Bulletin 57

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
JOSEPH A. HOLMES, DIRECTOR

SAFETY AND EFFICIENCY
IN
MINE TUNNELING

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WASHINGTON
GOVERNMENT PRINTING OFFICE
1914

First edition. March, 1914.

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SAFETY AND EFFICIENCY IN MINE TUNNELING.

By David W. Brunton and John A. Davis.

INTRODUCTION.

PURPOSE OF REPORT.

During the past few years great progress has been made in the United States toward safer, more efficient, and more economical tunneling methods. This advance is partly due, no doubt, to the recent increase in the number of tunnels and adits driven for developing and draining mines and transporting ore. The Bureau of Mines during 1911 and 1912 made a special examination of this phase of mining operations, in connection with an investigation of mining methods and means for preventing accidents. The details especially studied were the provisions for the safety of employees, the kinds of equipment, the methods of driving, and the costs of construction. The results and conclusions obtained from that investigation are discussed and summarized in this bulletin.

Up-to-date information concerning tunneling methods is difficult to obtain. There are few books on the subject; and much of the material they contain, although interesting and of value historically, is now obsolete. The engineering periodicals, it is true, endeavor to keep abreast of the times, and several in nearly every issue present some article bearing upon tunnel work. But the very multiplicity of these articles prevents one from reading them all, and the foreman or superintendent in charge of a tunnel, or the mining engineer designing one, and especially the business man financing the project, has no time for a lengthy search after scattered articles in order to determine the present status of tunnel work. Then, too, knowledge of new methods travels slowly. Valuable inventions and improvements in tunneling as well as in all the other industries frequently remain in the notebook of the investigator, or as theses are buried in university libraries, or are published only in journals of small scientific societies. New tunneling methods and equipment that are proving safe, efficient, and economical may be totally unknown outside the district in which they originate. This paper, based on a special examination both in the field and in engineering literature, is intended to supply data concerning modern and recent tunneling practice in the United States, and to make suggestions that, it is hoped, will result in a saving to the mining industry of energy, capital, and life.

In most of the published accounts of tunnel work the writers do not attempt to criticize the methods they are describing. The articles usually present accurate descriptions of equipment and of various phases of working operations, with occasional figures showing the cost of the work; rarely do they include a discussion of the means for preserving the health and life of the employees or data bearing on the choice and efficiency of equipment, or an analysis of methods and costs. As a result, the reader who endeavors to draw conclusions is dependent wholly on his own resources. In this bulletin, on the contrary, the making of such analyses will be a primary consideration, but as the purpose of the bulletin is to give an impartial disinterested report, the criticisms made are intended to be constructive rather than destructive. For that reason emphasis is placed on safe, efficient, and economical methods and on good points of equipment, whereas bad practice and obsolete machinery are ignored, except, perhaps, as examples of what was inadvisable or as having some bearing historically. Thus the authors hope to set forth a guide for future work rather than an unillumined record of past or present achievement.

SCOPE OF REPORT.

This bulletin is confined chiefly to a discussion of tunnels and adits for mining purposes, such as drainage, transportation, or development, but it also discusses those used to carry water for power, irrigation, or domestic use, in which the essential features are practically identical with mine tunnels.

Most tunnels of the sort discussed are driven through rocks at least fairly hard in contrast to ordinary soil, quicksand, and other heavy material of a treacherous nature, and practically none is driven through recent river-bed deposits. Therefore descriptions of the special methods and equipment for tunnel work in such materials are omitted. A distinction is made between tunnels or adits for which the excavation is wholly or in a large part in material containing no ore and those that follow a vein. As far as possible the discussion is limited to the former, because the methods employed in driving along a vein are usually more akin to the distinctive operations for removing ore and are therefore not so apt to be good examples of tunnel practice.

It has been suggested by prominent authorities that the word "tunnel" be restricted to the designation of such nearly horizontal passageways as extend completely through a mountain or hill from daylight to daylight, and the words "adit" and "drift" be used only for nearly horizontal galleries that enter from the surface and serve to drain a mine or furnish an exit from the workings but do not continue entirely through the hill. Such definition is eminently desirable from strict technical consideration, and would contribute to precision of usage, but, although the suggestion was made over 30 years ago and has been repeated several times since, such usage is not widely established. The American practice of referring to any horizontal gallery as a tunnel, without regard to whether it extends completely through a hill, is so firmly fixed in mining literature and among mining men in this country, even being embodied in the United States mining laws, that the use of a more precise definition has been thought scarcely justifiable in this report.

ACKNOWLEDGMENTS.

In the preparation of this bulletin the writers are deeply indebted to the officials of the New York Board of Water Supply, of the Los Angeles Aqueduct, and of the United States Reclamation Service, and to the officers, managers, superintendents, and foremen at the different tunnels for favors granted, for information supplied, often at no little inconvenience, and above all, for a hearty cooperation that has been an unfailing source of inspiration. Many thanks, also, are due the manufacturers of equipment and materials used in tunnel work for their promptness and courtesy in furnishing catalogues, data of tests, and similar material, and in supplying photographs, blue prints, and cuts. Acknowledgment is made of many valuable suggestions obtained from articles in engineering periodicals and from books on tunneling and related subjects.

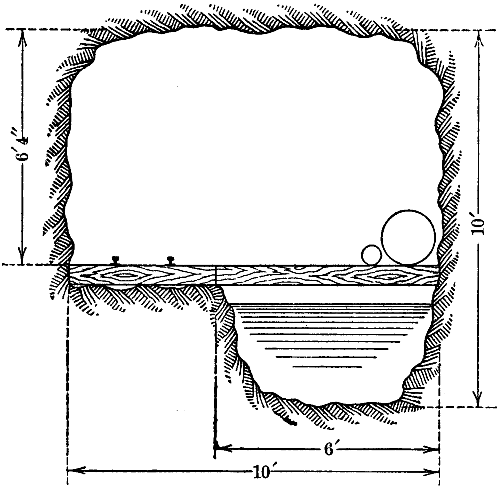

FIGURE 1.—Cross section of Roosevelt Tunnel.

TUNNELS VISITED.

The tables below briefly summarized the chief features of the tunnels and adits actually visited in the special field work upon which this report is based. Complete information was obtained wherever possible concerning surface and underground equipment, provisions for safety of the men, use of explosives, and methods employed in driving with regard to efficiency, cost, and other data bearing on construction.

Lack of space prevents a full presentation, but the tabulation conveys an adequate idea of the location, purpose, and magnitude of the different tunnels.

TABLE 1.—*Description of tunnels visited.*

Name of tunnel.	Location.	Operator.	Purpose.	Shape of cross section.	Size.	Length.	Character of rock penetrated.	Remarks.
						Feet.		
Burleigh	Silver Plume, Colo.	Dives-Pelican & Seven-Thirty Mining Co.	Mine drainage and development.	Rectangular	6 feet wide, 7 feet high.	3,000	Granite and gneiss.	Started by hand in fall of 1868; Burleigh drills introduced early in 1869. This was the first use of power drills for driving a mine adit in this country.
Carter	Ohio City, Colo.	Carter Mining Co.	Mine drainage and transportation.	Rectangular, with arched roof.	5½ feet wide, 7½ feet high.	7,600	Gneiss, granite, and porphyry.	Started July, 1897; on Nov. 1, 1911, driven 6,550 feet. Part of intervening time spent in drifting laterals; 3 years shut down entirely, and 5 years only three men at work.
Central	Idaho Springs, Colo.	Big Five Tunnel, Ore Reduction & Transportation Co.	do.	Rectangular	2,500 feet of tunnel, 12 feet wide, 8 feet high; remainder 5 feet wide, 7 feet high.	[a] 9,000	Idaho Springs gneiss.	
Gold Links	Ohio City, Colo.	Colorado Smelting & Mining Co.	do.	Rectangular, with arched roof.	6 feet wide, 8 feet high.	3,900	Gneiss, intruded granite, and porphyry.	Started May, 1906. Driven intermittently until completed, December, 1912.
Gunnison	Montrose, Colo	U. S. Reclamation Service.	Irrigation	Horseshoe	10 feet wide at bottom; 10 feet 6 inches at spring line; 10 feet high at spring line; 12⅓ feet high at center of arch.	30,645	Chiefly metamorphosed granite with some water-bearing clay and gravel, some hard black shale, and a zone of faulted and broken material.	Begun Jan. 11, 1905; headings holed through July 6, 1909; trimmed to full size February, 1910.
Laramie-Poudre.	Home, Colo.	Laramie-Poudre Reservoir & Irrigation Co.	do.	Rectangular	9½ feet wide, 7½ feet high.	11,306	Close-grained red and gray granite.	Started Dec. 25, 1909; completed July 20, 1911.
Lausanne	Mauch Chunk, Pa.	Lehigh Coal & Navigation Co.	Mine drainage	Arched roof	12 feet wide, 4 feet high at spring line; 8 feet high at center of arch.	20,000	Mauch Chunk red shale, Pottsville conglomerate, slate, and anthracite coal.	
Lucania	Idaho Springs, Colo.	Lucania Tunnel & Mines Co.	Mine development and transportation.	Square	8 by 8 feet	[b] 12,000	Hard granite	Started in the fall of 1901 and driven intermittently up to 1907, suspended from summer of 1907 to summer of 1910; tunnel driven up to Dec. 1, 1911, 6,385 feet.

Marshall-Russell.	Empire, Colo.	Marshall-Russell Gold Mining, Milling & Tunnel Co.	Mine drainage, development, mining, and transportation.	Rectangular	8 feet wide, 9 feet high.	*b* 11,000	Granite and gneiss.	Started October, 1901; 1,000 feet driven intermittently up to 1907; since then regularly; 6,400 feet completed Nov. 1, 1911.
Mission	Santa Barbara, Cal.	Board of water commissioners, city of Santa Barbara.	Water supply	Trapezoid	4½ feet wide at top; 6 feet wide at base; 7 feet high.	19,560	Shale, slate, and hard sandstone.	Completed, 1912.
Newhouse	Idaho Springs, Colo.	Argo Mining, Drainage, Transportation & Tunnel Co.	Drainage and transportation.	Square	8 by 8 feet	22,000	Idaho Springs gneiss.	Started by hand September, 1893. Machines installed Janaury, 1894. Operations suspended September, 1897; resumed Oct. 1, 1899; suspended Mar. 31, 1902; resumed June 20, 1904; suspended Apr. 30, 1905; resumed Apr. 1, 1906; suspended Feb. 22, 1907; resumed Dec. 16, 1908; suspended Dec. 31, 1909; resumed Apr. 1, 1910; completed Nov. 17, 1910.
Nisqually	Alder, Wash.	Nisqually Contract Co.	Hydroelectric power for city of Tacoma.	Rectangular with arched roof.	9½ feet wide; 11 feet high at center; walls 9 feet high.	10,000	Rhyolite	Started May 1, 1910. Progress Nov. 1, 1911 (two headings), 8,923 feet; completed 1912.
Ontario	Park City, Utah.		Mine drainage	Trapezoid	5 feet wide at base; 4 feet wide at top; 6 feet high above track; ditch 1½ feet deep, 5 feet wide, below tracks.	24,000	Porphyry, quartzite, limestone, and granite.	Started July 25, 1888; suspended several times for 1 to 14 months. Still unfinished.
Rawley	Bonanza, Colo.	Rawley Mining Co.	Mine drainage and development.	do	8 feet wide at base; 7 feet wide at top; 7 feet high.	6,235	Andesite	Started May 27, 1911. On Nov. 1 driven 1,940 feet, and on Dec. 1 2,335 feet. Completed October 1912.
Raymond	Ohio City, Colo.	Raymond Consolidated Mines Co.	do	Square	9 by 9 feet	3,200	Gneiss to 1,600 feet, then granite and schist with porphyry inclusions.	Started December, 1903; suspended March, 1908, to June, 1910. Length of tunnel driven Dec. 1, 1911, 2,900 feet. Completed 1912.
Roosevelt	Cripple Creek, Colo.	Cripple Creek Drainage & Tunnel Co.	Mine drainage	See figure 1		15,700	Pikes Peak granite with some phonolite schist near end.	Started June, 1907; work progressed steadily until January, 1911, when discontinued at 15,743 feet. Resumed October, 1911, and advanced 375 feet (to 16,118 feet) on Dec. 1, when work still in progress.
Siwatch	Leadville, Colo.	Siwatch Mining & Tunnel Co.	Development	Rectangular	6 feet wide by 7½ feet high.	*b* 5,000	Granite.	

a Approximate. *b* Projected.

TABLE 1.—*Description of tunnels visited*—Continued.

Name of tunnel.	Location.	Operator.	Purpose.	Shape of cross section.	Size.	Length. *Feet.*	Character of rock penetrated.	Remarks.
Snake Creek	Heber, Utah	Snake Creek Mining & Tunnel Co.	Mine drainage and development.	Rectangular with ditch on side.	9½ feet wide by 6½ feet high.	[a] 14,000	Diabase	Started May, 1910; suspended July, 1911, at 4,100 feet, because of bad ground near 3,000-foot station requiring reinforced concrete, which was completed about Jan. 1, 1912, and driving resumed.
Stilwell	Telluride, Colo.	Liberty Bell Gold Mining Co.	do	Square, with ditch at side.	7 by 7 feet	2,590	Conglomerate and andesite.	Started September, 1900; completed in 1905.
Strawberry	Strawberry Valley, Utah and Wasatch Counties, Utah.	U. S. Reclamation Service.	Irrigation	Straight bottom and walls and arched roof.	8 feet wide by 9½ feet high.	19,100	Limestone and interbedded sandstone for 10,000 feet, then sandstone and interbedded shale.	Started 1906; driven intermittently with temporary equipment until beginning of 1909; since then steadily from west heading only. On Oct. 1, 1911, had driven 11,165 feet. Completed in 1912.
Utah Metals	Tooele, Utah	Utah Metal Mining Co.	Transportation	Rectangular	10 feet wide, 8 feet high.	[b] 11,780	Quartzite	Started Nov. 27, 1906; suspended Apr. 13, 1909, to Dec. 9, 1909. Until Apr. 26, 1910, being enlarged from 4½ by 7 feet to 8 by 10 feet. Continued 8 by 10 feet since Apr. 27, 1910. On Nov. 1, 1911, had driven 6,050 feet.
Yak	Leadville, Colo.	Yak Mining, Milling & Tunnel Co.	Transportation and development.	Square	7 by 7 feet	23,800	Sandstone, limestone, shale, porphyry, and granite.	Started 1886. From 1888 to 1898 operated intermittently; from 1898 to completion in 1910 work practically continuous.

[a] Projected.

TUNNELS OF CATSKILL AQUEDUCT.

The Catskill Aqueduct (fig. 2), in Ulster, Orange, Putnam, and Westchester Counties, and the city of New York, N. Y., and operated by the board of water supply, New York City, includes the following tunnels, of which those examined in the field are typical:

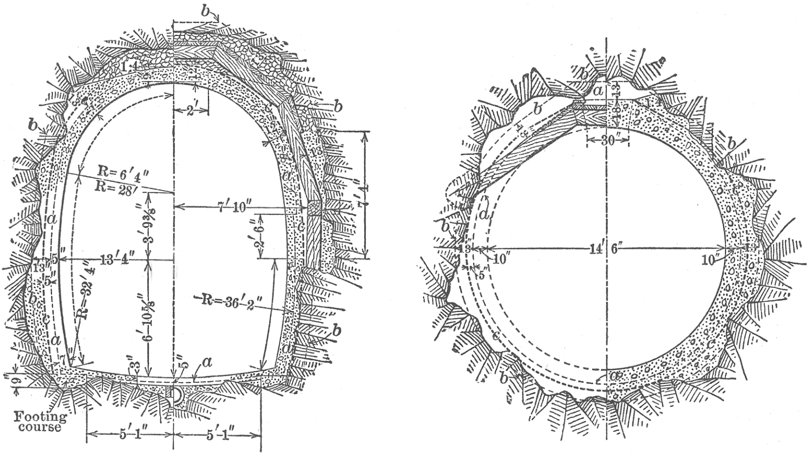

FIGURE 2.—Cross sections of typical tunnels in the Catskill Aqueduct. In the figure *a* is the line within which contractor was not permitted to leave projecting rock, *b* is the ideal average line of excavation, from which excavation, masonry, and packing quantities were computed, and *c* is the line of effective thickness of masonry, and trimming line, except for points on limited areas.

TABLE 2.—*Tunnels of Catskill Aqueduct.*

Name of tunnel.	Length.	Character of rock penetrated.	Started.	Completed.
	Feet.			
Peak	3,470	Hard rock	November, 1908	November, 1909.
Rondout Siphon	23,608	Onondaga limestone, Binnewater sandstone, Hudson River shale, Esopus shale, High Falls shale, Shawangunk grit, Hamilton and Marcellus shale, Helderberg limestone.	March, 1909	May, 1911.
Bonticou	6,823	Hudson River shale, sandstone.	November, 1908	February, 1911.
Wallkill Siphon	23,391	Hudson River shale	May, 1909	December, 1910.
Moodna Siphon	25,200	Hard sandstone, granite, and Hudson River shale.	February, 1909	June, 1911.
Hudson Siphon	3,022	Granite	December, 1910	January, 1912.
Breakneck	1,054	Granite, gneiss	September, 1910	April, 1911.
Bull Hill	5,365	Granite	June, 1909	January, 1911.
Garrison [a]	11,430	Hard gneiss	June, 1907 [b]	9,900 feet, December, 1911.
Hunters Brook [a]	6,150	Hard rock, soft in places; schist.	September, 1909	5,963 feet, December, 1911.
Turkey Mountain [a]	1,400	Manhattan schist	October, 1909	December, 1910.
Croton Lake Siphon [a]	2,639	Manhattan schist, Fordham gneiss.	July, 1910	January, 1912.
Croton [a]	3,000	Manhattan schist	August, 1909	December, 1911.
Chadeayin [a]	700	.....do	November, 1909	September, 1910.
Millwood [a]	4,750	Very hard rock, except 170 feet soft rock and earth; gneiss.	May, 1910	2,908 feet, December, 1911.

[a] Not visited in the field; data obtained from the New York office of the Board of Water Supply.
[b] Suspended November, 1910, to April, 1911.

TABLE 2.—*Tunnels of Catskill Aqueduct*—Continued.

Name of tunnel.	Length.	Character of rock penetrated.	Started.	Completed.
	Feet.			
Sarles [a]	5,230	Gneiss, hard rock, schist	February, 1910	3,558 feet, December, 1911.
Harlem R. R.[a]	1,100	.....do	June, 1910	January, 1911.
Reynolds Hill [a]	3,650	Schist	October, 1910	682 feet, December, 1911.
East View [a]	5,388	.....do	April, 1910	5,252 feet, December, 1911.
Elmsford [a]	2,375	Soft rock, schist	May, 1911	438 feet, December, 1911.
Yonkers Siphon	12,302	Yonkers gneiss	June, 1910	11,923 feet, December, 1911.
Van Cortland Siphon	1,809	.....do	July, 1910	September, 1911.
City	[b] 18.11	Fordham gneiss, Manhattan schist.	December, 1911	

[a] Not visited in the field; data obtained from the New York office of the Board of Water Supply.
[b] Miles.

TUNNELS OF LOS ANGELES AQUEDUCT.

The Los Angeles Aqueduct (fig. 3), in Inyo, Kern, and Los Angeles counties, Cal., and operated by the department of public works, Los Angeles, for water supply, power, and irrigation, includes the following tunnels, which were examined in the field:[a]

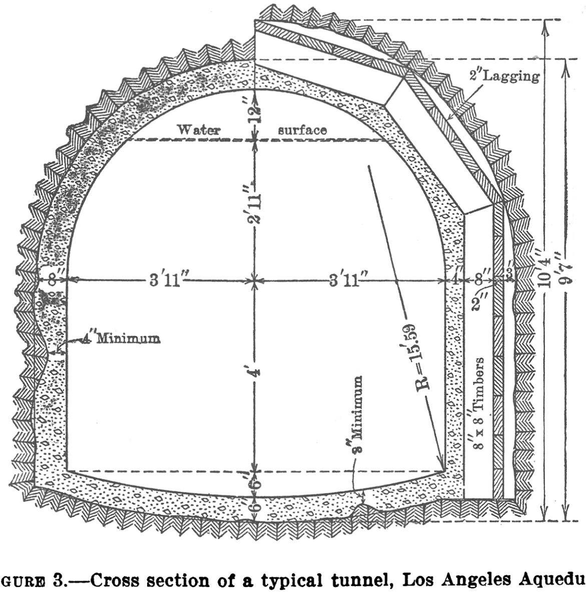

FIGURE 3.—Cross section of a typical tunnel, Los Angeles Aqueduct.

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[a] This aqueduct includes a number of other tunnels that were not visited because they were either completed at the time of field examination or were being driven by hand drilling through soft material.

TABLE 3.—*Tunnels of Los Angeles Aqueduct visited.*

Name of tunnel.	Length.	Character of rock penetrated.	Started.	Completed.
Little Lake division:	*Feet.*			
Tunnel 1B	1,918	Medium hard granite	South heading, June, 1909; north heading, July, 1909.	December, 1909.
Tunnel 2	1,739	Medium hard granite, very wet.	May, 1909	September, 1909.
Tunnel 2A	1,322	Medium hard granite	do	September, 1909.
Tunnel 3	4,044	North heading, medium hard granite; south heading, granite of varying hardness with pockets of CO_2 gas.	South heading, March, 1909; north heading, May, 1909.	July, 1911.
Tunnel 4	2,033	Medium hard to hard granite.	February, 1909	November, 1909.
Tunnel 5	1,178	Medium hard to very hard granite.	do	July, 1909.
Tunnel 6	411	Medium hard granite	do	May, 1909.
Tunnel 7	3,596	Variable, soft and swelling in parts.	North heading, March, 1909; south heading, November, 1909.	July, 1911.
Tunnel 8	2,560	Medium hard to hard, swelling in parts.	North heading, November, 1909; south heading, December, 1909.	August, 1911.
Tunnel 9	3,506	Medium hard to hard	November, 1909	February, 1911.
Tunnel 10	5,755	Medium hard granite	South heading, December, 1909; north heading, January, 1910.	August, 1911.
Tunnel 10A	5,961	Medium hard to hard	Both headings, March, 1910.	December, 1911.
Grape Vine division:				
Tunnel 12	4,900	Hard granite	July, 1909	May, 1911.
Tunnel 13	1,958	do	North heading, July, 1909; south heading, May, 1909.	April, 1910.
Tunnel 14	859	Hard rock	North heading, April, 1909; south heading, May, 1909.	February, 1910.
Tunnel 15	895	do	North heading, May, 1909; south heading, July, 1909.	December, 1909.
Tunnel 16	2,723	do	North heading, July, 1909; south heading, April, 1909.	February, 1910.
Tunnel 17	3.024	Hard granite	March, 1909	November, 1910.
Tunnel 17½	1,364	do	January, 1910	Oct. 31, 1910.
Tunnel 17A	5,330	do	Both headings, January, 1910.	February, 1912.
Tunnel 17B	9,220		Both headings, March, 1910.	(a)
Elizabeth division:				
Elizabeth Lake Tunnel.	26,870		South heading, Oct. 5, 1907; north heading, Nov. 1, 1907.	Feb. 28, 1911.

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[a] Dec. 1, 1911, north heading had been driven 3,800 feet; south heading, 3,634 feet; completed in 1912.

AMERICAN TUNNELS DESCRIBED IN ENGINEERING MAGAZINES.

The following data are comparable with those above, and give practically similar information concerning certain American tunnels that were not examined in the field but are rather fully described in engineering periodicals. Although the information contained in the various accounts is, perhaps, somewhat less complete than similar data obtained at other tunnels actually visited, nevertheless it is generally sufficient for each tunnel to convey a good idea of the main features of the work done.

TABLE 4.—*Comparative data of American tunnels described in engineering magazines.*

Name of tunnel.	Location.	Operator.	Purpose.	Shape of cross section.	Size.	Length.	Character of rock penetrated.
						Feet.	
Buffalo Water Works [a]	Buffalo, N. Y	Department of Public Works, Buffalo.	Water supply	Nearly rectangular.	15 feet wide, 15½ feet high, as excavated.	6,575	Limestone.
Chipeta Adit [b]	Ouray, Colo		Mining	Square	7½ feet by 7½ feet	[c]2,000	Hard.
Cornelius Gap [d]	Near Portland, Oreg	United Railways Co	Electric railway	Arched roof	17½ feet wide, 22½ feet high (above ties).	4,100	Basalt.
Fort William [e]	Fort William, Ontario.	City Water Works	Water supply	do	5 feet wide, 6½ feet high	4,820	Do.
Grand Central Terminal sewer. [f]	Grand Central Station to East River, under Forty-sixth Street, New York City.	New York Central & Hudson River R. R. Co.	Storm-water discharge.	Circular	6 feet diameter inside of lining.	3,000	Hard gneiss.
Joker (drainage) [g]	Red Mountain, Colo	Red Mountain Railroad, Mining & Smelting Co.	Mine drainage and development.	Rectangular	12 feet wide, 11 feet high, as excavated.	5,055	Very heavy, requiring much timber.
Kellogg [h]	Shoshone County, Idaho.	Bunker Hill & Sullivan Mining & Concentrating Co.	Mine development	Arched roof	9 feet wide, 11 feet high	9,000	Quartzite.
Northwest [i]	Chicago, Ill	City of Chicago	Water supply	Horseshoe	Area equivalent to 14-foot circle.	21,180	
Ophelia [j]	Cripple Creek, Colo		Mine drainage and development.	Rectangular	9 feet by 9 feet	8,500	Granite and breccia.
Second Raton Hill [k]	Raton Pass, Colo.-N. Mex.	Atchison, Topeka & Santa Fe Ry.	Railway	Horseshoe	22 feet wide, 29 feet high, as excavated.	2,790	Shale, sandstone, and a 3-foot bed of soft coal.
"Spiral" [l]	Selkirk Mountains, British Columbia.	Canadian Pacific Ry	do	Arched roof	22 feet wide, 27 feet high, as excavated.	([m])	Crystalline limestone

[a] Eng. Rec., June 25, 1910, p. 802.
[b] Min. and Sci. Press, July 11, 1908, p. 60.
[c] Approximate.
[d] Eng. News, June 29, 1911, p. 783.
[e] Eng. and Contrng., May 25, 1910, p. 472.
[f] Eng. Rec., Apr. 11, 1908, p. 496.
[g] Mines and Minerals, May, 1907, p. 470.
[h] Mines and Minerals, October, 1901, p. 122.
[i] Eng. Rec., Aug. 7, 1909, p. 144.
[j] Mine and Quarry, May, 1907, pp. 118–122.
[k] Eng. Rec., Apr. 4, 1903, p. 461.
[l] Eng. News, Nov. 10, 1910, p. 512; Comp. Air Mag., February, 1911, p. 5931.
[m] Tunnel No. 1, 3,200 feet; Tunnel No. 2, 2,890 feet. Both curved; 573 feet radius.

CAUSES AND PREVENTION OF TUNNEL ACCIDENTS.

Data collected by the Bureau of Mines show that an average of nearly 4 men for each 1,000 employed in and about the metal mines of the United States were killed during the year 1911, as compared with 3.8 per 1,000 in coal mining during the same period. Although complete figures for accidents in tunnel driving can not be obtained, a study of such data as could be collected indicates that the number of deaths per year per 1,000 men employed has been somewhat greater than the above figures, the result obtained by averaging data extending over periods of 1 to 10 years for 16 representative tunnels being 4.7 deaths per year per 1,000 men employed. In addition to the men killed outright by accidents in tunnel work, nearly 3 times as many more have been seriously injured or perhaps maimed for life, and almost 13 times as many slightly injured. By far the largest part of these deaths and injuries was caused by falling ore or rock from the roof or walls of the tunnels, but explosives, haulage, electricity, and other causes have each contributed their quota of casualties.

Are these accidents preventable? Not entirely, because some elements of danger are inherent in the work of driving tunnels; such, for example, as the danger from some unforeseen falls of roof,, from the derailment of tunnel cars, or the risk involved in handling even the least dangerous explosives by the most approved methods. But it is equally true that much of the present mortality and injury is the result of ignorance or gross carelessness, and can be avoided. When, for instance, a man sees fit to thaw frozen dynamite in a frying pan or by a candle flame, there is nothing accidental about the explosion that ensues, except, possibly, the fact that a man so ignorant or reckless should have been intrusted with so dangerous a substance. Nor is the responsibility for accidents all on the part of the miner. The manager and his representatives are in many cases either ignorant of the precautions that should be taken for the safety of the men or most negligent in seeing that they are properly and consistently carried out. The following discussion of causes of tunnel accidents is presented in the hope that, by bringing these matters once more squarely to the attention of the men interested, much of the needless death and suffering may be prevented.

CAUSES OF ACCIDENTS.

FALLS OF ROOF.

There are many causes that combine to make falls of rock from the roof by far the greatest source of danger in tunnel work, but perhaps the chief of these is the common practice of greatly overloading the holes with explosives. Extremely heavy charges shatter and crack rock that would ordinarily stand without any danger of falling, and render it extremely dangerous to the men working underneath. Of course it is essential to efficient work in tunnel driving that the blast should completely " break bottom " without any necessity for a second loading and firing; still every foreman and superintendent should see that the smallest amount of dynamite that will do the required work is employed in the holes near the roof. Economy of explosive demands this, all other considerations aside; but the dangers, also, of the heavier charges should be thoroughly appreciated by the superintendent and, when such charges seem imperative, extra vigilance should be exercised and extra precautions taken for the safety of the men.

Another prolific source of accident is the fact that men sometimes return to the tunnel face, after shooting a round, without thoroughly testing the roof just exposed by the blast. It should be the duty of every man employed in the tunnel to examine the roof under which he must work, and especially in that part of the tunnel newly exposed after shooting; the foreman, upon reaching the heading after the blast, should at once detail one or two men (or as many as prove necessary) to clean down thoroughly all the loose pieces of overhead rock. Fortunately, this is done regularly at all well-organized tunnels and it is a practice that can not be too highly recommended for universal adoption.

It must be admitted that from a roof declared by experienced men to be sound, a large block may suddenly and without warning crash into the tunnel. This occurrence will undoubtedly be claimed to have been purely accidental; yet even the danger from such a block (which perhaps was perfectly solid when first exposed, but became loosened by the concussion of subsequent blasting) is, in many cases, overlooked because of the lack of illumination in which all tunnel work must be done, and may be discovered in time if there is a systematic and regular examination of the entire roof of the tunnel. Some one has pointedly observed, " The fall of a slab of rock weighing anything less than 1 ton should at once be charged to carelessness."

It should be said in this connection that the " sound " of the roof is not a proper criterion of its safety, because there are numerous cases on record where the sound of the roof was satisfactory and indicated rock that seemed solid even to experienced men, although a big block or bowlder was actually loose. The better method of testing the roof—one used by many large mining companies and recommended by the Bureau of Mines[a]—is to strike it with a pick or a heavy stick, at the same time touching the doubtful piece with the free hand. If any vibration is felt the rock is unsafe and should

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[a] Rice, G. S., Accidents from falls of roof and coal: Miners' Circular 9, 1912, p. 8.

be taken down or supported at once. If the roof is too high to reach with the hand, a stick should be held against the doubtful piece while it is being struck, and if it is loose the vibration can be felt through the stick.

Prompt and adequate timbering is extremely important. But timbering is a laborious process and it either takes the men of the tunnel crew from their regular work, or it requires extra men. Extra men, however, add to the confusion in the heading and, as their work is done simultaneously with the other work of the tunnel, it seriously hinders either the drillers or the shovelers, or both. So it has become recognized among tunnel men that in most cases timbering seriously impedes the progress of driving, and therefore, although it may be well understood that the roof is dangerous, there is almost always a tendency on the part of those responsible to delay timbering as long as possible. Perhaps the American willingness to "take a chance"—a trait particularly noticeable in the Western States—may be a contributing cause; but the fact remains that the work of timbering is too often delayed until a so-called "accident" brings the necessity forcibly and unavoidably to the front. It is impossible to urge too strongly that all necessary timbering be done promptly, that it can not be done too soon, and that any delay seriously jeopardizes the lives and limbs of the men who have to work under a roof improperly supported.

It is true that in many tunnels the weight of the roof or pressure against the walls has been too great even for the strongest and heaviest timbering, and although such breakage can not always be prevented, it may often be alleviated by means discussed in the section on timbering. The important consideration in these cases as regards safety is the fact that actual failure of the timbers and caving of supported ground rarely comes without warning. Either the timbers will at least be bent appreciably before they break, or, as is usually the case, they will crack and splinter and so give unmistakable warning to the miner that the time is approaching when they will collapse. With such warning any subsequent accident is chargeable to carelessness or negligence in heeding the danger signal. It may be said in this connection that, other things being equal, timber that has a fiber that will split, crack, or splinter out, rather than that which has a fiber that will break off short under a transverse strain, is on this account more desirable for such work.

Falls of rock are also caused by cars becoming derailed and knocking out the supporting timbers under a heavy or loose part of the roof, allowing the roof to fall and kill or injure any men who happen to be underneath. Such accidents are in many cases unavoidable because of the difficulty of preventing derailments. Owing to the lack of illumination, it is usually impossible to see whether the track ahead is clear, and it is therefore necessary to run somewhat blindly and assume that nothing has fallen upon the track since the previous trip; and the mere work of keeping the roadbed of a tunnel track in such shape that its unevenness would no longer cause the cars to jump off would be enormous. The only way, therefore, to lessen these accidents (which fortunately are not so numerous as from other causes) is to keep the track in as good condition as possible, and to use all reasonable watchfulness and caution in tramming, and to avoid in particular running trips at a high speed over bad track.

USE OF EXPLOSIVES.

Next in importance as a cause of injury in tunnel work is the careless, reckless, improper, or ignorant use (or rather misuse) of explosives. Such accidents are of various kinds, the most frequent being those arising from handling, storing, and thawing dynamite, from premature blasts, from misfires, or from poisoning by gases from explosives. The subject of the proper ways to handle, store, and thaw dynamite is treated at some length in the chapter on blasting, but as it is impossible to place too much emphasis upon the necessity for care and caution in the use of explosives, a recital here of the precautions to be taken is well warranted.

PRECAUTIONS AS TO HANDLING.

Don't forget the nature of explosives, but remember that with proper care they can be handled with comparative safety.

Don't smoke while handling explosives and don't handle explosives near an open light.

Don't shoot into explosives with a rifle or pistol, either in or out of a magazine.

Don't attempt to manufacture any kind of an explosive except under the supervision and direction of a trustworthy person who is skilled in the art. Many serious accidents, which have destroyed lives or inflicted injury on persons and property, have been caused by such attempts.

Don't carry blasting caps or electric detonators in the clothing.

Don't tap or otherwise investigate a blasting cap or electric detonator.

Don't attempt to take blasting caps from the box by inserting a wire, nail, or other sharp instrument.

Don't try to withdraw the wires from an electric detonator.

PRECAUTIONS AS TO STORING.

Don't leave explosives in a wet or damp place. They should be kept in a suitable, dry place, under lock and key, and where children or irresponsible persons can not get at them.

Don't store dynamite so that the cartridges are on end, as this position increases the danger of nitroglycerin leaking.

Don't store or handle explosives near a residence.

Don't open packages of explosives in a magazine.

Don't open dynamite boxes with a nail puller, or powder cans with a pickax.

Don't store or transport detonators and explosives together.

Don't keep electric detonators, blasting machines, or blasting caps in a damp place.

Don't allow priming (the placing of a blasting cap or electric detonator in dynamite) to be done in a thawing house or magazine.

PRECAUTIONS AS TO THAWING.

Don't use frozen or chilled explosives. Most dynamite freezes at a temperature between 45° F. and 50° F.

Don't thaw dynamite on heated stoves, rocks, sand, bricks or metal, or in an oven, and don't thaw dynamite in front of, near, or over a steam boiler or fire of any kind.

Don't take dynamite into or near a blacksmith shop or near a forge.

Don't put dynamite on shelves or anything else directly over steam or hot-water pipes, or other heated metal surface.

Don't cut or break a dynamite cartridge while it is frozen, and don't rub a cartridge of dynamite in the hands to complete thawing.

Don't heat a thawing house with pipes containing steam under pressure.

Don't place a "hot-water thawer" over a fire, and never put dynamite directly into hot water or allow it to come in contact with steam.

LOADING PRECAUTIONS.

Don't allow thawed dynamite to remain exposed to low temperature before using it. If it freezes before it is used, it must be thawed again.

Don't fasten a blasting cap to the fuse with the teeth or flatten the cap with a knife; use a cap crimper. The ordinary cap contains enough fulminate of mercury to blow a man's head or hand to pieces.

Don't "lace" fuse through dynamite cartridges. This practice is frequently responsible for the burning of the charge.

Don't explode a charge to chamber a hole and then immediately reload it, as the bore hole will be hot and the second charge may explode prematurely.

Don't force a primer into a bore hole.

Don't do tamping with iron or steel bars or tools. Use only a wooden tamping stick with no metal parts.

Don't handle fuse carelessly in cold weather, for when it is cold it is stiff and breaks easily.

Don't cut the fuse short to save time. Such economy is dangerous.

Don't worry along with old broken leading wire or connecting wire. A new supply will not cost much and will pay for itself many times over.

FIRING PRECAUTIONS.

Don't explode a charge before every one is well beyond the danger line and protected from flying débris. Protect the supply of explosives also from the flying pieces.

Don't hurry in seeking an explanation for the failure of a charge to explode.

Don't drill, bore, or pick out a charge that has failed to explode. Drill and charge another bore hole at least 2 feet from the missed one.

PREMATURE EXPLOSIONS.

It is often difficult to determine just what were the causes of premature explosion, because the persons responsible for the explosion rarely survive to tell the tale and even eyewitnesses are scarce; but carelessness in handling the dynamite in the heading is no doubt the most potent factor. In many cases the so-called accident does not result from the first instance of carelessness or recklessness, but is the disastrous climax of a series of practices that have become habitual; hence persons knowing the common disregard for dynamite on the part of the men who handled it and were killed are able to draw accurate conclusions as to the probable cause of the "accident." As an example might be cited the case of two men who were accustomed to throw sticks of dynamite to each other along the tunnel, over distances of 15 or 20 feet, especially if visitors with "nerves" were present. But even at other times, perhaps because of long familiarity with dynamite and hence a contempt or disregard of its true dangerousness, the sticks were thrown to one another rather than carried the few intervening feet. However, the practice as far as these two personally were concerned, was finally stopped by a disastrous explosion in which they were blown almost to atoms. The subsequent appearance of the tunnel indicated that the explosion was caused by the detonation of a stick falling near the full supply for the entire round.

Another cause of premature explosions is the practice of carrying dynamite to the face of the tunnel in a box or sack and dropping it rather roughly to the ground at the end of the journey. This contempt is also bred, no doubt, by familiarity. It is true that oftentimes gelatin dynamite is not as sensitive to direct shocks as one might imagine, and that many times it will stand very rough usage without detonation; but in other cases, and there are very many of them on record, serious explosions have ensued as a result of inexcusable carelessness in handling. It is neither safe nor advisable to rely in any degree whatsoever upon the "inertness" of dynamite. Nor is it possible to condemn too strongly the practice of carrying detonators or primers (sticks of dynamite containing a detonator and a fuse) in the same bundle with the rest of a supply of explosive for a round. The detonators should always be brought in separately and should under no circumstances be placed in the same box cr even near together in the heading. Many serious accidents have resulted through disregard of this rule.

A certain risk must always attend the loading of a bore hole with dynamite, especially during the insertion of the primer, but much of the danger that often needlessly accompanies this work can be minimized or avoided by proper care. Efficiency of course demands that there shall be no air spaces in the charge of explosive when it is finally ready for detonation; hence the dynamite must be rammed down so that it fills all the unequal spaces in the bore hole; but tamping should always be done by pressure rather than impact. Never use a tamping bar as if it were a javelin. But even in pressing down the charge, great care must be taken that too much force is not employed, especially if a cartridge seems to stick in a hole; for should it become suddenly loosened the miner might not be able to recover himself in time to prevent its being rammed hard against the bottom with disastrous results. Anything more than light pressure should never be given the primer and under no circumstances should it or the succeeding cartridge be struck a blow with the rod.

Irregularity in the rate at which fuse burns is also a cause of premature explosions. Different makes and brands of fuse burn at different rates, and a miner accustomed to a slow-burning fuse will perhaps not realize the necessity of cutting the faster fuse longer, so that he may have time enough to reach a place of safety before the detonation takes place. There are several causes of variations in the burning rate even of the same brand of fuse. For example, experiments conducted by the Bureau of Mines[a] show that mere confinement in a closed vessel is sufficient to cause a fuse to burn three or four times faster than its normal rate. It is true that under ordinary conditions of mining, variations of this magnitude are not apt to be reached, but irregularities of 20 or even 30 per cent are quite possible, and in long bore holes in which a quantity of tamping is used, especially of a type impervious to the escape of the gases (such as closely packed wet clay), the variation may be much greater. Therefore, with such tamping, the rate of burning may be increased to a dangerous extent, unless due allowance be made for the extra speed. But even more important is the effect produced by mechanical injury,

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[a] Snelling, W. O., and Cope, W. C., The rate of burning of fuse as influenced by temperature and pressure: Technical Paper 6, Bureau of Mines, 1912, 28 pp.

which is more apt to be a common occurrence. Mere bending of fuse (if it is in proper condition for use), such as might result from coiling it near the collar of the hole to prevent its being struck by flying rock from other blasts, or even placing it with some force within the hole, has little if any effect upon the rate of burning; but abrasion, blows, or too great pressure produces serious variations in this rate and in some cases may even cause fuse to burn almost instantaneously. It is therefore essential that none but fuse in good condition ever be brought into the heading, and that care be taken while it is there to see that it is not injured by rocks or tools falling on it, and that it is not abraded or otherwise injured with the tamping bar while the hole is being loaded.

Mention should be made of the seemingly obvious danger of reloading a bore hole before it has had time to cool off sufficiently from a previous blast. In tunnel work this danger occurs in connection with the "guns"—the ends of holes that have not broken to the bottom with the first explosion.

MISFIRES.

Many deaths and injuries are caused by the subsequent detonation of a charge of dynamite that failed to explode at the proper time. Such misfires do not, however, cause accidents unless the charge is detonated unexpectedly. Sometimes this happens by drilling into it during preparations for the next round, or by striking it in the muck pile, where it has been thrown by the blast from a neighboring hole, or perhaps by the sudden explosion of a delayed shot from a fuse that has long been smouldering.

Many misfires can be traced directly to some injury to the fuse. The insertion of the primer into the hole, fuse end first, often causes fuse to crack at the sharp bend thus made; the danger of such cracking is especially great when the fuse is cold or the hole is full of cold water. Sudden and rough uncoiling of the fuse in cold weather will usually cause it to break. Obviously, therefore, cold fuse should not be bent, twisted, or roughly handled. It is claimed by some persons that misfires are caused through fuse being cut off ahead of the fire by the explosion of a neighboring hole, so that the charge fails to explode. There is some question whether this really happens or not; but, if it does, it is a pretty strong argument that the hole was misplaced, for if a hole is properly placed, only in rare instances, if ever, will enough of it be shot away to cut off the fuse ahead of the fire. It is also claimed, and with somewhat more reason, that the fuse is apt to be torn out by flying pieces of rock from the explosion of other holes, but this result can be largely obviated if the fuse is properly coiled close to the mouth of the hole before it is "spit."

The failure of a fuse properly to ignite a detonator is often the result of improper storage. When the asphalt waterproofing composition used in some fuses gets too hot it becomes viscid and agglomerates the powder grains in the core of the fuse and thus delays, and in some cases actually prevents, the fuse from burning. Experiments conducted by the Bureau of Mines[a] indicate that prolonged exposure at a temperature of 60° C. is sufficient to cause a marked retardation in the rate of burning of fuse. It follows, therefore, that fuse should not be stored near boilers, steam pipes, or other sources of heat, where the temperature is apt to be high. Cold is likewise deleterious, for it renders the asphalt composition brittle and liable to crack, and these cracks either decrease the rate of burning by permitting the gas from the powder core to escape more readily than usual, or, if they are large enough, they may stop the travel of the fire entirely. The fuse should be carefully protected from moisture during storage for, with waterproof fuse of the type almost universally employed in tunneling, if the dampness once gets into the powder train its removal is difficult. As the fuse burns, the moisture is driven ahead of the fire in the form of steam and even if it does not accumulate in sufficient quantity to quench the fire in the fuse, enough of it may be driven into the detonator to prevent ignition and thus cause a misfire.

Many misfires originate from improperly prepared primers. Before the fuse is inserted into the detonator, an inch or two should be cut off and thrown away, for gunpowder (which forms the core of the fuse) is somewhat hygroscopic, and the end of the fuse may have gathered moisture enough to quench the burning powder or prevent the ignition of the cap. This cut should be made with a sharp-cutting tool, squarely across the fuse, for if made diagonally the point may curl over the end of the fuse when inserted in the detonator and thus prevent the spit of the powder train from reaching the detonating composition in the cap. Care should also be taken that the powder grains in the end of the fuse do not leak out after the fuse has been cut, for this would tend to weaken the force of the spit into the detonator and might prevent its ignition. The open end of the cap should be carefully crimped around the fuse with a proper crimping tool, so that it will be tight enough to hold the detonator and the fuse together and keep out moisture, but the crimping should not be tight enough to cut off the powder train in the fuse. This is particularly liable to happen with a narrow crimping tool that presses a narrow groove in the detonator and the underlying fuse. There are tools on the market that have a crimping face of at

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[a] Snelling, W. O., and Cope, W. C., The rate of burning of fuse as influenced by temperature and pressure, Technical Paper 6: Bureau of Mines, 1912, p. 19.

least a quarter of an inch, and the extra price of these tools would be no more than the cost of the explosive wasted by a single misfire—to say nothing of the loss of life that might arise therefrom. It is, of course, obvious that the teeth or a knife should never be used for crimping, for, as previously stated, there is enough explosive in an ordinary detonator to blow a man's head or hand to pieces. After it is crimped, the detonator should be buried in the end of the stick of dynamite, with its axis parallel to that of the stick, and the top of the detonator should be flush with the top of the dynamite. For if the cap is buried deeper, the explosive is liable to become ignited from the side spitting of the fuse before it is properly exploded by the detonator, a result that not only destroys the efficiency of the explosive, but causes a larger amount of gases, especially those most dangerous to the men who must breathe them. It is also important to use a detonator of sufficient strength. Although 3X blasting caps were considered strong enough for "straight" nitroglycerin dynamite, the less sensitive gelatin dynamite requires a much stronger detonator to explode it properly. For this reason nothing weaker than 5X caps should ever be used with gelatin dynamite, and the universal experience is that better results have been obtained when a change has been made to even stronger detonators. These insure the complete detonation of the explosive and thus produce only a minimum amount of dangerous gases.

It is very difficult to count the explosions during blasting and be sure that the charges have all been detonated, so it is not always possible to determine whether there has been a misfire. For this reason the face, or as much of it as is not covered by the débris resulting from the blast, should be inspected for evidences of missed holes, and it should be carefully watched during the removal of the muck. If a missed hole is discovered, under no circumstances should an attempt be made to pick out the material. If no tamping has been used, a stick of dynamite containing a detonator should be inserted in the hole and exploded. If tamping has been employed, another hole should be drilled and blasted at least 2 feet from the missed one. In picking down the muck pile the pick should be handled as if it were a hoe and not like a sledge hammer; that is, the material should be pulled or scraped down and never struck violently with the point of the pick. In this way, should there happen to be a piece of unexploded dynamite in the débris, there is much less danger of its exploding. The importance of this precaution can not be too strongly emphasized. Should a piece of dynamite be discovered in the muck, it should be removed carefully and handed to the foreman who should at once take it to a safe place, and extreme care should be used if a piece of fuse accompanies it or is discovered near it, for this would indicate that an unexploded detonator may possibly still be inside of the stick of dynamite, the danger of which is obvious. Under no circumstances should a new hole be started in the remnants of a hole that has ever held dynamite; for although the inference is always, of course, that the dynamite has been detonated, still there remains a chance that detonation has not occurred—a chance not as slight as ordinarily might be supposed, to judge from the number of accidents traceable to this source. And even if a rod be used to test the hole, it might encounter a small obstruction and by thus seeming to show the bottom of the hole fail to reveal the dynamite beneath.

GASES FROM EXPLOSIVES.

Poisoning from the gases produced by explosives is common in tunnel work. The ailment is familiar to most miners; in its mild form it is usually called "powder headache" and produces little more than temporary inconvenience, but in severe cases it has been known to produce death within a very short time. In the section on blasting it is explained that the harmful gases resulting from the complete detonation of dynamite under normal conditions are usually carbon dioxide and carbon monoxide; that although carbon dioxide will not support respiration, and when present in sufficient quantities may cause unconsciousness and even death, it has no very injurious effects when sufficiently diluted; that carbon monoxide is exceedingly dangerous and even small amounts of it may prove fatal if breathed for a sufficient length of time. This gas probably causes the familiar symptoms after a dose of "powder smoke." By reference to the table on page 153, it will be seen that gelatin dynamite, the explosive almost universally used in tunnel work, under proper conditions generates comparatively little of the more dangerous gas. Experiments conducted by the Bureau of Mines[a] indicate that even this can be obviated by a slight modification in the chemical composition of the gelatin dynamite. But when even such a dynamite is not completely detonated (either through the use of too weak a detonator or any other cause), and especially when it burns rather than explodes, a much greater volume of monoxide is formed, and in addition there are a number of other harmful gases developed, including the dangerous peroxide of nitrogen. It is therefore essential that the detonators employed be strong enough to explode the dynamite completely, and that every precaution be taken to prevent the dynamite from taking fire through the side spitting of the fuse or in any other manner.

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[a] Hall, Clarence, and Howell, S. P., The selection of explosives used in engineering and mining operations: Bull. 48, Bureau of Mines, 1913, 50 pp., 3 pls., 7 figs.

The deadliness of the gases resulting from explosives improperly detonated may be illustrated by describing an accident that is known to have cost 9 lives. A study of the attendant circumstances, as described to the writers, indicates that the explosive, or at least a large part of it, must have burned rather than detonated. Gelatin dynamite was employed and the charge was even smaller than previous blasts of which the men had inhaled the fumes without serious effects, but in this case the fumes are described by the men as being brownish yellow rather than the usual grayish or bluish white. After igniting the blast the men retired about 500 feet to wait for the smoke to clear, and while they were waiting the smoke drifted slowly over them and then, owing to some change in the current, drifted slowly back again. The men soon felt the usual symptoms of carbon monoxide poisoning—slight choking, nausea, profuse perspiration, and headache—but they all revived upon reaching the open air about an hour and a half after the blast was fired. Within a short time, however (and in one case before the man could walk to the bunk house), the men began to cough up bloody mucus and to exhibit other symptoms of nitrogen peroxide poisoning, and in less than three days 9 of the 13 men who had been in the tunnel and exposed to the fumes had died. The 4 who escaped were either not exposed to the gas for the full time, or else found some other source of air supply which served partly to dilute the gases; but some of these men as well as those who went in with the motor to bring the men out were ill for days and even months after the catastrophe.[a]

It is the opinion of physicians who have studied the matter that many swift deaths among miners, formerly diagnosed as pneumonia, may really have been caused by the inhalation of gases from burning dynamite.

GASES FROM OTHER SOURCES.

Although any carbon monoxide encountered in tunnel work is liable to be a result of the use of the dynamite, there have been cases where this dangerous gas has been generated by the combustion of oil and grease in the air receiver and transmitted to the heading by the compressed-air pipe. The causes of such combustion are fully discussed in the section on air compressors, but mention is here made that the ignition of accumulated oil and grease is generally due to faulty valves in the compressor. These permit warm compressed air to leak back into the cylinder; this air upon being recompressed becomes still hotter, so that after a time the temperature of

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[a] A full discussion of the customary symptoms that accompany poisoning from nitrogen peroxide and carbon monoxide and their comparison with the symptoms exhibited by the men is contained in the April, 1911, number of Colorado Medicine: "An Unusual Powder-Smoke Fatality," by Carl Johnson.

the air in the receiver may be far higher than the ignition point of the lubricant employed. If an explosion does not then ensue, the oil on the sides and bottom of the receiver will burn and produce carbon dioxide or carbon monoxide, either of which jeopardizes the safety of the miner in the heading. It is therefore necessary to inspect the valves of the compressor regularly; moreover, dependence should never be placed on the compressed-air line for tunnel ventilation.

There are several tunnels in which bodies of gas have been encountered, the gases most frequently found being carbon dioxide and hydrocarbon gases. The former is, of course, chiefly dangerous because of the possibility of the men being suffocated, but this can be largely obviated by proper ventilation. In one of the tunnels of the Los Angeles aqueduct, flows of carbon dioxide were encountered in a series of crevices across a zone about 150 feet wide. In order to make it possible for the men to work in the tunnel this zone, including 300 feet on either side, was tightly sealed with concrete; in addition it was found necessary to leave in the center of the gas zone back of the concrete an annular space to which an exhaust "blower" was connected that constantly drew off the gas during the driving of the tunnel, while an additional blower forced fresh air in to the men. If either of these machines stopped the men had to get out of the tunnel as fast as possible, but as long as the machines kept running the air was sufficiently pure.

The chief danger from hydrocarbon gases lies in their explosibility, but they are so commonly encountered in coal mining that precautions to be taken in their presence are fairly well known. However, a rather unique although highly dangerous method of dealing with them was employed in one of the tunnels examined by the writers, and is well worth describing.

The gas was encountered in a zone approximately 2,300 feet in extent, through about 500 feet of which oil could be distilled from the rocks, although there was no seepage. The gas was highly explosive, and had an odor of kerosene or gasoline rather than of crude petroleum. The largest quantities of it came into the tunnel immediately after blasting, and the maximum accumulation was approximately 30,000 cubic feet. There did not appear to have been any particular seepages in the gaseous zone, but rather there was always an unknown quantity ahead of the work. As the gas was highly explosive extra precautions had to be taken for the safety of the men at work. The mere requirement of safety lamps in the tunnel was not considered sufficient, because the very nature of the rock was such as to cause dangerous sparks from a pick or from the starting of a drill hole, which it was thought would be sufficient to ignite the gas and produce an explosion. The expedient adopted was to explode the accumulation after each blast and to burn any new gas as fast as it appeared in the tunnel during the remainder of the work.

For this purpose the tunnel was wired from the portal to the heading with a 550-volt circuit, into which there were introduced at intervals of about 200 feet throughout the entire gas-bearing section a number of arcing devices. Any ordinary street arc lamp could have been adapted for this work, provided that the carbons were not exposed for more than 2 inches; otherwise the concussion from ordinary blasting, as well as from the gas explosions, would have broken them. The use of one soft and one hard carbon was found to give the best results. The system was operated as follows:

Immediately after blasting, a fire boss and his helper took charge of the tunnel. After waiting 30 minutes after the blast had been fired they turned a current of electricity through the arc line by means of a switch at the portal. The arcs were purposely placed in series in order to make certain that if any one of them burned they would all burn; an ammeter was placed at the control switch to show whether they had lighted. If the arcs did light, an explosion generally ensued, sometimes a severe one. But whether or not there was an explosion the switch was always opened for 15 minutes and then closed a second time as an edded precaution, although a second explosion never resulted. When the line was dead once more two men carrying safety lamps proceeded to a protected station approximately halfway to the heading, where they again sent a current through the arcs. A few explosions resulted from this practice, but they were unusual rather than customary. After having made this test the fire boss and his helper proceeded to the heading, testing the entire tunnel for gas by means of the safety lamps they carried. They would ordinarily find in the heading an accumulation of gas extending back a distance of 125 to 150 feet, because the nearest arc could not be placed much nearer to the heading than 150 feet on account of the danger of the carbons being broken by the concussion from the blasting. The fire boss would then take an arc kept 150 feet from the face and attached to the circuit by an armored cable and place it over the muck pile; the two men would return again to the midway station and once more close the circuit and ignite the remaining gas. Then, and then only, with all the arcs burning, they would return to the heading and place torches as near the roof as possible at intervals of about 150 feet throughout the gaseous section. The torches were lighted from the arcs, and the men were not permitted to light them in any other way, or, indeed, to carry into the tunnel any other means of lighting them. By this time all the seepages that were strong enough to support a steady flame would have been lighted and would be burning, and the gas that came from pockets that could not sustain a flame would be ignited by the torches before it could accumulate in any quantity.

The fire crew then returned to the mid-station, where they extinguished a red light and lighted a white one, indicating that the tunnel was safe for the incoming crew, for no one but these two men were allowed in the tunnel beyond this point unless the red light was out and a particular white one burning, in order to obviate danger through any accidental extinguishing of the red light without the knowledge of the fire crew and before the tunnel was safe. The fire crew was allowed four hours for this work, although ordinarily that length of time was not required.

The working crew upon reaching the heading ordinarily found the muck pile too hot to be handled, if, indeed, it was not actually in flames, for it burned usually for one-half to two hours after each blast, and once at least it burned for 14 hours. After it had been cooled sufficiently by streams of both air and water, the machines were set up and the round of holes drilled in the regular manner. Any gas that developed during the drilling of a hole was lighted as soon as the hole had been completed, and if sufficiently strong to support a flame it would burn until the end of the shift. At one time as many as 6 out of 8 holes on the top round were burning like blowtorches, giving flames 6 to 18 inches in length. When the round had been finished the holes had to be cooled before loading. This was accomplished by turning water and air lines through ordinary blowpipes, both into the holes and over the face of the tunnel. The flames were, of course, extinguished by this process, and as soon as the gas had accumulated in the tunnel sufficiently to become apparent in a safety lamp placed near the roof about 30 feet from the heading, it was ignited by a torch and the resulting flames were at once put out again by air and water. This process was continued until the holes were cool, when they were at once loaded as rapidly as possible and fired, the fuses being always lighted from near the bottom of the tunnel.

Although the fact that there were no accidents in driving through the gas-bearing zone after the installation of the "safety arcs" shows that this system was efficacious in this particular instance, it is not one that can be recommended unqualifiedly for general use. In the opinion of engineers who have made a special study of the question of safety in mining, the use of anything but safety lamps or their equivalent in mines or tunnels where explosive gases are known to exist is never without risk, whereas the practice of burning the gases as fast as they make their appearance is in itself extremely hazardous. Indeed, the fact that no disastrous explosion occurred under this system seemed to them remarkable. Moreover, it is obvious that long delays were necessary before the men could start to work, and even after they had reached the heading the heat must have greatly decreased their possible efficiency. A less dangerous method of handling the gas, and one that would probably prove more economical in the end, would be the installation of a ventilating system large enough to dilute to harmlessness several times the amount of gases ordinarily encountered. Safety lamps only should be allowed in the tunnel and all blasts should be fired by electricity.

HAULAGE.

A large proportion of the injuries attributed to tramming is caused by the practice of riding on the cars, especially loaded ones. When riding on the top of a full trip, a man is always in danger of a serious injury at every low place in the roof, and if he is riding between the cars (or any place but the rear end), he is liable to be jarred from his foothold and dragged under the cars, and in case of derailment he has little chance of escape. The risk of derailment is unavoidable in tunnel work, partly because of the insufficient illumination under which tramming is generally carried on, and partly because of the difficulty of keeping the roadbed in good condition or the track clear of small obstructions. Even when riding on empty cars there is serious risk whenever the miner sits on the ends or sides and allows his feet to hang over; the safest way is to sit inside of the car and to crouch low enough to avoid being struck by any jutting place in the roof. The arms and hands should be kept inside of the car to avoid the possibility of being caught between the car and the wall at a tight place. The driver or "mule skinner" is usually compelled to ride on a loaded trip and sometimes at the front end of the train in order to be near the animal he is driving; the extra hazard of this position should be fully realized and extra precautions taken. The dangerous practice observed on the part of some drivers, of riding with one foot on the bumper and the other on the chain by which the mule pulls the trip, is very obvious and can not be too strongly condemned. This act should be made sufficient cause for instant dismissal. It ought not to be necessary to mention the danger of attempting to jump on or off a moving trip of cars, because the chances in such a case of a man missing his footing and being caught or dragged under the cars, or of breaking an ankle or leg in the uncertain light, should be so clearly seen that no one ought to consider the risk worth taking; but the number of injuries arising from this cause shows only too well that this precaution is habitually disregarded.

Great care is necessary during the operation of placing a derailed car back upon the track. It is very easy for a miner to strain or otherwise injure himself if he attempts to do this without getting some one to assist him. Also in handling a derailed car that is full of rock there is danger of the block or crowbar slipping and allowing the car to drop suddenly on the miner's foot or hand, if indeed it does not topple over completely and crush him against the side of the tunnel.

Failure to allow sufficient room to a passing trip of cars is also a frequent source of injury. Before going into a strange tunnel the miner, if he is not accompanied by some one familiar with the tunnel, should always ascertain upon which side of the track there is the most room, and in meeting a passing trip should always give the animal pulling it all the space possible, so as to avoid being tramped on or kicked, or being caught between the cars and the walls of the tunnel. It is also advisable to hide any light when meeting a horse or mule, for there are some animals that are especially afraid of the high-powered acetylene lamps that are coming to be used almost entirely in tunnel work. If the animal balks when coming toward a light a serious mixup may occur, as the cars behind can not always be stopped at once. In a tunnel, as on the surface, attention should always be given the heels of animals whether moving or at rest, and it is best to speak to animals when approaching them from behind, for many serious injuries have been caused by passing too close to nervous animals without warning. When turning a horse or mule around in a heading, the driver should watch carefully to see that he is not stepped on; inane as this advice sounds, many really serious accidents have resulted from just this simple cause.

ELECTRICITY.

An examination of reports of electrical accidents in tunnel work shows that in most cases the shocks were caused by the trolley wire. This is not surprising when one considers the many factors that unite to make an electrically charged wire especially dangerous underground. The earth is almost always used to complete the return circuit and, therefore, if the miner inadvertently touches any part of an electrical apparatus that is charged with current, and if he is not well insulated from the ground, he will certainly get a shock, the intensity of which will depend on the voltage or pressure of the electric current and the incompleteness of his insulation from the earth. Some trolley wires carrying a current as high as 600 volts have no insulating or protecting covering whatsoever and most of them are without a guard or shield of any sort, although they are sometimes placed less than a man's height from the floor and directly over the rail. Then, too, tunnels are generally damp or wet, so that a man is rarely well insulated from the ground. As the light at best is poor, one can not always see the wire as he approaches it, and the space is so restricted that a man walking in the tunnel must keep his head close to the wire when at the same time the most of his attention must needs be given to his footing. Moreover, in climbing into or riding in the cars, most of which in tunnel work are of metal and furnish excellent electrical connection with the rails, one's head must pass close to the live wire. The carrying of metal tools, such as crowbars or drill steel, also picks and shovels with wet wooden handles, is also the cause of many shocks through their accidental contact with the trolley, especially if such tools are carried on the shoulder. It is therefore important, when walking in a tunnel where a trolley wire is installed, constantly to bear its existence in mind and take every precaution to avoid contact with it either by hand, wet clothing, or tools.

In addition to the trolley wire there are in tunnel work other sources from which electrical shocks may be received. Wherever the heading is illuminated by electricity, the lights are usually grouped in a cluster and connected to the main circuit by means of a flexible cable, so that they can be removed easily to prevent breakage during blasting. The wires of the cable are, of course, insulated, but owing to rough usage the insulation is often damaged or scraped off, leaving the bare wire exposed. Even a slight damage of the insulation is often sufficient to permit a considerable leakage of current from which a person handling the cable may receive a severe shock. Such wires are the more dangerous because, supposing them to be protected, one is more apt to handle them carelessly. The men who remove these wires preparatory to blasting and afterwards replace them or otherwise adjust them should examine them closely and not touch any place where the insulation has become damaged.

Shocks are also caused by motors, transformers, or other pieces of electrical equipment that are supposed to be safe but have accidentally become charged, and by switches and other similar devices during adjustment or repair. In handling apparatus of this sort a workman should carefully insulate or otherwise protect himself from the current and should try to handle the apparatus in such a manner that any involuntary muscular reaction from a shock will throw him clear of its live parts, rather than bring him more closely in contact with them. Although electric locomotives are usually in such perfect contact with the rails that a person touching any charged part of the frame will rarely receive a shock, there are times (as, for example, when there is a considerable amount of dirt or sand on the rails) when the locomotive is almost completely insulated from them; in such a case anyone coming in contact with a live part of the frame or of the drawbar, or even with one of the cars coupled to the locomotive, may receive a severe shock, which is apt to be all the more serious because it is unexpected. For this reason the touching of such equipment should be avoided unless actually necessary.

Mention should be made here of the immediate steps to be taken in case a man has received a severe electric shock and is perhaps lying unconscious and seemingly dead, for it is often possible by prompt treatment to revive and restore a man in this condition who might otherwise fail to recover consciousness. Methods recommended by the bureau are described in Miners' Circular 5.[a]

FIRE.

The chief danger from fire to the men in a tunnel is the possibility of the buildings at the surface becoming ignited. These structures are, of course, subject to the same causes of fire as ordinary buildings, such as the careless handling of matches or lights, spontaneous combustion of oily waste wherever it is allowed to accumulate, or the short-circuiting of electric wires, not to mention the risk of forest fires in heavily timbered regions. At a large majority of tunnels now being driven the blacksmith shop, the storeroom, the boiler house, and other buildings are situated much closer than the 200 feet that should separate them from the tunnel portal, and in many districts, especially where the winter snowfall is heavy, they are directly connected with the tunnel by snowsheds constructed of wood. At such tunnels, also, means of exit other than the portal are seldom provided, so that in case of fire in these buildings men are penned up in the tunnel and, in the customary absence of a fire door, are in serious danger of suffocation from the gases and smoke produced by the conflagration. It is therefore essential, and in some States it is required by law, that in all tunnels where combustible structures must be erected nearer to the portal than 200 feet there should be a separate exit at least 200 feet away, and that a fireproof door that can be closed from a distance should also be provided. A sufficient water supply should always be maintained to put out a fire, and hydrants with a coiled 1½-inch hose and a nozzle should be placed not less than 40 feet and not more than 100 feet from each building or group of buildings.

Most tunnels, except where timbered, are practically fireproof, and hence underground fires are not common in tunnel work. It is nevertheless important to guard against the dangers of underground fire. Whenever such fires do occur they usually start in some small way, either from candles or lamps being placed too near the posts or caps of a timber set, or from a match or the coals from a pipe thrown into a pile of rubbish, hay or other combustible material that may in turn ignite the timbering. Although such fires can usually be extinguished before any great damage has

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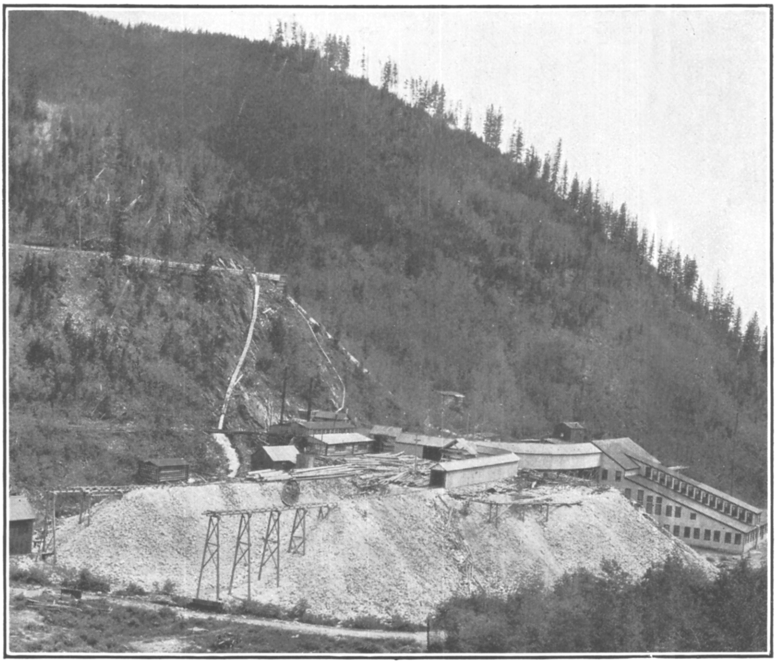
[a] Clark, H. H., Roberts, W. D., Illsley, L. C., and Randolph, H. F., Electrical accidents in mines; their causes and prevention, 1911, 10 pp., 3 pls.

resulted, provided their presence is discovered soon enough and there are means at hand to extinguish them, it is much better to prevent the ignition by obviating causes. Therefore, combustible rubbish should not be allowed to accumulate in the tunnel, and any supply of hay for the use of the mules or horses underground should be carefully stored in a shed provided for that purpose, and open lights or smoking should not be permitted in its neighborhood. Candles or torches should never be left burning near timbers, and the practice of wedging a lighted candle between two nails driven into a post should be sufficient cause for the instant dismissal of the guilty person.

WATER.

Water under pressure is another source of danger in tunnel work. Men may be hurt in jumping back to avoid the rocks and other débris often carried with it, or perhaps buried under an accompanying rush of mud and sand. A good example of this may be found in the records of a foreign railway tunnel, where a cleft filled with water, sand, and gravel was encountered and the ensuing sudden and violent inburst filled up more than a mile of the tunnel in a very few minutes, burying 25 workmen beyond all hope of recovery. A somewhat similar occurrence in one of our American tunnels was likewise due to water. The tunnel caved in at a point about 4,000 feet from the heading, but the men working there were warned in time to escape, although they had barely reached safety before the tunnel became entirely closed. When this happened the mass of muck, composed chiefly of soft clay and running shale impervious to water, formed a dam that cut off from the main part of the tunnel the flow of approximately 2,700 gallons per minute. As soon as the part of the tunnel between the cave and the heading became filled with water, the full pressure of the head in the mountain over the tunnel was exerted against the dam, forcing it down the tunnel until the pressure was relieved. The additional length of the débris then offered greater resistance, and it remained stationary until the pressure had again accumulated enough to move it. This process was repeated until 440 feet of tunnel had been filled. Several attempts were made at first to relieve the pressure by inserting a section of ventilating pipe at the top of the dam; but after several men had narrowly escaped being buried by the rush of mud as the dam moved forward this scheme was abandoned and the tunnel was sealed up by a concrete bulkhead, the men being protected by a temporary bulkhead of wood during the construction of the permanent one.

In driving through limestone and dolomite it is not unusual for a tunnel heading to tap immense caves filled with water, mud, and sand. The volume of the fluid mass flowing into the tunnel is deter-



A. END OF FLUME, MILL, DUMP, AND OTHER SURFACE FEATURES AT THE CARTER TUNNEL.

B. POWER HOUSE, PORTAL, AND PART OF FLUME, NISQUALLY TUNNEL, ALDER, WASH.

mined by the size of the opening, and its velocity is proportionate to the head. Under the pressure of a head of 300 or 400 feet the cutting action of the rock particles and sand carried by the water soon enlarges even a drill hole to a size that permits the filling up of the heading in an incredibly short time. When a round of shots breaks into a cave of this kind, the heading and perhaps the completed tunnel for a distance of hundreds and even thousands of feet back from the face may be filled so fast that the escape of the workmen is impossible if they are at the face. Fortunately, however, during shot firing, the time of the greatest danger, the men are always out of the heading.

When an underground cave or reservoir filled with water, mud, sand, and loose rock is tapped in a tunnel heading, one of two things occurs; generally the cave or reservoir empties itself completely into the tunnel and, after the flow is over, the solid matter that the flood leaves behind can easily be shoveled up and hauled out. Sometimes, however, the volume of solids is so great that the tunnel is completely choked before the reservoir is emptied. In these cases, when the flow of water ceases, the men are usually set to work cleaning up the material with which the tunnel has been filled, but when this cleaning process advances sufficiently to weaken the dam that holds back the flood a new outburst occurs and, because the passageways have already been opened, the second outbreak is often more violent and dangerous than the first. If this operation were repeated often enough, the cave or reservoir would, of course, be drained and the heading be regained, but in many instances the operation of attempting to regain the heading has been found so dangerous that it has been abandoned and a curved tunnel has been bored to pass around the danger point.

In the dolomite in the Cowenhoven Tunnel, at Aspen, caves of this kind filled with water and dolomite sand were frequently encountered. It was no uncommon thing after a round of shots to have the tunnel completely filled for hundreds of feet back from the face. As soon as the water from the cave that had been tapped drained off, the mud and sand were easily loaded and work in the face was resumed.

An immense cave was tapped by a drill hole in a long crosscut that was being driven from the tunnel to the Della S. mine. The drill hole, under the pressure and cutting action already described, enlarged so rapidly that the men fled from the face, and a few seconds after the opening must have enlarged to a size that permitted the filling of the tunnel with such rapidity that the tunnel cars were hurled back and flattened against the posts. Several unsuccessful attempts were made to regain the face, which finally had to be bulkheaded and the tunnel run around it, as at the Simplon Tunnel in Switzerland.

Numerous caves were encountered in the 1,200-foot level of the Free Silver mine, which was also run through dolomite; but, fortunately, although they must have extended to great heights, their horizontal cross section was very much less than that of the caves 1,200 feet above. When these reservoirs were tapped with a drill hole the water would spout out with such velocity that it was impossible to stay in the face, and in a short time the opening would be worn to a size which sometimes increased the amount of water to be handled by the pumps to 3,000 and even 4,000 gallons per minute. At first the noise from the inrushing volume of water was exceedingly terrifying to the men, but in a short time whenever a cave of this kind was tapped the men simply joined hands to assist each other in maintaining their footing and waded back with the torrent as they would do in crossing an extremely rapid stream. Many narrow escapes occurred, but, due to the precautions taken by the management and workmen, no serious accidents occurred during any of these inrushes.

INTOXICATION.

Although few accidents in tunnel work are traced directly to intoxication of employees, the extent to which it contributes to many mishaps that are attributed to other causes is perhaps too little appreciated. The fact that a man who has put an " enemy into his mouth to steal away his brains " is then much more liable to be careless or negligent of his own safety and the lives of the men around him is so true as to be almost axiomatic. Even a slight amount of intoxication, which might be allowable if the work was to be done on the surface, is dangerous under ground, where it is very apt to be aggravated greatly either by the lack of fresh air or by the heat, neither of which is unusual in tunnel headings. Therefore it is essential that a man in such a condition should not be permitted under ground and, if discovered there, should immediately be sent out of the tunnel by the foreman. Repeated offenses should result automatically in dismissal.

PREVENTION OF ACCIDENTS.

In discussing the prevention of accidents in tunnel work, little is to be gained by arguing whether the manager, the foreman, or the miner is solely to blame for their occurrence. The greater responsibility lying, as ever, with those who have the broader vision, the manager or the superintendent is in duty bound to see that the place where the men are to work shall be made as safe as possible and to insist that they themselves exercise the greatest care and caution in conducting their work. Upon the foreman falls the responsibility of carrying out the manager's orders, of seeing that the men are instructed in the proper precautions to be taken, and that they are constantly and consistently exercised, and, if necessary, of discharging either temporarily or permanently any man who willfully or habitually disregards them. As for the miner, whose business is shown by statistics to be a hazardous one at best, it is only through the most extreme care on the part of each man, not only for his own welfare but for the safety of his coworkers, that he can hope to escape from the dangers that surround him. Each one, therefore, has his share of the responsibility, and it is only by cooperation between all parties concerned that any progress can be made toward the prevention and reduction of the fatalities and the injuries now encountered in tunnel driving. As it is impossible to reiterate too often the methods of obviating accidents, the following paragraphs are addressed directly to the parties most concerned, in the hope of bringing home to them once more some of the more important preventive measures.

PRECAUTIONS FOR THE MANAGER OR SUPERINTENDENT.

Insist that necessary timbering be done promptly, and always keep an adequate supply of lumber at hand so that no delay may ensue from the lack of it. See that the minimum quantity of explosive is used (in order to prevent unnecessary shattering of roof and walls) and inaugurate a systematic and regular examination of the roof to insure the timbering of loose pieces at once. Have all bent or breaking posts or caps promptly replaced by new ones.

Provide suitable magazines and thaw houses for explosives.[a]

Do not permit any disregard of the proper precautions as to handling, storing, and using explosives, and see that each man is provided with a copy of such precautions. Do not permit the transportation of detonators or primers to the heading in the same bundle with the remaining supply of explosive for the blast. Have careful tests of the burning rate of the fuse made periodically, especially when a different brand of fuse is purchased, and warn the men of any discovered irregularity. Destroy any damaged fuse at once. Do not store fuse near any source of heat. Prohibit the reloading of a bore hole before it has had time to cool from a previous blast. Give the man who makes the primers the necessary equipment and tools and have him carefully taught how to prepare and waterproof the primers.

Do not purchase caps weaker than 5X for use with gelatin dynamite. See that the proper precautions are taken whenever a missed hole or evidences of one are discovered.

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[a] Specifications for such buildings recommended by the Bureau of Mines are to be found in Technical Paper 18 of the bureau: Hall, Clarence, and Howell, S. P., Magazines and thaw houses for explosives, 1912, 34 pp., 1 pl., 5 figs.

Institute a regular and frequent inspection of the valves on the air compressor and insist that any defective valve be promptly and properly repaired even at the cost of a possible shutdown, that there may be no explosion of gas or burning of grease in the receiver or pipe line to produce harmful gases and jeopardize the safety of the men at the heading. Do not delay the installation of adequate auxiliary ventilating equipment when natural accumulations of harmful gases are encountered in the tunnel, particularly when such gases are of an explosive nature. When explosive gases are present, none but safety lamps or their equivalent should be permitted under ground.

Prohibit the men from riding on loaded trips, and, whenever possible, provide special cars for their use, either propelled by hand or drawn by a motor. Do not permit the men to jump on or off moving cars, nor the drivers to " ride the chain." Tell all new men the proper side of the tunnel to take when meeting a trip, and caution them to shield any bright light when so doing.

If there is a trolley wire or other electrical apparatus in the tunnel, caution the men against its danger, and do not allow them to carry tools on their shoulders when passing in or out. See that the cable or wires leading to any temporary or movable cluster of lights in the heading is kept in good repair. Instruct the men, especially the foremen, as to the proper methods of resuscitation in a case of electric shock.

Prohibit the accumulation of combustible rubbish any place in the vicinity of buildings or timbering and see that the supply of hay is properly confined to prevent danger from fire. Unless absolutely necessary, do not construct any wooden buildings nearer than 200 feet to the mouth of the tunnel. If wooden buildings must be built near the mouth, provide a separate exit from the tunnel at least 200 feet away, with a fire door that is arranged to be closed from a distance. In either event, provide an adequate water supply, with hydrants and hoses, at suitable distances from the several buildings.

Exercise great precaution when driving toward a place where a flow of water is likely to be encountered that might carry with it a rush of mud, sand, gravel, or other débris, and take immediate steps for the safety of the men as soon as such a flow is struck.

Prohibit the drinking of intoxicating liquors on property controlled by the tunnel company and institute a system of inspection to prevent any intoxicated man from working in the tunnel, discharging habitual offenders against this rule.

PRECAUTIONS FOR THE FOREMAN.

Insist that the least amount of dynamite required for loading "back" holes shall be used. Do not return to the face after blasting nor permit the men to return without first examining the new roof. Upon arriving at the heading immediately detail as many men as may be required to clean the roof before attempting any other work under it. When passing in or out of the tunnel never fail to inspect the roof, testing any doubtful piece for possible vibration. See that any loose piece of rock is either pulled down at once or properly supported, and never take any chances by postponing the work of timbering no matter how pressing other matters may be, because a few minutes' delay in timbering may cost several lives. Have any timbers showing the effects of too great pressure relieved properly as soon as they begin to fail. When timbering is necessary close to the face, see that the front sets are thoroughly braced and blocked before firing. When the roof "breaks high" fill the space between the lagging and the roof with broken rock or blocking to prevent a large rock from crashing through the lagging upon the men beneath.

See that the men read the precautions to be taken in handling explosives, or have a copy read to them. Do not permit any instance of careless or reckless handling of explosives to go unchallenged and do not fail to discharge men for the first grave offense of this character. Never permit a man to handle dynamite recklessly, either for the purpose of scaring someone or for any other reason. See that the detonators and primers are transported to the heading in boxes separate from the rest of the supply and that they are not placed side by side after arriving. Insist that proper care be used in loading holes and that the tamping be done by pressure rather than by impact. Never allow anything but wooden bars to be used for this purpose. Do not permit a bore hole to be loaded before it has had sufficient time to cool completely from the previous blast.

Warn the men of any change in the rate of burning of fuse. See that they do not mutilate the fuse by rough handling and that they do not crack or break it by placing the primer in the hole fuse end first, or by uncoiling the fuse roughly in cold weather. Do not use fuse that has been stored or kept near a boiler, steam pipe, or other source of heat, or that has been exposed to moisture. See that the fuse is properly coiled close to the hole before blasting, in order that it may not be torn out by the blasts in a near-by hole. Instruct the men as to the proper way to prepare a primer. See that the fuse is cut squarely; that an inch or so of it is discarded; that the grains of powder do not leak out of the end that is inserted into the detonator; that the crimping is done carefully with the proper tool; that the detonator is not buried too deeply in the dynamite; and that caps of sufficient strength are used.

Always count the holes as they are blasted and never fail to inspect the new face for evidences of missed holes. See that any such are detonated properly as soon as they are discovered, even at the possible cost of some delay. Insist that the shovelers use their picks properly when picking down the muck pile. Keep a close watch for any unexploded dynamite in the muck and have the men do likewise. When such is found remove it carefully to a place of safety and be particularly cautious when a piece of fuse accompanies it. Never start a new hole in the remains of one that has ever held dynamite.

When the presence of any amount of dangerous gases, either from explosives or from natural sources, is suspected see that the men are supplied with fresh air, either by opening the compressed-air line or by breaking into the ventilating pipe, if the current is in the right direction. Do not knowingly remain or permit the men to remain in any atmosphere that will not support a candle flame, because there is no way to determine how bad it may be after the light becomes extinguished. See that the men do not use anything but safety lamps, or their equivalent, in tunnels where explosive gases are encountered, and do not permit any matches or other means of striking an open light to be carried into such a tunnel.

Have the track and roadbed kept in as good condition as possible in order to lessen the risk of derailments. Do not permit men to ride upon loaded trains unless it is absolutely necessary, and in such cases warn them carefully as to the risk being taken. Insist that the men riding in an empty car keep their feet and hands inside of the car and that they watch carefully for low places in the roof. Never fail to discharge any driver caught "riding the chain." See that the men give an approaching train of cars plenty of room, and if animals are used to draw the cars see that the men hide their lights when the animals approach.

Warn the men of the danger from the trolley wire. Familiarize yourself with the proper means of resuscitation after an electrical shock. See that the men are not permitted to carry on their shoulders tools or other instruments that are conductors of electricity. Inspect regularly any cables or wires used for carrying electricity to lights in the heading, or any others that have to be moved frequently, and see that all worn parts are covered with insulating material or replaced if necessary. Do not permit the men to ride on electric locomotives.

See that no piles of combustible rubbish are allowed to accumulate underground, and do not permit the use of candles or torches in the vicinity of hay or other inflammable substances. Do not fail to discharge any men guilty of leaving candles or torches burning near timbers, especially when a candle is wedged between two nails driven into a post.

Exercise special precautions when approaching a place where an inrush of water is to be expected.

Be particularly cautious about drunkenness. Note the men when coming on shift and do not permit a man even slightly intoxicated to go underground. If such a man is discovered in the tunnel send him to the surface at once. Discharge those who are habitual offenders in this respect.

PRECAUTIONS FOR THE MINER.

Do not return to the face of the tunnel without testing the newly exposed roof for loose rocks, and, if any such are discovered, either clean them down yourself or report them to the foreman. Form the habit of carefully examining the roof as you pass in and out of the tunnel, testing doubtful places for vibration; call the foreman's attention at once to any ground that you think should be timbered or to any timbers that need relieving to prevent their breaking.

If you are called upon to use dynamite, do so with great care, observing the precautions outlined in previous paragraphs. Never attempt to scare anyone by reckless handling of explosives and never treat dynamite with roughness. Never place or carry detonators or primers and the rest of the supply of dynamite for the round in the same box or bundle. If it is your duty to assist in the loading of the holes, do this with care, using pressure rather than a blow to tamp the powder in the hole, and be careful never to use too much force in pushing it.

Inquire as to the rate at which the fuse burns, especially when a new brand is being tried, and see that the fuse is cut long enough to give you and your companions time to reach a place of safety. Protect the fuse from mechanical injury, such as scraping, blows, or too great pressure either from falling rocks or from the tamping bar; never use a fuse that has been thus damaged. Never reload a bore hole before it has had time to cool. Do not use fuse that you know has been stored near a boiler, steam pipes, or other source of heat or one that has been exposed to moisture. If you prepare the primer, see that an inch or so is cut squarely from the end of the fuse before it is put into the detonator; that no powder runs out of the end of the fuse during this process, and that the detonator is properly crimped around the fuse. Under no circumstances use anything but the regular crimping tool for this purpose.

Always inspect each new face for evidences of a misfire, and, if one is discovered, call the foreman's attention to it immediately, so that he may have it detonated. Never attempt to pick out the material in such a hole; either explode it with a primer or, if this can not be done, drill and fire another hole at least 2 feet away. Use great care in removing any unexploded dynamite from the muck pile, and be especially cautious if a piece of fuse is discovered near it, for this may mean that there still is a detonator in the cartridge. Never handle a pick like a sledge hammer; pull or scrape the material down rather than strike it with the pick. Do not start a new hole in the remnants of a former one that has ever held dynamite, for there is always a chance that the dynamite may not have been detonated.

Whenever you feel that you are inhaling fumes from dynamite that has burned, or any other harmful gases, try to get to fresh air as soon as possible; the quickest way to do this is often to open the compressed-air line, or to break the ventilating pipe, if you know that the current is in the right direction. Never use anything but a safety lamp or a portable electric lamp in a tunnel where explosive gases are known to exist, and do not carry any other means of striking a light into such a tunnel.

Never attempt to ride upon a full car or a loaded trip; and when riding in empty ones see that your feet and hands are well inside and that your head is low enough to clear the roof at all places. Learn which side of the tunnel has the most room, and when a trip of cars approaches allow yourself as much clearance as possible. If the trip is drawn by an animal, hide any bright light you may be carrying. If it is your duty to drive a horse or mule or to run a locomotive, try to do everything possible to prevent derailments; report any places where the track or roadbed is in bad condition. Remember that the front end of the trip is the most dangerous place you can stand, so that if this is necessary, you must take extra care; never under any circumstances ride with one foot on the chain by which the cars are being pulled. Take care that the animal does not step on you or kick you, and speak to him before approaching him from the rear. In placing a derailed loaded car back upon the rails, take care not to strain or otherwise injure yourself in so doing; keep your feet and hands in a safe position, and see that the car does not topple over and crush you against the sides of the tunnel.

Bear constantly in mind that the trolley wire is dangerous and that you must pass within a few inches of it when going in and out of the tunnel, often when your attention must be given to your footing. This danger should be especially avoided when climbing into cars. When you are in a tunnel where there is a trolley wire never carry tools, drill steel, or anything else that is metal or wet on your shoulders. Do not handle any electrical equipment unnecessarily nor ride on electric locomotives. Never cause anyone to receive an electric shock; it is never possible to foretell its results. If it is your duty to repair electrical apparatus, see that you are properly insulated, or that the current is cut off and can not be turned on without your knowledge; keep your hands and body in such a position that a recoil from an accidental shock will throw you clear of any charged part of the apparatus. In removing and replacing the temporary cluster of electric lights in the heading be careful not to touch any bare or injured place in the wires, and call the foreman's attention to any damaged place you may discover. Familiarize yourself with the methods of reviving a person injured by electric shock and put them into practice as soon as possible whenever necessity occurs.

Do not smoke or throw a lighted match near any pile of inflammable rubbish either in a building or near timbering, and do not carry a candle or a torch near any piles of hay. Never wedge a candle between two nails on a post or other piece of timber; many disastrous mine fires have started in just this way.

Never take a drink of liquor before or during working hours, and do not hesitate to report any man you see doing so or who is in an intoxicated condition; your safety and perhaps your life may be sacrificed to his carelessness when under the influence of liquor.

SURFACE EQUIPMENT FOR DRIVING TUNNELS.

The equipment on the surface at tunnels that are being driven usually centers around a power house. Ventilating machinery for supplying fresh air to the men working in the tunnel is usually placed here, with the blacksmith and repair shops near by. Other surface equipment consists of storehouses and buildings in which the tunnel crew are housed and fed.

POWER EQUIPMENT.

SOURCES OF POWER.

Although the power for driving a tunnel may be obtained from various sources, in general practice at present it is produced primarily from either steam or flowing water. Although, as far as could be ascertained, the gas producer used in connection with internal-combustion engines has been installed at only one tunnel, nevertheless it offers a possibility as a source of power that will have to be seriously considered in the design of future plants. Within the last few years, as its advantages have become better appreciated, the gas producer has developed rapidly—so rapidly, in fact, that few people realize that it is to-day as reliable and substantial a piece of apparatus as an ordinary boiler, that its consumption of fuel is only one-third as great, and that the labor to operate it need not be one whit more skilled. The gasoline engines occasionally used to furnish power in tunnel operations have been confined either to temporary plants or to small and isolated units of machinery. In localities where petroleum is cheap it is probable that an oil engine of the Diesel type, with its wonderful fuel economy, may be found the cheapest means of producing power. Electricity, especially where it is used at tunnel plants to operate prime moving machinery, is sometimes considered a source of power; but as the current so employed has to be generated elsewhere, usually from steam or water, or possibly from producer gas, petroleum, or gasoline, electricity is merely a convenient form of power for transmission, not a source of power.

PRODUCTION OF POWER.

WATER POWER.

In tunneling, the machines most frequently employed for the utilization of water power are of the impulse type. Such a wheel is driven by the force of a stream of water issuing from a nozzle acting against vanes or buckets on the circumference of the wheel, and is well adapted for use with a relatively small volume of water under high head. The efficiency of the machine is dependent upon the way the vanes or buckets reverse the direction of the water discharged upon them; hence they usually conform to a curve which is very carefully designed to avoid loss of power through eddies and friction as the water strikes the vane. There may be more than one nozzle in order to obtain greater power or, if high rotative speed is desired, a small wheel with multiple nozzles may be substituted for a large one. In order to obtain the best results the peripheral speed of the cups or vanes should be between 42 and 48 per cent of the speed of the water issuing from the nozzle. Impulse wheels are manufactured in many different designs, sizes, and speeds, adapted for working under widely diverse conditions. Those observed at different tunnels examined for this report were, as far as could be learned, giving very satisfactory service.

The turbine wheel, in which the force of the water is made to act through suitable guides upon all the vanes or blades simultaneously, affords another means of utilizing water power and may be designed for either high or low heads. Its use is limited, however, especially with high heads, to localities where clear water is available, because of the abrasive action of sand and grit upon the guides. With low heads this action is not so marked. As the water power available in the vicinity of tunnels and adits is in most cases from streams furnishing high heads and at certain seasons of the year carrying large amounts of sediment, the use of turbine wheels for such plants is not feasible unless large settling basins can be provided.

The hydraulic compressor, converting the energy of water directly into compressed air, offers a third method for utilizing water power. The earliest type operates on the principle of the hydraulic ram, in which a column of water is allowed to acquire velocity and is then suddenly checked, developing intermittently for a short space of time pressure much greater than that due to the head of the column of water. This pressure is employed in compressing air. Sommeiler, in about 1860, designed a machine of this type for use at the Mount Cenis Tunnel. Such compressors require rather high heads and have low efficiency. Although conditions might be such as to make the use of compressors of this type desirable, the water power they require can generally be utilized more advantageously in some other manner.

The hydraulic compressor recently developed by C. H. Taylor,[a] introducing air into a column of water and compressing it as the air and water fall together to the bottom of a shaft where the air is

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[a] See article on Taylor hydraulic air compressor, Compressed Air Mag., June, 1910, p. 5675.

separated and collected, is very efficient and requires only a small amount of attention, although the cost of construction prohibits its use except for installations much larger than those ordinarily required for tunnel work. The latest installation of this system, which was completed in June, 1910, is situated at Ragged Chutes on the Montreal River, and supplies air to the mining district near Cobalt, Ontario.[a]

At this plant a concrete dam diverts water from the river above the rapids to the tops of two circular shafts 8½ feet in diameter, where, by means of suitable apparatus, a large quantity of air is introduced into the water in the form of bubbles. The mixed water and air descend the shafts (350 feet deep) and start through a passage 1,000 feet long. The passage, as shown in figure 4, is so designed that the compressed air is permitted to rise to the surface of the water and is collected partly along the top of the passage and partly in a large collecting chamber excavated near the end of the passage. The waste water then rises 298 feet through a shaft 22 feet in diameter and is discharged into the river below the rapids. The air is drawn from the top of the chamber at a pressure of 120 pounds to

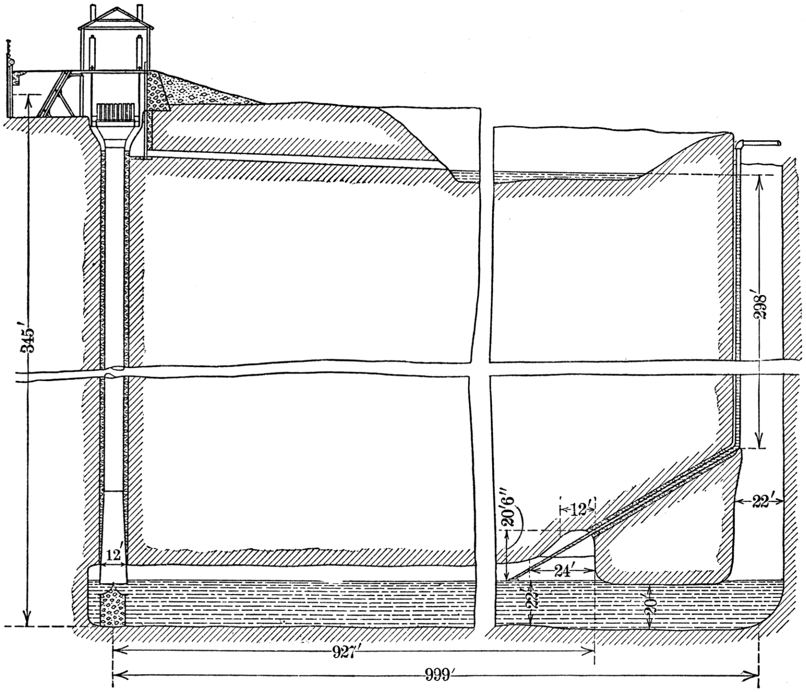

FIGURE 4.—Cross section of Taylor hydraulic compressor at Cobalt, Ontario.

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[a] Bateman, G. C., The Cobalt Hydraulic Company: Eng. and Min. Jour., vol. 92, Nov. 18, 1911, p. 998.

the square inch and is transmitted by a 20-inch main to the mines 9 miles distant. The capacity of the plant is the compression of 40,000 cubic feet of free air per minute to 120 pounds per square inch.

The familiar overshot, breast, and undershot wheels are not used to drive machinery for tunnel work because of their large size in proportion to the power developed and because of the trouble of their maintenance.

The following table, which is based upon actual results, shows the efficiency of different types of water motors:

Percentage of theoretical horsepower realized by various water motors.

	Per cent.
Impulse wheels	70 to 85
Turbine wheels	75 to 85
Overshot wheels	60 to 65
Breast wheels	50 to 60
Undershot wheels	30 to 50

Results obtained at the testing flume of the Holyoke (Mass.) Water Co., whose tests are taken as standard by American engineers, show efficiencies for turbine wheels under favorable conditions of over 80 per cent,[a] but this is unusual, the figures above being much nearer the efficiency obtained in ordinary practice. The efficiency of hydraulic compressors of the ram type is about 30 to 40 per cent, whereas the Taylor compressor at Cobalt is said to utilize at least 75 per cent of the theoretical power of the water.

STEAM ENGINES.

Steam engines are of two types, reciprocating and turbine. In the reciprocating engine power is developed by the pressure and expansion of steam in a cylinder acting against a moving piston. Such engines may be either simple or compound, both forms being used in tunnel plants. In simple reciprocating engines the total expansion of the steam and consequent reduction of pressure takes place in one cylinder, whereas in the compound type a part of the expansion takes place in the first cylinder and the steam, under somewhat reduced pressure, is expanded further in a second cylinder, necessarily larger because of the lower pressure of the steam.

The steam turbine is similar in principle to the water wheel, except that steam instead of water is the motive fluid. Among its advantages are economy, small size per unit of power, and freedom from vibration, and its use is steadily increasing on both land and sea. Modern steam turbines in sizes of 250 to 500 horsepower, with a steam pressure of 150 pounds and a 28-inch vacuum, will develop

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[a] Larner, C. W., Characteristics of modern hydraulic turbines: Trans. Am. Soc. Civ. Eng., vol. 66, March, 1910, p. 322.

a kilowatt-hour with a consumption of 18 to 20 pounds of steam. A recent series of shop tests[a] on a 300-kilowatt Swiss condensing turbine showed that with 112½ pound of steam and a 96.6 per cent vacuum a kilowatt-hour was produced with 16.1 pounds of steam. The difficulty of reducing the high rotative speed of the turbine engine down to the restricted speed of reciprocating machinery has prevented until recently the use of turbine engines in tunnel installations, but with the advent of the turbocompressor we may expect to see them dividing the field, or, perhaps, entirely displacing the cumbersome reciprocating plants now in vogue.

INTERNAL-COMBUSTION ENGINES.

Internal-combustion engines develop power from the pressure produced by the explosion or rapid combustion (confined in a suitable cylinder) of a mixture containing the proper proportions of air and a gasified fuel. The source of the fuel gas may be gasoline, kerosene distillate, or even crude petroleum, or the gas may be generated from coal by distillation in a retort or by a gas producer. The engines are usually designated by the kind of fuel for which they are adapted, as, for example, oil engines, gasoline engines, or producer-gas engines. As far as could be ascertained, the latter two are the only types now used in tunneling.

Although the gasoline engine has been developed with wonderful rapidity during the last 25 years in connection with the automobile industry, the use of engines of this type for tunnel work has been confined to a very limited field, namely, the operation of machinery in isolated or not easily accessible localities. As prime movers for tunnel plants of any magnitude they can not compete, under most circumstances, with machines using other sources of power, and on this account their application has been confined either to enterprises small in scope or to the temporary and early development stage of larger projects, for which they are sometimes installed to begin the work at places by nature inaccessible for nonportable units of heavier machinery, pending the construction of a special roadway or a transmission line. Most manufacturers of air compressors have recently begun to supply air compressors directly driven by internal-combustion engines, although as yet only the smaller sizes of gasoline engines are being used. With suitable adaptations the principle might be applied equally well to the larger sizes using oil or gas as fuel. Within the past three years gasoline locomotives have been successfully employed in coal mines, and, according to the manufacturers, there are now over 100 of them in operation. These machines are equally

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[a] Described in a letter from the manufacturer to the authors.

suitable for use in tunnel work and will, no doubt, be employed extensively for this purpose in the near future.

The only use of producer-gas engines in tunnel work to date, so far as learned, was in England at the power plant for the tunnel recently completed under the Thames River, connecting North and South Woolwich. As the tunnel was driven by the use of air locks absolute reliability in the power plant was required to avoid a decrease of pressure that might result in serious damage to the tunnel. At this plant, as described in "The Engineer,"[a] three engines, each of 150 brake horsepower, when running at 180 revolutions per minute, were supplied with gas from suction producers using Scotch anthracite coal and were connected to a central shaft which transmitted power to 4 air compressors and 4 dynamos. The plant was operated continuously from July, 1910, until the end of December, 1911, except during October and November, 1910, when after the vertical shaft had been completed the plant was purposely stopped while preparations were being made to start tunneling.

The gas producer is discussed in detail in Bureau of Mines Bulletin 13.[b] The reader is also referred to the bibliography accompanying this report and to the bulletins noted below, which are issued by the Bureau of Mines:

Bulletin 4. Features of producer-gas power-plant development in Europe, by R. H. Fernald. 1910. 27 pp., 4 pls., 7 figs.

Bulletin 7. Essential factors in the formation of producer gas, by J. K. Clement, L. H. Adams, and C. N. Haskins. 1911. 58 pp., 1 pl., 16 figs.

Bulletin 16. The uses of peat for fuel and other purposes, by C. A. Davis. 1911. 214 pp., 1 pl., 1 fig.

Bulletin 31. Incidental problems in gas-producer tests, by R. H. Fernald, C. D. Smith, J. K. Clement, and H. A. Grine. 29 pp., 8 figs. Reprint of United States Geological Survey Bulletin 393.

Bulletin 55. The commercial trend of the producer-gas power plant in the United States, by R. H. Fernald. 1913. 93 pp., 1 pl., 4 figs.

Although, as far as could be learned, oil engines of the Diesel type have not been employed in tunnel power plants their marked success in other fields warrants the discussion of the possibility of their use in tunnel work. The Diesel differs from other internal-combustion engines in that instead of drawing into the cylinder an explosive mixture of air and a combustible gas (such as producer gas, gasoline vapor, kerosene, or even crude petroleum previously volatilized by heat) and then compressing this mixture and exploding it by means of an electric spark, or some other suitable device, the Diesel engine compresses air alone, and when this is under high pressure (approxi-

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[a] The Engineer (London), Temporary power plant for Woolwich footway tunnel: Jan. 12, 1912, pp. 46–47.

[b] Fernald, R. H., and Smith, C. D., Résumé of producer-gas investigations, Oct. 1, 1904, to June 30, 1910. 1911. 393 pp., 12 pls., 250 figs.

mately 500 pounds per square inch, which is much greater than that usually attained in other types of internal-combustion engines) injects into the cylinder a spray of finely atomized oil. During the compression of the air to the required pressure it reaches a temperature of more than 1,000° F., more than sufficient to ignite the oil instantly without the use of an electric spark, hot plate, or other similar device.

The chief advantage of the Diesel engine is economy of fuel. It is a well-known fact that the rapidity and completeness of combustion are increased by pressure; it is not surprising therefore that under the high pressures in this machine excellent results can be obtained from a small amount of oil. The extremely fine atomization of the fuel, due to its being injected by compressed air under a pressure of 300 to 500 pounds per square inch higher than that in the cylinder, is undoubtedly another important factor. And again, as scavenging is effected by air only instead of by a mixture of air and fuel, as is the case in other types of internal-combustion engines, there is no possibility of fuel being lost through the exhaust valves during this process, a saving that is extremely important in engines designed for a two-stroke cycle.

In addition to the advantage of fuel economy, however, the Diesel engine does not require frequent cleaning, as is the case with oil engines depending upon a hot plate or a similar device for the ignition of the explosive mixture. It also dispenses with the carbureter, so necessary for the gasoline engine and always a source of trouble and annoyance. In addition, as the mixture being compressed in the cylinder of a Diesel engine is not explosive, allowance does not have to be made in the design of the cylinder and other parts for undue stresses and strains that might result from a premature ignition of the charge, of not infrequent occurrence in other engines. Although some provision can, of course, be made for these shocks, their force and violence can not always be foreseen and there have been many instances of disastrous results arising from them.

The principal disadvantage of the Diesel engine, on the other hand, is that of high first cost, and this would prohibit its use for tunnel work of short duration. However, the price has recently been greatly reduced abroad and it is certain to be reduced in America, now that manufacturers in this country are equipped with suitable apparatus and prepared to execute the high class of workmanship required in its construction; hence in the near future this drawback may disappear. But even now, if the time required for the completion of the work is to be long enough or the amount of power to be used is great enough to warrant a heavy initial outlay in order to effect a saving in operating cost, the choice of a Diesel engine should be seriously considered.

ELECTRIC MOTORS.

Electric motors may be designed either for direct or alternating current. Where used as prime movers at the tunnel plants visited they were of the alternating-current type only and operated at comparatively low voltages, usually 440 volts. Their power was generally transmitted to the machinery by means of belts, but at one or two places on the New York Aqueduct direct-connected electric-driven air compressors were noticed.

TRANSMISSION OF POWER.

Electricity, because it can be economically transmitted long distances, is the favored means of transmitting to a tunnel the power generated at some remote station. It possesses one well-known disadvantage, that of occasional interruption, especially where distances are great and tension high, because of unavoidable hindrance to service through storms. On the other hand, producer gas has possibilities that deserve serious consideration. Its transmission has been recently taken from the realm of mere conjecture and demonstrated as practical in the Pittsburgh (Pa.) district where natural gas is piped for distances of 200 miles, and in England, where producer gas is supplied to points within a radius of 160 miles with transmission losses even less than those of electricity. In its application to tunnel operations, producer gas can be generated in a plant conveniently situated on a railroad siding or some other readily accessible place, and the power piped to internal-combustion engines at the tunnel portal.

Where distances are comparatively short (that is, less than 5 miles) electricity is rivalled by compressed air, and the competition grows more keen as the length of the transmission system decreases. This is possible because compressed air, in spite of its low efficiency and high cost, is necessary where pneumatic drills are employed; and even were electric transmission chosen, air would have to be compressed by electricity at the tunnel. Hence it is the usual practice to produce the compressed air at once, thus avoiding the extra machinery and the additional operating losses of electric transmission.

CONSIDERATIONS INFLUENCING CHOICE OF POWER.

A number of factors enter into the choice of power for tunneling operations. To begin with, the plant is usually short lived. Then, too, such local conditions as accessibility, distance from a railroad, the availability of water power, etc., must be considered. Each method of deriving power has also certain peculiarities that render it particularly adaptable to some conditions. Among these may be mentioned the cost of installation, of labor, of fuel, of interest and depreciation, and other operating expenses. Aside from all this, it is often necessary to decide between the production of power at the plant or elsewhere and the purchase of power from an established hydroelectric company. Some of these factors are discussed briefly below.

DURATION OF PLANT.

At many tunnel power plants, in direct contrast with those used in manufacturing, the equipment is required only for the comparatively short time of actual tunnel construction. Thus it becomes a delicate problem to determine just how far one is justified in the purchase of machinery and apparatus for utilizing all the various economies that may be effected in the production of power. It is difficult to decide whether it would not be better in the end to install less costly machinery that would necessitate slightly higher expense in operating and maintaining than to tie up extra capital in equipment that would be of no further use when the tunnel is completed. Of course, the shorter the probable life of the plant the more would one be justified in such a course. Although, if the equipment can be transferred upon the completion of the tunnel to other projects, this would so prolong its period of usefulness that the original expenditure of capital could properly and with true economy be greater. A notable instance of this was observed on the Los Angeles Aqueduct, where, as far as possible, upon the completion of one of the numerous tunnels, the equipment was transferred and used in power plants at other tunnels whose construction had not yet begun. If a central station for generating a large amount of power is being considered, the purchase of apparatus with regard to ultimate economy in operation is again the far-sighted policy. This was the case at the Rondout Siphon Tunnel, but at the average mining tunnel the converse is more often likely to be true.

ACCESSIBILITY OF TUNNEL.

Many tunnels are driven at places very difficult of access. They may be so situated as to make the installation of heavy machinery no easy matter, as, for example, where the road from the nearest railroad is poor and has steep grades; or they may be at a great distance from the nearest siding, so that if the power plant requires coal, the delivery of this fuel is not only costly, but also uncertain and difficult in some seasons of the year. Such conditions are favorable for the adoption of power transmission in some form from a waterfall or rapid, if there is one near enough, or, lacking these natural advantages, from a fuel plant installed at some point more readily accessible.

COST OF INSTALLATION.

The cost of installing water wheels depends entirely on local conditions, which are never twice alike. Where high heads are available and the quantity of water required is not large, water can be conveyed to the water wheel by small flumes or pipes, which are comparatively inexpensive. For example, at the Carter Tunnel (Pl. I, *A*), with an available head of 145 feet, a flume 16 by 48 inches inside and 5,000 feet in length is sufficient to supply the 200 horsepower developed. At the Laramie-Poudre Tunnel, with a static head of 268 feet, 400 horsepower is conveyed to the tunnel plant by a wooden-stave 22-inch pipe line, 8,500 feet in length. The Utah Metals Tunnel obtains water from two sources: The first gives a 700-foot head, requires 2,500 feet of 12-inch, 1,900 feet of 10-inch, and 100 feet of 8-inch spiral, riveted steel pipe, and furnishes 170 horsepower; the second, giving a 750-foot head, requires 2,000 feet of 12-inch, 1,000 feet of 8-inch, and 3,000 feet of 6-inch pipe to produce 55 horsepower.

Where heads are low retaining dams are usually necessary. At best these are expensive, and their cost increases enormously with their height. With low heads larger flumes are required to convey the greater quantity of water. At the Nisqually Tunnel (Pl. I, *B*), a low dam and a 6 by 8 foot wooden flume 1,200 feet long are used. The water is delivered under an effective head of 29 feet to a turbine wheel that generates 1,000 horsepower. One has only to consider some of the very expensive dams on the larger rivers, furnishing power for manufacturing purposes, in order to realize how great the cost of installation may be where low heads are utilized. It is fortunately true, however, that where water power is obtainable for tunnel work the heads are usually high and the less expensive flumes or pipe lines of moderate length can be utilized.

The cost of the machinery actually within a tunnel power house is greater for steam than for water power or electricity; but if, as should be done to make the figures truly comparable, the cost of the dam and the flume (or of the transmission line for electricity) be taken into consideration, the advantage is usually reversed. It is somewhat cheaper to install a steam plant than one using producer gas and having engines of the same capacity, but the difference is not great. Fernald,[a] after a study of many tables of costs, applying to other uses, however, than tunnel work, concludes that "complete producer-gas installations for the larger plants, say, from 4,000 to 5,000 horsepower, cost about the same as those of first-class steam plants of the same rating. With smaller installations the balance is probably in favor of the steam plant." As it is not customary in

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[a] Fernald, R. H., The commercial trend of the producer-gas power plant in the United States: Bull. 55, Bureau of Mines, 1913, p. 26.

tunnel work to install machinery designed to effect all the refinements of steam economy found in permanent plants, it is probable that the first cost of the average steam plant for tunnel work is less than those upon which Fernald's estimates are based, in which case the comparison would be even more favorable to steam. This is partly offset by the fact that the price of gas producers and engines is constantly being lowered, and by the fact that the cost of actually placing the machinery would be less for the gas producer.

The initial expense of installing any of the various systems for transmitting power is dependent upon two factors: The cost of the machinery required to produce the power, to convert it into the form suitable for transmission, and to reconvert it into the form adapted to the machines using it; and the cost of the transmission line. Except for a slight increase in capacity, to provide for losses in transmission, the factor of machinery cost under any given conditions is independent of the distance over which power is to be delivered, but the cost of the line increases somewhat faster than its length, other things being equal. If the power is required ultimately for the operation of air drills, practically the same size of compressor will be necessary whether electric, air, or gas transmission is employed, and the cost of boiler, engine, and foundations, in the case of electricity or air, will approximately balance the cost of the producer, engine, and foundations for gas. Air transmission requires practically no other machinery than that just mentioned, but gas, on the other hand, needs a blower of some sort to force it through the line, and electricity requires, in addition, a generator, motor, transformers, and the extra foundations. A list of the three forms of power transmission arranged in the order of increasing machinery cost would be air, gas, and electricity.

The cost of an electric transmission line may be divided into three parts: First, the conductor and connections; second, insulation; and third, erection of the line. Although a detailed discussion of this subject is beyond the scope of this bulletin, it may be stated that, for a given power loss and a given distance, the weight of the conductor required to transmit a definite amount of power is inversely proportional to the square of the voltage employed. On the other hand, the cost of insulation increases rapidly with the potential, and the cost of erection, complicated by steel towers and special precautions, is greatly augmented at high voltages. Thus the economical transmission of a given amount of power for a stated distance is limited by the maximum voltage that may be used without the increased cost of installation and erection destroying the saving in the cost of copper.

Relative to the most advantageous voltage for the short distances and small amounts of power commonly employed in tunneling operations, the following rule of thumb will give results closely approximating those of the most careful calculations: Multiply the distance to be traversed in miles by 1,000 and select the voltage of the commercial size of transformer nearest to this figure. The standard voltages of transformers now in use are 220, 440, 660, 1,100, 2,200 6,600, 11,000, 22,000, 33,000, and 66,000. If the distance from the power station to the tunnel plant is 5 miles, select a voltage of 6,600; if the distance is 10 miles, a voltage of 11,000. If the distance falls midway between transformers steps, use the voltage that will permit most ready sale of the apparatus when the work is completed.

As there are certain difficulties in the construction of direct-current generators for voltages higher than 600, alternating current is generally employed for transmission lines. An important additional advantage of alternating current is the ease with which the potential may be changed from low to high, or vice versa. Where electricity is transmitted at high tension it is usually generated at a comparatively low voltage, "stepped up" by transformers to the desired potential for the line, and "stepped down" by transformers at the tunnel plant.

The following figures, which show the installation cost of an electric-transmission line for different voltages and distances, assuming approximately 10 per cent drop in the line, are based upon data kindly furnished by the General Electric Co.:

Cost of installation for electric-transmission line for different voltages and distances.

1. 200 horsepower; 1 mile; 440-volt, direct current:	
Poles, cross arms, insulators, and fittings (poles spaced 100 feet)	$375
33,000 pounds copper cable, 500,000 circular mils (4 conductors required), at 18¼ cents per pound	6,025
Cost of erection	300
Total	6,700
2. 200 horsepower; 1 mile; 440-volt, 3-phase, 60-cycle, alternating current:	
Poles, cross arms, insulators, and fittings (poles spaced 100 feet)	415
34,000 pounds copper cable, 350,000 circular mils (6 conductors required), at 17¾ cents per pound	6,035
Cost of erection	375
Total	6,825
3. 200 horsepower; 1 mile; 1,100-volt, 3-phase, 60-cycle, alternating current:	
Poles, cross arms, insulators, and fittings (poles spaced 125 feet)	385
5,100 pounds copper cable, B. & S. No. 0 (3 conductors required), at 17¾ cents per pound	905
Cost of erection	265
6 transformers, 1,100 to 440 volts, with switches, etc., erected	2,900
Total	4,455

4. 200 horsepower; 5 miles; 1,100-volt, 3-phase, 60-cycle, alternating current:

Poles, cross arms, insulators, and fittings (poles spaced 125 feet)	$1,870
122,000 pounds copper wire, B. & S. No. 000 (9 conductors required), at 17¾ cents per pound	21,650
Cost of erection	1,580
6 transformers, 1,100 to 440 volts, with switches, etc., erected	2,900
Total	28,000

5. 200 horsepower; 5 miles; 6,600-volt, 3-phase, 60-cycle, alternating current:

Poles, cross arms, insulators, and fittings (poles spaced 125 feet)	1,870
6,500 pounds copper wire, B. & S. No. 6 (3 conductors required), at 17¾ cents per pound	1,150
Cost of erection	1,080
6 transformers, 6,600 to 440 volts, with switches, etc., erected	3,700
Total	7,800

6. 200 horsepower; 25 miles; 6,600-volt, 3-phase, 60-cycle, alternating current:

Poles, cross arms, insulators, and fittings (poles spaced 125 feet)	9,350
103,000 pounds copper wire, B. & S. No. 1 (3 conductors required), at 17¾ cents per pound	18,300
Cost of erection	5,150
6 transformers, 6,600 to 440 volts, with switches, etc., erected	3,700
Total	36,500

7. 200 horsepower; 25 miles; 22,000-volt, 3-phase, 60-cycle, alternating current:

Poles, cross arms, insulators, and fittings (poles spaced 125 feet)	9,900
33,000 pounds copper wire, B. & S. No. 6 (3 conductors required), at 17¾ cents per pound	5,860
Cost of erection	5,190
6 transformers, 22,000 to 440 volts, with switches, etc., erected	5,200
Total	26,150

Freight, right of way, surveying, and engineering costs are not included in the above data.

The Pneumo-Electric Machine Co.[a] has estimated that if compressed air were used to transmit 200 horsepower 1 mile, 10 per cent loss at 80 pounds pressure being allowed, an 8-inch pipe would be required, which, at $1.78 per foot, would cost $8,400. Calculations show that in order to transmit the same amount of power in the form of producer gas containing 120 British thermal units per cubic foot, the required pipe would have to be only 4 inches in diameter, costing, at 70 cents per foot, approximately $3,700. To both these values should be added the expense for laying the line, but this figure would be small compared to the cost of the pipe.

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[a] Cost of power transmission; electricity versus compressed air; Min. and Sci. Press, May 14, 1910, p. 700.

Where the power is ultimately required for use in air drills and power is to be transmitted only for short distances compressed air is the cheapest of the three methods as regards installation cost, the higher cost for the machinery required by the other systems more than balancing the expensive air-pipe line. Producer-gas transmission, with its cost for machinery slightly greater than that for air, yet less than that for electricity, and its line factor just the reverse, is best suited for the medium distances—beyond the economical range for air, but still too short, on account of extra machinery, for electricity. For long distances, on the other hand, electric transmission at high tension is, of course, preeminent.

LABOR REQUIREMENTS.

Tunnel power plants are generally not large enough to require the entire time of even one operator; hence it is impossible to prevent their being overmanned. Consequently the amount of labor required does not as a rule seriously affect the choice of power. At a tunnel plant using water wheels, hydraulic air compressors, or electric motors as prime movers one man per shift is sufficient. Even then, as was the case at the Laramie-Poudre Tunnel, it is not unusual to make the shifts 12 hours long, thus requiring only two men per day; or, as at the Carter Tunnel, to assign the engineer other work for a part of his time. If the results obtained from its use in other work be accepted, a producer-gas plant would require no more exacting attention, one man per shift at plants that develop as high as 750 or 1,000 horsepower being required. A steam plant of the same capacity, on the other hand, would require at least two firemen in addition to the engineer. In larger steam installations the labor required per horsepower generated is naturally not so great. For example, at the Rondout siphon tunnel 8 men per 8-hour shift were able to attend to a steam plant rated at 4,000 horsepower and containing 10 air compressors, each having a capacity of 2,400 cubic feet.

FUEL CONSUMPTION.

If the charge for delivering fuel be included the cost of fuel at most tunnel plants is high; hence the quantity required is of great importance. Steam plants require much more coal than gas plants of the same size, for although large steam plants, with every means for effecting thermal economies, may be operated with as little as 2 pounds of high-grade fuel per brake horsepower-hour, in small plants such as are used in tunnel work a fuel consumption as low as 3 pounds would be exceptional, and 4 and 5 pounds is more probable. With producer gas, on the other hand, it has been repeatedly demonstrated that internal-combustion engines can be operated on less than 1 pound of coal per brake horsepower-hour, and at the best plants this figure runs as low as three-fourths of a pound. The consumption at the Woolwich Tunnel plant during a test was 0.727 pound of Polmaise Scotch anthracite per brake horsepower-hour. The small internal-combustion engine has the additional noteworthy characteristic of being decidedly efficient in small sizes. A gas engine of 50 to 60 brake horsepower has little greater fuel consumption per horsepower than a large gas engine of 500 or 1,000 brake horsepower.

THERMAL EFFICIENCY OF ENGINE.

The comparatively high fuel consumption of the steam engine is due to its low thermal efficiency. Although large and economically operated steam plants may realize 12 to 15 per cent of the energy of the coal, 5 per cent is much nearer the value generally obtained in tunnel work. The following table shows the distribution of the average heat losses for one year at a well-conducted steam plant where the thermal efficiency at the flywheel was 10 per cent:

Loss of theoretical heat energy at a steam plant.

	Per cent.
Losses due to imperfect combustion, heat absorbed in ashes, moisture, etc., heat in flue gases, radiation, etc	25
Loss due to latent heat in exhaust steam	60
Loss in steam pipes and auxiliaries	3
Loss due to friction in steam engines	2
	90

The producer-gas engine, on the other hand, has a much higher efficiency, 20 to 30 per cent being not unusual in actual practice. Recent exhaustive shop tests of a number of first-class foreign-built producer-gas engines ranging in power from 70 to 120 horsepower gave thermal efficiencies at full load of 31.3 to 34.9 per cent and a coal consumption of 0.72 to 0.623 pound per brake horsepower-hour. The following table shows the distribution of losses in a producer-gas plant operating with similar economy to the steam plant cited above:

Thermal losses in producer-gas plant.

	Per cent.
Loss in gas producer	15
Loss in water jacket	21
Loss from radiation and friction	4
Loss in exhaust gases	35
	75

COST OF ELECTRIC CURRENT.

If the line of an established electric-power company runs near enough to the tunnel plant, it may be cheaper to buy power than to generate it. Such a concern is in a position to utilize a waterfall too distant to warrant its development for a single-tunnel project, and by distributing a large amount of power among many permanent customers is enabled to sell it very cheaply. The price of current usually ranges from 1½ to 2 cents per kilowatt-hour. On the Los Angeles Aqueduct the power used at all the tunnel plants was obtained from a private transmission line operated by a separate department of the aqueduct organization, and a flat rate of $1\frac{7}{10}$ cents per kilowatt-hour for power was charged against each tunnel, which was estimated to be sufficient to operate the system and eventually pay for its installation. At a tunnel in Colorado a flat rate of $2.50 per horsepower-month is charged, to which is added $1\frac{3}{10}$ cents per kilowatt-hour used. On a 24-hour day basis this is equivalent to 1¾ cents per kilowatt-hour. At another tunnel in Colorado, 2 cents per kilowatt-hour is the price of current. At a third, the power for the compressor costs $5.50 per horsepower-month, which is equivalent to 1 cent per kilowatt-hour on a 24-hour day basis; but at the same tunnel 2 cents per kilowatt-hour is charged for the current used in the motor generator set that operates the trolley system, making the average cost for the total power used approximately 1½ cents per kilowatt-hour. At one tunnel plant using much power the current is said to have cost only ⅞ cent per kilowatt-hour, an exceptionally low figure, but in this case other considerations were involved that really made the cost of the electricity greater.

The following schedule is used by a number of western hydroelectric companies, who claim that this rate and method of making a charge is "fair and rational."

Schedule of charges for electricity.

Horsepower.	Fixed charge per month per horsepower of maximum demand.	Energy charge.
For the first 100	$3.25	Add for all energy used as shown by meter 13 mills per kilowatt-hour for the first 40,000 kilowatt-hours used each month, and 5 mills per kilowatt-hour for all additional energy.
For the next 400	2.25	
For the next 500	1.75	
For all additional	1.00	

The maximum demand is determined by the company by meters, disregarding starting peaks and peaks due to short circuits or accidents to user's apparatus.

COST DUE TO INTEREST AND DEPRECIATION.

The charge for interest per unit of power is dependent upon the amount of capital invested, but that for depreciation depends on several factors. In the case of water power a dam or a ditch would have little salvage value after the completion of the tunnel; more might be realized from a pipe line or flume, and still more from the machinery in the power house. A similar analysis may be made for other means of producing power. Both interest and depreciation charges are dependent also on the number of hours the plant is used daily, it being evident that if a plant be used 24 hours instead of 12 the same total cost for interest and practically the same total loss by depreciation will be distributed over double the number of horsepower-hours, and hence be proportionally less.

CONCLUSIONS.

In choosing the power to be used for tunnel plants, a waterfall or rapid, if either is available, should have first consideration. The chief arguments in favor of water power are as follows: No fuel is required; the cost for attendance and repairs is a minimum; the power is comparatively reliable, obviating losses from interruptions of service. The factor that may prohibit its selection is a high cost of installation and resulting large charge for interest and depreciation per unit of power. This consideration, dependent entirely upon local conditions, usually determines the adoption or rejection of a possible water-power plant. Again, where water power is not obtainable directly at the tunnel plant, if it can be procured from a waterfall in the neighborhood, the essential factors remain the same, except that a means of transmitting the power, such as air or electricity, must be selected and the cost of the transmission system must be included in the cost of installation. Another possible means of obtaining the advantages of water power is current purchased from an established hydroelectric company. In such case, to the price of the power should be added the cost of attendance at the tunnel plant and the interest and depreciation charges on the necessary equipment. Allowance must be made for interruptions to service, which are neither unusual nor avoidable, in long-distance electrical transmission.

The choice of machinery for utilizing water power is also largely governed by local conditions. As high heads, for which impulse wheels are especially adapted, are generally to be found where water power is available for tunnel work, this type of machine is properly chosen in most instances. Turbine wheels may be used where the water is clear or can be settled in a reservoir. The hydraulic compressor, although practically automatic and entailing only a small operating expense, is so costly to build that it is scarcely to be considered, except for plants much larger than those designed for tunnel work.

If water power were not available and electricity were not purchasable, a steam plant would, according to usual practice, be installed. This choice is difficult to understand unless it be attributed to the supposed unreliability of the gas producer. The usual steam plant for tunnel purposes is, as has been shown, very inefficient in its utilization of the energy of coal and has a fuel consumption rarely less than 4 or 5 pounds per horsepower-hour. As regards cost of installation, the balance is slightly in favor of steam, but not sufficiently so to overcome the disadvantage of higher operating cost.

The producer-gas plant on the other hand, is several times more efficient in its utilization of heat energy, producing a brake horsepower per hour at some plants with as little as 1 pound of coal. With a producer it is also possible to utilize cheaper grades of fuel. The manufacturers of air compressors have recently adapted their machines for use with internal-combustion engines. It would seem, therefore, if a plant using fuel were necessary that the installation of a producer-gas plant under most conditions would be more desirable that a steam plant.

As a means of transmitting power for any great distance the balance is preponderantly in favor of electric transmission at high tension. In tunnel work and over comparatively short distances compressed air is able to compete with it, because the air drills require this form of power for their operation. When it is necessary to obtain power from coal there seems to be a field for producer-gas transmission in the medium distances, where the cost of the line and the power losses in transmission prohibit the use of air, but where the cost of the extra electrical machinery is still not warranted by the saving in cost of line.

AIR COMPRESSORS.

Many factors should be considered in selecting an air compressor. Motive power and capacity have to be considered first; then, perhaps, the type of the compressor. The methods of regulation under varying load likewise deserve attention, as do cooling and devices for removing moisture from the air.

The most familiar types of air compressor consist essentially of a cylinder in which air is compressed by a piston. Automatic devices are provided for admitting and delivering air, and a flywheel is required to keep the piston moving smoothly. When steam or internal-combustion engines are the prime movers they are usually, although not necessarily, incorporated with the compressor, the power and air pistons being connected by a common piston rod or engine shaft. Where water or electricity is employed the compressor usually is belted to the prime motor, the flywheel of the compressor serving as a pulley. But there is a growing demand for the direct-connected electrically driven machine, in which the electric motor forms an integral part of the compressor, the armature serving as a flywheel. Such machines are now supplied by all the leading manufacturers.

Direct-connected air compressors driven by water power are also obtainable; the water wheel serves as a flywheel for the compressor. Air compressors of an entirely different type, operating on the principle of the reverse turbine, have recently been placed on the market. They are especially adapted to take advantage of the high rotative speed of electric motors and steam turbines. Although the Taylor hydraulic system is, strictly speaking, an air compressor, it has been described (pp. 47–49) as a means of utilizing water power and will not be discussed here.

POWER.

Although the kind of motive power selected for a given plant is generally predetermined by local conditions, the amount of power required is worthy of brief discussion. Rix[a] states that in compressing air from atmospheric pressure to 90 or 95 pounds,[b] 20 brake horsepower must be delivered at the flywheel shaft of a reciprocating compressor for every 100 cubic feet per minute of piston displacement, this figure being deduced as the average result of a number of tests of air-compressor plants, the capacities of almost every kind of compressor being compared with the motive power required. Rix also states that the figures of power required as given in trade catalogues are usually somewhat lower than this value, but such figures are theoretical and do not take into consideration the mechanical or volumetric efficiency of the compressor. The following tables are computed from the catalogue of two leading manufacturers of a popular type of compressor and show the rated brake horsepower per 100 cubic feet of cylinder displacement when the final gage pressure is 100 pounds.

Relation between required brake horsepower and capacity of 2 air compressors, each compressing to 100 pounds per square inch.

Compressor A.		Compressor B.	
Capacity (cubic feet per minute).	Brake horsepower required for each 100 cubic feet of displacement.	Capacity (cubic feet per minute).	Brake horsepower required for each 100 cubic feet of displacement.
144	18.7	248	19.3
247	18.6	338	19.2
372	18.4	537	18.1
534	18.3	680	18.1
704	18.1	873	18.0
1,051	18.0	1,056	18.0
1,312	17.8	1,188	18.0
1,692	17.7	1,414	17.9
2,381	17.7	1,845	17.9

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[a] Rix, E. A., Compressed-air calculations: Address before the mining association of the University of California, Feb. 19, 1908; reprinted in Compressed Air Mag., June, 1908, p. 4894.

[b] Throughout this report when air pressure is mentioned the figures given are those above atmosphere; that is, gage pressure.

It will be observed that the table supports the statement made by Rix, and that even in spite of the increased final pressure, the values are somewhat less than the one he proposes. They also show that in machines of large capacity, proportionally less power is required.

The following table, based on published figures, shows the amount of power required or provided per 100 cubic feet of free air actually compressed at several turbocompressor plants:

Power consumption of turbocompressors.

Pressure, pounds per square inch.	Capacity, cubic feet of free air per minute.	Rated horsepower of motor or engine per 100 cubic feet of free air when compressing to stated pressure.	Actual horsepower required to compress 100 cubic feet of free air per minute to stated pressure.
90	4,600	21.8	
118	21,250	18.8	17.0
135	20,000		18.5
170	22,000	18.2	

CAPACITY.

The capacity of compressors is rated in free air,[a] and in reciprocating machines is based equally on speed and on the piston displacement—the number of cubic feet of cylinder space swept by the piston each minute at the given speed. However, there are unavoidable losses in volume because of clearance, piston speed, leakage, and expansion, the sum of which may amount to as much as 30 per cent of the rated capacity in a single-stage compressor at 100-pound pressure. The capacity of turbocompressors is based on the volume of free air drawn into the intake per minute. Although some of the more carefully designed reciprocating compressors may give a volumetric efficiency as high as 90 per cent, for compressors such as are customarily employed in power plants for tunnels 80 per cent is more likely to be the figure, and although the tables in manufacturers' catalogues of air drills are in the main fairly accurate for new drills, the air consumption may be greatly augmented as parts become worn. Moreover, provision must be made for pipe-line leakage and for air required by drill sharpeners, blacksmith forges, and an extra small drill, if one is used, for blocking and trimming. It is therefore most desirable to have the air compressors considerably oversized as based on catalogue rating, and in tunnel practice the oversize usually ranges from 100 to 150 per cent. The following table shows a comparison between the rated compressor capacity and the desig-

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[a] Free air is air at atmospheric (14.7 pounds) pressure and at a temperature of 60° F.

nated air consumption for the drills employed at several tunnels and adits:

Relation between compressor capacity and air consumption of drills.

Tunnel.	Catalogue figures for compressor.		Drills.		Altitude.	Oversize of compressor.[a]
	Speed.	Capacity.	Number in heading.	Stated air consumption.		
	R. p. m.	*Cubic feet per minute.*		*Cubic feet per minute.*	*Feet.*	*Per cent.*
Carter	150	868	2	230	9,000	280
Laramie-Poudre	165	602	3	250	8,000	140
Elizabeth Lake	160	736	3	185	3,000	300
Lucania	150	544	3	250	8,000	120
Marshall-Russell	175	487	2	200	8,000	140
Mission	190	247	1	100	1,200	150
Rawley	175	427	2	190	10,000	120
Snake Creek	165	680	2	300	6,000	125
Strawberry	175	427	2	300	7,000	40

[a] Not including drill sharpeners, forges, or leakage in pipe lines.

The loss of effective compressor capacity through leaky pipe lines is not fully realized at many plants and no steps are taken either to determine the amount of this waste or to prevent it. If the pipe lines are constructed with great care and covered so as to protect them from accident or from extremes of temperature, the loss by leakage may be almost negligible, but if they are not well built or if the surface lines are exposed to injury from numerous causes—not the least being diurnal and seasonal variation in temperature—and those underground are liable to be struck by falling rock or derailed cars, the leakage is likely to be considerable, and the lines should be tested at short intervals in order that the loss may be ascertained and stopped. If reciprocating compressors, driven either by steam or water power, are used, the leakage can be easily ascertained by closing all of the outlet pipes from the line and noting the number of strokes per minute necessary to maintain the desired pressure; but if turbocompressors are used, unless the output at different speeds and pressures has been carefully determined and tabulated, the leakage can best be ascertained by stopping the compressor when the receiver and lines are filled, allowing the pressure to drop to, say, 50 per cent, then starting the machine and noting the time required for the pressure to reach the original point. Although this course does not give exact results, still it will furnish a useful index to the rate of leakage, and is so simple that one might think it ought to be in general use. Unfortunately, the habit of workmen, especially the "chain gang," seems to be to assume that all air lines are much freer from leaks than they really are. A few years ago at a large western mine, where a great number of drills, drill sharpeners, and pumps were driven by compressed air, when the required pressure could not be maintained, bids for a new and expensive compressor were asked for, but the management thought of testing the pipe lines by the method indicated above, and it was discovered that 1,100 horsepower was being used to supply the loss by leakage. In this case the "chain gang," instead of the machine shop, "got busy" and in a week the waste of air was so reduced that instead of another compressor being bought one of the largest of those in use was shut down.

TYPES.

Reciprocating air compressors may be divided into two general types: "Straight line," sometimes called "tandem," and duplex. Either type may be single stage if the air reaches its final pressure in one cylinder, or multistage if the compression is completed in a second, third, or even a fourth cylinder.

STRAIGHT-LINE COMPRESSORS.

In a tandem compressor, if it be driven by steam or an internal-combustion engine, the power and air cylinders have a common piston rod and the power is applied in a straight line. The flywheels, of which there are usually two, may be at either end or between the cylinders and are connected to the piston rod by a crosshead and ordinary connecting rods. If the tandem compressor be driven by electricity or water, practically the only change is the omission of the power cylinder.

DUPLEX COMPRESSORS.

A duplex compressor consists of two tandem compressors placed side by side and having between them a flywheel on a common shaft. The two sides are connected to the flywheel shaft by cranks set at 90°, so that when one side is encountering maximum resistance the other is working under the lightest load. Many different combinations are possible. The steam cylinders[a] may or may not be compounded, and the air cylinders may be single stage or multistage. Again the steam cylinders may be omitted and the power transmitted to the machine by a belt or by a direct-connected motor.

TURBOCOMPRESSORS.

The working principle of a turbocompressor (fig. 5) is that of a reversed turbine, air, instead of water or steam, being the fluid acted upon. Such a compressor consists essentially of a revolving impeller, not unlike that of some centrifugal fans, surrounded by stationary discharge vanes supported in a suitable casing. The

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[a] Internal-combustion engines have not as yet been applied to this type of compressor.

discharge vanes recover the major part of the energy that exists as velocity in the air leaving the impeller, which is roughly almost one-half of the total energy supplied from the driving machine, and convert this velocity into available pressure. In a centrifugal fan, which has no such vanes, this energy is lost as heat produced by eddies and friction; hence it is not difficult to see why a turbocompressor is more efficient. Single-stage turbocompressors are employed chiefly in metallurgical plants, at blast furnaces and cupolas, and could be used for mine ventilation; but, if a high pressure is required, such as that needed by rock drills and other pneumatic machinery, several impeller units are mounted in series on a common shaft within a common casing, the air from the first set of discharge vanes going to the intake of the second impeller, and so on. Compressors producing a pressure of 170 pounds and having as many as 29 stages have been constructed, but if so many stages are employed the impellers are usually mounted in groups of four to ten.

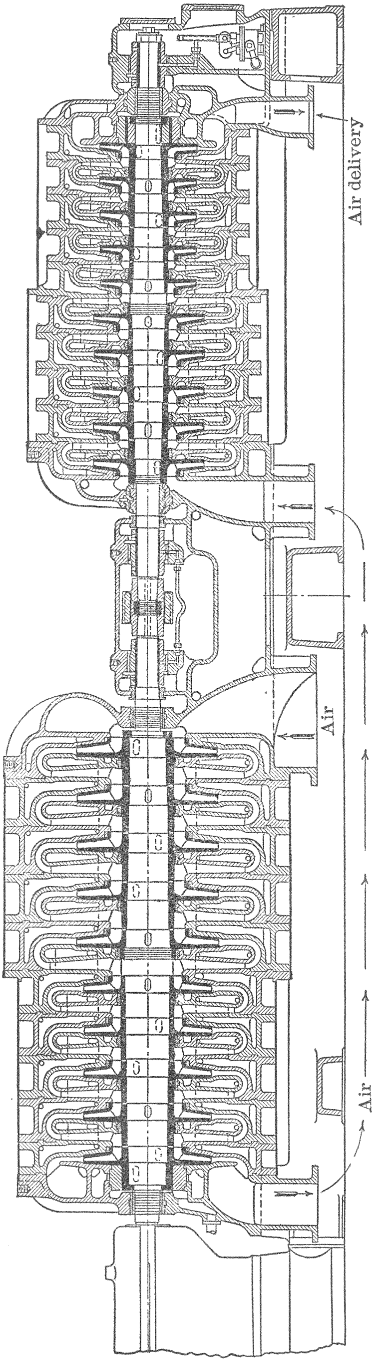


FIGURE 5.—Section of a turbocompressor.

Turbocompressors are just beginning to be made in this country, but they have been in use for several years in Germany, where their design and manufacture have already reached a high degree of perfection. The first large turbocompressor was built in 1909 for the Reden mines near Swarbrucken. It is driven by a 1,000-horsepower, mixed-pressure steam turbine at 4,200 revolutions per minute, and compresses 4,600 cubic feet of free air per minute to a gage pressure of 90 pounds to the square inch. Rather recently six motor-driven compressors of this kind were built in Germany to supply power for the Rand mines in South Africa, each of them being driven by two 2,000-horsepower synchronous motors running at 3,000 revolutions per minute. The compressors have a rated capacity of compressing 21,250 cubic feet of free air per minute at 68° F. to 118 pounds pressure per square inch. In a test, when compressing 23,750 cubic feet of free air per minute to 100 pounds pressure, the energy of consumption per 100 cubic feet of free air was 17 horsepower, and the highest isothermal efficiency obtained was 67.04 per cent. The first large turbocompressor built in this country went into service in May, 1911, and has been in continuous operation ever since; it is driven by a steam turbine at 4,700 revolutions per minute and has a capacity of 3,500 cubic feet of free air per minute delivered at a pressure of 105 pounds per square inch.

For material relative to other of the numerous makes of air compressors the reader is referred to the trade catalogues of the manufacturers, who will be glad to supply information and to render assistance in the selection of a compressor.

COMPARISONS.

The chief advantages of the straight-line compressor are strength, simplicity, compactness, and ease of installation. It is usually self-contained, being mounted on a single bedplate, and requires relatively inexpensive foundation. The frictional losses in a good machine of this type are not large, and it may have a fairly good power economy at or near full load with moderate pressures. These features make it advantageous for less accessible plants or those that are somewhat temporary.

A great advantage of the duplex type is the facility with which either steam or air cylinders may be "compounded" without increasing materially the number of parts. Hence the duplex type can take advantage of the great saving in power resulting from compound steam cylinders as well as of the economy resulting from two-stage air compression. Practical experience with the two types of machines fully confirms the theoretical investigations of their comparative efficiency, and carefully conducted tests extending over long periods of time have established the economical superiority of the duplex type. This type, also, if properly designed, shows little, if any, greater mechanical losses through friction, etc., than the straight line, and it is much more easily regulated under varying loads. Most duplex compressors are now made with a substantial subbase that gives them a strength and rigidity comparable with those of the straight-line type, reduces the outlay for foundation, and thus meets some of the conditions that have until recently been so much in favor of that type. The result is that, with perhaps a half dozen exceptions, the air compressors at the tunnel plants examined were of the duplex type.

The entire absence of valves, reciprocating parts, and sliding friction in the turbocompressor, together with its freedom from vibration, its high capacity in proportion to weight and to floor space occupied, and its ability to take advantage of the high rotative speeds of electric motors and steam turbines, is certain to bring it into general use. Using live steam, condensing or noncondensing turbine-engine turbocompressor units can compete successfully with the highest grades of reciprocating-engine compressor plants; moreover they can successfully utilize exhaust steam from engines, pumps, or other apparatus, which forms one of the cheapest possible sources of power, because, by utilizing steam that would otherwise go to waste, the cost of fuel is practically nothing.

Another advantage, and one that may easily be overlooked, is the fact that the turbocompressor practically eliminates the danger of explosions in air receivers and pipe lines. The cylinders of piston compressors must be lubricated, and the lubricating oil, which becomes finely divided, has ample opportunity to mix with the air being compressed and to be carried with it into the receiver, but in the turbocompressor only the bearings require lubrication, and to these the air being compressed has no access.

When an electric motor or water wheel is the source of power, the turbocompressor may be easily connected to either without loss from speed reduction and friction, a feature that renders the combination highly desirable. The turbocompressor is readily adapted to automatic control and may be regulated for the delivery of a constant volume or constant pressure, as required. Its efficiency is maintained over a wide range of load within a few per cent of the maximum, and the efficiency does not decrease with continued service.

REGULATION.

STEAM-DRIVEN COMPRESSORS.

Although changing the load changes the speed of any steam-driven machine, such change, especially with high air and steam pressure, is to the advantage of the straight-line compressor, because this type will not run satisfactorily at low speeds on account of the insufficient momentum of the flywheel. To avoid stoppage, either the steam cut-off must be lengthened, involving a loss of steam as the machine speeds up to supply more air, or the steam must not be decreased below a fixed limit, and when the demand for compressed air falls below that supplied regularly by the machine the excess must be permitted to escape through a safety valve. Either of these operations entails loss of power. For this reason the straight-line compressor can not operate economically much below 40 per cent of full load.

In the matter of regulation of steam-driven compressors the duplex has an unquestioned advantage over the straight-line type. The quartered cranks, in addition to minimizing strains and reducing extremes, enable one cylinder to come to the help of the other just at the time when that help is most beneficial. There can be no dead center and the machine will run so slowly as hardly to turn over if the compressed air in the receiver is not being drawn upon, and will speed up rapidly as the demand for air increases, no change in the cut-off being necessary. The duplex machine, therefore, has about the same steam economy over the full range of load, without any loss of compressed air at the safety valve.

The regulation of the turbocompressor when steam-turbine driven is merely a matter of controlling the steam admitted to the turbine.

WATER-DRIVEN COMPRESSORS.

Compressors driven by impulse wheels may be regulated by several devices, including the deflecting nozzle, the needle nozzle, and the cut-off. The deflecting nozzle is provided with a ball-and-socket joint and is controlled by air-receiver pressure in such a manner that a part of the stream of water may be shifted on or off the buckets of the wheel, thus increasing or decreasing the power developed. The nozzle may be stationary and a steel plate may be made to deflect the stream of water. The needle nozzle is merely a discharge valve in which the movement of a conical needle diminishes or increases the amount of water passing through. The cut-off, also, regulates the quantity of water discharged, a tight-fitting plate being shifted across the nozzle tip. The deflecting devices can vary the power rapidly, but, of course, waste water; the reverse is true of the other devices. For tunnel plants, however, as water economy is rarely an essential consideration and variations in load are frequent and sudden, the deflecting devices are most suitable.

ELECTRICALLY DRIVEN COMPRESSORS.

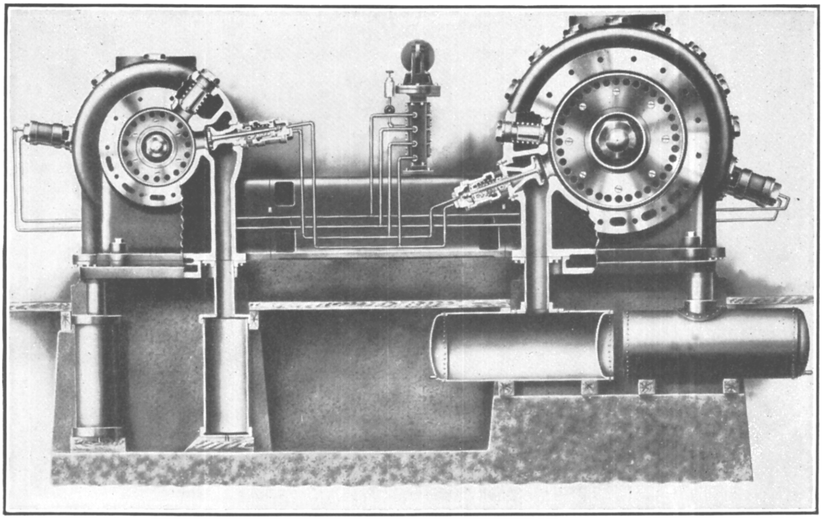
In any reciprocating machine the volume of air compressed varies with the number of piston strokes per minute. In the turbine compressor this volume depends on the rotative speed. In electrically driven reciprocating compressors, whether direct-connected or belted to a motor, the speed has to be reasonably constant and can not be varied to meet fluctuations in the demand for air, and as economy obviously forbids the discharge of excess compressed air through a safety valve, "unloaders" must be provided to overcome the difficulty.

Unloaders.—The more common type of unloader limits the amount of air admitted to the compressor. A valve in the free-air intake pipe is controlled by the pressure in the air receiver, so as to throttle the admission of air when the load is light, and allows more to enter when the demand for air increases. The device may be employed successfully with turbocompressors, but with riciprocating machines it has objections, because when the machines are running with a partly throttled inlet, the air drawn into the cylinder, being of smaller volume, is rarefied and on the return stroke of the piston is compressed through a greater range of pressure, so that the temperatures are higher than ordinary and may reach unsafe limits, especially if the terminal pressure is great. This feature is not so important with turbocompressors, because the temperatures, on account of the repeated cooling between stages, never become so high as in reciprocating machines.

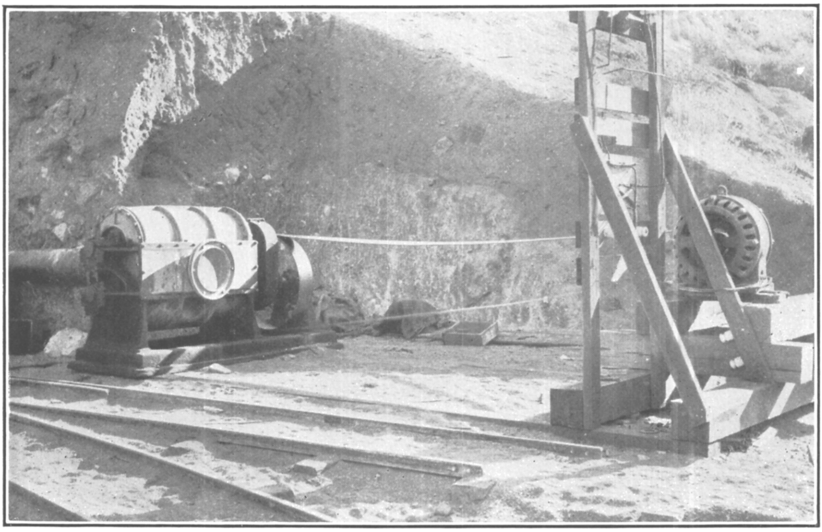
On some piston compressors the unloader employed is of almost exactly opposite type, consisting of a device for holding the intake valves open whenever the air pressure reaches a predetermined point; it avoids the high temperatures developed with the type previously mentioned.

In one type of unloader for reciprocating machines the excess air is forced automatically into clearance tanks, the process being controlled by a predetermined receiver air pressure. A view of this device, partly in section, is shown in Plate II, *A*. Under normal full load the controller is inoperative, but at less than full capacity some of the compressed air is forced into the tanks instead of through the discharge valve, thus reducing the output. On the return stroke this air expands and returns its stored energy to the piston. There are 8 tanks in all, and five equal and successive unloading stages are possible by throwing in respectively 1, 2, 4, 6, or all of the tanks. The regulation is said to be unaccompanied by shock from sudden variations in load, and heating from excessive compression ratios is avoided. In fact, as there is a little radiation from the clearance tanks, the air is probably returned to the cylinder slightly cooler than when it left.

Another method of unloading is by holding open the discharge valves of the compressor, and permitting compressed air instead of free air to fill the cylinder as the piston retreats so that the pressure on both sides of the piston is balanced. Although this procedure unloads the compressor completely it has a serious drawback. As the load is resumed, the balance of pressure is disturbed, one side of the piston being subjected to something less than atmospheric pressure, the opposite side being exposed to the full pressure of air in the receiver, the difference in pressure being thrown on the piston instantly and maintained throughout the entire stroke. The resulting heavy strain on the compressor prohibits the use of this unloader except in the smaller sizes. Still another type releases the partly



A. ELECTRICALLY DRIVEN AIR COMPRESSOR WITH FIVE-STEP CLEARANCE REGULATOR.



B. VENTILATING BLOWER USED AT LOS ANGELES AQUEDUCT.

compressed air during its passage from the low to the high pressure cylinders, but little can be said for this method except that it is not so wasteful as releasing high-pressure air.

HEAT FACTORS.

HEAT PRODUCED.

Heat is produced during the compression of air, the rise in temperature being largely dependent upon the ratio of the initial to the final pressure. For instance, if air at 60° F. be compressed in a single stroke from atmospheric pressure to 100 pounds gage, the temperature attained would be 485° F., assuming no loss by radiation during the process. On the other hand, under the same conditions, if the final pressure were only 25-pounds gage, the air would be heated to only 233° F., and if it were then cooled again to 60° F. and further compressed from 25 pounds to 100 pounds gage the final temperature would be approximately 250° F. The effect of the increase in temperature is to cause the air to expand to a larger volume; hence more work is required to compress it. If the air could be used at once to operate a motor, before any of the heat escapes through radiation, etc., this work could be obtained again from the air; but as in mining work the heat is almost without exception entirely dissipated in the pipe line before the air reaches the drills, the production of heat during compression entails a serious loss of power.

DANGERS OF HIGH TEMPERATURES.

Aside from the item of power waste, the temperature reached during compression has an important bearing on the subject of explosions in air lines. It can readily be imagined that if the discharge valves are not working properly and some of the highly heated compressed air is allowed to reenter the cylinder with the fresh intake air, compression may begin at a temperature much higher than normal. In this case, even with two-stage machines, the final temperature of the compressed air may be gradually built up from 250° to 500°, 600°, or even higher. The temperature is often high enough to volatilize lubricating oil, the vapors of which, mingling with the air, may form an explosive mixture. If the temperature then becomes high enough to ignite the mixture an explosion must result.

REMOVAL OF HEAT.

The ideal way to prevent the evil effects of heat would be to devise some means of removing it from the air as fast as produced during compression. This is fairly approximated in turbocompressors. With piston compressors such a course is unfortunately impossible in practice, but various devices partly accomplish the result. A familiar one is to surround the cylinder with a jacket of cooling water, the piston also being sometimes cooled in this way. But when one considers that air is a very poor conductor of heat, and that at the time when it is hottest it occupies only the minimum volume in one end of the cylinder, and even then for only a short time, it will readily be seen that this method can not be very effective. In some modern compressors the inlet valves are placed in the piston and the discharge valves in the ends of the cylinders instead of in the heads, thus permitting the heads to be fully water jacketed, a practice that is to be most highly commended. As water jacketing is the only means used to cool the air during single-stage compression, such machines are not economical of power.

In two-stage compressors, however, a part of the heat is actually removed during the process of compression by means of intercoolers provided for this purpose. The air is only partly compressed in the first cylinder, the heat produced being practically all removed during the passage of the air through an intercooler on its way to the second cylinder. By removing the heat in the intercooler, the final temperature of the air is kept much lower than with single-stage compression; hence there is less expansion of the air to be overcome, resulting in a consequent saving of power. In a properly designed two-stage machine compressing to 100 pounds gage this saving of power is approximately 13 per cent, and it increases with the higher terminal pressures. If the pressure is less than 80 pounds the saving is hardly great enough to be a serious consideration and single-stage machines are customarily employed in such cases, but for pressure higher than 100 pounds, two-stage compression is imperative because of the high temperatures that are otherwise produced. As shown by the following table, the pressure ordinarily employed in tunnel plants ranges from 80 to 120 pounds, averaging about 100 pounds:

Compressed-air pressures at different tunnel plants.

Plant.	Pounds.	Plant.	Pounds.	Plant.	Pounds.
Carter	112	Marshall-Russell	110	Roosevelt	110
Central	120	Mission	100	Siwatch	80
Gold Links	100	Moodna	95 to 100	Snake Creek	110
Gunnison	90	Newhouse	110	Stilwell	100
Laramie-Poudre	120	Nisqually	90 to 95	Strawberry	85
Mauch Chunk	100	Rawley	100	Utah Metals	110
Los Angeles Aqueduct	100	Raymond	90	Walkill	110
Lucania	115	Rondout	100	Yak	90
				Average	102

Because of the many stages required with turbocompressors when delivering air for use in drilling, the difference between the pressures of the air on entering and leaving any one stage is extremely small compared with the pressure of two-stage reciprocating machines. Hence the resulting increase of temperature in any one step is not great, and the comparatively small amount of heat generated can be effectively removed by the use of a suitably designed water jacket. Some idea of the efficiency obtainable with such a cooling system may be had from considering the fact that the air delivered into the receiver at 105 pounds pressure from the turbocompressor mentioned on page 69, has a temperature of only 120° F.

INTERCOOLING.

The efficiency of two-stage compression is dependent upon the intercooler. It is an essential part of this type of machine and usually consists of a shell, generally cylindrical in shape, containing a number of tubes, similar to those in a tubular boiler, through which cold water is made to circulate (fig. 6). The heated air from the low-pressure cylinder enters near one end, passes through the nest of tubes, its passage being obstructed by baffle plates to insure the maximum contact between the air and the cooling surface, and is delivered at much lower temperature to the high-pressure cylinder at the other end. The success of the intercooler depends upon several considerations. In order that the least dependence need be placed upon the heat conductivity of the air itself, which is notably poor, the intercooler must subdivide the air completely and insure that the maximum amount is thrown in contact with the cooling surfaces. This is accomplished by properly spaced water tubes and baffle plates. At the same time the cross section of the cooler must not be too small, lest the velocity of the air past the cooling surface be so great that sufficient time be not allowed for the water to absorb all the heat. It is desirable also to have the water and air flow in opposite directions in order that the final cooling of the air may be effected by the entering, and consequently the coldest, water. Theoretically the cooling surface should be sufficient to absorb all the heat in the air passed

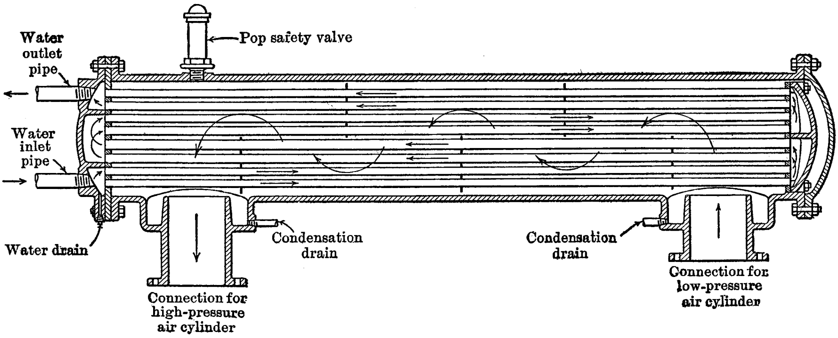


FIGURE 6.—Section of an intercooler.

over it, and thus reduce the temperature to that at which the air entered the low-pressure cylinder, but even in good practice, owing possibly to mechanical difficulties, intercoolers usually fail to do this within 5 to 10 or even 40 degrees.

MOISTURE FACTORS.

The intercooler assists also in removing water from the air. Normal air always contains some water vapor, the amount absorbed being determined by the temperature and pressure. In the air compressor the vapor would be condensed out of the air as the volume is reduced were it not that the increase in temperature more than offsets the effect of the decrease in volume, so that actually the vapor-holding capacity of the confined air is slightly increased. As the air passes through the intercooler the temperature is lowered greatly and the air gives up moisture which is precipitated in a finely divided state and does not settle for some time; hence only a part of it can be collected in the intercooler and drawn off through the drains provided unless the intercooler is so made as to act also as a mechanical separator. Some moisture is swept along with the air to the higher pressure cylinder and revaporized by the temperature therein.

ACCESSORIES.

PRECOOLERS.

Cooling the air before its admission into the air compressor may assist in removing some water from it, and there are a number of precooling devices. One[a] improvised device consists simply of a number of odd pipes set between two wooden boxes. The pipes are wrapped with cloth; arrangement is made to have water drip on them constantly, so that the air is cooled by evaporation as it is drawn through the pipes from one box to the other on its way to the compressor intake. At a plant in Johannesburg the air for the compressors is obtained through a subway leading to the center of a building with air-tight roof and floors, and with walls of constantly wetted cocoa matting. At another plant the sides and roof of a similar structure were covered with burlap, both inside and out. A cooler of this type also filters the dust and grit that might seriously injure the cylinder or piston of the compressor, and can not be too strongly recommended in dusty situations. Precooling the air, and thus reducing the volume at a given pressure, increases the capacity of the compressor; hence a greater weight of air is drawn in and compressed at each stroke.

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[a] Described in Engineering and Mining Journal, issue of Nov. 27, 1909, p. 1081.

AFTERCOOLER.

The aftercooler[a] is not generally employed in tunnel plants. In cooling the air as this comes from the high-pressure cylinder it precipitates some of the water vapor and also reduces the volume of the air and practically eliminates the danger of explosion in the air line. The water vapor is usually so finely divided that all of it does not at once fall out because of the reduced temperature, part being swept along with the air and deposited both in the air receiver and in the pipe line. There should therefore be provision for draining this water. The reduction in volume is somewhat problematical and is probably not a serious consideration.

AIR RECEIVERS.

The air receiver, a cylindrical shell of steel provided with inlet and outlet pipes and usually a safety valve, is supposed, according to the popular notion, to store, cool, and dry the air and to equalize irregularities in its production and use. In actual practice, however, the receiver probably does little more than partly to equalize irregularities. As the receiver ordinarily used in tunnel plants rarely has a capacity greater than one minute's run of the compressor it can not possibly furnish much storage space, and as the air in the receiver is being renewed each minute, if the compressor is in operation, its velocity through the receiver must be enough to prevent much cooling. The air near the shell loses some heat, of course, but this loss is small compared to the heat in the mass of the air nearer the center of the receiver, so that the air leaves with a temperature only slightly, if at all, less than that at which it entered. Furthermore, as there is practically no cooling of the air there can not be much precipitation of water vapor. In practice, although most air receivers are provided with a drain of some sort, only a ridiculously small quantity of water is ever drawn off. On the other hand, some receivers have actually become combustion chambers by reason of oil and grease collecting on the inside of the shell and being ignited when the temperature of the air became high enough. However, the receiver and the pipe line, which may be considered as supplying auxiliary storage space, assist greatly in equalizing the pulsations of the air delivered from the compressor and of that used by the drills, and in this way it reduces the strains on the compressor. By regulating the flow it does not permit the air to attain a high velocity in the pipes, and hence power is saved, as the friction losses increase greatly with the velocity. To obtain the maximum benefit from this factor a second receiver is often installed

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[a] In design and principle the aftercooler is practically the same as an intercooler, and when used it is generally placed between the compressor and the air receiver.

as near as possible to the place where the air is to be used. The second receiver assists materially in maintaining a steadier air pressure at the drills and makes it possible to use a smaller pipe line. A tubular boiler, which may often be bought cheaply at second hand, makes an excellent receiver and a very efficient cooler. With a vertical tubular boiler it is necessary only to remove the fire and ash doors to provide for ventilation, whereas a horizontal tubular boiler should be placed on an incline sufficiently steep to insure a rapid draft of outside air through the flues.

DRAINS.

As practically the entire cooling of the air after it has left the compressor takes place in the pipe line, most of the water is precipitated there and causes serious inconvenience in several ways. During cool weather, through continued deposition and freezing, the pipe line may become closed altogether or so restricted as to cause serious drop in pressure or loss of power; or the water getting into the exhaust from the drills may prevent their operation through freezing at the low temperature of the expanded air. The obvious remedy is to remove the water, which is done by draining the low places in the line where the water collects. This can be accomplished automatically by the use of any good float-design steam trap, but if the pipe is exposed to low temperatures the trap should be placed in a small pit or otherwise protected to prevent freezing. If necessary further provision for the elimination of moisture from the compressed air and water from the pipes can be had by placing in the line any high-class standard steam separator, fitted with an automatic trap, as described above. For equally satisfactory results a larger size of separator is, however, necessary than for steam, as the carrying power of the more dense compressed air is greater than that of steam.

SUMMARY.

A brief summary of the factors that enter into the problem of selecting an air compressor follows. The power required for both reciprocating and turbine machines is approximately 18 to 20 brake horsepower for every 100 cubic feet of free air compressed to 100 pounds gage. The values given in trade catalogues for reciprocating compressors are generally a little below this figure, but it is a safe one to use in estimates. Such compressors ordinarily have a volumetric efficiency of approximately 80 per cent, and as they are rated on the basis of free air and as it is necessary to make allowance for loss due to clearance, etc., provision for increased air consumption above the catalogue rating for drills as they become worn, and for air used in sharpening machines and forges, must also be made with either reciprocating or turbine machines; hence it is advisable to select an air compressor considerably oversize. In practice the oversize, based upon drills only, ordinarily ranges from 100 to 150 per cent.

Of the two types of reciprocating compressors the duplex is preferable to the usual straight line, in spite of the latter's simplicity and easier installation, because of the former's more economical and efficient use of power and the facility of its regulation, especially when driven by steam under high pressure.

As the air pressure at tunnel plants is rarely below 80 pounds, in three out of every four being 100 pounds or greater, two-stage compression is desirable because of its economy of power. Indeed, such compression may be imperative because of the high temperatures that might otherwise be attained.

Although the manufacture of turbocompressors is just beginning in this country, they possess a number of advantages, especially for use with steam turbines and other rotary engines operating at high speeds, and will doubtless be more generally used in the future. Their development should therefore be closely watched. Steam-driven compressors are regulated by varying their speed; but, as in some power-driven machines, the speed is necessarily constant, other means, of which the throttle inlet and the clearance controllers are the two most used, must be provided for that purpose. Heat is produced during compression and, by expanding the air, causes loss of power. Some of this loss is obviated in stage compression by removing the heat during its passage through an intercooler between the cylinders. The numerous stages in the turbine machine enable this heat to be removed effectively by water-jacketing. Another evil attributable to this heat is the danger from the explosion of volatilized lubricating oils; but in the turbine machine this danger is eliminated because there are no sliding surfaces to require lubrication. Among the accessories that are designed to prevent or neutralize the effects of heat in piston machines are the precooler, the intercooler, the after-cooler, and the air receiver. The air receiver also equalizes the pulsations of the air and reduces friction losses. The devices mentioned assist also in freeing the air from water which often causes serious inconvenience. The major part of the water is apt to be deposited in the pipe line, however, if this is the coldest part of the system, and provision must be made for its removal.

VENTILATION EQUIPMENT.

MACHINERY.

Either "blowers" or fans are employed ordinarily for ventilating tunnels and adits. In every revolution of a positive blower a certain quantity of air is trapped between the impellers and the inclosing casing and has no means of escape except through the discharge pipe, as is indicated in figure 7. For this reason blowers are often styled "pressure" blowers and "positive-blast" machines. Plate II, *B*, shows one of these blowers in operation on the Los Angeles Aqueduct.

If fans are employed in tunnel ventilation they are almost without exception of the centrifugal type, the disk form, similar to the ordinary desk fan, being rarely used. In the centrifugal fan the air enters near the center in a direction approximately parallel to the axis of the shaft and is forced by the centrifugal action of the rapidly revolving blades toward their periphery, where it is discharged. This type has many modifications designed to prevent loss of efficiency through friction as the air strikes the back plate and changes direction or to prevent eddies due to the greater density of the air at that point caused by its momentum upon entering the fan.

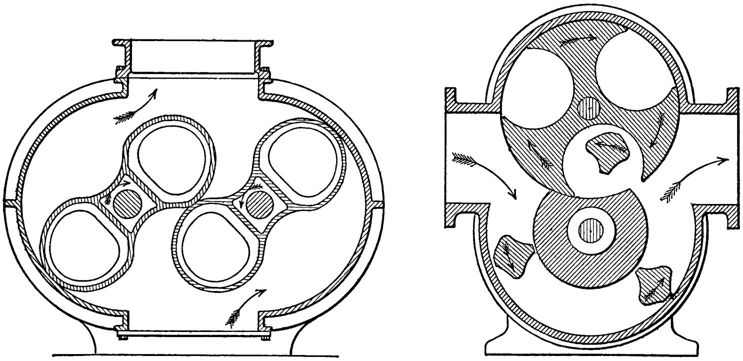

FIGURE 7.—Cross sections of pressure blowers, indicating the course of the air.

Turbocompressors in which by one, two, or even several stages air can be delivered at any required pressure have been employed as blowers for blast-furnace and foundry work at a number of places. The capacities of those manufactured for this purpose thus far are too great for the requirements of tunnel work, but their greater efficiency as compared with centrifugal fans, and the possibility of designing them to obtain any required pressure, will doubtless soon lead to their being made in sizes suitable for tunnel work, in which they should have a large field.

At one tunnel vitiated air was removed from the heading by the use of a jet of highly compressed air which was directed into the ventilating pipe, but this method in addition to being expensive is inadequate, and is, therefore, not to be advised, except as a temporary expedient and for short distances. On short levels and crosscuts, however, or on larger work pending the installation of more expensive and efficient machinery, jet blowers can often be used to good advantage. They can be operated by either compressed air or water under pressure and, though far from being as efficient as the mechanical types of ventilating machinery, will in many cases perform an extremely useful function. Jet blowers can frequently be used to move large volumes of air for short distances against low frictional resistances with good results, and their extreme economy in first cost makes them an excellent accessory in preliminary work.

DIRECTION OF CURRENT.

By a proper adjustment of the ventilating pipe the fan or blower ordinarily installed for tunnel work may be made to exhaust the air from or deliver it to the heading. One of the chief advantages of exhausting the air is that the dangerous gases and smoke produced in blasting are removed from the tunnel or adit promptly, and it is therefore unnecessary for the workmen to pass through the thick bank of smoke that would otherwise travel very slowly to the portal. On the other hand, when fresh air is blown in, it passes very much faster through the pipe and is cooler and fresher than if it has worked its way slowly in through the tunnel or adit and become heated from contact with the walls and contaminated by odors from the floor. The men, therefore, feel more comfortable and are able to do better work when this method is employed. The advantages of both methods, however, may be readily obtained by an arrangement of pipes similar in principle to the one shown in figure 8, which permits the air to be exhausted for a few minutes after blasting by opening gates *a* and *b* and closing *c* and *d*, assuming the current through the fan or blower to be in the direction of the arrow, while at other times by reversing this arrangement air may be forced into the heading. Table 5 shows the direction of the air current at various tunnels visited, from which it may be seen that, almost without exception, it is customary to exhaust the smoke, after blasting at least,

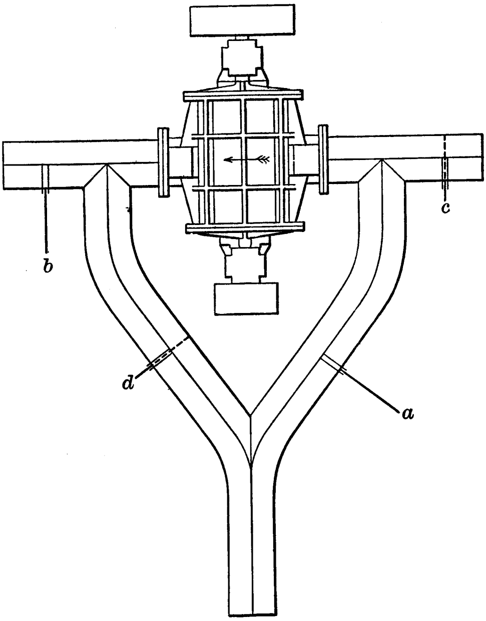

FIGURE 8.—Arrangement of gates and pipes for changing direction of ventilating current.

although at many places the ventilating current is reversed at other times. This arrangement is reported as giving excellent results, and its use is strongly recommended by the authors.

TABLE 5.—*Direction of air current at various tunnels.*

Tunnel.	Ordinary current.	Current after shooting.
Carter	Exhaust	Exhaust.
Central	do	Do.
Gold Links	do	Do.
Gunnison, east portal.	do	Do.
Gunnison, West Portal.	Pressure	Exhaust for 2 hours.
Laramie	Exhaust	Exhaust.
Lausanne Paudre.	Pressure	Pressure.
Los Angeles Aqueduct:		
Elizabeth Lake.	do	Exhaust 20 to 25 minutes.
Little Lake	do	Exhaust for 1 hour.
Grapevine	do	Exhaust ½ to 1 hour.
Lucania	Exhaust	Exhaust.
Marshall-Russell	Exhaust	Exhaust.
Mission	Pressure	Exhaust ¾ to 1 hour.
Newhouse	Exhaust	Exhaust.
Nisqually	do	Do.
Rawley	do [a]	Do.
Raymond	Pressure	Exhaust for 2 hours.
Rondout	do	Exhaust "for a while."
Roosevelt	Exhaust	Exhaust.
Siwatch	do	Do.
Snake Creek	do	Do.
Stilwell	do	Do.
Strawberry	do	Do.
Utah Metals	do	Do.
Wallkill	Pressure	Do.
Yak	Exhaust	Do.

[a] Intermittent.

CAPACITY.

There is no absolute method for determining the quantity of air needed to renew that vitiated by the respiration of men and animals working in tunnels. For coal mines many States have provided a legal minimum that ranges from 85 to 300 cubic feet per minute for each man and from 300 to 600 cubic feet for each animal. These figures, however, have practically no bearing on tunnel work, because in coal mines a much larger volume of air than that actually needed by the men must be supplied in order to dilute and render harmless the inflammable and dangerous gases given off from the coal. In many States the laws provide that even the requirements mentioned may be increased at the discretion of the mine inspector. Conditions in metal mines, on the other hand, are more closely akin to those in tunnels, but, unfortunately, most existing legislation merely stipulates that the ventilation must be "adequate." Arizona specifies the amount considered harmful by chemical analysis.

Richards[a] considers that the following air quantities are sufficient for proper ventilation in metal mining work:

Per light, 1 cubic foot per minute.

Per man, 25 cubic feet per minute.

Per animal, 75 cubic feet per minute.

The mining-regulations committee of the Transvaal, on the other hand, provide (for metal mines) a minimum of 70 cubic feet per man

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[a] Richards, R. H., Mining notes, vol. 2, 1905, p. 142.

per minute.[a] When a person is sitting in repose as in a theater or meeting hall, 20 cubic feet of fresh air per minute is considered adequate provision by engineers making a specialty of ventilation, but much larger quantities are of course required when the men are working. The following table giving the results of a test conducted by Bernhardt Draeger,[b] shows the amount of air breathed in the first minute after performing various kinds of work.

Quantity of air actually breathed in first minute after exertion.

Kind of work.	Subject A.	Subject B.	Subject C.	Average.
	Liters.[a]	*Liters.*[a]	*Liters.*[a]	*Liters.*[a]
Sitting 10 minutes	8.5	8.25	9	8.58
Walking 270 yards	10.5	11.3	11.7	11.2
Marching 550 yards	14.3	17.5	13	14.9
Running 270 yards	30	30	30	30
Rolling barrel weighing ¾ hundredweight	38	33	40.5	37.2
Running 550 yards	38	42	38	39
Race, 270 yards	[b] 52	[c] 61	[c] 59	[d] 51

[a] 1 liter = 0.0353 cubic foot. [b] Time, 40 seconds. [c] Time, 42 seconds. [d] Time, 41 seconds.

These figures give the amount of pure air actually inhaled and exhaled, but, of course, in order that the products of respiration may be diluted sufficiently for the air in the confined space of a tunnel to be kept pure, a much larger quantity must be supplied. Assuming that 20 cubic feet is sufficient for a man at rest, and applying the ratio deduced from Draeger's table, it would appear that the following volumes of air (in cubic feet per minute) should be supplied for ventilation if the forms of exercise indicated were undertaken in a small room or in a tunnel: Sitting, 20; walking, 26; marching, 35; running, 70 to 90; rolling barrel, 85; racing, 130.

Although some members of the tunnel crew, such as the shovelers, ordinarily work as hard as men running or rolling a barrel, there are times when the work of the drillers more closely approximates the exertions required in walking; so, taking everything into consideration, it would seem that 75 cubic feet per minute per man should be adequate provision for tunnel ventilation. Assuming that a horse or mule requires two to three times the air needed for a person, 150 to 200 cubic feet per minute should be furnished each. At mine adits where any attempt is made for even moderate proggress 8 to 15 men, and possibly 2 animals, are employed in or near the heading. Under these conditions 600 to 1,500 cubic feet of fresh air per minute would be required for purposes of respiration. It is true that some air is furnished by the exhaust from the drills, but

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[a] (Editorial), Carbon dioxide criterion for ventilation: Eng. and Min. Jour., vol. 90, Nov. 5, 1910, p. 899.

[b] Glückauf, vol. 40, 1904.

their action is intermittent and the supply never adequate, so that much dependence can not be placed upon it; on the whole, it is much better simply to ignore this possible source when deciding upon the capacity of ventilating machinery.

Although the supply mentioned above is sufficient for ordinary requirements, a much greater, and indeed the maximum, demand for ventilation occurs immediately after blasting, when it is obviously important to remove the gas and smoke quickly so that the men may resume work with the least loss of time. The volume to be removed depends largely upon the amount of explosive employed; for customary charges under normal conditions it would probably not vary greatly from 60,000 cubic feet, a figure representing the average result of practical experience at tunnels where information bearing on this question was obtainable. It is true that ordinarily the air is seldom contaminated by the blast for more than 150 feet from the face. In a heading with a cross section of 70 square feet the blast would have a volume of only 10,500 cubic feet, and it might appear that the removal of this amount of bad air would clear the tunnel. Such might be the case provided the smoke would be removed instantly, but this result is, of course, not attainable in practice. The readiness with which gases become diffused must be taken into consideration, especially as it is customary immediately after blasting to turn a jet of highly compressed air into the heading. Such a practice is necessary because, to avoid injury from flying rock, the ventilating pipe rarely extends nearer the breast than 100 feet; so to remove the gases they must be forced out of the extreme end of the tunnel into the influence of the suction of the ventilating pipe. The result is that as a part of the bad air is removed its place is occupied by fresh air, which quickly becomes contaminated. It is therefore necessary to remove nearly six times the amount of foul air to clear the tunnel. In order to be considered good practice under ordinary conditions removal should take place in 15 minutes, requiring an exhauster capable of removing 4,000 cubic feet per minute.

This capacity, however, is necessary for only a few minutes after blasting. It is desirable, therefore, to have the fan or blower so arranged that it can exhaust for a short time at full load and then be run at a lower speed and supply the heading with the smaller volume needed for respiration. Such an exhauster was used at the Laramie-Poudre Tunnel. It was directly connected to a water wheel and ordinarily removed approximately 1,300 cubic feet by running at 100 revolutions per minute, but immediately after blasting the blower was speeded up to 300 revolutions per minute, when it exhausted nearly 3,900 cubic feet per minute, clearing the heading usually in 15 to 20 minutes.

At the Rawley Tunnel an attempt was made to obtain the same result by operating the blower intermittently at or near full load. Although the operation of the blower or fan at full load for one-third of the time supplies the heading with an equal amount of air as when running at one-third capacity all the time, different results are obtained in practice. The purity of the air is not maintained so nearly constant with the intermittent system, and as the starting and stopping of the blower is usually dependent upon some man, its operation may be forgotten or neglected. This method of ventilating, therefore, can not be commended.

PRESSURE.

It is, of course, essential that the required amount of air be actually delivered to or removed from the heading. To do this pressure is necessary in order to overcome the frictional resistance to the flow of air in the pipe. This pressure must be generated by the fan or blower and may be either positive when forcing air in or negative when exhausting it. In either case the amount required depends upon the quantity of air passed and the size and length of pipe. Although the relations between these several factors are somewhat complicated, they are shown in the following formula advocated by G. S. Hicks, jr.:[a]

$$q=44.72\,d^{\frac{5}{2}}\sqrt{\frac{(P^2-14.7^2)}{lg}}$$

Where q = quantity of air in cubic feet per minute.
d = diameter of pipe in inches.
P = absolute initial pressure in pounds per square inch.
l = length of pipe in feet.
g = specific gravity of gas referred to air as unity.
p = required pressure in pounds per square inch.

From which, by transposing:

$$p=\sqrt{216.10+\frac{q^2l}{2000d^5}}-14.7$$

In which $p = P - 14.7$,
And $g = 1$.

It must be borne in mind that the formula is theoretical and does not take into consideration leakage, the extra friction due to elbows in the pipe, etc., but it is said to be based on good general practice

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[a] Engineering Tables, p. 12, P. H. & F. M. Roots catalogue 41.

for air and gas transmission and to give fairly satisfactory results. The table following, calculated from the formula, shows the pressure in pounds per square inch required to pass air through various sizes and lengths of pipe, assuming the quantity of air to be 4,000 cubic feet per minute—the value derived above as a suitable maximum capacity for a ventilating blower or fan.

Table 6.—*Loss of pressure when forcing 4,000 cubic feet of air per minute through various lengths and sizes of ventilating pipe.*

Diameter of pipe.	Length of pipe in feet.									
	1,000.	2,000.	3,000.	4,000.	5,000.	6,000.	8,000.	10.000.	12,000.	14,000.
Inches.	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*	*Lbs. per sq. in.*
6	20.20	....	....	....	....	....	....	....	....	....
8	6.75	11.80	....	....	....	....	....	....	....	....
10	2.52	4.69	6.65	8.45	10.10	....	....	....	....	....
12	1.06	2.05	2.90	3.87	4.71	5.52	7.06	8.63	9.87	....
14	.50	.98	1.45	1.90	2.32	2.77	3.60	4.40	5.16	5.90
16	.26	.51	.76	1.02	1.25	1.48	1.95	2.40	2.84	3.27
18	.14	.29	.43	.58	.70	.84	1.11	1.38	1.64	1.89
20	.085	.17	.25	.34	.42	.50	.67	.83	.99	1.15

If the pressure can not be increased to correspond with the length of pipe, the volume of air delivered is diminished, the size of the pipe remaining the same. This is illustrated in Table 7, in which a maximum pressure (P—14.7) of 1 pound per square inch is assumed.

Table 7.—*Maximum air capacities of pipes of different sizes and lengths when the initial pressure is 1 pound per square inch.*

Diameter of pipe.	Length of pipe in feet.									
	1,000	2,000	3,000	4,000	5,000	6,000	8,000	10,000	12,000	14,000
Inches.	*Cu. ft. per min.*	*Cu. ft. per min.*	*Cu. ft. per min.*	*Cu. ft. per min.*	*Cu. ft. per min.*	*Cu. ft. per min.*	*Cu. ft. per min.*	*Cu. ft. per min.*	*Cu. ft. per min.*	*Cu. ft. per min.*
6	685	485	....	....	....	....	....	....	....	....
8	1,410	1,000	815	705	630	575	....	....	....	....
10	2,465	1,745	1,425	1,235	1,105	1,005	870	780	710	660
12	3,890	2,750	2,245	1,945	1,740	1,590	1,375	1,230	1,125	1,040
14	5,720	4,045	3,300	2,860	2,560	2,335	2,020	1,810	1,650	1,530
16	7,985	5,645	4,610	3,990	3,570	3,260	2,825	2,525	2,305	2,135
18	10,720	7,580	6,190	5,360	4,795	4,375	3,790	3,390	3,095	2,865
20	13,950	9,865	8,055	6,975	6,240	5,695	4,930	4,410	4,025	3,730

Table 8 shows the calculated pressure required to overcome frictional resistance in passing a volume of air equal to the rated capacity of the ventilating machine through pipes of the sizes adopted to convey the air to the headings of some of the tunnels visited in the field work when completed to their proposed length.

TABLE 8.—*Pressure required to force quantity of air equivalent to catalogue rating of ventilating machine to proposed length of tunnel through pipe chosen.*

Tunnel.	Rated capacity	Diameter of ventilating pipe.	Stated length of ventilating pipe when tunnel is completed.	Pressure required.
	Cu. ft. per min.	*Inches.*	*Feet.*	*Lbs. per sq. in.*
Carter	1,560	15	7,600	0.41
Central	5,540	19	9,500	1.93
Laramie-Poudre	3,900	14½	9,200	3.34
Los Angeles Aqueduct:				
Elizabeth Lake	6,350	18	13,000	4.14
Little Lake	2,500	12	[a] 3,000	1.23
Grapevine	2,500	12	[a] 1,500	.63
Lucania	3,120	18½	12,000	.87
Marshall-Russell	4,160	12½	11,000	8.30
Mission	2,500	10	13,000	10.25
Nisqually	2,400	14	5,000	.87
Rawley	2,500	12½	6,200	2.02
Roosevelt	4,800	16½	15,700	4.38
Siwatch	1,560	10	5,000	1.94
Snake Creek	4,650	16	14,000	4.27
Strawberry	4,000	14	19,000	7.50
Utah Metals	4,880	12	11,800	13.24

[a] This division contains a number of tunnels. The distance given is the maximum.

It will be observed in these examples that the pressures needed ordinarily range from 1 to 5 pounds, 2 pounds being roughly the average. At two of the tunnels in this list, in order to obtain the extra pressure required to furnish sufficient ventilation, it was necessary to use a "booster," as it is called—that is, to install a second blower some distance within the tunnel; by operating the two together the pressure otherwise attainable would be virtually doubled. At the Mission Tunnel, the "booster" was situated near the 5,500-foot station. At the Strawberry Tunnel, both machines had been placed in the tunnel at the time of examination, the first one at 4,000 feet and the second at 11,000 feet. Two other tunnels at the time visited had not penetrated far enough to require such additional equipment, but doubtless extra provision for obtaining pressure will become necessary with continued progress.

SIZE OF PIPE.

The necessity for high pressures, and hence the use of "boosters," may be obviated in large measure by the choice of ventilating pipe having diameters of sufficient size. The difference between a 12-inch and an 18-inch pipe often exerts a great influence on the ventilation of the heading, but even aside from added cost, indiscriminate enlargement is undesirable, every inch of space in the average tunnel being required for other purposes.

Transposition of the formula on page 85 gives—

$$d=\sqrt[5]{\frac{q^2lg}{2{,}000(P^2-14.7^2)}}$$

which indicates the necessary diameter of the pipe in terms of the other variables. Table 9 following shows a number of solutions of this formula (assuming again that $q=4{,}000$ cubic feet of air per minute to be passed), and from it may be found the proper size of pipe for use with various pressures and distances.

TABLE 9.—*Diameter of pipe required in order to deliver 4,000 cubic feet of air per minute with different initial pressures and for various distances.*

Pressure.	Length of pipe in feet.									
	1,000	2,000	3,000	4,000	5,000	6,000	8,000	10,000	12,000	14,000
Ounces per sq. in.	*Inches.*	*Inches.*	*Inches.*	*Inches.*	*Inches.*	*Inches.*	*Inches.*	*Inches.*	*Inches.*	*Inches.*
1	21¼	24¼	....	....	....	....	....	....	....	....
2	18½	21¼	23	24¼	....	....	....	....	....	....
3	17	19½	21¼	22½	23½	24½	....	....	....	....
4	16¼	18½	20	21¼	22¼	23	24½	....	....	....
5	15½	17¾	19¼	20¼	21¼	22	23½	24½	....	....
6	15	17	18½	19½	20½	21¼	22½	23½	24½	....
8	14	16¼	17½	18½	19¼	20	21¼	22¼	23	23¾
10	13¼	15¼	16¾	17¾	18½	19¼	20¼	21¼	22	22¾
12	12¾	14¾	16	17	17¾	18½	19½	20½	21¼	21¾
Pounds per sq. in.										
1	12¼	14	15¼	16	16¾	17½	18½	19¼	20	20¾
1½	11¼	13	14	14¾	15½	16	17	17¾	18½	19
2	10¼	12¼	13¼	14	14½	15	16	16¾	17¼	17¾
3	9½	11¼	12	12¾	13¼	13¾	14¾	15¼	15¾	16¼
4	9	10½	11¼	12	12½	13	13¾	14½	15	15½
5	8¾	10	10¾	11¼	11¾	12¼	13	13½	14	14½
6	8¼	9½	10¼	11	11½	11¾	12½	13	13½	14
8	7¾	9	9¾	10¼	10¾	11¼	11¾	12¼	12¾	13¼

COMPARISON OF CENTRIFUGAL FANS AND BLOWERS.

Within certain limits the speed at which centrifugal fans are operated determines the volume of air delivered and the pressure generated, but the fans are incapable of producing pressures much greater than 1¼ pounds per square inch, and many of them are limited to 8, or even 5, ounces. Therefore as the frictional resistance against which air is to be forced or exhausted becomes greater through increasing lengths of pipe the pressure generated in the fan must be increased, by greater speed to the maximum limit at which the fan may be operated, and after that limit is passed the volume of air delivered necessarily becomes diminished with increased length of pipe. The positive blower, on the other hand, is capable of much higher pressures—8 pounds per square inch is easily attainable, and 15 pounds is possible with some makes—and in tunnel work, in which distances are as a rule great, the ability to deliver air against high resistance is an important consideration in favor of the blower. It operates also at a much lower speed when delivering the same volume of air against an equal pressure (1:10 is considered a fair ratio), and this adjustment lessens the wear and tear upon belts and machinery. Because of its higher pressure, the blower makes it possible to choose a smaller diameter of pipe, a factor worthy of consideration, because not only the initial cost but also the space occupied must be taken into consideration. The first cost of the fan, on the other hand, is less than that of the blower, and to economize room and obviate the wear on the belt the fan may be connected directly to electric motors, the greater cost of low-speed motors tending to prevent this possibility with a blower.

SUMMARY.

In most cases a machine of the blower type, capable of high pressure, is better adapted for tunnel ventilation in which resistances are apt to be great. For the best results the ventilating pipe should be so arranged that the direction of the air current may be alternated at will, exhausting for a short time after shooting and blowing for the remainder of the time. The blower should be adjusted to operate at two capacities—a lower one, supplying 600 to 1,500 cubic feet per minute, as determined by the number of men and animals, and a higher one capable of exhausting approximately 4,000 cubic feet per minute. This adjustment would make it possible under ordinary conditions for the men to resume work in the heading about 15 minutes after shooting. The pressure generated in the blower must be properly adjusted to the size of the pipe and the length of the tunnel in order that the determined volume of air shall be actually delivered to or removed from the heading. The pipe chosen should be of such size that only a moderate pressure at the blower is required, due consideration being accorded such items as cost of pipe and the space such pipe must occupy.

INCIDENTAL SURFACE EQUIPMENT.

In connection with the blacksmith and repair shops mention should be made of the drill-sharpening machine and the compressed-air meter. The use of the former is quite common, being employed at a majority of the tunnels visited; but the latter, so far as could be learned, has been used only in one or two places, although there appears to be a field for its employment in tunnel plants.

DRILL-SHARPENING MACHINES.

Several types of drill-sharpening machines are used in the United States, each consisting essentially of a frame on which two cylinders are mounted, one vertically, the other horizontally, each containing a reciprocating piston. Compressed air is employed as the motive power, the consumption, according to figures given by the manufacturers, ranging from 30 to 100 cubic feet per minute at a pressure of 85 to 100 pounds. Some device is necessary to hold the drill steel firmly in place. The sharpening is accomplished by means of suitable dies or dollies, which are either attached to or struck by the proper piston and forge the hot steel into the desired shape. The piston and die acting vertically are used for drawing out the corners of a broken or a very dull bit, or swaging out the grooves between the points, or insuring that the bit is of the required gage, whereas the horizontal die sharpens the cutting edges. With a suitable set of dies the machine may be used also for the construction of new bits from ordinary drill steel.

The use of a sharpening machine results in some saving of labor cost, for only one operator of medium skill is required. Such a man can ordinarily turn out several times the work of a skilled blacksmith and helper sharpening bits by hand. One manufacturer claims that his machine when handled by an expert is capable of sharpening 250 drills per hour, but he states also that half that number, under normal conditions, is satisfactory. With another type the capacity is given as 60 to 100 sharpened drills per hour. The lowest of these figures is more than ample for the usual requirements of tunnel work as, according to figures obtained at tunnels visited, the number of drills ordinarily sharpened ranges from 100 to 200 per day, although in hard ground the use of as many as 400 was reported.

The labor saved in the blacksmith shop is only a minor consideration, however, for the real superiority of the machine over hand sharpening lies in its ability to turn out perfect bits. As the progress in tunnel driving is often largely determined by the time required to drill a round of holes, this important part of the work deserves careful attention. It has been demonstrated repeatedly by practical experience that on comparing the cutting qualities of a machine bit with one sharpened by hand there is a marked difference in favor of the former. This is due to the fact that the bits come from the machine true to gage, thus greatly reducing the danger of binding or sticking in the hole; there is, therefore, less delay in drilling and a smaller loss of time from this cause for the driller and helper, or perhaps the entire crew, and there is less likelihood of " lost " holes. Then, too, the bits, being correctly shaped and properly sharpened, not only " stand up " better and stay sharp longer, but they also drill faster, and it is not necessary for the drill crew to change steel so often, thus reducing another source of delay. The use of drill-sharpening machines at the ordinary tunnel plant is, therefore, strongly recommended not only for its saving of time and labor both in the blacksmith shop and in the heading, but also for its ability to make bits whose superior drilling qualities will easily pay a large return upon the money invested in the machine.

AIR METERS.

Air meters are of various types. In one of them the volume of air is measured by causing the air to impinge consecutively upon a number of turbine wheels mounted on a common shaft connected with a registering device by a master gear. The machine is calibrated to read in cubic feet per minute of free air and is claimed by the manufacturer to give accurate measurements of air under varying pressures. A second type operates upon the principle that, with a uniform difference of pressure on both sides of an orifice and a constant initial pressure and temperature, the quantity of air passed is proportional to the size of the orifice. In this machine the difference in pressure on the two sides of the diaphragm is kept uniform by the constant weight of a taper plug that closes the orifice until the difference in pressure is sufficient to raise the plug and support it. The taper is so designed that the amount of air passed through the orifice is directly proportional to the rise of the valve, and this movement is multiplied and transmitted to a needle which records it upon a moving sheet of paper, thus affording a means of measuring the volume of air passed. A third type involves a device for determining the pressure due to the velocity of the flow of air in a pipe, which is proportional to the amount of air passed if the temperature and initial pressure are constant, and for transmitting that pressure to one arm of a U tube filled with mercury. The tube is balanced on knife edges, and as the pressure causes a flow of mercury to the other arm, the balance is disturbed and the tube is deflected, the amount of deflection being commensurate with the pressure and hence with the flow of air. The deflection is transmitted by levers to a recording needle. In a fourth type, although only a proportional volume ranging from 0.125 to 8 per cent is actually measured, the recording device registers in terms of the full 100 per cent volume.

Any of these meters may be used to determine the quantity of compressed air delivered to a purchaser. Their most important use, as far as tunnel work is concerned, is in determining the quantity of air used by rock drills. It is well known that all pneumatic rock drills show an increased air consumption, due to leakage, etc., as the various parts become worn through use. This fact is quickly discovered in practice, and many actual tests bear out the statement that after the steady use of the ordinary rock drill for six months or a year this loss will range from 20 to 40 per cent. This additional air is not only expensive to compress but, what is of more importance, the efficiency of the drilling machine is lowered at the same time, and the man operating it is unable to do as much effective work, thus entailing further loss. If the drill repairman has to guess at the air consumption, it is very difficult for him, even though he be an expert mechanic, to send from the repair shop back to the heading a drill that will do as good work as when it was new. But if the shop is provided with some means of determining the air required by the drill the repairman is much better able to remedy the defects and make the proper repairs. This results in a saving of expensive power and increases the efficiency of the drill and the amount of work done by the driller. Moreover, every time the drill leaves the repair shop it is very desirable to keep a record not only of the cost of its repair but also of its present air consumption, in order that upon its next return a comparison may be made with the last record as well as with the nominal air requirements. By such a course, if the air consumption is excessive necessary repairs may be made that would perhaps have been unsuspected otherwise, and at the same time the manager may keep an accurate statement of drill repairs and weed out inefficient drills. The following sample drill record[a] gives a rough outline of such a system:

DRILL RECORD.

Tool, *Piston drill.* Maker ____________________ Size, *2¼ inches.*
Purchased, *2/1/10.* Serial No. *123456.* Shop No. *12.*
Normal air consumption *90* cubic feet per minute, at *75–80* pounds.

Date.	Air consumption.	Pressure.	Repairs.
February 24, 1910	94	76	2 side rods.
March 10, 1910	99	78	2 pawl springs. 1 leather cup. 1 chuck bolt. 1 chuck key.
May 10, 1910	[b] 128	75	1 air chest and valve.[b]
After repairs	96	....	2 piston rings.

[a] By courtesy of the Excelsior Drill and Manufacturing Co.
[b] Excessive air consumption corrected by repairs indicated.

UNDERGROUND EQUIPMENT FOR DRIVING TUNNELS.

Underground equipment for driving mine tunnels includes rock drills, cars for removing broken rock, and some means of hauling them to the surface, together with the necessary tools, track, pipe, hose, lamps or candles, and other incidentals. It may also include telephones or other means of communication with the portal and, if the tunnel is being driven down grade or from a shaft, pumps to remove water.

ROCK-DRILLING MACHINES.

TYPES.

As a rule rock-drilling machines are classified primarily according to the motive power by which they are operated. The great majority of those used in tunnels are of the pneumatic type, but hydraulic drills have been employed and electric drills are coming into favor. For surface work steam is sometimes substitued for compressed air by making a few minor alterations in pneumatic drills, and machines using gasoline power are also to be found on the market; but the difficulties of disposing of exhaust steam with the former and of the products of combustion with the latter prevent any extensive use of these types underground. Some of the principal features of the various rock drills employed in tunnel work are described below.

PNEUMATIC DRILLS.

The pneumatic rock drill consists essentially of a cylinder containing a piston or a hammer that is reciprocated by the proper admission, application, and release of compressed air. In the piston type of air drill, a drill steel provided with cutting edges is made alternately to strike and recede from the rock by the movement of the piston to which the steel is firmly attached. In the hammer drill (Pl. III, *A*) the steel does not reciprocate, but is held loosely against the rock to which it merely transmits blows received from a moving hammer. Piston drills are almost without exception mounted in a cradle that may be attached to some rigid support while the drill is in operation, but may be easily removed when necessary; a screw thread is provided also, permitting the drill to be fed forward in the cradle as the hole grows deeper. In some types of hammer drills, especially those used for stopping and trimming, the cradle is omitted, and the drill either is held in the hand or is provided with a telescoping feed operated automatically by compressed air. In either type some device is required to rotate the drill steel in order that the cutting edges of the bit may not strike repeatedly in exactly the same place. In cradle-mounted drills this result is generally accomplished by a mechanism, consisting of rifle bar, ratchet, and pawls, that is arranged to turn the piston or hammer, this in turn rotating the chuck holding the drill. If the telescoping feed is employed the entire machine must be rotated by hand. Figure 9 shows a section through a piston pneumatic rock drill and gives a list of the principal parts.

Pneumatic drills are often differentiated by the method employed in controlling the admission of air to the cylinders. This may be accomplished by tappet, air-thrown, or auxiliary valves, or by direct regulation of the air supply by the movement of the piston or hammer itself.

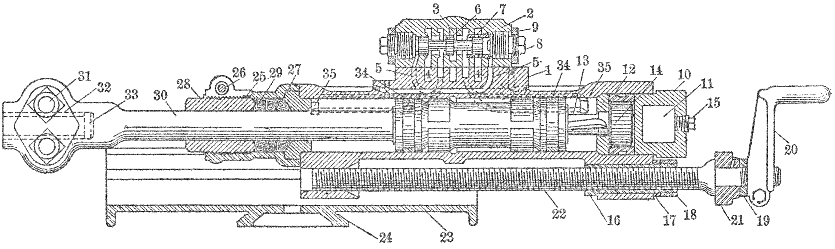
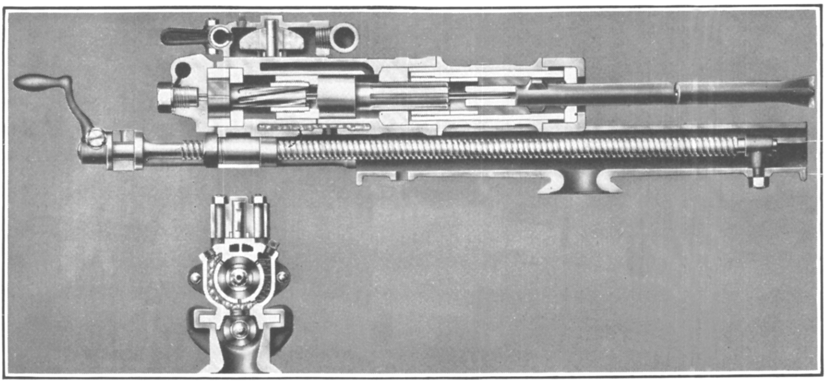


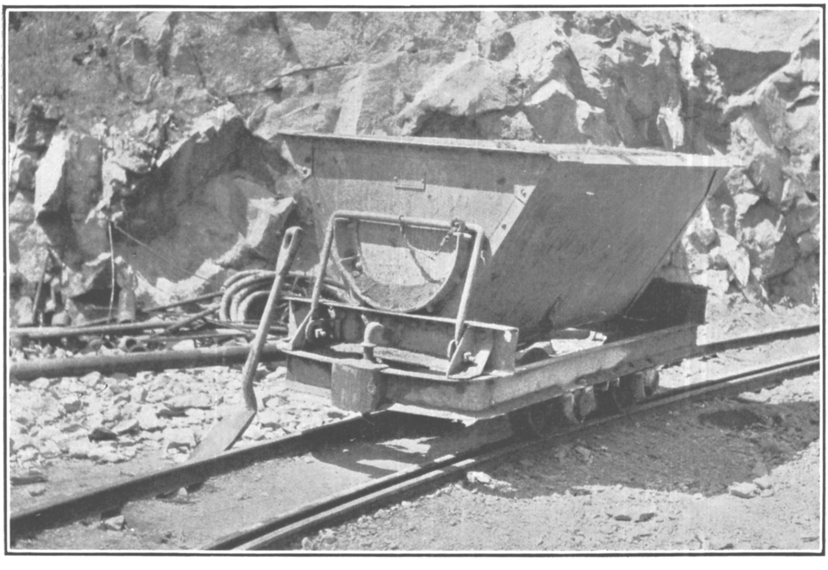
FIGURE 9.—Section of piston rock drill. 1, Cylinder; 2, steam chest; 3, inlet port; 4, exhaust ports; 5, reverse ports; 6, valve; 7, valve bushing; 8, buffer; 9, check nut; 10, top head; 11, oil chamber; 12, ratchet ring; 13, rifle bar; 14, ratchet; 15, plug; 16, feed nut; 17, lock washer; 18, check nut; 19, washer; 20, feed handle; 21, yoke; 22, feed screw; 23, shell; 24, trunnion; 25, lower head; 26, clamp bolt; 27, bushing; 28, gland; 29, packing; 30, piston; 31, clamp bolt; 32, chuck bushing; 33, chuck button; 34, piston rings; 35, cylinder ports.

The action of the tappet valve is indicated in Plate IV, *A*, which shows a section through a drill equipped with such a valve. As in operation the piston moves from the position shown in the figure toward the lower end of the cylinder, the crank end of the tappet rises and the other end drops into the depression of the piston, thus producing around the tappet pin a slight rotation sufficient to move the slide valve. This movement admits air against the lower end of the piston, at the same time connecting the upper end of the cylinder with the exhaust pipe. The piston therefore starts in the other direction and a similar, but reverse, process takes place.

The operation of the air-thrown valve is somewhat more complicated than that of the tappet valve. In figure 9, showing a drill equipped with the usual form of air-thrown valve, the action of the valve may be traced as follows: In the figure the piston is indicated as just starting on the down stroke, the valve being so placed that live air is entering the top cylinder port 35 from the air inlet port 3 by way of the connecting passages indicated by dotted lines, while



A. SECTION OF IMPROVED TYPE OF HAMMER DRILL FOR TUNNELING.



B. TUNNEL CAR USED AT NISQUALLY TUNNEL.

at the same time the front of the cylinder is connected with the exhaust 4 by the lower cylinder port and its airways. The upper end of the "spool" of the valve is connected with the lower end of the cylinder, and hence with the exhaust, by the reverse port 5. As soon as the piston in its travel uncovers the other reverse port 5, shown by dotted lines, pressure from the upper end of the cylinder will be transmitted to the lower end of the spool and will throw it against the upper end of the valve chest, and this movement will alternate the connection of the ports for live air and exhaust, thus reversing the piston. A similar process is then repeated on the upstroke.

In a recent modification of the usual air-thrown valve the spool is replaced by a cylindrical shaft carrying two flat wings, which somewhat resemble those of a butterfly. The operation of this valve is indicated in figure 10. In the figure, upper view, the piston P is represented as about to start on the forward stroke. The valve is thrown so that live air is permitted to enter through the supply ports S, S2, and SS2, while the spent air in the front end of the cylinder is exhausting through the ports EE1, E1, and the exhaust E. As soon as the piston in its forward movement uncovers the exhaust port EE2, live air will pass through EE2 to E2 and its pressure on the valve at E2 will balance its pressure on the opposite wing of the valve facing port S2. The valve will then be in equilibrium, but will be held stationary, with the ports S2 and E1 open because of the impact of the air opposite S2. Near the end of the stroke, however, the piston closes the exhaust port EE1 and in passing from EE1 to F1 it compresses the air which is trapped in the clearance space at the end of the cylinder. This cushion pressure, communicated through the cylinder ports SS1 to S1, is sufficient to throw the balanced valve to the position shown in the middle view. Live air is then admitted through S1 and SS1, the exhaust ports EE2 are opened, and the piston starts on the return stroke.

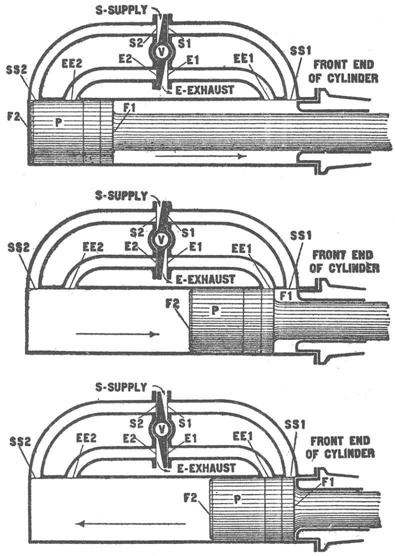


FIGURE 10.—Three views of butterfly valve, showing its action.

One form of auxiliary valve used on a well-known piston drill is described as a mechanism in which the strains, shocks, and jars to which the tappet or rocker is subjected are transferred from the main valve, with its vital and delicate functions, to a smaller auxiliary valve weighing only a few ounces, especially designed to withstand this service. This drill is illustrated in Plate IV, B.

When the drill is in operation one end of the auxiliary valve projects slightly into the cylinder and is thrown by the piston in its travel. The movement is perfectly free and very short—only enough to uncover a small port and release pressure from one end of the main valve, which is at once thrown by the resulting unbalanced pressure, opening wide the main port and admitting compressed air to the other end of the piston for the return stroke. The auxiliary valve is simply a trigger that releases the main valve.

In another form of auxiliary valve (fig. 11), the main air-thrown spool is controlled by two auxiliary valves consisting of steel balls that are positively actuated by the movements of the piston. In figure 11 the piston *a* is represented as having just started on the down stroke. Compressed air is entering the upper end of the cylinder through the port *g* and the spent air in the lower end is escaping through the port *h* and the exhaust chamber *j*. At the end of the stroke the ball *c* will drop on its seat and the ball *d* will be raised, thus allowing the air in the end of the valve chest at *f* to exhaust past *d* through the port between the upper and lower balls. The unbalanced pressure thus produced throws the valve to the other end of the chest, which reverses the connections between the cylinder chambers and the inlet and exhaust ports. The piston therefore starts on the return stroke and a similar but reverse process takes place.

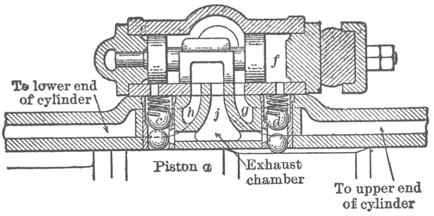
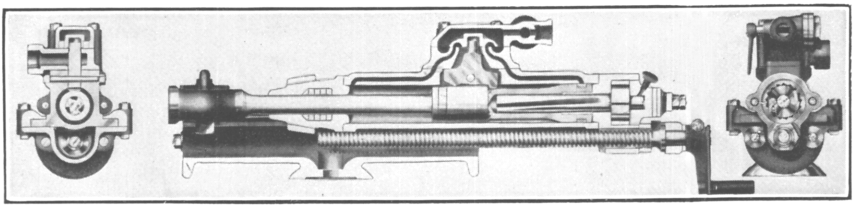
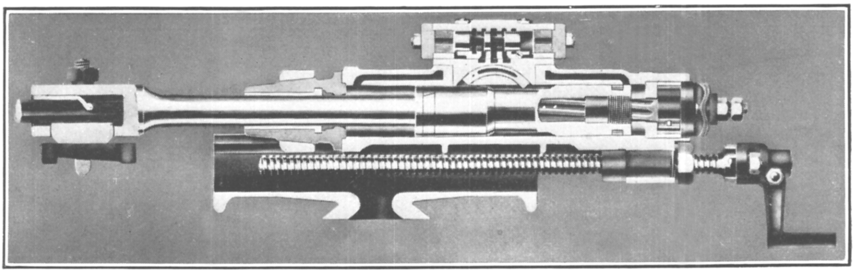


FIGURE 11.—Section of steel-ball auxiliary valve.

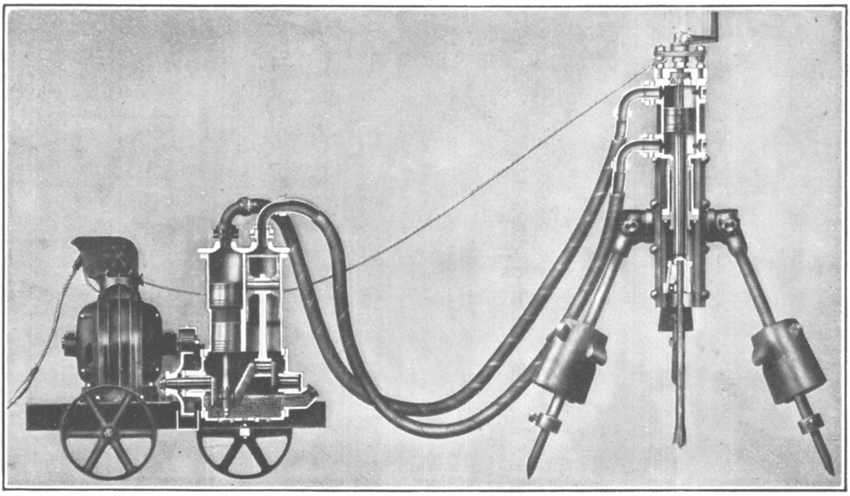
The valveless air-regulating mechanism, in which the movement of the piston itself covers and uncovers various ports, is employed almost exclusively on drills used for stoping only. Although rarely chosen for tunnel work, a brief description of this method of regulating air supply is warranted by its extensive use in its own field. The principle of operation is illustrated in figure 12, which shows cross sections through the cylinder of one make of valveless drill. In the position shown in the upper view of the figure air under pressure enters from the feed cylinder through the port *a* and passes to the front of the piston, where it exerts pressure at all times. The piston is forced back until the port *e* is uncovered, when compressed air passes through the port *f* and exerts pressure on the top of the pis-



A. SECTION OF A DRILL EQUIPPED WITH TAPPET VALVE.



B. SECTION OF TAPPET AUXILIARY-VALVE DRILL.



C. ELECTRICALLY DRIVEN AIR DRILL.

ton. As the area of this face is greater than that of the striking end, the piston starts forward. Live air is shut off when the port *e* is closed, but the piston is pressed forward by the expansion of the air until the exhaust port *b* is opened just as the blow is struck on the drill steel.

HYDRAULIC DRILLS.

The best-known hydraulic rock drill is, perhaps, one of the rotary type developed for use in the Simplon Tunnel, which consisted essentially of a hollow steel tube armed with teeth that were held firmly against the rock by hydraulic pressure while at the same time the tube was slowly revolved by a water-driven motor. This drill (fig. 13) is described by Prelini,[a] as follows:

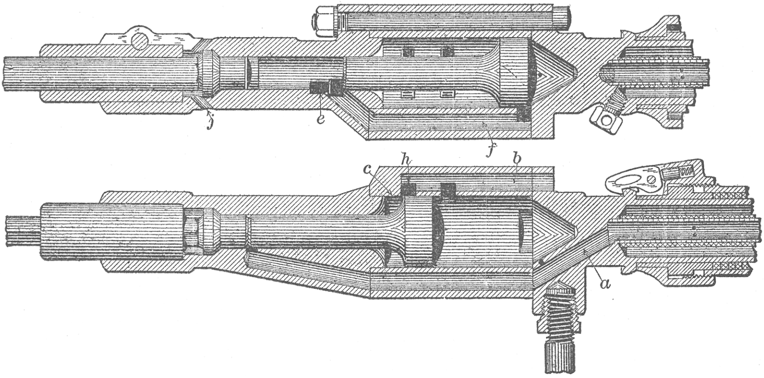


FIGURE 12.—Cross sections of valveless drill.

> This rotary motion is given by a twin-cylinder single-acting hydraulic motor, *e*, the two pistons, of 2⅞-inch stroke, acting reciprocally as valves. The cranks are fixed at an angle of 90° to each other on the shaft, which carries a worm, gearing with a worm wheel, *q*, mounted upon the shell *r* of the hollow ram *i*, and this shell in turn engages the ram by a long feather, leaving it free to slide axially to or from the face of the rock. The average speed of the motor is 150 to 200 revolutions per minute, the maximum speed being 300 revolutions per minute. * * * The pressure on the drill is exerted by a cylinder and hollow ram, *i*, which revolves about the differential pistons, which is fixed to the envelope holding the shell *r*. This envelope is rigidly connected to the bedplate of the motor, and, by means of the vertical hinge and pin *t*, is held by the clamp *v* embracing the rack bar. When water is admitted to the space in front of the differential piston the ram carrying the drilling tool is thrust forward, and when admitted to the annular space behind the piston the ram recedes, withdrawing the tool from the blast hole. The drill proper is a hollow tube of tough steel 2¾ inches in external diameter, armed with three or four sharp and hardened teeth, and makes from 5 to 10 revolutions per minute, according to the nature of the rock. When the ram has reached the end of its stroke of 2 feet 2½ inches the tool is quickly withdrawn from the hole and un-

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[a] Prelini, Charles, Tunneling, 1902, p. 103.

screwed from the ram; an extension rod is then screwed into the tool and into the ram, and the boring is continued, additional lengths being added as the tool grinds forward; each change of tool or rod takes about 15 seconds to 25 seconds to perform. The extension rods are forged-steel tubes fitted with four-threaded screws and having the same external diameter as the drill. They are made in standard lengths of 2 feet 8 inches, 1 foot 10 inches, and 11¾ inches. The total weight of the drilling machine is 264 pounds and that of the rack bar when full of water is 308 pounds. The exhaust water from the two motor cylinders escapes through a tube in the center of the ram and along the bore of the extension rods and drill, thereby scouring away the débris and keeping the drill cool; any superfluous water finds an exit through a hose below the motors and thence away down the heading. The distributor, already mentioned, supplies each boring machine and the rack bar with hydraulic pressure from the mains, with which connection is effected by means of flexible or articulated pipe connections, allowing freedom in all directions. The area of the piston for advancing the tool is 15½ square inches, which under a pressure of 1,470 pounds per square inch gives a pressure of over 10 tons on the tool, while for withdrawing the tool 2½ tons is available.

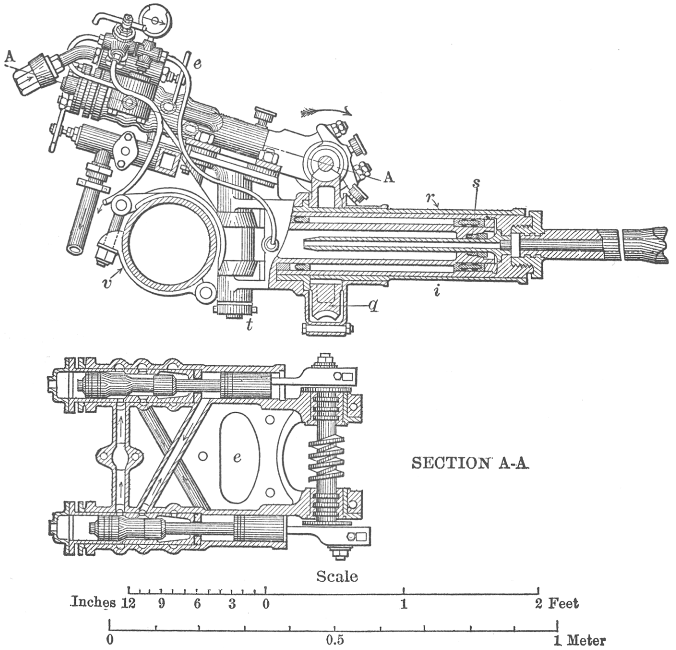

FIGURE 13.—Section of rotary hydraulic rock drill.

A recently invented percussion hydraulic drill is described fully in the Engineer,[a] from which figure 14 and the following brief abstract are taken:

The drill consists essentially of a cylinder, in which is a piston, *c*, free to move, while at the other end of the cylinder is a flap valve, *d*, which is kept open by a spring. The interior of the cylinder is in communication with a

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[a] New hydraulic rock-boring drill: The Engineer (London), Jan. 7, 1910, p. 24.

"striking tube," *fg*, at the end, *f*, of which is an air vessel. When the valve *h* is opened, water flows through the apparatus, out past the valve *d*, into the waste pipe *e*. The rush of water past the valves causes the pressure on the under side to be less than the pressure on the upper side, where the velocity is less. * * *

* * * When the velocity attains a certain value the difference of pressure is sufficient to close the valve, and the column of water in the striking tube is suddenly stopped. The kinetic energy of the water in the tube is communicated to the piston *c*, which is impelled forward with high velocity, and the drill which is at the end of it strikes a heavy blow on the stone or rock being bored.

The pressure in the interior of the cylinder is diminished by the moving out of the piston *c* * * * enough for the valve to open. Water then streams through the open valve. The piston is meanwhile being brought back to its original position by springs, but before it is right back * * *, the valve, *d*, closes, and the direction of motion is reversed by the hydraulic shock. The drill then strikes another blow as before. The actual apparatus is shown in section and plan in figure (14 of this bulletin) which is roughly to scale, the over-all length being about 4 feet.

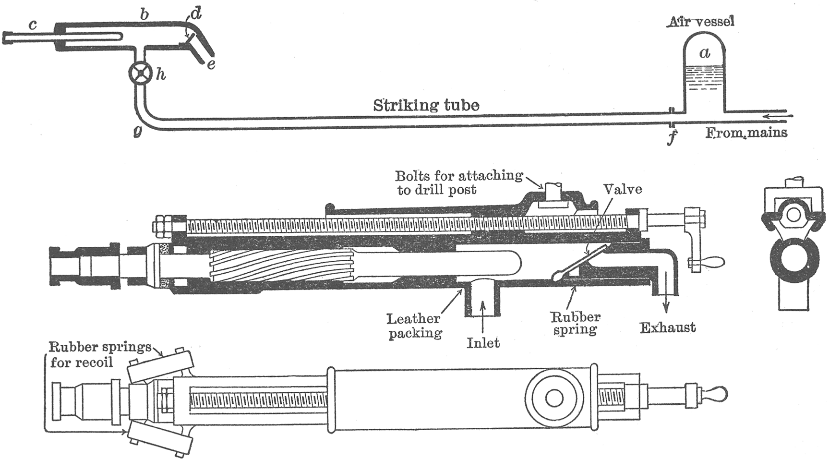


FIGURE 14.—Sections and connections of hydraulic percussion rock drill.

The actual magnitude of the blow depends primarily upon (1) the weight of the striking column; (2) the velocity of the water when the valve closes; and (3) the weight of the chisel and boring bar.

The velocity of the column is fixed by the velocity at the valve required to produce the necessary difference of pressure to close the valve—i. e., it is fixed by the stiffness of the spring controlling the valve. The rapidity of the blows is limited by the fact that after each blow the striking column is brought to rest, and it must be accelerated up to the requisite velocity before the valve will close. The rapidity of working depends, therefore, upon the pressure which is urging the column forward—i. e., it depends on the pressure in the supply mains. The actual magnitude of the blow is said to be unaffected by the varying pressure in the mains, and to depend only on the weight of the striking column and the strength of the spring controlling the valve. The inventor claims that machines of the type described strike from 20 to 30 blows per second, while the maximum speed of percussion machines of existing types is from 3 to 5 strokes per second.

One of these machines has recently undergone a series of tests at the Millbank pumping station of the London Hydraulic Power Co. The pressure used was 450 pounds per square inch. * * * The tests were carried out on a block of hard Portland stone. The diameter of the drill used was 2⅝ inches, and on an average progress was made in the stone at the rate of 10½ inches per minute. This is equivalent to the removal of 46 cubic inches of stone per minute. The drills stood up to the work so well that after holes aggregating about 25 feet in depth had been drilled it was not necessary to do anything to the edge. A stream of water plays on the chisel the whole time and serves the threefold purpose of keeping the chisel cool, of rinsing the bore hole, and of allaying the dust.

ELECTRIC DRILLS.

An electric rock drill consists primarily of an electric motor and a means of applying the power developed in it to the work of drilling rock. In some machines the motor is mounted directly upon the drill frame, but in others it is removed a short distance and connected to the drill by a flexible shaft or some similar device for transmitting power. Provision must also be made for preventing the shocks and jars developed by the impact of the drill steel upon the rock from being transmitted back to the motor, which is a machine incapable of operating for any length of time under such conditions. In many of the earlier models springs or cushions or some elastic material such as rubber were used for this purpose. These devices failed to give satisfaction either because of inability to do the work required or because of excessive wear, breakage, and annoyance. In two or three of the early models an ingenious attempt was made to avoid these troubles by taking advantage of the fact that if an electric current is passed through a spiral coil of wire a suitably placed bar of soft iron will be drawn into it. By providing two such coils or solenoids and causing the current to flow through them alternately an iron piston carrying a drill steel was made to reciprocate between them. In order to have the blow sufficiently smashing to be effective, however, a prohibitive weight of copper wire was needed for the solenoids. To-day practically all electric drills use compressed air in some manner to cushion the reaction of the blow. The compressed air possesses the very desirable characteristic of extreme elasticity and is not affected by wear and tear. In one machine, however, a hammer is made to strike the end of the drill steel by centrifugal force, the rebound giving the necessary flexibility.

One of the successful electrically driven rock drills that has been on the market for over five years is illustrated in Plate III, *C*. In this machine the drill piston is reciprocated by alternating pulsations of compressed air, created by a double-cylinder air compressor driven by a standard electric motor. Two short lengths of hose connect the air compressor to the drill, each running from one of the compressor cylinders to opposite ends of the drill cylinder. The air in the system, which acts as an unwearing cushion between the pulsator and the drill, is never exhausted, but is simply used over and over. The drill is very simple—merely a cylinder containing a piston and rotating device—valves, chest, side rods, buffers, and springs being omitted, and the compressor has neither valves nor water jackets. The motor may be designed for either direct or alternating current as desired, and it is mounted with the compressor on a wheeled truck for easy handling.

A second air-cushioned electric drill of the piston type, but one in which the motor is mounted directly on the drill frame, is illustrated in figure 15. In this drill the motor *m*, which can be readily detached from the rest of the machine whenever the drill is moved, is connected by reducing gears to the crank shaft *s*, which drives the connecting rod *r*. This is attached and gives a reciprocating motion to the cylinder *c*, which slides in suitable guides and contains a piston *p* provided with a chuck for holding a drill steel. As the cylinder moves forward air is compressed in the chamber *b* behind the piston and makes the piston move forward, which causes the drill bit to strike the rock. During the return stroke of the cylinder the compression of air in the other chamber, *f*, brings the piston back again with it. Rotation is obtained by means of standard spiral nut and ratchet. Details of the feed screw, the carriage, and other features are shown in the illustration.

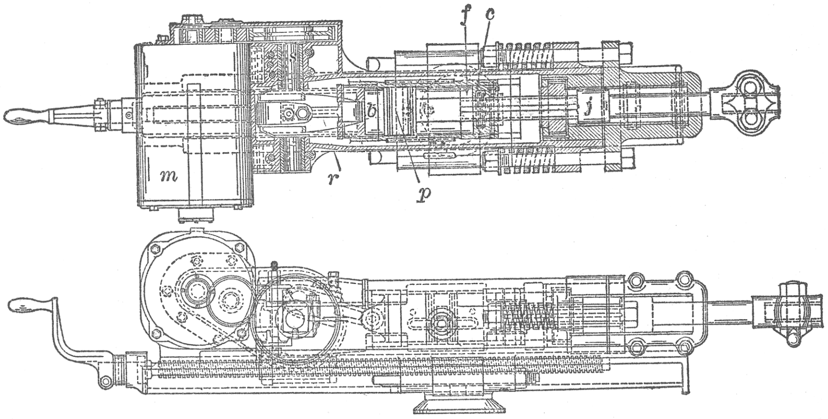


FIGURE 15.—Cross section of electric air-cushioned piston drill.

In an electrically driven air-cushioned rock drill of the hammer type (fig. 16) power is transmitted by suitable gears and cranks from the motor to a piston and causes it to reciprocate in an air cylinder. The same cylinder contains at its other end a hammer which, however, is in no manner directly connected with the piston. As the latter starts on the down stroke it compresses the air in the space between it and the hammer, which is projected forward until it strikes the end of the drill steel. Just as it does so it releases the compressed air by uncovering an exhaust port controlled by a poppet valve. When the piston starts on the return stroke the exhaust valve closes and the partial vacuum created pulls the hammer toward the piston. The latter in its travel uncovers an inlet port, also poppet controlled, admitting new air which destroys the vacuum. The momentum of the hammer would cause it to strike the piston, which again starts on the down stroke, were it not for the compression of the air entrapped by the closing of the poppet valve as soon as the vacuum is destroyed. The drill steel is rotated by the motor through a shaft, gearing, and ratchet. Water is forced to the cutting edge through hollow steel by a small pump supplied with the drill; but if water under pressure is already available the pump may be disconnected. Another feature of this drill is the automatic chuck which is adapted for using steel as it comes in the bar, thus obviating the necessity of forging shanks.

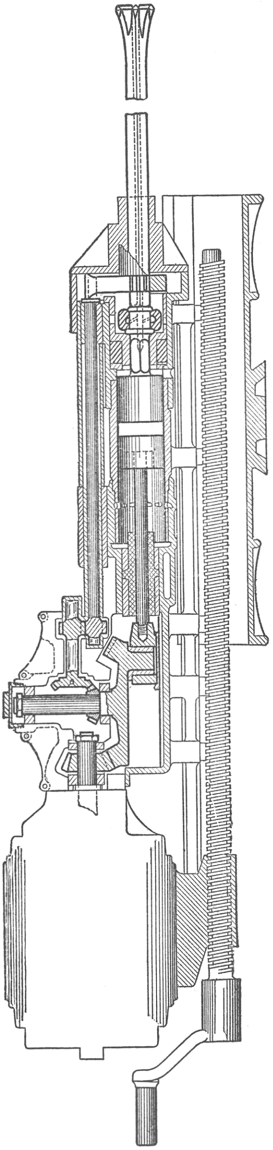
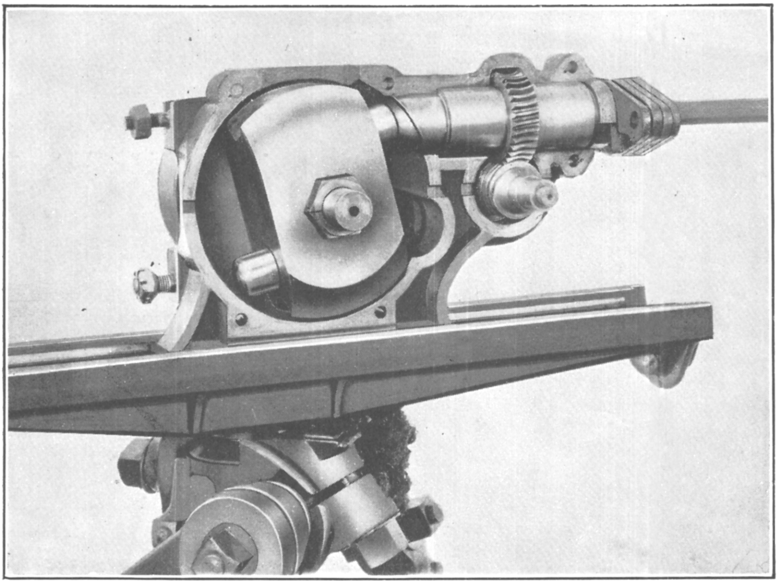


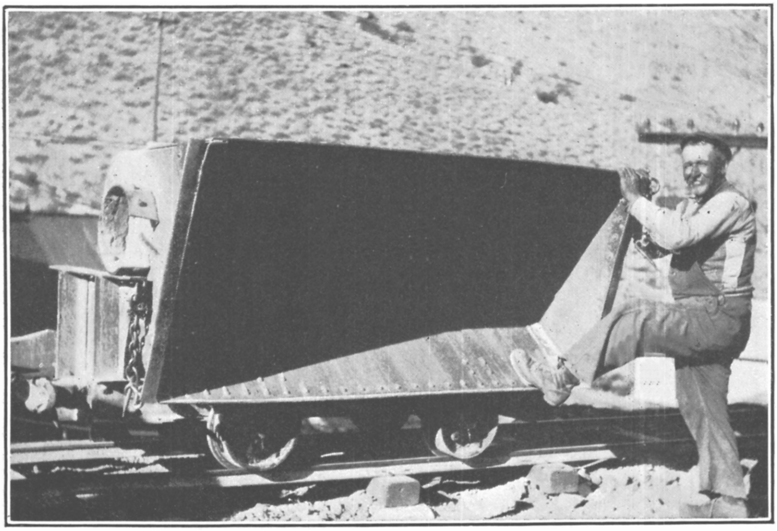
FIGURE 16.—Cross section of electric air-cushioned hammer drill.

A fourth electric drill, also having an air-cushioned hammer, is illustrated in figure 17. In this drill, as the yoke *a* moves forward the piston *b* compresses the air in the chamber *c*, forcing the cylindrical hammer *d* against the anvil block *e*, which transmits the blow to the drill steel at *f*. On the return stroke of the piston the compression of air in the chamber *g* brings the hammer back in readiness for another blow. Water is forced through hollow steel by a small pump, whose plunger reciprocates with the drill piston.

So far as could be learned, the only electric drill in service to-day that does not use an air cushion is the one illustrated in Plate V, *A*. In the plate are shown two hammers which, although free to slide in their sockets in the revolving disk, are thrown out by centrifugal force and strike the anvil block which transmits the blows to the



A. ELECTRIC REVOLVING HAMMER DRILL, WITH MOTOR AND PART OF CASING REMOVED.



B. METHOD OF DUMPING ROCKER-DUMP TUNNEL CAR.

drill steel. The steel, which is held in a chuck rotated by a worm gear as indicated, is of the auger type, the spirals acting in the capacity of a conveyor for removing broken rock from the hole.

GASOLINE DRILLS.

Because the difficulty of disposing of the waste products of combustion, which are not only hot and disagreeable, but also contain gases injurious to the health of the workmen, makes the gasoline drill hardly suitable for service underground, and because, as far as could be learned, it has never been used in tunnel work, its design and construction will not be discussed here.[a]

MERITS OF EACH TYPE.

PNEUMATIC DRILLS.

The chief advantage of the pneumatic rock drill is its ability to withstand rough usage and still perform efficient service. The work of a rock drill is done necessarily under conditions that would quickly destroy almost any other type of machinery. It is subjected to constant and severe vibration when in operation, for although it is usually held firmly and securely, it can not be mounted rigidly. Lubrication, when supplied at all, is often administered in large doses most irregularly, and it is impossible to prevent sand and grit from getting into the machine, thus adding greatly to the wear and tear. In many cases men who operate it have no conception of its construction and ignorantly subject it to shocks and strains for which it was never designed, their first impulse when things go wrong being to seize a sledge hammer and hit the machine in the most convenient place. All drill runners of course do not belong to this type, but the description fits a much too large percentage of them. Everything considered, the rock drill, as some one has so aptly put it, must be

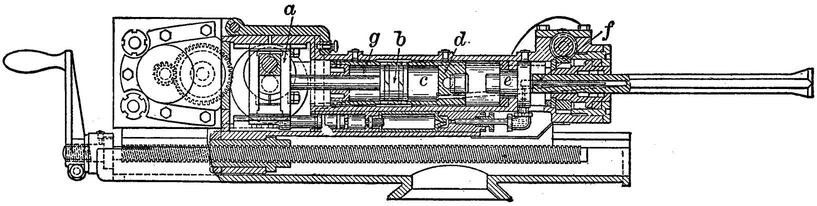


FIGURE 17.—Another type of electrically operated air-cushioned hammer drill.

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[a] For description of one of these machines having two explosion cylinders, see Eng. and Min. Jour., vol. 86, Nov. 21, 1908, p. 1008; Eng. News, vol. 60, Nov. 26, 1908, p. 575; and Min. and Sci. Press, vol. 97, Dec. 19, 1908, p. 852. For description of a drill, of English manufacture, in which a cam, driven by a gasoline engine, trips a spring-actuated piston, see Engineer (London), vol. 110, Sept. 30, 1910, pp. 365, 366, and Eng. News, vol. 64, Nov. 17, 1910, p. 538.

capable of being "cleaned up with a sledge hammer, wiped off with a scoop shovel, and still stay with you." This requirement necessitates the elimination of all unsuccessful features, the rejection of complicated parts that are not absolutely essential, the determination of the proper size and strength of those remaining, and the selection of materials having the proper stability and wearing qualities. It is only natural that the pneumatic drill, which has been undergoing development for more than 50 years, should be able better to cope with the conditions and to operate more steadily, with fewer interruptions and a lower cost for repairing broken or worn parts than any of the newer types.

Among other advantages of the pneumatic drill may be mentioned the fact that it furnishes some ventilation, that it does not require the introduction underground of electricity at comparatively high voltages (oftentimes a source of danger), nor does it require pipes strong enough to withstand the pressures needed for the rotary hydraulic drill. The air drill should not be relied upon too strongly for ventilation, however. In the first place, the supply of air is intermittent and is not forthcoming while the drill is stopped for the purpose of changing steel or moving it into position for a new hole, etc.; in the second place, the drills are not in operation immediately after the blast—the time ventilation is most needed—although it is true that the use of pneumatic drills makes it possible to direct a jet of compressed air into the heading at this time to assist in removing the smoke; and, finally, there are cases on record in which the exhaust from the drills not only did not deliver fresh air, but even filled the heading with carbon dioxide and other dangerous gases produced by combustion of oil and grease in the receiver, resulting, in one instance at least, fatally for several men. Again, at tunnels using electric haulage, the adoption of electric drills would simply add to a danger already present rather than introduce a new one, and in such cases the advantage of the air drill in this respect is not so important.

The most important disadvantage of the pneumatic drill, on the other hand, is its well-known lack of power economy. Since, as stated by Rix,[a] "the tables set forth in the trades catalogues for the consumption of standard piston rock drills are fairly accurate," let us determine from them the power required for rock drills by using his estimate of 20 brake horsepower per 100 cubic feet of free air per minute. The lowest figure given for any type of rock drill used at the tunnels examined for this report is 65 cubic feet per minute at 100 pounds pressure, whereas drills using as much as 150 and even 175 cubic feet were very numerous. On this basis, then, without any

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[a] Rix, E. A., Compressed-air calculations: Address before the mining association of the University of California, Feb. 19, 1908.

allowance for loss of power through friction in the pipes or leakage in the machines when they become worn, pneumatic drills require the application of 13 to 35 brake horsepower at the compressor during the time the machine is operating. Although the rotary hydraulic drill employed in the Simplon Tunnel required as much as 13 horsepower[b] (exactly the minimum figure just deduced for air drills) it is by comparing the power used in air drills with even the maximum of 6 horsepower for electric drills, many of which run on less than 2, that the large difference in power consumption is revealed.

As regards the different types of pneumatic drills used in tunneling, the piston machine has somewhat the advantage over the hammer type as regards reliability and efficiency in drilling holes vertically or nearly vertically downward. This reliability may be attributed, without doubt, to its simpler construction. It does not contain any mechanism for introducing a water spray through a hollow drill steel, it is not troubled by crystallization of metal parts from the repeated shocks of rapid blows, and it has a much greater range of feed. The last feature is important when the machine is handled by an inexperienced operator, giving as it does greater latitude before the piston begins to strike the front head. These considerations make the piston drill more nearly "fool-proof," and hence better adapted for use by ordinary drill runners—especially those in the Eastern States, who, as a rule, are neither as intelligent nor as careful as those in the West. Complexity of construction, however, should not be confused with the number of parts, for if this were taken as the standard and every screw, bolt, or nut counted separately it could be shown that the hammer drill is the simpler machine.

The greater efficiency of the machine in drilling holes that point downward was clearly brought out in the recent extensive drill competition in the Transvaal, according to the committee conducting the test, who reported that one of the main reasons for the better showing made by the piston drills underground was the fact that practically all of the holes drilled there were pointed downward. This is substantiated in several instances at tunnels in this country in which the excavation is accomplished by the heading-and-bench method; in such tunnels the piston drill is reported to have given better satisfaction in drilling the vertical holes required for the removal of the bench.

The hammer drill of the type used in tunnel driving, on the other hand, requires less power than the piston machine, is lighter, smaller, more easily handled in a restricted space, and is able to drill with greater speed holes that are horizontal, or nearly so, especially those pointing slightly upward such as are necessary under the ordinary

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[b] Comstock, C. W., Great tunnels of the world: Proc. Colo. Sci. Soc., vol. 8, 1907, p. 363.

methods in driving tunnel headings. In hammer drills the air consumption, and hence the power required, varies from 65 to 100 cubic feet per minute at 100 pounds pressure (catalogue rating) as compared with 135 to 175 cubic feet for piston drills. The weights of hammer drills range from 115 to 170 pounds, whereas the piston machines weigh from 280 to 400 pounds, and the dimensions of the former are approximately four-fifths those of the latter. The shorter length of the hammer machines makes it possible to start the cut holes nearer the side of the tunnel, thus obtaining a wider angle between each pair with a consequent increase in the chances of breaking the full length of the round of holes. The rate of drilling is of course largely dependent upon the character of rock penerated, but the data of Table 10 below (in which it will be seen that piston drills, even in shale and sandstone, rarely drilled over 10 feet per hour, whereas the hammer drills in granite and other hard rock rarely fell below that figure, 15 and even 20 feet being not uncommon) seem to warrant the general statement that the hammer type has the greater speed in drilling the holes required in tunnel headings.

TABLE 10.—*Comparative drilling speed of piston and rock drills as reported at various tunnels.*[a]

Name of tunnel.	Type of drill.	Character of rock penetrated.	Drilling speeds per machine per hour.	Remarks.
			Feet.	
Carter	Hammer	Granite	10	Approximate.
Catskill Aqueduct:				
Rondout	Piston	Shale	8	A fair average.
Wallkill	do	do	10.5	Normal conditions.
Central	Hammer	Gneiss	8–16	
Fort William (water)	Piston	Trap	2	Phenomenally hard rock.
Gold Links	do	Gneiss	8–10	Approximate.
Joker	Hammer	Breccia	12–15	
Laramie-Paudre	do	Granite	15	Ordinary conditions.
Los Angeles Aqueduct:				
Little Lake	do	Hard granite	15.84	Average of 15 accurately timed shifts.
Grape Vine	do	Granite	10	Estimated average.
Lucania	do	do	13	Average of 3 drills.
Marshall-Russell	do	do	10–20	
Mission	do	Shale and sandstone	30	Medium ground.
Newhouse	do	Gneiss	10	
Nisqually:				
Headworks end	Piston	Rhyolite	8–10	
Discharge end	Hammer	do	10	Rock is much harder than at other end.
Ophelia	Piston	Granite	8–10	
Rawley	Hammer	Andesite	15–20	Average of 4 accurately timed drill shifts, 19.7 feet.
Raymond	do	Granite	8–12	
Siwatch	do	do	10–15	
Snake Creek	Piston	Diabase	[b]8–12	
Stilwell	do	Andesite	5–10	
Strawberry	do	Shale	13	Test run.
Utah Metals	do	Quartzite	6–8	

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[a] Includes time used in setting up and tearing down column or bar, in shifting machine to new holes, and in changing steel, but does not include time used in mucking for set up or in loading, blasting, and clearing smoke.

[b] During a competition test in which both drills were mounted on the same bar in order to obtain identical conditions of rock, etc., the piston machine drilled 21 feet per hour, whereas a hammer drill made only 20 feet. The conditions were unusual, however, because water under pressure was encountered in practically every hole drilled, and doubtless influenced the results greatly.

It is difficult to determine just how much of the greater speed of the hammer drill is due to the manner of attack, the water feature, the greater ease and speed in replacing a dull steel with a sharp one, or to the nonreciprocating drill steel, but there is little doubt that all these factors enter into the result. The piston machine when attacking the rock strikes comparatively slow, heavy, smashing blows that soon dull the cutting edges of the bit, especially if the rock be hard. Then until the steel is changed the penetration must be accomplished by crushing. Conversely, the more frequent blows of the hammer type, being lighter, do not dull the bit so quickly and penetration is effected by a chipping action which is speedier as well as more economical of power. The application of water through a hollow steel to the face of the drill hole, in addition to cooling the drill bit and preserving the temper of its cutting edges, affords a positive means of removing the cuttings promptly from the front of the bit. This not only prevents the recutting and grinding of material already broken, with a consequent saving of power, but it increases the efficiency of the machine because it enables the drill bit always to strike an uncushioned blow on "live" rock. Hammer drills having the water feature, however, are said to make a poor showing when drilling vertical holes. This is doubtless due to the fact that the velocity of the rising current of water in the drill hole is not sufficient to prevent the rock grains from settling against it to the bottom of the hole and interfering with the work of the drill.

The plunger action of the piston drills, on the other hand, although it is probably no more efficient in actually removing the rock grains, keeps them stirred up enough partly to obviate the difficulty. Anyone who has experienced the trouble and delay of changing steels with the usual chuck in piston drills will appreciate the saving in time and energy resulting from the use of a chuck into which the drill needs only to be inserted. As in the hammer drill the steel does not reciprocate, the elimination of friction against the sides of the drill hole effects a considerable saving of power and prevents a retardation of the blow, even though, as has been argued, this advantage is partly offset by the loss of power in heating the hammer and drill end and in overcoming the inertia of the steel. An additional advantage of a nonreciprocating drill steel is the fact that it may be held against the rock at any desired point and a drill hole started wherever necessary without loss of time, a feature especially important where the face of rock is oblique to the drill.

Piston and hammer drills employed in tunneling are seemingly on an equal footing to-day as regards cost of drill repair parts, although until quite recently the piston drills had somewhat the advantage. From September, 1905, to March, 1906, hammer drills were employed at the Gunnison Tunnel with a drill-repair cost per machine of 13 cents per foot of hole drilled; but when piston drills were substituted the repairs were reduced to 3 cents per foot.[a] Two years later (September, 1907, to August, 1908), in driving the last 3,000 feet of the Yak Tunnel, the cost of materials only for repairs to the hammer drills employed was only 1¾ cents, approximately, per foot of hole. At the Marshall-Russell Tunnel where hammer drills were employed, the average cost of drill repairs from June, 1908, to June, 1911, was only 1½ cents per foot drilled. Piston machines were used at the Strawberry Tunnel from January, 1909, to the time of the authors' visit in September, 1911, the cost for repairs being nearly 2½ cents per foot drilled. On the Little Lake division of the Los Angeles Aqueduct, where hammer drills were employed, the average cost of drill-repair materials from July, 1909, to May, 1911, as shown by the following record was only 24 cents per foot of tunnel excavated. As each of the two machines in the heading drills approximately 8 feet of hole for every foot of tunnel excavated, the cost per machine per foot of hole is 1½ cents.

Cost of repairs for hammer air drills, Little Lake division, Los Angeles Aqueduct, July 1909, to May, 1911.

Name of tunnel.	Distance excavated.	Total cost of drill repairs.	Cost of drill repairs per foot of tunnel.
	Linear feet.		
1B, south	1,030	$160.59	$0.156
2, north	926	180.72	.195
2, south	419	64.75	.154
2A, north	460	46.28	.100
2A, south	375	55.50	.148
3, north	864	113.60	.131
3, south	2,149	505.01	.235
4, north	448	67.03	.149
4, south	725	215.48	.297
7, north	1,911	399.70	.209
7, south	1,024	493.46	.482
8, north	225	146.56	.651
8, south	1,334	530.52	.398
9, north	777	230.51	.297
9, south	2,479	404.94	.163
10, north	2,626	585.78	.223
10, south	1,776	577.24	.325
10A, north	1,373	303.06	.221
10A, south	1,756	359.27	.204
Average			.24

For 1910 and the first half of 1911 the repair cost of hammer drills at the Carter Tunnel was 2 cents per foot drilled. At the Lucania Tunnel the repairs cost one-half cent per foot drilled, but the hammer drills had been in use only one month at the time the tunnel was visited. The hammer drills at the Rawley Tunnel were new also,

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[a] In addition to the cost of materials these figures include also a charge for the labor of the machinist making the repairs, which is not embraced in any of the values which follow. This fact must be considered in making comparisons.

the repairs for June and July, 1911, averaging 1 cent per foot of hole. These figures, which are based upon estimates furnished by managers or others in charge at the various tunnels, do not pretend to other than approximate accuracy, but they give a basis of comparison such as has been hitherto unattainable, although in making such comparisons the type of rock must of course be duly considered.

In spite of the development of other types of valve mechanism for air drills, the tappet valve, which was one of the pioneers in the field, possesses advantages that still keep it in demand for use on piston drills intended for certain kinds of work. As it is unaffected by condensed moisture, which greatly interferes with the action of some other types, it is especially adapted for use with steam or with air containing a large quantity of water vapor. Its distinctive advantage, however, is that its movement is positive—if the piston makes a stroke the valve must be thrown—hence there is no uncertainty or " fluttering " in the action of the drill.

The tappet drill is at a disadvantage when working in ground that will not permit the use of a full stroke, because the piston must travel far enough to throw the valve and hence too short a stroke is not permissible. Then, too, because some air will necessarily be trapped in front of the piston and compressed after the valve is thrown, the drill strikes a cushioned blow. This is not always a disadvantage; in elastic and " springy " rock an uncushioned blow will not give the best cutting effect, whereas in sticky material compression assists the piston in starting on the return stroke. The tappet is subject to strains and wear, which necessitate specially hardened material, not only in the tappet itself but in the bearing surfaces of the piston.

Under conditions that require a snappy, vicious blow with high air pressure, the ordinary air-thrown valve gives the best results. This valve is particularly applicable to hammer drills in which, because of the small size and weight of the hammer, it is essential that there shall be no cushioning of the blow, and the air-thrown valve is customarily employed on such of these machines as are not of the valveless type. When used with piston drills the air-thrown valve permits a variable stroke; it renders possible a change at will in the length of piston travel and the force of blow. The short stroke and light blow possible with this type of drill make it adapted to starting a hole or to drilling through seamy rock. After the hole is under way, or if the rock is solid, a full stroke is used to get the best efficiency from the machine. The air-thrown type of valve is not positive in its action, however, and is apt to be somewhat sluggish with air or steam containing much water. The claim is made for the butterfly type that it avoids this difficulty as well as most of the troubles caused by freezing and that it has a positive and at the same time a flexible action which permits much higher speed than other valves.

The auxiliary valve is designed to combine the advantages of the tappet and the air-thrown valves while avoiding their defects. The lightness of the tappet auxiliary is said to prevent the injury or retardation of the piston and also to obviate the rapid wear of rings, piston, and cylinders caused by crowding against the opposite cylinder wall, due to an unbalanced tappet not readily moved. A drill equipped with this type of valve has a wide variation of stroke and delivers an uncushioned blow. The main advantage of the steel-ball auxiliary valve is its great resistance to wear and the cheapness of replacing its wearing parts. The claim is made for this valve that it assures a positive action of the drill without sticking or fluttering and yet possesses the necessary flexibility.

The valveless regulating of admission and exhaust has the advantage of simplicity and lighter weight owing to the elimination of the valve and valve chest. It also uses air expansively, which should result in economy of power. It entails a cushioned blow, however, thus reducing the drilling power where the rock is hard and tough; but for medium hard rocks, especially with high air pressure, the difference is said to be less pronounced because the lighter and more rapid blows chip rather than pulverize the rock and enable the drill to penetrate readily. One real disadvantage is the fact that as the cylinder becomes worn there is a leakage of air past the piston, increasing the air consumption and interfering with the accurate working of the drill.

HYDRAULIC DRILLS.

Among the advantages and disadvantages of the rotary hydraulic drill used at the Simplon Tunnel should be mentioned the fact that the power was delivered to the cutting edge without the shocks, jars, and strains due to percussion, thus eliminating one source of wear and tear. The machine also utilized a very high percentage of the power stored in the motive fluid, its efficiency being given by one authority as 70 per cent. Again, by passing a part of the waste water down the boring tube, chips and débris were promptly removed from the cutting edge, thus insuring the maximum boring power. On the other hand, the pressure required for operating this drill was enormous, ranging from 450 to 1,200 pounds per square inch, according to one writer, and according to another, 1,470 pounds. In any case the piping necessary to transmit the water under such high pressures must have been expensive to install and maintain. To withstand the back pressure, the drill also required extremely heavy and rigid mountings, which made it cumbersome and hard to move, so that it could not be easily placed for a new hole.

The percussion type of hydraulic rock drill can not as yet be said to have been demonstrated to be a practical success. It is an interesting possibility, however, because, like the hydraulic ram, it utilizes the shock that occurs in pipes at every stoppage of a moving column of water.

ELECTRIC DRILLS.

Among the advantages claimed for an electric drill of the pulsator type are saving of power, rapid drilling speed, simpler construction, and less trouble with fitchered holes. The motors used to operate the pulsator require 3 to 5 horsepower, a very small amount compared with the necessities of the ordinary pneumatic drill. Although it is true that the cost of power used by a drill is not the only item that determines its efficiency, such a marked difference in power consumption is an important consideration. Moreover, the pulsator electric drill has the full drilling speed of any corresponding standard air rock drill and has practically the same cost for wages and fixed charges. The pulsator type also eliminates many parts, such as valves, springs, and side rods, which are sources of trouble and unreliability in other rock drills. It is able also to strike a heavy blow because the pressure of air back of the piston is greatest just at the time of impact; and should the drill steel become caught in the hole from any cause, the machine does not cease running, as is the case with air drills, but the pulsator continues to exert several hundred alternate pulls and pushes on the drill steel per minute, which in most instances are sufficient to loosen the drill at once.

On the other hand, the combined drill and pulsator are cumbersome and occupy a large space, every inch of which is precious in the tunnel heading—a disadvantage that increases directly with the number of drills needed for the work. For tunnel work it is necessary either to expend extra time and energy in placing the truck and pulsator upon the muck pile, where they are subject to breakage if the muck is being removed simultaneously with the drilling; or to wait until the tunnel is cleared of débris before starting to drill, a procedure that is impracticable if speed in driving is required. But if there is no particular haste or if drilling and mucking are alternated, this disadvantage is not so serious.

The piston electric drill described on page 101 does away with the need of a pulsator, truck, and connecting hose, thus making a compact machine and one more comparable with an air drill. It is, however, rather heavy (weighing 490 pounds with the motor attached and 350 pounds without it) and is somewhat difficult to handle and move in a small heading. It has a marked advantage over air drills in power economy, operating as it does on 4 horsepower, and actual results show that its drilling speed is fully up to that of standard piston pneumatic drills. At the Elmsford Tunnel of the Catskill Aqueduct these drills are reported to have attained a speed of 100 feet in 6 to 8 hours when drilling in a comparatively soft mica schist, but in the harder Fordham gneiss of the City Tunnel the rate was only 60 feet per shift of 8 hours. This drill is still in the process of development; the small defects that always appear in any newly designed machine when put to actual use need to be corrected, but the results attained with it on one part of the City Tunnel were very encouraging. One of the machines is reported to have operated there for more than five weeks, drilling over 4,000 feet of holes with only minor breakages, such as pawl springs.

The weight of the air-cushioned hammer drill and motor described on pages 101–102 is about 150 pounds less than that of an electric piston drill and motor. With the motor removed, although it weighs more than a pneumatic hammer drill, it is little heavier than a piston air drill of corresponding capacity. Its power consumption is rated at 2¼ horsepower, and in the tests on the Catskill Aqueduct 6 to 8 feet per hour was the average drilling speed attained in ordinary work, including delays. This speed was undoubtedly increased as the delays from breakdowns became less frequent. The drill was still in the process of being perfected at the time of examination, so no data could be obtained as to its reliability.

The other air-cushioned hammer drill (see p. 102) has been employed in several mines in Colorado, where it is reported to be performing creditable service.

The average power consumption of the rotary hammer drills (pp. 102–103) is about 1 kilowatt per hour (1⅓ horsepower). They were employed on the Elmsford contract of the Catskill Aqueduct and were reported as particularly efficient in comparatively soft rock, drilling at times as high as 100 feet per machine in an 8-hour shift.

CHOICE OF DRILL.

The factors to be considered in the selection of a rock drill for tunnel work are, on the one hand, the cost of power, attendance, maintenance, and fixed charges, and on the other the rate of drilling, the best drill being the one that combines all these factors in such a way as to develop the greatest drilling speed for the least cost. The power cost should embrace not only the actual power at the tunnel plant, including charges for labor, fuel, interest, and depreciation, but all losses in generation, in transmission, and utilization in the drill. The wages of the drill runners and all helpers required are just as much an item of operating cost as the charge for power. The cost for maintenance includes the cost of repair parts for the drill and the charge for the time of the machinist, together with the cost of sharpening drill steel. The fixed charges should include interest and depreciation on the cost of the drills and a proportion of the administrative expenses. The rate of drilling, on the other hand, should not be based upon the speed of penetration while the drill is actually hitting the rock, but should include all delays caused by the drill, such as loss of time in preparing the set up, in shifting the drill into position for new holes, in changing drill steels, and any other interruptions properly chargeable against the machine.

Applying the specifications mentioned to the various rock-drilling machines, the hammer pneumatic drill is seemingly the one best adapted for use under ordinary conditions in driving mine adits and tunnels. Its power consumption is more than that of electric drills, but is about equal to that of the hydraulic and is less than that of the piston air drill. In the matter of attendance it has somewhat the advantage. Both the piston air and the electric types usually require at least two men—a drill runner and a helper—to operate each drill, and the hydraulic machine requires five men.[a] With the hammer drill a runner is necessary, of course, but one helper often is able to attend to two drills, or two helpers to three machines. We have just seen that there is practically no difference between the piston and hammer air drills and some of the electric drills as to repair cost. The multiplicity of parts in the rotary hydraulic machine, however, is said to have been a source of much trouble in this respect. Theoretically the hammer drills do not dull the steel so rapidly, and hence should call for less of the blacksmith's time for sharpening bits. Practically this difference is not important, because under ordinary conditions the blacksmith is rarely overtaxed, and hence the extra labor of sharpening a few bits more or less is not noticeable on the cost report. The fixed charges are such a small part of the total cost of drilling that any discrepancy in them is rarely, if ever, large enough actually to decide the question. The rate of drilling is really the greatest factor in favor of the hammer type ordinarily used in tunnels. Not only does it penetrate faster when actually drilling but, as it has not the excessive back pressure of the piston machine, it can be employed with a lighter set up, with a consequent saving of time. Then, too, its ability to start a hole at any desired point and to drill rapidly holes that point upward enable it to be used advantageously on a horizontal bar, with a saving of the one-half to one and one-half hours required to remove the débris before setting up the vertical column generally used in tunnel headings for piston air drills. The hammer drill not only demands less time for changing drill steels but conserves energy as well.

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[a] Prelini, Charles, Tunneling, 1902, p. 105.

If large tunnels are being excavated by the heading and bench method and if a large number of holes are being drilled downward, or if, because of acidulous mine water or some other reason, the water feature of the hammer drill would be unsatisfactory, the piston pneumatic drill would doubtless give equally if not more satisfactory results. Or if speed is not especially required and the drilling and mucking shifts can be alternated, the pulsator electric drill, with its large power economy, might prove the most efficient. Moreover, if the self-contained electric drills continue to be improved as they have been recently, their greater economy of power will without doubt soon outweigh their lower drilling speed and present higher maintenance charges, especially at places where electricity is readily available. On this account their development should be closely watched.

HAULAGE.

TUNNEL CARS.

Most students of tunneling methods concede that an essential, and possibly the chief, feature of the problem is the rapid removal of débris produced in blasting; but it is commonly not so well recognized that the speed with which this may be accomplished is greatly influenced by the size of the tunnel car. Large cars even when empty are heavy and cumbersome, but when full of rock they can be handled only with the greatest difficulty. To remove such a car from the heading and replace it with an empty one requires either several extra men to assist in the work, or a horse or mule must be provided for the purpose. In addition to being unwieldy, large cars occupy a greater proportion of the actual space in the heading, constricted enough at best, thus preventing the shovelers from working to the best advantage; the added height involves a waste of energy because each shovelful of rock must be lifted a greater distance, making it impossible for the men to handle as much material in a given time. With large cars it is necessary to maintain a switch or siding near the end of the tunnel in order to permit the empty cars to pass the loaded ones, and time and labor must be expended frequently in relocating the switch nearer the heading to keep pace with the tunnel advance. The smaller car, on the other hand, when empty can be tipped off to one side out of the way and replaced easily when needed, thus giving a clear track for a loaded car and obviating the necessity for a switch. In case of derailment, by no means rare in practice because of the poor condition of most tunnel tracks, the large car, even when empty, is harder to replace, and when full must sometimes be entirely unloaded before it can be put back on the track. Although a larger number of the smaller cars, each of which occasions some delay in its arrival and departure, is necessary to remove the same amount of débris, the authors' observations, based upon a study of actual conditions at a large number of tunnels, are that with proper organization greater progress is attainable by using smaller cars, preferably having 15 to 25 cubic feet capacity. The tendency at many American tunnels is toward the use of much larger cars, especially where electric haulage is employed; but the use of large cars, when analyzed, has been shown to be a handicap rather than a contribution even in those tunnels in which creditable progress has been made.

In design the cars at a majority of the tunnels visited follow the standard mining types with tilting bodies, but at a few of them other types were employed to meet special conditions. A car with a side-dumping, tilting-box body used in the west end of the Gunnison Tunnel is illustrated in figure 18. End-dumping cars are similar to this, except in that the hinge is transverse instead of longitudinal and the door is situated at the end instead of the side. The car used at the Laramie-Poudre Tunnel, which is shown in figures 19 and 20, was of the turntable type, which permitted dumping from both sides of the track as well as between the rails. As the system of car handling in the headings at this tunnel necessitated throwing all of the cars over on their sides once, and nine-tenths of them twice, on each trip, the connections between the trucks and bodies of the cars were made unusually strong. The turntables were fitted with two concentric rings (fig. 20), and the locking mechanism for securing the bodies to the trucks was so designed that when the releasing lever was fastened in place the cars were as rigid as if the bodies were riveted to the axles. A car of the rocker type (fig. 21 and Pl. III, *B*) was used with very satisfactory results in the tunnels of the Los Angeles Aqueduct. At the Nisqually Tunnel a similar car (Pl. III, *B*) but with a slightly different locking device was employed. In order to obviate the tilting body, the car at the Utah Metals Tunnel was constructed with the floor permanently inclined

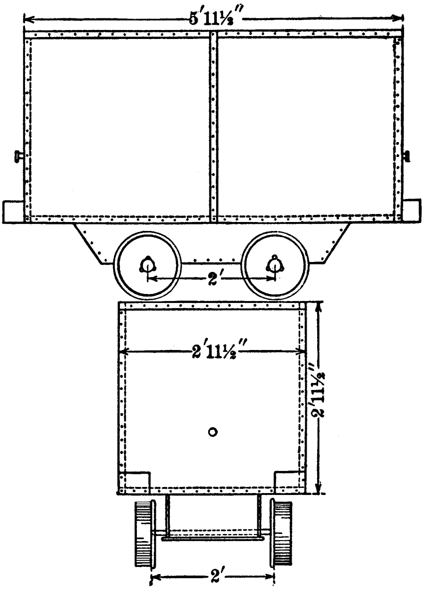


FIGURE 18.—Tunnel car used in the west end of the Gunnison Tunnel, side and end views.

toward the side door; at the Carter Tunnel the car used was of the gable type, in which the floor slopes away from the center toward doors on each side of the body. At the east end of the Gunnison Tunnel a simple open box car with the body bolted directly to the truck was employed, and similar cars are now in use in the Strawberry and Newhouse Tunnels. Although this car is ideal as regards

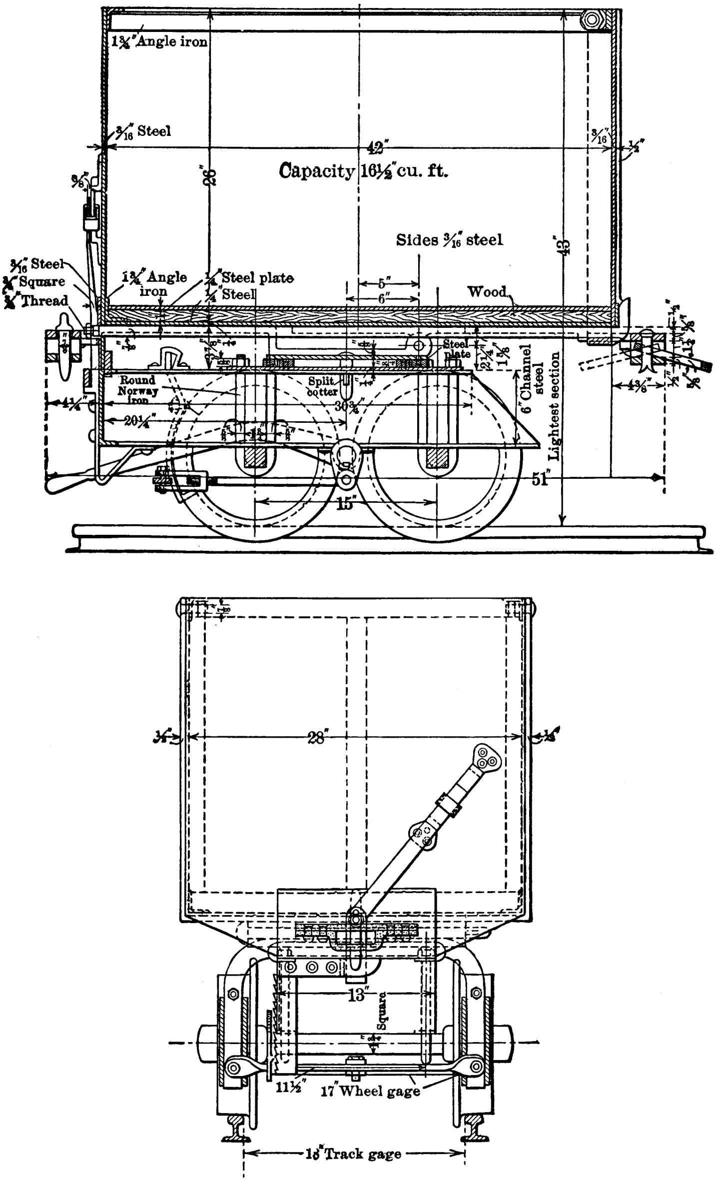


FIGURE 19.—Tunnel car used in Laramie-Poudre Tunnel, side and end views.

its simplicity, it requires special equipment for dumping, because the entire car must be turned completely over. The following table

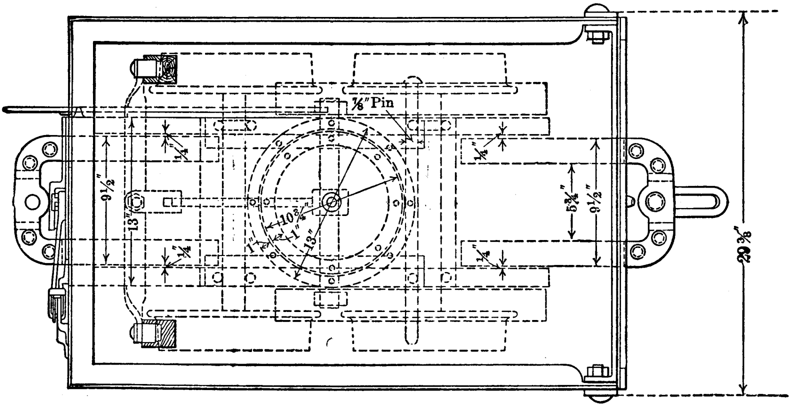
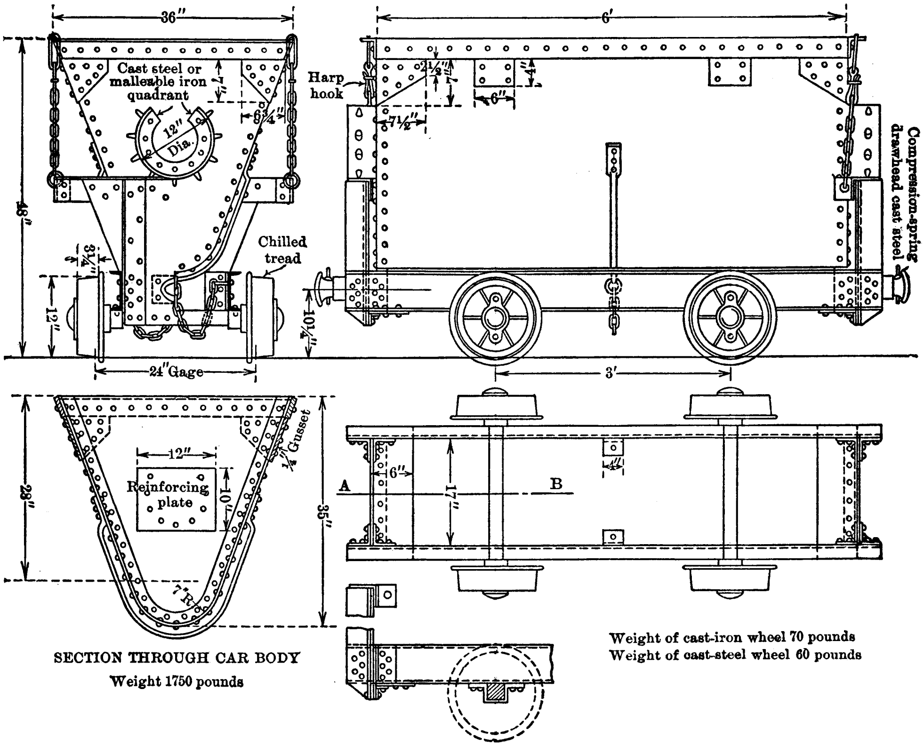


FIGURE 20.—Tunnel car used in Laramie-Poudre Tunnel, plan.



SECTION THROUGH CAR BODY
Weight 1750 pounds

Weight of cast-iron wheel 70 pounds
Weight of cast-steel wheel 60 pounds

SECTION A-B

FIGURE 21.—Rocker-dump tunnel car used on Los Angeles Aqueduct.

contains suggestive data concerning the cars used in tunnels and adits in the United States:

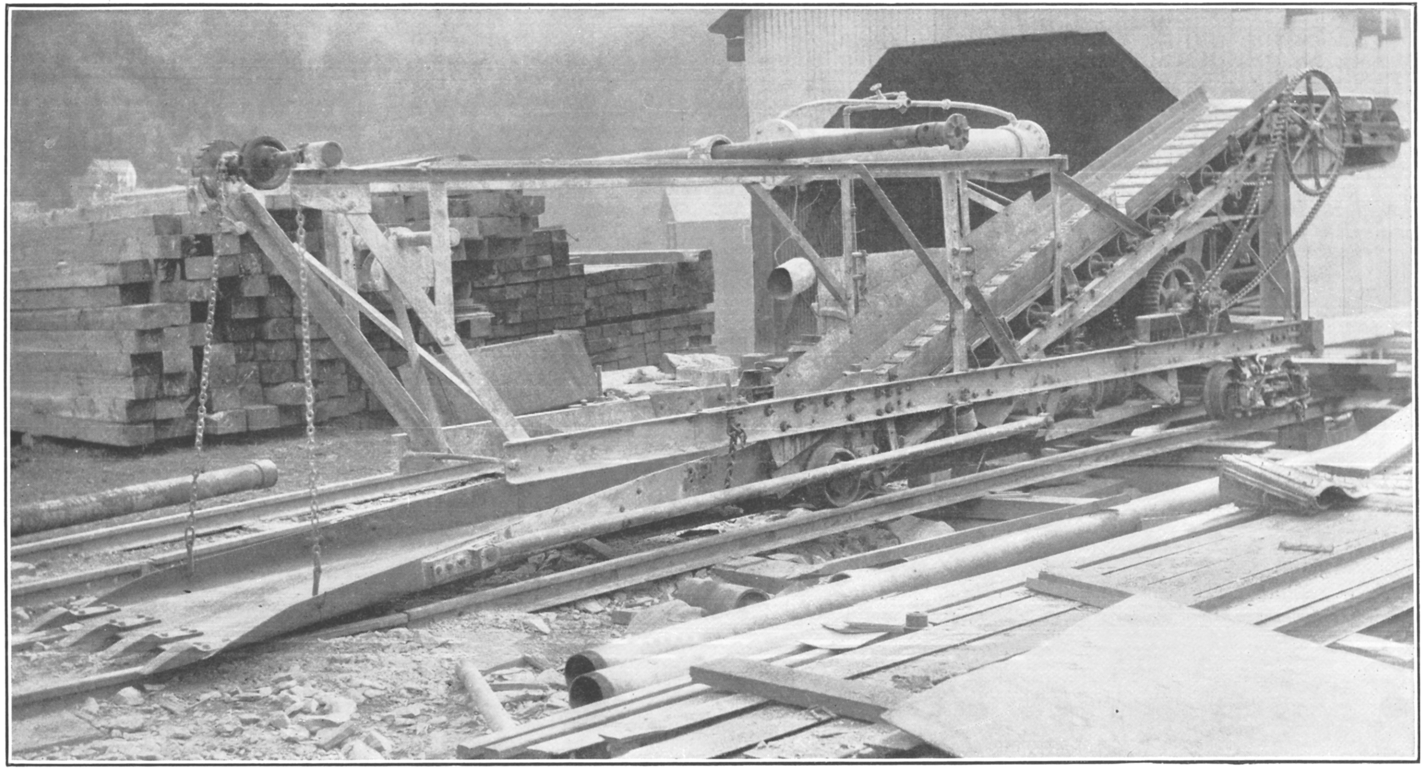
TABLE 11.—*Data concerning tunnel cars used in the United States.*

Name of tunnel.	Type of car.	Capacity.	Track gage.	Weight of rail.	Means of haulage.	Remarks.
		Cubic feet.	*Inches.*	*Pounds.*		
Buffalo (water)	Rocker dump	27	24	25	Electric	Trolley and storage battery.
Carter	Gable dump	21	18	12	Horse	
Catskill Aqueduct:						
Rondout Siphon	Side dump	40	30	25	Mules	
Wallkill Siphon	do	40	30	25	Electric	
Moodna Siphon	End dump	40	36	30	Mules	Automatic cage dump.
Central	Turntable, end dump.	30	24	20	Electric	Storage battery.
Chipeta		20	18		Mules	
Cornelius Gap		81	36		Electric	
Fort William (water)		18	18	12	Animals	
Gold Links	Turntable, end dump.		24	20	Horse	
Grand Central sewer	Bucket and flat car.	14	24		Hand	
Gunnison, west	Side dump	35	24	16		
Gunnison, east	Box	54	24	16	Electric	Derrick dump.
Laramie-Poudre	Turntable, end dump.	16	18	16	Mules	
Lausanne	End dump	80	42	30	Electric	Cradle dump.
Los Angeles Aqueduct:						
Little Lake	Rocker dump	32	24	16	Mules and electric.	
Grape Vine	do	32	24	16	Electric	
Elizabeth Lake	do	32	24	35	do	
Lucania	Turntable, end dump.	22½	18	20	Horse	
Marshall-Russell	do	28	24	16	do	
Mission	Side dump	22½	18	25	Electric	
Newhouse	Box				do	Revolving dump.
Nisqually	Rocker dump	27	24	16	do	
Rawley	Turntable, end dump.	17	18	16	Mules	
Raymond	do	32	18	16	Horse	
Roosevelt	do	16	18	20	Animals	
Siwatch	Side dump	33	22	30	Electric	
Snake Creek	End dump	20	18	20	Horse	
Stilwell	Side dump	22	24	30	do	
Strawberry	Box	47	24	25	Electric	Derrick dump.
Utah Metals	Side dump	32	24	34	do	Sloping floor.
Yak	do	33	19	30	do	

LOADING MACHINES.

Many attempts have been made to utilize machinery for loading tunnel cars. In several of the larger tunnels, intended for railway purposes, power shovels similar to those used in grading or in open-cut mining have been very successfully used in removing the broken rock of the bench after blasting. The ordinary steam shovel is generally employed, a few minor alterations being made so that it can be operated by compressed air. Power shovels operated by compressed air are also employed in some of the mines in the Joplin (Mo.) district.

The "mucking machine" illustrated in Plate VI was used successfully during the excavation of the Hummingbird Tunnel at Burke, Idaho. Its principal feature is an oscillating trough or shovel armed with teeth and driven by a compressed-air piston in such a manner that the forward stroke is appreciably faster than the return. When in operation the teeth rest upon a steel plate under the muck pile,



"MUCKING MACHINE" USED AT HUMMINGBIRD TUNNEL, BURKE, IDAHO.

and as the shovel is fed forward the broken rock is forced by the jerky motion backward along the trough and discharged upon a belt conveyor which delivers it to an ordinary mine car at the rear. The entire machine is mounted upon a wheeled track or framework and is fed forward by a second compressed-air piston connected with a crossbar that can be jacked against the sides of the tunnel. It is essential that the area of this piston be smaller than the one that drives the shovel, for then if the latter encounters a bowlder or other obstruction too solid for it to dislodge, the entire machine can move forward and back with the stroke of the larger piston. By this means the machine is not only prevented from injury before the obstruction is removed, but in many cases it will work the bowlder aside without any assistance. One man is required to operate the machine and two more are required to tram the car to and from the end of the conveyor and to shovel the rock out of the corners of the tunnel into the trough, for the machine does not swing from side to side, but merely cuts a swath down the center of the tunnel and hence leaves some material piled on each side. The machine is reported to have reduced the time required to clean the tunnel from six to two and one-half hours and to have made possible a material increase in speed of driving.

A power loader of a somewhat different type was introduced in the excavation of the bench at the Yonkers Siphon. It consisted of a chain and bucket conveyor, similar to that used in mill elevators and on some gold dredges, which delivered the material to a hopper whence it was carried to the tunnel car by a flat endless belt.

Owing to the hardness of the rock and the prevalence of huge bowlders, some weighing over a ton and necessitating frequent stops for repairs, this machine was unable to compete satisfactorily with hand loading. The machine was used on the surface for loading rock for use in concrete construction, and at this work it is said to have given excellent satisfaction. The material was taken from the dump pile produced in excavating the heading of the tunnel, in which the rock was broken more uniformly into small fragments than the material produced in blasting the bench. However, the size of this machine precludes its use, without considerable modification, in a small tunnel or heading.

MEANS OF HAULAGE.

In practically all tunnels of any length in the United States, either animals or electric motors have been or are employed to haul the tunnel cars. In Europe, notably at the Simplon and the Loetschburg Tunnels, compressed-air locomotives were used successfully. But although those machines are employed to some extent in this country in mining and industrial work, they have failed to give satisfaction at tunnels where they have been tried, chiefly because of the cost of high-pressure air, the maintenance of charging stations, the time lost in charging, etc. Many mines also are equipped with cable haulage; but because of the constantly increasing length of haul as the heading advances, the use of this system in tunnel work requires such frequent delays and loss of time in extending the cable system that it is hardly suited for tunnel practice. On the other hand, gasoline locomotives, which have recently proved most successful for coal mining, are in most particulars especially well adapted for tunnel work and deserve full consideration.

The principal advantage of animal haulage is the smaller cost of installation; moreover, it requires no special intelligence on the part of the driver, and the ability of the animals to step across the track at the tunnel headings obviates the necessity of a switch. On the other hand, the costs of maintenance and operation for animal haulage not only are high but go steadily on whether the animal is working or not and are influenced only slightly, if at all, by the size of tonnage handled. For these reasons animals are not economical for use in long tunnels, because the saving in installation expense is soon destroyed by the increased operating costs. Then, too, the odors arising from the track are offensive and disagreeable when animals are employed and their respiration vitiates the underground atmosphere, necessitating more ample ventilation. As far as efficiency is concerned, there is little if any difference between horses and mules, although the latter are considered by some to be the sturdier. Mules, however, are better fitted for work in low tunnels, because they are usually somewhat smaller than horses and, being less nervous, do not throw their heads violently up and back when anything touches their ears.

Electric mine locomotives may be divided into two classes—those operated from a trolley system and those obtaining their electrical current from a storage battery. The former are so familiar as hardly to require description. They generally consist of two motors, ruggedly constructed to withstand rough usage and protected from dust and moisture, mounted upon a cast-iron or structural-steel frame, which also carries the trolley, controller rheostat, and other accessories. The sides of the frame may be placed either inside or outside of the wheels. When placed outside, more space is available for the motors and other equipment and the various parts of the machine are more readily accessible. When the frame is placed inside the wheels the car has a smaller over-all width and is therefore more suitable for narrow tunnels.

The storage-battery locomotive is similar in most respects to the trolley machine, except that provision must be made for carrying the necessary batteries. In most cases the batteries are carried directly upon the motor itself, but the locomotive installed at the Central Tunnel is somewhat unique, in that the batteries are placed upon a separate car or tender. When the machine is handling cars in the tunnel it obtains its current from the battery; upon reaching the tunnel mouth the tender is left on a side track, where it is accessible for recharging, and a trolley, with which the locomotive is also equipped, is employed for switching.

Electric locomotives are compact and simple in construction and do not emit smoke, gas, or disagreeable odors. They are more rapid and are capable of hauling a much greater load than either a horse or a mule, and the cost of the power used is not nearly so great as the cost of forage. But, on the other hand, they require the installation of extra machinery in the power plant, an expensive trolley wire, or a troublesome storage battery, and the roadbed and track must not only be heavier in construction, but usually the rails must be bonded to make them good electrical conductors. The disadvantage of the cost of the extra electrical machinery is, of course, partly offset by the fact that it can be utilized also to operate the ventilating machinery and to furnish illumination for the tünnel. The use of trolley wires in the restricted tunnel space, however, introduces the grave danger of serious injury to persons coming in contact with them.

Gasoline locomotives consist essentially of a frame, usually of cast iron, upon which are mounted the gasoline engine (usually 4-cylinder), the necessary transmission system containing gears and clutches, together with the carburetor, magneto, cooling system, and other accessories. In external appearance they are not unlike the electric locomotives described above. Two forward and two reverse speeds are usually provided in the machines manufactured in this country, a lower one of 3, 4, or 5 miles per hour and a higher speed double that of the lower. The drawbar pull ranges from 1,000 to 4,000 pounds, according to the size of the locomotives. In some of the machines the exhaust gases from the engine are passed through a tank containing a solution of calcium chloride, which cools the gases and is said to remove all offensive odors from them. In a German-made machine the exhaust gases are sprayed with water to produce the same effect.

The gasoline locomotive combines most of the advantages of both electric and animal haulage. It is self-contained and independent of a central station or any other outside source of power. It is fully as rapid as the electric motor ordinarily used in tunnels and is capable of handling an equal load. The fuel for a gasoline locomotive can be obtained readily in almost any locality, and the machine does not consume fuel when it is not running, a matter of great importance in tunnel work, where interruptions necessarily occupy a large percentage of the time. Another advantage, although perhaps not so important for tunnel work, is the fact that the haulage system may be expanded by the addition of extra units without alteration in the power plant.

The following table, based upon replies from operators and users of gasoline locomotives received in answer to inquiries sent out by this bureau early in 1912, shows the cost of haulage with these machines:

TABLE 12.—*Operating cost of gasoline haulage.*

Name of operator.	Months in use.	Average number of trips made by each locomotive.	Average gross tonnage, loaded trip.	Average gross tonnage, empty trip.	Average length of haul, one way.	Average number ton-miles daily.	Fuel used per day.	Cost of fuel per day.	Cost of labor per day.	Cost of lubricating oil per day.	Operating cost, exclusive of repairs.	Cost per ton-mile, not including repairs.	Average daily cost for repairs.	Cost per ton-mile, including repairs.
					Feet.		*Gals.*							
Alden Coal Co., Alden, Pa.	5	20	50	13.5	1,750	1,270	9	$0.86	$4.48	$0.20	$5.54	$0.004	(a)	......
Breese-Trenton Mining Co., Trenton, Ill.	9	30	30	11	2,200	515	20	2.00	5.90	.50	8.40	.016	(b)	......
Durham Coal & Iron Co., Chattanooga, Tenn.	9	12	20	6	1,000	59	15	1.65	3.50	.85	6.00	.101	$1.70	$0.13
Henderson Coal Co., Pittsburgh, Pa.	7	12	40	12	5,200	615	8	1.04	5.30	.32	6.66	.011	.33	.012
H. B. Swope & Co., Madera, Pa.	11	20	32	10	3,000	475	15	2.25	6.24	.25	8.74	.018	.57	.020
Pocahontas Smokeless Coal Co., Welch, W. Va.	14	15	35	10	2,400	307	12	1.44	4.50	.35	6.29	.02	.25	.021
Midvalley Coal Co., Wilburton, Pa.	12	7	60	27	11,500	1,320	15	1.50	4.14	.12	5.76	.004	.33	.005
Munroe Coal Mining Co., Barnum, W. Va.	5	20	21	6	3,000	300	20	2.20	3.75	.15	6.10	.02	1.00	.024
Roane Iron Co., Rockwood, Tenn.	18	9	36	14	5,800	600	13	1.30	3.50	.41	5.21	.009	(c)	......
Shade Coal Mining Co., Mount Pleasant, Pa.	4½	12	61	9	2,000	740	15	2.20	2.50	.86	5.56	.0075	.22	.008
Tennessee Consolidated Coal Co., Tracy City, Tenn.	14	35	19.5	7	2,500	440	15	2.50	4.10	.65	7.25	.016	.50	.018
Vaughen Coal & Coke Co., Roderfield, W. Va.	11	10	27.5	14	10,500	830	16	3.60	4.50	.90	9.00	.011	(b)	......

[a] None as yet. [b] Not given. [c] Not excessive.

Practically the only disadvantage of the gasoline locomotive is the character of the exhaust from the engine, but this can be eliminated by proper and adequate ventilation. If the gas were confined in a small unventilated space the air would soon become unfit for breathing, but as the greater part of the time the motor is traveling back and forth in the tunnel and as a large volume of air is, under proper management, being supplied by the ventilating blower, the exhaust gases from the engine are quickly diluted to harmlessness. It is essential, however, that the blower be arranged to deliver air to the heading through the ventilating pipe rather than through the tunnel, in order that the air may reach the workmen as pure as possible. Even if it should be necessary to operate the blower at full load, the added cost of doing so would be more than repaid by the saving effected by the gasoline haulage.

DUMPING DEVICES.

The box cars used at the Strawberry Tunnel were dumped by an electrically operated stiff-leg derrick. The hook in the derrick block carried a bail that engaged trunnions, one at either end of the car. The trunnions were placed in such a way that when an empty car was picked up by the bail the weight of the running gear was sufficient to hold the car upright, but if the car was loaded its center of gravity was above the trunnions. A spring-actuated pin, placed in one leg of the bail and engaging a hole in the car body above the trunnion, prevented the car from overturning until it was swung out over the place where the rock was to be deposited, when by pulling a rope the attendant could disengage the pin and permit the car to turn over and deposit its contents. It would then automatically right itself and could be swung back on the track. The derrick was mounted on wheels so that it could more easily be moved ahead, but moving was necessary only at intervals of three to six months.

Among the advantages claimed for this system of dumping was that it could be operated by the train crew, the motorman running the hoist and the brakeman adjusting the bail, thus saving the labor of a dumping gang. Then, too, it gave a much larger dumping area, with a consequent saving of the time that with the ordinary mine car is lost in shifting tracks, etc. But this saving was offset in part by the settling of the dump and on this account the moving of the derrick was accomplished with great difficulty. It is probable that some of this annoyance could be avoided in a future installation by using very wide wheels similar to the type used on roller trucks for moving houses. However, the derrick is expensive, costing when erected at the tunnel approximately $3,600, of which hardly more than $1,500 could be realized from its sale after the completion of the work. For this reason its use must extend over a considerable length of time in order that the saving in wages may repay the original cost.

At the Newhouse Tunnel the loaded cars were rolled into a cylindrical steel framework having rails at the bottom and a set of angle-iron guides at the top with just enough clearance space between them to hold the car firmly. The entire apparatus was then revolved by an electric motor until overturned, emptying the contents of the car, and it was then righted by continued revolution and the car removed. Although used here only for ore cars, the dumped material falling into a bin for shipment, it offers a satisfactory and reasonably inexpensive means of dumping the more durable solid-body and truck cars and could doubtless be applied to tunnel dumps by the use of a light trestle or similar structure.

Almost any one of the various cradle dumps used at many coal mines can readily be adapted to tunnel work by mounting it upon a stout frame of logs or large timbers, which could be pushed forward along the top of the rock pile, as necessary. By this means it is possible to eliminate hinges and turntables between the body and the truck of the car, thus simplifying and strengthening its construction. One of these cradle dumps was used at the Lausanne Tunnel. It was not expensive and saved considerable time in dumping cars and in keeping the rock pile in proper condition. It was pushed forward by the motor every two or three days, requiring only a few minutes for the operation. Figure 22 illustrates a similar dumping device used at the Cameron mine, Walsenberg, Colo. It has the added advantage of being mounted on a turntable, thus giving nearly double the top width of dump attainable with ordinary cradle devices. As described in Mines and Minerals,[a] the dump consists essentially of three plates of $\frac{1}{8}$-inch iron 3 by 4 feet in size. To the top plate are bolted a pair of mine rails with the ends bent up into horns. This upper plate revolves on the mine car axle *e*, the bearings for which are supported upon a mine rail and bolted to the middle plate. A piece of channel iron, *d*, is bolted to the middle plate, and upon it the dump falls back after a load of rock has been discharged. The upper plates as a unit revolve upon the two annular pieces of iron, *b*, 22 inches in diameter. The king-pin, *c*, is 1 inch in diameter and the plates, where it passes through them, are reinforced by a piece of $\frac{1}{2}$ by 3 inch bar iron. The lower plate is supported by four short lengths of 12-pound mine rail.

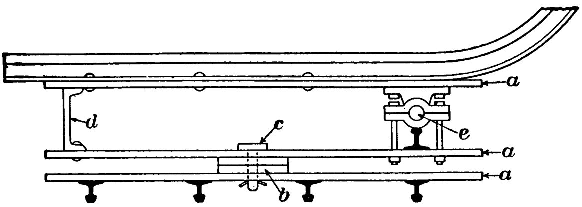


FIGURE 22.—Turntable cradle dump used at Cameron mine.

INCIDENTAL UNDERGROUND EQUIPMENT.

TUNNELING MACHINE.

Although tunnels have been constructed for mine drainage, irrigation, and supplying water to cities for thousands of years, they were so few in number during ancient times and constructed at such

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[a] October, 1911, p. 158.

irregular intervals that there was no great incentive to improve upon the methods ordinarily employed in building them. However, with the advent of the steam railroad it was soon realized that the desirability of maintaining level gradients would necessitate the driving of many tunnels, and the active minds of inventors were immediately directed toward the problem of making a machine that would do this work more or less automatically.

The first tunneling machine of which any record could be found was constructed at Boston in 1851 for use in the Hoosac Tunnel. It weighed 70 tons and was designed to cut in the face of the tunnel a circular groove 13 inches wide and 24 inches in diameter by means of revolving cutters. The trial of this machine in the tunnel proved unsuccessful, and a distance of only about 10 feet was cut with it before it was abandoned. In 1853 the Talbot tunneling machine, which was designed to make an annular cut 17 feet in diameter, leaving a cylindrical core to be removed by blasting, was tested near Harlem, New York, but also proved unsuccessful. Later a smaller machine was constructed, adapted to cut an 8-foot annular groove. The smaller machine, although less unwieldy than its predecessor, also proved a complete failure after $25,000 had been expended upon it. Numerous machines constructed upon almost every conceivable principle have been experimented with since 1853, but most of them have been entirely discarded.

However, it is not safe to predict that a tunneling machine will not be constructed to perform this work in the future because, difficult as the problem of designing such a machine appears, the obstacles in the path are no greater than they have been in scores of other instances where slow and costly manual methods have been superseded by less expensive and more expeditious mechanical processes. The invention of some new rock-cutting device, or the material improvement of some of those now known, may simplify the problem to such an extent that the construction of a successful tunneling machine will be rendered comparatively easy. The simple device of putting an eye in the point of a needle made the sewing machine possible; the breech-loading gun was a complete failure until the brass cartridge was invented; and not even the genius of a Langley or a Wright could construct a flying machine until the internal-combustion engine had reached its proper stage of development.

One of the great obstacles encountered by legitimate investigators in this field has been the great difficulty of obtaining funds, for with this machine, as well as with any other new and complicated machine built to operate under difficulties and strains that can not be measured in advance, costly experimental work is necessary. It must be remembered that machines of the size and strength necessary to cut the entire face of a tunnel as a single operation are of necessity costly, and their maintenance during the trial stages is extremely expensive. For this reason success can hardly be expected unless the inventor, or the company back of him, commands large funds. The failure of one badly designed and inadequately financed machine after another and the suspicions aroused in the minds of possible investors by the untruthful and flamboyant " literature " that has been issued by too many alleged tunnel-machine companies in their efforts to " work the public " have caused most people to look upon machines of this kind with extreme distrust, much of which is indeed just, for even a casual scrutiny of the claims of many of these concerns shows clearly the fictitious character of the statements.

ILLUMINATION.

With few exceptions, illumination for tunnels and adits in the United States at the present time is furnished by electricity, acetylene gas, or candles. The smoky open-flame miners' oil lamp is occasionally used in tunnels situated in the coal-mining districts, and, of course, under conditions that prohibit the use of an open flame, safety lamps must be employed. When acetylene gas is employed it is usually generated in portable lamps, but during the work on the water conduit for Washington, D. C., in 1899, this gas was manufactured at a plant on the surface and carried by pipes underground, where it was burned in jets at regular intervals. Coal gas was similarly employed at the Mount Cenis Tunnel, which was started in 1857 and opened for traffic in 1872. The following table, however, shows the present practice with regard to means of illumination:

TABLE 13.—*Means of illumination at various tunnels.*

Name of tunnel.	Method of illumination.
Buffalo (water)	Electric lamps.
Carter	Acetylene lamps and candles.
Catskill Aqueduct	Electric lamps at intervals and usually a cluster of lamps in the headings.
Central	Acetylene lamps.
Fort William (water)	Electric lamps (16 candlepower) every 75 feet and one 32-candlepower in heading.
Gold Links	Candles.
Gunnison	Electric lamps, cluster in heading, and candles.
Joker (drainage)	Electric lamps.
Laramie-Poudre	Acetylene lamps for drillers, candles for muckers.
Lausanne	Miners' oil lamps and safety lamps.
Los Angeles Aqueduct	Electric lamps and candles.
Lucania	Acetylene lamps.
Marshall-Russell	Do.
Mission	Electric lamps every 200 feet, cluster in heading, candles.
Newhouse	Electric lamps at stations; acetylene lamps in heading.
Nisqually	Electric lamps every 75 feet; cluster in heading.
Ophelia	Candles.
Raymond	Electric lamps every 200 feet; cluster in heading.
Rawley	Acetylene lamps.
Roosevelt	Electric lamps.
Siwatch	Electric lamps every 200 feet and candles.
Snake Creek	Acetylene lamps.
Stilwell	Electric lamps in heading, candles.
Strawberry	Electric lamps every 135 feet; cluster in heading.
Utah Metals	Electric lamps at switch; acetylene lamps in heading.
Yak	Electric lamps.

Neither candles nor the open-flame oil lamp can be recommended as a means of lighting a tunnel or adit during construction. Practically all that can be said in their favor is that they require a much smaller initial outlay than electricity or acetylene, yet they are more expensive per unit of light than either acetylene or electricity, consume a greater amount of oxygen, and give off a correspondingly greater amount of noxious gases. Candles not only do not give enough light, but what they do supply is flickering and unsteady unless there are no drafts, and as they are quickly extinguished by the exhaust blasts from air drills they can not be placed to light properly the work of the drillers; hence with their use the efficiency of a high-priced drillman is greatly reduced. Candles are often wasted or dropped into the muck pile, an item of loss that may amount to a considerable sum in the long run. The open-flame oil lamp can not be prevented from giving off soot and smoke, which obscure the light thrown on the work, and the soot, collecting in the miner's throat and lungs, irritates the mucous membranes and renders them easily susceptible to disease.

Electric incandescent lamps possess a number of advantages for tunnel work. They give a brilliant and steady light, one that is not affected by drafts and neither pollutes the air with soot nor vitiates it by consuming the oxygen. By combining several of them in a cluster plenty of light in the heading is obtained for the drillers and shovelers, tending toward efficiency. To offset this advantage, however, the fact remains that unless they are used in connection with electric locomotives, drills, or similar machinery, the cost of lamp installation is almost prohibitive; even with the electric appliances in use, the extra wiring and the lamps themselves are expensive, the lamps entailing considerable loss through breakage. Electric lights are also at a disadvantage because they are not easily portable, and the removal and replacement of bulbs and wires in the heading before and after blasting complicate an already involved situation. Moreover, this means of illumination is uncertain, especially in wet tunnels, because the chance occurrence of a short circuit through moisture, accident, or carelessness throws the entire work in darkness, and, if other means of lighting are not at hand, stops all work until the trouble can be remedied. Again, whereas the use of electricity underground is always attended with some danger, this is especially true in the case of lighting appliances. The supposition is that the wires are protected, but the rough usage to which they are subjected soon destroys insulation, rendering persons who handle them (as they must do frequently) subject to severe shock.

One is tempted to say that the ideal means of tunnel illumination is found in the portable acetylene lamp, combining as it does the advantages of other illuminants while avoiding most of their defects. It may be obtained on the market to-day in a number of different designs and sizes adapted for practically every kind of work; the one most generally observed at the tunnels visited was about the size of an ordinary fruit can and capable of burning for 8 to 10 hours on one charge of carbide and water. Although too large for use on a cap, it was provided with a hook so that it could be suspended from any convenient place. Lamps suitable for wearing on a miner's cap are obtainable and will burn for two or three hours without recharging—an operation easily performed in two or three minutes. The initial expense of an acetylene lamp is not high and, with the possible exception of the electric arc, it furnishes the brightest known artificial light used for underground work, consuming only one-fifth as much oxygen as candles. It is ordinarily provided with a reflector, which not only concentrates the light upon the work where it is needed, but shields the flame from drafts so that it will burn steadily unless placed directly in front of the exhaust from an air drill. Extensive use by some of the larger mining companies in this country has shown that the cost of the carbide is much cheaper than either oil or candles, the use of acetylene lamps cutting the cost of light practically in half. At the Saginaw mine, Menominee Range, Mich., the cost is reported as only 2 cents per shift of 10 hours. Such lamps require practically no attention, are completely portable, and are not subject to breakage as are incandescent lamps. By giving the workman plenty of light his efficiency is not only increased but he is better able to guard against the dangers of underground work, such as an insecure roof, an unexploded stick of dynamite in the muck pile, or any other of the many dangers to which he is at all times exposed.

TELEPHONES.

Although it has been repeatedly stated in newspapers, engineering periodicals, and even by State legislatures that every mine should be provided with a telephone system, the importance of telephones in tunnel work can not be too often reiterated, not alone because of the greater safety they insure but on the ground of efficiency and economy as well. The sources of accident in tunnel work are too numerous to mention—falls of roof, caves, premature or delayed explosions, flooding, and noxious gases being some of the more common. When an accident occurs in a tunnel that is equipped with a telephone system, not only can assistance be summoned quickly, but provision can be made beforehand for the care of injured men upon reaching the surface; if professional help can be summoned and due preparation made while the men are still on the way from the heading, invaluable time is saved, for there are many instances where prompt medical attention has decided the question of life or death. Then, too, failure to obtain a proper round of holes in the given time, difficulty in blasting them to the full depth, or any of the many problems that commonly arise in tunnel driving call for a decision on the part of the foreman as to the method of procedure. Ordinarily the man intrusted with this position is capable of meeting such conditions as they arise, but it stands to reason that the work of the shift will be more efficient if the foreman can be in touch constantly with the superintendent and when in doubt receive suggestions and advice from the more experienced man's better judgment. Delay can be avoided in good part if the tunnel is equipped with a telephone, because the necessity that involves sending for fresh materials, tools, powder, etc., can either be foreseen and provided for promptly from the outside without the loss of a man from the heading crew, or when unexpected emergencies arise only half the usual time is necessary to obtain the needed supplies. Causes of accident and delay can not always be foreseen, it is true, but they can be met promptly and further damage to men and property can be prevented by the use of the telephone; that these advantages are appreciated is shown by the fact that most of the tunnels and adits examined in the field were equipped with telephones.

The type of telephone equipment should be carefully chosen because every telephone is not suited for underground use. For use in tunnels the instrument must be waterproof, dustproof, and to be useful, it must be placed as near the heading as possible; it must be designed to withstand the frequently recurring concussions of blasting. In the most successful types of mine telephones the mechanism is placed in a heavy metal casing in such a way that the essential parts are instantly accessible upon opening the outer door, but are tightly sealed when it is closed. The more delicate mechanism is guarded further by an inner door, also of iron, and the wires are protected so that water can not enter the casing. The bells must necessarily be placed outside, but they are protected by a metal hood, which, however, does not prevent their being heard for a considerable distance.

The telephone line for tunnel work is somewhat simpler than a similar line on the surface, because no poles are required and the wires can be strung from ordinary glass or china insulators fastened to plugs in the roof or to light cross timbers. Ordinary bare iron wire can be used, but much better results are obtainable where rubber-covered wire is employed; and for the same reason a full metallic circuit is desirable, although the telephone may be operated with only one wire by using a ground connection for the return. But as the usefulness of a telephone system is measured entirely by its reliability, the best is in the end by far the cheapest.

To guard against noise and the effects of concussion the telephone should not be placed nearer the heading than several hundred feet. Although such an installation is convenient for anyone in the tunnel desiring to call up the office, it makes difficult and sometimes even impossible the obtaining of any response to a call originating on the surface. To obviate this difficulty, the use of an extension loud-ringing call bell is recommended, which, if placed behind a jutting rock or in some similar protected position, apprises the foreman at the heading instantly of any call at the telephone. Such a bell should be connected with the telephone circuit by a flexible insulated cable mounted upon a reel in such a way that the bell may be advanced regularly to keep pace with the tunnel progress and need be never farther than 200 feet from the heading. When the cable is extended to full length, perhaps 1,000 feet, the telephone should be advanced to a point as near the heading as practicable and the extra cable reeled up once more.

INCIDENTALS.

Among the many devices used to save time and promote efficiency underground are the hose supporter and the drill rack, both of which can be made readily by any tunnel blacksmith. The hose supporter consists merely of two telescoping pieces of iron pipe, the length of each being about three-fourths of the width of the tunnel. In operation the hose is placed over the pipes, which are then extended until their pointed ends fit into convenient niches on either side of the tunnel near the roof; the pipes are clamped into position firmly by a threaded key which is provided for this purpose. By using two or three of these spreaders the hoses are kept clear of the shoveler, who is thus saved no little trouble and annoyance and is able to work to better advantage. The drill rack is simply a rack for separating different lengths of drill steel. A satisfactory form consists of an **A** frame made of 4 by 4 inch timbers, into which iron pegs are driven at convenient intervals. The segregation of the sharp drill steels on this rack enables the helper to pick out the proper length with assurance and dispatch.

TUNNEL-CONSTRUCTION METHODS.

The following discussion of methods of tunnel construction is restricted chiefly to those in which the entire cross section is excavated in one operation. The majority of tunnels and adits driven for mining work and many tunnels intended for irrigation and water supply are small enough to be driven in this manner; but in the construction of the larger undertakings, such as are required for railroad or similar purposes, it is customary to drive a pilot tunnel or heading, as it is sometimes called—although the term is also employed to designate the advancing end of any tunnel—previous to the main body of the work which then consists in enlarging the smaller excavation to full size. The latter method, in addition to lowering the average cost of the entire work, because the process of enlarging is much easier and less expensive than that of driving the heading, also gives a valuable preliminary insight into the conditions that must be encountered later by the main tunnel, and enables the constructor to anticipate emergencies and make provision for them in his plans, thus aiding to prevent accidents and loss. However, as this bulletin is to be confined chiefly to mine adits and small tunnels, a discussion of the various phases of the "heading-and-bench" system can not be treated here as such, although the methods used in excavating in one operation the entire section of a small tunnel are in most cases applicable to the driving of headings for larger tunnels. Local conditions at each project necessarily modify methods to such an extent that it is impossible to make a general analysis to fit all cases, but the discussion following is intended to bring out some of the more important features of the methods employed in the various operations of drilling, blasting, mucking, and timbering as they are applied to the driving of mine adits and tunnels.

The methods employed in the operation of drilling, aside from the mere running of the machine, are concerned chiefly with the number of drilling attacks, the mounting of the drills, and the number, depth, and direction of the holes. The choice of ammunition, including the fuse and caps as well as the explosive, and the manner of loading and firing the holes are the main factors influencing the efficacy of blasing. Considerations of safety, however, demand that suitable provision be made for storing the explosive and thawing it as needed and that proper precautions be observed in its use. The number of men and the positions in which they work, the method of handling the muck cars in the heading, and the use of steel plates upon which to shovel are the salient features of mucking, whereas timbering is concerned chiefly with the materials employed and the different types or methods of using them.

DRILLING.

NUMBER OF SHIFTS.

Relative to the number of shifts preferable for drilling, one of the chief advantages claimed for the single shift per day is economy. By having the débris cleared from the heading by the shoveling crew at a separate time, the drillmen, upon reaching the face, are able to start immediately to work setting up the machines and preparing to drill the round; there is therefore no waste of time or labor on the part of these men or the helpers in shoveling out débris preparatory to mounting the drills. This method is especially economical when vertical columns are employed. During the process of drilling, the operators and their helpers are not interfered with or hindered in any way by the shoveling crew, and there is therefore a saving of that loss of motion that can hardly be prevented when two crews are working simultaneously in the heading. Moreover, as there is no delay in getting started, the round of holes can ordinarily be completed within the allotted time, and even if this can not be done plenty of extra time is available without delaying the following shift. The drilling and mucking shifts can be distributed so that there is no loss of time and wages while the men are waiting for smoke and gases produced in blasting to be removed from the tunnel—of cardinal importance where the provisions for ventilation are inadequate. These considerations go to support the contention that the actual excavation cost per foot of tunnel is lower with this method than with other systems.

On the other hand, by employing a single drill shift the daily progress in driving the tunnel is necessarily limited to the advance gained from the one attack, and therefore the completion of the work must inevitably be delayed. Most tunnels are practically worthless until completed. If their construction is not pushed as rapidly as possible not only is the capital invested in the equipment, tools, etc., tied up much longer than necessary and the cost for interest and the depreciation charges proportionally increased, but there is also a delay in the realization of the benefits to be derived from the tunnel, which in most cases is more than sufficient to offset any saving in excavation cost. For example, if an adit is being driven to drain a mine the extraction of additional ore below water level is greatly delayed; or if the adit is intended to lower the cost of transporting the ore to the surface the loss on the additional tonnage handled in the old way, owing to the delay in its completion, should be charged against this system of operation. Similarly, with an irrigation tunnel, the entire season's crops may be lost from the longer time required to complete the tunnel if it is excavated by the one-shift method. Then, again, the cost for administration and many other of the fixed charges are operative during the period of construction, independent of the number of shifts per day, and as the daily progress increases with the additional attacks per day, the proportionate charge against each foot of tunnel driven will be smallest when the greatest number of shifts are employed. Although by a saving of the time and wages of workmen there is a seeming economy in the cost of excavation by the one-shift method, when factors that reach deeper are considered, in most cases the ultimate cost of the tunnel will be lowered by methods that make for speedier completion.

Greater progress is undoubtedly attained with two shifts per day than with one, and if the work is properly organized there need be only little added excavation cost. The usual custom with this system is to have the shovelers start work somewhat in advance of the drillers and to work at first in removing the broken rock directly at the face, so that the drillers may set up their machines promptly. At some adits and tunnels where two drill shifts were used the drilling and mucking took place simultaneously, the drillers themselves attending to the work of clearing out for the set-up. Of these two methods the former is preferable, not only because it economizes the time and exertion of higher-priced men but also because the length of time when both crews are at work together in the heading—and consequently the inevitable amount of interference and interruption—is thereby lessened. At a few places three crews of shovelers were required to remove the rock broken by two drilling attacks. This system is obviously expensive because the cost of the extra shovelers must be charged against a footage only slightly, if at all, increased by their efforts, and it entails for two of the three shifts the disadvantage of simultaneous work just mentioned; it is therefore not desirable.

The consensus of usage at tunnels and adits where the best results in driving have been achieved, both in this country and abroad, leads to the conclusion that the three-shift system of attack is the most desirable. This method has a number of opponents who claim against it four chief disadvantages: (1) That time is lost on the part of the drillmen in getting the machines set up and in operation; (2) that the greater number of men crowded in the restricted space of the heading are in each other's way and therefore unable to work to the best advantage; (3) that the men must be paid for time wasted in waiting for the smoke and gases produced in blasting to be cleared from the heading; (4) that the system makes no provision for delays due to adverse conditions. As is pointed out subsequently, the time consumed in setting up the machines can be made negligible by the use of suitable methods of mounting and by properly directing and blasting the round of holes. Some crowding is, of course, unavoidable, but it is more than offset by the gain in efficiency resulting from the various incentives that can result only from the three-shift method. To begin with, the shovelers have constantly before them the necessity of removing the waste rock before the drillers have finished their work and are therefore unconsciously speeded up by the competition. At the same time the drill men endeavor to have their holes finished by the time the tunnel is cleared in order that no delay may be attributed directly to them. Both crews are inspired to better work by the knowledge that a competing shift is to follow immediately upon their heels, taking their places and performing similar work. Then, too, after the holes have been drilled the extra men from the shoveling crew are of great assistance in taking down the machines and removing them and their mountings, hose tools, and other articles that must be taken to a place of safety during blasting. As to the time wasted in clearing the tunnel of smoke, if the tunnel is adequately equipped with ventilating apparatus this operation should require little more than 15 minutes—just long enough for the men to eat their lunches—time that would have to be lost at any rate. Delays, of course, can not always be prevented, but the men are encouraged by rivalry to reduce these to the minimum, knowing that their work is to be compared with that of the shift to follow. These answers to the various objections are in no sense theories, but are deductions from actual observation and a study of conditions as they existed at tunnels where some of the most efficient work in this country was being performed.

The ideal results of the three-shift method, to be sure, are obtained only through perfected organization and good management, but they utterly disprove the contention that efficient work is not possible under those conditions. That the method may produce the most rapid progress has never been gainsaid, and with proper handling the actual cost of excavation per tunnel-foot need be little, if any, greater than with other methods; moreover, as has been shown, in most cases the system affording greater speed is within limits ultimately the more economical one. For these reasons, unless the conditions are indeed exceptional, the employment of three drilling shifts per day is recommended, and the following discussion of other phases of tunneling methods will, unless otherwise noted, be predicated upon the assumption that three drilling shifts are being employed.

METHODS OF MOUNTING DRILLS.

American tunnel practice is almost equally divided between the horizontal-bar and the vertical-column methods of drill mounting. The horizontal-bar mounting consists essentially of an iron pipe, 4 to 6 inches in diameter, a little shorter than the average width of the heading, and provided with a solid head at one end and a jackscrew with a capstan head at the other. The vertical-column mounting, which is rarely employed with more than two drilling attacks per day, is not greatly dissimilar except that it is usually provided with a yoke and two jackscrews at one end, and its length is somewhat less than the height rather than the width of the heading. In several notable European tunnels a drill carriage was employed, however, so that a discussion of this method of drill mounting should not be omitted.

The horizonal-crossbar method of mounting rock drills can perhaps be best illustrated by a description of the procedure used at the Laramie-Poudre Tunnel. As soon after the blasting as ventilation permitted (ordinarily 10 to 15 minutes), the workmen returned to the face from a position of safety 1,500 to 2,000 feet away, bringing with them an ordinary tunnel car containing the crossbar, drilling machines, tools, hose, etc. The three drillers, sometimes with the assistance of the foreman, first removed from the roof or walls loose rocks that might have fallen later, possibly causing injury. This accomplished, they next cleared a space in the top of the rock pile, for 2 or 3 feet back from the face of the tunnel and perhaps 4 or 5 feet from the roof, in order that they might have room to work when drilling. Because of methods of blasting especially employed for this purpose the rock pile usually occupied only a small part of this space, so that ordinarily little work was required to clear it out. In the meantime the helpers were expected to unload the bar and machines from the car, placing them on the rock pile conveniently at hand, and connect the hose to the air and water mains. As soon as a proper space was cleared out the bar was picked up by the drillers and helpers and held in position transversely across the tunnel at a measured distance from the face and roof, as directed by the foreman, where it was blocked, wedged, and finally screwed as tightly as possible in place. The drillmen then placed the machines upon the bar and started drilling as soon as the helpers completed connecting the hose to the drills. The necessary holes having been drilled from this position of the bar and the waste rock having been removed in the meantime by the shovelers (an operation carried on simultaneously with drilling and ordinarily accomplished before the drillers had finished), the machines were taken off, the bar lowered and set up again about 18 to 24 inches from the floor, the drills replaced, and one or two holes drilled by each machine from this position of the bar. The machines and the bar were then placed in a tunnel car and removed from the heading during the blasting. This method, sometimes slightly modified, was used at several other tunnels and adits with almost equally good results.

The procedure with the vertical-column method of mounting is similar to this in some respects, but there are also some important distinctions aside from that of upright position. Owing to the vibration produced by the drills, neither method of mounting will give satisfaction unless the bar is firmly jacked against solid rock. The vibration is intensified and the need for a substantial foundation is much greater with the vertical column, because the drills are usually mounted on cross arms projecting from the columns at right angles, thus affording a leverage for any movement of the drill. It is therefore necessary to remove all of the waste rock from the space immediately in front of the face of the tunnel prior to drilling, in order that the foot of the column may rest upon the solid floor, which, at the two or three tunnels where this method was employed with the three-shift system, caused considerable delay even under normal conditions. But in the majority of places where this method of mounting was employed not more than two drilling attacks were attempted per day, and the extra work of clearing away was performed by the crew of shovelers before the drillers started work.

The best results with the carriage mounting for drills were obtained during the construction of the Loetschberg Tunnel through the Bernese Alps. In the first type of carriage employed there, the horizontal bar carrying the drills was mounted at the end of a steel beam which was pivoted to a truck and counterbalanced at the other end by a heavy weight. Before this carriage could be brought sufficiently near to the face, even with the long beam, for the crossbar to be jacked in position, it was necessary to clear a rather large passageway through the center of the rock pile down to the floor. In doing this, part of the material was carried away and the remainder piled on either side of the tunnel to be carried away during drilling. When the passage was finished, however, the carriage with the crossbar and drills mounted upon it and extending longitudinally was quickly rolled to the face, the bar swung around and jacked into position, and the drills were at once started to work.

This carriage was superseded by one in which the counterbalanced beam was abolished and the drill bar carried directly upon a short post mounted on the truck. With this device practically all of the broken rock had to be removed from the heading before the carriage was brought to the face, after which, however, the drills started at work promptly.

One of the most important factors to be considered in choosing a method of mounting for tunnel work is the time required to get the drills in operation after blasting, including not only the actual time employed in setting up the necessary apparatus, but also the time consumed in the preparatory work of clearing away débris. The time spent in waiting for the smoke to clear is, of course, independent of the method of mounting and can therefore be ignored in this connection. With the horizontal-bar system used at the Laramie-Poudre Tunnel, the time normally employed in mucking back was rarely more than 15 to 20 minutes. Jacking the bar in place occupied 5 to 10 minutes and attaching the drills and making the water and air connections usually required 10 to 15 minutes. The entire operation thus consumed, under ordinary conditions, 30 to 45 minutes, but it was not at all unusual for the drills to be in operation within 20 or 25 minutes from the time the drillmen reached the heading. At other tunnels and adits using this system the time required for similar work was reported as 30 to 60 minutes. Owing to the much greater quantity of material to be cleared out when the vertical-column method is employed, the time consumed in getting the drills in position to start work at adits and tunnels where the three-shift system was used ordinarily ranged from 2½ to 4 hours, and even under the most favorable circumstances was rarely less than 2 hours.

The time spent in the Loetschberg Tunnel in removing the waste rock was approximately 1½ hours with the first type of carriage used and 1½ to 3 hours with the later model, but in order to attain such speed nearly twice as many men were employed at the work as are usually found in American tunnel headings. After the Loetschberg Tunnel had been cleared of the necessary amount of débris, however, the machines could ordinarily be started in 5 to 10 minutes.

Aside from the question of the time consumed in clearing, the amount of waste to be removed has another bearing on the problem of choosing a mounting. In order that there may be no delay in getting the drills at work, usually the attempt is not made to remove the waste rock entirely from the heading before setting up the mountings, much of it being merely shoveled to one side and removed later. This preliminary work is often performed by the drillmen, especially with the three-shift system; and where (as in the case of the vertical-column method) there is a great deal of it to be done, by the time they have the machines set up and are ready to start drilling these men are pretty well tired out and consequently can not work as rapidly and efficiently in drilling the required holes. Even if the work is performed by the regular shoveling crew, the men certainly are not stimulated by the knowledge that they are performing dead work, and that every shovelful handled in clearing back must be moved again later. This disadvantage obtains not only in the three-shift system but in many cases where two shifts are employed and the shoveling crew start ahead of the drillmen and commence work clearing away the face for a vertical-column set-up. The horizontal-bar and, to a lesser degree, the drill-carriage methods have the advantage of requiring a much smaller proportion of duplicated work.

The adaptability of the mounting for the work required of it after the drills are in operation is another factor to be reckoned with. The advocates of the vertical-column method claim that it enables the holes to be placed to better advantage, and this is quite truly the case when piston drills are employed. But hammer drills mounted on a horizontal bar can place the holes just as effectively, if not more so. With either type of machine the drill carriage is badly handicapped. It was discovered with those used in the Loetschberg Tunnel—and the same disadvantage was experienced at an adit in this country where a similar drill carriage was tried and soon abandoned—that it was impossible to point the inclined holes in such a way as to obtain the maximum efficiency from the explosive used. Therefore, in order to make the holes break to the bottom it was necessary to use heavier charges of explosive, and the holes were not drilled as deeply as they might otherwise have been. The shallower holes necessitated greater labor in the unproductive preparatory work of setting up and tearing down the drills and increased the opportunities for delays in blasting. Then, too, it is impossible with one set-up of a horizontal bar, such as was used in the carriage method of mounting, to make the holes near the bottom of the tunnel sufficiently horizontal to insure an even floor, necessitating trimming, and causing trouble in maintaining the proper tunnel grade.

The fact must not be overlooked, however, that with the carriage method the drills are subject to less wear and tear because they are kept on the bar continually and are not thrown around on the floor and muck pile. When this is permitted the drills are apt to become filled with sand, grit, etc., which, because of friction and abrasion, increases the cost of repairs. Nor should the facility in changing to a new hole possessed by the horizontal bar and the drill carriage be disregarded. When these methods of mounting are employed, all that is necessary in starting a new hole is to slide the drill along the bar and clamp it in place, but with the vertical column not only the machine but the cross bar as well requires adjusting; as the adjustment is vertical instead of horizontal, the entire weight of both drill and cross arm must be lifted or sustained at nearly every change.

Taking into consideration all of the factors mentioned, the horizontal bar proves to be the method of mounting drills best adapted for tunnel work. Its use enables the drills to be put in operation with the least loss of time and by the smallest number of men. It requires the rehandling of the minimum quantity of waste rock so that the drill men are not fatigued before they start drilling or the shovelers disheartened by dead work. It permits directing the holes in such a way that the maximum strength of the explosive is utilized, drilling deeply so that too great a part of the time need not be spent in preparatory work, and placing the holes to insure the breaking of the roof and floor smoothly and at the desired grade. It is especially adapted for use with the more rapidly drilling hammer machines and lends itself readily to removal when necessary. In common with the vertical type it is subject to the danger of allowing grit to become lodged in the machines, but this can be partly prevented by care in handling.

PROPER NUMBER OF HOLES.

Any determination of the proper number of holes to be used in driving a tunnel or adit of a given size is dependent upon several factors. A large number of holes in which a large charge of explosive may be placed expedite the operations of driving, because the heavier blast tends to hurl the rock farther away from the face, not only saving time in setting up the machines but also giving the shovelers more room and enabling them to work to better advantage on more widely scattered material. But at the same time, holes that are not strictly a necessity entail an extra expense not only for the explosive used in them, but also for the time required in drilling, especially if the drilling work requires more time than the operation of removing the rock, as any extra holes delay both crews. If the proper number of holes is being used, the major part of the rock should be broken into fragments small enough to be shoveled readily, although an occasional bowlder, because of the relaxation it affords the workman from the steady grind of shoveling is said to expedite rather than retard the speed with which the spoil can be loaded into the tunnel cars.

The central factor, however, in a just determination of this question is undoubtedly the physical character of the rock being penetrated, which is never twice alike in different localities, preliminary experimentation being usually necessary in order to discover what number of holes will produce the best results. Generally speaking, igneous rocks require more holes than sedimentary rocks, but there are wide divergences in both classes. The holes must be more closely spaced for a tough rock that is close grained and massive than for one that is brittle and easily shattered, even though it may be harder and more difficult to drill. Bedding or joint planes or joint cracks are of great assistance, and a rock in which they occur will be more easily broken and hence require fewer holes. The following table shows the number of drill holes used in American tunnels penetrating different classes of rock:

TABLE 14.—*Number of holes used in driving tunnel headings in various American tunnels.*

Name of tunnel.	Number of holes.	Character of rock penetrated.	Approximate area of heading.	Square feet of heading per hole.	
				Sedimentary rocks.	Igneous rocks.
			Square feet.		
Burleigh	16	Granite and gneiss	42		2.6
Buffalo (water)	22	Limestone	120	5.5	
Carter	10-11	Gneiss, granite, and porphyry.	41		3.7-4.1
Catskill Aqueduct:					
Rondout Siphon	22	Limestone, sandstone, and shale.	120	5.5	
Wallkill Siphon	24	Shale	120	5.0	
Moodna Siphon	24	Sandstone and shale	120	5.0	
Yonkers Siphon	21	Gneiss	120		5.7
Central	18-24	do	35		1.5-1.9
Chipeta	15-19		57		3.0-3.8
Fort William (water)	14-20	Basalt	35		1.7-2.5
Gold Links	12	Gneiss and granite	48		4.0
Grand Central sewer	18	Gneiss	40		2.2
Gunnison	24	Altered granite	60		2.5
Joker	19-21		130	6.2-6.9	
Laramie-Poudre	21-26	Close-grained granite	70		2.7-3.3
Lausanne	15-21	Shale, conglomerate, and coal.	85	4.0-5.6	
Los Angeles Aqueduct:					
Elizabeth Lake	25	Granite	145		5.8
Little Lake division	14-16	Medium hard granite	90		5.6-6.4
Grape Vine division	20-21	Hard granite	90		4.3-4.5
Lucania	25	do	65		2.6
Marshall-Russell	18-20	Granite and gneiss	72		3.6-4.0
Mission	12-14	Shale and slate	37	2.6-3.1	
Newhouse	19	Gneiss	65		3.4
Nisqually	18	Rhvolite	95		5.2
Northwest (water)	22	Sedimentary rock	110	5.0	
Ophelia	20-24	Granite	80		3.6-4.0
Rawley	25-27	Andesite	55		2.0-2.2
Raymond	14	Gneiss and granite	80		5.7
Roosevelt	24-26	Hard granite	60		2.3-2.5
Siwatch	12	Granite	45		3.7
Snake Creek	16	Diabase	65		4.0
Spiral	21	Limestone	175	8.4	
Stilwell	16	Conglomerate and andesite	50	3.1	
Strawberry	16-18	Limestone, sandstone, and shale.	50	2.8-3.1	
Utah Metals	12-16	Quartzite	80	5.0-6.6	
Yak	18	Limestone, sandstone, shale, and granite.	50	2.8	

DIRECTION OF HOLES.

Chiefly because of the great influence of local conditions, the arrangement of drill holes is rarely identical in driving at any two tunnels. For reasons to be explained later, however, it is customary to drill a part of the holes (called the "cut" or "cut holes") in such a manner that when blasted they will remove a core of rock from the face and thus decrease the work to be done by the remaining holes. Practically all of the various means of arranging drill holes in the headings of American tunnels may be summarized as follows into three main types according to the kind of cut employed.

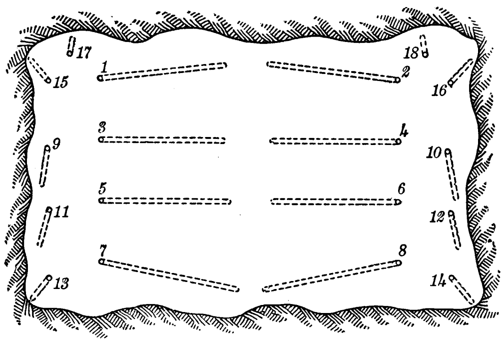

FIGURE 23.—Wedge-cut round of holes.

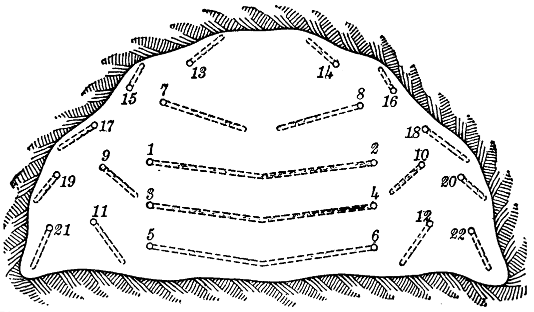

FIGURE 24.—Modified wedge-cut round for arched heading.

The wedge or **V** cut is the one most commonly employed in tunnels in this country. It consists essentially of several pairs of holes drilled from opposite sides of the heading in such a manner that when properly charged and exploded they will break out a wedge-shaped core of rock, usually extending from the roof to the floor of the tunnel. Figure 23 shows a typical wedge-cut round similar to the one employed in driving the Buffalo Water Tunnel. Holes 1 to 8 comprise the cut and were blasted simultaneously by electricity, whereas 9 to 14 are the side holes, and were next fired together, and 15 to 18 are the back or dry holes and were exploded last. Such a round must necessarily be changed somewhat if the heading is arched or semicircular. Figure 24 illustrates such a round, similar to those used in driving the heading of the large siphons on the Catskill Aqueduct. In this case holes 1 to 6, comprising the cut, were blasted together, followed by holes 7 to 12, which are called relievers, and finally by 13 to 22, which are called trimming holes.

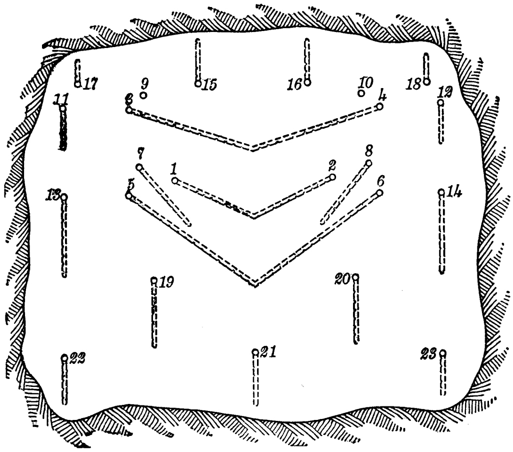

FIGURE 25.—Wedge-cut round drilled from a horizontal bar mounting.

Either vertical columns, as used in the cuts just cited, or a horizontal bar may be used to mount the machines when drilling this type of round; but if the majority of the holes are to be drilled from one position of a horizontal bar, the position of the holes must necessarily be somewhat modified, although the general arrangement still remains a wedge-cut round. Figure 25 shows such an arrangement similar to the one employed at the Laramie-Poudre Tunnel. Holes 1 and 2 were called "short-cut holes," 3 to 6 "long cuts," 7, 8, 9, 10, 19, and 20 relievers, 11 to 14 sides, 15 to 18 backs, and 21 to 23 lifters, the numbering indicating the order of blasting. The lifters and two relievers (holes 19 and 20), which were used only in hard ground, were the only holes drilled from the lower position of the bar. Three machines were employed in drilling this round. The holes drilled by each and the order of drilling, the machines being lettered A, B, and C from left to right when facing the heading, are shown in the following:

Order of drilling for each machine at Laramie-Poudre Tunnel.

BAR IN UPPER POSITION.

Machine A.	Number of drill hole.	Machine B.	Number of drill hole.	Machine C.	Number of drill hole.
1	17	1	16	1	18
2	11	2	10	2	12
3	5	3	15	3	6
4	13	4	9	4	14
5	7	5	3	5	8
6	1	....	....	6	2

BAR IN LOWER POSITION.

7	19	....	....	7	20
8	22	6	21	8	23

A somewhat similar round was used with a horizontal bar at the Rawley Tunnel, and there are, of course, many other variations of the V-cut arrangement of holes, but the figures given illustrate the principles underlying the more common arrangements employed in tunnels and adits in this country.

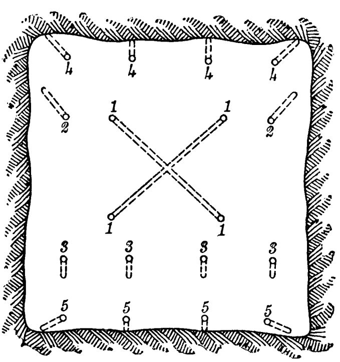

FIGURE 26.—Pyramid-cut round of holes.

The second general type of cut frequently employed may be designated as the pyramid cut, consisting usually of four cut holes drilled in such a manner that they meet, nor nearly meet, at or near a common point—generally near the axis of the tunnel—and when properly blasted remove a somewhat pyramidal core. Figure 26 shows a round of this type similar to the one employed at the Yak Tunnel in which the cut holes, No. 1, were blasted simultaneously, followed by the remaining holes in the order indicated. In most of the instances observed by the authors the pyramid cut has been employed with vertical columns, but it can be drilled just as efficiently with the horizontal bar by drilling two or possibly three holes with each machine in the lower set-up. Figure 27 shows such a round.

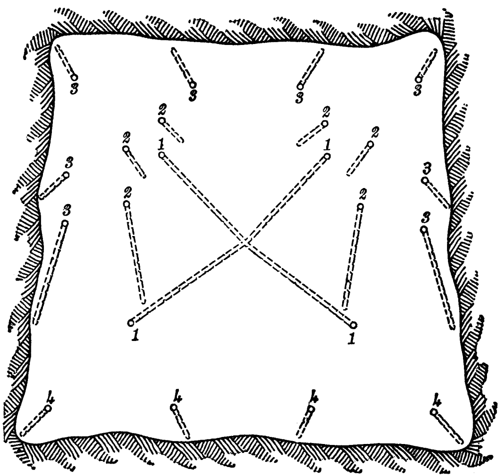

FIGURE 27.—Pyramid-cut round for use with horizontal-bar mounting.

The third type is the bottom or draw cut which was employed at several places visited, the one at the Carter Tunnel, illustrated in figure 28 being typical. The holes were blasted in the order indicated, Nos. 1 to 3 comprising the cut.

It can easily be proved theoretically that if a bore hole is drilled in a homogeneous mass of rock the maximum efficiency can be obtained from a suitable charge of explosive placed in it when the line of least resistance (by which is meant the shortest distance from the charge to a free surface of the rock) is at right angles to the axis of the bore hole; and that the minimum efficiency will be obtained when the two are coincident. Practically, also, although a homogeneous rock is a rarity, and hence the actual results will be influenced quantitatively somewhat by the various features of rock texture, such as joints, cracks, fault fissures, and bedding planes, the results have been found to agree in the main with the theoretical deductions. Obviously, therefore, in the heading of a tunnel or adit, where only one free face can be obtained, it is impossible to drill and blast a single hole in such a manner that the maximum efficiency can be obtained from it. But by drilling a number of holes arranged according to any of the preceding systems, and blast-

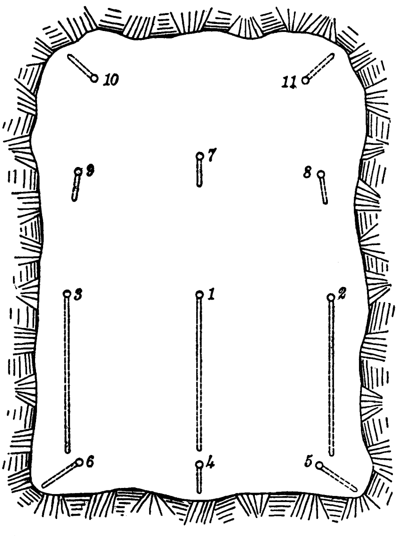

FIGURE 28.—Bottom-cut round of holes.

ing the cut first so as to create more free surface much better results can be obtained from the holes that remain. It is for this reason that the position and direction of the holes comprising the cut are generally considered the most important feature of the work, the spacing of the remaining holes being admittedly merely a matter of having them sufficiently close together to break the rock into fragments of the required size for easy handling.

When the wedge or V cut is employed, the several pairs of holes should be placed close enough together for them to be of some mutual assistance, especially if the entire cut is exploded simultaneously. What this distance shall be is controlled almost entirely (as in the determination of the proper number of holes) by the character of the rock, its texture, toughness, the presence of cracks and bedding planes, etc. The separating distance is often determined by the foreman in charge, and if he is a man of wide experience, satisfactory results may follow; but the general efficiency of the work will often be increased greatly if experiments are made at the outset to determine just what combination will give the best results for the particular rock being encountered. It follows, of course, that such experiments should be repeated whenever a marked change in the nature of the rock is observed.

In order that the line of least resistance may approximate as closely as possible the perpendicular to the axis of the drill hole, the angle between opposite holes in the cut should be as large as can be obtained with any given depth of round. Hence, it follows that the drill holes should start as near as possible to the opposite sides of the heading; but obviously the full width of the heading can not be utilized because provision must be made for the feed screw and crank of the drill, which usually extend 3 to 4 feet from the face. This necessary allowance works especially to the disadvantage of narrow headings, because in them a greater proportion of the actual width must be sacrificed. But with broad headings the marked advantage of a wide angle is easily obtained and possibly offers an explanation of the popularity of the wedge-cut system in such headings.

Of even greater importance than the necessity of procuring a wide angle between opposite holes is that of drilling them so that they meet, or at least bottom near enough to one another to be detonated simultaneously by the one first to explode. Owing to mechanical reasons, the width of the drill bit, and hence the size of the holes, must be decreased with each successive change of steel, and as a result the hole is necessarily smallest at its bottom end—the place where the explosive is most needed and where it is extremely desirable that the holes should be as large as possible. Omitting from consideration the expedient of chambering—that is, the enlargement of the bottom of the hole by the explosion of a small primary charge before loading it with the main charge of the explosive—which consumes entirely too much time to be considered for rapid tunnel driving, the defect can be overcome to a surprisingly large degree by the simple resort of connecting the drill holes, which concentrates the explosive at the point of the V. When fuse firing is employed, it is essential that the holes be so directed that they are intercommunicating (or so nearly so that both holes will detonate together) or the desired effect will not be gained, but when electric firing is employed direct connection, although very desirable, is not so absolutely essential.

In addition to the mere concentration of explosive thus obtained, the combined efficiency of the two charges is much greater than when exploded separately. If it be assumed that the holes are drilled in homogeneous rock and that they make equal angles with the shortest line from their junction to the free face, and if both are loaded with identical charges of explosive and detonated simultaneously, their maximum breaking effect will be exerted along the resultant of their combined forces, which in this theoretical case coincides with the shortest distance to the free face (the line of least resistance). Practically, of course, this result will be somewhat modified; but it is a well-established fact that if the ground is tough and difficult to break, much better results are obtained if the cut holes are directed and drilled to intersect—although, unfortunately, this is not widely known, as evidenced by the too great number of cases observed where no attempt was made to connect the cut holes.

Practically the same conditions prevail with the pyramid cut. The number of holes comprising it may vary from three to six, or even eight, according to the nature of the ground, and the proper number can best be determined by experiment. It is just as necessary to drill the holes with the widest possible angle between them, and it is even more essential that they meet in a common point, because one of the main advantages of this cut is the concentration of a greater quantity of explosive at the narrow apex of the core of rock to be removed. This advantage is lost if the charges of explosive in the different cut holes are not detonated simultaneously.

The bottom cut, as usually drilled in practice, although it often enables the attainment of a wider angle between the axis of the drill hole and the line of least resistance disregards entirely the important advantage to be obtained from connecting drill holes, and this, in the opinion of the authors, should be sufficient to prevent its use under any but exceptional conditions. For narrow mine adits, however, in which it would be impracticable, if, indeed, possible, to drill an effective wedge or a pyramid cut round, the bottom cut furnishes the only solution of the difficulty. In drilling in such a tunnel it is recommended that the cut holes be drilled from as near the top of the heading as possible and directed in such a manner that they will connect with holes that are usually considered lifters and that both be detonated together.

DEPTH OF HOLES.

During the past four or five years there has been some difference of opinion among students of the problems of tunnel driving, as to the proper depth for drill holes in tunnel headings. In view of some of the remarkable results attained in driving the Simplon and Loetschberg Tunnels, where, as is agreed by everyone, the holes were much shallower than those in American practice, the question has been raised as to whether the holes in the tunnels of this country are drilled too deep. Numerous tables have been prepared in support of this argument, from which it appears that at most European tunnels the progress is much greater (in some cases more than twice greater) than that of tunnels in America. At the same time consideration is not always given the fact that in many instances the records are by nature in no wise comparable; for in Europe, at the majority of tunnels cited, the work was conducted throughout the entire 24 hours of each day, whereas in America in many instances only two shifts (and, indeed, in some only one) were employed daily. Then, again, the nature of the rock exerts an all-important influence upon progress, and in many cases this has been to the advantage of the European tunnels. A notable example of the influence of the rock encountered is found at the Loetschberg Tunnel, where the same methods and practically the same equipment were employed at the different ends, the north end working in limestone and the south end in gneiss and schist. The progress attained at the south end was much less than that at the north, in some months the progress in the north end being nearly double that in the south. Other considerations, also, especially the labor and the cost of driving, enter into the problem in such a manner as to make it impossible to say (when everything is taken into account) that the greater speed in European tunnels is due solely to the use of extremely shallow holes. That in many instances the holes in American tunnel headings are too deep, however, is impossible of denial, and hence a discussion of the factors that enter into the determination of the proper depth of holes is extremely desirable.

One of the chief advantages arising from the use of shallow rounds is (when the holes are properly directed) the increased efficiency obtainable from a given charge of explosive; for, as the width of the heading is for all practical purposes constant, the angle between the line of least resistance and the axis of the bore hole becomes a function of the depth of round, the width of the angle increasing with shallow holes. This advantage obtains especially with the wedge-cut and with the pyramid-cut, and it should be a fundamental consideration with the bottom-cut method of drilling the holes. Strangely enough, however, in the Loetschberg and the Simplon Tunnels, which are so often cited as examples of the "highly desirable" European practice of using shallow holes, this advantage was almost if not entirely thrown away because the holes were drilled in vertical rows and were nearly parallel to the bore of the tunnel. In such a case the line of least resistance and the axis of the bore hole are nearly coincident, a condition that results in the production of the least possible efficiency from the charge of explosive, and it can not be gainsaid even by the advocates of this method that a much greater quantity of explosive was required to break the same amount of rock than is usual in American practice. If to this is added the fact that such a system utterly ignores the advantage to be obtained from connected drill holes by the concentration of explosive at the apex of the core of rock to be removed, there is strong ground for rational suspicion that the extreme shallowness of the holes used in these tunnels was adopted from necessity rather than from desirability; with this system of drilling and directing the holes the difficulty of blasting out the rock with deeper rounds could not fail to be greatly increased.

Among other advantages of the use of reasonably shallow holes may be mentioned the fact that such a method allows the holes to be of larger diameter at their farther end, increasing their capacity for explosive and enabling its concentration at the point where it is most needed. This feature makes possible the European practice of employing extremely shallow holes, but it can hardly be denied that much more effective results in blasting might be accomplished by a change in the direction of the cut holes. Then, too, as in America at least, the holes are rarely charged with explosive to their full extent; the mass of rock between the ends of the charges of explosive in the different holes and the free face of the heading (which can be considered as a measure of the amount of resistance to be overcome) is not so great with the shallow holes. This fact or the customary use of relatively heavier charges in shallow holes may explain, perhaps, why in such cases the major part of the rock is usually thrown farther down the tunnel instead of being piled high immediately in front of the new face, with the double advantage of making loading of the rock easier and saving time in getting the drills mounted. It is fairly well established, also, that the rock tends to break into smaller fragments if shallow holes are employed. Again, if deep holes are not employed the same care in starting them exactly at a given point is not required, nor is it necessary to direct them with such great accuracy, although of course the need of connecting the cut holes must not be overlooked.

The principal and unavoidable disadvantage in using the shallow hole round, on the other hand, is the fact that in order to obtain the same daily advance a proportionately greater number of drilling attacks must be made. This results in a waste of time in drilling, for it is possible under ordinary circumstances to drill one hole of a given depth more rapidly than it is two holes of the same aggregate footage, because of the time lost in changing to a new position, starting, etc. But even granting that the difference in drilling time (perhaps because it is too small or because in either case the drilling can be completed before the heading can be cleared of débris) is not an appreciable factor, each extra drilling attack required to obtain the same progress causes a corresponding loss of time in loading and blasting the holes, in waiting for the smoke and gases to be removed, in clearing the débris from immediately in front of the face, and in setting up the drills, all of which is ordinarily dead work and can not be avoided. This loss was seriously felt at the Loetschberg Tunnel, because in the endeavor to compensate for it four drills had to be employed in the heading (6 by 10 feet), and as a result the holes had to be drilled nearly straight, with disadvantages already described, because otherwise the drills in the center interfered seriously with the operation of those at the side.

On the other hand, if the holes are too deep the angle between the cut holes may be so narrow and the mass of rock in front of the charge of explosive may be so great that it will be impossible for the cuts to break bottom on the first blast, and thus the entire round is spoiled. The usual remedy in such cases is to blast the cuts separately and not to fire the remainder of the round until inspection has shown that the proper depth has been reached by the cut holes. Some delay can not be avoided when this method is employed, even if the holes break to the end, for it is never possible to return to the breast for such inspection immediately after the cuts have been detonated. But if the cut holes fail to break, the delay is greatly increased, because the remaining parts must be cleaned out, reloaded, and fired, with an additional delay in waiting for the smoke to clear.

This system was used at one of the Colorado tunnels, which at the time of first examination was being driven through some very tough rock. A round of holes slightly deeper than the average width of the heading was used, and it had given satisfactory results in the somewhat more frangible ground previously penetrated, the round being drilled and blasted in an 8-hour shift without difficulty, but when the harder rock was struck it became necessary to blast the cuts separately, and frequently to reload and shoot them for the second and occasionally for the third time, the cycle being lengthened to about 10 hours, and several times at least 14 hours was needed. If three drilling shifts had been employed at the time, such a condition would have been fatal, but as only two attacks were being made the difference was not so noticeable, though even in this case the cost of the extra explosives required and the overtime wages of the men added a considerable expense to the tunnel work. Shortly after the first examination of this tunnel by the authors, however, the depth of the rounds was reduced to about 75 per cent of the width of the heading.

This made it unnecessary to load and shoot the cuts separately, and instead of getting two 7½-foot rounds in 20 to 22 hours, by working three 8-hour shifts it was possible to drill and blast four, and sometimes five, 5-foot rounds per day, thus increasing the daily tunnel progress from 15 to nearly 23 feet with only a small extra cost for labor. The consumption of explosive, a considerable item with the old system, was also decreased fully 25 per cent, and the total cost of the tunnel per foot was considerably reduced.

The disadvantage of too deep holes was strikingly brought out in the construction of the Laramie-Poudre Tunnel. During the first part of the work a 10-foot round was drilled in a heading 9½ feet wide, but the round was later changed to one of 7-foot depth with much better results. To be more specific, during the seven months from April 1, 1910, to October 31, 1910, at the east end of the tunnel, 3,171 feet was driven, an average of 453 feet per month, with a 10-foot round; but during the next 8⅘ months, from November 1, 1910, to July 24, 1911, when the tunnel holed through, 4,798 feet was driven, or an average of 545 feet, with a 7-foot round. This in an increase of over 20 per cent in spite of the fact that the greater speed was made when the work was at a greater distance from the portal; and, as there was no essential change in the methods or the equipment, or in the character of the rock penetrated, the increase is attributable solely to the use of shallower holes. When the 10-foot holes were employed to obtain an advancement of 8½ to 9 feet it was unusual to be able to drill and blast more than two rounds in 24 hours, and oftentimes not that many, as the average of 14½ feet daily testifies; but with the 7-foot round not only could three attacks be made, advancing on an average of 6.5 feet per attack, but a comfortable margin of time was left to provide for delays, and under favorable conditions this extra time meant extra footage. Thus, in March, 1911, the American hard-rock record of 653 feet, or over 21 feet per day, was established. This advantage of being able to complete an entire cycle of operations during a single shift should be given the weight it deserves in the problem. If crews of men could be found who would work as well without rivalry and without special incentive to push the work, it might be perfectly feasible to choose a depth of round that would require 10 or even 12 hours for preparation, but under the present working conditions, where it is necessary to have some accurate measure of the work performed by each crew, a round is required for which the entire cycle can be completed during a single shift, with a sufficient margin of safety to provide for any ordinary delay.

It is, of course, impossible to set any definite standard or guide for the proper depth of hole that will be applicable to all cases. There are too many variables influencing the result. The proper depth must be determined by experiment in each individual case. However, from an extended examination of the results obtained from the methods employed in American practice, from a careful analysis of European practice as outlined in available published accounts, and from a study of all other procurable modern authority, the authors are of the opinion that for the majority of cases the proper depth of drill hole, the one that most equitably balances the advantages and disadvantages inseparable from the problem, is 60 to 80 per cent of the width of the tunnel heading. The following table gives an analysis of American practice in this respect:

TABLE 15.—*Depth of drill holes used in American tunnels.*

Name of tunnel.	Type of cut.	Height of heading.	Width of heading.	Average depth of cut holes.	Average depth of other holes.	Average depth of round drilled.	Percentage of width of heading of average round.	Character of rock penetrated.
		Feet.	*Feet.*	*Feet.*	*Feet.*	*Feet.*		
Buffalo (water)	Wedge....	8	15	8	7	7	46.5	Limestone.
Carter..........	Bottom...	7½	5½	9	8	8	[a] 106	Gneiss, granite, and porphyry.
Catskill Aqueduct:								
Rondout Siphon.	Wedge....	8	14	10	8	8	57	Limestone, sandstone, and shale.
Wallkill Siphon.	...do.......	8	14	12	10	10	71.5	Shale.
Moodna Siphon.	...do.......	8	14	10	8	8	57	Sandstone and shale.
Yonkers Siphon.	...do.......	8	14	8	6	6	48	Gneiss.
Central..........	...do.......	7	5	8	7	7	140	Do.
Fort William (water).	Bottom...	6½	5	6	5	5	[a] 77	Basalt.
Gold Links.....	...do.......	8	6	6	5	5	[a] 62.5	Granite and gneiss.
Gunnison......	Wedge....	6	10	7	6	6	60	Altered granite.
Joker(drainage)	...do.......	11	12	10½	9	9	75	
Laramie-Poudre.	...do.......	6½	9½	8	7	7	74	Close-grained granite.
Lausanne......	...do.......	8	12	8	7	7	58	Conglomerate, shale, and coal.
Lucania........	...do.......	8	8	9	8	8	100	Hard granite.
Marshall-Russell.	Pyramid..	9	8	10	9	9	112	Granite and gneiss.
Mission........	Bottom...	7	5	8	7	7	[a] 100	Shale and slate.

[a] The height of the heading, instead of its width, is considered in this ratio when the bottom cut is employed.

TABLE 15.—*Depth of drill holes used in American tunnels*—Continued.

Name of tunnel.	Type of cut.	Height of heading.	Width of heading.	Average depth of cut holes.	Average depth of other holes.	Average depth of round drilled.	Percentage of width of heading of average round.	Character of rock penetrated.
		Feet.	*Feet.*	*Feet.*	*Feet.*	*Feet.*		
Mission (hard ground).	Pyramid..	7	5	8	7	7	140	Sandstone.
Newhouse......	...do.......	8	8	6½	5½	5½	69	Gneiss.
Nisqually......	Bottom...	11	9½	8	6½	6½	[a] 59	Rhyolite.
Northwest (water).	Wedge....	10	13	10	9	9	69	Sedimentary.
Ophelia........	...do.......	9	9	7	6	6	67	Granite.
Rawley........	...do.......	7	7½	9	8	8	106	Andesite.
Raymond......	...do.......	9	9	12	10	10	111	Gneiss and granite.
Roosevelt......	...do.......	6	10	7	6	6	60	Hard granite.
Siwatch........	Bottom...	7½	6	5	5	5	[a] 67	Granite.
Snake Creek....	Wedge....	6½	9½	6½	5½	6	63	Diabase.
Spiral..........	...do.......	10	16	12	10	10	63	Limestone.
Stilwell........	...do.......	7	7	6½	6	6	86	Conglomerate and andesite.
Strawberry.....	...do.......	6	8	7	6	6	75	Limestone, sandstone, and shale.
Utah metals....	Bottom...	8	10	6½	6	6	[a] 75	Quartzite.
Yak............	Pyramid..	7	7	5	4	4	57	Limestone, sandstone, shale, and granite.

[a] The height of the heading, instead of its width, is considered in this ratio when the bottom cut is employed.

BLASTING.

SELECTION OF EXPLOSIVE.

To be suitable for use in tunnel work, as distinguished from surface blasting operations, an explosive should not produce any great quantity of poisonous gases and should not easily be affected by moisture. In common with other usages, a substance is required here that is stable in composition and not rapidly deteriorated by frequent changes in temperature or other causes; it must not be so sensitive to shock that safe transportation and handling is well-nigh impossible, as, for example, is the case with liquid nitroglycerin. Although under some circumstances, especially in tunnels that are not wet, an explosive called ammonia dynamite can be employed, the one that best fulfills the necessary requirements and the one that is almost universally used in tunnel work is known as gelatin dynamite.

Gelatin dynamite is a combination of a certain quantity of blasting gelatin (varying according to the strength desired) and a suitable absorbent. The former is made by adding a small percentage of guncotton (nitrocellulose) to liquid nitroglycerin, thus producing a jellylike mass that has greater explosive qualities than either of its constituents, but is much less sensitive to shock than nitroglycerin. The absorbent is usually some combustible material (wood pulp is frequently employed) to which has been added sufficient sodium nitrate to supply the necessary oxygen for its combustion. By the use of such a combustible absorbent, instead of the inert one formerly employed with "straight" nitroglycerin dynamite, the gases generated by the burning of the wood pulp add to the volume produced by the detonation of the explosive constituent, and the extra heat generated in this combustion adds greatly to the total intensity of the reaction. Ammonia dynamites, which were discovered somewhat more recently, consist of a combination of ammonium nitrate and nitroglycerin absorbed in a so-called "dope" similar to that just described. The following tables[a] show typical compositions of commercial samples of these two kinds of dynamite:[b]

Typical compositions of commercial gelatin dynamite.

Ingredient.	Strength.						
	30 per cent.	35 per cent.	40 per cent.	50 per cent.	55 per cent.	60 per cent.	70 per cent.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Nitroglycerin	23.0	28.0	33.0	42.0	46.0	50.0	60.0
Nitrocellulose	.7	.9	1.0	1.5	1.7	1.9	2.4
Sodium nitrate	62.3	58.1	52.0	45.5	42.3	38.1	29.6
Combustible material[a]	13.0	12.0	13.0	10.0	9.0	9.0	7.0
Calcium carbonate	1.0	1.0	1.0	1.0	1.0	1.0	1.0

[a] Wood pulp (used with 60 and 70 per cent strength), sulphur, flour, wood pulp, and sometimes resin used in other grades.

Typical compositions of commercial ammonia dynamite.

Ingredient.	Strength.				
	30 per cent.	35 per cent.	40 per cent.	50 per cent.	60 per cent.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Nitroglycerin	15	20	22	27	35
Ammonium nitrate	15	15	20	25	30
Sodium nitrate	51	48	42	36	24
Combustible material[a]	18	16	15	11	10
Calcium carbonate or zinc oxide	1	1	1	1	1

[a] Wood pulp, flour, and sulphur.

The harmful gases usually resulting from dynamite are carbon dioxide and carbon monoxide. Although the former will not support respiration, and when present in sufficient amount may cause unconsciousness and even death from strangulation, it has no very injurious effects when sufficiently diluted. Carbon monoxide, how-

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[a] From a paper by Clarence Hall before the Am. Inst. Chem. Eng., Washington, D. C., meeting of Dec. 20, 1911.

[b] For further discussion of the nature and composition of explosives, which is hardly within the province of this report, the reader is referred to the following bulletins published by the Bureau of Mines:

Bull. 15: Investigations of explosives used in coal mines, by Clarence Hall, W. O. Snelling, and S. P. Howell, with a chapter on the natural gas used at Pittsburgh, by G. A. Burrell, and an introduction by C. E. Munroe. 1911. 200 pp., 7 pls.

Bull. 17: A primer on explosives for coal miners, by C. E. Munroe and Clarence Hall, 61 pp., 10 pls. Reprint of U. S. Geol. Survey Bull. 423.

ever, is not only exceedingly dangerous, but its effects are also cumulative; indeed, if air containing even a small amount of it is breathed for any length of time, serious and often fatal results will follow. The fact that gelatin dynamite (and perhaps ammonia dynamite, which approaches it very closely in this respect) produces under proper conditions the least amount of carbon monoxide is one of its chief advantages for use in tunnel work. Even with this explosive, however, if the blasting cap is not strong enough to cause a complete detonation, especially if the dynamite burns rather than explodes, much greater amounts of carbon monoxide are formed; in addition there are many other harmful gases produced, among which may be mentioned the dangerous peroxide of nitrogen and hydrogen sulphide, the former being especially virulent.

The following table shows the results of tests conducted by the Bureau of Mines concerning the kind and amount of gases produced by the detonation of samples of various kinds of commercial dynamites. In making the tests a charge of 200 grams (approximately 7 ounces) in the original wrapper was exploded in a Bichel pressure gage, the gaseous products being retained and analyzed.

Gaseous products from explosives.

Kind of explosive.	Carbon dioxide.	Carbon monoxide.	Oxygen.	Hydrogen.	Methane.	Nitrogen.	Hydrogen sulphide.	Volume of gas.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Liters.*
40 per cent "straight" nitroglycerin dynamite	27.3	26.9	0.0	18.0	0.4	27.4	....	88.5
60 per cent "straight" nitroglycerin dynamite	22.2	34.6	.0	23.2	.8	19.2	....	128.9
40 per cent strength gelatin dynamite	50.8	3.0	.0	1.8	.8	39.5	4.1	60.3
40 per cent strength ammonia dynamite	41.4	3.8	.0	3.1	.8	45.5	5.4	65.6
FFF black blasting powder (300 grams).	49.7	10.8	.0	1.8	.6	28.4	8.7	67.8

A further distinctive feature of gelatin dynamite, winning for it the advantage over ammonia dynamite for most tunnel work, consists in its practically waterproof quality, a condition largely due to the insolubility of the blasting gelatin which can be freely immersed in water with little if any of it dissolving. Ammonia dynamite, on the other hand, being hygroscopic, has a great affinity for moisture, and hence not only can not be used in wet work (or even in damp work if the original paraffined paper covering has to be split), but greater care must be used in selecting a dry place for storing it.

Gelatin dynamite is somewhat less sensitive to direct shocks than other dynamites, and, unlike them, the sensitiveness does not increase with the strength; much stronger detonators must therefore be used, even with the higher grades, in order to insure complete detonation. This fact is often not sufficiently appreciated by practical mining men, many of whom are not aware of the greater ultimate economy obtainable if the more powerful, although somewhat higher-priced, detonators are used with gelatin dynamites.

The strength of nitroglycerin dynamites as they are made to-day is generally rated according to the percentage of their nitroglycerin content in spite of the fact that both the volume of gases and the temperature (and hence the disruptive force) are augmented somewhat by the combustion of the absorbent material. Although 40 per cent is the strength most generally employed, they may be obtained in the following grades: 15, 17, 20, 25, 30, 33, 35, 40, 45, 50, 60, 70, 75, and 80 per cent. In the ammonia dynamites a part of the nitroglycerin is replaced by ammonia nitrate; but, as will be seen from the table on page 152, the rated strength of this dynamite is nearly the sum of the percentages of these two constituents. Ammonia dynamite is prepared in the same grades as nitroglycerin dynamite between 25 and 60 per cent. Owing to the strength of the blasting gelatin being greater than either of its constituents, the rated strength of gelatin dynamite is somewhat greater than the percentage of its explosive element. The usual grades of this dynamite correspond to those of nitroglycerin dynamite between 35 and 80 per cent. but it may also be procured in "100 per cent" strength.

The proper grade for use at any particular tunnel must be determined solely by local conditions. Such widely divergent results are obtained at different localities, although the same grade of explosive is used, and although the rock, as far as can be determined from its physical appearance and structure, is identical, that it is impossible to be dogmatic even with minute knowledge of local details. Generally speaking, however, a tough, close-grained, igneous rock will require a stronger explosive, whereas a sedimentary rock, or an igneous rock that has been altered and weathered, or perhaps shattered and broken, can be blasted just as effectively with a lower grade of dynamite. A notable example of the use of an extremely high-grade explosive is that of the Roosevelt Tunnel, where in the tough, close-grained, Pikes Peak granite "100 per cent" gelatin dynamite was required before satisfactory results were obtained. This is reported to have been the first "100 per cent" dynamite put to use in tunnel work. At the beginning of all work it is advisable to experiment with explosives of different strengths in order to determine which grade is best suited for the particular rock being penetrated, and it is, of course, obvious that similar experiments should be repeated whenever, owing perhaps to a change in the character of the rock, the dynamite being used fails to give satisfactory results.

The following table shows the grades of dynamite employed at the tunnels visited:

TABLE 16.—*Dynamite used at tunnels visited.*

Name of tunnel.	Kind.	Strength. *Per cent.*	Remarks.
Carter	Gelatin	40	Some 80 per cent.
Catskill Aqueduct:			
Rondout Siphon	do	60	
Wallkill Siphon	do	60	
Moodna Siphon	do	75	
Central	do	40	
Gold Links	do	40	A small amount of 60 per cent.
Gunnison	do	40 and 60	Mostly 60 per cent.
Laramie-Poudre	do	60	Some 100 per cent with the 60 per cent in cut holes.
Lausanne	do	60	
Los Angeles Aqueduct:			
Little Lake	do	40	Some 25 per cent and some 60 per cent.
Grapevine	Ammonia	40	Some 60 per cent and 75 per cent gelatin.
Elizabeth Lake	Gelatin	40	Tried 60 per cent and 70 per cent also.
Lucania	do	50	
Marshall-Russell	do	40 and 80	80 per cent also.
Mission	do	40 and 60	
Newhouse	do	40	100 per cent with 40 per cent in cut holes occasionally.
Nisqually	do	40	
Rawley	do	40 and 60	60 per cent in cut holes and lifters.
Raymond	do	40 and 60	
Roosevelt	do	40, 60, and 100	
Siwatch	do	40	
Snake Creek	do	40	Some 35 per cent and some 60 per cent.
Stilwell	do	40	
Strawberry	do	40	
Utah Metals	do	40 and 60	
Yak	do	40	

The practice of loading the bottom part of the hole with 80 and even 100 per cent dynamite and using 40 or 60 per cent in the remainder is not now uncommon, especially in tunnels and adits in the Western States. It has the advantage of producing a greater disruptive force at the bottom of the hole, where such force is most needed, and at the same time it reduces somewhat the cost of explosives, especially as compared with an excessive amount of lower-grade dynamite. There is entailed, of course, the trouble of handling two different kinds of dynamite, not only in the heading but in the thawing house as well. Although in some tunnels where this procedure was tried the same results might possibly have been achieved by the use of shorter rounds, an alteration in the type of cut, or some other change in method, still the combined charge is useful, especially for exceedingly hard, tough rock.

It is obviously impossible to make any set rule for the determination of the proper quantity of explosive to be employed in tunnel work. There are entirely too many variable factors governed solely by local conditions. Various writers have derived from theoretical consideration formulas for the calculation of the proper charge of explosive for a blast hole, but the application of these rules is limited to other types of blasting, such as quarrying or general mining, and they are not suited to the practical and actual conditions of tunnel work. For this the determination of the proper quantity of explosive is often left to the judgment of the foreman in charge, who, if he be widely experienced, can often produce excellent results; but the proper quantity can best be ascertained by a series of experiments in which the effects produced by different quantities of explosive are studied and compared.

It is very essential, however, that the charge of explosive be large enough. If it is too small and the cut holes fail to break bottom or the rest of the holes do not blast out their full share of rock it will be necessary to reload the parts remaining; this procedure not only requires fully as much explosive as if the holes had been properly charged in the first place, but also occasions a loss of time and footage, both of which are expensive. For this reason in a number of the tunnels visited it was customary to load the cut holes nearly to the collar. Although this is perhaps extreme as far as insuring that the cut holes break bottom is concerned, the extra dynamite helps to shatter the rock in finer fragments, thus making it easier for the shovelers to handle. Also, as no stemming[a] is usually employed in such cases, a certain amount of the explosive probably acts in that capacity and increases the efficiency of the remainder of the charge. The very common practice of loading the lifters entirely full has a very different object—that of throwing the major part of the débris some distance away from the new face of the heading, thus making it easier for the drill men to get their machines at work promptly, and as the rock is scattered over a greater area the shovelers can attack it to better advantage. Such a practice is highly to be commended.

Data as to the exact amount of explosive actually employed in practice are difficult to obtain, chiefly because at many places an accurate record of powder consumption is not kept, but figures were obtained wherever possible at the tunnels visited. At the Gunnison Tunnel an average of nearly 30 pounds of 40 per cent and 60 pounds of 60 per cent gelatin dynamite was employed per round. This quantity is equivalent to approximately 5.5 pounds per cubic yard excavated. In driving the south heading of the Elizabeth Lake Tunnel the average for 1909 was 32.09 pounds[b] of explosive per foot of tunnel, equivalent to 6 pounds per cubic yard. This figure, however, in-

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[a] In order to differentiate clearly between the material placed on a charge in a bore hole and the act of placing this material, the Bureau of Mines adopted the practice of designating the material as " stemming " and the act of placing it as " tamping." Readers should note this distinction which has been observed in the bureau's publications dealing with the testing or use of explosives.—Ed.

[b] Aston, C. W., The Elizabeth Lake Tunnel: Mine and Minerals, September, 1910, p. 102.

cludes the dynamite used in trimming, so it is somewhat higher than the amount actually needed in driving. At the Rondout Siphon 175 to 200 pounds per round were required to drive an average of 10 feet[a] with a heading approximately 120 square feet in area, equivalent to 3.9 to 4.5 pounds per cubic yard of rock excavated.

In advancing the heading of the Buffalo Water Tunnel, 3 pounds of 60 per cent dynamite were required per cubic yard.[b] At the Laramie-Poudre Tunnel the powder consumption per cubic yard for March, 1911, was 3.9 pounds; for April, 4.7 pounds, and for May, 4.9 pounds. The average on the Little Lake division of the Los Angeles Aqueduct for May, 1911, was 4.5 pounds per cubic yard. At the Wallkill Siphon Tunnel the average powder consumption per cubic yard ranged from 4.3 to 4.6 pounds. At the Yonkers Siphon Tunnel the powder consumption was approximately 4.5 pounds per cubic yard excavated.

The figures for the explosive used in the Simplon and the Loetschberg Tunnels indicate that the European average is somewhat higher than the American in this respect. At the Simplon Tunnel the charge was 6.5 pounds per cubic yard,[c] and at the Loetschberg Tunnel the charge per round to obtain an average advance in the 6.5 by 10 foot heading of approximately 3.5 feet was 53 to 57 pounds,[d] equivalent to 6.5 to 7 pounds per cubic yard.

The usual means of firing blasting charges, especially in tunnels and adits in the Western States, is by the use of a safety fuse. The term safety fuse originated from the fact that when properly used under working conditions this fuse burns at a uniform rate and does not flash or explode, as was often the case with the means employed for igniting blasting charges previous to its invention; but the term is somewhat misleading, because the fuse is not, nor has it ever seriously been claimed to be, safe for use in gaseous coal mines. The fuse used for tunnel work is composed of a core of gunpowder surrounded by various layers of waterproofing material.

Under ordinary conditions a safety fuse burns at a uniform rate, the variation rarely being greater than 10 per cent. In the European countries the normal rate is approximately 30 seconds per foot. In tests conducted by the Bureau of Mines with 14 samples of triple-tape fuse purchased for the Isthmian Canal the average rate of burning was determined as 26 seconds per foot for 3-foot lengths and 24.5 seconds per foot for 50-foot lengths. This rate is much faster than that of the fuse commonly employed in Western tunnel work,

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[a] Hogan, J. P., Progress on the Rondout Pressure Tunnel: Eng. Record, Jan. 1, 1910, p. 26.

[b] Saunders, W. L., Rock tunnel records: Eng. Record, Aug. 27, 1910, p. 224.

[c] Saunders, W. L., Tunnel driving in the Alps: Bull. Am. Inst. Min. Eng., July, 1911, p. 515.

[d] Saunders, W. L., op. cit., pp. 538, 539.

40 to 45 seconds per foot being the customary rate of burning for fuse used there, although those figures are not the results of tests.

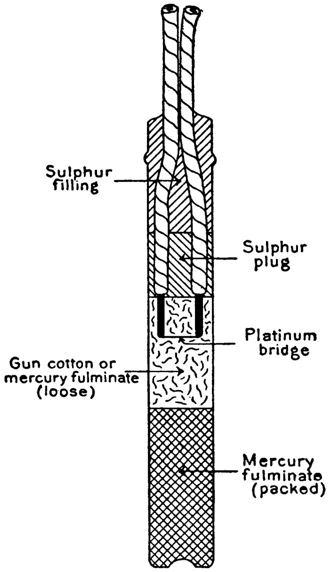

FIGURE 29.—Section of an electric detonator.

Experiments conducted by the Bureau of Mines[a] prove conclusively, however, that the normal rate of burning of fuse is greatly changed by a number of conditions. Excess of pressure greatly accelerates it, and if the gases are sufficiently confined, the increase may be as great as 300 to 400 per cent. Although as great an increase as this would rarely be obtained in practice, the use of stemming that is too tightly packed or is impervious to the escaping gases may produce sufficient pressure to increase greatly the rate at which the fuse burns. When fuse is exposed to low temperature for a short time the rate is slightly increased, but if it is stored at temperatures below freezing and handled before being warmed, cracks are apt to result in the waterproof composition which will permit the gas to escape and, by reducing the normal pressure, retard the speed of the fuse. Storage at high temperatures, however, causes a marked retardation which is apt to to cause delayed shots and misfires; fuse should therefore never be stored near boilers or other places where the temperature is high. Moisture seriously impairs the efficiency of fuse, which should be carefully protected from it. Although the train of powder is covered with a waterproof covering throughout its length, the powder exposed at the end readily absorbs moisture and the cotton or hemp threads in the center act in the capacity of sponges, so that the fuse for a foot or so from the end may be impregnated with moisture. When the fuse is lighted this water is driven ahead of the fire in the form of steam and delays the burning of the fuse, and if there is enough of it may become concentrated and extinguish the fuse entirely. Although twisting and bending seemingly have little effect upon the rate of burning, mechanical injury, such as pounding or crushing by falling rock, and abrasion, such as might result from the use of the tamping stick for consolidating the charge, greatly increase the rate. The results of the bureau's experiments show most conclusively that the greatest care should be

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[a] Snelling, W O., and Cope, W. C., The rate of burning of fuse as influenced by temperature and pressure: Technical Paper 6, 1912, 28 pp.

taken in the storage and handling of fuse to prevent accidents from premature or delayed explosions.

A detonator or blasting cap consists of a copper cylinder closed at one end and about the diameter of an ordinary lead pencil, into which is packed some dry mercury fulminate and potassium chlorate. When used with a safety fuse, the end of the fuse is inserted in the open end of the copper cylinder, which is then crimped around the fuse by the use of suitable pliers. Under no conditions should anything but the proper tool be used for this purpose, because the fulminate of mercury is extremely sensitive to very slight shocks, and there is sufficient strength in a single detonator to produce disastrous results if discharged accidentally. In some tunnels, more especially those in the Eastern States, the detonators are ignited directly by an electric current. For this purpose special electric detonators are required in which the fuse is replaced by two suitably insulated copper wires joined at the inside end by a bridge of fine platinum or other high-resistance wire capable of becoming incandescent during the passage of an electric current. These are inserted in an ordinary detonator into which some guncotton has previously been placed. Caps prepared in this manner are called electric detonators and may be obtained from the manufacturers with wires of varying lengths as required.

Figure 29 shows the component parts of one of these detonators.

The strength of detonators is determined by the weight of mercury fulminate they contain and they are generally designated as triple X, quadruple X, etc.

The following table shows the weights of charge used in the different grades:

Weights of charges used in detonators and electric detonators of various grades.

Weights of detonator charges.		Weights of electric detonator charges.	
Commercial grade.	Weight of charge.	Commercial grade.	Weight of charge.
	Grains.		*Grains.*
3X or triple	8.3	Single strength	12.3
4X or quadruple	10.0	Double strength	15.4
5X or quintuple	12.3	Triple strength	23.1
6X or sextuple	15.4	Quadruple strength	30.9
7X or No. 20	23.1		
8X or No. 30	30.9		

The detonator chosen for blasting in tunnel work should be strong enough to produce complete detonation. With "straight" nitroglycerin dynamite, 3X caps were considered heavy enough, but with gelatin dynamites much stronger ones should be used, because this dynamite is not nearly as sensitive (being, of course, safer to handle).

The jellylike mass of the gelatin dynamite also has a tendency to retard the explosive wave as it passes along the bore hole and, therefore, requires a much stronger initial explosion to carry the wave with the same force through the entire length of the charge. The leading manufacturers all recommend nothing weaker than 6X caps for gelatin dynamite; although 5X caps have given results that were thought to be sufficiently satisfactory at some tunnels; at others, when a change was made to those of greater strength, the universal experience has been that the better results more than warranted the change, the common report being that "it pays." It is, therefore, here recommended that nothing weaker than 6X detonators (or double-strength electric detonators) be used with gelatin dynamite, which is practically the only kind employed in tunnel work. In addition to the less effective results produced by the lower-strength caps, the composition of the gases is greatly changed when detonation is not complete, and unsuspected and dangerous constituents may result. As we have seen, with complete detonations the gases are mainly carbon dioxide and nitrogen, with perhaps a small amount of carbon monoxide. With incomplete detonations a much greater percentage of dangerous carbon monoxide is formed from the nitroglycerin, and in addition the highly toxic peroxide of nitrogen is produced in larger or smaller amounts according to varying degrees of completeness of the reaction.

LOADING.

There has been much discussion lately regarding the proper position for the primer (as the particular cartridge of dynamite containing the detonator is called) when loading a blast hole, some arguing that it should be the last cartridge to be placed in position, whereas others claim the only proper place for it is at or near the bottom of the hole. One of the more common arguments for placing it at the top instead of the bottom of the hole is the fact that by so doing one removes the danger of igniting the dynamite from the side spitting of the fuse, which not only lessens the efficiency of the explosive, but produces dangerous gases. For it is obvious that when the fire in the fuse is compelled to travel past the full length of the charge, there is danger of the flame bursting through the waterproof covering and igniting the powder. However, an expert connected with one of the leading explosives-manufacturing companies states that gelatin dynamites (the kind generally used in tunneling) are less liable to be deflagrated in this manner than any of the others. Nevertheless, the objection, which is not applicable when electric detonators are used, is worthy of serious consideration when safety fuse is employed.

A second argument in favor of placing the cartridge last is the fact that the dynamite charge is much more apt to be packed firmly, thus eliminating air spaces which decrease the effectiveness of the explosive. For when the primer is the first or the second cartridge in the hole, the remaining cartridges and especially the one immediately following can not properly be pressed in place with safety, and air spaces are likely to be left. This objection, of course, applies equally to charges detonated by safety fuse or electricity.

A third argument is the fact that the detonation of a charge of dynamite in a bore hole takes place in a series of steps which follow each other with almost inconceivable rapidity but are nevertheless distinct. The first of these is the explosion of the cap, which in turn detonates the dynamite in the primer and causes what may be termed the primary explosion, which is in turn communicated to the remainder of the charge. Although, by employing a strong cap, the amount of dynamite detonated in the primary explosion can be greatly increased, it can never be large enough to disrupt the rock completely, the greater force of the secondary explosion being required for this purpose. If, then, the cartridge containing the detonator is placed at the bottom of the hole and fired, the primary explosion consumes the dynamite ineffectually at the bottom of the hole where the full force of the blast should be used; consequently there is much greater danger of losing the last 8 or 10 inches of the round, with a consequent decrease in daily advance. It is also claimed that in placing the primer at the bottom, the explosion of the detonator tends to force a certain amount of the charge from the hole; this contention, however, is debatable. But if the primer is placed at the top, the primary explosion does not consume dynamite that is essential, and the full strength is developed from the remaining part of the explosive, except in so far as it is influenced by the previously mentioned hindrance to the explosive wave caused by the use of gelatin dynamite. And again, by placing the primer at the top, during the primary explosion a certain amount of pressure is developed and the remainder of the charge is, therefore, detonated under that greater pressure, hence its effectiveness is increased. This applies particularly where stemming is employed, and it requires only 2 or 3 inches of clay stemming to produce the results. If clay stemming is not used an extra stick of dynamite placed on top of the primer, which acts partly in the same capacity, is of great assistance.

On the other hand, it is claimed that when the primer is placed on top of the charge the collar of the hole is apt to be knocked off by the explosion of a neighboring charge. There is some question as to whether this really would happen, but if it did it would be a very strong indication either that the hole was misplaced or was too heavily loaded; for, as some one has said, " certainly the collar of a hole is no place for dynamite." And as to the objection that when the primer is placed at the top the fuse is liable to be torn out by flying rocks, the remedy is a very simple one—that of coiling the fuse carefully close up to the hole. And, finally, if the wave does not travel with enough force to the bottom of the hole, the matter can be remedied by the use of a strong detonator or by employing a higher grade of explosive, which would have the double effect of producing a greater primary explosion and of lessening the length of the charge, because less explosive would be required.

In view of the several considerations outlined above, it would appear that the primer should be placed at or very near the top of the charge. It is clearly recognized by the authors that, in some sections of the country at least, miners are accustomed to place the primer at the bottom. But it does not necessarily follow that a practice is correct because the miners so consider it, for many of them also think that dynamite exerts a greater influence downward than in any other direction.

The use of stemming in tunnel work is also a mooted question. When low explosives, such as blasing powder, having a slow rate of explosion are employed stemming is, of course, absolutely essential for confining the gases long enough for their full strength to become effective. But with dynamite, whose detonation is extremely rapid, almost instantaneous, many persons believe that stemming is not required, and this belief appears to be warranted by the results obtained without its use. A possible explanation for this is the fact that in tunnel work the holes are generally overloaded and hence the pressure produced by the extra few inches of dynamite charge tends to confine the gases generated by the remaining and effective part of the charge, which is, of course, also the function of stemming. Another probable reason is that the inertia of the column of air in the bore hole acts as an incomplete substitute for stemming of some more solid material. This can be demonstrated by the effect produced by exploding uncovered dynamite on some flat surface in the open, and this effect doubtless has given rise to the belief that dynamite exerts more force downward than in any other direction. It is unanimously agreed, however, by experts who have studied the subject that better results can be obtained from any properly loaded hole when more substantial stemming is employed. The quantity of stemming required depends, of course, upon the rate of detonation of the explosive. With black blasting powder it may be necessary to fill nearly all the remainder of the hole in order that the stemming may not be forced out before the reaction is complete and the full strength of the gases produced; but with ordinary charges of gelatin dynamite 2 to 6 inches of well-packed clay stemming will in most cases be fully sufficient.

The use of stemming in tunnel work has several disadvantages. In the opinion of many, if indeed not a majority, of tunnel men they more than counterbalance any gain in efficiency of explosive from its use. In the first place it causes delay in loading the holes at a time when every minute is precious. Again, the majority of miners, and especially those in the Western States, are strongly biased against it, and anyone who has tried to overcome one of their prejudices will appreciate the difficulty that would be experienced in getting them to use stemming, although, of course, if stemming were absolutely essential to good results, mere prejudice on the part of anyone should not be allowed to stand in the way of its adoption; still, as this is not the case, the wishes of the miners are usually deferred to. But a more serious disadvantage of the use of clay or similar material for stemming is the danger attending its removal from a missed hole, one of the most prolific sources of accident in tunnel work. But if the stemming consists of an extra stick of dynamite, as is usual in tunnel work, the simple insertion of a primer on top of the unexploded charge is all that is needed to prepare the hole for refiring. For these reasons, then, although clay stemming is essential in a bore hole that is not overloaded, for tunnel work in which it is customary to use more rather than less "powder" than is required, it is not so necessary that clay, sand, or similar stemming be employed.

Another thing to be considered in connection with the loading of a blast hole is the necessity of having the dynamite properly thawed. Thawing is required not only because of due regard for the safety of the men (which alone should be more than sufficient), but also because frozen dynamite can not be properly packed in the hole, and air spaces can not, therefore, be avoided, and because frozen dynamite does not develop its full strength on detonation, so that there is a decided loss in effectiveness.

It is desirable that the cartridges have as nearly as possible the same diameter as the drill hole, and that they be slit (carefully, of course) along the side with a sharp knife just before they are placed in the hole. Slitting enables the explosive to conform to any irregularities in the shape of the hole. The position of the detonator in the priming cartridge deserves attention also. Experiments have shown that the maximum force from a detonator is developed in the direction of its length. For this reason the detonator should be inserted into the end of the cartridge and not obliquely in one side, as is often done in tunnel work. Nor should the fuse project into or be laced through the cartridge because of danger of setting fire to the cartridge instead of detonating it properly with the cap.

FIRING.

When the number of holes to be fired is large, the work of lighting the fuses is generally done by two men, but when there are only a few holes in the round one man is sufficient. If two men do this work, it is customary for each man to light the fuse of a corresponding hole (for example, opposite cut holes) at the same time, each calling out the hole as he lights it. The actual ignition of the fuse (which should be closely coiled and the free end split for one-half to three-fourths of an inch to expose the powder train) is accomplished in various ways. At some places a candle is used and at others an acetylene lamp. The much better practice, however, is to use what is called a "spitter"; that is, a short piece of fuse that has been slashed and partly severed at regular intervals of perhaps one-half inch so as to expose the powder train. When the end of a spitter is ignited, the fire travels along the spitter, and as it reaches one of the cuts it spits violently out of the side; if the spit is directed toward a fuse it is almost certain to cause proper ignition. Each fuse should be ignited separately, and no attempt should be made, as it sometimes is, to bunch several and ignite them all at once. When it is necessary to protect the ends of the fuse from water, an empty powder box may be employed.

In extremely wet tunnels it is sometimes necessary to use a fuse igniter. One form of igniter that has given good results at a number of places consists of a short cylinder of celluloid, closed at one end and having the same inside diameter as that of the outside of standard fuse, and containing a small quantity of gunpowder or similar explosive. In use the open end is slipped over the free end of the fuse, which, instead of being split, is cut square. The igniter fits the fuse tightly enough to be held in place by friction. When being lighted the igniter is, of course, protected from dropping water, and the celluloid is set on fire by a candle or other flame; as the igniter is unaffected by mere dampness, it burns until the powder charge is reached, when the flash seldom fails to start the fuse. These igniters are not expensive and are exceedingly useful in wet work.

When ordinary electric detonators are employed, the only operations required are those of connecting the wires and passing a current through them by closing an electric-light circuit, or by generating a current in a so-called "battery" which consists of a hand-operated magneto or dynamo. All the holes so connected are exploded simultaneously, the chief and most serious disadvantage of electric firing for tunnel work. As has been mentioned, blasting in tunnel headings to be effective must take place in several steps—the cuts first, followed by the relievers, backs, sides, and lifters. Therefore, with electric blasting, although it has the advantage of shooting the cuts simultaneously, it is necessary for some one to return to the heading and connect the wires leading to the charges in the holes to be fired in each of the succeeding steps; and as some time must always be allowed in order to permit the smoke to clear, and oftentimes no little shoveling is required to uncover wires that have been buried by a previous round, blasting a round in this manner takes much more time.

To overcome this defect, manufacturers of blasting supplies are trying to perfect a delay-action detonator which resembles an ordinary electric detonator except that the platinum bridge, instead of directly igniting the mercury fulminate, sets fire to a short train of gunpowder, so that an appreciable, although short time elapses before the flame reaches the detonating part. By making the powder train of two different lengths two delays are obtained, so that if the special detonators be used in connection with a detonator not containing a powder train, the blasting can be performed in three stages from a single connection of the wires and from only one closure of an electric current. When these devices are used in tunnel work an instantaneous detonator is usually placed in the cut holes, a "first delay" in the relievers, and the "second delay" in the remaining holes. Unfortunately, the special detonators have not as yet been perfected for more than two delays, and this limitation has undoubtedly prevented their more extensive use in tunnel headings. For such blasting three stages hardly give satisfactory results, because a fourth stage is essential for the lifters, whose function is to throw the material broken by the other holes from the immediate front of the new face. Moreover, with the horizontal-bar mounting, a fifth step is also desirable in order to permit one lifter to go off after the others and throw the material away from that side of the tunnel where the capstan end of the bar is to be placed, so as to afford plenty of room for the jack bar and to permit it to be screwed tightly in place.

However, the delay-action electric detonators that are now found on the market permit the blasting to be conducted in almost any number of steps that may be required. Like the ordinary electric detonators, they consist of a platinum wire bridge inclosed in a metal cylinder by a waterproof composition. In the other end of the cylinder (which, instead of being closed and containing mercury fulminate, is left open) a short section of ordinary safety fuse is inserted, crimped in place, and the joint waterproofed. An ordinary blasting detonator is placed on the outer end of this piece of fuse and an electric current is passed through the wires leading to the platinum bridge, when the fuse takes fire and burns until it ignites the detonator. By making the pieces of fuse inserted in the blasting caps of different lengths, any desired number of delays may be obtained from one connection of the wires and one closure of the electric current. This device, therefore, overcomes the one great disadvantage of electric firing.

Chief among the advantages of electric firing is the certainty of detonating all of the cut holes simultaneously. Although, of course, if two holes are connected they may explode as one, it is impossible to make several pairs of cut holes—in a wedge cut, for example—explode together when fuse firing is employed. There will always be enough variation in the rate of burning of the fuse to prevent it, no matter how exactly the lengths of the fuses are cut. But when the cut holes are detonated simultaneously, as can be done with electric firing, each can assist the other, with a resulting increased effectiveness from the explosive. It is, of course, true that holes fired on the first and second delays can not be made to detonate absolutely at the same instant, even with electric delay-action detonators; but this is not essential, as the work of the succeeding holes does not approximate that of the cuts.

Another advantage of electric firing is the absence of smoke and dangerous gases caused by the burning of fuse. A large percentage of these gases is carbon monoxide, as is shown by the following analysis of gases obtained from burning fuse.[a]

Analysis of gases produced by the burning of fuse.

[A. L. Hyde, analyst.]

	Per cent.
Hydrogen sulphide	0. 8
Carbon dioxide	32. 7
Oxygen	1. 4
Carbon monoxide	23. 4
Hydrocarbons	4. 1
Nitrogen	23. 8
Hydrogen	13. 8
	100.0

STORING.

The place used for the storage of explosives should be substantially constructed, well ventilated, protected as much as possible from fire or lightning, and should be kept locked to prevent the entrance of children or other irresponsible persons. At mines or quarries the ideal magazine is, of course, one of cement or of brick, but at most tunnels, where the work is usually of a somewhat temporary character, the cost of such a building is not always justified; the dynamite is stored usually in a short drift in the side of the hill or

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[a] Snelling, W. O., and Cope, W. C., The rate of burning of fuse as influenced by temperature and pressure: Technical Paper 6, Bureau of Mines, 1912, p. 8.

in a log house. If neither of these can be obtained, a frame house will answer the purpose, although of course not as well. When a house is used, it should always be covered with corrugated iron or some similar fireproof material and care should be taken to remove any small sticks and grass from immediately around it. If considerable quantities of explosive are to be stored, the magazine should be situated at some distance from the rest of the work, and in every case the powder should be kept far enough from the tunnel buildings to prevent serious damage to them, or to the persons working in them, in the event of an accidental explosion. Obviously, dynamite should not be stored near a dwelling.

More than one kind of explosive, as, for example, black blasting powder and dynamite, should not be stored together, but there is no particular objection to the storage of different grades of the same kind of dynamite in the same building, except the possibility of confusion that might result from such a practice. Detonators, electric detonators, and fuse should under no circumstances be stored in the same building with dynamite, nor should the operation of placing caps on safety fuse be conducted at or near the magazine or thaw house. Tools should never be permitted inside of the magazine, nor should the boxes of dynamite be opened there. The floor of the magazine should always be constructed of wood, and it should always be kept free from grit and dirt.

THAWING.

Most dynamite freezes at a temperature of 45° to 50° F., and if it is subjected to such temperatures it must be thawed before it can be used. In tunnel work dynamite is generally thawed by spreading the sticks on shelves in a warm room or small building, separate and preferably somewhat removed from the main magazine. If the power for the tunnel work is derived from a steam plant, the waste steam is very often used to heat the thaw house. In this case, however, it is very essential that the steam coils be boxed or screened in such a way that it will be impossible for a stick of dynamite to fall upon a steam pipe, as a serious explosion might result. Nor should the pipes be so placed that any nitroglycerin exuding from the cartridges can fall upon them. As most thaw houses are insulated from the cold by having double walls or high-banked earth against the sides, only a little heat is required to keep them warm; if electricity is available, a cluster of incandescent bulbs is often used for heating, with good results. At other tunnels a special heater was observed, which was composed of one or more coils of iron or other high-resistance wire stretched between insulators on a suitable framework, usually of wood. When a heater of this type is employed it must be protected from the danger of a stick of dynamite lodging upon the wires, because they are generally hotter even than steam coils (a red glow being not uncommon), and hence there is much greater danger of explosion from this source.

At one of the tunnels visited an unprotected heater of this type was employed, and when comment was made upon the fact that there was no protection, reply was made that this condition was intentional. The reason given was that any person entering the thaw house was supposed to turn off the current by means of a switch provided for that purpose, and that the knowledge that the coil was not protected would make the men more careful to see that this was done. Such reasoning is all right as far as it goes, but it does not provide for the contingency of a stick of powder falling off the shelves when no one is in the building, nor is the thaw house safe for some little time, even after the wires have been disconnected, as they do not cool off instantly. Taken in connection with the fact that this particular thaw house was only a short distance from the tunnel portal, that it had no lock, and that all timber and other tunnel supplies had to be hauled past it, the situation should have been marked, in the language of insurance, " extra hazardous." That it did not occasion some accident during the period of its use is truly marvelous.

PRECAUTIONS.

Carelessness in the use of explosives is one of the prime causes of accidents in tunneling; it follows, therefore, that every care should be taken in their handling. Although the following list of precautions, compiled from a number of sources, is not claimed to be complete, it is given here in the hope that it may once more repeat some of the precautions to be observed in the handling and use not of dynamite alone but of the accessories of blasting as well.

HANDLING.

Don't forget the nature of explosives, but remember that with proper care they can be handled with comparative safety.

Don't smoke while handling explosives, and don't handle explosives near an open light.

Don't shoot into explosives with a rifle or pistol, either in or out of a magazine.

Don't attempt to manufacture any kind of explosive except under the supervision and direction of a trustworthy person who is skilled in the art. Many serious accidents, which have destroyed lives or inflicted injury on persons and property, have been caused by such attempts.

Don't carry blasting caps or electric detonators in the clothing.

Don't tap or otherwise investigate a blasting cap or electric detonator.

Don't attempt to take blasting caps from the box by inserting a wire, nail, or other sharp instrument.

Don't try to withdraw the wires from an electric detonator.

STORING.

Don't leave explosives in a wet or damp place. They should be kept in a suitable, dry place, under lock and key, and where children or irresponsible persons can not get at them.

Don't store dynamite cartridges on end, as this increases the danger of nitroglycerin leaking.

Don't store or handle explosives near a residence.

Don't open packages of explosives in a magazine.

Don't open dynamite boxes with a nail puller or powder cans with a pickax.

Don't store or transport detonators and explosives together.

Don't store fuse in a hot place, as this may dry it out so that uncoiling will break it.

Don't keep electric detonators, blasting machines, or blasting caps in a damp place.

Don't allow priming (the placing of a blasting cap or electric detonator in dynamite) to be done in a thawing house or magazine.

THAWING.

Don't use frozen or chilled explosives. Most dynamite freezes at a temperature between 45° and 50° F.

Don't thaw dynamite on heated stoves, rocks, sand, bricks, or metal, or in an oven, and don't thaw dynamite in front of, near, or over a steam boiler or fire of any kind.

Don't take dynamite into or near a blacksmith shop or near a forge.

Don't put dynamite on shelves or anything else directly over steam or hot-water pipes or other heated metal surface.

Don't cut or break a dynamite cartridge while it is frozen, and don't rub a cartridge of dynamite in the hands to complete thawing.

Don't heat a thaw house with pipes containing steam under pressure.

Don't place a " hot-water thawer " over a fire, and never put dynamite directly into hot water or allow it to come in contact with steam.

LOADING.

Don't allow thawed dynamite to remain exposed to low temperature before using it. If it freezes again before it is used it must be thawed again.

Don't fasten a blasting cap to the fuse with the teeth or by flattening it with a knife; use a cap crimper. The ordinary cap contains enough fulminate of mercury to blow a man's head or hand to pieces.

Don't "lace" fuse through dynamite cartridges. This practice is frequently responsible for the burning of the charge.

Don't explode a charge to chamber a hole and then immediately reload it, as the bore hole will be hot and the second charge may explode prematurely.

Don't force a primer into a bore hole.

Don't do tamping with iron or steel bars or tools. Use only a wooden tamping stick with no metal parts.

Don't handle fuse carelessly in cold weather, for when it is cold it is stiff and breaks easily.

Don't cut the fuse short to save time. It is dangerous economy.

Don't worry along with old broken leading wire or connecting wire. A new supply will not cost much and will pay for itself many times over.

FIRING.

Don't explode a charge before everyone is well beyond the danger line and protected from flying débris. Protect the supply of explosives also from this source of accident.

Don't hurry in seeking an explanation for the failure of a charge to explode.

Don't drill, bore, or pick out a charge that has failed to explode. Drill and charge another bore hole at least 2 feet from the missed one.

MUCKING.

NUMBER OF MEN.

The number of men in the crew that removes the rock broken in blasting exerts an important influence upon the speed at which the tunnel can be advanced. If the vertical column or the drill carriage is employed and the remainder of the work can not proceed until the heading is cleared, every minute saved at this work can be transmuted directly into progress; and although with the horizontal-bar system, nearly all of the mucking is done simultaneously with the drilling, and the heading can ordinarily be cleared by the time the drillers have finished the upper round of holes, still if the crew of shovelers is not large enough to accomplish this promptly, the delay is serious. In any event it is essential for rapid progress that the muck be removed as speedily as possible, as there are always a great number of little things to be done by the shovelers, even after the main work of loading the débris has been accomplished. Further, it is obvious that the removal of the muck in the shortest space can be accomplished only by a nice adjustment to conditions and the employment of the exact number of laborers proper for the purpose; for it must be remembered that the space in the tunnel heading is most restricted, and if too many men attempt to work there simultaneously they will so seriously interfere with one another as to offset any possible gain from the employment of the extra men. On the other hand, if too few men are at work it will be impossible for them to remove the débris within the time allowable. Analysis of the most satisfactory practice at a number of tunnels shows that, under the conditions prevailing in the heading, a man shoveling requires 2½ to 3 feet of floor space. That is, for a tunnel 10 feet wide not more than four shovelers should be used simultaneously, whereas not more than two men can work to advantage side by side in a 6-foot heading. In addition to these, however, it is desirable to have a man or two at work picking down the rock pile in front of the shovelers, loosening bowlders, assisting in the handling of the cars, or doing any of the many other things that make for speed in loading the muck. Accordingly, at tunnels 6 to 10 feet wide the proper number of men in the mucking crew ranges from three to eight.

At the Loetschberg and other of the European tunnels two sets of muckers were employed, one of which would rest while the other was engaged in loading the car. This course was thought to be conducive to greater speed, because the men could work much harder for the few minutes it took to load a car if they had an equal time to rest during the loading of the next one. There is little doubt that this is true or that the heading can be cleared sooner when such a method is used, but because of the higher cost of labor in this country, especially in the Western States, it is greatly to be questioned whether the gain would be sufficient to make such procedure profitable. At the Loetschberg Tunnel the shovelers receive a daily wage of 80 cents[a] as compared with the $3 or $3.50 for like work in the western part of the United States, so that a double crew of only five shovelers (similar to those of the Loetschberg Tunnel) would entail in this country an extra cost of $15 to $17.50 per shift, as compared with $4 in Europe. As the advance per shift in America rarely exceeds 7.5 feet, the extra cost of the double-crew system would amount to at least $2 per foot of tunnel driven. This would be justified only if it obviated a corresponding delay, although even in that event the question could properly be raised whether a change or adjustment in some other phase of the work was not the better solution.

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[a] Saunders, W. L., Tunnel driving in the Alps: Bull. Am. Inst. Min. Eng., July, 1911, p. 532.

POSITIONS OF WORKING.

The advantage of giving the men a rest from the grind of steady shoveling can be obtained without the necessity of extra laborers by changing their positions regularly according to the system in use at several of the tunnels visited by the authors. At the Laramie-Poudre Tunnel, where one of the best examples of this method was observed, the six muckers worked according to the following cycle of operations: As soon as a car (1) was filled with waste two shovelers, who will be designated as A and B, took it at once to the rear, while two other shovelers, C and D, jumped to an empty car (2) near by (which had previously been thrown off the track on its side), set it upright on the track and pushed it into a position to be filled. In the meantime the remaining two, E and F, stopped picking down the rock pile, took the shovels left by A and B, and started at once to assist C and D in filling car 2. Another car (3) was then brought up by A and B as near as possible to the car (2) being filled, and was thrown off the track on its side in the position formerly occupied by the second car. A and B then picked down the rock pile for the other four during the remainder of the time consumed in loading car 2. When filled this car was removed by C and D, while E and F set up the third car and filled it with the assistance of A and B. The fourth empty car was meanwhile brought up by C and D, who then took their turn at picking down, and the cycle was completed when E and F took the third loaded car to the rear, returning with another empty, and then resumed their original position on the muck pile. It will be seen that by this method every man spent at least one-third of the time in tramming or picking down the rock pile, either operation being easier work than that of shoveling, and amounting virtually to a rest which, although perhaps not as complete as if no work at all had been done during that period, was still sufficient to relieve greatly the hard monotony of shoveling.

The regularity and mechanical exactness of procedure with this system is a still more important advantage. Each man soon learns precisely what is expected of him for each step of the operation and hence there is absolutely no confusion, no lost motion. There is rarely any occasion for the foreman to give an order to the men except under unusual circumstances, and in consequence he does not acquire the habit of shouting at the men constantly, an unfortunate phase of this work only too noticeable at some of the tunnels visited; nor, on the other hand, do the men form the time-wasteful habit of running to him for guidance in every minor contingency that arises. The statement that the little things make for success is not claimed as original, but it can nowhere apply better than in planning the utmost work attainable in the limited space of a tunnel heading; indeed, this seeming detail of eliminated friction and confusion warrants and deserves the most serious consideration.

In addition to advantages in organization, the speed attainable with this method leaves little, if, indeed, anything, to be desired. Cars of 16-cubic foot capacity were filled at the Laramie-Poudre Tunnel ordinarily in three or four minutes, and on one occasion (which, however, was somewhat exceptional, as the men realized that they were being timed) only one minute and thirty seconds was needed. At the Rawley Tunnel, where a similar system was used with only four muckers, 25 cars, having a capacity of 17 cubic feet, were filled in exactly 2 hours, and on a different shift 20 cars were loaded in 1 hour and 45 minutes. These figures are from an accurately timed record kept by one of the authors. The usual time required for mucking at the Rawley Tunnel is not far from the average of these figures (which include all ordinary delay incident to making the cars up into trains), a value somewhat less than six minutes per cubic yard of rock loaded. This does not suffer by comparison with the Loetschberg Tunnel, where five minutes was required to fill a cubic-meter car (35.5 cubic feet)[a] by a crew of 10 men, with an extra minute to remove it when full and replace it with an empty one.

HANDLING CARS.

The method of handling the tunnel cars is still another detail of consequence in the operation of mucking. One of the most common arrangements is to have them trammed from the face by hand to a siding or switch where they are made up into trains and hauled to the portal by whatever means are provided for that purpose. This system, however, possesses some disadvantages. The switch must be moved frequently at no little expense and trouble in order to keep pace with the tunnel advance, or else it will soon be so far from the face that it is practically worthless. There is considerable loss of time while the loaded cars are being removed and the empty ones are being brought to the face; even although every effort be made to reduce this lost time to the minimum, the switch can not well be located nearer than 100 feet from the face, whereas in practice 300 to 500 feet is more apt to be the actual distance. Moreover, the full car must usually be taken by hand the entire distance to the switch before the empty can pass it; when this system is employed heavy cars, almost without exception, are used (for reasons later shown), and on this account to move them any distance by hand entails heavy exertions on the part of the mucking crew.

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[a] Saunders, W. L., Tunnel driving in the Alps: Bull. Am. Inst. Min. Eng., July, 1911, p. 535.

At some tunnels this difficulty was obviated by extending two tracks all the way to the face and loading cars on each one alternately. Even this arrangement is not entirely satisfactory, because it requires the extra labor and trouble of laying two tracks instead of one, which must be done after the tunnel is cleared of débris and before the new round is fired and is therefore very apt to cause a serious delay in the whole work, especially if the shovelers are a little late in clearing the heading. In addition, the need for keeping the switch as close as possible to the face is not so apparent with this method as with the first, and hence this most necessary feature is apt to be neglected. In that event much time will be wasted in the course of a shift by the men tramming the cars an extra distance.

At one tunnel the necessity for a double track was avoided by covering the entire floor of the heading with steel plates for about 30 or 40 feet back from the face. The cars to be loaded could easily be jumped from the track onto the first of these plates and rolled as near the rock pile as necessary, and when one car was full an empty one could be shunted around it without difficulty and placed in position for loading while the full one was being rolled back upon the track and trammed to the siding when the trains were made up. Such a method is simple and effective and, except for the work of moving the siding ahead, requires little extra labor; most of the plates are needed in any event for the men to shovel from, so that the work of adding one or two more would scarcely be noticed. This procedure is recommended if for some reason it is necessary to employ cars of large capacity.

But in most cases, as was mentioned in the section on haulage equipment, it is much better to use cars of smaller capacity; then the empty ones are tipped off the track to allow the full ones to pass and can be righted when needed and placed back easily by two men, thus avoiding all the complications and extra work arising from the use of a siding or switch. The smaller cars are also easier to load, for, as they do not occupy as much space in the tunnel heading, there is more room for the shovelers to work; also, the sides of the smaller cars being lower, each shovelful of rock does not have to be lifted as high in order to get it into the car, saving both time and energy. They are likewise easier to handle in case of a derailment, and as fewer men are required for tramming them out of the heading when full, a larger percentage of the time of the shoveling crew can be spent in the actual process of loading.

When the smaller cars are used, however, the work of handling them must be thoroughly systematized in order to prevent waste of time through avoidable delays. Although similar to that in use at a number of the tunnels examined, the system employed at the Rawley Tunnel was, perhaps, more carefully planned in all the details than were any of the others. Upon arrival at the heading the empty cars were pulled as near as possible to the full cars waiting to be removed, which at this juncture ordinarily stood on the track some 75 or 100 feet from the face of the tunnel. The mule was then detached from the empty "trip" and used to pull the full cars back to the car being loaded, usually the last one of the previous empty trip; or if they had all been loaded the full cars were pulled back as near the face as possible. The empty cars were then hauled up to the full ones and tipped off the track on their sides out of the way. All of this work was performed by the mule driver alone, except when the shovelers had completed the loading of the empty cars, when they assisted wherever possible in order to expedite the work. After seeing that the cars of the full trip were properly coupled up, the driver then started with them for the dump and the muckers took the two empty cars nearest the portal, set them on the track, and trammed them to the face where one car was again tipped off on its side while the other was being loaded. Unless the mule driver was delayed in getting to the heading so that he did not arrive before all of the cars of the previous trip were filled, the operation of getting the loaded trip out of the heading and an empty car again in position to be loaded rarely occupied more than three to five minutes. The remainder of the cycle was similar to that just described for the Laramie-Poudre Tunnel, each full car being trammed a short distance beyond the last one of the empty trip, which was then taken up to the face and thrown on its side ready for use without delay when needed.

To recapitulate, the chief advantages of this system are: (1) It does away with a switch near the heading; (2) the cars do not have to be trammed any great distance by hand—only a little more than the length of one trip—and the distance is constant and does not vary with the tunnel advance; (3) the minimum time is consumed in getting the full car out of the way and replacing it with an empty one; and (4) little time is lost in making up the trains to be hauled to the dump. It can not, of course, be used satisfactorily unless the cars are small enough to be handled easily by two men, but this is a matter that can be provided for in purchasing the equipment, and, as has been shown, the smaller car has other advantages that make it desirable for tunnel work. A system similar to the one outlined, modified, of course, to fit local conditions, is highly recommended for future tunnel work.

STEEL PLATES.

The use of steel floor plates from which to shovel rock broken by the blast has become so general that mention of this feature of mucking should hardly be necessary. The authors were much surprised, however, to find at one or two tunnels, where otherwise there was little left to be desired in the line of organization and equipment, that the muck was being shoveled without the use of plates. Even the most cursory study of the efforts of the men in pushing the shovels into the rock pile along the uneven surface of the bottom of the tunnel, and comparing the time required to load a car with results at other tunnels where steel sheets were in use, soon made it evident that much energy and time were being wasted needlessly.

At one tunnel the plates were not used because it was necessary to excavate the floor on a curve instead of making it flat, though even here plates could have been employed by leaving a part of the waste material in the bottom of the tunnel to form a flat surface upon which the sheets could have been placed. Upon inquiry at another tunnel the following reasons were given for their nonuse: (1) The muck was so sticky that it would not be any easier to shovel from the plates than from the rock pile; (2) it was impossible to prevent the sheets from becoming bent, twisted, and jumbled up with the muck when the holes in the bench were blasted; and (3) it was a great deal of trouble to lay them in position before blasting and to handle them during the work of mucking.

Although it must be admitted in all candor that the stickiness of the muck in this instance made it difficult to handle under any conditions, there is no real reason to suppose that shoveling from a plate would have been more arduous than from the pile—quite the reverse. The second objection is somewhat more serious for it is true, especially with heavy blasts, that the sheets are sometimes caught up and twisted by the explosion and occasionally hurled for considerable distances down the tunnel. If, however, the plates are properly covered with waste rock from the previous round before blasting, such occurrences are so extremely rare as to be neglible, and this is ordinarily the remedy for such difficulty. At the particular tunnel under discussion the trouble was somewhat different. It was being driven with a heading nearly square, which was followed at a distance of 8 to 10 feet by a bench 3 to 4 feet high. The holes in the bench were drilled from the same set-up as those in the heading, and they “ looked ” down and away from the heading and hence toward and slightly under the position that would have been occupied by the steel sheets. It is not surprising that with this arrangement much difficulty should have been experienced in keeping the plates down during the short time they were tried. Objection should not have been taken to the steel sheets but to the design of the tunnel itself, which was being driven considerably higher than it was wide but would have served every purpose required of it equally well if the dimensions had been reversed. Such a change would not only have obviated the sheet trouble but would also have made driving easier in other respects. Aside from this, the tunnel was not high enough to warrant the removal of the material in two operations, as was being done at the time it was visited. Since then, however, the bench has been abandoned, all of the material being excavated at once, which has made practicable the mucking from steel plates.

The third criticism is entirely a question of economy and can best be met by inquiring as to whether it is not better for the muckers to spend 15 or 20 minutes while the drillers are loading the holes, and possibly as much more time during the mucking, in doing work that will save itself several times over. For it can not be denied that a man can work to much better advantage and handle more rock during a given time if he shovels from a smooth surface. When he is shoveling from the pile the shovel can rarely be pushed more than an inch or two without encountering a piece of rock too big to be shoved aside, and there must therefore be a distinct stop while the shovel, with any load upon it, is lifted clear of the obstruction, only, in all probability, to encounter another one almost immediately. It is not surprising, therefore, to find that the experience at a large majority of tunnels leads to the conclusion that steel plates for shoveling are among the chief economies.

TIMBERING.

MATERIALS.

If the rock through which a tunnel is driven does not possess sufficient strength and rigidity to carry the weight of the superincumbent mass, artificial supports for the roof, sides, and sometimes for the bottom, are necessary to prevent the rock from falling, crumbling, or squeezing into the excavation. These supports may be timber, brick, stone, metal, or concrete. Because of its cheapness and availability in many mining districts and the ease with which it can be cut to the required sizes and shapes and placed in position, timber is the support most generally employed; and even if masonry or concrete lining is required in the specifications of the completed tunnel, timber is almost always used as a temporary support until the more permanent material can take its place.

In most underground situations seasoned timber is preferable to green because it is better able to resist decay. The bark should invariably be removed from round logs, as the space between it and the wood affords an excellent breeding place for many forms of wood-destroying insects, and the bark itself collects moisture and thus encourages the growth of fungi, which are the chief wood-destroying agents. Round timbers when properly peeled and seasoned are more durable than square timbers cut from a similar log of the same size and age, because the corners of the latter are especially liable to decay. In young and small timber, such as is generally used for mining work, the outer half of the log is usually sapwood containing starch, sugars, proteids, and other soluble organic compounds—the foods upon which decay-producing fungi thrive, which are practically wanting in the heartwood. If, in the process of squaring up, the attempt is made to obtain the largest possible square timber from a given log, the corners consist largely of this easily infected sapwood and are accordingly most liable to conditions bringing about quick rotting. It is not surprising, therefore, to find in moist underground workings where square timbers have been in place for three or four years that the corners of the timbers have decayed to such an extent that they can be pried off down to the heartwood with a miners' candlestick or any other sharp instrument. It is, of course, true that the outer part of a round log also consists of sapwood, but the exposed surface has not been injured or bruised by the saw.

Although round timbers deteriorate much more slowly than square ones, they are not as easily handled in the tunnel, are harder to align properly, and it is much more difficult to reinforce them by the ordinary false sets. If timber must be transported long distances, the greater weight of the round sticks (especially if the logs are truncated cones—or "churn shaped" as the miners say—instead of being nearly cylindrical), the freight costs become excessive and square timbers must be used. Under these conditions the saving in transportation charges will often pay for some type of preservative treatment to be applied to the timbers before they are placed underground.

The best method of checking the growth of fungi, and so increasing the durability of timber, is to poison the source of their food supply. Although there have been many processes invented for this purpose, most of those in use to-day depend upon the injection of either zinc chloride or creosote. The former can not be used advantageously in wet situations, however, for as it is soluble in water it is soon leached out, leaving the timber just as susceptible to attack as before. But if creosote is properly applied it can not be washed out, no matter how much water passes over the timber, and for this reason it is the preservative generally employed in mining and tunnel work. It is, however, somewhat more costly than zinc chloride and, as it is a liquid, the transportation charges are considerably higher than on zinc chloride, which can be shipped in bulk.

Creosote can be applied as a surface coating by painting with a brush or simple immersion in a tank, or the log can be more deeply impregnated with the preservative by one of the more complicated processes involving heat, pressure, or vacuum. Although painting is the least efficient method, it has the advantage of cheapness, and if carefully done will give fairly satisfactory results. Although dipping or simple immersion results in little if any greater penetration of the preservative, it insures a more certain filling and coating of the cracks, checks, and other imperfections of the log and thereby affords better immunity from decay. After the necessary equipment has been obtained, it is also in most cases cheaper than painting, because it is more economical of labor, it being easier to run a number of sets of timber through a vat by some form of mechanical conveyor than to paint the same number by hand. For either method the timber must be fully dried and seasoned beforehand; otherwise cracks in the wood due to the evaporation of the moisture will break the protective covering, which is only a thin one at best, and thus give the fungi access to the interior of the stick. It is perhaps unnecessary to add that these, as well as any of the other treatments, should be given after the timbers have been cut to form, so that the ends and the mortise openings may be coated as well as the sides.

If the extra cost of a more thorough impregnation is warranted, the "Bethell" process is widely employed for timbers that are to be placed in wet situations. By this treatment the timber is for several hours given a bath of live steam at perhaps 20 pounds pressure, after which it is subjected to a vacuum for three or four hours more, when creosote, heated to a temperature of approximately 160° F., is applied under pressure until the desired amount of the preservative is forced into the wood. "Burnettizing" is a process practically identical with this, except that zinc chloride is used in place of creosote. There are also a number of methods less frequently employed that are designed to effect economy in the amount of chemical required and differ chiefly in the manner of their application. In one or two of them the interior of the timber is impregnated with the less expensive zinc chloride, which is in turn protected from the action of water by treating the outer zone with creosote. A more complete discussion of processes than can well be included in this report may be found in Forest Service Bulletins 78 and 107, "Wood Preservation in the United States," and "The Preservation of Mine Timbers."

For permanent tunnel linings brick or stone were formerly the chief materials employed, and most of the older tunnels, both in this country and abroad, are lined in this manner. Such linings are expensive, however, and require a higher class of labor to place them in position than does concrete, their modern substitute, nor do they afford the same imperviousness. Although metal beams and posts are employed advantageously as roof supports in the main entries and gangways of some coal mines, high cost prevents their use except for this or work of similar importance. The best modern material for permanent linings is undoubtedly concrete, and although its employment in this work has thus far been restricted chiefly to railroad, irrigation, or water-supply tunnels, its use in practically every important mining tunnel where a permanent lining is necessary must almost certainly follow.

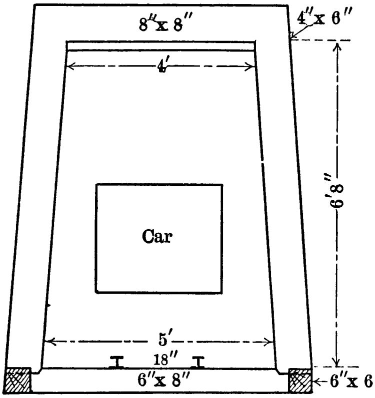

FIGURE 30.—Four-piece set for a small tunnel.

TYPES.[a]

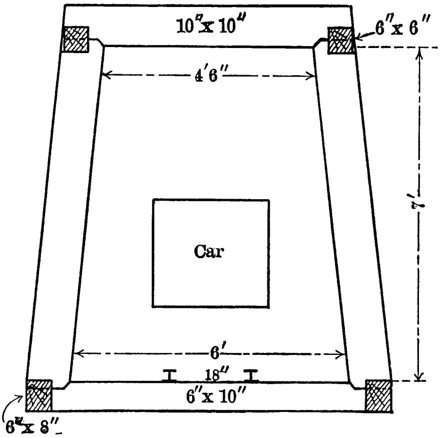

FIGURE 31.—Four-piece set for a medium-sized tunnel.

The simplest form of roof support is, of course, a single post, or a cap supported at either end by a "hitch" or recess in the side wall. If the sides of the tunnel are not strong enough to afford a hitch, the ends of the cap are supported by posts, and if the floor will not bear the weight of the posts a sill is placed for them to rest upon. Figure 30 illustrates such a four-piece set applied to as small a tunnel as it is usually advisable to excavate. The timbers are 8 by 8 inches, and instead of being partly beveled for withstanding side pressure, the posts are held apart by a 2 by 8 inch plank spiked to the cap. Figure 31 illustrates a common form of timbering designed for a tunnel of a convenient size for driving when a single track is all that is required. The timbers are 10 by 10 inches, and the joint between the cap and the posts is beveled at the corners so that the timbers can easily resist horizontal or vertical pressure without splitting. If heavier ground is encountered than can be held with

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[a] As tunnels and adits for the purpose covered in this report are rarely too large to be driven as a single heading, the many complicated and ingenious systems of timbering that are used in driving large railway tunnels, either with multiple headings or single heading and bench work, need not be considered here. For a discussion of these methods the reader is referred to the monumental works of Drinker ("Tunneling, Explosive Compounds, and Rock Drills," by Henry S. Drinker, 1878, 1025 pp.), of Prelini ("Tunneling," by Charles Prelini, 1902, 307 pp.), and of Stauffer ("Modern Tunnel Practice," by D. Mc.N. Stauffer, 1906, 300 pp.), and to the publications of the civil and mining engineering societies.

this set, the posts and cap can be made of 12 by 12 inch timbers, with 8 by 10 inch sills and 8 by 8 inch braces or simply "braces," as they are commonly called.

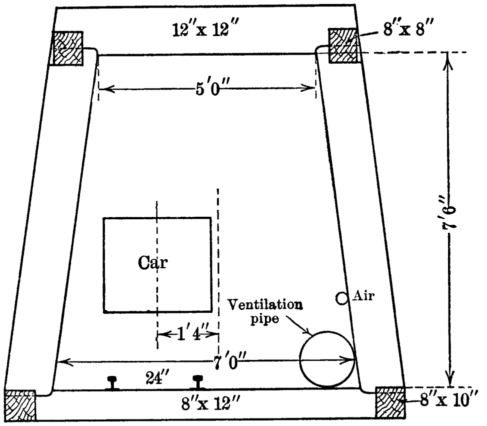

FIGURE 32.—Arrangement of timbering providing a manway at one side of the tunnel.

In single-track tunnels where there is considerable traffic it is often advisable to have the opening wide enough to give room for a manway and the ventilating pipe on one side and the car tracks on the other, as shown in figure 32. Or if the tunnel has to carry a considerable volume of water the design shown in figure 33 has been used in many instances and has given excellent satisfaction, notwithstanding the fact that both the opening and the timbers are unsymmetrical. The 6 by 6 inch sill, which also forms the rail tie, is not notched into the post on the right side, but is merely held in place by a 2 by 10 inch plank spiked to the face of the post, the upper end of the plank being recessed for a depth of 4 inches to receive the sill. If even larger volumes of water have to be provided for, the arrangement illustrated in figure 34 has given very good results at a number of places where it was tried. The amount of water and the grade of the tunnel of course determine the proper depth of the drain. The sill is supported by planks spiked to the posts, as in the preceding case.

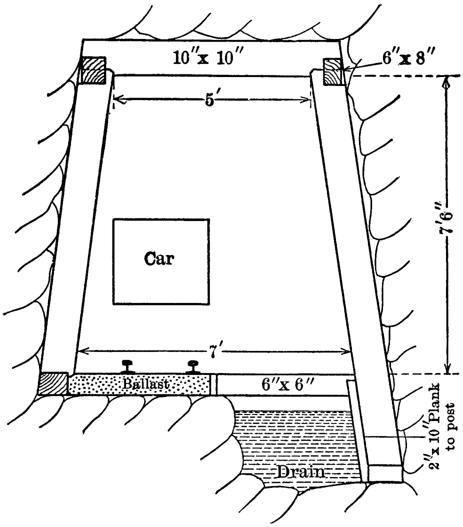

FIGURE 33.—Timbering for a wet tunnel.

Where the roof pressure becomes too great to be carried by a horizontal cap, what is known as the "arch set" is usually employed. Figure 35 shows the design of such a set for a tunnel 7 feet 6 inches high by 6 feet wide on the sills. The timbers are 8 inches square and the collar braces 4 by 6 inches. Instead of the braces being placed on the outer edges of the timbers, as is done on square sets where they can be slipped in from the outside, the collar braces on arch sets should be mortised into the face of the timbers in a central position, bisected by the joint, as shown in the illustration. By this means the level pieces forming the arch, being much more difficult to hold in place while being blocked than are square sets, are prevented by the braces from slipping. If more room is required in the upper part of the tunnel than is given in this design, vertical posts are often used (see fig. 36), a substitution that not only increases the width of the tunnel at the shoulders, but calls for all timber cuts to be at an angle of 30 degrees, making the sides and the top pieces of the arch interchangeable. Figure 37 illustrates the arch system of timbering as employed in medium heavy ground for a tunnel 8 feet in width by 7 feet 6 inches in height, which is about the minimum size for a double-track tunnel. If the walls of a tunnel are sufficiently firm to stand without timbers and only the roof requires support, the arrangement shown in figure 38 can often be used to advantage. This system makes a carefully constructed footing for the arch timbers necessary, but hitch-cutting with a modern hand pneumatic drill is comparatively a cheap operation, the cost of which will be repaid many times by the saving in timbers and in the smaller quantity of rock to be excavated.

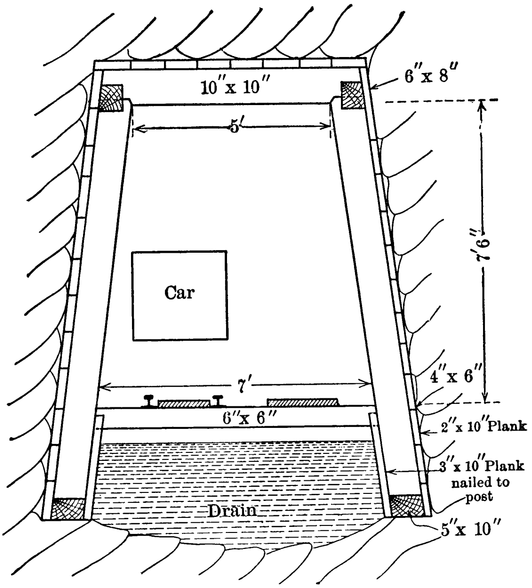

FIGURE 34.—Timbering for a tunnel producing a large volume of water.

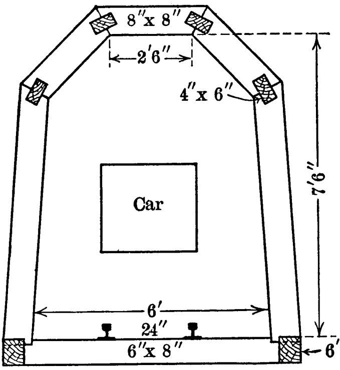

FIGURE 35.—Arch set for a small tunnel.

Swelling or creeping ground results from the exposure of certain rocks to the air, whereby they undergo chemical change and increase in volume so that the excavation not only closes in from the sides and roof but swells up from the floor as well. Under such conditions it may be necessary to design the timbers as shown in figure 39, the drain box and the track being protected by an inverted arch. If 12 by 12 inch timbers in this form will not resist the squeeze at the usual distance of 4 feet between centers, it is customary to close them up until they have sufficient resistance to withstand the pressure. Occasionally, however, zones of rock are encountered that can not be held even by this expedient, in which case the timbers can be kept from breaking by placing the sets about 3 or 4 inches apart; then whenever the pressure becomes too great it can be reduced by removing with a long-bladed pick whatever decomposed rock is in line with the open spaces, 4 or 5 inches back from the timbers. Swelling ground is usually so soft that this can be done without much trouble, and it is neither a difficult nor expensive matter to keep a wood-lined tunnel open until it can be lined conveniently with concrete; which, by preventing access of air to the rock, will remove much of the difficulty. The octagonal set (fig. 40) offers another means of holding such heavy ground, and it is often supplemented with concrete in front of and between the timbers, as shown on the left side of the illustration. To insure the safety of the tunnel after the timbers have decayed, the sets should not be spaced less than 12 inches apart, and 15 to 18 inches is still safe as the wider opening gives room for a stronger rib of concrete between the timbers. The section is exceedingly easy to handle, and if the flow of water is not too great for the drainage area underneath, nothing better could be adopted. All the

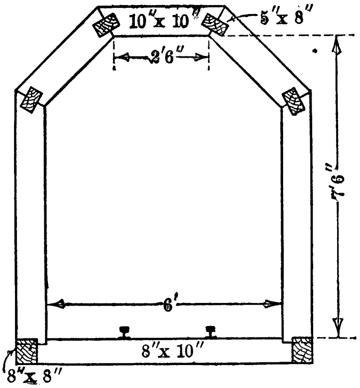

FIGURE 36.—Arch set with vertical posts.

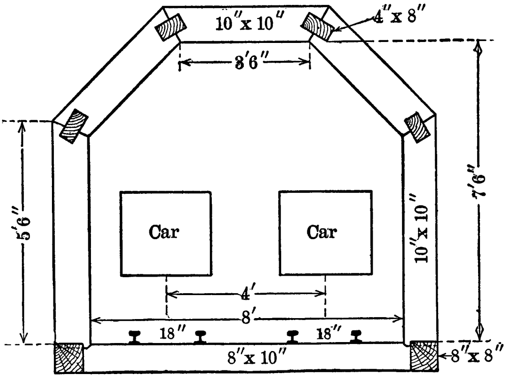

FIGURE 37.—Arch set for a double-track tunnel.

pieces in this timber set are exact duplicates, which is a great convenience not only in framing but also in storage and erection.

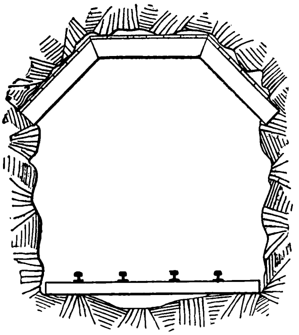

FIGURE 38.—Arch roof support and hitch.

An entirely different problem from swelling ground, and one temporarily much more difficult to handle, is often encountered where adits or tunnels have to be driven through shear zones, caved ground, or loose, crushed material that will not stand overhead without being supported as fast as it is opened. One of the oldest designs for driving through areas of this description is shown in figure 41, each alternate set carrying a double cap. This arrangement is simple, easily operated, and very satisfactory if the material in the roof or sides does not bring too much pressure on the spiling.[a] The method possesses, however, two grave disadvantages—it requires two different sets of timbers and the spiling used must be long enough to cover both sets. The latter difficulty, if the overhead material is heavy, may prove serious, as it is often difficult to drive spiling across one space, to say nothing of two.

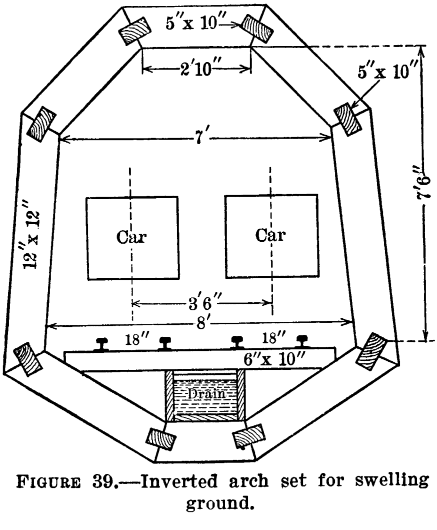

FIGURE 39.—Inverted arch set for swelling ground.

Under such conditions the tailblock system, illustrated in figure 42, is generally employed. As the timber sets are all the same size, it avoids one disadvantage of the preceding type, nor do the sets differ in any particular from those used with ordinary lagging. It offers, however, little improvement in the matter of driving the spiling in place. To be sure, the spiling does not have to cover two sets; but if the ground is heavy, the great pressure brought upon the tailblock by a comparatively small quantity of rock resting over the spiling as it is being driven forward creates an amount of friction that, added to the resistance in front of the spile, makes driving exceedingly

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[a] If the lining of a tunnel can easily be placed in position it is usually known as "lagging," but if it has to be sharpened to a chisel-shaped end and driven into position it is called "spiling" or "forepoling."

difficult even with the heaviest sledges that can be used. Although the greatest care be taken, the back end of the spile is often broomed and split by the heavy pounding required. This can be obviated in part by capping the back end of the spile with an iron shoe, and a heavy piston drill is sometimes employed to do the pounding. If the ground over the tunnel is very much shattered and weak, making the continued hammering on the spiling dangerous, it is safer to force the spiling slowly forward with light jackscrews.

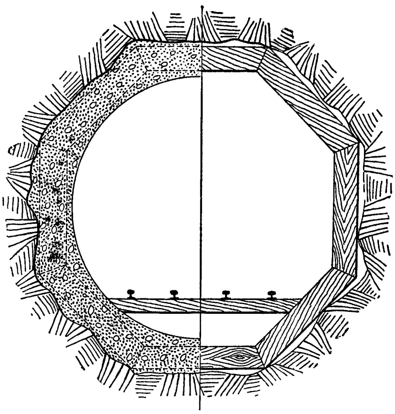
FIGURE 40.—Octagonal set for tunnel.

When an opening has to be driven for any considerable distance through soft material requiring immediate support, the work can be expedited greatly by the use of what is known throughout the West as the swinging false set, illustrated in figure 43. Like many other inventions, this system was the child of necessity, and was first used in the Cowenhoven Tunnel, at Aspen, Colo., where the overlying rock brought so much pressure on the spiles that it was almost impossible to drive them forward with an 18-pound sledge. With this method no tailblocks are employed, nor need the spiling be driven across two sets of timbers, as in figure 41. The weight on the front end of the spiling is carried directly on the swinging false set, and the spiling can be driven into place with a quarter of the hammering necessary under the tailblock system. As will be seen by inspection of the longitudinal section, the posts of the swinging false set rest and rotate on the sill of the permanent set and when first erected occupy the position shown by the dotted lines. They carry a circular steel cap, *b*, which supports the front end of the spile *a*, so that the only pressure to be overcome in driving is that of the rock immediately above and in front of the spile, which can consequently be driven forward much more easily than in the tailblock system, in which a weight of 100 pounds on the front of the spile would easily cause a pressure of five times that amount on its supports. As the spiling is driven

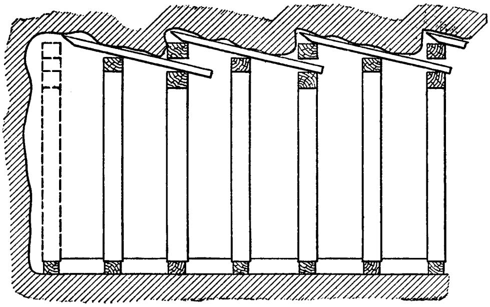
FIGURE 41.—Timbering for loose ground.

forward the turnbuckle *c* is slowly unscrewed, allowing the swinging false set to fall forward and carry the point of the spiling in a nearly horizontal line. When all of the spiles have been driven home and the supporting block *d* has been placed under them, the turnbuckle *c* is unscrewed still farther, permitting the hanging rods to be unhooked from the eyebolts and the false set to be advanced to a new position, one set farther ahead. The system requires that the timbers for at least five or six sets from the face shall be bolted together in very much the same manner as hanging bolts are used in placing shaft timbers; this, however, is a direct advantage rather than otherwise, for by bolting the timbers together and screwing them tightly against the braces they can be placed in position much more easily and quickly. Further, the timbers are held together so firmly that if hard ground is encountered in any part of the face much heavier charges of explosives can be used than if they were held in place merely with blocks and wedges. The swinging false set works equally well with square or arched sets, but if the latter are used the collar braces should be shifted from their normal position on the center line of the timbers, as shown in figures 36, 37, 39, and 40, to the outer end of the joints to make room for the greatest possible width of spiling; by this means the angle gap can be reduced to a minimum and the length of the "lacing," *e*, correspondingly reduced.

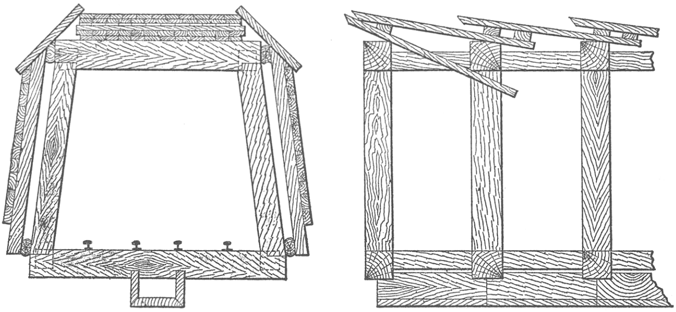

FIGURE 42.—Tailblock system of timbering.

In driving a heading where the character of the rock necessitates timbering close to the face, care must be taken to thoroughly brace and block the front sets before firing. If the roof "breaks high" and there is any possibility of large masses dropping out of it, the space between the lagging and the roof must be completely filled either with waste or blocking; otherwise a large piece of rock may drop from the roof and pass completely through the lagging and thus endanger the lives of the men below. Timbering close to the face always diminishes the rate of progress by compelling the use of shallower holes and lighter charges. An excellent plan to admit of heavier rounds under these conditions is to keep the last 6 or 8 sets of timbers firm and tight up against their collar braces by the use of tie bolts. These should be provided with center hooks to admit of their ready removal on the same plan as the hanging bolts that have so long been successfully used in shaft sinking. Only 6 or 8 sets of bolts are required, those from the rear being moved forward and used in the face. This system of tying the sets together has been found to be equally as advantageous in horizontal as in vertical driving.

The tunnel shield, such as is generally employed with or without compressed air for piercing subaqueous river-bed deposits, affords another solution of the problem of driving through soft ground. Although, in the matter of speed, safety, and economy, the modern shield leaves little to be desired, it has one great drawback, the initial cost of the installation, which practically bars it from use for the narrow zones or small areas of running ground usually encountered by the class of tunnels considered in this report.

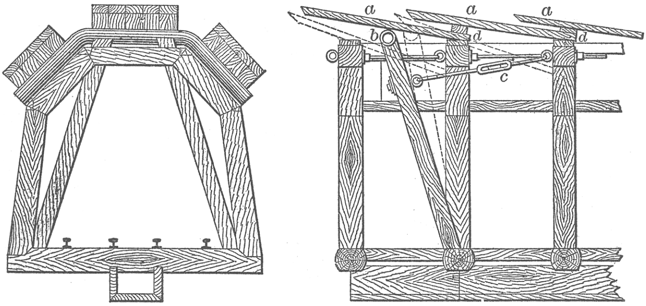

FIGURE 43.—Swinging false set for loose ground.

The system of timbering employed at the north end of the Elizabeth Lake Tunnel of the Los Angeles Aqueduct is of especial interest because of the ingenious and extremely effective means employed to drive through a comparatively hard rock which, however, was so shattered and broken that it could not be trusted to stand even temporarily without support; in addition, the speed and efficiency with which the timbering was placed in position were notable, enabling the north end of the tunnel to progress practically as fast as the south, although in the latter very little timbering was required. The following description is taken, with some condensation and rearrangement, from an article by Herrick:[a]

The main tunnel, approximately 12 by 12 feet, was preceded by a short pilot heading approximately 8 by 8 feet, in which the roof

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[a] Herrick, R. L., Tunneling on Los Angeles Aqueduct: Mines and Minerals, vol. 31, October, 1910, pp. 135–143.

was supported by "false" timbering. Assuming for convenience the time of inspection to have been at the end of a shift, with the drills removed from the breast preparatory to blasting the round, the position of the timbers would have been somewhat similar to that shown in figure 44, *a*, which represents a short-timbered section of the full-sized tunnel and the horizontal timbers used temporarily in supporting the roof of the pilot heading. The posts of the permanent sets were 8 by 8 inch square timbers, 8 feet 6 inches in length; and in the figure they are represented as spaced longitudinally at 8-foot intervals, although in practice they were often set irregularly, depending upon the weight of the roof. The width between the posts was 10 feet 2 inches in the clear, whereas the tunnel was broken as nearly as possible to a width of 12 feet. The collar braces were ordinarily 2 by 6 inch planks whose ends were supported either by wedges or timber ends spiked to the sets. The false posts in the heading were later used as permanent posts for the

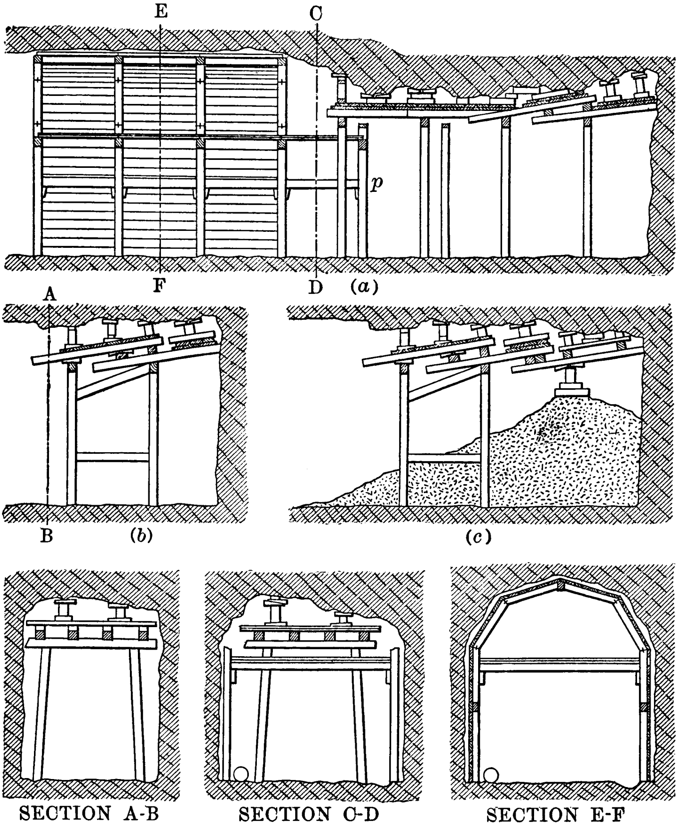

FIGURE 44.—System of timbering used at the Elizabeth Lake Tunnel.

full-size tunnel, and as one end of them was beveled to carry pieces of the permanent arch, the beveled ends were placed next the floor in the heading, whereas their squared ends supported the temporary caps, which likewise consisted of timbers already cut to form and later used as permanent posts. In this way there was no handling of heavy timbers not intended for permanent use. Just before blasting, the false timbers were carefully braced and wedged to the roof as tightly as possible, as shown in figure 44, *b*.

As soon as possible after the blasting, the timbermen went back to the heading to shore up the new roof temporarily from the top of the rock pile. For this purpose two horizontal timbers, supported from the broken rock close to the side walls and having transverse timbers and blocking resting upon them, as shown in figure 44, *c*, were placed by the timber crew, an operation that interfered little with the work of the muckers shoveling back from the face to allow the placing of the drills. The débris was next removed down to solid bottom to permit the setting of false posts, which, when capped, then carried the weight of the roof.

Timbering the tunnel during the enlargement to full size was not a difficult operation. Starting at the last permanent set and proceeding toward the face, new permanent posts (shown in figure 44, *a*) were placed in position as fast as the section was widened by picking down the side walls. Transverse spreader timbers, shown in view of section CD, figure 44, were then placed between these posts with their bottoms 14 inches below the joint and resting on timber ends spiked to the posts. Across these spreaders were laid two tiers of 2-inch plank, forming a floor 4 inches thick. The floor was some 2 or 3 inches below the bottoms of the caps resting on the false posts, so that it was easily laid while the false sets continued to hold the roof. Working from the end of this floor, the wedges and blocking transmitting the roof weight to the false set were next carefully knocked out and the shattered roof picked down on the floor, from which it was later shoveled into cars. By placing the permanent posts a foot or so in advance of the false posts, as in figure 44, *a*, the arch timbers of the permanent sets could be put in position as soon as the roof had been sufficiently removed. Lagging and wedging quickly followed, so that the roof was supported by the permanent sets shortly after the removal of the false blocking.

Although lining a mining tunnel with concrete is not, strictly speaking, a type of timbering, both have the same function, that of supporting the roof and walls. In this work the concrete is usually placed in the openings between the timbers and for a few inches in front of them, a disposition that, if the sets are not spaced too closely together, is generally sufficient even though later decay of the wood results in a corresponding weak spot in the lining. This defect can be avoided, however, by the use of posts and caps made of reinforced concrete in place of wood, a practice that has been recently introduced and is finding great favor wherever its added expense is warranted. The concrete posts and caps are made outside of the tunnel in a mold that gives them the identical form of the wooden pieces they displace, and by proper reinforcement they can be made equal if not superior to timbers in strength, a strength that is practically permanent.

In water-supply tunnels a concrete lining performs the additional function of obviating eddies and friction against the otherwise irregular walls, and for this reason such tunnels are generally lined throughout, irrespective of the needs of the roof for support. On the Los Angeles Aqueduct the tunnel lining was generally at least 8 inches thick in places where the tunnel was not timbered, although an occasional rock projecting into the concrete was not removed unless it came within 4 inches of the inside finished surface of the lining. In timbered ground concrete was placed between the timbers and for a minimum distance of 4 inches in front of them. The inverted siphons of the Catskill Aqueduct were lined with concrete which was ordinarily 2 feet thick, but solid rock was permitted to project without removal to within 10 inches of the interior surface. Owing to the great hydrostatic head, in places as high as 700 feet, to which these linings were to be subjected, every piece of timber was removed before the concrete was put in place, and if it was necessary to support the roof during the time the concrete was setting, steel roof supports were designed and placed for this purpose.

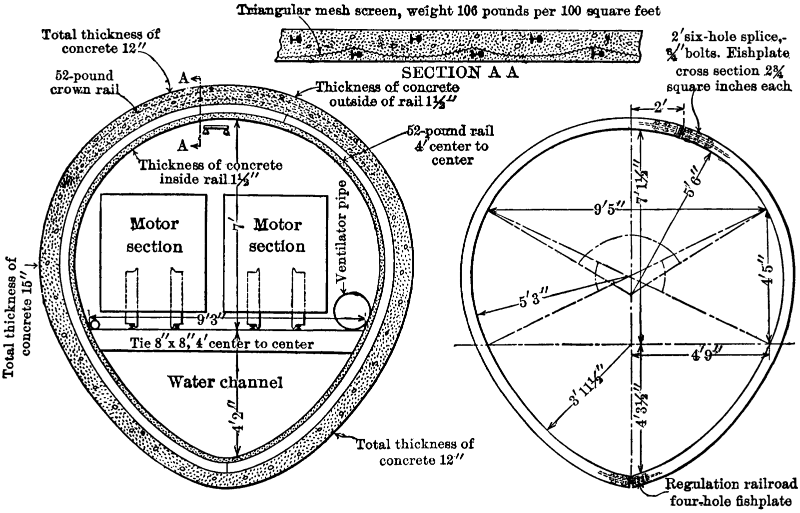

FIGURE 45.—Reinforced concrete lining at Snake Creek Tunnel.

At the Snake Creek Tunnel, where a zone of swelling ground was encountered which resisted all efforts to hold it in the ordinary way, the strongest timbers that could be obtained being crushed and broken in less than a month's time, a concrete lining reinforced with steel rails was installed. Figure 45, showing this lining, is practically self-explanatory.[a] If large volumes of water have to be carried through ground extremely difficult to hold, this design seems excellent, although its cost would be prohibitive for anything except important tunnels draining large areas of well-developed ground.

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[a] A complete description may be found in the Engineering Record for May 25, 1912, p. 565.

COSTS OF TUNNELING.

In regard to publicity, the cost of tunneling is perhaps the most neglected feature of the work. Although the past 10 or 15 years have witnessed a very considerable amount of tunnel driving and there is presumably much information relative to cost extant, and although the articles describing methods, equipment, and other features of many of these tunnels have been numerous, only very meager data regarding the cost of the work, which is a very practical means by which the efficacy of methods and equipment can be measured, have found their way into the ordinary channels of publicity—the engineering periodicals. This possibly is due in part to the prejudice entertained by some contractors and tunnel men against a publication of their cost data; or they may actually not know what the work has cost them, aside, perhaps, from the difference between their bank account at the beginning and at the end of the job; others, perhaps, are unwilling to go to the trouble (for it does involve considerable extra labor) of preparing such matter for the magazines or other publications.

In an attempt to remedy this condition somewhat, there are set forth in the following pages as complete and accurate data as could be obtained, showing the cost of various phases of tunnel work at a number of different tunnels. Although the writers have not had the advantage of auditing the books from which these figures were taken and hence can not vouch personally for the absolute accuracy of the figures, the data were in all cases procured from persons in charge or who were in a position to know what the work actually cost. Accompanying the figures is a brief list of the more important features of the tunnel, without which it is impossible to make even an approximate comparison between any two pieces of tunnel work.

GUNNISON TUNNEL.

Important details.

Location: Montrose, Colo.

Purpose: Irrigation and reclamation.

Shape of cross section: Horseshoe.

Size: 10 feet wide at the bottom, 10 feet 6 inches wide at the spring line, 10 feet high at the spring line, 12 feet 4 inches high at the center of the arch.

Length: 30,645 feet.

Character of rock penetrated: Chiefly metamorphosed granite with some water-bearing clay and gravel, some hard black shale, and a zone of faulted and broken rock.

Type of power: Steam.

Ventilator: Pressure blower.

Size of ventilating pipe: 17 inches.

Drills: At first, pneumatic hammer, 4 drills in the heading; afterwards, pneumatic, piston, 4 drills in the heading.

Mounting of drills: Horizontal bar for the hammer drills, vertical columns for the piston drills.

Number of holes per round: 20 to 24 in the heading (approximately one-half of the tunnel).

Average depth of round: 6 to 7 feet.

Number of drillers and helpers per shift: 4 drillers and 2 helpers.

Number of drill shifts per day: 3.

Explosive: 60 per cent gelatin dynamite, with some 40 per cent.

Number of muckers per shift: 5 to 8.

Number of mucking shifts per day: 3.

Type of haulage: Electric.

Wages: Drillers, $3.50 and $4.00; helpers, $3.00 and $3.50; muckers, $2.50 and $3.00; blacksmiths, $3.50 and $4.00; motormen, $3.00; brakemen, $2.50 and $3.00; power engineers, $4.00.

Maximum progress in any calendar month: 449 feet.

Average monthly progress: 250 feet, approximately.

Cost of driving.

	Cost per foot of tunnel.
10,019 feet driven by undercut heading and subsequent enlargement	$87.23
20,626 feet driven by top heading and bench	62.18
Average cost of excavation of entire tunnel	70.66

These costs include all labor, all materials, all repairs, all power, depreciation figured as 100 per cent on all equipment, with a proportionate charge for general (supervisory) and miscellaneous expenses of the entire reclamation project.

LARAMIE-POUDRE TUNNEL.

Important details.

Location: Home, Colo.

Purpose: Irrigation.

Cross section: Rectangular.

Size: 9½ feet wide by 7½ feet high.

Length: 11,306 feet.

Character of rock penetrated: Close-grained red and gray granite.

Type of power: Hydraulic at the east end, electric at the west.

Ventilator: Pressure blower.

Size of ventilating pipe: 14 and 15 inches.

Drills: 3 pneumatic hammer.

Mounting of drills: Horizontal bar.

Number of holes per round: 21 to 23.

Average depth of round: 10 feet at first, 7 to 8 feet later.

Number of drillers and helpers per shift: 3 drillers, 2 helpers.

Number of drill shifts per day: 3.

Explosive: 60 per cent gelatin dynamite, with some 100 per cent in the cut holes.

Number of muckers per shift: 6.

Number of mucking shifts per day: 3.

Type of haulage: Mules.

Wages: Drillers $4.50, helpers $4, muckers $3.50, blacksmiths $5, drivers $4.50, dumpmen $3.50.

Maximum progress in any calendar month: 653 feet, March, 1911.

Average monthly progress: 509 feet (for the 16 months when complete plant operated).

Special feature: Inaccessibility; the tunnel was located about 60 miles from the nearest railroad siding, and the roads were mountainous and very steep in places.

Cost of driving tunnel 11,306 feet.

	Cost per foot of tunnel.
Superintendents and foremen	$1.50
Drilling	4.47
Mucking and loading	4.92
Tramming and dumping	4.63
Track and pipe	.47
Power house	.35
Blacksmithing	.84
Repairs	.47
Bonus to workmen	1.75
Maintenance of camps, buildings, and fuel	.62
Machinery repairs	.12
Air drills and parts	1.33
Picks, shovels, and steel	.84
Explosives	4.50
Lamps and candles	.42
Oil and waste	.38
Blacksmith supplies	.53
Liability insurance	.81
Office supplies, telephone, and bookkeeping	.86
	29.81
Permanent equipment (less approximately 10 per cent salvage)	9.73
	39.54

The permanent equipment included power plant, camp buildings and furnishings, pipes, rails, etc.

LOS ANGELES AQUEDUCT.

LITTLE LAKE DIVISION, TUNNELS 1 TO 10A.

Important details.

Location: Inyo County, Cal.

Purpose: Water supply, power, and irrigation.

Cross section: See figure 3.

Size: See figure 3.

Type of power: Electric power purchased at a nominal cost per kilowatt-hour from a hydraulic plant constructed and owned by the aqueduct.

Ventilators: Pressure blowers.

Size of ventilating pipe: 12 inches.

Drills: Pneumatic hammer, usually 2 in each heading.

Mounting of drills: Horizontal bar.

Number of holes per round: Usually 14 to 16.

Average depth of round: 6 to 10 feet.

Number of drillers and helpers per shift: 2 drillers and 2 helpers.

Number of drill shifts per day: Usually 1, but sometimes 2.

Explosive: 40 per cent gelatin dynamite, with some 20 per cent and some 60 per cent. Ammonia dynamite also tried.

Number of muckers per shift: Usually 5.

Number of mucking shifts per day: Usually 1, but 2 when 2 drill shifts were employed.

Type of haulage: Tunnels 1 to 3–N, mules; tunnels 3–S, to 10A–N, electric; tunnel 10A–S, mules.

Wages: Drillers and helpers $3, muckers $2.50, blacksmiths $4, helpers $2.50, motormen $2.75, dumpmen $2.50.

Cost of driving Tunnel 1B–S for 1,341 feet.

[Driven through medium-hard granite at an average speed of 225 feet per month.[a]]

	Cost per foot of tunnel.
Excavation	$9.15
Engineering	.18
Adit proportion	.28
Permanent equipment (estimated)	2.35
Timbering (857 feet)	1.02
	12.98

In this tunnel, as in all of the tunnels of this division and of the Grapevine division, the cost of excavation includes the wages of shift foremen, drillers, helpers, muckers, motormen or mule drivers, dumpmen, blacksmiths and helpers, machinists, electricians (part), and power engineers; also the cost of powder, fuse, caps, candles, light globes, machine oil, blacksmith supplies and fuel, and machinists' supplies, and the cost of power and of repairs for power, haulage, compressor, and ventilating machinery.

"Engineering" includes the cost of giving line and grade, etc.

"Adit proportion" is a proportionate charge per foot of tunnel to defray the cost of an adit from the surface to the tunnel line.

"Permanent-equipment" costs were not segregated for each tunnel, but were compiled for the whole division, so the charge represents a proportionate charge per foot for the entire division cost, without salvage, of trolley and light lines, including freight and cost of installation; pressure air lines with freight and installation; ventilating lines with freight and installation; water lines with freight and installation; mine locomotives and cars, picks, shovels, drills and drill sharpeners, with repairs for the last four items.

Cost of driving Tunnel 2, length 1,739 feet.

[Driven through medium-hard but very wet granite at an average speed of 170 feet per month.]

	Cost per foot of tunnel.
Excavation	$8.81
Engineering	.19
Adit proportion	.34
Permanent equipment	2.35
Timbering (1,590 feet)	3.28
	14.97

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[a] The average speed given is computed on the basis of one heading per month.

Cost of driving Tunnel 2A, length 1,322 feet.

[Driven through medium-hard granite at an average speed of 150 feet per month.]

	Cost per foot of tunnel.
Excavation	$8.05
Engineering	.16
Adit proportion	.34
Permanent equipment	2.35
Timbering (1,322 feet)	2.51
	13.41

Cost of driving Tunnel 3–N for 1,148 feet.

[Driven through medium-hard granite at an average speed of 150 feet per month.]

	Cost per foot of tunnel.
Excavation	$10.10
Engineering	.23
Adit proportion	.51
Permanent equipment	2.35
Timbering (956 feet)	2.44
	15.63

Cost of driving Tunnel 3–S for 1,358 feet.

[Driven through granite of variable hardness, and containing pockets of carbon-dioxide gas, at an average speed of 155 feet per month.]

	Cost per foot of tunnel.
Excavation	$12.38
Engineering	.28
Adit proportion	.16
Permanent equipment	2.35
Timbering (1,244 feet)	3.28
	18.45

Cost of driving Tunnel 3 (3–N and 3–S), complete, 4,044 feet.

[Driven through decomposed granite of medium hardness, dissected by slips and talcose planes requiring timber where ground was wet, and also containing pockets of carbon-dioxide gas, making work difficult and requiring extra provisions for ventilation. Average speed, 140 feet per month.]

	Cost per foot of tunnel.
Excavation	$12.67
Engineering	.24
Adit proportion	.35
Permanent equipment	2.35
Timbering (3,570 feet)	2.71
	18.32

Cost of driving Tunnel 4, length 2,033 feet.

[Driven through medium hard to hard granite at an average speed of 145 feet per month.]

	Cost per foot of tunnel.
Excavation	$12.00
Engineering	.24
Adit proportion	.16
Permanent equipment	2.35
Timbering (1,705 feet)	2.16
	17.01

Cost of driving Tunnel 5, length 1,178 feet.

[Driven through medium hard to very hard granite at an average speed of 120 feet per month.]

	Cost per foot of tunnel.
Excavation	$11.10
Engineering	.21
Adit proportion	.08
Permanent equipment	2.35
Timbering (916 feet)	1.83
	15.57

Cost of driving Tunnel 7, length 3,596 feet.

[Driven through biotite granite of variable hardness at an average speed of 140 feet per month.]

	Cost per foot of tunnel.
Excavation	$13.55
Engineering	.27
Adit proportion	.13
Permanent equipment	2.35
Timbering (2,609 feet)	3.60
	19.90

Cost of driving Tunnel 8–8 for 1,334 feet.

[Driven through medium hard to hard granite at an average speed of 135 feet per month.]

	Cost per foot of tunnel.
Excavation	$12.82
Engineering	.19
Adit proportion	.18
Permanent equipment	2.35
Timbering (126 feet)	.39
	15.93

Cost of driving Tunnel 9 for 3,506 feet.

[Driven through medium hard to hard granite at an average speed of 195 feet per month.]

	Cost per foot of tunnel.
Excavation	$12.19
Engineering	.18
Adit proportion	.07
Permanent equipment	2.35
Timbering (305 feet)	.29
	15.08

Cost of driving Tunnel 10 for 5,657 feet.

[Driven through medium hard to hard granite at an average speed of 200 feet per month.]

	Cost per foot of tunnel.
Excavation	$13.50
Engineering	.19
Permanent equipment	2.35
Timbering (194 feet)	.11
	16.15

Cost of driving Tunnel 10A–N for 1,496 feet.

[Driven through medium hard to hard granite at an average speed of 165 feet per month.]

	Cost per foot of tunnel.
Excavation	$13.02
Engineering	.13
Permanent equipment	2.35
Timbering (24 feet)	.78
	16.28

Cost of driving Tunnel 10A–S for 2,200 feet.

[Driven through medium hard to hard granite at an average speed of 200 feet per month.]

	Cost per foot of tunnel.
Excavation	$12.37
Engineering	.20
Permanent equipment	2.35
Timbering (215 feet)	1.15
	16.07

GRAPE VINE DIVISION, TUNNELS 12 TO 17B.

Important details.

Location: Kern County, Cal.
Purpose: Water supply, power, and irrigation.
Cross section: See figure 3, page 14.
Size: See figure 3, page 14.
Type of power: Electric power purchased from aqueduct plant.
Ventilators: Pressure blowers.
Size of ventilation pipe: 12 inches.
Drills: Pneumatic hammer, usually 2 in each heading.
Mounting of drills: Horizontal bar.
Number of holes per round: Usually 18 to 20.
Average depth of round: 6 to 8 feet.
Number of drillers and helpers per shift: 2 drillers and 2 helpers.
Number of drill shifts per day: Usually 2.
Explosive: 40 per cent ammonia dynamite, but 60 per cent and 75 per cent gelatin dynamite were employed in hard ground.
Number of muckers per shift: 4 or 5.
Number of mucking shifts per day: Usually 2.
Type of haulage: Electric after the first 400 to 500 feet.
Wages: Drillers and helpers $3, muckers $2.50, blacksmiths $4, helpers $2.50, motormen $2.75, dumpmen $2.50.

Cost of driving Tunnel 12, length 4,900 feet.

[Driven through hard granite at an average speed of 185 feet per month.]

	Cost per foot of tunnel.
Excavation [a]	$22.10
Engineering [a]	.32
Permanent equipment [a]	2.35
Timbering (90 feet)	.08
	24.75

Cost of driving Tunnel 13 for 1,525 feet.

[Driven through hard granite at an average speed of 130 feet per month.]

	Cost per foot of tunnel.
Excavation	$20.60
Engineering	.10
Permanent equipment	2.25
Adit proportion	.37
	23.32

Cost of driving Tunnel 14, length 859 feet.

	Cost per foot of tunnel.
Excavation	$22.70
Engineering	.13
Permanent equipment	2.25
Adit proportion	.72
Timbering (22 feet)	.16
	25.96

Cost of driving Tunnel 15, length 895 feet.

	Cost per foot of tunnel.
Excavation	$23.28
Engineering	.11
Permanent equipment	2.25
Adit proportion	2.42
	28.06

Cost of driving Tunnel 16, length 2,723 feet.

[Driven through hard granite at an average speed of 145 feet per month.]

	Cost per foot of tunnel.
Excavation	$20.07
Engineering	.17
Permanent equipment	2.25
Adit proportion	.55
Timbering (18 feet)	.04
	23.08

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[a] These items include the same costs as for the Little Lake division. See p. 195.

Cost of driving Tunnel 17, length 3,024 feet.

	Cost per foot of tunnel.
Excavation	$20.47
Engineering	.21
Permanent equipment	2.25
Timbering (142 feet)	.22
	23.15

Cost of driving Tunnel 17½ for 1,345 feet.

[Driven through medium hard to hard granite at an average speed of 225 feet per month.]

	Cost per foot of tunnel.
Excavation	$19.56
Engineering	.31
Permanent equipment	2.25
	22.12

Cost of driving Tunnel 17A for 3,275 feet.

	Cost per foot of tunnel.
Excavation	$18.70
Engineering	.17
Permanent equipment	2.25
Timbering (441 feet)	1.18
	22.30

Cost of driving Tunnel 17B for 4,915 feet.

	Cost per foot of tunnel.
Excavation	$21.09
Engineering	.21
Permanent equipment	2.25
Timbering (163 feet)	1.90
	25.45

ELIZABETH DIVISION, ELIZABETH LAKE TUNNEL.

Important details.

Location: Los Angeles County, Cal.

Purpose: Water supply, power, and irrigation.

Cross section: Rectangular, with arched roof.

Size: 12 by 12 feet.

Length: 26,870 feet.

Type of power: Electric power purchased from aqueduct plant.

Ventilator: Pressure blower.

Size of ventilating pipe: 18 inches.

Drills: Pneumatic hammer, 3 in the south heading and 2 in the north.

Mounting of drills: Horizontal bar.

Number of holes per round: 25 in the south heading, 16 in the north heading.

Average depth of round: 8 to 10 feet.

Number of drillers and helpers per shift: 2 drillers and 2 helpers at the north end, 3 drillers and 3 helpers at the south end.

Number of drill shifts per day: 3.

Explosive: 40 per cent and 60 per cent gelatin dynamite.

Number of muckers per shift: 6.

Number of mucking shifts per day: 3.

Type of haulage: Electric.

Wages: Drillers and helpers $3, muckers $2.50, blacksmiths $4, helpers $2.50, motormen $2.75, dump men $2.50.

Maximum progress in any calendar month: 604 feet, April, 1910.

Average monthly progress per heading: 350 feet per month.

Cost of driving the north heading, Elizabeth Lake Tunnel.

[Driven through altered granite, requiring much timbering, 13,370 feet.]

	Cost per foot of tunnel.
Drilling and blasting	$11.25
Mucking and tramming	11.70
Engineering and superintendence	1.27
Drainage	.45
Ventilation	.22
Light and power	5.55
Timbering (13,031 feet)	8.48
Cost of auxiliary shaft	.93
Permanent equipment (full charge, no salvage; estimated)	3.70
	43.55

Cost of driving the south heading, Elizabeth Lake Tunnel.

[Driven through medium hard to hard granite, requiring but little timbering, 13,500 feet.]

	Cost per foot of tunnel.
Drilling and blasting	$14.65
Mucking and tramming	11.10
Engineering and superintendence	.86
Drainage	.17
Ventilation	.41
Light and power	4.93
Permanent equipment (without salvage; estimated)	3.70
Timbering (3,424 feet)	2.19
	38.01

LUCANIA TUNNEL.

Important details.

Location: Idaho Springs, Colo.

Purpose: Mine development and transportation.

Cross section: Square.

Size: 8 by 8 feet.

Length: 12,000 feet projected; 6,385 feet driven December 1, 1911.

Character of rock penetrated: Hard granite.

Type of power: Purchased electric current.

Ventilator: Pressure blower.

Size of ventilating pipe: 18 and 19 inches.

Drills: Pneumatic hammer, 3 in the heading.

Mounting of drills: Vertical columns.

Number of holes per round: 25.

Average depth of round: 8 to 9 feet.

Number of drillers and helpers per shift: 3 drillers and 2 helpers.

Number of drilling shifts per day: 1.

Explosive: 50 per cent gelatin dynamite.

Number of muckers per shift: 3.

Number of mucking shifts per day: 1.

Type of haulage: Horses.

Wages: Head driller $5, drillers $4, nipper $3.50, boss mucker $5, muckers $4, drivers $4, power engineers $4, blacksmith $5.

Maximum progress in any calendar month: 263 feet, September, 1911.

Average monthly progress: 125 feet per month for the first 4,800 feet, 240 feet per month for the last 1,575 feet.

Average cost of driving first 4,800 feet.

	Cost per foot of tunnel.
Labor	$8.86
Powder	7.86
Fuse and caps	.17
Candles and oil	.21
Horse feed and shoeing	.18
Power	1.64
Repairs	.14
Tunnel equipment	2.75
Surface plant	1.25
	23.06

"Tunnel equipment" includes the cost of materials and installation of the pressure air line, the ventilating line, rails, ties and fittings, and the drainage ditch. "Surface plant" includes buildings, compressor. blower, transformers, motors, and drill sharpener.

Cost of driving next 1,575 feet.

The contractor received $21.50 per foot to cover the cost of labor, powder, fuse, caps, candles, oil, horse feed and shoeing, power and repairs, and the installation of the tunnel equipment.

MARSHALL-RUSSELL TUNNEL.

Important details.

Location: Empire, Colo.

Purpose: Mine drainage, development, and transportation.

Cross section: Rectangular.

Size: 8 feet wide by 9 feet high.

Length: 11,000 feet projected; 6,700 feet driven January 1, 1913.

Character of rock penetrated: Granite and gneiss.

Type of power: Purchased electric current; also a small auxiliary hydraulic plant.

Ventilator: Fan.
Size of ventilating pipe: 12 and 13 inches.
Drills: 2, pneumatic hammer.
Mounting of drills: Vertical columns.
Number of holes per round: 18 to 20.
Average depth of round: 9 to 10 feet.
Number of drillers and helpers per shift: 2 drillers and 2 helpers.
Number of drill shifts per day: 1.
Explosive: 40 per cent gelatin dynamite; with some, 80 per cent.
Number of muckers per shift: 4.
Number of mucking shifts per day: 1.
Type of haulage: Horses.
Wages: Drillers $4, helpers $3, blacksmiths $4, helpers $3, muckers $3.25, trammers $3.75, dumpmen $3.25, power engineer $3.50, shooters $3.25.
Maximum progress for any calendar month: 187 feet, June, 1909.
Average monthly progress: 125 feet.

Cost of driving tunnel 6,700 feet.

	Cost per foot of tunnel.
Labor	$9.37
Powder, fuse, caps, and blacksmith coal	3.35
Drills, steel, and repairs (less 30 per cent salvage)	1.34
Power	1.41
Permanent equipment and general expense (less 30 per cent salvage on permanent equipment)	3.41
	18.88

MISSION TUNNEL.

Important details.

Location: Santa Barbara, Cal.
Purpose: Water supply.
Cross section: Trapezoid.
Size: 6 feet wide at the base, 4½ feet wide at the top, 7 feet high.
Length: 19,560 feet.
Character of rock penetrated: Shale, slate, and hard sandstone.
Ventilator: Pressure blower.
Size of ventilating pipe: 10 inches.
Drills: 1 pneumatic hammer.
Mounting of drills: Horizontal bar.
Number of holes per round: 12 to 14.
Average depth of round: 7 to 8 feet.
Number of drillers and helpers per shift: 1.
Number of drilling shifts per day: 3.
Explosive: 40 per cent and 60 per cent gelatin dynamite.
Number of muckers per shift: 4.
Number of mucking shifts per day: 3.
Type of haulage: Electric.
Wages: Drillers $3.50, helpers $3, muckers $2.75, blacksmiths $4, helper $3, motormen $2.75, dumpmen $2.50, power engineers $2.75.
Maximum progress in any calendar month: 414 feet, February, 1911.
Average monthly progress: 210 feet.

Cost of driving the south portal, Mission Tunnel, May, 1909, to September, 1911, 5,515 feet.

	Cost per foot of tunnel.
Administration	$1.14
Labor	9.20
Power	2.12
Explosives	1.97
Timbering (563 feet)	.30
Track and pipe	1.22
Miscellaneous supplies	2.46
Drill parts (including steel)	1.02
Bonus	.48
	19.91

"Administration" includes superintendence, office supplies, and general charges. "Miscellaneous supplies" includes candles, light globes, shovels, picks, blacksmiths' supplies and fuel, and machinists' supplies.

NEWHOUSE TUNNEL.

Important details.

Location: Idaho Springs, Colo.
Purpose: Drainage and transportation.
Cross section: Square.
Size: 8 by 8 feet.
Length: 22,000 feet.
Character of rock penetrated: Idaho Springs gneiss.
Type of power: Purchased electric current.
Ventilator: Pressure blower.
Size of ventilating pipe: 18 inches.
Drills: Pneumatic hammer.
Mounting of drills: Vertical column.
Number of holes per round: 14 to 22.
Number of drill shifts per day: 1 and 2.
Explosive: 40 per cent gelatin dynamite, with some 100 per cent in the cut holes.
Number of muckers per shift: 3.
Number of mucking shifts per day: 1 and 2.
Type of haulage: Electric.
Wages: Drillers $4 to $4.50, helpers $3.25 to $4, muckers $3.50, motormen $3.50, dumpmen $3, blacksmiths $3.50 to $4.50, helpers $3.

Cost of driving the Newhouse Tunnel

	Jan. to Aug., 1909, 2,233 feet.	Sept. to Dec., 1909, 1,098 feet.	Apr. to Aug., 1910, 693 feet.
Labor	$6.72	$6.98	$11.73
Explosives	4.15	3.52	4.57
Fuse and caps	.39	.36	.44
Transportation of materials broken	1.49	1.47	2.22
Power	1.99	2.16	2.82
Blacksmithing	1.57	2.61	2.00
Use of drills, repairs and steel	1.50	2.74	2.86
Equipment, ties, rails, pipe, etc	1.74	1.78	2.19
Sundries	.79	.80	1.85
	20.34	22.42	30.68

RAWLEY TUNNEL.

Important details.

Location: Bonanza, Colo.

Purpose: Mine drainage and development.

Cross section: Trapezoidal.

Size: 8 feet wide at the base, 7 feet wide at the top, 7 feet high.

Length: 6,235 feet.

Character of rock penetrated: Tough, hard andesite.

Type of power: Steam with wood for fuel.

Ventilator: Pressure blower.

Size of ventilating pipe: 12 and 13 inches.

Drills: 2 pneumatic hammer.

Mounting of drills: Horizontal bar.

Number of holes per round: 23 to 25.

Average depth of round: 8 to 9 feet at first, 5 to 6 feet later.

Number of drillers and helpers per shift: 2 drillers and 2 helpers.

Number of drill shifts per day: 2 at first, 3 later.

Explosive: 40 per cent and 60 per cent gelatin dynamite (in the proportion of about 2 to 1).

Number of muckers per shift: 4.

Number of mucking shifts per day: 2 and 3.

Type of haulage: Horses and mules.

Wages: Drillers $4.50, helpers $3.75, muckers $3.50, blacksmiths $4.50, drivers $3.50, power engineers $4.

Maximum progress in any calendar month: 585 feet, July, 1912.

Average monthly progress: Approximately 350 feet.

Cost of driving the tunnel 6,235 feet.[a]

	Cost per foot of tunnel.
Drilling and firing	$5.25
Mucking	2.16
Tramming	1.13
Track and pipe	.44
Miscellaneous underground expenses	1.44
Power plant	2.50
Blacksmithing	.73
Miscellaneous surface work	.83
General expenses	1.98
Permanent plant	3.24
Timbering (1,618 feet)	1.18
Boarding house, debit balance	.04
	20.98
Credit by salvage on permanent plant	1.11
	19.87

"Drilling and firing" includes labor, powder, fuse, caps, supplies, and repairs. "Mucking," "Tramming," and "Track and pipe" include labor and supplies. "Miscellaneous underground expenses" include wages of foremen, underground telephone, etc. "Power plant" includes labor, supplies, and

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[a] A more detailed statement of the cost of this tunnel may be found in an article entitled "A problem in mining, together with some data on tunnel driving," by F. M. Simmons and E. Z. Burns, Bull. Am. Inst. Min. Eng., March, 1913, p. 369.

fuel. "Blacksmithing" and "Miscellaneous surface work" include labor and supplies. "General expenses" include salaries, office supplies, telephone, etc. "Permanent plant" includes machinery and buildings, with labor of installation, steel rails, permanent supplies, and repairs. "Timbering" includes labor and supplies. The salvage of the permanent plant is approximately 50 per cent on salable articles, such as machinery, rails, cars, etc.

ROOSEVELT TUNNEL.

Important details.

Location: Cripple Creek, Colo.

Purpose: Mine drainage.

Cross section: Rectangular, with large ditch at the side.

Size: 10 feet wide by 6 feet high.

Length: 15,700 feet.

Character of rock penetrated: Pikes Peak granite, chiefly.

Type of power: Purchased electric current.

Ventilator: Pressure blower.

Size of ventilating pipe: 16 and 17 inches.

Drills: 3 pneumatic hammer.

Mounting of drills: Horizontal bar.

Number of holes per round: 24, usually.

Average depth of round: 6 to 7 feet.

Number of drillers and helpers per shift: 3 drillers, 2 helpers.

Number of drill shifts per day: 3.

Explosive: 40 per cent, 60 per cent, and some 100 per cent gelatin dynamite.

Number of muckers per shift: 4, usually.

Number of mucking shifts per day: 3.

Type of haulage: Horses and mules.

Wages: Drillers, $5; helpers, $4; muckers, $3.50; power engineer, $4; blacksmith, $5; helper, $3.50; dump man, $3.50; drivers, inside, $5, outside, $4.

Maximum progress in any calendar month: 435 feet, portal heading, January, 1909.

Average monthly progress: Portal heading, 300 feet; shaft headings, 270 feet; all headings, 285 feet.

Cost of driving tunnel.

Total cost of portal work	$111, 980. 06
Contractor's percentage	11, 404. 88
Cost of shaft headings	262, 126. 55
Total cost of tunnel	386, 421. 49
Number of feet driven	14, 167
Average cost per foot	$27. 27

Cost of driving the portal heading.

	Feet.	Cost per foot.
1908:		
February and March	514	$22. 690
April	262	30. 970
May	268	26. 760
June	187	35. 010
July	203	29. 600
August	300	21. 760

1908—Continued.	Feet.	Cost per foot.
September	351	$19.600
October	287	23.000
November	360	21.120
December	334	18.350
1909:		
January	435	16.410
February	290	22.206
March	340	21.745
April	316	21.266
May	402	18.762
June (8 days)	62	40.600

Cost of driving shaft headings.

1908:		
October (2 headings)	49	$105.52
November (2 headings)	141	44.38
December (2 headings)	177	40.11
1909:		
January (2 headings)	261	24.06
February (2 headings)	601	23.70
March (2 headings)	639	26.256
April (2 headings)	670	25.02
May (2 headings)	552	28.34
June (2 headings)	498	27.375
July (1 heading)	319	32.871
August (1 heading)	410	27.747
September (1 heading)	355	32.40
October (1 heading)	380	28.178
November (1 heading)	298	34.20
December (1 heading)	251	35.153
1910:		
January (1 heading)	282	28.82
February (1 heading)	259	30.636
March (1 heading)	344	27.62
April (1 heading)	376	25.313
May (1 heading)	393	24.856
June (1 heading)	373	26.616
July (1 heading)	350	25.247
August (1 heading)	372	25.029
September (1 heading)	342	28.45
October (1 heading)	372	27.361
November (1 heading)	192	27.786

Typical distribution of expenses, portal heading, July, 1908, 203 feet.

	Cost per foot of tunnel.
Machinery and repairs	$0.61
Air drills and parts	.99
Picks, shovels, and steel	1.90
Ditch men	1.09
Explosives	6.90
Candles	.36

	Cost per foot of tunnel.
Oil and waste	$0.09
Electric power	2.06
Blacksmith supplies	.09
General expense	.16
Liability insurance	.17
Lumber, ties, and wedges	.01
Horses and feed	.01
Compressor men	1.79
Drillers and helpers	4.21
Blacksmiths and helpers	3.43
Muckers and drivers	4.11
Foremen	1.50
Bookkeeper	.12
	29.60

Typical distribution of expenses, shaft heading, February, 1910, 259 feet.

	Cost per foot of tunnel.
Maintenance of buildings, tents, etc	$0.096
Machinery and repairs	1.158
Air drills and parts	1.930
Shovels, picks, and steel	1.930
Pipe and fittings	.193
Ditch men	1.480
Explosives	5.032
Lamps and candles	.217
Oil and waste	.252
Electric power	2.440
Blacksmith supplies	.150
Liability insurance	.213
General expense	.342
Lumber, ties, and wedges	.119
Horses and feed	.324
Machine men and helpers	4.050
Muckers	3.065
Blacksmiths and helpers	1.362
Engineers	1.300
Pipe and track men	.675
Drivers and dump men	2.355
Foremen	1.753
Mine telephone	.008
Bookkeeper	.193
	30.636

STILWELL TUNNEL.

Important details.

Location: Telluride, Colo.
Purpose: Mine drainage and development.
Cross section: Square, with ditch at side.
Size: 7 by 7 feet.

Length: 2,950 feet.

Character of rock penetrated: Conglomerate and andesite.

Type of power: Purchased electric current.

Ventilator: Fan.

Size of ventilating pipe: 10 inches.

Drills: Started with electric drills, finished with pneumatic piston drills, using 2 in the heading.

Mounting of drills: Vertical columns.

Number of holes per round: 16.

Average depth of round: 6 to 6½ feet.

Number of drillers and helpers per shift: 2 drillers and 2 helpers.

Number of drill shifts per day: 1.

Explosive: 40 per cent gelatin dynamite.

Number of muckers per shift: 3.

Number of mucking shifts per day: 1.

Type of haulage: Horses.

Wages: Drillers $4.50, helpers $4, muckers and trammers $3.50, blacksmith $4.50.

Maximum progress in any calendar month: 170 feet, August, 1904.

Average monthly progress: 150 feet (last 10 months).

Cost of driving the tunnel.

	Feet.	Cost per foot of tunnel.
1901	12	$23.88
1901–2	490	22.98
1902–3	377	27.94
1903–4	702	21.69
1904–5	1,077	21.19
1905	292	30.37
Average for	2,950	23.38

These costs include all labor, supplies, repairs, powder, fuse, caps, candles, tools, lubricants, and general expenses, and the total value of the electric-drill plant with which the tunnel was started, and the total value of the air-drill plant which succeeded it, together with tunnel buildings, pipe, rails, and the ventilator, with no credit for salvage on any of this permanent equipment.

The fiscal year dated from September 30.

The tunnel was driven in 1901–3 with electric drills, and the high cost for 1905 -- 292 30.37

STRAWBERRY TUNNEL.

Important details.

Location: Utah and Wasatch Counties, Utah.

Purpose: Irrigation and reclamation.

Cross section: Straight bottom and walls, with arched roof.

Size: 8 feet wide by 9½ feet high.

Length: 19,100 feet.

Character of rock penetrated: Limestone with interbedded sandstone, and sandstone with interbedded shale.

Type of power: Electric power generated in a hydraulic plant operated in connection with the tunnel. Distance of transmission from west portal to power house approximately 23 miles.

Ventilator: Pressure blower.

Size of ventilating pipe: 14 inches.

Drills: Piston pneumatic, usually 2 in the heading.

Mounting of drills: Vertical columns.

Number of holes per round: 16 to 18.

Number of drillers and helpers per shift: 2 drillers and 2 helpers.

Number of drill shifts per day: 3.

Explosive: 40 per cent gelatin dynamite.

Number of muckers per shift: 6.

Number of mucking shifts per day: 3.

Type of haulage: Electric after first 2,000 feet.

Wages: Drillers $3.50, helpers $3.25, muckers $2.75, motormen $3.25, brakemen $2.75, blacksmiths $4, helpers $2.75.

Maximum progress in any calendar month: 500 feet, November, 1910.

Average monthly progress: 320 feet per heading.

Cost of driving the tunnel.

	Feet.	Cost per foot of tunnel.
West heading:		
Previous to 1909	1,613	$60.05
During 1909	3,892	33.58
During 1910	5,021	30.56
During 1911	3,491	41.52
January to July, 1912	2,382	36.79
East heading, October, 1911, to July, 1912	2,682	33.04
Average for	19,081	36.78

Detailed cost of driving the west heading for the year 1909, 3,892 feet.

	Cost per foot of tunnel.	
Labor:		
Engineering	$0.49	
Superintendence	.73	
Shift bosses	1.22	
Timekeepers	.36	
Drillmen and helpers	3.15	
Miners (for handwork, trimming, etc.)	.23	
Muckers	2.96	
Track and dump men	.74	
Mule drivers	.39	
Motormen and brakemen	.44	
Electricians and blower men	.07	
Disabled employees	.19	
Timbermen	.22	
Miscellaneous	.40	
		$11.59
Materials:		
Powder, fuse, caps, etc	3.08	
Lumber	.29	
Oils, candles, etc	.22	
Ventilating pipe	.64	
Track, including ties	.68	
Pressure air pipe	.40	
Drill repair parts (including hose)	.18	
Miscellaneous	.19	
		5.68

	Cost per foot of tunnel.	
Repairs:		
Machine-shop expense (including labor and supplies)	$0.93	
Blacksmith-shop expense (including labor and supplies)	1.22	
		$2.15
Power (all purposes)		7.65
Depreciation:		
Haulage equipment	.09	
General equipment	1.00	
		1.09
General expense	3.96	
Camp expense	1.21	
Corral expense	.25	
		5.42
Total		33.58

"General expense" includes a proportionate charge for the expenses of the Provo office, such as salaries, stationery, telephone, and supplies; also a proportionate charge for the expenses of the Washington, the Chicago, and the supervising engineer's offices. The Provo office covers approximately 68 per cent of this charge, the Washington office 23 per cent, the Chicago office 2 per cent, and the supervising engineer's office 7 per cent.

Detailed cost of driving the west heading for the year 1910, 5,021 feet.

	Cost per foot of tunnel.	
Labor:		
Engineering	$0.61	
Superintendence	.60	
Shift bosses	1.25	
Timekeepers	.22	
Drillmen and helpers	2.85	
Miners	.28	
Muckers	2.93	
Track and dump men	.71	
Motormen and brakemen	1.49	
Electricians and blower men	.13	
Disabled employees	.16	
Timbermen	.28	
Miscellaneous	.07	
		$11.58
Materials:		
Powder, fuse, caps, etc	3.52	
Lumber	.22	
Oils, candles, etc	.20	
Ventilating pipe	.65	
Track, including ties	.74	
Pressure air pipe	.28	
Drill repair parts (including hose)	.24	
Miscellaneous	.07	
		5.92
Repairs:		
Machine-shop expense (including labor and supplies)	.90	
Blacksmith-shop expense (including labor and supplies)	1.23	
		2.13
Power (all purposes)		5.70

	Cost per foot of tunnel.	
Depreciation:		
Haulage equipment	$0.20	
General equipment	1.00	
		$1.20
General expense	3.32	
Camp expense	.63	
Corral expense	.08	
		4.03
Total		30.56

Detailed cost of driving the west heading, for the year 1911, 3,419 feet.

	Cost per foot of tunnel.	
Labor:		
Engineering	$0.45	
Superintendence	.82	
Shift bosses	1.65	
Timekeepers	.38	
Drillmen and helpers	4.07	
Miners	.37	
Muckers	5.13	
Track and dump men	2.00	
Motormen and brakemen	1.87	
Electricians and blowermen	.08	
Disabled employees	.48	
Timbermen	1.72	
Miscellaneous	.05	
		$19.07
Materials:		
Powder, fuse, caps, etc	2.61	
Lumber	.80	
Oils, candles, etc	.43	
Ventilating pipe	.77	
Track, including ties	1.52	
Pressure air pipe	.36	
Drill repair parts (including hose)	.34	
Miscellaneous	.25	
		7.08
Repairs:		
Machine-shop expense (including labor and supplies)	2.16	
Blacksmith-shop expense (including labor and supplies)	1.54	
		3.70
Power (all purposes)		5.20
Depreciation:		
Haulage equipment	1.85	
General equipment	.50	
		2.35
General expense	3.00	
Camp expense	1.10	
Corral expense	.02	
		4.12
Total		41.52

Detailed cost of driving the west heading, January to July, 1912, 2,382 feet.

	Cost per foot of tunnel.	
Labor:		
Engineering	$0.36	
Superintendence	.56	
Shift bosses	1.08	
Timekeepers	.26	
Drillmen and helpers	3.08	
Miners	.43	
Muckers	4.95	
Track and dump men	1.55	
Motormen and brakemen	1.33	
Electricians and blowermen	.18	
Disabled employees	.48	
Timbermen	2.59	
		$16.85
Materials:		
Powder, fuse, caps, etc	2.72	
Lumber	2.13	
Oils, candles, etc	.32	
Ventilating pipe	.70	
Track, including ties	1.51	
Pressure air pipe	.30	
Drill repair parts (including hose)	.32	
Miscellaneous	.39	
		8.39
Repairs:		
Machine shop (including labor and supplies)	1.39	
Blacksmith shop (including labor and supplies)	1.02	
		2.41
Power (all purposes)		3.75
Depreciation:		
Haulage equipment	2.20	
General equipment	.50	
		2.70
General expense	1.90	
Camp expense	.79	
		2.69
Total		36.79

Detailed cost of driving the east heading, October, 1911, to July, 1912, 2,682 feet.

	Cost per foot of tunnel.	
Labor:		
Engineering	$0.49	
Superintendence	.77	
Shift bosses	1.36	
Timekeepers	.31	
Drillmen and helpers	3.62	
Muckers	4.03	
Track and dump men	2.00	
Mule drivers	.89	
Timbermen	1.80	
Electricians and blowermen	.30	
Disabled employees	.09	
Miscellaneous	.21	
		$15.87

	Cost per foot of tunnel.	
Materials:		
Powder, fuse, caps, etc	$2.67	
Lumber	.93	
Oils, candles, etc	.36	
Ventilating pipe	.45	
Track, including ties	.56	
Pressure air pipe	.12	
Drill repair parts (including hose)	.38	
Miscellaneous	.21	
		$5.68
Repairs:		
Machine shop expenses (labor and supplies)	.62	
Blacksmith shop expenses (labor and supplies)	.65	
		1.27
Power (all purposes)		3.21
Depreciation:		
Haulage equipment	.47	
General equipment	1.02	
		1.49
General expenses	1.86	
Camp expenses	1.35	
Corral expenses	.95	
		4.16
Pumping (labor and material)		1.36
Total		33.04

THE HISTORY OF TUNNELING.

UNDERGROUND PASSAGEWAYS OF ANCIENT RACES.

The art of excavating underground passageways has been known to mankind for many centuries. The ancient Egyptians and Hindus employed it in the creation of many wonderful subterranean temples and sepulchers in hard rock, and similar monuments are found in the works of the Hebrews, Greeks, Etruscans, Romans, Aztecs, and Peruvians—in fact, of all ancient civilized peoples.

EGYPTIAN TEMPLES AND TOMBS.

It is not surprising that the Egyptians, with their wonderful knowedge of quarrying as well as of many other useful arts, should have been versed in methods of underground rock excavation. Remains of their work, some of which dates back to 1500 B. C., may be found in the grottos of Samoun, the tombs near Thebes and Memphis, the catacombs of Alexandria, and the temples of Ipsamboul. A gigantic tomb has been found at Abydos, which was cut in the solid rock during the XIIth dynasty by Senwosri III; also Rameses II, who is perhaps the best remembered personage of these ancient times, constructed, either because of vanity or the great length of his reign, many rock-cut temples, the grandest of which is probably that of Abu Simbel.

The work was performed with hand tools and the labor necessary to have fashioned monuments of such magnitude and grandeur must have been stupendous. For cutting granite and other hard rock, the workmen used saws of copper which were either fed with emery powder or were set with teeth of that abrasive. A similar method was employed as early as the IVth dynasty for circular holes, which were drilled by a tube having fixed teeth or fed with emery powder. For removing rock in a quarry or in a tunnel, grooves varying in width from 4 to 20 inches were made on four sides of a block, which was then broken out by the swelling action produced by soaking with water a number of wooden wedges driven into these grooves.

HINDU CAVES AND TEMPLES.

The excavations in India probably number at least 1,000, the majority of which are of Buddhist origin. They are usually of two types—chapels and monasteries. The chapels consist of a nave with a vaulted roof, separated from the side aisles by columns, and containing a small chapel at the inner circular end. The monasteries consist of a hall surrounded by a number of cells for the residence of monks and ascetics.

Most of the Indian excavations are of much later date than those in Egypt. The earliest, the Sudama or Nigope cave, was constructed probably about 260 B. C., the Lomas Rishi was built about 200 B. C., and those of Nassick about 129 B. C. These earlier caves imitated very closely contemporaneous timber-roofed temples, and for this reason the columns all slope inward, copying with great fidelity of detail the rafter supports of the wooden temples. In the Karli caves (about 78 B. C.) this feature is absent; the columns of the nave are quite plumb and the perfection of architecture and ornamentation is unsurpassed by any of the later Hindu rock temples. The galleries and rooms of the caves of Ellora contain a total of nearly 5 miles of subterranean work. Although the builders may possibly have known of gunpowder, it was not used in the construction of these tunnels, which, like all the preceding works, were accomplished laboriously with hand tools and probably by slave labor. The caves of Salsette belong to the sixth century A. D., whereas those at Elephanta were constructed about 800, and the Gwalior temples were excavated still later, during the fifteenth century.

GRECIAN TUNNELS AND MINES.

Modern archæological investigation indicates that tunneling was possibly known to the Minyæ, an ancient Grecian people dating back beyond 2000 B. C., whose cycle of myths includes, among others, that of the Argonautic expedition. A series of shafts, 16 in all, are to be seen near Lake Kopais, in Bœotia, which are supposed to have been constructed by these peoples for the ventilation of an ancient drainage tunnel. The shafts are 200 to 1,000 feet apart, 6 to 9 feet wide, and have a maximum depth of 100 feet. The tunnel was probably the enlargement of a natural watercourse, such as are commonly found in similar calcareous rocks. Krates, of Chalkis, a mining engineer who lived in the time of Alexander the Great, is credited historically with an attempt to drain this lake by utilizing and enlarging natural watercourses.

Although the exact date of the introduction of mining into Attica, probably from the Orient, is unknown, it seems to have been subsequent to the time of Solon (about 600 B. C.). By 489 it is certain that the silver mines of Laurium were yielding a highly satisfactory return, and at the instigation of Themistocles the net profits from them were applied by the Athenians to the construction of a fleet, so that these mines no doubt contributed largely to the prosperity and power of Athens. The workings, approximately 2,000 in all, consisted of shafts and galleries in which the rocks were hewn out with hand tools and brought to the surface on the backs of slaves. Air was supplied to the large underground stopes or chambers by ventilating shafts about 6 feet square and 65 to 400 feet deep.

Gold was mined in Macedonia and Thrace at least as early as the fifth century B. C., and Herodotus mentions a tunnel in the island of Samos, built in the sixth century, which was 8 by 8 feet in cross section and nearly a mile long.

AZTEC MINES.

The Aztecs were well acquainted with mining, and they obtained copper from the mountains of Zactollan; the mines of Tasco furnished silver, lead, and tin; and the extensive galleries and other traces of their labor were of great assistance to the early Spanish miners. With no knowledge of iron, although iron ore was very abundant, their best tools were made of an excellent substitute in the form of an alloy of copper and tin. With tools of this bronze they could not only carve the hardest metals, but with the aid of powdered silica they could cut the hardest minerals, such as basalt-porphyry, and even amethyst and emerald.

PERUVIAN MINES.

Although the mines of the ancient Peruvians were little more than caverns excavated in the steep sides of the mountains, nevertheless they knew of the art of tunneling, as is shown by the tunnels of their aqueducts and by the extensive tunnel that they built to drain Lake Coxamarco. They, too, had no knowledge of iron, and their tools were made of an alloy of copper and tin, which they probably discovered quite independently of the Aztecs, whom they rivaled also in the cutting of gems.

ROMAN TUNNELS.

The Romans, however, were undoubtedly the greatest tunnel builders of early history. They drove tunnels for passage, drainage, water supply, and mining, not only in Italy but wherever their conquests led them, as is evidenced both by records and by old workings left behind in the countries they dominated. One hardly needs to mention the numerous aqueduct tunnels and sewers of the ancient city of Rome, some of which are in use to-day, attesting the ability of the Romans in this branch of engineering. Remains of their work, many of them remarkably well preserved, have been found in France, Switzerland, Portugal, Spain, Algiers, and even Constantinople.

Their tunnels were of no mean size. A road tunnel near Naples, constructed, according to Strabo, about 36 B. C., was approximately 4,000 feet long, 30 feet high, and 25 feet wide. About 359 B. C. Lake Albanus, which lies about 15 miles southeast of Rome, was tapped for its supply of clear water by a tunnel over 1 mile long, 8 feet high, and 5 feet wide. Possibly the greatest Roman tunnel was driven by the Emperor Claudius to drain the overflow waters from Lake Fucinus, which is situated about 75 miles nearly due east of Rome and has no natural means of outlet. This tunnel, completed in 52 A. D., after 11 years' labor, is over 3 miles long and was designed to be 19 feet high and 9 feet wide; but it appeared to have been even larger than this when, in 1862, it was reopened to obtain valuable land beneath the lake.

These works seem all the more marvelous when one considers the primitive methods available at that time. Explosives were unknown and machinery was not then used in mining. Rock openings were usually made by chipping, by channeling and wedging as in Egypt, or by cutting large grooves around the block to be excavated, using hand tools made of iron, copper, and bronze, although it is quite possible that for certain classes of stone cutting diamonds or some similarly hard minerals were employed in conjunction with primitive tube drills and saws. These methods were often supplemented by fire setting, a method chiefly employed, however, in the large chambers or stopes and not well adapted for driving small tunnels. It consists simply of heating the rock to a very high temperature and quenching suddenly with water (or sometimes with vinegar in calcareous rocks), producing shattering and disintegration because of sudden contraction. Many writers have described the intense and fearful sufferings of men engaged in this work, usually slaves and prisoners of war, who perished by the thousands—a fact, however, of little concern to the ancient builders.

The value of Spain as a storehouse of precious metals, offsetting somewhat the influence of eastern wealth, was well appreciated by Roman leaders, and an armed force for the protection of the mines was maintained there constantly, in many cases at the cost of serious political and financial embarrassment at home. In southern Spain, where the numerous silver and copper mines contained much water, Roman tunnels are very common. They are remarkable for their small size, being usually about 5 feet in height and, where timbered, 16½ to 36 inches in width. One adit, as far as explored, has a length of 1,850 feet and a maximum depth of 183 feet, and another is 2,300 feet long and has a maximum depth of 215 feet.

As nearly as can be ascertained to-day from discoveries in them of various objects of interest, including coins, it is certain that these adits must have been driven very early in the Christian era. Toward the latter end of the period in which these particular tunnels were used by the Romans attempts were made to work the ore bodies below them by raising water from the lower stopes by means of slave-operated water wheels.

As artificial ventilation by means of blowers was at that time unknown, like most of the Roman tunnels these were ventilated by shafts which were spaced in the tunnel mentioned above at about 25-meter intervals; in order also to minimize the depth to which the shafts were sunk, the courses of the tunnels corresponded very nearly to those of the valleys or gulches above them instead of being straight, as is the usual modern practice. Like the adits, the ventilating shafts were remarkably small. Where timbered the adits were usually about 2 feet 10 inches square in the clear, and where the rock would stand without timbering they were circular and generally did not vary much from 2 feet 4 inches in diameter.

TUNNELING IN EUROPE DURING THE MIDDLE AGES.

With the fall of the Western Empire, tunnel work in Europe practically ceased for many centuries. Some excavations were made, it is true, for tombs and the crypts of monasteries, and underground passages to a secluded exit for escape in time of defeat were a necessary part of the equipment of each castle. Crude attempts at mining also were practiced in Germany. The Teutonic tribes, whose main occupation was warfare and who were barbarous and essentially nomadic at the time of the conquests of Julius Caesar, probably learned from the Romans the value of gold; later they began to search for precious metals and to pursue other peaceful occupations.

During the Middle Ages tunneling was devoted almost exclusively to the needs of war and was seldom employed in constructing aqueducts or other public works. There is, however, a record of a road tunnel begun in 1450 by Anne of Lusignan. It was intended to pierce the Alps at an elevation of nearly 6,000 feet and afford better means of communication between Nice and Genoa, but was never completed. Work was subsequently resumed in 1782 by Victor Amadens III, but was finally abandoned 12 years later after a total of nearly 8,000 feet of tunnel had been constructed.

DEVELOPMENT OF THE USE OF GUNPOWDER IN TUNNELING.

Although gunpowder in Europe, according to the consensus of opinion, was probably invented early in the fourteenth century, and by the end of the sixteenth century was commonly used in military operations for gunnery and for blowing up fortifications, it was not applied directly to mining or tunnel operations during this period. Agricola's "Bergwerck Buch," the third edition of "De Re Metallica," published by Basel in 1621, a complete English translation of which has been issued, pictures Roman methods and hand work and fire setting as the usual means of mining.

In the year 1613 Martin Weigel is said to have introduced gunpowder in mining. Gatschmann at this time describes the use of wooden plugs for stemming. The plugs were later (about 1685) supplanted by clay. August Bayer (" Das gesegnete Markgrafenthum Meissen," 1732) and Henning Calvör (" Nachrichten über das Berg- und Maschinewesen am Harze, etc.") also confirm the date of 1613 as that of the invention of drilling and blasting, but Honemann and Rössler make it 15 or 20 years later. Whatever may have been the date when blasts were first fired in mines it is certain that blasting had become fairly common by 1650, for powder is mentioned as having been purchased for the Harz mines as early as 1634, drill holes are reported at Düllen which bear the date of 1637, and blasting is known to have been introduced into the Freiberg district in 1643.

The use of gunpowder gave a new impetus to mining and a large number of men became skillful in overcoming the difficulties of underground drifting, so that it is not surprising to note that an increased number of tunnels for other purposes were begun soon after. The chief of these purposes was transportation, and in the eighteenth and early part of the nineteenth centuries many tunnels were driven for canals which, aside from wagon roads, were the only highways at that time. Later the development of steam railroads and the desirability of maintaining level gradients led to the building of a still greater number of tunnels. A brief summary of the features of the more important transportation tunnels constructed abroad and at home follows.

CANAL TUNNELS.

The first modern tunnel constructed for commercial transportation was the Malpas Tunnel on the Languedoc Canal in France. It was 515 feet long, 22 feet wide, and 27 feet high, and was built between 1679 and 1681,[c] by Riquet, a French engineer. Although it showed that canals could be constructed through country before thought impassable, no more canal tunnels were driven in France until nearly a hundred years later, the Rive de Gier Tunnel (1,656 feet long) being constructed on the Givors Canal in 1770, and the Torcy Tunnel (3,970 feet long) on the Centre Canal in 1787. The Tronquoy and the Riqueval Tunnels on the St. Quentin Canal were started in 1803, and the Noireu Tunnel (approximately 39,400 feet in length) on the

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[a] The writers wish to acknowledge their especial indebtedness to Henry S. Drinker, from whose monumental work on tunneling (Tunneling, explosive compounds, and rock drills, 1878, 1025 pp.) this and other valuable information concerning the earlier history of tunnel driving has been obtained.

same canal was begun in 1822. On the Bourgoyne Canal, the St. Aignan Tunnel was started in 1824, so that by the middle of the nineteenth century nearly 20 canal tunnels, with a total length of nearly 93,500 feet, had been constructed in France.

The earliest transportation tunnel in England was the Harecastle, on the Grand Trunk Canal, which was begun in 1766 and opened for traffic in 1777. This tunnel was 8,640 feet in length, 9 feet wide, and 12 feet high. There were originally four other shorter tunnels on this canal. The Harecastle Tunnel was found to be too small to accommodate traffic, and was replaced in 1824 by a parallel tunnel which was 16 feet high and 14 feet wide, 4 feet 9 inches of the width being used for a towpath. The Sapperton Tunnel on the Thames-Medway Canal was started in 1783. It was approximately 12,500 feet long and its construction took six years. The next large canal tunnel in England was the Blisworth (9,250 feet long), on the Grand Junction Canal. It was started in 1798 and required seven years for its completion. In 1856 there were over 45 tunnels on the various English canals, aggregating some 220,000 feet in length.

The first canal tunnel in the United States was the Auburn Tunnel at the Orwisburg landing on the Schuylkill Navigation Canal. The tunnel (which was 450 feet long, 20 feet wide, and 18 feet high) was begun in 1818 and opened for traffic in 1821. The hill it pierced is composed of red shale, and the highest point was only 40 feet above the top of the tunnel. The tunnel was shortened in 1834–37 and again in 1845–46, and was finally made an open cut in 1855–56. The "Summit Level," or Lebanon Tunnel, on the Union Canal, begun in 1824 and finished in 1826, was the second canal tunnel in this country. It was originally 720 feet long, 18 feet wide, and 15 feet high, being driven through argillaceous slate at a total cost of $30,464. It was followed by the "Conemaugh" and "Grants Hill" Tunnels, on the western division of the Pennsylvania Canal (1827–30); the Pawpaw Tunnel, on the Chesapeake & Ohio Canal (1836); and two tunnels on the Sandy and Beaver Canal, Ohio (1836–38).

RAILWAY TUNNELS.

The first railroad tunnel of which the authors have record was the Terre-noire Tunnel, near St. Etienne, France, on the Roanne-Andrezieux horse railroad. This tunnel, which was begun in 1826, was 4,920 feet long, 9.8 feet wide, and 16.4 feet high. Some 14 other tunnels were built on the road from St. Etienne to Lyons between 1826 and 1833. The first tunnels on a railroad using steam locomotives were those on the Liverpool & Manchester Railway, constructed between 1826 and 1830. It was on this road that the famous trial between the "Rocket," "Novelty," and "Sans Pareil" locomotives took place in 1829. The following summary of early railroad-tunnel building in Europe is quoted from Drinker's work on tunneling:[a]

> Tunnels, of course, multiplied rapidly in England with the extension of railways, and during the 12 or 15 years following the construction of the Liverpool & Manchester line there were a large number of tunnels built throughout the Kingdom, among them being the famous Kilsby, Box, and Woodhead Tunnels. The first tunnels on a steam railway in France were those built on the St. Germain line in 1837. Subsequently the ones on the Versailles, the Gard, and the Rouen lines raised the total length of tunnels in France in 1845 to 12,833 meters (42,105 feet). The report of the Corps des Ponts et Chaussées on tunnels for 1856 shows at that date a total on French railroads of 126 tunnels, of a total length of 65,106 meters. Among the noted early French tunnels may be cited the Nerthe, Arschwiller, Rilly, La Motte, Lormont, and Alouette. In Belgium the Cumptieh Tunnel, built in 1835, on the Chemin de l'Etat, seems to have been the earliest. In Germany (Prussia and other States) the earlier lines were so located as to not require much tunnel work; and the Oberau Tunnel (1839), on the Leipzic-Dresden line, in Saxony, was the first. In Austria Rziha gives the Gumpoldskirch Tunnel as the first. A tunnel at Eriebitz (perhaps the same), on the North line, is mentioned in the Ponts et Chaussées report (above cited) as an early Austrian one. In 1856 there were some 50 tunnels in Austria, of a total length of 13,522 meters. In Italy the Naples-Castelamare line, opened in 1840, had several tunnels. In 1856 the total Italian tunnels amounted to 10,181 meters. The Bologna-Pistoja line is especially remarkable for its semisubterranean character. Among the early Swiss tunnels especially to be noted is the Hauenstein, commenced in 1853 and finished in 1858.

The first railway tunnel in the United States was constructed between 1831 and 1833 on the Allegheny Portage Railroad in Pennsylvania. The tunnel (which was driven through slate) was 901 feet long, 25 feet wide by 21 feet high, and was lined throughout with masonry 18 inches thick. It was followed by the Black Rock Tunnel (1835–37) on the Philadelphia & Reading Railroad and the Elizabethtown Tunnel (1835–38) on what is now the Pennsylvania Railroad. After this time railroad-tunnel construction became so general that by 1850 as many as 48 tunnels had been completed on American railways.

MINE TUNNELS.

Among the early European mining tunnels driven with gunpowder and hand drilling mention should be made of the Deep George and the Rothschonberger Stollen in Saxony, the Joseph II adit at Schemnitz, Hungary, and the Ernest August Stollen, which was later driven under the Deep George. Several tunnels, of which the Taillades Tunnel was the most important, were also driven in connection with the Marseilles Aqueduct during this period.

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[a] Drinker, H. S., Tunneling, explosive compounds, and rock drills, 1878, p. 19.

The Deep George Stollen [a] was driven between 1777 and 1799. The total length of the main tunnel is 34,529 feet. Its various branches aggregate 25,319 feet more, and yet this immense undertaking, driven entirely by hand, was to obtain a drainage depth of only 460 feet. It passed through graywacke for nearly the entire distance.

Work began on the Joseph II mining adit, at Schemnitz, Hungary,[b] in 1782, but owing to various interruptions the tunnel was not completed until 1878. The portal is at Wornitz, on the left bank of the River Gran, about 10 miles west of Schemnitz. The tunnel is 10.27 miles long, 9 feet 10 inches high, and 5 feet 3 inches wide, and cost $4,860,000. It is used entirely for mine drainage, and the annual saving in pumping amounts to more than $75,000.

The Rothschonberger Stollen [c] was driven to drain the mines of Freiberg, Saxony; it was begun in 1844 and completed April 12, 1877. The tunnel starts in the Triebisch Valley, at Rothschonberg, about 12 kilometers above Meissen, on the Elbe. Its length on the line planned to Halsbrucke was 42,662 feet, but as completed to a connection with the Hirmmelfahrt mine was, including branches, 95,149 feet. The depth below the Anna Stollen was 308 feet. Hand drilling and black powder were used to the end of 1875, when Burleigh drills were introduced. The work was carried on by the State. The tunnel was 9 feet square and was driven from 18 headings, yet 33 years were required for its completion, the average rate of progress in each of the headings being only about 15 feet per month.

The Ernest August Tunnel [d] was driven below the Deep George Stollen in 1851–1864. The main tunnel is about 34,218 feet long, but the entire length of the adit and its branches is 74,452 feet, all driven in rock similar to that in the George Stollen. The tunnel is 11 feet high and 5½ feet wide, and is driven on a grade of 35.6 feet to the mile. Hand drilling and black powder were used, and with 7-hour shifts, the rate of progress was 50 feet per month; 4-hour shifts increased the rate of progress to 78.7 feet per month, and by crowding the miners to the limit the progress during the last three weeks was 75 feet, or at the rate of 107 feet per month.

Some idea of the importance the early German miners attached to drainage may be gathered from the fact that this colossal enterprise gave them an increased drainage depth of only 315 feet.

The Taillados Tunnel [e] on the Marseilles Aqueduct was begun in January, 1839, and completed at the close of 1846. It was driven from 14 shafts, and in their construction so much water was en-

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[a] Drinker, H. S., op. cit., p. 351.

[b] Wochens, oesterreich, Ing. Achitek. Ver., 1886, p. 284.

[c] Raymond, R. W., The Rothschonberger Stollen: Trans. Am. Inst. Min. Eng., vol. 6, 1877–78, pp. 542–558.

[d] Drinker, H. S., Tunneling, explosive compounds, and rock drills, 1878, p. 351.

[e] See Martin, Felix, M. de Mont. Richer et le Canal de Marseille, 1878.

countered that the work of sinking was difficult and at times seemed almost impossible. It was finally necessary to place at one of the shafts a steam engine of 100 horsepower in order to remove the water, which amounted to 3,300 gallons per hour. The cost of sinking the shafts was approximately $40 per foot, and the tunnel itself cost approximately $37 per foot, or, including the cost of the shafts, $48.50 per foot. The Assassin Tunnel on the same project was somewhat less difficult to drive and cost only $25.50 per foot for 11,400 feet, whereas the Notre Dame Tunnel, which was lined with masonry for its entire length of 11,500 feet, cost $32.50 per foot.

The first large mining tunnel in the United States was begun as early as 1824. This was the "Hacklebernie" Tunnel near Mauch Chunk, Pa.; it was driven by hand, and black powder was used. When work stopped in 1827 an opening 16 feet wide by 8 feet high had penetrated 790 feet through hard conglomerate. Work was resumed in 1846, and the tunnel was extended to a length of 2,000 feet.

DEVELOPMENT OF THE USE OF ROCK DRILLS AND HIGH EXPLOSIVES IN TUNNELING.

The invention of drilling machines, which occurred almost simultaneously with the discovery of high explosives, gave another great impulse to tunnel driving. The following table gives in chronological order some of the more important events connected with these two wonderful improvements.

A short chronological history of the development of high explosives and rock drills.

1847. Sobrero discovered nitroglycerin.
1849. J. J. Couch, of Philadelphia, patented on March 29 the first percussion rock drill.
1851. J. W. Fowle, of Philadelphia, patented on March 11 the first direct-action percussion drill.
1854. Schumann invented his percussion drill at Freiberg.
1857. Schumann drills used in Freiberg mines.
1857. Sommeiller invented a rock drill for use at Mount Cenis.
1861. January 1 Sommeiller improved drills commenced work in the Mount Cenis Tunnel.
1863. Nobel first applied nitroglycerin as a blasting agent.
1865. Guncotton tried at the Hoosac Tunnel by Thomas Doane, chief engineer.
1866. Nitroglycerin tried with great success at the Hoosac Tunnel by T. P. Shaffner.
1866. Burleigh drills tried and proved to be a great success at the Hoosac Tunnel.
1867. Nobel invented dynamite.
1868. Dynamite patented in America by Nobel.

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[a] Drinker, H. S., op. cit., pp. 54–55.

The first extensive utilization of these aids was in the construction of the Mount Cenis Tunnel in Europe and the Hoosac and Sutro Tunnels in this country. The success attained with them soon led to further activity in tunneling, not only for railroads but in connection with mining, drainage, and water supply as well—an activity culminating in the immense amount of such work undertaken within the last 10 or 15 years.

THE SUTRO TUNNEL.

The idea of draining the mines of Virginia City by a deep tunnel was first broached in the spring of 1860, when Mr. Adolph Sutro began negotiations with the mines, the State, and finally with the Federal Government for contracts, concessions, etc. Actual work was first begun at the portal of the tunnel in Carson Valley, 3½ miles from Dayton, on October 19, 1869. The work was carried on by hand until September, 1872, when diamond drilling was begun and tried rather unsuccessfully. In 1874 Burleigh drills were introduced, operated by compressed air generated in a compressor made by the Société John Cockerill, of Seraing, Belgium. The tunnel was completed July 18, 1878, when the Savage vein was cut 20,000 feet from the portal and 1,922 feet below its outcrop. The tunnel inside of the timbers was 10 feet high by 14 feet wide, divided into two passageways by a central row of posts. The rate of progress varied greatly, ranging from 19 to 417 feet per month, the average monthly rate from start to finish being 192.3 feet.[a]

THE TEQUIXQUAC TUNNEL.

The Tequixquac Tunnel, which now forms the most important link in the drainage system of the Valley of Mexico, was begun during the reign of the Emperor Maxmilian. The work was stopped, however, at the fall of the Empire and was not resumed until 1885; even then the want of funds prevented any material progress until March, 1888.

This tunnel is 6¼ miles in length, driven through a mass of sand, mud, and soft calcareous sandstone. It is brick-lined throughout, the section is ovoid, with an extreme width of 13 feet 9 inches and a height of 14 feet, and the tunnel has a gradient of 1 foot in 1,388. The calculated flow of water is 450 feet per second, or 200,000 gallons per minute. At first the headings were driven in the center, but the bottom-heading system was soon adopted of necessity. The greatest completed tunnel advance in any one month was 182 feet, and the

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[a] Report of commissioners, Sutro Tunnel, 1872, 987 pp., and Drinker, H. S., Tunneling explosive compounds, and rock drills, 1878, pp. 337–350.

greatest distance that any single heading was driven in a calendar month was 656 feet.[a]

THE SHOSHONE TUNNEL.

The Shoshone Tunnel, 1906–1910, is owned by the Central Colorado Power Co. Its intake portal is on the Grand River 12 miles above Glenwood Springs. The tunnel is 12,453 feet long, 12 feet high, and 16 feet 8 inches wide, and is driven wholly through hard metamorphic granite.

Where timber supports were necessary vertical posts and a three-piece arch were employed, all of which were afterwards completely covered by concrete lining. Driving was carried on from seven cross-cut adits, as well as from both the intake and the discharge ends.

The cost of the tunnel, not including concrete lining, $927,653, was divided as follows:

Construction costs of Shoshone Tunnel per linear foot of tunnel.[b]

Test drifts	$0.45
Drilling and blasting	20.66
Trenching and grading floor	1.15
Track work	1.76
Mucking and loading	17.28
Hauling	2.88
Dumping and maintenance	2.18
Blasting supplies	8.35
Drill steel	2.91
Sharpening and repairing	4.60
Timbering, temporary and permanent	3.87
Light and wiring	1.57
Ventilating	.59
Pipe, air hose and connections	.85
Power drills	2.94
Hoists and trestles	.96
Pumping	.21
Sundries	.28
Total construction costs	74.49
Overhead costs, including surveying, management, office, etc	30.91
Total cost per linear foot	105.40

TUNNELS IN JAPAN.

Following is a list of some of the more important Japanese mining tunnels:

The Tsude adit, Ashio mine, Shimotsuke district, was driven between 1885 and 1895. It is 11 feet high, 13 feet wide, and 10,000 feet

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[a] Campbell, A. J., and Abbott, F. W., Tequixquac Tunnel, Valley of Mexico: Trans. Am. Soc. Civ. Eng., vol. 32, pp. 171–194.

[b] Data furnished by L. P. Hammond, manager.

long, and is situated on the bank of the Watarase River, at Ashio. The mine contains an aggregate of 540,846 feet of levels.

The Omodani mine, Ono district, has five levels, aggregating 58,380 feet in length, the longest being 12,110 feet, whereas the drainage adit is 10,850 feet long.

The Yoshioka mine, Kawakami district, is opened by eight levels and crosscuts, totaling 134,281 feet, the main adit being 39,193 feet in length.

The Okawamae adit, which was constructed to drain the Kusakura mine, Nugataken, has a length of 10,000 feet.

The Sosuido adit drains the Innai silver mines and is 8 feet high, 10 feet wide, and 7,800 feet long.

SIGNIFICANT DATA FROM RAILROAD TUNNELS.

Although this report deals chiefly with the construction of mining tunnels, much can be learned from the study of tunnels driven for railroads. Under ordinary conditions the rate of progress in a railroad tunnel is limited by the speed at which the advance heading can be driven, and as these headings do not differ materially from mining tunnels the rates of progress attained in them are of great interest to the miner. A railroad-tunnel heading must be driven to line and grade, like a mining tunnel, and although the maintenance of a uniform width and height is desirable, it is not absolutely necessary, railway-tunnel headings having thus a slight advantage over mining tunnels. On the other hand, the multifarious operations carried on between the heading and the portal of a railroad tunnel, even under the best possible organization, often temporarily obstruct transportation; the continuity of the work is sometimes hindered by the shooting of the benches back of the face; and even when all the holes in the benches and headings are blasted together more time is needed to clear out the smoke from so many groups of shots than from a single round in the heading, as in a mining tunnel. On the whole, in similar rock and with equally good equipment and organization there should be little, if any, difference between the speed attained in driving a mining tunnel and a railway-tunnel heading, because, although the conditions for rapid progress are not exactly identical, the opportunities are practically equal.

RATE OF PROGRESS AND COST OF RAILWAY TUNNELS.

The history of the more important railway tunnels of the world also shows forcibly the rapid increase in the rates of driving and the lessening of the cost of construction since the introduction of rock drills and high explosives. The data in the following table are representative.

Progress and cost of some famous railway tunnels.

Tunnel.	Construction period.	Length.	Duration of boring.	Average daily progress in headings.	Cost per linear foot.
		Miles.	*Months.*	*Feet.*	
Mount Cenis	1857-1870	7.97	157	4.4	$356.00
Hoosac	1858-1874	4.75	....	[a] 3.0	398.00
St. Gothard	1872-1882	9.26	88	6.2	231.00
Arlburg	1880-1883	6.2	40	13.6	182.30
Simplon	1898-1905	12.4	78	[b] 13.69	239.40
Loetschberg	1906-1911	9.3	54	[c] 14.2	211.00

[a] Average of east and west headings, 1865-1873.

[b] Allowing only for days on which drilling was carried on, advance was 17.45 feet per day.

[c] Average for last 30 months, 17.1 feet.

MOUNT CENIS TUNNEL.

The Mount Cenis Tunnel was driven through the northern spur of the Cottic Alps to afford direct connection between the French and Italian railway systems. Work was begun on August 18, 1857, and the French and Italian headings met on December 25, 1870. The length of the tunnel as completed was 42,157 feet and the cost $15,000,000, or $356 per linear foot. The greatest depth below the surface was 5,275 feet, where the rock temperature was 85° F. The Sommeiller rock drill, driven by compressed air, was first used in this tunnel January 12, 1861, or five years before the introduction of air drills into the Hoosac Tunnel in the United States. The rate of progress varied greatly with the rock encountered, the total time consumed in driving being 13 years and 1 month, or an average daily progress in each heading of 4.4 feet.[a]

HOOSAC TUNNEL.

One of the most important early tunnels driven in the United States was the Hoosac, on the line of the Troy & Greenfield Railway. The project first came under consideration in 1825, but actual work did not begin until 1858. Hand drilling was employed until October 31, 1866, when Burleigh rock drills were first introduced; two months later nitroglycerin was substituted for black powder and the net result of these two most important improvements was greatly to increase the rate of driving. Many disheartening delays and interruptions occurred, due chiefly to failure of the earlier types of drilling machines and to change of engineers and contractors, but in March, 1869, a contract was let to Shanly Bros., of Toronto, who completed the work on December 22, 1874.

The tunnel had a total length of 4¾ miles, and most of it was driven through mica schist. The maximum speed attained in a single heading was 184 feet in one month of 26 working days, and the average speed in the east and west headings for the last six months was 4.2 feet per day. The cost was $10,000,000, or $398 per linear foot.[b]

ST. GOTHARD TUNNEL.

The great undertaking of driving the St. Gothard Tunnel was made possible by a joint treaty between Germany, France, and Italy, and on May 7, 1872, a contract for the tunnel was let to M. Favre,

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[a] Drinker, H. S., Tunneling, explosive compounds, and rock drills, pp. 354–357; Vernon-Harcourt, L. F., Alpine engineering: Proc. Inst. Civ. Eng., vol. 95, pp. 249–261.

[b] Drinker, H. S., op. cit., pp. 315–337.

of Genoa, who gave a bond for $1,600,000 for the successful completion of the work within a period of eight years. The tunnel is 48,887 feet, or 9.26 miles, in length, and for the most part is through schists. After tests of a number of drills, Ferroux drills were selected for the north side and McKean for the south side. The average rate of progress in the headings was 186 feet per month. In 1880 one of the headings passed through a zone of softened feldspar which, under the weight of the superincumbent rock, squeezed into the tunnel with such force that granite walls and arches 6 feet 7 inches in thickness were required to hold it in place. The maximum rock temperature encountered was 88° F., at a point 5,575 feet below the surface. The headings met February 29, 1880, but the tunnel was not completed until 1882, nearly two years after the time called for in the original contract. The total cost was £2,327,000, or $231 per linear foot.[a]

ARLBERG TUNNEL.

The success of the Mount Cenis and St. Gothard Tunnels, coupled with the desire of the Austrian Government to have a railway route to France that would not pass through Germany or Italy, led to the construction of the Arlberg Railway, which runs from Innsbruck, in the Tyrol, to Bludenz, near the Swiss frontier, a distance of 85 miles, piercing the Arlberg Range about 20 miles from Bludenz by a tunnel over 6 miles long. In the selection of the machinery and in planning the work advantage was taken of the experience gained in the Mount Cenis and St. Gothard Tunnels. In consequence the results obtained were as much in advance of those in the St. Gothard as the operations in that tunnel had been an improvement on those employed in the Mount Cenis. The driving of the Arlberg Railway Tunnel began in July, 1880, and the headings met on November 13, 1883, the average rate of progress being nearly 2 miles a year. The greatest temperature of the rock was 64° F., at a point 2,295 feet below the surface. Ferroux percussion drills, operated by compressed air, were employed in the eastern heading and Brandt rotary drills, worked by water pressure, in the western. The Ferroux drills drove 17,355 feet and the Brandt drills 14,880 feet, a difference of 2,475 feet in favor of the former. This variation was due more to the rock in the east and west headings being different than to any difference in the efficiency of the drills themselves, as is shown by the following figures of the average daily advance of the two drills:

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[a] Drinker, H. S., op. cit., pp. 359–370; Vernon-Harcourt, L. F., Alpine engineering: Proc. Inst. Civ. Eng., vol. 95, pp. 261–268.

Average daily advance of two types of drills, Arlberg Railway Tunnel.

Year.	Ferroux.	Brandt.
	Feet.	*Feet.*
1881	13.5	9.5
1882	17.2	15.1
1883[a]	17.85	17.82

[a] Ten and one-half months.

The figures show that as the nature of the rock became similar when the faces approached each other the efficiency of the Brandt drill was practically the same as that of the Ferroux. The Brandt drill was much more cheaply operated than the Ferroux and required the use of only 7 miners in the heading, as against 12 with the Ferroux.

The total length of the tunnel was 32,235 feet and its cost was $5,877,684, or $182.30 per linear foot.[a]

SIMPLON TUNNEL.

The Simplon Tunnel consists of two parallel, single-track, railway tunnels, 56 feet from center to center, driven from Brigue, Switzerland, to Iselle, Italy, a distance of 12.4 miles.

Operations began at Brigue November 22, 1898, and at Iselle December 21, 1898. The headings met February 24, 1905, but the tunnel was not completed and ready for use until January 25, 1906. Brandt rotary hydraulic drills were employed in both headings and the average rate of heading advance was 13.69 feet per diem, although when conditions were favorable speeds of 16 feet per diem in the Italian end and 20 to 21 feet in the Swiss end were readily attained. The rock was principally gneiss, with occasional beds of slate, granite, and marble.

In hard rock the cycle of operations was as follows: Bringing up and adjusting drills, 20 minutes; drilling, 1¾ to 2½ hours; charging and firing, 15 minutes; mucking, 2 hours.

More serious difficulties were encountered in driving this tunnel than in any other tunnel yet undertaken. Swelling ground was extremely common, and in places the pressure was so great that the roof and sides could be held in place only by steel I beams, with the spaces between rammed with rapid-setting concrete. A part of the tunnel where the pressure was the greatest is said to have cost $1,620 per linear foot. Many springs were encountered, and at times 17,000 gallons of cold water per minute flowed into the tunnel. Near the center of the tunnel large springs of hot water with a

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[a] Vernon-Harcourt, L. F., Alpine engineering: Proc. Inst. Civ. Eng., vol. 95, pp. 268–271; Charton, A. P., Le chemin de fer et le tunnel de l'Arlberg: Le Génie Civil, vol. 6, 1885, pp. 3–18.

total flow of 4,330 gallons per minute were encountered. One spring alone (temperature 116° F.) flowed 1,400 gallons per minute. The high temperatures engendered threatened to prevent further advance, but by bringing both cold water and cold air into the headings the temperature was reduced enough to permit work being resumed, although it took six months to drive the last 800 feet. The rapid average rate of progress maintained in the Simplon Tunnel, in spite of the difficulties encountered, was due to superb equipment and an organization so efficient and thorough that 648 men and 29 horses at the Swiss end and 496 men and 16 horses at the Italian end were advantageously employed.

Notwithstanding the care taken in ventilation and the precautions adopted for the health and safety of the workmen, 60 men were killed during the progress of the work. The total cost of the tunnel was £3,200,000, or $239.40 per linear foot.[a]

LOETSCHBERG TUNNEL.

The Loetschberg Tunnel was driven through the Bernese Alps in Switzerland, and forms the last link in the railway system connecting the city of Berne with the village of Brigue at the north end of the Simplon Tunnel. The desirability of connecting the Bernese Oberland with the Rhone Valley was discussed as early as 1866, and the present location of the tunnel was first proposed in 1889.

The railway begins at Frutigen in the Bernese Oberland, about 32.5 miles from the north portal; 50.5 per cent of this length is on horizontal curves. There are about 12 short tunnels on the line, aggregating 16,000 feet in length, one of which is a spiral tunnel 5,460 feet long with a 985-foot radius. The main tunnel is 47,678 feet long and was first planned to be run on a tangent, but a serious cave 1.6 miles from the north portal, which killed 25 men and filled 5,900 feet of tunnel, compelled the abandonment of the original line and the adoption of a curved tunnel to pass around the immense, peaty, mud-filled fissure that the heading had tapped.

At the south end Ingersoll-Rand air drills and compressors were used, whereas at the north end Myers drills and compressors were adopted. Compressed-air locomotives running on 30-inch gage tracks were used. In each heading four to six drills were employed, each mounted on a horizontal bar on a carriage, so that mucking out after firing was necessary before drilling could be commenced in the face. For the last 30 months of driving the average rate of progress in the south heading was 15.8 feet per day and in the north end, where the driving was much easier, 18.6 feet per day. On the north side, when

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[a] Saunders, W. L., Tunnel driving in the Alps: Trans. Am. Inst. Min. Eng., vol. 42, pp. 441–446; Fox, Francis, The Simplon Tunnel, Proc. Inst. Civ. Eng., vol. 168, pp. 61–83.

the heading was in limestone, it was advanced 5,623 feet in 6 months, or an average rate of 30.8 feet per day.[a]

BUSK-IVANHOE TUNNEL.

The Busk-Ivanhoe Tunnel, on the Colorado Midland Railway between Leadville and Glenwood Springs, is 9,394 feet long, and has an altitude of 10,810 feet at Busk and 10,944 feet at Ivanhoe, making it the second highest railway tunnel in the world. It is driven almost the entire distance in metamorphic granite with some softened shear zones that gave considerable trouble in driving and timbering.[b] The tunnel cost $1,250,000, and 30 men were killed in the progress of the work.

SEVERN TUNNEL.

The Severn Tunnel (1873–1887), which is on the line of the Great Western Railway in England and passes under the estuary of the Severn River, has a length of 4.35 miles and traverses a great variety of rocks, including conglomerate, limestone, carboniferous beds, sandstone, marl, and sand. The most serious difficulty encountered in driving was the great volume of water that came, not so much from the estuary above as from a huge spring on the land side. Several ineffectual attempts were made to bulkhead this spring, and before the tunnel could be successfully driven it was necessary to erect an immense pumping plant with a capacity of 45,000 gallons per minute, but the maximum amount pumped for any considerable period did not exceed 20,000 gallons per minute.[c]

TOTLEY TUNNEL.

The Totley Tunnel on the Dore & Chinley Railway, between Sheffield and Manchester, England, is 3.53 miles long. Work was begun in 1888, and the tunnel was ready for traffic in September, 1893. It is driven almost entirely through carbonaceous black shale which contained some beds of sandstone and grit. The progress of the work was greatly impeded by heavy inrushes of water, some of which carried vast quantities of sand and silt. For a time the discharge from the Padley heading amounted to 5,000 gallons per minute. At first the water was carried out of the tunnel in 12-inch pipes, but as these proved insufficient and liable to clog with sand, the headings were closed with water-tight bulkheads and center drains carried in

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[a] Saunders, W. L., Tunnel driving in the Alps: Trans. Am. Inst. Min. Eng., vol. 42, pp. 446–469; Bonnin, R., La Nature (Paris), vol. 37, 1909, pp. 147–157.

[b] (Editorial.) Progress of work on the Busk Tunnel, Colorado: Engineering News, Aug. 25, 1892, p. 171.

[c] Vernon-Harcourt, L. F., The Severn Tunnel: Proc. Inst. Civ. Eng., vol. 121, pp. 305–308.

from the portal. This work took six weeks, during which the pressure behind one of the dams rose to 155 pounds per square inch.[a]

ASPEN TUNNEL.

The Aspen Tunnel on the Union Pacific Railway, between Cheyenne and Ogden, although only 5,900 feet in length, is interesting on account of the obstacles encountered in driving, the difficulty of holding back the swelling ground, and the fact that mechanical loading of the broken rock was successfully employed in both headings. The tunnel was driven through carbonaceous shale, containing an occasional stratum of yellow sandstone, that dips 20° to 30° east, and the course of the tunnel is a little south of west. The opening is 22 feet 6 inches high and 17 feet wide in the clear, timbered with 12 by 12 inch timbers with vertical posts capped with a 7-segment circular arch. These timber sets were spaced 2 feet apart, 1 foot apart, or closer, as the weight of the ground demanded. In a part of the tunnel walls of solid 12 by 12 inch timbers would not stand the rock pressure, and the timbers were replaced by 12-inch steel I beams, which sometimes buckled sideways before the concrete filling could be rammed in place.

Small air-driven steam shovels with buckets holding ¾ cubic yard were employed for loading cars in the headings and effected a great saving in both time and cost.[b]

ARTHUR'S PASS TUNNEL.

Arthur's Pass Tunnel, South Island, New Zealand, sometimes known as the Otiro, is on the line of the New Zealand Government Railway which connects Christchurch on the east with Greymouth on the west coast and pierces the crest of the Southern Alps for a distance of 5⅝ miles. Work began in May, 1898, and the contract called for the completion of the work in a period of five years; price $5,000,000. Tunnel haulage was at first attempted with 16,000-pound benzine locomotives, but they were discarded on account of uncertain action and the annoying fumes, and electric locomotives were substituted.[c]

MONT ROYAL TUNNEL.

The Mont Royal Tunnel, which will be 3½ miles long, is now being driven through a mountain immediately behind the city of Montreal by the Canadian Northern Railroad Co. Work was begun in July,

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[a] Rickard, Percy, Tunnels on the Dore and Chinley Railway: Proc. Inst. Civ. Eng., vol. 116, pp. 117–138.

[b] Hardesty, W. P., The construction of the Aspen (Wyoming) Tunnel on the Union Pacific Railroad: Engineering News, vol. 47, March 6, 1902.

[c] Gavin, W. H., Arthur's Pass Tunnel, New Zealand: Engineering News, vol. 67, May 9, 1912, pp. 870–875.

1912. The rock is medium hard to hard limestone, intersected by numerous trap dikes. In each heading three to four drills, mounted on a horizontal bar, are used. One-half of the machines are equipped with a water-feeding device and take air at 100 pounds pressure. The rock cuts readily and as much as 20 feet 6 inches of hole has been drilled in one hour by a 3⅝-inch machine.[a]

OTHER TUNNELS.

Data relative to other noted railway tunnels are presented below:

Data relative to other noted railroad tunnels.[b]

"Govi" Tunnel, Genoa-Ronco Railway, Italy, 27,093 feet.

Marianopoli Tunnel, on railway from Catani to Palermo, Sicily, 22,453 feet.

Pracchia Tunnel, on railway between Florence and Bologna, 9,000 feet.

Biblo Tunnel, Italy, 13,907 feet.

Nerthe Tunnel, between Marseille and Avignon, 15,153 feet.

Balisy Tunnel, on railway between Paris and Lyons, 13,448 feet.

Kaiser Wilhelm Tunnel, on the Mosselle Railway near Kochem, 13,841 feet.

Standridge Tunnel, between London and Birmington, 15,383 feet.

Graveholtz Tunnel, on line of the Bergen Railway, Norway, 3⅓ miles.

Kojak Tunnel, on line of Northwestern State Railway, India, 2½ miles.

Stampede Tunnel, on Northern Pacific Railway, 9,850 feet.

Cascade Tunnel, on the Great Northern Railroad, through the Cascade Mountains in the State of Washington, 13,413 feet.

Tennessee Pass Tunnel, Denver & Rio Grande Railroad, between Leadville and Red Cliff, length 2,572 feet, altitude 10,239 feet.

Tunnel through the crest of the Andes on railway line between Buenos Aires, Argentina, and Santiago, Chile, 15,880 feet long, altitude 10,500 feet.

The summit tunnel on the Peruvian Central Railway, 3,596 feet long, altitude 15,781 feet.

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[a] O'Rourke, D. J., The Mont Royal Tunnel record: Mine and Quarry, July–August, 1913, p. 730.

[b] Encyclopedia Americana, vol. 19.

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PUBLICATIONS ON MINE ACCIDENTS AND METHODS OF MINING.

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