line for any particular burden until the column under the desired spacing is reached, the figure found will represent the number of tons of rock that must be moved for each foot of drill hole. If, for example, the burden is 14

feet and the spacing 11 feet, 12.93 tons of rock must be moved for each foot of drill hole. By multiplying this quantity by the depth of the hole in feet, the total tonnage for which explosive must be provided in this hole may be determined.

Obviously, the proper spacing and burden for a given size of drill holes may be determined in the same way. Thus it may be found that when the calculated amount of explosive is placed in a $5\frac{5}{8}$-inch drill hole the charge is too low in the hole to break the upper rock effectually. The difficulty may be overcome by separating the charge into two parts with intermediate stemming, by using a larger amount of a lower grade explosive, by increasing the burden and spacing of the drill holes and increasing the charge accordingly, or by reducing the diameter of the drill holes. When the explosive would fill the bore hole too high, it may be a good plan to chamber the hole. If the drill holes will not hold enough 40 per cent strength explosive to properly break the bottom, it is better to use 60 per cent strength explosive in the bottoms of the holes.

In order to judge the proper size of drill holes to use, the shot firer should know the number of pounds of explosive that can be placed in a drill hole of a given size. Table 10, compiled by one explosives firm,[a] indicates the quantity of various types of explosives that can be placed in each foot of drill holes of various sizes when the cartridges are slit and the charge well tamped.

TABLE 10.—*Quantity of explosive per foot that can be placed in drill holes of various sizes.*

Diameter of drill hole.	Gelatin dynamite.	Straight nitroglycerin dynamite.	Low freezing ammonia dynamite.	Extra ammonia dynamite.
Inches.	*Pounds.*	*Pounds.*	*Pounds.*	*Pounds.*
3	4.25	3.75	3.60	3.72
$3\frac{1}{2}$	5.68	5.10	4.89	5.07
4	7.55	6.60	6.33	6.62
$4\frac{1}{2}$	9.35	8.40	8.06	8.38
5	11.80	10.50	10.08	10.35
$5\frac{5}{8}$	14.94	13.20	12.53	13.00
6	17.00	15.00	14.40	14.90

The conditions of a shot may be varied by changing the amount of the charge, by changing the type of the explosive, and by varying the spacing and burden of the drill holes. Hence the problem is complicated, and good judgment and long experience are required on the part of the shot firer before maximum efficiency is attained.

[a] See E. I. Du Pont de Nemours & Co. catalogue: Explosives for quarry blasting, 1915, p. 15.

"BLANKET" OR "BUFFER" BLASTING.

The effectiveness of blasting may, under certain conditions, be greatly facilitated by "blanket" or "buffer" blasting, also termed "shooting against the bank." This method can be employed only when the face does not exceed 30 or 35 feet in height. In loading the shattered rock with a steam shovel a strip 14 to 20 feet wide is left next to the face. This acts as a buffer for the next blast and prevents the rock from being hurled into the excavation where it may bury tracks, damage equipment, and increase the cost of loading. Also, the buffer tends to confine the energy of the explosion and promote shattering. In several Illinois quarries buffer blasts thoroughly shatter the rock with so little displacement that the casual observer would not realize that a blast had been discharged. Some quarrymen claim that blanket blasting is more successful with a good floor seam than where no seam exists. The method is highly recommended for quarries with low faces, but can not be employed with a high face, for the shattered rock falls to the floor and leaves the upper part of the face exposed. In loading holes for a buffer blast, the charge should not extend to the upper part of the drill hole, because the buffer tends to direct the shock of the explosion back into the solid ledge, and "breakbacks" may complicate the drilling of the next line of holes.

ARRANGEMENT OF CHARGES IN DRILL HOLES.

THE CONTINUOUS CHARGE.

A continuous charge of one type of explosive that fills the hole except for the space at the top required for stemming is the simplest arrangement possible. Such a charge may be used where the rock is uniformly hard, the face vertical, and the toe well cleaned. The burden, spacing, and diameter of drill holes should be adjusted to conform with the quantity of explosive as governed by the rock tonnage effectively moved per pound of explosive. The explosive used may be 40 per cent strength dynamite or of higher grade. In no quarry observed was a grade lower than 40 per cent strength used for a continuous charge. The use of "40 per cent" dynamite is common, but "60 per cent" dynamite is used in a number of quarries. It is claimed that the more effective shattering obtained with the stronger explosive more than justifies the higher cost. However, opinions differ regarding the success of using the higher explosive. Some quarrymen claim that "60 per cent" dynamite pulverizes the rock adjacent to the drill holes but does not shatter the rest of the burden more than "40 per cent" dynamite. The action of the explo-

sive is probably controlled to some extent by the physical properties of the rock. In this, as in many other matters, definite rules can not be given. The quarryman must be guided to some extent by the characteristics of the rock, and the best methods can in most places be determined only by experience.

In general, the higher explosive breaks the rock into smaller fragments than that of lower grade. It must be remembered, however, that the results obtained depend on spacing of holes and size of charge, as well as on grade of explosive used. If the rock is extremely unsound—that is, intersected by numerous seams—a low-grade explosive will probably give the best results.

USE OF DIFFERENT GRADES OF EXPLOSIVE IN THE SAME DRILL HOLE.

The continuous charge is not always made up entirely of one type of explosive. In many quarries the burden is heavier near the quarry floor, consequently a greater volume of explosive, or a more powerful explosive, must be used at the bottom. If the hole is not sprung and the charge is continuous, the charge can not be concentrated in the bottom of the hole except by using a higher grade of explosive at that point. Under such conditions, "60 per cent" dynamite is commonly used in the bottom of the hole, and "40 per cent" dynamite for the rest of the charge. Occasionally as high a grade as "75 per cent" gelatin dynamite is used in the bottoms of drill holes where the lower beds are exceptionally hard.

Failure to employ sufficient explosive near the base of the quarry wall may later cause much delay and additional expense. An instance was observed where an unbroken bench about 8 feet in height was left at the quarry wall. This bench had to be cleaned, redrilled, and blasted before the cut could be completed.

The upper part of a shale or limestone deposit, especially in flat-lying rock, is usually thin bedded or jointed and is easier to break than the lower part. Consequently, a low-grade explosive, commonly of 30 per cent strength, is used for the upper part of the charge in such deposits.

In some quarries an effective means of breaking the rock is to throw out the base with a heavy charge and allow the upper part to fall, the impact of the fall assisting greatly in shattering the mass. Where this method is employed a high-grade explosive is usually placed in the bottom of the hole and a small amount of low-grade explosive in the upper part.

For rock having "slab cleavage," that is, rock in which the length and width of the average block is much greater than its thickness, this method is to be avoided. If the base is thrown outward and the upper beds fall almost vertically, the attitude of the fragments will

be similar to that illustrated in *A*, figure 7. The slab-like masses may be so bound in that great difficulty will be encountered in loading them. Not only is loading rendered difficult, but the loading tends to undermine the pile of blocks, involving great danger from slides. This danger is especially pronounced in the wintertime. However, if the charge is so adjusted that the upper beds are thrown farthest from the face and the lower beds successively to lesser distances, the rock fragments are not bound in, but lie in the most favorable position for loading, as shown in *B*, figure 7.

The use of different grades of explosive in the same drill hole is opposed by some quarrymen on the ground of lack of effectiveness. It is claimed that because the different grades of explosive have different rates of detonation the shot is not exactly simultaneous, the higher grade detonating an appreciable interval ahead of the lower grade, and that this lack of coordination between the different parts of the shot weakens the force of the explosion. This objection is probably not valid, for the detonation of each grade starts at the same moment, and as each grade is exerting its force on a different part of the rock mass it is probable that the time interval is too small to affect the result materially.

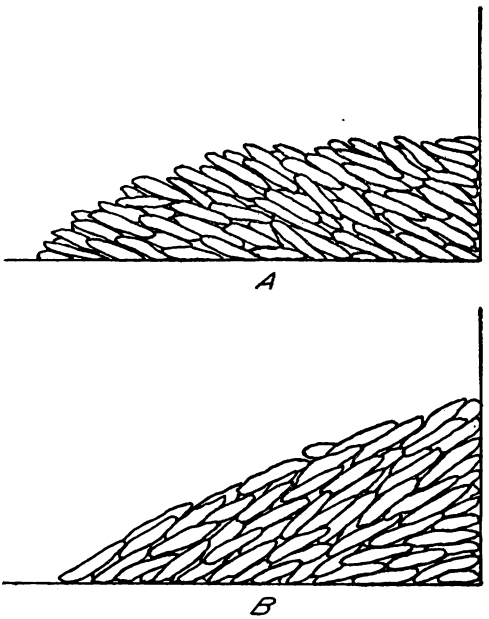


FIGURE 7.—Effects of different methods of placing charge in rock having "slab cleavage": *A*, Manner in which the slabs are bound in by blast that throws base farther than upper part; *B*, favorable position of rock from blast that throws upper part farther than lower part.

THE BROKEN CHARGE.

To obtain a successful blast, proper distribution of the charge in the drill hole is essential. To accomplish the desired distribution, the charge may have to be divided into two or more parts with intermediate stemming. Adjacent holes may be so loaded that explosive in one hole is at the same level as stemming in the next. The point should be emphasized, however, that where the rock is of uniform hardness, the same distribution may be attained by reducing the size of the drill hole and employing a continuous charge. The continuous charge should be used wherever possible.

In ledges where the successive beds of rock vary in soundness or hardness and therefore break with varying degrees of difficulty, proper distribution of the charge is best attained by the use of intermediate stemming. Where the rock varies widely in hardness, it is customary to put explosive in those parts of the holes that intersect the hard beds and stemming in the parts that intersect the softer beds. Where shale is interbedded with limestone, as a rule no explosive is used in that part of the drill hole that intersects the shale, because it is a soft rock and is easily broken. Explosive is therefore placed in the limestone and stemming in the shale.

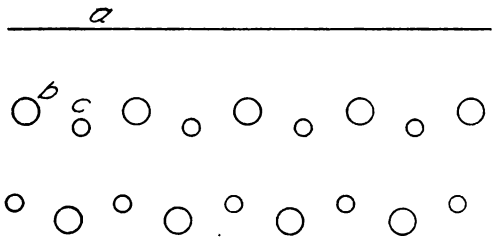


FIGURE 8.—Arrangement of drill holes in an Iowa quarry: *a*, Face; *b*, churn-drill holes; *c*, supplementary tripod-drill holes.

The separation of a charge into five or six parts, with intermediate stemming, has been noted. If electric detonators are employed to fire the charge, a number of detonators are required for a charge of this type.

EXAMPLE OF A BROKEN CHARGE.

In one limestone quarry in Iowa the lower beds are hard and fairly sound; the middle beds have numerous closed seams that cause the rock to break readily; and near the surface is a bed 2 to 6 feet thick of extremely hard blue limestone. Two rows of holes 5½ inches in diameter are made with a churn drill, the holes being staggered, placed 10 feet apart, and having 10 feet of burden. The churn-drill holes are supplemented by tripod-drill holes that are drilled into the hard blue limestone, but no deeper. It is stated that if such supplementary charges were not provided, the blue limestone would break into masses 30 to 40 feet in length and weighing many tons. The arrangement of the holes is shown in figure 8. The distribution of the charge in the churn-drill holes is illustrated in figure 9. Enough "40 per cent" dynamite is placed in the hole to reach to the top of the sound bed. This requires about 60 pounds and, to insure against misfire, two electric detonators are placed in this charge. The middle part of the hole is then filled with stemming up to the hard blue

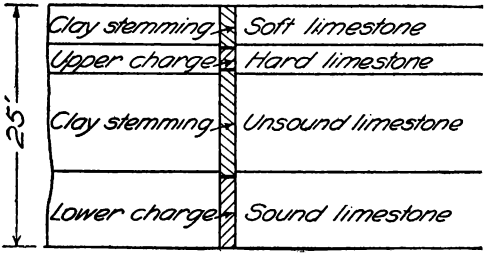


FIGURE 9.—Distribution of explosives in the churn-drill holes shown in figure 8.

bed. In the part penetrating this bed a second charge, consisting of 10 to 15 pounds of "40 per cent" dynamite, is placed. One electric detonator is used in this charge. The rest of the hole is tamped with clay stemming. This method is successful, especially since the introduction of the supplementary tripod-drill holes.

CHARGING HOLES WITH UNEQUAL BURDEN.

Rarely, if ever, are all the holes for one shot charged with the same amount of explosive, the charge being varied to suit conditions. A corner hole, for example, must be loaded heavier than the others to clean out the corner efficiently, or, as a quarryman would say, to "pull the corner." The amount of charge is also influenced by inequalities of the surface. A heavy toe or projections on the face require heavy charges in the holes immediately behind them. If on account of the presence of loose rock a hollow place in the face was not observed in time to modify the spacing of drill holes, the charge must be modified. The holes immediately behind the hollow place should be loaded lighter than the average, and those on either side a little heavier.

EFFECT OF WATER IN DRILL HOLES.

As drill holes are usually completed a considerable time before loading, they are as a general rule partly filled with accumulated water. In some places the explosive is loaded into the holes without removing the water. If a type of explosive designed for submarine blasting is employed, this procedure may be permissible. Usually the use of other types of explosives is desirable, hence removal of the water is advisable. An objection to the presence of water, aside from the possible injurious effect on the explosive, is the resulting decrease in the efficiency of the explosion. It is a well-known law of physics that when the temperature of a gas in a sealed vessel is increased, the tendency of the gas to expand causes a corresponding increase in the pressure on the walls of the vessel. Therefore, the force of an explosion is due in large measure to the high temperature of the gases evolved. Water in the drill hole keeps down the temperature of the gases and reduces their pressure, and therefore detracts from the efficiency of the blast.

REMOVAL OF WATER FROM DRILL HOLES.

All quarrymen are familiar with apparatus for the removal of water from drill holes. A portable hoist operated by a gasoline engine or by electricity is convenient for bailing out deep holes. With the ordinary apparatus employed all the water can not be removed, but the small amount left in the bottom of the hole usually occasions no disadvantage.

LOADING DRILL HOLES.

It is an established fact that the explosive should completely fill that part of the drill hole that it occupies. Even a small air space around the charge greatly weakens the force of the explosion. A common practice is to cut the cartridge and pour the loose dynamite into the holes through a large funnel. This is somewhat dangerous, as loose dynamite may be scattered around the holes and cause a premature explosion. Steps are being taken in one State to make loading of loose dynamite illegal. However, in loading irregular holes in unsound rock the loose charge is preferred by some operators. When cartridges are used they should be large enough to fit the drill hole without danger of jamming, and may be slit in such a manner that they may be easily expanded to fill the hole when pressed with a wooden tamping rod. The cartridges that contain the detonators should not be tamped.

STEMMING AND TAMPING.

It is assumed by some quarrymen that explosives like 40 per cent "straight" nitroglycerin dynamite or of higher grade act so quickly that no stemming is necessary. Experiments conducted by the Bureau of Mines[a] have shown conclusively, however, that the efficiency of 40 per cent "straight" nitroglycerin dynamite is greatly increased by the use of stemming.

Clay is the most common stemming employed, although sand or rock dust is sometimes used. Experiments conducted by the Bureau of Mines[b] show that tamped moist fire clay gave the best results with 40 per cent "straight" nitroglycerin dynamite in four out of six tests with varying amounts of stemming, while tamped moist sand stood second. In the other two tests this order was reversed.

Greenwell and Elsden[c] claim that untamped dry sand is the best stemming. The advantages of sand stemming are thus enumerated:

1. The time taken in loading is reduced to about one-fourth.
2. The effect of the shot is at least as good as with clay stemming.
3. It has been found that with sand stemming rock fragments are not thrown as far as with clay stemming.
4. Misfires are not as common as with clay stemming, as no tamping rod is used and therefore no injury to the fuse or wires can arise.
5. If a misfire should occur, the stemming may be readily removed with a wooden implement without danger.
6. Premature explosions during tamping are impossible when sand is used.

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[a] See Snelling, W. O., and Hall, Clarence, The effect of stemming on the efficiency of explosives: Tech. Paper 17, Bureau of Mines, 1912, 20 pp.

[b] Snelling, W. O., and Hall, Clarence, work cited, p. 19.

[c] Greenwell, Allan, and Elsden, J. V., Practical stone quarrying, 1913, p. 287–288.

7. Sand is usually easily obtained.

8. Sand is more convenient than clay for making shots in open joints.

9. Dry sand does not freeze like clay.

It is also stated that the quarry-industry association of Germany investigated the use of sand stemming, and after numerous experiments the above conclusions were confirmed.

A number of the advantages enumerated are obvious, but some of the claims may be questioned. For example, the statement that sand is as effective as clay is not borne out by the Bureau of Mines tests. According to the tests made by Snelling and Hall, previously referred to, untamped dry sand was much less effective than either tamped moist sand or tamped moist fire clay, particularly where the larger amounts of stemming were employed. However, in the tests made by Snelling and Hall a perfectly smooth bore hole in metal was employed. A drill hole in rock is less smooth and regular, and consequently in actual practice different results may follow.

The claim that with sand stemming rock fragments are not thrown as far as when clay stemming is used is doubtful.

Sand stemming has without doubt many advantages in its favor and its use is recommended.

For blasting in churn-drill holes 35 to 90 feet deep, the depth of stemming used varies from 10 to 25 feet. In one cement-rock quarry in Pennsylvania where 50 per cent nitrostarch blasting powder is used, an average of 22 feet of clay stemming is employed in holes averaging 85 feet deep.

METHODS OF TAMPING.

For holes of moderate depth, tamping may be done with a light wooden pole. A metal tamping bar should never be used.

Attachment of a tamping block to a drill bar and tamping with a drill is extremely dangerous and should never be permitted. On January 10, 1907, a premature explosion occurred at the site of the Pedro Miguel Locks on the Isthmus of Panama. Ten men were killed. As a result of an investigation by the Isthmian Canal Commission it was decided that this and other premature explosions had been caused by the use of tripod rammers, and their further use was prohibited.

For deep holes a tamping block 3 or 4 feet long may be attached to a rope. A convenient device in tamping deep holes is a light tripod with a pulley at the apex over which the rope attached to the tamping block may be operated. The use of a tripod not only makes tamping easier, but also permits the men to work at one side and not directly over the loaded hole. This lessens the danger of accident in case of premature explosion.

GENERAL PRINCIPLES FOR THE SHOT FIRER'S GUIDANCE.

As a rule, a high explosive shatters rock more effectually than a low explosive; consequently, in quarries where many large solid blocks are thrown down by the primary shot, the substitution of a higher grade explosive may give more satisfactory results. However, the use of high-grade explosives is not recommended, as they are expensive and more dangerous to handle. The use of a detonating fuse increases the effectiveness of low-grade explosives and thus permits their use where otherwise a high-grade explosive is required. A variation in the arrangement of drill holes may bring about the same result as changing the type of explosive, for it has been found that, within certain limits, a reduction in the burden and spacing of drill holes causes more thorough shattering of the rock mass.

Drill holes should be of the minimum diameter required to contain the necessary explosive. Intermediate stemming is advisable in certain deposits where enlargements occur in the drill holes, where the beds are variable in soundness, or where for any other reason an adjustment of the powder column seems to be justified. However, if the successive beds are fairly uniform, equally efficient results may be attained by reducing the diameter of the drill holes and using a continuous charge. By this means the charge is distributed throughout a greater length of hole and the inconvenience of employing intermediate stemming is avoided.

Water should be removed from drill holes before loading.

If there is a heavy toe or a poor floor seam it is better to place in the bottoms of the drill holes explosive of higher grade than the main charge.

DETONATING THE CHARGE.

METHODS OF FIRING CHARGES.

In order to obtain the most effective results, a number of holes should be fired simultaneously. As it is physically impossible to fire charges simultaneously with ordinary fuses, electric detonators and an electric current are required. One method of firing charges is to place one or more electric detonators in the dynamite of each hole, connect them all with wires, and fire with an electric current. Another method is to use a detonating fuse in each hole and to fire this fuse at the surface. The fuse itself detonates at a very high rate and fires all the charges.

ELECTRIC DETONATORS.

An electric detonator, also called an electric exploder, is a blasting cap containing a high explosive, such as mercury fulminate, which is

ignited by means of a wire through which the current of electricity passes. This wire is of small diameter and has such a high resistance that it is heated to incandescence, or fused, and thereby fires the priming. Electric detonators are made in various sizes. For ordinary blasts in quarries, No. 6 detonators are in common use. However, it has been proved in many quarries that the efficiency of a blast largely depends on the intensity of the initial explosion, and for this reason the use of No. 8 detonators is recommended. A more complete discussion of electric detonators is given by Hall and Howell.[a]

NUMBER OF DETONATORS FOR EACH CHARGE.

For small blasts in tripod or hammer drill holes one detonator is commonly used for each hole. While this is probably sufficient to fire the charge effectively, the danger of misfires is greatly reduced by using two detonators and wiring them in parallel for live-wire current and in series if a battery is used.

For large blasts in churn-drill holes two or more detonators are commonly used. In one Pennsylvania quarry, where 90-foot holes 5 inches in diameter are loaded with continuous charges of 40 per cent "straight" nitroglycerin dynamite, three No. 6 detonators are used in each hole, one near the bottom, one at the middle, and one near the top. They are wired in parallel and fired with the quarry current.

When broken charges with intermediate stemming are employed, each part of the charge should have at least one detonator, and to insure against misfires two detonators are preferable.

In one quarry in the South it is customary to place several unattached No. 6 detonators in the charge to act as "boosters" for the purpose of increasing the rate of detonation and thereby increasing the effectiveness of the shot. This practice is very dangerous, for a loose cap may lodge in a cavity or on a ledge on the drill-hole wall and may later be fired with the tamping rod, causing a premature explosion.

The results of tests made at the Pittsburgh station of the Bureau of Mines to determine the efficiency of such a method, described in Bulletin 59,[b] indicate that "extra detonators placed at 5 inches apart in a cartridge file of an insensitive explosive 40 inches long have a slight tendency to increase the propagation of the explosion wave, but that extra detonators placed 10 inches apart offer no advantages." It is pointed out that the explosion of the dynamite surrounding the unattached detonators may precede the explo-

[a] Hall, Clarence, and Howell, S. P., Investigations of detonators and electric detonators: Bull. 59, Bureau of Mines, 1913, 73 pp.

[b] See Hall, Clarence, and Howell, S. P., work cited, pp. 68–69.

sion of the detonator and that the latter would, under such circumstances, explode in the products of combustion and therefore offer little if any advantage in extending or increasing the rate of the explosion wave. Hence, the distribution of unattached detonators throughout the charge probably offers little if any advantage.

However, the presence of several electric detonators distributed throughout the charge and connected by wires offers a decided advantage. The detonators are all fired simultaneously and tend to make the detonation of all parts of the charge practically simultaneous, thus producing the maximum shattering effect.

The best type of "booster" is a brass tube 4 to 6 inches in length, containing trinitrotoluene detonating fuse with an electric detonator attached. It increases the rate of detonation and insures more complete detonation of a low-grade explosive. The advantage of using "boosters" with 40 per cent or 60 per cent "straight" nitroglycerin dynamite may be questioned.

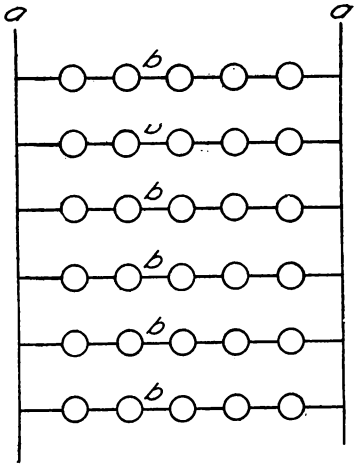


FIGURE 10.—Wiring in parallel series: *a*, Trunk lines; *b*, parallel series.

METHODS OF WIRING.

Two methods of wiring a number of shots to be fired at one time are in common use—wiring in "series" and wiring in "parallel." The methods are fully described in Technical Paper 111.[a] A third method, which is sometimes employed when many holes are fired at one time, is a combination of the above methods, and may be termed a "multiple series" or "parallel series" connection. When shots are fired in parallel a current of one ampere per detonator is recommended; therefore the use of a multiple connection for wiring a large number of holes would require a greater amperage than is usually available, and also would increase the difficulty of supplying suitable wiring. Connecting a large number of holes in series would require a high voltage and would involve the danger of a misfire in the event that insufficient voltage were available. Therefore, a method recommended is to connect small groups of holes in series and to connect these series in parallel, as shown in figure 10. When holes are connected in this manner it is important that each subseries should have the same resistance.

[a] Bowles, Oliver, Safety in stone quarrying: Tech. Paper 111, Bureau of Mines, 1915, pp. 21–22.

THE USE OF POWERFUL DETONATORS RECOMMENDED.

A point that requires special emphasis among quarrymen is the advisability of using strong detonators. As stated by Rutledge,[a] "when high explosives are detonated the stronger and quicker the action of the detonator the greater will be the shock given the explosive, and the more effective will be the explosive." A weak detonator may fire the charge, but may cause a very inefficient explosion, particularly when the more insensitive explosives are employed. The use of weak detonators simply because they are cheaper than strong ones is not wise. An imperfect detonation will probably result in a loss amounting to many times the price of all the detonators employed. No detonators weaker than No. 6 should be used with high explosives, and with the more insensitive explosives, such as gelatin or ammonia dynamite, or nitrostarch blasting powder, No. 8 detonators should be used. One blasting engineer advises the use of No. 8 detonators for all shots.

NUMBER AND ARRANGEMENT OF DETONATORS IN CHARGE.

Manufacturers of electric detonators claim that the priming exerts its greatest force in the direction of the loaded end. Consequently, the detonator shell should be so placed in the charge that its loaded end is toward the greater mass of explosive to be detonated. In any event the long axis of the shell should parallel the drill hole, and it should be entirely embedded in the explosive.

As a safety precaution, not less than two detonators should be placed in each charge. If a broken charge is employed, each separate part of the charge should have at least two detonators. Where heavy charges are fired in churn-drill holes, four or five detonators are commonly used in each hole, being distributed regularly throughout its length. Enough detonators should be used to insure sudden and complete detonation of the charge. Detonators cost only a few cents each, and the use of too few or too weak detonators may mean the loss of many dollars, and cause an accident if part of the charge is unexploded.

Each electric detonator should be tested with a reliable galvanometer. If any detonator shows abnormally high resistance, it should not be used when firing with a battery or where connected in series. The entire circuit should be tested in the same way when all connections are made except to the blasting machine or power circuit. In testing a circuit of holes with a galvanometer the instrument should have long lead wires and should always be placed at a safe distance. Ordinarily the current used in a galvanometer is much too

[a] Rutledge, J. J., The use and misuse of explosives in coal mining: Miners' Circular 7, Bureau of Mines, 1914, p. 12.

weak to fire an electric detonator, but some galvanometers are operated with a shunt circuit, and a defect in wiring or connection may throw the entire current from a cell battery through the shunt. Such a current has been known to fire an electric detonator. Consequently, for the sake of safety, the lead wires should be long.

For parallel connection there should be one ampere of current for each detonator, and for series or parallel series, at least two amperes for each series. When a blasting machine is used, the charges should always be connected in series.

The provision of ample current is important. Using a current much stronger than is required to insure against misfires has other advantages. Experiments by the Bureau of Mines[a] have shown that, owing to slight variations in electric detonators, those employed in a multiple shot do not all detonate at exactly the same moment; also, that when a weak current is used the time intervals between the detonations are longer than when a strong current is employed. Therefore, with a strong current the discharge of the various detonators is more nearly simultaneous, and consequently the blast is more effective.

DETONATING FUSE.

NATURE OF FUSE.

The type of detonating fuse known to the trade as "cordeau" is now in common use. It consists of a lead tube filled with trinitrotoluene and carefully drawn to uniform size. The diameter of the fuse commonly used is 6 mm., or $\frac{15}{64}$ inch. Also, 5.5 mm. size is used. Cordeau has been known for a number of years in France and England, but until recently it has been used in the United States only as a means of measuring the rate of detonation of high explosives.[b] During the last three or four years, however, cordeau has been used rather extensively in mining, quarrying, or other blasting work where large multiple shots are employed. It has been found especially advantageous in quarries where great masses of rock are thrown down with large charges of explosives in churn-drill holes.

METHOD OF USING DETONATING FUSE.

After the drill holes have been cleaned, and before loading is begun, a piece of detonating fuse is inserted into each hole. The fuse extends the full depth of the hole and 6 to 10 inches above the surface. After the holes are loaded and tamped, a main line of fuse is placed on the surface and attached to the branches from each hole. A detonator is attached to the end of the main line, and is fired either

[a] Hall, Clarence, and Howell, S. P., Investigations of detonators and electric detonators: Bull. 59, 1913, p. 71.

[b] See Hall, Clarence, and Howell, S. P., work cited, pp. 66–68.

by fuse or by electric current, thus firing the detonating fuse. The detonation wave flashes along the main line and into each drill hole. As the rate of detonation of the trinitrotoluene is very high, all the charges are fired practically at the same moment. The method of placing the fuse is shown in figure 11.

EFFICIENCY IN THE USE OF DETONATING FUSE.

It is claimed by Hall[a] that the use of detonating fuse is not as advantageous in the United States as in Europe, owing to the fact that the explosives used in the United States are more sensitive to detonation than those used in European quarries and mines. Explosives of the ammonia nitrate class used in the European countries are insensitive, and when detonators are used incomplete detonation may occur, whereas with cordeau detonation is more sudden and complete. In the United States, where nitroglycerin dynamites are com-

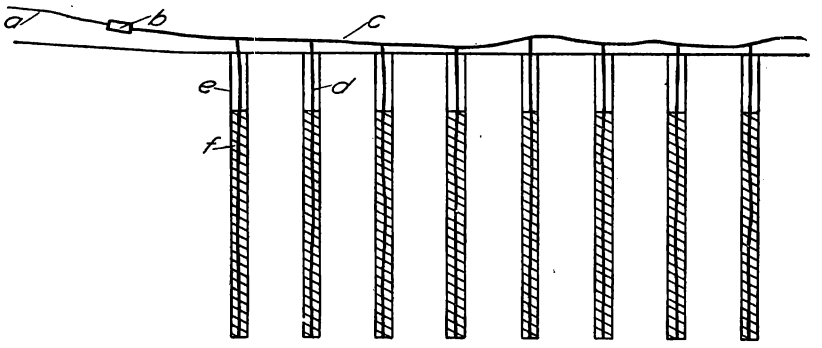


FIGURE 11.—Arrangement of detonating fuse in a circuit of holes: *a*, Wire; *b*, electric detonator; *c*, main line of fuse; *d*, branch line of fuse; *e*, stemming; *f*, charge of explosive in drill hole.

monly used, complete detonation may be easily brought about with electric detonators.

Also, the fact is generally recognized that for small shots in tripod or hammer drill holes the use of cordeau offers little if any advantage.

However, for deep-hole blasting, even where dynamite of the nitroglycerin class is used, detonating fuse has a number of advantages over electric detonators. Recognition of this fact has led to its recent introduction at a number of quarries in various parts of the United States. The different advantages may be summarized as follows:

Cordeau of 6-mm. diameter has a rate of detonation upwards of 15,000 feet a second. On this account the detonation of all the fuse used for a blast is practically instantaneous, and as fuse is in contact with each charge throughout the entire vertical depth it fires every

[a] Hall, Clarence, Discussion of paper, "A new safety detonating fuse," by Harrison Souder: Am. Inst. Min. Eng. Bull., April, 1915, p. 895.

part of the charge practically at the same moment. Not only is incomplete detonation impossible, but the simultaneous discharge increases the disruptive effect of the explosion. In general, therefore, for deep-hole blasting a smaller charge with cordeau will give as good results as a larger charge with electric detonators.

The rate of detonation of the fuse as compared with the rates of detonation of certain high explosives is given in the following table.[a]

TABLE 11.—*Rates of detonation of trinitrotoluene detonating fuse and of certain samples of high explosives.*

Class and grade of explosive.	Diameter of explosives tested.		Rate of detonation, feet per second.	Authority.
	Inches.	Millimeters.		
Trinitrotoluene detonating fuse	12/64	4	13,750	Bureau of Mines.[a]
Do	24/64	6	15,170	Do.
Do	.....	.....	17,000	Manufacturer.
60 per cent "straight" nitroglycerin dynamite	1¼	32	20,490	Bureau of Mines.
50 per cent "straight" nitroglycerin dynamite	1¼	32	18,850	Do.
60 per cent strength low-freezing dynamite	1¼	32	16,640	Do.
40 per cent strength gelatin dynamite	1¼	32	16,210	Do.
40 per cent "straight" nitroglycerin dynamite	1¼	32	15,650	Do.
30 per cent "straight" nitroglycerin dynamite	1¼	32	14,920	Do.
40 per cent strength ammonia dynamite	1¼	32	10,350	Do.
5 per cent granulated nitroglycerin powder	1½	38	3,340	Do.

[a] Data for Bureau of Mines tests are from Bulletin 48, The selection of explosives used in engineering and mining operations, by Clarence Hall and S. P. Howell, Bureau of Mines, 1914, p. 44.

It may be noted that the rate of detonation of the fuse is much higher than that of the 40 per cent strength ammonia dynamite, and about the same or a little lower than that of the 40 per cent "straight" nitroglycerin dynamite. One important advantage of the detonating fuse is that it increases the rate of detonation of any low-grade high explosive, for example, an explosive of the nitroglycerin class having a low content of nitroglycerin. Howell[b] has shown that the rate of detonation of a low-grade high explosive is increased to the rate of the detonating fuse extended through it. The disruptive effect of the low-grade explosive is thereby increased, but is not increased proportionally; that is, the disruptive force of a low-grade explosive detonated with cordeau is not as great as that of a similar type of explosive having a normal rate of detonation equal to that of cordeau. However, the disruptive force of the low-grade explosive is materially increased. Thus it may be seen that the quarryman may substitute low-grade and relatively cheap explosives for the higher grades ordinarily used and still maintain normal efficiency.

[a] Howell, S. P., Discussion of paper "A new safety detonating fuse," by Harrison Souder: Am. Inst. Min. Eng. Bull. 100, April, 1915, p. 897.

[b] Howell, S. P., place cited.

It is interesting to note, however, that even when 40 per cent "straight" nitroglycerin dynamite or other types of higher grade are used, better results are attained by using cordeau than by electric detonators.

In one Indiana quarry the writer observed the result of a blast where cordeau had been used for the first time in this quarry. The explosive employed was 40 per cent nitrostarch blasting powder. The charge was approximately as large as was commonly used with electric detonators. The effect of the shot was to blow out a deep trench and to throw the rock so far that it buried tracks in the quarry. Evidently the charge was too heavy.

The writer observed a subsequent blast for which the charge was reduced about 25 per cent. For this blast 41 holes 19½ feet deep were drilled, the holes being arranged in two rows and staggered. The first row had about 10 feet of burden, and the holes were 12 feet apart. The second row was about 18 feet from the face, and the holes were 12 feet apart. An average of 45 pounds of "40 per cent" nitrostarch blasting powder was placed in each hole. For the blast referred to in the previous paragraph, 60 pounds was the average charge for each hole. The holes were tamped with clay and the shot discharged with cordeau fired by a single No. 6 detonator connected with the quarry current. The explosion shattered but did not throw the rock. A considerable number of blocks were large, requiring excessive secondary blasting. In the writer's opinion the size of the charge was reduced too much; a reduction of 10 to 15 per cent, rather than 25 per cent, in the amount of explosive employed would undoubtedly have given better results. The necessary reduction in the amount of the charge as compared with that employed with detonators is an excellent demonstration of the increased efficiency that results from the use of detonating fuse with an insensitive explosive.

In addition to the gain in shattering efficiency, it is claimed that the time required for loading holes is greatly reduced by the use of the detonating fuse.

SAFETY IN USE OF DETONATING FUSE.

The detonating fuse is much safer than detonators in storing or handling. Howell[a] states that trinitrotoluene is much less sensitive than either 40 per cent "straight" nitroglycerin or mercury fulminate. Trinitrotoluene can not be exploded by friction, fire, or ordinary shock, but requires the shock of a detonator. The fuse is not affected by heat or cold, but may be affected by moisture at an open end or where connections have been made.

[a] Howell, S. P., work cited, p. 896.

The increased safety from the use of cordeau can best be appreciated by briefly reviewing the dangers that beset the ordinary method of firing a charge with electric detonators.

A detonator is usually placed at a point not far from the bottom of the drill hole. During all subsequent loading and tamping this detonator and others which may be successively placed in the hole are a menace to life and limb. Careless handling of the detonators or heavy, careless tamping of the charge may cause a premature explosion. In the event of a misfire, the presence of unexploded detonators along with dynamite in the rock mass is a source of extreme danger, as a blow from a hand sledge or from the tooth of a steam-shovel dipper may result in a serious or fatal accident. Occasionally "blind" misfires may occur; that is, one or more holes may not detonate, and, on account of the great mass of rock thrown down, this fact may be hidden from the quarryman. If the men are unaware of this danger no precautions may be taken and the risk will be greater than where misfires are known to have occurred. Incomplete detonation of a charge is also a source of danger. Unexploded dynamite in the rock, although less dangerous when no detonators are present, is a source of grave danger where metal tools are constantly employed.

When detonating fuse is used the drill holes contain no detonators, and the one used to fire the fuse need not be brought until everything else is prepared, the placing of the detonator being the last act in preparing a shot. Should a misfire occur, the lesser danger of the unexploded dynamite would be present, but the greater danger of unexploded detonators would be removed. However, misfires are rare when cordeau is used. They can occur only through serious defects in manufacture of the fuse, defective connections, or severance of the line in tamping. If due care is exercised, the danger of misfires is remote. "Blind" misfires can scarcely occur when cordeau is used, for the presence of the unexploded fuse could readily be observed.

When detonators are used misfires may be caused by attempting to fire so many holes that the firing machine is overloaded. With cordeau this danger is eliminated, for with proper connections an unlimited number of charges may be fired with one or two detonators.

Furthermore, the more efficient results from the use of detonating fuse permits in some places of the substitution of low-grade or insensitive explosives for those of higher grade. Such substitution decreases the danger, for it is well known that low-grade or insensitive dynamite is much safer to handle than high-grade dynamites, especially those of the "straight" nitroglycerin type.

PRECAUTIONS TO BE TAKEN IN USING DETONATING FUSE.

In placing detonating fuse in deep holes there is danger of the fuse breaking from its own weight. As pointed out by Howell,[a] the 6-millimeter fuse weighs about 10 pounds per 100 feet. Experiment showed that a piece of fuse 1 foot long would not support a weight of 34 pounds. Hence jerking or otherwise carelessly handling the fuse might easily break it. Such fuse is now prepared with a covering of fabric which greatly increases its strength. It is wise, therefore, to use covered fuse for deep holes.

A second danger is in tamping. The stemming employed should contain no sharp flakes or angular fragments of rock which might, when tamped, separate the fuse. Some misfires are presumed to have been caused in this way. Careless handling of the tamping stick itself might damage the fuse. The fuse protected with cotton covering is less liable to damage in tamping.

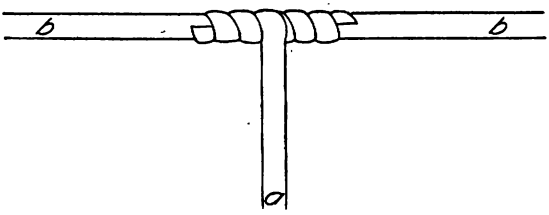


FIGURE 12.—Proper method of connecting branch lines and main line of detonating fuse: *a*, Branch line; *b*, main line.

It is important that connections between the main line and the branches be properly made. The end that projects from the drill hole should be split for a distance of several inches and the main line placed in the crotch of the split. The two halves are then twisted about the main line in opposite directions, as shown in figure 12. It is important that the branch lines should be split and not the main line. Tests by the Bureau of Mines[b] have proved that to detonate cordeau through the lead covering is very difficult. With the method described it may be seen that the lead-covered fuse is the conductor of the detonation wave, and when it explodes, the branches, which are split in such a manner as to expose the explosive within, are easily fired. The union may be wound with tape to make it more secure and to keep out moisture. The branch line should be conducted away from the main line at right angles, at least for an inch or two, but at greater distances curves cause no difficulty. After all joints are made they should be carefully inspected before an attempt is made to fire the charge.

Another method of making connections with the branch lines consists in eliminating the main line of cordeau entirely. A detonator is placed on the end of the fuse extending from each hole. Then the

[a] Howell, S. P., work cited, p. 898.

[b] Hall, Clarence, and Howell, S. P., Investigations of detonators and electric detonators: Bull. 59, Bureau of Mines, 1913, pp. 22–24.

detonators are wired in series and fired with an electric current. By this method each hole is an independent unit, and the danger of misfires is thereby increased.

The method of attaching a detonator to the detonating fuse is shown in figure 13. If the detonator is to be fired with ordinary fuse it must be crimped to the fuse in the ordinary manner. Usually, however, electric detonators are employed. The use of a No. 8 det-

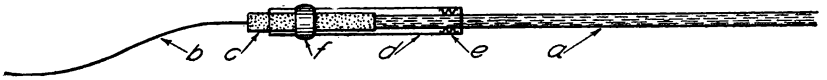


FIGURE 13.—Method of attaching electric detonator to detonating fuse: *a*, Fuse; *b*, wire; *c*, detonator; *d*, brass union; *e*, crimp in union; *f*, brass ring. After Souder.

onator is recommended. The first step in making the attachment is to cut off the end of the detonating fuse squarely. A brass union, which is slit at one end, is then slipped over the fuse, with the slit end projecting. The union is crimped to the fuse at the end opposite the slit. The detonator is then inserted into the union until it comes firmly against the end of the fuse. A space of one-eighth of an inch between the detonator and the fuse may be sufficient to cause a misfire. A ring is then slipped down over the union, and as the slits allow the union to close the detonator is held firmly in place.

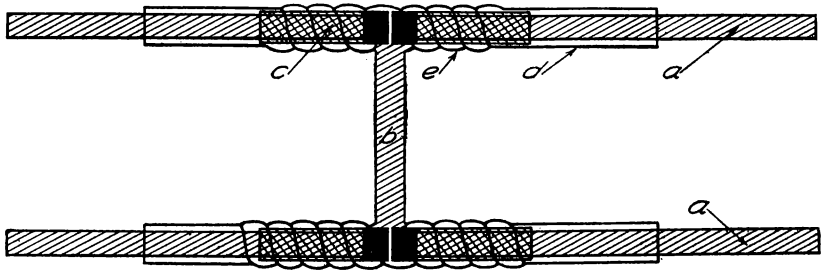


FIGURE 14.—Method of making cross connection with detonating fuse: *a*, Main lines of fuse; *b*, connecting branch of fuse; *c*, electric detonator; *d*, brass sleeve; *e*, winding of branch about main line.

If many holes are fired at one time, requiring two or more main lines, as a matter of precaution it may be advisable to make cross connections between the main lines. A method of making cross connections is described by Souder.[a] The main line is cut and electric detonators placed on the ends, which are then united by a brass sleeve firmly crimped to the fuse. The connecting fuse is then split and wound round the brass sleeve in the ordinary manner, as shown in figure 14.

[a] Souder, Harrison, A new safety detonating fuse: Am. Inst. Min. Eng., Bull. 94, October, 1914, p. 2555.

The arrangement of detonating fuse and method of wiring for a blast in six rows of holes in a Pennsylvania quarry is shown in figure 15. To insure against misfire two detonators are employed, and each loop of fuse is fired at both ends.

RELATIVE COST OF FUSE AND DETONATORS.

A price quoted for cordeau in 1916 was $57.50 per 1,000 feet, bare, and $62.50 per 1,000 feet, covered. For a 20-hole shot, with the holes 50 feet deep and arranged in two rows with 12-foot burden and 14-foot spacing, the total cost of cordeau, including a surface main line, would be about $78. Two electric detonators for each hole and

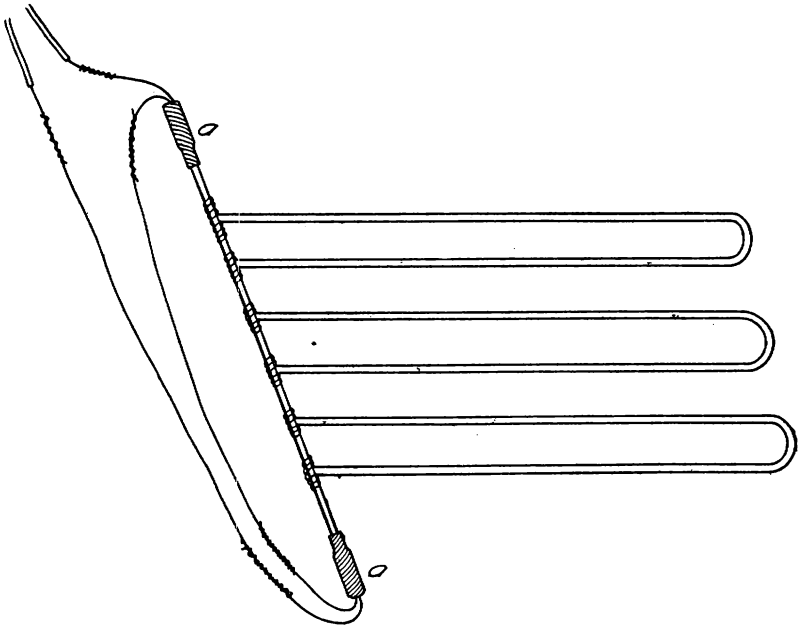


FIGURE 15.—Arrangement of surface lines of detonating fuse and wiring for a blast in a Pennsylvania quarry. Six rows of holes; double lines are detonating fuse; single lines are wires; *a a* are electric detonators.

the necessary wiring would cost approximately $20. Though cordeau is more expensive than detonators in firing a charge, the increased efficiency with cordeau permits a compensating decrease in the size of the charge, particularly when gelatin or ammonia dynamite or nitrostarch blasting powder are used. The use of a detonating fuse for deep-hole blasting is to be recommended on the basis of economy and of safety.

EXAMPLES OF SHOTS IN WHICH DETONATING FUSE WAS EMPLOYED.

The details of a shot observed by the writer in a quarry near Bath, Pa., are given below.

The shot was in cement rock—that is, argillaceous limestone. The charge was distributed in 20 holes averaging 85 feet deep, the average depth of clay stemming used being 22 feet. "Fifty per cent" nitrostarch blasting powder was used. As shown in figure 16, the

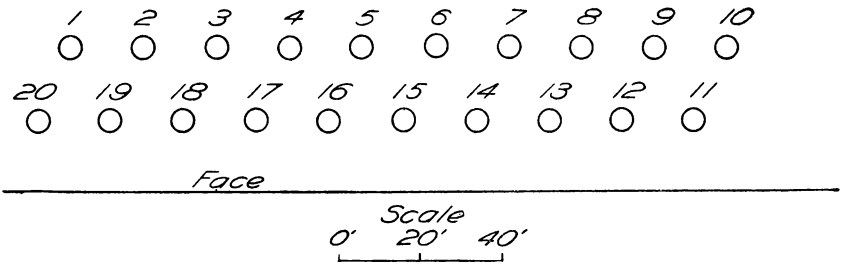


FIGURE 16.—Arrangement of drill holes for a blast in a quarry near Bath, Pa.

holes were arranged in two rows, staggered, and spaced 18 feet apart, the first row being 18 feet and the second row 36 feet from the face. The depth of the holes, depth of stemming, and number of pounds of explosive in each hole is shown in Table 12.

TABLE 12.—*Depth of drill holes, depth of stemming, and number of pounds of dynamite used for a typical blast in cement rock.*

Number of drill hole.	Depth of drill hole.	Depth of stemming.	Dynamite used.
	Feet.	*Feet.*	*Pounds.*
1	97	19	1,000
2	92	21	900
3	93	22	850
4	89	21	850
5	78	20	750
6	78	20	750
7	76	18	650
8	74	18	500
9	75	18	500
10	71	22	500
11	76	29	550
12	80	23	700
13	82	25	750
14	85	20	750
15	86	26	750
16	88	24	800
17	90	27	850
18	92	23	800
19	95	27	950
20	95	25	900

The shot was fired by means of cordeau in two main lines without cross connections. As far as the writer could judge by ear and eye, the explosion was simultaneous. A mass of rock of approximately 46,080 tons was thrown down and was so completely broken that little secondary blasting was required. The total cost of explosive, including fuse, was about 3.7 cents per ton of rock obtained. Approximately 3 tons of rock were thrown down for each pound of explosive.

A description of one of the largest blasts ever made in a limestone quarry in the United States will be of interest. *A*, figure 17, is a plan showing the arrangement, burden, and spacing of the drill holes, and *B* is the profile showing the depths of the drill holes. There were 46 vertical holes of an average depth of 91 feet 11 inches,

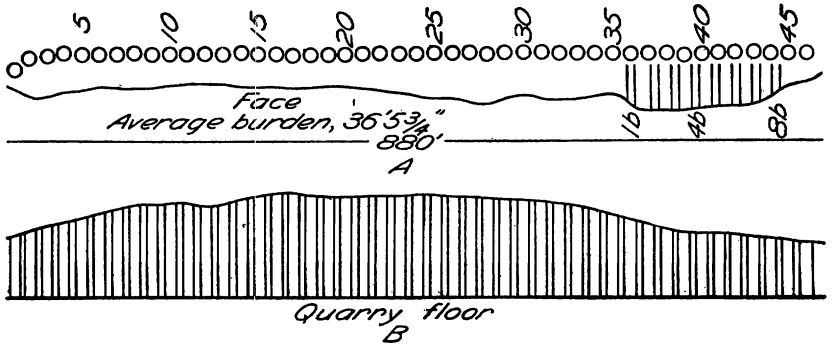


FIGURE 17.—Arrangement of drill holes for a large blast in a Tennessee quarry: *A*, Plan: *B*, profile.

an average burden of 36 feet 5¾ inches, and an average spacing of 19 feet 7 inches. Eight horizontal "snake holes" were drilled at that part of the face where the burden was heaviest. The length of the face was 880 feet. The spacing, burden, depth, and amount of explosive for each drill hole are shown in Table 13.

TABLE 13.—*Data for a large blast in a limestone quarry.*

Hole No.	Spacing.	Burden.	Depth.	Explosives used.			Cost.
				60 per cent gelatin dynamite.	60 per cent straight nitroglycerin dynamite.	Low-freezing dynamite.	
	Ft. in.	*Ft. in.*	*Ft. in.*	*Pounds.*	*Pounds.*	*Pounds.*	
1		35 0	66 5	.....	415	153	$70.90
	23 9						
2		38 0	70 1	.....	414	208	76.89
	18 8						
3		38 0	75 0	.....	315	263	70.10
	20 0						
4		37 0	77 6	.....	463	208	83.26
	21 6						
5		35 0	81 5	101	411	155	83.74
	18 3						
6		37 0	86 3	166	515	102	99.84
	21 10						
7		35 0	91 4	313	316	99	92.73
	17 0						
8		35 0	91 11	359	360	199	115.53
	16 0						
9		34 0	96 9	314	311	45	98.00
	20 6						
10		33 0	98 7	259	323	151	80.66
	20 6						
11		32 0	97 9	354	276	101	93.10
	18 3						

TABLE 13.—*Data for a large blast in a limestone quarry*—Continued.

Hole No.	Spacing.	Burden.	Depth.	Explosives used.			Cost.
				60 per cent gelatin dynamite.	60 per cent straight nitroglycerin dynamite.	Low freezing dynamite.	
	Ft. in.	*Ft. in.*	*Ft. in.*	*Pounds.*	*Pounds.*	*Pounds.*	
12	21 3	31 0	100 1	408	279	49	$94.74
13	20 6	29 0	103 6	412	370	47	106.86
14	20 0	26 6	104 5	412	321	98	106.15
15	17 3	30 0	106 2	307	375	170	107.51
16	21 0	31 0	107 7	357	503	20	114.00
17	18 6	29 0	107 9	314	415	100	105.85
18	19 10	31 6	108 9	310	369	65	95.47
19	21 6	31 0	108 1	309	429	157	113.34
20	20 0	24 0	109 2	257	431	101	100.64
21	21 8	25 6	109 8	312	339	103	96.05
22	17 6	29 0	108 8	314	369	151	105.53
23	20 3	31 0	107 9	309	361	152	103.96
24	19 10	32 0	107 0	362	364	101	105.58
25	16 0	35 0	106 1	408	372	98	112.25
26	19 2	39 0	105 4	421	335	101	109.48
27	20 0	40 6	104 10	474	382	48	116.61
28	20 2	43 0	104 5	472	374	263	139.14
29	21 6	44 0	104 2	391	247	152	99.79
30	20 3	44 0	103 6	413	374	262	131.36
31	23 10	43 6	103 2	417	431	49	115.69
32	20 0	42 0	101 2	356	279	47	87.75
33	16 0	40 0	98 7	416	185	146	94.33
34	21 0	40 0	94 11	419	330	99	108.34
35	21 6	39 6	90 0	412	237	48	89.69
36	22 6	39 0	84 0	412	233	101	95.05
37	20 0	.....	79 9	316	318	101	93.61
38	17 0	.....	79 1	415	229	98	94.58
39	19 6	.....	76 1	315	534	102	121.68
40	16 6	.....	73 3	258	386	188	104.56
41	15 0	.....	73 11	306	243	132	86.00
42	16 6	.....	71 0	260	305	108	85.42
43	16 0	.....	67 7	260	195	103	70.57
44	14 4	33 0	64 8	312	242	53	77.89
45	12 2	31 0	62 0	207	305	106	78.31
46	.....	31 0	60 4	206	241	106	69.86
1*b*	.....	.....	32 0	259	194	210	82.16

TABLE 13.—*Data for a large blast in a limestone quarry*—Continued.

Hole No.	Spacing.	Burden.	Depth.	Explosives used.			Cost.
				60 per cent gelatin dynamite.	60 per cent straight nitroglycerin dynamite.	Low-freezing dynamite.	
	Ft. in.	*Ft. in.*	*Ft. in.*	*Pounds.*	*Pounds.*	*Pounds.*	
2B	14 8	.....	35 6	311	336	160	$101.84
3B	13 0	.....	35 0	419	238	453	135.61
4B	13 6	.....	34 0	201	183	218	74.07
5B	15 0	.....	29 0	206	198	220	76.91
6B	16 0	.....	16 0	150	234	167	68.42
7B	14 4	.....	17 0	200	306	109	77.86
8B	13 0	.....	19 0	200	201	159	69.73
Total.	.....	.....	.....	16,061	17,711	7,205	5,188.99

The 12 snake holes required 548 pounds of "60 per cent" gelatin dynamite. A total of 7,130 feet of cordeau was used, the cost of which was $304.95. The following summary indicates the high efficiency of the shot:

Results of shot.

Total weight of explosives, pounds	41,525
Total cost of explosives, including dynamite, fuse, and caps	$5,568.37
Weight of rock shot down, tons	242,096
Cost of explosive, cents per ton of rock moved:	
Primary shot	2.30
Secondary shots	1.37
Total cost of explosives	3.67
Quantity of rock moved per pound of explosive, tons	5.83

Plate IV shows the quarry face prior to the blast, and Plate V, the new face and the mass of shattered rock resulting from the blast. Plate VI illustrates the blast itself, photographed at the moment of detonation.

GENERAL CONSIDERATIONS IN BLASTING.

PROPER BALANCE BETWEEN EXPLOSIVES COST AND DRILLING COST.

A given quantity of rock may be broken with less explosive by drilling a large number of small holes closely spaced than with a smaller number of larger and more widely spaced holes. Therefore, within certain limits the cost of explosives evidently will decrease as

the cost of drilling increases, and vice versa. If either factor is overemphasized, blasting efficiency is decreased. It is important, therefore, that the quarryman attain a proper balance between these two costs. The soundness of the rock, the rate of drilling, the ease or difficulty of shattering, the method of loading rock, and other considerations must all be taken into account. A ratio of drilling costs to explosives costs that might prove the cheapest in one quarry might not be suited to another quarry. The quarryman must determine by experiment the relative costs of drilling and explosive that will give the lowest aggregate cost of the two.

PROPER BALANCE BETWEEN BLASTING EFFICIENCY AND ROCK-LOADING EFFICIENCY.

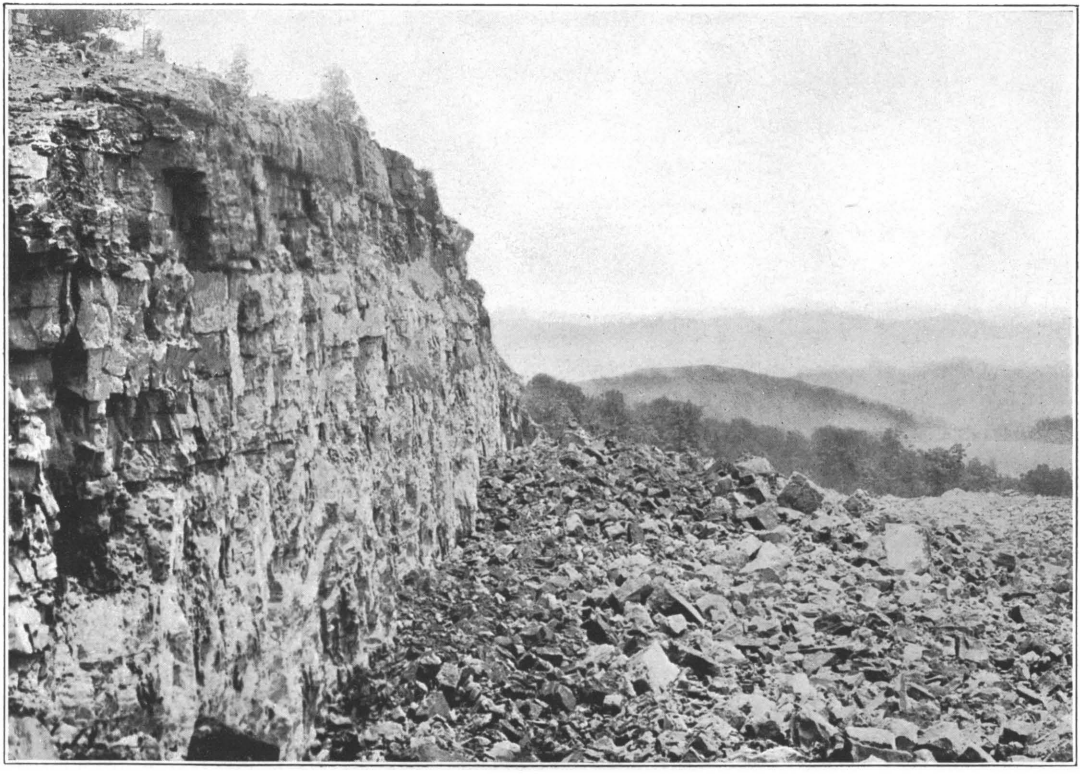
All quarrymen know that the size of a charge must be so regulated that the rock will not be thrown too far. Aside from obstructing and damaging of tracks, a wide, thin mass of rock is more expensive to load with a steam shovel than a thicker mass which covers a smaller area. It may be assumed that loading can be done most efficiently when the face of the rock pile stands at its maximum angle of repose, that is, as steep as the fragments will lie without rolling down. This condition is most easily attained in comparatively low benches where "blanket" blasting may be employed. The buffer of shattered rock prevents fragments scattering and thus aids in maintaining a steep face. With a high face the proper adjustment of the charge, both as to spacing of drill holes and quantity of explosive used, is a more difficult matter.

Moreover, there is an intimate relation between rate of loading and efficiency in blasting. If light charges are employed the explosives costs may be low, but the rock may be so imperfectly shattered that loading is repeatedly delayed while blocks ahead of the steam shovel are being broken by secondary blasting. Owing to such delays, the daily tonnage loaded with each shovel may be small.

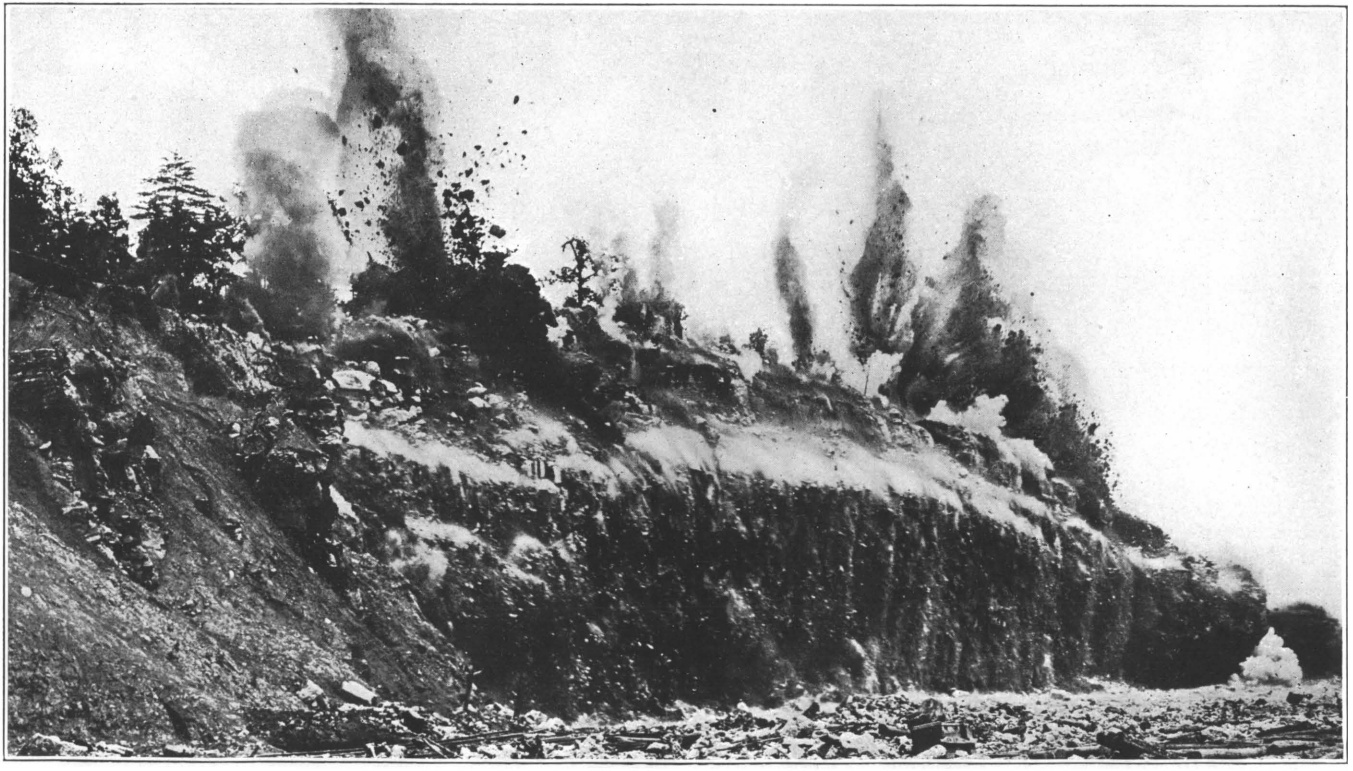
PROPER BALANCE BETWEEN SIZE OF BLAST AND SIZE OF STEAM SHOVEL.

In some quarries where "blanket" blasting is employed on a low face, blasting may precede steam-shovel loading by a long period of time. Under such conditions the two operations are independent of each other insofar as size of blast and width of steam-shovel cut are concerned. In other quarries steam-shovel loading follows blasting closely. In blasting on a high face the rock brought down by one blast is all removed before another mass is shot down, but in "blanket" blasting on a low face, a definite width of shattered rock may be left for a buffer. In either case the position of the steam-shovel tracks for the final cut is guided by the position of the face. As pointed out on a subsequent page (p. 123), one of the necessary

FACE IN TENNESSEE QUARRY JUST BEFORE A LARGE BLAST.



APPEARANCE OF THE QUARRY AFTER THE BLAST.



BLAST PHOTOGRAPHED AT THE MOMENT OF DETONATION. NOTE THE "SNAKE-HOLE" SHOT, IN LOWER RIGHT-HAND CORNER.

conditions for efficient loading is that each cut shall be the maximum width the shovel is capable of handling. Therefore it is important that the mass to be removed is of such width that it may be taken in full cuts. Usually the mass is removed in two or three cuts. The actual number of cuts varies because it depends on the size of the shovel and the magnitude of the mass that may be shot down efficiently at one time, but it is important that the mass shall not require a narrow cut to complete its removal. For example, if the mass to be removed is $2\frac{1}{2}$ cuts wide, the expense of moving the track and the number of moves for the shovel are the same for the half cut as for a full cut, consequently the cost of loading the half cut is excessive. Therefore the blasting foreman should so judge the magnitude of the charges that the mass of shattered rock resulting from the blast shall when lying at the maximum angle of repose be of such a width that it may be removed at one, two, or more full cuts. On the other hand if it is found that a blast of a certain size gives greater efficiency than those of larger or smaller size, then the size of the steam shovel should be such that it can load the entire mass in full cuts. The proper adjustment may therefore be made by varying either the size of the blast or the size of the steam shovel, the method of adjustment being governed by quarry conditions.

SECONDARY BLASTING.

DEFINITION OF SECONDARY BLASTING.

The term secondary blasting, as previously stated, is applied to the process of breaking with explosives the larger pieces resulting from the primary shots into sizes convenient for loading. The process is known locally as "blistering" or "bulldozing." Secondary shots are also occasionally termed "subsequent" shots.

CONDITIONS GOVERNING THE EXTENT OF SECONDARY BLASTING.

The amount of secondary blasting required depends on three main factors: (1) The nature of the rock, (2) the efficiency of the primary shot, and (3) the method of loading rock.

Rock that breaks easily and splits readily parallel with the bedding, like much of the cement rock of the Lehigh district, requires little secondary blasting, as the primary shots break the rock into small pieces. However, many limestone ledges break with difficulty, and many of the fragments resulting from the primary shot may be of large size. An excessive number of large blocks may also result from the presence of open joints which intersect the rock ledge and divide it into numerous independent masses. The masses in which the holes for the charges are drilled will probably be shattered sufficiently,

but those having no drill hole in them will probably be only slightly shattered, because the seams act as buffers and tend to check the force of the explosion. As a result, such masses may be merely pushed out and may not be broken. As a rule, the drill holes for primary shots should be more closely spaced in seamy than in sound rock, for such an arrangement promotes shattering and decreases the amount of secondary blasting necessary.

Also, light charges tend to throw the rock down in large fragments, whereas heavier charges would probably break it up much better. Furthermore, a low-grade explosive breaks the rock less efficiently than a high-grade explosive. In a number of quarries, "60 per cent" dynamite is used for all primary shots, for it is claimed that such an explosive breaks the rock much better than "40 per cent" dynamite. In quarries where the rock is difficult to break into fragments of convenient size, the quarryman should aim to use explosives that have a high disruptive effect rather than those the effect of which is mainly propulsive.

Secondary blasting is influenced greatly by the method of loading the rock. Rock loaded by hand must be broken into sizes that one man can handle, or can easily be reduced by sledging to the desired size, whereas steam shovels can load fragments of large size. Thus the expense for secondary blasting is usually much higher where hand loading is employed than where steam shovels are used.

METHODS OF SECONDARY BLASTING.

Two methods are in common use in secondary blasting, known as the "mud-capping" or "adobe" method, and the "block holing" method. The mud-capping method is known locally by other names such as "dobeying," "blistering," or "bulldozing," though some of these terms are applied to any method of secondary blasting. The method consists in placing a stick of dynamite, with a fuse attached, on the surface of the rock to be broken, and covering the dynamite with a mass of clay which tends to confine the explosion and direct it toward the rock. In the block-holing method, holes several inches deep are drilled in the blocks with hammer drills, and a stick, or part of a stick, of dynamite is placed in each hole. The rock dust that results from drilling is commonly used for stemming. Comparatively low-grade explosive is employed, "30 per cent" dynamite being commonly used. Whatever method is employed, a number of blasts are prepared, the fuses lighted, and the blasts thus discharged in rapid succession. In one quarry in the Lehigh district, electric detonators are placed in the charges, connected by wires, and fired simultaneously with an electric current.

ADVANTAGES OF "BLOCK-HOLING" OVER "MUD-CAPPING" METHOD.

Mud-capping is usually regarded as an inefficient and expensive method of secondary blasting. It is well known that black blasting powder discharged on a rock surface has little or no effect on the rock, whereas dynamite discharged in the same manner may shatter the rock to a considerable extent. For this reason it is a common belief among quarrymen that the explosive force of black blasting powder is directed upward, whereas that of dynamite is directed downward. It should be clearly understood, however, that all explosives exert equal force in all directions. The different results obtained from black blasting powder and dynamite are due to the different rates of combustion of the two explosives. Gases are evolved at a comparatively low rate of speed from black blasting powder and, consequently, they can readily push aside the overlying air and escape. When dynamite is discharged, however, the gases are evolved so rapidly that they can not readily escape and, consequently, exert a violent shattering force upon any body with which the dynamite is in contact. As this force is the same in all directions, it may be readily understood that when a stick of dynamite is discharged on the surface of a rock, a large proportion of the energy of the explosion is wasted and does no useful work. On the other hand, if the dynamite is discharged in a drill hole, a much greater proportion of the energy is expended on the rock.

As a result of tests made by the Bureau of Mines,[a] it was found that the block-holing method is at least 10 times as effective in breaking limestone as the adobe method for a given quantity of explosive. Similar conclusions have been reached by many quarrymen as a result of practical experience. As a consequence, the block-holing method is commonly employed for a major part of secondary blasting, the adobe method being used only under special conditions. In some places, however, the latter method is employed extensively and with low efficiency.

METHOD OF BLOCK-HOLE BLASTING.

In drilling for secondary shots, lines of hose are usually employed to conduct compressed air to the hammer drills from the main air lines on the quarry bank. Other methods have also been noted. In one quarry a motor-driven air compressor is mounted in a box car and can thus be readily shifted to the place where drilling is required, making long air lines unnecessary. In another quarry the steam

[a] See Snelling, W. O., and Hall, Clarence, The effect of stemming on the efficiency of explosives: Tech. Paper 17, Bureau of Mines, 1912, p. 15; Hall, Clarence, and Howell, S. P., The selection of explosives used in engineering and mining operations: Bull. 48, Bureau of Mines, 1913, pp. 37–39.

shovel is equipped with a compressor of the Westinghouse air-brake type. Where secondary blasting is carried on within a short distance of the steam shovel, this is a very convenient method.

If many large blocks are visible in the mass of rock thrown down by the primary shot, it is customary in some quarries to drill and blast these blocks ahead of the steam shovel. Where the mass is broken up more thoroughly by the primary shot such preliminary blasting is usually omitted. Even where the primary blasting is highly efficient blocks of large size are frequently encountered in loading with a steam shovel. Some of the blocks may be too large to be lifted with the dipper, or blocks that the steam shovel could handle may be too large for the crusher. In either event, it is customary to push such blocks to one side. As the steam shovel is advanced these large masses are left between the steam-shovel track and the face. They are later drilled and blasted and the resulting fragments are loaded with the next cut.

Where hand loading is employed the larger masses exposed at the surface are first blasted and other large masses are blasted subsequently as they are encountered in loading.

CIRCUMSTANCES THAT JUSTIFY THE MUD-CAPPING METHOD.

Occasionally in steam-shovel loading masses of rock too large to be moved with the dipper may effectively block further progress. For such blocks "mud-capping" is probably preferable to "block holing" to save time. The drilling might require only a few minutes, but during this time the steam-shovel gang and the train crew would be idle. The use of a few more pounds of explosive is cheaper than to keep 10 or 12 men, a locomotive, and a steam shovel idle for even as short a space as 20 minutes or one-half hour.

Another circumstance which may justify mud-capping is the obstruction of tracks with blocks of stone. As a result of a blast or a rock slide, large masses of stone may roll onto the tracks, preventing the passage of cars or locomotives. Stalling of loaded or empty cars may not only hinder the train crew, but also keep the steam-shovel gang or the hand loaders idle for lack of cars. As in the preceding case, time is of more value that a few pounds of dynamite. Therefore, mud-capping, which requires only a few minutes, is more economical than block-holing.

Where the large fragments are thin and slablike in form it may be difficult to obtain drill holes of effective depth, and mud-capping may give better results than the block-holing method.

The amount of blasting under the special conditions that may justify mud-capping, as outlined in the preceding paragraphs, probably represents a very small proportion of the total secondary blasting

required in any quarry. Except when these conditions arise, all secondary blasting should be by block-holing.

EXAMPLES OF EXCESSIVE SECONDARY BLASTING COSTS.

A number of instances have been noted where the cost of secondary blasting was excessive. It may be instructive to quarrymen to review the probable causes of the high costs, in order that inefficient methods may be recognized and avoided.

In one limestone quarry observed, more explosive is used for secondary than for primary blasting. Two causes may be assigned. First, the rock is seamy, and hence many large, solid masses are thrown down. Where such conditions prevail, the quarryman may find some useful suggestions on pages 50–51, where blasting in unsound rock is discussed. In the second place, the high cost is partly due to hand loading, for the rock must be broken into small pieces to be loaded by this method.

In a limestone quarry in the Middle West, excessive secondary blasting is required because the rock is very hard and is thrown down in large solid fragments by the primary shots. Tripod-drill holes are placed 5 feet apart and with a 6-foot burden on 32-foot benches. The explosive employed is 40 per cent "straight" nitroglycerin dynamite. Larger and more widely spaced holes made with a churn drill or wagon drill would probably give better results, especially if "60 per cent" dynamite were employed.

In one southern limestone quarry as much explosive is used for secondary as for primary blasting. Apparently, this is due chiefly to the fact that the rock is loaded by hand and must be broken into pieces much smaller than is necessary for steam-shovel loading.

In several quarries visited by the writer, the high cost of secondary blasting is due chiefly to the employment of the wasteful and inefficient mud-capping method. The introduction of the block-holing method would greatly reduce the amount of explosive required.

EXAMPLES ILLUSTRATING BLASTING EFFICIENCY.

The following selected examples are fairly representative of the degree of efficiency attained in American quarries where rock is obtained for the manufacture of Portland cement. Conditions vary so widely that the results obtained in different places can not be intelligently compared without understanding the controlling conditions. Consequently, the conditions in each quarry are briefly outlined. The results are expressed as the number of tons of rock obtained per pound of explosive, and not in terms of costs. When given in this form, the result is independent of fluctuations in price of explosive.

EXAMPLE 1.

Conditions.—Limestone is thin-bedded, with a few open joints; churn-drill holes are 5 inches in diameter and 90 feet deep, with 17-foot spacing and 26-foot burden, and are arranged in a single row, 12 to 20 holes being fired at one time; continuous charge of "40 per cent" dynamite is used, with three electric detonators in each hole; secondary shots are chiefly by block-holing method; steam-shovel loading.

Result.—Approximately 4 to 5 tons of rock is obtained per pound of explosive, including secondary shots.

EXAMPLE 2.

Conditions.—Limestone dips 35° to 40°; has some open bedding planes and a few joints; quarry face is perpendicular to strike; churn-drill holes are 60 to 80 feet deep, with 16-foot spacing and 20-foot burden, 12 to 20 holes being fired at one time in single row; charge is continuous, "60 per cent" dynamite being used in bottom of hole and "40 per cent" dynamite for main part, and is fired with detonating fuse; rock is loaded by hand.

Result.—Approximately 3½ to 4 tons of rock is obtained per pound of explosive, not including secondary blasting.

EXAMPLE 3.

Conditions.—Cement rock is thin-bedded in upper 20 feet, with solid ledge beneath; churn-drill holes are 145 feet deep, with 18-foot spacing and 20-foot burden, 10 to 22 holes being fired at one time in single row; continuous charge, mainly "40 per cent" dynamite, "30 per cent" dynamite used near top, four-electric detonators used in each hole; steam-shovel loading.

Result.—About 3¾ tons of rock is obtained per pound of explosive, including secondary shots.

EXAMPLE 4.

Conditions.—Limestone lies flat, is thin-bedded near top, but otherwise sound; 20-foot face; 8 to 20 5-inch churn-drill holes arranged in two rows and staggered are fired at one time; continuous charge of "40 per cent" ammonia dynamite, fired with electric detonators; steam-shovel loading.

Result.—Average for several years is 3.9 tons of rock per pound of explosive, including secondary shots.

EXAMPLE 5.

Conditions.—Limestone lies flat, is thin-bedded but fairly sound; 60 to 70 churn-drill holes 19 feet deep are shot at one time, the holes being in two. three, four, or six rows, and staggered; "blanket" blasting; continuous charge of "60 per cent" nitrostarch blasting powder, fired with two No. 8 detonators in each hole; steam-shovel loading.

Result.—About 4 tons of rock is obtain per pound of explosive, including secondary shots.

EXAMPLE 6.

Conditions.—Limestone dips at low angle, is thin-bedded, with few open and many closed seams; 25-foot face; about 20 5½-inch holes, with 10-foot spacing, arranged in two rows and staggered are fired at one time, supplemented by tripod drill holes in hard bed; broken charge of 40 per cent "straight" nitro-

glycerin dynamite is used, with intermediate stemming in soft bed, two electric detonators are placed in bottom charge, and one in top charge; steam-shovel loading.

Result.—About 3 tons of rock is obtained per pound of explosive, including secondary blasting.

EXAMPLE 7.

Conditions.—Limestone lies flat, is thin-bedded near surface, thick-bedded below; 30 to 35 foot face; 20 to 30 churn-drill holes, with 12-foot spacing and 12-foot burden, arranged in two rows and staggered are fired at one time; continuous charge of "60 per cent" gelatin dynamite, with electric detonators, is used; secondary blasting is mostly by "mud-capping" method.

Result.—About 2.7 tons of rock is obtained per pound of explosive, including secondary blasting.

EXAMPLE 8.

Conditions.—Limestone lies flat, is thick-bedded, with few seams; 22-foot face; an overburden of clay 2 to 4 feet thick is shot down with rock and included in tonnage; about 50 4-inch churn-drill holes, with 7 to 10 foot spacing and 12-foot burden, arranged in two rows and staggered are fired at one time with electric detonators, 60 per cent "straight" nitroglycerin dynamite used; steam-shovel loading.

Result.—About 2½ tons of rock is obtained per pound of explosive, primary blasting only.

EXAMPLE 9.

Conditions.—Limestone lies flat, is thick-bedded; 30 to 50 foot face; 20 to 30 churn-drill holes with 18-foot spacing and 18-foot burden are shot at one time in one row, with occasional holes closer to face and staggered; 60 per cent "straight" nitroglycerin dynamite and electric detonators are used for primary shots; secondary blasting is nearly all by block-holing method, using hammer drill and "40 per cent" dynamite; steam-shovel loading.

Result.—About 2.6 tons of rock is obtained per pound of explosive, including secondary blasting.

EXAMPLE 10.

Conditions.—Limestone lies nearly flat, is thick-bedded, has some open bedding seams and some open joints; 70 to 80 foot face; 20 to 40 churn-drill holes arranged in two rows and staggered are fired at one time; first row has 18-foot spacing and 15-foot burden, second row has 18-foot spacing and 14-foot burden; for primary shots broken charge of 60 per cent "straight" nitroglycerin dynamite with intermediate stemming, fired with detonating fuse; secondary shots are mostly by block-holing method, "40 per cent" dynamite being used; steam-shovel loading.

Result.—About 4 tons of rock is obtained per pound of explosive used for primary blasting only, and 3.6 tons per pound of explosive including that used for secondary blasting.

EXAMPLE 11.

Conditions.—Limestone beds dip about 30°, have a few open seams and clay pockets; 80-foot face; 18 to 40 churn-drill holes are drilled in single row, with 20-foot spacing and 20-foot burden, holes extend 3 to 5 feet below quarry floor; a little "80 per cent" dynamite is used in bottoms of holes, main part of charge is "60 per cent" dynamite and "40 per cent" dynamite, electric detonators are used; rock is loaded by hand.

Result.—About 5 tons of rock is obtained per pound of explosive, primary shots only, and 2½ tons, including secondary blasting.

EXAMPLE 12.

Conditions.—Limestone beds lie nearly flat, upper 20 to 30 feet are jointed, but rock beneath is sound; 40 to 60 churn-drill holes with an average depth of 95 feet, arranged in one row, or two rows staggered, spacing 16 to 22 feet, burden 36 feet, are fired at one time; charge is mostly "60 per cent" low-freezing dynamite, about one-fifth of the charge being of lower grade, and is fired with detonating fuse; steam-shovel loading.

Result.—About 5.8 tons of rock is obtained per pound of explosive, primary blasting only.

EXAMPLE 13.

Conditions.—Limestone is thick bedded and dips 10° to 15°; 24 churn-drill holes 5⅝ inches in diameter and 56 to 121 feet deep, with a spacing of 16 to 17 feet and a burden of 22 feet; also 34 "snake holes" 16 feet long are drilled; churn-drill holes contain broken charges with 12 feet of intermediate stemming, "60 per cent" gelatin dynamite and "30 per cent" dynamite are used in about equal quantities, the charge being fired with detonating fuse.

Result.—About 4.2 tons of rock is obtained per pound of explosive, primary blasting only, rock well broken.

EXAMPLE 14.

Conditions.—Limestone beds lie flat, have many clay seams; 50-foot face; 50 per cent "straight" nitroglycerin used in 5⅝-inch churn-drill holes, burden and spacing variable on account of seams; steam-shovel loading.

Result.—About 3.8 tons of rock is obtained per pound of explosive, including secondary blasting.

EXAMPLES OF QUARRY COSTS.

The following additional examples of actual quarry costs may be of interest:

EXAMPLE 1.

Conditions.—Bench quarry, no stripping, limestone dipping at a high angle, numerous open bedding planes; tripod drills employed, making holes 2½ inches in diameter and 16 feet deep; spacing 6 to 10 feet; "40 per cent" dynamite used; secondary blasting is by both "block-holing" and "mud-capping" methods, more explosive being used in secondary than in primary blasting; rock is hand-loaded; cars are hauled by mules.

Result.—Operating cost of delivering rock to crusher, 18 to 19 cents per ton.

The use of more explosive for secondary than for primary blasting is due to the fact that the useful work done by the explosive in the primary shots is lessened by the presence of open seams, and also to the necessity of breaking the rock into small pieces suitable for hand loading.

EXAMPLE 2.

Conditions.—Crystalline limestone, overburden 6 to 15 feet of clay or shale, many large clay pockets, rock variable in composition, churn-drill holes are 15 to 20 feet deep and 6 inches in diameter, explosive used is "40 per cent" am-

monia dynamite, electric detonators; secondary shots are mostly by block-holing method; rock is hand-loaded on four levels, locomotive transportation on heavy grades.

Result.—Operating cost of delivering rock to crusher, stripping included, is 32 cents per ton.

EXAMPLE 3.[a]

Conditions.—Quarry is situated in the Middle West; rock is an argillaceous limestone, fairly uniform in quality, with occasional seams and pockets; face is 40 to 60 feet high, and is shot down as a single bench; churn drill employed, drilling holes 6 inches in diameter and averaging 55 feet in depth, with a spacing of 10 feet and a burden of 8 feet; about 14 holes are shot at one time, "40 per cent" gelatin dynamite is used, about 150 pounds in each hole; rock is loaded with steam shovels.

Result.—The total operating cost of drilling, including repairs, is 3.15 cents per ton of rock. Assuming a price of 12 cents a pound for the explosive, the cost of blasting per ton of rock is 5.76 cents. The total cost of drilling and blasting is, therefore, 8.91 cents per ton of rock produced.

EXAMPLE 4.

Conditions.—Rock is hard limestone, with 3 to 8 feet of soil overburden; churn drills are used, drilling holes 5⅝ inches diameter and averaging 70 feet deep, main charges are 40 per cent "straight" nitroglycerin dynamite, with a little "60 per cent" dynamite in bottoms of holes; secondary blasting is mostly by "mud-capping" method; rock is loaded with three tractor shovels having dippers of ¾-yard capacity.

Result.—Cost of explosives (dynamite at 20 cents per pound) is 7 cents per ton of rock obtained; total operating cost of delivering rock to crusher is about 30 cents per ton.

EXAMPLE 5.

Conditions.—Limestone, flat-lying, thin-bedded, 25-foot face; blasts in churn-drill holes, in two rows and staggered, 8 feet apart, "40 per cent" ammonia dynamite used, buffer method of blasting is employed; secondary blasting is by block-holing method; rock is loaded by steam shovel; locomotive haulage; average output about 1,500 tons per day.

Result.—Total operating cost of delivery to crusher, stripping not included, is 12 to 13 cents per ton of rock obtained.

BLASTING RECORDS.

IMPORTANCE OF KEEPING RECORDS.

The preceding pages have shown that blasting is a complicated problem involving many interdependent factors. The progressive quarry operator constantly endeavors to reduce the cost of production. As a result, many changes in methods of blasting may be made, and the only way in which the operator can judge the relative efficiency of the various methods tried is to keep a careful cost record of each. Therefore the maintenance of such records is an essential factor in successful blasting.

[a] See McDaniel, A. B., Methods employed in drilling and blasting in the cement quarry: Concrete Cement Age, June, 1914, p. 70.

All records should be reduced to some convenient basis for comparison. Drilling may be expressed as cost per foot of hole drilled, and explosives as the amount of rock moved per pound of explosive used, or as the cost per ton of rock obtained. Usually all costs are reduced to a rock tonnage basis.

EXAMPLES OF BLASTING RECORDS.

The following tables are actual blasting records from a Pennsylvania quarry. Three records are given in order that the relative efficiency of the different methods may be compared.

TABLE 14.—*Examples of blasting records kept at a Pennsylvania quarry.*

BLAST 11.

Labor and material.	Itemized cost.		Total cost.	
	Amount.	Cost in cents per ton.	Amount.	Cost in cents per ton.
Primary blast.				
Labor:				
Drilling 624 feet, at 44 cents ... $274.56				
815 feet, at 42 cents ... 342.30				
	$616.86	1.62		
Cleaning holes	5.85	.02		
Handling powder and loading holes	30.58	.08		
			$653.29	1.72
Supplies:				
Dynamite, low-freezing—				
7,606 pounds, 30 per cent, at 9 cents per pound	684.54	1.80		
200 pounds, 40 per cent, at 11 cents per pound	22.00	.06		
1,350 pounds, 50 per cent, at 11 cents per pound	148.50	.39		
500 pounds, 60 per cent, at 12½ cents per pound	61.25	.16		
			916.29	2.41
Detonators—				
12, 40-foot	1.55			
10, 80-foot	2.38	.01		
11, 100-foot	3.20	.01		
13, 120-foot	4.50	.01		
			11.63	.03
Total, primary blast			1,581.21	4.16
Secondary blasting.				
Labor:				
Air drilling	14.17	.04		
Blasting	84.30	.22		
			98.47	.26
Supplies:				
Dynamite, 700 pounds, 40 per cent	77.00	.20	77.00	.20
Fuses, 1,100 feet	3.52	.01		
Detonators, 125 pieces	.88			
			4.40	.01
Total, secondary blasting			179.87	.47
Grand total			1,761.08	4.63

GENERAL DATA.

Holes drilled, 12.
Drilling hours, 419.
Feet drilled, 1,439.
Feet drilled per hour, 3.43.
Rock blasted, 37,994 gross tons.
Tons per pound of dynamite, primary blasting, 3.93.
Blast made July 1, 1915.
Drilling done by contract, 100 per cent.

TABLE 14.—*Examples of blasting records kept at a Pennsylvania quarry*—Con.

BLAST 12.

Labor and material.	Itemized cost.		Total cost.	
	Amount.	Cost in cents per ton.	Amount.	Cost in cents per ton.
Labor:				
Contract drilling—				
337 feet, at 42 cents	$141.54			
1,608 feet, at 40 cents	643.20			
Drilling by company, 512 feet	25.00			
	$809.74	1.13		
Cleaning holes	3.80	.01		
Handling powder and loading holes	32.33	.04		
			$845.87	1.18
Supplies:				
Nitrostarch blasting powder—				
5,050 pounds, 35 per cent, at 9½ cents per pound	484.80	.68		
15,900 pounds, 40 per cent, at 10 cents per pound	1,590.00	2.21		
1,650 pounds, 50 per cent, at 11 cents per pound	181.50	.25		
			2,256.30	3.14
Detonators—				
20, 50-foot	3.75			
20, 80-foot	5.63	.01		
40, 100-foot	13.80	.02		
Connecting wire and tape	1.15			
			24.33	.03
Total, primary blast			3,126.50	4.35
Secondary blasting.				
Labor:				
Air drilling	31.40	.04		
Blasting	144.50	.20		
			174.90	.24
Supplies:				
Dynamite, 900 pounds, 40 per cent	90.00	.13		
			90.00	.13
Fuses, 1,700 feet	5.44	.01		
Detonators, 200 pieces	1.40			
			6.84	.01
Total, secondary blasting			271,74	.38
Grand total			3,398.24	4.73

GENERAL DATA.

Holes drilled, 20.
Feet drilled by contract, 1,945 feet in 477 hours, 4.08 feet per hour.
Feet drilled by company, 512 feet in 100 hours, 5.12 feet per hour.
Rock blasted, 71,872 gross tons.
Tons per pound of explosive, primary blast, 3.18.
Blast made August 23, 1915.
Drilling done by contract, 79 per cent; by company, 21 per cent.

BLAST 14.

Labor and material.	Itemized cost.		Total cost.	
	Amount.	Cost in cents per ton.	Amount.	Cost in cents per ton.
Primary blast.				
Labor:				
Drilling by company, 1,850 feet	$137.91	0.31		
Cleaning holes	5.40	.01		
Handling powder and loading holes	24.99	.06		
			$168.30	0.38
Supplies:				
Nitrostarch powder—				
4,500 pounds, 50 per cent, at 11 cents per pound	495.00	1.13		
10,000 pounds, 30 per cent, at 9½ cents per pound	920.00	2.10		
			1,415.00	3.23
Cordeau, 5 spools	114.66	.26		
Detonators, 1 piece	.12			
			114.78	.26
Total, primary blast			1,698.08	3.87

TABLE 14.—*Examples of blasting records kept at a Pennsylvania quarry*—Con.

BLAST 14—Continued.

Labor and material.	Itemized cost.		Total cost.	
	Amount.	Cost in cents per ton.	Amount.	Cost in cents per ton.
Secondary blasting.				
Labor:				
Air drilling	$24.00	0.05		
Blasting	127.03	.29		
			$151.03	0.34
Supplies:				
Dynamite, 800 pounds, 40 per cent, at 10.08 cents per pound	80.69	.18		
			80.69	.18
Fuse, 2,300 feet	8.35	.02		
Detonators, 300 pieces	2.54	.01		
			10.89	.03
Total, secondary blast			242.61	.55
Grand total			1,940.69	4.42

GENERAL DATA.

Holes drilled, 15.
Feet drilled, 1,850.
Drilling time, 284 hours.
Feet per hour, 6.51.
Rock blasted, 43,945 gross tons.
Tons per pound of explosive, primary blast, 3.03.
Blast made February 1, 1916.
Drilling done by company, 100 per cent.

A noteworthy feature of the records is the information they convey regarding drilling costs. As noted in the general data for blast 11, all the drilling for that blast was done by contract. The cost of drilling was 1.62 cents per gross ton of rock obtained. For blast 12, when 79 per cent of the drilling was done by contract and 21 per cent by the quarry company, the cost was reduced to 1.13 cents per ton. For blast 14, in which all drilling was done by the company, the cost per ton was only 0.31 cents.

Similar comparisons may be made of the number of feet drilled per hour by each method, the effect on costs of the use of cordeau in blast 14 as compared with the use of electric detonators in blasts 11 and 12, and of other factors.

The chief purpose of such records is to provide the operator with information for intelligently comparing methods and determining which give the best results.

A complete blasting record should include not only a statement of labor and supplies for drilling and loading, but also a description of the blast, indicating number, depth, and arrangement of drill holes, and amount of explosive in each. The following record of a blast in a limestone quarry is given as an example. The form of the report may be modified to suit the quarry conditions. With such records before him, the quarry operator may deduce valuable information regarding the type of explosive and size of charge that does the best work, and the most efficient burden and spacing of drill holes.

TABLE 15.—*Form of daily blasting report adopted by a Michigan limestone quarry company.*

Fired 2.40 p. m., Oct. 13, 1916.

Hole No.	Burden.	Depth.	Dynamite used.	Per cent.	Kind.	Depth of stemming.	Fuses.		Dynamite, top load.	Per cent.	Kind.	Fuses.		Remarks.
							Number.	Length.				Number.	Length.	
	Feet.	*Feet.*	*Pounds.*			*Feet.*			*Pounds.*					
1	22	55	200	40	Ex.	41	1 1	56 50	33	20	FFF.	1	30	Total dynamite used, 2,836 pounds.
2	23	50	200	40	Ex.	38	1 1	56 50	28	20	FFF.	1	30	
3	23	51	200	40	Ex.	38	1 1	56 50	28	20	FFF.	1	30	
4	23	55	200	40	Ex.	40	2—	50	28	20	FFF.	1	30	Total short tons rock moved, 21,182.4.
5	23	55	200	40	Ex.	39	1 1	56 50	32	20	FFF.	1	30	
6	23	55	200	40	Ex.	35	1 1	56 50	32	20	FFF.	1	30	
7	23	55	225	40	Ex.	35	1 1	56 50	28	20	FFF.	1	30	7.4 tons per pound of dynamite.
8	23	55	225	40	Ex.	34	1 1	56 50	33	20	FFF.	1	30	
9	23	55	215	40	Ex.	37	2—	50	28	20	FFF.	1	30	
10	23	55	240	40	Ex.	38	2—	46	33	20	FFF.	1	30	
11	23	54	230	40	Ex.	37	2—	46	33	20	FFF.	1	30	
12	20	54	165	40	Ex.	39	1 1	50 46						
Total or average			2,500				7 12 5—	56 50 46	336			12	30	

.....

Blaster.

SAFETY IN BLASTING.

Manufacturers of explosives are familiar with the dangers involved in handling or storing them, consequently strict observance of the printed directions that accompany boxes of explosives is essential for safety.

Various safety precautions particularly applicable to limestone and cement-rock quarrying have been mentioned incidentally in the preceding pages. All men in charge of the handling or care of explosives at quarries should be familiar with the various publications[a] of the Bureau of Mines relating to safety in blasting. Constant vigilance is the price of safety in blasting, and all quarry operators should insist on shot firers being thoroughly familiar with all important safety rules and adhering strictly to them.

MINING ROCK FOR CEMENT MANUFACTURE.

EXTENT OF MINING.

Limestone and shale are usually worked by open-pit methods. However, in several places conditions are such as to justify underground working, which is herein termed mining to distinguish it from open-pit quarrying. Mining of raw materials for cement manufacture has been observed by the writer near Ironton, Ohio; Superior, Ohio; Independence, Mo.; Hannibal, Mo.; Wampum, Pa.; Manheim, W. Va.; Richard City, Tenn.; and Oglesby, Ill.

ADVANTAGES OF MINING ROCK.

Where the rock sought has little or no overburden open-pit quarrying is unquestionably cheaper than any mining method. As the depth of overburden increases, however, the cost of stripping increases proportionally, until a point is reached where mining would be cheaper if conditions are such that it can be substituted for the open-pit method. The greatest advantage of mining over open-pit quarrying is that the cost of stripping is saved. This saving is offset by the increase in cost of getting out the rock, and the quarryman must carefully balance these two factors in order to determine which method will be the cheaper.

A second advantage of mining is that the workmen are protected from inclement weather, and no time need be lost through adverse weather conditions, as in open-pit quarrying. As rain or snow can not enter the mine, the rock is dry except for underground water and condensation from the mine air.

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[a] See Munroe, C. E., and Hall, Clarence, A primer on explosives for metal mines and quarrymen: Bull. 80, Bureau of Mines, 1915, 125 pp.; Hall, Clarence, and Howell, S. P., Magazines and thaw houses for explosives: Tech. Paper 18, Bureau of Mines, 1912, 34 pp.; Bowles, Oliver, Safety in stone quarrying: Tech. Paper 111, Bureau of Mines, 1915, 48 pp.

Another advantage is that beds deep enough to be worked by mining usually have no pockets or seams of clay, therefore soil does not get mixed with the rock.

DISADVANTAGES OF MINING ROCK.

One disadvantage in mining rock is the high operating cost per ton of rock obtained. Drilling and blasting costs are higher than in open-cut work, and are becoming relatively higher each year, because the cheaper method of deep-hole blasting, now in general use in quarries, can not be applied in mines. In mines the rock must usually be loaded by hand, although in rare instances power shovels may be employed. Thus in most mines loading is more expensive than in open-pit quarries. The high operating cost is further increased by the expense for lighting and ventilation. Also, the men are subject to dangers from falls of rock from the roof. The extent of this danger depends on the nature of the roof and the care that it receives.

CONDITIONS FAVORING ROCK MINING.

In open-pit quarrying only a limited thickness of overburden can be economically removed. Where the overburden is so thick that open-pit methods would be unprofitable, the deposit must be worked by some method of mining or else remain unused. In order that mining may be safe and practicable certain favorable conditions must be present. The most important consideration is the character of the material that will form the roof. In general, the expense of timbering in limestone or shale would be so high that mining could not be conducted profitably. Consequently the roof should be strong enough to require no support other than pillars, and the rock that overlies the serviceable stone should be strong and sound.

Shafts have been observed at some rock mines, but entries are more common, for they permit cheaper removal of the rock. Beds that lie approximately flat are the most desirable for mining, as most rock breaks more smoothly parallel with the bedding, and a smooth, nearly level floor simplifies transportation problems.

METHODS OF ROCK MINING.

DEFINITION OF TERMS EMPLOYED.

In describing mining operations it is necessary to use terms that are not employed in connection with open-pit operations. Some of the more common technical terms used in mining rock for cement manufacture are defined in the following paragraphs.

Mine.—The term mine as used herein is applied to underground workings. It is used in contrast with the "open-pit" quarry.

Roof.—The roof is the undisturbed rock that overlies the workings.

Entry.—An entry is an approximately horizontal passage through which access is gained to the mine and through which the rock is taken.

Shaft.—The term shaft is employed where the opening to the mine is vertical or steeply inclined.

Drift.—The term "drift" as used herein refers to a chamber or room.

Breast.—The breast is the working face at the end of a drift.

Bench.—The breast is usually worked in successive steps, each of which is termed a bench.

Pioneer bench.—The pioneer bench is the first bench that is blasted. It is usually the bench nearest the roof of the drift.

Crosscut.—A crosscut is a passage connecting contiguous drifts.

MINING SYSTEMS.

Limestone and shale mines are usually worked on a plan somewhat resembling the "room-and-pillar" system, and consist of a main

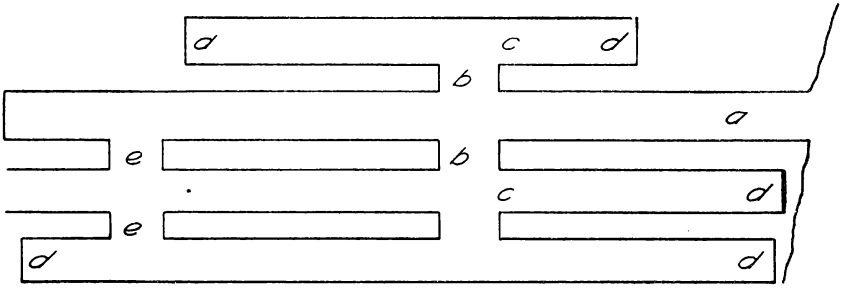

FIGURE 18.—Plan of workings in West Virginia limestone mine: *a*, Main entry; *b*, drift neck; *c*, drift; *d*, breast; *e*, crosscuts.

entry and a series of parallel drifts, or chambers, connected by crosscuts. Supplementary air passages may be necessary to insure adequate ventilation.

In some mines crosscuts are driven at such intervals that pillars about 20 feet square are left to support the roof. The drifts vary from 30 to 60 feet in width, depending on the strength of the roof. Thirty-five feet is the maximum height of drifts observed by the writer.

In a West Virginia mine the workings at the time of the writer's visit consisted of 10 parallel drifts, each 45 feet wide and 26 feet high, the pillars being about 30 feet thick. The rock mined is overlain with sound limestone 20 feet thick, which makes a secure roof.

In opening a new drift, a neck is turned off the entry and the drift is worked both to right and left, parallel with the entry, as shown in figure 18. This method provides for a large number of working places. At the time of visiting this quarry, mining was carried on at

16 breasts. Crosscuts for access from one drift to another and for ventilation are driven every 100 to 150 feet. The position of the crosscuts is governed to some extent by the position of open vertical cross joints; for, wherever possible, a joint plane is utilized to form one wall of the crosscut, as such a joint facilitates blasting.

At this mine the beds dip 17 feet in each 100 feet. The direction of the drifts with reference to the dip is important. The relative direction of dip, strike, and drift are illustrated in figure 19. The drift, although nearly parallel with the strike, runs up the dip at a low angle from the entry. This arrangement has two distinct advantages—gravity drainage and easy haulage for loaded cars. The lateral dip in the drifts has certain disadvantages, as the work of drilling holes and loading the rock is somewhat more difficult than on a level floor. Moreover, all tracks must be graded to a level bed, and this requires considerable labor. A vertical cross section of the drift workings is shown in figure 20.

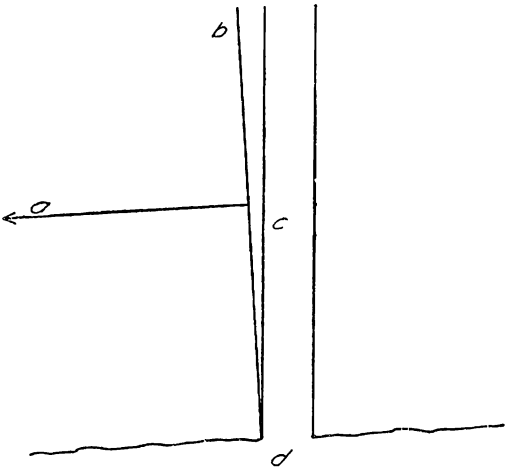

FIGURE 19.—Direction of drift with reference to dip and strike, in a West Virginia limestone mine: *a*, Dip; *b*, strike; *c*, drift; *d*, entry.

In an Illinois limestone mine the pillars at one side of the main entry are opposite the rooms, or chambers, at the other side. It is claimed that when the pillars are staggered in this way the roof holds much better.

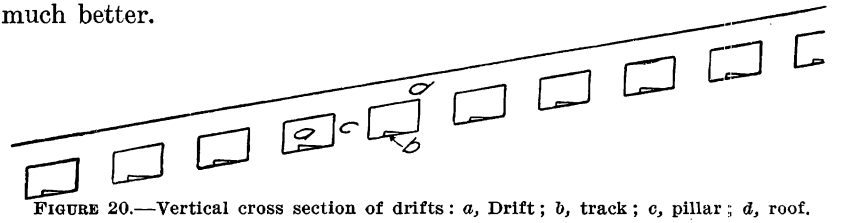

FIGURE 20.—Vertical cross section of drifts: *a*, Drift; *b*, track; *c*, pillar; *d*, roof.

The dimensions of the main entry, crosscuts, air course, and chambers in a Tennessee shale mine are given in a previous paragraph. On account of the weakness of the shale the pillars between the chambers are 50 to 75 feet wide.

SHAFT MINES.

Limestone mines with shafts are rare. The only one of this type observed by the writer is in southern Ohio, where a bed of very pure

limestone is worked through a shaft over 500 feet deep. The rock is mined by the room-and-pillar system and is hoisted to the surface. A current of air downcast by fans through a second shaft provides the necessary ventilation.

DRIFT MINES.

The more common method of opening a limestone mine is by horizontal entry. Some mines have a single entry from which various drifts or chambers are turned off; in others access may be had through a series of entries. Transportation of rock is easier through a level or slightly sloping drift than through a shaft. Also, where the underground workings are reached by approximately level entries, drainage is usually simplified and is commonly by gravity.

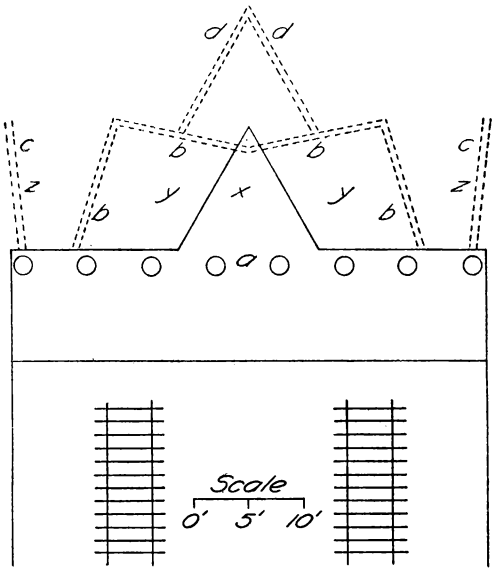

FIGURE 21.—Method of working breast in a West Virginia limestone mine: *x*, V or core; *y*, shoulders; *z*, sides; *a*, blast holes for 12-foot bench; *b*, blast holes for shoulders; *c*, blast holes for removing sides; *d*, blast holes for removing core.

METHODS OF DRIVING DRIFTS.

The usual method of working the breast in drifts is to blast out a pioneer bench, the top of which is at or near the roof level, and then shoot down the rest of the breast in one or more following benches. Details of methods of driving drifts observed in various places are given in the following paragraphs:

In a limestone mine near Manheim, W. Va., the drifts are 45 feet wide and 26 feet high. The pioneer bench is 14 feet high and extends the full width of the drift; the underlying rock is removed in one 12-foot bench. The pioneer bench is blasted as shown in figure 21. A V-shaped mass, or "core" as it is locally termed, is first blasted out as shown at *x;* the shoulders *y* are next shot out, and then the sides *z*. The 12-foot bench is blasted with a single row of vertical holes, *a*, spaced 6 feet apart and with 10-foot burden. A bedding seam at the floor makes it possible to blast this bench without the aid of "snake-hole" shots. The order of blasts is as follows: The vertical bench

holes *a* and the two sets of horizontal holes *b* on the shoulders are fired simultaneously; later the side holes *c* and the core holes *d* are fired simultaneously. There is an open bedding seam at the roof, and another 8 to 11 inches below the roof level. Separation takes place more easily along the latter than along the former. Also, this second seam acts as a cushion and prevents shattering of the roof by the shots. As a result the pioneer blasts breaks the bed at this seam, leaving the band of rock 8 to 11 inches thick attached to the roof. However, to leave this thin band up would be unsafe, therefore it is drilled with a stoping machine (vertical drill) and shot down with very light charges. The roof is then very safe, as it consists of a bed of sound limestone 20 feet thick, having very widely spaced vertical seams.

Formerly the bench at the quarry floor was made the pioneer bench, and the upper part was shot down later. It is said that the method gave good results in blasting, but involved dangers from roof falls; also, the expense for cleaning the roof was high.

In one eastern Missouri quarry, mining was practiced a number of years ago but was later abandoned. The drifts were 45 feet wide and 30 feet high. The pioneer bench was 8 to 10 feet in height, and the underlying rock was removed as a single bench by a blast in vertical drill holes. The shale floor provided a smooth parting and made removal of the entire bench possible without the aid of "snake holes." The pioneer bench and the second bench were blasted simultaneously, the former being kept one step ahead of the latter.

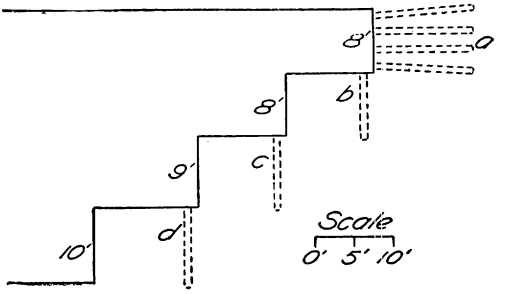

FIGURE 22.—Method of blasting face in drift in a limestone mine in western Missouri. Vertical section lengthwise of drift is shown. *a*, Blast holes for pioneer bench; *b*, *c*, and *d*, holes for second, third, and fourth benches.

In a western Missouri mine, the drifts are 45 to 60 feet wide and 35 feet high and are worked in four benches. The pioneer bench is 8 feet high and is advanced horizontally 10 to 12 feet with each successive blast. The other three benches are 8, 9, and 10 feet, respectively, in height, as shown in figure 22. Blasts *a* and *b* are fired at one time, and blasts *c* and *d* at one time. The drill holes for the pioneer bench are arranged in three groups as designated by *a*, *b*, and *c*, figure 23. The charges consist of "40 per cent" dynamite and are fired by means of fuses attached to detonators. The fuses for the holes *a* (fig. 23) are shorter than those for the holes *b*, and the

latter are shorter than those for the holes *c*. As a result, the blasts in holes *a* are fired first and force out the central V-shaped mass; following this the blasts in holes *b* remove the shoulders, and finally, the blasts in holes *c* throw down the side masses. Each succeeding bench is shattered by a blast in a single row of vertical holes, as shown at *b*, *c*, and *d*, figure 22.

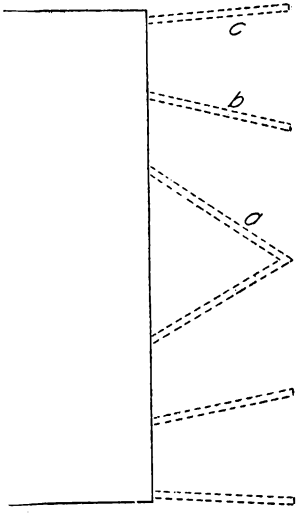

FIGURE 23.—Plan arrangement of blast holes in pioneer bench: *a*, *b*, and *c* are rows of drill holes.

The holes for the blasts *a* (fig. 23), which force out the V-shaped central mass, are known as the "cut holes." To be effective, blasts in the cut holes should be as nearly simultaneous as possible. When fuse is used each pair of holes should so nearly meet at the bottom that the first charge to detonate will detonate the charge in the adjacent hole and thus insure almost simultaneous action. Shots when fired with electric detonators are more nearly simultaneous and more effective than with fuse.

The blasts on the pioneer bench leave a mass of rock about 1 foot thick at the roof. Originally this band was left up, but later was found to be unsafe because a thin layer of shale separated it from the next overlying bed. Of late years it has all been removed and the roof is now in very safe condition.

A unique method of drift mining is practiced in a limestone mine in southern Ohio. The bed of serviceable stone, which is only 8 feet thick, is overlain with a thick bed of sandstone, and underlain with 8 inches of soft blue shale, as is shown in figure 24. The limestone is undercut in the blue shale with a coal-cutting machine, which cuts the shale rapidly, and is then blasted. The sandstone makes a strong roof.

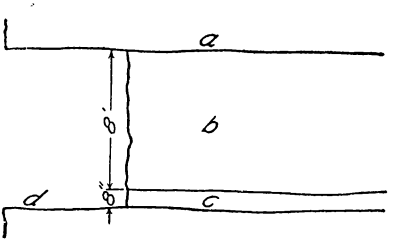

FIGURE 24.—Beds in limestone mine in southern Ohio: *a*, Sandstone roof; *b*, 8-foot limestone bed; *c*, soft blue shale, 8 inches; *d*, floor.

In another limestone mine in southern Ohio the drifts are 15 feet wide and about 22 feet high. The rock is mined in three benches, the pioneer bench being 10 feet high, the middle bench 9 feet, and the lower one 3 feet. The arrangement of drill holes is shown in figure 25. For the pioneer bench there are three series of holes, *a*, *b*, and *c*. The holes *a* are $7\frac{1}{2}$, *b* $6\frac{1}{2}$, and *c* $5\frac{1}{2}$ feet deep. The 9-foot bench is

blasted with the four vertical drill holes *d*, and the 3-foot lower bench with the four "drift" or "snake" holes *e*. The five sets of shots are fired in consecutive order at one operation by means of electric detonators.

In blasting the pioneer bench it is obvious that success can be attained only when the three series of shots, *a*, *b*, and *c*, follow each other in order, for the shots *a* must remove the central V-shaped mass before the shots *b* can throw the shoulders in, and the shoulders must be removed before the shots *c* can effectively throw down the sides. Consecutive shots are fired in some quarries by means of fuses of different lengths. However, in this quarry delay-action electric detonators are used. In a detonator of this type a short piece of fuse is ignited by the electric current, and the cap is fired

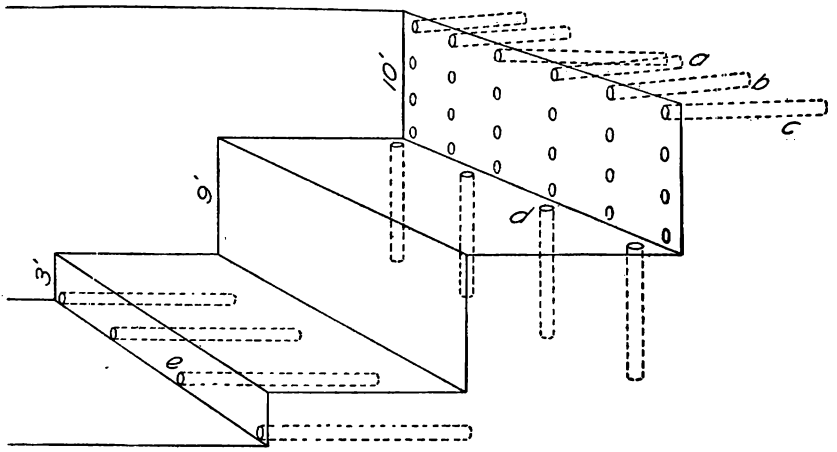

FIGURE 25.—Method of blasting face of drift in limestone mine in southern Ohio: *a*, *b*, and *c*, three sets of blast holes for pioneer bench; *d*, holes for second bench; *e*, holes for third bench.

by the fuse. The period of delay is governed by the length of the fuse. The charges *a*, *d*, and *e* are fired by instantaneous detonators, charges *b* by "first delay," and charges *c* by "second delay" detonators.

In an Illinois limestone mine the drifts are 21 feet high and 20 to 40 feet wide, depending on the strength of the roof. The limestone is removed in three benches of 7 feet each. Contrary to conditions in most mines, the middle bench is chosen for the pioneer bench. At the bottom of this bench is a clay seam which makes a plane of easy separation from the lower bench. A side view of the drift is shown in figure 26. The pioneer bench *a* is removed by three rounds of shots in horizontal holes, throwing out first a central core, then the shoulders, and finally the sides, as in the methods described in

preceding paragraphs. The upper and lower benches are then shot down with blasts in the vertical drill holes *b* and *c*.

The rock is loaded with electric shovels and compressed-air shovels, and is conveyed to the crusher with electric trolleys.

In a shale mine in Tennessee, operated in connection with an open-pit limestone quarry, the main entry, the air course, the drift necks, and the crosscuts are 8 by 8 feet. The drifts, or chambers, from which most of the production is obtained are 20 feet wide and 16 feet high. The rock in the chambers is shot down with three rows of holes, the bottom row having eight holes and the middle and upper rows three holes each. For the 8 by 8 foot work the lower row consists of four holes, and the upper rows of two holes each. The shattered rock at the walls is cleaned down with picks. Timbering is necessary, as the shale roof is weak and insecure.

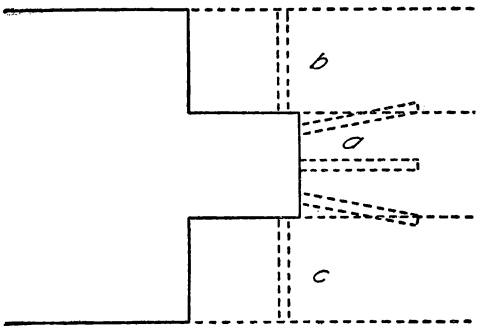

FIGURE 26.—Method of blasting face of drift in Illinois limestone mine: *a*, Pioneer bench, *b*, upper bench; *c*, lower bench; dotted lines represent drill holes.

STOPING FOR LIMESTONE.

What is known as a "shrinkage stoping" method was at one time employed in an eastern Missouri limestone mine. The rock was blasted down by overhand stoping, the working space being kept open by drawing off part of the broken rock from an opening beneath. The serviceable limestone in this mine is 35 feet thick, and is underlain with shale. A preliminary drift, or "heading," was run in the shale; both walls and roof were timbered; at suitable intervals raises or chutes were started in the roof of the heading and these enlarged into stopes, the walls of the stope diverging upward as shown in figure 27. Each stope was about 50 feet wide at the top and 150 feet long. The stopes were separated by "stop pillars," or masses of solid rock, to prevent loose rock from falling from one

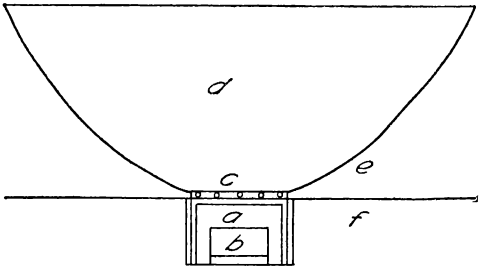

FIGURE 27.—Method of "shrinkage stoping" formerly employed in a limestone mine in eastern Missouri: *a*, Heading; *b*, mine car; *c*, chute; *d*, stope; *e*, limestone; *f*, shale.

stope into another. The broken rock was drawn off, as desired, into cars through chutes in the timbered roof of the heading. The original intent was that branch headings should be made and a number of stopes worked simultaneously; but before this stage was reached the method was abandoned. The plan was, however, scarcely carried far enough for a fair test. The method probably would be more successful when employed in a limestone bed more than 35 feet thick, as a larger tonnage of rock would be obtained for a given expense of driving and timbering a heading.

GLORY-HOLE METHOD OF QUARRYING.

The so-called "glory-hole" method has been employed to some extent in mountainous regions, particularly in West Virginia and Washington. A tunnel is driven into the ledge near its base at a point conveniently situated for transporting the rock to mill or railway. At the inner end of the tunnel a large chamber is blasted and is used as a storage bin for the rock to be removed from above. A slightly inclined shaft is then driven up to the surface. The rock is quarried around this opening in a circle of ever-increasing size, the hole assuming the shape of a funnel. At the bottom of the shaft the flow of rock into the chamber is regulated by segmental gates. The method resembles somewhat the stoping method described in the preceding paragraphs, except that the workings are open to the surface. In one quarry the lower opening is directly adjacent to and above the crusher house and the quarried stone drops through a slightly inclined chute into hoppers, which feed the crusher directly. The glory-hole method of quarrying was first developed in Germany. Its chief advantage is that gravity does the work of a steam shovel.

CARE OF THE MINE ROOF.

The amount of attention that should be given to a mine roof depends greatly on the nature of the rock that constitutes it. A thin bed of rock with a distinct plane of separation between it and the overlying bed makes an extremely dangerous roof. In one Missouri mine a man was killed a few years ago by a fall of rock from such a roof. In a limestone mine with a sandstone roof the presence of masses of limestone clinging to the roof is dangerous, as there is usually a plane of easy separation between the two types of rock. Bedding planes or joints in close proximity tend to weaken the roof, and careful attention should be given to discover signs of weakness, and all loose scale should be promptly removed.

Some companies have a systematic examination of the roof made at specified times. In a West Virginia mine a derrick mounted on a truck is employed for this purpose. The derrick is of light, though

strong, construction and is so designed that it may be lowered to pass under electric wires. Working from positions at the top of this derrick one or two men sound the roof and bar down all loose scale.

In one Tennessee mine the shale roof requires timbering to render it safe.

Where the mine entrance is in a steep hillside material dislodged from above may be a source of danger at the entrance. A noteworthy safety device observed by the writer is a timber roof extending from the face of the hill to a point 15 or 20 feet outside the main entry. Its purpose is to prevent falling débris from injuring workmen as they enter or leave the mine.

ROCK LOADING IN MINES.

Power shovels operated by compressed air are in use in mines near Independence, Mo., Oglesby, Ill., and Lowmoor, Va. In the former locality two shovels of the tractor type are employed to load rock into side-dump cars of 4-ton capacity.

In most limestone mines the breast is too small and work conducted in too many places to make power-shovel loading practicable. Hand loading is, therefore, the common method in such mines. Where the drift is wide, as at Manheim, W. Va., cars are placed on two tracks, one near each side of the drift. With every convenience provided, hand loading is usually less efficient than power-shovel work, and this factor has a distinct bearing on the high cost of rock-mining operations.

To insure the greatest degree of safety, benches should not be cleaned at the same time as loading. When for any reason hand loading is temporarily suspended this interval should be employed for bench cleaning.

ROCK TRANSPORTATION IN MINES.

In the shaft mine in Ohio referred to previously, cars of 2½-ton capacity are hauled to the main hoist with mules. The cars are then run into the cage and hoisted to the surface. Mule haulage is also employed in mines having horizontal entries. In one mine substitution of storage-battery locomotives for mules is contemplated.

An electric motor is employed for haulage in a southern Ohio mine. As the drifts are in places less than 6 feet in height, an objection to the use of a trolley is the danger of contact with trolley wires where there is not sufficient head room.

In one Missouri quarry, the loaded cars are hauled by horses to the main haulage way and brought to the mine entrance by locomotive. At the present stage of mining, the drifts are short, and

surface openings at several points provide adequate ventilation, even where steam locomotives are employed.

In a West Virginia mine gasoline locomotives are used. Three locomotives are employed, two being in active service and one held in reserve. The chief objection to their use is that they may be easily deranged and thus render unreliable service. Those who operate or contemplate operating gasoline locomotives in a mine should carefully consult the various published articles on this subject.[a] Efficiency and safety in the use of gasoline locomotives depend on the proper choice of a machine and on careful observance of the rules governing its operation.

ROCK TRANSPORTATION FROM MINES TO CEMENT PLANTS.

At shaft mines the loaded cars may be run directly from the hoisting cage to the plant by cable haulage, if the distance is short. Where electric motors, gasoline locomotives, or steam locomotives are employed in the mine drifts, it is customary to haul the cars to the crusher by the same means. An aerial tramway is in operation at one Pennsylvania mine. Where the crusher is situated some distance from the plant, the crushed stone may be brought to the plant with cable cars. In one mine the mine cars are run out on a trestle and are dumped into a storage bin, from which the rock is drawn off into hopper-bottom cars of 10 to 12 ton capacity. The latter are conducted to the plant down a steep incline by cable. In a West Virginia mine the entry is near the top of a high and rugged river bluff, and all rock is transported by cable cars down a steep incline many feet in length.

ROCK LOADING.

METHODS EMPLOYED.

After the rock has been stripped, drilled, and blasted the next step is to load the broken fragments ready for transportation to the crushers. As previously stated, two methods of loading are in common use, hand loading and steam-shovel loading. In about five-eighths of the open-pit limestone and cement-rock quarries visited, steam shovels are employed, and in three-eighths of them the rock is loaded by hand. Of the nine limestone or shale mines visited, power-shovel loading is employed in three, and hand loading in six.

[a] See Hood, O. P., and Kudlich, R. H., Gasoline mine locomotives in relation to safety and health: Bull. 74, Bureau of Mines, 1915, 83 pp.; also Hood, O. P., Gasoline locomotives in relation to the health of miners: Am. Inst. Min. Eng., Bull. 94, October, 1914, pp. 2607–2611, and discussion, Am. Inst. Min. Eng., Bull. 100, April, 1915, pp. 892–894.

HAND LOADING.

In the hand-loading method two men are usually employed at each car to pick up the pieces of rock and throw them into the car. Blasting is so conducted as to reduce the blocks to small size, any further breaking necessary being done by sledging. The fine material is thrown into the cars by means of specially constructed forks.

ADVANTAGES OF HAND LOADING.

When rock is loaded by hand it may be sorted and carefully classified. Where bands of dolomite occur in the rock ledge the fragments of dolomite may be thrown to one side and thus the magnesium content may be kept to a minimum. Where flint bands occur the flinty masses may also be thrown aside. Where the rock contains pockets or seams of clay that can not be easily removed prior to blasting, the clay can be separated from the rock during the loading. Accumulations of earth or inferior rock may be loaded into cars and removed to the dump.

The greatest advantage claimed for the hand-loading method is that it allows great freedom in selecting rock from various parts of the quarry. Usually hand loading is carried on at numerous points simultaneously. If the rock is of variable composition, cars are tagged in such a manner as to indicate the place where they were loaded, and thus the approximate composition of the rock in each car is known. The chemist may then designate the order of dumping the cars and the number of cars of each kind required to give the best mixture.

Hand loading has a decided advantage where a limited amount of capital is available, for the loading equipment is not costly.

DISADVANTAGES OF HAND LOADING.

Breaking rock into small fragments requires an excessive amount of blasting. Thus, the drilling and explosives costs are relatively high where hand loading is employed. Hand sledging, which is employed to supplement blasting, is a slow and costly method of breaking rock. Moreover, hand loading requires many more laborers than are required for steam-shovel work for a given output of rock. This is an important consideration at times when labor is scarce and wages high.

STEAM-SHOVEL LOADING.

Steam shovels have replaced hand loading in a majority of open-pit quarries where rock is obtained for cement manufacture. The shovels are of various types and sizes. Many 60 to 100 ton shovels, designed to operate on tracks and having $1\frac{3}{4}$ to 4 yard dippers, have

been observed by the writer. Others are of the tractor type, that require no tracks. They are smaller than those which operate on tracks, having dippers of $\frac{1}{8}$ to $1\frac{3}{4}$ yards capacity. Tractors give the best service where the quarry floor is firm and smooth. If the quarry floor is parallel with the bedding and coincident with an open bedding plane, a smooth floor is easily maintained and is therefore favorable for tractor-shovel operation or for laying tracks. If the floor is shale or other soft rock, the tractor wheels may require plank supports. For any type of shovel a smooth floor is desirable.

A steam-shovel gang consists of one runner, one craneman, one fireman, and three to six pit men. Some of the smaller tractor shovels require no craneman. The first cost of a steam shovel is high, but with proper care the cost of maintenance is not excessive.

The amount of material handled by a steam shovel per day varies widely, as it depends on efficiency of operation, the conditions of the rock mass, time required to spot cars, and other factors. Tractor shovels are capable of handling 400 to 700 tons of rock per day, while the larger types, with 4-yard dippers, may attain a maximum of 1,400 to 1,600 tons. A fair average for one of the larger types working in limestone is 600 to 1,000 tons per day.

ADVANTAGES OF STEAM-SHOVEL LOADING.

A steam shovel can load blocks of considerable size, hence blasting costs are relatively lower than for hand loading. Moreover, the slow and laborious hand sledging is unnecessary.

A second advantage is the rapidity with which material may be handled. One quarryman reports that a small steam shovel in his quarry loaded 57 6-yard cars of limestone in five hours. Coupled with rapidity of loading is the fact that only a small gang of men is required for a steam shovel, whereas a large gang of hand loaders would be required to handle the same amount of material. This advantage is most keenly felt when wages are high and labor is scarce.

DISADVANTAGES OF STEAM-SHOVEL LOADING.

When rock is loaded by hand it is usually obtained from many parts of the face simultaneously. The steam shovel, however, loads for a considerable time from one position. This is the chief objection to the use of steam shovels in quarries of variable rock, as it does not permit sufficient mixing of the rock from the various parts of the face.

A second disadvantage is the inability of the steam shovel to sort the material that it loads. If clay is thrown down with the rock it must be loaded with the rock. If bands of dolomite or flint occur in the ledge, in general such material must be loaded with the good

rock, although with careful manipulation it may be possible to set the larger undesirable masses to one side.

When the sum available for investment in equipment is small, the high first cost of a steam shovel may be an objection to its introduction.

DESIRABILITY OF CHANGING FROM HAND METHODS TO STEAM-SHOVEL LOADING.

At the time of writing, labor is one of the most difficult problems with which quarry operators are confronted, especially in Northern States. Consequently, any equipment designed to give efficient service with a minimum of man power appeals directly to the quarry operator's needs. Therefore, a comparison of the work accomplished per man by each method is desirable.

Figures obtained from 14 quarries where steam shovels are used indicate an average daily tonnage of about 112 tons of rock loaded per man. The men represented in this determination are pit men and steam-shovel men only. For 11 quarries where hand loading is employed, the average daily tonnage per man, loaders only, is 16. This indicates that about seven times as much rock can be loaded per man with steam shovels as can be done by hand. The cost of labor for hand loading by contract is 10 to 12 cents per ton, whereas the operating cost of steam-shovel loading, not including repairs or upkeep, is 4 to 6 cents per ton.

Neither the cost of loading nor the number of tons loaded per man daily are alone a measure of the relative efficiency of the two methods, for other factors enter into the problem. The blasting costs are higher for hand loading than for steam-shovel work, because the rock must be broken into smaller pieces. On the other hand, the cost of crushing hand-loaded material will obviously be lower than that of crushing the massive fragments loaded with steam shovels. Hence, we may assume that these two factors, namely, the higher cost of explosives for hand loading and the higher cost of crushing for steam-shovel loading, will balance each other, although the former figure is probably the higher of the two.

There must also be added to the cost of operating a steam-shovel the interest on the first cost of the equipment, a depreciation charge, and the expense of maintenance. It is evident that, even with the addition of these items, steam-shovel loading under normal conditions is cheaper than hand loading.

SOLUTION OF DIFFICULTIES IN CHANGING FROM HAND LOADING TO STEAM-SHOVEL LOADING.

It has been shown in the preceding paragraph that, in general, the use of steam shovels has decided advantages over the hand-load-

ing method. Hence, any possible solution of the chief difficulties in introducing steam shovels where hand loading is now practiced is to be desired.

Sufficient capital to purchase a steam shovel is, of course, a prerequisite condition.

In some quarries work is conducted on such a small scale that a steam-shovel equipment would be deemed unprofitable. The small tractor shovel is well adapted for small quarries. In general, quarrying can not be done as cheaply on a small scale as on a large scale, and it is doubtful whether cement could be manufactured profitably on a scale so small that the necessary rock could not be handled to advantage with a steam shovel, provided the bulk of the material is obtained from a single quarry.

In several quarries the overburden is shot down and loaded with the rock. Where the overburden varies greatly in thickness from point to point, a proper mix may be obtained by loading at various points simultaneously. In small quarries requiring only one steam shovel this is difficult if not impossible to do, consequently an excess of soil may be loaded at one time and a deficiency at another time. One quarry in Kansas overcame this difficulty by stripping all the soil off with one steam shovel, loading practically pure rock with another shovel, and mixing the clay and the rock proportionally after crushing. Another solution of the difficulty is to have a large storage bin in which the raw materials may be distributed uniformly and thus equalize any irregularities in loading.

Similarly, the presence of clay-filled seams or pockets is made an occasion for resort to hand loading, because of the inability of the steam shovel to sort the material properly. Hand loading of such material is a slow and costly process, and the quarryman is justified in adopting any reasonable expedient to make the steam-shovel method practical. The use of a clam-shell bucket on a derrick arm has been suggested as a means of removing clay from large pockets. If the material can be used as a cement constituent, the use of a large storage bin, as previously mentioned, will tend to equalize the proportion of rock and clay. If the content of the seams and pockets is undesirable, it may be possible to remove such material by installing washing equipment in the screens. The removal of sticky clay by washing would be difficult or impossible, but usually such clay may be used as one of the cement constituents and need not be removed.

The chief objection to the use of a steam shovel in quarries having rock of variable quality is the difficulty of obtaining a uniform and properly proportioned mixture. As a rule, the rock is fairly constant in a given bed, the greater variations occurring in passing from one bed to another. Many quarrymen deem it necessary to blast and

load the rock from the various beds separately and later mix it in definite proportions. However, the writer believes that in either flat lying or inclined beds an approximately uniform mixture may be obtained by developing a high face and shooting it down as a single bench.

In some deposits when a high face is shot down as a single bench the rock along the face varies considerably in chemical composition. In one Pennsylvania quarry of this type the steam shovel, by means of a special arrangement of tracks, loads at various points successively and thus supplies any desired rock mixture. Ordinarily a steam shovel works in cuts paralleling the quarry face, but in this quarry the main track parallels the face, and a series of branch lines meet the face perpendicularly, as shown in figure 28. When a sufficient amount of rock has been loaded from one branch line the steam shovel is moved to the main line and shifted to another branch, an operation that requires only a short period of time. The branch lines are so arranged that cars may be loaded on the branch adjacent to the one from which the steam shovel is working.

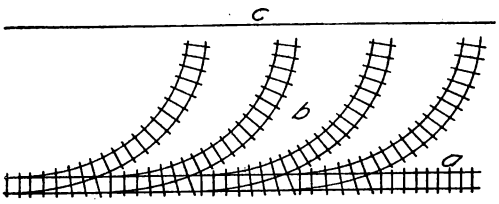

FIGURE 28.—Track arrangement suitable for quarries with high face: *a*, Main line; *b*, branch lines; *c*, quarry face.

In another quarry the tracks are similarly arranged to avoid loss of time in secondary blasting. When large blocks impede steam-shovel work, the shovel is backed to the main line and shifted to another branch, where loading is resumed with the loss of a few minutes only. By this method the shovel is not kept idle while rock masses are being drilled and blasted, but additional service in spotting cars is required. In ordinary steam-shovel practice, a train of cars may be loaded in rapid succession, whereas with the method described above, loaded cars must be switched to the main line and empties returned to the branch singly, a procedure which consumes much time.

As mentioned in a preceding paragraph, the problem of obtaining a uniform mixture is greatly simplified by having ample storage room for raw rock, and by distributing the rock uniformly in the storage bin.

Another difficult in changing from hand loading to steam-shovel loading is the necessity for modifying the car equipment to suit the more strenuous requirements. Hand loading is ordinarily easy on cars and small cars are adequate. When a steam shovel is used larger and stronger cars are necessary. When a steam shovel is introduced it is probable that the purchase of new car equipment

would be more satisfactory than any attempt to enlarge or otherwise remodel old cars.

EFFICIENCY IN STEAM-SHOVEL OPERATION.

In changing from hand loading to steam-shovel loading, the small crusher and the small cars commonly used in conjunction with hand loading may make the purchase of a small steam shovel seem advisable. Also, if the rock varies in composition in different parts of the quarry it may be deemed wise to employ two or more small shovels in preference to one large one. In some instances the use of small shovels may be justified, but the operator should, wherever possible, employ shovels of at least 95-ton burden. Rock loading is rough and heavy work, and repair bills for small shovels may be excessive. The employment of large strong shovels involves little expense for repairs or replacements. Moreover, the use of small shovels increases the costs of secondary blasting, as the rock must be broken smaller than for large shovels.

In order to utilize the loading capacity of a steam shovel to its full extent, certain conditions must be met. The chief conditions on which successful operation depends are discussed in the following paragraphs.

The steam shovel should be kept in good repair and should be operated by a skilled runner. An unskilled runner will delay the entire process of loading and transportation.

Rapidity of loading depends largely on successful blasting. Thorough shattering of the rock is essential. If the shovel is repeatedly delayed while large rock masses are being blasted the rate of loading will be slow.

Each cut made with the shovel should be of the maximum width it is capable of handling properly. With each advance of the shovel much time is consumed in moving and laying tracks; hence it is desirable to load a maximum amount of material from each position. When a narrow cut is made the quantity of rock obtained for each advance is smaller than with a full cut; consequently time is sacrificed. The same argument may be advanced with reference to the thickness of the mass of shattered rock to be loaded. Where the mass is thin a small amount of stone is obtained for each move. Moreover, the dipper is filled more easily where a heavy burden provides effective resistance.

Transportation facilities have a direct bearing on the rate of loading. An insufficient number of cars greatly retards the rate. An instance has been observed where a shovel was idle a large part of the time waiting for empty cars. Similar delays may be caused by unskilled handling of cars. Haulage should, if possible, be so conducted that a train of empties is ready to be placed immediately after the loaded cars are removed.

Undue loss of time may be caused by having to move the shovel when a train is partly loaded. If conditions are such that empty cars are not available immediately after the departure of a loaded train, the quarry operator should aim to so regulate the number of cars in the trains that one or more complete trainloads may be obtained from one position of the shovel. Tracks may then be moved and the shovel advanced during the interval when no empty cars are available.

The rate of loading is influenced to some extent by the type of crusher employed. The crusher should be large enough to take any size of block that the steam shovel can conveniently load. Too small a crusher not only delays the steam shovel by its inability to crush the rock as fast as it is loaded, but places on the shovel the additional work of setting aside for secondary blasting and subsequent loading many blocks that could readily be handled by a larger crusher.

ROCK TRANSPORTATION.

USE OF THE TERM.

The subject of rock transportation involves consideration of the means of removal of the rock from the quarry face to the cement plant. It includes the handling of loaded cars and empties.

QUARRY CARS.

TYPES OF QUARRY CARS.

Quarry cars should be designated to suit the nature of the work required of them. For hand loading, they need not have a capacity of more than 2 or 3 tons. They should be low in order to make loading easy. For steam-shovel work, larger and stronger cars are required. Cars of 5 to 10 ton capacity are commonly used, a maximum of 18 tons having been noted. A steam shovel handles many large and heavy blocks, and the cars must be substantially built to withstand the rough usage they receive in loading and unloading. One company uses metal cars of 5 or 6 ton capacity, mounted on two flexible trucks each having four small wheels.

When steam shovels are introduced to take the place of hand loading, it is usually necessary to change the car equipment, as the cars commonly employed in hand loading are too small to give efficient service with a steam shovel. One company found that cars enlarged to increase their capacity were so weak that repair bills became excessive. It is probably better to purchase new cars of suitable size and strength than to attempt remodeling of old cars. Standard hopper-bottom railway cars are employed in some quarries, though cars of smaller size are usually employed.

Various methods of unloading have been observed. Side-dump cars, the body of the car rotating on a pivot or rocking on a semicircular bar, are the most common. A car with one open side is an unusual type observed in one locality. It is so designed that the load is dumped when the car reaches a portion of the track that has one rail elevated. In several quarries the loaded car is placed in a revolving cage and the rock is dumped by a complete revolution of the cage. Cars that dump at the front by depression of the front of the car or elevation of the rear are also used, commonly in connection with a tipple. At one Illinois quarry, if the rock is so placed in the car that it fails to overbalance, the rear of the tipple platform is raised by means of an air piston. A type of car used in an eastern Pennsylvania quarry has a metal body fixed rigidly to a four-wheel truck. The track just beyond the crusher slants upward at an abrupt angle. When the car reaches this point it is suddenly tilted and the rock falls out at the rear, which is open. The crusher is situated beneath the tracks. The car is simple in construction and easy to keep in repair. One difficulty, however, is that frequently blocks too large to pass between the rails are dumped and may occasion considerable delay.

An adequate supply of cars is important. It is claimed that in many quarries shovels are not actually working 40 per cent of the time, the delay being due to lack of track and car facilities. In one quarry observed, loading was done with a 95-ton steam shovel having a 3½-yard bucket, and only eight 5-ton cars were provided. A locomotive hauled 4 or 5 cars at a time, when it could easily have hauled 8 or 10. So few loaded cars were in reserve that when any delay occurred in the quarry the crusher would be idle almost immediately. On the other hand, if the crusher was for any reason delayed, the lack of empty cars caused almost immediate suspension of loading.

MOTIVE POWER FOR CARS.

A gravity system of haulage, where practicable, is the cheapest that can be employed. If quarry conditions are such that movement of loaded cars is mainly down grade, gravity methods may be used to advantage. Horses or mules are commonly used in quarries where the tracks are level or nearly so, and where loading is by hand. Where steam shovels are used the rock cars are too heavy to be handled with animals, and locomotives are generally employed. Locomotives may be of the small type known as "dinkeys," or of standard size. Side-geared locomotives of the Shea or Heister type are recommended for use on heavy grades. The use of electric storage-battery locomotives is contemplated in at least one rock mine for underground work, but their actual operation has not been observed by the writer. Gasoline locomotives are used in at least two rock mines for underground

transportation. Haulage by electric trolley is found to be satisfactory and economical in some places. The most serious objection to the use of electric trolleys is the necessity for frequent moving of trolley poles.

A central-control electric three-rail system has been lately introduced in a northern Michigan quarry. The track is divided into sections with an independent current for each section, and thus cars on different sections are subject to independent control. The movements of all cars are controlled from a central tower. The system is somewhat complicated, but is so nearly automatic that few men are required to operate it.

In many quarries the grades are so steep that a cable-and-drum system must be used. Power may be supplied by steam or by electricity.

In one New York quarry a cable belt is driven at a uniform speed, and the cars are moved along the track by clamping them to this cable.

INCLINED RAILWAY.

Where quarries are situated at high elevations the rock may be brought down on inclined railways. If a car is attached to each end of the cable, an empty car ascending while the loaded car descends, the loaded car supplies all the power necessary. Where this system is employed a powerful and reliable brake is necessary if accidents are to be avoided.

In most quarries rock is obtained at a level much lower than the crusher platform, and the loaded cars have to be pulled up an inclined track. Under such conditions, some type of power hoist is necessary. In many places a single track is employed, the loaded car being hauled up by a cable, dumped, and returned under cable control. The power required for hauling cars may be greatly reduced by employing a double-track system and allowing the weight of the empty descending car to assist in elevating the loaded car.

OVERHEAD CABLEWAY.

The use of an overhead cableway supported by towers has been noted. The rock is loaded into skips or pans that are hauled on cars to positions beneath the cable. The pans are then hoisted vertically, attached to the cable, and conveyed to the cement plant. There are two objections to this method. First, the maximum load that can be hoisted is limited to about two or three tons, and therefore material can not be handled rapidly; second, the passage of the loaded pans across the quarry at a great height involves grave danger to the men from falling rock. At one quarry only has this method been observed, and its abandonment is contemplated.

AERIAL TRAMWAY.

The aerial tramway is another type of cable transportation, which is more satisfactory than the overhead cableway. The following information concerning an aerial tramway in operation in Alabama may be of interest:

A ¾-inch endless steel cable travels on supports which keep it 15 to 30 feet above the ground. The rock is carried about 8,000 feet, the cable being about 16,000 feet long. It is driven by a 35-horsepower electric motor. The drivewheel, which is about 8 feet in diameter, is provided with automatic clamps which prevent slipping. The buckets have a capacity of about 1,000 pounds each. There are 70 buckets on the cable at one time, 35 full, traveling toward the cement plant, and 35 empty, returning to the quarry. When an empty bucket reaches the quarry it is automatically unclamped from the cable and is suspended by the carrying device from a single-rail circular track, along which it is easily pulled by hand to a position beneath any one of six rock slides from the storage bin, where it is quickly filled. The loaded bucket is then pulled by hand around the track and is automatically clamped to the traveling cable. The cable makes one complete circuit in 36 minutes and delivers approximately 1 ton of rock per minute at the cement plant. The equipment has given satisfactory service for the past six years.

One important advantage of the aerial tramway system, well illustrated in the installation described, is the ease of conveying rock over hilly ground, thus avoiding excessive cost for grading. Where the cable passes over a hill the strain on it may be somewhat increased, but the power consumption is little, if any, larger than where no hill intervenes, for the buckets descending on one side of the hill balance those ascending on the opposite side.

At one Illinois plant, having a capacity of 4,500 barrels of cement a day, all the rock is transported across a river by aerial tramway. The method is satisfactory if the equipment is closely watched and all defects or weaknesses repaired without delay.

The rock for a New York plant is transported by aerial tramway a distance of 1 mile from the top of a high hill. There are 40 buckets on the line, each having a capacity of 1,300 pounds. The maximum capacity of the tramway is about 1,500 tons per day of 24 hours. Coal required at the quarry is carried in the tramway buckets that would otherwise return empty.

At another New York plant the rock is conveyed about 1,500 feet down a steep hill by aerial tramway. The buckets are of 900-pound capacity. The tramway carries, on an average, about 400 tons per day, although its maximum capacity is much greater.

MINE HOIST.

Where rock is obtained from shaft mines an ordinary mine hoist is employed. The loaded cars are usually hauled by horses or mules to the main shaft, placed in a cage, and hoisted to the surface.

QUARRY TRACKAGE.

LEVEL OR MODERATELY INCLINED TRACKS.

Quarry trackage is an important factor in rock transportation. Efficient rock handling depends largely on arrangement of track, elimination of heavy grades, firm roadbed, and uniform gage. As a rule, tracks are standard-gage width, although narrow gage is used in many places. In any event, the gage should be wide enough to insure reasonable stability of the cars in rough places.

An uneven quarry floor may require much grading, and makes tracklaying difficult. The problem of moving and laying track is greatly simplified if the quarry floor is smooth and uniform. In quarrying limestone beds that lie approximately flat, an open bedding plane at the quarry floor is a great advantage, for it provides the smooth floor desired.

The trackage system must be modified to suit conditions. Where steam-shovel loading is employed, the system is usually simple, as loading is conducted at only a limited number of places at one time. The larger the quarry the more complicated will be the track layout, and in quarries where eight or more shovels are used, a block system and tower-signal station may be employed.

Where rock is loaded by hand in large quarries many working places must be provided. A convenient system designed for hand loading in quarries having reasonably high faces with an ample supply of rock is that shown in figure 28 (p. 122). From a main line parallel with the face a series of branch lines extend to the face. One car is placed at the end of each branch, and usually two loaders are employed at each car. Loaded cars are collected by a locomotive from various branches in succession until enough cars to form a train are obtained. Empty cars are distributed to the various branches in a similar way.

For low-faced quarries with a limited supply of rock at each working place, the face is usually very long, and a branch to every working place would require an excessive amount of trackage. Therefore in such quarries the branch tracks usually are parallel with the face, as shown in figure 29. Several cars are placed on each branch and are loaded simultaneously.

In comparing the two methods it may be pointed out that where the branches parallel the face and a number of cars are loaded on each branch, some inconvenience may result from partly loaded cars being nearer to the main line than fully loaded cars. Then all the

cars on the branch may be taken to the main line, where the partly loaded cars are cut out and returned to the branch. Where the branches meet the face perpendicularly, conflict of loaded and empty or partly filled cars is avoided. On the other hand, as only one car is placed on each branch, more switching is required to collect a train of loaded cars or to distribute a train of empties. Where the branch lines parallel the face and three, four, or more cars are on each branch, the collection or distribution of cars is a relatively simple matter.

A trackage system employed in an Alabama quarry consists of branch lines meeting the face perpendicularly and connected with a circular main line provided with a crossover dump. The cars are hauled by mules. Formerly the quarry cars were hauled up the incline to the crusher. By means of the crossover dump they are now dumped into cable cars of much larger capacity. This arrangement has increased greatly the haulage capacity of the cable incline and has materially reduced quarry costs and increased the output of rock.

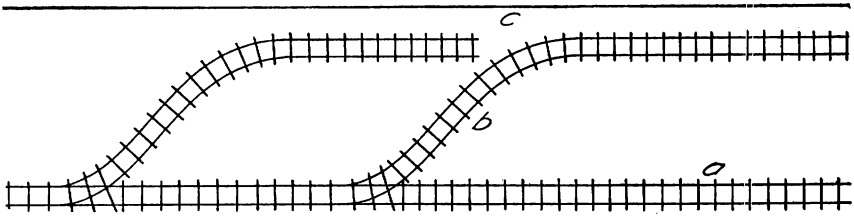

FIGURE 29.—Track arrangement suitable for quarries with low face: *a*, Main line; *b*, branch lines; *c*, quarry face.

Where the hand-loading method is employed in quarries worked on several levels, the trackage is more complicated. An actual example of the complexity in track arrangement that may occur in a quarry worked on four levels is shown in figure 30. The obvious difficulty of transportation under such conditions should encourage the adoption of steam-shovel loading and the working of fewer benches, if circumstances would permit such readjustment.

In many quarries the cars are taken from the quarry by locomotive. Where the quarry has a fixed floor level, this system may be permanent, but where the quarry is gradually deepened the grades may become too steep for locomotives. In some quarries the desire to avoid the expense of introducing a cable incline system has led the operators to retain locomotive haulage on very steep grades. This may make it necessary to purchase additional locomotives, for the trains travel more slowly on heavy grades and are made up of fewer and fewer cars. The installation of a cable system would probably be better than to adhere to locomotive haulage on excessively heavy grades.

Where there is sufficient room for the additional trackage required, a system of back switching may be employed to overcome heavy grades. The system occupies considerable space and involves the first cost and maintenance of relatively long stretches of track, but it permits the handling of a number of cars at a time. In the opinion of some quarrymen, back switching is preferable to cable inclines.

INCLINED TRACKS FOR CABLE CARS.

Where the gradient of the track exceeds that on which locomotives may be operated to advantage, a cable-and-drum system of haulage

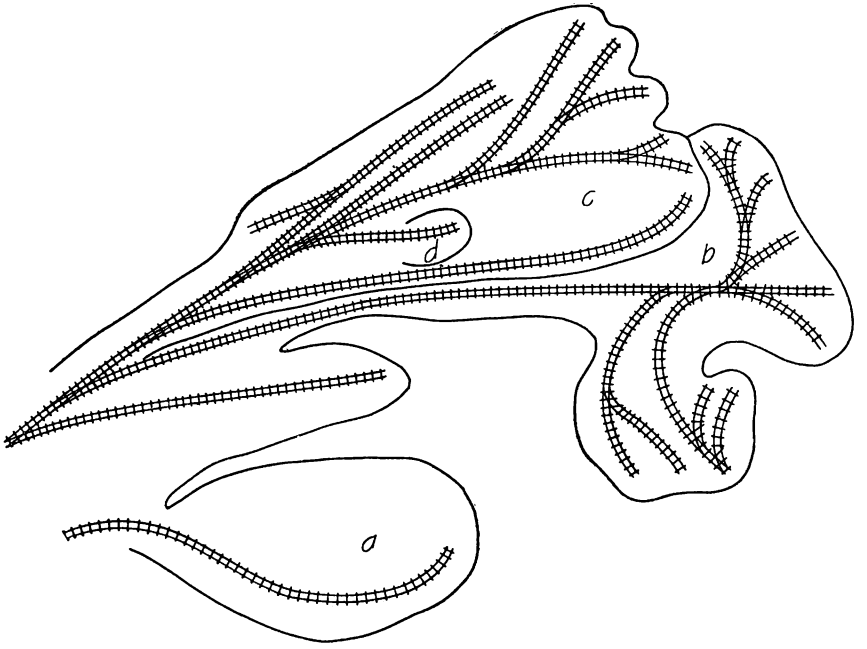

FIGURE 30.—Complex trackage system in a quarry where rock is loaded by hand on four levels: *a*, Upper level; *b*, second level; *c*, third level; *d*, lower level.

is usually employed. Single tracks are used in many quarries. In some quarries cars are operated independently on two or more tracks. In others, a car is attached to each end of the cable, one descending as the other ascends, a double track, being provided, or a three-rail track above the center switch and a single track below.

On a single track where the load is carried down grade some source of power is required to return the empty car. By the three-rail or double-track system, where two cars are operated by the cable, the descending loaded car pulls up the empty car, and no power is required. Similarly, when transportation is up the incline, the descending empty car helps to draw up the loaded car. The only power re-

quired is that needed to raise the actual weight of stone moved, plus the amount consumed by friction, whereas with the single-track system the additional power necessary to move the empty car is required. A considerable saving is, consequently, effected by utilizing the force of gravity, but up to the present time few quarry operators have taken advantage of it.

The "ground hog" or "barney" type of cable haulage is employed in several quarries. A heavy buffer, mounted on four wheels and attached to the cable, operates on a narrow-gage track situated between the rails of the car track. At some distance from the foot of the incline the narrow-gage track runs into a depression below the car-track level. When the cable is out the buffer rests in this depression. The loaded cars pass along the track over this excavation. As the cable winds on the drum the buffer comes up behind the car and pushes it up the incline. Usually one car only is taken up at a time. One advantage of this method is that the inconvenience and danger of attaching the cable to loaded cars and unhooking it from empties is eliminated.

An endless chain system is used as a substitute for a cable in some places. Cars are pulled up the incline by projections arranged at regular intervals on the chain.

TRANSPORTATION FROM POINTS OUTSIDE MAIN QUARRY.

For most cement plants a minor proportion of the raw materials is obtained outside the main quarry. Thus, where limestone and shale are used the plant is usually situated near the limestone quarry, and the shale may be obtained elsewhere. Where cement rock is used, a small amount of high calcium limestone is commonly shipped from outside points.

In some places the shale quarry has its own car tracks and incline, its transportation system being independent of that maintained at the limestone quarry.

Where raw materials are thus obtained from two sources, the tracks and storage bins should be so arranged as to avoid excessive rehandling. A convenient means of handling rock brought from a distance was noted at an Oklahoma plant. The rock is conveyed in hopper-bottom cars and dumped into a trough that opens on a belt conveyor which carries it to storage.

An example of very inefficient handling of material was noted at one Pennsylvania plant. The cement rock, which was quarried close to the plant, required the addition of a small amount of high calcium limestone brought from a distance. The limestone was carried in hopper-bottom cars to a trestle above the quarry pit, which at this point was 40 or 50 feet deep. It was dumped into the pit, loaded into quarry cars by hand, and hauled by mule to the incline up which the

cement-rock cars were taken. The cars loaded with limestone were elevated at such times as they were needed to maintain the desired mixture. It would seem that the maintenance at a high level of a limestone storage bin from which the rock could be drawn by gravity or conveyor would add greatly to the efficiency of this transportation system.

TPYICAL TRANSPORTATION SYSTEMS.

NECESSITY FOR COMBINATION OF TWO OR MORE METHODS.

In order to avoid the extra cost of rehandling cars an effort should be made to simplify the transportation system as much as possible. At most cement plants, however, the rock must be elevated from the quarry to the bank, from ground level to an elevated crusher, or brought from a high level to a lower level. This necessitates the employment of cable inclines for the hills or trestles, and thus involves a combination of two or more methods of transportation.

SINGLE METHODS OF TRANSPORTATION.

In a few localities the simplest system of transportation is possible, namely, the transfer of rock from the quarry face to the crusher by a single process. Thus, at New Castle, Pa.; Hannibal, Mo.; and Superior, Ohio, the crusher is on a hillside. The rock is quarried at higher levels and is brought to the crusher directly by locomotive or electric trolley.

COMMON COMBINATIONS OF METHODS.

The combination of methods most commonly observed is haulage of cars in trains by locomotives to the incline and cable control up or down the incline, as the case may be. In many quarries the cars are hauled to the incline with horses or mules.

In five quarries, having a gentle down grade from the face to the incline, the loaded cars are let down from the face to the foot of the incline by gravity. The empty cars are usually returned by horses or mules. Where the grade is light and the cars are small they may be pushed to the face by the men.

SPECIAL COMBINATIONS OF METHODS.

In a deep Pennsylvania quarry with somewhat limited floor space, loaded cars are hauled from the point of loading to the incline by means of a cable operated on a drum attached to the steam shovel. A cable-and-drum equipment, operated by electricity, hauls them up the incline.

A gravity method, where practicable, is the cheapest possible power that quarrymen can utilize. A description, therefore, of two

systems where gravity is employed effectively may be of value to the industry.

In one Pennsylvania quarry the loaded cars run on a down grade to the cement plant, where they are elevated, dumped, and returned by gravity on a trestle which descends on a gentle grade to a point near the quarry face. Thus gravity transports both loaded and empty cars.

In a quarry in western New Jersey both loaded and empty cars are moved by gravity to the foot of the incline. The trackage system is shown in figure 31. The incline has a double track, the empty car descending as the loaded car ascends. Thus the weight of the empty car assists in elevating the loaded car. The empty cars leave the incline

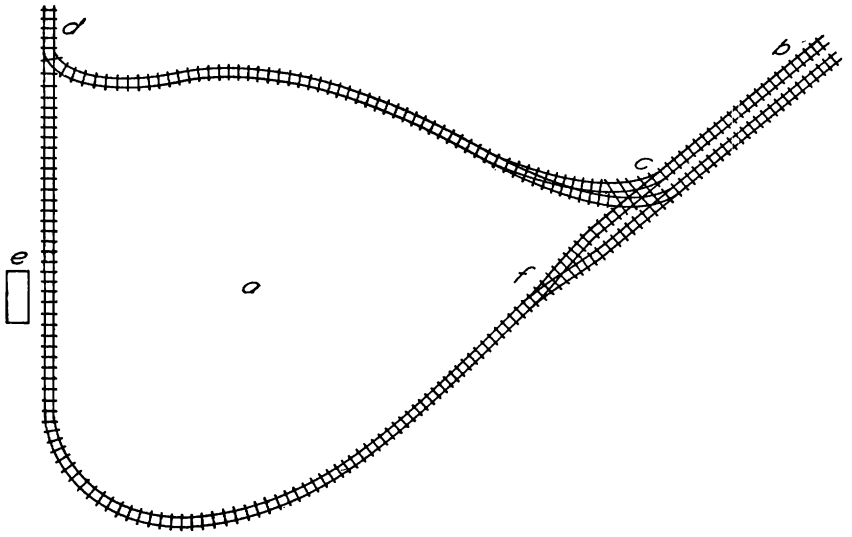

FIGURE 31.—An efficient track system used in a western New Jersey quarry: *a*, Pit; *b*, incline; *c*, point at which empties are switched off; *d*, steep incline where empties are stopped and reversed; *e*, steam shovel; *f*, foot of incline.

30 or 40 feet from the bottom, at *c*, and run on a gentle down grade to *d*, where the track is elevated abruptly. This elevation stops the car, and as it returns an automatic switch directs it to the track leading to the steam shovel, *e*. The car is thus turned around. It descends by gravity to the steam shovel, where it is stopped by placing a block on the rail. When the car is loaded the block is removed and the car again travels by gravity to the foot of the incline, *f*, where the hoist cable is attached. The success of this system depends chiefly on getting exactly the proper grade, which must, of course, be governed by the ease of running of the cars. If the grade is too gentle the cars may stop, and if it is too steep they may be stopped with difficulty or may jump the track. This method eliminates all

horses, mules, and dinkey engines from the quarry. The system is so nearly automatic that little labor is required. With a gang of only 16 men this quarry can produce about 1,200 tons of rock per day.

IMPORTANT FACTORS IN SUCCESSFUL TRANSPORTATION.

In order to attain high efficiency in transportation, the quarryman should aim to have strong cars, an ample supply of cars, a simple haulage system, a combination of as few methods of transportation as possible, and should utilize the force of gravity wherever practicable.

ROCK CRUSHING.

LIMITS OF DISCUSSION.

Rock crushing is a process of manufacture rather than a quarry process, consequently no attempt is made to discuss it herein except insofar as it influences or is influenced by quarry methods or rock structures. The features of special interest to quarrymen are size of crushers, rate of crushing, and situation of crushing plant with respect to rock transportation.

SIZE OF CRUSHERS.

The size of the crusher should be governed by the size of the rock fragments that can be loaded and transported. Thus, when hand loading is employed the small fragments obtained may be easily handled by a small crusher. In most quarries where the hand-loading method is employed a gyratory crusher of about No. 9 size is used. Where loading is with steam shovels larger crushers are desirable. Gyratory crushers of No. 18 and No. 21 sizes have been observed where steam shovels with 3½-yard or 4-yard dippers are employed.

The attempt to use a small crusher for breaking rock loaded with a steam shovel is likely to result in great loss of time. Many blocks that could easily be loaded with the shovel must be set aside and broken by blasting, because they are too large for the crusher. Also, if the crusher is too small, the larger pieces may jam the crusher and cause much delay.

At one quarry where rock loaded with a steam shovel was crushed with a No. 9 gyratory crusher, as much explosive was used for secondary as for primary blasting because the rock had to be broken into such small pieces.

The size of the crusher required depends to some extent on the character of the rock cleavage. Some rocks have "cubic cleavage," that is, the rock tends to break into cubical blocks. Others have a "slab cleavage," and tend to split out in relatively thin slabs. Thus, at one cement rock quarry the ratio of length to width to thickness of the average fragment is 5 to 3 to 1. A cubic block 2½ feet in size is approximately of the same volume as a slab 5 feet long, 3 feet wide, and 1 foot thick. The slab would pass through a rectangular opening only 3 square feet in area, whereas the cubic block would require an opening more than 6 square feet in area. Obviously, for crushing a

given volume of stone per day, a larger crusher should be provided for rock having cubic than for rock having slab cleavage. The following comparison of data may serve to illustrate the effect of rock cleavage on ease of crushing.

At one quarry where the rock has slab cleavage, the rock was loaded with a No. 61 Marion steam shovel and crushed with a No. 9 gyratory crusher. Three men were employed to dump the rock cars and conduct the rock into the crusher, and the average daily output was about 700 tons. At another quarry where the rock has cubic cleavage, the fragments were loaded with a No. 60 Marion steam shovel, and crushed with a No. 9 gyratory crusher. Four men were employed to force the rock into the crusher, and the daily output was about 400 tons. It will be noted that in the quarry where the rock broke in cubic blocks a smaller shovel was used, and consequently smaller fragments were loaded. However, notwithstanding this advantage, more men were required to force the rock into the crusher, and a much smaller daily output was obtained.

CAPACITY OF CRUSHER.

Where steam shovels are employed, the crusher should be large enough to handle any blocks that can be conveniently loaded, and also should be able to crush with sufficient rapidity to prevent delay in loading and transportation. Where gyratory crushers are used, slippery clay mixed with the rock may greatly decrease the capacity of the crusher during rainy periods. The effect of clay is less pronounced in large gyratory crushers. Large rolls provided with blunt teeth are used for crushing at several plants and give satisfactory service where the rock is not exceptionally hard. They may be operated in pairs, the rock being crushed between them, or singly, a baffle plate being substituted for the second roll. They have the advantage of having wide, hopper-like mouths that will take large-sized fragments. There is little danger of delay through jamming of rock fragments in roll crushers, although the rock may arch in the hopper if a large quantity is dumped in at one time. The larger sizes of rolls are capable of crushing rock very rapidly. A single roll 60 inches in diameter and 84 inches long was observed working in limestone that was loaded with steam shovels. When timed by the writer, it crushed the rock to 8-inch size or smaller at the rate of 11 tons per minute.

Jaw crushers are preferred by a number of quarrymen. On account of the wide mouth little labor is required in feeding the rock into the crusher, also the cost of secondary blasting is reduced. At one jaw crusher observed, a traveling apron governs the supply of rock and equalizes the flow, which would otherwise fluctuate greatly when large carloads are dumped into the hopper at intervals.

DEVICES FOR MOVING JAMMED BLOCKS.

Loosening blocks of stone jammed in a crusher by using a bar or sledge may delay and even temporarily suspend quarry production. Most well-equipped crushing plants are provided with mechanically operated hooks, by means of which a jammed block may be readily moved. In some plants the hook is raised with a rope tackle or with some form of cable hoist. The pneumatic hoist is probably the quickest and most convenient, but can be used only where compressed air is readily available.

SITUATION OF CRUSHER.

The method of transportation is governed to a large extent by the situation of the crushing plant. At a few quarries the crusher is so situated that little or no elevation of stone is required. At one New York quarry the crusher is placed on a hillside, at a lower level than the quarry floor, and the rock is discharged at a still lower level into a storage bin. This simplifies transportation, as no cable-car incline is required, and minimizes power consumption, for at no stage is the rock raised to a higher level. Some such method should be followed wherever possible, but in most places the topography makes elevation of the rock quarried unavoidable.

SAFETY AROUND CRUSHERS.

It is dangerous to work around crushers, especially those of the roll type. Adequate measures should be taken to prevent accident to workmen barring down rock in the crusher hopper. Attention of operators is directed to a safety device employed in an Indiana and a New York quarry. Each man who works close to the crusher mouth is provided with a leather belt to which a rope is attached at the back. The rope is fastened securely at the other end, and is of sufficient length to allow freedom of action, though short enough to prevent the workman from falling into the crusher.

MIXING THE RAW MATERIALS.

MIXING OF RAW MATERIALS IN RELATION TO QUARRYING.

The mixing of the raw materials of Portland cement is a manufacturing and not a quarrying process. However, the method of mixing has a pronounced effect on the quarry methods in many places. So complete is this interdependence that at a number of plants the chemist controls the quarry method to a considerable extent. In the preceding pages the saving that may be effected by employing modern quarry methods has been emphasized. However, the diffi-

culty of getting a proper and constant mixture of raw materials may impose such restrictions on the quarry that these methods can not be used. Consequently the mixing process, insofar as it affects quarry methods, is discussed herein.

IMPORTANCE OF MAINTAINING A PROPER MIXTURE.

Probably no part of the process of manufacturing Portland cement requires more care and vigilance than proportioning the mixture of raw materials. On proper mixing depends to a great extent the quality of the cement and, therefore, the success of any cement manufacturing enterprise. Variations in chemical composition within certain limits are permissible, but every cement company finds that a certain definite ratio of principal constituents gives a superior cement. Hence every reasonable effort should be made to maintain this composition as nearly as possible.

CONSTANT AND VARIABLE ROCK LEDGES.

There are two distinct types of quarries—those having rock of fairly constant composition throughout their full extent, and those in which the rock varies considerably in composition from point to point. With quarries of the first type, the method of quarrying has little or no influence on the mixing process, whereas with those of the second type mixing so depends on quarrying and loading that some cement manufacturers think that a proper mixture can not be made without quarrying the rock selectively from various parts of the excavation.

ADVANTAGES OF ROCK OF UNIFORM COMPOSITION.

In quarrying rock ledges of fairly constant composition the methods used have not been subject to limitations imposed by the cement-plant superintendents. As a consequence the quarry superintendents have been enabled to take advantage of the improved methods that have recently been developed in many limestone quarries where rock is obtained for concrete aggregate or road building. Deep-hole blasting and steam-shovel loading have, in such quarries, largely replaced blasting in tripod holes on shallow benches and loading the rock by hand.

DISADVANTAGES OF ROCK OF VARIABLE COMPOSITION.

In most quarries where the rock varies considerably in composition, the plant chemist insists on a certain number of cars of rock being taken from each bench or each part of the face in definite order. The necessity for picking and choosing the rock here and there

excludes the possibility of developing a high face, hurling down great quantities of rock at one time by the deep-hole blasting method, and loading it with a steam shovel. Instead, the tripod or hammer drill must be used, the rock must be worked in benches, car tracks multiplied, and the rock loaded by hand.

A COMPARISON OF QUARRY METHODS.

It has been found by careful investigation in many quarries that the quarry costs per ton of rock obtained are as a general rule much lower where the churn drill and steam shovel are employed than where smaller drills and hand-loading methods are used. This is shown more clearly by figures obtained from various quarries. The average total operating or working cost of delivering rock to the crusher—stripping not being included—for 11 quarries employing the hand-loading selective method was 27.4 cents per ton. The highest cost was 35 cents per ton and the lowest 18 cents. An average of the same costs for eight quarries employing steam-shovel methods was 21 cents per ton. The highest was 32 cents per ton and the lowest 12 cents. As indicated by these figures, the costs may be reduced about 23 per cent by the use of steam shovels. Many cement manufacturers, either through lack of appreciation of the advantages of modern methods or through inability to devise new mixing methods, still adhere to the old ways and are thereby seriously handicapped in attempting to quarry rock as cheaply as it is done in quarries having rock of constant composition.

NEED OF IMPROVED PROCESSES OF MIXING RAW MATERIALS.

Therefore, it is a matter of supreme importance to cement companies operating quarries of variable rock to determine whether the method of mixing the cement constituents can be so modified that the quarryman is not forced to retain slow and costly methods. If such a change can be brought about, quarry costs may be greatly reduced, the output readily increased, and the chemical work materially lessened. As a result of observations made at many plants, the writer is convinced that this can be done. No new and untried methods are suggested, but simply combinations of methods now in successful operation.

The object of this discussion is to show that mixing methods and quarrying methods can be so adjusted that the desired mixture may be obtained without sacrificing quarry efficiency. In other words, the author has attempted to point out methods of securing and maintaining a desirable mixture of cement ingredients from a quarry of variable rock, at the same time employing the churn-drill and steam-shovel method of quarrying.

MIXING THE MATERIALS IN THE QUARRY.

Instead of quarrying the rock selectively and later mixing the various types in definite proportion, it is suggested that the rock ledge be shot down in great masses, irrespective of any incidental mixing that may occur, and the desired mixture obtained by methods described in subsequent pages. The more thoroughly the rock is mixed in the process of quarrying, the easier can the proper mixture be obtained at later stages. A number of quarrymen maintain that the charge of explosive in deep churn-drill holes may be so adjusted that the rock in successive variable beds may be mixed with a fair degree of uniformity as it falls to the quarry floor after blasting. Blasting should, therefore, be so conducted as to bring about this result as fully as possible.

However, in many quarries incidental mixing through blasting can not be relied on to give even an approximately uniform mixture. In several quarries bands of shale interbedded with limestone are shot down and loaded with the limestone. If the shale is in bands of uniform thickness and is thoroughly mixed with the limestone, its presence may not be detrimental; but in some places the shale bands vary greatly in thickness, resulting in undesirable fluctuations in the composition of the quarry product. Also, the composition of the limestone itself may vary considerably from point to point.

If the rock, when thrown down by well-adjusted blasts, still varies considerably in different parts of the quarry, the employment of two or more shovels, each loading a different class of rock and all working simultaneously, will tend to increase the uniformity of the product when all the carloads are dumped into the same bin. This method is employed at Martins Creek, Pa.; Earlham, Iowa; Dewey, Okla.; and a number of other localities. Two shovels are commonly employed at the same time, one in high-calcium and the other in low-calcium stone. If the quarry is too small to justify the use of more than one large shovel, two or more small tractor shovels may give more satisfactory results. However, it is believed that the steps outlined subsequently will insure sufficient uniformity in the mixture even when a single shovel is employed. In one quarry observed a single large shovel was operated successfully in rock that varied from 42 to 90 per cent in its content of calcium carbonate.

MIXING PROCESSES IN CEMENT PLANTS.

THE THREE PROCESSES EMPLOYED.

According to the methods now followed in many plants, the maintenance of a uniform and desirable mixture would be impossible by the method of quarrying outlined in the preceding paragraphs. To

clearly understand the reason for such limitations, it is necessary to review briefly some of the methods of mixing the raw materials.

Three methods of mixing, the dry, the semiwet, and the wet, are in common use. The dry method is the most common for mixing raw materials obtained from quarries. The wet method is used almost exclusively in the manufacture of cement from marl, the material being excavated wet and kept wet until the moisture is driven out in the kilns. For the semiwet process the raw materials are at first in a dry state, but at some stage water is added, the subsequent steps being similar to those of the wet process.

DRY PROCESSES.

TYPICAL EXAMPLES OF DRY PROCESSES.

Some typical examples of dry methods may be briefly outlined. In one plant the limestone is crushed in a No. 8 gyratory crusher, and all fragments larger than 2-inch are recrushed in a No. 5 crusher. The rock is dried and then passed through a ball mill, after which clay is added. The clay is ground, dried, and added to the limestone in the desired proportion by weight. The resulting mixture is pulverized in Fuller mills. It is sampled every hour, and the proportion of clay needed is calculated from analyses of the samples.

At another plant observed, the crushed rock is conveyed to a storage bin, after which it is passed through driers and then through hammer mills. Shale is added and the mixture is pulverized in Fuller mills and is then ready for burning. As before, analyses of the mixture are made every hour.

At a third plant, the crushed and dried rock is passed through ball mills, the desired amount of shale added, the mixture pulverized in tube mills, and then burned.

A noteworthy feature of the methods outlined, which is common to most of the dry-mixing methods observed, is the fact that the proportions of the raw materials are usually fixed by analyses of preceding mixtures, and that there is no means of changing the mixture after it is once made, except insofar as a change in subsequent mixtures may affect it. In other words, one mixture is analyzed, and as a result of this analysis a change is made in a subsequent mixture, which may or may not need the change. If the raw materials are subject to only slight fluctuations in composition, the method gives satisfactory results, but if marked changes in composition occur an undesirable mixture is likely to result. The selective, hand-loading method of quarrying is the direct outcome of this inability of the plant chemist to cope with marked fluctuations in composition of the raw materials. In some plants, mixing has become a process of almost pure guesswork. The chemist becomes

very familiar with the rock, judges its composition with fair accuracy from its appearance, and regulates the mixture on the basis of this judgment. Needless to say, the method demands constant vigilance and careful quarry supervision.

METHODS OF OVERCOMING FLUCTUATIONS.

One fairly effective method of meeting the difficulty is to run the mixed and pulverized material into a large storage bin before it enters the kilns. If a supply sufficient for eight or nine hours' run is maintained the operator may compensate for any undesirable variation. Thus, if the mixture is found to be too high in calcium, an excess of clay or shale may be added for a time sufficient to reduce the average calcium content to the desired point. This method can succeed only where proper mixing of the materials may be brought about in the bin.

A more accurate method of mixing is followed by one Oklahoma plant where the limestone is sampled before the shale is added. The limestone is crushed, dried, and ground, and then by means of an intermittent sampler a sample is taken about every 65 seconds. All the samples for a given period of time are mixed, an average analysis obtained, and the proper proportion of shale calculated and added by weight. As clay stripping is shot down and loaded with the rock in the quarry, a small amount of shale only is added. The mixture is then pulverized in tube mills. The product of five tube mills is all conducted to a single bin, from which it is distributed to the kilns. Such mixing gives greater uniformity to the product.

THE BEST METHODS OBSERVED.

An admirable method observed of making a dry mix is successfully used by a company in the Middle West. Blasting in churn-drill holes and steam-shovel loading are employed. A clay overburden of good quality is shot down and loaded with the rock, and on this account the addition of shale is unnecessary. The rock is crushed, dried, and ground in a ball mill to 14-mesh size. The ground material is then taken to storage bins, and during transit a continuous sampler takes a representative sample, which is analyzed every 3 hours. A record is kept of the rate of entry, the number of hours that rock is discharged into the bin, and the average composition for each 3-hour period. From this data a fairly accurate analysis of the entire contents of the bin may be deduced. The ratio of silica and alumina to calcium is calculated by Newberry's formula that carbonate of lime=silica$\times$5+alumina$\times$2. For example, if the estimated analysis of the entire bin shows 12 per cent silica, 7 per cent alumina, and 72 per cent calcium carbonate, the desired percentage of calcium car-

bonate would be 74. As only 72 per cent calcium carbonate is present, the contents of this bin would be expressed as —2. Twelve such bins are maintained, and each is designated by a plus or minus figure, which indicates the excess or shortage of calcium carbonate.

In drawing off the material it is taken from not less than three bins at once, and a combination of bins is selected such that the + and — quantities will give a zero result. The first run is sampled and its composition approved before the mixture is allowed to go to the tube mills for final grinding. This method has advantages over methods of mixing pure shale and limestone. In the latter methods temporary obstruction of the flow of shale or limestone decidedly affects the composition of the mixture, whereas with the method outlined, the contents of the various bins differ so little in composition that the effect of obstructing any one outlet for a brief period would be small. For plants where shale is mixed with limestone, it is, therefore, desirable to add the shale before the material enters the storage bins.

An improved type of bin, having for its object more complete mixing of the materials, was observed by the writer at an Illinois plant. Four bins are provided for storing rock after it is crushed, dried, and ground to 20 mesh. Each bin is divided into four parts by vertical partitions. The parts are filled in succession, but the material is drawn off from all four simultaneously. This tends to equalize any fluctuations in composition. Uniformity in the final mixture is further promoted by having sufficient storage room at the kilns for 3 or 4 hours' supply.

Another improvement suggested is a simple device for proportioning the amount of material from each bin. It consists of a series of sprocket-driven conveyors, each having three speeds. Thus, if material is to be taken from three bins in the proportion of 1 to 2 to 3, the conveyor for the first bin could be operated at slow speed, the second at intermediate, and the third at high speed.

Another company quarrying limestone interbedded with shale maintains a two weeks' supply of rock. The proportions of shale and limestone in the ledge are such that if the entire face is thrown down as a single bench and the mass thoroughly mixed, the resulting mixture has approximately the proper composition for cement. The large storage bin assists in obtaining a uniform mixture. The rock is drawn from the storage bin, pulverized, and conducted to 16 concrete tanks. Continuous samples are taken, the samples being analyzed every hour. From the data thus obtained the average analysis of the material in each tank is determined. The material is drawn from one, two, three, or a maximum of four openings in each tank, and a combination of tanks is determined that will give the desired mixture.

One New York plant maintains six storage bins. The material may be drawn from each bin by two worm feeds, each controlled by a 3-speed cone pulley. Thus six rates of removal from each bin are possible.

Meade[a] suggests that the raw material, after it is passed through ball mills, be stored in tanks, the composition of the contents of each tank being slightly higher in lime than is desired. The material is sampled as it enters the tank and an analysis calculated for the entire mass. It is then an easy matter to calculate the small proportion of clay or shale that must be added to bring the mixture to the proper composition. At least three bins should be maintained for the mixture and one for shale.

A slight modification of this method is proposed by an Illinois cement company. The chemist of this company has found that the composition of the cinders from the boiler room is approximately the same as of the shale used in the cement mixture. It has been proposed, therefore, to run the mixture a little higher in calcium than is desired for cement, and to correct the mixture by means of an auxiliary cinder feed. By this means any fluctuation in composition could be corrected immediately, whereas by the method used at the time of the writer's visit the correction was made about 24 hours after the defective mixture has gone by.

THE SEMIWET PROCESS.

DESCRIPTION OF THE PROCESS.

Fluctuations in the composition of raw materials may also be overcome successfully by use of the semiwet process, also called the semidry process. Both terms are misnomers, as the materials are just as wet as in the wet process. The term "semiwet" implies the subsequent wetting of materials originally dry. The advantages claimed for the semiwet process are intimate mixture of the particles and ease of obtaining a uniform mixture of the cement constituents. This latter phase of the process will be considered here. The process is best described by reference to a concrete example.

At an Iowa plant where the semiwet process is successfully employed, the raw materials consist of limestone and shale. The limestone is variable in composition, the calcium carbonate content varying from 72 to 92 per cent. The rock is shot down by blasts in churn-drill holes and is loaded with two steam shovels. The quarry operators aim to keep one shovel working in the high-calcium and the other in the low-calcium stone, and thus equalize the product to some extent.

[a] Meade, R. K., Portland cement, 2d ed., 1911, p. 103.

The rock is crushed and passed through a ball mill, after which ground shale and water are added. The mixture is pulverized wet. The slurry, containing about 40 per cent of water, is conveyed by screw feed through horizontal troughs to storage tanks. Seven tanks are maintained, each having a capacity of about 600 barrels of cement. Samples are taken from the troughs with a continuous sampler and analyzed every hour. The composition of the contents of each tank is determined from these analyses. It is customary in some plants to maintain high-calcium and low-calcium tanks.

The method of making the final mixture is very similar to that employed for the dry process described on page 143. Knowing the analysis of the contents of each tank, the operator can easily work out a combination from two or more tanks that will give the desired composition. The eighth tank is reserved for mixing the slurry drawn from the storage tanks. An air agitator is employed to thoroughly mix the contents of the tank, and when so mixed the composition of the entire contents may be determined from a single analysis.

ADVANTAGES OF THE SEMIWET PROCESS.

When wetted to the consistency of a slurry, a very intimate mixture of the particles may be obtained. It is claimed that grinding or pulverizing the material wet requires at least one-third less power than when ground or pulverized dry, although the counter claim is made that the wet material wears the pulverizing machinery much more rapidly than the dry. Pumping the slurry is a very simple and cheap method of moving it from place to place. Also materials mixed and conveyed in a wet condition make no dust.

The greatest advantage, however, is the ease with which variable raw materials may be mixed together and a final mixture of the desired composition obtained, as outlined above. This feature of the semiwet process recommends it as an alternative method in plants working on materials of variable composition.

DISADVANTAGES OF THE SEMIWET PROCESS.

The greatest difficulty in connection with the semiwet process is getting rid of the 30 or 40 per cent of water that has been added. The process was first employed where the water could be evaporated at small cost, because of cheap natural gas. Where powdered coal is used for fuel, drying is more costly. The difficulty has of late years been overcome with fair success by the use of long kilns. In kilns 160 feet long, so it is claimed, all the moisture is driven out during the passage of the material through the first 35 feet of the kiln. Therefore, the same output may be obtained with slurry in a 160-foot kiln as by the dry process in a 125-foot kiln. It must not

be presumed, however, that the entire disadvantage of the semiwet process is the additional first cost and maintenance of long kilns. It is unwise to assume that the heat used in evaporating the water would, with a dry mixture in shorter kilns, be entirely wasted. By means of waste-heat boilers this heat may be saved and utilized.

DESIRABILITY OF PROPER PLANT DESIGN.

The realization that cheaper and more rapid methods of quarrying than the selective hand-loading method are to be desired has led a number of operators to seek some process of mixing raw materials that will permit the use of churn drills and steam shovels. The semiwet process has apparently appealed to several operators as the best means of obtaining a uniform mixture of variable raw materials. As a consequence a number of plants that were originally designed for dry mixing have been remodeled and the necessary equipment added to adapt them to the semiwet process. Several plants of this type differ in some important respects from one properly constructed. Thus in the attempt to retain as much as possible of the old equipment, water may be added at a much later stage than in plants originally designed to operate by the semiwet process. The grinding and pulverizing may be done while the materials are dry, the water being added subsequently. As a result such plants have the disadvantages of both processes, the slower and more costly pulverizing of dry materials, and the high fuel cost of burning a wet mixture. Also, it has been pointed out that long kilns are required in operating a semiwet process to advantage. In remodeling a plant to adapt it for this process, the introduction of new and longer kilns is an expense that few cement-plant operators would care to undertake.

In planning to remodel a plant in order to modify the mixing system in such a way as to encourage more advantageous quarrying, the operator should carefully consider the original design of his plant and the extent of remodeling demanded. Considered from every angle, it is probable that the dry method should be retained in plants designed for a dry process. It is evident that less modification would be required to introduce a system of bins as described on page 143, than is necessary to convert the plant into a properly equipped semiwet plant.

THE STORAGE BIN AS A MIXING DEVICE.

It has been pointed out that variable constituents may be mixed to some extent in the quarry by intelligent deep-hole blasting, and that the quarry products may at a later stage be mixed together as desired, either by the employment of a series of bins, the contents of which may be apportioned in accordance with their composition,

using the dry method; or by a similar apportionment of the contents of slurry tanks, using the semiwet process.

Final mixing of the materials may, however, be simplified by having a large storage bin for crushed rock. A mixture of remarkable uniformity has been obtained from a rock ledge of variable composition by this means. However, in order to attain the desired result the bin must be used intelligently. The following example will serve to illustrate the proper use of the storage bin:

One plant has a crushed-rock storage bin of 40,000-ton capacity. The rock is distributed uniformly over one-half the bin by means of a belt conveyor, until this half is filled. It is then drawn off while the other half is being filled. The rock is taken from beneath by two belt conveyors. It falls from the bin through 24 slides that are opened in succession for 5-minute periods. Thus through uniform distribution in entering the bin and equally uniform removal, fluctuations in composition of the rock as it enters the cement plant are very slight, and the proper proportion of shale to be added is easily maintained.

INCOMPLETE MIXING IN STORAGE BIN.

The essential factors in successful mixing are proper distribution of the materials in the bin, and removal of the contents in such a manner as to cause further mixing. At a number of plants observed the rock-storage bin is employed merely for storage purposes and its utility as a mixing device is not realized.

At one plant the crushed rock is distributed in the bin with a clamshell bucket and traveling crane, a type of equipment that gives excellent service when properly used. As observed by the writer, however, the material, instead of being distributed in various parts of the bin and removed by taking bucket loads from different points, was taken from the point where it entered the bin from the crusher, and was conveyed directly to the drier. As the rock was loaded at the quarry with a steam shovel it is obvious that by such direct transference mixing was imperfect. Such procedure is justified only where the rock is constant in composition, otherwise a distribution should be made in the bin and material for the drier selected from the mixed mass.

Where crushed rock is deposited centrally in a storage bin, as by a belt or bucket conveyor, the finer materials tend to remain near the point of entry, and the coarser fragments to roll toward the sides. If clay is quarried with the rock, the clay, being finer, tends to accumulate near the center, and the limestone toward the sides. If the outlet of the bin is also centrally placed, an excess of clay will be obtained when the outlet is first opened, and the proportion of limestone will gradually increase. This undesirable condition has been

observed in a number of plants. It is wise, therefore, to select for the outlets positions not directly beneath the point of entry. In the 40,000-ton storage bin referred to in a previous paragraph, the rock is distributed centrally by a belt conveyor, and removed from beneath through two lines of holes situated toward the sides of the bin.

Where dissimilar materials are used, such as limestone with clay that pulverizes to a finely divided condition, the tendency to segregate is marked. Where limestone and shale are employed, this tendency is less pronounced. A uniform mixture is therefore more readily obtained in a storage bin where materials of like physical character are employed, shale with limestone, and clay with marl.

Having thus obtained a fairly uniform mixture in a storage bin, the subsequent processes of mixing and proportioning the cement materials are greatly simplified. A large storage bin is particularly advantageous where the dry process is employed.

Where rock is stored in bins after crushing, it may be dried either before or after storing. With some materials, however, such as clay and some shales, the better plan is to dry first, as the damp materials may pack in the bins and make removal difficult.

The best construction for a storage bin is concrete and steel. The roof trusses may be made to rest directly on the bin walls, and the belt conveyors for bringing in the stone may be carried by the trusses.

SUMMARY OF CONCLUSIONS.

Mixing methods in their relation to quarrying may be briefly summarized as follows:

Churn-drill and steam-shovel methods of quarrying are, in general, to be preferred, and mixing of the quarry materials should be so conducted as to permit their use. Mixing may be done by using a number of storage tanks for approximately proportioned mixtures of ground rock, the contents of the tanks being determined by analyses. The final mix is made by drawing off a calculated proportion from three or more tanks. Thorough mixing of the quarry products may be attained by the intelligent use of a large rock storage bin. The semiwet process is an alternative method of obtaining a uniform and desirable mixture of diverse constituents.

ECONOMIC CONDITIONS.

CONDITIONS GOVERNING PLANT LOCATION.

In choosing a location for a cement plant the presence of an adequate supply of suitable raw materials is not the only consideration. Other conditions of equal importance must be satisfied, or the enterprise will be doomed to failure. The chief factors, other than raw materials, on which success depends are market, transportation, and fuel.

MARKET.

It is unwise to attempt the manufacture of cement in localities where the demand for it is limited. A cement plant should have, first, a strong local market area in which there is practically no competition, and, second, easy access to a larger competitive market area. As cement is a very heavy material, transportation charges are relatively high when compared with its value, and on this account the competitive market area is usually restricted. No excellence in limestone or shale can compensate for a small market.

TRANSPORTATION.

The available market area largely depends on transportation facilities. A high freight rate tends greatly to restrict the field. No plant should be entirely dependent on one railroad line unless the railroad is financially interested with the cement plant. At least two transportation routes should be available, one being preferably a water route.

FUEL

In general practice, 120 pounds of coal dust is required to burn one barrel of cement. If coal is also used for power 200 to 300 pounds are required for each barrel of cement. The fuel cost for an average plant is, therefore, 30 to 40 per cent of the total cost of manufacture. A hydroelectric plant may supply the necessary power to operate the machinery, but fuel is necessary for burning the cement. Hence, a good fuel supply at reasonable prices is a necessity.

RAW MATERIALS.

The success of any cement manufacturing enterprise depends largely on the nature, availability, and extent of the raw materials, particularly in areas where competition is keen. The plants that have raw materials that can be quarried, pulverized, and burned at relatively low cost always have an advantage over competitors. On the other hand, if there is a good local market and cheap fuel, a manufacturer may be justified in building a plant where raw materials are of inferior quality.

No plant should be built having less than 20 years' supply of raw material in sight. Each barrel of cement requires approximately 450 pounds of limestone and 150 pounds of clay or shale. Therefore, a plant making 1,000 barrels daily will use annually about 66,000 tons of limestone and 22,000 tons of clay or shale. This quantity is equivalent to almost 1,000,000 cubic feet of limestone and 250,000 cubic feet of shale. A 1,000-barrel plant should, therefore, have 20,000,000 cubic feet of limestone and 5,000,000 cubic feet of clay or shale available on its property.

EFFICIENCY OF OPERATION.

A cement plant established under the most favorable conditions may fail to yield a profit if operated with poor efficiency. The choosing of a site for the plant need be done but once, and the prospective operator may engage the services of a competent engineer to insure its proper location, but the operation of the plant demands constant attention every day. It is, therefore, supremely important that the cement manufacturer familiarize himself with all modern methods and equipment designed to increase efficiency of operation. Where competition is keen those who keep most closely in touch with each new advance are best enabled to reduce the cost of manufacture, and thus widen their markets.

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PUBLICATIONS AVAILABLE FOR FREE DISTRIBUTION.

BULLETIN 16. The uses of peat for fuel and other purposes, by C. A. Davis. 1911. 214 pp., 1 pl., 1 fig.

BULLETIN 45. Sand available for filling mine workings in the northern anthracite coal basin of Pennsylvania, by N. H. Darton. 1912. 33 pp., 8 pls., 5 figs.

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