# STEAM SHOVEL MINING

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*INCLUDING A*

## CONSIDERATION OF ELECTRIC SHOVELS AND OTHER POWER EXCAVATORS

### IN OPEN-PIT MINING

BY
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*Dedicated to*

POPE YEATMAN

to whom for his constant encouragement and example the author is deeply indebted.

# PREFACE

The purpose of this volume is to present in a collected and condensed form the general information covering the development and study of such mining problems as may best be solved by the application of open-pit methods involving the use of modern power-excavators. It is assumed that the reader has a general idea of mining practice.

Compared with the broad old science of mining, the application of the power-shovel is relatively new, and perhaps for that reason, information concerning its use is not so widely known. The majority of those in charge of such operations are practical men little given to writing. Many have been drawn from industrial and constructive engineering fields rather than from mining, and some are at times inclined to guard the details of successful methods which have been gained only after long experience. It is therefore the purpose here to analyze, compare and criticise such information on the subject as the writer has accumulated through reading, travel and operation of open-pit work, with the aim of making this information as helpful as possible to the profession of which the writer is happy to belong. It will be noted that emphasis is laid only on such power-shovel work as pertains to mining, rather than to include the broad field of general excavation work, though in many cases the operations are conducted in much the same way. Mining operations are however of a destructive character and much less attention need usually be paid to the aftermath of the excavations or to the final disposal of the debris. The direct object is the winning of the ore, efficiently and cheaply, not the construction of an engineering monument of utility.

The prices of steam-shovels and other machinery and supplies, given in this book, are based on 1915–1917 quotations and are believed to be more useful than present day prices, which are regarded as unstable. Firms contemplating purchases should solicit from manufacturers their latest quotations, but for comparative purposes the prices given in the text are believed to be safest.

I wish to make the fullest acknowledgment for the assistance that has been received in gathering this information. It has come from pit and shovel men, from their foremen and superintendents, from the builders of the machinery and equipment, from engineering periodicals and books, and from those engineers with whom I have had the great pleasure and advantage of association for the past twelve years. I also wish to express my gratitude and appreciation for the work done by my friend Mr. E. Coppee Thurston in reading and correcting the manuscript and proof sheets.

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## CHAPTER I

## THE POWER SHOVEL

### INTRODUCTION TO MINING OPERATIONS

**Early Open-Cast Work.**—Since the earliest time recorded, mining has been carried on from open-pits or "glory-holes." In some places the ore-bodies could be removed directly from the surface, but more often it was necessary to remove valueless material from the top before much of the ore could be won, and later, as depth was gained, valueless material had to be removed from the side slopes. Before the invention of excavating machinery, this work was slowly performed by human labor. In those early times much of the work was done by slaves, and all classes of human labor was indeed cheap in comparison with present labor. Without the assistance of perfected excavating machinery it would now be unprofitable to work many ore deposits which are to-day being profitably exploited, and many others would yield only a small part of the profit they now yield.

**Transition at Rio Tinto, Spain.**—One of the oldest and most interesting examples of open-cast work is found at Rio Tinto, Province of Andalusia, southern Spain. This district was known as Tarshish in the Bible and as Tartessus in classic days, and to it is attributed the silver and much of the gold which formed the mainstay of the wealth of Tyre. Thus its history dates from the remotest antiquity, when it was worked by the Phoenicians, and later by the Romans.

These ore deposits are essentially great lenticular bodies of copper-bearing iron pyrites, but their upper portions were so enriched by decomposition that they yielded good returns in silver and gold, and in enriched copper ores which were smelted in little furnaces. Great heaps of old slag found about the mines testify to the large amount of ore extracted and the excellence of the metallurgy, the labor of which must have covered a long period. Part of this work was in the form of open-casts, as they are termed, and part was underground galleries.

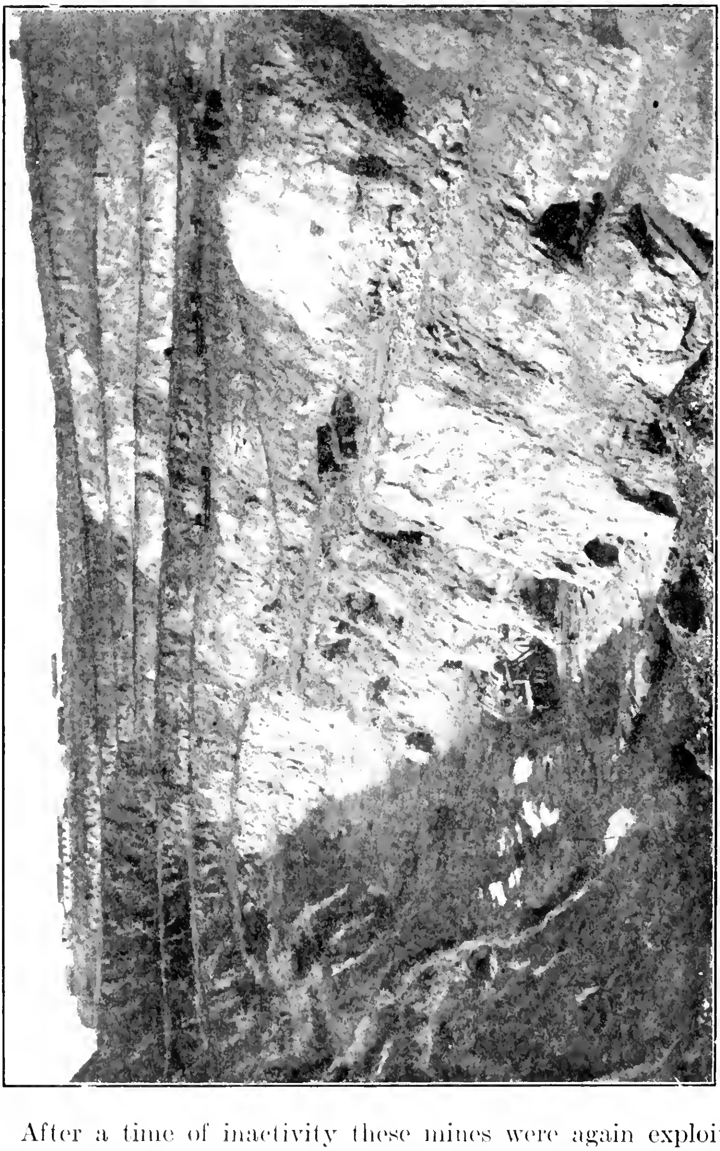


Fig. 1.—North Lode open-cast, Rio Tinto.

After a time of inactivity these mines were again exploited by the Spaniards, but they had a rather checkered career until 1872 when those at Rio Tinto were sold to London and

Bremen capitalists. The new owners continued to work the open-casts with hand-labor, but with the cost of production steadily increasing. The owners of other mines in the district employed the same method, and even until 1911, when the writer visited the district, the Zarza mine of the Tharsis Company was being so worked in the open-cast portion. Increasing depth and higher cost of labor have forced the operators to seek more economical methods with the result that the Rio Tinto Company has adopted steam-shovel methods; the Tharsis Company, because of its topography, has developed and adopted underground mining methods. A study of these mines serves well to exemplify the history of open-cast mining from the earliest to the present time. Figs. 1 and 2 illustrate the appearance in September 1911 of two of the Rio Tinto pits. The first is called the North Lode open-cast and illustrates the work done by hand-labor; the second is looking westerly into the Dionisio open-cast and shows the work being done with steam-shovels.

**Early Application to Mining.**—The steam-shovel has been in limited use since about 1865 and in general use since about 1884. At first it was employed in making railroad cuts and in excavating many classes of material for different constructive purposes. It is stated[1] that some hand stripping was conducted in the anthracite regions of Pennsylvania as early as 1864 and more extensively in 1874, but not until 1887 was the first steam-shovel introduced for this purpose. This shovel was one of the early types of Oswego shovel weighing 30 to 35 tons. Since then shovels have come rapidly into use for stripping anthracite coal veins. Not until 1892 does it appear that steam-shovels were introduced for mining ore, but in that year a steam-shovel was shipped to the town of Biwabik, on the Mesabi range, to be used for excavating the overburden covering deposits of iron ore. The Bucyrus Company states that its first shipment of steam-shovels to mining companies was in April 1890, when several were shipped to the Michigan iron country. The year following, and as iron-bearing properties continued to be developed in Minnesota and Michigan, the company shipped steam-shovels with reasonable regularity to many mining companies.

The first shovel sent to the Mesabi range was hauled in by wagons and was a small machine, but the following year a twenty-seven ton shovel was put to work; from that time until

[1] Warriner, J. B.: Anthracite Stripping, T.A.I.M.E., 1916, pp. 33–60.

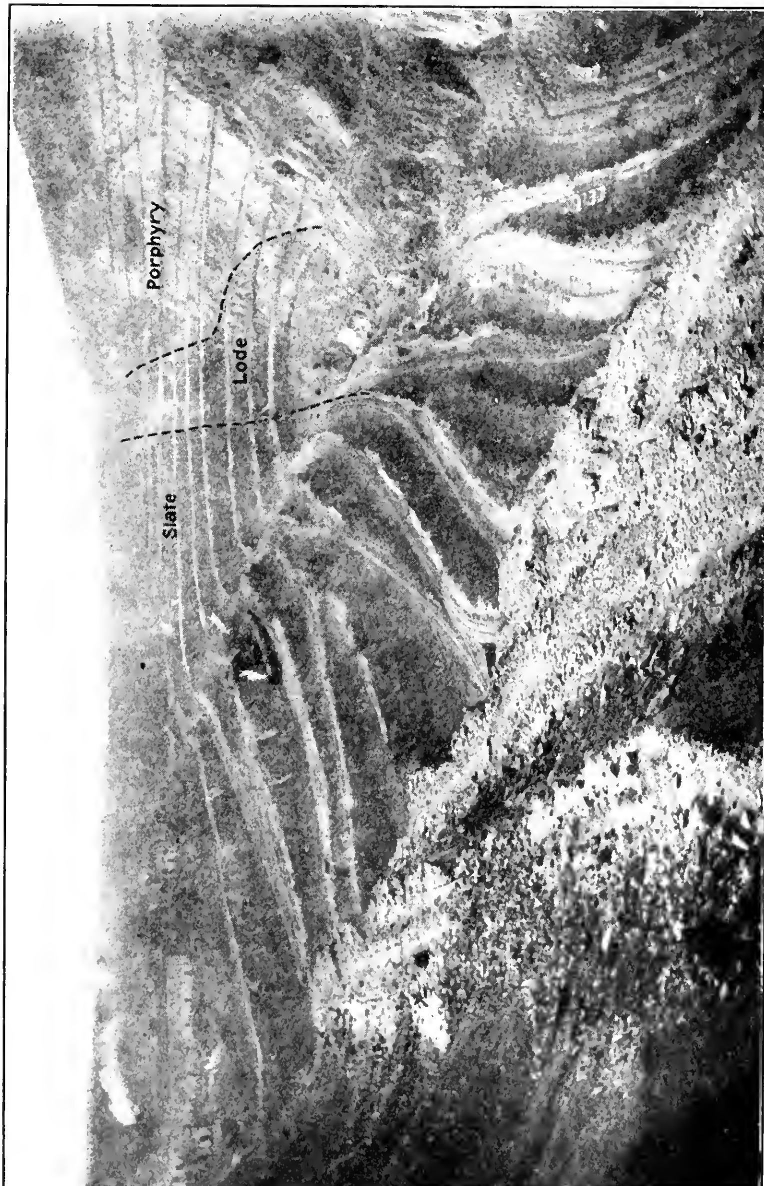



FIG. 2.—Dionisio open-cast, Rio Tinto.

the present, steam-shovels have taken a remarkable part in the uncovering and mining of iron ore.

The success achieved on the iron ranges was so marked that it was decided to introduce steam-shovel methods for mining some of the large deposits of low-grade copper ore. In 1904 the Bucyrus Company shipped some shovels to the Rio Tinto Company in Spain; in August 1906 shovels were started stripping overburden from copper-bearing porphyry ore at Bingham, Utah, now owned by the Utah Copper Company; in 1907 they were put on similar work near Ely, Nev., by the Nevada Consolidated Copper Company; and later at Santa Rita, N. Mex., by the Chino Copper Company. During the years 1914–1916 eight shovels, operated by electricity, have been shipped from the United States to Kiiruna, Sweden, for use in the magnetite mines of the Loussavaara-Kiirunavaara Aktiebolag; several have been shipped to the Belgian Congo for use in the copper mines of the Union Minière du Haut Katanga, at Kambove; several are at work on the great deposit of copper ore at Chuquicamata, Chile, owned by the Chile Copper Company; and many others have been started on mining work of a similar nature. A recent application of power shovels in mining is that of stripping and mining shallow deposits of coal in Kansas and Illinois. This was begun in 1911 near Danville, Ill., at the property of the Mission Mining Company, and has been widely extended with the perfecting of the great revolving shovels. In the anthracite fields of Pennsylvania, the shovel equipment has remained largely of the 70 and 80-ton type but the successful use of large drag-line excavators, weighing about 255 tons and operated electrically, would indicate a field for the large revolving steam-shovels such as have been installed near Steubenville, Ohio, and in the Kansas coal fields. Several large revolving shovels are also being used by the Hydro-electric Commission of Canada, digging the big power canal at Niagara Falls.

## MECHANICAL DEVELOPMENT

**Invention and Patents.**—The first steam-shovel is said to have been designed and patented by a Mr. Otis, about 1840, and a few of crude design were built about 1864 by the Otis Company of Boston. It was not until then that these excavators came into even limited use, and not until about 1884 that they began to play an important part in all classes of excavation. From that time to the present day, gradual but continuous improvements have been made in the design of the mechanical construction, boilers and engines. A history of the United States patents on them may be found in Sub-class No. 16, Excavators, Dippers, under No. 37, Excavating, which sub-class contains approximately 462 patents. Copies of these may be obtained from the United States Patent Office, Washington, D. C.

**Description of Standard Shovel.**—The power shovel is classed as an up-digging excavator, designed to excavate earth, broken or loosened rock and ores, gravel and other material. To accomplish this, the machine has been designed to imitate in a mechanical way, the motions gone through by a man shovelling. Reduced to the simplest form there are three movements; the first consists in advancing the excavating tool to contact with the material to be removed, and always acts in a vertical plane; the second (aided by the first) fills the excavator and elevates it, acting in a vertical plane; the third swings the loaded elevated excavator laterally and in a horizontal plane. These three motions are called crowding, hoisting and swinging, each is reciprocal, and each may act independently, or two or all three motions may act simultaneously. or with overlapping motion periods. For convenience in moving the machine from place to place it is usually equipped with a self-propelling mechanism which drives it backwards or forwards on its own wheels. Such movement is technically called "moving-up" or "moving-back." To discharge the filled elevated excavator, the bottom, which is hinged and latched, is tripped, permitting the material to fall through.

The general construction and arrangement of the various makes of standard power shovels is essentially the same. The following description, illustrated by Fig. 3, is intended to cover, in a general way, what may be classed as a standard steam-shovel, though many special features and variations in construction are found in the different types and makes. The principal parts of such a shovel are:

*Car Frame.*—Upon this rests the operating machinery and power equipment. It is subjected to great strain and shock, especially at the front end, and must be of the strongest construction. It is built of steel I-beam sills, running the full length of the car, and made rigid by cast-steel separators drawn tight by bolts passing entirely through the car from side to side. The deck is covered with steel plates. The front end may well consist of a heavy ribbed casting so built as to greatly strengthen the I-beams and give the deck great rigidity.

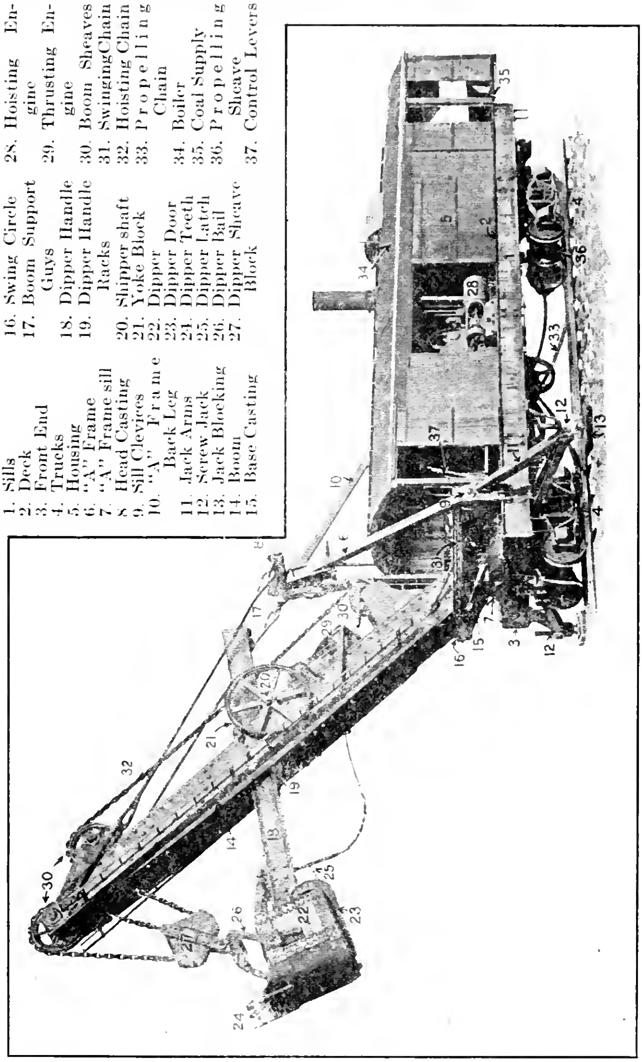



Fig. 3.—Standard or railroad type of steam shovel.

The frame is mounted on two all-steel extra heavy trucks of M. C. B. standard with diamond frames. The inside axles of both trucks, and sometimes all four axles, are keyed to sprocket wheels, chain-connected to a drive sprocket wheel for propelling the shovel.

The frame supports a housing of timber on a steel framework. The roof is often armored with steel plate to protect the crew and machinery from falling rocks.

*A-Frame.*—This is mounted on the front end of the deck, is supported by the A-Frame sill, and is guyed by the head-casting. It is built of two heavy square steel posts. The feet of the posts are drilled to take through-bolts for fastening to sill clevises. The heads are often joined to the head casting by being babbitted in the recesses provided for them. The A-frame is inclined slightly forward and is much shorter than the boom. Joined to the head casting is the A-frame back leg, a solid steel tension member which supports the A-frame and is fastened over the rear trucks of the car.

*Jack-Arms.*—These may virtually form a continuation of the A-frame posts, or, in very large shovels, may spring from the head-casting. They are stabilizers used to prevent the front end of the car from tipping sidewise as the boom and dipper swing from side to side. They are made of cast or structural steel; the upper or compression member is fastened to the A-frame supports, the lower or tension member is secured under the deck, and the lower ends of both members carry a screw-jack which is readily raised or lowered to get a bearing on the jack blocking. The wider spread obtained by the jack-arms is equivalent to widening the track gauge to the same distance, and thus makes for great stability.

*Boom.*—In front of the A-frame sill is the base casting carrying a large vertical journal, serving as a pivotal-bearing for the swing-circle and boom, and forming the axis of rotation for the third motion of the shovel. The lower end of the boom rests on the swing-circle and both revolve together through an arc of from 180° to 240°. The upper end is supported by the boom support guys, made of steel rods or bars. The boom is made of two parallel armored-wood (or all steel) members normally inclined at an angle of about 40° and so separated as to allow the free passage of the dipper handle.

A study of the figure formed by the deck, boom, A-frame, A-frame back legs and boom support guys shows it to be a diamond truss; when digging straight ahead the first two members are in compression and the last two are in tension, while the A-frame acts as a compression strut between the chords. When digging to the right or left of the shovel axis, the truss diagram changes. The A-frame leg farthest from the dipper becomes a tension member, the nearby leg remains in compression, and a compression strain is thrown across the deck. Fig. 4[1] gives a graphic idea of this, it being a strain diagram illustrating the strains in a 60-ton shovel.

*Dipper Handle.*—The dipper handle, or stick, is also an armored wood member, usually made up of two parallel parts, which plays between the two boom members. To the lower end is attached the dipper, and on its lower face are fastened two parallel manganese-steel racks by which it is run in and out by the manganese-steel pinions on the shipper shaft. The dipper handle

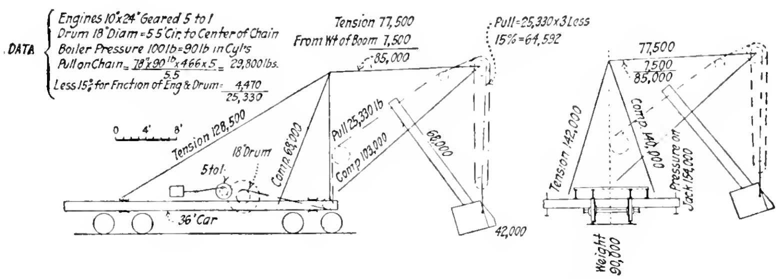



FIG. 4.—Strain diagram of standard 60-ton steam shovel.

is always held in contact with these pinions by means of the yoke block.

*Dipper.*—The dipper is scoop shaped, with a digging lower edge and an ingeniously hinged door for a bottom. It is subjected to the severest wear and so the front, or even the whole dipper, is often made of manganese-steel. The lip is either in the form of a cutting edge for soft material, or else is provided with four heavy manganese-steel teeth, or points, for hard digging. The bottom is hinged with special hinges fastened to the lower part of the back, and is closed by a spring lug-latch on the front side. It is opened by the craneman pulling on a light line operating a lever to which the latch is attached, and thus dumping the load where desired. The capacity of the dipper ranges from one-half to eight cubic yards, and depends on the size of the shovel and the character of the material being excavated. It is

[1] ROBINSON, A. W.: The Steam Shovel in Mining. P.L.S.M.I., Vol. IV, pp. 59–68.

attached to the dipper handle by steel arms and is provided with a hinged bail, which is fastened to the dipper sheave block, and by which it is hoisted. Fig. 5 illustrates the Vanderhoef all-manganese-steel 4-cubic yard dipper, which has given good results in severe work. Such a dipper will have a life of at least one million cubic yards of medium hard material.

*Engines.*—These usually consist of three separate engines of the double cylinder horizontal type. They control the three working movements, crowding, hoisting and swinging and are so

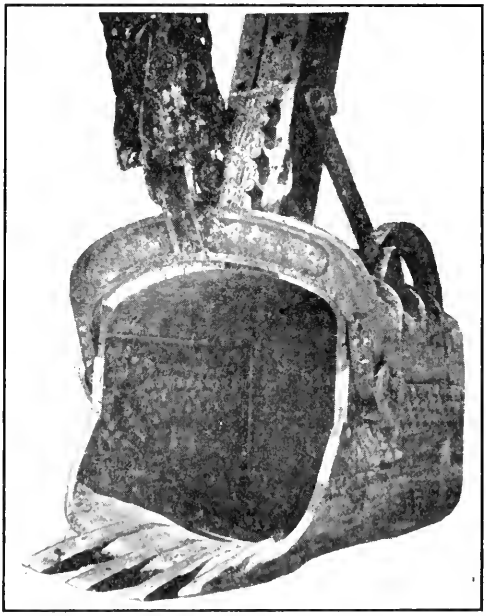


Fig. 5.—Vanderhoef all manganese-steel dipper.

termed. The crowding, or thrusting engine is mounted on the upper side of the boom, is reversible, and is controlled by the craneman. Its connecting rods rotate a crank shaft to the ends of which are keyed steel pinions; these engage with the shipper shaft gears rotating the shipper shaft. The shipper shaft is provided with manganese-steel pinions which track with the dipper handle racks and thus run the dipper in or out, giving it the first motion. The gears and pinions have square holes to fit the square section of the shipper shaft, thus doing away with troubles of keying.

The hoisting engine is of the double horizontal type, with Stephenson link motion. As it does heavy hoisting, it is the most powerful of the three. The engine shaft is fitted with counter-balanced discs and a pinion; the latter engages with the gear of the hoisting drum shaft. This drum is provided with housings at both ends, one for the hoisting band and the other for the brake band; also with bronze bushings for loose mounting on the drum shaft. To engage the hoisting drum, the hoisting friction is tightened by a small steam ram. An older method, used to drive the drum, was by positive gearing, but this was slower, less reliable under severe strain and operated less smoothly than the friction clutch.

The heavy hand-forged hoisting chain usually passes from the hoisting drum to the fair lead, or sheaves, below the swing circle, thence over the lower boom sheave, and along the boom to the upper boom sheave, thence over one of the boom point sheaves and around the dipper sheave block, returning to and around the second boom point sheave and back to the dipper sheave block becket where it is fastened. By this means the second, or hoisting motion, is accomplished.

To check the empty dipper when lowering, the band brake, operated by a foot lever, is used. Both the hoisting friction and check-bands are lined with asbestos or wood blocks.

The hoisting engine is also used to propel the shovel. This is accomplished by a propelling shaft driven by gearing from the hoisting drum-shaft. A jaw clutch engages the propelling shaft with the double pocket-sheave which is loosely mounted thereon. From the double pocket-sheave steel chains lead to sheaves on the two inner axles of the trucks, and, if desired, traction may be had from all eight wheels by providing additional sheaves and chains to connect the outer with the inner axles. By using square axles and split pocket-sheaves with square centres, the sheave halves may be bolted together so as to permit them to travel across the axle, need no keying, and yet be very secure.

The swinging engine is usually like the crowding engine except for the crank shaft and reverse valve. It drives the swinging-drum through an intermediate shaft. From the swinging-drum the two ends of the swinging chain, or cable, are passed across two sheaves, located at the front end of the car, and then around the swinging-circle and attached to the foot of the boom or to the circle. By rotating the swinging-drum in the proper direction, the swinging-circle and boom are caused to swing in the direction desired, and thus the third motion of the shovel is accomplished.

The hoisting and swinging engines are controlled by throttles operated by the shovel runner, but the crowding engine is controlled by the craneman, who also trips the dipper.

*Boiler.*—This is located over the rear trucks and generates steam for the engines. On the larger shovels the locomotive type is generally used, but on small shovels, the less economical vertical submerged tube type prevails. Attention should be given to provide boilers of ample capacity and good steaming qualities. Under-capacity causes surging in the boiler and drawing over of water into the engines. A working pressure of 125 lbs. per sq. in., is usual but they are tested to much higher pressures for safety. Lagging the boiler and pipes results in fuel economy and drier steam at the engines. Some fire boxes are equipped with shaking grates. All boilers should be provided with accurate pressure gauges, safety valves, duplex steam feed-pumps, injectors and whistles. Water tanks, holding about 1500 gallons, are placed on one or both sides of the boiler, and a rear platform carries a small amount of coal.

All of the machinery, except that exposed on the front end, is protected by the housing. The rear door swings upward to form a roof over the rear platform. A short bench, and vice and lockers are found convenient.

*Shipping.*—To ship a shovel across the country by rail the boom and dipper with handle are loaded on a flat car, and the A-frame back leg is adjusted to lower the A-frame top to clear at a height of about 14 ft. above the rail.

**Modern Standard Construction.**—Like most other mechanical inventions, the first steam-shovels were small and crude in design, and mechanically far from perfect. Constant study, usage and experiment, however, have resulted in wonderfully good design, and enormously increased size and strength with corresponding capacity. Contemporaneous with the development of the shovel has been the perfecting of various materials of construction entering into their manufacture. Among these are manganese-steels, generously used for such items as shipper-shaft pinions and racks, dipper fronts or entire dippers, dipper teeth and other parts subject to great abrasive wear; basic open-hearth cast steel and forgings instead of cast iron; rigid castings in place of structural steel for jack arms, swing circles and front ends;

and steel in place of wood for such items as the car frames and trucks. Most of the materials are carefully tested in the testing laboratory and must fulfill rigid requirements. The actual working up and balance of the material into the various members has been another contemporary improvement of great importance and is seen in the machine-cut or ground gears, the hammered steel shafting, the carefully forged chains, and the construction of boilers and other parts with a view to fuel economy. Much attention has been given to the scientific lubrication of all bearing surfaces, both as to the lubricant used and its positive but economical application. A point of great convenience in operation and of economy and accuracy in manufacture, is the interchangeability of spare parts. With the best of design many shovel parts are subject to breakage and wear so that provision must be made to replace them as quickly and as economically as possible. This is done by carrying in stock the interchangeable spare parts which are likely to be required and thus avoiding serious delays in keeping the shovels working.

In design, the required strength of the members is calculated as accurately as possible, but working conditions are so variable that actual usage in the field and under the severest conditions is considered the best criterion as a guide to the design of many parts. The engines, however, are subject to fewer unknown strains and may be designed with considerable precision. In this respect it is the aim of certain builders, such as the Marion, to design their engines to have sufficient power to work well under all normal loading, but in the event of great overload to have them stall rather than risk breaking some member of the shovel. Other builders, such as the Bucyrus, design their engines more powerfully, depending on the shovel runner to use reasonable care and judgment in operating. In the hands of competent and experienced runners, the higher powered machines are credited with somewhat greater capacity than those of lower power, but where operated by inexperienced or careless runners, the latter class will be broken down less frequently and over a given period may show an equal or greater task performed than the former. Both classes appear to be held in about equal esteem by the users. The design of the boilers provides for ample steaming capacity, fuel economy, and thorough testing for safety. For the heavier shovels the locomotive type is best but for light shovels vertical boilers are still general. The use of an outer shell, as in the Parker construction, provides an annular space for heating the feed water to a point where most of the scale-forming impurities are deposited. These solids may readily be removed from the outer ring, and the formation of scale in the boiler proper is greatly retarded, thus making for steaming efficiency and less frequent washing out. Longer usage will show to what extent these advantages apply. The use of pure water in the boilers is of great importance, regardless of the design.

Thus after more than fifty years of usage and study, the engineers and builders of modern shovels have settled on designs and materials of construction which may be called standard, meaning that they embody those characteristics which have proven best in practice. As the work thrown on them in hard digging is probably more severe than that done by any other class of excavator, and as their use is broader and the operating conditions more varied, their evolution will no doubt continue, but the present modern shovel must be considered as an eminently satisfactory machine. The general tendency in designs is constantly towards machines of larger size and capacity.

**The Railroad, or Standard Type.**—The type of machine illustrated by Fig. 3, is that generally employed for railroad and construction work, for stripping and mining large deposits of low grade ores, and for excavating canals and similar work. They are usually designed with railroad trucks for operating from a railroad track, but they may be equipped with broad traction wheels for operating in places where a track would not be desirable. The usual range in weight is from 25 tons to 135 tons and the capacity of the dipper ranges from one-half cubic yard to 8 cubic yards. The boom swings through a horizontal arc of from a little more than a half circle up to about 260°. The general description given on pages 6 to 12 is of this type.

The lighter sizes are used in brick yards, gravel pits, and in some instances in tunnels. They may be expected to load from 30 to 90 cu. yd. per hour. Those weighing about 40 tons and equipped with 1½-cu. yd. dippers are common on general con tract work, on narrow cuts, and are occasionally used in tunnels of large size. They will load from 60 to 180 cu. yd. per hour. The 70-ton shovel is used on railroad work and the heaveir classes of construction. With a 2-cu. yd. dipper it will load from

Table 1.—Dimensions of Bucyrus Standard Shovels

| | | Chain type shovels | | | | | | Wire rope shovels | |
|---|---|---|---|---|---|---|---|---|---|
| | | 110 C | 100 C | 88 C | 70 C | 68 C | 40 R or C | Class 80 | Class 45 |
| Effective pull on dipper | | 98000 lb. | 91000 lb. | 75300 lb. | 64000 lb. | 56000 lb. | 33000 lb. | 80000 lb. | 45000 lb. |
| Capacity of dipper | | $3\frac{1}{2}$–6 yd. | $3\frac{1}{2}$–5 yd. | 3–4 yd. | $2\frac{1}{2}$–3 yd. | $2\frac{1}{2}$ yd. | $1\frac{1}{2}$ yd. | 3 to 5 yd. | $2\frac{1}{2}$ yd. |
| Size of engines (double cylinder) | Main | 13″ × 16″ | $12\frac{1}{2}$″ × 16″ | 12″ × 15″ | 10″ × 14″ | 10″ × 12″ | 8″ × 8″ | 12″ × 12″ | 10″ × 10″ |
| | Swing | 9″ × 9″ | $8\frac{1}{2}$″ × 8″ | $8\frac{1}{2}$″ × 8″ | $7\frac{1}{2}$″ × 7″ | $7\frac{1}{2}$″ × 7″ | $5\frac{1}{2}$″ × 6″ | 9″ × 9″ | 7″ × 8″ |
| | Thrust | 9″ × 9″ | $8\frac{1}{2}$″ × 8″ | $8\frac{1}{2}$″ × 8″ | $7\frac{1}{2}$″ × 7″ | $7\frac{1}{2}$″ × 7″ | $5\frac{1}{2}$″ × 6″ | 9″ × 9″ | 7″ × 8″ |
| Car-body | Length | 44′ $9\frac{3}{4}$″ | 44′ 2″ | 42′ $7\frac{3}{4}$″ | 36′ $4\frac{1}{2}$″ | 37′ 7″ | 26′ $3\frac{5}{8}$″ | 42′ | 36′ |
| | Width | 10′ | 10′ | 10′ | 10′ | 9′ 3″ | 7′ | 10′ | 10′ |
| Wheel base | Traction | ....... | ....... | ....... | ....... | ....... | 16′ 2″ | ....... | ....... |
| | Truck | 35′ 6″ | 35′ $10\frac{1}{2}$″ | 33′ 10″ | 30′ $3\frac{1}{2}$″ | 29′ 4″ | 21′ $1\frac{1}{2}$″ | 36′ | 31′ |
| Width over traction wheels | | ....... | ....... | ....... | ....... | ....... | 15′ | ....... | ....... |
| Height of "A" frame | Extreme | 20′ $7\frac{3}{4}$″ | 19′ 3″ | 19′ 2″ | 19′ 0″ | 18′ 10″ | 14′ $1\frac{1}{2}$″ | 21′ 8″ | 19′ 6″ |
| | Lowered | 14′ 6″ | 14′ $6\frac{1}{2}$″ | 14′ 6″ | 14′ 6″ | 14′ 6″ | ....... | 15′ | 15′ |
| Boiler | Type | Loco. | Loco. | Loco. | Loco. | Loco. | Loco. | Loco. | Loco. |
| | Dimensions | 58″ × 18′ 3″ | 54″ × 18′ | 50″ × 18′ | 44″ × 18′ | 44″ × 17′ | 42″ × 13′ 6″ | 52″ × 21′ 4″ | 46″ × 20′ |
| Water tank capacity | | 1800 gal. | 1600 gal. | 1757 gal. | 1500 gal. | 1750 gal. | 700 gal. | 2000 gal. | 1950 gal |
| Weight in working order | | 130 tons | 113 tons | 103 tons | 87 tons | 79 tons | 48 tons | 101 tons | 73 tons |
| Shipping weight | Domestic | 116 tons | 101 tons | 91 tons | 75 tons | 68 tons | 42 tons | $88\frac{3}{4}$ tons | $64\frac{1}{2}$ tons |
| | Approximate gross weight boxed for export shipment | 121 tons | 104 tons | $93\frac{3}{4}$ tons | $77\frac{3}{4}$ tons | 70 tons | 43 tons | 96 tons | $71\frac{1}{2}$ tons |

80 to 240 cu. yd. per hour. The 80-ton size carrying a 2½-cu. yd. dipper is used for the same class of work—but has a somewhat greater capacity. The 90-ton size with a 3¼- or 3½-cu. yd. dipper will load from 120 to 350 cu. yd. per hour and is much used in large open-pit mines, and large cuts. The 100-ton shovel is used on hard and heavy work similar to the 90-ton machine, but will handle a 4-cu. yd. dipper and load more material. Shovels of great power weighing 120 to 135 tons are built and

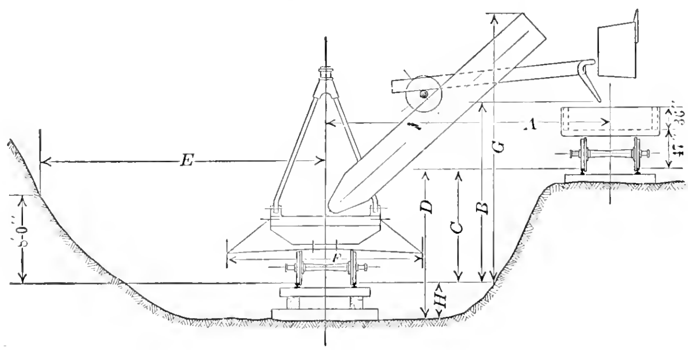



TABLE 2.—WORKING DIMENSIONS OF RAILROAD TYPE SHOVELS

| | | Chain type shovels | | | | | | Wire rope Shovels | |
|---|---|---|---|---|---|---|---|---|---|
| | | 110 C | 100 C | 88 C | 70 C | 68 C | 40 R or C | Class 80 | Class 45 |
| A | Dumping radius.. | 32' | 29' | 30' 3" | 27' | 26' 5" | 21' 6" | 33' | 27' |
| B | Height of dump... | 17' | 17' | 18' | 16' 6" | 16' | 12' | 18' 6" | 16' 6" |
| C | Depth of cut, shovel track to loading track.... | 10' | 10' | 9' 6" | 9' 6" | 9' | 5' | 11' 6" | 9' 6" |
| D | Max. depth of thorough cut..... | 16' | 15' 6" | 15' | 14' | 13' | 7' 9" | 16' 6" | 13' 6" |
| E | Digging radiusat 8' elevation..... | 33' | 33' | 33' 1½" | 30' | 28' 4" | 23' | 32' | 26' |
| F | Spread of jack screws......... | 22' | 20' | 20' | 18' 4" | 18' | 15' | 19' | 18' |
| G | Height of boom.. | 33' | 28' 9" | 28' 11½" | 27' 0½" | 26' 7" | 21' 3½" | 33' | 27' 7" |
| H | Depth of cut below rail........ | 6' | 5' 6" | 5' 6" | 4' 6" | 4' | 2' 9" | 5' | 4' |

are suitable for the heaviest service in hard digging. They have a capacity of 250 to 400 cu. yd. per hour. The capacities given here are merely approximate, as there are may factors to be considered later, that affect or control what the machines should be expected to do. Because of operating delays it will be safer to use the lower capacities in calculating what may be expected of a given shovel over a considerable period of operation.

Table 1 gives the general dimensions, and Table 2 gives the working dimensions of this type of steam-shovel. A column is also included showing these characteristics for the wire rope shovels.

**The Revolving Type.**—This type of shovel is designed to work in a full circle, turning in either direction, and operating in a manner similar to that of a steam-crane. The lighter sizes range from about 17 to 55 tons, are equipped with dippers of from ½ to 1¾ cu. yd., and will excavate from 25 to 100 cu. yd. per hour. Fig. 6 is a line drawing of one of the larger shovels built by the Marion Co. They are mounted either on trucks or traction wheels. Only one man is required to operate many of them, but a lower output must then be expected. These machines find their greatest use in street construction, basement and trench excavation, and in gravel and clay beds. Because they dig and then deliver material at any angle from the digging face, they often eliminate a loading operation. They are also useful when equipped with a clam-shell or an orange-peel bucket instead of the usual dipper. This change is made by changing booms.

A class of very large revolving shovels is made and used for stripping overburden from beds of coal, iron ore and similar deposits. These shovels (Fig. 6) have been highly developed and are the heaviest and of the greatest capacity of any built. They have a working weight of from 95 tons to 325 tons, carry dippers of from 1½ to 8 cu. yd., use booms from 45 to 90 ft. long, and dipper handles from 54 to 60 ft. long. To support them two tracks of 3 ft. gauge are generally employed, though they may be mounted on rollers, or caterpillar tractors. To equalize the frame stresses and give the flexibility of a three point suspension, a hydraulic truck equalizing device is used on each corner of the truck frame. They will build dumps as high as 65 ft., have a radius of cut at grade of from 34 ft. to 70 ft., and at 40 ft. elevation have a maximum radius of 101 ft. They will handle from 150 to 300 cu. yd. (place measure) per hour.

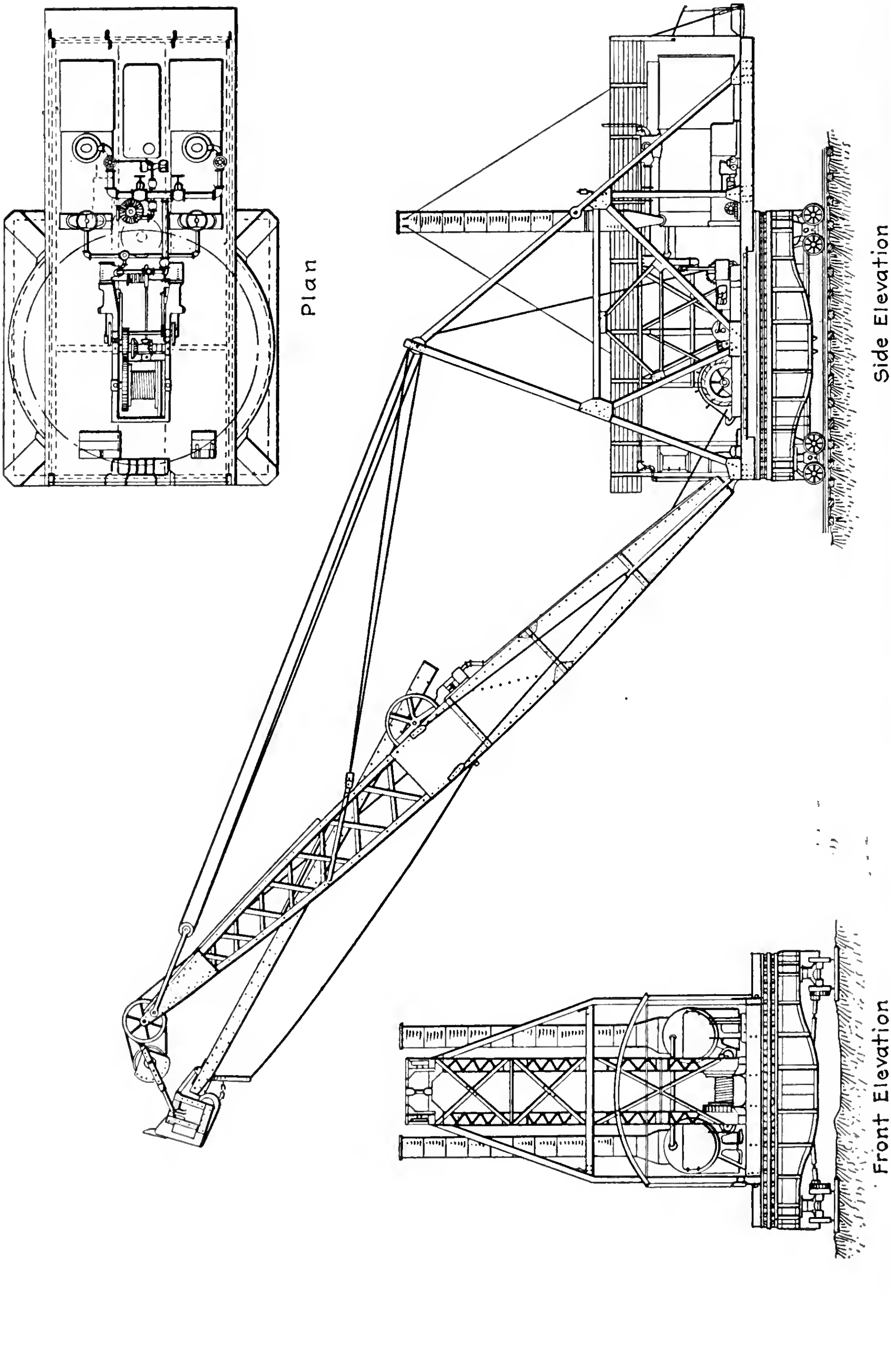



Fig. 6.—Large revolving steam shovel.

Equipped with long booms, they are able to strip wide cuts, dumping the excavated material at a sufficient height and distance to permit mining of the uncovered material without further handling of the overburden. With no loading or transporting of stripping required, this method effects a great saving in cost where conditions are favorable to its application. Once the overburden is removed, the underlying mineral deposit may be mined in any manner desired, following along in the wake of the stripping shovel. Besides having a great working range of depth and width of face, they leave the mineral deposit in good shape to obtain a clean and high recovery and at minimum cost.

They are also suitable for excavating big railroad cuts and large canals, where the material must be removed in cars and under conditions unfavorable for the use of dray-line excavators.

Table 3.—Table of Dimensions—Small Revolving Shovels

| | | 14–B | 18–B | 25–B | 35–B |
|---|---|---|---|---|---|
| Effective pull on dipper, lb. | | 12300 | 18250 | 25800 | 31490 |
| Capacity of dipper | | 2/3 cu. yd. | 7/8 cu. yd. | 1 1/4 cu. yd. | 1 1/2 cu. yd. |
| Size of engines | Main | 5″ × 6″ | 6 ″ × 7″ | 7″ × 8″ | 8″ × 8″ |
| | Thrust | 4″ × 5″ | 4 1/2″ × 5″ | 5″ × 6″ | 6″ × 6″ |
| | Swing | 4″ × 5″ | 4 1/2″ × 5″ | 5″ × 6″ | 6″ × 6″ |
| Revolving frame (over-all house dimensions) | Length | 15′ 3″ | 16′ 0″ | 17′ 9″ | 19′ 4″ |
| | Width | 9′ 0″ | 10′ 0″ | 10′ 0″ | 10′ 0″ |
| Wheel base | Traction | 7′ 3″ | 8′ 0″ | 8′ 9″ | 10′ 3″ |
| | Truck | 7′ 3″ | 8′ 0″ | 8′ 9″ | 10′ 3″ |
| Width over traction wheels | | 8′ 6″ | 9′ 6″ | 11′ 8 1/4″ | 12′ 4″ |
| Boiler | Type | Vertical | Vertical | Vertical Submerged | Loco. |
| | Dimensions | 42″×7′ 8 1/2″ | 48″×7′ 812″ | 54″×8′ 11″ | 48″×9′ 7″ |
| Water tank, total capacity | | 275 gals. | 350 gals. | 400 gals. | 500 gals. |
| Weight in working order | Railroad | 20 tons | 25 tons | 33 1/2 tons | 42 1/2 tons |
| | Traction | 21 tons | 27 tons | 37 1/4 tons | 46 3/4 tons |
| | Caterpillar | 26 1/2 tons | 30 1/2 tons | 44 1/4 tons | 50 3/4 tons |
| Shipping weight | Railroad | 17 1/2 tons | 22 tons | 30 1/2 tons | 38 tons |
| | Traction | 19 tons | 24 tons | 33 1/4 tons | 42 1/4 tons |
| | Caterpillar | 24 tons | 27 1/2 tons | 40 1/4 tons | 46 1/4 tons |
| Approx. gross weight boxed for export (1 ton—2000 lb.) | Railroad | 18 1/2 tons | 23 1/2 tons | 32 1/2 tons | 41 tons |
| | Traction | 20 1/2 tons | 26 1/2 tons | 36 tons | 46 tons |
| | Caterpillar | 26 tons | 31 tons | 42 1/2 tons | 51 tons |
| Approx. volume for export | Railraod | 1300 cu. ft. | 1750 cu ft. | 2350 cu. ft. | 3000 cu. ft |
| | Traction | 1400 cu. ft. | 1950 cu. ft. | 2750 cu. ft. | 3500 cu. ft. |
| | Caterpillar | 1800 cu. ft. | 2400 cu. ft. | 3250 cu. ft. | 3800 cu. ft. |

TABLE 4.—WORKING DIMENSIONS OF SMALL REVOLVING SHOVELS

| | Size of machine | 14-B | | 18-B | | 25-B | | 35-B | |
|---|---|---|---|---|---|---|---|---|---|
| | Standard mounting | Traction | | Traction | | Railroad | | Railroad | |
| | Angle of boom | Standard 45° | Special 52° | Standard 45° | Special 54° | Standard 45° | Special 50° | Standard 46° | Special 51° |
| A | Maximum dumping radius | 21′ 6″ | 20′ 8″ | 23′ 5″ | 22′ 3″ | 25′ 8″ | 25′ 0″ | 30′ 6″ | 30′ 0″ |
| B | Clear dumping height | 11′ 2″ | 13′ 2″ | 11′ 10″ | 14′ 10″ | 13′ 0″ | 14′ 4″ | 16′ 6″ | 17′ 1″ |
| C | Level floor radius (absolute level) | 15′ 0″ | 14′ 3″ | 17′ 0″ | 16′ 0″ | 18′ 2″ | 17′ 8″ | 21′ 3″ | 20′ 6″ |
| C1 | Level floor radius (dipper at max. rake) | 17′ 0″ | 16′ 0″ | 18′ 6″ | 17′ 9″ | 19′ 6″ | 19′ 6″ | 22′ 6″ | 22′ 3″ |
| D | Below level floor | 3′ 2″ | 2′ 7″ | 3′ 3″ | 2′ 6″ | 3′ 11″ | 3′ 5″ | 5′ 9″ | 5′ 2″ |
| E | Radius of boom | 17′ 3″ | 15′ 7½″ | 19′ 1″ | 16′ 7″ | 19′ 9″ | 18′ 4½″ | 23′ 2″ | 21′ 6″ |
| F | Height of boom | 18′ 0″ | 19′ 5½″ | 19′ 4″ | 21′ 5½″ | 20′ 6″ | 21′ 7½″ | 24′ 4″ | 25′ 9″ |
| G | Digging radius at 8′ elevation | 23′ 6″ | 22′ 6″ | 25′ 6″ | 24′ 0″ | 27′ 6″ | 26′ 10″ | 32′ 6″ | 31′ 8″ |
| H | Height dipper will cut | 16′ 6″ | 18′ 9″ | 18′ 0″ | 21′ 3″ | 20′ 6″ | 22′ 0″ | 24′ 4″ | 25′ 9″ |
| I | Rear end radius | | | | | | | | |
| | On roof | 10′ 8½″ | 10′ 8½″ | 11′ 4″ | 11′ 4″ | 12′ 8″ | 12′ 8″ | 13′ 3½″ | 13′ 2½′ |
| | On rev. frame | 10′ 4″ | 10′ 4″ | 11′ 0″ | 11′ 0″ | 12′ 5″ | 12′ 5″ | 13′ 0″ | 13′ 0″ ′ |
| J | Highest part with boom lowered | 13′ 0″ | 13′ 0″ | 13′ 10″ | 13′ 10″ | 14′ 2″ | 14′ 2″ | 14′ 4″ | 14′ 4″ |
| | Deduct from B. F. & H. for railroad mounting | 6½″ | 6½″ | 6″ | 6″ | .... | .... | .... | .... |
| | Add to B. F. & H. for traction mounting | .... | .... | .... | .... | 5½″ | 5½″ | 8¾″ | 8¾″ |
| | Add to B. F. & H. for Caterpillar mounting | 9″ | 9″ | 4″ | 4″ | 7″ | 7″ | 6½″ | 6½″ |

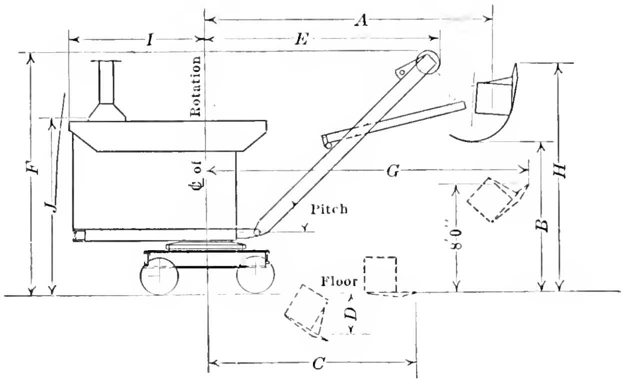



Table 5.—Table of Dimensions—Large Revolving Shovels

| | 150-B | 175-B | 225-B |
|---|---|---|---|
| Effective pull on dipper, pounds | 43000 | 60000 | 77500 |
| Capacity of dipper | $2\frac{1}{2}$ yd. | $3\frac{1}{2}$ yd. | 6 yd. |
| Length of boom | 60′ | 75′ | 80′ |
| Length of dipper handle | 38′ | 48′ | 58′ |
| Length of revolving frame | 37′ 3″ | 40′ $5\frac{1}{2}$″ | 48′ 8″ |
| Width over sub-base | 22′ | 28′ 2″ | 33′ 8′ |
| Distance C to C of trucks | 20′ × 17′ | 24′ × 24′ | 30′ × 30′ |
| Locomotive boiler | 58″ × 15′ | 64″ × 15′ 10″ | 76″ × 18′ |
| Water tank capacity, gallons | 1460 | 2500 | 3000 |
| Main engines, double | 10″ × 12″ | 12″ × 15″ | 14″ × 16″ |
| Swing engines, double | 8″ × 8″ | 9″ × 9″ | 10″ × 10″ |
| Thrust engines, double | $7\frac{1}{2}$″ × 7″ | $8\frac{1}{2}$″ × 8″ | 10″ × 10″ |
| Approx. net shipping weight, pounds | 248,000 | 363,000 | 521,000 |
| Approx. working weight, pounds | 316,000 | 428,000 | 624,000 |

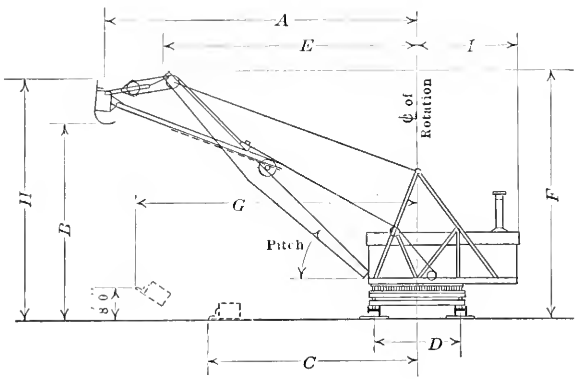



TABLE 6.—WORKING DIMENSIONS OF LARGE REVOLVING SHOVELS

| | | 225–B | 175–B | 150–B |
|---|---|---|---|---|
| | Pitch of boom | 45° | 45° | 45° |
| A | Dumping radius | 94′–6″ | 85′–0″ | 74′–0″ |
| B | Height of dump (dipper door open) | 61′–0″ | 52′–0″ | 40′–0″ |
| C | Level floor radius | 59′–0″ | 56′–0″ | 46′–0″ |
| D | Center to center of tracks | 30′–0″ | 24′–0″ | 17′–0″ |
| E | Radius of boom | 77′–0″ | 68′–0″ | 56′–0″ |
| F | Height of boom | 76′–0″ | 66′–0″ | 53′–0″ |
| G | Digging radius at 8′ elevation | 88′–0″ | 78′–0″ | 68′–0″ |
| H | Height of cut | 72′–0″ | 61′–0″ | 47′–0″ |
| I | Radius of rear end | 32′–4″ | 29′–0″ | 27′–0″ |

An ordinary railroad type shovel would often require 8 or 9 cuts, with all the attendant changes of position of the loading track, to make an excavation that one of these great revolving shovels could make in one large cut.

Table 3 gives the dimensions, and Table 4 the working dimensions of the small Bucyrus revolving shovels.

Table 5 gives the dimensions, and Table 6 the working dimensions of the large Bucyrus shovels.

Table 7 gives the dimensions of some medimum-sized Marion shovels.

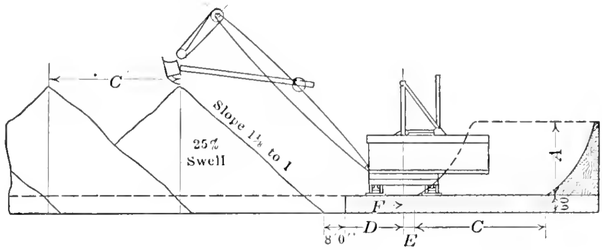



TABLE 7.—SPECIFICATIONS OF MARION COAL STRIPPING STEAM SHOVELS

**1⅛ TO 1 SLOPE OF SPOIL BANK**

MODEL 211
2-YARD DIPPER, 45-FOOT BOOM

| A | C | C′ | D | E | F |
|---|---|---|---|---|---|
| 14′ | 29′ | 29′ | 17′ | 4′ | 13′ |
| 16′ | 29′ | 29′ | 16′ | 4′ | 12′ |
| 18′ | 20′ | 20′ | 16′ | 4′ | 12′ |
| 20′ | 11′ | 11′ | 25′ | 13′ | 12′ |

MODEL 231
3¼-YARD DIPPER, 55-FOOT BOOM

| A | C | C′ | D | E | F |
|---|---|---|---|---|---|
| 20′ | 36′ | 36′ | 19′ | 4′ | 15′ |
| 22′ | 36′ | 36′ | 18′ | 4′ | 14′ |
| 24′ | 27′ | 27′ | 18′ | 4′ | 14′ |
| 26′ | 11½′ | 11½′ | 29½′ | 15½′ | 14′ |

**1¼ TO 1 SLOPE OF SPOIL BANK**

MODEL 211
2-YARD DIPPER, 45-FOOT BOOM

| A | C | C′ | D | E | F |
|---|---|---|---|---|---|
| 12′ | 29′ | 29′ | 17′ | 4′ | 13′ |
| 14′ | 29′ | 29′ | 16′ | 4′ | 12′ |
| 16′ | 20′ | 20′ | 16′ | 4′ | 12′ |
| 18′ | 12′ | 12′ | 24′ | 12′ | 12′ |

MODEL 231
3¼-YARD DIPPER, 55-FOOT BOOM

| A | C | C′ | D | E | F |
|---|---|---|---|---|---|
| 18′ | 36′ | 36′ | 19′ | 4′ | 15′ |
| 20′ | 32′ | 32′ | 18′ | 4′ | 14′ |
| 22′ | 22′ | 22′ | 18′ | 4′ | 14′ |
| 24′ | 14′ | 14′ | 27′ | 13′ | 14′ |

**Extra Long Booms and Dipper Sticks.**—To meet special or unusual requirements, the leading shovel builders are generally willing to recommend or design machines considered best suited to the conditions.

Some requirements are best filled by machines of standard design, but equipped with extra long booms and dipper-handles, and braced with special wide spread jack-arms. In brick manufacture, where a uniform mixture of material is desired mined directly from bottom to top of the working bank, such a shovel permits of very high continuous cuts. Again in digging canals and aqueducts, an extremely high lift is often desired, so that the excavated material may be loaded into cars on the bank or cast directly on the banks. By decreasing the size of the dipper and increasing the length of the boom and dipper stick such lifts may be effected. For digging to depths of from 10 to 30 feet below grade, one of these shovels may have its car-frame mounted on transverse sills, which are in turn mounted on trucks or rollers equipped to run parallel to and on both sides of the canal being excavated. The long booms and dipper sticks also permit, when desirable, the dumping of material at a considerable distance to the sides of the excavation. In practical working, it is usually undesirable to plan banks of a height much greater than the width that a given shovel may be expected to cut; this width being measured from the loading track to a point on the bank about 6 ft. above grade. Thus the length of boom and dipper stick should bear a direct relationship to the height of bank desired.

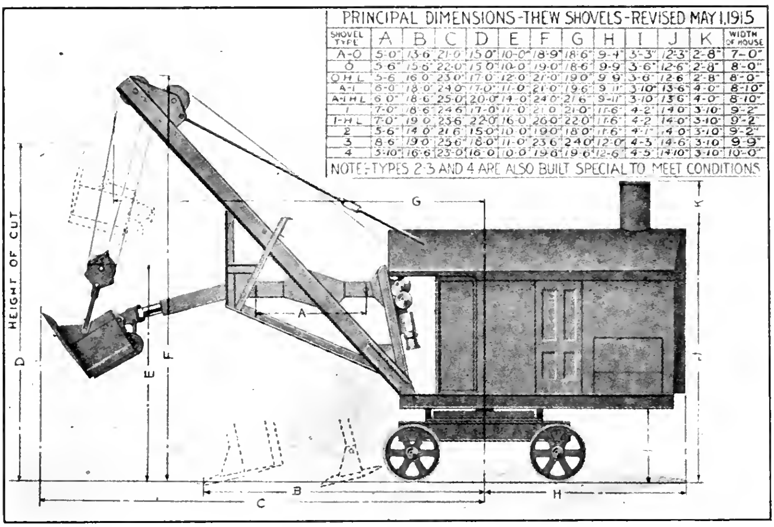
**Quarry, Tunnel and Stope Shovels.**—A light weight type of shovel is often used for loading rock broken in advancing tunnels or adits, in quarries or even in certain underground stopes. Shovels for such uses are made by the Bucyrus, Marion, Ball, and Thew Companies.

The Thew,[1] as illustrated by Fig. 7, has several unique features. It is a self contained machine, having the upright tubular boiler, boom, engine, small coal bunker and water tank all placed on a revolving platform. The earlier steam-driven machines and the present electric and gasoline ones are operated by one engine. On the steam machines this engine is of the double reversing type, runs continuously, and is controlled by a fly-ball governor. The later steam machines have independent engines for hoisting, swinging and crowding and have the usual control levers and throttles. One of the unique features is the horizontal crowding motion having a movement of about eight feet. This is accomplished by suspending the dipper by an adjustable arm hinged to a sort of trolley or carriage and arranged to move horizontally along a trackway and thus parallel to the grade. This permits broken material to be shovelled clean along a flat bottom for 8 or 10 ft. Another feature is that the dipper handle is built with a swivel clamp designed to permit

[1] Manufactured by the Thew Automatic Shovel Co., Lorain, Ohio. They are used in clay and shale working industries; underground by the American Zinc Lead & Smelting Co. of Joplin, Mo., in the ore quarries of the Granby Company, of Phoenix, B.C., and elsewhere. Over 1200 are rated in use on such work. The makers claim this shovel to be the pioneer of the full-circle swing type.

the dipper to turn or swivel when working its way through obstructions, thus avoiding concentrated strains. They are self-propelling and built to run on traction wheels or track, using jacks in the latter case. They carry dippers of from ¾ to 1¾ cu. yd. and handle from 25 to 40 cu. yd. of clay or shale per hour with a single operator, or from 40 to 60 cu. yd. with the addition of a fireman. One or two trackmen are usually provided.

The Bucyrus 27-D Coal-loader is an excavator which has found much favor in digging and loading coal from the thin flat beds of Kansas. These beds are previously uncovered by large


PRINCIPAL DIMENSIONS-THEW SHOVELS-REVISED MAY 1, 1915

| SHOVEL TYPE | A | B | C | D | E | F | G | H | I | J | K | WIDTH OF HOUSE |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| A-O | 5-0" | 13-6 | 21-0 | 15-0" | 10-0" | 18-9" | 18-6" | 9-4" | 3-3" | 12-3" | 2-8" | 7-0" |
| O | 5-6" | 15-6" | 22-0" | 15-0" | 10-0 | 19-0" | 18-6" | 9-9" | 3-6" | 12-6" | 2-8" | 8-0" |
| O H L | 5-6 | 16-0 | 23-0" | 17-0 | 12-0 | 21-0" | 19-0" | 9-9 | 3-6" | 12-6 | 2-8" | 8-0" |
| A-I | 6-0" | 18-0" | 24-0" | 17-0" | 11-0 | 21-0" | 19-6" | 9-11" | 3-10" | 13-6" | 4-0" | 8-10" |
| A-I H L | 6-0" | 18-6" | 25-0" | 20-0" | 14-0 | 24-0" | 21-6" | 9-11" | 3-10" | 13-6" | 4-0" | 8-10" |
| I | 7-0" | 18-6" | 24-6 | 17-0" | 11-0 | 21-0 | 21-0" | 11-6" | 4-2" | 14-0" | 3-10" | 9-2" |
| I-H L | 7-0" | 19-0 | 25-6 | 22-0" | 16-0 | 26-0" | 22-0" | 11-6" | 4-2 | 14-0" | 3-10" | 9-2" |
| 2 | 5-6" | 14-0" | 21-6 | 15-0" | 10-0 | 19-0" | 18-0" | 11-6" | 4-1" | 14-0 | 3-10" | 9-2" |
| 3 | 8-6" | 19-0 | 25-6" | 18-0" | 11-0" | 23-6" | 24-0" | 12-0" | 4-3 | 14-6" | 3-10 | 9-9" |
| 4 | 5-10" | 16-6 | 23-0" | 16-0 | 10-0 | 19-6" | 19-6" | 12-6" | 4-5" | 14-10" | 3-10" | 10-0" |

NOTE:-TYPES 2-3 AND 4 ARE ALSO BUILT SPECIAL TO MEET CONDITIONS



Fig. 7.—Standard Thew shovel.

revolving shovels. Usually the beds are cut up by seams and layers of fire-clay or shale, locally called "horsebacks," which must be kept out of the coal and amount to about 10 or 15 per cent. of the cut. The machine is well suited to this work. It has three motions similar to all shovels, but the crowding motion is kept in a straight line and may be held at any acute angle with the horizontal. The dipper may thus cut a level floor lifting the coal bed cleanly off its bottom but stopping at any seam, just about the same way that a man would operate a scoop on a shovelling floor. Having the dipper loaded, the boom is hoisted, and the machine is rotated to the desired dumping point where the dipper is tripped and discharges forward. The maximum radius is 30 ft. 6 in. Fig. 8 gives an idea of the appearance and construction of these loaders. They have a capacity of from 40 to 70 tons per hour in coal beds from 2 to 4 ft. thick. The machine is capable of making a complete cycle in 30 sec. The price of this machine mounted on caterpillar tractor was about $14,000 at factory in 1916. Its shipping weight is 30 tons and working weight 33 tons.

The Erie[1] shovel is a recent one having a unique crowding device which automatically permits the operator to cut a level

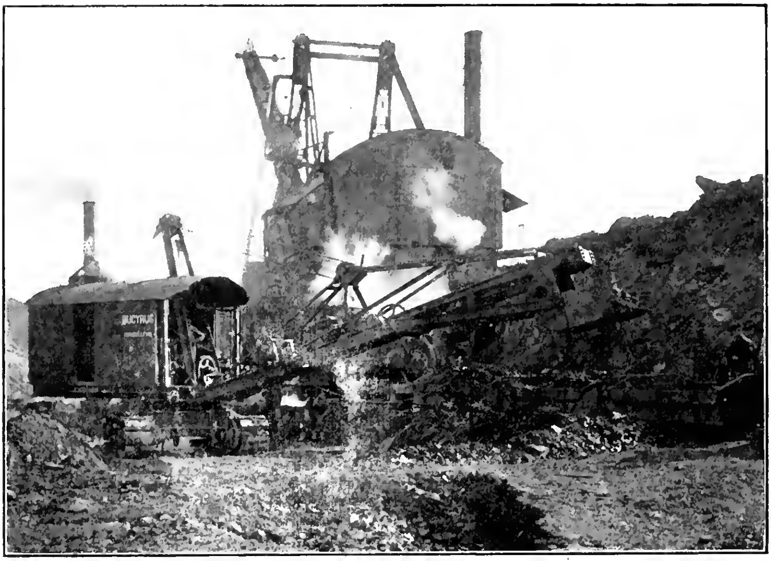



Fig. 8.—Bucyrus 27-D coal loader.

floor or a floor of a determined slope. This is useful on road and ditch work. A device is fitted to the ordinary crowding mechanism by means of which the crowding engine may be made to work automatically in definite relationship to the hoisting motion without having the operator constantly playing one motion against the other. This device simply consists of a small pinion, fitted to one end of the shipper shaft, which operates a small auxiliary rack traveling in and out with the dipper stock. On the end of the rack is a small roller traveling on a track which represents the desired bottom to be cut. This

[1] The Erie Shovel is built by the Ball Engine Co., of Erie, Penn.

mechanism is connected to the control valves governing the crowding and hoisting engines and thus controls their relative movement. By setting the roller track in the correct position the desired grade of bottom will be cut. If desired the device may be thrown out of use and the shovel operated in the usual way.

This shovel weighs 20 tons, carries a 17½-ft. boom and a 34-cu. yd. dipper, cuts a maximum floor width of 35 ft., is fitted with either traction wheels or rail trucks and is of the revolving type.

The conditions must be exceptional for the economic operation of these light shovels underground but under quite favorable conditions both the loading time and cost may be materially reduced as compared with hand-loading. The stope or chamber in which they are to be used must be at least 18 ft. high and 30 ft. in diameter and even then the machines are awkward to move around as the piles of ore are depleted. Coarse material must be broken as encountered to facilitate loading and considerable will usually be left scattered around which must be loaded by hand. Because of the disadvantages, the cost of mechanical shovel-loading underground may easily exceed that of hand-loading.

Before leaving these light tunnelling and loading machines, it may be mentioned that several so-called "shovelling machines"[1] have been designed and put on the market for such purposes as shovelling and loading the broken material in tunnel advances, in stopes and from stock piles. Here again conditions must be favorable for their use if they are to show a lower cost than hand work, though more often some saving in time may be effected. They are subject to hard usage and it has been found that the expense and delay in keeping them in operation is often a serious drawback. Portable loaders, which simply elevate rock by means of a conveyor belt, but do not "shovel," have been used in drifts with good results.[2]

**Wire-Rope Shovels.**—A steam-shovel of later design and known as the Robinson or Atlantic type, was developed by the American Locomotive Company and after trials covering 1908 and 1909, was put out by the American Equipment Company

[1] Among these are the Myers-Whaley, made at Knoxville, Tenn., and the Halby, made by the Lake Shore Engine Works, at Marquette, Mich.

[2] P.L.S.M.I. Sept. 6–9, 1915, Morris-Lloyd Mine.

in 1910 in the iron ore service of the Lake Superior iron ranges. It is now built by the Bucyrus Company, as illustrated by Fig. 9; Tables 1 and 2 give its characteristics.

This shovel is designed to do the same class of work as the heavier standard shovels; it has the distinctive feature of using a direct wire-rope hoist instead of the differential chain and pulley hoist. The hoisting engine is bolted to the base of the boom and the wire rope passes from the drum over one large twin grooved sheave directly to the dipper back. The hoisting drum is of large diameter and is driven by two machine-cut gears. The steel hoisting rope is made up of two parallel cables equalizing their load by passing around a thimble on the dipper. The placing of the hoisting engine at the end of the boom makes more room available in the body, part of which is used to house the extra large, efficient, locomotive-type boiler with its large water tank. This arrangement, however, throws considerable additional weight on the front end and turntable.

The Class 80 Atlantic shovel has a working speed of 3 to 5 dippers per minute. Bunker capacity is 8800 lb.

In addition to this heavy shovel, others are built designed to use dippers as small as 1 cu. yd. capacity.

The design, materials and workmanship are of a high class, and among the advantages claimed for the shovel are increased efficiency due to the elimination of the friction of the chain links passing over their several sheaves, lower coal consumption due to the use of a boiler of large steam capacity, the ability of the dipper to take large boulders and full loads and its simplicity due to the elimination of the bail and the increased speed and ease of the manœuvres making faster loading. In severe service it was found that the earlier designs were not strong enough, so that when competing with the chain-type of shovel, the delays due to breakdowns were a serious disadvantage. By strengthening the shovel to the ruggedness of the chain-type, its advantages of design, fuel economy, extra lift, ease of operation and increased loading speed when working in favorable ground, recommend it to favorable attention.

It may be mentioned that just prior to 1906 the Allis-Chalmers Company of Milwaukee brought out a wire-rope shovel. This had some unique features including a scheme to relieve the weight on the boom and turntable by so leading the ropes that the working strains are resolved into a lifting component, made

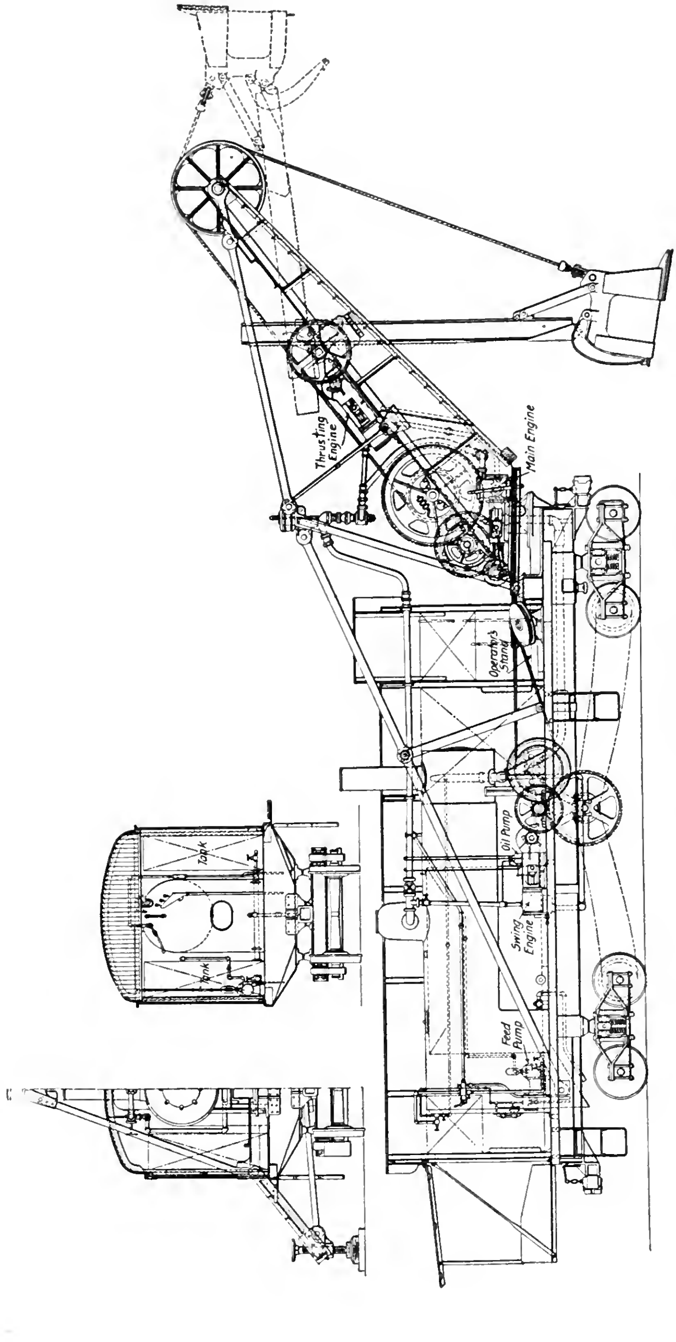



FIG. 9.—Atlantic wire-rope shovel.

effective at the foot of the boom, thus lessening the load on the turntable and reducing the strains and turning friction so that the swing is made more rapidly. To do this, the hoisting drum is geared directly to the engine, and from the drum the rope is lead over a sheave, suspended from the A-frame, to a differential drum placed on the boom. This differential drum is made up of three concentric drums, one of large diameter in the centre carrying the single rope from the hoist, and two drums of a common smaller diameter, rigidly attached to both sides of the former, carrying the twin ropes over the head sheave to the dipper back. Their respective diameters are about 7 ft. and 3½ ft. Winding the rope on the hoisting drum causes the hoisting rope to run off the large diameter of the differential drum, and the twin ropes atached to the dipper to wind on the smaller drums. This difference in drum diameters causes a slower but heavier pull to be exerted. The rope, running from the sheave suspended from the A-frame to the large differential drum, is located in the line of the turntable axis, so that no guide pulleys are required to act as guides for the rope twist. Thus when digging strains are exerted, the tendency of the rope between these sheaves to exert an upward pull on the differential drum and lower end of the boom relieves weight from the turntable. This is said to amount to about 15 tons when digging with a 3-cu. yd. dipper, and 7 tons with the dipper empty. All three engines are of the double reversing type. With a 40° normal angle and 15-ft. boom, a clear lift of 18 ft. is had. The boiler is of the locomotive type. The machine is self-propelling and the clutches are steam driven. There are also side bearings which screw up horizontally making a wedge-like contact between the tread of the truck-wheels and sides of the car bed. This gives the machine a rigid bearing on the rails across the full width of the track gauge and prevents rocking or tilting.

Wire-rope shovels of this class have not come into the general use enjoyed by those of the chain type. For the large revolving shovels, however, wire rope is exclusively used for hoisting.

**Electric Driven Shovel.**—Until recent years steam—or in a very few cases compressed air—was the motive power employed on all of these excavators, but the progress made in electrically driven shovels has been so marked that the present day electric machines leave little to be desired.

Electric shovels of all sizes are now built by the leading manufacturers and, where electric power is reasonably cheap, they are rapidly coming into general use. During the few years they have been in service they have been highly satisfactory and dependable.

These shovels are divided into three classes; the friction electric, operated by a single constant-speed motor with friction clutches; the three or four motor direct-current equipment; and the three or four motor alternating-current equipment. The first class is little used as it is much slower than steam, though about as cheap in operation.

Either direct or alternating current may be used. The tendency of present practice is strongly in favor of alternating current when it is available, and to date there is a large preponderance of alternating current machines in actual service. This preference is due to the greater ruggedness of the A. C. motor and to the elimination of all commutators and of the motor generator required to supply direct current.

The electrically operated shovel has a number of strong advantages over the steam-driven shovel. The operation is quieter, steadier, quicker, cleaner and safer from sparks and fire than that of the steam shovel. None of the troubles due to bad or interrupted water are encountered; no pipe lines or hauling of water or fuel are required; no fireman is needed; during cold weather the steam driven machine must be carefully watched, as must also the pipe lines, to avoid freeze-ups; on the steam machine some steam must be kept up, even while idle, and the usual " standby" losses of firing-up and after-use come in with the attendant expense for fuel and water. The electric machine may be started working as soon as the crew arrives, and current consumption ceases when the motors are stopped. It is of course necessary to provide transmission lines and power cables to the shovels, but the expense of maintaining them, or moving them about, is comparatively small. In some cases where current has to be supplied to electrically driven shovels from a long distance, or at an unsuitably low voltage or through an existing feeder of inadequate capacity, the operation of the electric shovel, which is intermittent and characterized by high peaks, may cause objectionable disturbance of the electric supply to other consumers on the same electric feeder.

Except for the changes required to install the electric equipment replacing the steam, the design of the machines is practically the same. Instead of the usual steam engines, independent motors are provided for the hoisting, swinging and crowding functions on all except the small machines, where the first two motions may be taken care of by a single motor. The hoisting and swinging motors are mounted directly on the rear of the shovel platform and are geared to the drums through reducing gears. The thrust motor is mounted on the upper side of the boom, and geared to the pinion-gears through proper reducing gears. Control of the motions is effected by individual electric controller levers operated in the same manner as in steam machines. To protect the motors and machinery from careless handling or severe overload, automatic cut-out relays are put in the line so that under such conditions, the motors will be stopped and the load held. The motors may then be restarted by means of the controllers.

Notwithstanding the fact that the electric shovel has been developed to a point where its performance, control, and dependability is fully equal to that of the corresponding steam machines, its general adoption has been retarded through a perfectly natural caution, remembering that the steam machines have been wonderfully developed, have shown enormous cost reductions over older methods of excavation and that the training of both the machine operators and those in charge of such work has almost entirely been with the steam machines. Furthermore, at many places where electric power is cheaply available, the companies have considerable capital invested in good steam equipment which still has a long efficient working life, and though in the long run the expense involved in changes in equipment might be fully justified, such transitions will be gradual, and should be made with balanced judgment.

In this connection it is interesting to note the shipments of electric shovels made by the larger manufacturers and some of the localities to which they were sent. Other makers have also furnished electric shovels.

Commencing in 1912 and extending to April, 1919, the Bucyrus Co. have shipped twenty electric railroad shovels, five large electric revolving shovels, and nine small electric revolving shovels. Shipment was made of three in 1912, two in 1913, three in 1914, two in 1915, nine in 1916, four in 1917, eight in 1918, and three in the early part of 1919.

Of the railroad shovels seven were Type 103-C, six were Type

100-C, two were Type 70-C, and the remaining five were one each of Types 60-C, 68-C, 40-R, 95-C, and Cl.B. Vulcan. All the railroad shovels use alternating current except the Vulcan which uses 220-volt direct current. These shovels were shipped, eight to the Luossavaara Kiirunavaara Akt., Sweden, four to the Chile Exploration Co., Chile, three to the Hydro-Electric Power Commission, Niagara Falls, and the remaining five singly to different purchasers.

The large revolving shovels are all Type 225-B, A-C. machines, three of which were shipped to the Hydro-Electric Power Commission, Niagara Falls, one to the Pittsburg and Midway Coal Co., Kansas, and one to the Chile Exploration Co.

Of the small electric revolving shovels, two were Type 14-B, three Type 25-B, three Type 35-B, and one Type 18-B. All use alternating current except one 14-B, which uses 550-volt direct current. Two were shipped to the Russian Reclamation Service in Turkestan, two to the Locust Mt. Coal Co., Penna., and the others singly to other purchasers.

The Marion Steam Shovel Co. from 1908 to 1919 inclusive has shipped nine railroad electric shovels, ten large electric revolving shovels and twenty-six small electric revolving shovels. Of these 45 shovels the year of shipment of six is not known, though it was quite recent, but the others were shipped, one in 1908, three in 1909, one each in 1910, 1913, and 1914, five in 1915, ten in 1916, three in 1917, ten in 1918 and four in 1919.

Of the railroad shovels three were Model 40 using A-C, and were shipped to the Los Angeles Board of Public Works. The others all use 550-volt direct current, two being Model 51, two Model 91, and one each Models 41 and 92, and were shipped one to Norway, one to Chile, two to Sanborn, N. Y., and one each to Cambria, N. Y. and Niagara Falls.

The large revolving shovels all use 230-volt direct current, one was Model 271 and nine were Model 300, and all were shipped to coal stripping operations in Ohio and Pennsylvania.

Of the small revolving shovels, six were Model 28 and one of these, shipped to the Government of the Phillippine Islands, uses 230-volt direct current, the others all using alternating current. One Model 28 was shipped to the United Verde Copper Co. in Arizona for use underground. Six small revolving shovels were Model 31, all using A-C except one 230-volt D-C machine, shipped to the Canadian Klondyke Mining Co., and fourteen were Model 36 all using alternating current and almost all shipped to coal mining operations in Ohio and Pennsylvania.

In most of the electric installations now being made, fuel is so expensive as to leave no doubt as to the economy of electric equipment, whereas electric power is of quite moderate cost, frequently being as low as ¾ c. per KWH. Based on results from shovels long in operation, it has been found that the average current consumption per ton of material excavated varies from 0.3 KWH to 0.5 KHW. In difficult digging or poorly shot banks, this power consumption will be exceeded, since most of the shovel movements do not deliver any tonnage. The above figures cover a reasonable average in hard rock.

A Comparison of Electric Shovels Operating in The Hard Iron Ores of Sweden is Quoted From The Bucyrus Company (The Power Consumption is Somewhat High)

| | German make | | Bucyrus make | |
|---|---|---|---|---|
| Maximum peak | 390 KW | 380 KW | 520 KW | 480 KW |
| Next 4 peaks | 370 KW | 340 KW | 460 KW | 420 KW |
| Amount of excavation | 36.5 tons | 36.5 tons | 37.4 tons | 36.0 tons |
| Average load | 105.4 KW | 105.4 KW | 157 KW | 164 KW |
| Time to load 5 cars | 20 min. | 22 min. | 9½ min. | 11¾ min. |
| KWH | 35.1 | 38.6 | 24.8 | 30.8 |
| KWH per ton | 0.96 | 1.06 | 0.662 | 0.855 |

The following comparison shows the estimated difference in the cost of operation per day of 10 hr. of a 100C Bucyrus shovel, whether operated by A-C. electric motors or with boiler equipped with oil burner or for coal.

| | Electric | Oil Burning | Coal |
|---|---|---|---|
| Runner | $5.00 | $5.00 | $ 5.00 |
| Craneman | 3.60 | 3.60 | 3.60 |
| Fireman | ..... | 2.40 | 2.40 |
| 5 Pitmen | 7.50 | 7.50 | 7.50 |
| Oil (18 bbl. @ $1.50–42 gal./bbl.) | ..... | 27.00 | ..... |
| Electricity (1400 KWH @ 1¢) | 14.00 | ..... | ..... |
| Coal (4 tons @ $5.00) | ..... | ..... | 20.00 |
| Oil and waste | 1.00 | 1.50 | 1.50 |
| Repairs and renewals | 1 50 | 2.50 | 2.50 |
| Water | ..... | 2.00 | 2.00 |
| Total | $32.60 | $51.50 | $44.50 |

It is assumed that 168 gal. of oil is equivalent to 1 ton of coal. Under favorable conditions the power consumption could be reduced all around to three fourths of the above. If electric power costs 2c. per KWH the operating cost of the electric shovel would be increased to $46.60 per shift and would, therefore, be higher than the steam shovel fired by coal, but less than if fired by oil. Fuller costs will be given in another chapter.

Fig. 10[1] interestingly shows the comparative operating power-costs with variable cost for coal and electric power.

It may be here mentioned that the price of an electric shovel of the 100C class was about $20,500 at the works in 1915. This covered motors and controllers sutable for alternating, 3-phase,

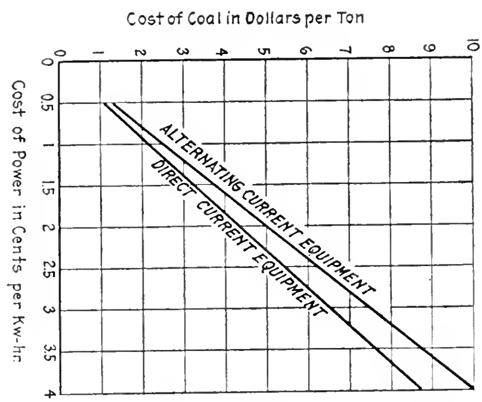



FIG. 10.—Comparative power cost curve.

60-cycle, 440-volt current but no transformers. This price was about one-third higher than for the same class of steam machine.

**Electro-Hydraulic—Driven Shovels.**—Desiring to simplify the control and reduce to a minimum the heavy surges of current taken by the motors, the electro-hydraulic shovel has been designed. This first consisted in replacing the steam machinery of a standard shovel with a motor-driven centrifugal pump, a pressure tank, an air tank, a small air compressor and water cylinders with plungers, pistons and valves. Later it was found that the tanks and air compressor were not required, as sufficient capacity could be obtained without the use of compressed air;

[1] ROGERS, H. W.: The Application of Electric Motors to Shovels, T.A.I.-M.E., pp. 299–309, Feb., 1914.

the capacity of a centrifugal pump increasing rapidly with a decrease in head.

The shovel so equipped by the Penn Iron Mining Co., of Vulcan, Mich.[1] operates as follows: The dipper is hoisted by means of one large single-acting cylinder and plunger. Double hoisting ropes pass around two sheaves at the outer end of the plunger and these ropes have one end fastened to the dipper while the other is anchored to the front flange of the hoisting cylinder. With this arrangement the dipper travel is just twice that of the plunger. The weight of the dipper pulls the plunger back on the exhaust stroke. Swinging the boom is effected by means of a double-acting cylinder with a piston rod extending through each cylinder head and with a sheave at each end of the rod. Passing around the front of the swing circle is a rope, each end of which is led around one of the sheaves on the ends of the rod and then anchored to the car body. The thrusting is done by the four piston rods of four thrusting cylinders directly connected to the dipper handle in such a way as to give perfect balance around the shipper shaft. Swivels and sleeve-joint piping permit the boom to operate freely. To trip the dipper, an ingenious solenoid tripping device is placed near the front-end of the dipper handle and works the tripping latch. This shovel is operated entirely by one man; with one lever he controls the thrust, with another he hoists, while by lowering his right hand on the first lever he touches a button which causes the solenoid to trip the dipper. Foot levers control the swing of the boom to the right or left, and, when equally compressed, the valves automatically centre the boom. The controller handle for operating the motor which drives the centrifugal pump supplying power to the shovel is conveniently located near the right hand lever.

Loading from a stock pile, 3000 tons per 10-hr. shift has often been handled with only fair train service. The wattmeter record shows the dipper speed to be from 3 to 4 per minute and the power consumption to be from about 80 to 130 KW.

The advantages of this shovel are: its few, simple and slow moving parts; absence of gears, clutches, brakes and drums; comparative cleanliness and silence of operation; need of but one man to operate; greatly reduced peak load when operating;

[1] F. H. Armstrong: T.A.I.M.E., Feb., 1916. *Iron Trade Review*, Feb. 27, 1916, p. 393.

no power cost when idle; smooth and accurate control. As the leakage of the liquid is small, an hydraulic oil that will not freeze can be safely used in cold weather. While yet somewhat in the experimental stage, for the use to which this shovel has been put, it seems to have much to recommend it.

**Oil-Engine-Driven Shovels.**—Oil engines of the various types are supplied if desired by several of the shovel manufacturers, especially for the lighter machines. So far, this class of motive power has not come into very wide use and the conditions where it shows to advantage are exceptional.

## COMPETING MACHINES

**General Field.**—Power-shovels have been competing with aerial tramways, machine-scrapers, clam-shell, orange-peel and bucket excavators, dry-land dredgers and many other similar devices in the general excavating fields, but the results obtained with shovels have usually been much more satisfactory in mining.

**Dredges.**—Such machines as the great floating dipper and bucket dredges, the hydraulic and deep-water dredges, occupy as a rule a field distinct from that of the shovel.

The dipper dredge, designed to carry dippers of from 2 to 15 cu. yd. capacity, is used under some conditions for drainage and irrigation ditch excavation, and for large canal, inland lake and harbor dredging. It is especially useful and economical on subaqueous rock work as great power can be concentrated at one spot. In the Panama Canal there were a number of 15-cu. yd. dipper dredges at work on the slides.

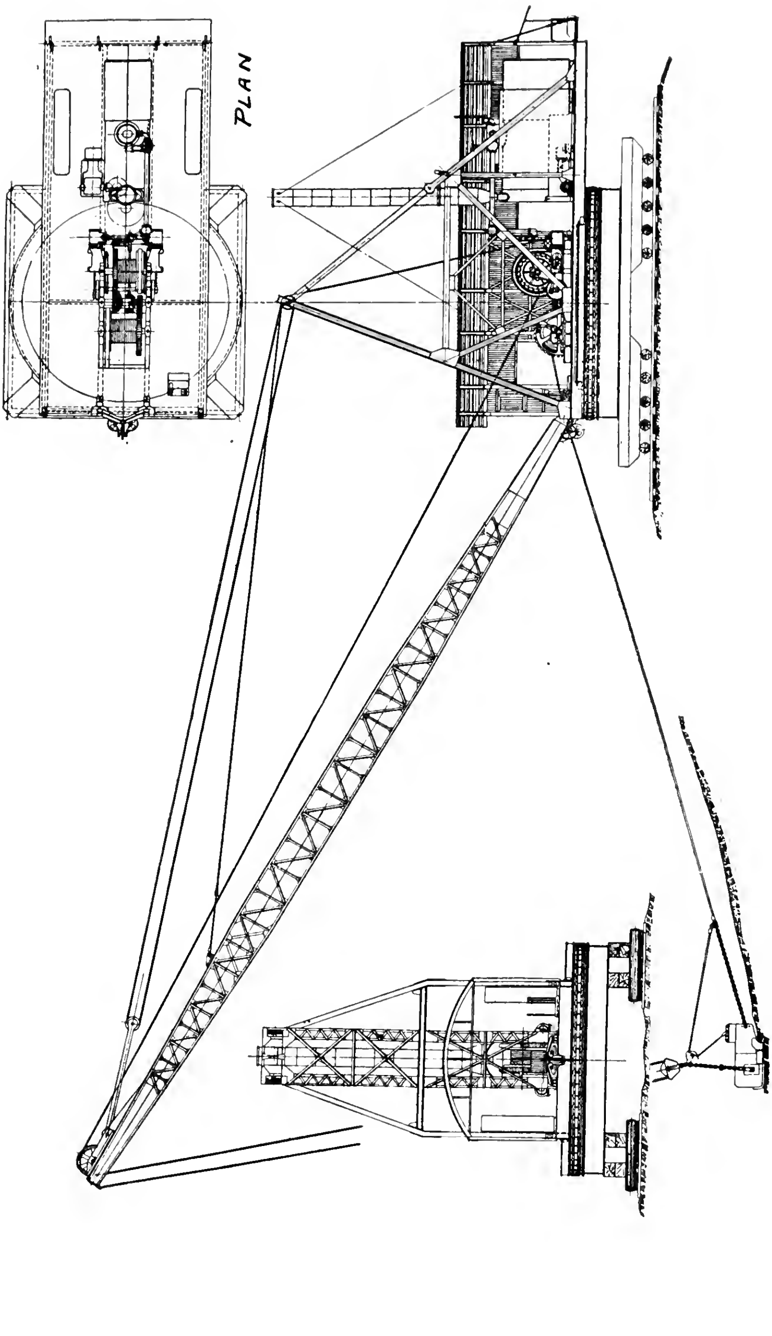
The hydraulic or suction dredge is the most economical device for removing great quantities of sand, loam, clay or gravel from river beds, lake bottoms or harbors. It has a much greater capacity than any other type of dredge, and is able to dispose of the material by means of pipe lines, at great distances from the point of excavation. The usual sizes have suction pipes of from 12 to 36 in. diameter, and are used on the Great Lakes, New York State Barge Canal and many other places.

Placer-dredges, built with buckets of from 2½ cu. ft. to 16 cu. ft. capacity, are employed in many parts of the world for digging gravels, hard-pan and decomposed rock forming the bottoms of present and former water courses. They have great capacity—some over 300,000 cu. yd. per month—and do their work cheaply

—at times as low as 2½ c. per cu. yd. Their greatest mining use is in dredging for gold, and they may be found on this work in California, Alaska, South America and other countries.

In removing overburden from some of the brown-coal deposits in Germany a continuous-bucket excavator, similar in principal to the Parsons trench excavator is used. The ladder and bucket arrangement is designed so that the empty buckets descend on the upper side of the ladder, mouth down, while the loaded buckets are drawn up on the lower side. The buckets discharge the spoil, upon passing around the upper sprocket wheel, into a hopper, which in turn discharges into the spoil cars running beneath. This machine is self-propelling and supported by trucks which run on a double track along the pit edge. Between the excavator tracks runs a third track on which the spoil cars are run. These machines are built in several sizes weighing from 12 to 70 tons and suitable for making cuts ranging in depth from 15 to 45 ft. with buckets of from 1¼ to 8½ cu. ft. capacity and digging capacity in medium soil of from 17 to 200 cu. yds. per hour. The continuous-bucket excavator is satisfactory where the overburden is soft and free from boulders and rock. In this respect it is similar to the drag-line excavator and has some additional features of advantage in its operation, but it has limitations in radius of action and disposition of overburden. In Germany the cost of removing overburden with these machines was estimated to be from 6 to 10 c. per cu. yd.

**Dragline Excavators.**—Probably the nearest direct competitor of the power-shovel in America is the dragline excavator. One of Marion design is illustrated by Fig. 11. It has been extensively used in irrigation and drainage ditches, levee and dam building, making railroad fills and in stripping quicksands and gravel overburden from bodies of ore and coal. It has a remarkably wide radius of action and can deposit material a long distance from the cut. Thus it may travel parallel to its work, digging from one side and depositing on the opposite, without throwing much weight on a weak bank. It has the advantage of digging far below the level on which it stands, so that in case of floods or high ground-water level, the excavation may be continued, whereas a steam shovel would be "drowned out." Its wide reach eliminates frequent moving and may even eliminate hauling of the excavated material. These machines work around a complete circle, as do the revolving steam shovels.




Front elevation

Side elevation

Fig. 11.—Drag-line excavator.

They are economical of labor as only one runner and one fireman are required. The character of material which it handles, however, must be much looser or softer than that which can be excavated with a steam shovel without blasting, and it is doubtful if its daily capacity will come up to that of the shovel. These two factors may have a very important bearing on the total cost of the problem.

The machinery is designed along the same line as on shovels. The bucket however, is of different design from the shovel dipper and is hoisted by a three-part line, one part of which leads to the bail. By braking this line dumping is effected, as the continued hoisting of the other two lines raises the back of the bucket and causes it to tip upside down. The fairlead consists of two horizontal sheaves, mounted on a casting at the front sill of the machine, and two vertical sheaves carried in a swinging frame pivoted to this casting. This frame takes the direction of the dragline and maintains a straight-lead at all times.

By making a few changes, a clam-shell or orange-peel bucket may be used if desired instead of the usual bucket.

Standard sizes in this machine, as built by the Bucyrus Company, are given in Table 8.

The longer the boom the smaller is the capacity of the bucket for any given class. Some especially large machines have been built with booms up to 150 feet long and some handle buckets holding as much as 8 cu. yds. Their shipping weight ranges from 75 to 135 tons. Some are mounted on skids and rollers, which is considered standard; others on caterpillar traction, which eliminates the carrying of plank and rollers and facilitates moving over difficult ground; still others are mounted on trucks, of the four-wheel equalizing type. The second method is self-propelling, while with the other two, the machine usually pulls itself ahead by means of anchoring the bucket and then pulling on the dragline. The standard power is steam, but electric motors or even oil engines may be substituted if desired. Table 8 gives the abstract specifications, and Table 9 the working dimensions of standard Bucyrus dragline excavators.

TABLE 8.—ABSTRACT OF SPECIFICATIONS OF DRAGLINE EXCAVATORS

| | Class 7 | Class 9½ | Class 14 | Class 20 | Class 24 | Class 175 |
|---|---|---|---|---|---|---|
| Turn table, diameter | 7′ | 9½′ | 14′ | 20′ | 24′ | 24′ |
| Standard length boom | 42′ | 45′ | 60′ | 85′ | 100′ | 125′ |
| Standard size bucket | 1 yd. | 1¼ yd. | 2 yd. | 2½ yd. | 3½ yd. | 3½ yd. |
| Double cylinder main engines | 7″ × 8″ | 8″ × 8″ | 8″ × 10″ | 9″ × 12″ | 10½″ × 12″ | 12″ × 15″ |
| Digging rope diameter | 1″ | 1″ | 1⅛″ | 1¼″ | 1⅜″ | 1½″ |
| Dragline drum, pitch, diameter | 18″ | 21⅛″ | 26½″ | 28¼″ | 33″ | 32″ |
| Double cylinder swinging engines | 5″ × 6″ | 5″ × 5″ | 5½″ × 6″ | 7½″ × 7″ | 8″ × 8″ | 9″ × 9″ |
| Boom suspension | Permanent Adjustment | Drum Geared to Main Engine | | | | |
| Boiler, dimensions | 48″ × 9′ 7″ | 50″ × 10′ 0″ | 54″ × 9′ 7″ | 54″ × 13′ 7″ | 58″ × 15′ | 64″ × 15′ 10″ |
| Boiler, feed | 2 Injectors | 1 Injector, 1 Duplex Steam Pump | | | | |
| Water tank, capacity | 400 U. S. gal. | 450 U. S. gal. | 570 U. S. gal. | 850 U. S. gal. | 1100 U. S. gal. | 2500 U. S. gal. |
| Shipping weight, skid mounting | 30 tons | 35 tons | 56 tons | 80 tons | 115 tons | 157 tons |
| Approximate additional weight of special mounting | Caterpillars 9 tons | Caterpillars 17 tons | Caterpillars 29 tons | Propelling trucks 15 tons | Propelling trucks 18 tons | Propelling trucks 16 tons |
| Packed for export shipment. Approximate gross weight | 34 tons | 38 tons | 54 tons | 84 tons | 120 tons | 163 tons |

NOTE—Domestic shipping weight does not include counterweight on Class 20, 24, or 175 machines. Export weight does not include counterweight on any size, or panelled house on Class 20, Class 24 or Class 175.

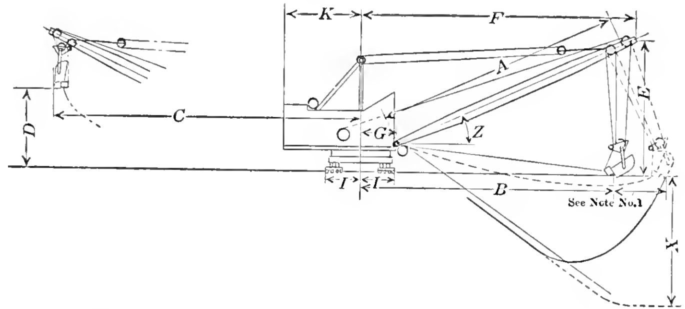



TABLE 9.—WORKING DIMENSIONS OF DRAGLINE EXCAVATORS

| Measurements Boom—A | Class 7 42 feet | | Class 9½ 45 feet | | Class 14 60 feet | | Class 20 85 feet | | Class 24 100 feet | | Class 175 125 feet | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Angle—Z | 25° | 40° | 25° | 40° | 25° | 40° | 25° | 40° | 25° | 40° | 25° | 40° |
| B | 40′– 0″ | 34′– 0″ | 43′–6″ | 37′– 6″ | 61′–8″ | 53′–0″ | 87′–9″ | 75′– 9″ | 104′–0″ | 90′– 0″ | 125′–3″ | 109′–0″ |
| C | 42′– 0″ | 36′– 0″ | 48′–6″ | 40′– 6″ | 65′–9″ | 58′–0″ | 92′–2″ | 80′– 0″ | 108′–6″ | 94′– 7″ | 129′–9″ | 113′–8″ |
| D | 10′– 6″ | 19′– 8″ | 13′–6″ | 23′– 6″ | 15′–6″ | 28′–0″ | 26′–6″ | 45′– 4″ | 28′–8″ | 50′–10″ | 40′–4″ | 67′–4″ |
| E | 23′– 0″ | 32′– 7″ | 27′–9″ | 38′–3″ | 34′–0″ | 47′–9″ | 45′–9″ | 65′– 1″ | 51′–3″ | 74′– 0″ | 62′–0″ | 90′–4″ |
| F | 43′– 6″ | 37′– 6″ | 49′–6″ | 42′–11″ | 67′–2″ | 58′–6″ | 92′–0″ | 79′–11″ | 108′–4″ | 94′– 5″ | 130′–2″ | 113′–0″ |
| G | 4′– 2½″ | 4′– 2½″ | 7′–2″ | 7′– 2″ | 8′–8″ | 8′–8″ | 11′–7″ | 11′– 7″ | 13′–2″ | 13′– 2″ | 12′–10″ | 12′–10″ |
| I | 6′–10″ | 6′–10″ | 6′–9″ | 6′– 9″ | 8′–5″ | 8′–5″ | 11′–9″ | 11′– 9″ | 13′–5″ | 13′– 5″ | 13′–10″ | 13′–10″ |
| K | 14′–10⅝″ | 14′–10⅝″ | 17′–8″ | 17′– 8″ | 19′–1″ | 19′–1″ | 23′–6″ | 23′– 6″ | 27′–2″ | 27′– 2″ | 27′–2″ | 27′–2″ |
| (Note 2) X | 14′–18′ | | 15′–18′ | | 20′–25′ | | 30′–35′ | | 35′–40′ | | 45′–50′ | |
| (Note 3) Y | 22′– 6″ | 14′ | 20′ | 13′ | 32′ | 20′ | 58′ | 40′ | 58′ | 36′ | 69′ | 43′ |

NOTE 1—A good operator can throw the bucket from 10 to 40 feet beyond the end of the boom depending upon the size of the machine and the conditions under which it is working.

NOTE 2—The digging depth "X" does not conform to any fixed rule which can be drawn out. It increases with the length of boom, but varies greatly with the material being dug. In neavy clay the machines will dig as shown in the table. Under conditions where they can be held close to the edge of the bank and where the material fills the bucket easily these figures may b considerably larger. The sketch shows in a general way how digging from the side and from the end of a cut may affect the digging depth.

NOTE 3—"Y" is the depth below level of machine to which the standard length of hoisting rope will allow the bucket to reach.

The dragline excavator, under favorable conditions, will give a good account of itself, but it generally occupies a different field from that held by the power shovel.

In concluding this chapter of description, it may be said that while other types of excavators are more suitable for certain special conditions, the power shovel is to date by far the most suitable and efficient excavator for the greatest range of big excavation work.

# CHAPTER II

## MECHANICAL EQUIPMENT

### GENERAL CONSIDERATIONS GOVERNING THE SELECTION OF EQUIPMENT

Considering in a broad way the selection of equipment for open-pit mining, the factors of first importance are; the magnitude, output and probable life of the undertaking; the supply and character of the labor; the form of power to be employed and the conditions under which it will be obtained and operate; the amount of capital available or justifiable to expend; the possibilities of increase or fluctuations in the scale of operations; the actual working conditions expected to be met with in the pit—such as height and width of benches, grades and curvature of trackage, disposal of overburden and ore, class of material, climatic and topographic conditions, water supply and drainage; cost and conditions under which various supplies are obtainable and under which product is sold; any conditions relating to time allowance for work, delivery of product, or government laws to be complied with.

The actual operation of a shovel mine is largely a combination of mechanical and civil engineering. Much assistance in the selection of equipment can often be had by consulting with the engineers of the best operated open-pit mines and with the foremost manufacturers of shovel and other required equipment.

On a given job, it is as a rule better to adopt and then adhere to a make, type and size of machine, which may be called standard, than to install a number of different makes, types or sizes. Though experimentally interesting, different shovel designs add greatly to the stock of spare parts which must be kept on hand and cause some confusion in shop repair work and in interchanging the operating crews. The rule of standardization of equipment is applicable to almost all of the major equipment. If experimental work on equipment be undertaken, it should be done in a limited way and carefully watched, in which case the results may lead to progressive improvement and be of actual value. Operations carried on by others should be watched and analyzed.

It is sometimes possible to buy second-hand equipment, discarded from government or railroad work. If, on careful inspection, such equipment be found in good condition and well adapted to the requirements, some economy can be made on the first cost; but if the work be large and will cover a long time it is generally more economical in the long run to purchase new and efficient machinery fully adapted to the work.

It is an impressive fact that the general trend of practically all of the equipment is ever towards heavier and larger machines. First cost of equipment is less important than the unit cost of excavation.

The various equipment units will be considered individually and collectively.

## SHOVELS

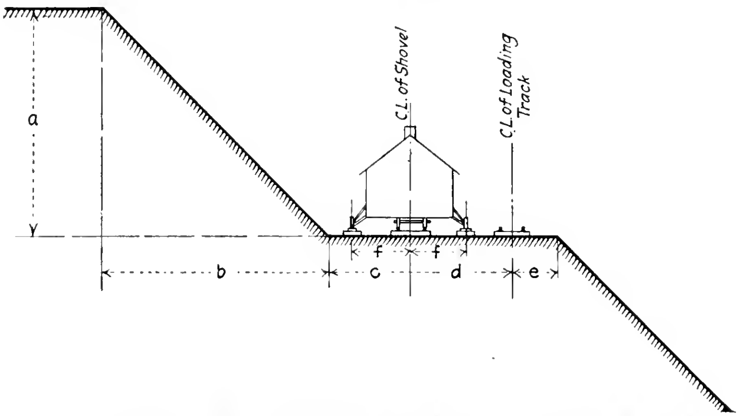
**Principle Governing Factors.**—The best shovel to select for a given piece of work will depend on the amount and character of material to be excavated, its disposal, the height and width of the cuts or benches, the class of operating labor available and the class of power that is most economical.

**Pit Working Conditions.**—If the excavation is to be a large one with deep cuts, heavy expensive shovels are justified because of their greater output and cheaper unit costs. A shovel excavating 1200 cu. yd. per shift will show a unit operating cost about two-thirds that of a shovel excavating only 800 cu. yds. per shift, assuming that efficient train service be afforded for the prompt loading and supplying of cars. If the material be hard and inclined to break coarse or if the shovel be expected to dig moderately compact material without the aid of blasting, a heavy rugged shovel with medium-sized dipper should give the best results. If the material be soft or easy to dig a larger dipper may be employed, and if the train service be good a fast loading shovel, such as the Atlantic, may give the greatest output. If the cuts are to be very light a small shovel may be more economical as it is more easily moved.

If working conditions permit, the weight of shovel, length of boom and length of dipper-stick should bear an approximate relationship to the height and width of the banks and benches it is proposed to carry. A working example of this relationship for straight bank-loading is shown on Fig. 12.[1] In box or

[1] Nevada Consolidated Copper Co., Dec. 15, 1915.

"thorough-cutting," or excavating below the loading-track, the relationship is worked out as shown in Fig. 13. As a rule, the best height of bank is about equal to the width of cut which the shovel can take, thus, with a 34-ft. boom, the cut will be about 45 ft. and the height of bank about 45 ft.; with a 40-ft. boom these dimensions can be increased to say 60 ft. More will be said

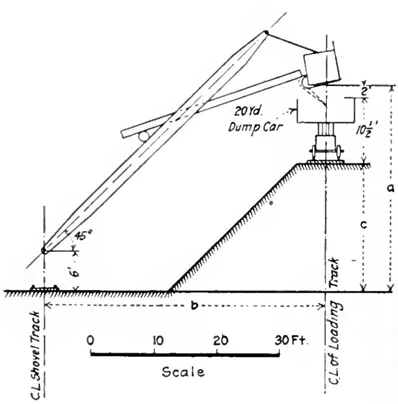



| Shovel | Weight, tons | Class | Length boom, ft. | Length dipper-stick, ft. | *a* | *b* | *c* | *d* | *e* | *f* |
|---|---|---|---|---|---|---|---|---|---|---|
| *G*-1 | 95 | *C* | 32 | 22½ | 61′ | 68′ | 18′ | 27′ | 80′ | 11′ |
| 1 | 100 | *B* | 34 | 24 | 45′ | 45′ | 22′ | 29′ | 43′ | 11′ |
| 2 | 70 | *B* | 28 | 18 | 50′ | 50′ | 19½′ | 24′ | 75′ | 11′ |
| 3 | 100 | *B* | 32 | 24 | 68′ | 73′ | 17′ | 31′ | 87′ | 11′ |
| 4 | 100 | *B* | 32 | 24 | 49′ | 41′ | 19′ | 30′ | 75′ | 11′ |
| 5 | 100 | *C* | 32 | 24 | 54′ | 54′ | 18′ | 32′ | 6′ | 11′ |
| 6 | 100 | *C* | 32 | 22½ | 48′ | 40′ | 20′ | 30′ | 6′ | 11′ |
| 7 | 100 | *C* | 32 | 24 | 46′ | 40′ | 18′ | 34′ | 8′ | 11′ |
| 8 | 100 | *C* | 34 | 24 | 60′ | 61′ | 21′ | 31′ | 11′ | 11′ |

FIG. 12.—Working example of shovel benches bank loading.

later as to the best height of banks, but if this point is decided then it must be kept in mind when drawing up shovel specifications. Practical working conditions very often cause this relationship to be totally disregarded but the results will then be less satisfactory.

**Class of Operating Labor.**—If the operating labor be inexperienced and careless and the digging difficult, the idea employed in the construction of some shovels, by which the engines are so designed that they will stall before breaking something, is a very good one. In the hands of such a runner the repairs may be much less on a shovel so designed. In the hands of a first-class runner the more powerful engines employed on other designs should give equally good results and perhaps show a greater capacity. The idea employed in still other shovels of exerting a horizontal and pivotal motion to the dipper gives very satisfactory results though these shovels are of comparatively smaller capacity.




| BOOM | DIPPER STICK | a | b | a WITH BLOCKS | c | c WITH BLOCKS |
|---|---|---|---|---|---|---|
| 50 FT. | 30 FT. | 33 FT. | 45 FT. | 41 FT. | 20½ FT. | 28½ FT. |
| 49 | 29½ | 32½ | 44 | 40½ | 20 | 28 |
| 48 | 29 | 31½ | 43 | 39½ | 19 | 27 |
| 47 | 28 | 31 | 42½ | 39 | 18½ | 26½ |
| 46 | 27½ | 30½ | 41½ | 38½ | 18 | 26 |
| 45 | 27 | 29½ | 40½ | 37½ | 17 | 25 |
| 44 | 26½ | 29 | 39½ | 37 | 16½ | 24½ |
| 43 | 26 | 28½ | 38½ | 36½ | 16 | 24 |
| 42 | 25 | 27½ | 38 | 35½ | 15 | 23 |
| 41 | 24½ | 27 | 37 | 35 | 14½ | 22½ |
| 40 | 24 | 26½ | 36 | 34½ | 14 | 22 |
| 39 | 23½ | 25½ | 35 | 33½ | 13 | 21 |
| 38 | 23 | 25 | 34 | 33 | 12½ | 20½ |
| 37 | 22 | 24½ | 33½ | 32½ | 12 | 20 |
| 36 | 21½ | 23½ | 32½ | 31½ | 11 | 19 |
| 35 | 21 | 23 | 31½ | 31 | 10½ | 18½ |
| 34 | 20½ | 22½ | 30½ | 30½ | 10 | 18 |
| 33 | 20 | 21½ | 29½ | 29½ | 9 | 17 |
| 32 | 19 | 21 | 29 | 29 | 8½ | 16½ |
| 31 | 18½ | 20½ | 28 | 28½ | 8 | 16 |
| 30 | 18 | 19½ | 27 | 27½ | 7 | 15 |
| 29 | 17½ | 19 | 26 | 27 | 6½ | 14½ |
| 28 | 17 | 18½ | 25 | 26½ | 6 | 14 |
| 27 | 16 | 17½ | 24½ | 25½ | 5 | 13 |
| 26 | 15½ | 17 | 23½ | 25 | 4½ | 12½ |
| 25 | 15 | 16½ | 22½ | 24½ | 4 | 12 |

FIG. 13.—Reach of shovels in thorough-cutting.

**Power.** *Steam.*—The power to be adopted for pit equipment may be steam, electricity, compressed air, or oil engines. The first method is the oldest, simplest, best understood by the average operator and probably the most dependable. The firing may be done with wood, coal or fuel oil, depending on the relative economy as applied to each particular installation. There are disadvantages sometimes found in the high cost of fuel, additional operating force required, difficulty in securing good boiler feed-water and in keeping the water supply-lines open in severe winter weather. Steam is still to be found in the great majority of installations, and unless other power can be shown to have decided advantages over steam, it is to be recommended.

*Electric.*—Electric power has been coming more and more into favor because of its economy and convenience. Perfecting the motors and the control and the cheaper cost of power have been responsible for this. Comparative operating costs of steam versus electric shovels were given in Chap. I., and were shown to depend largely on the relative costs of coal and electricity, though considerable operating labor is also dispensed with on the electric shovels. As the electric shovel has many of its moving parts rotary instead of reciprocating, the wear and tear on these is less, also boiler troubles are eliminated.[1]

A study of the relative cost of shovels operated respectively by steam and by direct and alternating current was given by Rogers[2].

This study was based on assumptions as to shovel output, capital cost, interest, amortization, labor, etc., and his conclusion is that the direct current shovel will be always cheaper than steam, but that saving for an A-C shovel over steam operation may come only from power saving, which of course will depend in turn on the relative cost of fuel and electric power.

The 103-C shovels used at Chuquicamata are operated by alternating current and require from .3 KWH to .7 KWH per ton according to the character of the digging, which is always, however, in rock. About 25 per cent. of this power is used in clearing the pit while waiting for cars.

They are fed at 5,000 volts, 3-phase, step-down transformers being located on each shovel.

The equipment on the latest shovels consists of one 250 h.p. 500 volt hoist motor and 100 h.p. thrust and swing motors with three 100 K.W. transformers. The control is partly automatic, consisting of contactors controlled by overload and jamming relays. The shovels are equipped with 4-cu. yd. buckets and can make 20-second cycles under favorable conditions.

Actual costs for these electric shovels show that labor and repairs cover about 80 per cent. of the operating cost, cost of power being only 10 per cent., with current at 2½ c. a KWH. Steam operating costs under similar conditions show a total more than double, though with very high coal and oil costs.

[1] Delays on account of shovel repairs on a Bucyrus 40-R electric shovel amounted to 3½ per cent. of the total time, and only a small part of this was spent in attendance on the motors and electrical equipment. Location, Granby Mine, Phoenix, B.C.

[2] Rogers, H. W.: The Application of Electric Motors to Shovels, T.A.-I.M.E., Feb., 1914, pp. 300–309.

The steam shovel repairs were 150 per cent. of the electric shovel repairs and the labor somewhat higher than with the electric shovel.

The A.C. electric shovels cost approximately 50 per cent. more than the steam. As to whether the direct-current or alternating-current equipment is preferable, it was previously stated that considerable difference of opinion exists. Those advocating the latter contend that it is superior on account of greater mechanical simplicity and because the characteristics of the wound-rotor induction-motors are better adapted to shovel service than are the characteristics of the series motors. Both the direct-current series motor and the wound-rotor induction motor, when provided with proper resistance in the circuits, will nearly approach the performance of the steam engine; both have good starting torque, quick acceleration under light loads, and reduced speed under heavy loads. The direct-current control is somewhat simpler but the alternating-current control, with the resistance automatically cut in and out by solenoid switch control, is now very satisfactory. The rotors will either start or stall at maximum torque, and automatic jamming relays prevent the current from reaching the "break-down" point.

It is desirable to keep the rotor inertia small, and for that reason two motors may, if necessary, be placed on the hoist instead of one larger one, but it is questionable whether except in the case of the largest machines the saving in power and time are not more than offset by the necessary addition of a motor and other parts.

With the direct-current motors a motor-generator set will usually be required, but with the alternating-current motors three stationery transformers is common practice. The latter arrangement makes for simplicity by substituting slip-rings for commutators, and stationary transformers for the rotaries. Under the severe conditions of jar and grit and dust found in all shovel service, this point is noteworthy. It is true that a larger excess in motor capacity must be provided for when using alternating-current. This may range from twenty five to forty five per cent.

It is not the intention to discuss here in great detail such subjects as are better left for expert argument, but from the foregoing and from the fact that a large majority of the most modern large electric excavators have been equipped with alternating-current motors, and that these are giving excellent satisfaction, it is judged that they are found in practice to be preferable.

*Compressed Air.*—Compressed air can be used for operating shovels in the same way that steam is employed and with the same engines and their mechanical advantages. Air is in many ways much superior to steam. It is cleaner and does away with boiler and water troubles, at least at the shovel; it requires no fireman, no coal passer, no watchman and no fuel-teaming. When electric power is used to operate shovels, a competent electrician must always be kept available, and this will be somewhat of an added expense in small installations. Compressed air can be transmitted long distances without any appreciable pressure loss, provided the lines are of ample size and kept tight. With electricity there are considerable losses in line transmission and in transformers or motor-generator sets, while the losses by condensation in long steam lines are usually heavy. A compressor plant can be located at a convenient point for handling coal and water for the boilers, if it be steam-driven, or at some central point if operated electrically. The air will then be distributed through mains to the shovels and may also be used for driving rock-drills for blasting, and almost any other auxiliary machinery. Why it is not used more in excavations of great magnitude is probably due to prejudice, the expense of installation of the plant and mains, and the expense of maintaining the pipe-lines. There is considerable expense in winter in maintaining water lines to steam-shovels and it is doubtful if the air lines would be any more expensive. The maintenance of electric lines is much simpler and comparatively light. It is probable that there is a field which may be more fully developed, for air driven shovels, but at present there is little information about them because of their very limited use.

*Oil Engines.*—Oil engines are used on some of the smaller shovels. These engines are efficient, but unless other sources of power are unattractive the internal combustion engine is not often chosen. They too, however, offer a field for future development.

The following is an example[1] of the comparative operating costs of steam and gasoline-driven, 18-ton revolving shovels.

[1] Thew Automatic Shovel Company—Shovel Capacity rating 40 to 50 cu. yd. per hour.

| | Per Shift | |
|---|---|---|
| | Steam | Gasoline |
| Operator | $5.00 | $5.00 |
| Fireman | 2.50 | .... |
| Two laborers in pit | 3.50 | 3.50 |
| Fuel—1000 to 1200 lb. coal | 2.00 | .... |
| 20 gallons gasoline | .... | 2.50 |
| Oil, supplies and repairs | 1.00 | 1.00 |
| Total (exclusive of hauling coal or water) | $14.00 | $12.00 |

Mechanically the operation is quite satisfactory. The boiler and steam engine, or a twenty horse-power electric motor, is replaced by a thirty-five horse-power gasoline engine.

The advantages claimed, over steam, for shovels driven by oil-engines are; that the water consumption is negligible and in cold weather a freezing mixture can be used; in some places the cost of fuel is reduced as gasoline or distillate, being a more concentrated fuel, can be more cheaply transported; there is an economy in labor, as no fireman or coal and water teaming is required; the power is quickly available and quickly shut off; there are no boiler troubles nor inspections; and a licensed engineer is not required.

The electro-hydraulic shovel was described in Chap. I. It is still in the experimental stage but has some very attractive possibilities.

**Competing Machines.**—In Chap. I the various types of excavators, which may be classed as competitors, were briefly described and their fields of usefulness were outlined Among these was mentioned the dragline excavator. It is a rare case where the dragline is able to compete successfully with the large revolving shovel for ordinary stripping purposes. For some work, as stated, the dragline is better adapted, but for stripping, the large shovels with long booms are usually superior. The large revolving shovels are built so that they can be converted into the dragline machines by making a few alterations, thus making them suitable for both kinds of work.

**Cost of Shovels.**—The following will give a general idea of the cost of shovel equipment early in 1917, (f.o.b. eastern factories).

*Standard Railroad Shovels.*—The prices below cover machines on standard mounting, that is, railroad trucks under shovels.

The prices for electric machines are for motors operating on 3 phase, 60 cycle, 440 volt alternating-current. For other types of alternating-current there is an additional charge of about 3 to 4 per cent., and for machines operated by direct-current motors, an additional charge of about 5 per cent.

| Domestic shipping weight, lb. | Working weight, lb. | Standard dipper, cu. yd. | Rated capacity in cu. yd. per hr. | Approximate cost | |
|---|---|---|---|---|---|
| | | | | Steam | Electric |
| 48,000 | 54,000 | 3/4 | 30 to 90 | $8,000 | |
| 81,000 | 95,000 | 1 1/2 | 60 to 180 | 10,200 | |
| 118,000 | 139,000 | 2 | 80 to 140 | 13,000 | |
| 141,000 | 166,000 | 2 1/2 | 100 to 300 | 14,300 | $22,000 |
| 160,000 | 188,000 | 3 1/4 | 120 to 350 | 16,100 | 25,500 |
| 188,000 | 220,000 | 4 | 150 to 400 | 18,200 | 28,500 |
| 210,000 | 246,000 | 5 | 200 to 500 | 19,700 | 30,750 |
| 235,000 | 276,000 | 6 | 250 to 550 | 21,500 | 34,500 |

*Large Revolving Shovels.*—The prices of these large stripping shovels, steam driven, ranges from $23,000 to $55,000 according to their size and weight. Their shipping weight runs from 250,000 lb. to 500,000 lb. The largest of these, when electric driven, costs about $67,600.

*Dragline Excavators.*—The prices below cover machines with standard mounting, that is skids and rollers, under all except the last one, which is mounted on propelling equalizing trucks.

The electric machines are for the same class of current as given for the shovels, and changes in current specifications add to the prices in the same way.

| Domestic shipping weight, tons | Standard boom length, ft. | Standard bucket capacity, cu. yd. | Diameter turntable, ft. | Approximate cost | |
|---|---|---|---|---|---|
| | | | | Steam | Electric |
| 35* | 45 | 1 1/4 | 9 1/2 | $10,000 | $13,000 |
| 55* | 60 | 2 | 14 | 13,750 | 17,250 |
| 93* | 85 | 2 1/2 | 20 | 20,350 | 24,000 |
| 112# | 100 | 3 1/2 | 24 | 25,850 | 30,000 |
| 180# | 125 | 3 1/2 | 24 | 36,850 | 43,175 |

* Including counterweight.

The combination shovel-dragline machines cost a little more than the draglines.

*Light Shovels.*—The Thew shovel, type 0, weighs 36,000 lb., carries a $\frac{3}{4}$-cu. yd. dipper and has a rated capacity of 40 to 50 cu. yd. per hour. Equipped with steam power it sold for about $4950, with gasoline power for about $5050.

## LOCOMOTIVES

**General Governing Condition.**—On all large excavations, traction by animal power has been replaced, where possible, by mechanical power. In some cases where a gradual approach is not possible, or where the pit is so deep that hauling on gradients is not economical, inclined planes with hoists are provided; in other cases material may be hoisted through vertical shafts placed outside of, but connected by adits with the pit area. For the great majority of open-pit workings, however, some class of locomotive is employed to haul trains out over graded road-beds.

The selection of locomotives for such service is governed by the class of power or fuel adopted: by the length, gauge, grades and curvature of the track; and by the weight of trains and output requirements. Modern practice is constantly toward heavier and more efficient units and it is always desirable to provide a reasonable amount of surplus power, so that the locomotive will not constantly be worked to full capacity. Reserve power is economical because it reduces repair costs, fuel and oil, lengthens the working life of the motive power and provides a reasonable reserve for emergencies.

It pays well to buy locomotives of proper design to meet requirements and to keep them in good order. In such service as mining, where the track must be frequently shifted, ideal conditions are generally impractical, but bad grades, sharp curves, neglected road-bed and track and run-down rolling-stock should be thoughtfully studied and avoided wherever practicable.

The majority of locomotives now in this service are steam-driven and coal-fired, though wood or oil may be substituted. Most of these are of the rod or direct-acting type, either with water-tank and fuel-bunker mounted on the locomotive, or trailed on a tender. For heavier grades than are negotiable with rod locomotives the Shay or Hesler type of geared locomotives is sometimes employed. It should be remembered, however, that in theory geared locomotives can haul no heavier trains nor climb steeper grades than direct-acting locomotives of the same weight with all of their weight on the driving wheels, and carrying no separate tenders. In other words, considering a locomotive of each of these types, both having the same weight on the driving wheels, and both properly designed, they will start the same loads, because the rail adhesion is the limiting factor in both cases. It is a fact, however, that the direct-acting locomotive has more tendency to slip its wheels in starting trains because of the position of the crank-pins. On the other hand, the geared locomotive will make less mileage and handle less tonnage per day, and has less advantage from train momentum in overcoming grades because of its speed and motion. The torque of the geared machine is more even and better distributed in starting, but the introduction of the gearing is in other ways a disadvantage.

Compressed air and electric locomotives are used in some places and will be described later.

**Direct-connected Steam Type.**—As previously mentioned, the majority of locomotives now employed in open-pit mining is of this type.

Compounding permits the steam to expand through a greater range of pressure than is possible in a single cylinder, and also reduces the amount of temperature change and consequent condensation in each cylinder, thus economizing fuel. But in this class of equipment it is not often resorted to.

Superheating is another method used to avoid cylinder condensation with its loss in efficiency. By superheating steam with a fire-tube superheater, the steam is ordinarily raised about 200°F. above the temperature of saturation, and in this state condensaton in the cylinders is not only largely avoided, but there is also another gain in efficiency, because the volume per pound of superheated steam is greater than that of saturated steam at the same pressure, and hence each pound of water evaporated forms a larger volume of steam, which means that fewer pounds of steam are required to fill the cylinders.

The economies resulting from compounding or superheating are best realized in locomotives working at high power for sustained periods of time, such as on long heavy mountain grades

As these conditions are not usual in open-pit work, the single expansion locomotive without superheater is generally considered best. It is therefore in this class of engine that we are interested in the following discussion.

*Determination of Tractive Force.*—The hauling capacity of a locomotive is determined by the relation between the tractive force developed and the resistance of the train, and both of these factors are dependent on the speed. At starting speeds, a locomotive will usually develop at the rims of the driving wheels its rated tractive force. This is calculated from the dimensions of the engine by the following formula:[1]

$$T = \frac{C^2 \times L \times 0.85P}{D}$$

Where $T$ = the rated tractive force at rim of drivers in pounds.
$C$ = the diameter of the cylinders in inches.
$L$ = the length of stroke of the pistons in inches.
$P$ = the boiler pressure in pounds per square inch.
$D$ = the diameter of the driving-wheels in inches.

*0.85* = 85 per cent of the boiler pressure in pounds per square inch. (From tests made, it has been found that well designed locomotives lose about 15 per cent, on account of drop of steam pressure between boiler and cylinders, due to condensation of steam, to leakage and to internal friction of the locomotive machinery. This is when the machine is working at full-stroke and at a speed not in excess of the speed at which it can haul its heaviest load).

*Factor of Adhesion.*—The factor of adhesion of a locomotive is the ratio between the working order weight on the driving-wheels and the tractive force. and hence is found by dividing the weight on the driving-wheels by the tractive force. If this factor is too high the locomotive cannot slip its driving-wheels and is called "logy," and if the factor is too low, the driving-wheels will slip too easily and the engine will be called "slippery." For saddle-tank locomotives carrying water and fuel wholly or partly over the driving-wheels, this factor should be figured on the basis of the tank and bunker being only about half full. In a well designed locomotive the cylinders will be so proportioned that there will be sufficient power to just over-

[1] Most locomotive builders will supply Tractive Force Tables covering their product.

come the adhesion on good clean dry rails. For this class of service the factor of adhesion may be taken at from 4.70 to 4.80.

*Draw-Bar Pull.*—The draw-bar pull of a locomotive is simply the tractive force minus the power required to move the locomotive itself. It is the net power available for pulling the load attached to the locomotive, and is a variable quantity because the amount of power required to move the locomotive itself, under different conditions of grade and track, is variable.

*Resistance due to Grades.*—When a train is hauled up-grade, there is the resistance due to lifting the train against gravity. When the grade is stated in per cent (or number of feet rise per hundred feet of haul), this resistance equals 20 lb. per ton of 2000 lb. for each per cent. If the grade is stated in feet per mile, the resistance per ton of 2000 lb. will be 0.3788 lb. per foot of rise per mile. Grades generally account for the greatest percentage of the total resistance.

*Resistance due to Rolling Friction*—The resistance due to rolling friction, or coefficient of rolling friction, varies greatly with the character and condition of the rolling-stock and track. Poorly laid track and crooked rails increase the resistance indefinitely. The resistance is increased by overloading the cars, although the resistance per ton hauled is less for properly loaded cars than for empties. In cold weather the resistance is greater. Much care should be given to lubrication in all seasons. It may be taken, however, that the following resistances per ton of 2000 pounds will be about averages.

| | |
|---|---|
| With extra good cars and track.................. | 5 to 6½ pounds |
| For reasonably good conditions.................. | 8 to 12 pounds |
| For bad cars and track.......................... | 20 to 40 pounds |
| For hard-running cars and very rough track........ | 60 to 80 pounds plus. |

For cars with wheels fast on axles and suitable bearings and oil-boxes such as are used in pit service, this resistance should not exceed 8 to 12 pounds, and 10 pounds may be taken as a general average. This is equivalent to a grade of 0.5 per cent. *i.e.* a car, once started, that will just keep in motion on a 0.5 per cent. down-grade would have such a resistance. From this it will be seen that poorly built cars and bad track are costly to contend with.

It should be mentioned that this resistance is not constant but varies with the speed, acceleration and weight of the cars hauled; however the foregoing assumptions will be approximately correct for this class of work.

*Resistance due to Curves.*—A curve is said to have a radius of so many feet, or degrees. By the former is meant that the center line of the track is described as an arc of a circle having a radius of so many feet; the latter expresses the angular deflection from the tangent measured at stations 100 ft apart; *i.e.* the number of degrees of central angle subtended by a chord of 100 feet represents the "degree curve". As one degree of curvature is equal to a radius of 5730 feet, the number of degrees divided into 5730 gives the radius in feet; or the number of feet radius divided into 5730 gives the number of degrees. This rule is sufficiently accurate for curves up to 15 degrees.

The resistance due to curves is considerable but extremely variable. The shorter the radius, the longer the wheel-bases of the locomotive and cars, the greater the speed, the greater the length of the train on the curve and the greater the length of the curved track, the greater will this resistance be. The elevation of the outer rail, the condition of the track and rolling stock, the weight of the cars and the proper widening of the track gauge to prevent the wheels from binding against the rails are other factors which enter into this resistance. Excessive or irregular curves, and very sharp curves, especially on steep grades, are obviously to be avoided if the locomotive is to pull efficiently and if wear and tear on the track and rolling stock is to be kept down.

It is generally assumed that this resistance amounts to from 0.7 to 1.0 pound per ton per degree of curvature, the lower figure being used for large capacity cars and the higher figure for small capacity cars, as in the latter case there are more wheels per ton of weight than in the former.

Because of this resistance it is customary, when a curve occurs on a grade, to reduce the grade of the curved part of the track so that the combined resistance of the lighter grade and curve will not exceed the resistance of the heavier grade on the straight part of the track. This is called compensation and many roads allow 0.035 per cent in grade for each degree of curve. For sharp curves and long wheel-bases this may be increased to 0.05 per cent per degree. This resistance is usually nearly twice as great for the locomotive as for the cars because of the longer wheel base of the former.

To reduce binding, the usual amount of clearance between the rail and wheel flanges is increased on curves. On curves up to eight degrees widening may be omitted,but for each two degrees or fraction there of over eight degrees, the gauge should be widened $\frac{1}{8}$ inch until a maximum of 4 feet $9\frac{1}{4}$ inches is reached for standard gauge tracks.

Elevating the outer rail on curves is desirable to counteract the centrifugal force tending to tip over the rolling stock. It is customary to elevate the outer rail about $\frac{1}{2}$ inch per degree of curvature for train speeds of 25 to 35 miles per hour, but this elevation should not exceed, say, eight inches.

*Hauling Capacity of Locomotives.*—From the foregoing can be determined the hauling capacity of any locomotive, as it is simply the tractive force of the locomotive divided by the sum of the resistances due to gravity, rolling fiction and curvature, minus the weight of the engine (and tender, if any). Expressed as a formula this is:

$$H = \frac{T}{G + R + C} - E$$

Where $H$ = the hauling capacity in tons of 2000 lbs.
$T$ = the tractive force of the locomotive in pounds.
$G$ = the resistance of gravity in pounds per ton.
$R$ = the resistance of rolling friction in pounds per ton
$C$ = the resistance of curvature friction in pounds per ton
$E$ = the weight of the locomotive in tons.

*Example.*—What is the hauling capacity of a saddle-tank locomotive weighing 65 tons, with tank and bunker half full, all the weight on the six driving wheels, cylinders 18 inches in diameter with 24-inch stroke, driving wheels 44 inches in diameter, boiler pressure 175 lbs. per sq. in., operating on a 3 per cent. grade, 10° curve, and rolling friction of cars 10 pounds per ton.

Substituting in the formula for Tractive Force, we have

$$T = \frac{18^2 \times 24 \times 0.85 \times 175}{44} = 26{,}288 \text{ lbs.}$$

Assuming a factor of adhesion of 4.7 gives a force of adhesion of

$$\frac{130{,}000 \text{ lbs.}}{4.7} = 27{,}659 \text{ pounds.}$$

The locomotive will therefore be able to exert its full tractive force before slipping. In the formula for hauling capacity

$G$ will equal 60 pounds
$R$ will equal 10 pounds
$C$ will equal, say, 0.8 pounds $\times$ 10, or 8 pounds.
$E$ will be 65 tons

Then

$$H = \frac{26,288}{60 + 10 + 8} - 65 = 272 \text{ tons as the hauling capacity.}$$

Further, if it be assumed that the load is to consist of loaded cars, weighing 20 tons each and carrying 50 tons of material or a total weight of 70 tons, this locomotive should start

$$\frac{272}{70} = 3.88 \text{ loads.}$$

If the rolling friction resistance rate is assumed to be greater for the locomotive than that assumed for the cars, (as is usually the case), then the hauling capacity may be computed by first deducting from the tractive force the total resistance to be overcome by the locomotive in handling itself under the above conditions and then dividing the remainder, or draw-bar pull, by the sum of the rates of the various resistances assumed for the cars.

It will be noted that the foregoing is really the starting load, since it takes no account of the speed at which the load is to be hauled, and as the prolonged tractive force depends on the speed and heating surface of the boiler, this would have to be modified for a prolonged effort at moderate speed. At 15 miles per hour probably only 85 per cent. of the starting tractive effort could be maintained for any considerable length of time, but for short hauls it would not be reduced more than a few per cent. and should haul any load it can start.

Under actual working test it was found that this locomotive would pull four of these cars up a 3 per cent. grade and two up a 4 per cent. grade, though by double-heading, two of these locomotives would pull five loads up a 4 per cent. grade. Part of the track was on about 10° curves, and the speed of the trains was from 8 to 10 miles an hour at the beginning of the grade. On account of the drop in boiler pressure under this pull, the locomotives could only handle these loads for a distance of from 800 to 1000 feet.

TABLE 10.—SHOWING HOW MANY TIMES ITS WEIGHT ON DRIVING-WHEELS

Any locomotive can haul in addition to itself on straight track up various grades, at different allowances for frictional resistances. To be on the safe side the tractive power is assumed to be equal to one-fifth of the weight on the driving-wheels. This table is approximate only since the usual factor of adhesion is from about 4.25 to about 4.75 instead of 5. The figures in this table have a margin of about 5 to 10 per cent. on the safe side. Speed s not taken into account. Tenders and weight on pony trucks must be considered as if part of train weight.

| Number of times its own weight on driving-wheels any locomotive can haul | Frictional resistance of train in pounds per ton of 2,000 pounds | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 6½ | 7 | 8 | 9 | 10 | 11 | 12 | 15 | 20 | 25 | 30 | 40 |
| On absolute level | 60.5 | 56.1 | 49 | 43.4 | 39 | 35.4 | 32.3 | 25.7 | 19 | 15 | 12.3 | 9 |
| On grade 1 foot per mile | 57.1 | 53.2 | 46.7 | 41.6 | 37.5 | 34.1 | 31.3 | 25 | 18.6 | 14.7 | 12.1 | 8.9 |
| On grade 2 feet per mile | 54.1 | 50.5 | 44.7 | 40 | 36.1 | 33 | 30.3 | 24.4 | 18.2 | 14.5 | 12 | 8.8 |
| On grade 3 feet per mile | 51.3 | 48.1 | 42.7 | 38.4 | 34.9 | 32 | 29.4 | 23.6 | 17.9 | 14.3 | 11.8 | 8.7 |
| On grade 5 feet per mile | 46.6 | 43.9 | 39.4 | 35.5 | 32.6 | 30 | 27.7 | 22.6 | 17.2 | 13.8 | 11.5 | 8.5 |
| On grade 8 feet per mile | 40.4 | 38.8 | 35.2 | 32.2 | 29.6 | 27.4 | 25.5 | 21.1 | 16.3 | 13.2 | 11.1 | 8.2 |
| On grade 10 feet per mile | 37.9 | 36.1 | 33 | 30.3 | 28 | 26 | 24.3 | 20.3 | 15.8 | 12.8 | 10.8 | 8.1 |
| On grade 13²⁄₁₀ feet per mile— ¼ per cent. | 33.8 | 32.3 | 29.8 | 27.6 | 25.6 | 24 | 22.5 | 19 | 15 | 12.3 | 10.4 | 7.9 |
| On grade 15 feet per mile | 32.1 | 30.5 | 28.2 | 26.2 | 24.5 | 22.9 | 21.6 | 18.3 | 14.5 | 12 | 10.2 | 7.7 |
| On grade 20 feet per mile | 27.4 | 26.4 | 24.6 | 23.1 | 21.7 | 20.5 | 19.4 | 16.7 | 13.5 | 11.3 | 9.6 | 7.4 |
| On grade 25 feet per mile | 24 | 23.2 | 21.9 | 20.6 | 19.5 | 18.5 | 17.6 | 15.3 | 12.5 | 10.5 | 9.1 | 7.1 |
| On grade 26⁴⁄₁₀ feet per mile— ½ per cent. | 23.2 | 22.5 | 21.2 | 20 | 19 | 18 | 17.2 | 15 | 12.3 | 10.4 | 9 | 7 |
| On grade 30 feet per mile | 21.4 | 20.8 | 19.6 | 18.6 | 17.7 | 16.9 | 16.1 | 14.1 | 11.1 | 10 | 8.6 | 6.8 |
| On grade 35 feet per mile | 20.2 | 18.7 | 17.8 | 17 | 16.2 | 15.5 | 14.8 | 13.1 | 11 | 9.4 | 8.2 | 6.5 |
| On grade 39⁶⁄₁₀ feet per mile— ¾ per cent. | 17.6 | 17.2 | 16.4 | 15.7 | 15 | 14.4 | 13.8 | 12.3 | 10.4 | 9 | 7.9 | 6.25 |
| On grade 45 feet per mile | 16 | 15.6 | 15 | 14.3 | 13.7 | 13.2 | 12.7 | 11.4 | 9.8 | 8.5 | 7.5 | 6 |
| On grade 52⁸⁄₁₀ feet per mile— 1 per cent. | 14.1 | 13.8 | 13.3 | 12.8 | 12.3 | 11.9 | 11.5 | 10.4 | 9 | 7.9 | 7 | 5.7 |
| On grade 55 feet per mile | 13.6 | 13.3 | 12.8 | 12.3 | 11.9 | 11.5 | 11.1 | 10.1 | 8.8 | 7.7 | 6.8 | 5.5 |
| On grade 60 feet per mile | 12.6 | 12.4 | 12 | 11.6 | 11.2 | 10.8 | 10.5 | 9.6 | 8.3 | 7.3 | 6.6 | 5.7 |
| On grade 66 feet per mile— 1¼ per cent. | 11.7 | 11.5 | 11.1 | 10.8 | 10.4 | 10.1 | 9.8 | 9 | 7.9 | 7 | 6.25 | 5.15 |
| On grade 70 feet per mile | 11.1 | 10.9 | 10.6 | 10.2 | 9.9 | 9.6 | 9.4 | 8.6 | 7.6 | 6.7 | 6.1 | 5 |

| | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| On grade 79 2/10 feet per mile— 1½ per cent... | 9.9 | 9.8 | 9.5 | 9.2 | 9 | 8.7 | 8.5 | 7.9 | 7 | 6.25 | 5.7 | 4.7 |
| On grade 85 feet per mile........ | 9.3 | 9.2 | 8.9 | 8.7 | 8.5 | 8.2 | 8 | 7.5 | 6.7 | 6 | 5.4 | 4.5 |
| On grade 90 feet per mile........ | 8.9 | 8.7 | 8.5 | 8.3 | 8 | 7.8 | 7.7 | 7.2 | 6.4 | 5.7 | 5.2 | 4.4 |
| On grade 92 4/10 feet per mile— 1¾ per cent... | 8.6 | 8.5 | 8.3 | 8.1 | 7.9 | 7.7 | 7.5 | 7 | 6.25 | 5.7 | 5.15 | 4.3 |
| On grade 100 feet per mile........ | 8 | 7.9 | 7.7 | 7.5 | 7.3 | 7.2 | 7 | 6.5 | 5.9 | 5.3 | 4.9 | 4.1 |
| On grade 105 6/10 feet per mile— 2 per cent... | 7.6 | 7.5 | 7.3 | 7.1 | 7 | 6.8 | 6.7 | 6.25 | 5.7 | 5.15 | 4.7 | 4 |
| On grade 110 feet per mile........ | 7.3 | 7.2 | 7 | 6.8 | 6.7 | 6.6 | 6.5 | 6 | 5.5 | 5.1 | 4.6 | 3.9 |
| On grade 118 8/10 feet per mile— 2¼ per cent... | 6.77 | 6.7 | 6.5 | 6.4 | 6.25 | 6.1 | 6 | 5.7 | 5.15 | 4.7 | 4.3 | 3.7 |
| On grade 120 feet per mile........ | 6.7 | 6.6 | 6.5 | 6.3 | 6.2 | 6 | 5.9 | 5.6 | 5.1 | 4.6 | 4.2 | 3.6 |
| On grade 130 feet per mile........ | 6.2 | 6.1 | 6 | 5.9 | 5.7 | 5.6 | 5.5 | 5.2 | 4.8 | 4.4 | 4 | 3.5 |
| On grade 132 feet per mile— 2½ per cent... | 6.1 | 6 | 5.9 | 5.7 | 5.6 | 5.5 | 5.4 | 5.1 | 4.7 | 4.3 | 4 | 3.4 |
| On grade 140 feet per mile........ | 5.7 | 5.6 | 5.5 | 5.4 | 5.3 | 5.2 | 5.1 | 4.8 | 4.5 | 4.1 | 3.8 | 3.3 |
| On grade 145 2/10 feet per mile— 2¾ per cent... | 5.5 | 5.45 | 5.35 | 5.25 | 5.15 | 5.06 | 4.97 | 4.71 | 4.33 | 4 | 3.7 | 3.21 |
| On grade 150 feet per mile........ | 5.3 | 5.2 | 5.15 | 5.07 | 5 | 4.9 | 4.8 | 4.6 | 4.2 | 3.8 | 3.6 | 3.1 |
| On grade 158 4/10 feet per mile— 3 per cent... | 5.02 | 4.97 | 4.88 | 4.80 | 4.71 | 4.63 | 4.55 | 4.33 | 4 | 3.71 | 3.44 | 3 |
| On grade 160 feet per mile........ | 4.96 | 4.90 | 4.82 | 4.74 | 4.66 | 4.58 | 4.50 | 4.28 | 3.95 | 3.66 | 3.41 | 2.97 |
| On grade 170 feet per mile........ | 4.64 | 4.60 | 4.52 | 4.45 | 4.34 | 4.27 | 4.20 | 4 | 3.71 | 3.45 | 3.21 | 2.81 |
| On grade 180 feet per mile........ | 4.35 | 4.39 | 4.32 | 4.25 | 4.18 | 4.11 | 4.05 | 3.87 | 3.58 | 3.34 | 3.11 | 2.73 |
| On grade 184 8/10 feet per mile— 3½ per cent... | 4.23 | 4.20 | 4.13 | 4.07 | 4 | 3.94 | 3.88 | 3.70 | 3.45 | 3.22 | 3 | 2.64 |
| On grade 190 feet per mile........ | 4.09 | 4.06 | 4 | 3.94 | 3.88 | 3.82 | 3.76 | 3.59 | 3.35 | 3.12 | 2.92 | 2.57 |
| On grade 200 feet per mile........ | 3.86 | 3.83 | 3.77 | 3.72 | 3.66 | 3.61 | 3.56 | 3.40 | 3.18 | 2.97 | 2.78 | 2.45 |
| On grade 211 2/10 feet per mile— 4 per cent... | 3.62 | 3.60 | 3.55 | 3.50 | 3.45 | 3.40 | 3.35 | 3.21 | 3 | 2.81 | 2.64 | 2.33 |
| On grade 220 feet per mile........ | 3.45 | 3.42 | 3.39 | 3.33 | 3.28 | 3.24 | 3.19 | 3.06 | 2.87 | 2.69 | 2.53 | 2.24 |
| On grade 230 feet per mile........ | 3.27 | 3.25 | 3.20 | 3.16 | 3.12 | 3.08 | 3.03 | 2.92 | 2.73 | 2.57 | 2.41 | 2.14 |
| On grade 237 6/10 feet per mile— 4½ per cent... | 3.15 | 3.12 | 3.08 | 3.04 | 3 | 2.96 | 2.92 | 2.81 | 2.64 | 2.48 | 2.33 | 2.08 |
| On grade 240 feet per mile........ | 3.10 | 3.08 | 3.04 | 3 | 2.96 | 2.92 | 2.88 | 2.77 | 2.60 | 2.45 | 2.31 | 2.05 |
| On grade 250 feet per mile........ | 2.95 | 2.93 | 2.89 | 2.86 | 2.82 | 2.78 | 2.75 | 2.64 | 2.49 | 2.34 | 2.21 | 1.97 |
| On grade 260 feet per mile........ | 2.81 | 2.77 | 2.74 | 2.70 | 2.67 | 2.64 | 2.59 | 2.51 | 2.36 | 2.22 | 2.10 | 1.87 |
| On grade 264 feet per mile— 5 per cent... | 2.75 | 2.74 | 2.70 | 2.67 | 2.64 | 2.61 | 2.57 | 2.48 | 2.33 | 2.20 | 2.08 | 1.85 |
| On grade 270 feet per mile........ | 2.69 | 2.68 | 2.63 | 2.59 | 2.56 | 2.53 | 2.50 | 2.41 | 2.27 | 2.14 | 2.03 | 1.81 |
| On grade 280 feet per mile........ | 2.55 | 2.54 | 2.50 | 2.47 | 2.44 | 2.41 | 2.39 | 2.30 | 2.17 | 2.05 | 1.94 | 1.74 |

TABLE 10. *(Continued)*

| Number of times its own weight on driving-wheels any locomotive can haul | Frictional resistance of train in pounds per ton of 2,000 pounds | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | $6\frac{1}{2}$ | 7 | 8 | 9 | 10 | 11 | 12 | 15 | 20 | 25 | 30 | 40 |
| On grade $290\frac{4}{10}$ feet per mile— $5\frac{1}{2}$ per cent... | 2.43 | 2.41 | 2.39 | 2.36 | 2.33 | 2.31 | 2.28 | 2.20 | 2.08 | 1.96 | 1.85 | 1.67 |
| On grade 300 feet per mile................ | 2.33 | 2.31 | 2.29 | 2.26 | 2.23 | 2.21 | 2.18 | 2.11 | 1.99 | 1.88 | 1.77 | 1.60 |
| On grade $318\frac{8}{10}$ feet per mile— 6 per cent... | 2.16 | 2.15 | 2.12 | 2.10 | 2.08 | 2.05 | 2.03 | 1.96 | 1.85 | 1.76 | 1.67 | 1.50 |
| On grade 325 feet per mile................ | 2.08 | 2.07 | 2.05 | 2.03 | 2 | 1.98 | 1.96 | 1.90 | 1.80 | 1.70 | 1.61 | 1.45 |
| On grade 350 feet per mile................ | 1.87 | 1.86 | 1.84 | 1.82 | 1.80 | 1.78 | 1.76 | 1.72 | 1.62 | 1.54 | 1.46 | 1.32 |
| On grade $369\frac{6}{10}$ feet per mile— 7 per cent .. | 1.73 | 1.72 | 1.70 | 1.68 | 1.67 | 1.65 | 1.63 | 1.58 | 1.50 | 1.42 | 1.35 | 1.22 |
| On grade 375 feet per mile................ | 1.69 | 1.68 | 1.66 | 1.64 | 1.62 | 1.61 | 1.60 | 1.55 | 1.47 | 1.39 | 1.32 | 1.20 |
| On grade 400 feet per mile................ | 1.53 | 1.52 | 1.50 | 1.49 | 1.47 | 1.46 | 1.44 | 1.40 | 1.32 | 1.26 | 1.20 | 1.08 |
| On grade $422\frac{4}{10}$ feet per mile— 8 per cent .. | 1.40 | 1.39 | 1.37 | 1.36 | 1.35 | 1.33 | 1.32 | 1.28 | 1.22 | 1.16 | 1.10 | 1 |
| On grade 450 feet per mile................ | 1.26 | 1.25 | 1.24 | 1.23 | 1.21 | 1.20 | 1.19 | 1.15 | 1.10 | 1.05 | .99 | .90 |
| On grade $475\frac{2}{10}$ feet per mile— 9 per cent .. | 1.15 | 1.14 | 1.13 | 1.11 | 1.10 | 1.09 | 1.08 | 1.05 | 1 | .95 | .90 | .82 |
| On grade 500 feet per mile................ | 1.04 | 1.03 | 1.02 | 1.01 | 1 | .99 | .98 | .95 | .91 | .87 | .80 | .75 |
| On grade 528 feet per mile—10 per cent... | .94 | .93 | .92 | .91 | .90 | .89 | .88 | .86 | .82 | .78 | .74 | .67 |
| On grade 550 feet per mile................ | .86 | .86 | .85 | .84 | .83 | .82 | .81 | .80 | .75 | .70 | .66 | .60 |
| On grade $580\frac{8}{10}$ feet per mile—11 per cent .. | .77 | .76 | .75 | .74 | .74 | .73 | .72 | .70 | .67 | .63 | .60 | 54 |

The above table is useful for the selection of a locomotive of suitable weight to do any given work, since it shows the relative power of a locomotive on any practicable grade and with varying allowance for frictional resistance. Example: How heavy a locomotive is needed to start 50 tons on a 3 per cent. grade with contractors' cars and poor track involving a frictional resistance of 20 pounds per ton? By following the column for 20 pounds resistance to the intersection of the horizontal line for 3 per cent. grade, the figure 4 is noted, *i.e.*, a locomotive under these conditions can haul four times its weight on driving-wheels; since 4 times $12\frac{1}{2} = 50$, a locomotive $12\frac{1}{2}$ tons, or 25,000 pounds in weight with all the weight on driving-wheels, would be needed.

The chart, Fig. 14, graphically shows how the hauling ability of a locomotive varies under variable conditions of grade and curvature of track. For example, on straight track and 4 per cent. grade, its hauling ability for the entire train, is but 11 per cent. of that on straight level track; on level track and 20° curve its ability is but 33 per cent.; on 4 per cent. grade and 20° curve its ability is but 9 per cent.

Table 10[1] is here reproduced as a handy reference in selecting locomotives of suitable weight to do any given work. It also shows the effect of grade and train resistance.

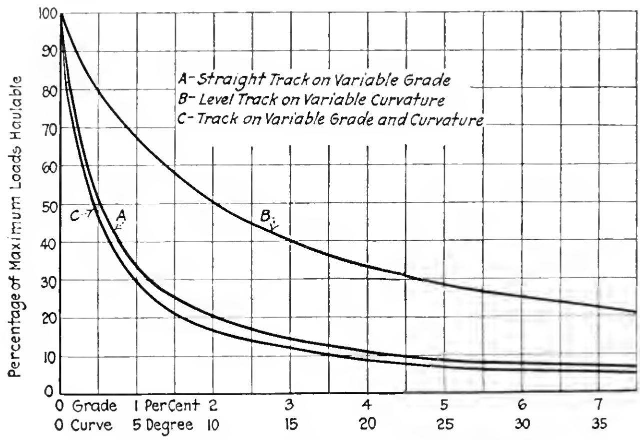



FIG. 14.—Hauling capacity of locomotives.

*Horse-power.*—There are times when it is useful to know the h.p. of a locomotive, though generally this is not of great significance. Since one h.p. is the ability to lift one lb. against gravity 33,000 ft. in one minute, then

$$HP = \frac{T \times S}{375}$$

Where $HP$ represents the h.p. of the locomotive,
$T$ represents the tractive force in pounds,
$S$ represents the speed in miles per hour at which the locomotive can haul its heaviest load.

$S$ may be determined from the following formula:

$$S = \frac{D}{F}$$

[1] H. K. Porter Co.—12th edition catalogue, p. 105.

Where $D$ represents the diameter of driving wheel in inches,
$F$ represents a factor taken from the table below,

| | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Stroke in inches... | 8 | 10 | 12 | 14 | 16 | 18 | 20 | 22 | 24 | 26 | 28 |
| Factor............ | 5 | 5.2 | 5.4 | 5.6 | 5.8 | 6 | 6.2 | 6.4 | 6.6 | 6.8 | 7 |

Thus, in case of the 65-ton engine mentioned

$$S = \frac{44}{6} = 7.33 \text{ miles per hour, and}$$

$$HP = \frac{26{,}288 \times 7.33}{375} = 513 \text{ horse power}$$

The boiler h.p. is dependent on many factors of heating, but with good bituminous coal, 3 sq. ft. (with oil, 2 sq. ft.) of heating surface will ordinarily create steam for one h.p., provided there is at least 1 sq. ft. of grate area for every 60 sq. ft. of heating surface.

*Fuel and Water Consumption.*—A locomotive will ordinarily consume about 4 lbs. of coal and 30 lb. of water per horse-power hour. The quantity of water required (in gallons) for a run of one mile about equals the average resistance (in pounds) overcome, divided by 104 or, say, one per cent. of the resistance to be overcome. As approximately one gallon of water is evaporated by one pound of good coal, the foregoing answer also represents the pounds of coal consumed. Under poorer conditions 1.6 pounds of coal are sometimes required per gallon of water evaporated. Short runs and intermittent service increase both the coal and water consumption per mile. Good coal will liberate about 14,000 B.t.u. per pound while poor coal may run below 10,000 B.t.u.; fuel oil will give about 19,000 B.t.u. per pound. (This runs about 7.6 lb. per gallon and 42 gal. or 319.2 lb. per barrel for crude petroleum). Wood fuel averages about 5500 B.t.u. per pound under ordinary conditions.

*Usual Sizes for Open-pit Work.*—Some contractors use four-wheel locomotives with cylinders as small as 5 in. × 10 in. and weighing only about 5 tons, but for general use in and around shovel mines locomotives are seldom lighter than 35 tons with cylinders 14 in. × 18 in. The heavier four-wheel connected saddle-tank engines run up to 60 tons and have cylinders 18 in. × 24 in.

Six-wheel connected saddle-tank locomotives range from 7-ton, with 6 in. × 10 in. cylinders, to 85-ton machines with 21 in. × 26 in. cylinders. The heavier types, say, from 50 tons with 16 in. × 24 in. cylinders up, are preferred where practicable, for although they require heavier and better track, they haul heavier loads and are more economical in labor and operating costs per ton mile. As they carry about 3 tons of coal and 2000 gal. of water they are good for a haul of several miles.[1] Side-tanks may be substituted for saddle-tanks but the latter are usually preferred. A two-wheel truck may be added either forward or back, or both forward and back, of the drivers to carry overhang or assist in rounding curves. These locomotives are built for moderate speeds and are very satisfactory where the run is not long enough to require a separate tender. In the iron mines of the Mesabi, locomotives with cylinders 19 in. × 26 in. and 20. in × 26 in. with separate tenders are in general use. They seldom operate on grades over 2 per cent. They are equipped with 8½ in. × 10½ in. cross compound air pumps for operating the air dump cars; also with two camel-backs, a piece of 1½ in. hoisting chain and 20 ft. of 1 in. chain for replacing cars and locomotives on the track. Present practice generally calls for standard gauge, viz., 4 ft. 8½ in.

Such companies as the Baldwin Locomotive Works, American Locomotive Company, H. K. Porter Company and others manufacture these locomotives and are glad to furnish specifications and advice on selection, but it is advisable to have same carefully passed upon by someone competent to judge and familiar with the local conditions of service for which they are to be used. It is not the purpose of this work to consider such items of equipment in minute mechanical detail as too much space would be required.

*Cost of Locomotives.*—Recent quotations on the cost of locomotives delivered at the point of manufacture seem to be from about 9 cents to 10 cents per pound. Thus a locomotive with cylinders 19 in. × 26 in., six 46 in. driving-wheels and a two-wheeled forward truck, burning coal, and with iron flues, steel tires, two feed-water injectors and a 2200 gallon tank on boiler

[1] The consumption of supplies per 9 hr. shift, for 65-ton locomotives of this class has been found to average about as follows: coal, 3½ tons; cylinder oil, 5 pints; engine oil, 4 pints; kerosene, 2 pints; cup grease, ¼ lb.; cotton waste, ½ lb.; calcium carbide for night illumination, 8 lbs.; water, 6,000 gallons.

is quoted at $15,380 at Philadelphia. Former prices have ranged from $7\frac{1}{2}$ cents per pound for 100-ton engines to about 9 cents per pound for 50-ton engines. The builders are always glad to furnish full specifications and prices to prospective buyers.

**Geared Types.**—If pit grades and approaches cannot be kept down, to, say 4 per cent or under, rod locomotives will operate only at great inefficiency. Under such conditions it may be better to substitute geared locomotives of the Shay or Hesler type, or even a rack type. These machines have the greatest tractive power, consistent with their weight, of any locomotive and tender. They are better adapted to heavy grades, sharp curves, temporary track and light rail than the rod type. Due to the greater number of cylinder reciprocations they exert a steadier pull and give lower fuel combustion. They can move at slower speeds and can start loads without danger of stalling It will be noted that all of the weight of engine and tender is on the drivers, and that these consist of trucks of short wheel base which are free to swivel, enabling the engine to pass sharp curves with the least possible friction and strains to track alignment. They are not adaptable to speeds of over, say, 15 to 20 miles per hour and the repairs and maintenance are generally high.

The Shay locomotives, built by the Lima Locomotive Works, of Lima, Ohio, and the Hesler locomotives have been used in some of the open-pit mines on the iron ranges, and at other places.

Where grades of over, say, 7 per cent. would be unavoidable some means of hoisting on inclined plane or through shafts must be resorted to for handling heavy tonnages. In a few instances cableways have been installed, but owing to the necessity of track shifting and their lower capacity they are rare.

**Compressed Air Locomotives.**—*General Considerations*—Compressed air haulage is often employed in underground mines, but seldom if ever in open-pit workings. Where mines are dusty or gaseous, as in some coal seams, or where ventilation is poor, or the workings wet or heavily timbered, or the roof bad, or where accident by electric shock would be serious, compressed air locomotives are of marked advantage, but in open-pit mines these conditions are not met. A few remarks regarding them, however, will probably be of interest. The two-stage compressed-air locomotive with an atmospheric interheater located between high and low pressure cylinders has shown such increased efficiency over the single-expansion locomotive that it is much preferred. Comparative working tests show a gain in efficiency of over 50 per cent, and a saving in compressed air consumed of over 30 per cent. The air is charged into the main reservoir at a pressure of 700 to 1200 pounds per square inch; it then passes through a reducing valve which maintains a constant pressure of 250 pounds in the auxiliary reservoir; from here it is fed to the high pressure cylinder and partially expanded to about 50 pounds pressure, becoming much colder than the surrounding atmosphere; it is then passed through a tubular interheater in which it is heated to nearly atmospheric temperature by contact with the surrounding air; the expansion is then completed in the low-pressure cyinder. In practice approximately equal amounts of work are thus done between 250 pounds and 50 pounds pressure, and between 50 pounds and atmospheric pressure.

Besides the compressed-air locomotives, there must be provided charging stations at convenient points, each station consisting of valve, bleeder valve and flexible metallic coupling. These serve for connecting the locomotive with the air supply, and, when properly arranged, permit the locomotive to be stopped, charged and restarted inside of a minute and a half.

Next there is provided stationary storage to permit the rapid charging of the locomotives and the continuous operation of the compressor. This storage is of such capacity that the locomotive can be charged without material reduction in pressure. If the stations are some distance from the compressor, it is usually best and most economical to take care of this capacity by using pipe of sufficient size to give the required volume, rather than a combination of pipe of smaller diameter and storage tanks.

Lastly there is the compressor, which usually compresses air to about 1000 lb. per sq. in. A high pressure is required to store sufficient quantity of air in the locomotive to drive it the necessary distance without recharging and not have the reservoirs excessively large. The size of the compressor depends upon the quantity of free air which must be compressed to do the total amount of work in a given period. A thorough investigation of the work to be performed by the locomotives is needed to determine the proper size.

This layout complete usually makes the first cost of installation of compressed-air locomotives from two to three times as great as for steam locomotives, and perhaps about the same as a complete trolley-type electric locomotive layout. Therefore, compared with steam locomotives, one or more of the following features must be of sufficient weight to justify the additional expenditure:

1. Absolute insurance against fire or explosion due to sparks, flame or heat emitted or caused by locomotives.

2. Cleanliness.

3. Great economy of power due to central generation from cheap fuel, even though at considerable distance from the charging stations; or where the compressor can be run for a short time and the stationary storage built up to run the locomotives for some considerably longer period.

4. Locomotives needed for instant service, or without licensed engineers.

5. Great saving in operating and contingent expenses, fixed charges and depreciation, when all are fairly considered.

As previously mentioned it will be an exceptional case indeed in open-pit work where any of the above features will be essential.

Compared with trolley-type electric locomotives, the air machines are safer from shock, do not require trolley lines nor bonded rails, operate better in wet places and have about the same operating power cost. They are fully as reliable and flexible and will meet all ordinary requirements of operation as well as any type of locomotive of the same weight.

*Cost of Installation and Operation of Air Haulage.*—As a matter of approximate comparative interest the cost of an installation, capable of handling 3000 tons over a 2 per cent. grade for a distance of one mile in 9 hr,. is estimated:

| | |
|---|---|
| 2 two-stage 20-ton compressed air locomotives | $10,800. |
| One 800 cu. ft. 1000-pound compressor | 9,000. |
| One 300 h.p. boiler at $15.00 per hp | 4,500. |
| 5000 feet of 5-inch special air line @ 70 cents | 3,500. |
| 3 charging stations complete | 400. |
| Fittings, valves and miscellaneous | 800. |
| Foundation and installation of machinery | 500. |
| Installation of pipe line | 600. |
| Total | $30,100. |

The cost of operation of the above plant per day of 9 hr. is estimated to be:

| | |
|---|---|
| 2 locomotive drivers @ \$4.50 | \$ 9.00 |
| 2 brakesmen @ \$4.00 | 8.00 |
| Compressor attendance | 4.00 |
| Oil for compressor and locomotives | 1.50 |
| Repairs for compressor, boiler, locomotives and pipe lines | 4.00 |
| Coal, (4½ lbs. per h.p.-hr., say 3000 h.p.hrs.) @ \$5. per ton, say | 34.00 |
| Interest and depreciation @ 15 per cent | 12.50 |
| Total | \$73.00 |

Moving 3000 tons one mile = 3000 ton miles

Cost per ton mile..............2.433¢

Operating two shifts would show a reduction in the above cost by about 0.3¢ per ton mile.

**Electric Locomotives.**—*Trolley System.*—Like compressed-air haulage, electric locomotives operated by overhead trolley lines, have found a very practical field in underground mines, but are seldom, if ever, used in open-pit mines. The cost of installation of such a system would probably be about as expensive as compressed-air and much greater than for steam locomotives. Provision would have to be made for the trolley lines and bonding of rails, and, for work requiring much shifting of track, this would be a serious disadvantage and expensive to maintain. The principal advantages would be found in cheaper power cost, elimination of firemen and somewhat better operation on heavy grades. The disadvantages of higher first cost of installation and subsequent maintenance of overhead feeders and track bonding on nonpermanent track would in most cases certainly be much greater than the advantages gained. There is also a lack of flexibility in train movement, as the locomotives cannot operate (except for short distances when supplied with reeled feeder) on lines not electrified.

### Centrally Controlled Systems

*The Woodford System.*[1]—The Woodford system is a centrally controlled electric haulage system which has been installed in a number of shovel workings of the quarry type and is giving satisfactory service.

It is described as a system of haulage for "operating a multiplicity of cars from a central controlling tower by remote control, and is applicable to such industrial haulage as is found in stone quarries, open-

[1] Woodford, F. E. Proceedings of the Engineers' Society of Western Pennsylvania, vol. 31, p. 583–608, Sept. 21, 1915.

mines, clay and gravel-pits and other places where the distance is not too great and where the operation of the cars can be seen for at least part of the time from the controlling tower.

"Each car carries its own motive power and control apparatus, taking current from a sectionalized third rail, which allows the car to be started and stopped, or have brakes applied at the will of the tower operator and the car to have its direction reversed at certain fixed points. Control apparatus on the car is arranged so that as the car descends grades the motors can be connected across resistance banks and the regenerative effect used for retardation purposes.

"The third rail which extends over the whole territory covered by the haulage system, is divided into long or short sections, as conditions require, and an independent feeder is extended from the control tower to each section. Tower operator has two voltages, 250 volts and 90 volts, which he can apply to various sections of the third rail at will. 250 volts is used for haulage purposes and 90 volts is for braking purposes. The regular car rails are used as a return circuit.

"The tower apparatus consists of a switching mechanism by means of which the tower operator may put haulage or braking voltages on sections of third rail, or allow them to stand without applied voltages. In the tower machine there are also, when desired, control switches for operating motor-driven track switches, reversing switches and other special apparatus located at distant points on haulage system.

"Each car truck is fitted with two railway type, 250 volt, direct-current series motors, mounted upon the axles and supported by suspension springs and connected with the axle through single reduction gears in the ordinary manner adopted by electric traction practice. These motors have their shafts extended at the commutator ends and carry a specially designed solenoid brake, which is applied by the action of the solenoid and released by springs. The car truck also carries three other pieces of apparatus necessary for the operation of the cars from a central point. They are a bank of resistance, a double-pole double-throw switch for reversing the direction of the motors, and an electric relay or selector switch. This selector switch is a solenoid-operated switch having two contacts and two positions. It is locked in its gravity position with one contact open and one contact closed. In its position induced by the solenoid it is also locked, opening the gravity circuit and closing the other circuit. The solenoid will not respond to a voltage lower than 175, but after being closed will hold in with 50 volts, or even less.

"The motors are operated with the selector switch in its excited position. The circuit in the gravity position is connected to the brake solenoids, and the relay will remain locked in gravity position while the brakes are applied. While the relay is in the gravity position it also establishes a circuit from the motors on the car to the bank of resistance which the car carries. This resistance is adjusted to dissipate the current generated by the motors running as generators while the car is traveling down a grade thus furnishing a dynamic brake for the car which is not under the control of the operator. This feature forms a safety device should the circuit breaker go out while the car is ascending the grade, as upon circuit breaker opening the car will merely coast back down hill at a not excessive speed. While the operator has control of the brakes at all times, this feature also relieves him of governing or speeding the car while it is traveling down grade. This feature renders the system semi-automatic, arranging its operation so that it is necessary for the operator to give very little attention to the running of the cars except where they are stopped for loading or unloading. . . From the central controlling tower placed at any convenient position, we are able to excite any section of third rail with a voltage ranging from 30 to 100 for controlling the brakes, as well as with the starting and running current for the motors.

"A reversing switch is placed on the car so that it is operated by a lever protuding from the side of the truck in such a manner as to engage track cams placed at desired locations along the track. These track cams are stationary forgings so formed as to throw the lever in the proper position and placed beside the track switch where all the cars are to be reversed. At points along the track where it is necessary for only part of the cars to be reversed this reversing cam consists of forgings and levers having two positions and operated by a solenoid. Should it be necessary to reverse the car, this solenoid is excited from the tower by a foot switch. It will readily be seen that by this method operation in any manner to meet any conditions may be accomplished, and with track cams at the top of all grades it is impossible for a car to enter upon that grade without the reversing switch and selector switch being in the proper position to cause motors to generate current as they are propelled by the momentum of the car, and furnishing the dynamic brake, maintaining a fixed speed at all times.

"At the dumping hopper or crusher another form of reversing cam has been adopted, which is thrown into different positions by a motor-driven movement. This cam carries parallel bars some ten or twelve feet in length between which the reversing lever of the car passes so that the car may be spotted in any particular position for dumping, arriving from either direction and leaving in either direction. This motor-driven cam is also operated by push-button switches from the tower.

"The elements described above together with centrally controlled, electrically operated track switches, with which all are familiar, render this system of control sufficiently flexible to meet any and all conditions and bring its operating levers within the reach of a single man. A car may be switched to any track; may be run into a passing track from the main line; may be stopped and started from any point and, in fact, operated in the same manner as though there were a motorman upon each car.

"The design of a car to carry this apparatus is not materially different than the standard construction of dump-cars to which in many cases the motors and other apparatus have been applied."

Special care has been given the design of the cars so that they may well withstand the severe service of shovel loading.

The following advantages are claimed for the system:—reduction in power cost, reduction in labor cost and decreased amount of rolling-stock required. It is stated that the capacity of the steam-shovel is materially increased by supplying cars individually as the shovel is not kept waiting between trains. The power cost reduction is claimed by virtue of dispensing with locomotives and placing the load on top of the tractive machinery.

The labor cost for operation is reduced because the entire system is operated by a very small crew, which may be as low as one operator where only a few shovels are involved and where all parts can be seen from one point. A smaller amount of rolling-stock is required because locomotives are eliminated and cars need not be tied up in trains awaiting loading and dumping. Each car consequently makes many more trips than do the cars hauled in trains. The design of the car units enables them to negotiate curves much sharper and grades much steeper—10 per cent. and over—than can be taken by locomotives.

The operating and upkeep costs are given at an average of not over one-half cent per ton mile, in all except very small installations; the upkeep of cars and apparatus being less than that of locomotives and cars for an equal amount of haulage. The elimination of water and fuel supply is an advantage.

"The installation cost compared with locomotive haulage has been found to be practically equal with a capacity of 3000 tons in ten hours. For smaller capacities the installation cost of an electric haulage system has been found to increase gradually above that of locomotive haulage, while above 3000 tons capacity it has been found to be somewhat less. This, of course, is in cases of locomotive haulage using cars of equal durability and equal weight, and while most of the locomotive systems in use at the present time are operating with cars of lighter construction, the upkeep of these light cars is very much greater."

An advantage noted is the elimination of a certain risk to men working on the rock trains. This has reduced casualty insurance rates as well as being commendable. Another advantage is in the flexibility of handling material from end or butt shovel-cuts, where one car loading is necessary. The 10-cu. yd. cars weigh net about 15 tons and carry about 14 tons.

Some of the companies using the system are: The Casparis Stone Company, of Kenneth, Indiana; Dolese and Shepard Company, Gary, Illinois; Hohan Stone Company, Maple Grove, Ohio.

Mr. O. P. Chamberlain of the Dolese & Shepard Co. states:

"I am satisfied as now developed our rate of maintenance is lower than for steam locomotive operated plants. . . Our current is purchased . . . at a cost averaging 2.65 cents per kilowatt hour."

Mr. George W. Patnoe of the Hohan Stone Co., states:

"We installed twenty-five 10-yard cars to supply a plant that was producing from 600 to 700 tons per hour. We have never used over two-thirds of the equipment."

At another quarry, he states: "We installed the Woodford system with ten 6-yard cars. They would be loaded with seven, and sometimes eight yards. Six of these cars supply this plant at an average output of 2,500 tons in ten hours. It would take at least three locomotives and from thirty-five to forty 6-yard cars to do the same work we are now getting done with a centrally controlled electric haulage system. The work of haulage of this stone to and from a quarry, a mile haul each trip, is all accomplished with two operators, and there is an elevation from the crusher to the bottom of the quarry of forty feet. . . This equipment has 50 per cent. greater capacity, and the cost of labor is not 25 per cent. what it would be with locomotives. . .

"I have operated this electric equipment for the last six years or more. . . It is more economical in every way; first cost, cost of operation, cost of maintenance, and the convenience of having the power-plant centrally located under one roof is very worth while." Its operation in winter, doing away with boilers and water lines, is favorably commented on.

"A steam-shovel loading single cars has a capacity at least 15 per cent. greater than when the cars are placed in trains by locomotives." The reason given is that the track is kept clean of stones without separating cars.

"Another feature that makes it more convenient is that in the source of power there is a greater number of units. If one is disabled, the chances are that there is another to take its place. The plant is not liable to as much delay as if it were depending on two or three locomotives.

"The cost of operating the electrical equipment is only about 25 per cent. of the cost of operating a steam-haulage system. . . It is applicable for any haulage system within a reasonable distance, such as gravel pits, clay pits, iron ore, etc.

Mr. W. R. Casparis, of the Casparis Stone Co., states:

"I give the following reasons why I personally favor electric haulage where the conditions are permanent, such as a stone quarry, brick plant, etc.

*First*: The great saving of men and labor.

*Second*: The steep grades which this class of equipment can easily negotiate. It is not an uncommon thing for us to operate a car over 6 to 10 per cent. grades.

*Third*: The rapidity with which the cars operate over fair, and even bad track, owing to the spring construction, and having only four wheels, they quickly equalize. (This speed is from six to twenty miles per hour).

*Fourth*: The electric haulage affords the maximum movement of material with a minimum amount of equipment. The eight (10-yard) cars we have in operation are capable of hauling 3000 ton of stone in 10 hours.

*Fifth*: The comparative ease with which electric haulage is operated, as there are no locomotives to coal, water and watch at night."

Considerable space has been given to the foregoing system as it is one of much merit where conditions permit its use. It would not be adaptable for very slow loading, or where heavy fog or dense smoke was liable to cut off the towerman's view from the important points along the track. If the distance from the loading points to the crusher or dumping point was very considerable it would lose its advantages. There would be more costly equipment tied up in long distance travel and a higher percentage of tare weight, than if long trains of large capacity cars were hauled with heavy locomotives. At many open-pit mines the ore is loaded directly into ore cars which are hauled out of the pit and switched into an assembly yard where they are made up into heavy trains for transportation many miles to discharging points. In this case there is no transfer of the ore and the system would not be adaptable, but if a crusher was located within a mile or two of the loading points, or if the material was to be placed on stock piles or waste dumps, the system might be profitable to adopt, and should be considered.

Following is a list of installations of Woodford haulage systems serving power-shovels in stone quarries, with daily capacities, etc;—

Dolese & Shepard Co., Chicago Ill., 4000 tons, average haul one mile, three shovels. Installed in 1909.

Missouri Iron Company, Waukon, Iowa. 1500 tons, average haul one half mile, one shovel. Installed in 1910. Plant not in operation.

Laurin & Leitch Eng. & Constr. Co., Beaver Hall Sw., Montreal, Canada. 3000 tons, average haul one half mile, two shovels. Installed in 1911.

Casparis Stone Company, Kenneth, Ind. 3000 tons, average haul one half mile, two shovels. Installed in 1914.

Temescal Rock Company, Corona, Cal. 1000 tons, average haul one half mile, one shovel. Installed in 1915.

Michigan Alkali Company, Alpena, Michigan. 6000 tons, average haul one mile, five shovels. Installed in 1916.

Virginian Limestone Corporation, Ripplemead, Va. 2000 tons, average haul one half mile, one shovel. Installed 1916.

Laurin & Leitch Eng. & Constr. Co., Montreal, Canada. 3000 tons average haul one half mile, two shovels. Installed 1918.

Bessemer Limestone Co., Youngstown, Ohio. 1000 tons, average haul one half mile, one shovel. Installed in 1919.

Oklahoma Portland Cement Co., Ada, Okla. 2000 tons, average haul one half mile, two shovels. Installed in 1919.

The Woodford Company is now building a system for the Marble Cliff Quarries Company of Columbus, Ohio, which is expected to serve four shovels on a two mile haul.

## CARS

**Stripping Cars.**—The cars employed for the removal of overburden are of numerous types and designs, but they may all be classed either as of the platform, the gondola or the dumping type.

*Platform Type.*—The platform cars are the simplest, consisting of a heavy platform of wood, sometimes armored, supported on two four-wheel standard trucks. The sides of the platform are provided with stake-pockets to receive short vertical posts which guide the unloading device, and at one end of each car is a hinged sheet-steel apron as wide as the platform and long enough to bridge over the coupler space to the next car, forming, as it were, a continuous support or bed for the unloader. These cars are about 34 ft. long and 7 ft. wide, and carry about 10 cu. yds. of earth or broken rock. The brake wheels must be placed at the side of the car.

When arranged in this way they can be unloaded by a plow-like contrivance which rests on the platform of the rear car. A steel cable is attached to the plow and to the locomotive, and, when the dumping place is reached, the aprons are lowered and the locomotive is detached and slowly moved ahead, drawing the plow along the length of the train and cleaning the car platforms of material. To avoid detaching and moving the locomotive ahead, the Lidgerwood Manufacturing Company has designed a plow with the cable attached to a single-drum reversible engine located at the front end of the first car close to the locomotive and deriving steam for operation from the locomotive. This does the unloading in the same way. The material may be unloaded either to the right or the left or on both sides simultaneously depending on the set and shape of the plow. The engine is capable of exerting a pull on the cable of about 25 tons and winding at a speed of 125 ft. per minute, thus unloading the train quickly and cheaply.

*Gondola Type.*—The ordinary gondola car is of similar construction to the platform car but has permanent sides and ends about 3 ft. high. They hold from 25 to 40 cu. yds. and are convenient for hauling some classes of material long distances. They must be unloaded by hand, (or with some type of derrick excavator) and as this is slow and expensive they are seldom used for overburden removal, but often for coaling shovels and locomotives. For coaling purposes specially constructed cars of this type are sometimes advantageously used.

*Dumping Type—Air or Hand Operated.*—By far the greatest number of cars in stripping service are of the dumping type, of which there is a large variety on the market.

For the purpose of removing overburden side-dump cars are made in capacities of from 1½ cu. yds. to 30 cu. yds. level-full or "water measure". They are built to dump on one side or both, the smaller sizes being dumped by hand, and the larger sizes, say, from 12 cu. yds. up, either by hand or by air. The smaller sizes, say, up to 10 cu. yds., are equipped with two two-wheel trucks and are of wood construction; the larger sizes have two four-wheel trucks and are built, on M. C. B. specifications, of wood, wood and steel or all steel. There are also used for some purposes bottom-dump and end-dump cars built in various sizes, designs and materials.

In present practice on large works, side-dump cars carrying from 12 to 30 cu. yds. of all steel construction are proving most satisfactory. Some cars of this type have a line of pivot pin-bearing dumping-centres and others dump on rockers located on either end of the car over the bolster. The latter method gives easier action with less shock to the car but does not compact the material dumped so well, nor keep the far side of the bottom so clean as when it is dumped more violently. Consolidating the dumped overburden is an advantage in dump building when the dump-track is later to be extended thereon, and keeping the car clean is essential, so that if the pin-bearing car is well designed with ample provision made for the shock and impact of dumping, there is no serious mechanical objection to this type.

Another varying feature is the movement of the side doors, some having rising doors and some having folding or drop-doors. The drop-door virtually forms a stepped-down extension to the car floor and greatly assists in discharging the material farther from the car and track. If the rising-door type is provided with extended bottom-plates, on which the doors may rest, this outward disposal is similarly accomplished. If the loading and dumping can be so arranged that the cars are required to dump from one side only, they have decided mechanical advantages in construction.and are considered safer in mining, as, on being righted after dumping, they cannot come all the way over and hit the men.

It is still a debated point as to whether air-dumped or hand-dumped cars are most efficient and economical, and the answer will generally depend on the local conditions of the particular problem for which the cars are intended. Hand-dumped cars up to 20 cu. yds. capacity, level-full, may be dumped without difficulty, but for larger sizes, the car is likely to be so heavy that hand-dumping ,or rather righting, may be awkward.

The air-dumping device is operated by compressed air from the locomotive. This is derived from the air-brake pump and is conveyed to the car reservoirs and dumping cylinders by special independent train lines similar to the air-brake lines. Some types of air-dumped cars are operated by a single lever for each side which controls the unlocking, dumping, righting and relocking of each car. Each lever governs the action of an air-piston situated at its respective side of the car and between the trucks. In operation, the piston at one side pushes the car-body over, the piston at the opposite side pushes it back. By means of a system of valves through which the air is admitted to the dumping cylinder the air is automatically shut off when the right amount for dumping the car has passed, avoiding air wastage. The air is thus applied directly to the car and no power is wasted. The doors are automatically raised in dumping and are held in the proper position for free clearance of the dumping load. The doors may be so hung or pivoted that they are free to swing out when struck by rocks, making for less liability of choking or of overturning the car. All steel, 16 or 20-cu. yd. cars of this type are in general use in the Mesabi iron mines. They are built by the Western Wheeled Scraper Co., and the Kilbourne & Jacobs Company, those of the latter having the following general dimensions:—

| | | |
|---|---|---|
| Type | R–20 | C–16 |
| Angle of dump | 45° | 45° |
| Gauge | 4 ft. 8½ in. | 4 ft. 8½ in. |
| Height—top of rail to top of body | 8 ft. 5½ in. | 7 ft. 10½ in. |
| Height—top of rail to center of draw bar | 34½ in. | 34½ in. |
| Length over couplers | 32 ft. 0½ in. | 30 ft. 0½ in. |
| Length over end sills | 29 ft. 5½ in. | 27 ft. 6¼ in. |
| Body depth | 2 ft. 4 in. | 2 ft. 0 in. |
| Length inside | 26 ft. 0 in. | 24 ft. 0 in. |
| Inside width | 8 ft. 10 in. | 9 ft. 0 in.. |
| Maximum width | 10 ft. 6 in. | 10 ft. 6 in. |
| Truck centres | 16 ft. 0 in. | 16 ft. 0 in. |
| Truck wheel base | 5 ft. 6 in. | 5 ft. 4 in. |
| Approx. weight, lbs | 48,000 | 42,500 |

Another dump-car, known as the Goodwin, is constructed of steel and divided into two compartments which can be unloaded separately because of the dividing steel diaphragm. The discharging mechanism can be operated either by air, steam, electricity or hand-power. It can be discharged on either or both sides, or in the center, according to the different manner of manipulating the bottom-plates and side-doors. Discharging can be done with safety while running at any speed, and in this way the material can be spread for a considerable distance from the track. Dumping all or any number of the cars is done by one man in any part of the train. For stripping service, however, these cars are seldom used as they are not sufficiently rugged and simple.

A hand-dumped car of 20 cu. yd. capacity and dumping from one side only, which has been found to give very satisfactory service, is illustrated in Figs. 15 and 16. Desirable features will be noted in the all-steel construction, extended bottom, one-way dump, wood buffer-sills, steep dumping angle, pivoted door and self-righting balance. The one-way dump adds to its mechanical strength and the steep 47° angle assists in discharging sticky or partly frozen material. Such cars cost about $2500 each. Where

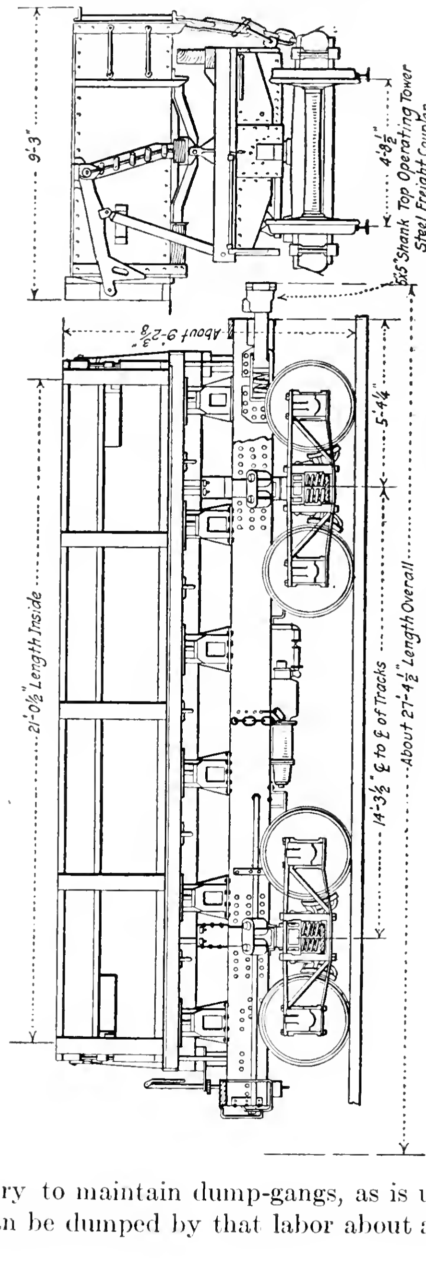



FIG. 15.—20-cubic yard dump car, general arrangement.

it is necessary to maintain dump-gangs, as is usually the case, these cars can be dumped by that labor about as quickly as the

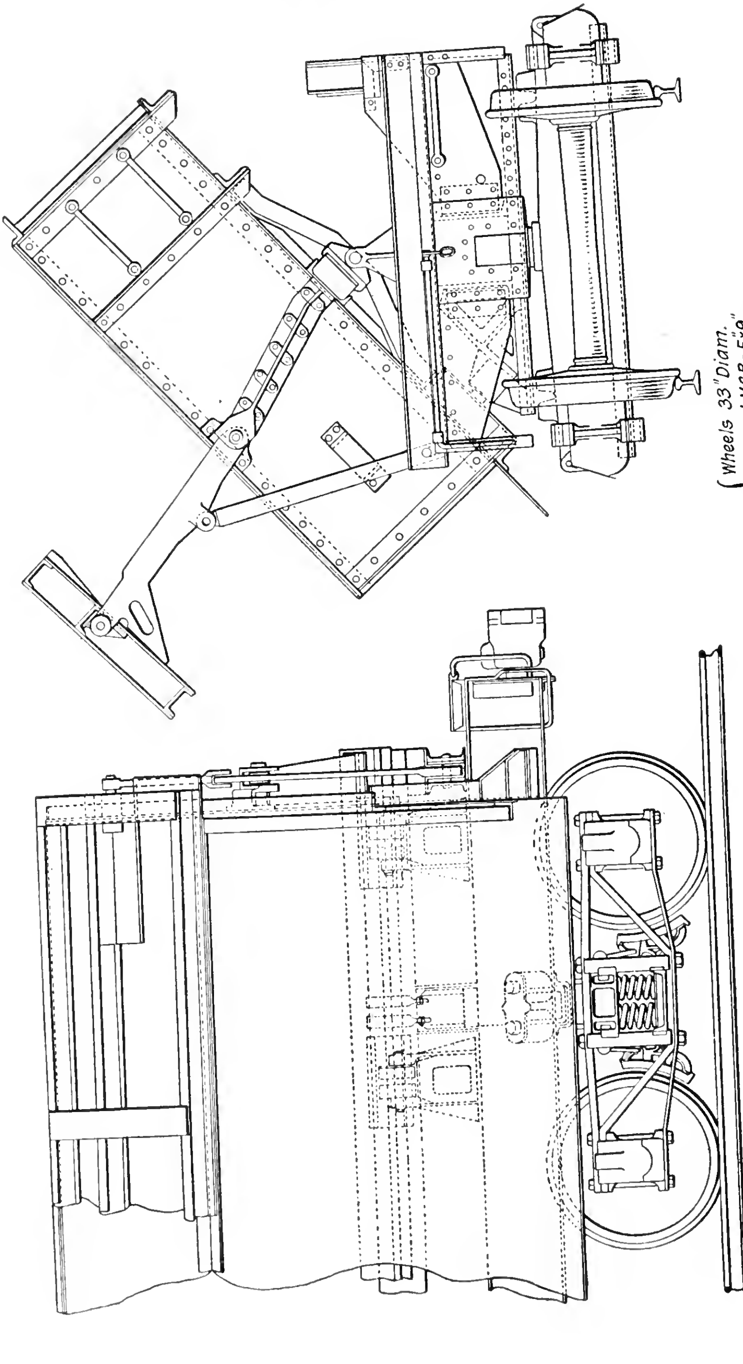



FIG. 16.—20-cubic yard dump car, dumping position.

air-dumped type; furthermore it is often necessary to "respot" the train to dump the material where desired and in such cases there is no advantage in being able to dump all cars simultaneously. Under such usual conditions it will be found that this car, being of simpler construction, will stay in service longer, require fewer repairs and be more economical to operate than those dumping by air. The company using this car has made many changes in the size of dump cars since operations were started. The first cars used were 6-cu. yd. two-way dump-cars. next came 12-cu. yd. cars with one-way dump, these were discarded for 18-cu. yd. cars and now the 20-cu. yd. car is standard. The factors governing the size of the cars are the height of centre of gravity, the amount of dead weight per cubic yard capacity and the number of cubic yards which can be hauled per trip. Among the builders of both types may be mentioned the Kilbourne & Jacobs Manufacturing Co., Columbus, Ohio; Oliver Manufacturing Co.; Western Wheeled Scraper Co., Aurora, Ill.; Continental Car and Equipment Co., Highland Park, Ky.; Clark Car Co., Pittsburg, Pa.,

**Ore Cars.**—*Hopper Bottom and Gondola Types.*—For loading and transporting ores short distances dump-cars may be used, but if the distance be considerable a different type of car should be adopted. For hauling iron and copper ores heavy all-steel cars, constructed on standard M.C.B. specifications and carrying from 50 to 70 tons are the usual sort. These are designed with a variety of automatic or hand-dumping bottoms or valves. They are usually required to dump over a bin, trestle or dock. The Ingoldsby type is of this variety and is a very satisfactory car. Such cars are built by the Pullman Company and other standard car builders. They cost about $2800 each for those of 60 tons capacity. Having been loaded directly by the shovels, they are usually hauled from the pit to an assembly yard by the regular pit locomotives, and there made up into standard trains for distant transport by heavy freight locomotives of the consolidation or Mallet type. To stand the strain of shovel loading and hard heavy service these cars must be of very strong construction. For this reason some companies have preferred to use for ore service heavy steel cars of the gondola type eliminating all dumping mechanism. For this type it is usual to provide a revolving or mechanical tipple of some sort, the use of which requires the uncoupling and recoupling of the train and is there-

6

fore slower. Also the tipple must be operated and kept in repair, so that while the upkeep and first cost of the ore cars is less, this advantage may be largely offset by the slower dumping and the upkeep and first cost of the tipple. A study of the conditions under which the ore trains are to operate will usually indicate the better type to adopt.

## TRACK

**General Pit-Track.**—As a rule in pit approaches and assembly yards, and even track on some of the benches, may be considered sufficiently permanent in character to warrant the spending of sufficient money to put it in first class condition. Such work should if possible, be laid out with not over 2 per-cent. compensated grade, and yet not necessitate unduly steep pit-tracks. On large work such track should be laid with 4ft. 8½ in. gauge and 75 or 80 lb. rail. Care should be taken to keep good grade and alignment and provide proper ballast. The curves should be made as easy as posible; as a rule 10° is considered the sharpest desirable and 20° the maximum allowable. In the pit itself 50° curves are sometimes unavoidable and can be operated over without a great deal of trouble when carefully laid and maintained. For the more or less permanent track 2½ per cent. grade should be the maximum although 3 per cent. is sometimes necessary. The pit grades are likewise kept down as low as practicable, but at times 5 per cent. and even up to 7 per cent. has been found unavoidable for very short runs. The great increase in motive power required upon heavy grades has been emphasized, and, while it is very desirable to keep within the approved limits of grade and curvature, nearly every mine has at times found it possible and necessary to exceed what may be considered practical railroad conditions.

In planning pit-trackage, the elipse or spiral is to be preferred to switch-backs, since lighter grades and easier curves and turnouts are secured; but to employ this system the pit must be large and roughly uniform in plan. Long, narrow or irregular pits or faces of ore must usually be worked with switchbacks.

It is just as essential to have competent and experienced track foremen for this class of work as for main-line railroad work.

**Definitions and Rules.**[1] *Track.*—"Track" consists of ties,

[1] Definitions and recommendations according to American Railway Engineers Association.

rails and fastenings; with all parts in their proper relative positions.

Track is a subject given the closest attention in good railroad practice but often neglected in mining operations. It is impracticable to give shovel-mine tracks the same great care and attention that are demanded for main-line railroad work because of their temporary character, but nevertheless it will be found highly economical to equip and maintain pit and yard tracks in as good shape as the conditions warrant. Poor track causes many expensive wrecks and derailments, produces much wear and tear on the rolling equipment and may very seriously cut down the efficiency of the motive-power.

The problems of grades and curvatures were previously discussed with their effects on the hauling capacity of locomotives. The resistance due to rolling friction was shown to vary over wide limits, say, from 6 to 30 pounds per ton, and to be dependent on the condition of the rolling-stock and the track.

*Alignment.*—By "alignment" is meant the horizontal location of a railway with reference to curves and tangents. It is desirable to keep the alignment uniform in direction over tangents (straight track) and of uniform variation over curves. For permanent work "witnesses" should be placed at points of tangent, points of spiral, points of change of curvature, summits and such other points along the line as will enable the alignment to be identically reproduced with a transit. For temporary track such refinements are impractical but by proper use of such auxiliary fastenings as nutlocks, tie plates, rail braces and various anti-creeping devices track can be maintained in fair alignment.

*Curves.*—A "curve" is a change in direction by means of one or more radii. The curve is simple when the change in direction is made by means of a single radius. The curve is compound when the change consists of two or more simple curves of different radii, but in the same general direction, joining one another at points having a common tangent. The curve is reverse when two curves have opposite general directions, but joining one another at a common tangent point. The curve is vertical when used to connect intersecting grade lines. The degree of curve is the central angle subtended by a 100-foot chord. Easement of curve is effected when a curve having a radius of regularly varying length connects a tangent to a simple curve, or connects two simple curves. Such a curve is also spoken of as a transition curve. "Elevation", as applied to curves, is the amount by which the outer rail is raised above the inner rail.

To determine the degree of a curve closely enough for practical purposes, take the middle ordinate of a 62-foot chord laid off on the inside rail; the length of this middle ordinate in inches approximately equals the degree of curvature.

The amount by which the outer rail should be elevated on a curve may be determined from the following formula:

$$E = 0.00066DV^2$$

Where E = elevation of outer rail in inches.
D = degree of curve
V = velocity of train in miles per hour.

An elevation of eight inches should not be exceeded.

*Gauge.*—The "gauge" of track is the distance inside between the heads of the rails, measured at right angles thereto and at a point $\frac{5}{8}$ inch below the top of the rail. Gauge is said to be "standard" when this distance is 4 ft. $8\frac{1}{2}$ in.

The spread of rails on curves of 8° or under should remain standard, but for each 2° or fraction thereof over 8°, the gauge should be widened $\frac{1}{8}$ in up to a maximum of 4 ft. $9\frac{1}{4}$ in. Gauge, including widening due to wear, should never exceed 4 ft. $9\frac{1}{2}$ in.

The installation of frogs upon the inside of curves is to be avoided where practicable, but when unavoidable, the above rule should be modified to make the gauge of the track at the frog standard.

The sharpest curve to which two fixed pairs of flanged wheels will adjust themselves, depends upon their distance apart, the diameter of the wheels and the size and shape of the flanges. Assuming the M.C.B. standard for flanges and rails and that the gauge is not widened on the curve, a sufficiently accurate formula for all practical purposes is as follows:

$$R = \frac{W}{2\sin a}$$

Where $R$ = radius in feet of sharpest curve that can be passed,
$W$ = wheel-base in inches
$a$ = angle the flanged wheels make with the rails.

The value of sin a, for various diameters of wheel, is given below:

| | | | | | |
|---|---|---|---|---|---|
| Diameter of wheels (inches) ... | 20–24 | 25–30 | 31–40 | 41–50 | 51–60 |
| Value of sin a ................. | 0.117 | 0.107 | 0.099 | 0.08 | 0.075 |

If intermediate wheels are introduced between the two pairs of flanged wheels, their relation with the rail requires a separate consideration.

When a truck is used the swing must be sufficient to allow the locomotive to pass the curve.

Adjustment of curves and refinement in laying them out, should be done by a competent trackman or engineer. The theory of curves will be found in such handbooks as Searles, Crandall, Holbrook and Talbot.

*Maintenance.*—"Maintenance" means preserving the track on proper grade, in alignment and in repair. The surface is maintained with some kind of ballast, such as earth or clay, cinders, burnt clay, broken stone or gravel, and the ties are tamped. The tools required for tamping are shovels, tamping-bars and picks. The method suggested for ordinary work is to tamp each tie from 18 inches inside of the rail to end of tie with shovel-handle or tamping-bar. It is best to tamp the end of the tie outside of the rail first and then let a train pass over before tamping inside of the rail. Give particular attention to tamping under the rail and tamp the center loosely with the blade of the shovel. The dirt between the ties should be placed in layers and firmly packed with the feet or otherwise, so that it will quickly shed the water; the earth should not be banked above the bottom of the ends of the ties; and the ballast filling between the ties should not touch the rail, but in the track centre should be as high as, or higher than, the top of the ties.

When not surfacing out of face, as when picking up low joints or other low places, the general level of the track should not be disturbed. Where the rails are only slightly and not abruptly out of level, the track may be safely operated over until such time as it would ordinarily be surfaced out of face.

To maintain the gauge, or prevent spreading of the track and canting of rails on curves, various devices are resorted to. Tie-plates are recommended in all cases for the preservation of ties, and in general better maintenance will result from their use. Shoulder tie-plates are recommended in preference to rail-braces, except for guard-rails and stock-rails at switches, where the latter should be used. For heavy or even medium traffic, shoulder tie-plates should be used on all ties on curves of over three degrees. For light traffic on easy curves the outside of rails on such curves should be double spiked.

In gauging track the standard gauge tool should be used; slight variation of gauge from the standard, say up to one-half inch, is not seriously objectionable if the variation is not abrupt. Wide gauge, due to worn rail and up to, say, one-half inch, is not objectionable, but when excessive should be corrected by closing in.

Where track is to be spiked to standard gauge, the rail should be held against the gage with a bar while the spike is being driven. The spikes should be started vertically and square, and so driven that the face of the spike comes in contact with the base of the rail; the spike should never have to be straightened while being driven. The outside spikes of both rails should be on the same side of the tie, and the inside spikes on the other. The inside and outside spikes should be spaced as far apart as the face and character of the tie will permit. The ordinary practice is to drive the spike 2½ inches from the edge of the tie; old spike holes should be plugged.

With standard gauge track the width of standard flangeways for all frogs and between main-rails and guard-rails should be 1¾ inches, measured at the gauge line.

The widening of gauge on curves has been mentioned.

The allowance that should be made for expansion for 33-foot rails is as follows (the temperature is to be taken on the rail, and the openings given are between the rail ends.)

| | | | | | |
|---|---|---|---|---|---|
| Temperature-degrees, Fah... | −20 to 0 | 0 to +25 | 25 to 50 | 50 to 75 | 75 to 100 |
| Allowance-inches.......... | 5/16 | 1/4 | 3/16 | 1/8 | 1/16 |

At over 100°F. rails should be laid close but without bumping. The expansion should always be uniform and by observing this and using care in placing plates and in spiking much can be done to prevent creeping track.

In the pits and on the dumps there is almost constant track shifting to be done. This is still largely done by hand, but track shifting machines have been devised which speed up and cheapen this operation. One satisfactory type of shifter used on the Mesabi range resembles a wrecker in general appearance and has two booms about 40 feet long, one for lifting the track and the other, on a level with the base of the machine, for side-pulling it. The machine is self-propelling, has one hoist of simple type and one horizontal boiler. The machine has a side-throw of 6 feet, but the usual method employed is to shift over about 4 feet and then move the shifter back 30 feet and repeat the operation. In doing the work thus gradually serious bending or twisting of the rails is avoided. It is claimed that as much as 3000 feet of track has been shifted 10 feet in half a day whereas to do this by hand 40 or 50 men would have been required for a full day. These machines are desirable labor savers when much track shifting, as on dumps, is to be done.

**Materials and Equipment.**—In the following table are given the requirements for one mile of track.

When using railroad type of shovels, each shovel will be equipped with about 6 ft. track sections, 36 ties, 2 or 3 pairs of good rail clamps, 2 large and 2 small jack blocks.

*Rail.*—The sustaining power of the same weight of rail varies greatly with different tie-spacings, roadbeds, etc., but it has been

TABLE 11

| Tie centers, inches apart | No. ties per mile | No. tie plates per mile | Four spikes to tie | | Six spikes to tie | | Length, rail ft. | No., rails per mile | No., splices per mile | No., bolts per mile | | Fence posts | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | No., spikes | No., kegs | No., spikes | No., kegs | | | | Four hole splice | Six hole splice | Feet apart | No., one side |
| 24 | 2,640 | 5,280 | 10,560 | 31 | 15,840 | 47 | 20 | 528 | 1,056 | 2,112 | 3,168 | 8 | 660 |
| 23 | 2,755 | 5,510 | 11,020 | 33 | 16,530 | 49 | 22 | 480 | 960 | 1,920 | 2,880 | 12 | 440 |
| 22 | 2,880 | 5,760 | 11,520 | 34 | 17,280 | 51 | 24 | 440 | 880 | 1,760 | 2,640 | 14 | 377 |
| 21 | 3,017 | 6,034 | 12,068 | 36 | 18,102 | 54 | 26 | 407 | 814 | 1,628 | 2,442 | 16 | 330 |
| 20 | 3,168 | 6,336 | 12,672 | 38 | 19,008 | 56 | 28 | 378 | 756 | 1,512 | 2,268 | 16½ | 320 |
| 19 | 3,335 | 6,670 | 13,340 | 40 | 20,010 | 59 | 30 | 361 | 728 | 1,456 | 2,184 | 18 | 293 |
| 18 | 3,520 | 7,040 | 14,080 | 42 | 21,120 | 63 | 33 | 326 | 652 | 1,304 | 1,956 | 19 | 278 |

Note. Figures for number of rails, splices and bolts are based on some shorts in rails 30 and 33 ft. long.

found that 1 lb. of rail per yard will sustain from 225 lb. of driving-wheel for the lighter rails to 300 lb. for the heavier.

While any figures regarding carrying power of rails must be approximate, those given in the following table represent the limit of good practice, and should not be exceeded.

TABLE 12

| Weight of rail per yd., lb. | Greatest weight per axle, lb. | Weight of rail per yd., lb. | Greatest weight per axle, lb. | Weight of rail per yd., lb. | Greatest weight per axle, lb. |
|---|---|---|---|---|---|
| 12 | 4,000 | 45 | 25,000 | 75 | 48,000 |
| 16 | 6,400 | 50 | 28,000 | 80 | 52,000 |
| 20 | 9,000 | 55 | 33,000 | 85 | 57,000 |
| 25 | 11,400 | 60 | 38,000 | 90 | 62,000 |
| 30 | 15,000 | 65 | 40,000 | 95 | 66,000 |
| 35 | 18,000 | 70 | 44,000 | 100 | 72,000 |
| 40 | 22,000 | | | | |

Notes: If the weight on each axle is not the same, the heavy weight should be taken.

Rails weighing 80 lb. per yard are in general use for open-pit trackage in many places.

For the inspection and identification of rails all rolling mills brand and stamp their product. The raised brand gives the name of the manufacturer, a number or abbreviation by which the rail section is designated, the month and year of manufacture, and, if the metal is open hearth steel, the letters "O.H." are added.

The rails while red-hot, but after having been completely rolled and sawed to length, are stamped on the web with the number representing the heat, blow or melt of steel, and the letter to indicate the position of the rail in the ingot.

There are many rail sections, but those in most common use are classed as A.S.C.E. sections and A.R.A. sections, the initials meaning American Society Civil Engineers and American Railway Association. The section desired should be specified. Old and worn rail is frequently used for dumps and little-used track.

In purchasing re-rolled rails great care should be used as they often prove very defective, unreliable and dangerous. Some unscrupulous dealers have attempted to sell them as rail made from new steel billets.

### Fastenings and Accessories

*Spikes.*—Cut-spikes are commonly used in the United States but in other countries screw-spikes are common. The latter are being more often recommended for permanent track because they help prevent the premature destruction of the wood fibres of the tie caused by driving the ordinary square cut spike, and because they give a greater holding power than the driven spike. The sizes of both kinds vary for different rail weights and conditions. Cut spikes range from $3\frac{1}{2}$ in. $\times$ $\frac{3}{8}$ in. for 16-lb. rail up to $5\frac{1}{2}$ in. $\times$ $\frac{9}{16}$ in. for 60-lb. to 80-lb. rail.

Tie plugs may be had of any wood desired.

*Tie Plates.*—There are many varieties of tie-plates from simple flat punched-plates to corrugated shouldered flanged-plates. The shouldered flange tie-plate with corrugated rail bearing surface and not less than 6 in. wide by $\frac{3}{8}$ in. thick, is to be recommended for soft-wood ties. They may be punched with two, three or four spike-holes as desired and are of various widths, lengths and thicknesses depending on the rail section and use they are to serve. Such tie-plates are made for both cut and screw-spikes. The three-hole punching permits the use of the plate for either right or left hand side, while a three or four-hole punching may be specified to serve two rail sections of different base width. In ordering intermediate tie-plates, give the rail base, size of spike, number of holes wanted and width and thickness of plate; for shoulder tie-plates add description of angle-bar used.

Tie-plates may be inserted under the rail after track has been laid but naturally at much greater convenience than if assembled with the track. They may also be punched with special holes in the field by means of a portable tie-plate punch.

*Angle Bars and Fastenings.*—Various forms of angle bars have been devised in an effort to secure a continuous rail-joint but the standard bar is used in pit service work. The track bolts with square nuts, used with lock nuts, are standard. Where it is necessary to bond rails for signal purposes, No. 8 iron wire or No. 6 copper wire is generally used with size $\frac{9}{32}$ in. tin-plated channel-pins. Where bonding is not practical for signal purposes, various rail-deflection and vibration devices have been designed by the signal manufacturers.

Simple derailing devices are made for attachment to any rail-section it may be desired to protect.

*Tools.*—The track tools consist of shovels, picks, gauges, drills, track-chisels, spike-mauls, double-face hammers, napping-hammer, tie-plate swages, track-punches, tie-plug punches, claw-bars, spike-pullers, pinch-bars, lining-bars, tamping-bars and picks, rail-tongs and track-wrenches. They may be had from any railway supply company.

*Ties.*—Timber ties are used, both treated and untreated. With increasing scarcity and cost of timber suitable for ties, treating is becoming more general; some steel ties are also being used, but they have not yet met with general favor for this class of work for a number of reasons.

The usual methods of treating ties are by injecting zinc chloride into the fibre, called Burnettizing; injecting zinc tannin, the Wellhouse process; zinc creosote, the Card process; creosoting by the Lowry or Rueping processes; and Wood creosoting.

The estimated life of untreated and treated ties in the United States is given[1] as follows:

TABLE 13

| Species | Estimated life (all ties properly tie-plated) | | |
|---|---|---|---|
| | Untreated years | Treated with 10 lbs. creosote per cu. ft. years | Treated with 0.5 lbs. zinc chloride per cu. ft., years |
| Black locust | 20 | | |
| Redwood | 12 | | |
| Cedar | 11 | | |
| Cypress | 10 | | |
| White oaks | 8 | | |
| Longleaf pine | 7 | 20 | |
| Chestnut | 7 | 14 | 11 |
| Douglas fir | 6 | 15 | 11 |
| Spruce | 6 | 14 | 11 |
| Western pine | 5 | 17 | 12 |
| White pine | 5 | 14 | 10 |
| Lodgepole pine | 5 | 16 | 11 |
| Tamarack | 5 | 15 | 11 |
| Hemlock | 5 | 15 | 11 |
| Red oaks | 4 | 20 | 12 |
| Beech | 4 | 20 | 12 |
| Maple | 4 | 18 | 12 |
| Gum | 3 | 16 | 11 |
| Loblolly pine | 3 | 15 | 10 |

[1] U. S. Dept. of Agriculture Bull. No. 118, Nov. 9th, 1912.

The approximate cost of this preservative treatment for ties (7 in. × 9 in. × 8 ft., or 42 board feet) is given by the same authority as follows:

| Kind of treatment | | Cost of treatment | | | | | |
|---|---|---|---|---|---|---|---|
| Preservative | Amount injected, lbs., per cu. ft. | Seasoning | Labor | Fuel | Maintenance | Chemicals | Total |
| Creosote...... | 10 | $0.01 | $0.06 | $0.01 | $0.015 | $0.279 | $0.375 |
| Creosote...... | 6 | .01 | .06 | .01 | .015 | .168 | .263 |
| Zinc chloride.. | .5 | .01 | .06 | .01 | .016 | .070 | .166 |
| Creosote<br>Zinc chloride | .3<br>.5 | .01 | .06 | .01 | .016 | .154 | .250 |

The economic avantage of this treatment is not only in more than doubling the life of the tie, but in sawing the cost and inconvenience of replacement.

The care of ties in storage, as well as in usage, should be considered. They should be piled in open-crib or some way as to be well ventilated yet protected from ground water, decayed grass, weeds and wood. The piles should be plainly marked and spaced for inspection and seasoning. Zinc-treated ties must not be placed in service until they have seasoned the full time prescribed for that purpose.

When the ties are placed in service, note that the year rings of growth point downward.

The use of tie-plates is of great importance to prevent the mechanical wear between the rail and the tie.

*Frogs and Switches.*—Standard track material is generally used in pit-work and can be secured from manufacturers of such material to suit every purpose. It should be noted that for frogs, switches and crossings the use of manganese-steel, both of the insert and solid type design, has shown great economy and is strongly recommended. It will usually be found more satisfactory to purchase such track material from specialists in its manufacture than to try to build it in the plant shops. A full description of the conditions under which the part is to operate will usually result in securing from the manufacturer most satisfactory designs and workmanship.

There has never been any question about the life of a spring-frog as compared with a rigid-frog, but the former has not always been considered safe. Safe designs are obtainable now and should be used where practicable.

Guard-rails are usually placed opposite frogs on one or both tracks. They are usually simply spiked and braced, or may be held with adjustable clamps and division-blocks and bolts. Satisfactory guard-rails are also made of manganese-steel, all in one piece, with braces, spike-lugs and plates ready for installation.[1] With sharp turnouts, guards are recommended to be ¾ in. higher than the track rail. The toe end of a frog on a curved run should be braced with shoulder-plate or track-brace on the first two or three ties forward of the ½ in. frog-point to maintain track gauge. These require but little upkeep.

Very rugged frogs are also manufactured of solid manganese-steel in one piece and with wheel-flange protective flanges.[2] With their use it should not be necessary to guard-rail the tangent track.

The split-switch should be used wherever possible though stub and lap switches may sometimes be necessary. The switch should be well reinforced on both sides of the web by bars, or should be made of manganese-steel. Where the points are well reinforced it is not necessary to have more than two bridle-rods, and the rail is less liable to fracture or part. The bridle-rods may be attached to the web or base of the rail by bolts or patented safety devices, but good die-formed clips attached to the web of the rail, and holding the rods in a vertical position, are very satisfactory. The head-rods are sometimes made adjustable by means of a clevis or screw.

Switch-stands are of numerous designs ranging from simple ground-throw stands through pony-stands to high main-line stands, or even the high ladder-stands. Their selection is largely a matter of personal choice and adaptability. Stands with single targets are to be preferred to those with dual targets of different colors.

*Protective Devices.*—These should be employed wherever possible when danger exists. Crossing-bells are now in service on most railroads and are reliable both day and night. They should possess loud-ringing bells and attractive visual signals for both day and night. For dangerous curves, switches and yards where there is possibility of collision or accident, block signals of an

[1] Ajax Forge Company.

[2] Connelly Frog Company.

approved type will be found a great economy. Safety devices and their application are a very important and broad subject which cannot be but touched upon here, but if given careful study by pit operators they will find it more than recompenses every effort they make in safeguarding their men, equipment and continuous operation.

## DRILLS

Preparation of the ground to be excavated frequently requires more or less loosening or breaking up before it can be efficiently handled. This operation is usually done by the aid of explosives placed most advantageously. The drilling of holes to accommodate such charges of explosives is one of the common methods. The drills used for this work may be divided into two classes, viz. churn drills and piston drills. This class of drilling is not to be confused with prospect drilling.

**Churn-drills.**—Churn drills have to a large extent displaced the tripod piston drills. The reason is that it has been found more economical to sink a few holes, say, 6 in. in diameter to the full depth of the cutting, say, 50 to 150 ft., than to drill a large number of small shallow holes at various levels. This has generally resulted in speeding up the breakage with all attendant economies and convenience, and also in greater safety. In some cases the cost of drilling and explosives have also been reduced. By this method benches and faces are usually broken in one stage. Churn drills are best adapted to work where the height of the face exceeds 25 ft. and where the formation, if stratified, is more or less flat. Holes 5⅝ in. in diameter are recommended for the harder rocks because more explosive can be concentrated in the bottom of the hole, where it is usually most needed, and the larger hole permits a greater spacing interval and thus a lesser drilling cost. In the softer rocks, such as shale or sandstone and even some limestones, or when working a shallow face, the use of 4 in. or 4½ in. bit may be more economical because distribution of the charges rather than concentration is desired. Thus a 5⅝ in. hole requires, say, 15 lb. of dynamite per foot whereas a 4 in. hole requires only about 7½ lb. Economy in explosives may thus be gained where the conditions warrant the lighter charges.

Drills suitable for this work are made by the Keystone, Star Cyclone and other driller companies. They range in sizes from those drilling holes of 4½ to 8 in. in diameter and may be traction or non-traction, and steam, gasoline or electric driven.

For general blast hole work, the Keystone steam traction size No. 3½, drilling a 5⅝ in. hole, is well suited; for very light work, the non-traction size No. 1, drilling 4½ in. holes, will serve; but for hard usage and heavy work such as is found at most of the copper mines, the traction size No. 5, drilling a 6 in. hole, will be found most serviceable.

Selection of motive power will depend to some extent on local conditions, as steam, electric and gasoline driven rigs have all been satisfactorily developed. The steam drill is the speediest, handiest and most reliable, but presents the questions of water supply, cost of fuel and fireman. The electric drill is lighter, moves about under its own power, within range of its wiring, and requires no fireman or fuel and water care. Cyclone electric rigs have given satisfactory service at the Chile Copper Company's mines.

The gasoline drill has many of the advantages of the two former but it is not as flexible or reliable. Experience with gasoline engines leads to this conclusion. In some cases the steam rigs have been run with fair success by compressed air. Everything considered the steam traction rig is most widely preferred.

The Keystone No. 3½ traction steam machine weighs about seven tons and the No. 5 about nine tons. The Star drills are heavier; those at Utah Copper had 16 h.p. engines with 8 in. ×8 in. cylinders. The cranks are adjustable but run on about 18 in. stroke. The weight of the tools on these drills is 2000 lb. They are used principally for prospect drilling and not for shallow holes. As the holes drilled on this work are comparatively shallow (50 to 150 ft.) they are drilled by "spudding." Jars are seldom required, consequently the strings of tools are short and a shorter mast may be used, thereby lowering the centre of gravity. On some works it has been found advantageous to convert the operating rods into levers, placing them on the inside of the bed; the walking beams have been reinforced and the spudding wheels on the beam have been attached to I-beams; in some parts cast-steel has been substituted for cast-iron. Such changes have been made to withstand rough usage.

Where eight or ten drills are employed each drill carries the following equipment:

200 feet of 2-inch manila drill cable (hawser-laid)
150 to 160 feet of $\frac{3}{8}$ in. wire sand-line.
1—standard Keystone rope-socket.
1—4 in. or $4\frac{1}{2}$ in. $\times$ 20 ft. drill-stem.
1—$5\frac{5}{8}$ in. No. 100 Mother Hubbard bit.
1—12 ft. sand-pump (may be made from $4\frac{1}{4}$ in. casing).
1—10 in. Keystone steam-driven blower (with steam rigs).
1—forge, consisting of a pipe 2 in. $\times$ 36 in. and a wood box 32 in. $\times$ 36 in.
1—right-hand tool wrench for $3\frac{1}{2}$ in. squares.
1—left-hand tool wrench for $3\frac{1}{2}$ in. squares.
1—single-acting floor-jack and circle.
2—No. 6 Barrett lever hoisting jacks.
1—spectacles.
1—anvil billet and block (consists of a piece of 75 lb. rail one foot long spiked to a 2 in. $\times$ 12 in. plank.)
3 to 7—50-gallon water barrels (3 used in summer and 7 used in winter).
2—gasoline torches.
2—16-lb. sledge hammers.
1—set small tools, such as machinist hammers, wrenches, files, oilers, etc.

Such small repair parts as extra water glasses, gaskets, pipe-fittings, etc. are usually kept on hand, as well as a fair amount of extra tools, spare parts and casing.

It has been found that 200 feet of drill cable is the best length. At least 125 feet are needed if the mast is to be lowered or raised. The rope will wear most where it is in contact with the spudding pulleys and sheave pulley, and in time must be spliced. Three splices will shorten the rope about 40 feet. After a rope has been worn out there is about 60 feet of good rope left on the drum, and generally this 60 feet can be spliced on to a rope having three or four splices, making a practically new rope for that rig while the other drill receives a new rope. In this way much valuable rope can be saved.

The drilling crew consists of one driller and one tool-dresser.

Coal is supplied to the drills from piles conveniently placed along the benches.

When a drill must be moved from one bench to another, or for a considerable distance, much time may be saved and wear on the drill avoided by loading the drill onto a flat car by means of a locomotive crane. This is illustrated in Fig. 17 at the Nevada Consolidated Copper Company. To do this, three chains of proper length are attached to the drill, one just above the front axle and the other two to the two rear wheels. The mast is not lowered nor are the tools taken off the machine; the fire is not drawn nor is the boiler blown out. A barrel of water is put on the bed of the drill and all other equipment is loaded on the flat car. When the drill is unloaded it moves to its new spot and begins drilling. The barrel of water serves until the necessary water connections are made.

This method obviates preparing a road; requires but 3 men on the crane beside the drill crew, instead of at least 10 men on roads and moving; causes no delays or damage on haulage tracks passed over or along; and relieves the drill of most of the wear and tear of moving. When it is necessary to move a drill on its own power, the tools are tied beneath the rig and

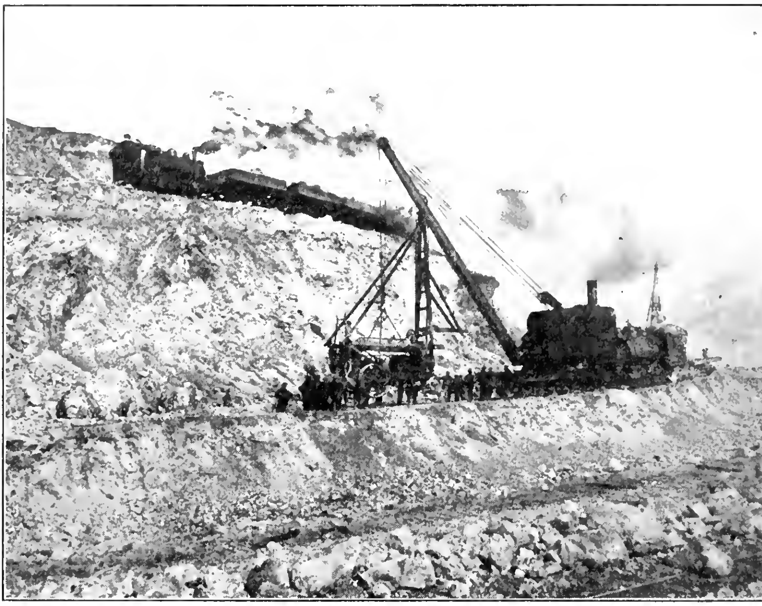


Fig. 17.—Moving drill with locomotive crane.

dragged along the ground; a barrel of water is put on the bed of the machine and the remainder of the equipment is carted along on a push car or by teams.

These drills carry a working steam pressure of 100 lb. per sq. in. They consume per shift about 600 lb. of coal, twelve 50-gallon barrels of water (for boiler and drill hole); $1\frac{1}{2}$ pints of cylinder oil; 1 pint dark lubricating oil; $\frac{3}{4}$ lb. cup grease; $\frac{1}{2}$ gallon of gasoline for night lighting; and $\frac{1}{4}$ lb. cotton waste.

With hard mining usage they require many renewals and repairs but by a little careful study the weaker parts can be strengthened by increasing their size or changing the material. Repairs are made in the field until the drill loses its efficiency, when it is taken to a shop, entirely dismantled, examined and rebuilt.

The footage drilled and cost per foot of this work has been found to be about as follows:

| Character of ground | Footage per 10 hr. shift | Average depth of holes, ft. | Average cost per ft. |
|---|---|---|---|
| Soft altered porphyry | 60 | 20–60 | 25¢ to 35¢ |
| Harder altered porphyry | 40 | 25–60 | 35¢ to 40¢ |
| Cement quarry limestone | 40 | 45–70 | 25¢ to 30¢ |
| Flat limestone | 35 | 50 | 30¢ to 40¢ |
| Hard trap or granite | 20 | 50 | 40¢ to 60¢ |

Under average conditions these drills should make 30 ft. per shift at a cost of 30 cents per foot.

The percentage of time in drilling may be estimated at 85 per cent.; in moving 5 per cent.; in repairing and other stoppages, 10 per cent. About 75 per cent. of the cost is for labor and 25 per cent. is for supplies. Keystone drills, sizes 3½ and 5 steam tractors, cost about $1200 and $1400 each respectively at the factory.

**Tripod Drills.**—Tripod drills are used in cases where relatively shallow holes with light powder charges are best suited; where benches are too narrow for the use of churn drills; where it is necessary to shoot the toes of high banks, or midway points in such banks. Such holes may be drilled at any angle, whereas the churn drill hole must, of course, be vertical. These holes average about 1½ in. in diameter and run from 10 to 25 feet in depth. The drills may be operated by air or steam; single or in batteries. They are built by the Ingersoll-Rand, Sullivan, Denver, Rockhill and other rock drill manufacturers. The care of the drills and steel is the same as for underground practice.

Both the reciprocating piston machines and the hammer or Leyner type are used. The latter require a water-feed attachment but are faster drillers. The 18-A and 248 Leyners and the Waugh Turbo are the newest examples of the hammer and turbo types. The 18-A costs about $255, the 248 about $335 and the Turbo about $360, without tripods or steel.

The cost of drilling ranges from 10 to 25 cents per foot depending on the character of rock, costs of labor, fuel or power, oil, etc., number of drills in operation and nearness to source of power and water. The fewer the drills, the higher is apt to be the cost per foot. The footage made per 10 hr. shift ranges from 40 to 60 ft. in rocks of medium hardness, such as limestone, and from 20 to 30 ft. in traps and granites.

Large drills of the piston type have been specially built for some companies, notably at Rio Tinto, Spain, where slope holes were necessary. These special machines are very heavy and powerful, suitable for drilling 2-in. holes to depths of 25 ft. or more. They are moved about on push cars.

In operating drills of this type power is supplied them by a central compressor plant with attendant air lines or by steam boilers near the drills; long steam lines are inefficient. Electric drills, with the exception of the Temple-Ingersoll, have not been sufficiently perfected for this service.

**Drills for Block-holing.**—After a bank has been shot, there frequently remain many fragments or boulders too large to be handled by the shovels. Again it sometimes happens that a hole will not completely break the bottom of the bench, so that it must be further shaken up.

For this work powder may be placed directly on the face of a boulder or "bottom," covered with mud or dirt and exploded; or holes may be drilled in the face and the powder charged as usual. The former method is called "mud-capping," "dobying," "bull-dozing" or "burning bottom," and the latter "block-holing."

Block-holing is much more effective and economical in powder but may not be desirable for other reasons. If block-holing is found advantageous the drilling may be done by hand or with light plugging or rotating drills. Of these small drills the Jackhammer BCR is the most satisfactory. In case no air or steam line is available, it is an easy job to equip the shovels with a small air compressor similar to those used for locomotive train service. These have a capacity of 150 cu. ft. free air per minute, are entirely self-contained and will operate three Jackhammer drills. The outfit supplied by the Westinghouse-Pacific Coast Brake Company is made up as follows:

One 10½ in. cross compound air compressor; one S-6 governor; one sight-feed lubricator; 1½ in. steam valve; one reservoir drain cock; one air gauge; one $30\frac{1}{2}$ in. × 72 in. reservoir; one compressor stand. The cost of the outfit is about $400.

The BCRW Jackhammer drills cost about $140 each.

Such equipment is recommended where local conditions do not interfere.

## MISCELLANEOUS EQUIPMENT

*Pumps.*—In many open-pit mines arrangements must be made to take care of more or less surface and ground water.

Pumps are generally employed for dewatering simple pit sumps. In some instances in the Lake Superior iron ore pits, a system of drainage drifts is run beneath the pit. These lead to a pump shaft on the edge of the pit. Such pits often handle from 500,000 to 2,000,000 gal. of water per day. As pits increase in size and depth, the water to be handled is likely to increase. There is usually a wet season to contend with, and in some instances even cloud bursts have occurred which have done considerable damage to the workings and equipment. Again, as in the case of copper mines, the pit waters may be very corrosive, necessitating pre-treatment or special pipe lines and equipment. All of these points must be considered in determining what dewatering equipment will be required. With a fair idea of the quantity, lift and nature of water to be handled, any of the better known pump makers will be able to advise the most suitable equipment for the work. With a little care a pit drainage system can generally be so arranged that the pumps and pipe lines will require but little shifting and may be given more care in installation. Electric power, if available, is usually by far the most economical and satisfactory. Whether the pumps be centrifugal or reciprocating, geared or belted, will depend on local conditions. Some sort of housing should be provided for them and reasonable care given to lubrication and repair. For short temporary work a light pump mounted on a wheeled bed is very convenient.

*Illumination for Night Work.*—Most open pit mines are required to work night shifts in order to fulfill production requirements. When this is the case, the problem of adequate illumination must be carefully considered.

Permanent locations about the property will usually be served with modern electric incandescent outdoor-type equipment. For less permanent locations, such as along edges of dumps, movable incandescent electric light tripod stands, built of light pipe and served with flexible armoured cable, will be found very convenient. These stands should be about 14 ft. high and carry one 100 c.p. lamp, with open reflector, at the goose neck. The frequency or spacing of the stands will depend on dumping conditions, but there should be sufficient of them to enable the workmen to move about with safety. Flood lights may at times be used to advantage if properly placed and mounted. Current for all of this lighting equipment is presumed to come from the general property circuit.

For night work with the shovels, adequate illumination is imperative, not only on the bank from which the shovel digs, but around the shovels themselves, to permit inspection and adjustment and to afford safety to the workmen. Acetylene was first used but in comparison with electricity, it has been found more troublesome, costly, and less safe. At one time the Utah Copper Company illuminated its benches by three powerful searchlights set on the opposite side of the canyon. Current for this lighting may be taken from local pole lines through a reeled cable to the shovel, but from the nature of the work such lines will be troublesome to maintain, and the voltage is usually higher than that desired for use with incandescent lamps having the toughest filament. A better means of supplying current is to mount a 1 KW. turbo-generator outfit in a convenient place on the shovel, and drive it with steam from the boiler. One of these sets will operate about forty 25-watt incandescent lamps or their equivalent. They require little attention and are rugged and accessible for inspection. They operate to full rating with steam pressures between 90 and 250 lb. A governor keeps the speed uniform and the light steady. A fixed resistance regulates the voltage and a rheostat is provided to adjust the voltage when the number of lamps is varied.

A comparison on one property of the relative cost of lighting with acetylene and by these turbo-generators, was estimated as follows:

Carbide: Average carbide used per steam shovel per month; 2800 lb. at \$.0415 = \$116.20

1 KW. turbo-generator: Average steam consumption 200 lb. per hr. at $6\frac{2}{3}$ lb. per lb. of coal = 30 lb. coal per hr., at

9 hr. × 30 shifts = 8100 lb. or 4.05 tons, at $5.65 = $22.88 per month.

In both cases attention and maintenance must be added, and the carbide plant requires more care.

For locomotive lighting the same arguments may be applied. A turbo-generator of 1/3 KW. is sufficient for illumination. This will supply current for a 250-watt incandescent headlight plus 4 cab lamps, or two 150-watt headlights plus 2 cab lamps. The steam consumption at this load is about 90 lb. per hr. The voltage regulation is entirely satisfactory. A running test with these headlights gave a distance of 705 ft. with the 150-watt lamp, and 1163 ft. with the 250-watt lamp. The test was made at 34 volts; weather conditions favorable.

The cost of operating carbide lights compared to the turbo-generator was estimated as follows:

Carbide: Average use per locomotive per month, 232 lb. at $0.415 = $9.63.

1/3 kw. turbo-generator: Average steam consumption 90 lb. per hr. at 6 lb. per lb. of coal = 15 lb. coal 1 hr., at 9 hr. × 30 shifts = 4050 lb. or 2.025 tons, at $5.65 = $11.44 per month.

Attention and maintenance must be added in both cases.

The cost of the 1 KW. unit is about $125 plus lamps, plus $20. for attachment case. The 1/3 KW. unit costs about $75. The Westinghouse Electric and Schroeder Headlight Companies manufacture such units.

Illumination for drill rigs is usually with acetylene or gasoline torches, the latter being preferred.

*Telephones and Signals.*—It will be found well worth while to have an adequate telephone system covering all principal outdoor and indoor operations. This class of mining usually covers so much ground that means of quick communication is very desirable for efficiency and safety of operation. The location of the telephones must be left to the good judgment of the management. An efficiently operated central switch-board is very desirable and should be "on duty" at all times. Some installations using automatic telephones have been quite satisfactory, but it is questionable whether they show much economy in operation over the manual board for work of this kind. Care should be given in the selection of instruments exposed to the weather. High-class boards, lines, cables and instruments will be cheapest to maintain and most satisfactory in the long run. It is well to allow for additions and extensions when installing equipment. Such companies as the Western Electric and Tromberg Carlson are prepared to study conditions and supply or install such equipment.

In this connection note that fire or accident sirens may be so installed as to be operated directly or indirectly from the telephone central in case of emergency.

*Locomotive Crane.*—One or two self-propelling locomotive-cranes of about 30 tons capacity will be found indispensable for clearing up wrecks, unloading and loading coal machinery and supplies, moving drills and much other work. For heavy work a short goose-neck boom should be provided, also clam-shell and orange-peel buckets. These machines are useful in doing a certain amount of excavation work where the cut is too shallow or the job too small for a shovel. Such jobs would be slower and more expensive if done by hand.

*Dump Plows.*—These have been profitably used in dump building on the Mesabi. They vary in size from those having a a spread of 5 feet, called "dozers," to large dump spreáders which cut 18 to 24 inches below the dump track and distribute the dirt for a width of 20 to 30 ft.

*Repair Car.*—A master mechanic's repair car will be found of great convenience in going about the pits making repairs to shovels and washing out boilers on the job. In this way such equipment will not have to be brought into the shops, except for general overhaul, and a saving in time can be affected in much of the repair and upkeep work.

The following equipment fitted into an ordinary box car with two-ton coal bunker will be found satisfactory.

1—25 hp. Economy boiler, return flue, equipped with one ½ in. and one 1 in. injector.

1—10½ in. cross-compound air-compressor, with air receiver 24 in. dia. by 48 in. long.

1—air motor for drilling and boring.

1—air hammer.

1—Vulcan welding outfit complete.

1—6 in. vise and 1—3 in. pipe-vise fitted on work bench.

1—3½ in. × 4 in. × 3½ in. duplex water pump.

1—hand drill-press.

1—1 ton and 1—2½ ton chain blocks.

2—50 ton Norton jacks and 2—15 ton Barrett jacks.

2—12 in. and 2—6 in. screw-jacks.
Hose and nozzles for boiler washing.
Such miscellaneous hand-tools as wrenches, hammers, chisels, etc.

*Employees Car.*—Time may be saved in getting crews to work by fitting up a car to distribute and collect them at their points of work. By this means, they may be brought in from work to a central mess point where a hot lunch is provided. For this purpose a box car, fitted with a stove, benches and straps, and hauled about by one of the locomotives, is often of considerable convenience.

*Coaling Car.*—A specially designed steel coaling car may be provided so that after loading from a coal stock pile by means of a clam shell, it can be run around to the different shovel points, and a shift of coal drawn from it from side chutes.

*Powder Cars.*—Cars of the refrigerator type may be employed to distribute powder from magazines to the blasting points. In such a car the powder will be kept in proper thawed condition in the coldest weather, until it is actually required for loading. In warmer weather a flat car covered with a tarpaulin may be used for explosive distribution.

*General Utility Cars.*—From four to six flat cars will be found handy for transporting all sorts of parts and materials to and from the works. Two or three light four-wheel push-cars should also be provided.

*Wagons and Trucks.*—Some means of transportation must be provided for distributing materials to points not accessible by rail. Where the haul is fairly long and the roads fairly good, motor trucks can be employed to advantage. Where the haul from rail to such points is comparatively short or the roads very poor, teams and wagons may prove more satisfactory and the cost of operation cheaper. At some properties both methods are used because of varied local conditions.

**Shops.**—Well equipped adequate work shops are the principal tools with which operations are carried on. They should be planned with much care and then built and equipped as early in the construction period as possible. They will then be available to serve during the construction and opening up period as well as later Their importance is secondary only to the proper housing and care-taking of the operating personnel.

For shovel mines there are required machine, blacksmith, engine and car, electrician and carpenter shops. For a property producing say from 10,000 to 20,000 tons per day the following equipment should prove adequate.

*Machine Shop.*—The building will have a floor plan of, say 120 ft. × 180 ft., steel construction, well lighted, heated and ventilated, and will be served with single or double track connections from the outside. It will be equipped with modern tools. Individual electric drive is for the most part preferred. The makes of tool are optional and subject to personal opinion but those given will be found satisfactory. It is best to see that lists drawn are up to date as the following list is intended to be suggestive only.

1—25 ton capacity travelling crane with 50 ft. to 60 ft. span, and with a 5-ton auxillary hoist. N. B. P. Co.

1—air compressor 500 cu. ft. min. class P. E. 2 I–R Co., motor driven.

1—high pressure blower—motor driven No. 4 B. F. S. Co., with suitable reservoir.

1—1100-lb. single frame steam (or air) hammer with cast steel anvil and anvil cap. No. 1518 N. B. P. Co.

1—1½ in. head and forging machine for bolts, rivets and small forgings. A. M. Co.

1—bending rolls, capacity 10 ft. wide × 1 in. thick. Motor driven with separate motor for vertical adjustment of top roll. Size No. 7 N. B. P. Co.

1—bending rolls "pinching type" capacity 6 ft. wide by ¼ in. thick. N. B. P. Co. size No. 1.

1—plate flanging clamp 12 ft. × 4 in., to be raised, lowered and clamped by air. N. B. P. Co.

1—48 in. double punch and shears. Size No. 1 L. & A. Co.

1—bar cold saw cutting-off machine, high duty type, 36 in. blade with regular clamps and air clamps, with automatic saw sharpening machine No. 2471; No. 503 N. M. T. Co.

2—patent inserted tooth milling saws with high speed teeth fine pitch for structural iron. H. B. S. M. Co.

1—ditto with teeth coarse pitch for bar iron. H. B. S. M. Co.

1—100-ton hydraulic forcing press, to be operated by hand; used in a pit 4 ft. deep × 6 ft. wide × 30 ft. long; used to straighten car doors, dipper sticks, car sills, I-beams and any large long pieces. V. I. W. Co.

1—96 in. 400-ton hydraulic driving-wheel press, inclined type N. B. P. Co.

1—3 hp. U. S. grinding stand; U. S. E. T. Co.

1—5 hp. floor grinder.

1—arbor press, No. 6 G. T. E. Co.

1—Economy power saw, No. 4. W. R. M. & F. Co.

1—6 in. pipe threading and cutting machine with standard equipment. L. M. Co.

1—2½ in. double bolt cutter, equipped with lead screw on one side. A. M. Co.

1—2 in. double bolt cutter without lead screw. A. M. Co.

1—24 to 36 in. upright special pattern, shaft driven high speed drill with tapping attachment, round table, variable speed motor drive. C. P. T. Co.

1—28 in. upright high speed shaft driven drill, round table tapping attachment. C. P. T. Co.

1—5 ft. high speed full universal radial drill. A. T. W. Co.

1—7 ft. high speed full universal radial drill. A. T. W. Co.

1—25 in. crank shaper. S. & M. Co.

1—42 in. standard planer, 10 ft. table, with inside heads, reversing motor drive. N. B. P. Co.

1—No. 5 universal high power milling machine—motor driven by silent chain. 14 in. universal dividing head. Plain vise. Toolmakers universal vice. All steel machine, vice 10 in. × 2 in., oil pump, and the following arbors: 90–B, 91–G, 92–G, 93–J, 94–J, 95–J, 96–J, 101–J, 102–J, 103–J, 209–A, 210–A, 211–A, 212–A, 213–F and 232–A. Shell end mills: 401, 403, 405, 407, 410, 412, 601, 602, 603, 604, 605 and one 12 in. standard face mill. C. M. M. Co.

1—universal grinding machine for grinding all kinds of tools and cutters.

1—36 in. 20 ft. bed, selective head standard engine lathe, taper attachment, steady and follower rests.

1—24 in., 16 ft. bed, ditto lathe.

3—20 in., 12 ft., bed, ditto lathes.

1—20 in., 10 ft. bed, ditto lathe.

Above Lathes by L. & S. M. T. Co.

1—36 in. all steel independent lathe chuck.

1—24 in. all steel independent lathe chuck.

1—22 in. all steel independent lathe chuck.

4—18 in. all steel independent lathe chucks.

The 24 and 36 in. chucks for 36 in. lathe—U. M. Co.

If locomotive tire turning is to be done by the mine, and not at some railroad shops, a wheel lathe of a size to do the work will be required. This is an expensive tool but can be used for special large work not accomodated by the 36-in. lathe. N. B. P. Co. Omitting the wheel lathe, a boring mill can be used for large work but not as broadly.

For the blacksmith shop, add a forging press. This will be found especially useful and economical in making bolts, grab-irons and numerous other similar parts.

The foregoing only includes the large tools; there are a great many small tools, including pneumatic riveters, which should be selected by the shop foreman or master mechanic, but need not be listed here.

References:

N. B. P. Co.—Niles Bement Pond Co. N. Y. C.
I. R. Co.—Ingersoll-Rand Co., N. Y. C.
B. F. S. Co.—B. F. Sturtevant Co., Boston, Mass.

A. M. Co.—Acme Machine Co., Cleveland, O.
L. & A. Co.—Long & Allstatter Co., Hamilton, O.
N. M. T. Co. Newton Machine Tool Co., Philadelphia, Pa.
H. B. S. M. Co—Huther Bros. Saw Mfg. Co., Rochester, N. Y.
V. I. Co.—Vulcan Iron Works, Chicago, Ill.
U. S. E. T. Co.—U. S. Electric Tool Co., Cincinnati, Ohio.
G. T. E. Co.—G. T. Eames Co., Kalamazoo, Mich.
W. R. M. & F. Co.—W. Robertson Machine & Foundry Co., Buffalo, N. Y.
L. M. Co.—Landis Machine Co., Waynesboro, Pa.
C. P. T. Co.—Cincinnati Pickford Tool Co., Cincinnati, O.
A. T. W. Co.—American Tool Works, Co.
S. & M. Co.—Smith & Mills Co., Cincinnati, O.
C. M. M. Co.—Cincinnati Milling Machinery Co., Cincinnati, O.
L. & S. M. T. Co.—Lodge & Shipley Machine Tool Co., Cincinnati, O.
U. M. Co.—Union Manufacturing Co., New Britain, Conn.

*Blacksmith shop and forge.*—This may be conveniently placed adjoining the machine shop but should be partitioned off in such a way as to prevent dirt and smoke from passing into the machine shop, yet affording easy communication for transfering parts under repair. The railroad track should extend into the forge. The steam hammer may be placed in this shop or a second one provided. The forging press above mentioned will be placed in this shop. It will contain four or five forges, blown with low pressure air and served with one or two crawls with chain-blocks for supporting heavy work. There will be provided bench room with all necessary vises and hand tools. Convenience in storage and handling coke and coal will be considered. An oxy-acetylene welding outfit should be provided. In case tripod rock-drill equipment is to be used provision must be made for the testing and repair of the machines and for the forming and sharpening of the drill bits. This equipment may include a Paynter drill testing machine, bit muffle-roasters and automatic drill sharpeners.

*Engine and car shops.*—The first may be planned to house and repair motive power and the second to overhaul and repair rolling stock. Both will be provided with the necesary tracks repair pits and should be served with a suitable crane. Few tools are required in these shops because most of the work will be done in the machine, forge and wood shops. A flue cleaner, pipe and machinist's vices, screw-jacks, hose and nozzles for boiler washing and miscellaneous hand-tools such as wrenches, hammers, chisels, etc., will be provided. A sand house should be located near the engine shop Here sand will be dried and kept ready for use.

*Electricians' shop.*—The size and equipment of this shop will largely depend on the quantity and type of electric equipment used on the job. Even where this equipment is limited some space should be alloted to this work and the installation should be considered by the chief electrician.

*Carpenter shop.*—This shop will take care of woodworking and wood repairs. It will be convenient to have the following equipment: Universal woodworking machine, band saw, circular saw table, swing cut-off saw, grindstone, band saw sharpener, and an assortment of hand and bench tools. Care will be given to the wiring and drives of the machines to avoid fire risk.

*Foundry.*—At some isolated properties it has been found economical to put in a foundry for the casting of iron, brass and other parts. If such a plant is deemed advisable it should be planned by the master mechanic and a capable foundryman.

*Sampling and assaying.*—Equipment will not be detailed here because, while important, it is a subject belonging to mining in general, and not power-shovel work in particular. The same may be said of engineering and surveying paraphernalia. In the case of bank sampling, the assay laboratory should be prepared to make quick returns as they may govern the classification and routing of material.

**Balance in Equipment.**—One of the most important problems in power-shovel mining equipment is the selection of the proper number of machines of each kind to most efficiently balance all operations. Such a balancing of equipment must be determined for each property. The governing factor is the desired output. Theoretically, if the ore production is to be a given amount and the ratio of overburden to ore is known, the amount of yardage to be excavated per day can be stated. Then by estimating the capacity of the type of shovel selected when working under the local conditions, the number of shovels required can be determined. The shovel will then be considered as the governing unit of equipment and the endeavor will be made to reduce to a minimum delays to its operation. With this in mind, the proper number of locomotives, cars and drills will be estimated. Most of these conclusions will require considerable experimental work because there are so many factors which enter into each individual problem that they cannot be expressed by a mathematical formula. Some 10 to 20 per cent. additional equipment should be added for spares.

At one important efficiently operated property, having eight 95-ton steam shovels, it was found that the greatest efficiency resulted using five shovels, moving about 1200 cu. yd. solid per shift, and thirteen 65-ton locomotives. Of the thirteen locomotives, eleven were kept serving the shovels, one was doing general switching and other duty, and one was up for repairs. The capacity of the locomotives depended on the grades from and to the shovels and the length and grade of track to the dumps. About seventy-five 20 cu. yd. dumps cars could have been used to advantage· but there were only about sixty; of these about fifty-five were kept in service and the other five were on the rip track for repairs. Perhaps an average of 1½ shovels were kept working in ore and consequently loading directly into ore cars, while an average of 3½ shovels were working in overburden requiring the service of the 20-cu. yd. dump cars. There were periods when an increase in ore output was called for so that it became necessary to put more shovels in service at the expense of efficiency. At other times it became necessary to remove overburden at a number of places simultaneously in order to uncover sufficient ore for the normal daily requirements, and to ship material for treatment that would approximate the character of "average ore." This required spotting shovels at a larger number of paces which increased the number in service and decreased the efficiency. Under ordinary stripping conditions it was found that two locomotives should satisfactorily serve one shovel, each making 7 or 8 round trips per 9 hour shift, and that 7 or 8 overburden cars were required for each locomotive.

It was found that about eight No. 5 Keystone drills, drilling an average of 50 ft. of 6 in. hole per 9 hour shift, were required to keep ore and overburden broken ahead for 7 shovels.

At another property, working in rather harder material, there were eleven 100-ton shovels; twenty-one 50-ton locomotives; seventy-five dump cars, of which about fifty were small 12 cu. yd. cars, and twenty-five were of 20 cu. yd. capacity; and ten churn drills on blast-hole drilling. Some supplementary drilling was done by air drills. There was a shortage of at least thirty 20 cu. yd. cars. Here also the shovels working in ore loaded directly into ore cars.

A third property operated with twenty 95-ton shovels; thirty various sized locomotives, and about 250 dump cars averaging about 12 cu. yd. capacity. The ore was loaded directly into ore cars.

A Mesabi stripping operation called for 3 type 91 (120 tons) Marion steam-shovels; 10 standard locomotives with 19 in. × 26 in. steam cylinders; 110 20-cu. yd. K. & J. automatic air-dump cars; 1 wrecking crane; 1 dump plow; and 2 flat cars. Here no drilling and blasting was required except for breaking boulders.

All of the above mentioned equipment is of standard gauge and uses steam power.

There are openpit operations, such as those in the Kansas coal-fields, where the overburden need neither be blasted nor transported beyond the reach of the shovels. In such cases the ratio of stripping equipment to coal removal equipment is more simple.

On some works, where large revolving 300-ton shovels are used, the best possible train service to shovels is further emphasized. On an important canal job in Ontario four trains were provided for each 300-ton electric shovel Each of these trains consisted of one 50-ton electric locomotive and ten 20-cu. yd. automatic air-dump cars. The haul from shovel to dump was from 2 to 2½ miles on 1 per cent. adverse grade. As many as 290 of the 20 cu. yd. cars, (actually carrying about 17 cu. yd. water measure) were loaded per 10 hours. The shovel record was about 8500 solid cubic yards of earth excavated in two shifts. This material required no blasting. The canal section in earth ran about 160 ft. wide at the top, 50 ft. wide at bottom and 45 ft. deep.

**Life of Equipment.**—The life of equipment depends not only on the class of work upon which it is put but on the care with which it is handled and the thoroughness and promptness of making repairs.

On the iron ranges, the useful life of steam shovels should average from 12 to 15 years; their annual shop bill for repairs will run from $3,500 to $6,000. In winter, working on overburden, the general repairs to shovels has run as high as 10 cents per cu. yd. of material moved. Obviously a properly designed new shovel, or one just thoroughly shop-overhauled, should show less repair cost than a shovel old in the service.

On moving 25,000,000 cu. yd. of material at Panama, the average repair cost was 2.815 cents per cu. yd. with a range of from 1.35 cents to 3.37 cents.

Some allow the useful life of a shovel to be 20 years, assume that the first cost of a shovel will be on a basis of $200 per ton and the scrap value will be $10. per ton. This would show a straight line depreciation of 4.75 per cent. per annum on the first cost of the shovel. Shovels have been rented at from $250 to $400 per month, depending on their size and condition.

**Coarse Crusher Plants.**—In more recent practice it hasbeen found that the reduction in site of matzrial to be loaded by shovels has been very desirable before shipping to the treatment plants. In some cases it has fallen to the lot of the mine to effect this reduction and in such cases the mine has been charged with the installation and operation of a coarse crusher plant. Notable illustratons of this are to be found at the mines of the Chino Copper Company and Biwabik Iron mine. These will be briefly described and serve to illustrate the subject.

*Chino Coarse Crusher Plant.*—The Chino Copper Company selected a good site for a coarse crusher plant at a point about one half mile west of the principal orebodies. It was designed to handle very coarse pieces of ore encountered in shovel operations in certain portions of the mine. It has been found that breaking these fragments by the usual means of dobying or drilling and breaking by hand involved not only serious delay to the shovels but also considerable increase in mining cost. On the other hand, if they were loaded without breaking and sent to the mill, the difficulty and expense of handling them through the ore bins and coarse crusher plant at that point was even greater. The machinery selected for the mine crusher was intended to be of sufficient capacity to receive the largest pieces of rock that could be loaded by a steam shovel and to crush it to a size convenient to handle at the mill. It was intended to put only such tonnage through this crusher as came from the harder areas of the orebodies and which in the ordinary process of blasting broke too coarse to go direct to the coarse orebins and crushing department at the concentrator. By removing this cause of delay at the shovels in ore, it was expected to increase the shovel efficiency and decrease the cost of mining and shipping. The construction of this crusher was started in the latter part of 1913 and completed and put in operation in August 1914. It immediately began to give the expected results in reducing mining costs and in substantially decreasing the unloading force at the mill. The management stated that mining costs for the year 1915 had been reduced by about 2.66 cents per ton due to the operation of this crusher plant.

A description of this crusher plant follows:

The oversize ore is hauled from the shovels to the crusher in standard dump cars which are dumped on a grizzly about 50 feet wide, accomodating two cars, and sloping at an angle of 45° for a length of 30 feet. The grizzly bars are 8-in. I-beams set 8 in. apart and covered with manganese-steel plates to protect them. The undersize passes through the grizzly and into a bin from which it is fed onto a 48-in. pan conveyor operated by a 100 h.p. motor and located at a point below the crusher, and on this conveyor it joins the crushed ore which has passed through the crusher. The oversize material passes over the grizzly, its fall is broken by a buffer and it is fed into the mouth of a Power & Mining Machinery Co. jaw crusher 48 in. × 60 in. in size. About 200 h.p. is required to drive this crusher. Its discharge opening is set to about 8 in. and the crushed material is fed onto the pan conveyor. This conveyor elevates the material into a bin from which it is loaded into railroad cars. The feed to the conveyor is irregular, coming in rushes as the cars are dumped, and for this reason the speed of travel of the converyor is relatively high, viz. 45 feet per minute. The bins are of steel and have about 500 tons capacity, being about 15 feet wide, 40 feet long and 20 feet deep. The ore is drawn off from each side of the bin, by aproned arc gates, into cars which may be assembled by gravity on tracks laid on 1½ per cent. gradient. When the ore is moist or clayey it funnels badly and must be assisted out of the bin if the bin is to be emptied. The cost of this plant has been about $150,000. The complete cost of crushing, from the dumping of the cars to the loading from the bin, exclusive of administration expense, runs from 2 to 3 cents per ton, being about two-thirds for labor and one-third for supplies, etc. About 14 men are employed on all work in connection with the plant. Power costs about $50.00 per h.p. year. About 3000 tons of coarse material are treated on the day shift only. This is roughly one-third of the total production. The improvement in results at the mill crusher and in elimination of delays at the shovels has been marked. The cost of unloading coarse ore at the mill has been considerably reduced. With the irregular feed mentioned the capacity of this plant, actually crushing only about one-half of the material dumped on the grizzly, is somewhat in excess of 3000 tons per shift. It seldom has to handle pieces larger than 2 feet in diameter but sticky wet fines when mixed with the coarse material tend to clog up the chutes.

The Swedish magnetite mines at Kiiruna also have a jaw type of breaker, built by the Power & Mining Machinery Company, with a rated capacity of 500 tons per hour and weighing approximately 150 tons.

*The Biwabik Mine Crusher Plant.*—This property, on the Mesabi range, was the first to introduce the steam shovel, systematic sampling and mixing of ore and preliminary crushing of the ore at the mine.

A No. 24 Allis-Chalmers breaker, with 48 in. receiving opening, capable of handling the largest and heaviest pieces of ore that a 3-cu. yd. shovel can take, was installed. This is driven by a 200-h.p. belted induction motor. In designing the plant, the engineers considered the merits of jaw and gyratory crushers, and chose the latter because it was believed that the crushing would be more continuous; less shock requiring cheaper foundations; less liability to breakage; and that the large circular receiving opening would permit the discharge of an entire carload into the breaker without having chutes leading to it for feeders. The results have been very satisfactory, and to put through 1000 tons per hour, is usually a question of getting the cars to the dumping platform fast enough. The ore is dumped over a grizzly 14 ft. wide, with bars spaced 2 in. apart; the undersize falls through and into a spout inclined at 50° leading to one of the shipping bins. The receiving opening of the crusher is 48 in. clear between crushing surfaces and is 125 in. long. The crushed product passes through a revolving screen with 2-in. holes. It is 6 ft. dia. × 13 ft. long, inclined 1¾ in. per foot, and driven. by a 30-h.p.back-geared motor. The undersize from this 2 in screen joins the undersize from the grizzly. The oversize from the screen, ranging from 2 in. to 5 in. falls into a separate shipping bin. Good provision has been made for all lubrication. The electric power for the plant is steam generated on the spot. As some of the chunks of ore handled by the shovels weigh as much as 6 or 8 tons it will be seen that the crusher was built for heavy duty.

The approximate cost of the plant, with all machinery installed and including electric equipment was stated to be less than $75,000. The average cost of crushing is about 1.5 cents per ton. The operating crew consists of 10 men, viz. a foreman who looks after the electrical equipment, 5 dump men who unload cars and tend to the feeding and 4 brakemen who look after the discharge and car loading. In 1914 this company shipped 255,255, tons of ore, and this installation resulted in a minimum of reblasting in the pit. The ores are as a rule quite hard, requiring machine drills in blasting.

**Ore Dryers.**—Another type of equipment which an open-pit mine is sometimes called upon to install and operate is an ore dryer. It would seem that it had little connection with open-pit mining, but a brief description will show its mission.

The function of a dryer is to drive off free moisture; it does so by the application of heat and is thus distinct from a dehydrator, which extracts moisture by mechanical means, and from a calciner, which drives off combined moisture and other volatiles by roasting at high temperature. Many classes of materials are successfully dried by this means, and among these are some of the iron ores of the Mesabi.

Many ores are now being dried to enable them to be concentrated by electrostatic separation and other methods; some are dried because it is less expensive to evaporate the water from the ore than to pay the freight charges for its transportation; often there is a considerable saving in handling the dried product, if when wet it is liable to freeze in the cars or stock piles, or seriously interfere in crusher feeding operations or furnace charging. In the case of iron ores, there is not only a saving in freight, but a higher premium is obtained, because elimination of the moisture naturally brings up the iron content in the product as sampled.

Thus an ore containing 20 per cent. moisture and 48 per cent. Fe. in its natural state may be raised to 56.6 per cent. Fe. when dried to 6 per cent. moisture. Furthermore, the dried product is often better to treat in the furnaces, as fuel and linings can be saved and the capacity can be increased.

Without going into details, actual installations of driers have been shown to utilize from 75 to 80 per cent. of the thermal value of the fuel consumed. The Ruggles-Coles dryers are double shelled revolving cylindrical furnaces built in different sizes, from 80 in. to 104 in. in diameter and from 45 ft. to 75 ft. long. The moisture is exhausted by means of an exhaust fan.

In practical drying operations at a particular property it has been found, in reducing the moisture content from 18 per cent. to 8 per cent., using as fuel Pittsburg bituminous dock screenings costing $3.00 per ton and including 3 to 5 cents per ton as the cost of transporting the wet ore to the bins before treatment begins, that the cost has varied from 13 to 25 cents per ton with 18 cents as an average.

Installations of this type may be seen on the Mesabi at the Whiteside mine of the Shenango Furnace Company, near Buhl and at the Brunt mines of the Pittsburg Iron Company near Virginia, Minn.

In conclusion it may be said that there is a legitimate field for the ore dryer, and, under proper conditions, its use will result in a good saving to the mine operator.

# CHAPTER III

## METHODS OF ATTACK

### GENERAL PROBLEMS

Shovel excavation may be classified under four general problems, viz., Bench work, Thorough-cut, Casting-over and Course-stacking. A problem may be made up of a combination of these. Their difference lies principally in the method of dis-

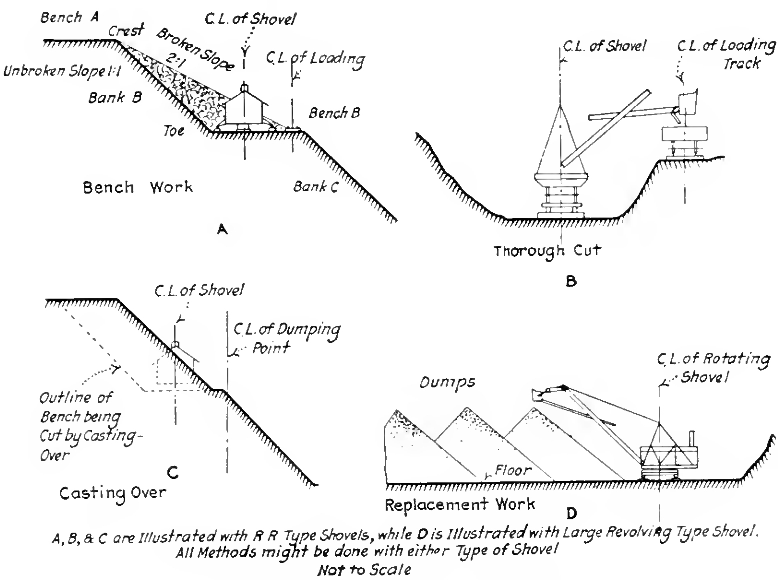



FIG. 18.—Four problems of excavation.

posing of the spoil. Fig. 18 A–B–C–D, illustrates these four general problems.

**Bench Work.**—This method of attack is widely used in mining large iron and copper deposits. It consists in digging from bank or face on one side or directly ahead of the shovel course, and loading the spoil into cars on a passing track on the same level and the opposite side of the shovel. In Fig. 18-A, bank B is being attacked at the expense of bench A, but at the same time bench B is being widened. Thus if the upper benches are to be maintained their banks must likewise be worked back to keep pace with the lower encroachment. With some deposits it is possible to do the stipping in one bench, but where the thickness is great it is necessary to work it off in a series of benches or terraces.

*Height of Banks.*—One of the first questions to decide in bench work is what shall be the best average height to carry the banks. The principal factors in determining this are, the total thickness of the deposit, its physical character, climatic conditions, the method of blasting and degree of fragmentation desired, and the specifications of the shovels to be used. In case ore and waste are both to be mined, careful consideration in planning banks must be given to the line of demarkation between the two, otherwise a mixture will result. If this line is very irregular the problem becomes difficult for that horizon, and considerable variation in bank height may become desirable in order to keep both products as clean as possible. Sorting ore from waste with a shovel from a bank can to a certain extent be done, but it greatly reduces the efficiency of the shovel, and there is sure to be some admixture of the two materials.

Several of the factors involved will be more or less interdependent. Careful consideration must be given to the safety of the crews and equipment as well as to the economics of the problem. Banks in some ground stand well for great heights while in other ground they crumble and slough off, assuming rather flat slopes. The higher the bank the flatter will be the safe slope. In working very high banks there is always more or less danger to men and equipment working below from pieces sloughing off. In blasting them greater burden is thrown on the toe, so that shovels and tracks are sometimes buried.

The following remarks may be taken to apply to both overburden and ore where both classes of material are to be mined.

A flat-lying deposit having a fairly regular contour and a moderate but fairly even thickness, may be mined out in one bench. Such deposits are, however, rare, and it will generally be necessary to divide the work into the fewest number of benches that will fit the characteristics of the deposit and the methods to be adopted in working it.

If the material is of a soft, rotten or highly shattered character it will generally break quite well when blasted by any method that will serve to shake it up. If, however, it is hard and not much broken by natural fracture or cleavage, high banks, blasted in one stage, are almost sure to break with a large proportion of "oversize." It is generally found simpler or more convenient to blast the banks in one stage than in several, as will be further explained under "Drilling and Blasting," Chap. IV. It is usually very undesirable to have banks break with a large amount of oversize. This material causes delays in loading, and may later be found very annoying at the reduction works. Large loosened masses may be left hanging in the banks, and must be carefully guarded against lest they endanger crews and equipment. For these reasons, in the case of hard tight ground, it is desirable to keep the height of banks low enough to insure reasonably good fragmentation and safety. Even under special conditions, where hard large material is desired, it will generally be found better to work banks of moderate height both on account of safety and general working control. It may be noted that hard rock is often cut by prominent slips and faults, and where these occur they should be given proper attention in planning the work.

Climatic conditions may have an important bearing on the best height to which the banks may be carried. For example, material that is soft, decomposed or highly shattered may be found to work down very well from banks of great height in dry or arid climates. The same material, when wet, may have a decided tendency to run or come down in rushes, thus causing loss of control of the banks. Examples of this class of material have been seen where the banks stood perfectly well during the dry summer months, but when saturated with rain or melting snows and frost, they would so slush down as to cover the benches and tracks and thus give much trouble. Decomposed porphyry and talcose material is especially apt to act in this way. Some examples have been noted in plastic ground where, although the banks stood up fairly well, the bench bottoms were badly squeezed up or out by the superincumbent weight of the banks. In working such material under wet climatic conditions, it is better to keep the height of banks moderately low, and to allow them to stand at rather a flat angle of repose if the benches are expected to be kept open for sometime. In regions of heavy snowfall and wind, it is found that pit banks offer good snow-brea ks; hence the benches are often heaped with snow-drifts, and they must be cleared out before traffic can be resumed. With high banks the drifts may at times come up to the tops and completely cover stretches of the benches. Thorough cuts are even more apt to become snow filled, and are more difficult to clean out.

The relationship between height of bank and shovel dimensions was pointed out in Chap. II, p. 46.

Using shovels of the railroad type, it has been found in practice that about the most satisfactory and economical bank is one having a height equal to the horizontal distance measured from the center-line of the loading track to a point on the bank six feet above the rail. Reference to Chapter I, table 2, shows this to be $A + E - 2$, or for a type 100*C* shovel, this would be 29 ft. + 33 ft. − 2 ft. = 60 ft. Using shovels of the largest revolving type—which might be done on very wide benches—it is not considered advisable to carry banks over 50 feet high in tight unshot material, or, in other words, much above the reach of the dipper, because if so, the bank would tend to be undercut. If, however, these large shovels were to be placed on loading from soft banks, or banks well blasted, where the material would naturally tend to feed down to the shovel, the height might be much increased, as there would be no tendency to undercut. It might be necessary, however, to keep "trimming" them to even slope so that the material would not come down in heavy rushes.

In theory, on a multiple bench job, the higher the banks, *i.e.*, the greater the vertical distance that can be allowed between benches, the more economical will become the removal of overburden, and the more efficient the shovel operation. There will be a smaller total yardage of overburden to remove because less horizontal area will have to be exposed to allow for the area of the benches eliminated. In other words, a steeper average slope from crest to toe of stripping can be realized. With deep or steeply dipping deposits the volume of slope yardage may become formidable. Reference to Fig. 2, in Chapter I. will illustrate this. Also the shovel operation will be more efficient because there will be less time consumed in moving the shovel ahead, and it can work in one spot, so that its loading time may be correspondingly increased.

In practise, however, the height of banks should be kept down to reasonably safe working conditions. It may furthermore be desirable to provide additional benches so that there will be plenty of working faces, or separate faces as natural divisions between different classes of ore or between ore and waste. Consideration of trackage arrangements may also require a reduction in bank heights because in running from bench to bench the gradients must be kept reasonable. The total rise possible is the product of the grade by the allowable length of track. The problem is thus a compromise between theoretical economy and practical conditions, but there are reasonable limits in both cases.

Benches less than 12 feet high are not economical, from the standpoint of shovel operation, because there is too much time spent in moving ahead; and benches over 75 feet high are liable to be difficult to control. Most work can be carried on somewhere within this range.

In the anthracite coal regions of Pennsylvania, rock cuts are usually carried from 22 to 25 feet high, though more recently these have been reduced to 12 or 15 feet, depending somewhat on the nature and hardness of the rock. The lower heights are recommended for very hard rock because it has been found that they give better results in blasting. The higher banks, and more especially their upper portions do not always break well with the system of blasting employed, and hence the material is more difficult to load. A saving of as much as 25 per cent. in the cost per cu. yd. of stripping has been claimed for the lower banks.

On the Mesabi, benches of from 25 to 30 feet in height are preferred.

In the open-pit copper mines, they are generally carried 45 to 60 feet high. The Utah Copper Company has even worked successfully one particular bank of ore 240 feet high. This was not done from choice but because of production requirements and particular physical conditions. Banks of this height are very likely to over shoot the loading tracks, bury the shovel, and give a large amount of oversize material which must be reshot. They are not under the same control as lower ones and must be handled with particular care and kept well trimmed. The method of shooting must also be different.

The slope or angle of repose of a bank depends on the nature of the material, its height, and if blasted, how it was blasted. This angle of repose is also called the slope ratio. In this case the vertical height is taken as unity and stated last; thus a slope of 2 to 1 ratio would indicate a base of 2 and vertical height of 1, and a 1 to 1 slope would be at 45°.

*Width of Benches.*—The width of benches is subject to less variation than the vertical distance between them. They should be wide enough to carry the loading track and provide for the shovel course. The distance between these track centre lines, or the dumping radii, for the various types and sizes of shovels is given in the tables in Chapter I and by Figs. 12 and 13, Chapter II. If the ground requires blasting the width should be increased so that the benches may be drilled and the banks shot without danger of caving-in the loading track above or covering up the loading track on the bench below. Ordinarily broken material will repose at a slope of about 1½ to 1 but rock blasting usually kicks it out flatter, or say to 2 to 1.

If the banks are shot well ahead of shovel operations it will be possible to do the drilling and shooting more methodically. The shovels may also work along with less interruption or fear of damage due to blasting. It may be necessary to widen some benches in places to allow for switches or passing tracks. During operation they will seldom be less than 50 feet wide, but where advance blasting or additional trackage is required may be considerably wider.

*Pit Slopes.*—The height and width of a series of benches will give the general pit slope and, as was pointed out, it is usually highly desirable to conclude the work leaving the slopes as steep as practicable in order to avoid excess overburden removal. During operations it may be found best to carry the pit slopes at a relatively flat angle, but as the work draws to a close it will often be found possible to narrow the benches and finally to lose upper benches by working the lower ones progressively up to them. For example, in the final stages of operation if may be quite possible to work the upper two 50-foot banks into one 100-foot bank, then do the same thing with the next two lower banks and so on, until further consolidation is no longer safe or convenient. Such bank consolidation may be broken up at safe intervals by simply leaving bench remnants of sufficient width, say 30 feet, to act as safety berms. These will not only serve to stop falling material which may become loosened, but may still be used as inspection ways.

When the pit plan is finally laid out such problems should be considered with the aid of working templates cut to scale.

Upper benches should not be carried beyond the final crest of stripping when so determined, and final drilling and blasting of banks should be carefully done so that the resulting banks will be left in the firmest and best condition to stand well and safely for the necessary time after consolidation. In this way it may be possible to increase the final pit slopes by from 5° to 10° or more, resulting in a great saving of yardage removal and disposition.

*Slides.*—In open-pit work slides often occur in wet weather. To guard against their interference with operations berms of from 20 to 30 feet may be left at suitable intervals, or the toes of banks may be cribbed up with poles or ties in rip-rap fashion, or dry-walling or facing may be resorted to. In working very high banks they must be carefully watched and trimmed to guard against slides.

**Casting-over.**—It sometimes becomes necessary to re-establish lost benches or to cut up a high bank. In so doing there may not be room for a loading track so that the shovel will simply cut its course discharging the dipper over the side of the bank. See Fig. 18-C. The cast-over material may have to be reloaded from the bench below. Usually after making one cut by casting-over, a loading track can be laid and bench-work continued.

**Thorough Cut.**—These are often called box cuts. In commencing stripping operations, running approaches or excavating ditches, the shovel is required to dig below grade and down grade. The dipper is discharged into cars on a passing track above grade or the material may be dumped on the bank along the shovel course. See Fig. 18-B. Reference to Table 2, Chapter I. shows that the depth of cut below rail for standard equipment is from 4 to 6 feet and the shovel must be cribbed up accordingly. By working down grade progressively this equipment can cut thorough-cuts to maximum depths of from 8 to 16 feet.[1] Having finished one such cut, the loading track may be relaid in the first cut and a second one started along side of the first. In this way a cut will be established with banks on each side and worked until sufficient depth and length is reached to begin bench work.

[1] For special work on canals and ditches special equipment set on rollers to straddle the cut, and having special long dipper sticks can be had, but they are not generally used in mining work.

Fig. 19 illustrates a method of dividing a 100-foot bank into two 50-foot banks by a system of thorough-cuts. Note the difference between this method and that of casting-over.

*First Cut.*—The location and method of the first shovel cut will depend on the local conditions. On some jobs of irregular

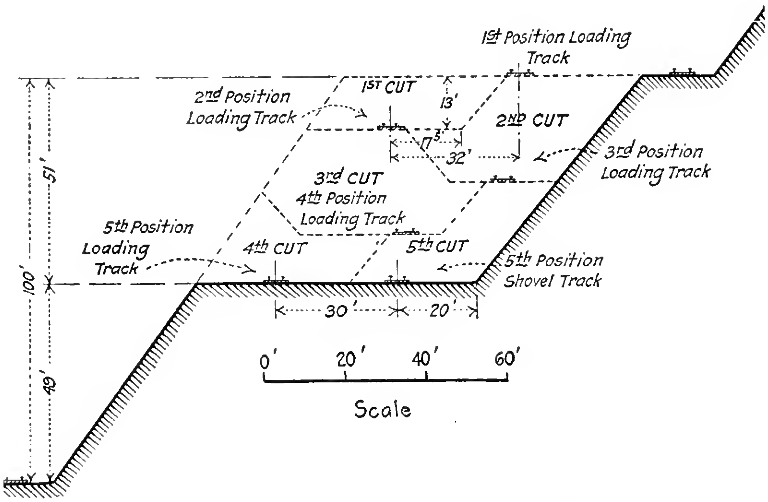



FIG. 19.—Dividing bank, thorough-cuts.

surface the first cut is run through the pit with the shovel cutting on grade and casting the spoil to the sides. In this grade cut a loading track is then laid and may be used for the next two cuts, one on either side. If the ground is not too rough the first loading track may be laid on the surface but care must be given

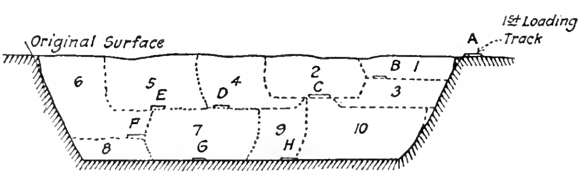



FIG. 20.—Thorough-cut work.

that the shovel cut does not tend to follow the undulations of the loading track resulting in an irregular profile. Care should also be given to the loading track to prevent undue wear and derailment of equipment.

On some jobs the large 300-ton revolving shovels are employed on thorough-cuts. An example from the iron ranges is given on Fig. 20.[1]

Here the loading track was laid on the surface of the ground at A; the shovel reach was sufficient to keep the furthest rail clear of spill. The small divisions numbered 1 to 10 indicate the successive cuts which would have been required had a 100-ton standard shovel been used to make the one large cut. The positions of the loading tracks would have been as shown by the letters A to H.

In some instances it is desirable to run a thorough-cut by laying the loading track alongside of the shovel in the cut. The depth of such a cut is not then limited by the lift of the dipper. This method, however, is slow because only one waste car can be spotted at a time at the shovel and as soon as a car is loaded it must be switched out and another one run in. The delays to shovel operation are great and the method is seldom employed. It is sometimes spoken of as "butting the ut."

**Course-stacking.**—In this system the haulage of earth is entirely eliminated as the shovel simply excavates from one side, swings about, and discharges the dipper on the opposite side. See Fig. 18–D. The first cut on a job of this sort will be a thorough-cut started on the extreme edge of the area to be stripped and probably at a place where the overburden is shallowest. The spoil from this cut will be deposited on ground not to be stripped. The deposit exposed by this first cut will then be removed, the shovel will be moved over for the next cut, which will be a bench cut, and wider as there is more room for operation, and the waste from this second cut will be deposited and stacked on the area of the first cut. This replacement operation will be repeated for the third cut and so on. There is a swell of about 25 per cent. in the volume of the broken spoil and as it is stacked in piles the crests of these piles will be considerably higher than the original ground unless the underlying deposit to be won is thick. This method is largely used in stripping certain coal deposits. Reference to Table 7, Chapter I, shows that stripping up to 48 feet can be excavated and stacked with the largest of the revolving shovels. Fig. 21 shows a shovel working in the first or thorough-cut and Fig. 22 shows a later cut being made as the operation is advanced. The material shown here consists of flat beds of

[1] Steam-shovel Mining on Mesabi Range. L. D. Davenport E. M. J. March 2–30th, 1919.

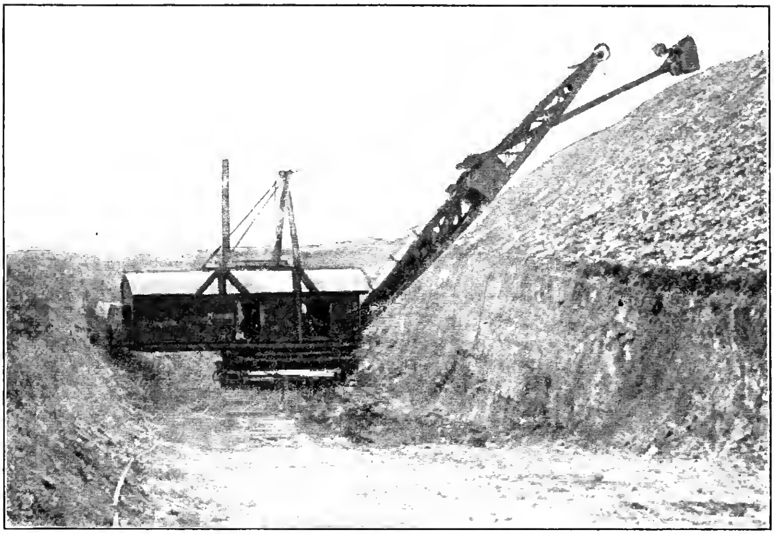


FIG. 21.—Steam shovel working in thorough-cut.

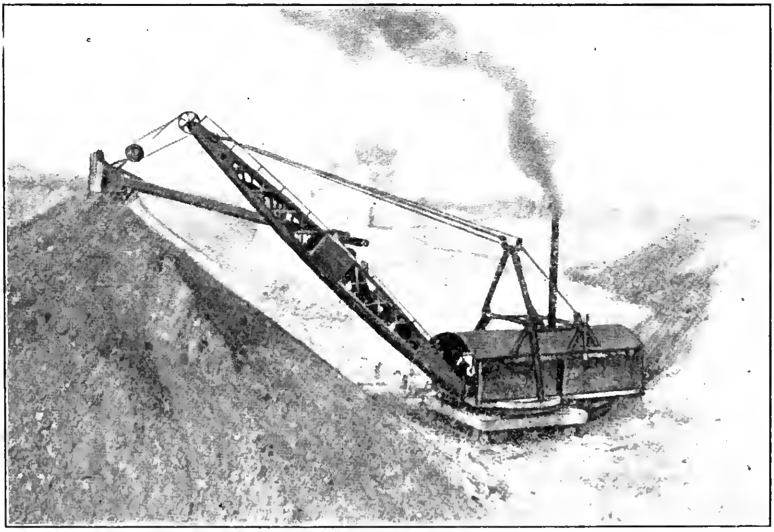


FIG. 22.—Steam shovel in thorough-cut after operation has advanced.

shale and clay seldom requiring any blasting. The benches stand almost vertical.

In some places the crests and troughs of the stripping rows have been balanced by hydraulicking. This leaves the land less broken, but it cannot be considered of much value.

## PIT LAYOUTS

As stated in Chapter VI, before a complete coordinated system of working an open pit can be planned, all the data, covering the shape, size, texture, structure and relationship of the deposit, to the enclosing formation should be known; also the detailed conditions for waste disposal, such as location, elevation and capacity of dump sites.

Depending on these conditions, pits may be planned on systems of spirals, switchbacks, or tees and wyes.

**Spirals.**—Where the deposit is more or less horizontal and regular in outline and of considerable area, the spiral system is usually adopted. A favorable location will be chosen for the approach and from this, spirals will usually be started from both sides. Occasionally separate approaches and independent spiral systems will be planned for overburden and ore removal. The care required in planning curves and grades will be mentioned in Chapter VI. Some examples of spiral pit systems are to be seen at the Shenango, Buffalo, Hull-Rust, Mahoning and other mines on the Mesabi iron range, and at the Nevada Consolidated Copper Company's mine.

**Switchbacks.**—Where the deposit is long and narrow, irregular in outline, or inclined, the switchback system is usually employed. Inportant examples of such pits may be seen at the Stevenson, Fayal and other mines on the Mesabi, at the Utah Copper, and at the Dehesa and Dionisio mines of the Rio Tinto Company in Spain.

The Chino Copper Company's ore bodies form an irregular annular ring and are worked by spirals and switchbacks.

The switchback tail-tracks waste horizontal distance without gaining elevation. Switchbacks tend to slow up traffic because of having to stop and reverse direction. On the other hand, the problem of curvature is usually simplified.

**Tees and Wyes.**—Such systems are often used in mining coal in Kansas, Ohio and Oklahoma. Fig. 23 illustrates a typical example of this system as employed near Pittsburg, Kansas. Here the first cut is made at right angles to the main haulage-way; the spoil is cast on the barren edge and a strip of coal about 50 ft. wide is uncovered. A loading track is then branched off from the main haulage way. This is laid on the far edge of the

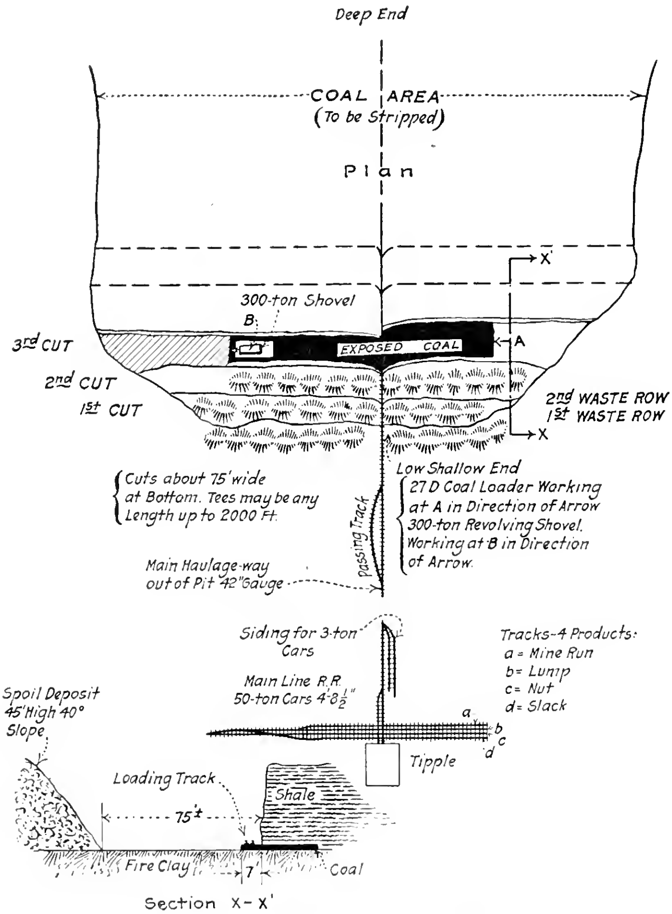



FIG. 23.—Tee pit layout; Kansas.

coal strip. The coal loader is then put to work at one end of the coal strip and advances toward the main haulage-way as it removes the coal. In the illustration, the 1st and 2nd cuts have been stripped and over two-thirds of the 3rd is finished. All of the coal has also been removed from the 1st and 2nd cuts and a start is being made on the 3rd strip. The 3rd cut will be completely stripped and the stripping shovel moved over to the 4th cut before much of the coal has been mined from the 3rd cut.

It is an advantage to commence operations at the lower side of the deposit since in that case haulage of the loads will have a favorable grade, and drainage will be simpler.

In these pits the overburden averages about 22 ft. but may run up to 44 ft. The coal seams are quite flat and average about 36 in. but may run up to 42 in.

Such a pit could be worked in a circular form but not so advantageously. There would be more narrow thorough-cut work, less flexibility and more time to reach maximum production. The shovel, however, would not have to be moved back

In the Danville, Illinois, coal district similar stripping is done. Here the coal seam (No. 7) is practically flat and about 6 ft. 4 in. thick. At the Carbon Hill property about 40 ft. of overburden is being removed by a No. 270 Marion revolving shovel equipped with a 90 ft. boom and 5-cu. yd. dipper. The shovel makes a 30 ft. cut for a length of 1200 ft. The coal is loaded into cars by model 31 shovel. At another property, a layer of shale 18 to 30 ft. thick, used for brick-making, is removed from above the coal, by a 70-ton Bucyrus shovel with 1½ cu. yd. dipper. This shovel, instead of making radial or parallel cuts, works in an approximate circle, going round and round a given area, and does not have to be turned. A similar practice may often be noted in the Mesabi range pits.

**Tunnels and Shafts.**—It sometimes happens that the topography of a deposit is such that it cannot be served in whole or part by the usual thorough-cut approach. It may be more economical to drive a tunnel into the pit or to connect the pit bottom to an outside shaft by means of a drift. In some cases the overburden will be removed through an approach, but the deeper ore will be worked by the "milling system" and drawn out through drifts and shafts.

The Balkan mine near Crystal Falls, Michigan, was stripped of loose sandy drift with drag-line excavators for a length of, say, 800 ft., width 300 ft., and depth 100 to 125 ft. The buckets dumped into portable hopper bins, so built that standard cars ran under and loaded from them. The loads were then pulled out by dinky engines over a spiral track. The underlying iron ore was harder, requiring blasting. When the pit gets too deep and the slopes too expensive to push further out it is expected to drift under the orebody from an outside shaft; from the drifts raises will be run up to the top of the ore, and down these the ore will be put for transport to the surface.

The Alpena mine at Virginia, Minn., is partly a pit mine, partly an underground mine and partly a "milling" mine. The pit portion includes ore carrying about 1 cu. yd. of overburden per ton. The pit is tracked with switchbacks and some of the grades are $4\frac{1}{2}$ per cent. so that they have to double out with two 75-ton locomotives hauling four 50-ton cars.

The Genoa mine, in the same district, but now worked out, utilized shaft extraction.

The Zarza lode of the Tharsis Company in southern Spain is in part mined open-cast by hand. The topography is such that both overburden and ore are hoisted. This is done by twin vertical shafts placed at a safe distance back from the brink of the open-cast.

At Rio Tinto, Spain, inclined planes are used to elevate ore from the bottom of the Mass No. 1 open-cast to railroad benches. Such methods are laborious and expensive.

**Inclined Planes.**—The methods used in brown-coal mining in Germany[1] (Leipsic, Bonn, Halberstadt, Cologne, etc.) are stripping with the continuous-bucket excavator, and open-pit mining in which the loading is effected by the "milling" system. Here the overburden is sand or soft sedimentaries and the mechanical excavator is more economical than the shovel. These deposits present a great variety in size and shape. They range from relatively thin beds, 9 to 30 ft. thick, up to great basin deposits 100 to 300 ft. thick. They are contained in clays, sandstones and marls. Many of them are covered by glacial drift consisting of fine sand, clay and light gravel. This may be only a thin layer from 20 to 25 ft. thick, or may range from 75 to 300 ft. thick. It is usually remarkably free from boulders. The topography of the country is usually flat.

Approaches to these pits are invariably short steep inclines served by the chain-haulage system. The reasons assigned for using inclines instead of long gentle slopes as used in America are the smaller outputs, greater value of surrounding land, necessity of preparing the coal before it can be marketed, greater economy in haulage and smaller capital investment required.

[1] Young, George J., P. L. S. M. I., Feb., 1916.

From one to three cuts or benches each about 25 ft. deep and sloped at 45° are necessary in stripping. As soon as the coal is exposed a steep incline is excavated in the coal and extended until a working face of from 50 to 100 ft. is obtained.

From the floor of the pit thus established, drifts of small cross-section are driven at intervals of 50 ft. into the bench, and at intervals of 25 ft. along the course of the drift, chutes are constructed to the surface. At the mouth of each chute a crater is started and the coal is worked by hand into the chute by the milling system.

The general layout for opening up a pit depends on the shape and size of the deposit. Two general systems are used on the larged and thicker deposits; usually the initial cut is started at the foot of the main incline and is extended parallel with and on the longer axis of the deposit. This can be done by starting three or four parallel drifts from the foot of the incline and extending them parallel with this axis. Mill holes are developed and follow up the drifting. The ribs are taken out and the floor of the pit cleared. At right angles to the first cut, parallel cuts are extended at intervals of 50 ft., and as rapidly as the flanking walls of the initial cut permit. Craters are then started in the flanking walls. In the other system the initial cut is made transversely to the main axis, and from the pit floor thus developed, drifts are driven parallel to the main axis of the deposit, and the line of advance of the craters is parallel to the main axis. The sequence of the stripping somewhat influences the method used in laying out a pit.

In many cases in the stripping operations of the anthracite region of Pennsylvania,[1] the stripping is considered too deep to be removed by locomotives and hoisting planes are resorted to. These are all single-track planes operated by small geared hoisting engines with a capacity of about 150 dump cars per day, or about the output of one shovel. The practical problem involved in putting these planes down along the steep sides of the average pit is often a serious one, as some of them are anchored on a slope of 50° to 60° pitch by bars sunk into the solid rock to which the road-bed is tied. These small hoists are suitable for a one shovel stripping job, but where

[1] Excerpts from Warriner, J. B., T. A. I. M. E., Feb., 1917. Also see "Mining the Mammoth Vein with Steam Shovels," Helms, D. C., Coal Age, Feb. 19, 1916.

two or more shovels are in operation on larger jobs it would be decidedly economical to use planes equipped with hoists capable of handling 300 or more cars per day. Such planes may be either single or double tracked, but the grade should be maintained at about 20°, which is the present average for the single-track planes. Some figures have been worked upon the comparative cost of the two types and are here quoted.

Length of plane 300 ft.

| | Single track | Double track |
|---|---|---|
| Hoist | $500 (Second hand) | $5,000 (New) |
| Tracks, track material, ropes, etc. | 700 | 1,100 |
| Grading for hoist and plane | 1,000 | 3,000 |
| Hoist house, pipe lines, etc. | | 800 |
| | $2,200. | $9,900 |

The operating personnel and costs are given as follows:

| | Single track | Double track |
|---|---|---|
| Top-men | 2 | 3 |
| Bottom-men | 2 | 3 |
| Locomotive engineer | 1 | 2 |
| Hoist engineer | 1 | 1 |
| Ore dump men | 5 | 8 |
| Costing: | | |
| Labor per day | $17.88 | $26.21 |
| Power per day | 4.30 | 6.48 |
| Interest and depreciation, 15 per cent | 1.00 | 4.00 |
| | $23.18 | $36.69 |
| Cost per car @ 150 and 300 cars per day | $ 0.155 | $ 0.122 |

or a difference of 3.3 cents per car.

In laying out these stripping pits, the location of the limits of stripping are set on a line where the normal slope of overburden from the bottom of the final cut intersects the surface. Slopes are calculated on 1:1 for earth of a clayey nature or shaley rock; 1½:1 or 2:1 for sandy ground; vertical for single cut rock work; and ½:1 for deeper rock work. It is very important to have the foot of the stripping slope well back from the bottom rock of the coal in order to prevent the washing of overburden by rains into the exposed vein. The standard width of such berms is from 10 to 15 ft.

Fig. 24 illustrates this work and Figs. 25, 26 and 27 illustrate crop, basin, and anticline stripping, into which divisions all of these anthracite strippings fall. Fig. 25, showing the crop stripping, is interesting in that it also shows the chain pillar left in early mining under the surface wash, which here was 40 ft. or more in thickness. Breasts were driven up in the early

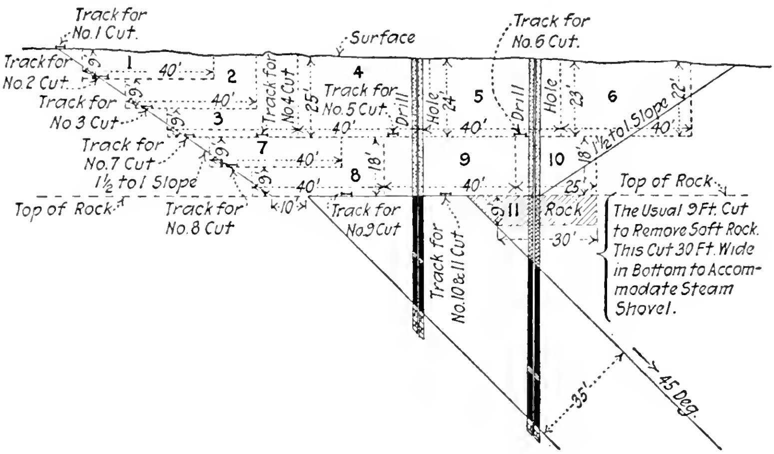



FIG. 24.—Stripping anthracite coal vein.

days until the roof caved in, and were then abandoned. The width of the chain pillar is at least 150 ft. Here the object is

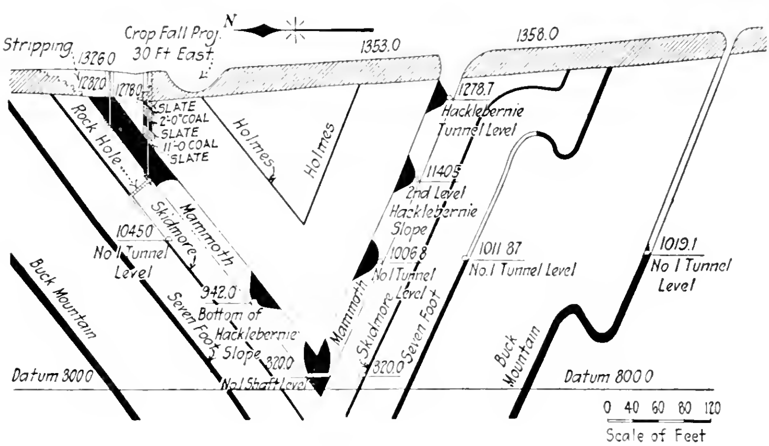



FIG. 25.—Stripping anthracite coal veins.

not to uncover all the coal, but merely to remove enough of the clay and rock to permit the mining of the coal from inside with minimum loss. To do this it is impossible to drive chutes up in the old vein and, therefore, a gangway is driven in a small underlying vein from which chutes are driven up to a point opposite the lowest edge of the chain pillar and rock holes are then driven through into this pillar.

Fig. 26 is of a large basin stripping, which was operated for several years (1900 to 1915) and shows the various stages of

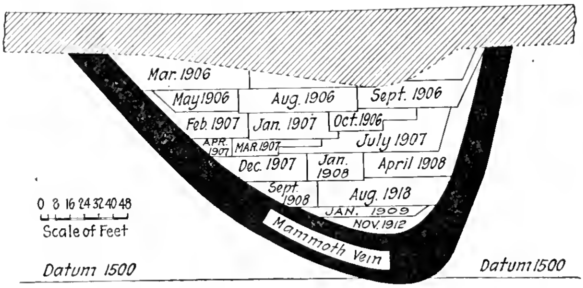



FIG. 26.—Stripping anthracite coal vein.

excavation characteristic of strippings of this kind. This is of a virgin vein; the width is 300 ft., length 4,800 ft. and maximum depth of cover 100 ft. The ratio of cubic yards of cover per ton of coal is 3.46 to 1.

Fig. 27 illustrates an anticlinal stripping of a worked-over area in which it is estimated that 60 to 70 per cent. of the

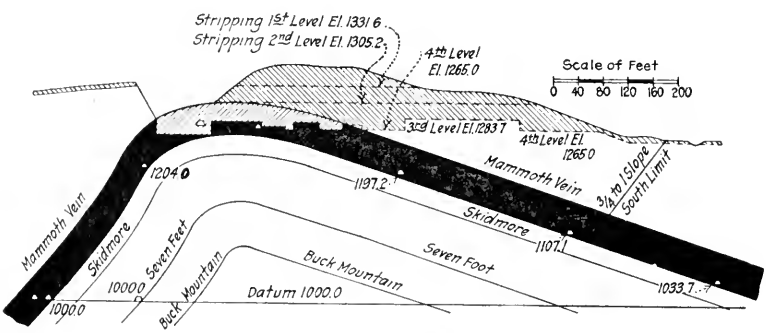



FIG. 27.—Stripping anthracite coal vein.

coal remains. Upon this estimate depends its profitableness, as it has been undertaken primarily to form a final barrier against a fire that has been raging to the east of it for many years. The vein is 55 ft. thick and is on a 20° pitch. It has

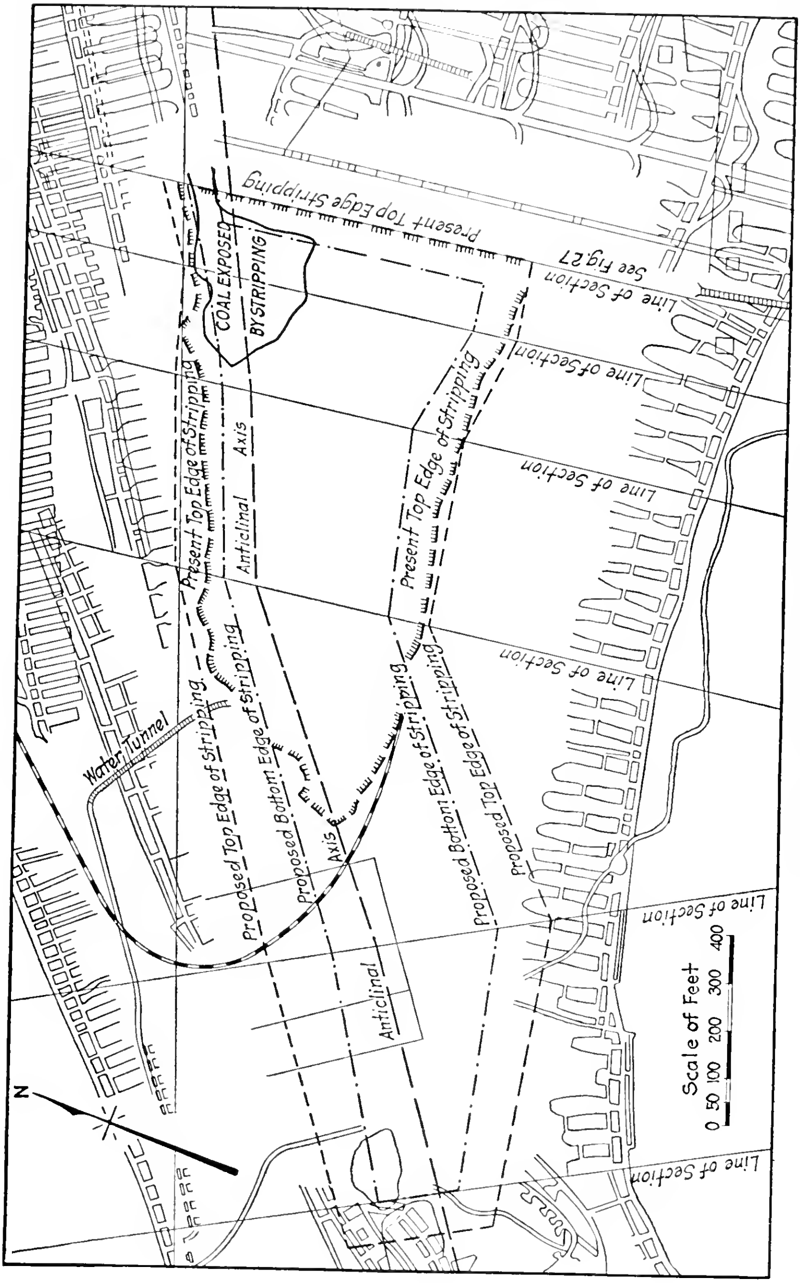



FIG. 28.—Plan; stripping anthracite coal veins.

been robbed and re-robbed, and robbed again, but because of its thickness and unhandy pitch, as well as the time of mining in the early '50's, it is thought that not over 35 per cent. of the coal has been extracted. Fig. 28 shows a plan of this stripping.

In passing it may be mentioned that many of the open pit coal mines have inclines leading from the pit bottom, or from the end of the haulage system, to the tops of tipples, from which the coal is graded, cleaned, and loaded into standard gauge cars.

*Milling System.*[1]—The best condition for the application of the system of milling is believed to be where an orebody is of medium size suitable for stripping, but where it would not admit of economical locomotive haulage of the ore. The relative depth of overburden and ore, the space and facilities for trackage and approaches and the relative outlay for equipment and development, are the determining factors for choosing between "milling" and "shoveling." The advantages of the former are smaller initial expenditure, as the entire orebody need not be stripped, a comparatively small amount of stripping to expose enough ore to begin production, less waste room is required and the approach to deep ore is avoided. Offsetting these are the expense of shaft, drifts, raises and attendant equipment, liability of flooding the mills with sand and slime during heavy storms and irregularity of production due to weather conditions and occasionally due to hanging up of mills.

Milling permits the recovery of practically all of the ore, is more economical than underground mining, and is perhaps a little safer. It is subject to the accidents incident to its limited underground work and to the blasting in the open, but the greatest danger is when men are carried into the chutes by slides, notwithstanding their attachment ropes.

In some cases shovels are used to mine the bottom layer of ore, or to dig the ore remaining in the hog-backs and cones, dropping it into the mills.

In other cases the mills are eliminated and the shovel loads directly into cars which are then trammed to a central raise feeding a shaft pocket. Fresh shovel cuts are started in pits of this kind by bringing in drifts from the shaft at the desired elevation and making an opening for the shovel to begin a a new cut.

[1] It is assumed this system is understood. See "Iron Mining in Minnesota," C. E. Van Barneveld.

## CHAPTER IV

# DRILLING AND BLASTING

Material to be excavated with power shovels should be loose enough to be dug easily and without causing excessive strains and wracking on the shovels. Some material occurs naturally in this condition so that no further breaking up is necessary. Other material can be dug without preliminary loosening but the saving in the cost of loosening is more than offset by the decreased output and by increased wear and tear on the equipment. Still a third class of material must be blasted in order to be excavated at all. Some class of explosive is now universally used to accomplish this, and, in order to prepare chambers for loading, hand-drills, machine-drills, churn-drills, gopher holes or pit-shafts and drifts are employed.

**Hand-drills.**—On the iron ranges of Minnesota the usual overburden is a glacial drift, frozen in the winter time, and often containing huge hard granite boulders ranging from 3 to 12 ft. in diameter. These are "chained out" by the shovels and placed on one side in their wake to be block-holed. These boulders are drilled by hand, using 7/8 in. steel. The holes are from 12 to 18 inches deep and are loaded with a stick of 60 per cent. dynamite. The drilling is done on contract at 20 cents per foot, and an average day's work is from 12 to 15 feet. When the overburden is frozen jumper drills, heated to a dull red, are used to put down shallow holes called "top" holes. In deeply frozen ground steam-points have been used. These holes are put down from 3 to 6 ft. through the frost. They are charged with from six to eight pounds of Du Pont black powder. In some cases the surface of the ground is broken up before frost sets in by drilling holes 3 to 5 ft. deep and shooting them with light charges.

In most of the open pit iron mines requiring blasting, it is customary to blast the ore banks ahead of the shovels. Top holes are used for this purpose, excepting where there is rock capping over the ore. In the latter case gopher holes are drilled. Top holes are "jumped" or churned by gangs of drillers working in groups of 2 or 4 men. The drills are made of 1 in. round or hexagonal steel, chisel pointed on both ends. A heavy iron cross handle or yoke 26 in. long is slipped over the drill and fastened in place with a 6-inch steel wedge.

The spacing of these holes depends on the hardness and texture of the ore, the height of the bank and the width of the cut to be taken. In average ore with banks from 15 to 25 ft. high, the holes are usually spaced 15 to 20 feet apart and about the same distance back from the crest of the bank. They should bottom a foot or two below grade so that there will be no "tight" ore on the bottom. The limiting depth of these holes is 22 ft. In average ground a 15 ft. top hole when finished, would be sprung with 2 to 6 sticks of 60 per cent. dynamite, then loaded with black powder and fired. If several holes of this depth were to be blasted in series, the charge would consist of 1 to 1½ kegs (of 25 lb.) of black powder. If the holes are to be shot separately, as is sometimes necessary when the bank is high and the loading track close in to the toe, the charge should be slightly increased. Top holes are usually fired in series of not more than five at a time, using a blasting machine. In some cases safety regulations require that holes be fired separately.

A gang of jump-drillers will average 50 ft. of hole per man per day in ore; good jump-drillers are, however, rather scarce. Hand-drills are generally being replaced by machine drills.

It may be mentioned further regarding the spacing of holes (hand or machine drilled) that rules have been made whereby both the spacing back from the face and the distance apart have varied from distances equal to the depth of holes to one-half their depth; or their distance apart may be even further decreased. Such rules will serve to experiment with until the material is broken to a suitable size. Obviously the spacing will have a marked effect on the cost of breaking as it involves both the cost of drilling and cost of explosive per cu. yd. A fair range of examples of such work may be found carrying from 0.25 to 1.25 ft. of hole and from 0.30 to 0.70 lb. of 40 per cent. dynamite per cu. yd. of material broken.

**Machine-drills.**—At the Utah Copper and Chino Copper mines machine drills have been extensively used. Fig. 29 shows the method of drilling and blasting a 70-foot bank and Fig. 30 the remarkable 240-foot bank, both at the Utah mine. Churn-drills will be substituted for some of this work as the levels are widened out to 80 or 100 feet. The general method now is to drill toe holes with a maximum depth of 25 feet, spaced from 15 to 30 ft. apart (average 20 feet), depending on conditions. It is usual to shoot one hole at a time, taking advantage of the face of rock facing the shovel and facing the main bank, thus giving two open faces; also the broken rock lies closer in, which is a decided advantage on these narrow benches. Frequently two, three or even as many as six holes are shot at a time, advantage being taken of seams, slips and the appearance of the face to be broken. In single or multiple shots there is no appreciable difference in the condition of the blasted ground, the material can be handled with equal ease and the banks are equally safe. The holes are drilled ahead of the work, and preceding shots

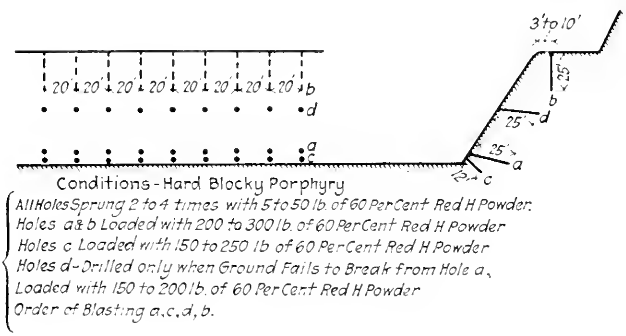



FIG. 29.—Drilling and blasting 70-ft. bank; tripod drilling.

do not destroy holes already made. One of the most important points is to see that the hole is drilled a little below grade, and that the powder is charged at the end of the hole. At this mine, it has been found that single-hole shots bring down plenty of material to last one shift so they do not affect the delays to the individual shovels. Blasting is done at stated times, viz., in the morning, at noon and at night. The shovels are brought to a safe position when shots are made.

Details of the drilling and loading of the holes are shown on Figs. 29 and 30. About two cubic yards of rock are broken per pound of powder consumed, including the powder used in breaking up large boulders. The records show that $4\frac{1}{2}$ to 5 tons of ore and waste are broken per pound of powder consumed. The question of block-holing large rocks versus dobying is entirely one of delays to operations. Whenever convenient, block-holing is preferred, but generally the large rocks are laid aside and dobied at the regular times for shooting.

At the Chino property the bank bottoms are drilled with air drills, but most of the regular bank shooting is done from churn-drill holes.

In digging the Panama Canal many variable mining conditions were met; subaqueous work with drill barges; drilling in the dry for dredging where the water was turned in after blasting; blasting for shovels through different cuts; blasting rock to be crushed for concrete; core rock for breakwaters and large rock for their armoring.

In working on 50-foot banks the method was to drill lifting holes 26 ft. deep with 7 foot spacing; then from the top and about

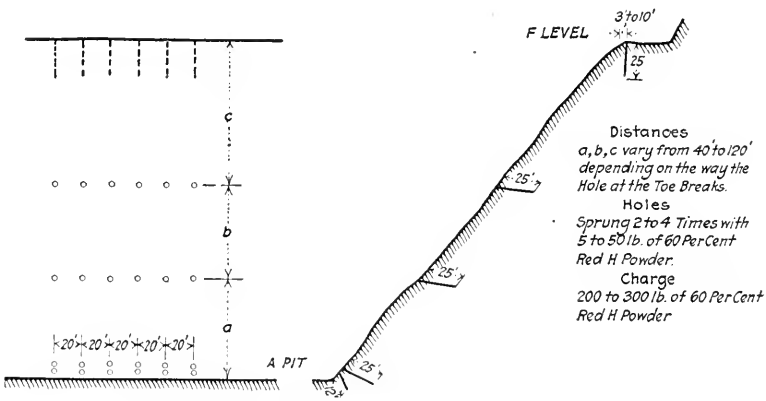



FIG. 30.—Drilling and blasting 240-ft. bank; tripod drilling.

14 ft. from the face perpendicular holes 28 ft. deep with 8 ft. spacing. This assumes the toe to be 26 ft. from the charge, if the down hole was projected to bottom grade. The charges were of 60 per cent. dynamite. The material was satisfactorily broken for shovel and crusher handling.

At Sosa Hill quarry, where the highest point on the bank is about 250 ft., as much as 25,000 cu. yd. were broken at one blast. Here the lifting holes were 26 ft. deep and the bank was cut into 4 or 5 benches from which 28 ft. vertical holes were drilled. The lifting holes were loaded with from 100 to 200 lbs. of 60 per cent. dynamite, and all the down holes with from 20 to 30 lbs. of 45 per cent. dynamite. After these blasts, all of the rock was dug from the bottom level.

In comparing single and multiple hole shots, it was the opinion at Panama that the latter were most advantageous; that single holes required at least one-third more powder; that the blasted ground was much easier to handle when holes were shot in multiple; that in case one hole in a larger series was caved or lost, the holes flanking it broke up the ground enough for the shovel to dig it and that the cost of drilling the next hole was less because there was no chance of one hole breaking into the ground where the next one was to be drilled, or, if the ground ahead had already been drilled, of spoiling holes. This was especially true in using both lifting and down holes; the blasted bank was left in better shape and the shovel output was much greater. Multiple shots carrying as many as 180 twenty-six foot tripod drill holes and 150 fifty-five foot churn drill holes were shot as once. Single holes were only used for loosening up hard spots found in front of the shovel, or when the shot was less than 100 ft. from valuable structures, or where a quantity of water might break through with a heavy shot.

It was also considered that dobying saved time but block-holing was safer. Large pieces which the shovels could not handle were broken at once, usually by dobying.

On the iron ranges a number of mining companies are using machine drills instead of "jumper" drills for putting down top holes.

The use of air machines in large open-pits necessitates either the building of a compressor plant and extending pipe lines through the pit or equipping each shovel with a compound air pump. The latter arrangement does away with the shifting of pipe lines as the benches are worked back, and eliminates any trouble incident to the freezing of air lines. Furthermore, and individual air pump is advantageous when the shovel is moved from ore to stripping, in that it can furnish power to operate jackhammer drills for block-holing boulders, obviating the necessity of using hand drills or dobying.

With the D-113 Ingersoll-Rand drill an average of 90 feet per shift has been made in the Mesabi district. This footage is equivalent to 45 ft. per man per day—two men being on the drill—as against 50 ft. per man per day with the jumper drills. The advantages with the machine drills, however, are that they drill through rock seams readily, and can drill to a depth of 28 or 30 ft. as against 22 ft. with jumpers; also it is easier to obtain crews for the machine drills.

The loading and blasting of these holes is the same as was mentioned above under hand drills.

On stripping rock in the Pennsylvania anthracite region, holes 12 ft. or less in depth are drilled with steam tripod drills which make about 7 ft. per hour in solid rock. The holes are usually arranged in 3 parallel rows with a staggered spacing of from 12 to 20 ft. They are fired in batteries of 15 to 25 or more. From 3 to 8 sticks of 40 per cent. dynamite are used to spring each hole, which operation may be necessary two or three times. They are then charged with black powder, filling the chamber and about 2 ft. up the remainder of the barrel. The holes are tamped with clay or coal dirt. For 25 to 30 ft. holes "Star" churn drills are employed using a 4 in. bit. These make about 3½ ft. per hour in solid rock. The average cost for drilling and blasting per cu. yd. of rock excavated is given as

| | |
|---|---|
| Labor drilling and charging, depreciation, equipment, etc. | \$0.045—\$0.065 |
| Powder | 0.055— 0.080 |
| | \$0.100—\$0.145 |

Temple-Ingersoll percussion drills, operated by a portable electric-air pulsator, have been successfully used in drilling holes for loosening up the stripped coal beds in the Danville, Ill. district.[1] In one case on a coal bed 5½ ft. thick, 1¾ in. holes were placed 5½ ft. back of the coal face and spaced 5 to 6 ft. These were each charged with 1½ to 2 pints of black powder. On another job these machines drilled 3 in. holes spaced 6 to 7 feet apart, and the charge in each was about 1½ quarts of black powder. A push-down blasting machine fires the charges. These holes are so loaded that they merely loosen the coal but do not throw it about. A 25 lb. keg of powder loosens about 100 tons of coal, which is then loaded into mine cars with a small shovel. At another mine, an Ingersoll-Rand portable electrically driven compressor furnishes air for Jackhammer drills which put down 1¾ in. holes about 6 ft. deep. Here the coal averages 6 ft. 4 in. in thickness. The holes are spaced 6½ ft. apart and are placed 6½ ft. back of the face.

**Churn drills.**—Where conditions permit the use of churn well-drills, they have largely replaced hand and tripod drills.

[1] "Steam-shovel Coal stripping in the Danville District."—*Coal Age*, March 11, 1916.

At the Utah Copper Company's mine few churn drills are used because of the narrowness of the benches. An interesting combination of the use of tripod drills and churn drills is shown on Fig. 31. Here the 240-foot bank is shot by means of churn drill holes from the bench supplemented by tripod drill holes at the toe and part way up the bank. Details of the arrangement and loading are given on the figure.

At the mines of the Chino Copper Company most of the blasting is done by means of churn drill holes. Only one of these is blasted at a time, as it has been found that large blasts of several holes each are not advantageous here. The bottoms

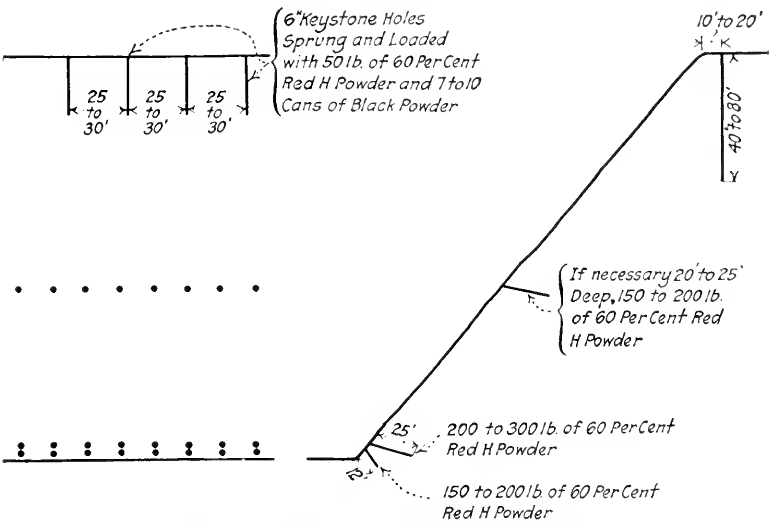



FIG. 31.—Drilling and blasting 240-ft. bank; well drilling.

are drilled with tripod drills. The churn drill holes are put down from 25 to 60 feet deep, and "sprung" by exploding several light charges of dynamite in the bottom. This chambering is done four or five times with increasingly larger amounts of powder each time; *e.g.* the first time with 5 sticks, the second with 10, the third with 20 and the fourth with 25 to 30 sticks. Generally 40 per cent dynamite is used for this purpose. By aid of a hand-mirror the sunlight can be reflected into a hole to a depth of 25 ft. or more and this is of assistance to the loader. After the hole has been chambered sufficiently, it is loaded for blasting. The charge depends on the depth of the hole, hardness of the ground, burden carried and track conditions. Two average charges are given as follows; (1) a 28 ft. hole on a terrace of ore, 138 lbs. of 40 per cent. dynamite and 100 lbs. Trojan granulated powder No. 2; (2) a 50 ft. hole on a higher terrace of ore, 170 lbs. of 40 per cent. dynamite and 200 lbs. of Trojan No. 2. The holes are usually drilled from 3 to 5 ft. below grade but in case the ground breaks badly, horizontal tripod drill holes are drilled and blasted to level up the surface for the advancing shovel track. Warning of a shot is given by whistle signals from the shovel and the crews in the vicinity seek shelter behind portable steel shields. Some interesting detonation experiments were carried out at this property between 1¼ in. × 8 in.—40 per cent. Ropauno Gelatin Dupont powder and 1¼ in. × 8 in. Trojan powder. No. 8 blasting caps were used and placed as nearly as possible in the center of the stick of powder used as a primer. The tests were carefully conducted. The conclusions reached were that the Gelatin powder if relatively easy of detonation and would explode in all cases where the powder is close enough together for a portion of the stick to be destroyed; on the other hand the Trojan detonates with difficulty and requires to be in close contact with the primer or other detonated sticks. This is based on small charges unconfined. These conclusions are of interest where the shooting out of missed holes would be attempted by the usual method of drilling and shooting another hole in close proximity to the missed hole. It seems doubtful if this could be successfully accomplished with Trojan powder. Again in badly ravelled or broken holes, it would seem that there might remain unexploded sticks which had not been loaded in close contact with the powder immediately surrounding the primer. Actual conditions of confinement in a drill-hole, however, might modify these conclusions.

The open-pit mines of the Nevada Consolidated Copper Company are in a leached highly altered monzonite porphyry. Practically all of this material requires blasting and this is done by putting down 6 in. churn drill holes. These are all drilled from 3 to 5 ft. below grade, *i.e.*, a hole 55 ft. deep is drilled for a bank 50 ft. high. This is important as it removes most of the danger of leaving unbroken rock on the bottom. The holes are spaced about two-thirds of their depth or say 37 ft. apart for 55 ft. depth. The slopes of the banks average about 45°, thus the edge is at a horizontal distance back of the toe equal to the height of the bank. Holes are drilled as close to the upper edge as convenient or, say, 10 ft. back, making them bottom 60 ft. back of the bank toe. When a drill has moved onto a hole location, the rear wheels are leveled transversely by running one wheel on blocking if necessary. The wheels are then blocked to prevent the drill from moving and the traction pin is removed. Two track jacks are placed beneath the front end of the bed, and are used to relieve the front wheels of the weight they support and to level the front end transversely. Such leveling is imperative in order to keep the belt on the pulleys when running. No attention need be paid to longitudinal leveling as drills are sometimes operated with the tools hanging almost to the end of the "A" frame tool guide, and, again, with scarcely any room between them and the front of the machine, but it is desirable to have at least 2½ ft. between the front of the drill and the tools. Much time, however, should not be spent in longitudinal leveling. A four-foot piece of 7⅝ in. casing called a "conductor" is used to guide the tools when drilling is started. This has two coils of old 2 in. drill cable wrapped around it to prevent it from sinking into the hole, and also to furnish a hold for removing it when the hole is finished. As mentioned above, the ground is blasted from 3 to 5 feet below grade, so for a few feet the holes are drilled in broken ground remaining from previous shooting, and the "conductor" protects this section of the hole from caving. The drill is run slowly until the hole is 4 or 5 ft. deep, then it is speeded up to about 58 drops per minute. The hole is baled out every 2½ or 3 ft. and the sludge is allowed to run down the bank below, unless it is to be sampled. After the hole is completed the tools are pulled up and are tied to the bed of the machine to prevent them from swinging while the drill is moving. The dart of the bailer is tied to the rope on the conductor which is then pulled out of the hole. The bailer, or sand pump, is hauled up into the "A" frame, to prevent it from swinging enough to do damage. The track jacks are removed, the traction pin put in place, the blocks removed from the wheels and the drill moved to the next spot.

The drill crew is followed by the blasting crew. The powder foreman carefully notes the depth of the hole, friability of the material and the burden carried, and from these decides what charge shall be used. Chambering is started with a light charge and the amount tripled for each succeeding chambering charge.

For a hole 45 ft. deep, say, 10 sticks of 60 per cent. semi-gelatin Red Cross powder would be used for the first charge, 30 sticks for the second chambering and 90 sticks for the third. When using black powder for the blasting charge, about 25 lbs. were allowed per foot of hole. For holes 90 ft. in depth, chambering is started with about 25 lbs. of 60 per cent. R.C. powder and increased by doubling or tripling the amount for successive chambering shots. Water is used for tamping these chamber charges, say, 20 gal. for the first charge and triple the quantity for each succeeding charge. About 25 lb. of black powder were allowed per foot of hole in the blasting charge.

In determining the progress of chambering it has been found helpful to use a sounding rope. This consists of a piece of window-pulley spot cord about 75 ft. long to one end of which is attached a piece of wood about 4 in. in diameter by 3 ft. long. This club is let down to the bottom of the hole and permitted to fall over from side to side in such a manner that the bottom enlargement may be felt out.

In charging holes a 50 ft. length of 6 in. diameter rubber-lined tuyere tubing is slipped over the cylindrical spout of a galvanized sheet iron funnel. This provides a smooth lining for the wall of the hole preventing powder from lodging in seams or fissures and also preventing fragments of the wall from being knocked off by the falling powder. After placing this sleeve in the hole, the sounding line is fed down, and then as the powder is run down, the sounding line is jigged up and down on top of the powder assisting it to be closely seated but not tamped. This line constantly shows to what extent the bore has been filled with powder and indicates the extent of chambering. When the required charge has been loaded, the sounder and sleeve are removed and the remainder of the bore is filled to the top with fine dirt or sand.

It was found that a blast-hole would break the ground in back of it about one-quarter of the distance it broke in front or on the side offering the least resistance. All blasting and chambering is done using No. 8 electric exploders.

Fig. 32 illustrates a hole being sprung.

The following percentages of explosives were used in the earlier operations of 1912 and later operations of 1916, by the Nevada Consolidated Copper Company.

| *Explosive* | 1912 per cent. | 1916 per cent. |
|---|---|---|
| F. F. black powder | 27 | 4 |
| Stick Trojan (3½ in. × 3½ in., mostly) | 23 | 69 |
| Bag or granular Trojan | 15 | 4 |
| Hercules R. R. P. | — | 8 |
| Hercules 3½ in. × 3½ in.—40 per cent | 30 | 3 |
| Hercules 1¼ in. × 1¼ in. Red "H" | — | 7 |
| Hercules 40 per cent (1¼ in. × 4 in., mostly) | — | 4 |
| Gelatin and semi-gelatin 60 per cent | 2 | — |
| Red Cross 40 per cent | 2 | — |

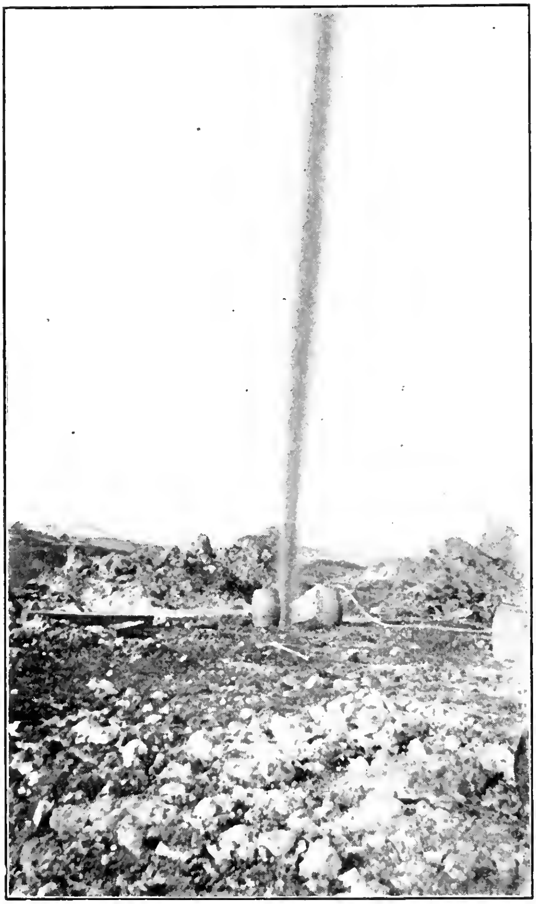


Fig. 32.—Drill-hole being sprung.

It will be noted that the black powder has been displaced by the more efficient Trojan powder. Experiments carried out using Trojan 40 per cent. vs. Hercules 40 per cent. indicated that the breakage per pound of Hercules was higher than the Trojan; that the cost per pound of the Trojan was less; that the cost per yard of material broken was about the same in both cases; that the Trojan worked well in cold weather, requiring no thawing, and was held in consequence to be more convenient and safer to handle.

About 0.6 lb. of powder were required per cu. yd. of material broken, including explosives used in dobying boulders.

Varied and interesting experience in the loading and shooting of well-drill holes has been given by S. R. Russell C. E.[1]

Some of the following brief records of blasts in different sections of the country are quoted from him. An average of 4 to 6 tons of stone per pound of explosive is about what should be expected in blasting deep holes.

A blast in a limestone quarry in Tennessee, stone used for ballast and commerical purposes, consisted of sixteen 5⅝ in. holes of average depth of 75 feet. Holes were spaced 18 ft. apart with average face burden of 22 ft. Charge, 3750 lb. of 60 per cent. and 3700 lb. of 40 per cent. L. F. dynamite. Produced 5.7 tons of rocks per pound of explosive.

A blast in cement rock in Pennsylvania consisted of fourteen 5⅝ in. holes; average depth 86 feet; spaced 18 feet apart with a face burden of 30 feet charge, 4850 lb. of 60 per cent. and 3250 lb. of 40 per cent. dynamite. Produced 55,000 tons or 6.8 tons per pound of explosive.

A blast in a Kentucky limestone quarry, stone used for ballast, consisted of nine holes; average depth 50 ft.; spaced 18 ft. apart and 25 ft. back. Charge, 3250 lb. of 40 per cent. dynamite, produced 16,200 tons of rock or 5 tons per pound.

A blast in an Oklahoma quarry, stone used for railroad ballast, consisted of eight holes 95 ft. deep; spaced 28 ft. apart; average face burden 33 ft. Charge, 2200 lb. of blasting gelatin, 3350 lb. of 60 per cent. and 1250 lb. of 40 per cent. dynamite. Produced 62,000 tons for 6800 lb. of explosive, or 9 tons per pound.

The above blast is criticised as being badly balanced, necessitating use of very strong expensive explosive at bottom, making the cost per ton as high or higher than had the blast been better balanced.

[1] Blast Hole Drilling—Keystone Driller Magazine First Edition and Du Pont Magazine, August, 1916.

A blast made in iron ore consisted of twenty-six 5⅝ in. holes of average depth of 84 feet; spacing 15 ft. × 15 ft. Holes were triple loaded, fired with Cordeau and one electric blasting cap. Charge, 8500 lb. of 40 per cent. dynamite were used, producing 50,000 tons of ore or 5 tons per pound of explosive.

A blast made in cement rock in New Jersey consisted of eleven holes; average depth, 102 ft.; spacing 20 ft. × 22 ft. Charge, 2040 lb. of 60 per cent. gelatin and 4475 lb. of 40 per cent. gelatin. Produced about 40,000 tons of stone, or 6 tons per pound.

A very successful blast, made in a West Viriginia ballast quarry, consisted of twenty-four 5⅝ in. well-drill holes varying in depth from 56 to 121 ft. Also about the center of the face at the bottom were drilled thirty-four 16 ft. snake holes, which were loaded and fired with the main shot. The snake holes were drilled to relieve a heavy toe at that point. Well holes were spaced 16 to 17 ft. apart and had an average burden of 22 ft.; R. C. gelatin 60 per cent. and R. C. Extra 33 per cent. dynamite were used; of the former 7900 lb. was used in the bottoms of the well-drill holes, and of the latter 7300 lb. in the tops plus 300 lb. in the snake holes. Nearly all holes were double-loaded, usually with a 12 foot break. Cordeau-Bickford was used in each hole to detonate the explosive. There were 64,000 tons of stone shot down, or 4.1 tons per lb. The stone was well broken and distributed.

A blast made in a West Viriginia quarry, stone used for ballast, consisted of seven holes; average depth, 60 ft.; spacing 18 ft. × 20 ft.; charge, 1600 lb. of 50 per cent. gelatin and 1200 lb. of 33 per cent. ammonia dynamite. Produced 13,000 tons of stone or 4.6 tons per pound.

Blasting in a cement quarry in New York had been done, with tripod drills putting down 1½ in. holes, 6 ft. back from face, 6 ft. apart and 12 ft. deep. This method was replaced by 6 in. well-drill holes, 20 ft. back, 20 ft. apart and 65 ft. deep. Charge is 500 lbs. of 40 per cent. R. C. dynamite per hole, and holes are shot in series of from four to twelve. Explosive consumption per ton and fragmentation of product same in both methods, but other economies derived by churn drills make the system decidedly preferable.

In a good many instances it has been observed that powder consumption per ton of rock has been about the same whether tripod or well drill holes were employed.

TABLE 15.—THIS TABLE GIVES NUMBER OF CUBIC YARDS OF ROCK DISPLACED PER FOOT OF HOLE AT DIFFERENT SPACINGS

Distance Apart of Holes in Feet

| Distance Holes Are Set Back From Face in Feet | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 5 | .92 | 1.11 | 1.3 | 1.49 | 1.66 | 1.85 | 2.04 | 2.22 | | | | | | | | |
| 6 | 1.11 | 1.33 | 1.55 | 1.77 | 2.0 | 2.22 | 2.44 | 2.65 | | | | | | | | |
| 7 | 1.3 | 1.55 | 1.81 | 2.0 | 2.33 | 2.7 | 2.85 | 3.11 | | | | | | | | |
| 8 | 1.49 | 1.77 | 2.0 | 2.37 | 2.65 | 2.96 | 3.26 | 3.55 | | | | | | | | |
| 9 | 1.66 | 2.0 | 2.33 | 2.65 | 3.0 | 3.33 | 3.66 | 4.0 | | | | | | | | |
| 10 | 1.85 | 2.22 | 2.7 | 2.96 | 3.33 | 3.7 | 4.1 | 4.44 | 4.81 | 5.18 | 5.55 | 5.92 | | | | |
| 11 | .... | .... | .... | 3.26 | 3.66 | 4.1 | 4.48 | 4.88 | 5.3 | 5.7 | 6.11 | 6.52 | | | | |
| 12 | .... | .... | .... | .... | 4.0 | 4.44 | 4.88 | 5.33 | 5.77 | 6.22 | 6.66 | 7.11 | | | | |
| 13 | .... | .... | .... | .... | .... | 4.81 | 5.3 | 5.77 | 6.26 | 6.74 | 7.22 | 7.70 | | | | |
| 14 | .... | .... | .... | .... | .... | 5.18 | 5.7 | 6.22 | 6.74 | 7.26 | 7.77 | 8.30 | | | | |
| 15 | .... | .... | .... | .... | .... | 5.55 | 6.11 | 6.66 | 7.22 | 7.77 | 8.33 | 8.88 | 9.44 | 10.0 | 10.55 | 11.11 |
| 16 | .... | .... | .... | .... | .... | .... | .... | 7.11 | 7.70 | 8.30 | 8.88 | 9.48 | 10.07 | 10.66 | 11.3 | 11.85 |
| 17 | .... | .... | .... | .... | .... | .... | .... | 7.55 | 8.18 | 8.81 | 9.44 | 10.07 | 10.70 | 11.33 | 11.96 | 12.59 |
| 18 | .... | .... | .... | .... | .... | .... | .... | 8.0 | 8.66 | 9.33 | 10.0 | 10.66 | 11.33 | 12.0 | 12.66 | 13.33 |
| 19 | .... | .... | .... | .... | .... | .... | .... | .... | 9.15 | 9.85 | 10.55 | 11.3 | 11.96 | 12.66 | 13.37 | 14.07 |
| 20 | .... | .... | .... | .... | .... | .... | .... | .... | 9.63 | 10.37 | 11.11 | 11.85 | 12.59 | 13.33 | 14.07 | 14.81 |
| 21 | .... | .... | .... | .... | .... | .... | .... | .... | .... | ..... | 11.66 | 12.44 | 13.22 | 14.37 | 14.77 | 15.55 |
| 22 | .... | .... | .... | .... | .... | .... | .... | .... | .... | ..... | 12.22 | 13.03 | 13.85 | 14.66 | 15.48 | 16.30 |
| 23 | .... | .... | .... | .... | .... | .... | .... | .... | .... | ..... | 12.78 | 13.63 | 14.48 | 15.33 | 16.18 | 17.03 |
| 24 | .... | .... | .... | .... | .... | .... | .... | .... | .... | ..... | 13.33 | 14.22 | 15.11 | 16.0 | 16.88 | 17.77 |
| 25 | .... | .... | .... | .... | .... | .... | .... | .... | .... | ..... | 13.88 | 14.81 | 15.74 | 16.66 | 17.60 | 18.51 |
| 26 | .... | .... | .... | .... | .... | .... | .... | .... | .... | ..... | 14.44 | 15.74 | 16.37 | 17.33 | 18.30 | 19.26 |
| 27 | .... | .... | .... | .... | .... | .... | .... | .... | .... | ..... | 15.0 | 16.15 | 17.0 | 18.0 | 19.0 | 20.0 |
| 28 | .... | .... | .... | .... | .... | .... | .... | .... | .... | ..... | 15.55 | 16.6 | 17.63 | 18.52 | 19.7 | 20.74 |
| 29 | .... | .... | .... | .... | .... | .... | .... | .... | .... | ..... | 16.1 | 17.18 | 18.26 | 19.33 | 20.4 | 21.48 |
| 30 | .... | .... | .... | .... | .... | .... | .... | .... | .... | ..... | 16.66 | 17.77 | 18.88 | 20.0 | 21.1 | 22.22 |

To reduce to tons:

| | | | |
|---|---|---|---|
| For limestone | multiply | by | 2.27 |
| For traps, syenites, etc. | " | " | 2.52 |
| For granites | " | " | 2.3 |
| For shale | " | " | 2.18 |
| For glass sand or gravel | " | " | 1.55 |

A blast in a Texas limestone quarry consisted in putting down twenty-four 6 in. holes, spaced 10 ft. apart and 20 ft. back from the face; depth varied from 45 to 70 ft., all run 5 ft. below grade. Charge, 6000 lb. of 40 per cent. dynamite averaging 250 lb. per hole, though some holes were loaded heavier than others depending on their burdens. The shot was very satisfactory although the holes were spaced closer together than was necessary. One successful blast was made with 9 holes, 50 ft. deep, loaded with 5150 lb. of quarrymen's special 5 per cent. powder. Later work showed that from 7½ to 10 cu. yd. of rock were thrown down per foot of hole drilled. The explosive cost varied from 9 to 7 cents per ton depending on character of rock and height of face.

The foregoing blasts will serve to give a general idea of what should be expected in well-drill work.

*Spacing of Holes.*—Regarding the spacing of holes, Table 15 is here reproduced from Russell. Table 16, from the same authority, gives the number of pounds of various explosives which can be loaded per foot in holes of different diameters, if the cartridges are slit and well tamped.

TABLE 16.—EXPLOSIVE CHARGES IN POUNDS PER FOOT OF HOLES OF DIFFERENT DIAMETERS

| Dia. of hole in inches | Gel. Dyn. | Straight N. G. Dynamite | R. C. Str. or L. F. Dyn. | R. C. X. or L. F. Am. Dyn. | Ex. or Am. Dyn. | Arctic or N. S. Explosives | Judson R. R. P. | Black Blast Powder |
|---|---|---|---|---|---|---|---|---|
| 3 | 4.25 | 3.75 | 3.68 | 3.60 | 3.72 | 3.25 | 3.25 | 3.00 |
| 3½ | 5.68 | 5.10 | 5.0 | 4.89 | 5.07 | 4.42 | 4.42 | 4.08 |
| 4 | 7.55 | 6.60 | 6.53 | 6.33 | 6.62 | 5.72 | 5.72 | 5.28 |
| 4½ | 9.35 | 8.40 | 8.38 | 8.06 | 8.38 | 7.28 | 7.28 | 6.7 |
| 5 | 11.80 | 10.50 | 10.2 | 10.08 | 10.35 | 9.10 | 9.10 | 8.42 |
| 5⅝ | 14.94 | 13.2 | 12.8 | 12.53 | 13.0 | 11.3 | 11.3 | 10.40 |
| 6 | 17.0 | 15.0 | 14.7 | 14.4 | 14.9 | 13.0 | 13.0 | 12.04 |
| 6½ | 19.5 | 17.5 | 17.25 | 16.8 | 17.49 | 15.2 | 15.2 | 14.0 |
| 7 | 23.1 | 20.4 | 20.0 | 19.6 | 20.28 | 17.7 | 17.7 | 16.3 |
| 8 | 30.2 | 26.7 | 26.13 | 25.6 | 26.49 | 23.1 | 23.1 | 21.4 |

Proper spacing of holes depends on the character of the material and the depth of the face. It is not subject to arbitrary ruling. A nice spacing for holes 35 ft. deep is about 12 ft.

apart and 15 ft. back. Holes 60 ft. deep can be spaced 16 ft. apart by 20 ft. back, and holes 100 ft. or more deep, 20 ft. apart and 25 ft. back in most rocks. It is rarely advisable to space more than 20 ft. apart and in hard rocks it is best to begin with rather closer spacing, say 15 ft. × 15 ft. and work up.

Holes should be drilled 3 or 4 ft. below the quarry floor, unless there is a natural parting at that level, in which case, if holes are sunk to grade, the bottoms will come clean.

*Systems of Shooting.*—As a usual practice well-drill holes are put down in one line more or less equidistant from the face. These are shot and the material is removed before the next line is shot. Occasionally, in very hard rock, better results have been obtained by staggering the holes. Another method of drilling and blasting is known as shooting against the bank, or "buffer" or "blanket" shooting. By this method a line of holes is shot down, the broken material is not moved but another line of holes back of the first line is shot. The débris from the first shot thus blankets the second shot. It is well adapted to limestone formations in which the stone is flat, thinly laminated or disintegrated on top, and where the face is not over 40 ft. high. The method eliminates the necessity of moving the shovel as often as when clear bank method is used. Care must be taken not to load too high in this method, as often the break-back makes it difficult to drill the next line of holes.

In bench mining as carried out by some of the porphyry copper mines, it has been the practice to do the drilling and blasting of the bank in front of the shovel. On account of narrowness of the benches it has not generally been possible to carry out the blasting operations very far in advance of the shovel requirements. Great delays to shovel operation have often resulted from this practice, viz., the shooting of but one hole at a time in front of the shovel. These delays have been due to unfinished holes, to moving the shovel back while the bank is shot, to cleaning off the track after the shot and to breaking up large boulders.

If the benches are widened out by an additional 50 ft., more money will, of course, be tied up in advanced stripping, but great advantages in blasting will result for a system may be used whereby several holes will be shot at a time well *back* of the shovels. The apparent advantages of this system are:

1. The shovel will not have to wait for a hole to be shot but will have broken ground ahead of it at all times.

2. The shovel will not have to move back for the blast.

3. The loading track, on the bench being blasted, will be the width of the shovel cut away from the toe of the bank at the time of blasting, and will not be covered with rock. After the blast it can be thrown into the broken material and made ready for the next cut.

4. A big proportion of the oversize boulders can be broken in the most convenient and efficient way before the shovel reaches them, as there will be ample time. Further, blasting them will not jar up the shovel as is sometimes the case with close heavy doby shots.

5. It is believed that shooting holes in series will show some economy in explosives. Certainly some economy could be effected in shooting the boulders by block-holing.

6. The loading of the holes may be done in a quiet systematic way, without any rush because of a waiting shovel. The sparks from the shovel will not endanger the lives of the powder-men. The work may also be cheaper.

7. A more systematic method of drilling can be used and at less cost per foot of hole.

With further reference to point 5, there is a difference of opinion, perhaps based largely on local conditions of the work, but the following questions have often been put:

(*a*) How does the powder consumption per cubic yard compare, using single-hole vs. multiple-hole shots?

(*b*) Which method leaves the broken ground in the best condition for handling?

(*c*) How does the cost of drilling compare in the two cases?

(*d*) Is the blasted bank left in better or safer condition in the one case than the other? This refers to large blocks which may roll down the bank endangering workmen or the shovel, and to the partly blasted portions of banks which may cause some trouble when removed by the shovel or when the next set of holes is drilled in portions so affected.

(*e*) How does the daily shovel output compare in the two cases? This assumes that the capacity of the shovel varies inversely as the delays. Thus if delays due to blasting amounted to 6 per cent. of the total operating time, the shovel output could be expected to be increased by roughly this amount if these delays were entirely eliminated, and the cost per cubic yard of handling material would be reduced by a smaller amount.

In some of the examples of mining operations given above these questions have been answered but not always in the same way. They must be decided on each individual job.

It is not usually desirable to break ground too far in advance of shovel operations, for, in addition to the premature expenditure of the cost, heavy rains may so settle the material that it is hard to handle, and in freezing weather the broken material may freeze together so firmly that reblasting may be necessary. Judgment must be used in determining how far ahead blasting should be carried.

*Breaking Oversize Material.*—In so far as economy of powder is concerned in breaking oversize there is no doubt but that block-holing is by far the best method. Elaborate tests were carried out by the United States Bureau of Mines to determine conclusively the comparative energy utilized by exploding powder under water and in the air. In a few words these tests showed that much greater breaking effect was obtained with one-half the powder by block-holing than by the best mud-capping. In some cases, where 80 per cent. of the total powder is used for bank shots and 20 per cent. for breaking oversize by mud-capping, it may be well worth while to consider block-holing.

Table 17 shows the actual saving in powder that can be effected with block-holing over mud-capping and snake-holing.

TABLE 17

| Weight of boulder, pounds | Size cu. yds. | Approximate number of 1¼ in. × 8 in cartridges required. | | |
|---|---|---|---|---|
| | | Mud-capping | Snake-holing | Block-holing |
| 100 | .02 | 0.5 | 0.5 | 0.25 |
| 500 | .12 | 1.5 | 1.0 | 0.25 |
| 1,000 | .23 | 2.0 | 1.5 | 0.50 |
| 2,000 | .47 | 3.0 | 2.5 | 0.67 |
| 3,000 | .70 | 3.5 | 3.0 | 1.00 |
| 4,000 | .93 | 4.0 | 3.5 | 1.25 |
| 5,000 | 1.16 | 4.5 | 4.0 | 1.75 |
| 7,500 | 1.74 | 6.0 | 5.0 | 2.50 |
| 10,000 | 2.33 | 8.0 | 6.0 | 3.50 |

*Various Explosives Used.*—There are many explosives manufactured, nearly all of which differ in some way; some are slow, others quick, some dense, others bulky, some good only in dry work, others in wet work, and some in very cold weather. To determine the best explosive for a certain set of conditions usually requires individual experimenting.

After deciding what explosives are best suited to a job, it is advisable to standardize these. Too many varieties are likely to be inconvenient to carry or confusing to work with.

In mining work the explosives generally used are gelatin dynamites, straight dynamites, extra or ammonia dynamites, Judson powder, nitro-starch powders, and diminishingly, black blasting powder. The straight nitro-glycerine dynamites are quickest and most shattering, also the most sensitive and dangerous to handle. Blasting gelatin, containing about 93 per cent. nitro-glycerine and 7 per cent. gun cotton is the most concentrated and powerful. The so-called gelignites and gelatin dynamites, varying in strength from 30 to 75 per cent. rating, are coming into wider use. These and blasting gelatin are comparatively safe to handle and are well adapted for very wet work. It is considered best to detonate them with a straight nitro-glycerine primer or Cordeau. The strongest detonating caps should be used.

The ammonia dynamites and nitro-starch explosives are slower than the straight dynamites and are not well adapted to wet conditions where the powder is to be exposed to water for a considerable time before shooting. They are less sensitive, however, and well suited where great shattering effect is not desired.

Judson powders and blasting powders are unsuited for hard rock excavation or wet work. They are all right where a slow heaving action is desired, as in earth, shale, sand and laminated material. When used in chambered holes in hard rock a large amount of oversize boulders result.

The more recent low freezing nitro-glycerine and gelatin dynamites do not differ in action from the high freezing powders and are just as efficient. Ordinary dynamite freezes at about 46°F. and must be thawed with care in cold weather.

Mr. Russell does not recommend springing well drill holes in quarry work, finding it a slow, tedious, expensive and somewhat dangerous operation, and prefers reducing the spacing of the holes and using a more concentrated powder to bring down the rock.

*Calculation of Charges.*—No rule can be given as to the amount of powder to load in holes of given depth as much depends on local conditions. It is very important to select the proper explosive and then it is necessary to "cut and try."

Before loading it is usual to calculate the number of tons or cubic yards available in the blast, and then to get about 5 tons of rock per pound of explosive, varying the load per hole according to the burden or local conditions. As a general rule some hold that a hole of any depth should be filled at least half its length with explosive, *e.g.* a 40 ft. hole should have 20 ft., or a 60 ft. hole 30 ft. of explosive.

To obtain maximum breakage and proper distribution of the explosive in the hole, the charge should come up in the hole to at least 30 ft. from the top, no matter how deep the hole may be.

One group of powdermen made up a formula for charging holes as follows:

$$P = \frac{A^2 \times H}{2}$$

P = pounds of powder to use (40 per cent Hercules)
A = distance from charge to toe (feet)
H = height of bank (feet).

Allowances were then made for any unusual conditions of burden or material.

All such rules must be considered as arbitrary and local, but they may be of some help in starting operations.

In deep holes considerable saving can be made and equally good results obtained by breaking the load two or three times. The object in breaking the load, besides saving explosive, is to distribute the charge where the rock is hardest, skipping seams and weak points where it is not needed. In shooting a series of holes it is held best to arrange the breaks so that they are not all on the same level. This amounts to the same thing as shooting two or more benches simultaneously. In loading deep holes the paper need not be removed as loose powder will be scattered along the walls of the hole and at the mouth. The sleeve used at Nevada Con. is to be recommended for loading powder in any condition.

In hard rocks a combination of 60 per cent. and 40 per cent. dynamite is recommended. A little 60 per cent. should be loaded in the bottom of each hole and 40 per cent. used on top. In softer rocks 40 per cent. will usually be found strong enough and often a lower grade can be used on top.

*Detonators.*—If holes are double or triple loaded at least two electric exploders should be used in each charge unit to assure thorough detonation and to afford a way out in case one should be damaged in tamping. Electric exploders with duplex wire leads have been found a convenience in deep holes because of their handiness and strength. Sometimes short-length electric exploders are used with connecting wire splices, if so the splices should be carefully made and well taped. On one operation, wires with red and blue colored insulation are used. In multiple shots, either parallel or series, there is thus less chance of mistake in final connections.

Strong detonators are recommended, at least No. 6 and preferably No. 8. In all holes at least two detonators should be used. A good rule is to place a detonator every 25 ft. in the explosive charge. Electric exploders should be tested with a reliable galvanometer before placing in the hole and again after the hole has been loaded and tamped. The entire circuit should be tested when all loading is completed and all connections have been made except to the blasting machine or power circuit. Connections should be made only in series if a blasting machine is used and tests should be made to see that the resistance of the circuit does not exceed the capacity of the blasting machine. If a power circuit is used for firing, either series, series-parallel or parallel connections may be made. If connections are made in parallel, at least $\frac{1}{2}$ ampere per exploder should be provided. If in series or series-parallel, 2 amperes should be allowed for each series.

A detonator of great merit in certain classes of work is Cordeau or Cordeau-Bickford detonating fuse. This consists of a lead tube 0.22 in. in diameter filled with a high explosive compound which is perfectly safe to handle or knock about and can only be detonated by direct contact with a blasting cap. It comes in spools of 150 to 500 ft. and is used as follows: the end of the Cordeau is laced through a dynamite cartridge, which is lowered to the bottom of the hole, allowing the Cordeau to extend the full length of the hole. The hole is then charged in the usual manner and the Cordeau is cut allowing about a foot to extend above the collar of the hole. After the series of holes to be shot are loaded in this way, the projecting ends of Cordeau are split down about 3 inches and forked Another length of Cordeau is laid across these forked ends, which are twisted tightly around it, and all are thus connected up. A blasting cap is attached to the end of the connecting Cordeau by means of a brass union and when this is exploded the entire line of Cordeau is detonated. It thus causes a thorough detonation the whole length of each explosive charge simultaneously; any number of holes can be connected with certainty of firing.

*Tamping.*—A few comments may be made upon the tamping of powder in the holes. In most metal mining work this is not done, except as the powder tamps itself in falling or as aided by a sounder. The dirt shovelled in on top of the charge is seldom tamped. A certain amount of water is often used, especially in springing, as tamping. Results thus obtained have generally been satisfactory but if it is found convenient to tamp the charge, the additional confinement, especially in unsprung holes, may show better results.

If tamping is done the tamping block should be of wood about 4 to 5 ft. long and just easily clear the hole, say, 5½ in. diam. for a 6 in. hole. It may be given more weight by babbitting the end. This block will be attached to a rope so that it hangs straight in the hole, and it can then be jigged up and down directly by one or two men, or a light tripod and sheave may be set directly over the hole if it is very deep. From 10 to 25 lb. of powder may be dropped in the hole at a time and then tamped. Tamping should never be done with the drill stem by screwing a wooden block in the end, nor should any metal parts (except the babbitt) be used on the tamping rig in the hole. Such practice has caused many accidents.

Whether the powder is tamped or not, the remainder of the barrel of the hole should always be carefully filled to the top with sand, clay or screenings, and especial care should be taken not to injure the exploder wires or Cordeau.

*Safety Rules.*—In loading holes a few rules should always be kept in mind and enforced. Many of these, of course, apply to any kind of blasting.

1. Permit no smoking in the vicinity of loading.
2. Be certain that a sprung hole is cool before re-springing or loading it.
3. Guard against sparks from steam shovels, locomotives or other drills falling near the loading operations.
4. See that the men assisting in the work have no friction matches in their clothing.

5. Do not allow loose powder to collect around the mouth of the hole.

6. Do not have more than sufficient powder to charge one hole in any one pile and keep this carefully covered with a tarpaulin until time to commence loading. If the hole is to be sprung the pile should be at a reasonably safe distance from it. It is best not to have the piles out over night.

7. If possible, complete the loading and firing without interruption. All the loading and shooting should be done on the day shift if possible and the powder crews should not work over eight hours except when unavoidable to finish a blast.

8. Do not connect up any lead wires until just before firing. Wires to be left for any length of time should have the ends buried in the earth. Electric storms have been suspected of setting off blasts.

9. Be sure dynamite is properly thawed before using. Do not cut, break or try to load frozen dynamite in a bore hole.

10. Be sure that all signals between blasting crew and other pit crews are well understood and obeyed by all. A sufficient time interval to protect crews and equipment should be provided for by these signals.

**Gopher Holes**[1].—This method may be called snake-holing on a big scale. It is often used in high cliff quarry work where the strata are irregular, or where drilling is inaccessible or inconvenient. This is a very old method of blasting developed in Europe and used to some extent in the western coal states and in the copper mines. One method is to drive a tunnel, say, 4ft. × 4 ft. in section, with jackhammer drills, from 40 to 50 ft. back from the bottom or toe of the face, and then to drive cross-headings Tee fashion from this of varying length according to the burden to be shifted. No explosives are placed in the main entrance legs but only in the cross-headings. The explosives may be loaded in recesses or sumps cut at certain intervals in the cross headings, but loading at grade gives good results.

If the face is very high, and it is possible to do so it is a good plan to sink well-drill holes on the surface from near the outer edge of the bench above, and load lightly with explosives. This greatly assists in breaking up the top ledges and gives the bank a safe slope. It is recommended that these holes extend about one-third the way down. A maximum height of about 150

[1] S. R. Russell, DuPont Magazine, Sept., 1916.

ft. to 175 ft. is all that should be expected with one system of adits, but cases where the bank was 240 ft. high have been successfully blasted. The method is not economical if the face is less than 70 or 80 ft.

The explosives generally used in gopher-holes are, say, twenty per cent. of 40 per cent. to 60 per cent. dynamite and eighty per cent. of DuPont R.R.P. or black powder. The explosive charges are placed 15 to 20 ft. apart and, in loading, it is best to remove the cartridges from the boxes so that they can be packed in better. Dynamite is laid on the bottom and R.R.P. on top.

A unit charge may consist of a few hundred up to several thousand pounds, depending on the burden. From two to three electric detonators should be primed in each unit. As a safety precaution against misfires, a line of Cordeau should be strung all along the adit and drifts connecting all explosive units.

For tamping between the units, screenings from the spoil of driving may be used. At the intersection of the legs and at least half way out the main leg, lean concrete should be used for tamping. This should be allowed to set about 48 hours before the blast is fired.

There are several methods of carrying the wires and making the connections. The wires may be put through pipes or grooved boards, but it is recommended that they be strung through eye bolts driven in the roof. A separate pair of wires should run from the portal to each unit; No. 14 gauge or heavier copper wire should be used Each pair of wires should be carefully tagged to indicate the unit it leads to. Tests should be made progressively to insure that the circuit is intact. When a power circuit is available parallel connections should be made.

Very good fragmentation is obtained by this method of blasting and under certain conditions it is the most economical method to employ. From five to six tons of material can be obtained per pound of explosive. While this method is applicable to many operations, yet if the face can be economically drilled it is advisable to recommend it.

At the mines of the Utah Copper Company, the great 240-ft. bank was blasted by gophering and well-drilling combined. Fig. 33 illustrates the method and details of the work and shows the powder charges employed.

Gopher-holing, when first used on the Mesabi iron range, consisted in making the holes large enough to permit a man to enter and work, but frequent accidents caused this system to be abandoned, and gopher holes now have an average diameter of about 15 in.

The benches are drilled and blasted by regular crews of "gopher-holers" made up of 10 to 30 laborers, working in groups of two men. The benches are 15 to 25 ft. high and are riddled with a series of holes 15 to 25 ft. deep, spaced 15 to 25 ft. apart. The general rule is to make the horizontal distance between the centre of the loading track and the chamber of the gopher hole 5 or 6 ft. less than the reach of the shovel. The collar of the

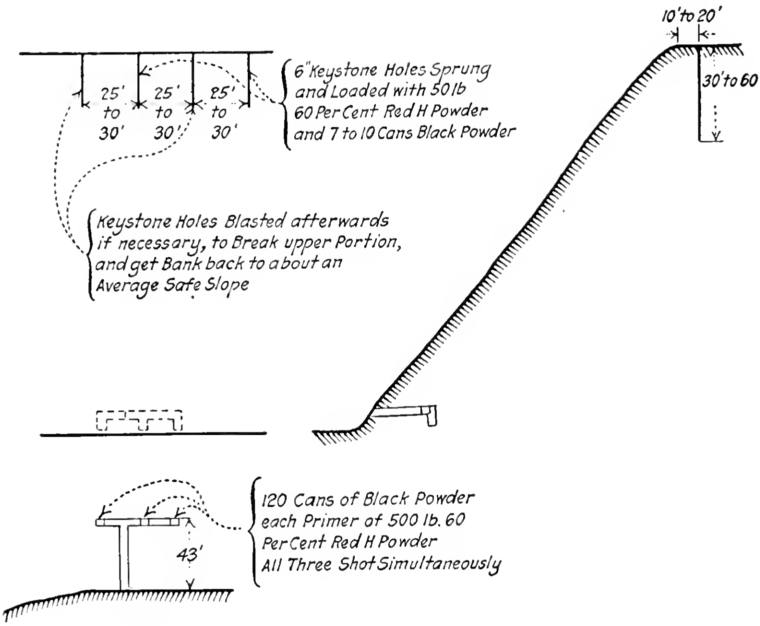



FIG. 33.—Drilling and blasting 240-ft. bank; gophering.

hole is at the base of the bank and the hole points downward at an angle of from 10° to 20° from the horizontal. The spacing of the holes depends on the hardness of the material and the burden imposed. The holes are made with a long spoon shovel, made by slightly turning up the edges of the blade of a No. 2 round-pointed shovel and fitting to it a 25 ft. handle of 2 or 3 in. diameter. When a hard seam is encountered it is drilled with a 24 ft. augur, or a moil, and is sprung by pushing in one or two sticks of powder with a pointed loading stick, and then firing with a blasting machine. The loose ground is then removed with the shovel. If a boulder is struck in the hole, repeated blasting with 60 per cent. dynamite will often shatter it sufficiently to allow the hole to be continued. If this is not successful, the hole is bottomed against the boulder and a new one started a few feet away. From 2 to 12 hr. is consumed per hole and costs from $2 to $8 for labor.

In winter the top of the banks freezes as deep as 8 ft., and unless this crust is broken by top drilling before gopher-holing is done, the latter usually undercuts the bank, causing slabs of frozen ground to slide down and bury the loading track.

The powder boss determines the size of the powder charge from the height of the bank and the material taken from the hole. With a 25-ft. bank, 15 to 25 sticks of dynamite are used to spring the hole, which is then loaded (after cooling) with from five to ten 25-lb. kegs of DuPont black blasting powder. The powder is fed in through a long wooden launder, about 2 in. × 2 in. inside cross section, fitted with a covered hopper at one end. A keg of powder is emptied into the hopper, the cover is closed and the loader is oscillated by a 12-ft. cross handle, causing the powder to run down the launder into the chamber of the hole. The long cross handle allows the powdermen to stand 6 ft. on either side of the hole instead of in front of it. The closed hopper protects the powder from flying sparks. Another method is to attach a hand blower to the launder by means of a rubber hose, and as the powder is blown in the launder is gradually pulled out. Both methods are quite safe. Wooden spoons, 3 in. × 3 in. in cross section, and 2½ ft. long, fitted with 25 ft. handles, have been used but are not as good.

The detonator consists of from 2 to 5 sticks of 60 per cent. dynamite with exploders, and is placed about two-thirds of the way down the charge. The two exploders may both be electric, or may be one electric and one ordinary cap with fuse. The latter combination is in more general use, since tamping sometimes injures the cap wires. Tamping is essential, and is done by filling the holes to the collar with sand or gravel. Holes are fired in sets of 3 to 5 at a time.

A unique method of gopher blasting has been worked out at the mines of the Chile Copper Company, at Chuquicamata, Chile.[1] Instead of the entrance to the gopher drifts being from

[1] E. E. Barker, M. & S. P., Sept. 30, 1916.
Howard W. Moore, M. & S. P., July 8, 1916.

the working face, it is placed well back of the face, this being accomplished by means of a shaft and connecting drift. Blasting by means of churn-drilling was first tried but the ground proved quite hard and the gopher hole method was adopted. The latter has resulted in lower costs per cubic yard of material blasted because less footage has to be driven between charges, and because the expense of springing operations, used in well-drill holes, is eliminated.

Some of the details of this work may here be interesting.

With the method of blasting by well-drill holes, 5⅝ in. holes were drilled about 25 ft. apart, in rows from 40 to 50 ft. apart, and to a depth of 4 to 5 ft. below the grade of the benches, which were 40 to 100 ft. high. These holes were sprung from 5 to 7 times using 60 per cent. or 75 per cent. dynamite, and a chamber 6 to 8 ft. in diameter was formed, which was then loaded. This loading was done in a manner similar to that used by the Nevada Consolidated Copper Company.

The cost of the drilling was unusual, viz., from $2.50 to $3.60 per foot. On this work Keystone electric drills made about 17 ft. per shift and Cyclone electric drills made about 23 ft. per shift. The latter machines proved very satisfactory. Lower footages were, of course, made on deeper prospect holes.

The method of blasting from tunnels is as follows: shafts are sunk at several convenient parts of the bench to a depth of 3 meters below grade, from the bottoms of these shafts cross-cuts are run parallel to the short axis of the orebody and about normal to the bench face, and from these cross-cuts, drifts are run every 15 to 30 meters, parallel to the long axis of the orebody. See Fig. 34. The cost of this work is about $10.00 per ft. The spacing has been increased to 30 meters, where the banks are at least 1½ times as high. In these drifts the charges of explosives are placed at 10 meter intervals. In calculating the loading charges, cross sections are taken to scale through the loading points and normal to the bench face as illustrated by Fig. 35. Scaling the line of least resistance indicates the approximate charge required and it has been determined that from 463 to 600 lb. of black powder, depending on the material to be blasted, should be loaded in each charge for every meter as measured on the line of least resistance. To translate the black powder, which is manufactured in Chile, into 40 per cent. dynamite the resultant charges may be divided by the factor 1.50, or into 60 per cent. dynamite by the factor 2.25.

For compactness the powder is loaded in sacks of 100 lb. each and these sacks are closely piled from floor to roof of the drift, the interstices between the sacks being filled with sand or muck. In the centre of each charge are placed two boxes of 60 per cent. dynamite which serve as primers. One electric exploder is carefully placed in each box of dynamite. See Fig. 36. The lead wires from the primers are carried along the floor of the drift in grooved wooden stringers (2 in. $\times$ 3 in. with $\frac{3}{4}$ in. groove) provided with $\frac{1}{2}$ in. covers. After a chamber is charged and

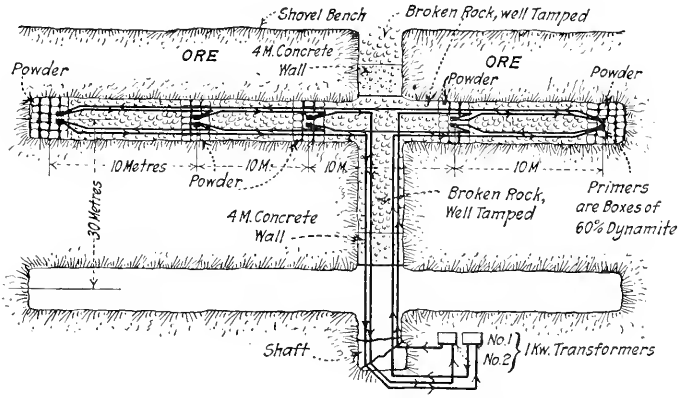



FIG. 34.—Plan of tunnel blasting, Chuquicamata, Chile.

connections made, broken rock is used to fill up the drift closely. Two separate electric circuits on different transformers are provided as a precaution against misfires. From careful experiments it was found that for a series of 20 exploders, 0.75 amperes current under 110 volts e.m.f., was the least current that would explode the series. It was, therefore, determined never to use a safety factor of less than 4, or in this case to provide 3 amperes. Each exploder plus about 30 ft. of fuse-wire, showed a resistance of about 2 ohms. See Fig. 37.

After the drift is loaded, the cross-cut leading back to the other workings is filled with broken rock to within 4 meters of the first drift back, and at this location a solid concrete bulk-head is put in, as illustrated in Fig. 34. The principal reason for this is to keep the gases after the shot, out of the workings back of the shot.

The handling of such large quantities of explosive requires constant close care and supervision. Further experimental work and experience will be required before the most efficient spacing,

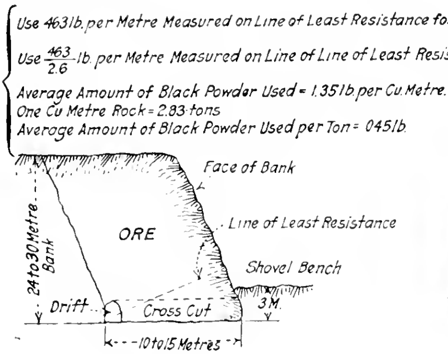



Fig. 35.—Method of calculating powder charge.

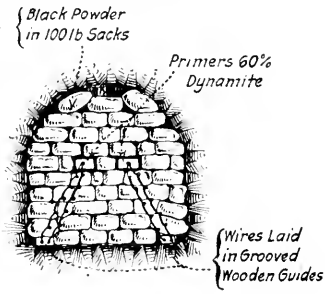



Fig. 36.—Method of loading tunnel for blasting.

charges and other details are finally determined, but the results obtained to date are considered very satisfactory.

About 1¼ lb. of black powder and 0.02 lbs of 40 per cent. dynamite are consumed per ton of material broken. About 400 lb. of powder is the load per lineal foot of tunnel loaded. More

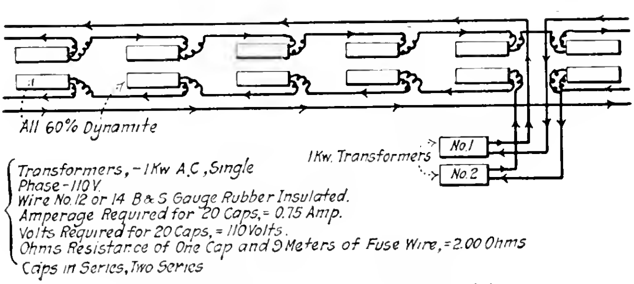



Fig. 37.—Diagram for tunnel blast wiring.

recently the black powder has been reduced to as low as 0.6 lb. per ton of ground broken.

Mixtures of clay and gravel used for ballasting have been loosened up for handling with steam shovels by a simple method of gopher-holing. In one case holes 20 inches wide by 26 inches high were extended into the gravel bank a distance of 26 feet and then turned at right angles for 10 feet. The excavations were made by a man lying down and working the material loose with a very short handled pick. The charge of black powder was placed in the extreme end of the 10 foot leg, the remaining distance .of which, and a few feet at the end of the 26 foot drift, was then refilled with gravel. This mixture was very compact, being about 75 per cent. small gravel, 20 per cent. clay and 5 per cent. sand. The method of loosening was cheap as about 80 ballast cars of material were loosened per blast and the complete drifts were dug at a cost of about $18.

**Storing and Thawing Explosives.**—All of the important powder manufacturers issue detailed instructions and will plan proper facilities for the storing, handling and thawing of explosives. It is advisable to consult with their engineers in planning this operation.

On the Mesabi range, powder magazines of heavy sheet steel are furnished by the powder company supplying the explosives. Sheet iron magazines for caps and fuses are usually built by the operators. Thawing houses are built according to recommendations furnished by the DuPont Powder Comapny, and are 5 ft. × 8 ft. outside. Shallow drawers, with 1 in. holes perforating the bottom, are provided in which to store the cartridges, and are accessible from the outside of the building. A radiator in the back part of the thawing house is supplied with hot water through a 1 in. pipe from a heater in a separate building. This heater is a small water-jacketed coal-fired stove. The housing for the heater is 4 ft. × 4 ft. and is placed not less than 10 ft. from the thawing house.

At some of the copper properties larger powder magazines have been built of heavy logs, stone and concrete. If it is necessary to keep large stocks of explosives on hand, it is best to distribute them in several magazines, at a safe distance apart and from dwellings or townsites. They should, however, be quite accessible and open to observation or protection in times of labor or other trouble.

The use of low and extra low freezing powders has in many places relieved much of the trouble of thawing. The use of the refrigerator type of powder car for transporting thawed powder about the pits has also been found useful.

It is often necessary to conform to state or federal regulations regarding the storing and transportation of explosives, and such regulations should be carefully observed. The Interstate Commerce Commission's requirements for the storage of explosives in magazines is quoted in the following tabulation from the Annual Report of Chief Inspector of the "Bureau for the Safe Transportation of Explosives and other Dangerous Articles," dated Feb. 1, 1911 (Tables revised Nov. 30, 1912, to include intermediate quantities).

Table 18.—Minimum Distances

| Lbs. of explosives | Distances, inhabited building, barricaded[1] (feet) | Distances, public railway, barricaded[1] (feet) | Lbs. of explosives | Distances, inhabited building, barricaded (feet)[1] | Distances, public railway barricaded (feet)[1] |
|---|---|---|---|---|---|
| 50 | 120 | 70 | 60,000 | 1,565 | 940 |
| 100 | 180 | 110 | 65,000 | 1,610 | 970 |
| 200 | 260 | 155 | 70,000 | 1,655 | 995 |
| 300 | 320 | 190 | 75,000 | 1,695 | 1,020 |
| 400 | 360. | 215 | 80,000 | 1,730 | 1,040 |
| 500 | 400 | 240 | 85,000 | 1,760 | 1,060 |
| 600 | 430 | 260 | 90,000 | 1,790 | 1,075 |
| 700 | 460 | 275 | 95,000 | 1,815 | 1,090 |
| 800 | 490 | 295 | 100,000 | 1,835 | 1,100 |
| 900 | 510 | 305 | 125,000 | 1,900 | 1,140 |
| 1,000 | 530 | 320 | 150,000 | 1,965 | 1,180 |
| 1,500 | 600 | 360 | 175,000 | 2,030 | 1,220 |
| 2,000 | 650 | 390 | 200,000 | 2,095 | 1,255 |
| 3,000 | 710 | 425 | 225,000 | 2,155 | 1,295 |
| 4,000 | 750 | 450 | 250,000 | 2,215 | 1,330 |
| 5,000 | 780 | 470 | 275,000 | 2,275 | 1,365 |
| 6,000 | 805 | 485 | 300,000 | 2,335 | 1,400 |
| 7,000 | 830 | 500 | 325,000 | 2,390 | 1,435 |
| 8,000 | 850 | 510 | 350,000 | 2,445 | 1,470 |
| 9,000 | 870 | 520 | 375,000 | 2,500 | 1,500 |
| 10,000 | 890 | 535 | 400,000 | 2,555 | 1,535 |
| 15,000 | 975 | 585 | 425,000 | 2,605 | 1,560 |
| 20,000 | 1,055 | 635 | 450,000 | 2,655 | 1,590 |
| 25,000 | 1,130 | 680 | 475,000 | 2,705 | 1,620 |
| 30,000 | 1,205 | 725 | 500,000 | 2,755 | 1,655 |
| 35,000 | 1,275 | 765 | 600,000 | 2,935 | 1,760 |
| 40,000 | 1,340 | 805 | 700,000 | 3,095 | 1,855 |
| 45,000 | 1,400 | 840 | 800,000 | 3,235 | 1,940 |
| 50,000 | 1,460 | 875 | 900,000 | 3,355 | 2,015 |
| 55,000 | 1,515 | 910 | 1,000,000 | 3,455 | 2,075 |

[1] "Barricaded," as here used, signifies that the building containing explosives is screened from other buildings or from railways by either natural

**Care in Use of Explosives.**—All operators know that detailed care in the use of explosives cannot be over-emphasized. Nevertheless, many accidents occur each year to which definite simple causes can be assigned, and many others occur, the cause of which is impossible to ascertain. In recent years a great deal of interest has been taken in safety work and large sums have been spent by the big companies in an endeavor to make the operations as safe as possible. Safety regulations have been issued by many companies, printed in pamphlet form and in the languages of all workmen.

Special care has been given to the selection and training of powder foremen, for upon them rests the direct supervision and common sense of the work. Both foremen and men are impressed with the motto "Safety First" and taught to consider themselves individually a committee of safety. They are invited at all times to submit suggestions as to how the operations can be better or more safely conducted, and from such suggestions many rules and devices have been improved. In starting a new property experienced men should be entrusted with the training of the personnel required for powder work.

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or artificial barriers. Where such barriers do not exist, the distances shown should be at least doubled.

"An artificial barrier shall be held to mean an artificial mound or properly revetted wall of earth of a minimum thickness of not less than three feet, of such height that any straight line drawn from the top of any side wall of the building containing explosives to any part of the building to be protected will pass through such intervening artificial barrier, and any straight line drawn from the top of any side wall of the building containing explosives to any point twelve feet above the center of the railway will pass through such intervening artificial barrier. The foregoing definition as to height shall also apply to any natural barrier.

"For quantities not given in the above table use distances shown for nearest tabulated quantity, or if extreme accuracy is desired take proportionate figures."

## CHAPTER V

# DISPOSAL OF MATERIAL

### TRANSPORTATION

**Trackage Arrangements.**—Every effort should be made to supply a proper balance of train service to keep the shovels operating as continuously as practicable. The efficiency of this service will depend on the amount, type and condition of equipment available, length of haul, track layout, weather conditions and good "railroading."

Ideal conditions would be where the track was so arranged that a train of empties could follow into loading position at the side of the shovel just as a loaded train pulls out. In some large pits a ladder track layout is arranged where the empty or return track from the yards branches into a number of loading tracks that feed the individual shovels. At the opposite end of the pit these tracks converge into a single track connected to the main track out of the pit. More commonly there is a passing track provided for each shovel. Where it is not practicable to get a passing track in, a stub track or "lie-by" may be laid just long enough to take an empty train while the loaded train passes by. The trackage arrangement must continually be kept in mind as it is constantly changing.

**Delays.**—Delays due to train service vary widely, depending on the layout and how well the crews are organized. Delays may be due to blasting, covered track, moving shovel, changing trains, spotting cars, dumping or simply waiting for trains. The nearer the shovel the empty train is and the more cars hauled per train, the less time is lost in changing trains. Track conditions affect the train length in several ways, for example, at the end of a cut there may be insufficient room ahead of the shovel to accommodate the whole train, which then must be cut up and switched in and out in short units. When this sort of loading is required the switch should be as well up with the shovel as possible. The effect of curvature, grades and weather on the length of trains has already been taken up in Chapter II. The advantages of the Woodford centrally controlled system has been fully described there.

**Distribution of Equipment.**—The transportation equipment for handling waste is usually kept quite separate from that for handling ore, and is usually of a different type. On the iron ranges and at the porphyry copper mines the ore cars used are furnished by the railroads. These are brought to an assembly yard and from this yard pit engines draw on the supply as required. The pit engines haul the empties to the loading positions of shovels working in ore and return with the loads to the assembly yard where the ore trains are made up for railroad transport to the docks or mills. The stripping equipment is handled entirely by the mine's organization or by the stripping contractor.

**Routing of Overburden.**—The waste haul depends on the location of the dumps. As a rule there will be several dumps in use, and it is the duty of the engineer to prepare a schedule of waste routing which will allocate the stripping from various areas and elevations in the pit to those dumps best suited to take care of the waste. Such a routing schedule should be made up monthly and clearly indicate what parts of the pit it is expected to strip, the yardage expected and the dumps to which each component portion is to go, as well as the remaining capacity of each of the dumps. It is also of interest to know the relative cost of moving overburden to the different dumps.

**Distribution of Crews.**—On the Mesabi Range, the overburden is transported from two to four miles. The main haulage tracks are on grades of from ½ per cent. to 2 per cent. and are well kept up. Rod engines of from 50 to 60 tons are generally used, and the dump cars range from 7 to 20 cu. yds. capacity. The dump crews consist of three gangs. One gang is on the dump and attends to the unloading of the cars and cleaning the track; another gang shifts the track; the third gang is stationed at a point where the dump's tracks converge on the main line, and here the men right and lock the cars in position for loading.

At the porphyry copper mines, the one crew at the point of dumping usually takes care of all of these duties. In this way all hands are available for any operation and very little time is lost. The train is quickly dumped, righted and locked, and the track cleared to enable it to return to the pit. The dump crew then devotes its time to further clearing of track and bank, or to shifting tracks.

**Use of Salt Solution.**—To assist dumping in freezing weather, when handling moist material, which has a decided tendency to build up on car bottoms, a hot salt solution is recommended from Mesabi practice. The salt water tank is made of wood, holds about 2500 gal. and is filled with water and salt to form a saturated solution. This solution is heated to the boiling point by a steam pipe or small vertical boiler. In freezing weather the car bottoms are periodically sprinkled with this solution from a hose. The result is satisfactory and the expense not great. The tank should be conveniently located to prevent delays to equipment being treated.

**Economy in Large Cars.**—At one of the porphyry copper mines an attempt was made to determine the relative economy of loading 20-cu. yd. and 18-cu. yd. (water measure) dump cars; these cars having a capacity of 16.1 and 14.5 place cu. yds. respectively. The advantages in hauling, spotting and dumping the 20-cu. yd. cars seemed quite evident, but it was desired to ascertain just what advantage was gained in loading time. Under conditions as nearly alike as could be found, covering a period of three months, and eliminating all time factors but the loading time, the results seemed to show that there is an advantage in using larger cars and that under similar conditions more yardage can be loaded into large cars than small ones in the same space of time.

**Estimating Cost to New Dump.**—When a new dump-site is proposed it is often desired to know what the cost of transportation to the new site will be as compared to that to other active dumps. A fair estimate of this may be made by noting, over a week's time, the average number of cars per locomotive-hour which are hauled to the active dump, and by observing the operations of the trains and proportioning the time for each phase of the work. These results can then be used to estimate the cubic yards per locomotive-hour which could be hauled to the new dump, making allowance for difference in length of haul, passing of trains or other conditions. The data may be set down as follows:

Estimated Cost of Transportation to Dump A.

| | |
|---|---|
| Length of haul, round trip | 9200 feet |
| Grades—1250 feet 4 per cent. in favor of loads. | |
| 2600 feet 2 per cent. adverse to loads. | |
| Average number of cars per train | 5 (or 72.5 cu. yds.) |
| Average number of cars per locomotive-hour | 5 |
| Overhead cost per locomotive-hour | \$3.00 |
| (includes supt., foreman, yardmaster, laborers, etc., but not dump gangs.) | |
| Labor cost per locomotive-hour | \$1.28 |
| Supplies cost per locomotive hour (9/8 labor) | 1.44 |
| Total cost per locomotive-hour | \$5.72 |
| Total cost per cu. yd | \$0.0789 |

Distribution of Time Consumed

Average time per round trip for five-car train equals 60 minutes.

| | Minutes |
|---|---|
| Loading train | 18 |
| Time train in motion | 15 |
| Switching | 6 |
| Dumping cars | 5 |
| Waiting at switch to meet other train | 4 |
| Delays at shovel | 6 |
| Other delays | 6 |
| Total | 60 |

Average running speed is $9200 \div 15 = 613$ ft. per minute. It takes 20 sec. to start or stop a train and requires a distance of 200 ft. On this run there are seven stops and starts per round trip. The speed, when underway may be found as follows:

$14 \times 20$ sec. $= 280$ sec. or say 5 min. in starts and stops.

$14 \times 200$ ft. $= 2800$ ft. distance consumed in starts and stops.

$(9200 - 2800) \div (15 - 5) = 640$ ft. pr min.

Using the foregoing as a basis, the data on the proposed dump may be estimated.

**Pit Haulage on Mesabi and Elsewhere.**—Some examples of pit haulage on the Mesabi are noted in the following. The Shenango iron mine has an average waste haul of 2½ mi. The pit is 150 ft. deep and the overburden varies from 50 to 80 ft. The track system has sharp curves and the grades run up to 3 per cent. Four locomotives are used in the pit, and in addition to the regular locomotives for handling the cars, two Lima geared engines are used as boosters on each ore train. Because of the curves and grades, one 50-ton rod engine and the two Limas can handle only four 50-ton cars from the bottom of the pit.

On a contract let for moving some eight million cubic of overburden from the Buffalo & Susquehanna iron mine, the dump distances varied from 2½ to 3½ mi. To keep four shovels busy, ten 55-ton locomotives operated ten trains, each made up of ten 12-cu. yd. air dump cars.

At the Agnew iron mine, both mining and waste dumping are confined to 40 acres. The overburden is 50 to 60 ft. thick, and the pit is 135 ft. deep, and covers about 15 acres. The waste dump is 150 ft. high and covers about 25 acres. Here a 50-ton rod engine can handle only five 7-cu. yd. cars, and to reach the top of the dump four switchbacks are used. The remainder of the ore at this mine will have to be won by other methods.

At one of the Swedish magnetite mines, the shovels load ore into 12-ton side dump cars which are hauled to the scales and breaker in five-car trains by 12-ton switching locomotives. This is unusually light equipment for such service, but the haul is very short.

The Utah Copper Co. at one time used small narrow gauge dump cars on the upper benches. These were loaded with ore and hauled to chutes where the ore was dumped down to a lower level and reloaded by shovels into standard cars. The steep hillside made it difficult at first to run standard gauge equipment to the highest benches, but permitted the chute arrangement.

Where pits are served by inclined planes the transportation problem is entirely different; it becomes one of hoisting, and as such may be considered common to any system of mining. It may be mentioned in passing, that a continuous traveling chain or cable haulage system has been used in Europe and South Africa for gathering small cars over large areas. In the brown-coal pits in Germany,[1] small cars are loaded at a number of points and are moved to a common dumping point. The cars are hand trammed from drifts and crosscuts over relatively short lengths of track, which may converge at a common point or which may be parallel and terminate in a common track transverse to the drifts. A double track equipped with chain haulage (Kettenbau) serves the secondary tracks or the common point. The chain, consisting of 4 in. links, is motor driven by a sprocket-wheel and gears at the end of the run. A plate, with a V slot, projects up-

[1] YOUNG, J. G.: Former reference.

ward from the end of each car and engages with the links of the chain. The chain is supported by the cars, spaced at 20 to 40 ft. intervals, but it is carried around corners on guide sheaves. At all turns the track is so graded, that the cars gain speed on the chain and disengage themselves, the chain here being elevated to permit this. They then run by gravity around the turn and are again picked up by the sag of the chain to be hauled along the next tangent. One or more runs may be necessary in a given pit. Loads are taken out on one track and empties returned on another. The incline giving access to the pit is also served by a chain. The system is almost automatic, delivering the loads to the tipple and returning the empties with but little manual assistance except at the points where the cars are attached and removed.

## DUMPS

The disposal of oberburden removed from orebodies often presents one of the most difficult problems in open pit work. It must be transported as cheaply as possible, and dumped on areas proven to have no underlying open-cast deposits, and where it will not have to be rehandled. Available dump are as are often restricted due to topography, local ordinances, rights of way or ownership by others. Sometimes provision must be made for the segregation of certain material for probable future treatment. These factors require that dumps be built in various ways. It is advisable to secure if possible, all probable dump area requirements well in advance of operatons.

On the Mesabi[1] dumps are classified according to their position, the manner in which they are started and the trackage arrangements and operation. There are side hill or escarpment dumps, trestle dumps, slush dumps, muskeg or lake dumps and caved ground dumps.

**Hillside and Escarpment Dumps.**—A hillside or escarpment often makes an excellent starting place, as the track may be laid on the contour or edge and dumping started. The track is then thrown horizontally outward on the dumped material and the dump height is rapidly built up. The Ontario Hydro-Electric Power Commission started a dump from the edge of a river escarpment averaging 65 ft. high. The two mile track from the canal to the dump was laid on a 1 per cent. up grade, permitting

[1] Davenport, L. D., previous reference.

a 50-ton electric locomotive to haul ten 20-cu. yd. cars. Cheap dumping was thus easily and quickly acquired. The Nevada Consolidated and Utah Copper Companies have utilized side hill dumps to good advantage.

**Mesabi Dumps.**—The details of dumping operations on the Mesabi vary with each stripping job and depend on the equipment used, size of job, type of dump and other conditions.

If possible, a dump is selected on a low piece of ground sloping away from the initial dumping point, and at a good elevation to take care of the proposed over-burden. The dump is started by backing the trains out on the dump track and dumping the cars alternately on both sides of the track. Next the track is jacked-up and the operation repeated until the desired dump height is attained. Fanning-out is then started by throwing the track sideways 4 ft. at a time and thus widening the dump. The best length of this type of dump seems to be from 1200 to 1500 ft. The outer rail of the track is elevated a little to prevent cars going over in dumping. If the cars are hand-dumped the rail elevation must not be so much as to make them difficult to dump, but if the cars are loaded slightly heavy on the dumping side, this will be counter-balanced. Straight dumps are sometimes preferred to curved ones if sufficient length can so be maintained. The reason for this is that the track is easier to throw and does not bind, but the interpolation of switchpoint rails at, say 75 ft. intervals, gives sufficient flexibility to the track, so that little difficulty is really experienced.

The three methods in common use on the Mesabi are described as follows:

*First method:* A side plow or "dozer" is used to level off the dump to the height of the track for a width of about 5 feet. The track is then jacked up, lined over 3 or 4 ft. and blocked up so that it will carry the cars but not the locomotive. Dumping is then started at the end nearest the pit, and a shoulder is carried toward the further end of the dump so that the track has ballast and can support the locomotive as the shoulder is advanced. When the limit of the dump is reached, all the material that can be dumped is placed on the end length and the remaining track is filled to the limit, working back to the beginning of the dump. The dozer is then used again and the operations are repeated.

*Second method.*—A plow having a spread as wide as 30 ft. is used to level off the dump 18 in. below the track. The dump is then refilled and the spreader used until the limit of spread has been reached. The last plowing is made level with the track, which is then lined over 12 to 15 ft., and the operations can then be repeated. This method is used with 20-cu. yd. cars, heavy equipment and a track shifter.

*Third method.*—The track is made safe for both cars and locomotive, and the first train out is dumped. The dump crew then levels off the dirt, and lines the track over a foot or more if possible, along that part of the dump just filled. After the track is ballasted, the next train is dumped further along and the next section of track lined over as before, working toward the end of the dump. With a high dump, several trains may be emptied before a sufficient shoulder is formed to allow the track to be lined over. The crew levels the dirt and throws the track over between trains. With this method there is always room to dump a train.

With the first and second methods, a dump crew, consisting of a foreman and one or two men, is required on each dump both shifts. A track crew is also required, consisting of a foreman and fourteen or more men, working day shift only. Under ordinary conditions a crew of this size can handle the track work on four dumps. With the third method a foreman and six men on each shift can handle all the work required for one dump. Each dump is provided with a shanty 6 ft. × 8 ft. for sheltering the dump crews. There is also a 16 ft. × 16 ft. shelter-house centrally placed within easy reach of all the dumps, where the track crews can eat and spend their lunch hour. At night stripping dumps are lighted by kerosene or gasoline torches, powerful acetylene lamps of the portable type or electric lights when practicable.

**Trestle Dumps.**—Trestle dumps are used in many cases on the Mesabi, where the topography is flat. They are somewhat expensive to start and may be troublesome. These may be constructed so that the unfilled trestle work carries only the empty cars, or the entire train. The former are more usual, and the Mesabi practice is to build these trestles of round timber with 3 or 4 post bents, spaced on 16 ft. centres, and from 16 to 25 ft. high. The trestle legs are legs from 8 to 12 in. in diameter, stringers 10 to 14 in., braces 3 to 5 in., and 8 ft. railroad ties are used for caps. The legs are set on cross-sills, are given a batter of 2½ in. per ft., and are cross-braced and longitudinally braced.

The stringers are placed above the caps and are spiked with drift bolts. Bents less than 4 ft. high are not built, blocking or cribs of old ties being used for the transition from earth to trestle. Filling is started by pushing cars out on the trestle ahead of the locomotive and dumping several on one side one at a time. As they are dumped the empties are pushed out on the trestle and others are dumped on the opposite side. Heavy dumping on one side will cause undue strain on the legs. As soon as the trestle has been filled for its entire length, the track is shifted to one side and dumping continued so that the dump is widened. The trestle is usually so located that the dump can be fanned out from both sides. If sufficient room is available, the trestle is continued far enough to start several dumps at intervals along its length. It is considered good practice to keep the edge of the dump straight, as this facilitates the throwing of the track. When the limiting horizontal distance has been reached, it is common practice to throw the track back, raise it up and make a new level by working back over the area already filled. Another method is to build a second trestle on the first dump and then to start a second deck in the same manner as the first. The cost of such dumps has been given at from \$2.50 to \$5.00 per linear foot.

**Slush Dumps.**—A slush dump may be made by the use of a trestle, strong enough to carry the loaded train, and fitted with an apron 6 to 8 ft. wide on the dump side. The material is dumped from the cars onto the apron, and from here it is washed down the bank by water. To do this a 3 or 4 in. pipe line, perforated with 1/4 in. to 1/2 in. holes at short intervals, is laid along the upper edge of the apron against the ends of the ties. In some cases 2 in. hoses have been used for sluicing. At one property, on the shore of a lake, a permanent trestle was built at an elevation of about 100 ft. above the lake and capacity for 150,000 to 200,000 cu. yds. was provided without shifting tracks.

The advantages of the method is that considerable material can be moved without shifting track, and hence no track crew is required, and the dumping space is always ready to receive the cars. Although the pipe may be laid in sections so that water will not be wasted, considerable is required. During freezing weather other dumping locations must be used.

**Swamp and Lake Dumps.**—Muskeg swamps are to be avoided as dump sites. The dumps often slide or settle suddenly in spots leaving the track hanging or taking the track and train over the edge. In attempting to fill a dump trestle across a muskeg swamp, the surface of the swamp is often bulged up as high as the settled dump, making dumping impossible, but by building the trestle along the edge of the swamp, the muskeg can sometimes be forced ahead of the dump. When used intermittently they are considered less dangerous.

With dumps that are fanned out in ponds or lakes there is considerable danger of sudden settling along the edge. The action of the water, agitated by the material being dumped, so undercuts the face of the dump that it suddenly sloughs off. Water as shallow as 5 ft. has been known to cause this trouble which may be aggravated by the sliding or settling of ooze and mud.

**Dumps on Caved Ground.**—On the Mesabi, caved ground, above underground workings, is sometimes used for dumps. It is decribed as follows: The additional weight of the dump does not greatly affect the weight on the underground timber, and filling the caves prevents surface water from collecting and breaking through into the lower workings. The expense of pumping water from the caves, frequently necessary in older underground mines, is eliminated by this method of filling the sunken areas. Blasting underground rooms causes the dump to settle, but advance information to the dump foreman on this permits him to plan his work accordingly. Dumps of this kind are usually started from a trestle.

**Copper Mine Dumps.**—At the great western porphyry copper mines, the building of dumps is largely governed by the terrain. The country is hilly or mountainous for the most part, and advantage is taken of slopes, gulches and side hills. Overburden is taken from a much greater range of altitude than on the Mesabi, and this requires that dumps be located at different elevations, otherwise grades would be excessive for reasonable distances. Overburden from the different benches is consigned to the dump sites best suited to give low costs. Such ground is often limited and consequently the areas and heights and trackage arrangement of the dumps vary widely. Very little trestle work is used in starting these. In some cases the same dump sites have been used to take care of overburden from different bench levels by carrying them in two or more decks; material from the lower benches being used for the lower deck and that from upper benches for the upper deck. The tracks on the dumps are usually on an ascending grade of at least 1 per cent. but may be higher. When the position and area is suitable the track may form a horse shoe so that the loaded trains come in, dump and continue on around to the return track. This, of course, turns the train around, so that if the relative position of cars and locomotive is to be kept the same at the shovel, the train must run in on a wye spur elsewhere before returning to the shovel. If the dump track is dead ended, as is usual, the end rails are elevated and supported on a cribbing of railroad ties. Several ties are lashed over the end rails to act as a wheel stop in case a car should be backed in too far. This end cribbing is moved out as the dump is fanned out and in so doing the length of the track is slightly shortened at each move on account of the dump slope. The use of switch point rails at frequent intervals permits the track to be flexed out over a long arc so that the end cribbing need not be moved too frequently. Every effort should be made to avoid train congestion or delays on the dumps, and for this reason it is often economical to double-track the lines to a big dump as well as have several separate independent dumping points.

**Height of Dumps.**—The question as to what is the best height to carry dumps depends on such things as the character of the material, the weather conditions, the weight of equipment and the type of dump.

The Mesabi waste dumps, often covering from 80 to 200 acres or more, range from 25 to 60 ft. high. At Hibbing the surrounding country is quite flat and the waste dumps are of more or less uniform depth. Here it is considered that the most economical height is not exceeding 50 feet, because when they exceed this, the dirt has a tendency to collect on the slopes and form shoulders. In time these become quite heavy, break loose and slide to the bottom, and in doing so, it not infrequently happens, that the dump breaks back of the edge 20 to 30 ft. and takes the track out with it. It is also found that high dumps settle much more than lower ones, and it requires a larger amount of work to keep up the track. When it is necessary to build higher than 40 or 50 ft., a second 30 or 40 ft. dump is put on top of the first after it has thoroughly settled. Higher dumps in level country may also require heavier adverse grades and increase the transportation cost. Under average conditions, dumps of this material from 25 to 40 ft. high seem to give the best results, but from necessity some have been carried in part up to 87 ft.

The dumps at the porphyry copper mines are extremely variable in height because the terrain is very steep and irregular and because of the necessity of utilizing to the fullest extent all suitable ground. In Nevada the dumps range from 25 to 150 ft. high and give little trouble. The material is altered, decomposed porphyry and does not have as much tendency to shelve off or settle as the Mesabi overburden of glacial drift and gravel. Also the dumps are not subjected to heavy rains. The type of dump cars used drops the material from a greater elevation so that it is more firmly packed. However, if ideal conditions could be chosen, perhaps these dumps would be carried about 75 ft. high. There would be little danger of shelving off or settling with this height, and at the same time the amount of track shifting would be kept fairly low. In Utah the material is somewhat similar, and the dumps have been formed wherever there was available area. Cañons and gulches were filled up without regard to height of dump, as the all important thing was to find a place to put the waste in this difficult mountainous location.

In the anthracite region dumps are made of all heights and sizes, though there is said to be less maintenance cost with heights of about 25 ft. Dumps of greater height settle and slip easily, especially in wet weather.

Hard, rocky material deposited in an arid climate can evidently be carried to heights of 100 feet without inconvenience, and will require less track shifting; while soft earthy material, or sand, clay and gravel deposited in a wet climate, should evidently be kept down to heights of 25 to 40 ft. because the saving in track shifting in using greater heights will be more than offset by settling and bank troubles.

**Hydraulicking.**—Hydraulicking has in some instances been used to dispose of waste material and to build dumps. In one of the smaller brown-coal pits in Germany, the upper portion of the overburden, consisting of sand and soil, was stripped by hydraulicking and back-filled into the lower part of the pit. Retaining dams were put in on the toe of the spoil slopes, and the sluices discharged back of them. The remainder of the overburden, however, consisted of moderately compact shales and clays, and was removed in two benches by undercuttings, caving and shoveling by hand into cars.

Hydraulicking has been considered as a possible system for stripping in the anthracite region, but scarcity of water and of areas on which to deposit and settle the refuse render the method impractical for most sections of the country. It has been suggested, however, that where the refuse could be flushed into mine openings to support the surface, a double value would be obtained. This has been tried with fair success.

In stripping coal in the Danville district of Illinois the method has been successfully used and given low costs. Here the superficial glacial deposit is hydraulicked by two nozzles 1¼ in. and 1½ in. in diameter connected to the same 4 in. main delivery hose. Each nozzle is operated by one man and delivers about 1000 gal. per min. under pressure of 135 lb. per sq. in. at the pump. The water supply is furnished by a duplex steam pump, of 2000 gal. capacity, installed at a nearby river.

The spoil is washed into an adjoining valley or into old excavations. The work is not carried on in freezing weather, but sufficient stripping is accomplished in the summer to permit the mining of shale for brick the entire year. The amount of soil moved per 8-hr. shift by this method is said to vary from 100 cu. yd. in tight ground to 2000 cu. yd. in loose ground.

For hydraulicking to be successful there must be an ample supply of water, suitable terrain for the disposal of the spoil and reasonably soft or loose material to work. The operations are confined to the open season. Under these conditions very low costs per cu. yd. of overburden removed can be attained.

## CHAPTER VI

## THE DETERMINATION OF A POWER SHOVEL MINE

### PRELIMINARY DATA REQUIRED

**Maps and Sections.**—It will be assumed that an orebody has been discovered and that the development work on it has progressed far enough to delineate its physical characteristics. The next step then is the selection of that method of mining, which, applied to the deposit, will result in yielding the highest economic returns. Such a selection can be made only after careful consideration of the particular conditions surrounding the problem by men of experienced engineering training, sound business judgment and an honest understanding of the conservation of both labor and natural resources. Given an orebody with sufficient development work to show its probable shape, size, texture, structure, unit value, boundaries and relationship to the enclosing formation, the solution of the mining problem may then be undertaken.

The occurrence and character of the deposit may be such that an experienced engineer will be readily able to correctly class it as a deposit requiring extraction by some underground method or extraction by open-pit work. On the other hand it may be of such a nature as to suggest investigation by both methods, and it is such a problem that will now be discussed.

The first work to be done consists in the preparation of accurate maps and sections of the deposit. These will be based on underground workings, drill holes and the general geology and topography. The maps should show the contours of the surface and of the top and bottom of the orebody. The sections (best made on cross-section paper) should be taken, if possible, at regular intervals through the orebody—say generally not closer than 50 ft. nor farther apart than 300 ft., with 100 ft., as a good average. Two sets of vertical sections, one set at right angles to the other, are desirable and may best be taken parallel to the two axes of the orebody. These sections will show graphically the actual and relative thickness of both ore and overburden, and may also

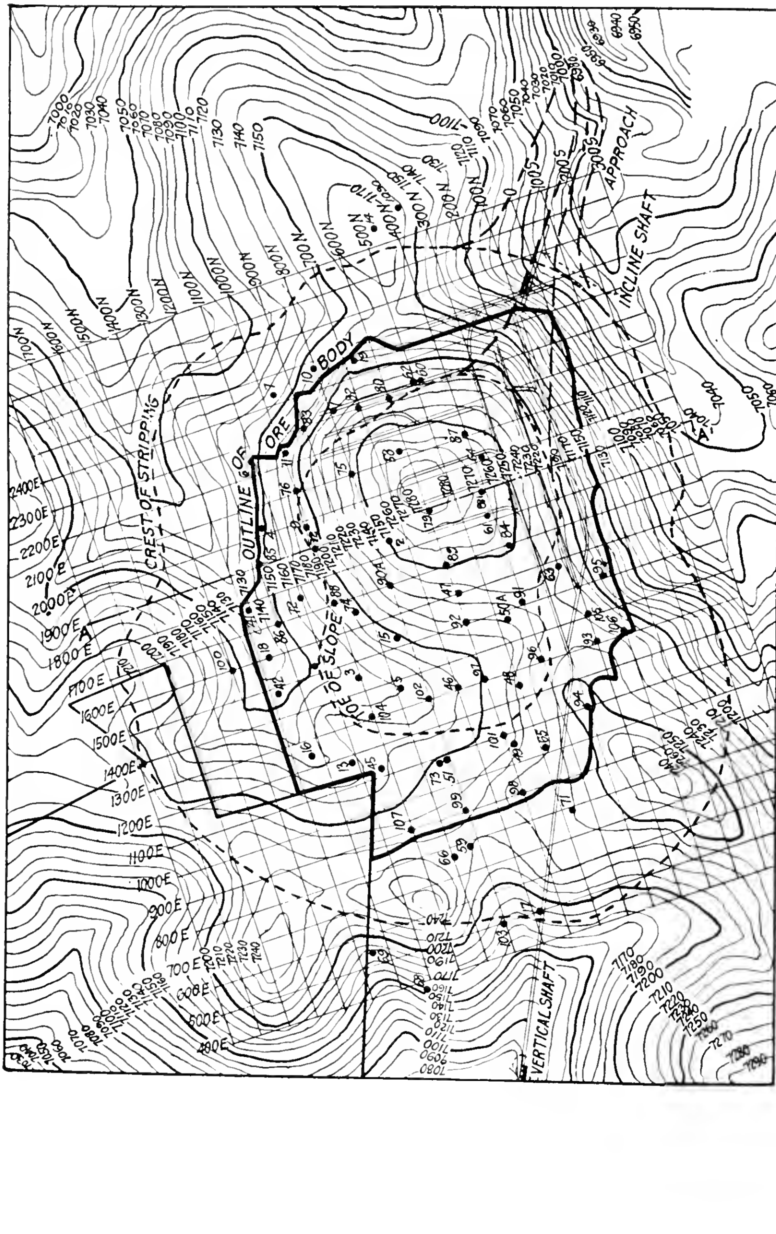



Fig. 38.—Topographic map over ore-body.

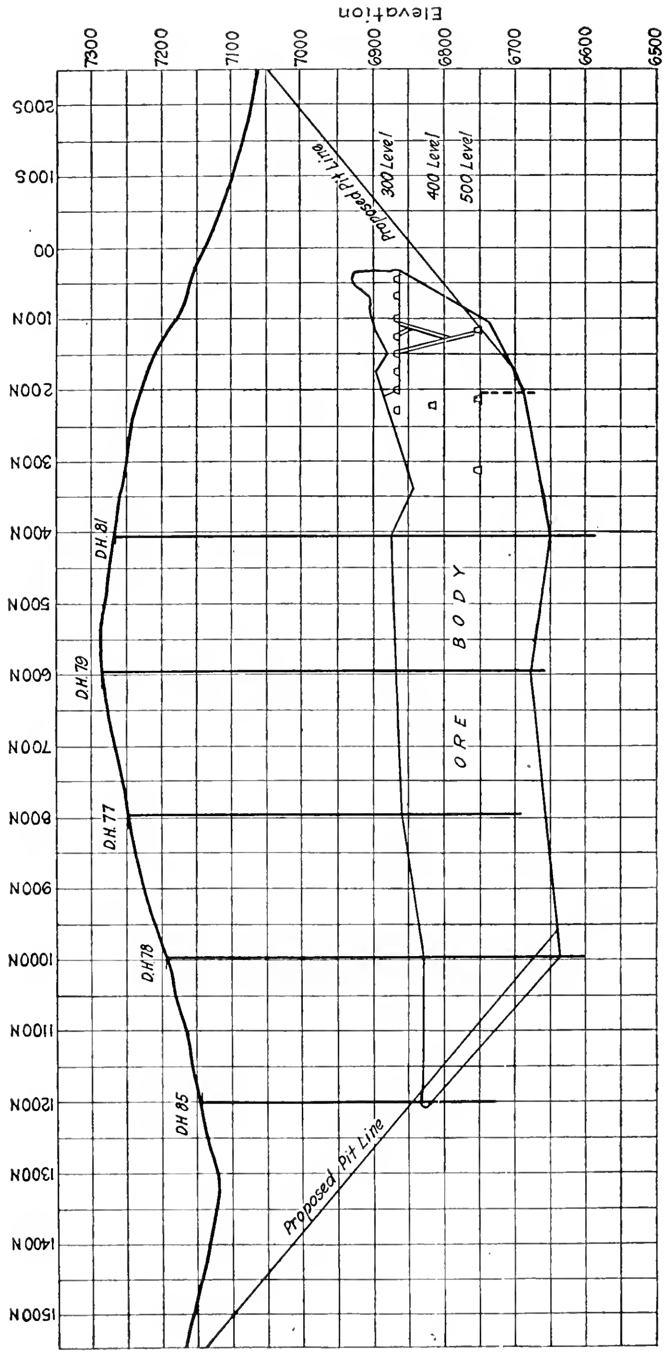



Fig. 39.—Vertical section through ore-body.

be used for indicating slope lines and shovel benches and banks of a preliminary nature. It may also be desirable to prepare sections through the deposit horizontally at definite elevations on which will be shown the outline of the ore intercept. These are especially useful in planning definite extraction levels, shovel benches or pit-bottom elevations.

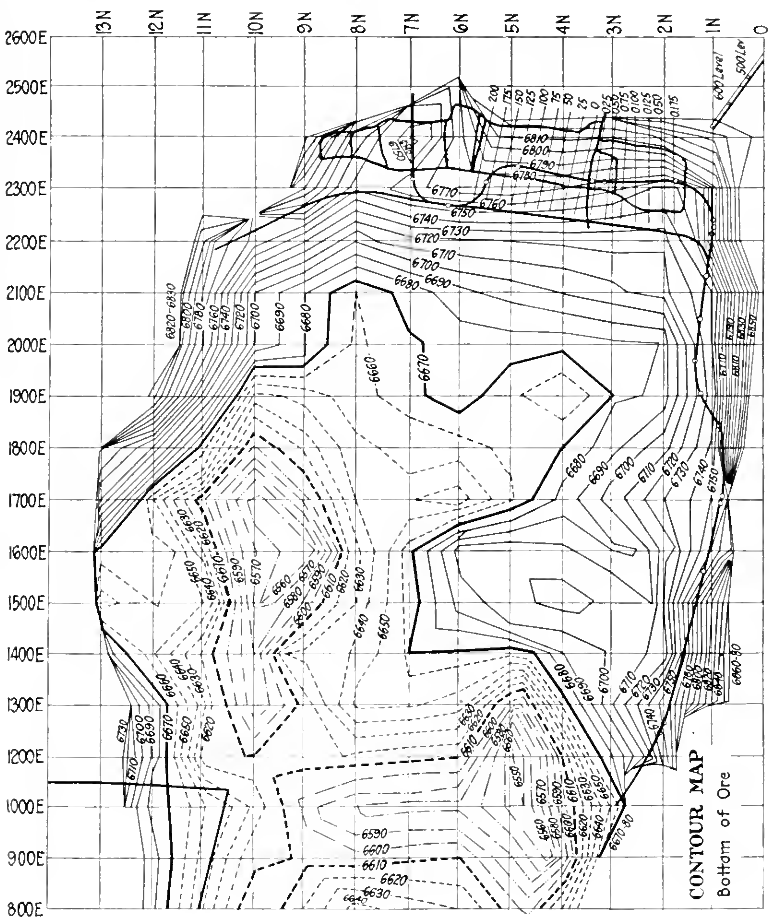



FIG. 40.—Contour map; bottom of ore-body.

Let us illustrate this work by an example in which Fig. 38 is a typical topographic map of the surface covering the orebody. On it will be found surface contours, location of all drill holes, projection of any important underground workings, location of cross-sections, outline of the developed orebody, possible outline of shovel pit bottom and outline of shovel pit edge. These last two outlines would in practice be straightened out to allow for practical pit working conditions. This map also shows where the orebody would probably first be attacked for shovel work

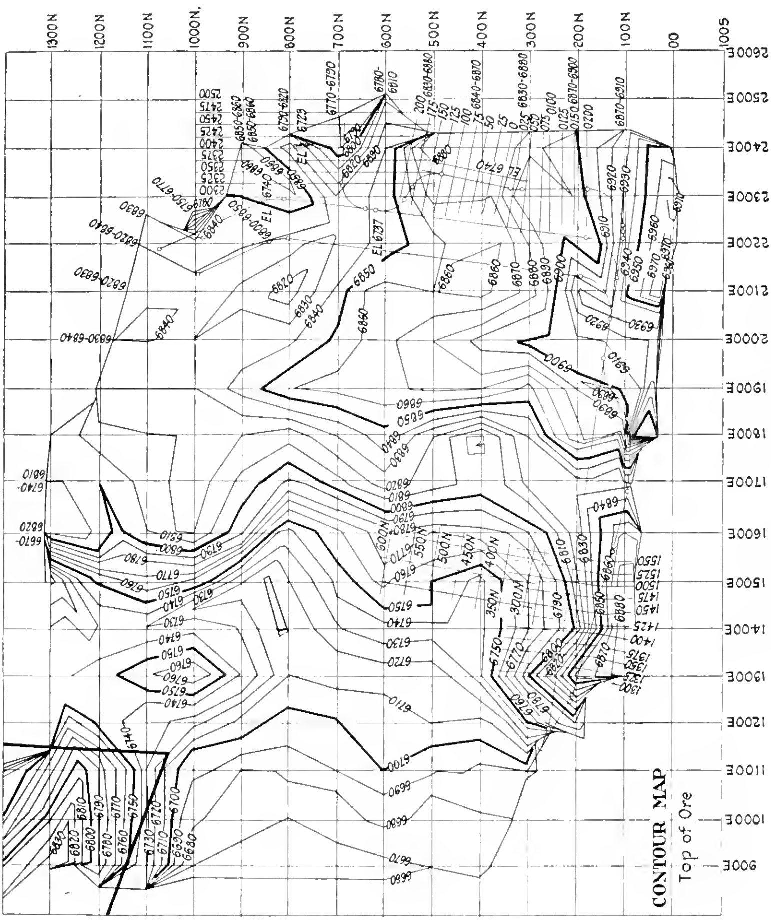



FIG. 41.—Contour map; top of ore-body.

because the overburden is here thinnest and the approach is simplest.

Fig. 39 is a typical vertical section, taken at $A$-$A'$ on Fig. 38, of the orebody and shows the surface, the drill holes and underground workings, the boundaries of the determined orebody and the proposed average pit slope lines.

Fig. 40 is a map showing the contour of the bottom of the orebody, and is useful for either pit or underground work. Contours of the top of the orebody (Fig. 41) may by tracings be superimposed to advantage.

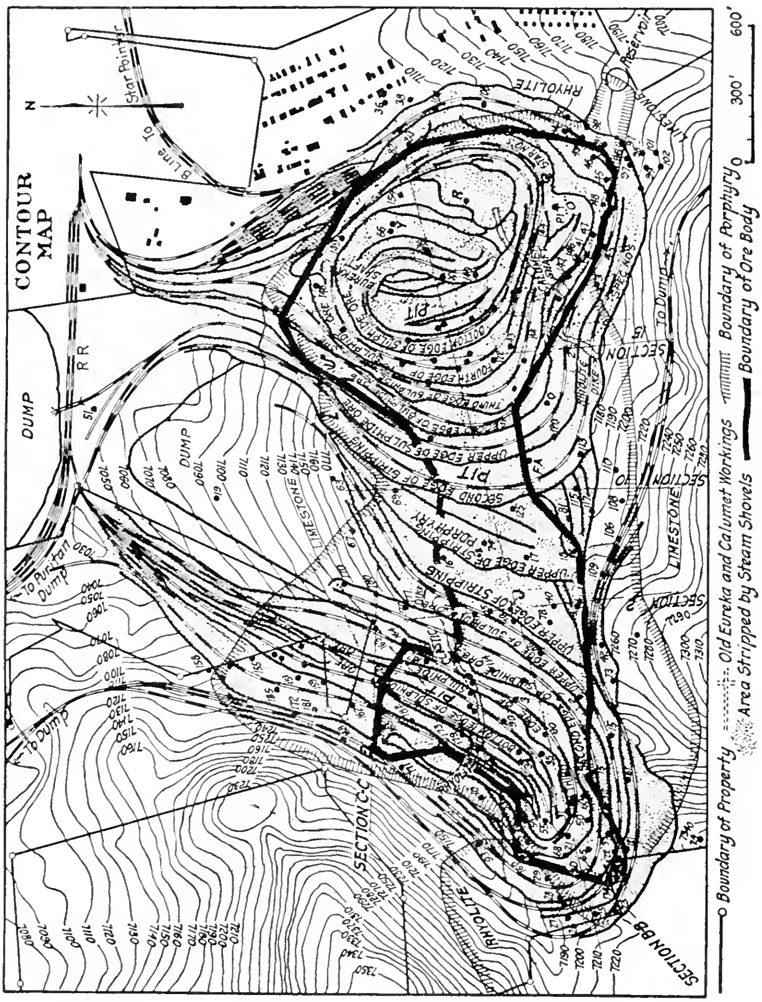



FIG. 42.

To further illustrate the use of such maps, Figs. 42, 43 and 44 are here given showing some actual well known pit operations.

Fig. 42[1] is a plan of the pits of the Nevada Consolidated Copper Company, of Ely, Nevada.

[1] Eighth Annual Report, Nevada Consolidated Copper Company.

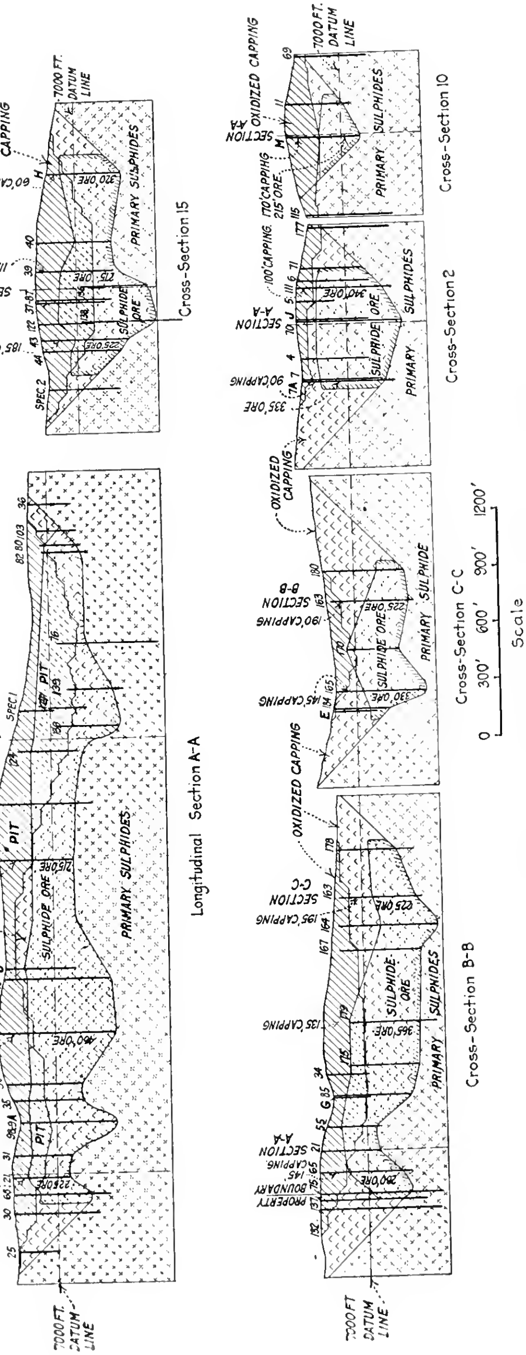



Fig. 43.—Sections through Nevada Con. Copper Co.'s pits

Fig. 43 is a series of vertical sections taken through the pit workings and orebody. These show clearly the advanced state of the work as it appeared on January 1, 1915, some seven years after the beginning of operations.

Fig. 44 is a plan of the pit, dumps and general surface plant of the Commodore mine of Virginia, Minnesota.[1] The operations here were made unusually severe on account of the limited area (40 acres), depth and shape of the orebody, and yard facilities. Seven hundred thousand tons of ore had been mined through underground operations when it was decided to change to open pit mining. To complete the stripping it was necessary to dump 800,000 cu. yd. on the Commodore 40 acres, over 20 acres of which composed the stripping area of the open pit. The waste dump finally reached a height of 87 ft. and, as the top of the deepest ore stripped was 114 ft. below the dump bottom, some of the overburden was elevated 201 ft. with the trackage shown. From shovel to dump there were four switch-backs on a 5 per cent. grade.

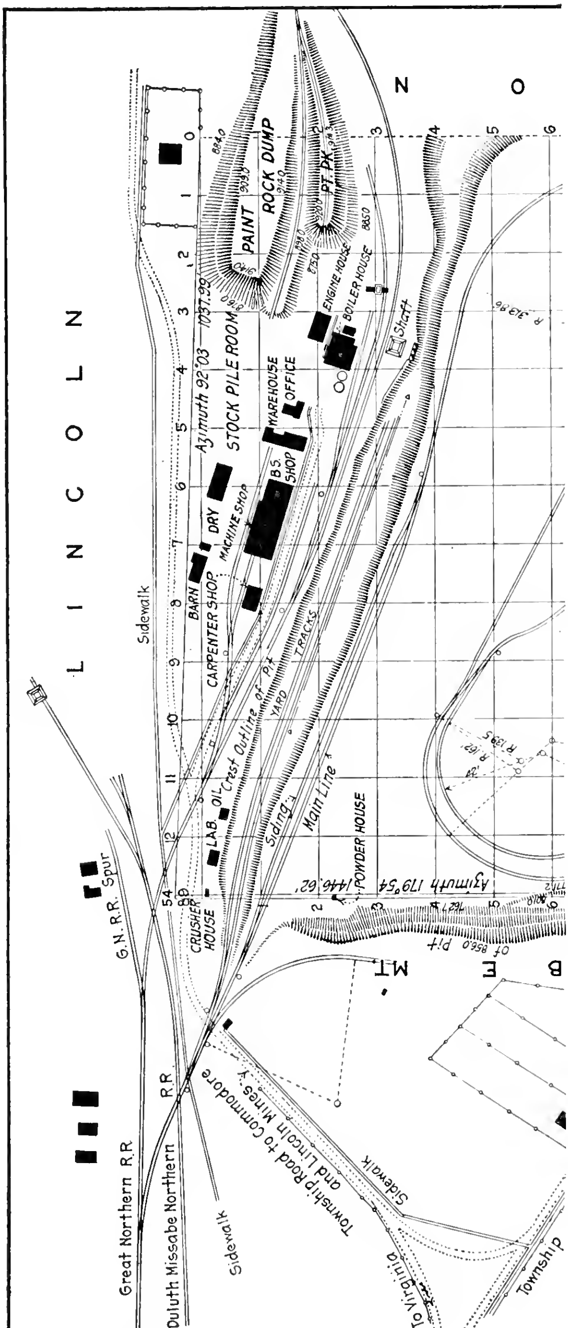
With the aid of such plans and sections as illustrated by Figs. 38, 39 and 40, computations may be made which will show the ratio of overburden necessary to mine the ore. A convenient form of keeping the calculations made up from these cross-sections is given on page 190.

Such maps are also used to show the work necessary for proper approaches and yards, the capacities of possible dump areas and

[1] Bayliss, M'Neil & Lutes—Mining Methods on the Messabi Range. T. L. S. M. I. 18th Annual Meeting.

TABULATION OF MAIN OREBODY

| Sec. no. | Distance between sections feet | Total sq. ft. of ore | Average per cent. Cu | Total tons ore | Average per cent. Cu | Tons ore × Cu per cent. | Sq. ft. unprofitable | Tons unprofitable | Tons remaining | No. samples |
|---|---|---|---|---|---|---|---|---|---|---|
| 1. N.S. (Complete series of sections, say, North and South) | | | | | | | | | | |
| Totals | | | | | | | | | | |


LINCOLN
MT.
Great Northern R.R.
Duluth Missabe Northern R.R.
G.N. R.R. Spur
Sidewalk
PAINT ROCK DUMP
PT. PK.
STOCK PILE ROOM
Azimuth 92°03' 1037.99
WAREHOUSE
OFFICE
ENGINE HOUSE
BOILER HOUSE
Shaft
B.S. SHOP
DRY
MACHINE SHOP
BARN
CARPENTER SHOP
LAB.
OIL
CRUSHER HOUSE
Crest Outline of Pit
YARD
TRACKS
Siding
Main Line
POWDER HOUSE
Azimuth 179°54' 1446.62'
Township Road to Commodore and Lincoln Mines
To Virginia
Township

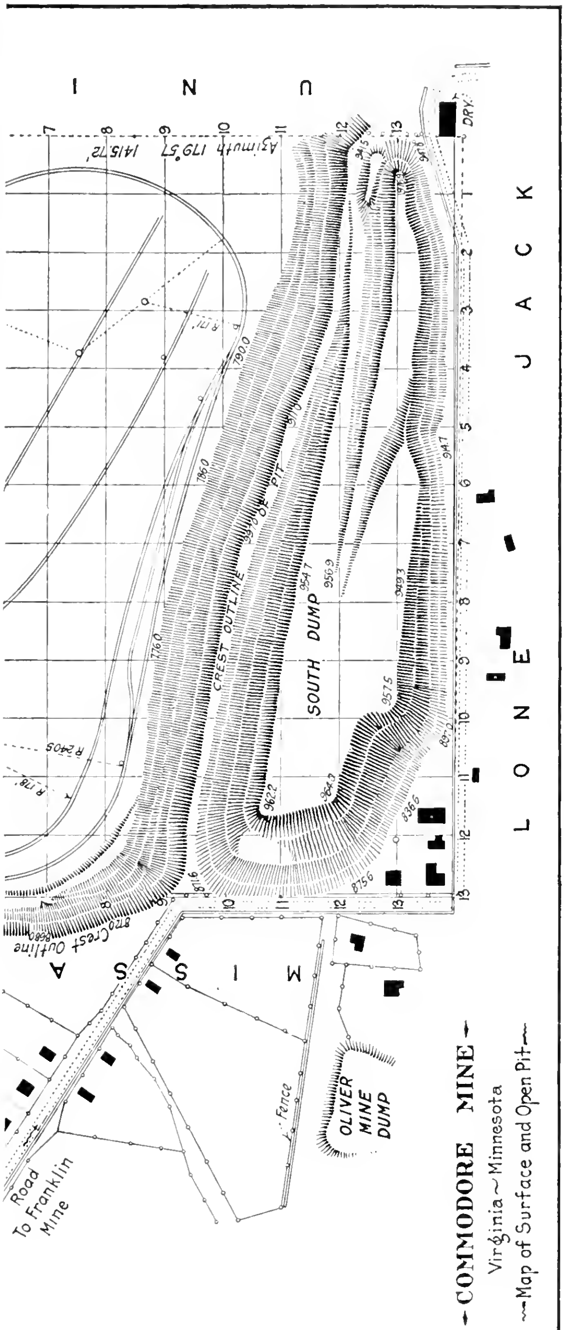



FIG. 44.—Plan of workings, Commodore mine, Minn.

Tabulation of Overburden

| Sec. no. | Distance between sections feet | Total sq. ft. waste | Cubic feet waste | Cubic yards waste | Ratio waste cu. yd. to 1 ton ore | Remarks |
|---|---|---|---|---|---|---|
| 1 N.S.<br>Totals | (Comp | lete se | ries of | sectio | ns, say, Nor | th and South) |
| E<br><br>Totals | (Comp<br>easter | lete se<br>n N–S | ries of<br>sectio | sectio<br>n) | ns at right | angles to the extreme |
| W<br><br>Totals | (compl<br>weste | ete se<br>rn N– | ries of<br>S secti | sectio<br>on) | ns at right | angles to the extreme |
| Total Ends | | | | | | |
| Corners<br>NE<br>SE<br>NW<br>SW<br>Total Corners | <br>(four,<br>care of | <br>three-<br>by th | <br>quarte<br>e abo | <br>r secti<br>ve sec | <br>ons of trunc<br>tions) | <br>ated cones not taken |
| Grand Totals | | | | | | |

such other data as may be required in case open-pit work is to be considered in detail. They will be used to estimate the amount of development work required in order to extract a given thickness and area of the deposit and to indicate where this work should be done, the surface area likely to be affected by caving-mining operations and such other data as may be required in case underground methods are to be considered in detail. In either class of mining, they are indispensable to accurate work.

**Local Costs and Conditions.**—The next study is that of the probable operating cost to be obtained, employing such methods of mining as, under the local conditions, suggest themselves as adaptable. The computing of results which may later be fulfilled in practice, is largely based on past performance, experience, and proper allowance for the effect that such local conditions as climate, topography, abundance and efficiency of labor, cost of supplies, nearness to market, transportation facilities, adjoining holdings and any other local factors that may have a bearing on the results.

**Initial Time and Capital Required.**—Often the initial period of time, and very generally the initial capital, required to bring a mine to a state of self-support, are factors of great weight in determining the method of attack.

**Other General Considerations.**—In case the product is subject to wide fluctuations in selling price, then flexibility of output without sacrifice of efficiency is a very desirable feature, and should be given weight in the decision. Again, it may be that one method of mining will, with little or no additional expenditure, make available for future use a large amount of contiguous material of low present value but possibly great future value, which may be rendered probable by new and improved methods of treatment or other factors reasonable to expect. As an example of this, mention may be made of the low-grade oxidized overburden and ore fringes which are now being removed from the large steam-shovel deposits of copper ore of this country. Such material may be put on dumps from which it is easily and cheaply available for future improved methods of treatment.

Another class of material from these mines, which may later be treated at a profit, is the deeper low-grade primary sulphides. This may at first be classed as waste, but may usually be left so exposed after open pit mining as to be attacked cheaply after the richer material has been removed; whereas underground mining usually leaves it more difficult and often impossible to economically reclaim.

Where two methods of mining a deposit are open for adoption and both seemed to indicate approximately equal profits, that one should generally be adopted which would yield the product of greatest gross value. In other words it would be conservation of resources without unfair sacrifice of net returns.

A factor of great importance in mining many deposits is the physical character of the ore. In the case of lenses of massive sulphide ores, great losses have been incurred in the past because they caught fire through heat generated by their own crushing. If such a contingency seems at all probable, it is evident that many of the factors mentioned may be of much less importance than the elimination of the fire risk. Again, if for some reason it be essential to preserve the ore in a very clean state, then a method of extraction must be selected which will insure this, even though in other ways it be less desirable.

**Sound Conclusion on Method to be Recommended.**—It is apparent that no fixed rules can be laid down for the selection of a mining method. This must be made only after careful consideration of all of the individual peculiarities and conditions of each deposit. The wonderfully rapid, cheap and efficient work done by the steam-shovel in recent years has rightly attracted much enthusiastic commendation and attention. This has not infrequently led to the expression of hasty opinion that properties, which should really be worked by underground mining, are "steam-shovel propositions." Upon full and careful investigation, the classification of some of these problems will be properly changed and serious initial mistakes will be avoided.

The general statement may be made that power-shovel methods are rarely justified except where the amount of excavation to be done is relatively large. An approximate idea of the amount of work required to warrant their employment may be had by estimating the total cost, including all necessary plant equipment, of removing a given yardage with power-shovels (or some other similar means), or of removing a much smaller yardage by some underground method. It is more apparent each year that machine-work is becoming cheaper than hand-work whenever the amount of work to be done is large enough to warrant the necessary initial capital outlay.

The operating cost of power-shovels is subject to considerable variation dependent on local conditions. Operating efficiency of the shovels is of prime importance and depends largely on a thorough coördination of the whole working scheme, as well as upon the skill and coöperation of the shovel-crews. In arriving at an estimate of the unit cost of excavating in a new problem, the local conditions likely to affect costs will be studied. In case the new property is located in a district where power-shovel mining has been going on for some time, the engineer will have considerable advantage in making an estimate. He will ascertain what costs are being obtained under "present good practice," and then, making allowance for structural variations of the deposit, grades, dump-sites, future improvement and other such conditions, will calculate the probable cost for the deposit under consideration. In such a locality it will usually be easier to get started and to secure efficient crews. On the other hand, if the property be located in a foreign country and far from any work of this class, the solution of the cost problem becomes more uncertain. The human factor in the complete organization must be more fully considered and the local conditions will all have to be worked out. Under such conditions a larger "factor of safety" should be employed in estimating the cost in order to cover unforeseen contingencies. Cost estimates seem to have a provoking tendency of being too low, rather than too high, when put to the test.

## ILLUSTRATIVE EXAMPLE OF THE SOLUTION OF A PROBLEM

**Hypothesis.**—As an example illustrating the solution of such a problem, let us assume that we have an orebody, such as shown by Figs. 38, 39 and 40, of the following characteristics: first, it has been well developed by churn-drill holes and underground workings and has been carefully mapped and sectioned; second, this work shows it to be a large flat-lying lenticular deposit, covered by a leached overburden; third, the ore occurs as disseminated copper and iron sulphides in a partly decomposed porphyry; fourth, it is of low grade and, in order to show a profit, will require a cheap method of extraction; fifth, the formation enclosing the orebody is of similar character to the ore except that the overburden is harder and more silicious while the sides and bottom are less decomposed and firmer; sixth, the few underground workings show that they require close timbering to be kept open for any length of time; seventh, the ground, though very heavy and inclined to squeeze, is fairly dry and the indications are that the ore would cave readily if properly undercut.

**Engineering Calculations.**—Estimates made from the plans and cross sections are as follows:

| | |
|---|---|
| Length of long axis | 1800 feet |
| Length of short axis | 1100 feet |
| Average thickness of *direct* overburden | 245 feet |
| Shallowest thickness of *direct* overburden | 70 feet |
| Greatest thickness of *direct* overburden | 300 feet |
| Average thickness of main orebody | 96 feet |
| Least thickness of main orebody | 40 feet |
| Greatest thickness of main orebody | 115 feet |
| Horizontal area of main orebody | 30 acres |

Assuming that practically all of the valuable metal in the ore is copper, it is decided, in estimating the ore tonnage, to draw the line between ore and waste at assay returns carrying less than 1.25 per cent. Cu. However, inasmuch as some material carrying less than 1.25 per cent. Cu, but not less than 1 per cent. Cu., would have to be removed in the event that shovel mining were employed, and furthermore as such a method would leave exposed an additional underlying tonnage of material which could be considered as ore if no stripping expense were charged against it, these second and third tonnages are also computed. The reason for drawing the line between ore and waste at assays under 1.25 per cent. Cu. for the main orebody, and at assays under 1 per cent. Cu. for the second and third bodies will be explained later.

In calculating the overburden yardage, the first thing required is the preparation of a preliminary working pit layout. Considering all development work, a line is drawn to include and circumscribe the orebody it is proposed to mine with shovels. This boundary is called the "crest of ore" in the proposed pit. A second line is drawn say about 20 ft. outside of this, and called the "toe of stripping." Study of the ground, and exposures in some railroad cuts in similar material, indicate that slopes at 45° should stand well for long periods. It is assumed that banks on this slope could be worked to a final height of 100 ft., provided that between such banks, berms of 30 ft. be left to catch any slides or loose material and prevent them from endangering the workmen and equipment working lower down in the pit. It will be assumed that in the earlier stages of the work benches would be carried only 50 ft. in height, but as the pit limits are gradually reached, two 50-ft. banks would be run together to form one 100-ft. bank. In this way steeper average slopes from the "toe of stripping" to the "crest of stripping" (viz., the final pit edge) can be carried, with an attendant considerable reduction in overburden. The sections thus prepared show an average slope inclination of 40° to the horizontal, and are taken the same for both ore and overburden.

These final benches are drawn on the ore-sections, while for the ends, sections are taken at right angles to the first and last of the ore-sections. The corners between the two sets of sections are computed as being ¾ sections of truncated inverted cones. From the completed sections, the outline of the edge of the final pit is then drawn on the topographic plan. This crest of stripping line is next adjusted to give a practical working pit, allowing for practical track-grades and curvature. The sections are then modified to conform with the final crest of stripping line. A study of the sections gives a good idea of the relative ratio of overburden to ore, and incidentally indicates where more development work might be done to advantage. In this case the study does not indicate that probable tonnage additions will materially alter the relationship of ore to overburden. The area of the final pit-plan is found to cover 65 acres. The average thickness of overburden over this area is 200 ft.

A summary of the tabulated results taken from the sections is as follows:[1]

(1) Ore assaying 1.25 per cent. Cu. or better.................. 12,000,000 tons; Avge. 2.0 per cent. Cu.
(2) Ore assaying 1.00 per cent. to 1.25 per cent. Cu.............. 600,000 tons; Avge. 1.15 per cent. Cu.
(3) Ore assaying 1.00 per cent. to 1.25 per cent. Cu.............. 2,000,000 tons; Avge. 1.10 per cent. Cu.
(4) Overburden required to be removed to mine (1) = 24,000,000 cu. yd.

Note: (1) is concentrating sulphide ore and is designated as the main orebody.

(2) is made up largely of material carrying copper oxides and carbonates, and overlies (1). It is not considered amenable to hydraulic concentration, though it is readily leached.

(3) is the low-grade sulphide material underlying (1). The assays show it to be richer than (1) in iron sulphides. While it is amenable to hydraulic concentration, both the concentration ratio and the copper content of the concentrates are low; consequently the treatment cost per pound of copper produced from such material would be somewhat higher than for the main orebody.

[1] For tonnage estimates of this character, the writer prefers accurate cross section methods to others. They are graphic, more comprehensive, fit in well with the actual pit plans, leave a record clearly understood and easily added to as desired, and are of great aid in studying the trend, form and general relationship of the ore deposit. If the work be accurately plotted and if the volumes be calculated on the theory of the mean proportional, or by the prismoidal formula, the results will be quite accurate. As a check on such estimates, the triangular prism method may be employed, and will be found quite precise.

Considerable has been written on methods and general principles of estimating the tonnage and value of orebodies, but it is not the intention here to recapitulate them. It is assumed that the reader has a fair knowledge of such special work. It cannot be stated too emphatically however

Because of the character of the material classed under (2) and (3) its extraction would not be profitable by underground mining. In the event of mining (1) with power-shovels however (2) would have to be mined as a part of the overburden, while (3) would be left exposed after the removal of (1). In this event it could be considered that neither (2) nor (3) be required to carry any stripping expense, all of this having fallen on (1). Furthermore (2) would not be charged with any mining expense (except for a possible later rehandling charge from dumps) since it would be carried as overburden removal from (1). A mining charge would have to be made against (3) however.

**Consideration of Shovel Methods.**—It is assumed, after a careful study of the cost of shovel work in this and other districts and after making due allowance for the local conditions, that this material (ore and overburden) can be excavated for an average cost of 32 cents per cu. yd. in place.

It is desired to bring the property up to a production of 5,000 tons per day at a reasonably early date. To insure a steady output of this amount it is estimated that in using shovels, ore must be exposed for a length of about 1,000 ft. and for a width of 100 ft. Study of the plans and sections indicates that this can be best accomplished by beginning to strip at the east end where the overburden is shallowest. The required exposure can here be made by cutting out a series of from three to four benches of overburden having heights of 50 ft. and leaving 30-ft. berms between them. Five million cubic yards would be removed in this initial work.

The actual opening up of a shovel mine involves other civil and mechanical engineering problems, as applied to each individual case, and they will next be taken up, as follows:

1. Proportion of overburden to ore, taking account of the depth and thickness of both, and the volume of overburden in the slopes and in any irregularities.

2. Location of the most desirable approach, or approaches, considering both overburden and ore removal, and the volume of material to be removed in this work.

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that success in mining open-pit deposits depends on a thorough examination showing conclusively that there exists a sufficient tonnage of ore to warrant the project. The more complete and precise this preliminary information is the more correct will be the conclusions and the better will be the working-plans evolved.

3. Location and cost of most desirable assembly yard.
4. Location and layout of overburden dumps, giving consideration to length of haul, grades, capacity and differences in elevation between which the material must be transported.
5. The drainage of the pit and ore, if important.
6. System of trackage and haulage to be employed.
7. Selection of the mechanical equipment to be employed.

These items have all been discussed in the preceding pages so that time and cost studies will conclude the present analysis.

Three means of approach were studied. The first had the disadvantage of long heavy grades; the second was unfavorable to a well located assembly yard; the third had an expensive 200-ft. cut at the pit entrance, but the grades were easy, and connections to the proposed assembly yard were convenient. It does not seem likely that this third approach would have to be materially altered for late operations, as the bottom of the orebody could be reached by means of spirals or switchbacks requiring about 10,000 ft. of trackage with three per cent. adverse grades. This approach requires the removal of 1,000,000 cu. yd. Although haulage of material out of the pit over the proposed spiral would cost considerable, the topography is such that to materially lower the approach involves the removal of so much additional yardage that the expense would not be justified by the somewhat lower transportation costs, and furthermore, initial production would be considerably delayed.

Incidentally an approach by means of a tunnel was considered but this was found to be much more costly and slower.

The third approach having been definitely decided on, the assembly yard was planned at its lower end with four tracks, each 1,000 ft. long and able to accommodate a full train of 22 standard 50-ton steel cars.

Study of the topography of the nearby country indicates that preliminary dumps, sufficient to hold six million cubic place yards of the initial overburden can be built up with four miles of trackage. These track positions were roughly located.

The drainage problem is often quite serious when working pits in wet countries, sometimes requiring a pump shaft near the edge of the pit, supplemented by a system of drainage drifts beneath the pit. Fortunately in this vicinity the climate is comparatively dry, so that a 100-gal. per min. pump, placed near a pit sump, will be able to dewater the pit of all catchment waters.

Next a time study was made to see about how long it would take to bring the mine up to the required production. Conditions are such that it is believed that over this estimated interval, an average of four 100-ton shovels can be worked two shifts per day and move an average of 1,000 cubic place yards per shift. In the beginning four shovels cannot be placed to advantage but, as the work advances, this number can be used, and in the later stages of the work an additional one or two shovels can be worked. With this, and the auxiliary equipment, it is assumed that two million cubic yards can be moved per year. As the initial overburden contains five million cubic yards, and the approach contains one million cubic yards, a total of six million cubic yards will have to be removed, which at the average rate of two million cubic yards per year, will require three years to accomplish.

A cost study of the task is estimated as follows:

| | | |
|---|---|---|
| *Approach:* | | |
| Earthwork, 1,000,000 cu. yds. @ 50¢ | $500,000. | |
| Trackage, 3,500 ft. @ $6,000/mile | 4,000. | |
| Building 3 miles of R. R., 2 per cent. grade, | 60,000. | $564,000. |
| *Assembly yard:* | | |
| 4 tracks—4,000 ft. | 5,000. | 5,000. |
| *Preliminary stripping:* | | |
| (To expose ore 1,000 ft. × 100 ft.) | | |
| Removing 5,000,000 cu. yds. @ 35¢ | 1,750,000. | 1,750,000. |
| *Preliminary dump and supply trackage:* | | |
| (Allows for 4 dumps and 1 supply track and totals 19,000 ft. of trackage) | | |
| Excavating 60,000 cu. yds. | 40,000. | |
| Track and construction | 20,000. | 60,000. |
| Total | | $2,379,000. |
| *Plant-buildings and equipment:* | | |
| (As per detailed estimate given later) | | 500,000. |
| Total initial expenditure | | $2,879,000. |
| or say in round numbers | | $3,000,000. |

In this problem it will be assumed for simplicity that no purchase price is placed on the property, that no indebtness remains from the old development work, and that ample ground for dump-storage has been acquired; thus there will be built up a suspense account of $3,000,000 before the property is brought to production. Of this amount $1,750,000 is for initial overburden removal and $1,250,000 is for building, equipment, and general working facilities. By some method, this initial capital expenditure will have to be returned as the ore is extracted. If evenly apportioned over the 12,000,000 tons of shovel ore developed, the latter amount equals without interest 10.4 cents per ton. Allowing three years for the preliminary work and seven years for extracting the orebody, the life of the operation would extend over a period of ten years. This sum of $1,250,000 would probably be spent during the first two years of the operations, so that interest charges must be calculated for a period of, say, nine years. Interest tables[1] show, that the amount which must be set aside at the end of each year for nine years,—so that at the date of the last payment this sum and its interest (taken at 6 per cent. and interest being compounded annually on the balance) will be paid—will be $183,775. The total sum to be set aside then equals $1,653,975, or at the rate of 13.8 cents per ton of ore developed.

It will further be assumed that an *average* sum of $1,250,000 will be tied up in a deferred stripping suspense account over a period of eight years of the operations. This is due to the necessity, during the earlier years of the work, of partly stripping a considerably larger area than that actually exposing ore. In other words to expose a given amount of ore for extraction it is not only necessary to completely strip such ore but also to partly strip a large amount of contiguous territory in order to allow for proper working-slopes and benches. In the later years of the life of the property this suspense account can gradually be reduced until finally absorbed. In the meantime some provision should be made for the interest value of this sum in suspense. Interest tables[2] show that at the end of eight years, $1,250,000 at 6 per cent.—interesting being compounded annually—amounts to $1,992.312. As a proper charge per ton of ore extracted, to carry its proportion of the stripping expense, has been determined, the $1,250,000 principal will automatically be absorbed

[1] Robinson, J. W.—Robinsonian Building-Loan Interest Tables; 6th Edition, Table 6.

[2] Robinson, J. W.—Robinsonian Building-Loan Interest Tables; 6th Edition, Table 1.

thereby, but the interest difference, amounting to $742,312, must be charged against the ore tonnage. This amounts to 6.2 cents per ton and, when added to the initial plant equipment and general working facilities charge of 13.8 cents per ton, makes a total of 20 cents per ton, of which about 9.6 cents per ton is required solely to cover interest on the principal.

It was previously estimated that to strip the 12,000,000 tons of ore, 24,000,000 cu. yds. of overburden had to be excavated, or an average of 2 cu. yds. per ton of ore; also that the average cost of excavating this material (both overburden and ore) would be 32 cents per cu. yd. in place. The estimated complete cost per ton of mining the orebody is therefore as follows:

| | |
|---|---|
| Removing 2 cu. yds. of overburden @ 32¢ | $0.64 |
| Removing ½ cu. yd. of ore (equal to 1 ton) @ 32¢ | 0.16 |
| Initial expenditure and interst charge redemption | 0.20 |
| Total cost | $1.00 |

**Consideration of Underground Methods.**—Of the many methods of underground mining, those well adaptable to a soft low-grade deposit of this class are not numerous. The following ones however are considered worthy of study.

*Top-Slicing.*—This method, as employed at several well operated mines, has given good results. When carefully worked it gives a high percentage of ore recovery because the general control is good, and shallow portions and fingers of the orebody can be closely followed out. The product is clean and the method is safe for the miners. It has the disadvantages however of a rather high cost and limited flexibility of output. Under unfavorable conditions the cost may reach $2.00 per ton while under very favorable conditions it may be as low as 80 cents per ton. A large amount of timber is required, both for supporting the openings and to form a heavy timber-mat between the ore and caved overburden. Ventilation is usually difficult, and this results in heating, which lowers the efficiency of the workmen. The output in tons per man-shift is lower than with some other methods. As timber and labor are both relatively expensive at this property it is estimated top-slicing would cost about $1.30 per ton.

*Shrinkage-Stope Methods.*—These methods are adaptable to deposits of considerable thickness, horizontal extent and regularity where the ground is self-supporting enough to permit the working up of stopes of convenient size without overhead support or protection, and where it is believed the extraction drifts (for drawing off the broken ore from stopes and caving pillars) may be kept open without excessive repair costs. Under these conditions the method is safe, the cost reasonably low—say from 70 cents to $1.00 per ton,—a high percentage of fairly clean ore may be expected and, once the deposit is well opened up, the operations may be conducted on a large scale with considerable flexibility of output. Much preliminary development work is required. Care must be used in carrying up the shrinkage stopes and especially in seeing that the pillars cave completely. The ore must be drawn off systematically to prevent chimneying, otherwise there results a bad mixing of waste and ore, lowering the grade and leaving ore unrecovered. It is essential that the capping follow the ore down with the drawing, as otherwise large unfilled chambers will be left which may later result in dangerous air-blasts upon the sudden collapse of large areas of capping. The method is not adaptable to very soft or running ground, or to deposits of great irregularity or thinness. Considerable capital is tied up in broken ore in the stopes before much reserve drawing can be started. This may add as much as 5 per cent. to the mining cost due to interest charges. Allowing for these factors, the deposit under consideration would lend itself to this method, provided the ground proved to be sufficiently self-supporting to permit of safe stopes and reasonably cheap maintenance of the extraction drifts. The deposit is however very soft and decomposed and contains a high percentage of kaolin, so that considerable doubt is felt about this last point.

*Block-Caving.*—This method is considered because it is known to give low costs, running from 60 cents to 90 cents per ton. When applied to deposits where the ore is of good grade and where the fringe-rock and capping are poor or valueless, there is serious objection to the method because of the lack of control over the drawing down of the ore. A large amount of waste invariably gets intimately mixed with the ore and this either seriously lowers the grade, or else involves heavy loss of good ore which has become entrained or badly diluted with waste. Unless sections of the orebody can be drawn fairly evenly these losses will be very serious. Because the control points governing the draw are usually so few and draw each from a number of divergent feeder points, it has been very difficult to obtain satisfactory results. Consequently it cannot be recommended for the orebody under discussion.

*Branch-Raise System.*—This system has been successully employed for mining several of the low-grade copper deposits individually very dissimilar as to their shape, dip, structure and ore texture. The reported costs have been remarkably low, ranging say from 35 cents to $1.00 per ton of ore delivered to surface. The method requires considerable initial and continuous development work, but it is simple, safe and flexible. The control gates differ from those used in block-caving as they are here placed close together in the "finger raises" (10 to 12½ ft. apart), draw only from one feeder point, and give easy access for inspection of the draw. For this reason even drawing may be effected, so that with the close definite control, a high percentage of tonnage recovery, without a serious drop in grade, may be expected.

Careful consideration of these various methods leads to the belief that the branch-raise system will be the best underground method to consider in competition with the open-pit method.

Detailed estimates indicate that an average mining cost, including current development work, of 80 cents per ton can be attained under present local conditions.

To develop and equip the property for a production of 5,000 tons per day by underground mining, it is estimated will require about two years. This will cost:

| | |
|---|---|
| For buildings and equipment.............. | $500,000. |
| For mine development work........... | $700,000. |
| Total............................... | $1,200,000. |

Proportioned over the tonnage developed, this amounts to 10 cents per ton.

After the property has been brought up to production, viz., at the end of two years, the current cost of ore extraction will be expected to carry the further cost of development work. To avoid wide current cost fluctuations, the development charges per ton will be based on the estimated average cost of such work over the life of the mine, although this figure may require periodic adjustment. It will be assumed then that this initial expenditure of $1,200,000 will be carried in suspense over eight years (one during development and seven during extraction) but in gradually decreasing amounts until finally extinguished at the conclusion of operations. Interest tables show that the annual payment at the end of each year which will, at the date of last payment, pay this debt and its interest at 6 per cent. to that date—interest being compounded annually both on amounts paid and the amounts due—is \$193,248; or a total of \$1,545,984 for the eight payments. This is equivalent to an average of 12.9 cents per ton of ore developed, or 2.9 cents per ton to cover only the average interest charges. (A further refinement in the apportionment of interest charges is not now practical).

The complete mining cost will therefore be 80 cents plus 10 cents plus 2.9 cents, or a total of 92.9 cents per ton.

Simply from the foregoing calculations it appears that the main orebody favors extraction by underground methods rather than by shovels, for these weighty reasons; first, the initial expenditure which must be made before full production can be expected is only a little more than one third; second, the time required to bring the property up to production is two years instead of three years; and third, the calculated cost of mining is about 7 per cent. less.

There are, however, other factors which must be considered before reaching a final decision. Some of these will now be taken up.

**Estimated Grade of Ore which can be Worked and Yield a Profit.**—Certain further assumptions, based on experimental work, experience and current practice, must here be made and are as follows:

| | |
|---|---|
| Ratio of concentration in milling: 12 to 1......... | 74 per cent. extraction |
| Extraction in roasting, smelting and converting... | 95 per cent. extraction |
| Assumed plant recovery: 74 per cent. × 95 per cent. | 70 per cent. extraction |
| Cost of coarse crushing at mines............ | \$0.03 per ton of ore |
| Transportation of ore to mill............... | 0.12 per ton of ore |
| Cost of concentrating...................... | 0.45 per ton of ore |
| Cost of roasting or nodulizing concentrates.. | 0.45 per ton of concentrates |
| Cost of smelting concentrates.............. | 2.00 per ton of concentrates |
| Cost of converting........................ | 8.00 per ton of blister copper |
| Cost of freight, refining, selling and miscell.. | 0.016 per lb. of copper |

In mining low-grade copper ores it has been found from experience that the average per cent. of copper in the ore mined is usually lower than called for by the assay plans. This is partly due to mining some lower grade ore than at first contemplated, partly to intermixture of low-grade capping and wall rock and partly to the tendency of sampling to give higher rather than lower results. For these reasons, it will be assumed that the grade of the ore will be lowered by 5 per cent. or to an average of 1.9 per cent., if mined with shovels; and by 10 per cent., or to an average of 1.8 per cent., if mined by the branch-raise method. It will be safer to allow no increase in tonnage expectancy because of this drop in grade.

The complete mining, treatment and marketing costs will be as follows:

| | Cost per ton of ore | |
|---|---|---|
| | By shovels | By underground |
| Mining | $1.00 | $0.93 |
| Coarse crushing | 0.03 | 0.03 |
| Transportation to mill | 0.12 | 0.12 |
| Concentration | 0.45 | 0.45 |
| Roasting or nodulizing | 0.04 | 0.04 |
| Smelting | 0.17 | 0.17 |
| Total | $1.81 | $1.74 |

The copper yield per ton of ore will be 70 per cent. of 38 lb. or 26.6 lb. with shovel mining; and 70 per cent. of 36 lb. or 25.2 lb. with underground mining. The production cost per pound of copper will then be:

| | Cost per pound of copper | |
|---|---|---|
| | By shovels | By underground |
| For the above items | 6.8 cents | 6.9 cents |
| For converting | 0.4 cents | 0.4 cents |
| For freight, refining, selling, etc | 1.6 cents | 1.6 cents |
| Total | 8.8 cents | 8.9 cents |

Assuming the selling price of copper to average 15 cents per pound over the life of operations, and 2 cents per pound to remain fixed as the cost of converting, refining and marketing, there will be left 13 cents per pound from which to pay all other expenses and profits. As the expense of operations using shovel mining is $1.81 and using underground mining is $1.74, a minimum yield of $1.81 ÷ 0.13, or 14 lb. for the former and $1.74 ÷ 0.13, or 13.4 lb. for the latter, is required to meet operating expenses. With a plant extraction of 70 per cent. this means that the ore must carry 20 lb. for the former and 19.2 lb. for the latter, or assay 1 per cent. and 0.96 per cent. copper respectively. Considering that waste admixture and other reasons mentioned may reduce the grade of ore shown by development sampling by 5 per cent. for shovel mining, and 10 per cent. for underground mining, the grade of 1.25 per cent. copper, adopted as the dividing line between ore and waste, will be reduced to 1.19 per cent. and 1.12 per cent. as against the grades of 1.0 per cent. and 0.96 per cent. respectively, required to cover operating expenses, or to "break even." As the object of the operations is to show a profit, material of a development grade of 1.25 per cent. (which should show a profit of from 40 to 50 cents per ton) is considered as lean as should be sought. As a further refinement, it

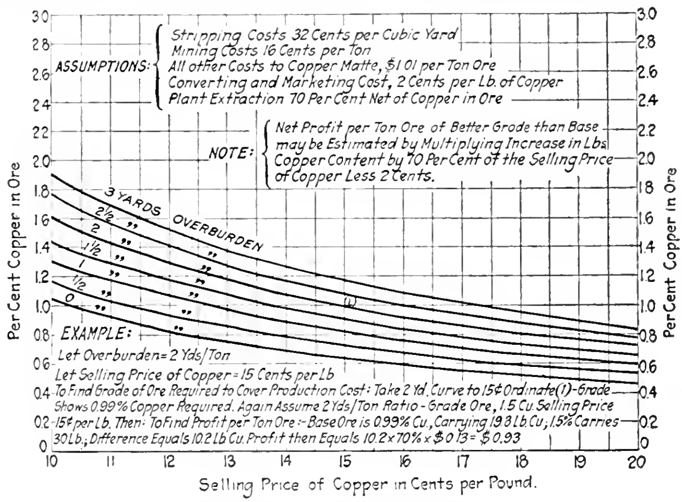



FIG. 45.—Chart showing approximate yardage of overburden a ton of ore of given grade can carry, with given selling price of copper, and just cover complete production costs.

might be shown that the plant extractions and working costs per pound of copper produced will be less favorable the further the grade of the ore falls below the average.

To quickly determine what approximate amounts of overburden can be profitably carried by various grades of copper ore and under various metal prices, a chart similar to the one shown in Fig. 45, which is based on the present example, may be constructed. A similar chart may also be used for underground mining when the cost of such mining is decided upon. Similar graphic representations may be drawn for other classes of ore and will be found approximately correct and useful for the operating departments. Obviously the higher the market price for metal, or the lower the mining and treatment cost attainable, the lower will be the grade of ore possible to work at a profit. It should be remembered, however, that the plant extraction and production cost per pound will be less favorable as the grade falls.

**Consideration of Profits from Lower Grade Material.**—The tonnage estimate showed that in the event of shovel mining a certain tonnage of mixed carbonate and sulphide material would be removed in stripping the main orebody; also that another tonnage of low-grade primary sulphide material would be left stripped below the main orebody. On the former there will be no mining or stripping charge, and on the latter a mining charge, but no interest or equipment charge.

Assuming the per ton treatment costs to be the same as those estimated for the main orebody, the cost of working the copper-bearing overlying material would be \$1.00 per ton less and the cost of handling the underlying material would be say \$0.80 less, allowing \$0.20 per ton for mining. This would make the corresponding costs:

\$1.81 – 1.00, or \$0.81 per ton; and \$1.81 – 0.80, or \$1.01 per ton. With 13 cents net to cover these costs, the copper yield required is $0.81 \div 0.13 = 6.2$ lb., and $1.01 \div 0.13 = 7.8$ lb. respectively.

Allowing a 5 per cent. drop in the development grade of this material there would be:

600,000 tons of material containing 1.09 per cent. Cu. (equal to 21.8 lb. per ton)

2,000,000 tons of material containing 1.04 per cent. Cu. (equal to 20.8 lb. per ton)

Assuming a plant extraction of 60 per cent. on this mixed and low grade material, some of which would probably have to be leached, the yield per ton from the above would be 13.1 lb. for the former, and 12.5 lb. for the latter. As this is in excess of the poundage to cover actual cost requirements, the profits per ton would be: 13.1 lb. – 6.2 lb. = 6.9 lb. × \$0.13 = \$0.897 for the carbonates. 12.5 lb. – 7.8 lb. = 4.7 lb. × 0.13 = \$0.611 for the sulphides.

Under these conditions it is indicated that this material may be expected to yield a fair profit, and that material assaying even as low as say 0.7 per cent. copper might be handled without loss provided it bore no mining or stripping charge.

As a matter of proper economic policy, a plant of limited capacity would avoid, if possible, the treatment of such material until after the more profitable ore had been exhausted.

**Comparative Resumé of Estimated Profits Derivable.**—The above discussion may now be summarized as follows:

| | Profit per lb. | Yield per ton | Profit per ton | Total profit |
|---|---|---|---|---|
| *Main ore body:* | | | | |
| By shovels: | 6.2 cents | 26.6 lb. | $1.65 | $19,800,000 |
| By underground: | 6.1 cents | 25.2 lb. | $1.54 | $18,480,000 |
| *Low grade carbonate:* | | | | |
| By shovels: | 6.8 cents | 13.1 lb. | $0.90 | $540,000 |
| By underground: | (could not be mined at a profit) | | | |
| *Low grade sulphide* | | | | |
| By shovels: | 4.9 cents | 12.5 lb. | $0.61 | $1,220,000 |
| By underground | (could not be mined at a profit) | | | |

Summary of all Important Factors

| | By shovels | By underground |
|---|---|---|
| Total profit derivable | $21,560,000 | $18,480,000 |
| Initial capital required | $ 3,000,000 | $ 1,200,000 |
| Time required to reach production | 3 years | 2 years |
| Flexibility of efficient output: | | |
| Tonnage | 50 per cent. | 25 per cent. |
| Grade | limited | considerable |
| Labor force required: | | |
| Crew | 325 men | 500 men |
| Percentage of total operating cost | 45 per cent. | 60 per cent. |

Availability—Ample in both cases but fewer high-priced men will be required with shovels.

Local conditions:

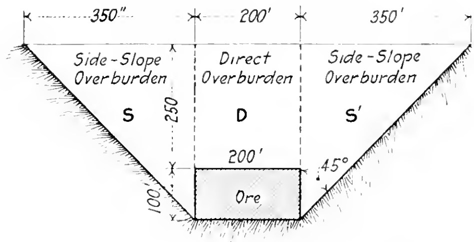
Climate—Generally favorable; open-pit work may be somewhat, though not seriously, interfered with by storms for three months of the year.

Supplies—All required can be readily secured in both cases. Of the total operating cost, supplies will amount to about 55 per cent. of steam-shovel mining and 40 per cent. for underground mining.

**Conclusion and Remarks.**—In consideration of the foregoing estimates, shovel work would probably be recommended, for although much more initial capital is required and a longer time will elapse before arriving at full production, the open-pit method will yield larger profits, (even omitting the doubtful carbonate and low grade sulphide material), have greater flexibility of output, deliver a cleaner product, give a higher percentage of the tonnage expectancy and be subject to rather better labor conditions. The shovel work is more comprehensive and there is more assurance of the correctness of the observations.

It is evident that the work is of sufficient magnitude to fully justify shovel methods, notwithstanding the attendant heavy initial expenditure.

Although the example chosen is one of a low-grade copper ore, any other class of deposit, such as a body of iron ore, or a coal bed or vein, may, with individual modifications, be worked out in much the same way.




Area Ore: 100'x 200' = 20,000 Sq Ft. = 1.0.
" Sts: 350'x 350' = 122,500 " " = 6.12 } = 8.6+
" D : 250'x 200' = 50,000 " " = 2.50 }

FIG. 46.

It is interesting to note that a different conclusion might have been reached had the ratio of overburden to ore been less favorable—say, $2\frac{1}{2}$ cu. yd. per ton of ore—or had some of the other factors been less favorable to pit-mining. Briefly, such changes will be further illustrated.

The foregoing example may be considered as a "borderline" proposition, and it was especially chosen as such because in it could be brought out most of the factors that must eventually be considered in the solution of such a mining problem. Many problems can at the outset be quite obviously classified as underground or shovel propositions, and can be more quickly solved, since many of the minor details may be omitted.

To emphasize this, suppose that this deposit had covered about the same area and carried about the same tonnage of ore, but that its principal dimensions had been much different. For example, suppose the short axis or width had been 200 ft. instead of 1100 ft.; the long axis or length, 8000 ft. instead of

1800 ft.; and the average thickness about the same, or, say, 100 ft. while the direct overburden remained at about 250 ft. Under such changed conditions, the simple section herewith, Fig. 46, indicates something like 8.6 cu. yd. of overburden per cu. yd. of ore, or, say, 4.3 cu. yd. per ton of ore. Note that of this overburden about 70 per cent. is in the 45° side-slopes alone, with but 30 per cent. as direct overburden. Thus the stripping cost alone, at 32 cents per cu. yd., would amount to $1.38 per ton of ore, as against the charge of $0.64 in the original problem. While the solution of the second problem is not precise, it is quite close enough to indicate that shovel methods need not be given further consideration.

Now consider if the orebody chosen in the "border line" illustration had been simply tipped up to a vertical position about either of its axes, and then covered with but a light capping. It will be seen that in this case the side-slope overburden representing practically 100 per cent. of the total, would be enormous, and the ratio of ore to overburden such as to decisively prohibit open-pit mining.

Again, had both the average thickness of the orebody and the direct capping been doubled, the final ratio of overburden to ore would have been much less favorable than in the first case, simply because of the large amount of side-slope removal, and thus the decision would have been against shovel methods. On the other hand, had the average thickness of orebody and direct overburden been halved, the ore-overburden ratio would have been more favorable to shovel work, although in that case the ratio of output and the initial capital investment would be reconsidered for downward revision.

Finally, it will be perfectly obvious that had all other conditions remained the same, but the average thickness of capping been only 125 ft. instead of about 250 ft., the problem would have been a steam-shovel one without any question.

## SPECIAL PROBLEMS

There are some special problems which require rather a different point of view than the one assumed in the foregoing.

It not infrequently occurs, especially on the iron ranges, that a deposit is found which must be analyzed in separate parts rather than as a whole. The overburden may be very thick over one part but thin over another or the ore-overburden ratio may vary widely. Again, the shape of the deposit or character of the ore may be irregular. Under such conditions, a combination of methods, open-pit for one section and underground for another, may be much more advantageous than either one exclusively for the whole deposit. In such a case, a careful study is required to indicate where to draw the dividing line.

In shovel-mining, a certain amount of ore is often left in the side-slopes that could not be economically mined by any method because of inaccessibility or lower grade than the average. Such slope-ore that to take out with shovels would involve the removal of too great an additional amount of overburden, can, however, often be profitably mined with some semi-underground method carried on from the pit bottom or benches.

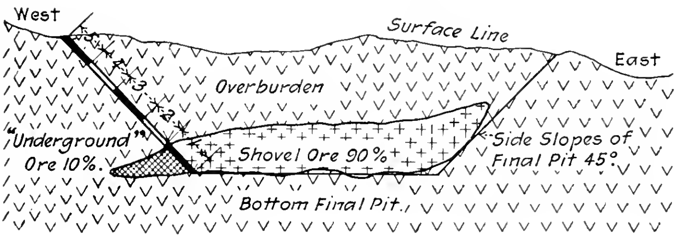



Fig. 47.

This classification of ore is a most important point; its recognition and application extends from leaving a small percentage of the total orebody as "fringe-ore" in the pit side-slopes to segregating the entire deposit into different classes of ore as regards extraction. To illustrate its application a few examples will be given.

The sketch, Fig. 47, is a rough cross-section of an ore body which contained about 40 million tons, 90 per cent. of which could be mined by shovels and yet keep the overburden-ore ratio under 2 cu. yd. per ton of ore. In most places the ratio was lighter than this, but where it was found to be heavier for any noteworthy tonnage, it was carefully noted by a different color. Such tonnages were found to lie principally in the side-slopes as indicated in the sketch, and it was estimated that this ore could be mined by slicing or caving for about 80 cents per ton. This amount was about the same as the cost of mining by shovels when the overburden-ore ratio did not exceed four to one. A simple approximate way to apply this dividing ratio was to project the slope line through the ore where the ratio did not exceed four in waste to one in ore, as shown on the west side of the section. In this way it was found that there were about 4 million tons of ore in the side-slopes which it was decided to mine by semi-underground methods. It may be mentioned that to have lumped this side-slope ore in with the rest of the orebody would have given an overburden-ore ratio considerably lighter than four to one, but it is obvious that such a method is not fair to the ore more favorably situated, and would have reduced the final profits.

A second illustration is given in Fig. 48, where 60 per cent. of the orebody was mined by shovels and 40 per cent. by slicing methods. Here the overburden-ore ratio dividing line was

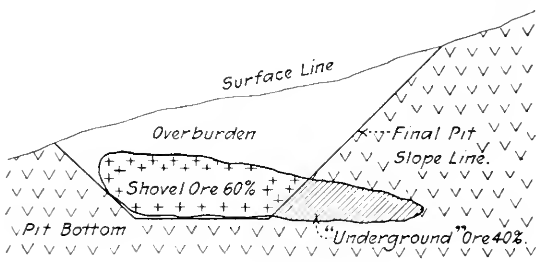



Fig. 48.

drawn at $1\frac{1}{2}$ cu. yd. of overburden to one ton of ore, it being estimated that with a heavier ratio the underground method would prove more economical.

A third illustration is given in Fig. 49. In this case the ore body is worked by open-cast methods which will be continued until the constantly increasing side-slope overburden makes it economically necessary to adopt another method of mining. It is estimated that this point will be reached when the overburden-ore ratio reaches, say, about three yards of overburden to one ton of ore. This will not be entirely caused by side-slopes, but also through the necessity of deepening the approach. The next method will probably be to adopt a milling system, and continue with that as deep as the side-walls will permit. After that, it will be necessary to adopt some straight underground system, and, on account of the character of the ore being a more or less massive sulphide, the chamber-and-pillar method may be adopted. In this last method, the worked-out chambers are closely filled with waste, just as compactly as possible, before the pillars are attacked; the object being to avoid too much pressure being thrown on them, as in that event, they begin to crush and heat with the very serious danger of the ore catching fire. It is expected that after shovel work is no longer practicable, the ore will be hoisted to the surface through a shaft, as is roughly indicated on the sketch.

From this illustration it will be seen that in the case of a long narrow orebody, or a steeply pitching one, it may pay better to strip and mine only the upper portion with shovels, because so to mine below a certain level would involve the removal of an excessive amount of overburden in order to provide safe working slopes or trackage spirals of reasonable grade to haul the ore out

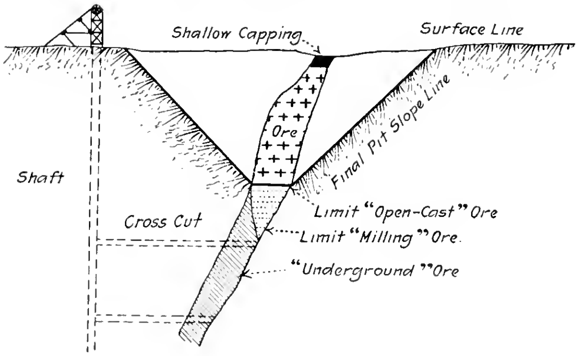



Fig. 49.

of the pit. In such cases the lower portion of the orebody will be at least partly stripped, and it may often be possible to continue extraction with an open-cast milling system. In this system, the ore in the pit bottom can be mined into a checkerboard series of mill-holes feeding into a net-work of underground extraction drifts, which in turn connect with a hoisting shaft.

In some cases it has been found desirable to strip an orebody with shovels, or drag-line excavators, and then mine the ore by a milling or underground method. Such a scheme may be made desirable where the overburden consists of very wet material, such as quicksand, resting on impervious material and under such conditions that the orebody cannot be properly drained for satisfactory underground mining. Such cases were seen in the iron ranges of Minnesota.

It was previously mentioned that certain classes of material, such as large bodies of massive pyrites, are frequently mined by open-cast methods because of the serious fire risk involved in mining them by underground methods. In this case, the comparative direct mining costs by different methods may be of secondary importance. Examples of this sort will be found at Rio Tinto, Spain.

In the anthracite coal regions of Pennsylvania, the older methods of mining were generally extremely wasteful of the coal. It has been found,[1] after the veins had been mined and the pillars then robbed, and even re-robbed, that from 75 per cent. to 50 per cent. of the original coal had been left. These losses were in the form of pillars, neglected portions of the veins, and in abandoned areas due to "runs" of loose material. The loss by fire has also been considerable. More modern practice has reduced these losses but they are still very heavy. Although the problem is still to produce the largest quantity of coal for the least expenditure of money, these losses have caused many operators to resort to stripping operations. This method may not always reduce the unit cost of production but it often permits mining a much larger percentage of the tonnage and in a manner so much more comprehensive that a higher yield of prepared sizes of coal is obtained, the product is cleaner, and the output, though flexible, may be kept steady. Furthermore, there are deposits which, because of the way they have been worked in the past or because of the loose friable character of the coal, can now only be recovered by stripping operations. In such cases the cost may be considerably higher than the usual underground cost. Lessors of coal lands often lower their royalty rates in order to induce the stripping of areas that otherwise would remain unmined. In addition to these reasons for stripping, there is the grave question of the conservation of these resources and when so considered, wasteful underground methods are generally at a serious disadvantage.

Some of the special problems will be further considered later.

J. B. Warriner, chairman of the committee representing the larger users of stripping methods in the anthracite region, gives two illustrations of stripping problems, with possible fundamental errors that may occur in preliminary calculations.

The first illustrates the economic limits of stripping in area and

[1] Warriner, J. B., Anthracite stripping; T. A. I. M. E., Feb., 1917, pp. 33-60.

depth. It is assumed that it has been decided to expend, on coal recovered from the stripping shown in Fig. 50, an amount per ton equal to the average margin of profit of the colliery, the return on the investment being considered to be secured by certain factors or advantages that do not lend themselves readily to calculation in exact figures. Then this cost per ton figure is translated into a ratio of cubic yards of overburden removed per ton of coal uncovered, and amounts for example to 2¾ cu. yd. per ton. Then the limits in area and depth are marked out to give this ratio. These limits at first may appear satisfactory unless the problem is resolved into its component parts, as shown on the figure by the shaded areas. Part A is the lowest component part which comes within the limits of the ratio, viz., 2¾ cu.

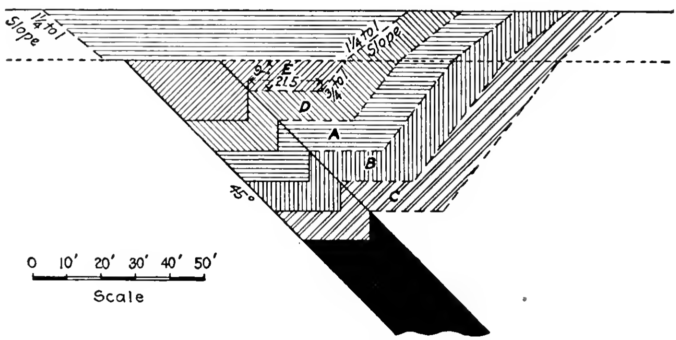



Fig. 50.—Anthracite stripping problem.

yd. per ton uncovered, while part B carries 3 cu. yd. and C carries 3½ cu. yd. These then are worked at a loss, regardless of the fact that parts D and E are operated at a considerable profit. To justify the removal of B and C areas there must be gained some marked advantages not included in the factors used in setting the 2¾-cu. yd. ratio; otherwise a considerable amount of the coal reserve would be depleted at no profit.

The second example, illustrated in Fig. 51, is a crop stripping for a virgin area where the clay and gravel overburden must either be removed or a chain pillar of coal left below the surface to prevent the contamination of the prepared coal. In this case it would be necessary to leave a 60 ft. chain pillar unless it is stripped. The coal below the chain pillar can be mined as cheaply per ton, for cutting and loading, as can all the coal from gangway to surface if stripped. By the latter method, lower unit development costs would be obtained, but this would amount to but a few cents per ton of coal. Against assuming the $2\frac{3}{4}$ to 1 ratio as the economic limit, it is found that in this case there are 4 cu. yd. to 1 ton of coal in the chain pillar, and

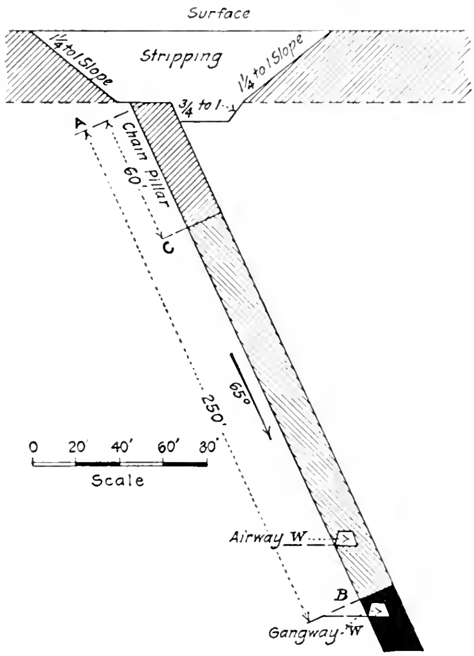



FIG. 51.—Anthracite stripping problem.

1 cu. yd. to 1 ton of coal in the entire area between gangway and surface. Therefore, the coal that can properly be classed as stripping coal is mined at a loss, and unless there are other substantial factors in its favor, the stripping should not be undertaken.

## CHAPTER VII

## COST OF SHOVEL WORK

The cost of excavating and disposing of material worked by shovels is subject to wide variations dependent on all the factors previously mentioned. Examination of what the costs have been at various well operated properties, taking proper account of particular local conditions, often serves as a useful basis in estimating what figures may be expected at new properties. It is to be noted that the cost of labor and supplies of all sorts has a decided upward tendency, and it has only been by great improvement in methods, equipment and general coördination of work that the general efficiency has been increased to such an extent as to approximately offset the higher prices paid for labor and all commodities. What has been done in the past may perhaps be expected in the future, although it now seems probable that both the cost of production and the selling price of the commodity produced will have an upward tendency in the future as compared with pre-war times. In examining the following examples it seems fair to assume that shovel costs during the years 1917 and 1918 were abnormally high, owing to general war conditions, but whether such costs will ever again be reduced to pre-war figures is doubtful. The efficiency of operating labor is much lower when the men are new at their work than when they have become seasoned to it.

As illustrative of the way in which the cost of open-pit work has risen the following comparative data are interesting.

A sliding scale was adopted depending on the selling price of copper. The figures given here are for copper selling at 18 to 19 cents per lb., which is the lowest basis. All labor received $12\frac{1}{2}$ cents more per shift for each increase in the selling price of copper of 1 cent per lb. up to 26 cents per lb., when for example, steam shovel engineers would receive \$7.83 per shift. Common labor, however, received an increase of 7.5 cents per shift per

1 cent increase in selling price until the selling price reached 22 cents, thereafter the increase was at the rate of 12.5 cents. It has been found that labor costs represent about 50 per cent. of the total cost of this work.

Comparative Wage Scale, Nevada Consolidated Copper Company Pits

| Occupation | Rate | |
|---|---|---|
| | Summer, 1915[1] | May, 1919[2] |
| Steam shovel engineers | $6.10 | $6.83 |
| Steam shovel cranemen | 4.42 | 5.16 |
| Steam shovel firemen | 3.25 | 4.00 |
| Pit men | 2.30 | 2.95 |
| Locomotive engineers | 4.50 | 5.25 |
| Locomotive firemen | 3.25 | 4.00 |
| Locomotive switchmen | 3.50 | 4.25 |
| Yardmen | 3.75 | 4.50 |
| Well drillers | 4.10 | 4.85 |
| Tool dressers | 3.25 | 4.00 |
| Track and dump men | 2.20 | 2.85 |
| Metal workers | 4.50 | 5.75 |
| Carpenters | 4.75 | 5.75 |
| Bricklayers | 6.00 | 7.25 |
| Electricians | 5.25 | 6.00 |
| Mechanics' helpers | 3.25 | 4.50 |
| Common labor, American | 3.00 | 4.25 |
| Common labor, foreign | 2.20 | 2.75 |

It is safe to say that the cost of supplies has risen in 1919 from 25 to 50 per cent. higher than the cost in 1915. Quoting from the 1917 Chino Copper Company report the general manager states: "In 1915, $1.00 moved 2.83 cu. yd. of material in place, in 1916, 2.63 cu. yd. of material, while in 1917, $1.00 moved only 1.97 cu. yd. Taking 1915 as the standard, $1.00 moved 100 per cent. in 1915, it moved 92.93 per cent. in 1916, and only 69.61 per cent. in 1917. In stating the above costs, taxes, other administrative and general charges have been included as usual." The experience of other companies no doubt closely parallels that of the Chino Company.

[1] In 1915 the shift was nine hours and straight time was paid for all overtime.

[2] All wages in 1919 based on eight hours.

## EXAMPLES OF COSTS

NEVADA CONSOLIDATED COPPER CO., RUTH, NEVADA

| Year | Cu. yd. overburden removed | Cost per cu. yd. | Tons of ore removed | Cost per ton |
|---|---|---|---|---|
| 1911 and previous | 6,038,683 | $.3630 | 7,137,416 | $0.157 |
| 1912 | 2,732,976 | .3364 | 2,596,991 | 0.1735 |
| 1913 | 3,100,661 | .3399 | 2,889,389 | 0.1775 |
| 1914 | 3,044,966 | .3171 | 2,513,241 | 0.1517 |
| 1915 | 2,758,350 | .2887 | 2,991,782 | 0.1524 |
| 1916 | 3,988,650 | .3009 | 3,337,570 | 0.2370 |
| 1917 | 2,998,025 | .3443 | 3,076,285 | 0.3338 |
| 1918 | 2,617,771 | .4120 | 2,711,743 | 0.4169 |

Totals to Jan. 1, 1919: 27,280,088 cu. yd. of waste and 26,855,857 tons of ore had been removed.

Under the cost of overburden are included all charges complete to waste dumps.

Under the cost of ore are included all charges such as labor, supplies, repairs, management, proportion of general and New York expense, etc., and taxes. Transportation of ore from the assembly yards to the mills is, of course, excluded. The item of taxes alone amounted to 4.08 cents in 1915; 11.28 cents in 1916; 14.48 cents in 1917, and 15.81 cents in 1918. This charge is entirely arbitrary and beyond the control of mining operation. The figures are taken from the annual reports of the company. One cubic yard is equivalent to 2.16 tons.

The stripping cost may be analyzed approximately as follows:

| | Cost per cu. yd. | Cost per cu. yd. |
|---|---|---|
| Steam shovel operation: | | |
| Shovel labor | $0.014 | |
| Fuel | 0.010 | |
| Supplies and repairs | 0.010 | |
| Pit labor | 0.011 | |
| | | $0.045 |
| Locomotive haulage and train crew service: | | |
| Locomotive labor | 0.025 | |
| Fuel | 0.030 | |
| Supplies and repairs | 0.003 | |
| Yard and train crews | 0.006 | |
| Dump labor | 0.025 | |
| Track maintenance | 0.025 | |
| Car repairs | 0.015 | |
| | | $0.129 |

| | | |
|---|---|---|
| Supervision: | | |
| Superintendents and foremen | 0.005 | |
| Engineering | 0.002 | |
| | | $0.007 |
| Drilling and blasting: | | |
| Churn drills | 0.015 | |
| Labor | 0.006 | |
| Explosives | 0 066 | |
| | | $0.087 |
| General and miscellaneous: | | |
| Water supply | 0.005 | |
| Building repairs and miscel | 0.003 | |
| Fund for renewal reserve | 0.030 | |
| General expense and insurance | 0.010 | |
| | | $0.048 |
| Total per cubic yard in place | | $0.316 |

The mining cost may be analyzed approximately as follows:

| | Cost per ton | Cost per ton |
|---|---|---|
| Breaking ore: | | |
| Churn drills | $0.007 | |
| Labor | 0.003 | |
| Explosives | 0.033 | |
| | | $0.043 |
| Steam shovel operation: | | |
| Shovel labor | 0.008 | |
| Fuel | 0.006 | |
| Supplies and repairs | 0.006 | |
| Pit labor | 0.007 | |
| | | $0.027 |
| Locomotive haulage and train crew service: | | |
| Locomotive labor | 0.008 | |
| Fuel | 0.010 | |
| Supplies and repairs | 0.002 | |
| Yard and train crews | 0.002 | |
| Track maintenance | 0.011 | |
| Car repairs | ..... | |
| | | $0.033 |
| General and miscellaneous: | | |
| Water supply | 0.003 | |
| Supervision and engineering | 0.006 | |
| Building repairs and miscel | 0.003 | |
| Renewals reserve fund | 0.015 | |
| Pit-pumping | 0.002 | |
| General expense and insurance | 0.011 | |
| | | $0.040 |
| Total mining cost per ton | | $0.143 |

To this must be added government taxes as mentioned in the foregoing. The ore is also charged with a certain amount per ton for the redemption of prepaid stripping. The cost of transportation of ore to mills is, of course, carried separately.

The coal consumed per steam shovel shift and per locomotive shift ran about 2.15 and 2.5 tons respectively; as the pit deepened the figure for locomotives has increased by about 2.2 tons. An average of about 1000 cu. yd. per shovel shift and 500 cu. yd. per locomotive shift were moved. About 60 tons of ore and 35 cu. yd. of waste were broken per foot of churn drill hole shot. About 20 lb. of coal were consumed per foot of hole drilled. The powder consumption was about 0.28 lb. per ton of ore and 0.55 lb. per cu. yd. of waste. Elimination of the black powder earlier used on waste, materially reduced the quantity required per cu. yd. of waste and it was found advantageous to use Trojan powder for all classes of material. Using about 50 per cent. cheap black powder and 50 per cent. of a 40 per cent. powder, the consumption per cu. yd. of waste was about 0.75 lb. per cu. yd. About 3 gal. of lubricating oil and 1.5 lb. of grease were consumed per 1000 cu. yd. of material moved.

Utah Copper Company, Bingham, Utah

| Year | Cu. yd. overburden removed | Cost per cu. yd. | Tons concentrating ore removed | Cost per ton |
|---|---|---|---|---|
| Prior to 1910 | 4,347,810 | .... | | ..... |
| 1910 | 2,814,746 | $0.40 | 4,340,245 | $0.2767 |
| 1911 | 5,450,604 | 0.371 | 4,680,801 | 0.2461 |
| 1912 | 4,676,568 | 0.376 | 5,315,321 | 0.2635 |
| 1913 | 4,835,479 | 0.425 | 7,519,392 | 0.2094 |
| 1914 | 5,708,836 | 0.368 | 6,470,166 | 0.2262 |
| 1915 | 5,961,367 | 0.285 | 8,494,300 | 0.1661 |
| 1916 | 5,911,455 | 0.291 | 10,994,000 | 0.2024 |
| 1917 | 4,271,868 | 0.372 | 12,542,000 | 0.3688 |
| 1918 | 4,064,091 | 0.468 | 12,160,700 | 0.4285 |

Totals to Jan. 1, 1919: 48,042,824 cu. yd., of overburden (covering 258.67 acres, of which 132.16 acres had been completely stripped) and 79,381,400 tons of ore had been removed.

Under the cost of overburden are included all charges complete to waste dumps.

Under the cost of ore are included all general and tax charges, but not the proportion of stripping or development charges.

The tax charges were high in 1917 and 1918 as also were labor and supplies. Stripping costs have been charged off on the basis of 7½ cents per ton of ore mined, it being estimated that approximately four tons of ore would eventually be mined for each cu. yd. of overburden removed. An average of about 0.8 cent per ton has been charged as the cost of development of the ore. The above figures are taken from the annual reports of the company.

The stripping cost may be divided up approximately as follows:

| | Cost per cu. yd. |
|---|---|
| Drilling and blasting | $0.085 |
| Steam shovel operations | 0.095 |
| Locomotive haulage and dump labor | 0.085 |
| Track maintenance | 0.080 |
| Dump car repairs | 0.015 |
| Overhead expense | 0.010 |
| General mine charges | 0.010 |
| Freight | 0.020 |
| Total per cu. yd. in place | $0.400 |

The mining cost may be divided up approximately as follows:

| | Cost per ton |
|---|---|
| Drilling and blasting | $0.050 |
| Steam shovel operations | 0.055 |
| Locomotive haulage | 0.055 |
| Track maintenance | 0.025 |
| Overhead expense at mine | 0.005 |
| General mine charges | 0.020 |
| General office, administration and insurance | 0.015 |
| Total mining cost per ton | $0.225 |

To this government taxes must be added; these are widely variable but approximate 7.5 cents per ton. As mentioned, 7.5 cents per ton is charged for redemption of prepaid stripping.

Chino Copper Company, Santa Rita, New Mexico

| Year | Cu. yd. overburden removed | Cost per cu. yd. | Tons ore removed | Cost per ton |
|---|---|---|---|---|
| 1911 and previous | 2,000,000 | $0.3168 | 600,000+ | $0.15+ |
| 1912 | 2,223,678 | 0 2761 | 1,301,463 | 0.1652 |
| 1913 | 3,082,174 | 0.3343 | 1,976,572 | 0 2313 |
| 1914 | 3,173,717 | 0.3347 | 2,114,910 | 0.2213 |
| 1915 | 3,133,916 | 0.3428 | 2,600,271 | 0.1947 |
| 1916 | 3,564,623 | 0.3882 | 3,216,065 | 0.1988 |
| 1917 | 3,712,414 | 0.3710 | 3,607,825 | 0.41 ± |
| 1918 | 3,264,556 | 0.51 ± | 3,749,238 | 0.51 ± |

Totals to Jan. 1, 1919: 23,863,755 cu. yd. of waste and 18,943,755 tons of ore have been removed.

The costs for 1915 may be analyzed about as follows:

| | Mining cost per ton | | |
|---|---|---|---|
| | Labor | Supplies | Total |
| Crushing at mine ($\frac{1}{3}$ of ore) | 0.75 cts | 0.25 cts | 1.00 cts |
| Drilling | 0.80 | 0.20 | 1.00 |
| Blasting | 0.25 | 1.50 | 1.75 |
| Hauling | 4.25 | 2.25 | 6.50 |
| Loading | 4.25 | 2.50 | 6.75 |
| | 10.30 cts | 6.70 cts | 17.00 cts |
| Administration | ..... | .... | 2.50 |
| Total mining cost per ton | ..... | .... | 19.50 cts |

| | Stripping cost per cu. yd. in place | | |
|---|---|---|---|
| | Labor | Supplies | Total |
| Drilling | 1.40 cts | 0.40 cts | 1.80 cts |
| Blasting | 0.40 | 3.60 | 4.00 |
| Loading | 7.00 | 4.20 | 11.20 |
| Hauling | 9.50 | 3.50 | 13.00 |
| | 18.30 | 11.70 | 30.00 |
| Administration | ..... | ..... | 5.00 |
| Total cost per cu. yd | ..... | ..... | 35.00 cts |

Under the cost of overburden are included all charges to waste dumps.

Under the cost of ore are included a proper apportionment of fixed and general charges of all kinds. The high costs in 1913 and 1914 were due largely to the limited areas and inconvenient or intermittent way in which the ore shovels had to operate. They had to avoid interference with the stripping shovels, and the standard railroad ore car service was not as steady or effective as was the dump car service supplying the stripping shovels. Furthermore the ore shovels were often held up awaiting the breaking of large pieces so that oversize material would not be delivered to the mill. The stripping shovels often avoided much of this delay. Larger working areas and the installation of a primary crusher at the mine did away with many delays and inconveniences and resulted in the delivery of a more uniform character of ore and better all round operating economies. The high costs in 1917 and 1918 are again due to war conditions and taxes. One cu. yd. is equal to about 1.95 tons. The figures are taken from the annual reports of the company.

Stripping costs are charged to operations on the basis of 30 cents per ton of ore mined.

Chile Copper Company, Chuquicamata, Chile

The mining costs at this property in 1919 were as follows: (Tons here are metric tons of 2204 lbs.)

| | Cost per ton shipped |
|---|---|
| Breaking ground | \$0.148 |
| Shovel operation | 0.143 |
| Breaking ground in front of shovels | 0.033 |
| Tramming | 0.114 |
| Sampling and assaying | 0.007 |
| Supervision | 0.009 |
| Total | \$0.454 |

Both oil and coal were used for firing shovels during this period. Electric power at the mines costs about $1\frac{1}{2}$ cents per KWH and the consumption is about 0.4 KWH per ton of material loaded by electric shovels. About $1\frac{1}{4}$ lbs. of oil is burned per KWH. Oil costs about \$24.00 per ton at the mine and is more economical than coal. Coal costs about \$30.00 per ton and 7.6 lb. of coal is consumed per ton of material loaded. Rating coal at \$40.00 per ton and electric power at 2 cents per KWH, a saving of about 8 cents per ton was effected by the use of electric shovels.

Blasting is done both by churn drill holes and tunnelling as explained in Chap. IV.

The distribution of shovel time ran about as follows:

| Loading, per cent. | Repairs, per cent. | Blasting, per cent. | Waiting for cars, per cent. | Other delays, per cent. |
|---|---|---|---|---|
| 27 | 13 | 8 | 17 | 35 |

The locomotives used were both coal and oil burners, the former consuming about 1.8 metric tons of coal per shift and the latter one-half as much as the oil, although the oil contained about

18,000 B.t.u. per lb. as against 13,000 B.t.u. for the coal. The locomotive service distribution was about as follows:

| Shovel service, per cent. | Coaling and oiling, per cent. | Taking water, per cent. | Yard service, per cent. | Round house, per cent. | Idle in yard, per cent. | Waiting for empties, per cent. | With crane, per cent. | Miscellaneous services, per cent. |
|---|---|---|---|---|---|---|---|---|
| 60 | 2 | 1.3 | 3.6 | 8.8 | 6.5 | 3 | 2 | 11 |

The cost of explosives per ton of material broken was about 6 cents, black powder costing 6 cents per lb. and 40 per cent. dynamite, 26 cents per lb. Only about 2 per cent. of the charge consisted of the latter.

**Mesabi Iron Ore.**—Usual stripping costs on the iron ranges vary from 15 cents to 30 cents per cu. yd., depending largely on the size of the job, local operating conditions and class of equipment employed. Contract prices for such work have usually been let at from 25 to 32 cents per cu. yd. on jobs carrying one half million cu. yd. or more. The cost sheets covering such work show a range of from 14 to 20 cents per cu. yd. under favorable conditions and from 20 to 26 cents per cu. yd., or even higher, under unfavorable conditions. In winter weather these costs may be increased by from 5 to 6 cents per cu. yd. To this must be added the interest charges on the capital investment tied up in prepaid stripping. For example, a 40 acre tract may require stripping to a depth of 100 ft. costing from $1,250,000 to $2,000,000 and covering a period of 3 years to complete. Assuming the annual interest charge to be $100,000, charged against eight million tons of ore, to be mined at the rate of one million tons per year, the stripping charge would be about 25 cents per ton of ore, which must be added to the mining cost.

Some of the more favorably situated properties show a cost of ore on the cars, including stripping, mining and local overhead charges, of from 15 to 18 cents per ton.

The cost of mining ore varies principally with the hardness, amount of rock sorting required, regularity of iron and phosphorus content, and transportation charge. Taking the Mesabi as a whole, the cost of open-pit ore on the cars varies from 15 to 75 cents per ton under very favorable and very unfavorable conditions respectively. This cost includes all local and outside overhead charges including stripping, but not royalty or interest on fee investment. The average mining cost of central Mesabi open-pit ores loaded on cars will fall within 40 cents per ton under average conditions and within 30 cents per ton under favorable conditions. The ores from the Mahoning-Hull-Rust orebody are exceptionally favorably situated as the deposit is very large, the stripping is light and the operating conditions are very good. The cost here, shipping about 35,000 tons per day, is about 15 cents per ton, of which less than 10 cents is for direct mining. Including these ores the average cost of mining the central two thirds of the open-pit Mesabi ores will average about 30 cents per ton, of which about 15 cents may be taken for the cost of removing overburden and 15 cents as the cost of shovel mining including bringing cars into the pit, loading them and returning them to the yards. As compared with earlier hand-loading work, the shovel loads at from 1/20 to 1/30 the cost. This is under good operating conditions; under very adverse conditions hand-loading has actually at times been cheaper.

The shovel crews from 8 to 10 men as follows:[1]

| | | |
|---|---|---|
| 1 runner, wages | $5.77 per 10-hr. day. | Bonus $25. for 26 days. |
| 1 craneman, wages | 4.04 per 10-hr. day. | Bonus $20. for 25 days. |
| 1 fireman, wages | 2.50 per 10-hr. day. | |
| 4 to 7 pitmen, wages | 2.35 per 10-hr. day. | |

Locomotive engineers were paid $4.10 per 10-hr. day with a $20 maximum bonus for continuous good work for the month. The direct labor cost at the shovel runs from $23.45 to $30.50 per day, assuming full bonus is paid.

The shovel supplies run about as follows:

Coal—2½ to 3½ tons per 10-hr. shift.
Lubricants—black oil 5 gal., cylinder oil 5 gal. per 24 hrs.
Illuminants—gasoline 10 to 15 gal. per night; kerosene 2½ gal. per night.
Water—12,000 to 15,000 gal. per 24 hrs.

Much of the ore of the Lake Superior region is mined on lease with royalties ranging from 10 cents to $1.35 per ton. The average for the region is between 30 and 50 cents, the higher figures appearing in later leases. The Mesabi range bears the highest general average of royalties, viz., from 25 cents to $1.00 per ton. The royalty rate is a measure of the value of the ore in the ground to the fee owner, who generally demands as high a price as the leasor can afford to pay. The higher the grade of the ore and the lower the mining and transportation cost, the

[1] These were the conditions in the middle of 1915.

higher will be the royalty. The Minnesota State Tax Commission for 1915 valued the ores for the purpose of taxation at an average of 18 cents per ton, viz., 50 per cent. of what was regarded as the real present value (36 cents per ton) of the ore in the ground.

**Miscellaneous Work.** *Granby Mine, Phoenix, B. C.*—A Bucyrus 40 R. electric shovel started at this property in 1914, handled about 500,000 tons of material at a cost of about 45 cents per ton. The conditions of mining were rather difficult.

*An Ontario hydro-electric canal job,* involving the removal of about eleven million cu. yd. of earth and four million cu. yd. of rock, removed the former for about 23 cents per cu. yd. complete. Electric shovels, of the 225 B Bucyrus make, were used and power cost about ¾ per cent. KWH. The cut averaged about 160 ft. at the top, 84 feet at the water line and 50 ft. in width at the bottom. Below this came rock, which was taken out in box cut with other shovels of the 110-ton R. R. type. The waste was hauled about 2½ miles.

*Anthracite Strippings.*—The following is quoted from a paper by Mr. J. B. Warriner.[1]

"Anthracite strippings are a notable example of the way in which labor-saving devices have held down operating costs in the face of steadily advancing labor costs. In the early day when $1.10 was the regular wage for a 10-hr. day, unit stripping costs were around $0.26 per cu. yd. for clay excavation and $0.60 for rock. With present wages nearly double the early rate, stripping costs range from $0.16 to $0.20 for clay and from $0.35 to $0.40 for rock. This is due largely to the use of steam shovels and the increase in the size of the shovels employed."

Further on, in speaking of the ability of large drag-line excavators to do this work more cheaply than shovels, it is stated: "Whenever it is possible to cast the excavated material to one side, rather than to load it into cars, labor is largely dispensed with. The labor item in ordinary strippings is probably 70 per cent. of the total cost of stripping, but in such an operation as described this percentage is reduced to about 50 per cent. From the results obtained to date it is believed that present stripping costs can be reduced by these advanced methods by at least 10 cents per cu. yd., allowing amply for interest and deprecia-

[1] T. A. I. M. E. Anthracite Stripping. N. Y. Meeting, Feb., 1917.

tion items. The cost for power has been proven to be only 1 cent per cu. yd. and the first cost of the equipment is little greater than for an ordinary 70-ton shovel operation requiring a full complement of locomotives, dump cars, rails, etc. The principal item of cost is the moving of so large a machine from one stripping operation to another, or to and from the railroad." It was evidently the intention that these machines would be used on the clay strippings but not on the rock work.

*Brown-coal Mining in Germany.*—The estimated cost of removing overburden is from 6 to 10 cents per cu. yd., and of mining open-pit coal deposits (not including stripping charge), of from 25 to 60 ft. thick, from 9 to 10 cents per short ton. The stripping is done with continuous-bucket excavators previously described, and the coal is mined by the milling system. In 1913 about 87,000,000 tons of this coal were mined.

*Yukon Gold Company.*—In providing a seventy mile conduit for the water-supply for the Klondike hydraulic mines[1] of the Yukon Gold Company, 37 miles of open ditch was constructed. The top layer of ground is frozen "muck" composed of very fine black silt and ice, frozen to a depth of from nothing to several hundred feet. This is overlain by a blanket of moss often a foot or more in thickness, which has the effect of preserving the muck in frozen condition throughout the year. Thirty miles of the open ditch was dug with six 30-ton steam shovels, the remainder with horse scrapers. As the shovels required a working space at least 16 ft. wide, no side-sloping was done with them. Small quantities of frozen material were removed with 40 per cent. dynamite, but the larger patches were passed over, and after thawing were taken out with scrapers. The shovels worked two 10-hr. shifts and burned about three cords of wood per day. The record performance for one shift was 410 lineal feet of ditch or 1200 cu. yd., and the average performance was 14.7 lineal feet or 51 cu. yd. per working hour, equivalent to 11.2 lineal feet or 38.7 cu. yd. per available hour. The season lasted from June 15 to about October 15, with work usually slow the first and last weeks, 100 working days being about the true season length. The actual cost per lineal foot was about the same for shovel as for scraper work, but the scarcity of horses would have greatly delayed the work had they been used exclusively and it would have been expensive to import them. The cost of this excavation was some-

[1] H. H. Hall, M. & S. P., Aug. 28, 1915.

times as great as $5 per cu. yd. Common labor was paid $4 per day and board, and horses were hired at $100 per month including found.

*Panama Canal.*—The Canal Records of Nov. 8, 1911, and Feb. 7, May 8 and Aug. 7, 1912, Parts II, give the cubic yards excavated and the costs per cu. yd. of steam shovel work in the Central Division for the four quarters of the fiscal year ending June 30, 1912. From these figures the average costs for the year have been calculated as follows:

| | Cost per cu. yd., cents |
|---|---|
| Drilling | 5.36 |
| Blasting | 6.22 |
| Loading | 4.92 |
| Tracks | 8.85 |
| Transportation | 7.34 |
| Dumps | 4.78 |
| Pumps | 0.41 |
| Maintenance of equipment | 8.43 |
| Plant arbitrary | 3.94 |
| Division expense | 1.45 |
| Administration | 3.64 |
| Total | 55.34 |
| Quantity excavated | 16,917,662 cu. yd. |

**Keeping of Cost Records.** *Graphic Records.*—Graphic representation showing the efficiency of shovel operations is always found helpful to the operating officials and to the crews. Such representation is more easily understood than lists of figures by these men, and it is always to be encouraged. Auditing is a matter of past history, but when the results are clearly presented and can be followed from day to day, there is sure to be developed an incentive for improvement. In addition to production and service records, some empirical cost graphs were worked out[1] for the Nevada Consolidated to show the effect of varying daily production on cost. Figs. 52, 53, 54 and 55 illustrate the method, and by the use of such charts a daily unit cost of production can be figured to a fraction of a cent. The operating costs are classified under indirect or overhead expense, shovel-operating expense and locomotive-operating expense. To illustrate the use of the charts assume that six shovels, each operating two shifts, handle in 24 hr. a total of 12,000 cu. yd. of material, or an aver-

[1] J. M. Anderson, E. & M. J., Jan. 27, 1917.

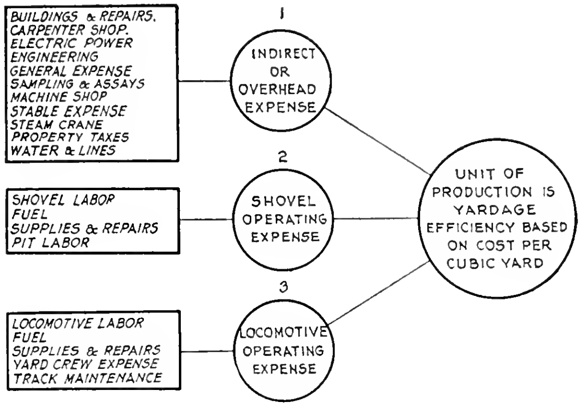



Fig. 52.—Steam shovel cost distribution chart.

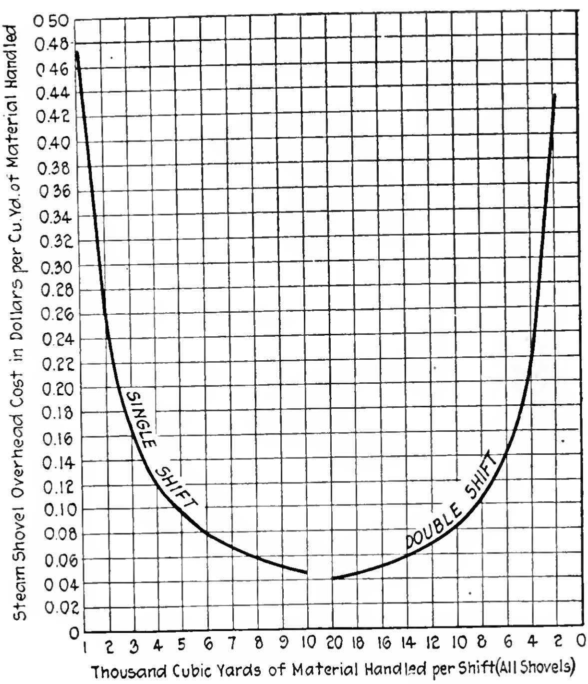



Fig. 53.—Steam shovel overhead cost chart.

age of 1000 cu. yd. per shovel shift. Reference to Fig. 53 shows that with all shovels working double shift (viz., six) loading an average of 12,000 cu. yd. per shift, the overhead expense of production at this rate is $0.072 per cu. yd. This cost chart is based upon the total overhead cost for six shovels per single-shift month of $14,217.98, or $25,645.66 per double-shift month, or of $473.93 per single-shift and $854.85 per double-shift day. The items aggregating these totals are as follows. The component factors are expressed in percentage of single and double shifts.

Steam-shovel Overhead Costs

| | Single-shift Per cent. | Double-shift Per cent. |
|---|---|---|
| Water and water lines | 2.63 | 8.20 |
| Building repairs | 3.50 | 7.34 |
| General expense | 63.00 | 59.80 |
| Stable expense | 2.74 | 2.20 |
| Machine shop | 9.60 | 7.17 |
| Electric power | 4.43 | 2.18 |
| Engineering and surveying | 4.22 | 3.40 |
| Sampling and assaying | 1.62 | 3.77 |
| Steam-crane expense | 2.64 | 2.34 |
| Carpenter shop | 0.40 | 0.26 |
| Property taxes | 5.22 | 3.34 |
| | 100.00 | 100.00 |

The operating cost is found from the chart, Fig. 55. At the average rate of production of 1000 cu. yd. per day the cost per cu. yd. is $0.051. The cost curve is based on the average cost per steam-shovel shift for the entire year 1914, during which a total of 4617 shifts were worked at the following cost.

Steam-shovel Operating Costs

| | Total cost | Cost per shift |
|---|---|---|
| Shovel labor | $64,868.57 | $14.05 |
| Fuel | 65,941.92 | 14.28 |
| Supplies and repairs | 55,258.73 | 11.96 |
| Pit labor | 51,523.68 | 11.16 |
| | $237,592.90 | $51.45 |

The locomotive service cost per steam-shovel shift is found from the chart of Fig. 54. Assuming that an average of two locomotives attend each shovel the cost of this service is $0.084 per cu. yd. of material. The chart is based upon average costs, which include all accounts chargeable to transportation, for the entire year 1914, as follows:

LOCOMOTIVE COST PER SHOVEL-SHIFT

| | Cost per 9-hr. shift |
|---|---|
| Locomotive labor | $11.28 |
| Fuel | 14.30 |
| Supplies and repairs | 3.83 |
| Yard crew expense | 0.88 |
| Track maintenance | 11.42 |
| Total cost of one locomotive | $41.71 |
| Cost of two locomotives | $83.42 |

The cost per cubic yard then becomes: overhead, $0.072; operating, $0.051; locomotive service, $0.084; total, $0.207.

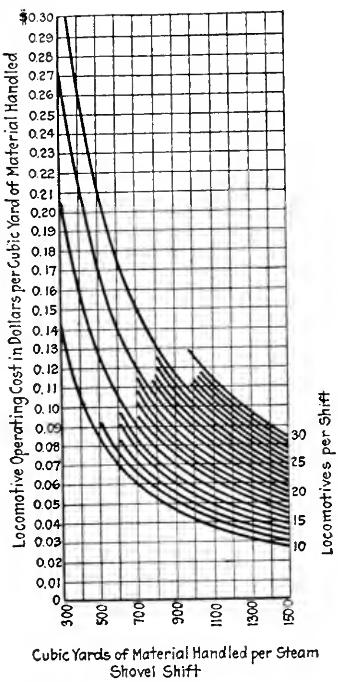



FIG. 54.—Locomotive operating cost chart.

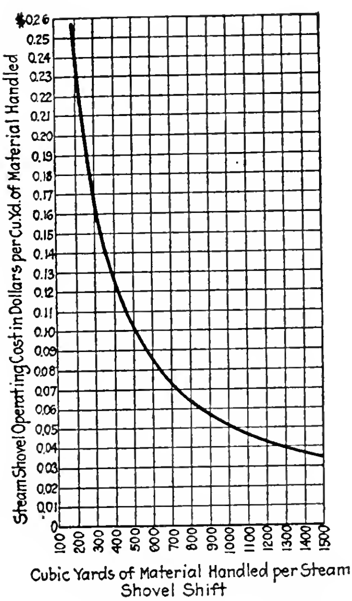



FIG. 55.—Steam shovel operating cost chart.

This cost is not the complete cost, however, as such items as drilling and blasting and dump labor must be handled separately for ore and waste. The charts should be replotted as often as governing cost changes warrant, otherwise they would give erroneous results. From these results a summary chart may be plotted and kept handy for ready inspection, which will show from day to day and cumulatively how the cost and efficiency of the work is going.

It will be noted that variations in the numbers of shovels operating, due to repairs or other causes, will affect the results from these curves. Furthermore the best balance of equipment may be determined in this way, since it will be seen that in case the train service to any shovel is poor, the yardage of that shovel will be materially reduced, running up the cost per cubic yard for that particular shovel; and again, if too much train service is assigned to a shovel it may be quite possible to increase the yardage from that particular shovel, but only at a high total train service expense, so that the cost per cubic yard will again be higher than where a better balance is maintained. With graphic charts of this kind it is not difficult to educate foremen to a keen appreciation of the economics of the problem, and they are quick to put this into practice.

For the use of operating officials and for means of comparison a consolidated daily graphic chart may be kept showing the following data.

1. The total operating cost per cubic yard of stripping and per ton of ore. The total ordinates of such a curve will be built up of:

(*a*) Supervision and engineering...............—Color "A"
(*b*) Drilling and blasting (based on average cost)—Color "B"
(*c*) Steam shovel operations...................—Color "C"
(*d*) Pit haulage...............................—Color "D"
(*e*) General expense...........................—Color "E"
(*f*) Renewals..................................—Color "F"
(*g*) Water supply and miscellaneous............—Color "G"
(*h*) Taxes (to be added in case of ore bearing same)..................................—Color "H"

2. Cubic yards per steam shovel shift (day and night).
3. Cubic yards per locomotive shift.
4. Delays in percentage of total time in service.

The delays may likewise be plotted with cumulative ordinates in colors representing delays due to:

(*a*) Blasting..................................—Color "a"
(*b*) Water, moving, repairs and miscellaneous...—Color "b"
(*c*) Waiting for ore cars.......................—Color "c"
(*d*) Waiting for waste cars....................—Color "d"

5. Added to 4 may be a line indicating percentage of overtime worked
6. Ratio of locomotive shifts to steam-shovel shifts.
7. Average number of cars in service per locomotive.
8. Total cubic yards overburden removed.
9. Total tons ore removed.
10. Average grade of ore removed.

At the end of the month and end of the year, the averages of these results may be struck for comparison.

**Office Tabulations.**—For the purpose of office records monthly tabulations of results may be made up which will show both the total costs and the costs per cu. yd. and per ton. Such statements will be found very useful.

TABULATION I

This will show the results by months and by years of the following items:

Yardage—total cubic yards overburden (or tons of ore shipped).
Steam shovels—total shifts and average number in service.
Steam shovel operation.
  Total shovel labor and cost per cubic yard or ton.
  Total fuel and cost per cubic yard or ton.
  Total supplies, repairs and cost per cubic yard or ton.
  Total pit labor and cost per cubic yard or ton.
Locomotive and train crew service.
  Total locomotive labor and cost per cubic yard or ton.
  Total locomotive fuel and cost per cubic yard or ton.
  Total supplies and repairs and cost per cubic yard or ton.
  Total yard and train crews and cost per cubic yard or ton.
  Total dump labor and cost per cubic yard or ton.
  Total track maintenance and cost per cubic yard or ton.
  Total car repairs and cost per cubic yard or ton.
Supervision.
  Total superintendents and foremen and cost per cubic yard or ton.
  Engineering and surveying and cost per cubic yard or ton.
Drilling and blasting.
  Total drill expense and cost per cubic yard or ton.
  Total labor and cost per cubic yard or ton.
  Total explosives and cost per cubic yard or ton.
Water supply—total cost and cost per cu. yd. or ton.
Miscellaneous maintenance, buildings, repairs, etc.—total cost and cost per cu. yd. or ton.
Renewals, reserve fund—total cost and cost per cu. yd. or ton.
General expense—total cost and cost per cu. yd. or ton.
Total orebody stripping cost deferred and cost per cu. yd. or ton.
Total track or other costs deferred.
Total cost deferred.
Less deferred costs charged to mining.
Ledger balance deferred charges account.

To the ore account may be added:

Total taxes and cost per ton (dry weight).
Total pit pumping and cost per ton.
Total operating cost and cost per ton.
Total ore cost and cost per ton.
Totals and averages to date.

A different arrangement and division of the detailed cost of operations may be carried from the accounts to tabulations prepared to show the details of Labor, Supplies and General and miscellaneous expenses distributed to the various operations in the following way:

The headings for the different columns will designate the different operations: Steam shovels, Locomotives, Track repairs, Blasting, Well drills, Water lines, Car repairs, Dumps, Pumping, General expense, Miscellaneous, and the last two columns will be Total cost and Average cost per cu. yd. (or per ton). Each of these columns will carry on their right a column called Acct. No. —, which will give the number of the account from which the amount is taken.

Running down the chart in the margin will be, first, the Labor items: Supt. and foremen, Shovel engineers, Shovel cranemen, Shovel firemen, Pitmen, Locomotive engineers, Locomotive firemen, Locomotive brakemen, Yardmasters and switchtenders, Trackmen, Drillers and blasters, Dumpmen, Blacksmiths and helpers, Carpenters and helpers, Machinists and helpers. Laborers, Teamsters. A horizontal line will here be drawn and the total Labor cost and cost per cu. yd. (or ton) will be footed up for each of the columns. Following under this in the marginal column items of Supplies will come: Pipe & fittings, Iron & steel, Explosives, Steam shovel and locomotive parts, Oil, waste, etc., Tools, Railroad and drill supplies, Fuel, Dump car parts, General mine supplies. A second horizontal line drawn here will foot up the total cost of Supplies and show the cost per cu. yd. (or per ton) for each of the columns. Following under this in the marginal column will come the General and Miscellaneous items: Stable expense, Shop expense, Steam crane, Electric power, Building repairs, Water lines, Engineering and surveying, Taxes, Steam shovel and locomotive renewals, General expense, Sampling and Assaying and Miscellaneous. A third horizontal line will be drawn here and the General and Miscellaneous total amounts and amounts per cu. yd. (or ton) will be footed up. A fourth and final horizontal line will then be drawn and the grand total amounts and amounts per cu. yd. (or ton) will be footed up for all items.

A recapitulation of the above may be added at the bottom in this form.

| | Costs this month | |
|---|---|---|
| | Total Amount | Per cu. yd. (or dry ton) |
| Pay roll.................................. | | |
| Supplies.................................. | | |
| General expense........................... | | |
| Miscellaneous............................. | | |
| Total cost................................ | | |
| Number of | Quantities this month | |
| Cu. yd. overburden........................ | | |
| (Dry tons ore)............................ | | |
| Shovels employed (individual shovel numbers).. | | |
| Cu. yd. per shovel........................ | | |
| (Dry tons per shovel)..................... | | |

**Monthly Statistics.**—A monthly statistical chart, giving the information indicated in Table 19 below will be found very useful and interesting. These items may be rearranged or added to in any way desired by the local management. Careful intelligent study and analysis of costs will invariably repay all effort and expense so spent and is one of the best methods of picking out weak spots, reducing the costs and increasing the efficiency of the work. The accounting and engineering departments should work in close harmony with the management in studies of this kind.

All of the foregoing data will form the basis on which periodical reports will be submitted by the mine management to the general management, and then in more condensed form to the property owners.

TABLE 19.—MINING DEPARTMENT, STEAM SHOVEL MINING DATA AND STATISTICS

| | 12 months, last year | This year | | | |
|---|---|---|---|---|---|
| | | Jan. | Feb. | March | April |
| **Yardage** | | | | | |
| Ore—dry tons shipped | | | | | |
| Ore—cu. yd. (1 cu. yd. in place = 2.16 dry tons) | | | | | |
| Overburden to dumps (cu. yd. in place) | | | | | |
| Total yardage moved (cu. yd. in place) | | | | | |
| **Steam Shovels** | | | | | |
| Steam shovel shifts, 9 hr. in ore | | | | | |
| Steam shovel shifts, 9 hr. in waste | | | | | |
| Average yardage per shovel shift operating in ore | | | | | |
| Average yardage per shovel shift operating in waste | | | | | |
| Tons of coal used by steam shovels | | | | | |
| Cost of shovel coal per ton (F. O. B.) | | | | | |
| Tons of coal used per shovel shift | | | | | |
| Lb. of coal per ton of ore shipped | | | | | |
| Lb. of coal per cu. yd. of waste moved | | | | | |
| **Locomotive** | | | | | |
| Total shifts of 9 hr. in ore | | | | | |
| Total shifts of 9 hr. in waste | | | | | |
| Tons of coal used on locomotives | | | | | |
| Tons of coal per locomotive shift | | | | | |
| Cost of locomotive coal per ton (F.O.B.——) | | | | | |
| Lb. of coal per ton of ore shipped | | | | | |
| Lb. of coal per cu. yd. of waste moved | | | | | |
| Cost per ton-mile of material handled | | | | | |
| **Well Drills** | | | | | |
| Drill shifts of 9 hr. worked | | | | | |
| Number of blast holes drilled in ore | | | | | |
| Number of blast holes drilled in waste | | | | | |
| Number of feet drilled in ore | | | | | |
| Number of feet drilled in waste | | | | | |

| Well Drills (cont.) | 12 months, last year | This year | | | |
|---|---|---|---|---|---|
| | | Jan. | Feb. | March | April |
| Lb. of coal per foot drilled.......... | | | | | |
| Lb. of coal per ton of ore shipped.. | | | | | |
| Average footage per 9 hr. shift...... | | | | | |
| Total number 6 in. holes drilled..... | | | | | |
| Total number 6 in. holes shot....... | | | | | |
| Tons of ore broken per foot of hole drilled......................... | | | | | |
| Cu. yd. waste broken per foot hole drilled......................... | | | | | |
| Explosives | | | | | |
| Lb. of principal H. E. per 1000 cu. yd. waste.................. | | | | | |
| Lb. of each other type used per 1000 cu. yd. of waste................ | | | | | |
| Lb. of prineipal H. E. per 1000 tons of ore shipped.................. | | | | | |
| Lb. of each other type used per 1000 tons of ore shipped............... | | | | | |
| Cost exploders per 1000 cu. yd. waste or ore.................... | | | | | |

| Lubrication | 12 months, last year | This year | | | |
|---|---|---|---|---|---|
| | | Jan. | Feb. | March | April |
| Gal. lubricating oil per ton ore shipped......................... | | | | | |
| Lb. grease pĕr ton ore shipped...... | | | | | |
| Cost of lubricating oil and grease per ton ore shipped.................. | | | | | |
| Gal. lubricating oil per eu. yd. waste........................... | | | | | |
| Lb. grease per eu. yd. waste........ | | | | | |
| Cost lubricating oil and grease per cu. yd. waste................... | | | | | |

Record month for material moved
Best record for one shovel in one month

| | | | | | |
|---|---|---|---|---|---|
| Tons of material handled per man per shift...................... | | | | | |

The mine accounts will be carried on cost sheets, the following outline of which may serve as a fair example.

1. Pit stripping deferred.
2. Equipment and miscellaneous construction deferred.
3. Pit mine cost sheet.
4. General expense.
5. Shovel and locomotive renewals.
6. Taxes.
7. Profit and loss.
8. Accounts receivable.
9. Inventory of supplies—material in transit and reconciliation of supply ledger.
10. Boarding house.
11. Machine shop.
12. Electric shop.
13. Stable or truck operations.
14. Churn drill prospecting and developing.
15. Steam crane expense.
16. Pay roll distribution.
17. Segregation of costs as desired.

**Annual Reports.**—The annual reports of mining companies are widely variable but some of the best of them prepare accountant's reports which show the following information, and the records kept should be such that this may readily be obtained.

Exhibit "A"

Statement of Assets and Liabilities.

Exhibit "B"

Statement of Operations.

This will briefly show operating revenue, operating expenses, miscellaneous income and receipts, other charges, and a surplus or deficit from such operations.

Exhibit "C"

Current assets and current liabilities and investments.

The investments will be detailed at face and book values.

Under Exhibit "A" may be prepared a schedule showing composition of increase or decrease in surplus funds. Also a schedule showing "Prepaid ore expense" due to stripping of overburden ahead of ore removal.

Any additions to property or plant should be set forth.

A careful study of the annual reports of the companies whose costs have been herein illustrated will serve as an excellent guide as to the material required for an intelligent clear report to property owners.

# CHAPTER VIII

## ADMINISTRATION

**Introductory.**—The administration of open pit mines is of course based on the same general lines as is that of any other mining enterprise, but it is more usual to find the life of open pit mines longer and better defined than in the case of most underground mines. In other words they are usually more fully developed than underground mines. This makes it possible to treat them more as great industrial enterprises, and as such more care and money can be expended on the plans for their operation.

There are a number of general problems which should be determined as early as convenient, which will have a bearing on the administrative policy of the property. Among these may be mentioned the ratio of output desired to be maintained with a known developed and probable tonnage, and as a corillary to this how much capital may justifiably be spent in equipment to realize such a ratio of output; the lowest grade of ore that can be considered profitable to treat when by treating it the treatment of higher grade ore is postponed; the amount of prepaid stripping that may justifiably be carried to insure steady working conditions and an average grade of ore. The labor problem will also require the constant care of the management.

**Ratio of Output.**—Theoretically, the value of a property is directly proportional to the speed with which its latent value may be converted into actual money. The present value of a dollar payable in one year is double that of a dollar payable in twelve years, and four times that of a dollar payable in twenty-four years (interest at 6 per cent.). Likewise, a value which cannot be liquidated in less than forty years is not worth ten cents on the dollar today. Furthermore extensive large scale operations usually result in low production costs because of a large divisor for all fixed and general charges not proportional to production.

On the other hand there are practical and physical limitations to maximum production which usually necessitate some sort of a balance or compromise with the theoretical view. The greater the planned production, the greater will be the amount of invested capital required. This will be tied up in all sorts of equipment, in the necessary extensive development, and in many cases in a large amount of prepaid stripping. It must be remembered that this bears interest also, so that a dollar invested today at 6 per cent. will be worth two dollars in twelve years and so on.

Next the physical characteristics of the mine may easily be such that too intensive a production would only tend to inefficiency due to cramping or crowding of working conditions in the available territory. This might entail losses of valuable ore which could not be taken out in keeping pace with the general program.

Third, there is the question of supply and demand of the commodity produced. The price of most commodities is subject to considerable fluctuation and many properties find it necessary to somewhat govern their production in an effort to maintain a fair price for the product. With an excess of production over demand, prices usually "soften" so that it may easily take three tons of ore to show the same profit as would be had from two tons if the demand was just met by the supply. At the present time taxation is such that excess profits, which might in certain cases be made by maximum production, would be considerably reduced. In so far as market prices are concerned, a property might be operated at a high rate of production with a view to storing the commodity during low prices and selling heavily at high prices, but there will then be interest charges to consider on the cost of production as well as the actual money so tied up and this will tend to cause wide fluctuations in the dividend paying power of the property. Stock in most metal producing companies is rather widely held and subject to such various policies of administration that concerted action cannot be expected even if anti-trust laws were not in effect to prevent collective action.

In the final analysis therefore a production figure will usually be decided upon which will be high enough to secure good operating cost and permit of first-class equipment, insuring a good rate of interest on the total estimated and available capital investment. In planning such an average output it will, of course, be wise to so arrange the operations and plant that they have some economic flexibility, and, if later conditions warrant, that the plant and output can be extended economically.

*Example.*—The solution of a problem of this kind presents some interesting features and a hypothetical example will be given to illustrate the method used by the writer. In this problem it is assumed that we have a property producing about 9000 tons per day and having a developed tonnage of about 60,000,000 tons,

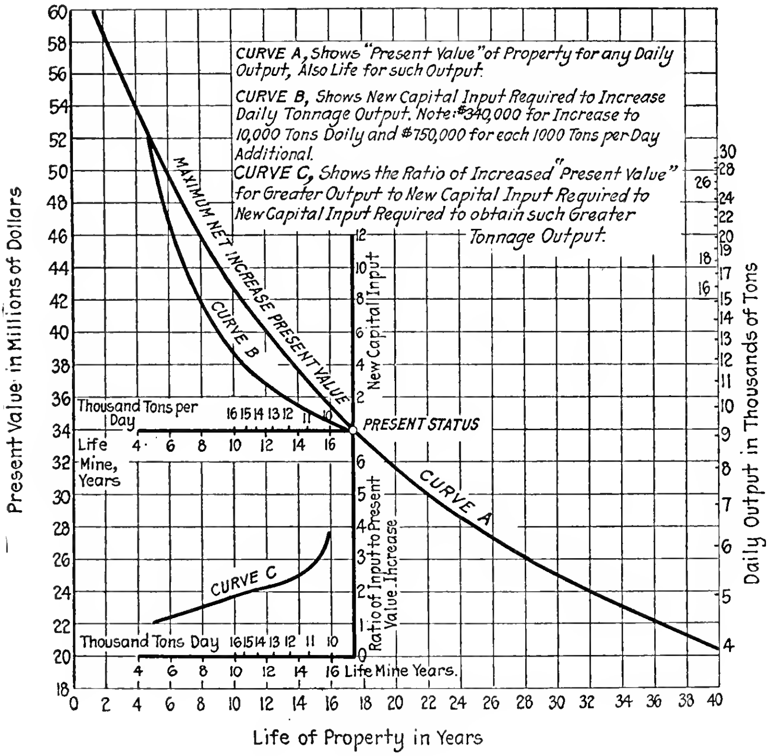



FIG. 56.—Valuation chart; ratio of output.

which would give it an assured life of about 18 years and a present value of about $34,000,000.[1] It is further assumed that with an expenditure of only $340,000 the plant could be increased to an output of 10,000 tons per day, and that for each additional expenditure of $750,000 it could be increased in increments of 1000

[1] It is assumed that the method of arriving at the "present value" of a property is understood.

tons per day. It is desired to determine whether or not it is justifiable to increase the production and if so to about what extent. The curves on chart, Fig., 56 have been drawn to show the solution graphically. Curve *A* of this chart is so plotted that by noting any ordinate on the right margin and then running over to the left to the intersection of this curve, and from this point dropping down to the bottom abscissa, the life of the property in years will be found; or, by simply continuing across to the left hand margin, the "present value" of the property will be found. For example, with a daily output of 12,000 tons, the life will be about 13 years, and the present value will be about $38,000,000.

A set of co-ordinates was drawn from a point on the curve corresponding to the present output of the property of 9000 tons per day. The auxiliary ordinate is plotted to represent the new capital required to increase the daily output and the auxiliary abscissa to represent the life of the mine with various daily outputs. On this basis curve *B* was plotted. The difference in the ordinates (on the left hand margin) between curves *A* and *B*, shows the net increase in the present value which can be made by additional plant capacity up to 30,000 tons per day. At 30,000 tons per day, curve *B* crosses curve *A* and would there show no gain in present value and a higher output would show a loss. Curve *B* shows that the maximum net increase in present value is reached when the output is increased to 16,000 tons per day. This net increase is about $4,000,000, for an expenditure of $5,000,000, or the gross increase in present value is $9,000,000. Curve *C* has been plotted merely to show the maximum increased present value accruing with new capital input increments. In other words, curve *C* simply shows the ratio of the net increase in present value to the necessary new capital expended in increasing the output to increase the present value. For example, by increasing the output by 10 per cent. with the expenditure of $340,000 the present value is increased by $1,400,000 or the ratio is over 4 to 1. Beyond 'this output, the expenditures are assumed to be heavier per 1000 tons increase capacity, so that the curve flattens out. It is shown, however, that by increasing the tonnage output to 14,000 tons per day, this ratio is still as high as 2 to 1, and for 12,000 tons per day it is 2.3 to 1; in other words for every new dollar spent in increasing the plant capacity to 12,000 tons over 9000 tons, the present value of the property is increased $2.30. These curves automatically take care of the interest charges on the new capital invested, except for the period between beginning the expenditure and the time at which benefit from the increased production can be realized. This would be a relatively insignificant amount.

In the same way a set of curves may be plotted covering any problem when the necessary data are as accurately known. In this example, unless some practical reason due to operating conditions forbids, it would be advisable to increase the output of this property up to, say, 16,000 tons per day, at which figure the maximum net increase in present value is shown to be attained.

**Lowest Grade of Ore that Should be Worked.**—It is necessary to determine the lowest grade of ore that can be mined at a profit under given costs of mining and overburden ratio; or stated another way it is necessary to know how much can be expended on mining and stripping ore of a given tenor. In all pits there are variations in the ore-overburden ratio and in the tenor of the ore; there are also variations in the selling price of the product which may materially effect the economics of the question, but to plan work ahead, price of commodity and cost of working must to a certain extent be assumed. In Chap. VI this subject was discussed and curves were given to illustrate. It is not only necessary to avoid shipping material on which an operating loss is borne, but it is also necessary to avoid shipping material which will not show an average profit sufficient to pay fair dividend requirements. In fact fair dividend requirements may be considered in truth a part of the operating expense. A copper mine might meet all operating expenses nicely on an ore carrying 1 per cent. copper, but to pay dividends to its stockholders it might require an ore averaging 1.5 per cent. copper.

It may be that in mining and stripping, much "border" ore will be loaded by the shovels and it becomes a question whether to put it on the dump or send it to the mill. The expense of mining it has already been met, to put it on the dump will entail some additional expense and it will therefore only have to stand trans-transportation and treatment costs over and above the cost of wasting it. The mine may have plenty of capacity to load all mill requirements and the mine management may not be especially interested in what becomes of the product after it has been moved, but it must be remembered that the rest of the plant has a limited capacity and that for every ton of poor material put through, a ton of average material has been displaced. All of these factors must be considered if the earnings are to be kept up. The ideal arrangement would be to keep the mill full of high grade material as long as it lasted and as this became exhausted lower grade material could then be treated. In this way, other things remaining constant, the maximum money would be produced in the shortest time, and this would be available to draw interest. Such an arrangement cannot be broadly met due to the very nature of the mining operations and to the necessity of conserving ores which are to a fair degree profitable. Even with mines having a great length of life, the justice in a reasonable policy of conservation comes into consideration along with the academic economics. It sometimes happens, however, that low grade material can be segregated in dumps apart from the straight waste, and these may be reserved for later treatment. If so, the plant may be enabled to produce more wealth in a short time, making the property of greater "present value," and the low grade material may also be saved and some day may add considerable value to the property. In this question, as in most others, it is a question of compromise, but the compromise should be based on a full understanding of the consequences. The graphic solution of the problem as explained in Chap. VI has been found very helpful in educating mine operatives to an appreciation of the consequences and when they are understood there is much less tendency to ship material known to be poor.

**Amount of Prepaid Strippings.**—This problem is somewhat comparable to the extent of development in vein mines, not so much in the light of adding to ore reserves, which may better be called prospecting, as to the opening up of tonnage ready for extraction. It has been the policy at some of the Mesabi iron mines to completely strip the ore body before extracting the ore. Such work has often been done by contractors or equipment not needed elsewhere has been put to work on such jobs. Under certain conditions this policy may be entirely justifiable and it is certainly advantageous in planning the ore mining, but it does tie up considerable money, often in excess of that actually required to carry on with the required ore extraction. Furthermore, stripping banks left exposed to the weather for a number of years have a tendency to slough off into the pit and may require more or less cleaning up and repair. Stripping should, however, be carried far enough ahead to insure safe, clean and unhampered mining of average grade ore. The waste should be moved far enough back to permit the ore benches being carried in the best practical manner for blasting and loading. Furthermore, with the stripping carried well ahead of mining operations so that it could be discontinued for say six months without inconvenience, it will be found of much advantage in the event of labor troubles, or an unexpected shortage of labor or supplies, or accidents to shipping equipment. To justify carrying it beyond such requirements or contingencies, there must be some good local reasons; otherwise, the interest charge on the sum so spent will not be warranted. In some localities, due to climatic or other local conditions, stripping can best be done periodically and where these conditions are met, the advance stripping may at times be considerable. It is usually found cheaper to crowd stripping in in summer and work it but lightly in winter. Also it may be found that labor or supplies are cheaper at certain periods than at others and it may then pay to push the work when these conditions are most favorable. It may also prove desirable to have several pits or portions of a deposit opened simultaneously, in which case the prepaid stripping will probably be greater than if the entire tonnage was taken from one pit. The objections to excess stripping are simple and against them only must be weighed local conditions. Care and study should be given that the advance stripping is removed from the most advantageous areas.

**Engineering Work.**—The careful surveying and mapping of the property should be kept up. All of the month's shovel work, both in ore and waste, is cross-sectioned every month. Stadia work is sufficiently accurate for the regular monthly reports, but once every six months the work should be done by triangulation. The cross-sections are taken at intervals of from 27 to 100 ft., the latter is the practice at the copper properties, while 40 ft. is the distance used on Mesabi iron mines. The 27-ft. interval is convenient for direct yardage results. These results are plotted to scale on cross-section cloth, and the areas are determined by planimeter or triangles. The cross-sections are always taken at the same places and are plotted in different colors so that each month's work is easily seen. The engineer's estimates are checked by the railroad weights for the ore, and with the yardage capacity of the waste cars. Much of the cost and technical data are based on the engineering work. Future stripping plans will usually depend in large measure on an accurate knowledge of the orebody. For this purpose sectional models are very useful. Such a model may be made up of 1-in. boards set vertically on a scale of 1 in. per 100 ft. (horizontal and vertical). On one face of each board may be pasted a blue print cross-section of the deposit at that point. Such cross-sections will have been made up of the drill hole and surface data. As the excavation progresses these boards may be drawn from the model box and cut out to conform to the pit conditions. Such a model will also serve to indicate grade of ore expectancies. Bench sampling should be carefully and frequently done and the results of the assays quickly reported. When a shovel is working in "border" ore the samplers should be most vigilant. This work will invariably save the treatment of unpayable material and often the wastage of good ore.

Time studies of certain phases of the work will often be useful in securing data for new estimates.

**Labor.**—Most of the labor employed about open-pit mines is unionized, and, with the exception of pit and dump gangs, is of an intelligent type. The work of the different services is closely interdependent so that inefficiency in one is almost sure to affect the others. As a rule the shovel-workers' unions have displayed a spirit of fair play. The train crews are similar to those found on railroad operation. These men may be encouraged by a system of bonus payments such as used on the Mesabi range. At times this is effective in raising efficiency. Profit-sharing in various ways is now being tried by some companies but that is a subject in itself. Reports may be prepared each month showing the average yardage handled by each shovel runner, each locomotive crew; the footage of hole drilled by each driller; and the condition in which each operator has kept his machine. These are very good both in keeping up the discipline of the work through inspection and in being able to rate the efficiency of the crews through the records.

It is highly desirable to keep the labor turnover to a minimum. New men are as a rule of decidedly lower efficiency than the old crews, familiar and educated to the requirements of the job. New men must become "acclimated," as it were, to all of the local conditions, and many even to the ways of performing their tasks. For this education the company must pay.

The betterment of quarters, food and places of amusement or recreation for workmen has a very marked effect in securing the best men, especially if they have families, and married men with families are usually more contented and steadier. Wherever possible, the family is to be encouraged. The foremen should be kept from too close contact with their crews when off duty; it makes for better discipline.

The safety of the workmen is a subject which must constantly be looked to. There is always present a certain percentage of men who are careless or ignorant of their dangers. Safety engineers are employed by some companies to make careful studies of all accidents, of exposed machinery parts, shops, handling and care of explosives, railroad grade-crossings and many other similar things. These men should work in close co-operation with the medical officers as well as with the managment. The elimination of intoxicants has been found to decrease accidents as well as increase the general efficiency, prosperity and dependability of the men. In open-pit work it is especially necessary to protect men from unnecessary hardship in bad weather. Arrangement for their distribution to and from work and for reasonable shelter while at work must be planned. In the case of foreign employees who understand but little English, it is well to provide them with small instruction pamphlets written in their native language. All such rules as have to do with the handling or use of explosives, warning signals for blasting, trimming banks, repairing equipment, use of intoxicants, sanitary rules, hours of work, time of meals, recreation provision and similar subjects should be freely published and placed in all hands by the welfare engineers. These engineers should keep careful records which may be submitted periodically in condensed form to the management. It is interesting to note that the Minnesota mine inspector's report for 1910, referring to fatal accidents in underground and open-pit mining, showed 3.32 per 1000 for the former and 4.59 per 1000 for the latter. One reason given was that open-pit work more nearly resembles railroading than mining, and hence the greater percentage of fatalities. Whether this proportion would hold true for other districts is not known, but it is true that a large proportion of open-pit accidents are chargeable to the train service; misfires and premature explosions would probably come second. The time has arrived when workmen are seeking betterment of conditions, and aside from the justice or ethics of their collective stand, it is highly desirable that they be accorded as just treatment as is humanly possible. The great majority are not unmindful of a spirit of fair dealing and their loyalty and contentment is generally reflected to some degree in their work.

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