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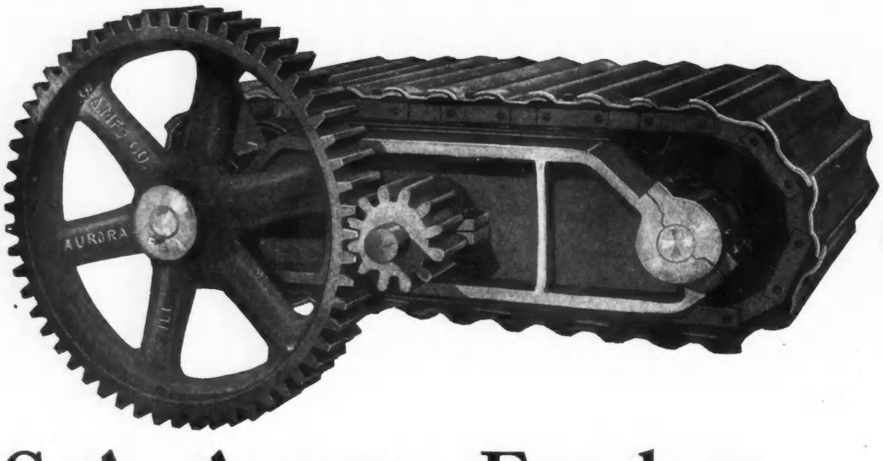

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Engineering and Mining Journal

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Methods of Underground Metal Mining

By F. W. Sperr*

SYNOPSIS—A diagrammatic classification of 162 different mining methods together with definitions of the primary and secondary subdivisions of these various methods, with the object of clarifying the existing muddle in the use of terms describing the systems used.

The universal term comprehending all methods of mining is digging. Digging coal is coal mining; digging gold from gravels is placer mining; digging stones is stone mining or quarrying; digging ores from open surface workings is openpit mining; and digging ores and metals from underground workings is mining in the most ordinary use of the term, but the specific act is called stoping and the excavation is called a stope. The supreme object of mining is profit. Where there is no anticipation of profit of some sort, there is no mining; and, as in all industrial enterprises, there is a certain element of risk to invested capital and consequent liability to reverse the attainment of the supreme object. For the attainment of his object the miner takes the risk of the liability to bodily

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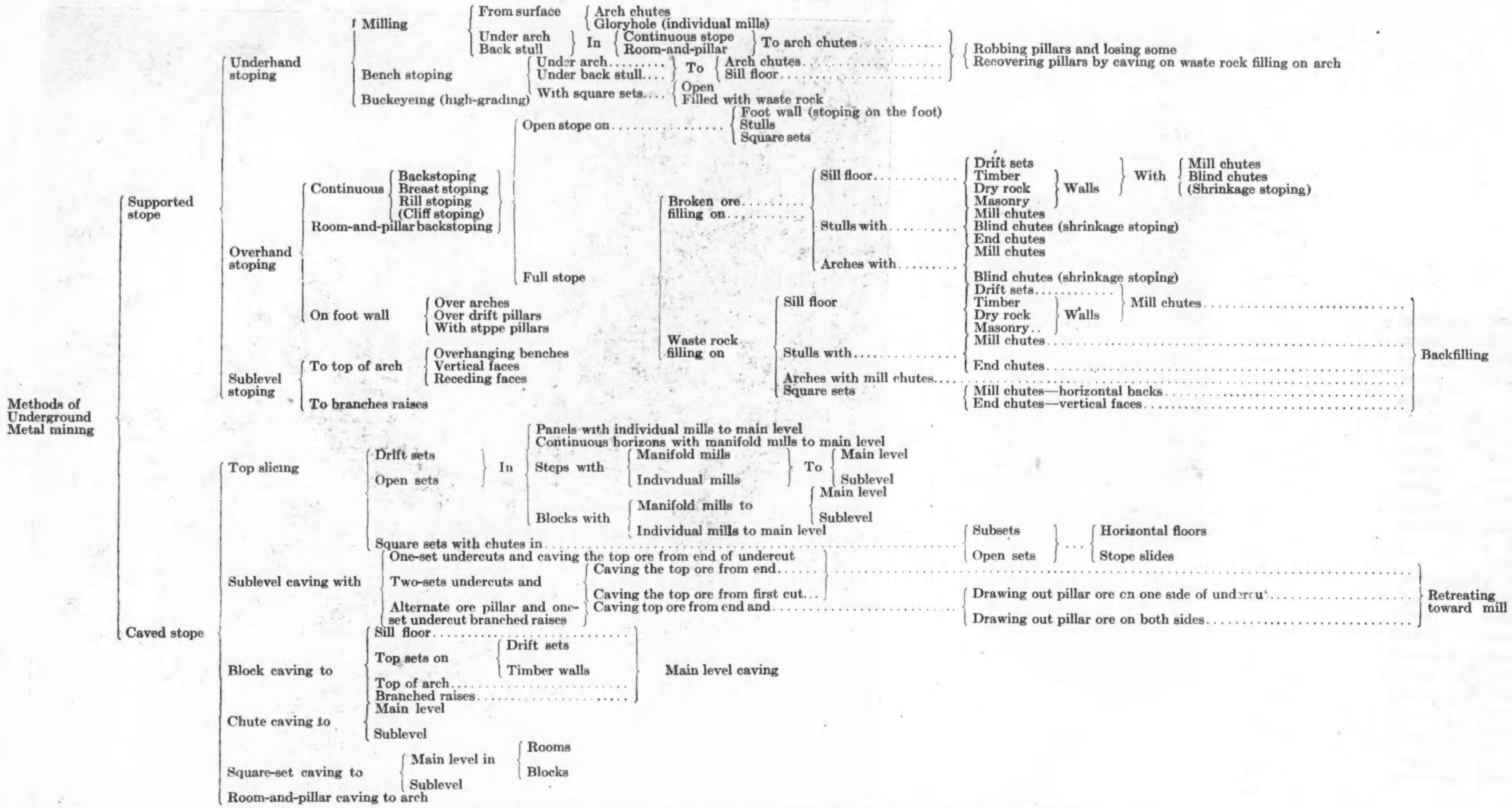
FIG. 1. UNDERHAND BENCH STOPE IN CLEVELAND CLIFFS IRON CO. MINE



DIAGRAMMATIC CLASSIFICATION OF UNDERGROUND METAL-MINING METHODS

injury and loss of capital, and therefore the supreme effort on the part of the miner, the operator or the manager will be to increase the profits and reduce the liability to bodily injury, and without sacrificing safety for profits.

The primitive miner excavated his material from the earth by simply digging a cavern. The necessities for supporting the cavern to make it safe and for making the greatest possible profit have led to the devising of a great variety of methods of mining. An excavation is self-supporting in the weakest of rocks if it is not too large and if it is not required to remain open too long. On the other hand, there is some span over which the strongest of rocks ceases to be self-supporting, and the limit of span becomes less with the extension of time. Different devices for preventing loose rock from falling do not properly constitute different methods of mining, but the different means employed under different tendencies in the rocks to close up the excavation lead to the division of methods of mining into two general systems—the supported-stope system and the caved-stope system.

The supported-stope system includes all methods in which the rocks above the ore and on the sides are required to remain standing in place until after the operations of removing the ore are completed. The caved-stope system includes all methods by which the overburden or the ore, or both ore and overburden, are made to cave and follow the excavation downward as the ore is being removed from underneath.

The differences in different orebodies in regard to mode of occurrence, texture of ore, character of the inclosing rocks and depth of the ore from surface naturally lead to the subdivision of the two principal systems into nine different principal methods and to the resubdivision of the principal methods into various minor methods.

The Principal Methods of Mining

The principal methods may be enumerated and defined as follows:

1. Underhand stoping is started from the top of a body of ore and is carried downward, the excavation being enlarged at the top with downward extension.
2. Overhand stoping is started at the bottom of a body of ore and is carried upward.
3. Sublevel stoping is carried on from small sublevel drifts above the main level or haulageway, each main level when its operations are finished becoming the top "sub" of the next main level below.
4. Top slicing is removing the ore from the top of an orebody downward in successive slices in unit operations and allowing the overburden to cave to the floor of each successive slice.
5. Sublevel caving is caving the ore from sublevel to sublevel by means of undercuts supported by drift sets on each successive sublevel, each main level in succession becoming the top "sub" of the next main level below.
6. Block caving is caving the ore by a single undercut for each main level, the undercut being made on the main level (sill floor), or on an arch above the main level, or at the top of a system of branched raises from the main level.
7. Chute caving is caving the ore in funnel-shaped stopes from individual chutes either on the main level or on sublevels, escape holes being maintained from the tops of the stopes for the access and retreat of the miners.
8. Square-set caving is mining the ore on square sets until the weight of the overburden or capping rock becomes excessive; then the legs of the bottom sets are blasted out and the overlying rock is allowed to cave into the excavation.
9. Room-and-pillar caving is backstoping rooms on broken ore (by shrinkage stoping) and caving the room pillars and the overlying arch, together with the stopes full of broken ore, drawing the ore through arch chutes at the main level.

Among the secondary subdivisions of mining methods the following definitions will lend clarity to the subject:

Milling is breaking ore by underhand stoping into mill-holes that lead to loading chutes on an underground level or haulageway. The stope may start from surface on an outcrop of ore or on an orebody from which the overburden has been stripped, or under an arch of the ore or capping rock, or it may start under a back stull built in for protection against the falling of loose fragments from the arch.

Arch chutes are loading chutes on a haulage level that are fed through mill-holes which ultimately become small, funnel-shaped, underhand stopes in the top of the arch above the level. At first the mill-holes are placed as far apart as is necessary to secure a maximum run of ore to each chute; then other mill-holes are raised half way between the preceding ones, until the ridges of ore (hog-backs) between chutes are reduced to small dimensions.

A gloryhole is a large underhand stope developed from surface in mining a lenticular- or massive-shaped orebody through an individual mill-hole to a chute under the inverted apex of the orebody.

Bench stoping is breaking the ore in a succession of more or less well-defined horizontal benches, one under the other, from the top of the stope to the bottom. It is similar to milling from which it differs most when the ore is broken to the sill floor and shoveled into the tram-cars instead of being drawn from chutes; and when square sets are used, either open or filled, to sustain the overlying rocks.

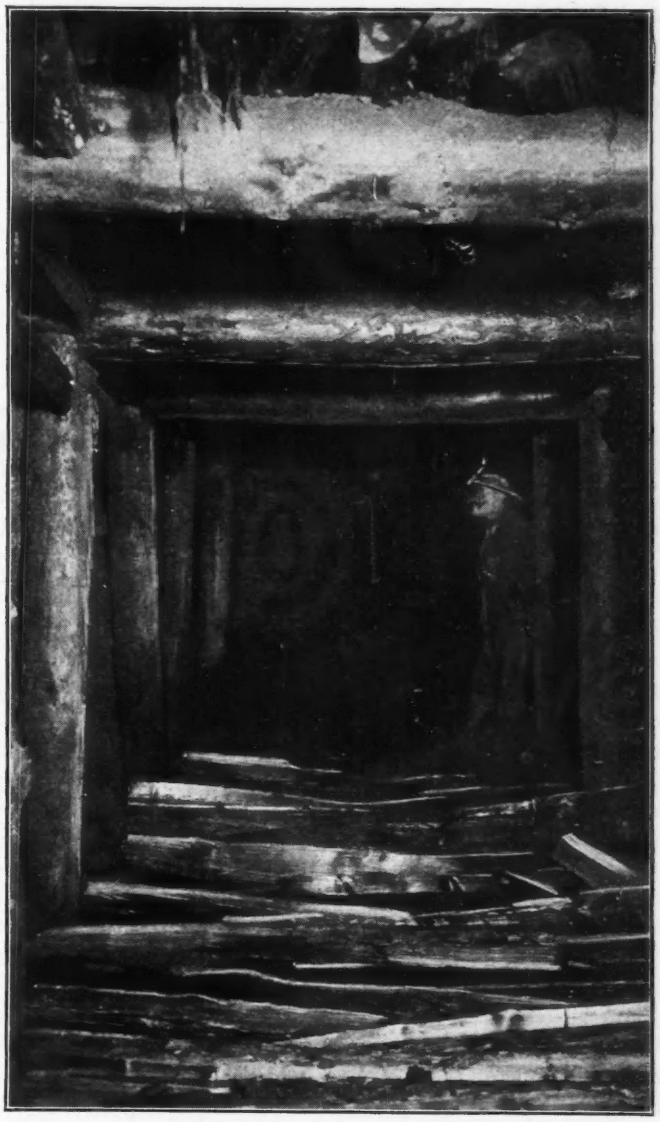


FIG. 2. SHOWING METHOD OF LAGGING FLOOR IN TOP-SLICING DRIFT

Raise stoping is extending upward the apex of an overhand stope. This operation in itself does not constitute a method of mining. Back stoping is cutting successive horizontal slices from the back or top of the stope, starting from a raise or raise stope. Breast stoping is breaking the ore in successive steps by means of breast holes inclined slightly downward. Rill stoping or cliff stoping is like back stoping, except that the face of the stope is inclined instead of horizontal, and the drill holes are pointed at a high angle instead of horizontally or slightly upward as in back stoping. Breast stopes and rill stopes are carried continuously along the lode from end to end, or from shaft pillar to shaft pillar. Back stopes may be continuous along the lode or in rooms and pillars across the lode.

Continuous stoping may temporarily make use of shaft and floor pillars, and of arches in lodes with an angle of dip of, say, 50 deg. or more. Provision is supposed to be made for the recovery of all such pillars. Room-and-pillar stoping is systematically stoping the ore in rooms of definite dimensions with intervening pillars of certain size. Provision should always be made for the recovery of all pillars, under this method of mining, or else collapse with probable disaster is sure to follow if mining is continued.

FIG. 3. OPEN PIT IN MINNESOTA IRON MINE, SHOWING MILL HOLES

Pillar-on-foot (wall) stoping is adapted to lodes having a low angle of dip, say, 40 deg. or less. Above and around the pillars the method may be back stoping, or breast stoping, or rill stoping, but not room-and-pillar; it is not open stope, because of the pillars; and it is not full stope. The complete recovery of the pillars in this method of mining, is seldom practicable. Usually the pillars are robbed and partly lost.

Buckeyeing and highgrading is gouging out high-grade material which may be discovered in any of the workings, and it is one of the favored methods for "digging the eyes out of a mine"—hence the terms.

Gophering and coyoting are indescribable methods in irregular burrows, which cannot be classified, but which belong to the supported-stope system.

There has been much confusion in the use of terms to designate methods of mining, different terms having been applied to the same method and the same term to different methods. The accompanying diagram is designed to give a consistent scheme of classification of methods. There are 162 different methods represented on the diagram taking no account of the different ways of developing, hoisting, tramming or hauling, drilling and blasting, holding up loose rock, filling stopes, clearing chutes, dealing with ore pillars, and the many special devices for executing the details of the different methods.

The methods indicated on the diagram for dealing with arch, room and floor pillars, in some of the underhand-stoping operations, are the same for dealing with such pillars when they occur in any of the supported-stope methods.

There is hardly any limit to what could be written on these mining methods and descriptions of their various combinations. A few illustrations of types will here suffice. Fig. 1 shows an underhand bench stope where the drilling is being done from a tripod rigged on a ladder platform. This method is adaptable to very hard rock. In Fig. 2 is shown a very good example of top slicing with drift sets. The method of setting legs, caps and studdles is obvious, also the manner of lagging the floor on which the gob will fall when the legs are blasted out and the weight of the capping above begins to crush the sets. The slicing is continued in width, set against set, until the pressure indicates imminent movement from above, when the legs are blasted.

In Fig. 3 is shown an openpit at Eveleth, Minn., showing underhand milling from surface. The top of the arch is 20 to 30 ft. above the haulage level. The mill holes, showing funnel shaped in the illustration, have loading chutes below. The ridges between the mills are called hogbacks. Such arches may be developed by many different methods of mining: By underhand milling; by any of the methods of overhand stoping except backfilling; by sublevel stoping to arch; and by block caving to arch.

A case of a sublevel acting badly is shown in Fig. 4. The pressure from above is developing too slowly, and the stope sets in the undercut are falling out of place, owing to the gob not caving and following the ore. Such conditions are liable to develop a sudden fall of rock, causing an air-blast that may do damage, besides the danger of caving the sublevels below.

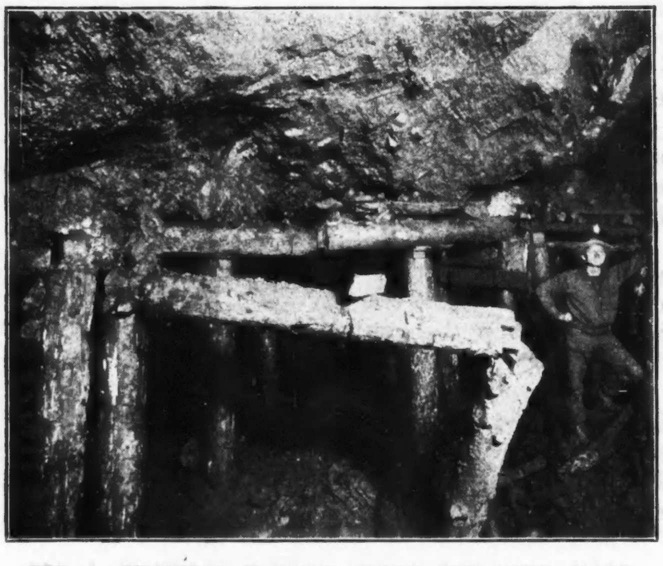


FIG. 4. SHOWING EFFECT WHEN SUBLEVEL ROOF DOES NOT CAVE PROPERLY

The success of a mining venture will often depend upon the selection of a proper mining method. An improper choice leads to high mining costs, loss of orebodies and often loss of life. It is too much to hope that any fixed system of nomenclature for mining methods may soon be adopted but it is an aim worth while and should be borne in mind by engineers before attempting to devise new descriptive titles for their work and adding to the present confusion of terms.

The Coronado Top-Slicing Method

By Peter B. Scotland*

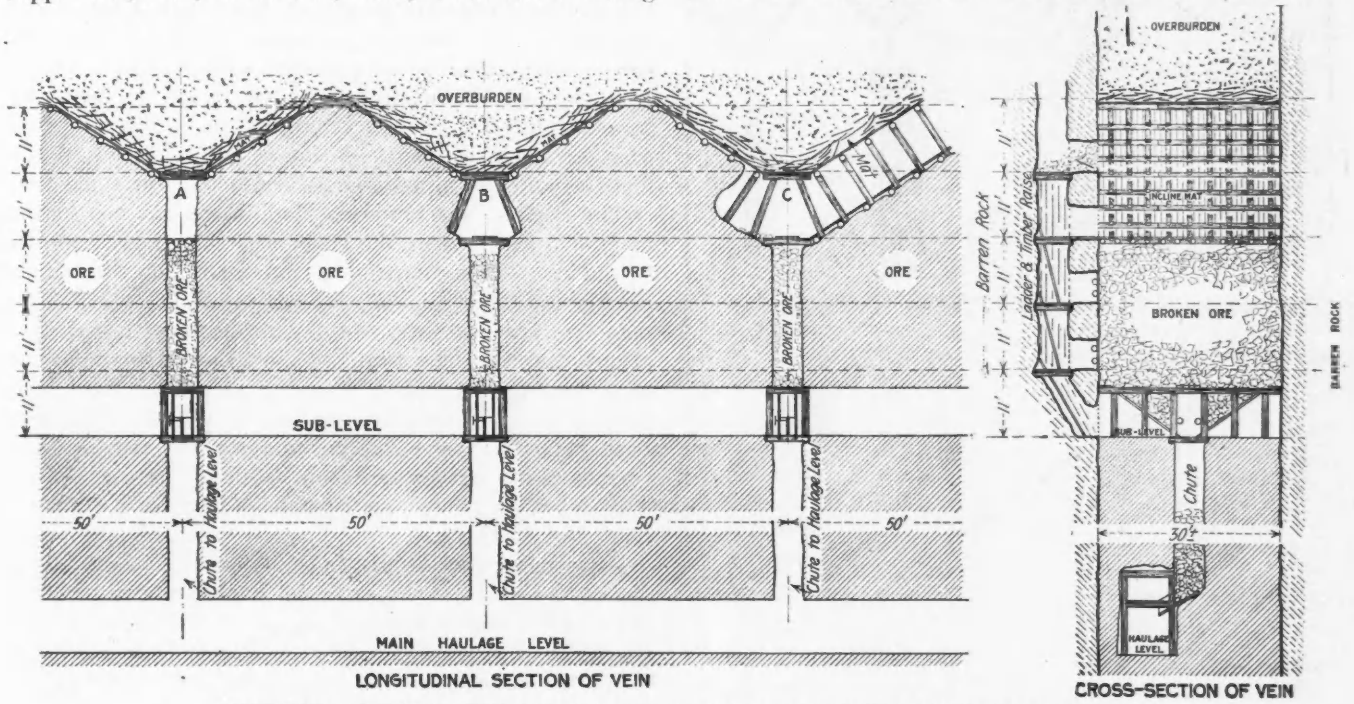
SYNOPSIS—A method of top-slicing developed by the Arizona Copper Co., to do away with the shoveling and wheelbarrow work incident to the ordinary top-slicing method of mining. The slice is developed on an incline greater than the angle of repose of the broken ore, so that it may be drawn from pockets without mucking or tramming.

A great disadvantage of the ordinary top-slicing method of mining is the necessity for much shoveling and frequent wheelbarrow work to get the broken ore into the stope chutes. To reduce this labor, chutes are made at close intervals; this, in turn, involves much extra work to make and maintain these chute raises.

A stoping method has been worked out and is now in successful operation at the Coronado mine of the Arizona Copper Co., Ltd., at Metcalf, Ariz., where the shoveling of the broken ore is practically eliminated by inclining the floor of the top-slice stopes. This is a combination of the shrinkage, rill and top-slicing methods of mining. The various steps in the working of this incline top-slicing are: Chutes are raised through the orebody, which is in porphyry, at intervals of 50 ft. along the strike of the vein and as nearly as possible equidistant between the walls, which in this mine are of granite. Sublevels are driven at vertical intervals of 55 ft. above the haulage level to connect these raises and allow access to the slice stopes above. Crosscuts are then made from wall to wall, on the sublevel, opposite the mouths of the haulage-level chutes. These crosscuts are timbered with square sets, and in these sets sloping plank slides are erected and ore gates fitted, forming an ore bin on each side of, and emptying into, the haulage-level chutes.

Commencing on the top of these sets, an opening the full width across the vein and about 4 ft. in size is stoped to the mat above. This opening, or shrinkage stope, is kept filled with broken ore, and access is obtained from the ladder and timber raise made in one of the walls of the vein. By opening the ore gates on the sublevel, enough broken ore is drawn out of this shrinkage stope to allow access and working room.

Referring to the longitudinal section shown, the part marked *A* shows conditions after the slice has been worked out and blasted down. A timber mat of 2 x 12-in. plank covers the floor of the slice, separating the overburden from the solid ore and is in place to form the roof of the next slice beneath. The mat is nailed to sills of 10-ft. stulls placed at 5-ft. intervals. These sills will form the caps of the timber sets supporting the mat on the next slice beneath. The broken ore in the shrinkage stope has been drawn down 11 ft. preparatory to beginning the next slice.



SECTIONS SHOWING THE CORONADO INCLINE-TOP-SLICING METHOD OF MINING

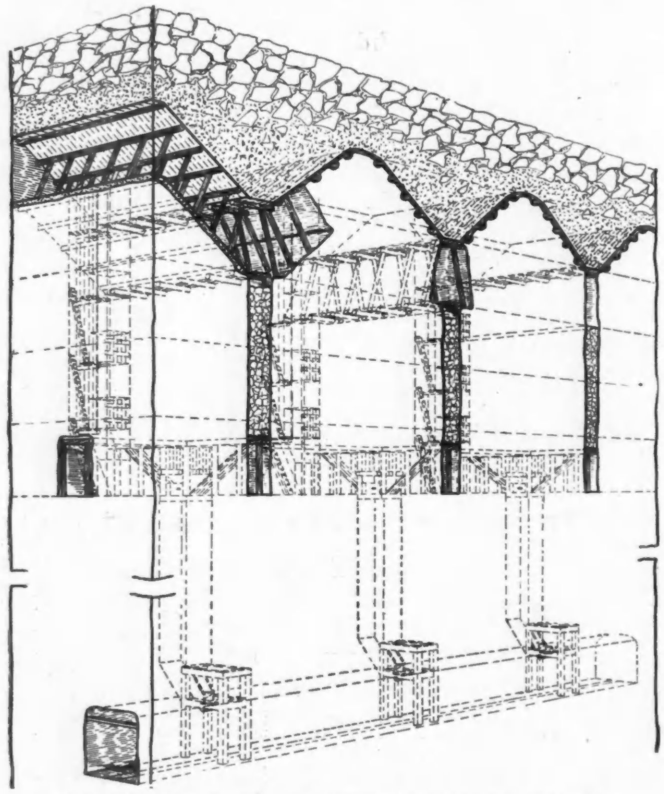
In part *B* of the same section the stulls which formed the chute grizzly of the exhausted slice have been caught up by two inclined posts standing on hitches in the walls of the shrinkage stope. A grizzly is then laid over the mouth of the opening and stoping begins. Jackhamer drills are used, and the ore, when blasted, rolls directly through the grizzly into the shrinkage stope below. The floor of the stope is kept sloping at an angle of about 33 deg. from the horizontal. As the sills of the mat above are exposed by mining, they are caught up by posts 10 ft. long. New sills of 10-ft. posts are laid down and covered by 2-in. plank 12 ft. long, forming an ore slide for this stope and a roof for the next slice below. When the top of the incline is reached, mining ceases. The floor is completely boarded over, the shrinkage stope left full of broken ore, and the posts blasted down. The mat

*General superintendent of mines, Arizona Copper Co., Ltd., Morenci, Ariz.

and superincumbent overburden crushes in, and a new slice can now be started 11 ft. lower down.

The amount of broken ore left in the shrinkage stope is easily controlled by the ore gates opening over the main-haulage chutes. For safety, the stopes should never be drawn empty. The men employed in each slice are one machine miner, a timberman and a timberman-helper. All the broken ore rolls down the incline to the shrinkage stope, shoveling being practically eliminated.

A floor slope of 33 deg. has been found to give the best results. It is steep enough for the ore to roll on, but not steep enough to seriously hamper the movements of the men in their work. If the width of the ore is too great to mine in one slice, it can be worked out in



PERSPECTIVE VIEW OF THE CORONADO INCLINE SLICING METHOD

panels and each panel blasted down when completed. Little trouble has been experienced in mining along the side of a caved panel.

Should the weight of mat above become too great when one slope of a slice has been worked out, this slope can be blasted down and the other slope worked out later, without special difficulty. The consumption of timber is no greater than in the flat top-slice method. Much credit is due to W. G. Scott, the superintendent of the Coronado mine, and to his foremen and shift bosses, for devising and working out this new mining method.

Mining Activity at Caen, France

By Charles R. Flandreau*

It is well known to the close observers of international mining operations that there are extensive deposits of iron in the territory comprising Norman Basin in the consular district of Havre, France. Prior to the outbreak of the European War these deposits were being exploited on a large scale and at a good profit by Thyssen Bros. German capitalists, who controlled, in large measure, the metallurgical operations in that section. It may also be said that the existence of these rich deposits of iron were in no small way responsible for the rapid growth of the port of Caen.

*District National Bank Building, Washington, D. C.

At the outbreak of the war it was but natural that the French government should sequester the German interests in this field. The mobilization of the French army also greatly hindered the development of these mines because nearly all the able-bodied miners were called to the colors. At that time there were 21 mining concessions in this district with an annual output exceeding 600,000 tons. The exact locations of the principal mines were: In Calvados—Saint-Remy, Saint-Andre, May-sur-Orne, Soumont and Jurques; in Orne—Halouze, La Ferriers-aux-Etange and Larchamps; and in Manche—Dielette, Bourberouge and Moratin. It will be seen, therefore, that the mining industries of this region were of no small importance.

Prior to the war the work of extraction and reduction of the iron ore in the Norman Basin was in the hands of the Societe des Hauts-Fourneaux et Acieries de Caen, a concern backed by French and foreign capital, and by the Societe Francaise de Constructions Mecaniques. These concerns had at that time begun to equip the blast furnaces of Caen with modern apparatus. The war, however, caused a sudden cessation of all activities in that district, and the mines remained inactive for some time.

The Societe Normande de Metallurgie has now recommenced operations at the stage where work was stopped by the war. This company has enlarged upon the original plan of the former operatives, and rapid progress is being made toward realization of the projects. Present operations comprise the working of the mines of Soumont; the construction of a railway crossing the entire basin, connecting with the metallurgical works at Caen and terminating at the port of Caen; and the construction of a private port on the canal from Caen to the coast, which will, when completed, be capable of receiving vessels of 8000 tons. The value of this last plant should not be underestimated for it will not only permit vessels to discharge their cargoes of material necessary for the operation of the mines, but will also allow the exportation of the finished product of the mines direct from Caen.

The mine equipment is up-to-date and comprises coke ovens, installations for the recovery of byproducts, blast furnaces, each of 450 tons' capacity and able to produce 600,000 tons of castings per year, Thomas steel apparatus, Martin steel apparatus, and a complete series of rolling mills for rails, beams, bars, sheets and marketable steel. Power for the operation of the mines will be furnished from the central station, which will be fed by the gas of the coke and blast furnaces. The first battery of coke ovens is already lighted, and three others are nearing completion and will be in operation as soon as possible. The plants adjoining are completed, and it is expected that these will soon be able to furnish benzol, toluol, naphthalene and phenol to the powder service of the government. The first blast furnace is nearly completed, and the others will be in operation shortly after this one. These furnaces will first product hæmatite casting for the manufacture of the steel to be furnished by the Thomas and Martin steel mills and rolling mills. Other castings and rolled products will be produced as the works are improved later on.

Mining Methods Illustrated in Miniature

By W. R. Crane*

SYNOPSIS—A general discussion of mining methods and underground support illustrated by the use of clay models, which are conducive to clear understanding of underground conditions, correlating the workings in various parts of the mine and presenting a good mechanical picture of the development as a whole. This method is particularly valuable for showing students the principles of the many varied systems of mining.

The value of models as a means of conveying ideas and creating mental imagery of works and processes is beyond question. Models are employed in courts as exhibits illustrating industrial and domestic problems and in connection with great enterprises, as the building of the Panama Canal. No class of work furnishes a more profitable field for the use of models than does mining, and this is now well recognized by the mining engineer, the mine manager, the stockholder and the student of mining. It is to be hoped that some good may result from our efforts to describe modern mining methods as they look, not to the man in the mine who can see but a few feet at a time, but rather to him who, being acquainted with methods, sees the mine in its entirety.

Lack of time and opportunity prevent many from securing a comprehensive grasp of the methods employed in widely separated localities, and the nomenclature used, the correct understanding and use of which is desirable. Models, like sketches, bring out the special features that should be emphasized, and place in the background or omit entirely the mass of details that tend to confuse and draw attention from the main points of interest. The details may be, and often are, the most attractive part of a mine setting, and the modeler must guard against the desire, if he is an artist, to make a model attractive rather than instructive.

As models are intended to show principles only, it is desirable that the work should be laid out as symmetrically as possible. In actual mining, great irregularity often exists owing to conditions of occurrence of mineral in the orebodies and methods of working. Further, all mine openings as shown in models are of necessity made large and consequently out of proportion to size of deposit. The same is true of timbering, but instead of being a disadvantage, the enlargement may be in fact a positive advantage in that it accentuates and emphasizes the features represented.

Development of Mines

The purpose of development is primarily to connect the deposit to be worked with the surface by openings suitable for the passage of men, the handling of ore and supplies and for drainage and ventilation. The opening of a mine is accomplished by the use of various forms of passages driven either in the deposit or in the inclosing wall-rock, as typified in Fig. 1. As to whether tunnels, slopes, inclined or vertical shafts shall be used depends upon the dip of the orebody.

The limits set by common practice are as follows:

	Angle Made with Horizontal
Drift or tunnel	0° to 3°
Slope or plane	3° to 25°
Inclined shaft	25° to 85°
Vertical shaft	90°

Except in special cases there can be no doubt as to the choice of a drift, tunnel or slope; but with inclined and vertical shafts, or when highly inclined deposits are to be developed, there may be some doubt as to which would be better. Probably the principal limiting condition is where the cost of driving crosscuts from a vertical shaft to the inclined deposit is prohibitive, which condition is reached when the deposit stands at an angle of 36 deg. with the horizontal. Second only in importance is the fact that with flatter inclines the ore will not run down the slopes to the loading chutes.

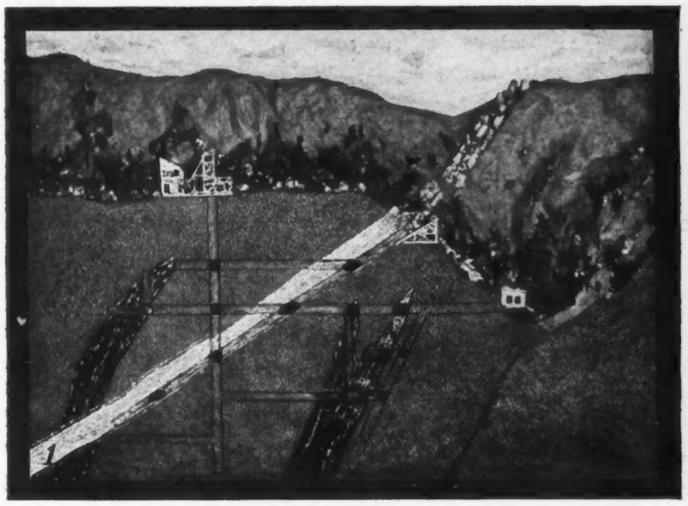


FIG. 1. SHOWING METHODS OF OPENING UP A MINE AND RELATION OF WORKINGS TO OUTCROP

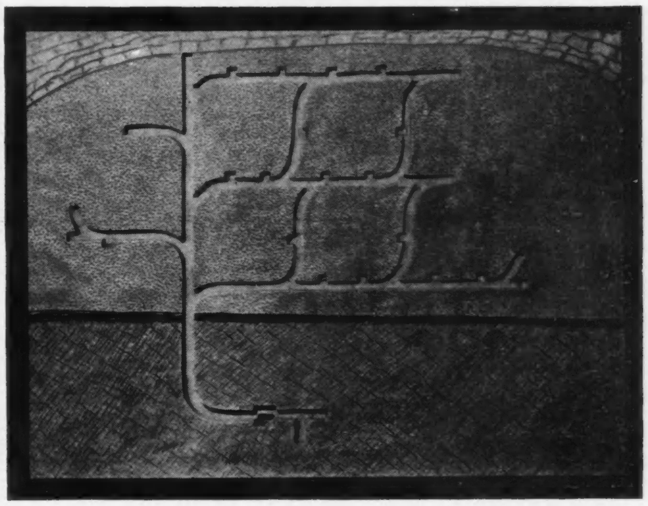


FIG. 2. PLAN OF DEVELOPMENT ON LEVEL PREPARATORY FOR TOP-SLICE CAVING METHOD

Following the preliminary work of connecting the deposit with the surface, the second stage of development work is begun and consists in subdividing the workable portions of the deposits by means of passages driven practically horizontally into blocks which can be attacked

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and the broken ore handled to the best possible advantage and at minimum cost, as in Figs. 2 and 4. The horizontal passages in the deposit are usually spoken of as levels and are spaced from 50 to 300 ft. apart vertically. These passages are in turn connected with other passages driven vertically at right angles to them and are called raises or winzes depending upon whether they are driven upward or sunk, noted in Figs. 3 and 4. By means of these development passages in the deposit the ore is blocked out and rendered accessible to the miners, and they at the same time give a means of determining the value of the ore thus blocked out.

A well-developed vein or highly inclined bedded deposit may be likened to a high building, thin but wide, the floors of which are served by elevators. A massive deposit may similarly be likened to a factory covering considerable area, the floors of which are also served by elevators with well-defined passages for handling cars and trucks as well as for the movement of laborers. In a similar manner a horizontal bedded deposit of moderate thickness, when developed, resembles the streets of a city, and not only with respect to the working places in the buildings and the handling of products in the streets, but with respect to lighting and drainage.

Careful and well-planned development systems are necessary for the proper working of mineral deposits, and while the expense may be great, the saving in operating cost and the high percentage extraction of ore together with the prevention of dilution by the admixture of waste are sufficient warrant for the extra expenditure and time involved.

Among the various factors that determine the method of support to be employed in a particular case, that of cost is probably of first importance. The cost of materials of support, their preparation for use, and especially the cost of development that will permit of their use, must be considered. The increased cost of timber and the constantly diminishing supply have been responsible for radical changes in methods of support—as from an extensive use of stulls and square-sets to caving and filling methods, and to the arching of stope backs to render them self-supporting.

Methods of support in metal mines depend primarily upon the character of vein-filling, nature of the hanging and foot walls, and inclination of the vein. While these three factors are of practically equal importance, yet the inclination or dip is the one particular factor that demands attention under all circumstances; the other factors may be so intimately interrelated in certain instances as to diminish or increase the conditions of support from a maximum to a minimum or vice versa. Figs. 5 and 8 illustrate this.

With horizontal beds or veins the superimposed strata must be supported, while with vertically standing beds or veins the vertical and horizontal acting components have changed places with respect to the deposit and a minimum amount of support is required to maintain excavations in the deposit. Methods and materials of support may be grouped under the following heads: Pillars of ore or waste; timber, consisting of props, stulls, cribs and square-sets; filling with ore or waste; caving methods, and arching of roofs. Pillars may reach a height of 100 ft. or more and stand in veins with inclinations as high as 50 deg., but the height must of necessity diminish with increase of dip. They are especially applicable to moderately wide veins in which open stopes are maintained, and in massive deposits worked by underhand stoping.

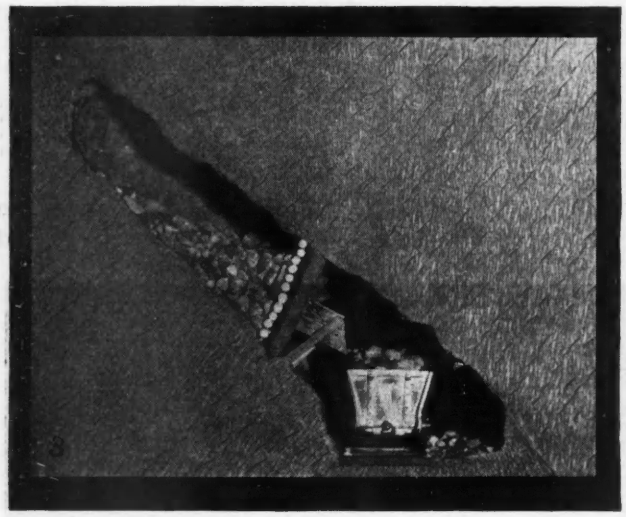


FIG. 3. VERTICAL SECTION, SHOWING THE USE OF STULLS ON LEVEL

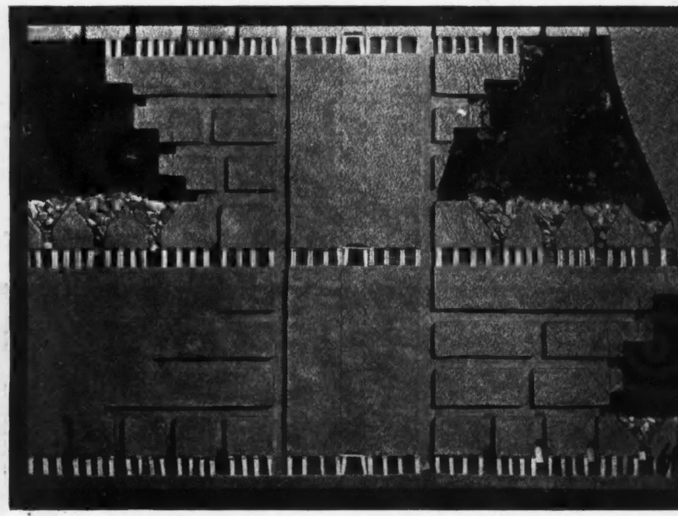
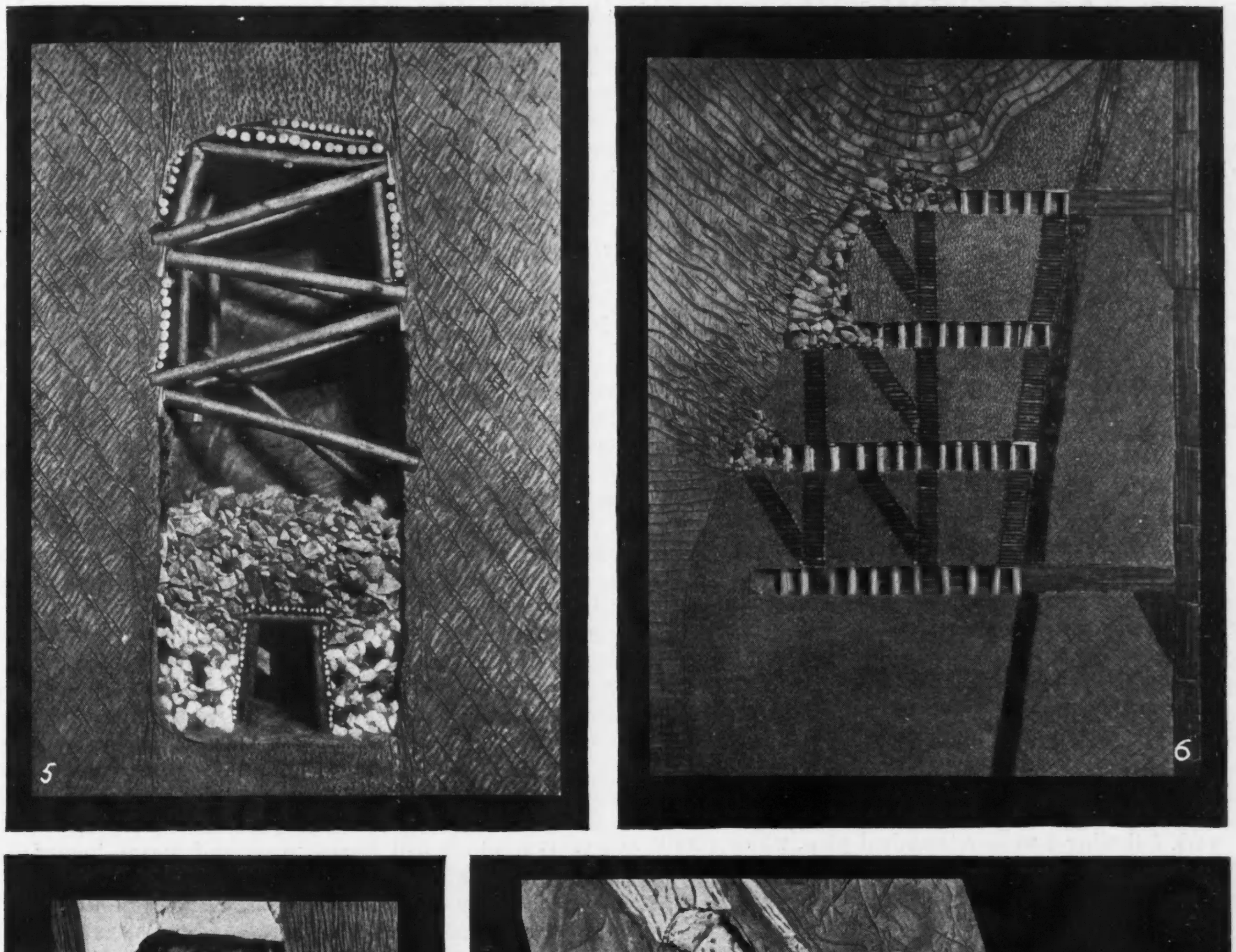


FIG. 4. VERTICAL SECTION, SHOWING METHOD OF DEVELOPMENT BY SUBLEVELS

Props constitute the universal method of support with timber, particularly on inclinations not exceeding 10 to 15 deg. They are placed normal to the surface to be supported and require no special preparation except the use of caps and occasionally lagging. Stulls differ little from props, but are adapted to steeper dips, as from 10 deg. to the vertical, as shown in Fig. 3. Owing to the fact that they are usually employed in inclined beds and veins, it is necessary to provide special excavations in the foot wall, known as "hitches," to hold them in place. Further, the angle that the stull makes with the hanging wall is somewhat greater than a right angle above and less below. The angle of "underlie" is that made by the stull and a normal to the hanging wall, being about one-fourth of the angle of dip of the vein.

Cribs are not extensively employed except in certain localities and are particularly suited to large stopes where the walls are weak and the vein-content difficult to hold up, as in Fig. 7. Cribs may be used with or without filling and under exceptional circumstances are filled in around



FIGS. 5 TO 8. SHOWING VARIOUS TYPES OF MINING METHODS AND UNDERGROUND SUPPORT

Fig. 5—Use of stulls for support of weak but nearly vertical walls. Fig. 6—Vertical section, showing development for top-slicing caving method. Fig. 7—Use of cribs in veins with weak walls. Fig. 8—Showing use of square-set timbering

with waste rock. Square sets were formerly extensively used in all forms of deposits, and so long as timber was plentiful they constituted a method of support applicable under a wide range of conditions. Square sets are shown in Fig. 8. When reinforced by waste filling, square sets have all the advantages of either used separately.

Use of Broken Rock as Ground Support

Ore and waste may be employed as support in wide veins and massive deposits, the former being temporary in character and the latter more or less permanent. Broken ore standing in stopes serves a useful purpose as support for drills, and at the same time forms an ore reserve that can be drawn upon to maintain regularity of output. Waste filling furnishes permanent support except in those cases where it is drawn from one stope to another as the work of removal of ore permits. It has its source in waste sorted from the broken vein-filling, from special excavations in the hanging and foot walls, from quarries on the surface and waste products from concentrating plants. The main objections to its use are the tendency to shrink, and to become quick and flow, thus jeopardizing the workings.

Caving is usually employed as a supplementary method following some well-defined system of working and support (see Fig. 6). It may be used in working massive deposits of shallow depth, the broken ore being drawn off below as it caves upward; or in similar deposits developed in floors, certain definite portions may be systematically caved, in which case considerable timber is often employed to regulate and render safer the handling of the ore. The use of caving methods necessitates continuous and rapid work following a well-systematized plan, which results in a large tonnage extracted at low cost. Caving is particularly adapted to large deposits of low-grade ore, such as copper and iron.

The arching of roofs or stope-backs may be classed as an indirect method of support, such arched excavations being commonly spoken of as "domes of equilibrium." It has been observed that the roofs of underground excavations will in time cave, forming a natural arch which under normal conditions will stand for some time, the weight of the rock and the resisting elements being in a state of equilibrium. When the conditions of vein-filling permit, such arches are usually formed and allow large quantities of ore to be removed with no support whatever or, at most, support of the most temporary character.

A Prospectors' Convention

By J. F. Kellock Brown*

A prospectors' convention is to be held in Fort William, Ontario, about Easter time. This represents something new in mining, and it is probably the first convention of its kind to be called together. Credit for the conception of the idea is due to J. T. Hewitson, of that city, a successful mining promoter and operator who has been forced to the conclusion that something should be done in the western Ontario district to improve the status of those who carry on the initial work in mining.

Port Arthur and Fort William, the twin cities of the center of Canada, are peculiarly situated with regard to contributary country. For hundreds of miles to the north, northeast and northwest, the country is practically unknown. The geology of this district is largely Archæan and Keweenawan. Farm land is scanty, with the exception of the clay belt through which the transcontinental railway runs. In any case most of the intending settlers having come so far, would pass this timbered country, and endure another day or two of train-traveling to reach the rolling prairie. The country has many waterways, and these have served to make it a land of the trapper, the prospector, and, latterly, the lumberman.

The twin cities are the natural center of this part of the country and also the natural outlet for its mineral and lumber products through the Great Lakes and their shipping trade. To these cities the prospector must turn, and it is there that he finds the men willing to "grubstake" him in his summer and winter work. The prospector's knowledge of the work is often scanty, and the town merchant has none at all. The prospector is the man who does the hunting for the minerals and finds the "color," but he needs financing through the period of development of the property to the stage when it becomes worthy of attention of outside capital. Obviously, the only men who are in a position to do that, and to whose interest it is to do it, are the merchants of the two cities. Gradually a feeling has grown that the future of Port Arthur and Fort William lies in the development of the surrounding country. This must be in the character of mineral development owing to the nature of the country; and as soon as this idea was realized, it was felt that there ought to be some organization to encourage and to carry on the work.

The result has been the proposal for a prospectors' convention which will discuss ways and means toward developing the search for minerals and will attempt to provide a reasonable system of aiding the prospector with proper information. The merchants on the other hand, will be given some ideas on the elements of mining, and an endeavor will be made to bring local capital into line with the requirements of the project, which is likely to develop along three main lines. First, it will try to arrange some system whereby the prospector will be able to get information and a preliminary report on his samples. This may be accomplished through the establishment of a small mining school and laboratory. The establishment of a clearing house for local "grubstakers" is the second consideration, where prospectors can obtain a list of men willing to undertake the early stages of mining, and where the men willing to do the "grubstaking" may have some guarantee that the prospector is bringing forward a reasonable proposal. The convention, or the organization resulting, will endeavor to make certain that the prospector's statements are reliable. Finally, some organization will be needed to take care of the projects that have been developed and proved by local capital at the stage where outside capital becomes a necessity to further expansion.

The project is worthy of assistance, and it is hoped that the prospectors will attend the convention in large numbers and that the merchants of the town will show their interest. While Port Arthur has the credit of initiating the prospectors' convention, Fort William, not to be outdone, has evolved a department of its Board of Trade under President W. A. Dowler, which intends to proceed upon somewhat similar lines. It is difficult to anticipate what may develop out of these two movements.

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Mine-Pumping Equipment

By Whitman Symmes*

SYNOPSIS—The choice of mine-pumping equipment depends on a variety of factors. There are two general types of pumps, plunger and centrifugal, which may be operated by steam, compressed air or electricity. Cornish pumps, air lifts and hydraulic elevators have a minor application. The operating conditions under which a type of pump works best are given and means for the adjustment of different types to the various conditions.

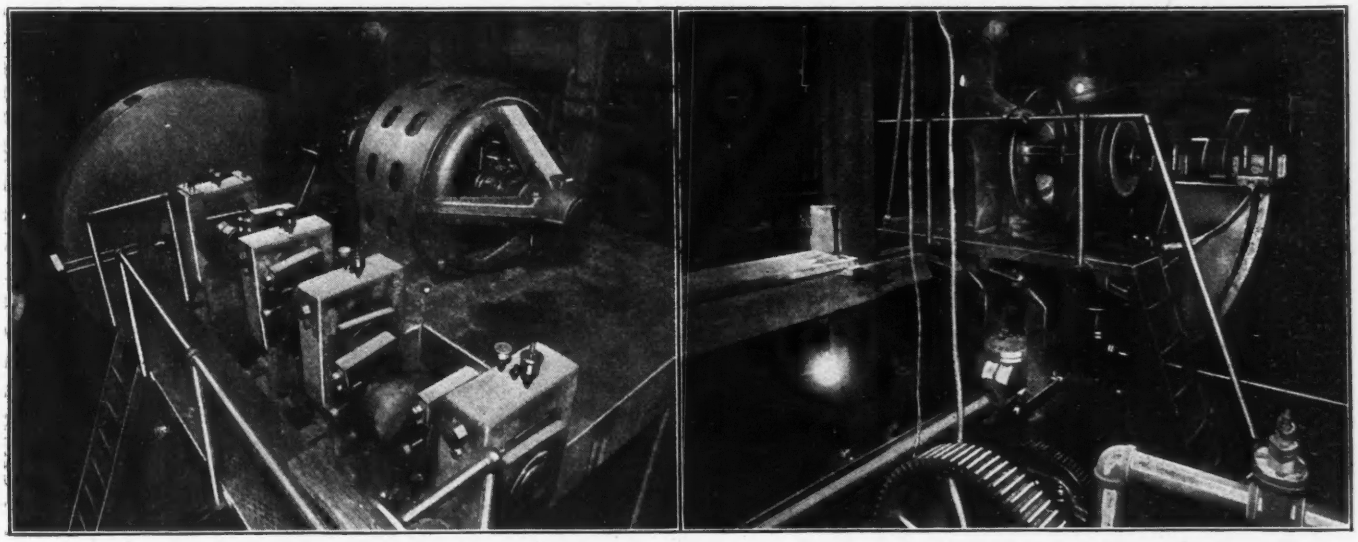
The principal types of mine pumps in use at present are steam- or air-driven plunger pumps, electrically driven plunger pumps and electrically driven centrifugal pumps. To a less extent pumping is accomplished by means of Cornish bob pumps, air lifts and hydraulic elevators. Centrifugals are coming more into use at the expense of the other types, notwithstanding the fact that the capacity and lift of any one pump can be varied only within narrow limits. Electricity for operation has also shown a large extension through the mining field, so that now it can be considered as the standard power for mine pumps. This extension is taking place notwithstanding the fact that fixed-speed motors must generally be employed, thus confining the pump to one fixed duty while in operation.

Cornish bob pumps are now but little used. The original Cornish pump was steam-driven and was condensing. California practice changed the motive power to a water-driven impact wheel with gear connection, as being far more economical in most places in that state. Later the waterwheel was replaced by an electric motor. The Cornish pump formerly used in Western metal mines has been generally replaced by electrically operated centrifugal and plunger pumps.

The use of steam-driven plunger pumps is principally confined to the coal-mining regions. They are also used to some extent in Colorado and other localities where coal is obtainable at comparatively low prices. On the Pacific slope they are now used only where the mines have not attained any great depth and where there is no convenient source of electric power available. There has been little change in steam-driven pumps within recent years. The station pumps used in the coal mines are generally of compound- or triple-expansion types, often with a flywheel. Those used in Western metal mines are generally of simple construction. There would seem to be a field for steam turbine-driven centrifugal pumps wherever the use of steam underground is advisable. Air turbines for driving centrifugals are too expensive in power for any known conditions.

Air-driven plunger pumps are still in use everywhere, although their field has been greatly encroached upon by the electrically driven plunger type, the power cost of which is only a fraction of that required for air operation. The type of pump used in the West nearly always has a simple cylinder and therefore can be used for either steam or air. Compound air-driven pumps require, of course, a different valve adjustment.

The deep-well pump, which is a half-brother of the Cornish pump, appears to have occasional applications to mining work. It operates by the pull on the rod, whereas the Cornish pump operates by the weight of the rod. Two deep-well pumps were recently installed in the shaft of the Great Western mine at Tonopah as a temporary expedient and have been found most serviceable. They are placed in the sump, and the power head is placed on the lower pump station and operated electrically. They have given additional sump without having to be lifted when the power failed, as would have been the case with any other motor-driven type.



TWO ALDRICH QUINTUPLEX PUMP INSTALLATIONS ON THE 13TH LEVEL OF THE BUNKER HILL MINE

Electric power, where used, is nearly always in the form of alternating current, and therefore the motors have a fixed speed and the capacities of the pumps, whether centrifugal or of the plunger type, cannot be varied. The amount of water pumped is made to agree with the influx either by intermittent operation of the pump, or by

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the use of a bypass. In the latter case, of course the pump is operated at full capacity and consumes power accordingly. Were direct current available, it would be far superior to alternating current for operation of mine pumps, because with direct-current motors the speed can be varied. A scheme for changing alternating to direct current for pumping the Comstock was abandoned because of the greatly increased expense of installation.

In the case of plunger-driven pumps there appears to be only two possible exceptions to the rule given. On the Comstock, two-speed motors have been used. By throwing a switch the motors can be made to operate the pumps to throw either 1000 or 500 gal. per min. By occasionally changing the capacity of the pumps by this means, regulation is obtained without excessive tank capacity, which, under the local conditions, would have been too expensive. In the other case regulation is obtained by means of changeable gears. Triplex shaft pumps were constructed with a double gear reduction and with three sets of gears, so that each pump has capacities of 75, 88 and 104 gal. per min. The gear ratio used is that which gives the least excess capacity possible over the influx at the winze where the pumps are installed, a further regulation being obtained by intermittent operation of the pump or by bypassing. The advantage of having the pump operated at the least possible speed to handle the influx is considerable, because gear-driven units operated at high speed require far more attention and repairs than those run at lower speeds. This advantage, however, is now becoming of less moment because of the introduction of herringbone gears and the use of the Drake locknut.

The regulation of centrifugals is obtained by throttling the discharge or by bypassing or by intermittent operation of the pump. The last is the only method by which the power consumed is in accordance with the water pumped. A large sump is therefore necessary and, unless the ground is bad, will pay for itself many times over, provided the electric power is purchased according to meter readings. Bypassing of course does not save any power worth mentioning and throttling saves very little power. Even when the throttle valve is entirely closed, the saving of power will be less than 10% of that used when the pump is operated at its full capacity. When pumps made for a high lift are used for a lower lift (as in beginning to unwater a shaft) a pressure gage should be used on the discharge column and the pump should be kept well throttled, otherwise it will probably take power in excess of the capacity of the motor, which may thus be burned out.

Regulation by Means of the Runner

In Comstock practice a further amount of regulation is obtained from time to time by changing the runner. In case the influx is less than the capacity of the pump, a runner is employed with a narrower opening. This cuts down the capacity of the pump and also cuts down the power consumption more or less in proportion. Small variations in the required lift, from that for which the pump was originally built, are met by slightly increasing or decreasing (turning down) the size of the runner. Generally it is advisable to change the liners to correspond.

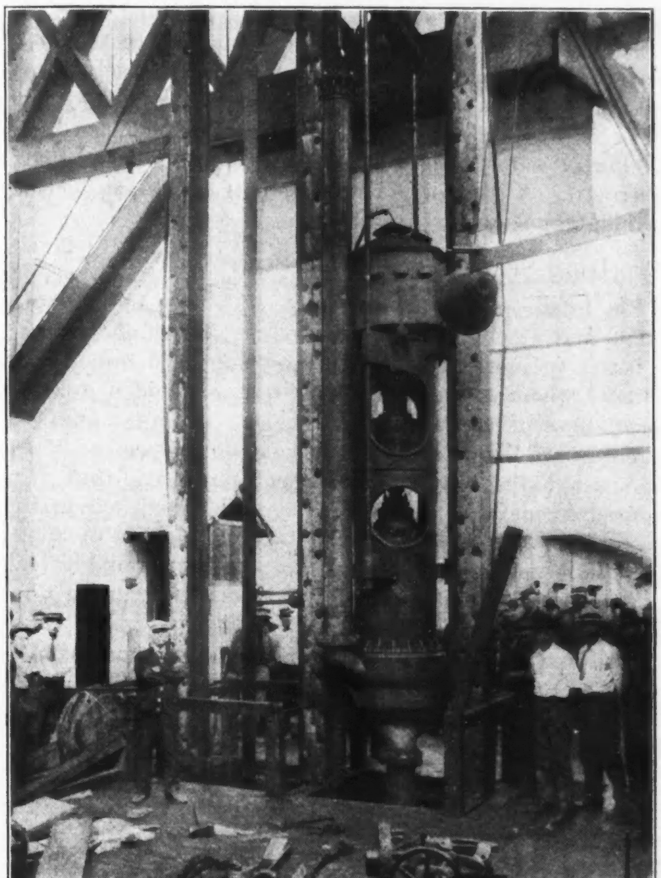
As regards lift, the plunger pump has an advantage over the centrifugal, its lift being dependent only upon the power of the motor and the strength of the machine. The centrifugal pump, however, must be made for a certain lift and a certain capacity. As soon as this lift is increased, if only by a few feet, the capacity is considerably reduced. The lift also varies with the slightest change in the speed of the motors. During the winter storms, when power is irregular, the lower levels of the Comstock are frequently flooded, because of a drop in the frequency of only two or three cycles per second. Under such conditions the plunger pumps operate with scarcely any perceptible change in capacity.

Plunger Pump More Efficient Than Centrifugal

The efficiency of the modern electrically driven plunger pump is in all cases superior to that of the centrifugal. The efficiency of the former can be assumed at about 82% or a little better. The efficiency of centrifugal pumps 4 to 10 in. in size can be assumed at 60 to 75%. In smaller sizes with high lift, the efficiency decreases rapidly. A 2-in. four-stage pump will give only about 35% efficiency. These statements are of course only approximate, because different manufacturers making apparently similar pumps, be they plunger or centrifugal, will deliver machines that vary by 5%, or even 10%, in efficiency. Comparative efficiencies from wire to water can generally be assumed at about 72% for the plunger pumps and at about 65% for the best centrifugal turbine pumps. Plain centrifugals (without diffusion vanes), when properly designed and used, may be nearly as efficient as the turbine type. The electrically driven plunger pump is superior to the electrically driven centrifugal for small quantities of water against a high lift. With increase of capacity and increase of lift, which requires heavier construction in the plunger pump and greater attention to packing and repairs, the advantages of the centrifugal become of increasing importance. Therefore, notwithstanding its inferior electrical efficiency of 7% or more (often much more) in the majority of instances, and especially where the mining is speculative and of uncertain duration, it is the most economical pump for shaft and station work.

For shaft sinking, the plunger pump, whether driven by steam, air or electric motor, has the field all to itself. Centrifugal pumps cannot be used except under unusual circumstances and never with satisfaction. The centrifugal mine pump must work on clear water. This applies especially to the turbine type. Sand and grit rapidly cut it out. In shaft sinking it is almost impossible to keep the water well down and at the same time keep the pump primed. In Comstock practice, where centrifugal shaft pumps are generally used for unwatering the old workings, as many precautions as possible are taken to handle the minimum amount of sand and grit. Work is pushed as rapidly as possible, and new runners and liners are kept at hand. As soon as a level is reached, new liners and runners are put in, sand boxes are installed, and the pumps are then operated as station pumps. Unwatering a mine, whether the shaft be caved or open, is very much easier on a pump than sinking a new shaft.

The type of electrically driven plunger pump now most largely used has either three or five vertical plungers. The vertical-plunger pump will go into fork without hammering, while in the horizontal pump the hammering is considerable and at high speeds is often dangerous. On the triplex pumps an air chamber is always employed, but on the quintuplex pumps the makers generally supply an alleviator only. It is better, however, to have an air chamber, even on the quintuplex type. An air chamber consisting of a horizontal 8-in. pipe connected to each of the five plunger discharges has been applied to quintuplex pumps on the Comstock at small expense and with considerable improvement in the running. Air chambers are best charged by compressing air in a large pipe by means of water from the pump column. Where the conditions are favorable to the use of a belt, a belt-driven pump will run smoother and better than one that is gear-driven. Vibration means expense. Only occasionally, however, will underground conditions justify a belt installation, and they are rarely seen. Silent steel belts are occasionally used. They make it easier to change a gear ratio, but have little else in their favor, and their expense is much against them. Single-reduction gears are generally used, and with the recent herringbone type of gear, with pinion cut from a solid shaft, a reduction of speed as high as 20 to 1 can be obtained. Pumps are often lined to protect them from acid water, either with cement or $\frac{5}{8}$-in. wood. Lead could also be used. Pump columns also are sometimes of necessity lined with lead or wood.



VERTICAL DIRECT-CONNECTED PUMP AT C. & C. SHAFT, COMSTOCK LODE

Electrically driven plunger pumps always employ a pulley or gear reduction between motor and pump. A direct drive similar to that now largely used for air compressors was tried in the high-speed plunger pumps made for the Ward shaft on the Comstock. The 800-hp. motor made 196 r.p.m. The pump ran well for short periods of time, but as soon as any trouble developed, the pump and pump column were subjected to terrific hammering and vibration. Although the rotor of the motor was made 9 ft. 7 in. diameter, and weighed 11,000 lb., in order to obtain a reasonable speed, the valves had to open and close $3\frac{1}{4}$ times per second. After numerous trials the pumps were discarded. These high speeds in plunger pumps are practicable if the pumps are properly designed, but there is really no necessity for their use. Slower-speed plunger pumps cost less money and require less attention. The Riedler pump is the most successful high-speed type and has mechanically actuated valves. Those at the C. & C. shaft make 113 r.p.m. with 24-in. stroke and have been in continuous use since 1903.

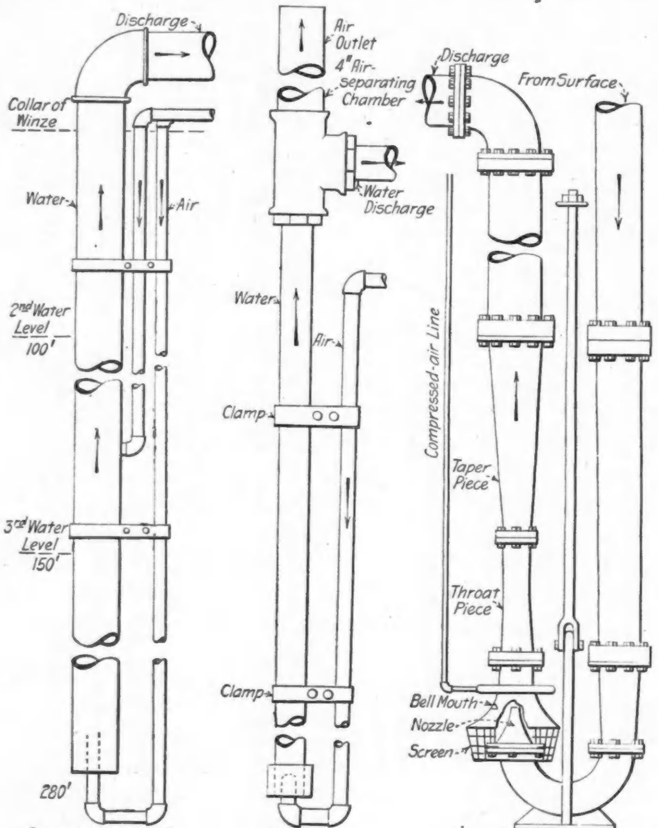
Where motors of 50 hp. or more are used, there should be an outboard bearing beyond the pinion, and in large installations, where herringbone gears are not used, a flexible coupling between motor and pinion should be employed. The shafts of both pumps and motors, except in the small sizes, are best made of nickel steel to resist fatigue. The manufacturers generally make the valves in nests of five or more to be placed in each pot. In Comstock practice an annular valve is used in each pot. It was developed from the Riedler valve and has given excellent satisfaction. On lifts of 500 ft. or less the Riedler leather seal is not necessary and the valve can be made entirely of metal. For higher lifts the valve can be supplied with a rubber buffer, which also acts similarly to the leather seal.

Plungers are generally made of chilled cast iron. Monel metal, although very expensive, has been used on the Comstock with excellent results. The usual Comstock practice, however (except in small pumps, where iron is occasionally used), is to line the plungers with brass tubing. This can be bought in $\frac{3}{8}$-in. sizes. The plunger is turned to fit it exactly, and it is then slipped on and fastened in place. Where heavy grit is not handled, it gives excellent results.

Characteristics of Centrifugal Pumps

Centrifugal pumps are inferior in power efficiency and are of a fixed capacity and fixed lift, and they lack the positive suction of the plunger pumps. Their use is further limited by the fact that they cannot be used for sinking purposes except temporarily, nor for handling gritty water. Their great advantages, however, consist in their small size, with correspondingly small pump station; in their high speed, which takes a smaller and cheaper motor; in the fact that they can more cheaply be made of brass or bronze to resist acid mine water; in their ease of installation; and, when properly used, in their practical freedom from all repairs. Two stations at the Ward shaft, in Virginia City, Nev., each contained three 200-hp. quintuplex pumps with a combined capacity of 3000 gal. per min. (500-ft. lift), and each station is 100 ft. long, 18 ft. wide and 18 ft. high. The 2310 station at the C. & C. shaft, in Virginia City, contains three 200-hp. centrifugal pumps with a combined capacity of 4500 gal. per min. (325-ft. lift) and is only 17 ft. wide, 10 ft. high and 30 ft. long. The 2000 station at the same shaft contains three Riedler pumps with a combined capacity of 4500 gal. per min. (395-ft. lift) and is 119 ft. long by 20 ft. wide by $19\frac{1}{2}$ ft. high.

Centrifugal pumps can be constructed equally well in either the vertical or the horizontal type. The latter is more often employed for station use, but it is Comstock practice to use the vertical type for station pumping on the lower levels. On the 2700 and 2900 levels the pumps formerly used in unwatering the lower workings have been swung out into the stations with the advantage that in case of an interruption to the power, which is always to be guarded against, the entire level must be flooded before the motor is reached and the pump incapacitated. The pumps are suspended on a trolley and I-beam extending from the winze a short distance into the station, and during interruption of the power the pumps (weighing, with motor, 13,000 lb.) have been easily swung out into the winze and hoisted. Upon the resumption of power they were again used to unwater the workings and were easily replaced upon the station. Such pumps, when used in shaft or winze work, are hung from worm-driven winches operated by compressed air. They run



Compound-air Lift FIG. 1 — Air Lift FIG. 2 — Hydraulic Elevator FIG. 3

FIGS. 1 TO 3. AIR LIFT AND HYDRAULIC ELEVATOR TYPE

on guides, but require no blocking, and have no other support than the cable.

The early types of centrifugal pumps were lacking in balance. It was sought to keep the runners centered by means of bearings and bolts. As soon, however, as the pump began to wear, the shaft and runners would start working toward one end or the other, and nothing could hold them in place. The modern pumps are designed to keep the runners in place by means of a balanced water pressure against their sides. Balance is the most important characteristic of a good centrifugal pump, and some makers have developed pumps that are practically perfect in this respect. Some vertical pumps have runners that make them entirely self-balancing, and in operation the runners will not only center themselves, but will lift the weight of the superimposed rotor. Other vertical pumps depend upon metallic thrust bearings, fed with oil and grease, or else balance the weight upon a disk fed with clear water from the surface. Centrifugal pumps should generally have clear water fed into their bearings. This allows less pressure on the glands and saves power. It also greatly increases the height of suction that can be maintained.

Centrifugal pumps have not as positive a suction as plunger pumps, and it is difficult to keep them in condition for a suction of more than 10 to 15 ft. Higher suctions can be obtained, but are not advisable. The strength of suction depends principally upon the design of the pump. Centrifugal station pumps, wherever possible, should be arranged with a positive flow from tanks to pumps. Where this is not feasible, the suction should never be more than a few feet, and no foot valve should be employed. Foot valves are not generally desirable upon shaft pumps and should not be employed unless the pumps are strongly constructed—otherwise when the power stops, the shock of the receding water column will split the pump casing. For a similar reason a quick-closing valve should never be used upon the discharge column of a centrifugal pump. Comstock practice is to use suctions without any foot valves when the pumps are for station use. When shaft pumps are in use for unwatering, however, they are fitted with foot valves in order to make progress with fewer changes. A check valve is placed in the discharge column and is arranged to be quicker acting, if possible, than the foot valve. If neither foot valve nor check valve is used, when the power stops, the receding column of water will drive the motor backward and may destroy it. A gate valve, for throttling, is always placed in the pump column.

High-Speed Cenrtifugal Pumps in Europe

The usual speed of centrifugals is 1740 or 1140 r.p.m. (1800 and 1200 synchronous speeds). The higher speed is better in most cases, the lower being used only occasionally where a high lift is not required and a greater smoothness of operation is desirable. Pump manufacturers would like to build centrifugals for a speed of 3500 r.p.m., but the electrical manufacturers have thus far refused to manufacture motors of such speed, except in the smaller sizes. However, motors and pumps of 3600 r.p.m. are now in use in Europe, and we could often gain many advantages by following such practice in this country.

Multistage centrifugal pumps are generally made for a lift of 150 to 225 ft. per runner. If the runners are made for a greater lift, the friction is apt to become excessive. Multistage pumps (both horizontal and vertical) are now nearly all built with split casings so that the runners can be easily removed. For very high lifts, where a large number of runners is necessary, the pump should be separated into sections, so that bearings may be placed between, as when more than six or seven runners are used between a single set of bearings, the shaft deflection is likely to become excessive. The voltages generally used on the Comstock in driving pump motors are 2200 for station pumps and 440 for shaft pumps. It is now general practice to carry high-tension current underground. In wet places it is advantageous to obtain motors which have the so-called "Panama Insulation," which will stand almost any amount of heat and moisture. This costs scarcely any more than the ordinary in ulation, but the manufacturers seem to require a much longer time for delivery. Could prompt deliveries be obtained, it would be the best insulation for any mine motors operated at 440 volts. It has not been made for higher voltages.

When motors have been drowned, the Comstock practice is to dry them out with the heat from a cluster of electric bulbs and with current at 38 volts, next 110 volts and finally 440 volts.

In shaft sinking, two pumps should always be used, each with its own discharge column and with its own air or steam pipe or electric cable. A telescope connection can generally be made at the mine from two pieces of pipe. Discharge connections of fire hose or of flexible copper tubing, are nearly always worth while in ordinary sinking. Where the mine plant does not include a separate hoist or winch to handle the pumps, wooden rollers are used to carry the main hoist rope between the centers and into the pump compartment. Where a hoist is not of sufficient capacity, the power can easily be doubled by attaching a 14-in. (or larger) sheave to the pump.

Air Lifts and Hydrualic Elevators

Air lifts in certain particular cases may afford the best method of removing water from the mine. They can only be used, however, when the situation is such that the necessary submergence can be obtained. For economical operation the submergence should be nearly twice as great as the lift. They will operate when the submergence is equal to the lift. Pumping will generally cease when the ratio of submergence to lift is 45 to 55. As compared with electrical pumps, air lifts are very extravagant in the use of power.

Air around a mine is generally carried at a pressure of 90 to 105 lb. Ninety pounds pressure corresponds to 207 ft. of water column. Therefore, if the air-nozzle of the lift is submerged more than 207 ft., it cannot be started by air at 90-lb. pressure. In order to pump to greater depth than 100 ft. (50% submergence) a compound air lift is used, having two or more compressed-air pipes, each controlled by valves at the collar of the shaft. When the water has been reduced by means of the upper nozzle and the pressure against the air correspondingly reduced, the air may then be turned in to the second air pipe and the level thus reduced below what would be possible by use of the upper nozzle alone. The air lift used temporarily in unwatering the North End Comstock mines is shown in Fig. 1. Where the water pumped by the air lift is to be taken away by gravity through a pipe, an air-separating chamber should be supplied at the top of the lift, as shown in Fig. 2.

Water jets or hydraulic elevators are occasionally used in mining operations. Small jets were formerly quite common in California mines. They will generally lift to a height equivalent to about 25% of the available water pressure, but the amount of water lifted depends largely upon the design of the jet. The hydraulic elevator formerly used in unwatering the North End Comstock mines is illustrated in Fig. 3. The nozzle was made of steel, and the water pressure at the nozzle when the elevator was at its last position at the 2500 level was 2900 ft. The throat and taper pieces were made of bronze, and the taper piece was lined with aluminum bronze. The linings lasted about 18 days. On account of its extreme vibration the elevator did not prove to be a practical success until compressed air was introduced into the throat-piece. This elevator could not be allowed to go into fork without shaking itself to pieces, and its efficiency decreased rapidly with the wear of the throat-piece liners. Generally, three gallons of pressure water would lift two gallons of mine water from the 2500 to the 2000 level. When first installed, this elevator was an economical method of pumping, as the delivery was then directly into the Sutro tunnel on the 1600 level. After electric pumps were installed on the 2000 level, it was temporarily employed below that point, but then became an expensive piece of machinery, because both the mine water and the power water had to be handled by the electric pumps. It was replaced by electrically driven centrifugal turbine shaft and station pumps.

Hydraulically operated plunger pumps were formerly much in use upon the Comstock and proved excellent machines. In the Combination shaft they were installed on the 2400, 2600 and 3000 levels and were used in conjunction with the Cornish pump. They lifted 2000 gal. per min. to the Sutro tunnel (1600 level), while the former was handling 1500 gal. Smaller hydraulically operated pumps were also used in the winzes in the North End mines. A hydraulic pump, using a Lakenan valve, has been operating in the North Star mine at Grass Valley, Calif., for many years at a minimum of expense. Such pumps, however, can never find more than a limited use for mining.

In designing mine-pumping plants, the greatest care should be used to combine a maximum of convenience and efficiency. A reasonable additional expense for convenient arrangement is nearly always regained the first time that any serious accident or lack of power occurs. Sand boxes should always be used. With centrifugal turbine pumps they are an absolute necessity. Recording instruments should be employed so that the power consumed by each pump can be noted. If they are not used, the pumpman should be required to take readings on the ammeter at frequent intervals. The water should be measured either by a weir at the discharge, or in the case of a large installation it can be measured and recorded by a venturi meter. The latter, however, must be placed either in the shaft or at some other point 20 ft. or more below the discharge.

Successful mine pumping is as much a matter of detail as any other modern engineering work. Each district and each mine has its own peculiar conditions, which should be taken care of in the design of the pumping plant.

Leasing Corporate Property in Colorado

By A. L. H. Street*

The following quoted Colorado statute has been interpreted by the United States Circuit Court of Appeals, Eighth Circuit, in the case of Elder vs. Western Mining Co., 237 Federal Reporter, 966, as invalidating a lease of a mining corporation's property where the lease has not been approved in the manner indicated in the statute:

"The board of directors or trustees of a mining or manufacturing corporation shall not have power to incumber the mines or plant of such corporation, or the principal machinery incident to the production from such mine or plant until the question shall have been submitted at a proper and legal meeting of the stockholders and a majority of all the shares of stock shall have been voted in favor of such proposition; and any mortgaging or incumbering of such property, without such consent, shall be absolutely void."

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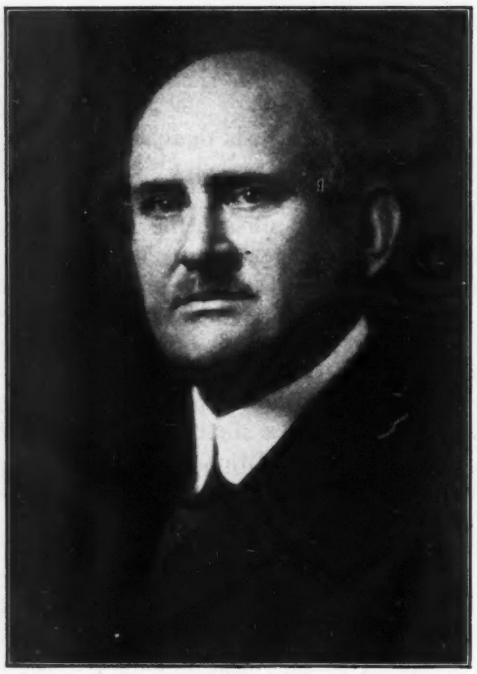
Distinguished Mining Administrators



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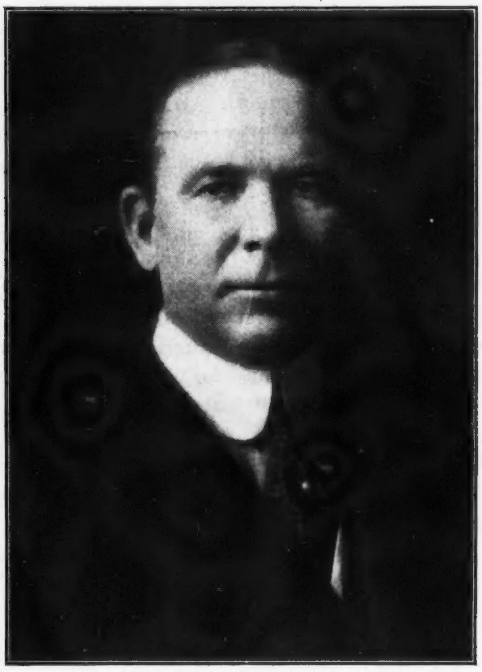
Vice president, Anaconda Copper Mining Co.



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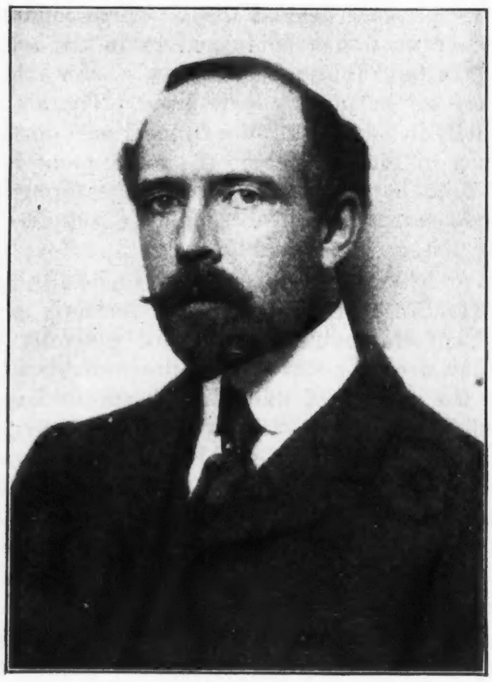
Managing director of the Hayden-Stone-Jackling properties

WALTER DOUGLAS
President, Phelps Dodge Corporation



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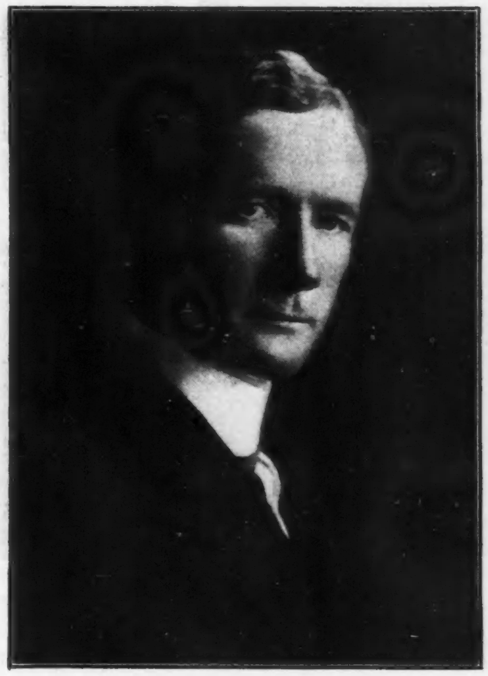

C. B. LAKENAN
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Underground Churn Drilling at Old Dominion

By Guy N. Bjorge*

SYNOPSIS—The initial use of a churn drill underground brought out defects in its design and the drill was practically rebuilt in the company's shops. After this its operation was successful and uninterrupted. Results and costs of the work before and after rebuilding show that with proper equipment churn drill holes for ventilation and prospecting purposes are cheaper than the same amount of sinking or raising.

In December, 1914, a churn drill designed for underground operation was purchased by the Old Dominion Copper Mining and Smelting Co., Globe, Ariz., to drill holes to serve two purposes; namely, for prospecting and for ventilation. There were several areas in the mine where prospecting by vertical holes ranging in depth from 100 to 250 ft. from the lowest level would give results that might save large amounts in the development of new levels. Large holes could be drilled from the 16th level to be used for the ventilation of long development drifts to be driven on the 18th level. The possibility of its use for ventilation led to the purchase of a churn drill rather than a diamond drill, and recent work has shown that a churn drill can be made to serve a useful purpose in the ventilation of new workings.

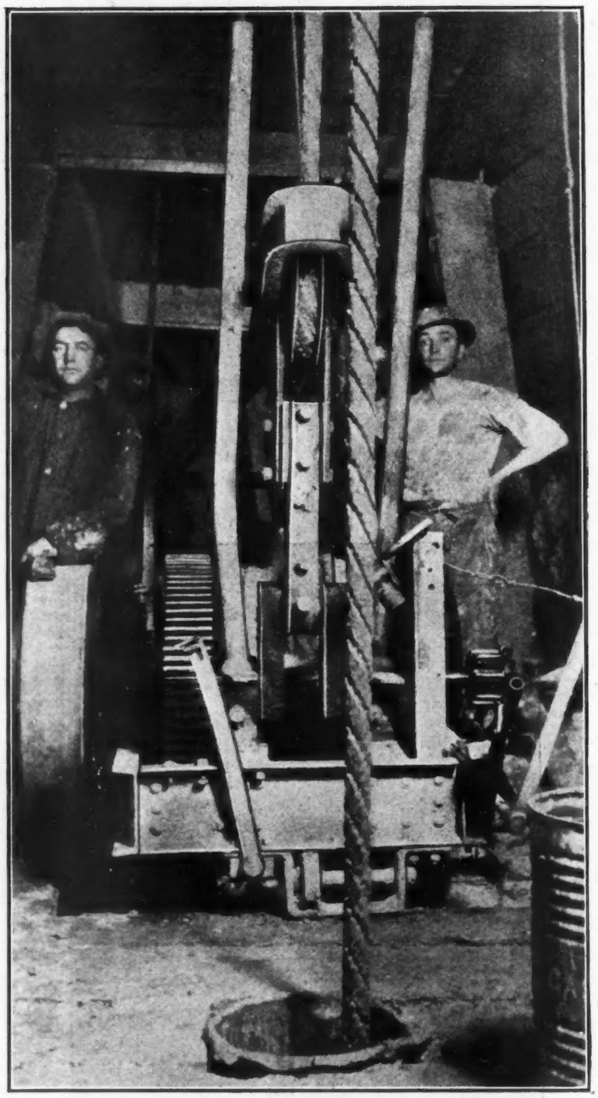


FIG. 1. UNDERGROUND CHURN DRILL AT THE OLD DOMINION MINE

*Mining geologist, Box 2434, Globe, Ariz.

Type of Machine and Changes Made

The machine was electrically driven and was so constructed that it could be moved on 18-in. gage tracks through ordinary drifts from one part of the mine to another. The frame was of 7-in. standard steel channels. The motor and controller were mounted on a separate truck, which was bolted to the drilling machine and securely braced when set for operation and could be easily detached for facility in moving. The mast was of 6-in. standard channel of ladder type with 3-in. standard channel braces and carried a crown sheave of 18-in. diameter. This mast was not rigid enough and was used only a short time. Where the drill is operating in good ground, the crown sheave is supported on sprags in the raise, which must be put up for headroom, and in other cases a spliced mast of 12 x 12-in. timber has been used.

The power was furnished by a 6-hp. General Electric special variable-speed motor with belt drive from a small pulley on motor countershaft to a 4-ft. band-wheel on the crankshaft. The machine was designed to drill 8-in. holes to a maximum depth of 300 ft., entirely by spudding.

Actual drilling began in February, 1915. During the next five months work was done on three holes. The machine soon developed a number of weaknesses, and the work was so hampered by frequent breakdowns that at the end of July, 1915, while drilling hole No. 3, it was decided to suspend operations and rebuild the drill. The weaknesses and disadvantages of the original machine were: (1) The band-wheel clutch, which was an internal-expansion friction clutch, kept slipping and breakages were frequent; (2) the crankshaft was too light and was bent; (3) the crankshaft bearings were too light; (4) the drive was from a small pulley on the motor countershaft to a band-wheel of 4-ft. diameter, and the belt kept slipping on the drive pulley; (5) the band-wheel was heavy and cumbersome; (6) in general, the parts were not strong enough and the frame not sufficiently rigid.

The machine was almost wholly rebuilt in the company's shops, and drilling was resumed late in October. Since that time the drill has been in continuous operation for 15 months and has given satisfactory service. The remodeled drill is shown in Fig. 2. The principal changes made were: (1) The frame was made of 8-in. standard steel channels with more bracing and was stronger and more rigid. (2) For the band-wheel and friction clutch there was substituted a gear on the crankshaft, and a new countershaft and pinion were added to drive the crankshaft. (3) The friction clutch for operating the spudding beam was replaced by a jaw clutch. The pinion is loose on the countershaft and forms half of the jaw clutch. A sliding collar on the countershaft works on a feather key which is operated by the clutch control lever.

(4) The power transmission is a gear transmission from motor to motor countershaft, belt transmission from motor countershaft to clutchshaft, with both pulleys about 2 ft. diameter, and gear transmission from clutchshaft to crankshaft. (5) The crankshaft is heavier and has three bearings instead of two. These bearings are heavier.

The over-all horizontal dimensions of the machine are 12 ft. 6 in. x 3 ft. 6 in., and 25 ft. headroom is required for the mast. The machine can be operated in an ordinary drift, though a little wider space at the head of the machine makes the operation more convenient. A 4 x 4 ft. raise to a height of 25 ft. from the rail gives sufficient space for the mast and crown sheave. Where such raises have been in good ground, the crown sheave has been supported on sprags in the raise and no mast has been used. The machine, in operating position, is shown in Fig. 1. In this case a 12-in. hole for ventilation purposes is being drilled. The cost of preparing sites has ranged from nothing for hole No. 9, where an old raise directly over the drift was utilized, to \$1.895 per foot of hole drilled for hole No. 12, where the entire space was cut out of solid ground. No. 12 was a ventilation hole, and the same space was later used for the blower and motor, so a portion of the cost could justly be charged to ventilation direct. Exclusive of hole No. 12, the cost of preparing sites has averaged \$0.351 per foot of hole drilled.

Up to date 11 holes have been drilled to an average depth of 160 ft., giving a total of 1760 ft. Table I gives general data on operation and a summary of performance and costs. In summarizing the performances and costs, the work has been considered in three periods, each representing certain conditions of operation.

TABLE I. GENERAL DATA AND SUMMARY OF COSTS

	First Period— Holes 1 to 3		Second Period— Holes 4 to 10		Third Period— Hole 12		Total	
	Hours	% of Total	Hours	% of Total	Hours	% of Total	Hours	% of Total
Moving	137.0	9.50	247	15.95	58	9.87	442.0	12.35
Drilling	504.5	35.02	1,032	66.62	457	77.72	1,993.5	55.72
Fishing	342.0	23.74	29	1.87	13	2.21	384.0	10.73
Repairing	239.5	16.62	127	8.20	26	4.42	392.5	10.97
Casing	95.0	6.59	105	6.78	2	0.34	202.0	5.65
Reaming	123.0	8.53	9	0.58	32	5.44	164.0	4.58
Total	1,441.0	100.00	1,549	100.00	588	100.00	3,578.0	100.00
Summary of Performance								
Total footage	441.500		1,124.000		195.000		1,760.500	
Footage per 8 hrs	2.451		5.806		2.653		3.936	
Footage per 8 hrs., actual drilling	7.001		8.713		3.415		7.058	
Summary of Costs								
	Cost per Ft.	% of Cost	Cost per Ft.	% of Cost	Cost per Ft.	% of Cost	Cost per Ft.	% of Cost
Repairing sites	\$0.306	3.96	\$0.369	8.21	\$1.895	27.05	\$0.522	9.34
Operating	5.951	77.07	3.126	69.54	4.111	58.67	3.944	70.65
Raising to recover tools	0.465	6.02	......	......	......	......	0.116	2.10
Proportion of initial equipment	1.000	12.95	1.000	22.25	1.000	14.28	1.000	17.91
Total	\$7.722	100.00	\$4.495	100.00	\$7.006	100.00	\$5.582	100.00

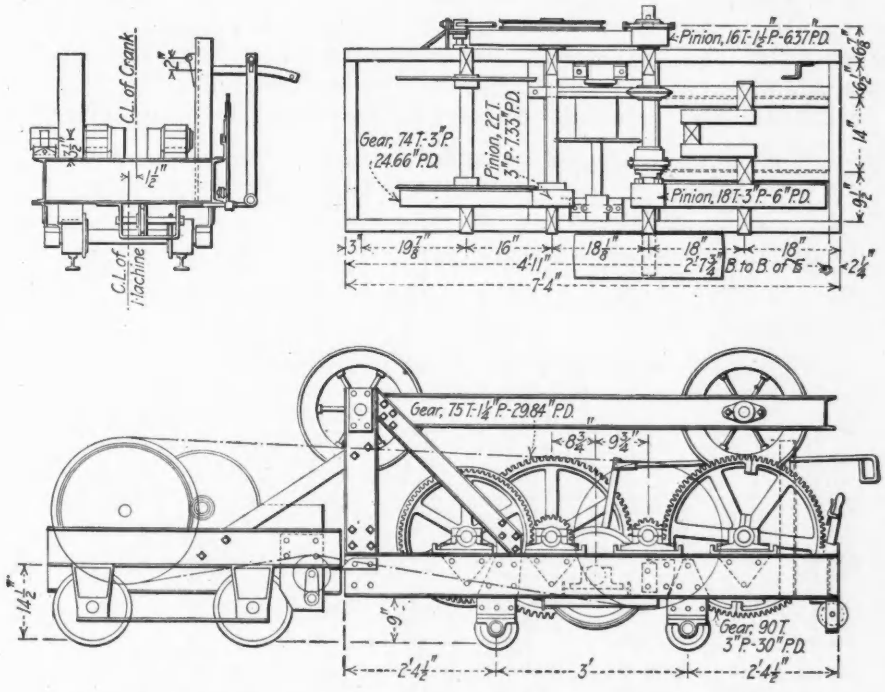


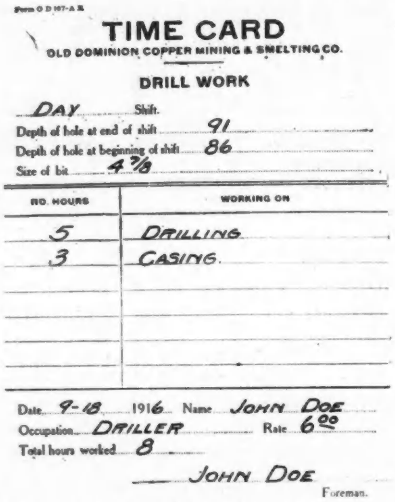
FIG. 2. DETAILS OF THE REMODELED CHURN DRILLING MACHINE

The machine is set for operation and held rigidly in place by small drill columns butting against the overhead timbers wedged and screwed up properly.

The drilling was done by one driller and one helper, working one 8-hour shift per day. The wages were on a sliding scale based on the selling prices of copper and averaged \$6.29 for the driller and \$5.29 for the helper for the total period. As the labor item constituted over 70% of the total operating cost, the high wage scale was a large factor in the cost. Bits were sharpened at the mine blacksmith shop on the surface. The log was kept by the driller on card forms, which were turned in at the end of each shift. The form is shown in Fig. 3.

The first period covers the first three holes, when the work was still largely experimental and while it was hampered by numerous breakdowns. Detailed costs of operation during this period are given in Table II. Hole No. 3, included in this period, gave more than an ordinary amount of trouble. The tools were lost twice, and the second time two sets of fishing tools were lost in attempting to recover the drilling tools and it finally became necessary to raise from the level below to recover the tools. The item of repairs was abnormally large during this period, amounting to more than it has for the entire time since. The second period covers holes Nos. 4 to 10. These were all prospect holes and were started with an 8¼-in. bit and finished with smaller bits where casing became necessary. In the area to be prospected, the vein has an average dip of 70 deg., and two holes were located in each crosscut on the 16th level so as to cut the vein at depths of 100 ft. and 200 ft. respectively. In this area both the hanging wall and the foot wall are diabase and the drilling was all in diabase and vein material, consisting largely of altered diabase with some vein quartz and including some blocks of silicified sediments. While in the wall rock, small samples were taken at 5-ft. intervals for geological examination. When the vein was encountered, regular 5-ft. samples were taken for assay. The full sludge was cut down to 10 to 15-lb. samples. This period gives a fair idea of the performance that may be expected from a machine of this type when used for prospect holes which are sunk about 200 ft. deep and starting with 8¼-in. and finishing with bits 6-in. diameter.

The third period includes only one hole, No. 12. This was a ventilation hole drilled from the 16th level to furnish ventilation for a new drift on the 18th level. The upper 33 ft. was diabase and the remainder a hard silicified limestone, and the hole was drilled with a 12-in. bit throughout. From a comparison with the second period



Form O D 107-A

TIME CARD

OLD DOMINION COPPER MINING & SMELTING CO.

DRILL WORK

Day Shift.

Depth of hole at end of shift 91

Depth of hole at beginning of shift 86

Size of bit 4 7/8

NO. HOURS	WORKING ON
5	Drilling
3	Casing

Date 9-18 1916 Name John Doe

Occupation Driller Rate 6.00

Total hours worked 8

John Doe Foreman.

FIG. 3. TIME CARD OF UNDERGROUND CHURN-DRILL OPERATION

it will be seen that the cost of drilling a hole with the large bit is approximately $1 per ft. more than the average cost of drilling the smaller holes.

The total cost of the initial equipment was $2382; the alteration to machinery cost $876, and the additions to equipment cost $492, making a total of $3750. The initial costs are being charged off at an arbitrary rate of $1 per ft. of hole drilled. As the initial equipment charge amounts to about 20% of the total cost, there will be a material decrease in costs when the equipment account is finally closed.

In opening the 18th level from the main operating shaft, the main drift was to be driven about 2500 ft. before a connection would be made which would serve to ventilate this level. The station at the 18th level was hot, and the air supply that could be drawn on from this point was poor. It was therefore decided to ventilate this drift through churn-drill holes from the 16th level. The first hole was started at a point 1000 ft. from the shaft and drilled to a depth of 196 ft. with a 12-in. bit. As soon as the 18th-level drift reached this point, a Baker rotary pressure blower, No. 4½, driven by a 20-hp. General Electric motor at 1025 r.p.m., set at the collar of the hole, was started, and in only a few hours the 18th-level drift, which had been the hottest place in the mine, became one of the coolest and best working places. As the drift was advanced, the air was carried to the face through a 12-in. galvanized-iron fan pipe. A second hole is now being drilled at a distance of 800 ft. from the first, and this will serve to ventilate the 18th-level drift until raises have been put up to the 16th level.

The advantages of churn drill holes for ventilation of new levels under such conditions as given are several: (1) the hole can be drilled through and ready to serve its purpose as soon as the drift reaches it, and the delay incident to raising for ventilation is avoided; (2) if in an area where no raise is needed for subsequent operation, there is a material saving over raising; (3) with 200 ft. between levels, driving a 200-ft. raise, which would be extremely hot and also expensive, is avoided.

The remodeled machine was designed by the company's mechanical-engineering department under the supervision of C. E. Mendelsohn. Thanks are due to C. E. Mendelsohn, mechanical engineer, and H. L. Norton, chief engineer, for material for portions of this paper and to P. G. Beckett, general manager, and I. H. Barkdoll, mine superintendent, for helpful suggestions and criticism.

TABLE II. DETAILED COSTS OF UNDERGROUND CHURN DRILLING

	First Period—Holes 1 to 3			Second Period—Holes 4 to 10			Third Period—Hole 12		
	Cost	Cost per Ft.	% of Cost	Cost	Cost per Ft.	% of Cost	Cost	Cost per Ft.	% of Cost
Labor:									
Moving	$288.91	$0.656	11.02	$552.68	$0.492	15.74	$65.65	$0.337	8.20
Drilling	602.50	1.364	22.92	1,612.27	1.434	45.87	484.65	2.485	60.45
Fishing	379.25	.858	14.42	55.08	.049	1.57	19.70	.101	2.45
Repairing	289.66	.656	11.02	179.30	.159	5.09	41.25	.212	5.16
Casing	110.35	.250	4.20	158.93	.142	4.54	15.00	.077	1.87
Reaming	151.00	.342	5.75	13.27	.012	.38	42.00	.215	5.23
Total	$1,821.67	$4.126	69.33	$2,571.53	$2.288	73.19	$668.25	$3.427	83.36
Supplies:									
Power	$56.09	$0.127	2.13	$73.24	$0.065	2.08	$19.30	$0.099	2.41
Rope	102.93	.233	3.92	13.98	.012	.38			
Repair parts	32.53	.074	1.24	103.54	.092	2.94			
Casing	58.61	.132	2.22	131.31	.117	3.74			
Miscellaneous	81.91	.186	3.13	24.56	.022	.70	1.75	.009	.22
Total	$332.07	$0.752	12.64	$346.63	$0.308	9.85	$21.05	.108	2.63
Shops:									
Sharpening bits	$256.58	$0.581	9.76	$309.58	$0.275	8.80	$93.00	$0.477	11.60
Mine machine	19.18	.043	.72	15.81	.014	.45	15.00	.077	1.87
Carpenter	8.66	.020	.34	8.75	.008	.26			
Electric	52.99	.120	2.02	196.67	.175	5.60	.70	.004	.10
Machine	36.18	.082	1.38	62.82	.056	1.79	3.60	.018	.44
Blacksmith	100.01	.227	3.81	1.80	.002	.06			
Total	$473.60	$1.073	18.03	$595.43	$0.530	8.16	$112.30	$0.576	14.01
Total operating cost	$2,627.34	$5.951	100.00	3,513.59	$3.126	100.00	$801.60	$4.111	100.00
Raising to recover tools	205.15	.465					369.55	1.895	
Preparing Sites:									
Labor				$259.80					
Supplies				117.80					
Air drills				23.12					
Timber				13.10					
Total	$134.95	$0.306		$414.32	$0.369				
Proportion of initial equipment	$441.50	$1.000		$1,124.00	$1.000		$195.00	$1.000	
Total cost	$3,408.94	$7.722		$5,051.91	$4.495		$1,366.15	$7.006	
Holes drilled	3			7			1		
Total footage	441.50			1,124.00			196.00		
Average depth of holes, ft	147.17			160.57					

Underground Tramming

By David B. Scott*

SYNOPSIS—Methods of underground tramming have developed through hand and animal power to systems using compressed air, electricity and internal-combustion engines as sources of power. The cost has diminished as more efficient mechanical methods have been applied. Compressed air and electricity are the most widely used powers.

Mine tramming has developed principally along the mechanical side of operation during the last few years, and there has been a consequent tendency toward the substitution of mechanical types in the place of those employing man and animal power. The development of mechanical tramming is at its best in the mines handling large daily tonnages, notably the low-grade coppers and the iron mines. Tramming by animal power still has a wide field in mines large and small, where concentrated handling of ore is impossible or where the tonnage does not justify expensive mechanical equipment. The cost of handling ore by the different means shows the principal reason why the mechanical types are favored. The cost ranges in general are as follows, for transportation alone: Man-power tramming, 20 to 30c. per ton; mule haulage, 8c, to 12c. per ton; mechanical haulage, 2c. to 5c. per ton.

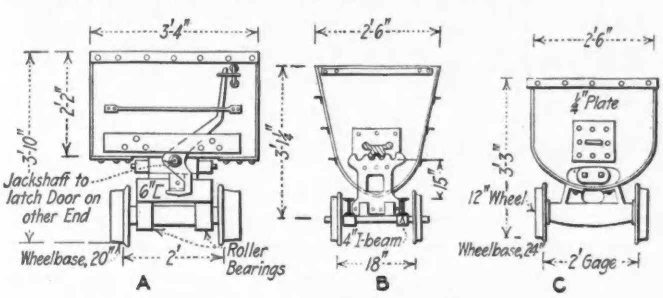


FIG. 1. END VIEWS OF WESTERN TYPES OF MINE CARS
A—30-cu.ft. end-dump car. B—Round-bottom, side-dump car, North Star mine type. C—1-ton tipple dumping car, no truck

Hand-tramming requirements have developed the car with small unit load to as high a degree as that reached by the large-capacity cars. In the small-car type the weight does not exceed one-fourth the capacity; in the heavy mechanical-haulage types the weight is usually one-third or more of the capacity. Car designs have certain distinctive features when the practice in Western and Eastern districts is compared. Western cars are almost universally of the short-body, narrow-gage type. The 18- and 20-cu.ft. cars are the common standards in Western mines, a capacity of 30 cu.ft. being rarely exceeded for hand-tramming. Track gages of 18, 20 and 24 in. prevail, and where constant duty is required, the 16-lb. rail proves the most satisfactory. The shape and dumping arrangements are largely a matter of local opinion, and the types now in use are box or round-bottomed, with end or side dump. Sketches of a few types are shown in Fig. 1.

Hand-cars in use in the Michigan and Minnesota country and in other metal-mining districts of the East differ from the Western cars in long, low bodies, wide gage and short wheelbase. Many districts use cars with one or both ends open. In the Copper country the usual track gage is 3 ft. 4 in., occasionally 4 ft., made necessary by the large capacity of 2 or 2½ tons. The car height above the rail ranges from 2 ft. 6 in. to 3 ft. commonly. Many mines use cars with one end open, but several of the Lake Superior copper mines make use also of the door type. Mechanical haulage is rather uncommon in the Lake Superior copper district, the character of the vein deposits being such as to make haulage concentration difficult. Hand-tramming with large open cars over long leads is the general practice. According to published figures, for example, the Calumet & Hecla mines, two years ago, had an average tram of 609 ft., with a shift output of 14.8 tons trammed per man.[1] As practically all tramming is done to shafts, the situation of the latter is governed considerably by tramming considerations. Some Eastern car types are sketched in Fig. 2.

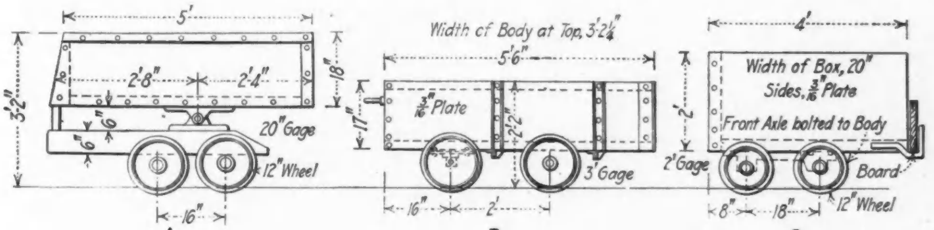


FIG. 2. SKETCHES OF EASTERN TYPES OF HAND CARS
A—Open-end 20-cu.ft. sublevel car. B—1½-ton iron-ore car as used at Mineville, N. Y. C—15-cu.ft. open-end car for slices

The use of cars with unit loads greater than 2 tons is required for the most economical operation of mechanical tramming systems. The design of these cars is governed by the dumping requirements. For continuous and simple dumping, the gable-bottomed car has been highly developed in the Michigan iron mines, in the Missouri lead districts and in a few of the Southwestern mines. Where tipple dumping is practiced, the rigidly mounted flat-bottomed car is required. Conspicuous examples of this type are found in the Ray Consolidated and Inspiration mines of Arizona, where 5-ton cars of this type are used to handle a combined daily tonnage for the two mines of more than 25,000 tons. The gable-bottom is not applicable to sticky ores such as are found in Bisbee or Butte, but is highly successful with the hard and soft ores found in the Miami district, Arizona. The deep, flat-bottomed tipple car is open to the objection that fine ore packs in the bottom, leaving a "skull," or shell, that is not freed by tippling.

The gable-bottomed car, evolved largely in the United States Steel Corporation mines of the Northern iron districts, and used considerably in the West, is com-

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[1] "Eng. and Min. Journ.," Nov. 21, 1914, p. 908.

monly of 3- or 4-ton capacity. Its load is released by hand through side-swing doors. As the security of the load is dependent on the locking device, it is interesting to note the various means employed. Three successful and considerably used types are shown in Fig. 3, from the St. Joe mine in Missouri, the Miami copper mine in Arizona, and the Oliver iron mine of Michigan. In Fig. 3 *A* locks with a hand lever at each end of the car, *B* with a lever at one end only and *C* with a swinging lever which, when at dead-center, keeps the doors shut. Cars of this type are swung on springs, while the tipple-car pattern cannot conveniently use springs because unequal loading of the car will not permit entrance into the tipple frame.

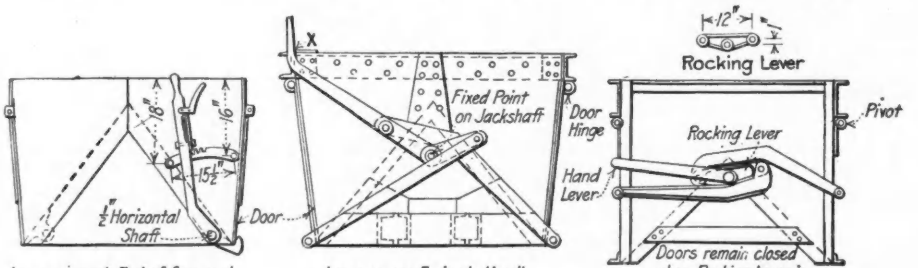
The use of locomotives deriving power from electricity, compressed air or gasoline engines is constantly being extended, even where concentrated haulage is impossible. Mine locomotives are easily capable of 500 ton-miles duty per shift; the displacement of mule haulage with its maximum duty of 150 ton-miles per mule shift is readily explained. In some districts, as in the Missouri lead country, mules are used for gathering the cars which are trammed to the shaft by motors.

Electric haulage was the pioneer in locomotive haulage, one of the earliest installations being in 1886 at the Hillside Coal and Iron Co. It is now in use in nearly every mining district, in both coal and metal mines, the dominating type being the trolley motor operating at 250 volts or less. Haulage by compressed air is a more recent development that is competing successfully with electric systems in all classes of service. It has displaced the electric locomotive in gaseous coal mines and some of the large metal mines. While it is not within the scope of this article to discuss in detail the merits and performance of these two types of haulage power, it can be said that safety is usually the deciding factor in making a choice. In mines having very large equipment, it is said that the cost of installation of the two systems is about the same. The operating cost gives neither type a definite advantage unless cheap hydro-electric power makes a power-house equipment unnecessary, in which case the electric type is favored. Elimination of the exposed trolley wire and the fire danger from feeder lines make compressed air attractive to many operators.

The power consumption of electric and compressed-air systems per ton-mile (reduced to kilowatt-hours for compressed-air systems using electric power for compressing air) can in general be rated within the following limits: Electric trolley systems, 0.8 to 1.0 kw.-hr.; electric storage-battery systems, 1.6 to 1.8 kw.-hr.; compressed-air systems, 0.8 to 1.0 kw.-hr. The power requirements of the electric and compressed-air systems show little difference in efficiency. The power consideration is, however, second in importance to maintenance, upkeep and flexibility.

Electric mine locomotives range in size from the small 3-ton units to the 30-ton Baldwin-Westinghouse locomotives at the Charleroi mine in Pennsylvania. Metal mines as a rule do not use a larger size than the 10-ton locomotive. The Butte mines commonly employ 3-ton motors operating at 250 volts on an 18-in. gage track. Many of the Missouri lead mines use 6-ton, 250-volt locomotives. Several of the Arizona mines, such as the Copper Queen and the Miami, use 6-ton sizes operating at from 210 to 250 volts. The Ray Consolidated has discarded 10-ton locomotives in favor of compressed-air haulage. An important feature of the trolley locomotive is the low body and compactness of design that permits wide drift clearances. This makes possible the use of small track gages and relatively small drifts, as in the vein mines having heavy ground such as are found in Bisbee, Butte and many of the iron mines. Electric haulage is almost universal in the Michigan iron-mining country, among the plants having considerable installations being the Newport, the Chapin and other mines of the Oliver Iron Mining Co. and the mines at Iron River.

Trolley locomotives can be classed according to the motor types—single-motor or two-motor. Illustrations of these two types as furnished by the Goodman Manufacturing Co., are shown in Figs. 4, 5 and 6. The single-motor type is characteristic of metal-mine equipment, as it permits the use of narrow gages without the sacrifice of motive power. In arrangement the armature lies along the long axis of the locomotive or parallel with the rails and is geared to both axles. Double-end control with controller boxes at each end of the motor is usual in single-motor types and is a very useful feature.



Levers on each End of Car, each Lever operating one Door
A

Lever on one End only, Handle latched at X
B

C

FIG. 3. EXAMPLES OF SIDE-DUMP DOOR LOCKS FOR HAULAGE CARS

The two-motor locomotive is employed principally in coal mining, where wide track gages and heavy grades prevail. This type with independent motor drive for each axle requires a heavier construction, and the range in weights is usually from 5 to 15 tons. These motors ordinarily have single-end control.

A more recent application of electricity to mine haulage is found in the storage-battery locomotive. This type has been used in many districts both as an auxiliary and as a main unit in the haulage systems. Like the compressed-air locomotive the storage-battery motor eliminates the trolley wire and track bonding and can operate wherever the track is laid. Unlike the compressed-air locomotive, it is not adapted to handling heavy loads on long hauls. Its field is principally in development and advanced headings. Because it carries its power source on its back, the storage-battery motor has a smaller hauling capacity than the trolley motor of the same size; in other words, more trips are required to handle the same tonnage. It is therefore best adapted to conditions requiring small unit loads, and to service conditions that will permit battery recharging for at least 9 out of 24 hours. For additional capacity per locomotive, batteries are occasionally mounted on a trailer. One of the largest systems recently in use, at the Big

Five tunnel in Colorado, permitted the handling of 80-ton trains by a single locomotive which had a trailer car for additional battery power.

Recent practice indicates that storage-battery locomotives range in weight from 2½ to 6 tons. In the Jeffrey, Westinghouse-Baldwin, General Electric and other types, the battery is mounted on the top of the locomotive and contains usually from 60 to 80 cells. The voltage is therefore lower than in trolley systems, ranging from 125 to 150 volts, which eliminates some of the commutator-sparking troubles found in trolley motors. Typical installations in Western properties are found at the Bunker Hill & Sullivan, the Alaska Treadwell and the Alaska

tives on a 30-in. track. Two-stage types of locomotives are used almost exclusively, since they are more economical, and the installation cost of the plant is less than that for the single-expansion type. The two-stage pattern has been developed since 1908. According to the H. K. Porter Co., the saving in compressed-air consumption was 30% as compared with the single-expansion engine. While a detailed description cannot be presented here, the chief features of the most up-to-date type of installation may be mentioned.

The air is expanded in two cylinders successively, and between the high- and low-pressure cylinders is interposed an interheater, the function of which is to raise

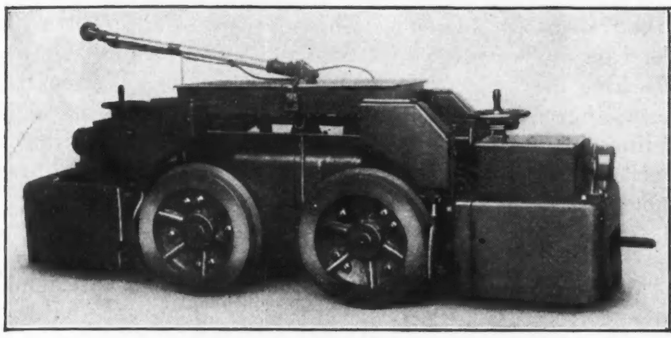


FIG. 4. SINGLE-MOTOR 6-TON LOCOMOTIVE

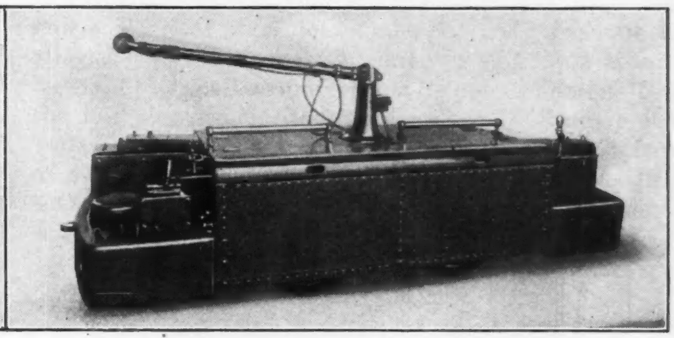


FIG. 5. SINGLE-MOTOR INDUCED-TYPE LOCOMOTIVE

Juneau mines. At the Idaho mines 2½- and 4-ton locomotives are used; at the Alaska properties the sizes are 4½ and 5½ tons. In Arizona the Miami Copper Co. has made successful use of the storage-battery motor in development work.

The use of compressed air in mine haulage has been developed entirely in the last 20 years. The question of safety is largely the factor in determining the choice between electric and compressed-air installations. As already noted, equipment costs compare favorably with those of electric systems of the same capacity. In opera-

the temperature of the air exhausted from the high-pressure cylinder before it is expanded again in the low-pressure cylinder. As the air from the high-pressure cylinder exhausts at a temperature of about —70° F., it is obviously a great gain in economy to bring this air to approximately the same temperature as the atmosphere before it enters the low-pressure cylinder. The locomotives made by the H. K. Porter Co. admit air to the high-pressure cylinder at 250 lb. and to the low-pressure cylinder at 50 lb. The storage or charging air is usually under a pressure of 900 lb. per sq.in., with a range from

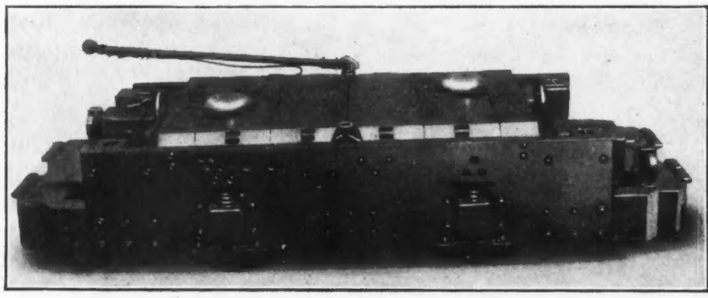


FIG. 6. TWO-MOTOR DOUBLE-END 13-TON LOCOMOTIVE

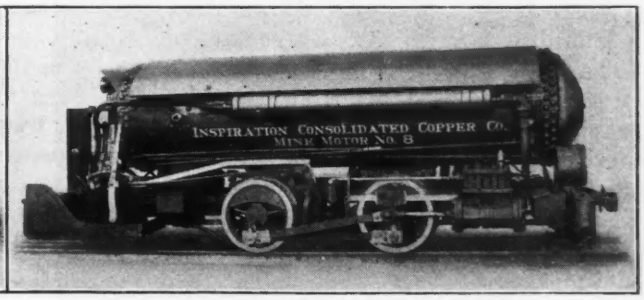


FIG. 7. 10-TON TWO-STAGE COMPRESSED-AIR LOCOMOTIVE

tion the compressed-air system has advantages in great flexibility, high acceleration, and speed of operating trains. In mines using the tipple type of car dumping, it has an advantage in eliminating the electric conductors around the tipples.

Compressed-air locomotives, while more bulky in size than electric motors, have the same range of application as the latter to metal mining; they are used in both the lode and large disseminated deposits. The common range of sizes now in use is from 4 to 10 tons. Haulageways do not necessarily have to be large to accommodate this type. The Homestake mine employed 4½-ton locomotives on 18-in. track; the Inspiration mine uses 10-ton locomo-

700 to 1200 lb. The employment of cutoff expansion with relatively low operating pressure makes it possible to use lighter cylinder construction than if the high charging-pressure air were admitted directly to the pistons.

Recharging of locomotives requires hardly 3% of the working time of each shift, as a rule. The operations at the plant of the Inspiration Consolidated Copper Co. can be taken as an example of the most recent practice. The 10-ton locomotives used here have a haulage capacity of about 50 ton-miles. The charging is done each round trip and consumes generally 45 sec. for the whole operation. With normal duty of 20 trips, this requires

15 min. out of the 8-hour shift. An illustration of a recent type of two-stage locomotive built by the Porter company is shown in Fig. 7.

For both hand-tramming and mechanical haulage, cast-iron or steel wheels are used with chilled-steel or manganese-steel tires. Coning of the wheels to keep the flanges away from the rail has, of course, special usefulness under the usually poor operating condition of mine tracks. The relations of wheels to axles brings in features not common in surface haulage. There are four general methods of wheel attachment: Both wheels rigidly attached to the axles; one wheel attached and the other moving freely on the axle; both wheels moving loosely on the axles, and the axle in halves, each with its attached wheel, called the split-axle type. The first type is applicable to straight track and is not greatly used in underground haulage unless the curves are of long radius. The second style with one loose and one fixed wheel, is widely used as the most satisfactory method of reducing friction; it reduces hub wear to almost nothing as compared with the first type. One

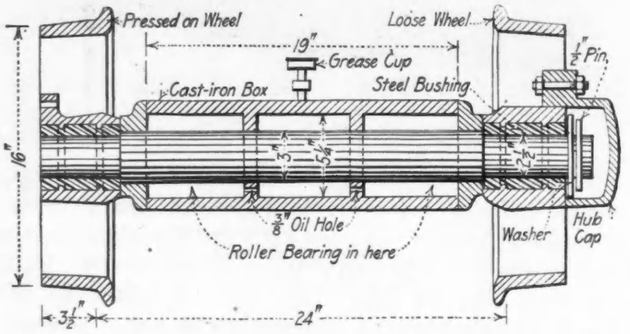


FIG. 8. ASSEMBLY OF ROLLER BEARINGS AND WHEELS, RIGID-BOX TYPE

wheel is pressed on at 6-ton pressure and the other is keyed on. The third type with two loose wheels is used in light tramming and for light trucks. The split-axle, or Anaconda type, with each wheel on a separate axle, is claimed to represent the best engineering practice.

The use of roller bearings has spread so rapidly in the last five years that it is not uncommon to find them in the smallest types of hand-cars. According to tests described by Mr. Liebermann,[2] the saving in draw-bar pull in favor of roller bearings is so large that a given locomotive can pull 90% more roller bearing cars on level track than plain bearing cars, at equal speeds. The point of application of rollers is important, and bearings can be classed as (a) wheel-hub bearings, (b) bearings in outside boxes, (c) bearings in inside separate boxes, (d) inside bearings in one rigid box. In general it is recommended that bearings in wheels be used for light tramming and that box bearings, inside or outside of hub, be used for heavy haulage. The type with bearings in the hub is used almost exclusively in Eastern soft-coal mines and in some of the Southern iron mines. Where service is heavy, the use of rollers in separate boxes has been found to be troublesome because of boxes getting out of alignment, with consequent roller breakage. The use of a rigid box extending from wheel to wheel and containing both bearings has proved a great improvement. The application of this type as used by the Miami Copper Co. and elsewhere is shown in Fig. 8.

[2]Am. Inst. of Mining Engineers, Bull. 114, 1916.

Lubrication is seldom required for the rollers oftener than once in six months, but systematic greasing once in two weeks has been found of great advantage in reducing friction between the box-cap and wheel-hub. For roller bearings a thin grease is required; for plain bearings, a black oil.

Couplings with light cars can be of the link-and-pin or chain-and-hook type. When loads heavier than two tons are moved in long trains, drawbar couplings are needed to prevent pulling of the trains apart. With the link-and-pin combined with drawbar couplings it is necessary to regulate the link length to the track curvature. With 25-ft.-radius curves for instance, a 12-in. link is needed, and with long trains the "take-up" in the train is considerable. This makes difficult "spotting" of cars at chutes, so that we have the usual practice of backing the train when loading from chutes. Automatic couplings are used in many large installations and eliminate much of this train stretching, the stretch per car being 3 in. as against 12 in. for link couplings under the same conditions.

Track-Equipment Materials and Methods

Mechanical tramming requires rails that will withstand the stresses of the heaviest unit, so that all calculations are based on the locomotive weight. The minimum rail sizes required, as published by various locomotive builders, are usually figured on an approximate formula: $W = \frac{10 \times L}{4}$ for 4-wheeled locomotives, where W is the rail weight in pounds per yard and L = locomotive weight in tons. In practice at least 40% is added to this weight, depending on the character of service, roadbed, etc. Rail of sufficient weight to keep the track alignment is useful only when proper attention is given to the spacing of ties, ballasting and drainage. For electric haulage heavier rails have decided advantages in keeping bonding in good condition. With ties spaced at 2-ft. centers with regular ballasting, 30-lb. rails, for instance, are satisfactory with 6-ton locomotives. It has been found, however, that better results are obtained under these same conditions by using 45-lb. rails.

Ties of pine, white cedar and creosoted pine are in common use—the last two varieties being used to prevent rotting. The softness of white cedar makes its use on sharp curves rather undesirable, as in a wet roadbed the spikes have a tendency to loosen.

With the rigid-wheelbase, characteristic of mine locomotives and cars, the minimum radius of curve is dependent on the wheelbase length and the play of the wheels. An approximate formula for determining minimum curve radius is used by some manufacturers as follows: $R = \frac{0.76 \times W}{2p}$, in which R = minimum radius in feet, W = wheelbase in feet, and p = wheel play in decimals of a foot.

The short, rigid wheelbase produces great wear on curves, so that short curves of long radius are preferable. Increase of gage on curves by a maximum of 1 in. is common practice. Adoption of standard haulage curves is now the practice in all large mines, the Copper Queen, for example, using 40-ft. to 80-ft. radius on a 20-in. gage; the Miami a minimum of 25-ft. on 24-in. gage; and the Inspiration, 60-ft. radius on 30-in. gage.

Managers of the Big Porphyry Coppers

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Chino Copper Co.

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Miami Copper Co.

LOUIS S. CATES
Ray Consolidated Copper Co.

ROBERT C. GEMMELL
Utah Copper Co.

The Testing and Application of Hammer Drills*

By Benjamin F. Tillson†

SYNOPSIS—Tests of hammer drills made by the New Jersey Zinc Co. resulted in the scrapping of the reciprocal drills and all of the work is now done by the one-man hammer drill. Thorough efficiency tests were made, developing more satisfactory conditions and forms of drills and bits. Records and comparisons are shown.

The hammer drill rightly receives the credit for having produced the one-man drill, and so many economies seem possible through the proper application of different types of hammer drills to various mining, quarrying and excavating operations that an indication of the economies effected by the New Jersey Zinc Co. at its Franklin mines may be of interest. When this company commenced its trials of hammer drills in 1907, these tools had not been developed to one-fourth the capacity and refinement they have since reached. At that time it was frequently said that such a small tool, drilling holes of less diameter than the reciprocating rock drill, could not drill enough holes in a shift to permit the placing of sufficient explosive to break a tonnage of ore comparable with that produced by the "rock drills"; that the placing of small holes inclined upward at angles steeper than 40 deg. above the horizontal could not be expected to produce results equal to the large flat, wet or dry, holes in the breasted back of an overhand stope and would only shatter the ground so as to make the back unsafe. In spite of these adverse opinions the hammer drills first showed their superiority over both heavy and light reciprocating drills in raising and in stoping, and then in drifting and quarry work. As a result, all of the reciprocating drills at the Franklin mines were scrapped three years ago, all of the mining work being accomplished with increased efficiency, as shown in detail in this article.

With the advent of the hammer drill in this property, it was considered advisable to make comparative tests of all the tools accessible, and it has since been the policy to investigate the merits of any advance of the drilling art in order to get the maximum amount of work from the tools. The necessity of devising some means of standardizing drill tests, and of measuring the consumption of compressed air as well as the drilling speed, was early realized.

The common test was to fill a measured air receiver with compressed air at a certain gage pressure, run the drill until the pressure had dropped to too low a figure, then compute, from the time, drop in pressure and capacity of the receiver, the cubic feet of free air used. This was not considered a fair indication of the drilling capacity of a machine, since the performance of some drills did not vary directly with the absolute pressures of the compressed air.

It was therefore found expedient to build a water-displacement air meter with which the drill test could be carried on for any length of time without serious variations in the desired air pressure. This apparatus, as shown in Fig. 1, consists of two tanks, half-filled with water, and made of 12-in. pipe with blind flanges, gage-glasses being mounted on one, a four-way cock connecting the compressed-air supply pipe with both tanks and the air line going to the drill. This device gives more accurate results than the common types of water or gas meters; and since any errors are due to the human element of reading the gage-glasses and reversing the four-way cock, they tend to be compensating throughout a number of tests.

The procedure is as follows: Air is drawn by the drill from the receiver *C*, which tends to trap any moisture carried over by the air from tanks *A* and *B* and assures a constant pressure while the four-way cock is reversed. In the arrangement shown in Fig. 1, the receiver draws its supply of air from the tank *B* and the water rises in

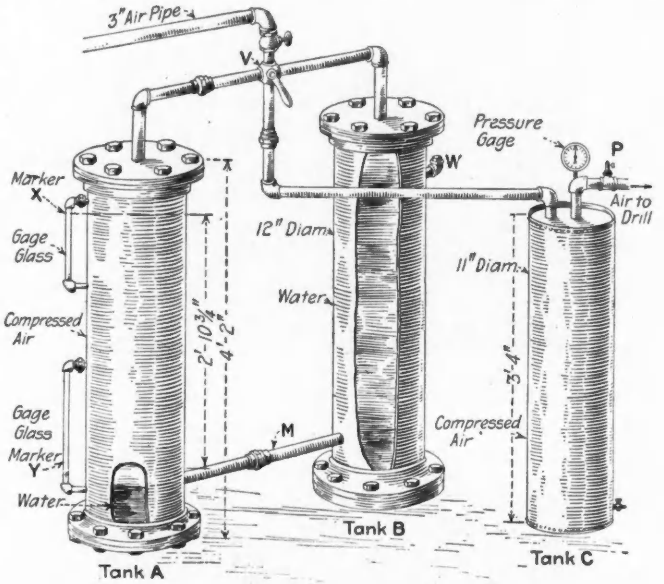


FIG. 1. ARRANGEMENT OF TANKS FOR TESTING AIR DRILLS

this tank by virture of the pressure of the air admitted from the air main through the four-way cock to the top of tank *A*, where the water is being forced downward and through the 2-in. connecting pipe to tank *B*. When the water has risen to a certain point near the top of the gage-glass in tank *B*, the four-way cock is reversed and the inlet air is supplied to the top of tank *B*; the drilling air is then taken from the top of *A*, the reversal of the cock again being made when the water in tank *B* has fallen to a point near the bottom of the gage-glass.

A pet-cock is placed on the top of tank *C* so as to permit the bleeding of air to bring the water columns to the desired point for starting a run, and another pet-cock is attached to the bottom of the same tank in order to permit the drainage of water. For convenience in measuring and computing, a run is made on the supply of compressed air indicated by a certain number of reciprocations of the water columns between fixed points on the gage-glasses; the pressures are measured on the air gage mounted on tank *C*, the length of time is taken by a stop watch, and the consumption of free air per minute is computed.

*This paper was originally presented at the February, 1915, meeting of the American Institute of Mining Engineers. For the purposes of this publication Mr. Tillson has amplified the paper materially and brought it up to date—Editor.

†Franklin Furnace, N. J.

Unless a pressure regulator is installed between the four-way cock and the receiver *C*, or else a globe valve at this point is operated manually to throttle the air so as to maintain a constant pressure, it is evident that the air pressure at the drill will vary in accordance with the water column supported by the inlet air pressure, but since the gage-glass marks are in this instance set 34¾ in. apart, the maximum variation in pressures is about 1¼ lb. per sq.in.; it is difficult to find a pressure regulator which will control a pressure of 90 or 100 lb. per sq.in. to a closer degree of accuracy, and the sensitiveness of drills to air pressures and the accuracy of time and distance measurements rarely exceeds this error.

It is obvious that in comparative drill tests the following factors must be considered: The nature of the rock drilled; the gage of the drill bits and their form and condition; the maintenance of equal compressed-air pressures; similar inclination and approximate depths of drill holes; equal vigor in the rotation of hand-rotated tools; and proper fit of the drill shanks in the chuck bushings, as well as their construction so that the blows are delivered on a plane surface of proper size at right angles to the axis of the drill steel and at the center of its shank end. In the tests summarized in Table I, 1¾ in. has been taken as the standard diametral gage of the bit, since it is a dimension which averages the gages of drill steels used in reciprocating rock drills and is fair in determining the performances of such tools; it also represents almost the largest gage necessary in hammer-drill stopers or block-holers, so that equal or even better performances may be expected from them as a hole is deepened with the smaller gages in a set of drill steel. At Franklin the testing rock is a compact coarsely crystalline white limestone, which greatly resembles a marble, and this rock proves a fair average of the various qualities of ore met in the mining operations. Although it is not hard, for a well-tempered drill bit can drill 3 or 4 ft. of hole before its cutting edges are materially dulled, and although it seems to chip freely, yet it possesses a compactness and toughness that is likely to prove surprising to one who has not previously tested a drill in it. Tests with various machines in Franklin and elsewhere indicate that this white limestone does not cut quite so fast as a sharp drill can achieve in Cripple Creek granite, is about on a par with Barre granite, and cuts slightly faster than Quincy granite. The chief difference is that a good drill will cut this limestone as fast in the second or third minute of its run, while it would have been dulled by the first minute's run in the granites, causing its cutting speed to fall off materially in the second and third minutes. Raised-center cross-bits are the standard type used with solid steel in these tests, and flat-faced six-point bits are generally used with the hollow steels; in general the cutting speed of these bits being about the same if the rotation of the drill steel is free. Fig. 2 shows the forms of the drill bits used at the Franklin mines.

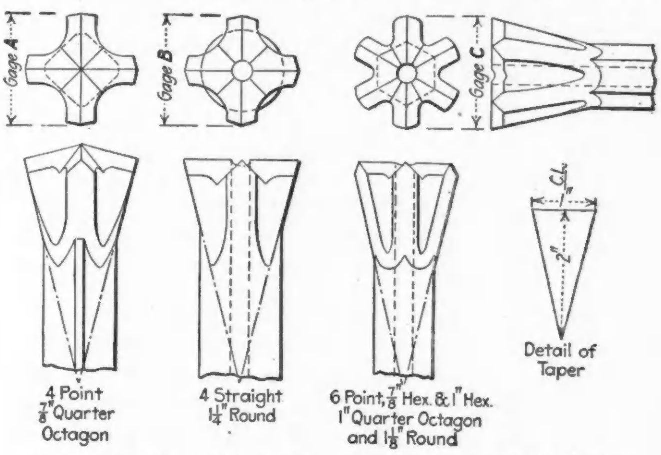


FIG. 2. DRILL BITS USED AT THE FRANKLIN MINES

Fig. 3 shows the forms of moils used with a pneumatic tool (built for a coal pick) which proves of great value in replacing the hammer and moil in cutting "hitches" in the rock as a footing for timber. The collar on the shank of the steel in conjunction with a spring retaining device on the pneumatic hammer prevents the moil from being "shot out" to injure a neighboring workman and absorbs the shock of the blows when the tool is away from the ground. This tool may therefore be handled with safety and comfort to the workman.

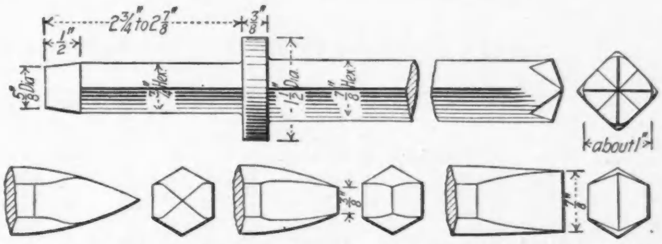


FIG. 3. FORMS OF MOILS USED WITH A PNEUMATIC TOOL

The results in Table I are fairly complete to date, and represent 45 different designs of drills made by 12 different companies, and indicate that the principal advances in the art, in so far as drilling speed and power economy are concerned, occurred in 1910 and 1911. It is believed that this table includes the best drills on the American market and so represents the development of the hammer drill from its infancy. During the last four years mining operators have been gradually learning to appreciate the value of the automatically rotated drill, and so the number of them has increased lately; but the majority are of the "rifle-bar" type (where the term is used generically to indicate that the piston is guided by a helix on one stroke and straight on the other). It seems that the possibilities of high drilling speeds and efficiencies are more limited in this type than where an auxiliary independent motor is used to rotate the drill steel, for in the latter instance the proper rotation of the steel is not influenced by sluggish piston action and neither is the force of the striking blow of the piston dependent upon the freedom of the rotation of the drill steel in the hole. In order to avoid invidious comparisons between the different makes of drills, symbols have been used to designate each certain make and design of drill. The following abbreviations have been used in the accompanying table.

Auto aux V = automatic auxiliary valveless control of rotations.

Auto rifle = automatic rotation caused by a piston reciprocating as though it were controlled by a rifle bar.

Dir. air = direct-air feed, or one in which the feed cylinder is rigidly attached to the hammer cylinder and in which the feed piston or plunger extends from the rear end of the machine by virtue of the air pressure applied to it.

Rev. air = reversed-air feed, or one in which the feed piston is rigidly attached to the hammer cylinder and the feed cylinder is free to extend backward, so readily adapts itself to the usual column mounting of stoping drills.

Some tests were included in this table for the consideration of points to be made later. Before studying the improvements in hammer-drill efficiency, it seems wise to explain the reasons for offering the figures in the last column of figures as representing a factor of "drill desirability."

In determining the relative merits of rock drills, whether of the reciprocating or hammer type, the logical basis is one of cost. Therefore, the drill which bores a foot of drill hole of standard cross-section at the lowest cost rate for drilling labor, power and maintenance (including amortization) would have the highest "factor of desirability," and a formula to express this may be developed as follows:

Let

F = "Factor of desirability";

D = Cost of drilling labor per foot of hole;

P = Cost of power per foot of hole;

M = Cost of maintenance per foot of hole.

Then

$$F = \frac{1}{D + P + M}$$

Let

t = Period of time for drilling-speed test, in minutes;

d = Depth of hole drilled in time t, in inches;

$S = \frac{d}{t}$ = Drilling speed during actual running of machine, in inches per minute;

L = Hourly wage of drilling labor, in cents;

O = Percentage of time spent in drilling to total operating time, including the changing of drill steels and shifting to new positions and starting of new holes.

TABLE I. SUMMARY OF TESTS OF REPRESENTATIVE DRILL TYPES MADE BY THE NEW JERSEY ZINC CO.

	Type of Drill											
Symbol for Drill	Date	Rotation	Control of Piston	Style of Feed	Weight	Air Press. Lb. per Sq.In. (Gauge)	Shape of Drill Bit	Free Air, Cu. Ft. per Min.	Drilling Speed, In. per Min.	Free Air, per In. Drilled	Factor, In. per Min. Divided by Free Air per In.	Condition of Bit and Remarks—General—Air measured by displacement of water. Rock was white crystalline limestone. Gage of bits was $1\frac{11}{16}$ to $1\frac{3}{4}$ in.
A	7/27/09	Hand	Valve	Dir. air	75	93	Raised crux	60.4	3.85	15.62	0.246	1 corner broken, with extension.
A_1	7/27/09	Hand	Valve	Dir. air	84	92	Raised crux	73.2	3.83	19.15	0.200	With extension.
B	7/27/09	Hand	Valve	Dir. air	76	90	Raised crux	90.0	3.20	28.20	0.118	
1B	7/27/09	Hand	Valve	Dir. Air	76	72	Raised crux	76.8	4.28	17.90	0.239	Different drill of above model.
C	7/27/09	Hand	Valve	Dir. air	64	90	Raised crux	63.0	5.10	12.37	0.414	
D	7/27/09	Hand	Valve	Dir. air	74	90	Raised crux	92.7	4.50	20.60	0.218	
D	7/27/09	Hand	Valve	Dir. air	74	85	Raised crux	86.2	5.52	15.6	0.354	Bit broken
E	7/27/09	Hand	Valveless	Dir. air	84	87	Raised crux	42.0	1.93	21.7	0.089	
F	7/27/09	Hand	Valveless	Rev. air	68	95	Raised crux	69.8	3.38	20.3	0.167	
G	10/1/09	Hand	Valve	Dir. air	78	88	Raised crux	75.8	5.65	13.4	0.421	
G_1	10/8/09	Hand	Valve	Dir. air	73	92	Raised crux	79.5	5.91	13.4	0.450	
G_2	10/25/09	Hand	Valve	Dir. air	73	88	Raised crux	79.5	6.00	13.2	0.455	
G_3	12/6/09	Hand	Valve	Dir. air	78	96	Raised crux	83.5	7.00	11.9	0.588	
H	10/20/09	Hand	Valveless	Rev. air	56	95	Flat hex.	29.8	2.46	12.1	0.203	No tappet
1H	10/21/10	Hand	Valve	Rev. air	..	93	Crux	66.7	2.10	31.8	0.066	
J	12/15/10	Hand	Valve	Rev. air	..	94	Crux	66.9	7.30	9.15	0.800	High run
J	12/15/10	Hand	Valve	Rev. air	..	94	Crux	66.3	6.70	9.92	0.675	Average 8 runs
J	2/25/11	Hand	Valve	Rev. air	..	94	Crux	68.8	6.25	11.01	0.567	Average 23 runs
K	3/2/11	Hand	Valve	Rev. air	..	92	Crux	77.6	7.29	10.65	0.684	
K	3/7/11	Hand	Valve	Rev. air	..	94	Crux	75.4	6.90	10.90	0.633	Average 10 runs
L	4/10/11	Hand	Valve	Rev. air	..	97	Crux	58.0	6.29	9.21	0.683	
L	4/10/11	Hand	Valve	Rev. air	..	97	Crux	58.5	6.19	9.45	0.655	Average 7 runs
L	4/18/11	Hand	Valve	Rev. air	..	93	Crux	55.8	6.55	8.52	0.769	Average 7 runs
M	4/21/11	Hand	Valveless	Rev. air	..	95	Raised crux	54.7	9.35	5.85	1.600	High run to $1\frac{11}{16}$-in. bit
M_1	6/24/11	Hand	Valveless	Dir. air	..	99	Raised crux	58.5	11.54	5.07	2.280	Bit a little soft
1M	6/24/11	Hand	Valveless	Dir. Air	..	99	Raised crux	58.5	9.09	6.45	1.410	Average 3 other runs, same machine
M_2	8/30/11	Hand	Valveless	Dir. air	..	100	Raised crux	65.0	10.75	6.04	1.780	
M_2	8/30/11	Hand	Valveless	Dir. air	..	99	Raised crux	62.9	9.56	6.59	1.450	Average 11 runs
N	9/27/11	Hand	Valve	Dir. air	..	99	Raised crux	36.0	5.28	6.82	0.774	
N	9/27/11	Hand	Valve	Dir. air	..	99	Raised crux	39.2	4.63	8.59	0.540	Average 11 runs
N_1	10/24/11	Hand	Valve	Dir. air	65	96	Raised crux	66.4	2.67	24.8	0.108	Average 14 runs
J	2/3/12	Hand	Valve	Dir. air	..	97	Raised crux	71.2	6.06	11.76	0.515	Drill "J" after 1 year's service
× M_3	2/20/12	Hand	Valveless	Dir. air	..	99	Raised crux	62.9	11.10	5.66	1.962	High run
× M_3	2/20/12	Hand	Valveless	Dir. air	..	100	Raised crux	63.5	10.48	6.06	1.739	Average 4 runs
× O	3/5/12	Auto aux. V.	Valveless	Dir. air	..	98	Flat hex.	79.1	10.34	7.65	1.350	Hollow bit, no air through it, high run
× O_1	6/3/12	Auto aux. V.	Valveless	Dir. air	87	100	Raised crux	85.5	9.55	8.95	1.068	With pressure control on feed ½ fd., bit with soft center, No. 6
× O_1	6/3/12	Auto aux. V.	Valveless	Dir. air	87	100	Raised crux	87.0	8.50	10.24	0.830	With pressure control on feed, fine bit, No. 12
P	2/16/12	Hand	Valve	Dir. air	..	100	Raised crux	67.0	8.43	7.94	1.062	Hard to rotate
P_1	5/13/13	Hand	Valveless	Dir. air	86	96	Raised crux	55.8	10.76	5.18	2.080	Rotates freely
P_1	5/13/13	Hand	Valveless	Dir. air	86	97	Raised crux	59.5	11.03	5.40	2.043	Rotates freely
P_3	5/13/13	Hand	Valveless	Dir. cir.	86	96	Raised crux	56.8	9.32	6.10	1.529	Average 9 runs
P_3	5/13/13	Hand	Valveless	Dir. cir.	86	96	Flat crux	57.1	7.98	7.17	1.110	Average 3 runs
Q	5/16/13	Auto rifle	Valve	Rev. air	..	91	Raised crux	51.4	6.12	8.39	0.730	
Q	5/16/13	Auto rifle	Valve	Rev. air	86	92	Db'le chisel	48.2	8.18	5.90	1.388	Hollow steel
R	9/25/13	Auto rifle	Valve	Rev. air	..	96	Raised crux	112.0	6.85	16.25	0.421	Air bled to control feed press
S	10/30/12	Auto rifle	Valve	Hand	37	98	Hollow six pt.	58.5	6.97	8.40	0.830	Av'ge 7 runs; block-holer
S_1	2/14/13	Auxiliary motor	Valve	Hand	90	95	Hollow six pt.	88.7	8.95	9.27	0.956	Av'ge 5 runs; sinker
S_2	11/5/14	Auto rifle	Valve	Rev. air	..	95	Six point	80.3	6.41	12.40	0.518	Stoper
S_3	5/1/14	Auto rifle	Valve	Hand Screw	147	91	Hol.flat crux	101.0	8.40	12.20	0.690	Column mounted water drifter. Wgt. includes 48-lb shell
T	1914	Auto rifle	Valve	Dir. air	95	93	Raised crux	71.5	7.49	9.55	0.785	Stoper
T_1	10/29/14	Auto rifle	Valve	Hand	40	93	Hollow six pt.	70.0	7.00	10.00	0.700	Av'ge 6 runs; block-holer
T_2	7/30/14	Auto rifle	Valve	Hand screw	148	90	Hol.flat crux	93.7	8.88	10.57	0.840	Av'ge 7 runs; column mounted water drifter, incl. 43-lb. shell
T_{20}	7/30/14	Auto rifle	Valve	Hand screw	148	90	Hol.flat crux	91.8	10.49	8.75	1.200	Maximum; incl. 43-lb. shell
T_3	8/7/15	Auto rifle	Valve	Hand	40	95	Hollow six pt.	85.2	6.65	12.80	0.520	Av'ge 5 runs; block-holer
U	3/4/15	Hand	Valveless	Dir. air	82	94	Raised crux	83.7	10.20	8.20	1.245	Av'ge 11 runs; stoper
V	1/25/15	Hand	Valve	Rev. air	..	92	Hol.r's'd crux	77.6	10.38	7.48	1.390	Av'ge 6 runs; water stoper
V_1	8/18/15	Auto rifle	Valveless	Hand screw	146	87	Hol.flat crux	88.6	9.44	9.40	1.003	Av'ge 12 runs; column mounted water drifter; incl. 57-lb. shell
V_{10}	8/18/15	Auto rifle	Valveless	Hand screw	146	84	Hol.flat crux	86.4	12.20	7.18	1.700	Maximum; incl. 57-lb. shell
V_{110}	8/18/15	Auto rifle	Valveless	Hand screw	146	93	Hol.flat crux	96.5	11.00	8.76	1.255	Maximum; incl. 57-lb. shell
V_2	8/17/15	Auto rifle	Valveless	Hand	89	84	Hollow six pt.	86.8	8.86	9.80	0.904	Av'ge 6 runs; sinker
V_3	8/16/15	Auto rifle	Valveless	Hand screw	..	87	Hollow six pt.	62.7	4.74	13.24	0.358	Av'ge 9 runs; column mounted water drifter
V_{30}	8/16/15	Auto rifle	Valveless	Hand	..	87	Hollow six pt.	66.0	4.59	14.40	0.319	Av'ge 6 runs; block-holer
W	8/10/15	Hand	Valve	Dir. air	87	86	Raised crux	66.8	10.11	6.63	1.510	Av'ge 5 runs; stoper
X	1/20/15	Auto rifle	Valve	Hand screw	..	91	Hollow crux	63.8	5.70	11.20	0.511	Av'ge 5 runs; column mounted water drifter
X_1	10/28/15	Hand	Valve	Dir. air	90	85	Raised crux	67.5	10.85	6.22	1.750	Av'ge 6 runs; stoper
X_2	3/8/16	Auto rifle	Valve	Hand	32	94	Hollow six pt.	68.3	5.06	13.70	0.369	Block-holer
Y	6/25/15	Auto rifle	Valve	Balanced air	89	87	Hollow six pt.	79.8	7.40	10.84	0.680	Av'ge 5 runs; column mounted water drifter; incl. 49-lb. feed
Z	3/28/16	Hand	Valveless	Dir. air	90	85	Six pt.	63.0	10.00	6.30	1.580	Av'ge 5 runs; stoper
Z_1	2/3/17	Auto rifle	Valveless	Hand screw	146	89	Hollow crux	80.3	9.63	8.05	1.200	Av'ge 5 runs; column mounted water drifter; incl. 57-lb. shell

Then

$$D = \frac{L}{\frac{60SO}{12}} = \frac{L}{\frac{5dO}{t}} = 0.2\frac{tL}{dO}$$

Let

p = Power cost to produce 100 cu.ft. of free air, compressed to standard drill-testing pressure, in cents;

v = Number of cubic feet of free air used in test by operating drill;

d = Depth of standard hole drilled, in inches.

Then

$$P = \frac{12pv}{100d} = 0.12\frac{pv}{d}$$

Substituting these values of D and P in the original equation we obtain

$$F = \frac{1}{0.20\frac{tL}{dO} + 0.12\frac{pv}{d} + M} = \frac{d}{\frac{0.2L}{O}t + 0.12pv + dM}$$

Since L is a constant for any particular mine, and O for a given number of steel changes with any particular type of drill—such as a column-mounted reciprocating drill, a column-mounted hammer drill, an air-feed hammer drill, a block-holing drill, etc.—we may simplify the equation by substituting

$$k = \textit{coefficient of drilling} = \frac{0.20L}{O}$$

Also, since p is a constant for any particular mine we may further simplify by placing

$$k' = \textit{coefficient of power} = 0.12p$$

and we then have the general equation for any particular mining conditions and type of drill,

$$F = \frac{d}{kt + k'v + dM}$$

However, the correct value for maintenance and amortization of any particular type and make of drill can be determined only after operations extending over months or years, so that this factor may well be left out of a formula which is to be used for classifying drills after drilling-speed and power-consumption tests, which may be completed in a short time. The consideration of the reduction of drilling speed and the increase of power consumption, which occur in a drill because of wear or any other normal results of service, may fairly be placed in the same class as maintenance. Judgment as to the materials, workmanship and design of any drill, as well as reports of its satisfactory service elsewhere, will lead to a rough estimate of the final desirability of a drill if it has shown a high standard, on testing based on drilling speed and power consumption.

The equation is thus simplified to the form,

$$F = \frac{d}{kt + k'v}$$

But other highly important factors enter into the problem of selection of a drill: The reduction in labor units, capital and overhead charges brought about by an increased drilling speed and increased tonnage per machine; the increased efficiency of supervision and work caused by the reduction and concentration of the number of working places; the possibility of producing a greater tonnage from any property with a limited number of working places; and the possibility of reducing the drilling equipment, with its attendant stock of spares, hoses and connections, and extensive air mains, if a drill with a greater drilling speed may be employed. It therefore seems that the following formula is more indicative of the actual merits of drills, although theoretically it has no derivation and must be considered empirical; it also possesses the virtue of reducing to a simple form. This formula for a "factor of desirability" has been used for the past eight years at the Franklin Furnace mines of the New Jersey Zinc Co. All coefficients have been omitted since the following drill tests have all been under the same standard conditions.

$$F' = \frac{1}{DPM}$$

Since M is treated separately, as has been previously suggested, the equation becomes

$$F' = \frac{1}{DP}$$

Now if the same values previously deduced for D and P are substituted,

$$F' = \frac{1}{\frac{kt}{d} \times \frac{k'v}{d}} = K\frac{d^2}{tv}$$

where K is a new coefficient equal to the reciprocal of the product of k and k'.

Therefore the "factor of desirability" equals the drilling speed, in inches per minute, divided by the power consumption in cubic feet of free air per inch drilled. It is quite evident that the factor gained from the quotient of inches drilled per minute divided by cubic feet of free air per minute (or the reciprocal of this quotient) gives merely the power consumption per inch of hole drilled and ignores the quantity of drilling which may be accomplished.

The application of both of these formulas for F and F' to a hypothetical problem may be of interest to show the comparative results within the limits of practice.

Let us assume that 30 hp. is required to compress 100 cu.ft. of free air per minute to 100 lb. per sq.in. gage pressure and deliver the same to a drill in the mine; that the power cost is 1c. per hp.-hr.; that a drill which shows a drilling speed of 10 in. per min. on test averages 20 ft. per hour under working conditions, and uses 60 cu.ft. of free air per minute on test; that another drill will show a drilling speed of 6 in. per min. on test with an air consumption of 36 cu.ft. per min. and will average 12 ft. per hour under working conditions, and that the wage scale for drill runners is 40c. per hour, then,

For the fast drill:

$t = 1$ min.

$d_1 = 10$ in.

$L = 40$c.

$$O_1 = 20 \div \frac{10 \times 60}{12} = 0.40$$

$$p = \frac{30 \times 1}{60} = 0.5\text{c.}$$

$v_1 = 60$

$$F_1 = \frac{d}{0.2\frac{L}{O_1}t + 0.12pv_1} = \frac{10}{\frac{0.2\times40\times1}{0.40} + 0.12\times0.5\times60}$$

$$= 0.424$$

$$F_1' = \frac{1}{\frac{0.20tL}{O_1d_1} \times \frac{0.12pv_1}{d_1}} = \frac{O_1d_1^2}{0.024tLpv_1}$$

$$= \frac{0.40 \times 100}{0.024 \times 1 \times 40 \times 0.5 \times 60} = 1.389$$

For the slow drill:

$d_2 = 6$ in.

$v_2 = 36$ cu.ft.

$$O_2 = 12 \div \frac{6 \times 60}{12} = 0.40$$

$$F_2 = \frac{6}{\frac{0.2 \times 40 \times 1}{0.40} + 0.12 \times 0.5 \times 36} = 0.271$$

$$F_2' = \frac{0.40 \times 36}{0.024 \times 1 \times 40 \times 0.5 \times 36} = 0.833$$

Thus the relative factors for the two drills by the first formula have a ratio of 0.424 to 0.271, or 1.56 to 1; and by the second formula (empirical) the ratio of factors is 1.389 to 0.833, or 1.67 to 1. In other words, by the empirical formula the fast drill is credited with about a 7% higher rating than by the theoretical formula, and this does not seem an undue allowance to cover the unestimated advantages previously enumerated.

Records made previous to July, 1909, have not been shown in Table I since much of the work done in 1907 and 1908 was distinctly experimental in determining the desirable cylinder diameters, lengths of strokes, piston weights, valve weights, etc.; but such records show drilling speeds of about 2 to 3 in. per min. with air comsumptions of from 40 to 70 cu.ft. of free air per minute at 90 lb. per sq.in. gage pressure. The listed tests made during 1909 cover most of the well-known American makes of hammer drills at that time, and if one excepts the drills denoted by symbols G, G_1, etc., since they were experimental tools, the design of which was developed by the New Jersey Zinc Co. at Franklin Furnace, N. J., it is noticeable that about 4½ and 5 in. were the highest drilling speeds obtainable at about 90 lb. pressure and with an air consumption of 60 to 90 cu.ft. of free air per minute; and for various drills the "factor" varied from 0.09 to 0.41. Those drills marked G, which were made exclusively for the New Jersey Zinc Co., increased the drilling speed about 40% above the best previous drill performances and remained unequaled in drilling speed for a year and unsurpassed for about a year and a half. The fact that a number of these drills were included in the equipment at Franklin accounts for part of the increased stoping efficiency during the year 1911, as cited later. Although it was then the opinion of some unprejudiced persons, well versed in the drilling art, that such tools had reached their practical limit of drilling speed as well as the limit of strengths of materials, yet 18 months later a new type of drill was developed to achieve 20% more drilling with twice as good a factor, and a renewed equipment of these other drills again increased the mining efficiency. Again a period of 18 months sufficed for the production of a hammer drill which still further advanced the drilling speeds 20%, and since the introduction of this drill we have been able to find several drills which surpassed it 10 to 20% in drilling speed.

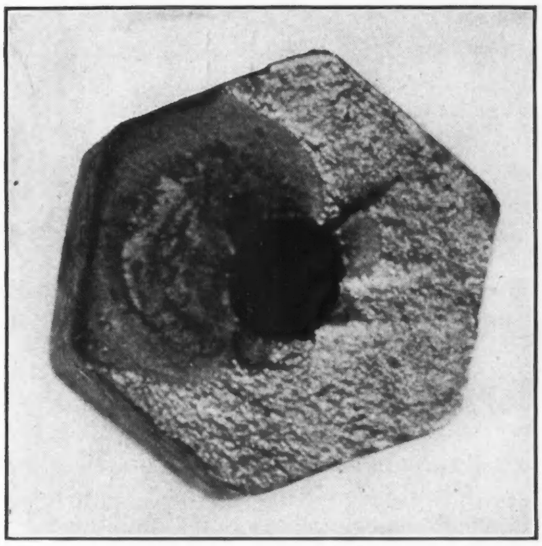
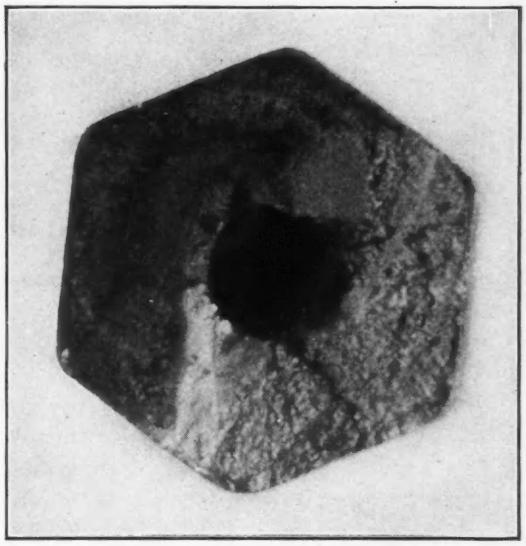
In Table I some seemingly freak runs are noticeable, which are included to call attention to the variability of results in presumably standard testing. For instance, under drills D it appears that a bit with two wings broken will drill faster and at a lower air consumption per minute than can be attained with a perfect bit; and again, with drill M_1, a bit which has proved a little soft and battered drills one-fourth more per minute than bits in proper condition and with the same air consumption. Furthermore, the tests of one person indicate that, when the size and form of the drill bits are the same, faster drilling can be done with short steels than with long ones, while another investigator shows a greater drilling speed with long steel than with short. The use of tappets or anvil blocks between the shanks of drill steels and pistons is generally estimated as causing a reduction of 20 to 30% in the drilling ability, but some tests do not confirm this and show even an increased cutting speed with the use of anvil blocks in a machine otherwise the same. With some drills the use of water to clean the cuttings from the hole seems to cause a cutting speed below that obtainable through the use of compressed air for the same purpose, but in other instances the advantages are reversed. In short, there seem to be so many variables in the drilling problem as to warrant a 10% variation in the results of supposedly standard tests, and a number of runs should be made to gain a fair average; or strict judgment of machines should not be made within this limit.

Consideration of Physical Phenomena

Perhaps the consideration of the physical phenomena relating to the process of drilling may prove of interest and value. When rock is excavated by a drill bit, three applications of forces seem to be involved—by abrasion, by crushing, and by severing or chipping. Although all these must take place to a certain degree, the greatest amount of useful work is performed when the percentage of force applied to chip reaches a maximum. But in rock it appears that chips can be produced in radically different ways—first, by the severing of molecules and, second, by the reflex forces produced in an elastic medium. To illustrate this, consider the chipping of a comparatively inelastic substance such as lead. With a hammer and a chisel whose axis is inclined considerably from the normal to the surface of a lead block it is possible to sever the lead and roll up chips, but if the chisel is normal to the surface of a thick block, only an indentation can be made and there probably will be a raised area about the indentation to accommodate a certain percentage of the displaced metal. On the other hand, with a highly elastic material, such as glass, the forces impressed by a normally positioned chisel will cause a compression of the molecules, whose elasticity will cause their expansion toward a free, unresisted surface. Since the greatest forces are developed at the surface, since the penetration of the chisel carries some forces to a depth below the surface, and since the chisel surface itself applies some forces at an angle to its axis and impedes the reëxpansion of molecules to the space it occupies, therefore the reflex forces produce more or less cone-shaped chips or flakes and leave a corresponding crater in the block of glass. Now, if the chisel is placed near the edge of a block of glass, the blow upon it will induce stresses to another free face and a correspondingly larger chip will be produced because of the tendency of the forces to seek relief in the shortest direction as well as because of the severing effect. The method of cutting of a drill bit is commonly shown as taking place in this last way with the progressive chipping of a series of benches or steps, but it is doubtful whether such a procedure exists, except in rare instances, for the speed and latitude of rotation between consecutive blows of the drill piston or hammer cannot be controlled with sufficient precision nor adjusted to the various rocks; and an inspection of the cuttings from a drill hole shows them to be flakes or a crushed and abraded powder.

In the formation of these flaky chips there may be a limiting force of blow for each velocity of impression in order to gain the most useful work (that is in the production of flakes), for it appears that beyond certain limits the blows increase the percentage of crushed material and the drilling speed does not vary with the force applied, so that some heavy-hitting drills accomplish more in medium-soft ground when a portion of their blows are absorbed by a tappet at the shank end of the steel or by a cushion of water intervening between the bit and the rock. If the force of the blows was lessened by a reduction in air pressure the speed of the piston would be slowed up and the drilling would suffer from the fewer number of blows per minute.

The transmission of the kinetic energy of the piston to the rock is also influenced by many factors. The blow may be delivered against the rock by the free drill steel which is driven forward through the intervening air or water by the impact of the piston, and the velocity of the steel will depend upon the relative masses of the drill steel and piston, the velocity of the piston and the coefficient of elasticity of the steel, in accordance with the well-known laws of mechanics which deal with elastic or partly elastic bodies and their impact. The drill steel in this way assumes the functions of a "jumper" drill which is driven against and rebounds from the rock at a high frequency, and its action is well seen in most all screw-feed hammer drills with the ringing or jingling of the steel in a drill hole. Another mode of force transmission is by compressional waves, traveling through the drill steel from the shank to the bit. This latter condition brings a cutting effect only when one end of the steel is tight against the rock, but then proves very efficient. Although the air-feed hammer drills usually chatter the steel against the rock, like a projectile shot from the chuck bushing by impacts of the piston, yet it seems possible to approximate the other working condition by designing the air feed so that the pressure is lowered as the piston is traveling on its back stroke (possibly by taking the supply air, for the back stroke, from the air feed) and so that the air-feed pressure builds up and forces the drill against the rock just before it is struck by the piston. The reversed-air feeds may sometimes approximate these conditions and then assist the machine to a higher drilling speed. If hammer drills were made so that the drill steels were always held firmly against the rock, when the piston strikes them, it seems unquestionable that the greatest efficiency of the blows of the piston would result, provided they were properly timed, for no energy would be lost by reason of the inertia of the drill steel, but only that due to heating, resulting from the imperfect elasticity of the metal. The question of the proper timing of the piston blows opens another phase of the matter; namely, the reaction of the rock upon the drill steel; and this effect is the more pronounced with harder rock. It tends to speed up the piston and is so noticeable in running a machine against a metal block as to invalidate, as too high, all air-consumption tests so conducted. The effect of these reactive vibrations upon the drill steel may prove very marked and serious. Where the reactive vibrations interfere with oncoming compressional waves, considerable energy is dissipated, and at times one may be so fortunate as to detect points of increased temperatur (probable nodes) upon a drill steel which is cutting ground; and it is no uncommon thing to see a drill steel, in service, break at two points (into three pieces) simultaneously, probably from fatigue because of these vibratory stresses. The accompanying photographic reproductions exhibit the peculiar conchoidal fracture resulting from the growth of vibratory rupture of a drill steel at two places simultaneously. It is noticeable that the fracture started at several points on the periphery of the hole and that opposite the large fracture growths on each end there are small growths on the other end. This may indicate a segregation of metal, or a concentration of stresses caused by scratches of the tools which originally punched or drilled the billets before rolling to the size of the ⅞-in. hexagon drill steel. The fractures seem to indicate a gradual growth and extension from the form of microscopic slip planes to cracks which extend as the steel fails because of vibratory stresses. Solid drill steels fail with similar fractures, but these usually start from a nick, bruise, punch or stencil mark at the surface; but some have been found which start in the interior of the steel. Of course any cut, nick or mark which suddenly reduces the section of a piece of steel (although only very slightly) causes any stresses from vibration or bending to concentrate at such points and encourage fractures to start there when the

RUPTURES OF DRILL STEEL SHOWING TWO ENDS OF A PIECE 1 IN. LONG metal is too greatly fatigued or stressed. Similarly, any sudden change in the constituency or structure of the metal would appear to produce the same results.

It is of interest that the railroads have had failures of their steel rails with fractures of the same appearance, as in both instances the metal is subjected to recurrent shocks of impact. On the other hand, if these vibrations synchronize at the bit, it is quite possible that the chipping forces are greatly augmented, and such an explanation may readily answer those puzzling drill tests in which a dull or broken bit exceeds a finely formed bit in drilling speed. For a long time at Franklin a tally was kept of the different individual drill steels which entered into the testing, with the hope of determining that some particular piece of steel produced the greatest cutting speed, but no conclusion could be drawn from the records, except that the changes in length due to resharpening probably masked any possibility of determining the suitable lengths for maximum efficiency. And it seems quite plausible that such a result should be expected if the possible wave lengths of the compressional vibrations in the drill steels are considered. Probably these reactive vibrations occur to a great extent, as well, in the process of drilling, where the steel dances in the chuck and against the rock, for steel breakage appears equally as high with such a type of machine as with the pneumatic feed, if not higher, and tests comparing these two types for such effects might prove interesting as well as instructive.

But still other factors influence the force delivered at the rock. If the anvil block or tappet is not in contact with the drill when the piston strikes, a considerable energy loss occurs through the transference of momentum to several pieces. If the steel is bound in the chuck bushing, a great amount of the energy is absorbed by the friction. If the steel is not straight, it loses energy because of the flexure. If the chuck is badly worn, the axis of the steel does not coincide with that of the drill and there is a loss due to the oblique, eccentric impact. If the steel is tight in the drill hole or if the friction against the side of the hole is great because of its depth, the velocity of the steel, as a projected body, is lessened and the drilling speed is reduced.

Length of Drill Steel

The length of the drill steel is an item generally credited as an important influence, and common opinion supports the idea that the cutting speed falls as the length of the steel increases, although some people, on the contrary, feel sure that the long steels drill the fastest. The tests conducted at Franklin do not lend an unqualified support to either view, for the peculiarities of different types of machines play so important a part. For example, if the air feed is very strong in a stoping drill the additional counteracting weight of a long and heavy steel may so improve the working conditions as to indicate a superiority for the long steel, and if the air feed is weak the reverse may be true; if the drill steel cuts by virtue of a dancing or "jumper" action, the mass added with length may so reduce its velocity against the rock as to bring it below the amount required for efficient chipping; if the piston normally delivers too heavy a blow for the rock, the drilling speed may be improved by the added inertia of the long steel; and if the steel is always against the rock when a blow is delivered, it is doubtful whether the length of the steel plays an important part unless the permitted decrease in the gage of the drill bits aids the cutting speeds. It is of course to be understood that the foregoing considerations of drill-steel lengths refer to the performances with bit gages of the same diameter.

The use of an anvil block is considered by some drill designers to necessitate a loss of from 20 to 30% of the power of a drill, but actual tests do not always indicate such a condition when the identical steel is tested in the same drill with and without a tappet. The results probably depend upon how frequently the tappet is struck when away from the shank of the steel, and also upon the suitability of the machine to the rock, for if its blows are too heavy the intervention of a loose tappet might reduce their force, with a benefit in drilling speed. The use of water at the bottom of the hole ordinarily consumes about 10% of the cutting speed if there is no tendency for the drill bits to lose their temper, and compressed air for cleaning the holes encourages a greater drilling speed provided the cushion of water in the bottom of the holes does not have a benign influence in reducing too powerful a blow upon the rock.

The manner in which a drill is rotated has a bearing upon the amount of work accomplished, and with hand-rotated tools, a vigorous rotation with a rapid and wide

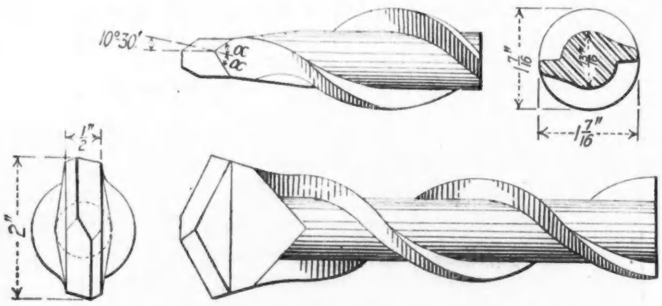


FIG. 4. SPECIAL DRILL WITH SINGLE BIT

arc of swing produces the best results; with power-rotated drills it is possible to reach such a speed as to abrade and dull the drill bits and consequently lessen the drilling speed. It seemed that, with a positive and constant rotation, the axial planes of the cutting edges of the drill bits should be at the same angle with the cut surface as the resultant velocity vector, as estimated for the rotative and striking velocities; and such a bit (see Fig. 4) was tried in an electrically driven hammer drill at Franklin without showing a change in cutting speed, probably because with either bit the chips came out in flakes, as previously described.

In view of the fact that the subject of hammer drills is more or less in its infancy and literature in regard to them is rather limited, it seems desirable to correct at the earliest opportunity any typographical or other errors which, if accepted without investigation, might work to the detriment of the art of drilling. In this connection it seems that some statements should be corrected in the 1910 edition of Eustace M. Weston's book, "Rock Drills," in the chapter "Philosophy of Process of Drilling Rock," under the subheading of hammer drills. In considering the kinetic energy of a blow he states, on page 139:

> In other words, to double the energy of a blow it would be necessary to double the mass, or weight, if the velocity is the same; but to double the energy, keeping the mass the same, the velocity must be increased four times. The weight of the piston hammer of the largest type of drill is 15 lb. The weight of piston, steel, etc., of a piston drill varies from 60 to 125 lb., so that a blow of equal force can be delivered by a hammer drill only by increasing the velocity of the hammer very greatly. This is asknowledged, for as one hammer-drill maker states, the weight of the piston is

one-fourth that of a piston drill; but the velocity is four times as great. To give a blow equal in power it should be sixteen times as great.

A mathematical error appears to have been made in the premises of Mr. Weston's argument and his consequent deductions as to the practical impossibility of hammer drills being able to compete with piston drills are quite logical, but probably at fault.

If the kinetic energy of a body, such as a drill piston, is designated by K, its velocity by V, and its mass by M, then,

$$K = \tfrac{1}{2}MV^2 \text{ and } K_1 = \tfrac{1}{2}M_1V_1^2$$

Now if M_1 equals M and K_1 is, say twice the value K, then,

$$V_1^2 = 2V^2 \text{ and } V_1 = V\sqrt{2}$$

therefore,

$$V_1 = 1.414V$$

So the velocity of the piston in a hammer drill need be only 1.4 times as great as that when the kinetic energy of the piston is cut in half.

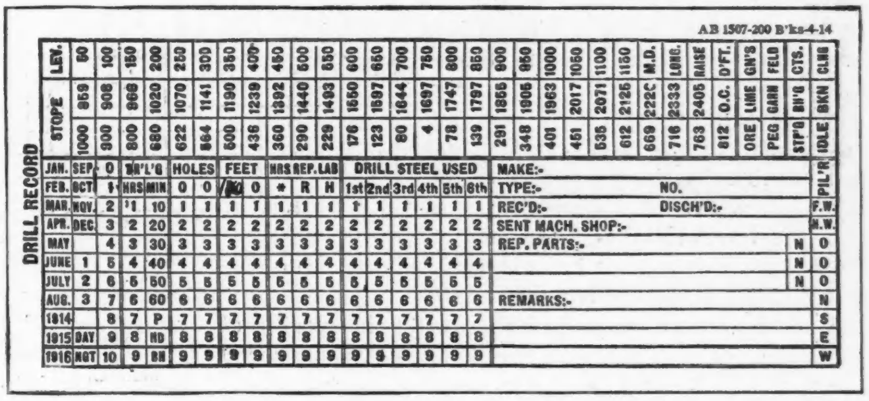
Again, in the example comparing the piston weights of piston and hammer drills, Mr. Weston appears in error in stating that the velocity should be 16 times as great, for if the piston of a hammer drill is one-fourth the weight of that of the piston drill, the velocity of the hammer-drill piston need be only twice as great as that of the piston drill in order to deliver blows of the same energy; and the hammer drill will also surpass the piston drill since it will strike twice as many of such blows per minute. Subsequent to the appearance of this article Mr. Weston granted the justice of the above contentions and stated that his book should have been corrected in accordance with the force theory here expressed. The necessity of using high air pressures in hammer drills is only incident to the peculiarity of certain drill designs and is not dependent upon the divorcing of the piston from the steel. If we are to consider the shock upon the parts of two drills of equal capacity, it is evident that with the shorter piston strokes in hammer drills, with the increased number of blows, whose final striking velocity is equal to that of a piston drill under comparison, the weight of the hammer-drill piston may be less and the energy in each individual blow may be less in order that the same amount of energy per minute be developed. Therefore the shocks upon hammer-drill parts are more frequent, but not as heavy as the shocks upon piston drills of equal capacity.

It is extremely difficult to get adequate figures as to the maintenance of drills unless some special forms are kept, which become to all intents a ledger account of each individual drill, for questions naturally arise as to the cost per foot of hole drilled, the length of time the machine has been in service and has been running, the number of holes drilled, the kind of rock encountered and the supply of steel used, as well as the drill parts replaced. The New Jersey Zinc Co. uses a system of punched slips for shift bosses' reports and "drill record" slips are a part of the scheme. These are punched in duplicate by the shift boss and filed at the mine office and main office, where the information is transferred to large sheets, of which each one accommodates the record of one machine for a month and the footings are carried forward so as to indicate the total work accomplished and maintenance of any machine "to date." The repair parts are designated as to whether they are new or old (second-hand) ones, and original and subsequent drilling-test records are noted on the same summary sheet.

Figs. 5 and 6 show the record slip and sheet. The positions of drills are by stope-slice coördinates, the class of work (whether raising, drifting, stoping, block-holing or drilling chutes) is indicated, the kind of rock (ore, limestone, gneiss, pegmatite, garnet or feldspar) is punched, and if the machine is idle, broken or being cleaned, those points are recorded.

The compressed-air rock drill made revolutionary changes in mining methods and in the reduction of mining costs in units of labor per ton of ore, and at Franklin even more marked savings have been made through the development of hammer drills. There, in the days of hand drilling, a total of 8 ft. in three drill holes with varying diameters of $1\frac{3}{4}$ to $1\frac{1}{4}$ in. was considered a fair 10-hour shift's work, and possibly 8 tons of ore would be broken per drill-shift or 4 tons per man-shift. With 3-in. reciprocating rock drills from 20 to 40 ft. of drill holes, ranging in diameter from $2\frac{1}{2}$ to $1\frac{1}{2}$ in., would be the average work for a 10-hour shift, although on rare occasions some men might drill as much as 80 or 90 ft. of holes in a shift, and possibly 20 tons of ore would be broken per shift, or 10 tons per man-shift, since two men were needed on a drill. It seems that as a rule a greater tonnage per foot of hole was obtained with hand-drilling because of the fact that, rather than dismount and reset heavy drill columns, machine men would tend to place as many holes as possible from one set-up, therefore many holes were placed disadvantageously for breaking efficiency. Another cause which would contribute to the same results would be the difficulty of starting holes with piston drills on uneven sloping faces, so that holes were frequently deflected from the direction in which they were supposed to be placed. These figures would lead to the rough estimate that $2\frac{1}{2}$ times as much tonnage per drilling man-shift was accomplished by piston drills as by hand-drilling.

With hammer drills 80 to 100 ft. of $1\frac{3}{4}$- to $1\frac{1}{4}$-in. drill holes are placed by one man in a 10-hour shift, about 150 to 200 tons of ore will be broken per drill-shift and the same amount per drilling man-shift, or 15 to 20 times the amount broken per man with reciprocating drills. Of course the entire credit for such increase in tonnage can-



AB 1507-200 B'ks-4-14

DRILL RECORD

LEV.	50	100	150	200	250	300	350	400	450	500	550	600	650	700	750	800	850	900	950	1000	1050	1100	1150	M.D.	LONG.	RAISE	D'FT.	GN'S	FELD	CTS.	CLNG
STOPE	959	908	968	1020	1070	1141	1190	1239	1392	1440	1493	1550	1597	1644	1697	1747	1797	1855	1906	1963	2017	2071	2126	222C	2333	2405	O.C.	LIME	GARN	BH'G	BKN
	1000	900	800	680	622	564	500	436	360	290	229	176	123	80	4	78	139	291	348	401	451	535	612	669	716	763	812	ORE	PEG	STP'G	IDLE

JAN.	SEP.	O	DR'L'G		HOLES		FEET		HRS REP. LAB			DRILL STEEL USED						MAKE:-		PIL'R
FEB.	OCT.	1	HRS	MIN	0	0	[illegible]	0	*	R	H	1st	2nd	3rd	4th	5th	6th	TYPE:- NO.		
MAR.	NOV.	2	1	10	1	1	1	1	1	1	1	1	1	1	1	1	1	REC'D:- DISCH'D:-		F.W.
APR.	DEC.	3	2	20	2	2	2	2	2	2	2	2	2	2	2	2	2	SENT MACH. SHOP:-		H.W.
MAY		4	3	30	3	3	3	3	3	3	3	3	3	3	3	3	3	REP. PARTS:-	N	O
JUNE	1	5	4	40	4	4	4	4	4	4	4	4	4	4	4	4	4		N	O
JULY	2	6	5	50	5	5	5	5	5	5	5	5	5	5	5	5	5		N	O
AUG.	3	7	6	60	6	6	6	6	6	6	6	6	6	6	6	6	6	REMARKS:-		N
1914		8	7	P	7	7	7	7	7	7	7	7	7	7	7	7	7			S
1915	DAY	9	8	ND	8	8	8	8	8	8	8	8	8	8	8	8	8			E
1916	NGT	10	9	BN	9	9	9	9	9	9	9	9	9	9	9	9	9			W

FIG. 5. TICKET PUNCHED IN DUPLICATE BY SHIFT BOSS TO RECORD DAILY DRILL PERFORMANCE

not be given to the type of drill, for improved organization, system of working and supervision have undoubtedly played an important part; but the greater mobility and flexibility of the light hammer drills have permitted and encouraged a more efficient placing of drill holes, have cut in half the labor necessary to run a drill and have permitted a more effective supervision and mining scheme. The actual tonnage broken per man working in a stope will not be so high, comparatively, since it has been found worth while to place additional men in stopes to sledge and block-hole large chunks of ore which were formerly often allowed to become buried and so proved obstacles to high tramming efficiency by blocking chutes through which the shrinkage stopes were drawn into tram cars.

Table II shows the gains that have been made with the adoption of hammer drills by the New Jersey Zinc Co. It is to be regretted that no records of tonnage and labor were available for earlier years, so as to cover the average efficiencies before the advent of hammer drills and back to the days of hand-drilling. The different divisions of mining work are classified in this table as drifting, raising, stoping, and opencut or quarry, and it may be interesting to summarize the important features, reducing the labor to an hourly basis, inasmuch as a change was made from a 10-hour to an 8-hour shift basis in July, 1913. Since 1913 labor conditions have been very unsettled and the force of workmen has been continually changing, with a considerable influx of incompetents, so that the comparison of efficiencies in the last three years does not properly indicate the economies which hammer drills might produce.

There have been no radical changes in the placing of the drill holes in drifts since the adoption of the air-feed hammer drill for this work, but one man with a single machine is now placed in a heading; he is instructed to "pull" a "round" each 2-hour shift, working overtime if necessary, and to accomplish an advance of 3½ to 4 ft. per round. Two men operating a reciprocating rock drill formerly made an advance of a 5 to 6 ft. round in five 10-hour shifts. So the drilling labor (runners and helpers) per foot of advance averaged 18.4 hours for the entire mine during the year 1910, when reciprocating rock drills were solely in use. As shown by the average for 1913, hammer drills have reduced this figure to 5.3 hours per foot of advance, or about one-third the former labor of drilling and blasting. The explosive costs have also been reduced by the use of hammer drills from the figure of $1.84 per ft. of drift during 1910 to $1.40 per ft. in 1913, for two probable reasons:

First, hammer drills permit the placing of drill holes smaller in diameter than those bored by reciprocating drills, so that an unnecessary amount of explosive is not required merely to fill the holes sufficiently to distribute the force of the explosion.

Second, the flexibility and ease of rigging the light hammer drills permit and encourage a more efficient placing of drill holes. The almost exclusive use of 1 x 8-in. explosive cartridges now, as contrasted with the 1⅛ x 8-in. cartridges formerly used, demonstrates the first contention, for in terms of 1-in. powder, the equivalent of 36.8 sticks per foot of drift was used in 1910, and 34.6 sticks per foot in 1913. The drill shifts per foot of advance were lowered from 0.74 in 1910 to 0.33 in 1913, and the corresponding drill hours from 7.4 to 3.1

The different drifts may vary in size from 6 x 7 ft. to 8 x 11 ft. in section, and perhaps 7 x 8 ft. is an average section. Because of the compact, tough nature of the ground, it requires from 20 to 30 drill holes in a round, and 24 would be a fair average, so the drilling operation is an important factor of the drifting costs. The following comparison of the average drifting costs for each year shows the saving which has been possible because of hammer drills; but only the cost of drilling labor and explosives is considered. The record drift in 1913 was driven for $2.06 per foot.

As covered by a footnote in Table II, the labor per foot of drift advance in 1916 should be increased from $0.58 to $0.78 if it includes the bonus of overtime for breaking a round in a shift. Such an overtime bonus was not included in the figures for previous years. The increased cost of labor, powder, and supplies accounts for the advanced cost of drifting per foot.

THE NEW JERSEY ZINC COMPANY

FRANKLIN, N. J.

DRILL RECORD

MAKE; .... TYPE; .... SHOP No. ....
PURCHASED ON REQ.; A B ...., 191 IN SERVICE; .... 191 .
CYLINDER DIAM.; .... inches PISTON STROKE; .... inches PISTON WEIGHT; .... lbs. .... oz.
AVGE. DRILLING SPEED (to date) .... inches per min. AVERAGE AIR CONSUMPTION; .... cu. ft. per min. free FACTOR; ....
ORIGINAL:— DRILLING SPEED .... inches per min. AIR CONSUMPTION .... cu. ft. per min. free FACTOR; ....
PRESENT ( ):—DRILLING SPEED .... inches per min. AIR CONSUMPTION .... cu. ft. per min. free FACTOR; ....

DATE		WORKING PLACE		DRILLING TIME		No. HOLES DRILLED	FOOTAGE DRILLED	DRILLING SPEED IN FT. PER HR.	MATERIAL DRILLED	PCS. DRILL STEEL USED						HRS. REPAIR LABOR			REPAIR PARTS (Maker's Symbols)		M'T'CE CHARGES		
		Lev.	Loc.	Hrs.	Min.					1st	2nd	3rd	4th	5th	6th	Special	$2.25	$1.85			SUPPLIES	MINE LABOR	MACH. SHOP LABOR
Brought Over																							
1st	DAY																			N			
	NGT																			O			
2nd	DAY																			N			
	NGT																			O			
3rd	DAY																			N			
	NGT																			O			
	DAY																			N			
	NGT																			O			

FIG. 6. DRILL-RECORD FORM IN USE AT NEW JERSEY ZINC CO.

	1910	1911	1912	1913	1916
Drifting cost per foot.........	$5.3	$4.92	$3.35	$2.70	$3.56 ($4.12)a

a—With bonus.

In 1908, using 2¼-in. piston reciprocating rock drills, 0.7 ft. of 6 x 6-ft. raise per 10-hour drill shift was made with a labor expense of 28.5 man-hours per foot of raise. About 27 ft. of drill holes were placed per shift, 24 holes were placed in a round, and 16 lb. of explosives were used per foot of raise advance, at a cost of $2.70 per ft. for supplies. Since labor was then paid $2 and $1.55 per 10-hour shift, the total cost of raising was approximately $7.50 per ft. of advance.

During the same year (1908) hammer drills were introduced, and an advance of about 1.5 ft. per drill-shift was made with a labor expense of 13.3 man-hours per foot of advance. About 50 ft. of drill holes were placed per shift, 24 holes per 5-ft. round, and 10 lb. of explosives were used per foot of advance, at a cost of $1.75 per foot for supplies and a total cost of $4.10 per foot of raise, or only 55% of the cost with the reciprocating rock drills.

The development of hammer drills with increased drilling speed permitted the reduction of the drilling labor to 7.8 hours per foot of raise advance, and the explosives cost to $1.54 per foot of raising done in the year 1910; and a further reduction to 4.8 hours of drilling labor during 1912, and an explosive cost of $0.93 per foot, although the wages were $2.20 and $1.70 per 10-hour shift. These costs rose slightly in 1913, since wages rose to $2.25 and $1.85 for 10-hour shifts, and in July of the same year the working hours were lessened from 10 to 8 and the hourly wage was increased to \$0.281 and \$0.231. However, the cost per foot was then only 5.2 hours of drilling labor and \$1.03 per foot for explosives. About 18 drill holes are now placed to pull a 5-ft. round and two men are expected

TABLE II. ANNUAL COMPARISONS OF MINING EFFICIENCIES WITH PISTON DRILLS AND HAMMER DRILLS

Drifting

Year	Footage Advance	Drill Shifts per Ft. Advance	Type of Drills, % H. D.	P.	% Size of Powder 50% Strength 1⅛ in.	1 in.	No. Caps per Ft.	Powder in Sticks per Ft. Advance	Drill Shifts	Explos. Cost per Ft. Advance	Run'rs and Helpers per Ft. Advance
1908[1]	494	1.45	...	....	..	...	....	36.4	25.2	.....	....
1909	4,890	1.31	...	....	..	...	....	32.7	25.0	.....	....
1910	4,909	0.74	...	99.2	73	27	5.43	31.1	42.5	\$1.84	1.84
1911	2,814	0.63	...	99.2	96	4	4.95	26.5	41.9	1.59	1.71
1912	374	0.44	100	....	11	89	7.40	36.4	89.6	1.61	0.84
1913	905	0.33	100	....	..	100	5.56	34.6	106.0	1.40	0.59
1916	6,011	0.26	100	....	..	100	6.45	42.0	162.0	1.94	0.58‡

[1]Last 4 months. ‡ 0.78 if bonus included.

Raising

Year	Footage Advance	Drill Shifts per Ft. Advance	Type of Drills, % H. D.	P.	% Size of Powder 50% Strength 1⅛ in.	1 in.	No. Caps per Ft.	Powder in Sticks per Ft. Advance	Drill Shifts	Explos. Cost per Ft. Advance	Run'rs and Helpers per Ft. Advance
1908	715	1.18	...	....	..	...	....	27.7	23.4	.....	....
1909	5,446	0.90	...	....	..	...	....	29.3	32.4	.....	....
1910	4,257	0.44	97.0	....	16	84	5.11	27.4	62.7	\$1.54	0.78
1911	1,865	0.23	98.5	....	8	92	4.38	21.5	92.5	1.04	0.65
1912	1,311	0.17	100.0	....	2	98	3.46	20.1	115.0	0.93	0.48
1913	2,306	0.20	100.0	....	1	99	3.94	21.4	106.2	1.03	0.58
1916	6,118	0.20	100.0	....	..	100	4.43	29.6	150.0	1.37	0.63‡

‡ 0.86 if bonus included.

Stoping (Active)

Year	Net Tons Ore Broken	Type of Drills, % B. H.	H. D.	P.	Net Tons Broken, Excl. B. H. Drill per Drill Shift	% Size of Powder. 50% St'ngth 1⅛ in.	1 in.	No. Caps per Ton	Powder in Sticks per Net Ton	Drill Excel. B. H. Shift	Explos. Cost per Net Tons Broken	Net Tons Broken per Man in Stope
1908	57,000§	...	....	....	23.4	..	...	.....	0.76	17.7 19.3	.....	....
1909	361,000	7.0	24.5	68.5	20.5	..	...	.....	1.01	20.7	.....	....
1910	337,000	12.7	62.3	25.0	38.1	16	84	0.310	1.025	39.0	\$0.055	13.2
1911	386,000	30.1	55.8	14.1	121.0	11	89	0.356	0.77	95.8	0.041	20.3
1912	427,000	35.5	62.0	2.5	195.0	6	94	0.415	0.86	108.3	0.045	25.6
1913	494,330	30.3	69.7	....	170.0	2	98	0.420	0.89	233.0	0.049	26.7
1916	636,920	18.5	81.5	....	119.0	..	100	0.520	1.15	136.5	0.061	33.0
§ Two months, estimated.												
1910	25,400	58.6	14.2	27.2	114.2	28	72	0.922	0.745	(35.2)¶ 85.3	\$0.051¶	2.44 (4.9)
1911	64,500	79.2	1.6	19.2	355.0	29	71	0.787	0.614	(45.3) 278.0	0.042	62.0 (50.6)
1912	107,800	62.0	11.4	26.6	228.3	29	71	0.571	0.719	(62.4) 164.5	0.040	54.2 (46.0)
1913	250,490	45.7	54.3	....	236.0	39	61	0.524	0.712	(91.3) 168.0	0.042	82.2 (51.0)
1916	148,450	31.7	68.3	....	133.5	..	100	0.148	0.477	(43,3) (63.4)	0.025	46.5 (18.3)

¶ Figures in parenthesis include block-holing shifts, and labor of stopers, runners, block-holers, and helpers; others rated against runners only.

to blast a round each 8-hour shift and are each paid 11 hours' time for performing the task.

As covered by a footnote in Table II and previously mentioned for drifting, the labor per foot of raise advance in 1916 should be increased from 0.63 to 0.86 if the overtime task bonus is included. The increased cost of labor, powder and supplies is largely responsible for the increased cost of raising.

The average raising costs for operating labor and explosives have been as follows:

	1910	1911	1912	1913	1916
Raising cost per foot.........	\$2.95	\$2.31	\$1.88	\$2.22	\$3.14 (3.65)a

a—With bonus.

The record short raise (of about 50 ft. in length) for 1913 had a cost of \$1.65 per foot, and the record long raise (about 100 ft. long) had a cost of \$2.09 per foot, with explosive costs respectively, of \$0.77 and \$.099 per foot of raise.

Stoping System

In 1909, when about 74% of those drills placing holes in the solid orebody were of the reciprocating type of 3-in. piston diameter, the ore production averaged about 20 net tons of ore broken from the solid per 10-hour drill shift with an equivalent of 1.1 sticks of 1 by 8-in. of 50% dynamite per ton of ore.

In 1910, when about 72% of the producing drills were air-feed stoping (hammer) drills, the tonnage per drill shift rose to 38 net tons with about the same amount of explosive (which cost \$0.055 per net ton of ore broken), and 13.2 tons were broken per 10-hour shift of men working in stopes, or 1.32 tons per hour. Although there is no record of the breaking labor prior to this year, the fact remains that in the actual running of the drills only one man was used with a hammer drill, while two men were employed with each reciprocating drill.

In 1911, when the hammer drills were about 80% of the total, the stoping efficiency profited by the improvements in the drilling speed of the hammer drills, and 121 net tons were broken from the solid per drill-shift with about 0.8 stick of 50% 1 x 8-in. dynamite per ton (at an explosive cost of \$0.041 per ton), and 2.03 net tons were broken per man-hour of men working in stopes.

In 1912, when about 98% of the stoping drills were hammer drills, 195 net tons were broken per drill-shift with about 0.87 stick of 50% 1 x 8-in. dynamite per ton (at an explosive cost of \$0.045 per ton), and at a rate of 2.56 net tons per man-hour in the stopes. In 1913, when all the stoping drills were hammer drills, the length of the working shift was reduced from 10 to 8 hours in the middle of the year and the tonnage broken per drill-shift fell proportionately to 170 net tons, but remained at approximately the same hourly rating as for the year 1912. However, the tonnage broken per man-shift in the stopes increased slightly to 26.7 net tons (at an explosive cost of \$0.049 per ton), with the consumption of 0.89 stick of 50% 1 x 8-in. dynamite per ton. The tonnage broken per man-hour was 2.97, which showed a steady gain over previous years.

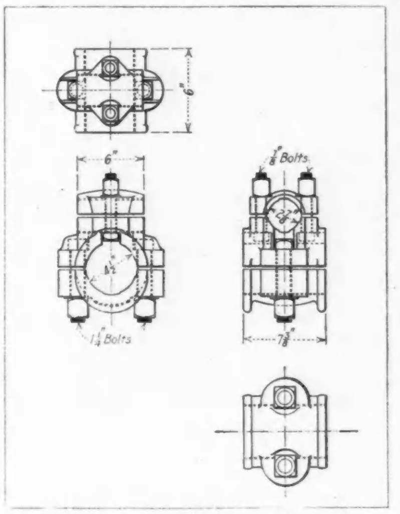


FIG. 7. IMPROVED COLUMN ARM CLAMP

It should be noted that the explosives charged against stoping include those used by the trammers in blasting ore in the chutes, and thus represent all the dynamite necessary to reduce the ore to the proper size for being handled through chutes and in the mill.

In order to provide broken rock for filling material to fill empty stopes to support the remaining orebody, "mill-holes" are developed in limestone country rock at the surface. For some years it was the practice to use 30-ft. bench-holes in the opencut for quarrying the rock, both 3-in. and 3½-in. reciprocating rock drills being used. It took steady work for two men to sink one 30-ft. hole in a 10-hour shift, and their work was hazardous because of the inconvenient places where set-ups were made, and because of the clumsy weight of their machines and the long, heavy drill steels which were handled. After the success of hammer drills in the underground mining operations, they were tried in the opencut work in 1912. Small holes were drilled to an average depth of 16 ft. and were given lighter burdens than had previously been the practice, for the object was to distribute the dynamite more evenly in the rock, as contrasted to churn-drill or mammoth blasts. In the tough, crystalline Franklin limestone this application of hammer drills to quarrying has proved superior to the heavy or mammoth blasts, for the same tonnage can be produced from a bench with a saving of labor and powder, since a great amount of expensive block-holing is avoided. A machine will drill about 100 ft. of holes in a shift with a heavy hammer drill and two men can drill only 30 ft. with a rock drill.

The light hammer drill, which may be mounted on a column to do sublevel drifting and to place the flat slicing holes, but which may be quickly removed and handled as a block-holer, has proved to be almost a necessity in top-slicing. The feed mounting, which has been superior in this service is of comparatively recent design and differs greatly from the shell and screw feed operated by hand. It is of a balanced pneumatic type so arranged as to permit the steel to be fed against the ground at various pressures, or held at any point, or retracted with various forces of pull. As a result it has been possible to start and drill holes through fissures and even crossing old holes at slight angles (which would not be attempted with the screw feed).

Another great convenience in drifting and top-slicing is the column fitting shown in Fig. 7. This special "cross" fitting permits a pieces of pipe to be used as an arm and allows an adjustment of its length to meet the requirements of pointing the drills without the likelihood of making another "set-up" of the drill column. A $4\frac{1}{2}$-in. column gives proper rigidity for the heavy drifting drills, but a $2\frac{3}{8}$-in. outside diameter arm (pipe) may be used satisfactorily and thereby save considerable weight in the arm and clamp required to fit the same.

Ore Occurrence as It Affects Mining Conditions in Bisbee

By W. B. Gohring*

It has struck me many times that mining conditions in Bisbee, Jerome and other camps where the ore is scattered make demands on the underground organization that differ radically from the demands made in the mines working on large "one-piece" orebodies, such as the porphyries and the big iron-ore deposits on the Mesabi range, for instance.

In the one-piece mines the orebody is outlined, shafts located, extraction drifts planned and a uniform mining method adopted by the management, often before a working opening is ever made in the ground; and when mining is finally started, the foreman and bosses are judged principally by the efficiency of their labor in getting out the ore as indicated on the daily reports. In Bisbee it is impossible for the management to lay out more than the most general plan of development, and the plan of attack on Bisbee ores is therefore a matter of almost daily conference between the shift bosses, foremen and superintendents.

Moreover, the management cannot accurately gage the efficiency of the underground work by study of daily reports in the office. Conditions, even in individual stopes, change so rapidly that what is good work on paper one week may not represent the same degree of efficiency the next, and the only way to judge of this is by personal contact with the underground changes in conditions. We have found that the law of averages works out in such figures and that they do represent the work fairly well over a period of time, but complex conditions render them unreliable in comparing one day or week with another.

The Bisbee foreman not only has to mine the ore, but has a part in finding it and in planning its extraction. He must not only keep up the daily routine output and keep his men efficiently at work, but must also keep the general development of his mine constantly in mind, with a view to finding new ore.

The ore occurrences are not along any regular lines; there is no symmetry or uniformity either in the situation of the different orebodies or in the physical or chemical nature of the ores themselves. A new orebody may be picked up in any part of the mine, resembling a fairly uniform vein. This orebody often will be made up of a hard pyrite ore with more or less hard and definite walls, and in the immediate vicinity the next orebody found may be a soft oxidized material of irregular outline and with its walls represented by nothing but the dying out of the copper contents. In other words, the walls may be the same material as the ore, apparently, but carry no copper and with no definite visible dividing line except the testimony on the daily assay sheets.

Another condition that constantly arises is when ore is encountered in new development work. It is explored by drifts and raises and a definite idea of its extent arrived at. A mining method is then adopted, and after the extraction drifts and loading chutes are carefully laid out and driven, the work of stoping begins. It is then not at all uncommon for stoping operations to show that our blocking-out work gave us an entirely erroneous idea ot the orebody. We find the main orebody to be off in a new direction not accessible to the extraction drifts we have cut, and it becomes necessary to lay out and drive new openings, and possibly to adopt an entirely different mining method.

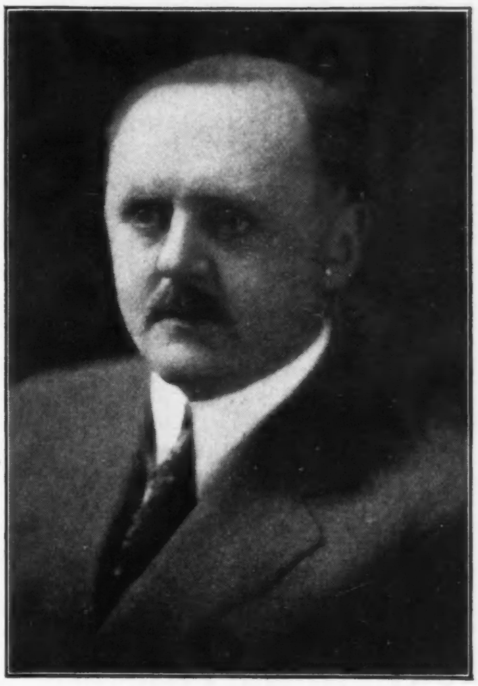
Again, considering any given mine as a whole, conditions of ore production change very rapidly. One year the ore may be coming chiefly from the north side of the shaft on two or three levels, and within a few months the bulk of the production from that shaft may be coming from the south side on entirely different levels. During this period changes will have been made almost daily in haulage drifts, loading chutes, air lines and what not, as well as in the organization taking care of that particular part of the mine.

Probably 50% of the output of the Calumet & Arizona mines comes from small, one-stope, isolated orebodies. Each of these presents a separate problem, and the method of stoping the ore and getting it to the shaft is decided by the underground organization for each one separately. In mining 73,000 tons of ore in January, 1917, an average month, the Calumet & Arizona Mining Co. worked 137 different stopes and used 18 modifications of four standard mining methods. It is obvious that under these conditions the management cannot plan the method of attack on its ores in detail and must rely on the judgment of its underground foremen to a great extent. It is obvious that these foremen must have the ability to handle routine work and get efficient results from men, but must further be men of thinking power and imagination.

*Superintendent of mines, Calumet & Arizona Mining Co., Warren, Ariz.

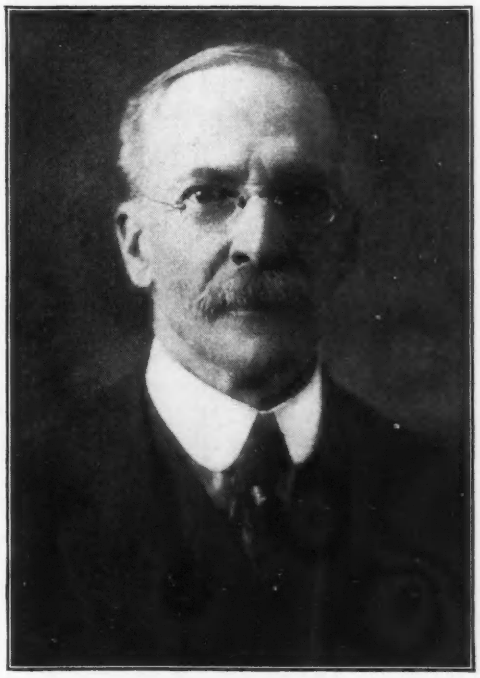
Well Known Mine Operators

GERALD SHERMAN
Superintendent, mining department, Phelps Dodge Corporation



JOHN KNOX
General superintendent, Calumet & Hecla Mining Co.

STANLY A. EASTON
Manager, Bunker Hill & Sullivan M. and C. Co.



CHARLES W. GOODALE
Manager, B. and M. department, Anaconda Copper Mining Co.

Loading Pocket in Burra Burra Mine

By Walter R. Hodge*

SYNOPSIS—The loading system of the Burra Burra mine consists of a series of raises delivering one into another and subsequently into a pocket, which in turn delivers the ore, through measuring capsules, into the shaft above the seventh level. The pocket is closed at the bottom by a vertical steel curtain in which are two openings provided with doors to control the flow of ore. Each of these openings delivers into a steel measuring capsule, also provided with a door. The door of each capsule opens into an inclined chute through which the ore runs into the skip.

The loading pocket here described is situated in the Burra Burra shaft of the Tennessee Copper Co.'s mines at Ducktown, Tenn. No claim is made for originality of design or construction. There is at least one pocket of similar design in operation at the Ray Consolidated mine, but the successful operation under the somewhat difficult conditions encountered in construction should warrant a rather detailed description of the work. It may be of interest to note that at no time during the cutting out of the rock, building the foundation or erection of the steelwork was regular hoisting from the shaft delayed for even one hour.

Details of the Pockets

A series of raises takes ore from each level above the sixth and delivers it into a pocket of 800 tons' capacity above the sixth level. This is an enlarged raise. It is stopped at the bottom by a concrete bin, or chute, which runs from the floor to the back of the shaft crosscut at the west edge of the opening to the lower pocket. In ground plan the outside dimensions are 8 x 8 ft. The south side of the chute is formed by the wall of the crosscut. The west wall is 18 in. thick, of concrete, reinforced with old rail and cable. In the north wall of the chute is

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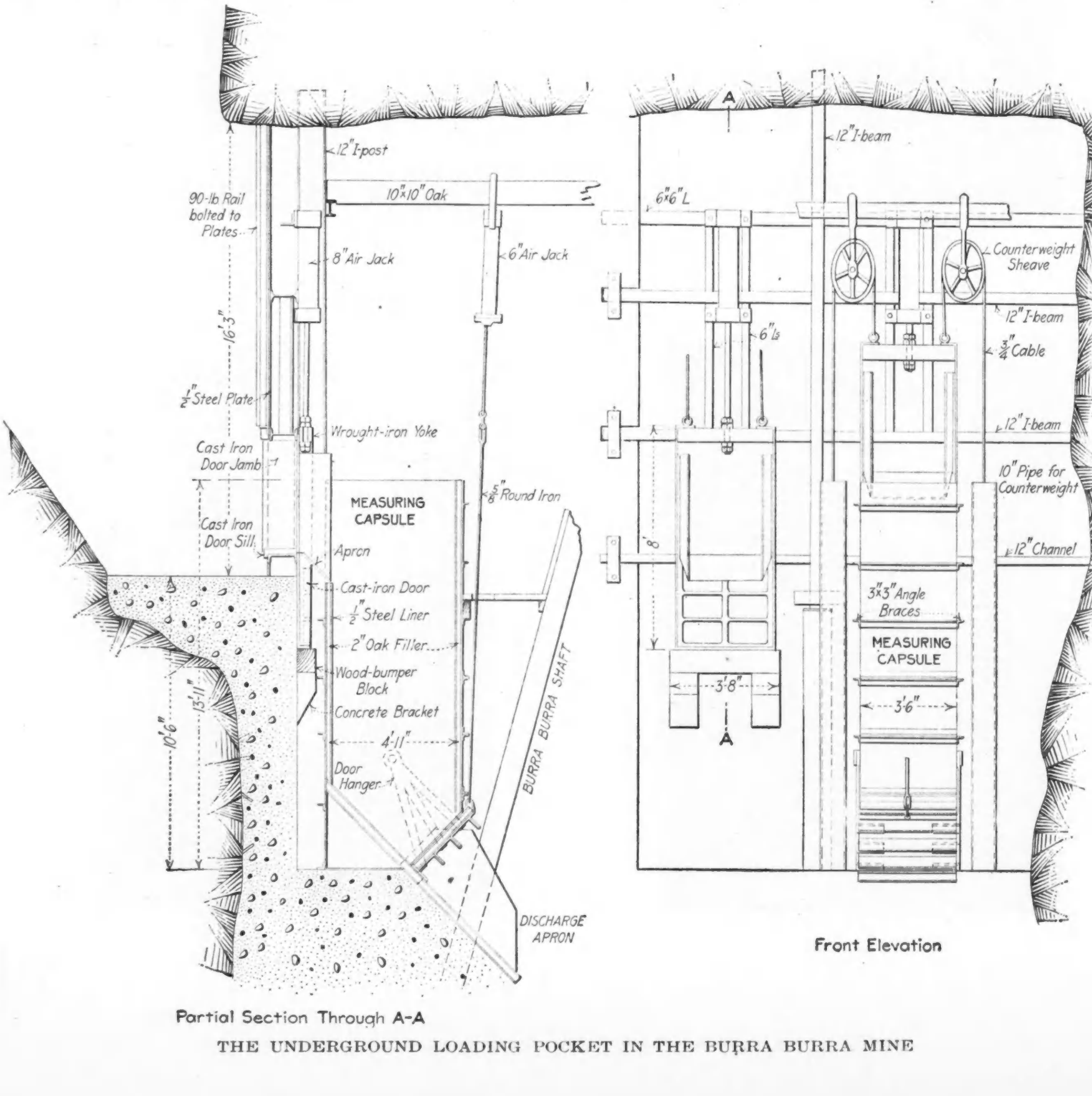

THE UNDERGROUND LOADING POCKET IN THE BURRA BURRA MINE an opening 3 ft. 3 in. square, to be used in emergency to load cars, but which is at present blocked with timber. On the east side, delivering into the lower pocket, is an opening 5 ft. wide by 4 ft. 4 in. high. The concrete about this opening is protected by cast-iron plates 1 in. thick, bolted to the concrete. Leading from the opening is a chute, 11 ft. long, of timber and 2-in. oak plank, lined with $\frac{1}{4}$-in. steel plate. This slopes down and away from the opening at about 30 deg. To control the flow of the ore, an arc gate, actuated by a rope and windlass, is placed at the end of the chute. The flat chute retards the flow of the ore sufficiently to prevent damage to the door, but still permits rapid discharge into the large pocket. All ore from above the sixth level passes through the chute here described. The sixth-level ore, hauled by an electric locomotive in side-dump cars, empties directly into the lower pocket.

The lower pocket is of 1500 tons' capacity, cut from the hanging-wall rock of the shaft, sloping at an angle of 55 deg. on the back and vertical on the front or shaft face. The lower opening is 17 ft. high and 20 ft. wide. It is stopped by a steel curtain. In the first few days of operation the fines in the ore packed into a hard mass with the natural slope of the ore—about 35 deg.—back from each door. This forms an excellent bottom which protects the concrete and is also self-renewing.

Details of Pocket Construction

The steel pocket consists of a frame-work of structural steel to which is riveted $\frac{1}{2}$-in. steel plate. This steel plate is stiffened and protected on the inside by 90-lb. steel rail bolted vertically to the plates on 6-in. centers. The rails directly over the door openings are supported by 6 x 6-in. angle-iron shelves. The other rails rest on the concrete floor of the pocket. There are three principal horizontal members to the framework. The lower member is a 12-in. channel, hitched and grouted into the rock at the south end of the opening and at the north end bolted to the concrete. The two upper members are 12-in. I-beams fastened in place in the same manner as is the channel. A vertical 12-in. I-beam runs from the floor concrete, to which it is bolted, to the rock back of the pocket opening, where a hitch and wedges secure it.

Through the curtain are two openings, with cast-iron door jambs, tops and sills 2 in. thick. The flow of ore through the openings is controlled by upcut doors. Each door consists of a heavy iron casting and a plate liner. The door is 3 ft. x 3 ft. 6 in. x 5 in. thick and is cored out on the back to reduce the weight somewhat. This casting weighs about 900 lb. Bolted to the front of the casting, with sixteen $\frac{3}{4}$-in. cone-headed bolts countersunk into it, is a steel-plate liner 1 in. thick. At each side of the door is a strap of 1 x 5-in. mild steel, which bends under the bottom of the door and is secured to it at the bottom and sides by 1-in. patch bolts. The straps extend 4 ft. 6 in. above the top of the door and are fastened by patch bolts to the top of a forged-steel yoke that is connected to the piston rod of an air jack. Above the door and bolted to it and to the side-straps is an apron of $\frac{1}{2}$-in. steel plate, 14 in. deep, which carries the ore over the back edge and liner of the measuring capsule and past the door guides. The door guides consist of 4 x 6-in. angles bolted to the floor concrete and riveted to the horizontal I-beam above the door opening. These guides are lined with 3 x 5-in. angles, easily renewable.

In its lowest position the door rests on wood-faced concrete brackets heavily reinforced with steel rail. Altogether, the door and appendages weigh about one ton. Through the yoke of the door on each side run 1-in. eye-bolts, and through each eye-bolt is clamped a $\frac{3}{4}$-in. cable, which runs up over a 24-in. sheave and down to the counterweight rod. The rod accommodates counterweights to over half the total weight of the assembled door. The counterweights are cast iron in 50-lb. units, 8 in. in diameter, provided with a vertical slot to accommodate the counterweight rod and cast with a bead on top which fits into a groove in the bottom of the next weight above and locks the weight securely to the rod. The column of counterweights swings in a 10-in. pipe attached to the capsule by straps of steel. The counterweight sheaves are kept well lubricated by means of grease cups fitted to the ends of hollow axles.

The air jack that actuates the door of the pocket is of 3 ft. 4 in. stroke and has an inside diameter of $7\frac{23}{32}$ in. The jack takes air at each end and so has a positive action both up and own. The two heads are cast iron and contain the air ports, and have lugs for bolting the air jack in position on the front of the pocket above the door opening. The cylinder is of 8-in. extra-heavy wrought-iron pipe turned to size inside. The piston cannot hit the lower head because the door is so adjusted that it rests on the brackets before the down stroke is entirely completed. The upper head is recessed to take a projection on the top of the piston, which it fits closely. On the up stroke an air cushion is formed, which prevents any serious jar. A hand-feed oil cup on the top of the cylinder furnishes lubrication to the cylinder, while the valves are lubricated by an automatic feed in the air line in front of the valves. A four-way valve operated by a 12-in. lever admits air to and exhausts air from the jack, and the single 1-in. pipe to each head of the jack serves alternately as inlet and exhaust. Air is used at about 80-lb. pressure at the cylinder.

Details of the Measuring Capsules

The measuring capsules are constructed of $\frac{1}{2}$-in. steel plate and 3-in. angles. On three sides the capsules reach to the top of the door opening, but at the back they come 11 in. below the sill of the door opening in the curtain. Inside the frame plates are 2-in. oak fillers on all four sides and liners of $\frac{1}{2}$-in. steel plate. The front and back liners and fillers are single pieces, but the liners on the sides are in sections 25 in. high, to allow easier handling. All bolt heads on the inside are countersunk, leaving a smooth surface in contact with the ore. The bottom of the capsule, which also is protected by a 2-in. oak filler and a $\frac{1}{2}$-in. steel liner, slopes at an angle of 45 deg. In a plane at right angles to the sloping bottom of the capsule is a discharge opening 3 ft. x 2 ft. 6 in. in the clear. This opening, except at the moment of discharge, is covered with a door suspended by three-pronged hangers from trunnions riveted to the sides of the capsule. The three-pronged hanger has the longest member at the bottom, so that the door, as it rises, draws away from the charge in the capsule. Except in case of a jam, it takes about two seconds to load the contents of the capsule into the skip; and in most jams a rapid up-and-down movement of the door suffices to start the ore running. The door of the capsule is of $\frac{1}{2}$-in. plate, 3 ft. x 3 ft. 6 in., with filler and liner, as in the capsule. It is heavily reinforced on the back with 6 x 6-in. angles. An iron rod, ⅝ in. in diameter, connects the door with an air jack swung on a timber above and in front of the capsule. This jack is 6 in. inside diameter and takes air only on the lower side of the piston for raising the discharge door. The door drops sharply of its own weight when the air in the jack is cut off.

An apron, wider near the capsule than the capsule door and at the opposite end narrower than the inside of the skip, directs the ore into the skip from the capsule. The apron, of ½-in. steel plate, lined with 2-in. oak filler and ½-in. steel plate, is bolted directly to the concrete floor of the capsule space and projects over the edge of the skip when the skip is on the dogs for loading.

Operation of the Loading Pocket

The skips operating in the Burra Burra mine are of about 5 tons' capacity and hoist in balance. The operation cycle of the pocket is as follows: No. 1 capsule is filled and dumped into No. 1 skip and the signal given to hoist. While No. 1 skip is being hoisted and No. 2 skip lowered, No. 2 capsule is charged and No. 1 capsule is recharged. Then No. 2 skip is loaded and rung up and No. 2 capsule is recharged. For periods of an hour at a time a skip a minute has been hoisted.

Pulling alternately from No. 1 and No. 2 doors imparts to the ore an oscillating motion which to a certain extent prevents jamming. However, in emergency a good tonnage has been hoisted from one compartment only. At times the ore will jam in the curtain doors or just above the tops of the doors. If this jam cannot be broken by thrusting bars against it, a short length of 4 x 6-in. timber is set against the key rock with its other end against the top of the lowered door. When air is applied to the door, the jam usually gives. If this fails, a small quantity of dynamite is used.

A 2-in. floor is laid just below the level of the pocket and surrounding the capsules. The space between the shaft and the curtain is well lighted by tungsten bulbs. A telephone connects the pocket with the main levels of the mine and with the surface. This makes the dumpers accessible from the eighth level, where, as on the seventh, the ore is still dumped directly into the skips from end-dump cars. On the third and sixth levels, the main levels, the ore is hauled in side-dump cars to the pocket system by a 3-ton electric motor handling six to eight loaded cars.

The Burra Burra shaft, a three-compartment foot-wall shaft inclined at an angle of 75 deg. to the horizontal, is sunk through the country rock, a mica schist, about 100 ft. from and parallel to the vein. The logical place for the loading pocket is between the sixth and seventh levels. From the center of the lower opening of the pocket to the seventh level the ground is badly shattered. This is particularly true of the ladderway (north) side of the shaft. Any attempt to cut out the rock as it stood would have put additional weight on the ground below and jeopardized the seventh-level station, so this ground had to be supported permanently before any breaking out of pocket was begun.

Before starting work in the pocket, this ground was supported by 12 x 12-in. posts and caps. The posts were set between the tracks delivering to No. 1 and No. 2 compartments and between No. 1 tram track and the ladderway compartment. The ends of the caps south of No. 2 track and north of the ladderway compartment were hitched into the walls of the station. Also a battery of 12 x 12-in. stulls, springing from the foot wall between No. 1 and No. 2 compartments and between No. 1 compartment and the ladderway, supported the brow above the station. When work commenced on the pocket, a post was set in the center of No. 1 tram track, supporting the cap, and all ore from the seventh level for a time was loaded in No. 2 compartment. Then the post between the compartments was removed. Two piers of concrete were built from the floor of the station against the back. The old timber caps were left in place, but the lagging was chopped out so that the concrete surrounded the caps, but was rammed solidly against the rock of the back. These piers had a ground section 28 in. x 5 ft. An arch between the two piers replaced the lagging. Old 20-lb. rails were placed vertically in the forms and wedged tightly against the floor and back to reinforce and tie together the piers. Rails bent to form were used to reinforce the arch. The rails here were placed on 14-in. centers. The forms were made of 2 x 6-in. oak plank with bracing of the same dimensions. The concrete was mixed by hand on the seventh level and shoveled into the forms. Near the back of the station the concrete was handled in buckets and placed with trowels and well rammed into place. No rock underground was suitable for aggregate, so all materials were sacked and taken underground between shifts and on Sundays. Suitable water was available at the station.

The Concrete Forms

When the piers and the arch joining them were in place, a mixing board was built at the head of the ladderway on the sixth level and a chute of 2-in. oak plank, 8 x 12 in. inside dimensions, was suspended down the ladderway. At the discharge end of the chute a flat distributing launder was swung. As the concrete rose in the forms above the seventh level, this chute was shortened and the distributing launder raised to a new position from time to time. The batches were about a half cubic yard in size and followed one another down the chute as fast as mixing permitted. Apparently the long fall, about 90 ft. maximum, of the wet concrete did not affect its quality, for after the concrete work had been completed, it became necessary to remove a part of it to provide for larger skips in proposed new equipment. At this time a close examination of the concrete broken out showed it to be a most-excellent and well-set mixture. However, this was probably due to the fact that it did not fall directly into place, but was run down the launder and shoveled into place in the forms, thus securing a remix of the material.

Above the piers on the seventh level the concrete was carried up and around the stulls supporting the brow of the station. From this point to 60 ft. above the level the ground had fallen out of the hanging-wall, leaving an irregular open space back of the shaft timbers, varying in depth from 10 ft. at the ladderway to nothing at the south end of the shaft and tapering above the pocket opening to nothing. This condition was of material assistance in placing the concrete. At a height of 18 ft. a flat arch was thrown across to the north wall of the ladderway and three legs, or dividers, were thrown from the foot wall to the face of the hanging-wall concrete. Of these legs, one was at the north end of the ladderway, rammed against the rock on its north side, one (18 in. thick) between the ladderway and No. 1 compartment and one (10 in. thick) between No. 1 and No. 2 compartments. The north leg was continuous vertically, but the other two were broken by the timber dividers, which were left in place and used as supports for the forms. Planks were nailed on the hanging-wall plates of No. 1 compartment and concrete filled in solid to 8 in. below the level of the capsule base. The hanging-wall plates were embedded in the concrete, but the studdles were removed to permit connecting the concrete dividers with the concrete of the hanging wall. A grillwork of old 20-lb. rails, running horizontallly and vertically, was used to tie the concrete together. Rails were embedded in the dividers and projected into the foot wall and set into the hanging-wall concrete.

The concrete gang consisted of a boss and four men at the mixing board. Another and often two more men were used to distribute and tamp the concrete in place. The same gang that placed the concrete built the forms. All forms in the shaft were built on Sundays and between shifts. There was little work directly in the shaft.

When the concrete was completed as previously outlined, the back of the shaft was heavily lagged with 6-in. round timber and the cutting of the rock began. Batter stulls were placed between compartments across the shaft to hold the brow of the pocket opening. These are permanent timbers. A drift, 14 ft. in length, was driven at the proposed top of the pocket opening. This was slow work, since at the start only light shots could be fired for fear of damaging the shaft timbers. From the end of this drift a raise was driven to the sixth level. Dropping back down the raise far enough to leave an 8-ft. floor in the station crosscut, a chamber was underhanded to the desired shape and size. This included cutting out rock below the opening drift and providing room for the capsules, etc. To obtain a maximum capacity with a minimum opening on the level and a stable floor to the station crosscut of the sixth level, the pocket was given roughly the shape of two funnels placed top to top, the front wall of the pocket being vertical and the back sloping at an angle of 55 deg. The opening on the sixth level was left the size of the raise originally driven and was covered with a door until the pocket was completed and the gate bin on the sixth level was ready for the removal of the forms. At that time it was enlarged to about 8 x 12 ft. The raise opening was just large enough to allow the passage of the largest piece of steel used in the construction of the pocket front and capsules. This piece, by the way, was the back of the capsule and was 10 ft. x 3 ft. 6 in. All waste rock from the cutting was dropped down a chute in the back of the ladderway to a bin on the seventh level, from which it was trammed to the skip.

Concrete in the Pocket

As soon as the rock portion of the pocket was cut to size and shape, concreting was resumed in the same manner as before. The concrete was brought to a level for the setting of the capsules and slopes into the shaft provided for the loading aprons. No bolts were set at this time except in the vertical wall back of the capsules for anchoring the capsules and for supporting the door guides in proper position.

Then a vertical wall 10 ft. 6 in. high was built from end to end of the shaft below the pocket opening and back of the position of the capsules. This was anchored to the rock by means of numerous pins set in holes angling into the rock. The wall was heavily reinforced with steel rails placed vertically and horizontally. The top of this wall was extended back into the pocket about seven feet to form a floor.

On the south side of the pocket the wall rock was firm and was merely trimmed to an approximately plane surface. On the north side, however, the rock was badly shattered and required substantial support. So from the floor of the pocket to the brow of the pocket opening, a concrete wall with a minimum thickness of 2 ft. was placed, reinforced as in the other work. This wall was also extended with a thickness of 18 in. across to the foot wall of the shaft. This forms a positive support to the fractured rock and takes the direct thrust of the north side of the steel curtain. The reinforcement of the part of this member in the shaft consists of old 1⅛-in. steel cable, burned to remove grease and laid in pairs in the concrete 18 in. apart vertically. Recesses were left in this member for the main horizontal members of the steel curtain.

Completing the Construction

Proceeding at the same time as the concrete work in the pocket, a series of raises had been prepared, delivering from the first to the sixth level. At the levels the raises had been funneled out to facilitate dumping and to afford protection against ricocheting fragments of falling ore. As an additional safeguard, heavy timber baffles were provided, from behind which the cars were dumped. When the raises were completed, the chute described above was built on the sixth level.

These raises, originally driven in the ore, were afterward replaced with raises driven in the foot wall. Or rather, they were replaced with a continuous raise delivering into the upper pocket at the fifth level and at the other levels provided with inclined openings into which ore from the cars is dumped.

The erection of the steel curtain and capsules began as soon as the concrete in the pocket floor and walls had set. All steel was lowered in the shaft to the sixth level and from there lowered through the raise opening to the floor of the pocket as needed. Hitches were cut in the south wall of the pocket to accommodate the main members of the curtain, 12-in. I-beams and channels. The north ends were slipped over the anchor bolts provided and fastened loosely, and the south ends were allowed to float free in the hitches. Not until the plates were all in place and riveted to the longitudinal members were the nuts on these anchor bolts turned home and the south ends and roof end of the vertical member wedged and grouted in place. The main members were first placed. Then the capsules were set up, riveted and set home on the anchor bolts in the back wall. Then the curtain was completed and the door jambs and sills bolted in place, and the air jacks and doors were hung, counterweights placed, liners and door fixed to capsules and air jacks for capsule doors hung. Then a floor was built around the capsules, piping and valves installed, telephone and signal bell put in, the apron chutes into the shaft bolted in place and the bottoms of the capsules bolted to the floor with split bolts. Finally, the raise opening on the sixth level was broken to its present dimensions and the control put on the concrete chute on the sixth level, and the pocket was ready.

The United Verde Underground Hoist

By T. D. Hawkins*

SYNOPSIS—Exposition of details of hoisting problem at United Verde and method of solution that lead to putting a new shaft in commission. The equipment of this shaft is described and hoisting and torque curves are given.

The problem of hoisting ore from the lower levels of the United Verde mine at Jerome, Ariz,. has been one that has received much consideration, with the result that a new shaft, No. 5, will be put in operation very soon.

The several shafts of the United Verde mine have been sunk just north of the town of Jerome. A tunnel has been driven northeast from Jerome, with its portal 1½ miles from the town and 1000 ft. below it, connecting all the shafts on this 1000-ft. level. This provides an outlet through which all the ore mined on all levels is hauled to a series of transfer bins one-quarter of a mile from the portal of the tunnel, where the ore is transferred from the mine trains into the bins, next to be loaded into the smeltery trains or to be crushed and treated in the crushing plant. A series of storage bins are situated at the other end of the tunnel near the several shafts now in use and almost adjoining No. 5 shaft. It may therefore be seen that all the ore mined above the 1000-ft. level is dropped by gravity to the storage bins on the 1000-ft. level, and all the ore mined below the 1000-ft. level must be hoisted to the 900-ft. level and dumped back to the bins on the 1000-ft. level, from which all outgoing ore passes; being loaded directly into standard-gage ore trains and hauled by underground electric locomotives to the transfer bins 1½ miles away. The mine railroad therefore operates for 1¼ miles underground, and ¼ mile on the surface, all on the 1000-ft. level. The foregoing outline of conditions shows the necessity for constructing an underground hoisting system.

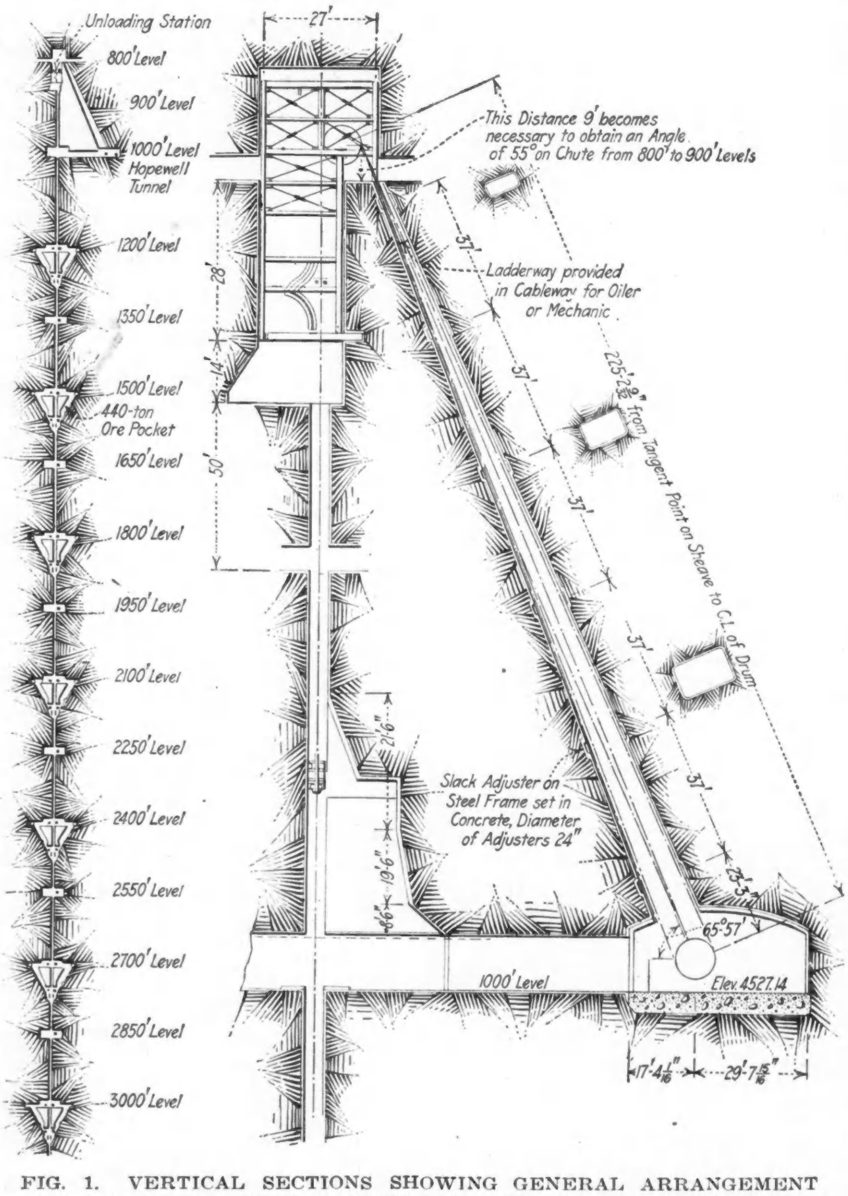

FIG. 1. VERTICAL SECTIONS SHOWING GENERAL ARRANGEMENT OF SHAFT AND DETAILS OF HOISTING AND DUMPING EQUIPMENT

In describing the method employed and the equipment installed in No. 5 shaft, attention may be invited to the following similar underground hoists of large capacity: First, two designed and built by August Kloone, in Dortmund, Germany, and in operation at the beginning of the European War. Second, to the single loading-station automatic hoist at Inspiration, Ariz. This No. 5 shaft contains three compartments — two skipways and a manway. Its present length is from the 800-ft. level to the 1850, and it carries three loading stations—one at the 1250 level, one at the 1550 level, and the last at the 1850 level—together with an unloading and selector station on the 850 level. It is contemplated to increase the number of loading stations below 1850 as smeltery conditions may require. Fig. 1 illustrates the general arrangement of the shaft and shows the several loading stations below the 1000-ft. level; the engine room and hoist station on the 1000-ft. level, the cable race leading from it to the headframe on the 800 level, and the unloading and selector station on the 850 level, with the gravity chutes back to the storage bins on the 1000-ft. level.

The shaft, together with all its loading and unloading stations as well as its engine room, cable race and chutes, is concrete-lined to insure a perfectly dry arrangement and long life of parts. There are no timbers whatever in any part of the shaft or its attending equipment, which is constructed entirely of concrete and steel.

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Provision is made to operate two skips of a modified Kimberley self-dumping type (each having a capacity of seven tons) and in balanced relation. These operate at an average hoisting speed of 1000 ft. per min. from a 900-hp. direct-current, double, drum-geared hoist.

Details of Construction of Shaft

The skipways are each 5 x 5 ft., and the manway is 5 ft. x 3 ft. 6 in. The three compartments are divided by two concrete curtain-walls each 10 in. thick. There are a series of pockets left in the concrete walls and on the sides of the compartments. The smaller pockets are for placing material when it becomes necessary to have men working on the repair of guides, etc., in the skipways. The large openings in the curtain-walls between the skipways are primarily intended to provide a means for the displacement of air from one compartment to the other when hoisting, and secondly to facilitate the passage of men working on guides from one compartment to the other. The pockets formed in the walls of the manway are provided for ladders, landings, etc.

Guide supports are placed every 5 ft. and consist of a section of 10-in. 25-lb. I-beam and two special handhold castings. The I-beams are punched in the web with three holes taking 3 x $\frac{3}{4}$-in. square twist reinforcing bars.

The guides are 4 x 8-in. Oregon pine treated with a decay-resisting compound and are attached to the guide support with two $\frac{7}{8}$-in. machine bolts staggered to provide a means for making a splice or joint in the guide at any guide support. The handhole recess formed by the casting provides means for connecting the nut and bolt, while a socket wrench is applied to the head of the bolt in a counterbored recess in the guide. The web of the I-beam provides an effective lock for the nut. All the bolts supporting the guide are recessed in a counterbore to provide clearance for the skip or cage shoes.

The manway comprises a complete set of ladder sections, each section 10 ft. long and inclined from one side of the compartment to the other, with landing stations every 10 ft. There are six $3\frac{1}{2}$-in. fiberduct conduits running the entire length of the shaft and concreted in the outboard wall of the manway. The manway is provided with junction-boxes at each level. All the power and light conductors run through these conduits and are tapped at each junction-box for the requirements of the particular level. Each junction-box is sealed with a gasket and clamp steel cover to exclude all moisture. Lighting is arranged with 10-watt incandescent lamps placed every 10 ft.; that is, at every landing station of the ladder. These lights are controlled by three-way switches from each level, in order that a man walking from one level to another may first switch the lights on in that section of the manway in which he is walking at the point from which he starts; and upon reaching the next level either above or below, he can switch them off and light the next section. In addition a master-switch on the 1000-ft. level controls all the lights in the entire manway. This system of wiring comprises an arrangement of three-way switches between each level and a master-switch on the 1000-ft. level to control the entire circuit; the master-circuit being used in the event of men working in the shaft.

The manway has anchor bolts set every 5 ft. in elevation in each outboard corner to provide for one 4-in. air-pipe, and one 8-in. standpipe for water (pumped), and one 20-in. pipe for water for drinking purposes, etc., all the pipes being held by straps to the anchor bolts.

The concreting of this shaft was accomplished by the use of steel forms, in 5-ft. sections for each set; there were sufficient forms to provide continuous pouring of the concrete. The concrete therefore had sufficient time to set before it became necessary to withdraw a form from its position to use again as the pouring continued upward.

The mixture of concrete used was in the proportion of 1:3:6—one part of portland cement, three parts of sharp sand (clean from dust), and six parts of clear gravel passing a 2-in. ring. The concrete was mixed on the 1000-ft. level and was conducted downward through a 4-in. pipe to a juncture for all points below the 1000-ft. level.

The following stresses relating to concrete and steel for structures were considered throughout the entire de-

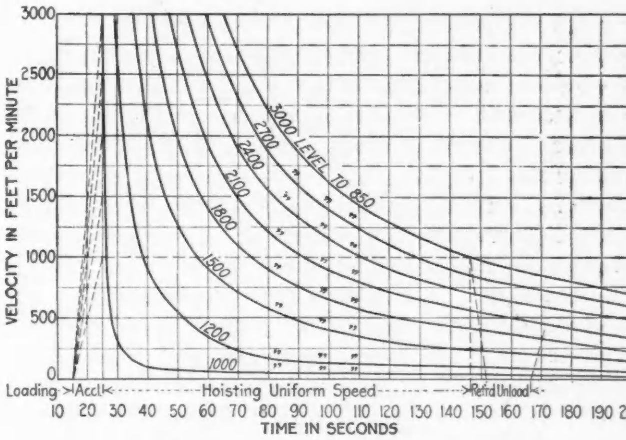


FIG. 2. SHOWING HOISTING CYCLE

sign of all work: Concrete mix, 1:3:6; tension, 0.00 lb. per sq.in.; shear, 150 lb. per sq.in.; compression, 300 lb. per sq.in.; formula for beams $\frac{WL}{10}$; bond, 0.00 lb. per sq.in. Concrete mix, 1:2:4; tension, 0.00 lb. per sq.in.; shear, 200 lb. per sq.in.; compression, 400 lb. per sq.in.; formula for beams $\frac{WL}{10}$; bond, 0.00 lb. per sq.in.

Steel: Standard structural sections 55,000 to 65,000 lb. ultimate strength per sq.in.—tension, 12,500 lb. per sq.in.; shear, 10,000 lb. per sq.in.; compression, 12,500 lb. per sq.in.; torsion in structure, 8000 lb. per sq.in.; rivets in shear, 6000 lb. per sq.in.; bolts in tension, 6000 lb. per sq.in.; max. $\frac{L}{R}$ of all columns or struts, 50.

The mixture in the proportion of 1:3:6 in the walls of the shaft provides ample strength to resist any slipping of rock, impact from skips, or outside influences tending to collapse them. The minimum section is 10-in. with no reinforcing in the outside walls. The curtain-wall has been reinforced every 5 ft. with three $\frac{3}{4}$-in. square twist bars. The resistance of these walls to collapse due to impact from side slap from a skip running at 1000 ft. per min., as well as from compression from the outside walls, is ample and has been well considered.

The skip used in this shaft is of a modified Kimberley type. It has a capacity of seven tons, and its weight is 12,600 lb. It is equipped on the base with a set of bronze bearings and a cradle seat bearing a locking-latch on the side bales, manganese trunnion wheels attached to a trunnion plate on the side, and hard white cast-iron removable liner plates inside the body. On the discharge side it has a 3-in. wooden pad in the bottom and is provided with air vents in the bottom to avoid any vacuum being created when wet or muddy material is dumped. The skip is made of structural steel, with the exception of the lower bearing pin-plates at the base of the bales, which are of nickel steel, the crosshead pin of chrome nickel steel, the lower bearing of cast iron, and the bronze journals with liners and trunnion wheel also noted. There is a clearance of $\frac{1}{4}$ in. all around between the guide shoes and the guides, and an adjustable distance of 1 in. between the guides and the wall of the shaft that provides an almost perfect alignment between the skip and the guides. All the guides on the skip are provided with removable wearing plates, thus avoiding the entire removal of the guide shoe when wear takes place.

The Hoist and the Power Plant

The hoisting outfit consists of the following equipment: One double-drum, double-brake, geared electric hoist in the engine room. The hoist is equipped with two 10-ft. flat drums, grooved to pitch a $1\frac{3}{8}$-in. round cable, with hydraulically operated brakes and driven by a 900-hp. direct-current reversible interpolar 500-volt series-wound motor. One of the drums of the hoist is over-wound with reference to the cable and the other under-wound. A set of controlling switches permits the changing of the relation of one drum to the other, with reference to the position of the skips in the shaft. Means are provided for running both drums in balanced operation, or one drum only and the hoist in unbalanced operation. Fig. 2 shows the hoisting cycle, while Fig. 3 shows the same with tabulations for torque on the hoist shaft at different points of the cycle for unbalanced operation, these points being indicated in the curve.

In addition to the hoist being equipped with adjustable controlling switches, it is arranged with an automatic over-wind release, operating to set the brakes if for any reason the skip is hoisted farther than its dumping station. A further automatic release operates to set the brakes and open all operating circuits if, for any reason, the discharge chutes from the dumping hopper become clogged or fill so as to impair dumping. Both of these automatic features will be dealt with under the description of the headframe and selector.

The power plant in connection with the hoist comprises a fly-type motor-generator set, a 900-hp. three-phase 60-cycle 2200-volt squirrel-cage induction motor on the same base, but separated from a 900-hp. 500-volt series-wound direct-current generator by a 20-ton 7-ft. diameter flywheel.

The control between the hoist motor and the generator on the motor-generator set are in series relation with each other, and the greatest flexibility obtainable with relation to speed and speed control can be insured by the arrangement of the combined field controls and with the least amount of energy expended.

A cable race is provided at an angle of 60 deg. from the hoist station to an outlet tangent with the sheaves on the headframe on the 800-ft. level. The cables running through this race are supported on guide idlers approximately 34-ft. centers in order to overcome any sweep or side slap that might be imparted to the cables upon quick reversal of the hoist, and for various other reasons.

The headframe comprises the general steel structure shown in Fig. 1 situated at and below the 800-ft. level. The headframe supports two 10-ft. sheaves operating in two bronze journals, and a 20-ton traveling crane operates above them. The dumping bin is three skip lengths below the center line of the sheave shafts. It has a capacity of seven skip loads and is equipped with a revolving selector. The latter is arranged to travel over an arc of 60 deg. and to stop over any one of three chutes connecting with three bins on the 1000-ft. level below. This revolving selector is electrically driven and is arranged to be automatically operated by means of a set of contactors on the engineer's

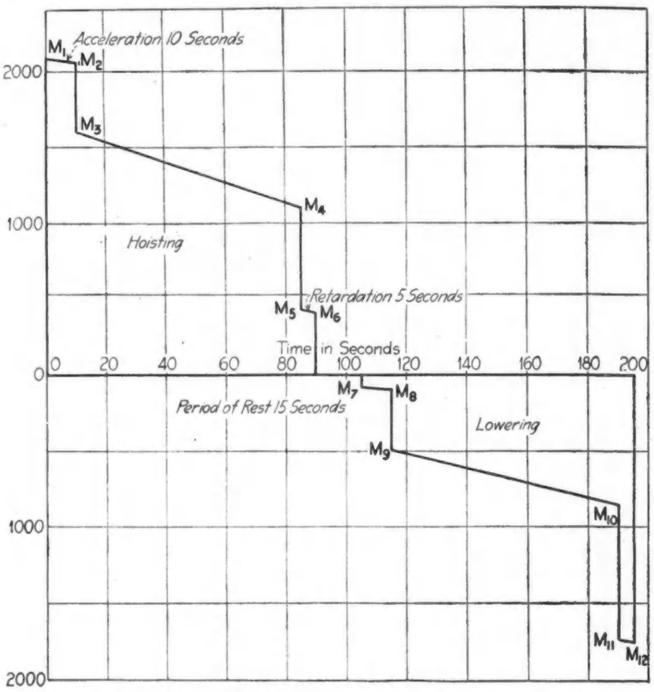


FIG. 3. SHOWING HOISTING CYCLE IN DETAIL

Ore 14,000 lb., ship 12,000, $1\frac{3}{8}$-in. diameter rope, drums and sheaves 10-ft. 0 in. diameter, depth 2,200 ft., rope speed 1,000 ft. per min.

TABULATION

M_1 =	229,082 ft.-lb. =	2,100 hp.	M_7 =	8,871 ft.-lb. =	81 hp.
M_2 =	227,652 ft.-lb. =	2,080	M_8 =	10,301 ft.-lb. =	94
M_3 =	179,115 ft.-lb. =	1,635	M_9 =	58,838 ft.-lb. =	538
M_4 =	143,310 ft.-lb. =	1,310	M_{10} =	94,643 ft.-lb. =	865
M_5 =	46,235 ft.-lb. =	422	M_{11} =	191,718 ft.-lb. =	1,750
M_6 =	45,520 ft.-lb. =	416	M_{12} =	192,438 ft.-lb. =	1,760

platform and a switchboard in the hoist station on the 1000-ft. level. The trunnion of the rising skip engages the dumping guides and begins to turn outwardly toward the hopper, the latch on the skip passing through a parting arranged in the guides at this particular point, dumping then being effected into the hopper. Should the skip be taken beyond its dumping point, an over-wind switch and hatch is provided covering the shaft. The skip on coming in contact with the over-wind switch operates through a relay to open the main circuit of the hoist motor and set the brakes, which cannot be closed until the skip has been lowered away from the over-wind hatch switch. The selector hopper is also provided with side diaphragm plates, and should it become filled as a result of choking chutes, the side thrust from the ore thus held in the hopper tends to expand the diaphragm plates, which operate through a relay to open the main circuit on the hoist motor and set the brakes. These cannot again be closed until the ore has been removed from contact with the diaphragms and the latter returned to their normal positions. From this it will be noted that no operator is required above the 1000-ft. level during operation of the shaft, excepting the periodical trips of an oiler. All the operation is carried out automatically in conjunction with, or independent of, control from the 1000-ft. level. A series of pilot lights on the switchboard in the engine room indicates the position of the selector.

General Features of Loading Stations

There are three loading stations connected with the shaft—one below the 1200-ft. level, one below the 1500-ft. level, and another below the 1800-ft. level. The bins commence from the level above at an angle of 60 deg. each converging toward the center line of the shaft. Below these bins and immediately adjoining the shaft is a measuring cartridge. Directly at the bottom of the bin, there is a circular gate which, when opened, discharges ore into a cartridge having a capacity of seven tons. At the bottom of the cartridge is another circular gate which, when opened, discharges ore directly into the skip hanging below the chute connecting with the gate and discharging into the shaft. The ore is discharged from the bins into the cartridge by opening the bin gate until it has completely filled the cartridge to the angle of repose. The bin or upper gate being an undercut gate, displaces part of the ore upon being closed, and thereby avoids the excessive friction of shear that would result, should it be necessary for the gate to completely cut off a solid shaft of ore in the loading cartridge. The gate at the bottom of the cartridge is an overcut gate and operates only to discharge the cartridge load into the skip. These gates are each operated pneumatically. The bin gate has a 16-in. x 5-ft. trunnion cylinder to operate it, and the cartridge gate has a 12-in. x 4-ft. trunnion cylinder for the same purpose. These cylinders are controlled by lever-operated valves situated in a room immediately above the loading cartridges. There are two of these cartridges corresponding to the two compartments of the shaft and two sets of gates, cylinders, and operating valves. Each set of operating valves is equipped with a master valve, and means are provided to assure the opening of only one gate at a time. If the bin gate is open the cartridge gate must be closed, and if the cartridge gate is open the bin gate must be closed, thus insuring a foolproof arrangement wherein the contents of any bin or any portion of any bin cannot be discharged into the shaft or in the sump unless it is deliberately intended that this should be done. Furthermore, the gallery is provided with a grating floor so that the operator of the station is in full view of the loading operations of the gates and cartridges at all times. Although the entire loading station is well lighted, a further precaution is taken against the closing of one gate before the other may be opened, by means of a series of bypassing valves connected with the gates, whose function is to insure the positive closing of the gate into the seat before the other gate can be opened, even though the operator should desire to open one gate before the other is closed. All the cylinders are self-cushioning in order to retard the speed of the piston at either end, by means of poppet valves with dashpot sleeves to avoid excessive jars and impact usually resulting in air-cylinder operation. Each station is provided with a small traveling crane, placed over the operating platform, for the purpose of removing any parts needing repair. Collapsible doors are provided in each shaft compartment to allow for the entrance of an operator from the skip or cage to the station, and for the moving or exchange of any parts needing quick repair. A passageway is provided from the operating room on one side to the operating room on the other side of the shaft, crossing through and over the manway. In this passageway the junction boxes for lighting and power control are placed, also the air and water control devices. Each operator is provided with a telephone and signal station both of which connect with the hoist station on the 1000-ft. level. One operator is usually needed in each station from which hoisting is done, and no more than two at any one time are required in the working of the shaft.

The hoisting capacity of No. 5 shaft in tons per 24 hours is approximately 5500; the number of men required for the total shaft operation is never more than four and seldom more than three, comprising the engineer in the hoist station on the 1000-ft. level, two operators, and possibly only one on the loading stations below the 1000-ft. level, and one oiler for all the levels and stations on the shaft. Precautions have been taken throughout the entire construction to insure the speedy repair of any damaged parts during operation. There is a 75-ton traveling crane in the hoist station arranged to take any part of any machinery from its place and deliver it on a standard-gage car on the tracks in the drift at the main entrance to the station, connecting with the main tracks in the 1000-ft. level tunnel. There is also a traveling crane above the headframe at the 800-ft. level, with a capacity of 20 tons, arranged to lift and move any part into the shaft and if necessary to lower it to the 1000-ft. level and to a car on the tracks mentioned. Cranes of 10 tons' capacity have been installed above the operating rooms in every hoist station for the same purpose. Each station on every level has immediate connection through drifts and by track to other shafts, thus facilitating the hoisting of broken parts for repair through one of the other shafts during a temporary shutdown of the hoisting machinery of No. 5 shaft. Each compartment of No. 5 shaft is provided on the 1000-ft. level with skip-changing and repair pockets to insure quick and easy means of changing skips. There are approximately 18,000 cu.yd. of concrete in the general construction, with 1000 tons of steel and 1100 tons of machinery. The period of construction covered over two years, and the maximum number of men employed at any one time of construction was 125.

Magnesite in Washington

The American Mineral Production Co., through its sales representatives, H. H. Brunt & Co., 662 Insurance Exchange Building, Chicago, has begun to market crude magnesite of a coarsely crystalline character, said to be similar to the standard Austrian magnesite, from a large deposit located about 50 miles north and west of Spokane, Wash., according to a report published in *Iron Age*. The company has let contracts to the Schaeffer Co., Tiffin, Ohio, for a calcining plant and, pending the erection and completion of this equipment, is preparing the deposit to be worked on a large scale. Shipments of the crude magnesite thus far for refractory purposes, have been limited to trial cars, but the company is submitting samples of its product to those desirous of acquainting themselves with the quality of the mineral. An average analysis shows: Silica, 0.8%; alumina, 0.4%; iron oxide, 2.7%; lime, 3.3%; magnesium carbonate, 92.1%.

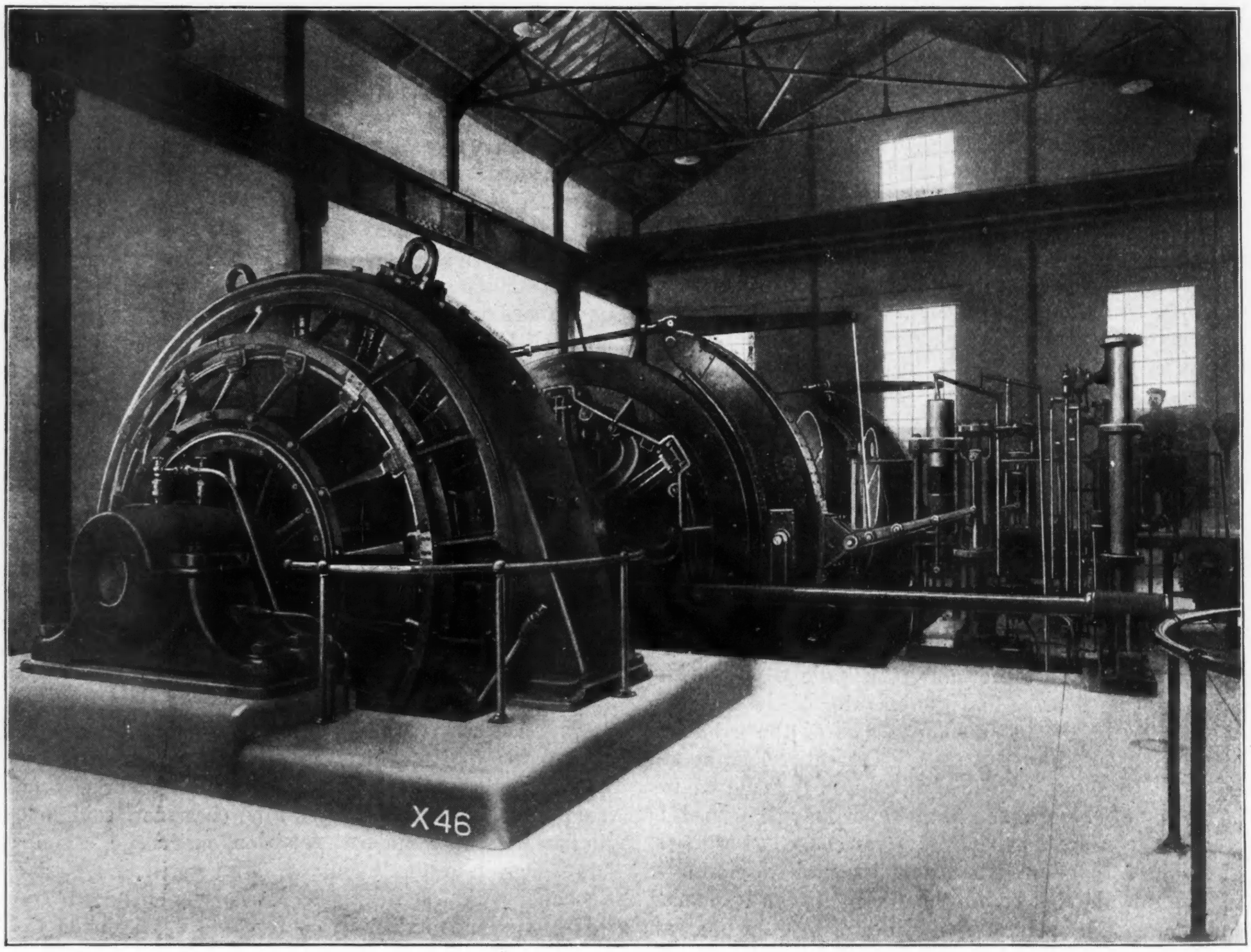
Recent Developments in the Design of Hoisting Engines

By Frederick W. O'Neil*

SYNOPSIS—Electric hoists have undergone great development recently due to the wide distribution of electrical energy at mines. Steam hoists are now built in the compound condensing type and are most economical in the use of steam. Electric-generated air-driven hoists have undergone improvement and refinement.

Recent advance in the design of hoisting engines has been marked chiefly by the great development of the electric hoist, due to the wider distribution of cheap electric energy in mining districts, and by improvements in steam hoists.

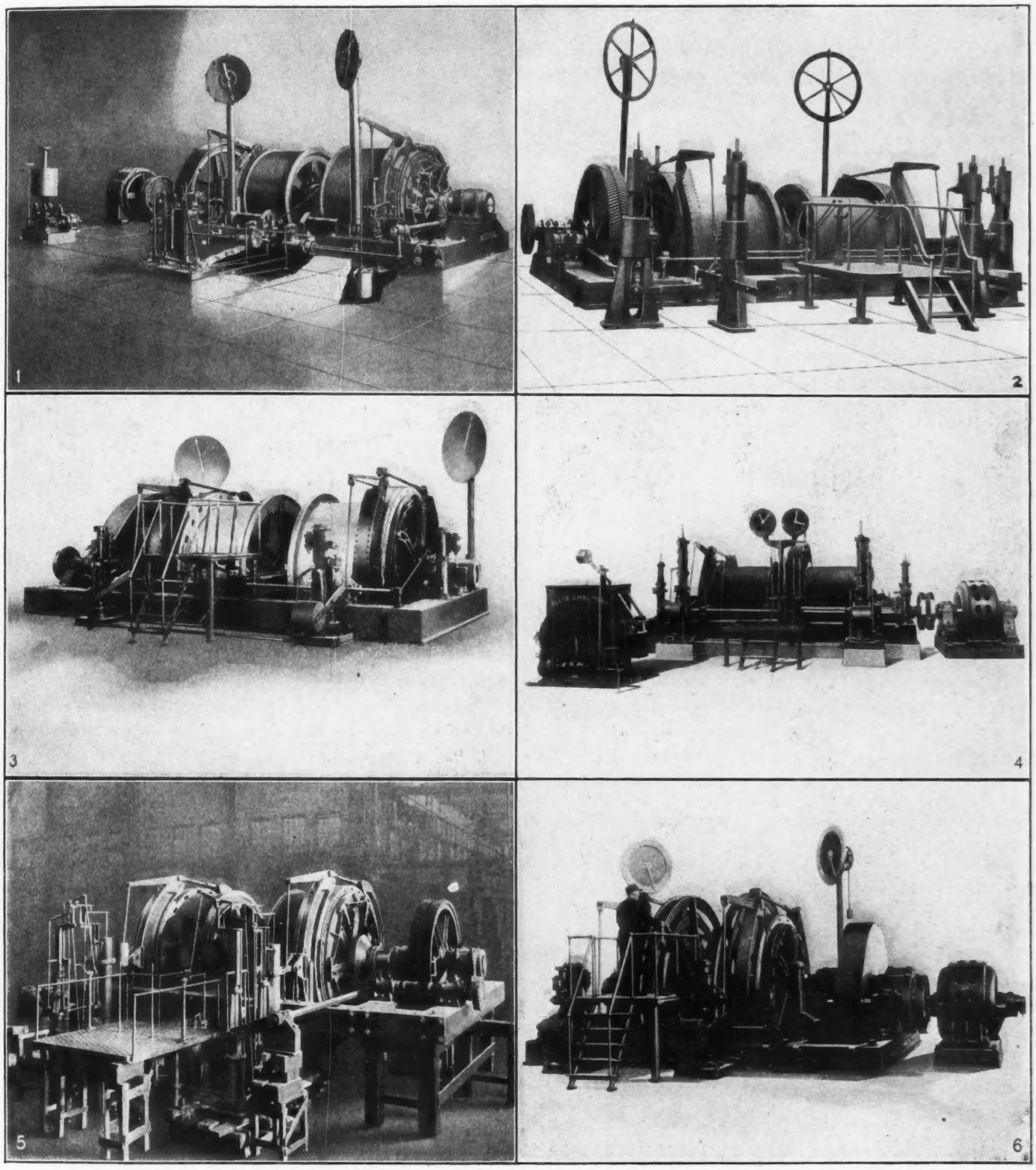
The type of electric hoist that is being most widely used is the induction-motor type in small and moderate sizes, using rope speeds up to, say, 1500 ft. per min. These hoists are usually geared in order to make use of high-speed commercial motors, and most of them can now be built with a single reduction in gearing, owing to the recent advances in the making of herringbone gears. Hoists typical of this class are shown in Figs. 1, 2 and 3. A hoist made by the Nordberg Manufacturing Co. for the Wakefield Iron Co. is shown in Fig. 1. It has drums 6 ft. diameter, 42 in. face, for 600 ft. of 1¼-in. rope. The total rope pull is 12,000 lb.; gear ratio, 10.7 : 1; hoisting speed, 500 ft. per min.; and it is driven by a 450-r.p.m. 200-hp. motor. The hoist is equipped with herringbone gears, flexible coupling, gravity post brakes and axial plate clutches; clutches and brakes are hydraulically operated. Electric control is of the contactor type with master controller. Overwinding devices are provided. In Fig. 2 is shown a hoist made by the Wellman-Seaver-Morgan Co. for the Empire mine, Grass Valley, Calif., to handle a load of four tons from a depth of 7000 ft. on an incline of 42 deg., at a hoisting speed of 1200 ft. per min. The drums are 6 ft. diameter, 5 ft. face, made of steel plate, each drum being equipped with Lane clutch and post brakes operated by air-thrust cylinders. The motor is of 750 hp., 435 r.p.m., and the control is of the liquid-rheostat type; safety devices are provided. The third example of this type of hoist is shown in Fig. 3,



LARGE ILGNER-TYPE ELECTRIC HOIST, BUILT BY NORDBERG MANUFACTURING CO. AND GENERAL ELECTRIC CO. FOR SENATOR CLARK'S ELM ORLU MINE AT BUTTE

*Assistant general sales manager, Ingersoll-Rand Co., 11 Broadway, New York.

which illustrates a machine made by the Allis-Chalmers Manufacturing Co. for the United Verde Copper Co., having a maximum rope pull of 33,500 lb., from a depth of 2200 ft.; rope speed, 1000 ft. per min. The drums are 10 ft. diameter, 5 ft. face, of steel plate, for 2200 ft. of

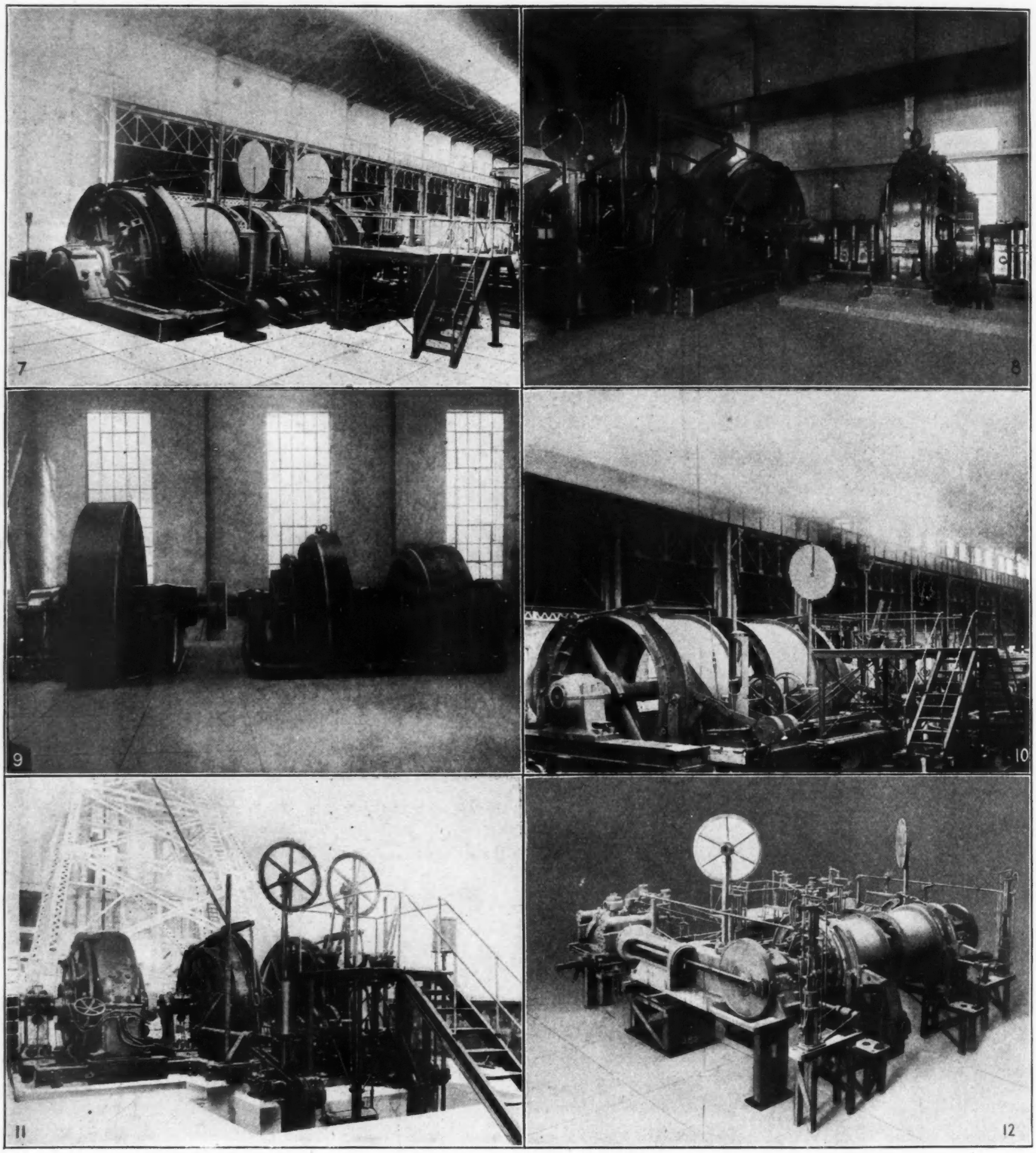


FIGS. 1 TO 6. VARIOUS MAKES OF ELECTRIC MINE HOISTS AT VARIOUS WELL-KNOWN MINES

Fig. 1—Wakefield Iron Co. Fig. 2—Empire Mines, Grass Valley. Fig. 3—United Verde Copper Co. Fig. 4—Allis-Chalmers hoist with liquid rheostat. Fig. 5—Furukawa Mining Co., Japan. Fig. 6—Allis-Chalmers reel hoist

1$\frac{3}{8}$-in. rope. The hoist is equipped with Brown axial multiple-arm friction clutch and gravity post brakes, hydraulically operated. It is equipped with single-reduction Wuest herringbone gears and flexible coupling, is driven by direct-current motor, and automatic safety devices are provided. Since this hoist is operated by a direct-current motor, automatic acceleration and retardation devices are also furnished. Other examples of this type of hoist are shown in Figs. 4 and 5. An Allis-Chalmers Manufacturing Co. induction-motor-driven hoist with liquid-rheostat control is shown in Fig. 4. This hoist is designed for 17,000 lb. rope pull, each drum being 7 ft. diameter, 5 ft. face, with 2000 ft. of 1$\frac{1}{4}$-in. rope, and is driven through single-reduction herringbone gears by a 200-hp. 580-r.p.m. motor. Each drum is equipped with Lane friction clutch and post brakes, operated by air-thrust cylinders, the hoisting speed being 650 ft. per min. A hoist for the Furukawa Mining Co., Ashio, Japan, is shown in Fig. 5. This was made by the Nordberg Manufacturing Co. and has a drum 10 ft. diameter, 42 in. face, equipped for 1800 ft. of $1\frac{1}{4}$-in. rope, with a total rope pull of 18,600 lb. A hoisting speed of 1000 ft. per min. is maintained, through a gear ratio of 11 : 1, driven from a 375-r.p.m. 375-hp. induction motor. Drums have axial plate clutch and gravity post brakes operated by air-thrust cylinders, and the hoist is equipped with the Welch-type safety stops.

Most of the hoists recently built are equipped with cylindrical drums for round rope, but a few have been designed with reels instead of drums for the use of flat rope. Such hoists when electrically driven have the advantage of a low starting torque, but have the disadvantage of all reel hoists in that the cost of rope maintenance is high. A reel hoist is shown in Fig. 6. This was built by the Allis-Chalmers Manufacturing Co. for a maxi-



FIGS. 7 TO 12. SHOWING TYPES OF ILGNER ELECTRIC AND AIR-DRIVEN MINE HOISTS
Figs. 7 to 9—North Butte Mining Co. Fig. 10—Canadian Copper Co. Fig. 11—Silver King Coalition Mines. Fig. 12—Anaconda Copper Mining Co.

mum rope pull of 20,000 lb., from a depth of 2500 ft., driven by a 400-hp. induction motor through Wuest gears, the control being of liquid-rheostat type. Each reel is equipped with Brown axial clutch and gravity post brakes hydraulically operated.

The chief advantages of the induction-motor hoist are its low first cost and high efficiency. It is the most efficient type of electric hoist. The chief disadvantage is the large starting current required, so that it becomes prohibitive to operate large hoists with this type of electric drive, owing to the immense draft of current from the line and consequent disturbances to the electric system. For such cases other devices must be used, as described later. Another disadvantage is that this type is not adaptable to automatic electric control of safety devices, and safety devices for such hoists have to be mechanical.

There are three different types of control used for induction motors. For hoists using motors up to 150 hp. a drum type of controller is satisfactory. For hoists larger than this the control must be of the master-controller type, where the primary current is not handled through the controller, but through master switches or contactors which handle the main current, and these switches are in turn magnetically operated from the master controller. In place of the contactor control liquid rheostats are also used to a great extent, but are generally more expensive than the former until a motor size of 350 or 400 hp. is reached. A liquid rheostat of this type is illustrated with the hoist in Fig. 4. For safety devices for electric hoists I believe that a modification of the Welch type of automatic stop is the best so far developed. This device consists essentially of a notched wheel or miniature geared positively to each drum and so constructed that a finger, when brought in contact with the notches, releases a latch. The position of this releasing finger is determined by a centrifugal governor, and when the latch is released it shuts off the current from the motor and applies the hoist brake by either mechanically opening switches or valves, which in turn control the current supply or thrust cylinders on the brakes. Such a device prevents the operator from overspeeding at any time and also causes him to do the following things: (1) At a predetermined and adjustable point in the shaft to start to slow down and to continue to slow down until the dump is reached; (2) to stop the hoist at the dump; (3) to reverse the hoist before starting. Hoists with hydraulically operated brakes are best suited to the application of this form of safety device, and in fact there are no satisfactory safety devices for electric hoists with hand-operated brakes.

The horsepower of an electric hoist is rather a misnomer and means little. A hoist motor must be capable of furnishing the large starting torque required, and after that to conform to the cycle of operation without overheating, as this is the criterion which determines the motor size. In other words, a 200-hp. motor means that a motor having a normal continuous capacity of 200 hp. will not overheat when following the hoisting cycle. This is an entirely different thing from the maximum power that the motor may develop in following the cycle. During the cycle the horsepower varies continuously, and the motor may have to handle a peak of 400 hp. Therefore, when selecting a motor for a hoist, it must be determined by its heating capacity after knowing the operating cycle by the well-known "root mean square" method of analysis.

Considering now the hoists where the loads and hoisting speeds necessarily give powers such that induction motors, if used, would give prohibitive starting current, we come to the type commonly known as Ilgner set, which consists of a hoist motor, generally direct-connected, of the direct-current type and operated by a motor-generator set consisting of an induction motor, direct-current generator and heavy flywheel mounted on a common shaft. These sets are usually designed to operate at about 15% slip when handling the peak of the load, and thus give up to the hoist about 30% of the energy of the flywheel, so that the current represented by this amount of energy does not come back on the line. The control is of the Ward-Leonard type. The main advantage of this type of electric hoist is to keep peaks off the line and also with this type the control is excellent; in fact, it is almost perfect. Further, dynamic braking can be used, because every position of the control lever corresponds to a fixed speed of the hoist motor, so that such hoists can be braked by motor as well as by brake.

One disadvantage of this type is its high cost, especially when the hoist motor is direct-connected, owing to its low speed. Another disadvantage is its low efficiency compared with induction-motor hoists. Charles Legrand, of Phelps, Dodge & Co., has made a good modification of this type by using a geared hoist which employs the higher speed and cheaper direct-current motor, but this of course cannot be used in hoists of great size. The operating condition under which Ilgner hoists will show their best efficiency is rapid and continuous hoisting; that is, where one hoisting cycle follows immediately after another and where there are no long periods of idleness. Unfortunately this is not a condition which obtains around most metal mines, and the stand-by losses are considerable. For example, the motor-generator set that serves the hoist shown by Fig. 7 takes 100 kw. to run idle. The ideal field for the Ilgner hoist, and a field which has not yet been invaded by it to any great extent, is in the coal mines of this country. The heavy loads, high speeds, short lifts and continuous service are ideal for the application of the Ilgner system.

Results of tests on large German hoists[1] show over-all efficiencies of from 28 to 53%, shaft horsepower to electric input to flywheel set. Shaft horsepower is the net work represented by weight of ore times height lifted. It should be borne in mind, however, that these German hoists, which consist of single reels of so-called Koepe type, using tail ropes between the cages in the shaft, give higher efficiencies than the large drum hoists used in American practice, due to our large unbalanced loads and the larger inertia effects. The hoist (Fig. 7) built by Wellman-Seaver-Morgan Co. and Westinghouse Electric and Mfg. Co. for North Butte Mining Co., is the largest unit of this type in operation. This hoist was designed to handle a load of seven gross tons of ore, at a hoisting speed of 2700 ft. per min., from a maximum depth of 4000 ft., under a cycle which will give a production of 200 tons per hour. The drums are 12 ft. diameter, 9 ft. 4 in. face, of cast-steel construction, each drum to handle 4000 ft. of $1\frac{5}{8}$-in. rope, the total rope pull being 42,000 lb. The clutches and brakes are operated by hydraulic cylinders, and in

[1]"Zeit. des Vereines Deutsch. Ing.," 1912, p. 1457.

addition to the main brakes each drum is fitted with an auxiliary band brake operated by hand.

The electrical equipment consists of an 1850-hp. 71-r.p.m. 550-volt direct-current motor, operated by a motor-generator set driven by a 1400-hp. 505-r.p.m. alternating-current motor. The flywheel on the motor-generator set weighs 100,000 lb. The hoist is provided with safety devices, so as to bring the control lever to a central position at the end of each trip and to automatically apply the brakes in case of overwind. The motor end of this hoist is shown in Fig. 8, while the type of motor-generator set used in this type of hoist is shown in Fig. 9. A similar hoist built by the Wellman-Seaver-Morgan Co. for the Canadian Copper Co. is shown in Fig. 10. This will handle 9 tons of ore from a depth of 1800 ft., at a hoisting speed of 2500 ft. per min., the drums being 12 ft. diameter, 7 ft. face, made of 1½-in. steel plate on cast-steel spiders.

Nordberg and Westinghouse are now building for the Butte & Superior Mining Co. the largest hoist of this type in the world. It will have two drums 12 ft. diameter, 10 ft. face, each to hold 5000 ft. of 1⅝-in. rope, the maximum rope pull being 52,000 lb. and the hoisting speed being 3000 ft. per min. At each end of the shaft there will be an 1800-hp. direct-current motor, or 3600 hp. total. A hoist of the reel type made by the Wellman-Seaver-Morgan Co. for the Silver King Coalition Mines is shown in Fig. 11. This is driven by a direct-connected direct-current motor. The hoist was made for a capacity of three tons hoisted from a depth of 1300 ft. at 1200-ft. rope speed. Each reel has a capacity of 1500 ft. of 4-in. by ½-in. flat rope. The hoist is operated by a 200-hp. 65-r.p.m. direct-current motor, driven from a motor-generator set. The illustration shows one of the advantages of flat-rope hoists; that is, to place them near the head-frame where the operator can see the ascending cages.

Electric-Generated Air-Driven Hoists

There is another way to use electricity indirectly for hoisting—applicable to hoists of large size—which prevents prohibitive peaks coming on the electric transmission lines. The system consists in compressing air with electrically driven air compressors and using this air in hoisting engines. A machine of this type, built by the Nordberg Manufacturing Co. for the compressed-air system of the Anaconda Copper Mining Co., Butte, Mont., is shown in Fig. 12. From this central system approximately 30 large hoists are operated.[2]

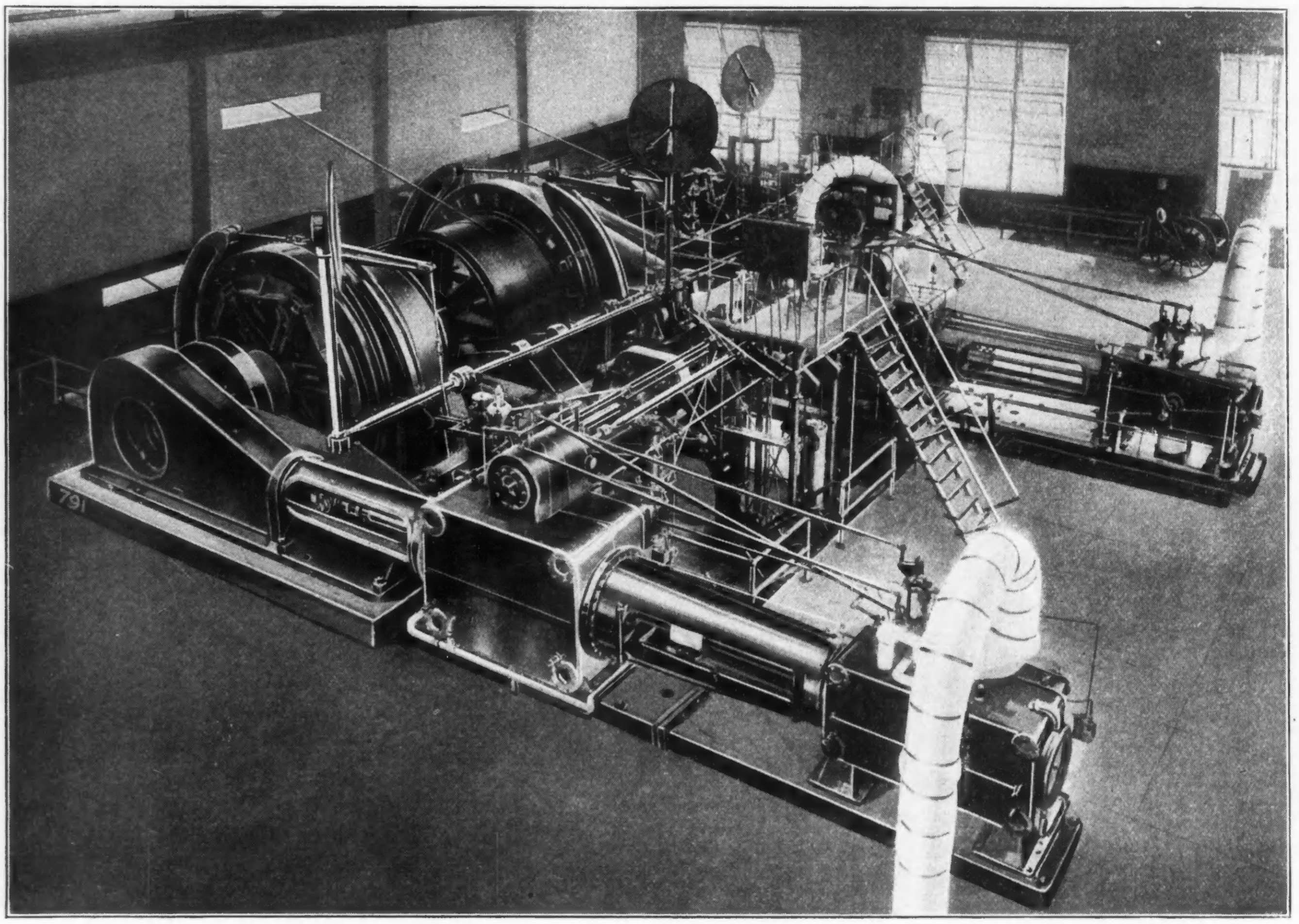


FIG. 13. SHOWING ONE OF TWO NORDBERG STEAM HOISTS BUILT FOR NEWPORT IRON CO.

The essential feature of the system consists in reheating the compressed air before it is used in the hoists. The hoist cylinders are of the convertible type, so designed that during the period of retardation the engine becomes an air compressor and sends the compressed air back into the system. Furthermore, the compressed-air storage is of the hydrostatic direct-displacement type; that is, the compressed air is held in tanks under a hydrostatic head equivalent to the air pressure, so that the stored air is

[2]This system has been fully described by Mr. Nordberg in "The Compressed-Air System of the Anaconda Copper Mining Co.," "Trans. A. I. M. E.," Vol. 46, p. 826.

directly displaced by water and is therefore all available. This system is well adapted to large installations and is as efficient as the large electric hoists of the Ilgner type. It is to be regretted that there are no more data of the comparative efficiencies of the two types of hoists, and it is to be hoped that in the future more of these data will be available.

Compound-Condensing Steam Hoists

No review of the developments of hoists would be complete without referring to the recent advance in building steam hoists of the compound-condensing type, and it may be said of the large hoists where the condensing water is available that this is the most economical type of any. With such hoists a performance of from 25 to 30 lb. of steam per shaft horsepower-hour is obtained.

One of the two hoists built by the Nordberg Manufacturing Co. for the Newport Iron Co. is shown in Fig. 13. It has cylinders 20 in. and 37 in. by 66 in., and drums 10 ft. diameter by 66 in. face, for 2700 ft. of 1¼-in. rope; total rope pull, 22,000 lb.; rope speed, 2400 ft. per min.; steam pressure, 150 lb. All auxiliaries are hydraulically operated. Such a hoist in addition to running condensing has the great advantage of using steam expansively right from the start and thus overcomes the waste of steam used in starting hoists of the high-pressure single-cylinder noncondensing type. Such hoists must be provided with special condensers which have large steam spaces so as to maintain the vacuum during the acceleration period.

The Nordberg Manufacturing Co. is now constructing for the Quincy Mining Co. the largest hoist ever built. This is of the compound-condensing type. There will be two sets of compound cylinders, each 32 in. and 60 in. by 66 in., the cylinders being set at an angle of 45 deg. from the horizontal. The drum will be of the cylindrical-conical type, 30 ft. diameter in the middle, 16 ft. at each end, to wind 11,000 ft. of 1⅝-in. rope up an incline which varies from 54 deg. at the top to 36 deg. at the bottom.

High-Voltage Motors for Mines

By Robert S. Lewis*

For a long time the standard motors for driving mining and milling machinery have been operated at approximately 220 or 440 volts. They are usually induction motors of the "squirrel-cage" or "phase-wound" type. As higher and higher voltages have been used in electrical work, it is only natural that higher-voltage motors should have been developed. Standard 2200-volt motors are now manufactured by the electrical companies and may be had as readily as motors of lower voltages.

The advantages of using 2200-volt motors may be illustrated by comparing them with 220- or 440-volt motors. There is a saving in transmission copper because, owing to the high voltage used, the line wires may be smaller in size. The amount of copper required varies inversely as the square of the voltage. Better voltage regulation can be secured. On account of the high cost of copper, there is a tendency to skimp the amount of metal put into the lines. They are often made so small that there is considerable loss in voltage; 5% to 10% is considered good regulation on lines to motors, but voltage drops of 15%, 20% or even 25% are not uncommon. With the high voltage, the lines may be of small wire and yet have less drop in voltage. This voltage loss results in a diminished output, since the power output of a motor is directly proportional to the speed, and the speed of an induction motor is somewhat dependent upon the voltage. This is an important fact that is frequently overlooked.

Better efficiency of transmission is obtained. For the same distance, same power transmitted and same size of conductors, the line loss varies inversely as the square of the voltage. Thus it is evident that high voltages are desirable. As a rough rule, for successful or satisfactory transmission with ordinary prices for copper, poles and other transmission-line parts, the pressure at the receiver end should be 1000 volts per mile of line. By this rule the proper voltage for various transmission distances may be selected. Thus for transmission of power up to one-quarter of a mile, use about 220 volts; for half a mile, use 440 volts; and for greater distances, up to two or three miles, use 2200 volts.

Transmission construction is cheaper and more compact when high voltage is used. Now that the individual-motor drive has become so popular for milling machinery, motors are scattered all over a modern mill. The high voltage makes it possible to use smaller conductors and conduits, consequently less building space is required, installation is easier and the construction is cheaper. Where power is generated at 2200 volts, it can frequently be directly transmitted to the motors without the use of transformers. This point is not so important where the current is sent long distances over high-tension transmission lines, but many milling plants have power-generating stations not more than two or three miles away, so that transformers would not be required for at least the larger motors. In the large mills using several thousand horsepower, there is a marked reduction in the quantity of copper required in the busbars and feeder lines distributing power from the mill substation to the various motors, when the pressure is 2200 volts.

The arguments against the use of high-voltage motors may be stated as follows: Slightly higher cost of motor; because of the high voltage, the insulation must be heavier and the winding more carefully done. The net result is a slight increase in cost. The sizes smaller than 20 hp. are now special, and where necessary, lower-voltage motors with transformers are used.

The weight is somewhat greater in the smaller sizes. For the larger sizes the weight is no greater than for 440-volt motors. Greater danger to employees has been suggested, but this objection has been disproved in practice. Owing to the greater precautions taken in insulating the motors and installing the leads and conduits, the danger to operatives is really not so great as with the less-protected motors of lower voltage. With high voltage, all motor parts are covered, the starters or compensators are metal-cased and grounded and are fitted with conduit connections.

The manufacturers have realized the possibility of the motors burning out and have constructed them with great care. The latest types of compensators are equipped with automatic overload cutout relays and ammeters. These motors have proved fully as reliable as the low-voltage machines. High-voltage motors have been installed in many of the new milling plants in the West.

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Uses and Costs of Aërial Tramways

By William C. Kuhn*

SYNOPSIS—Types of aërial tramways are described and classified, and the suitability of the system to solve problems of transportation under varying conditions is dealt with. Details of practice and method in installation and operation, together with approximate cost of material and construction, are used to favor the advocation of the system for general transport work. Applicability is shown to range from the distribution of articles in various stages of manufacture in factories, to the transport of large tonnages of ore for considerable distances across country.

Aërial tramways are adopted as a method of transporting materials in receptacles suspended from carriages running on overhead stationary track cables and their movement controlled by a running rope known as a traction rope. In Europe and other foreign countries these equipments are known as aërial ropeways to distinguish them from surface railways, which are frequently simply termed tramways.

There are really two types of aërial tramways. The original type was known as single-rope, in which the moving rope, being endless and running continuously in one direction, served the purpose both of supporting and propelling the carriers.

The second type is known as the double-rope system, which can be divided into two classes—the reversible tramway and the continuous tramway. In the reversible type there may be again two classes, the first, known as a single reversible, having only one bucket operating to and fro on a single line of track cable and controlled by an endless traction rope. The second class, known as a double reversible tramway, has two track cables and two carriers, each of which travels to and fro over its respective track cable, and both are controlled by an endless traction rope. On the single reversible tramway power is usually required to haul the carriers one way. On the double reversible tramway it frequently happens that no power is required, as the loading terminal when located at a sufficiently higher elevation enables the loaded carrier, when descending, to haul the empty carrier up on the opposite cable. The cycle being accomplished, the direction of the line is reversed.

The capacity of the above-mentioned types of tramways is somewhat limited, being controlled by the length of the line and the limit of weights for the individual loads.

The particular advantage of the reversible types is that they can be operated with a minimum amount of labor. As a rule only one man is required, stationed at the loading end. The carriers are generally arranged to automatically discharge their contents when they reach the discharge terminal.

With the continuous type of tramway there are two track cables and an endless moving traction rope. On this tramway there are a number of carriers all traveling in the same direction. The loaded carriers travel over a heavy cable, and the empties return on the opposite side of the tramway over a somewhat lighter cable.

The continuous system of tramways is more widely adaptable than any of the other types mentioned. It is

FIG. 1. LOAD OF LUMBER ON TRAMWAY NEAR QUINCY, PLUMAS COUNTY, CALIF.

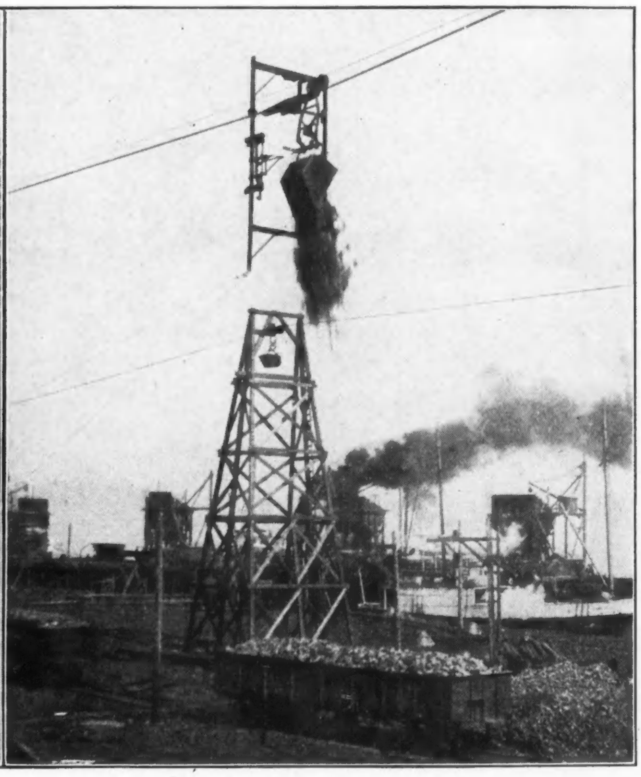

FIG. 2. AUTOMATIC DUMPING BUCKET AT TACOMA SMELTING CO., TACOMA, WASH.

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capable of transporting greater tonnages and can be constructed to greater lengths. For example, there are continuous tramways with capacities up to 300 tons per hour and lines have been constructed up to 22 miles long.

With the first-mentioned type, the single-rope system, the capacity is limited by the size and number of the individual loads that may be transported and the contour of the ground over which the tramway is to be constructed. If there are steep grades, this system is impracticable, as the bucket is suspended from a casting resting on the rope and held in position by friction only. The weight of the load is determined by the size of rope that can be satisfactorily operated by terminal machinery.

The question of relation of weight of individual loads to diameter and tension of supporting cable must be very carefully considered if maximum service is to be secured from cable equipment. Aërial tramways are

Track cables are installed at a predetermined tension, usually about 30,000 lb. per sq.in., customary practice being to have these tensions maintained by suspended weight-boxes attached to one end of the cable, the opposite end being anchored. These weight-boxes, depending on the size of the cable, are effective for a distance of from 4000 to 5000 ft. Therefore, when the length of the tramway is greater than, say, one mile, it becomes necessary to install intermediate tension structures for each additional 4000 or 5000 ft. These are placed at points where they will be most effective.

The intermediate supports (see Figs. 1 and 2) are placed at necessary points along the tramway. On level ground or ground having a uniform slope the supports will have an average spacing of about 250 ft., or about 20 or 22 to the mile. The heights vary from 10 to 80 ft. Where the tramway line passes over a sharp ridge or sum-

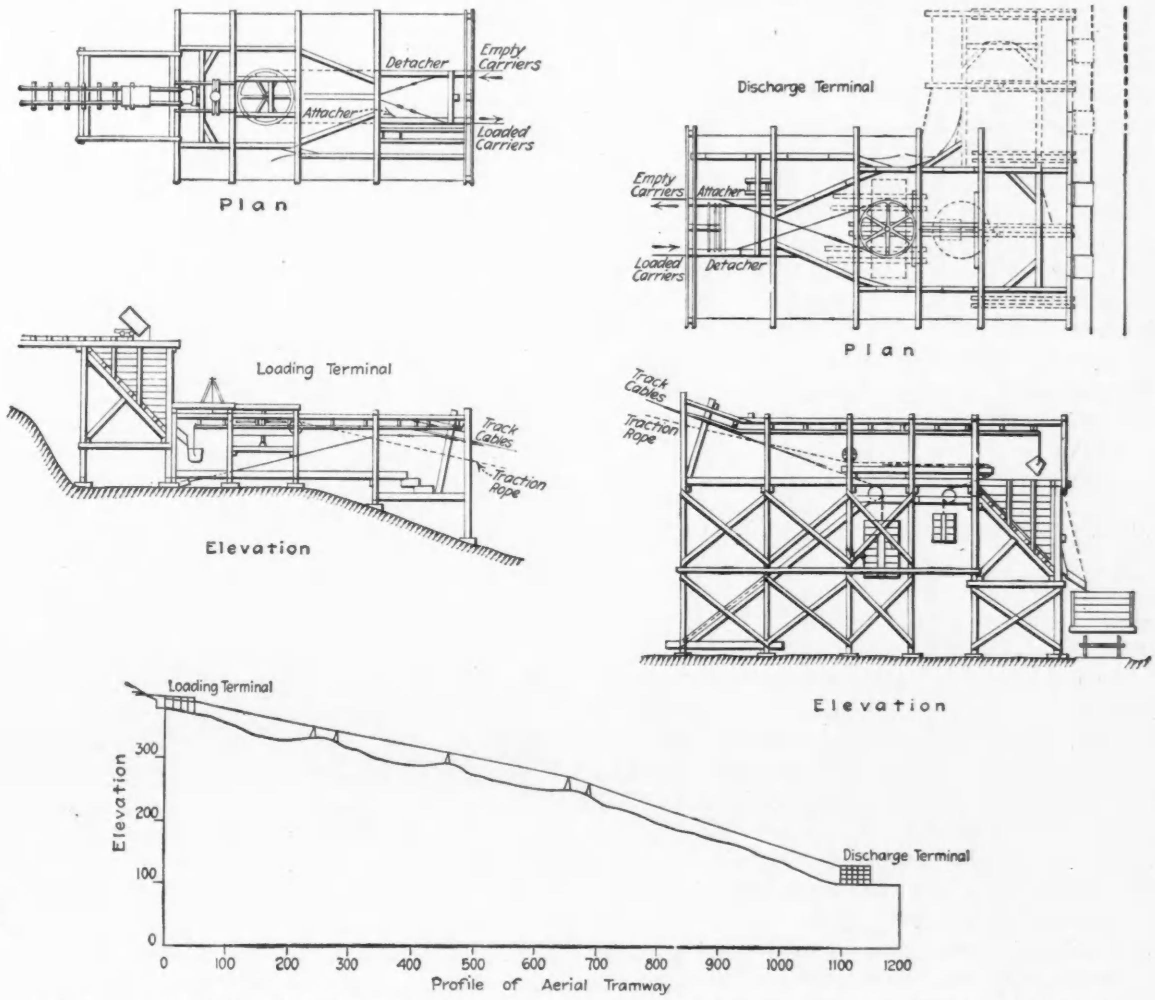


FIG. 3. TERMINAL DETAILS OF TRENTON-BLEICHERT CONTINUOUS TYPE AERIAL TRAMWAY

preferably constructed in a straight line from the loading to the discharging point. With the reversible type the lengths for satisfactory and economical operation vary up to one-half mile, with capacities up to 30 or 40 tons per hour for double reversible lines and about half as much for single reversible lines.

On the continuous system, however, it is sometimes found necessary or advantageous to change the direction of the line, horizontally, at one or more intermediate points. On such occasions an angle structure is installed, which deflects the traction rope. The track cables are replaced by overhead rails at these points so as to transfer the carriers from one section of cable to the next.

mit, it is sometimes necessary to substitute for the regular type of support, a curved-rail structure, through which the carrier passes without detaching from the traction rope. On these structures the track cable is replaced with a steel rail, thereby relieving the cable of excessive wear and strain due to severe bending and increased load from downward pressure of the traction rope. There are occasions, however, when it is possible to substitute a trestle instead of a curved-rail structure. The trestle type of structure consists of a series of bents each equipped with a saddle similar to the regular supports. This series of saddles permits of dividing the angle into small factors favorable to the track cables.

The terminal structures (Fig. 3), where the carriers are loaded and unloaded vary in design according to the type of tramway and the particular requirements of the purchaser. The design of these structures is such that the rails can be run to any desired point or points of loading or discharge.

The machinery for controlling the moving traction rope and carriers is generally placed at the terminal station. There is, however, an exception to this rule in the case of long lines divided into two or more sections. On such lines structures known as intermediate control stations are installed. Here the traction rope is divided and the carriers are transferred from one section to the other. The controlling machinery consists of sheaves of large diameters with their complement of brakes or power connections depending on whether power is developed or required. On lines developing a large amount of power, connections are usually furnished in addition to the brakes, so that the excess power developed may be utilized or controlled by a suitable device.

On continuous tramways the carriers are usually equipped with friction grips (Fig. 4), so that they may be attached to and detached from the traction rope. This is particularly advantageous, as it distributes the wear over the entire rope, the grips seldom if ever pinching the rope twice consecutively in the same place. Also, by detaching the carriers they can be shunted to any point in the terminals for loading or discharging. The loading of the buckets on tramways transporting ore, coal, stone, gravel, etc., is accomplished from bins through chutes at the loading end. They may be emptied into bins, crushers, etc., at the discharge end (see Fig. 3).

The Costs of Aërial Tramways

As to the cost of tramway equipment, this will vary according to type of line, capacity, length, peculiarities of the ground profile and terminal requirements. On a single reversible tramway the equipment will cost from about \$1100 for a line 500 ft. long at 5 tons per hour up to \$3600 for a line 2000 ft. and 10 tons per hour capacity. The shorter the line the greater the capacity, 30 tons per hour being possible on a 500-ft. line. On a double reversible tramway the equipment will cost from \$2250 for a line 500 ft. long at 8 tons per hour, to \$7000 for a line 2000 ft. long and 18 tons per hour. Up to 50 tons per hour can be transported on a 500-ft. line. The cost of installation will vary from one-third less to equal to the cost of material.

The cost of continuous tramway equipment will run from \$6000 per mile for cables, carriers, traction rope and machinery and bolts for intermediate supports, plus about \$2500 for terminal machinery, for a line of from 5 to 10 tons per hour capacity, up to about \$18,500 per mile, plus \$4500 for terminal machinery, for a line of about 200 tons per hour capacity. For instance, for a tramway 3 miles long at 10 tons per hour, the approximate cost would be $3 \times \$6000 = \$18{,}000 + \$2500 = \$20{,}500$, to which should be added the cost of timberwork, erection and installation, which would probably add another \$15,000 to \$20,000. These figures are rough approximations for a uniform line, but will furnish an idea of the outside cost for an equipment of this kind.

As to cost of operation, this will run from 1c. to 5c. per ton-mile, depending on the capacity and length of the line. The cost per ton-mile is lower on long lines than on short lines of equal capacity, as the labor requirements are practically the same in each case and the greater mileage proportionately reduces all costs. The number of men required will vary from two to ten, depending on capacity transported.

On first-class tramway equipments the cost of repairs and maintenance is a very small item. For the first two or three years it is practically nothing, and after this period it would average from 3% to 5% of an amount equal to about two-thirds the value of the cables, machinery and timber-work. Power costs are at a minimum because in the great majority of instances a tramway will operate by gravity and develop surplus power. On lines where power is required, it is still small for the reason that equipment to be operated consists of only the terminal machinery, requiring from 1 to 5 hp. and the moving traction rope

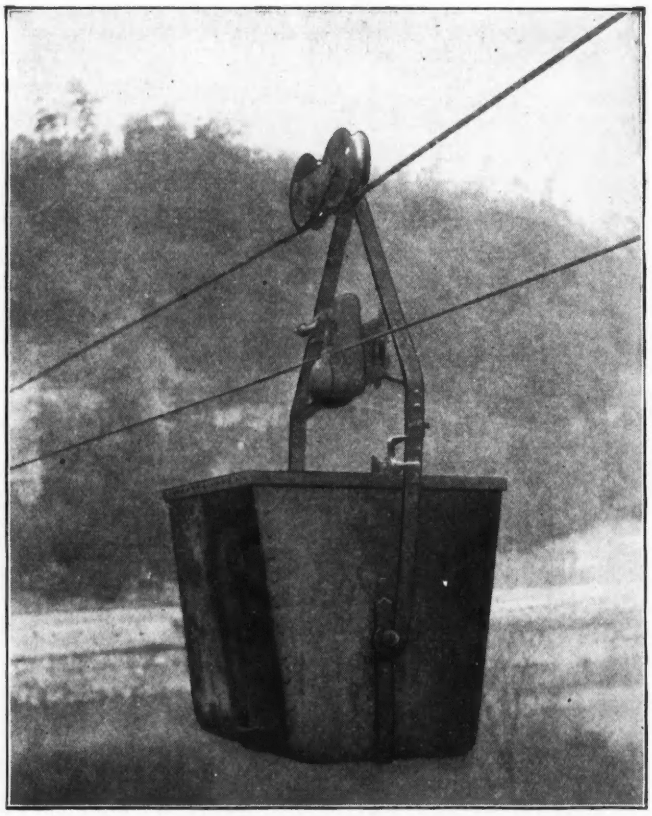


FIG. 4. TYPE OF SELF-DUMPING BUCKET FOR CONTINUOUS AËRIAL TRAMWAY

and carriers. Friction does not enter into the problem to the same extent as with a surface railroad using locomotives. By the use of what is known as a grip sheave (a large sheave with a number of grip jaws uniformly spaced around its periphery) it is possible with a comparatively low tension to grip the traction rope securely and prevent slippage. This provides a direct pull on the rope and carriers and permits of operating on grades up to 90%.

Aërial-tramway equipments, manufactured by reputable concerns, have passed the experimental stage. Experience has proved them to be a reliable and economical method of transporting materials even on level ground and comparing favorably with railroad haulage. As waste-disposal plants they are unsurpassed. The buckets being arranged for automatic dumping at any point along the line, the only labor required to operate is at the loading end (see Fig. 2). Transportation may be effected on any material of any size and weight convenient for loading and handling at the terminal stations.

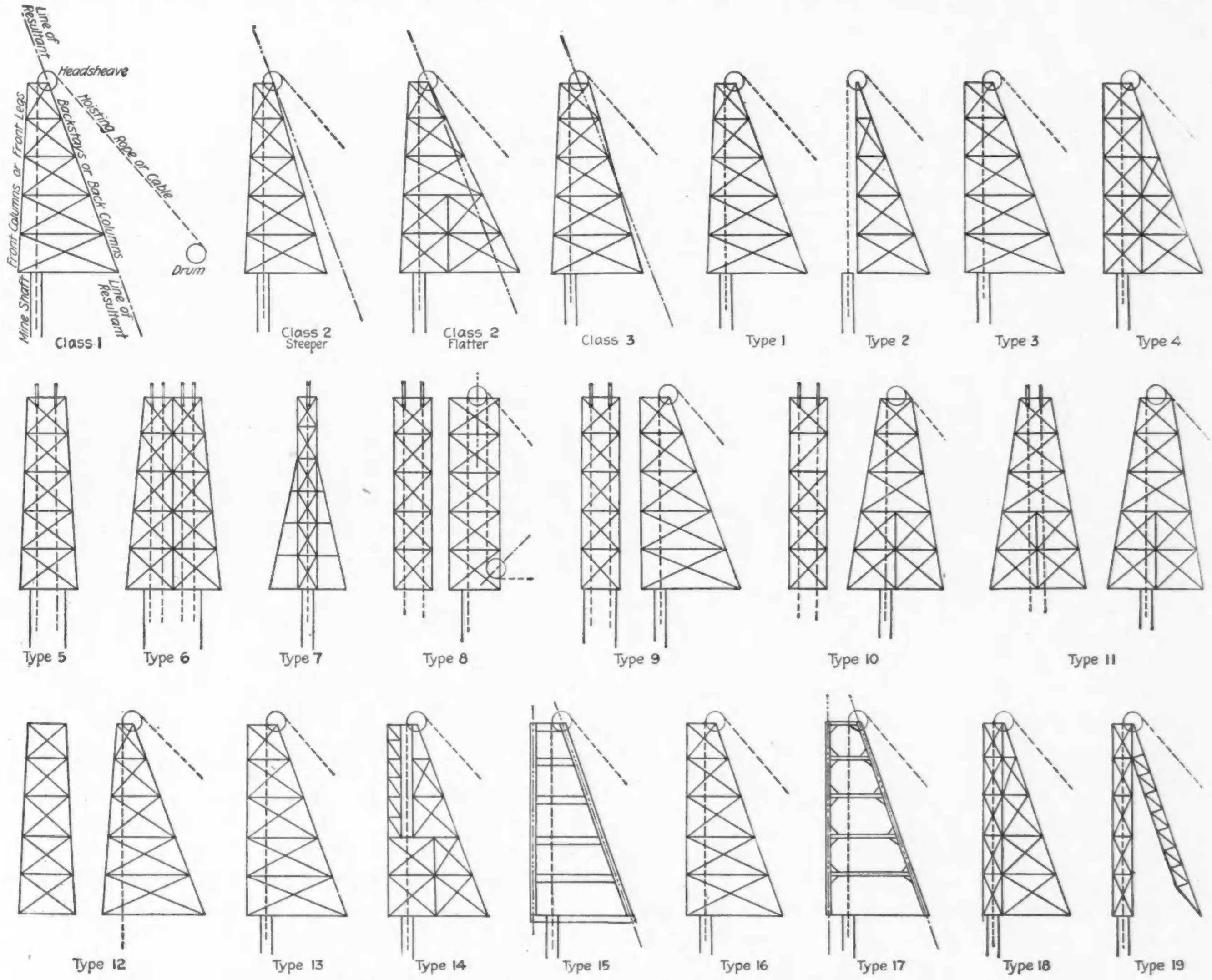
The Design of Headframes

By Floyd L. Burr*

SYNOPSIS—The various types and classes of headframes used under varying conditions, and constructed of different materials have different advantages. Each class and type is discussed in relation to its advantages, and the use of various materials is described. More frequent use of concrete headframes is urged, and the construction of one now being erected is described and illustrated.

When a new shaft is to be equipped with a headframe or an old one is to be so refitted, there are usually many matters to be considered before the purely technical structural design can be taken up. If these matters be neglected as far as possible, an unsatisfactory design is almost certain to result, particularly in case of a headframe and surrounding plant of a permanent nature. Even with the most careful study of present conditions and thoughtful allowance for possible future expansion and changes in local methods, some development in general economic or commercial matters or in the requirements in some associated industry may soon happen to render a well-nigh perfect layout decidedly imperfect from the new standpoint. Of course this phase of the effects of progress belongs in common to all manner of construction from the small dwelling house to the huge superdreadnought.

Each headframe, and each surface plant constitutes a special problem and for good results must be so studied. To be sure, such study will soon bring out points of resemblance or even of exact duplication, as well as of complete antitheses with other propositions. On account of this special nature of each proposition, the idea of standardized headframes does not make a strong appeal. It is, however, not intended to condemn standardization of details of construction such as shape or in some degree even size of structural members or the development of distinct types. The diversified conditions that render each case a special one are such as shaft dimensions and layout of the compartments, the natural and man-made topography and the particular requirements of production. Thus, sometimes the hoisting machinery has been placed or must be placed at a long distance from the



VARIOUS TYPES AND CLASSES OF HEADFRAME DESIGN

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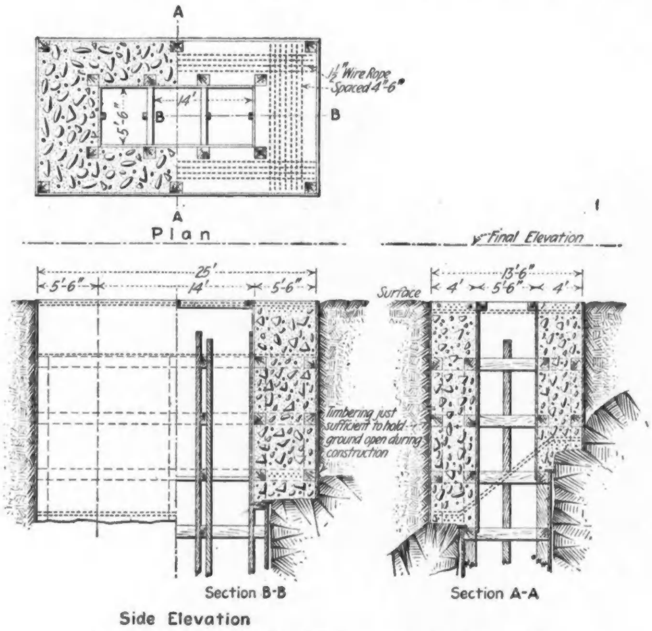
headframe and possibly not in alignment with the axis of the shaft, or the drums may of necessity be extremely close to the shaft; there are different types of hoist involving the use of flat or of round cables; there are all sorts of foundation conditions; there may or may not be occasion for the incorporation into the headframe structure of screens, chutes, storage bins, crushing machinery support or of allowance to provide for such future installation; speed of construction and earliest date of completion may be arbitrarily imposed by commercial conditions or by the arbitrary authority of some higher and perhaps distant official; there is the permanent or temporary or undeveloped nature of the ore deposits of the shaft itself; and there are an almost infinite number of other conditions that may have a bearing upon the design of a particular headframe or indeed upon any part of the whole plant. Some of the more concrete and definite conditions are layout of shaft, position of hoist, height at which the ore must be discharged by the hoist, the method of hoisting, as by car or skip, and the foundation conditions.

The Vertical-Shaft Headframe

Considering a simple vertical-shaft headframe, defined as one that does nothing else than support headsheaves and guides and resist hoisting strains, the position and requisite strength of the principal structural members will be determined largely by the direction and position and intensity of the resultant or series of resultants of the maximum strains in the hoisting cable or cables. Now the resultant or resultants may perhaps be under control in their direction and position. Probably in most cases the position of hoisting drums is already fixed by existing location or by general considerations, and the usual way is to let the cables lead directly in a straight, inclined line from drum to headsheave, thus producing for each cable a line of resultant which is inclined and which intersects the ground at some distance from the shaft. There is another scheme that has been used in the past, but has never to my knowledge come into very general use, by which a vertical-angle sheave is installed at the ground level at the side of the headframe and the cable rises vertically from this sheave to the headsheave. By this arrangement there are of course for each cable two resultants to be resisted. One of these is compression and vertical in direction and acts through the center of the headsheave over the shaft or shaft lining, while the other is tensile and inclined and acts through the center of the angle sheave at a point close to the ground, where it is generally comparatively easy to provide for it.

From the standpoint of the designer of the headframe, this arrangement of cables and sheaves and load resultants possesses certain advantages for many situations. It is easier and more simple to construct all columns vertical than inclined, and with this arrangement it is possible to do that. It has been used sometimes with the old pyramidal wooden headframe. There are many cases where it would not be applicable on account of the vertical portion of the cables interfering with some other features of the headframe, like skip dump, cage landing, trestle track, screens, bins and crusher or even some essential member of the structure itself. The cable from drum to angle-sheave would ordinarily be at comparatively little distance above the ground, where there would be the advantage of simple support of intermediate idler pulleys and great ease in oiling them. In some cases, no doubt, this same proximity to the ground would carry the disadvantage of interference with the most complete use of the surrounding yard. Of course one of the chief objections which would be urged would be adding of the extra sheave, which would mean an original outlay and would have to be maintained and kept oiled. The increased amount of bending to which the cable would be subjected would also appeal to some as being a disadvantage. The choice of this scheme would naturally depend upon the particular conditions, but it might often be found applicable and economical.

The principal determinable stresses that a simple headframe must resist are those due to the tension in the hoisting cables, the effect of wind, and the dead load of the weight of the structure itself with the sheaves and other accessories. But in addition there are undeterminable stresses which may be gradually destructive of the



DETAILS OF CONCRETE SHAFT COLLAR

integrity of the structure and which, at least, may give a disagreeable feeling of lack of confidence in its stability. These have to do with vibration and swaying due to the skip entering the dump, the ore falling into a bin or car, the rapid hauling of the car away on a connecting trestle, etc. Generally these visible effects are most noticeable in steel structures, owing to the elastic nature of the material and to the somewhat small amount of mass. Of course it is possible to make the base of the structure so large, to found it so well, and to design its members so massive and so well-braced at short intervals, that even in the case of a high frame of structural steel there will be little vibration apparent. But such construction absorbs lots of money, and the average mining company would be unwilling or unable to go so far beyond minimum requirements. The effect of wind is usually not very serious, but should be kept in mind and investigated. A conservative design for hoisting stresses will usually possess ample strength and stability to withstand the wind without much addition of cross-section to the members. By such a conservative design is meant one by which no member of the structure will be stressed up to its elastic limit by the breaking of any one of the cables of full new strength while the rest of the cables are all working at their maximum normal load and in which due regard has been paid to slenderness ratios of all compression members. It may be necessary sometimes to assume larger cables than will at first be used in order to provide for future changes and expansion.

Factors in the Design of Headframes

In entering upon the design of the structure one of the first questions to come up is as to the size of the base. In the ordinary type of headframe, which involves an inclined line of resultant passing through the head-sheave, the longitudinal position of the backstays or back columns will be fixed by consideration of this line of resultant (not necessarily by coincidence with it, however), and the resulting longitudinal dimension of the base with front columns in a natural position near the shaft will usually be ample for satisfaction of requirements for general stability. But the maximum width would be the corresponding dimension of the shaft or of that portion from which hoisting is to be done, and this would constitute a limit for the top of the structure as well. The width at the bottom may be limited on one or both sides by other structures or topographical features such as railroad tracks or driveways. Foundation conditions may also fix a limit on this width or on the position of the columns, although such limit is more likely to be one for minimum than for maximum width. Between these limits the width might sometimes be chosen to approximate some definite proportion of the height. This ratio would vary with the material of construction and possibly also with the shapes, sizes or types of structural members which the designer might prefer to use or which would be available. For steel structures a width, center to center of columns of one-fifth the height, base of columns to center of sheaves, might be considered a reasonable minimum, but it may be feasible to use a still smaller ratio, say down to one-sixth. The position and direction of the axis of the skip dump will have much to do with vibration of such a narrow frame. Even the hoisting is likely to produce a marked sway at the top of such a structure, but the shock of dumping a load of five or six tons of ore makes a vigorous bump, especially where the axis of the dump is normal to the longitudinal axis of the structure. If the base is made much wider than the necessary width at the top, the columns naturally will be made, to incline transversely. If the width at the bottom is made to just inclose the shaft, the columns will be vertical transversely. From the viewpoint of the fabricator and erector this is the simplest and therefore the most desirable condition, but such a design may be topheavy.

It must be kept in mind that these remarks apply to the simple headframe, while as a matter of fact a large proportion of the headframes are more or less complicated by functions other than the simple support of sheaves. These are so varied in character and combination that any generalization is difficult. Usually these accessories in the way of screens, bins, etc., will require extra space in the lower part of the structure only, and it may often be feasible to consider the simple headframe as the principal structure to which is applied an addition of partial height to serve these special purposes. It may sometimes be possible and wise to design such addition as an independent structure closely adjacent.

As with all engineering structures, the foundation is a very important part of a headframe. The foundation conditions immediately adjacent to the average shaft are very poor, inasmuch as the shaft lining is usually of timber often in more or less advanced stage of decay, the ground is often "made ground" for a considerable depth and contains various timbers and other forms of rubbish, which has been buried in the filling. The greater the depth down to solid rock or hardpan, the worse such conditions are. Any pressure near the shaft opening is liable to make the shaft timbers give way, especially when they decay, and allow complete or incipient caving into the shaft, thus settling any foundation in the immediate vicinity. It is obvious that it would not do to found a heavy permanent headframe on such a base. If the shaft has been lined with concrete from the ledge up a sufficiently long time before to allow of settlement around the lining, the condition is much better, since this lining will constitute a perfect and enduring retaining wall. As more and more shafts are being so improved, this condition may at no very distant date become common. It will often be wise to take up this matter of a concrete collar for the old shaft in connection with the design and construction of a new headframe. This was the procedure at the Curry shaft in 1908. The concrete collar was built up from the rock ledge some 20 to 25 ft. below, and at the ends the concrete wall was made wide enough to give a suitable bearing and anchorage for the front columns of the steel headframe which was designed to fit. The backstays rest upon simple piers built upon the rock surface, which was at a much higher elevation at this point than at the shaft. This arrangement produced a headframe 20 ft. in width, center to center of columns. The headsheaves are at an elevation of 87 ft. The headframe belongs to class 3, types 2, 3, 5, 9, 14, 16 and 18, according to classification given below and as shown in the illustration. In case of new shafts all these conditions can be controlled, but a large portion of headframe propositions are for old shafts. Where the rock is not buried more than 12 ft. or so, simple concrete piers—solid or cellular—can readily be built up from the ledge, and of course such concrete piers could be built up from greater depth if the expense were warranted.

Different Methods of Concrete Collar Support

Probably concrete piles could in some cases be driven in groups to furnish a support reaching down to or nearly to the ledge. But most mines would not be equipped for driving piles, and it might be impracticable to rig up for a very few piles. There are some cases where the "front legs" may be made to rest upon a concrete collar and lining built with especial reference to such use. There are other cases where the columns of a wide structure may rest upon ordinary piers with necessary bearing area of base or spread footing. But where a narrow structure is desired, it will often be best to use longitudinal or transverse girders to span the weak area and carry the column loads to piers or footings on good foundation ground. As has been remarked before, every shaft is a problem in itself. There may even be cases where a thin concrete lining already exists where a raft or concrete-slab foundation furnishing a common support for all the columns might be the best solution. Various combinations of these forms of foundation may be applicable to particular cases.

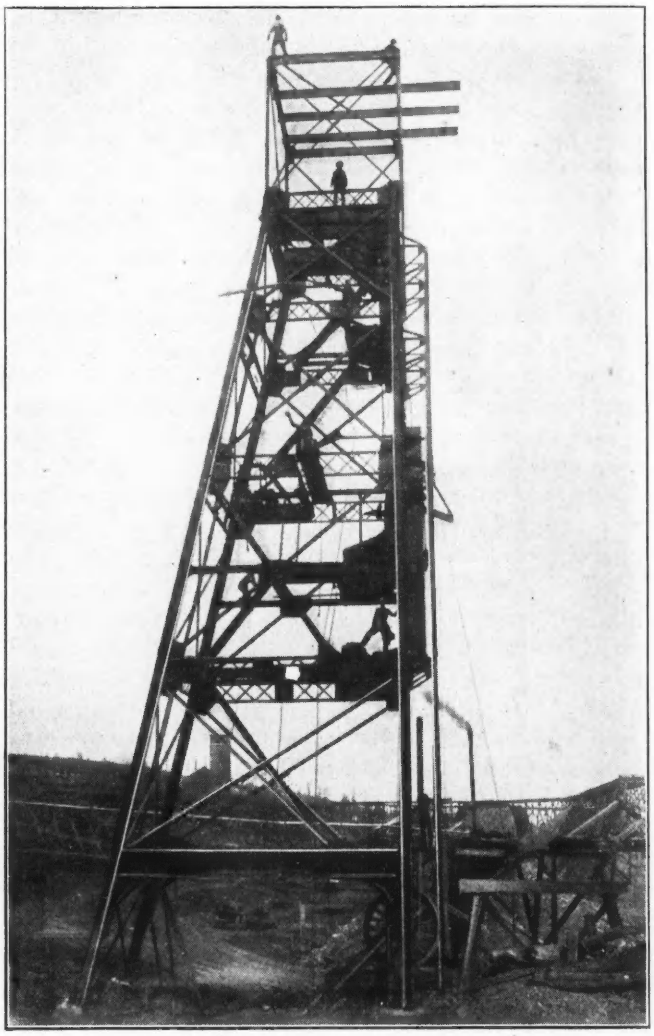
These matters of choice of foundation type are of course bound up with the design of the superstructure both as to its form and material. There are three principal structural materials that have been used for headframes—wood, steel and reinforced concrete. Wood has been used since mining began and has done good service in its way. Structural steel has been largely used for the last 20 years or more at new shafts and for replacements at old shafts. It possesses attractive properties of permanence not possessed by wood. Reinforced concrete has been little used so far, but it seems to possess qualities that make it superior to both steel and wood for many situations.

For small temporary headframes, timber will always have a place. It can undoubtedly be framed and erected, added to and remodeled with greater facility than other materials, and in many cases will serve the purpose perfectly while it lasts. But a structure of wood has two fatal defects which threaten its sudden or gradual demise—combustibility and liability to decay. Of course, selection of quality and species and chemical treatment may accomplish much toward the elimination of decay, but even then the item of combustibility remains. It costs money to obtain the best kind and quality of timber and apply a thorough chemical treatment, and generally that money might better be expended toward the higher cost of structural steel or reinforced concrete. While steel is not combustible, it can not withstand intense heat without fatal expansion and distortion. Now the ordinary shaft is wood-lined and subject to fire. Anyone who has seen the flames shooting skyward from a burning mine shaft can imagine what would become of a wooden or steel headframe. Concrete would probably withstand such a fire for a time without much injury. Set over a modern concrete-lined shaft, a steel structure is fireproof unless adjacent to or closely surrounded by large combustible structures. A wooden structure over a wooden shaft may even become the cause of a disastrous shaft or mine fire by becoming itself ignited from some surface source and dropping burning timbers into the shaft. So a wood headframe would be a safer proposition over a concrete shaft, but it is almost inconceivable that a permanent concrete shaft would be equipped with a headframe of such a temporary nature as wood. A steel or a concrete structure over a concrete shaft should be steel and concrete all through, so that there is practically nothing combustible in the structure. In such a case there would seem to be no occasion whatever for insurance.

Virtues of the Steel Headframe

From the viewpoint of the designer of structural details there is no question as to the superiority of steel over wood, because of the ease of producing in steel reliable, amply strong joints with economy of material. The uniform cross-section of steel shapes, the practically limitless obtainable lengths and great diversity of sizes and forms of rolled sections, the homogeneous nature and uniform quality of steel make a vivid contrast with the often variable dimensions of sawed timber, the limits in length easily obtainable, the low compressive values across the grain and general lack of reliability and uniformity in quality of wood.

Contrasted with reinforced concrete, steel possesses some points of superiority as well as some points of inferiority. Thus concrete is not thoroughly homogeneous nor is its quality so uniform and reliable. It is brittle to some extent; and for corresponding strength it is heavy. On account of these and other considerations concrete work cannot be designed with the same definiteness and small and uniform factor of safety as can steelwork. Concrete is manufactured on the ground, while steel is made and fabricated in a factory or shop and conditions for uniform and reliable work are likely to be better in the factory. Given the fabricated steel at the site, erection can probably be accomplished quicker than with concrete, but starting with only the general layout drawing completed, one could complete the plans and



CURRY STEEL HEADFRAME UNDER CONSTRUCTION IN 1909. IT IS OF CLASS 3, TYPES 2, 3, 5, 9, 14, 16 AND 18

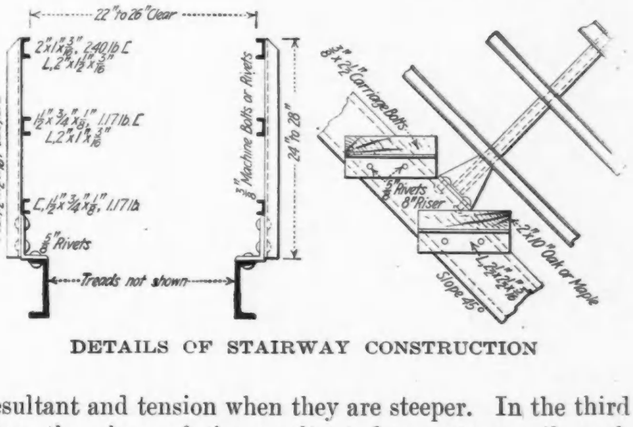
build in reinforced concrete in less time than would be required for detailed drawings and fabrication of steel. Reinforcement bars can generally be obtained for immediate delivery, and one may choose from many styles and manufacturers. Also one may generally find around the mine a considerable quantity of scrap steel that will work in as reinforcement. Form lumber can also usually be obtained with ease, so that before preliminary work, like excavation, is completed, materials for formwork are ready. From the natures of the two forms of construction, it is much simpler to make changes in concrete designs during the progress of the work than it is in steel designs. Likewise, one may easily work around temporary or permanent obstructions in concrete where in steelwork such would be almost impossible, and even where feasible would generally involve much effort, expense and delay. In comparing steel with concrete, the matter of painting should not be forgotten, for a steel structure is likely to need repainting after four or five years and this painting is likely to be a rather expensive and bothersome matter on account of accumulations of dirt, grease and rust and interference with the regular use of the headframe. It should also be noted that the riveted joints and other totally inaccessible surfaces may be just the ones that have the greatest need of the paint. To do away with the occasion for this paint, it may even be worth while to consider the coating of all principal members with cement "gunite." Maintenance and repairs are expensive and aggravating, and I am inclined to paraphrase and say "Millions for first cost, but not a cent for patching up."

The Concrete Headframe

There are certain properties of towerlike structures of concrete which should receive prominence in such a discussion. Concrete is heavy, and when built upon a good foundation a structure may, like a masonry stack, be very stable and yet be very narrow in proportion to its height. This property may be of greatest importance in the design where ground area is limited. Another point about concrete is its adaptability to the construction of cantilever work both for light loads and heavy ones. The great excess of stability of the structure proper is generally adequate to easily provide for these heavy eccentric loads, but if it should not be thus sufficient, one may readily counteract this eccentricity in large part by adding dead load to the opposite side of the structure. These things are possible because the dead load or weight of the structure is bound to be large as compared to the live loads. Of course this same matter of great weight makes it imperative that the structure be extremely well founded so as to avoid settlement or at least uneven settlement. Another feature of great importance in favor of concrete is the absence of diagonal braces. This leaves the many rectangular panels between girts entirely free and open for the passage of the hoisting cables or for the entrance of belts, tramcars, etc. At the ground this allows of the greatest possible freedom for traffic to and about the shaft. Layouts in structural steel are sometimes quite bothersome on account of the common use of diagonal braces.

In the case of some simple types of headframes in which the base area is kept down to a minimum, the Kimberly-skip dump extends outside of the plane of the outside columns and girts and the upturned skipbox travels even above the dumping position in the occasional overwind, half in and half out of the structure proper. This means that there must be in the side of the structure a long vertical slot or opening corresponding with the width of the skip. See type 14 in the drawing. The necessity of providing such openings is a nuisance to the designer, as the opening cuts through the natural position of such essential members as diagonal braces and girts, thus interfering with the integrity of the structure. Concrete seems to lend itself to the solution of such a problem better than wood or steel.

There are many types of headframe structures in wood and in steel, while of course concrete types are as yet undeveloped. Naturally the general form of concrete structures will, at least for some time, resemble that of steel or wooden structures, the difference now existing or later to develop being based on the great weight of concrete, the absence of diagonal braces, and the ease of executing almost any shape or size in concrete. All headframes fall naturally into three general classes. In the first class a pair or series of back stays is located approximately in the plane of the resultant or resultants of hoisting cable stresses, enabling these stresses to find their way in a single and most direct path to the foundation. There are many modifications and it is often impossible to design with strict adherence to this idea. In this class major hoisting strains reach the foundation only as compression through the back columns. Of course the plane of the back columns passes through the center of the headsheaves. In the second class the conditions are the same as in the first, except that the plane of the backstays does not coincide with the plane of the resultant and, consequently, the hoisting produces not only compression in the backstays, but also compression or tension in the front columns, such stress being compression when the back columns are flatter than the



DETAILS OF STAIRWAY CONSTRUCTION

resultant and tension when they are steeper. In the third class the plane of the resultant does not pass through the center of the headsheave and the distribution of stresses between front and back columns is indeterminate. Naturally one would prefer the first class on account of the simplicity of analysis, and certainty of distribution of principal stress, while the third class would be least desirable from the designer's point of view.

To enable one to make easy reference to various features of design of headframes, they may be classified according to the following described types. Any individual headframe will be then a combination of these types. Consideration of the relation of the headframe to the shaft opening, it serves, gives types 1 and 2; the number of columns, types 3, 4, 5, 6 and 7; the inclination of the columns, types 8, 9, 10, 11 and 12; the relation of the skip dump to the structure, types 13 and 14; and forms of bracing, types 15, 16, 17, 18 and 19, as shown.

Type 1: The headframe proper incloses the shaft.

Type 2: The headframe proper is on one side of shaft, and cageway and skipway guides are supported by brackets cantilevering out from the main structure.

Type 3: There are no intermediate columns situated longitudinally between back columns and front columns.

Type 4: There are such intermediate columns.

Type 5: There are no intermediate columns situated transversely between the two outside back columns or between the two outside front columns.

Type 6: There are such intermediate columns.

Type 7: There are side stays, or brace columns, leaning against the front columns to increase the lateral stability of an otherwise narrow structure.

Type 8: All columns are vertical.

Type 9: Only the back columns are battered or inclined and they only in the longitudinal plane.

Type 10: Both back columns and front columns are battered, but in the longitudinal plane only.

Type 11: All columns are battered equally and in both planes so as to form a truncated regular pyramid with vertical axis.

Type 12: All columns are battered and in both planes, but the longitudinal inclination of the back columns is flatter than that of the front columns.

Type 13: The path of the skip lies entirely within the structure as bounded by the planes of the exterior columns and girts.

Type 14: The path of the skip after entering the dump does not lie entirely within the structure, but intersects one of the outside planes of columns and girts.

Type 15: There are no braces in these exterior planes of columns and girts, thus leaving unobstructed open bays.

Type 16: The columns and girts are braced with diagonal brace members.

Type 17: This bracing is accomplished by use of knee braces.

Type 18: The backstay columns are connected to the front or intermediate columns by the use of girts and braces.

Type 19: The backstay columns are connected only at their upper extremities to the rest of the structure, being thus independent of girts and bracing and having no intermediate support throughout their length.

Steel or Concrete in Headframe Design

Steel headframes have been designed and built along the lines of all these types with the possible exception of type 15, which is a possible but not a likely arrangement for steel. Probably the majority of steel headframes conform to types 1, 3, 5, 12, 13, 16 and 18, but types 2, 4 and 19 are common. Type 9 deserves attention on account of its inherent simplicity, while type 14 is often a necessary consequence of an effort toward minimum dimensions and minimum cost. Type 7 has some merit for special cases. For home-built steel headframes it would seem that type 9 would be particularly attractive, as the batterwork is therein reduced nearly to a minimum, which makes for simplicity; and simplicity is important at the average mine shop, which is seldom equipped for elaborate work.

Type 11 has been the old standby for wooden construction at least in this region. Type 17 has generally been used in connection with 11, but 16 is often put in. Other types are not uncommon, as for instance 9, 12 and 19.

Reinforced concrete could be used in almost any type, but to keep down the cost and give full sway to the advantageous properties of concrete, one should design it in types 1, 2, 3, 5, 9, 14 and 18, where these types are applicable. If conditions are such that type 8 would be suitable, one could then attain great simplicity and consequent low cost, for vertical columns are much to be preferred to battered ones from the construction standpoint both as regards construction of forms and pouring of concrete.

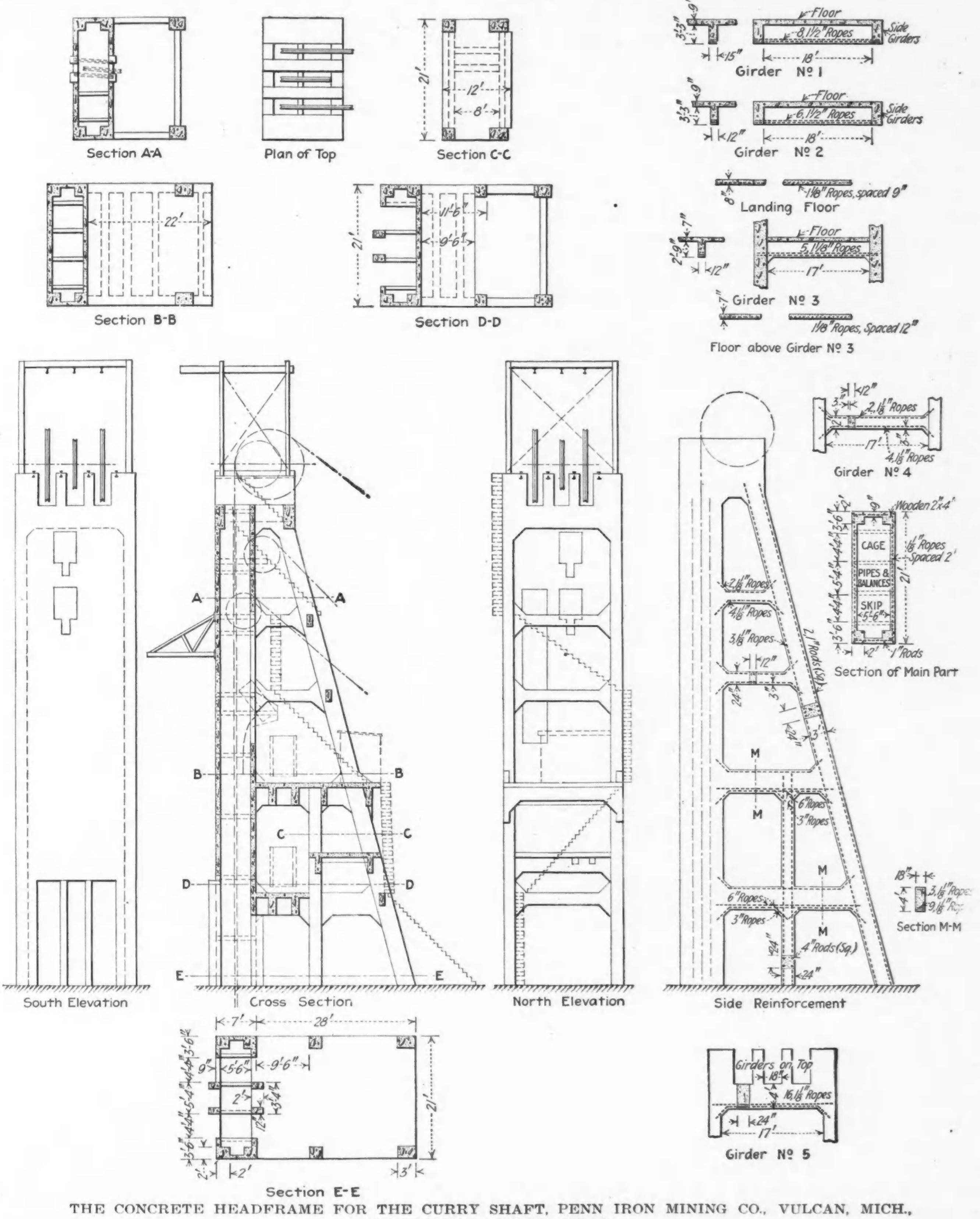
Generally, a steel headframe will be designed, detailed, fabricated and perhaps erected by some steel-construction company, but the mine engineer will have to make specifications as to the requirements. If the conditions are rather complicated, the elaborateness of the required specifications practically force the engineer to make a general design, since matters of clearances, working spaces and other items may have to be dealt with in connection with sizes, and form of members and other structural matters. It might be exceedingly difficult, in dealing with a distant structural designer, unfamiliar with problems of mining, to convey to him, in writing or drawing, the full idea of what are essential unbending requirements and what ones could accommodate themselves somewhat to his convenience in structural designing. To obtain an economical design, endless correspondence may be needed as the "ifs" and the "buts" begin to develop in the progress of the designing. It may therefore be preferable under some circumstances to have the design made quite complete by the mine engineer and then either send it to a steel-fabricating company or purchase the steel and work it up at the mine. In case it were to be built at the mine, it would be best to design it with that intention, keeping in mind all through such a choice of sections and details as could best be handled at the mine shops. Usually in such a case one would use complete rolled sections wherever possible in preference to built-up sections, in order to reduce shopwork to a minimum. Indeed it is a question whether steel companies could not profitably use fewer riveted sections.

It would be entirely impracticable to attempt to enumerate all the details of utility, convenience and safety that should be kept in mind in laying out a headframe, as they will vary with kinds of mining, methods of handling ore and supplies, size of proposition, etc. But there are a few points that may be mentioned. There must be means of handling everything that may need handling in or around the structure. This will include cages, cars, skips, sheaves, all kinds of pipes, pumps, timber, and miscellaneous supplies. Almost always one or more swinging jib cranes will be useful. These should be placed to best suit their purposes, and some of them will naturally be high up in the structure. They would usually be attached to one of the principal columns. In case of high structures a trolley crane or some other form of crane will be needed at the top to use for handling headsheaves.

In case of very low structures the need of such a crane is questionable, since it is easy to use a gin pole resting either on the ground or on the top of the structure, and the occasion for handling sheaves is not likely to arise more than once in three or four years. I have in mind a high headframe provided with a sheave crane which was erected and equipped in 1910 with new 12-ft. sheaves. These sheaves have been in constant use ever since, and there has never been any need of the crane since the original installation of the sheaves. Sometime, however, they will need replacement, and then it would be decidedly inconvenient to handle them 100 ft. up in the air without the crane.

Besides these regular cranes, there should be placed around through the structure at likely points, eye-bolts, clevises or other simple provision for hanging tackle whenever or wherever the need may arise. It is also easy to provide cantilever beams reaching out 5 or 6 ft. over points on the ground or wherever there may be occasion to unload wagons or handle materials. Such provision costs little money and may be very useful.

this specification. The surface of the ground should be sloped slightly away from the shaft and paved with concrete or macadam in order to do away with the usual



THE CONCRETE HEADFRAME FOR THE CURRY SHAFT, PENN IRON MINING CO., VULCAN, MICH., NOW UNDER CONSTRUCTION

Especial attention is needed to make sure of the easy approach close to the shaft compartments of all teaming traffic, conveying supplies to or from the shaft. Some of the diagonal bracing at the ground will generally have to be omitted and portal bracing substituted to meet

mud hole and to facilitate teaming and handling materials in bad weather. There must be ample clear headroom opening into the shaft compartments to allow of taking in long or high materials such as timber, pipe and rails.

It seems to be wise to inclose or box in the skipway from surface to the dump to avoid danger on the ground from the falling of occasional loose chunks of ore from the bale or edge of the skipbox during its upward flight. Suitable doors or gates should be provided at the entrance to the shaft compartments to prevent accidental or mischievous approach. Such inclosures and doors may well be high—even as much as 10 ft. in some situations. In some places hand-railings may be needed to guide one away from dangerous tracks or other points of danger, or at least to arrest progress and cause one to "stop, look and listen." There must be a system of substantial and safeguarded ladders and landings reaching up to the headsheaves and to other points where attendance is needed. Hand-railings should inclose the sheave wheels and protect the outside edge of all landings and floors.

Important Factors in Headframe Construction

In building a permanent and fire-resistant structure such as a steel headframe, it seems only logical to make the accessory portions also of time-enduring and fire-resistant materials. In accord with this idea, it is my practice to build all stairways of steel-channel stringers and I am not really satisfied with the 2 x 10-in. hardwood treads. These stairways are built with only sufficient width for one person, with a clear space between railings of only 24 to 26 in. All floors throughout the structure, including the stairway landings, are of reinforced concrete, generally 3 in. thick. These are usually reinforced with some form of expanded metal and cast on a wooden form, but there have also been used similar slabs with old steel rope for reinforcement and other slabs cast on "multiplex plates" and on "hy-rib" or "self-centering." The latter require plastering on the underside, and this plastering may be bothersome on account of comparative inaccessibility at some points. Such floors are usually made of wood and certainly cost less than the concrete slabs, but wood will require occasional renewal and it does not foster the feeling of security which goes with concrete construction. Grease and oil may accumulate, and such wooden floors may actually get afire, though such occurrences are rare.

Hand-railings may be of wrought-iron pipe 1 in. or more in diameter, but one built of small steel sections, either bolted or riveted together, seems preferable on account of greater ease of construction and lower cost. Some railings must be designed so as to be capable of ready removal to clear the path for removal of sheaves, etc. Steel railings may be built of angles, both for posts and rails, but the preferred type is constructed of "small steel channels" using for posts, $1\frac{1}{2} \times 1\frac{1}{2} \times \frac{3}{16}$ in. x 2.63 lb., and for rails, $2 \times 1 \times \frac{3}{16}$ in. x 2.40 lb. or $1\frac{1}{2} \times \frac{3}{4} \times \frac{1}{8}$ in. x 1.17 lb.; according to post spacing and required ruggedness. These steel railings may not be so attractive as the more conventional pipe railing.

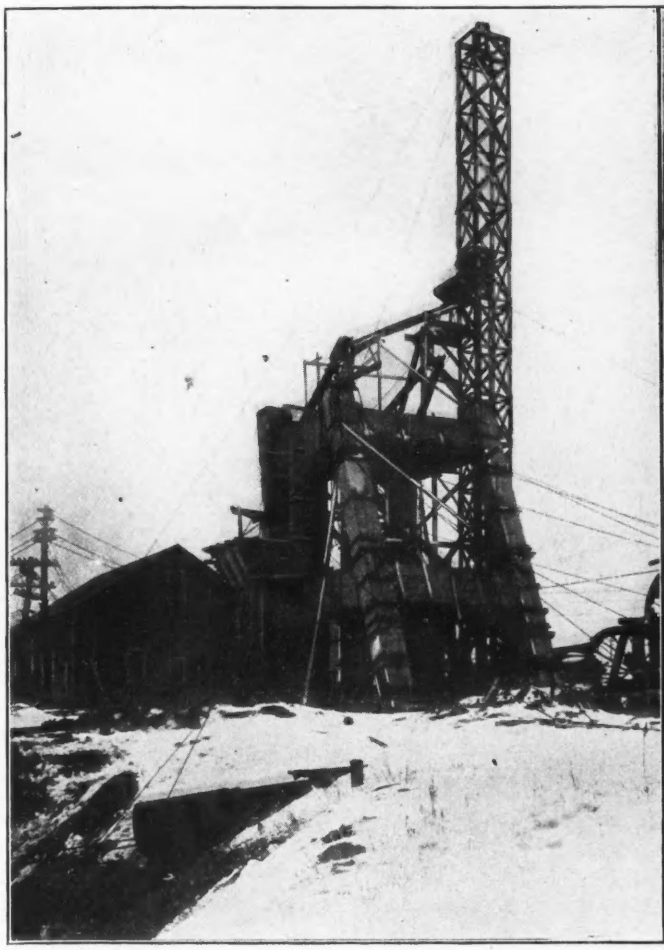
Any small buildings in the structure, such as landers' houses and toolhouses should be of fire-resistant construction. For buildings that are not to be kept heated, a satisfactory construction is corrugated sheeting on structural steel frame or even on a wooden frame. Where heating is necessary, a hollow-wall construction of cement stucco on steel frame or some other form of cement work will be in order. Windows for such buildings are preferably framed out in steel and provided with substantial steel sash glazed with wire-mesh glass.

As may be noted throughout this article, I believe that reinforced concrete should have consideration as a material for headframes, and the general arguments in favor of concrete are given added emphasis at the present time by the scarcity and high prices for steel and skilled labor due to wartime conditions. Concrete headframes have not in any sense been standardized yet and, in fact, very few have been built. But there should be no new principles involved and no particular reason for building differently from other towers which are subjected to lateral strains.

A New Concrete Headframe

The Penn Iron Mining Co. is at present constructing a concrete headframe under my direction, and while it is not intended to present a complete description, there may be some interest in a few remarks as to various details that are being used. The structure will be moderate in height, only 60 ft. to the center of the 10-ft. headsheaves. It replaces a decayed, wooden headframe over old "C" shaft. It covers a skipway, a cageway and a counterbalance way and belongs to class 1, being designed according to types 1, 4, 5, 9, 14, 15 and 18. There are four principal columns, all 30 in. square, connected by girts 15 in. wide by 48 in. deep. These girts are at elevations 16, 35 and 54 ft. Above the 54 ft. are sheave girders 36 in. deep surmounted by piers of the same height, which will support the sheave bearings. The girts were made 4 ft. deep in order to give great rigidity to the structure and 15 in. wide in order to allow room for working inside the forms and also to make lots of mass to retain heat, the structure being erected in the severe winter weather of the upper peninsula of Michigan. The columns were made large for the same reasons, though long, unsupported lengths in the case of the front columns would have required large cross-section for them in any case. Eight-inch floors at elevations 16 ft. and 35 ft. cover that portion of the area not over the shaft and serve as horizontal stiffening diaphragms. The dump and a pocket of 6 tons' capacity cantilever out from the south side. Reinforcement is entirely of old discarded steel hoisting cables from 1 in. to $1\frac{1}{4}$ in. diameter and, of various types and qualities. The girts are reinforced top and bottom to withstand bending moments from lateral forces such as wind and the effect of skip dumping. The whole structure is set upon and integral with a pair of 30 x 78-in. longitudinal concrete girders spanning the disturbed ground adjacent to the shaft. The reinforcement ropes of the columns start at the bottom of the foundation girders and reach continuously to the top of the structure without splices. It is to be noted that the columns are not truly "reinforced-concrete columns," since the flexible steel cables are not capable of aiding the concrete in its resistance to compression. Their function is to provide the tensile element to enable the columns to exert beam-like resistance to all lateral or bending loads. For forms there is used 2 x 6-in. tongue-and-groove hemlock, put together with 4-in. "economy double-headed nails." The reinforcement is attached to the inside of the forms by being wired at intervals to pairs of 3-in. double-headed nails driven into the form so as to hold the cable about 2 in. clear of the form. These double-headed nails serve the purpose well and after removal of forms, protruding points are nipped off flush with the surface.

The use of old steel rope is based primarily upon the low cost of a scrap material and upon the usual ease with which it can be obtained from neighboring mines when there is not a sufficient supply accumulated by discard. When straight and smooth and clean, it handles very nicely, and on account of its flexibility it can be placed in confined positions where a rigid rod would necessitate special provision. It should also be mentioned that it possesses the advantage of being obtainable and usable in any length. On the other hand, one never knows with any approach to precision just what its safe strength is and must use it at small tensile value always. He must "size it up" and establish in his mind a basis of evaluation, and he must keep on inspecting because one reel will not duplicate another and even on the same reel there may be all degrees of decay. It may require a scrubbing with steel-wire brushes for the removal of excessive amounts of rust or "rope dope."



"C" SHAFT CONCRETE HEADFRAME UNDER CONSTRUCTION

COLUMN-FORM YOKES OF SCRAP BOILER FLUES AND SCRAP ½-IN. STEEL ROPE

The column forms are yoked with a peculiar type of home-made yoke, a quantity of which was prepared some years ago for a similar job. These yokes consist of two pieces of scrapped 4-in. boiler flue about 44 in. long and two loops of old ½-in. steel rope, these loops being formed by joining the ends of proper lengths of rope by the use of "cold shuts." The loops are placed over the ends of the flues and twisted up by a short bar, untwisting being prevented by the insertion of a short rod or large spike.

A steel stairway will extend up one side of the structure supported by a few old steel rails "sticking out" of the concrete. The case of thus bracketing or cantilevering out from a concrete structure with structural-steel sections should be noted. Such a practice is most primitive but exceedingly satisfactory.

Every part of the structure is intended to be simple and substantial, and much of it will incidentally be far stronger than is strictly necessary. Practical considerations often make it seem wise to waste a considerable volume of concrete in order to save time or labor in formwork. The actual concrete is the cheapest part, the preparation of forms absorbing the large portion of the cost. There is opportunity for much development along lines of economical building and handling of forms as well as the making and placing of the concrete. There may be cases where sectional-steel forms would be applicable, but usually this would occur only where a whole plant is to be constructed of concrete and the forms could be used repeatedly. The special nature of headframes does not make steel forms seem very attractive.

Manganese in Nicaragua

Rapid progress is being made in the development of the manganese mine at Playa Real, Nicaragua, according to the U. S. Department of Commerce. The mine appears to have a large quantity of ore lying more like a blanket than as a vein formation. The principal mining method is that of stripping the surface. Large wharves that will permit the loading of the ore directly by overhead trolleys are contemplated.

World's Two Largest Steam Hoists

BY PAUL A. BANCEL*

The growth in the size of mines and the increase in the cost of fuel or in many localities the purchase of hydro-electric power has resulted in recent years in the development of three types of hoists, each of which has a particular field of application. The three types are the compound-condensing steam, the electric and the air-driven hoists.

The first of these is the twin-tandem or duplex-compound condensing steam hoist. Practically every modern steam hoist of any size installed abroad is of the condensing type, and the tendency among American mining men is to specify condensing operation whenever steam is used. It is sometimes incorrectly assumed that a condensing hoist is not a paying proposition except for a mine of great depth. It is argued that the saving due to the vacuum is obtained only when the hoist is running at normal speed and that during the period of acceleration the heavy steam consumption overbalances the saving during the remainder of the trip. This might be true if the condenser were not properly designed, so as to condense the large volume of exhaust at the start of the trip. Actually, with a properly designed condenser, the saving of about 50% due to the employment of vacuum is just as great during the period of acceleration as when running, and the proportionate saving by operating condensing is greater for shallow depth than for a great depth. Of course the average steam consumption for the entire trip is greater in the case of a hoist for shallow depth than for great depth, just as with noncondensing.

The problem of proper cylinder and condenser proportions, of valve-gear design and provisions to insure full starting torque and immediate acceleration, have been successfully worked out. Up to a few years ago, all these units were of the twin-tandem design, but recently a departure has been made in the case of some very large installations.

In the case of the Homestake Mining Co., the hoist was built duplex with inclined cylinders, 28 and 52 by 42 in., driving two shafts linked by side rods. The cylinders meet at the apex of the triangle as in Fig. 1. This hoist has been put into regular service recently and is the largest compound hoist in operation. It will be surpassed in size, however, by a new unit now being designed by the Nordberg Manufacturing Co. This will be not only the largest compound-condensing hoist yet constructed, but also the largest hoist in the world as regards capacity and depth, surpassing in size the great four-cylinder noncondensing hoists for the Tamarack Mining Co. built 16 years ago. It will have two high-pressure and two low-pressure cylinders 32 and 60 in. diameter, with a common stroke of 66 in., arranged at an angle of 45 deg., with the frames meeting at the apex of the base and driving a common crankpin, as shown in Fig. 2. The drum will be of the cylindro-conical type, 16 ft. small diameter and 30 ft. large diameter. This is 5 ft. greater than the diameter of the Tamarack drums. The hoist is designed to handle 10 tons of ore, from an ultimate depth of 10,000 ft., the shaft being inclined at an angle of between 35 and 54 degrees.

Where electric power, usually from hydro-electric generation, is available, two types of hoisting equipment are in use—the motor-driven and the air-driven. Rapid strides have been made within a comparatively short time in the use and development of motor-driven hoists, either with induction motor or direct-current motor drive and motor-generator set. In some instances electric hoists are arranged for full automatic operation, as in the case of the units at the Inspiration Consolidated Copper Co. which were described in a recent paper[1] before the American Institute of Mining Engineers.

One of the drawbacks to the use of the motor-driven hoist is the fact that the motor becomes large and expensive owing to the large starting current and the low speed, particularly in the larger sizes, where a first-motion hoist only can be considered. With great depth of shaft and load, large rope must be used and large drum diameter, which means slow speed and a large and expensive motor. Furthermore, the transformation of energy from the line through the motor-generator set, hoist motor, hoist and shaft, is inefficient. The air hoist, like the steam hoist, is capable of exerting great starting power, and is more efficient than the motor-driven hoist in large installations because the power otherwise wasted in braking is returned to the air-storage system, the engine working as a compressor when retarding. The growing custom of filling worked-out portions of the mine with rock calls for extensive lowering. Another feature

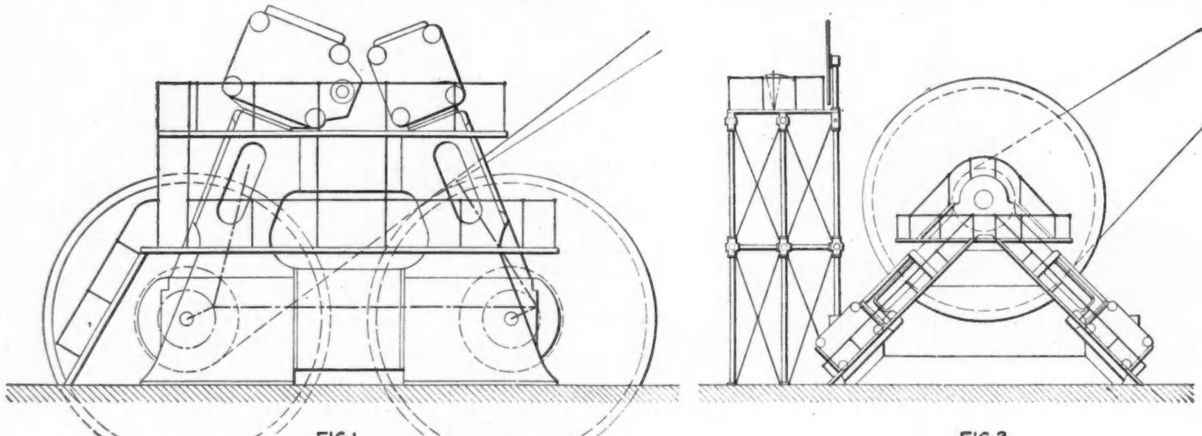


FIG. 1 FIG. 2

TWO TYPES OF INCLINE COMPOUND CONDENSING HOISTS

*Tribune Bldg., New York City.

[1]"Automatic Operation of Mine Hoists as Exemplified by the New Electric Hoist for the Inspiration Consolidated Copper Co.," by H. Kenyon Burch and M. A. Whiting, p. 583, Bulletin 111, A. I. M. E., March, 1916.

of considerable weight with many mining men is that in case of an interruption in the electric supply, the air hoists can be operated on the stored air for a considerable period during which the men can be hoisted from the mine.

The majority of air hoists in this country have been installed at the Anaconda mines in Butte, and during the last year four more have been built by the Nordberg Manufacturing Co. for the same concern. This system has been fully described in a comprehensive paper[2] before the American Institute of Mining Engineers. Other air hoists have been built for the North Star mines, Grass Valley, Calif., and the Mond Nickel Co., Sudbury, Ont. and at the present time there are several large mining companies giving serious consideration to the installation of air hoists larger than any that have been built to date.

Preventing Air-Driven Pumps from Freezing

By James Humes*

Three years ago, while sinking the Silver Hill shaft, Park City, Utah, we encountered a large flow of water, which necessitated the use of two large sinking pumps. As the shaft is nearly two miles from the portal of the tunnel and the power plant, compressed air was used to operate the pumps. As the compressors were within 300 ft. of the pump, the air was delivered warm, but nevertheless, the exhaust opening would freeze solid in less than five minutes after the pumps were started, which necessitated a continual stopping and starting of the pump. When the pumps stopped, they generally lost their water, and when they did run, the noise made by the exhausting air was enough to ruin the nerves of a strong man. After a study of the conditions at the bottom of the shaft, I devised the scheme, shown in Fig. 1, of placing a check valve at *B*, and at *A* I placed a T with a gate valve screwed into it. The reason for having the valves placed in this manner was so that the check valve would prevent the water from going back into the air end of the pump and so that when the pump was being started it would be necessary to open the gate valve and let it exhaust into the shaft; otherwise the exhaust air would find its way into the suction end of the pump and prevent a vacuum. Both these ideas, however, proved to be wrong. I visited the shaft again a few days after the scheme had been in operation and found that the pumps were working satisfactorily, but were requiring about 15 lb. more air to operate. The operation was noiseless, however, even the usual water-hammer in the column being absent. I timed the number of strokes the piston was making, then turned the exhaust in the shaft and found there was no difference in speed. I went up to the discharge end of the column and saw that the air was ejecting the water out of the end of the pipe for a distance of 15 ft., so it seemed that our sinking problem was solved.

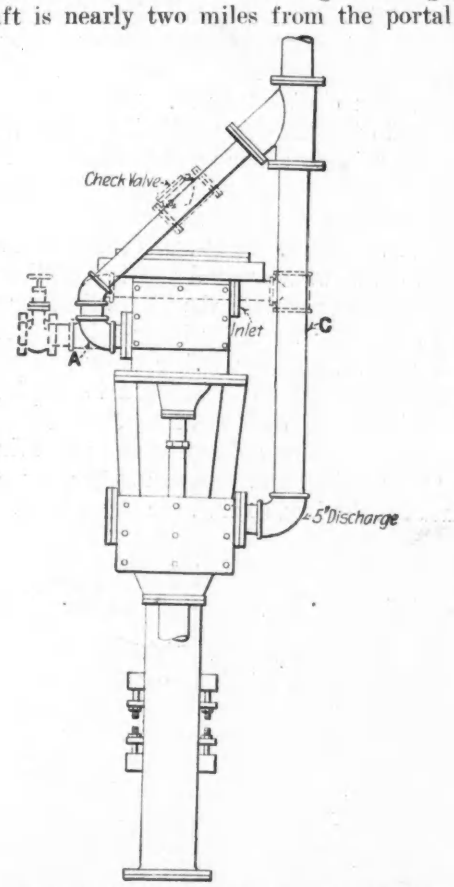


FIG. 1. EXHAUST PIPING FOR SHAFT PUMP

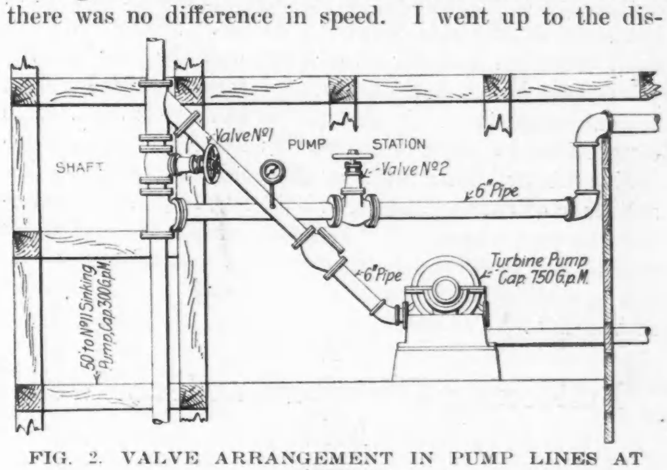


FIG. 2. VALVE ARRANGEMENT IN PUMP LINES AT STATION

Afterward, in repairing these pumps, we found the check valve broken, and from its condition we surmised that it was ineffectual at that point, so we discontinued its use entirely. The valve at *A* was also proved useless. In fact, we did not use any valve between the air cylinder and water column, and in the last year it became necessary to put in greater pumping capacity, and as we didn't have the 6-in. T on hand, we used two 90-deg. elbows at *A*, and carried the exhaust pipe across in front of the air cylinder to a T in the water column at *C*. We have sinking pumps from 80 to 650 gal. capacity, all of which are connected in this manner at present. Recently we cut a watercourse in the bottom of the shaft, and the flow increased so much that it raised the water 150 ft. in the shaft in spite of the fact that five pumps were running. Four of the pumps continued to run, but in the fifth, as long as the submergence was right, the air passed through the pump and brought up the water. We found that one of the reversing valves in this pump had become stuck, and it stopped the slide valve in such a position that it allowed the free passage of the air through the pump and into the water-discharge

[2] "The Compressed Air System of the Anaconda Copper Mining Co., Butte, Mont." By Bruno V. Nordberg, Trans. A. I. M. E., Vol. 46, p. 826.

*Superintendent, Silver King Coalition Mines Co., Park City, Utah.

column. It required six weeks to lower the water down to the pumps, and if they had been connected up in the usual manner, I doubt if we would have got as good results as we did.

At 200-ft. intervals we had a double installation of electric-drive turbine pumps, each having a capacity of 750 gal. per min. Fig. 2 shows such an installation, but this one was made in a hurry before we had reached the usual 200-ft. interval, on account of the sudden inrush of water that threatened to overwhelm our pumps. A peculiar condition existed at this point: When valve No. 2 was closed and the air-driven sinking pump was sending water to the pumping station above, the pressure gage showed but 28 lb. When neither of the pumps was running, the static pressure shown on the gage was 64½ lb., and when the turbine pump was throwing water to the same level as the sinking pump was the gage stood at 74 lb. When these data were taken, the sinking pump was at least 50 ft. below the station shown, and discharged into a wooden tank back of the pump, the piping being so arranged that this pump would throw the water to the next pumping station above.

The end of the pump compartment is shown in Fig. 2, but some of the equipment is omitted. There are three 6-in. columns, two for water and one for air; all the joints have bolted flanges, and each section is 10 ft. long, so that one of them can be replaced in a short time. The connections to the electric pumps are made with long-radius elbows. The tanks at the main pumping station are made of concrete and will hold 50,000 gal. The electric pumps are two- and four-stage, one having a capacity of 750 gal. against a head of 700 ft. and the other 750 gal. against a head of 250 ft., and both give the best of satisfaction.

Bulldozing Chambers at Treadwell Mine

By R. G. Wayland*

During the early years of the Treadwell mine, Douglas Island, Alaska, the lack of capital and the low margin of profit made it necessary to follow the best ore regardless of any regular system of pillars or supports. Timbering or waste filling was, and is, economically impossible. While pillars and levels were previously left in place, they were not situated to the best advantage. As soon as it became apparent that the mining would go to considerable depth, the pillars were left systematically; but although the strength of the ore and hanging wall was unusually great, the weakened condition of the old workings resulted in failure of pillars, except the line-pillars between adjoining properties. As mining operations attained greater depth and support was withdrawn from the weakened pillars near the surface, these pillars and remaining portions of the worked-out levels caved. At the same time large masses of waste from the surface gave way and followed the caved ore, so that the mined-out area was completely filled with caved pillars, caved level bottoms and waste. This material was chiefly ore in the lower portions of the mine and chiefly waste in the upper.

When an attempt was made to recover the caved ore, it was found that chute after chute would be permanently blocked, there being no safe or practical way to bulldoze or block-hole the large masses that came down. To meet this situation the bulldozing chamber was developed.

In Fig. 1 the drift A is one of the original drifts, B is the station for the finger chute, and F is the top of the original chute-raise nearest the foot wall. In cutting out a series of foot-wall bulldozing chambers another drift A', was run in the foot wall, approximately parallel to the wall, and was so placed that the top of the chute-raise D was from 30 to 35 ft. from the edge of the stope. From some convenient point a short manway raise was put up and an intermediate drift C driven and connected with the tops of the chute-raises D. This intermediate drift gave access to the chambers and also afforded ventilation. A large, flat raise was then put up from D to E at such an inclination that the ore would lie at its angle of repose from the center of the chute to the back of the chamber at the edge of the stope. Room was made so that the bulldoze man could walk around the top of the chute-raise.

When the ore runs well, the pieces that are too large to pass the chute can be easily block-holed or bulldozed between E and D, as the moving stream is not deep and large pieces are sure to be brought to the top. The ore below the line X-Y does not move.

Whenever a number of large blocks cause the broken ore to hang up, the bulldoze man can safely go up to E, drill, and blast out the key, so that the chute will again fill. It is good practice to leave some of the finer stoped ore in the bottom of the stope so that when the caved ore is very stubborn a few carloads can be drawn through F, thus causing a movement above, and disturbing any arch of larger masses of ore that may have formed in the throat of the stope raise.

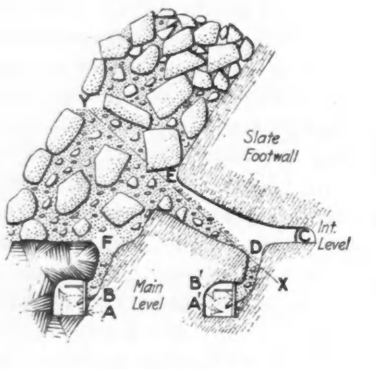


FIG. 1

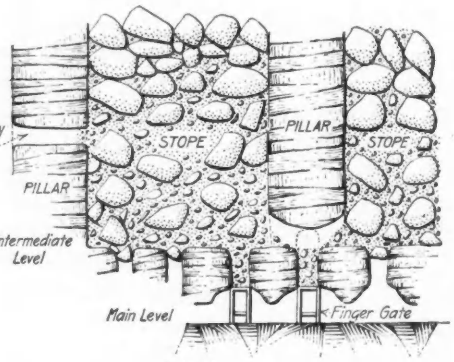


FIG. 2

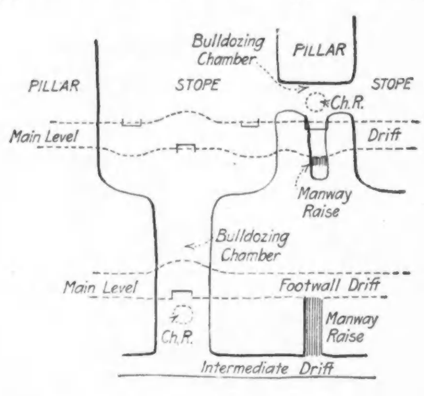


FIG. 3

SECTIONS AND PLAN OF BULLDOZING CHAMBERS IN TREADWELL MINE, ALASKA

*General superintendent, Alaska Treadwell Gold Mining Co., Treadwell, Alaska.

Where the dip is very flat, it is sometimes necessary to have more than one series of bulldozing chambers along the dip from one level to another. Fig. 2 shows one in a pillar between two stopes at some distance from the foot wall. It is a modification of the chamber shown in Fig. 1 and is not so safe or satisfactory, but reaches ore that the foot wall chambers would not touch. Fig. 3 is a plan showing both types of chambers, but placed closer together than in actual practice.

Ordinarily, two men work together in a bulldozing chamber and will handle from 250 to 400 tons in an 8-hour shift, depending on the size of material and the facility with which it runs. A good bulldoze man is a valuable man; he must be quick, agile and resourceful. After working for some time in a bulldozing chamber, he acquires a special knowledge of the behavior of broken rock and knows, instinctively, how to keep it moving with minimum of labor and explosive.

Winter Traveling in Western Ontario

By J. F. Kellock Brown*

Recently an inspection was made of a mineral property in western Ontario, under temperature conditions ranging downward to 50 and 55° below zero. Some experiences and sensations of a newcomer to this class of work may prove interesting.

The initial railway journey was made along the recently opened line of the Canadian Railway Co. reaching out of Port Arthur going eastward to Sudbury. This line passes through fresh territory approximately halfway between the lines of the Canadian Pacific and the Transcontinental. Although seen under winter conditions, the scenery along this route will almost certainly make it one of the most popular in Canada. Running eastward along the shore of Lake Superior and then northward along the Nipigon River and then along the banks of Lake Nipigon, one encounters some wonderful scenic effects, and in summer these spots must be perfect pictures of beauty.

The railway line was left at a siding about 100 miles east of Nipigon, and a tramp through the woods undertaken. As everything has to be packed in on men's backs in that country superfluous personal baggage should be reduced to a minimum. What you need are the clothes you stand in, a comb in your vest pocket, and some tobacco. For the rest, you can do your shaving when you return to the train going west or east, and have your bath at the nearest civilized stopping place. Scrubby chins are a mark of the prospector's independence. Pack sacks take the place of bags and grips. These are canvas holdalls designed to fit across the back and provided with straps over the shoulders and one band passing round the forehead. This provides a splendid exercise for the development of the neck. Everything was packed into the sacks the weights and shape adjusted and the party started off. The newcomer to this work had better go easy in his good-hearted offers to help, or it may end in his having to be packed in himself.

Proper clothes for the trip are essential. Beginning at the feet and working up, one should wear three pairs of socks (not carried in for a change but worn on your feet) a heavy pair of stockings over these and then a pair of moccasins. Two pairs of trousers, or a pair of mackinaw trousers with overalls, heavy underwear (the heaviest obtainable), coat, jersey, mackinaw coat, cap with earflaps, and a pair of leather covers over a pair of woolen mittens. An important point is to keep your feet warm, and this you cannot do in any other rig than that described; miners' leather boots and rubbers appear to be useless, neither of them keeping you warm enough, and the leather boots allowing you to continually slip, which adds greatly to the exhausting work of walking. Do not bring an ordinary overcoat, as it has tails and gets in the way, nor has it a collar sufficiently high and warm to go over the head. Woolen mittens that are separate from the leather covers are better than any combined type. The leather keeps out the wind and the inside mittens can be removed and dried readily.

Traveling in this case was easy, over beaten trails, but wherever the snow is untrodden, snowshoes have to be put on, since the effort of struggling through snow up to the waist (and over the head in some spots) renders progress impossible. A slip from the trail to either side means a snow bath, which is not entirely appreciated if the snow finds its way up one's sleeves and down the back of one's neck.

Three hours of almost steady tramping for those without a load brought the edge of a frozen lake in view, across which could be seen the camp, pitched in the only clearing in the dense timberland encountered since leaving the line of the railway. The camp proved to be one building, which of course served as the entire hotel; it berths eight men, houses the cookstove and resolves itself into a smoking room at night. There are no private suites with baths attached. So well constructed is the cabin that all day and during most of the night the heat inside is too great, and there are certain hours of the evening in which the fog is also so great that the men in the other bunks cannot be recognized. Attempts to lessen the heat by opening the doors result in filling the cabin with a flood of steam due to the contact of the inside air with the 40 to 50° below air outside. In the early part of the night it is far too hot for comfortable sleeping, more especially as undressing is more or less dispensed with and a pile of blankets added, but later on if the fire is allowed to go down it gets just about comfortable. Of other ills that may assail such a party, gentlemen who snore should be ejected, and those who love the "cup that cheers the day of past regrets" forbidden entrance into the Garden of Eden.

It is said that an army lives on its stomach, and this is equally true of a prospecting party in winter time. In the keen, clear frosty air it is a delight to know that a presentable and eatable supper is available in the evening. Sticklers for etiquette, those who object to seeing piles of potato jackets strewn over an oilcloth cover, who cannot get along with one knife and fork for all the courses there may be or those whose stomachs turn at the sight of other men mixing beans, toast, pie and tea into a sort of soup, are not invited to the feast. Where there is a good clean Norwegian cook who understands his business, some extremely appetizing meals may be encountered. But on the other hand, where the beef would do to mend the pack sacks, or a gruel appears misnamed beans, or the fried potatoes and the toast are indistinguishable from each other in the inky blackness of their being, or the bread is so full of hair that razors are flashing in the sun, then indeed does one wonder if life is worth prolonging.

*Mining engineer, with Arthur D. Little, Ltd., 137 McGill St., Montreal, Canada.

Washing in the morning is an operation that is undertaken in shifts—both space and water being limited. Hot water is prohibited, since that would merely be giving Jack Frost another chance to do damage. Stepping outside on a 55° below zero morning in proper rig, the only apparent difference is a feeling of sharpness in the air. Icicles begin to gather around one's mustache, and after a time these get tied onto the turned up lapels of the coats—anchoring one's head "eyes front." Walking briskly is enough to keep the circulation moving, and even standing about can be indulged in for the purpose of taking notes provided the stoppage is not for any length of time. Once the feeling of coldness gets in, it is difficult to get rid of it. Fall off the track, or into some hidden holes in a windfall, and the resultant struggle to regain the pathway will probably bring about a return of heat. Taking off and putting on mittens to work with bare hands on a notebook, should not be done—instead, learn to write and make sketches with them on. For this purpose have a notebook that holds the pencil to its side, so that when drawn roughly from the pocket, book and pencil will come together. At stopping places a fire is often kindled for heating purposes, but my experience has been that this makes one colder in the end than using a little energy walking round.

Ways of Obviating Winter Difficulties

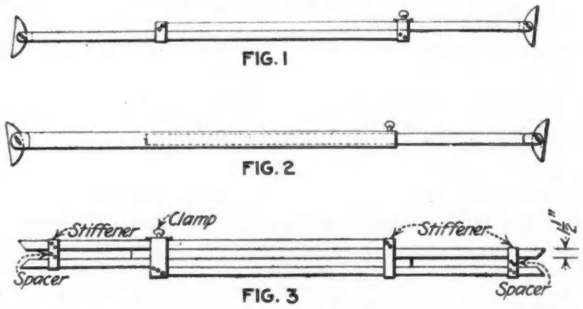
Naturally, where there is six feet and over of snow, surface work cannot be undertaken to any great extent, but if trails are well beaten out in advance, much ground can be covered and many trial shafts properly examined. Axes and snow shovels have to be added to the usual equipment. Skins of frozen water covering the rock faces are the most troublesome features, since they are so thin that they can hardly be distinguished, and they alter at times the appearance of the ore, causing examination to be rather tiresome. At the best, of course, all that can be done is in the end to report upon things seen, and other features that might have a bearing upon the project have to be allowed for as existing by proxy. Where there is much marshy ground in summertime and shaft sinking is to be done, the wintertime is about as good a period as any to carry on the work provided that care is exercised in keeping pipes from freezing. Some care may have to be taken to tide over the spring freshets, which are usually so great that work will probably be more or less interrupted for a few weeks in the early part of the year. Wintertime in the north country is ideal for transporting machinery. Then it can be taken in on sleighs with about one-tenth the effort that would be required to cover the same ground in summertime, even when the ground is at its driest. Swamps that are frozen are negotiable while the same water pools would engulf the plant during all the rest of the year.

The return from the scene of our labors meant in the end a 14-mile tramp partly through the timberland and partly along the line of the track, but fortunately a freight train hove in sight and consented to be flagged to a standstill with a branch of a tree and a red tobacco can, whereupon the remaining five miles was "hoboed" on the end of a car of lumber. This train deposited us at a divisional point in the center of the backwoods, where, however, much to the comfort of man and his internal machinery, a comfortable little hotel was unearthed, and a delightful evening spent listening to "Meet Me Down in Florida," with a snowstorm raging outside. Finally, after 15 hours' delay the lost westbound passenger train hove into sight in the small hours of the morning, and the return to Port Arthur was accomplished.

Stull-Measuring Devices

By Louis A. Rehfuss and W. C. Rehfuss*

The method of measuring for stulls in the average mine is fraught with inaccuracy. It is usually done by means of a tape or a pair of sticks, and in many cases the result is that the stull will not fit without further trimming of the hitch or timber, even then it seldom fits right. In hard ground where it may take a half a day or longer to cut a pair of hitches it is worth while to employ any means that will insure an accurate fit and save time in placing the stull. If a tape is used it takes a pretty good man to measure the distances between hitches, go to the surface or out to a station and measure the stick of timber selected with the same pull maintained in the stretching of the tape as he had employed in measuring



THREE TYPES OF STULL-MEASURING DEVICE

the hitches. No effort at all is made to obtain the batter of the hanging-wall hitch, the empty space being filled up with a mass of wooden wedges.

Where two sticks are used to obtain the stull length and batter, the miner will usually be found notching the sticks with a pocket knife to mark the distances. After the same sticks have been notched several times, he becomes confused in deciding which notches are the last ones and serious errors result.

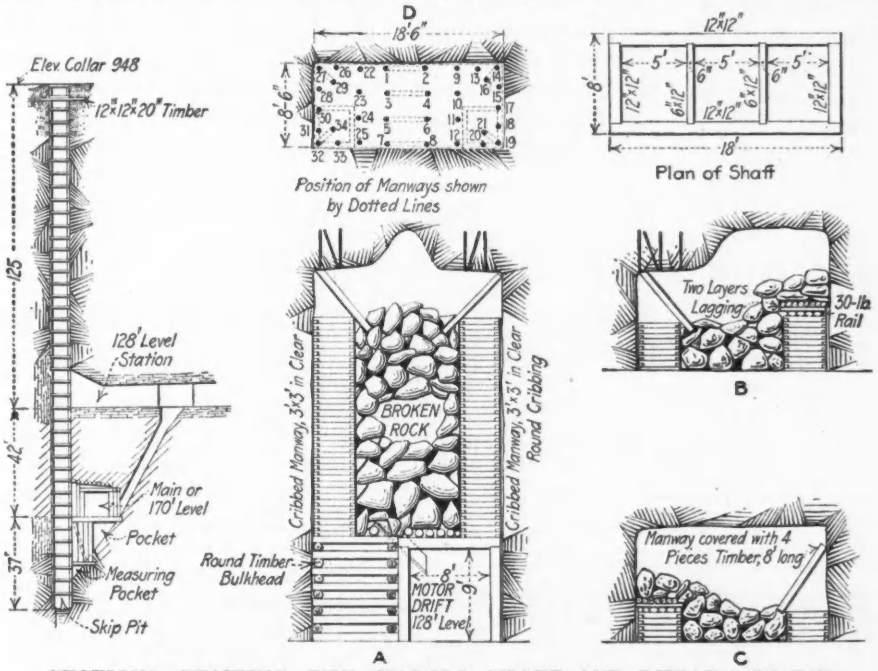
It was the consideration of these perfectly obvious, yet commonly committed errors, that led to the evolution of the stull-measuring stick, several forms of which are shown in the accompanying sketch. The most common form employed and the one most easily constructed is shown in Fig. 1, where the end clamping plates are used to get the batter of the hitches. Of course the end plates and the central clamp are fastened when the device is placed between the hitches, so that an exact reproduction of the stull required is obtained. The form shown in Fig. 2 is made of telescoping iron pipe and can be made in any mine blacksmith shop. The rotation of one pipe inside the other permits the measurement of hitches in walls that diverge or converge horizontally. The form shown in Fig. 3 is a new device that was brought to our attention by a miner who had been using notched sticks. One clamp locks the whole four sticks in position, enabling the measurement to be taken with little trouble and without assistance. The stiffness of the ends adds rigidity to the device. It is possible to use a system of figures on the sticks so that a series of hitches can be measured without making a separate trip for each.

*Mining engineers, Deerfoot Mine, West Point, Calif.

Shaft Raising at the Harold Mine

By Charles F. Jackson*

At the Harold mine, Hibbing, Minn., the shrinkage system was recently employed in raising the No. 2 shaft from the 128-ft. level of the old No. 1 shaft, which was about 650 ft. distant. It was not deemed safe to raise the shaft without supporting the walls; the use of the shrinkage system permitted raising without timber, thus securing the advantage of the lower cost in raising over that in sinking and at the same time avoiding transportation of



SECTIONS SHOWING THE HAROLD SHAFT AND DETAILS IN THE OPERATION OF RAISING

A shows condition after blasting cut holes. B shows condition after blasting west end. C shows condition at end of round. D shows plan of round

the timber through the old shaft and mine and hoisting by hand into place. The method of procedure is shown in the illustration. The material cut above the 128-ft. level was 20 ft. black slate, 85 ft. very hard taconite and 20 ft. surface. The taconite was hard to drill, but with the number of holes used, broke fairly well. The shaft was carried up full size, about 8 ft. 6 in. x 18 ft. 6 in. rock dimension.

A bulkhead was built at the level to take as much weight from the drift sets as possible and to support the slate wall, which was weak at this point. The work then progressed in three stages, as shown by *A, B* and *C* in the illustration. In the first stage, cut holes 1 to 8 (shown in plan *D*) and the corner holes 18 to 21 and 31 to 34 over the cribbed manways were drilled. The manways were then cribbed up within three or four feet of the back and the cut holes blasted. In the second stage (*B*) holes 9 to 12 and corner holes 13 to 16 and 18 to 21 were blasted, the manway on that side having been first cribbed as close to the back as possible and covered with short lengths of 30-lb. rails and lagging. In the last stage (*C*) the other side was drilled and squared up in the same manner. Sometimes the practice was varied by drilling all the remaining holes immediately after the cut had been blasted, but only one side was blasted at a time. Blasting was done with electric exploders, and after blasting it required 30 to 40 min. to blow the smoke out sufficiently for the men to return to the work.

Three eight-hour shifts were worked, the labor consisting of two miners, one helper and a shift boss part time. No Sunday work was done. The holes were drilled with two BC-21 Ingersoll-Rand butterfly stopers, air being supplied by a 10½-in. Westinghouse cross-compound air pump at 90-lb. gage pressure. One air line was carried in each manway. Holes were 4 ft. deep, and 30 to 35 holes were required per round. After each blast, seven to ten 3-ton carloads were drawn off to make room at the back and the rock was trammed by electric motor to No. 1 hoisting shaft. Before the top of the taconite was reached, two small test pits were sunk through the surface material to rock directly over the cribbed manways. The shaft was raised full size to within 15 ft. of the top of the taconite and then stopped temporarily, leaving a rock pentice, and the manways were holed through to the test pits. The bearers and collar sets were next placed, and the shaft was sunk through surface to taconite, regular sets being placed as sinking progressed. The dirt was hoisted with a one-ton sinking bucket by an air-driven 8 x 10-in. duplex hoist. When taconite was reached, the rock pentice was milled down through the connections driven over the manways. The shaft was then timbered to the level, the broken rock being drawn off and trammed to No. 1 shaft as the work progressed. When the 128-ft. level was reached the station was raised and timbered and sinking was resumed and carried a further distance of 79 ft.

*Chief engineer, Arthur Iron Mining Co., Hibbing, Minn.

Alaskan Timbers for Mine Props

Mine timbers of white spruce, Sitka spruce, white birch and western hemlock grown in the Chugach National Forest, Alaska, have been demonstrated by the Forest Products Laboratory at Madison, Wis., to be fully as good as Douglas fir from the Rocky Mountain region and superior to other Rocky Mountain species for use as mine timbers, according to the Forest Service, of the Department of Agriculture.

Tests made to determine the suitability of the Alaskan trees for mine timbers showed that in bending strength developed the four species named are from 20% to more than 100% stronger than lodgepole pine, alpine fir, Engelmann spruce, bristle-cone pine, and western yellow pine from the Rocky Mountain region. All these species are widely used for mine timbers and were tested.

Sitka spruce, white birch and western hemlock from Alaska proved to be considerably higher in average strength than Rocky Mountain Douglas fir. White spruce averaged nearly as well.

Redington Rock Drills at the Coniagas Mines, St. Catharines, Ont., according to the 1916 annual report of the company, drilled an average of 4.63 ft. per hour, compared with 3.89 for the year 1915.

Court Decisions in Regard to Unexploded Blasting Charges

By Chesla C. Sherlock*

The majority of courts hold that the master owes the duty of inspection after a blast has been exploded, to see that there are no unexploded charges, before sending employees to work in the vicinity. A Massachusetts case held it was the duty of a master to examine to see that all charges were exploded before sending laborers to the vicinity, when there was danger of injury to them by striking their tools on the unexploded charge. A North Carolina case held that a servant has the right to believe that the master has inspected a mine after the explosion and located unexploded charges, and if they existed, he would be warned against them. In a Minnesota case the court held the duty to be a more emphatic one where the master, to the knowledge of the servant, had been in the habit of making examinations but in the present instance failed to do so.

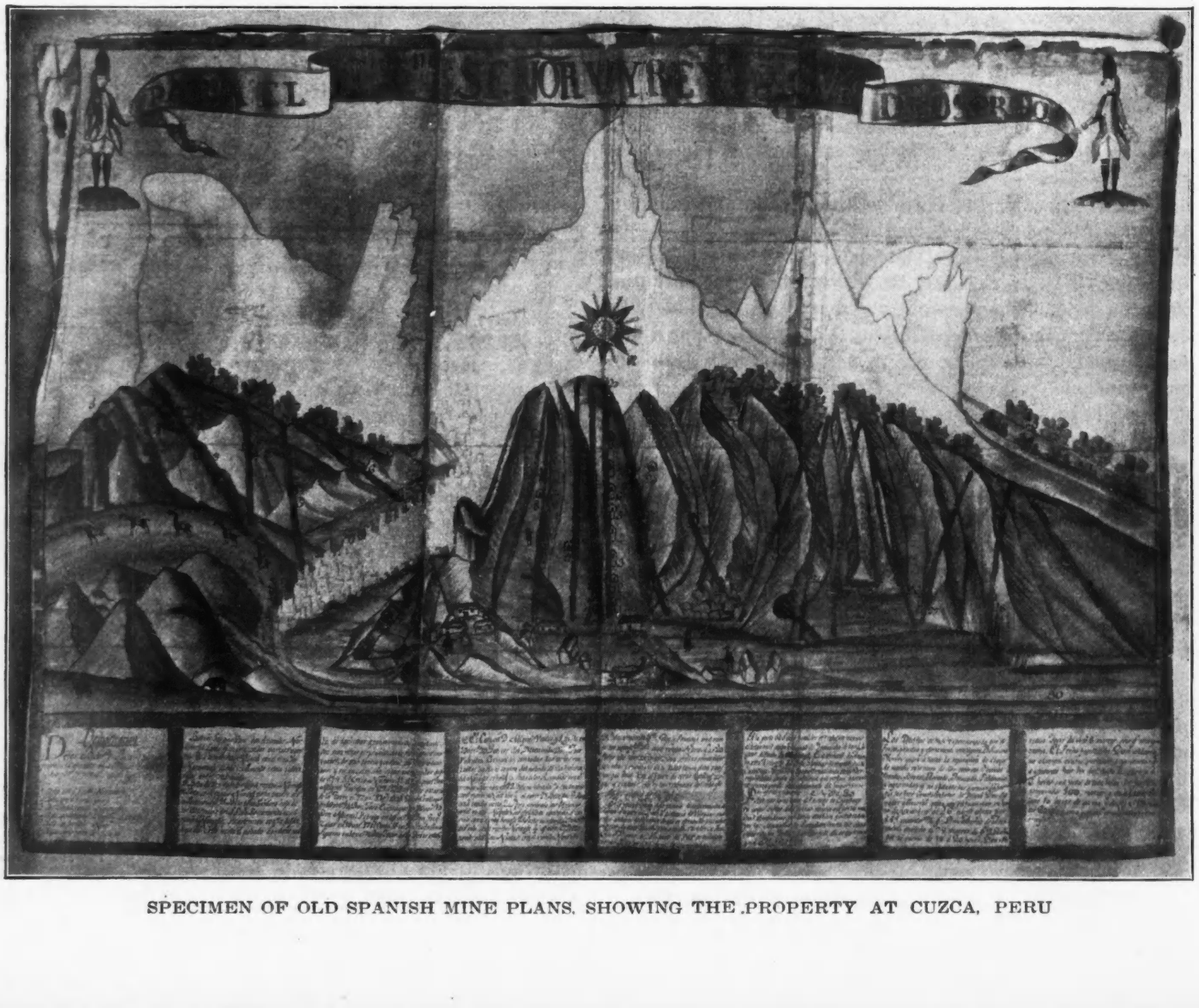
Other courts hold that it is the special duty of the master to warn the servant of the danger of unexploded charges when they are sent to work in the immediate vicinity. In Maine it was held not to be negligence, as a matter of law, to leave unexploded charges of dynamite in old holes in a quarry pit, while new holes are being drilled, but it is the duty of the master to warn his servant of this particular danger. Another Maine case held the master guilty of negligence for a failure to warn.

Many employers have been of the opinion that such a risk is a risk of the employment which the employee assumes. On the other hand, many of the cases hold that the risk of injury by unexploded charges is not an ordinary risk of employment, so a Washington case held such risk was not necessarily incident to the employment such as would be assumed by the servant where he undertook a contract of employment. The liability of the master, however, is limited in some instances. The master is not liable for unusual or extraordinary accidents, as where an old charge was cleaned out and a new one inserted, the explosion of which failed to set off a portion of the old charge that remained. Where a servant himself examined the hole and thought that the charge had gone off, he cannot recover from the master.

*Youngerman Building, Des Moines, Iowa.

Old Spanish Mine Drawing

The accompanying photograph, furnished by Lyman Chatfield, Buenos Aires, is a copy of mine plans made by Spanish engineers in 1797. These plans accompanied reports, made by mining engineers to the Viceroy of Peru, on the discovery of copper and silver deposits in the district of Chimboya, Peru. The documents and plans were sent from the office of the "Delegado de Minas del Cuzco" and addressed to the Viceroy, then the Marqués de Osorno. It is interesting to note the exaggerations and peculiarities of the graphics of that time.



SPECIMEN OF OLD SPANISH MINE PLANS, SHOWING THE PROPERTY AT CUZCA, PERU

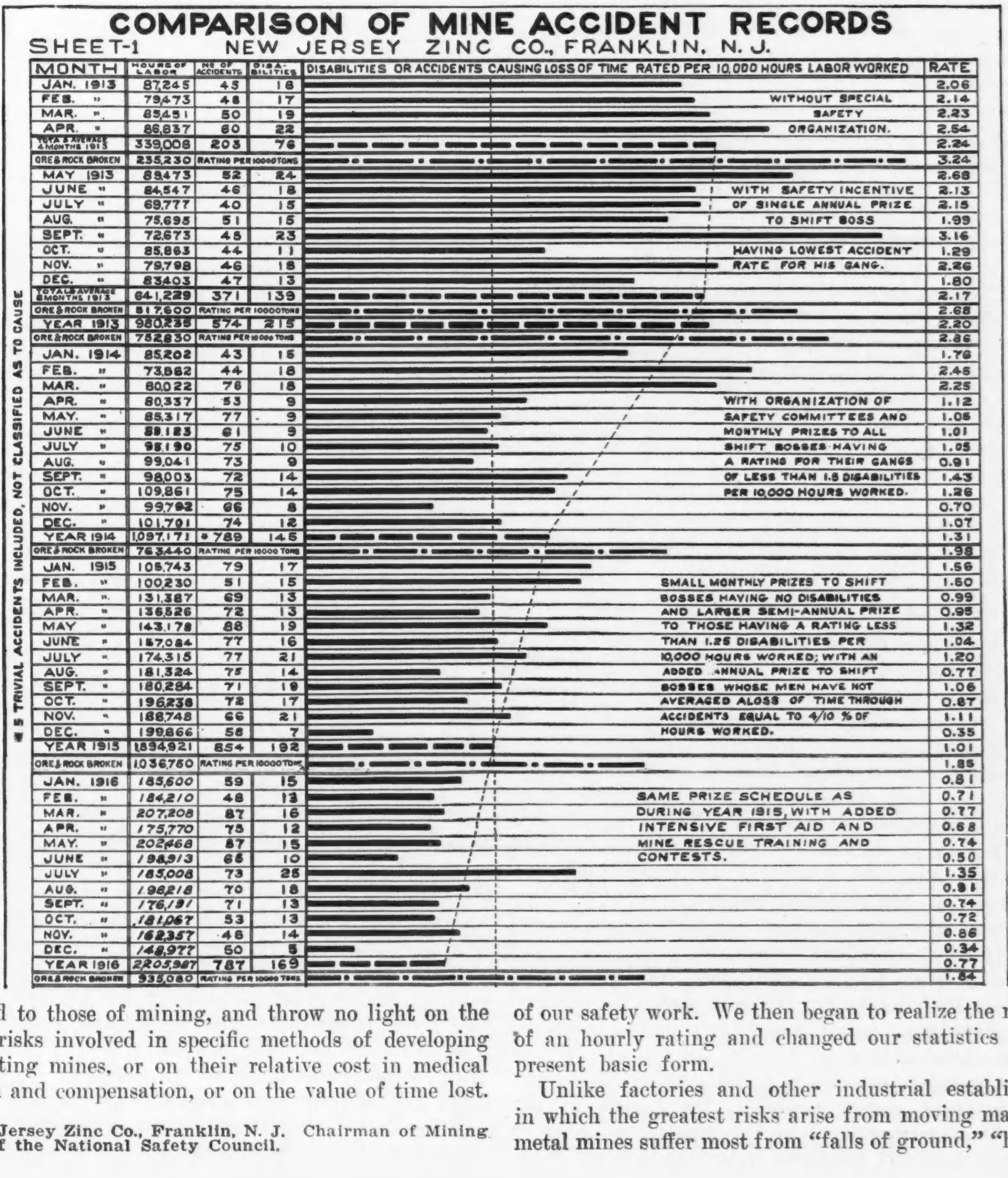
Recording of Accidents, and Safety Measures Employed by the New Jersey Zinc Co.

By Benjamin F. Tillson*

SYNOPSIS—Accident statistics as usually compiled fail to provide safe basis upon which hazards of other industries may be compared to mining. Methods of compiling and presenting figures were devised at the New Jersey Zinc Co. to obviate this. Accidents of all kinds decreased by intelligent coöperation of workmen.

The accident statistics collected by the Federal Government, state inspectors and labor bureaus, insurance companies and industrial corporations, fail to provide a safe basis upon which the hazards of other industries may be compared to those of mining, and throw no light on the relative risks involved in specific methods of developing or operating mines, or on their relative cost in medical attention and compensation, or on the value of time lost.

The mass of casualty statistics that have been compiled are, for the most part, wasted effort, for they have not been rated on the fixed basis of hours worked under the peculiar hazards involved and so have no common denominator. They, therefore, become a dangerous tool in the hands of those who may, without knowing the incompleteness of the data, draw deductions from figures that have not been collected on a comparable basis; they may, indeed, be used maliciously by those willing to distort them to support an argument. The seriousness of this pitfall was not brought home to us until we had overlooked some of these common short-comings for a year or two and had gained a distorted idea of the results of our safety work. We then began to realize the necessity of an hourly rating and changed our statistics to their present basic form.



Unlike factories and other industrial establishments in which the greatest risks arise from moving machinery, metal mines suffer most from "falls of ground," "handling

*New Jersey Zinc Co., Franklin, N. J. Chairman of Mining Section of the National Safety Council.

explosives," "loading of cars from chutes," "tramming," etc., in which supervision and education of the workman are absolutely necessary for his safety, since no mechanical contrivance can protect him. Realizing that any marked reduction in mining accidents would probably be most readily attained by educating the workmen to take an intelligent interest in their own safety; the New Jersey Zinc Co., in its mines at Franklin, N. J., in 1914 departed from its former system of depending entirely on foremen and bosses for safety-first and first-aid work and instituted a plan which seems to stimulate the interest of both men and bosses. It will perhaps be better to defer the details of this plan until after describing the methods previously employed and noting the condition to which it applied.

The mine at Franklin, N. J., during the period under consideration, has employed 350 to 1000 men (excluding the mill, machine shops and power plant) and has produced annually from 500,000 to 900,000 tons of zinc ore and from 150,000 to 250,000 tons of waste for rock fill. The ore is composed chiefly of the combined oxides of iron, manganese and zinc and the silicate of zinc. The waste rock is principally a highly crystalline limestone, so that the health of the workmen is not endangered by silicosis, as is the case at many Western metal mines.

Shrinkage methods of stoping are generally employed. The deposit is cross-sectioned by stope slices (having widths of 17 ft. and lengths equal to the horizontal distance across the orebody) leaving ore pillars averaging 35 ft. thick between every two slices. As a rule the mining of ore in stopes is carried upward to a height of 50 ft., leaving enough broken ore in the stope for its upper surface to be about 6 ft. below the solid back under which the miners are working. The broken ore is then drawn out, timbers are recovered, and waste rock is packed into the empty chamber up to the level above. Set timbers are next placed on the rock filling, and mining continues on top of these for another lift, the stope slices thus progressing upward from the bottom of the orebody. At some future time the intermediate pillars of ore will be recovered by a caving system of mining, working downward from the upper portion of the orebody; some of this work is now being practiced.

Enlisting the Aid of the Workingman

Until the beginning of 1913 the active first-aid and safety work devolved entirely upon the foremen and the bosses, who were trained in bandaging and other such operations as are necessary until an injured man can be sent to the hospital. It was naturally the duty of a boss to supervise the work of each man in his gang, and we felt that our shift bosses were more than ordinarily interested in the safety of their men. Yet we appreciated that failing in human nature which prompts a supervisor to feel that the results of an accident due to disobedience of explicit directions or to stupidity are well deserved and should not excite sympathy so long as no fatality or serious disability occurs.

Hence, to stimulate the shift bosses to such close attention in their safety supervision as would be expected of a kindergarten teacher toward her pupil, in April, 1913, we offered a prize of $200 at the end of the year to the mine shift boss or timber boss whose record was most free from serious accidents; the gravity of the several accidents was rated by the company's doctor in accordance with his prognosis rated against the total shifts of labor supervised by that boss. The table of prognosis values was formulated with due regard to the relative seriousness of fatalities, total disabilities and major injuries as shown in the New Jersey employers' liability and compensation laws, and the probable length of disability was considered in setting values for minor injuries. The accompanying table gives a list of injuries and the weights applying to them, among which is a demerit of 50 points for the failure of a boss to report an accident.

VALUATION OF INJURIES FOR SAFETY WORK

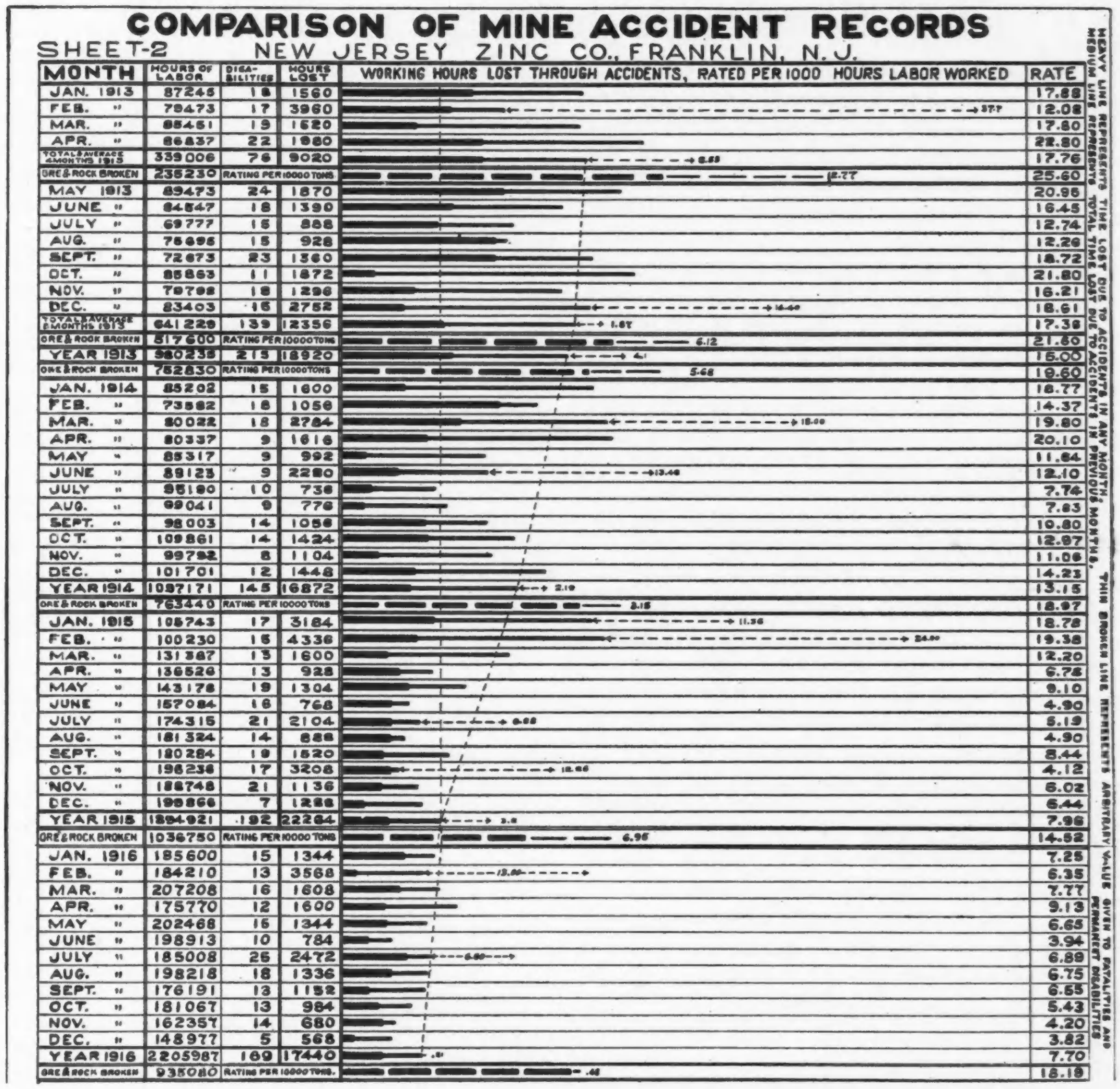
Diagnosis	Length of Disability	Demerits
Loss of life		200
Total disability		200
(a) Loss of both eyes		
(b) Loss of both arms		
(c) Loss of both legs		
(d) Broken back		
(e) Loss of one arm and one leg		
Loss of arm or leg		150
Loss of eye		125
Fracture of femur	3 months	100
Compound fracture of arm or leg	3 months	100
Loss of thumb or big toe		75
Simple fracture of arm or leg	2 months	60
Loss of fingers or toes		50
Injuries of joints	1 to 4 weeks	7 to 30
Lacerated wounds	1 to 3 weeks	7 to 30
Contusions	1 to 2 weeks	7 to 15
Abrasions	1 to 10 days	1 to 10
Scalp wounds	1 to 10 days	1 to 10
Dislocations	2 weeks	15
Failure to report accident		50

Although we realized that chance was involved in the winning of this prize, since many accidents have equal potentialities but result sometimes in trivial and sometimes in serious injuries, nevertheless, in view of the fact that the failure of a boss to win reward did not deprive him of anything that he had previously enjoyed, it seemed that such a prize ought to stimulate attention to safety precautions. It was gratifying to note that this prize went to a man who was conspicuous in his precautions for safety and whose work lay in a comparatively hazardous territory.

"Sheet" 1 shows graphically that during the last eight months of 1913, when this prize system was operative, accident prevention showed an improvement over the first four months of the year, when no such plan was in operation. The number of accidents per 10,000 hours of labor fell from 6 to 5.8, a reduction of 3.3%. The number of disabilities (accidents entailing any loss of time subsequent to the injury) per 10,000 hours of labor fell from 2.24 to 2.17, a reduction of 3.1%. The hours lost by accident disabilities per 10,000 hours worked in the mines per month, were 18.35 for the first four months, and 17.3 afterward, showing a decrease of 5.7%. During the first four months only 3 gang-months, totaling 24,694 hours, were entirely free from disabilities; while during the last eight months there were 12 gang-months, totaling 77,592 hours, with perfect scores. The record of the latter period was therefore about four times as good as that of the former.

It seemed probable that we could improve the system by awarding a monthly prize instead of an annual one, so as to reduce the element of chance. We first abandoned the elaborate evaluation of accidents by prognosis of injuries, and then arranged for a monthly bonus of $10 to each underground timber or shift boss, during 1914, who had less than 1.50 disabilities per 10,000 hours of labor in his gang each month; this rate was adopted arbitrarily as being possible of attainment, while indicating considerable improvement in safety work. The advantage of this scheme is that a boss can usually increase his salary by devoting his spare time and energy to the education and training of his men in safety measures; this encourages him to persist in his instruction of the stupid and ignorant, as well as in the disciplining of the careless and disobedient. That such has been the case is shown by computing the number of bonuses that would have been paid in the two periods of 1913 and until Dec. 31, 1914, if the monthly system had been operative for that whole time. During the first four months of 1913, 11 bonuses would have been paid out of a possible 32, a standing of 34.4%; and in the latter eight months of the year, 27 bonuses would have been paid from a possible 64, a standing of 42.2%. During 1914 we paid 66 bonuses out of a possible 110, showing that our bosses were then succeeding 60% of the time also, 41 times out of the 66 there were no disabilities. The ratio of hours worked by gangs having the foregoing disability ratings to the total hours of all underground gangs rose in a like manner.

A teacher rarely accomplishes much without the coöperation of his pupil, and neither will a boss be able to train his labor in safety methods without holding their interest. Hence, in January, 1914, we began to keep a monthly record of the number of hours lost through accidents (rated per 1000 hours worked) in each gang of the underground and surface mining operations; and of that gang having the best record each month every member was rewarded with a cigar, specially marked so as to denote that it had been given by the N. J. Zinc Co. for excellence in safe work. Although about 90% of our mine labor was of such nationalities as Russians, Poles, Slavs, Lithuanians and Hungarians, it was encouraging to note the pleasure and interest evinced by the men who received these cigars and their pride and understanding in regard to the matter. That this token means something to the men is shown by the fact that during 1914 there were 41 underground gangs who won the cigars by a clean record with no loss of time through accident, although 292,283 hours were worked. In the opencut gangs 14 won cigars, of which 12 gangs had a clean record with 62,560 hours of labor. As compared with 1913, the record of "no lost time" gangs in 1914 was over 125% better.



MONTH	HOURS OF LABOR	DISABILITIES	HOURS LOST	WORKING HOURS LOST THROUGH ACCIDENTS, RATED PER 1000 HOURS LABOR WORKED	RATE
JAN. 1913	87245	18	1560		17.88
FEB. "	79473	17	3960	37.7	12.08
MAR. "	85451	19	1520		17.80
APR. "	86837	22	1980		22.80
TOTAL & AVERAGE 4 MONTHS 1913	339006	76	9020	8.55	17.76
ORE & ROCK BROKEN	235230	RATING PER 10000 TONS		12.77	25.60
MAY 1913	89473	24	1870		20.95
JUNE "	84547	18	1390		16.45
JULY "	69777	15	888		12.74
AUG. "	75695	15	928		12.26
SEPT. "	72673	23	1360		18.72
OCT. "	85863	11	1872		21.80
NOV. "	79798	18	1296		16.21
DEC. "	83403	15	2752	14.40	18.61
TOTAL & AVERAGE 8 MONTHS 1913	641229	139	12356	1.87	17.38
ORE & ROCK BROKEN	517600	RATING PER 10000 TONS		6.12	21.60
YEAR 1913	980235	215	18920	4.1	15.00
ORE & ROCK BROKEN	752830	RATING PER 10000 TONS		5.68	19.60
JAN. 1914	85202	15	1600		18.77
FEB. "	73582	18	1056		14.37
MAR. "	80022	18	2784	18.00	19.80
APR. "	80337	9	1616		20.10
MAY "	85317	9	992		11.64
JUNE "	89123	9	2280	13.40	12.10
JULY "	95190	10	736		7.74
AUG. "	99041	9	776		7.83
SEPT. "	98003	14	1056		10.80
OCT. "	109861	14	1424		12.97
NOV. "	99792	8	1104		11.06
DEC. "	101701	12	1448		14.23
YEAR 1914	1037171	145	16872	2.19	13.15
ORE & ROCK BROKEN	763440	RATING PER 10000 TONS		2.15	18.97
JAN. 1915	105743	17	3184	11.36	18.78
FEB. "	100230	15	4336	24.00	19.38
MAR. "	131387	13	1600		12.20
APR. "	136526	13	928		6.78
MAY "	143178	19	1304		9.10
JUNE "	157084	16	768		4.90
JULY "	174315	21	2104	6.68	5.19
AUG. "	181324	14	888		4.90
SEPT. "	180284	19	1520		8.44
OCT. "	196236	17	3208	12.66	4.12
NOV. "	188748	21	1136		6.02
DEC. "	199866	7	1288		6.44
YEAR 1915	1894921	192	22264	3.8	7.96
ORE & ROCK BROKEN	1036750	RATING PER 10000 TONS		6.95	14.52
JAN. 1916	185600	15	1344		7.25
FEB. "	184210	13	3568	13.00	6.35
MAR. "	207208	16	1608		7.77
APR. "	175770	12	1600		9.13
MAY "	202468	15	1344		6.65
JUNE "	198913	10	784		3.94
JULY "	185008	25	2472	6.60	6.89
AUG. "	198218	18	1336		6.75
SEPT. "	176191	13	1152		6.55
OCT. "	181067	13	984		5.43
NOV. "	162357	14	680		4.20
DEC. "	148977	5	568		3.82
YEAR 1916	2205987	169	17440	.81	7.70
ORE & ROCK BROKEN	935080	RATING PER 10000 TONS.		.46	18.19

In the hope of perfecting our system so as to interest the shift bosses in the continued safety of their workmen, we made further changes at the beginning of 1915 in order to combat any lack of interest during the remainder of a month when a boss has been so unfortunate as to have "lost time" accidents early in that month. The revised scheme offered a monthly prize of

$5 to every underground boss who had not had a single "lost time" accident in his gang during that month; in addition, semiannual prizes of $20 may be gained by bosses of six months' standing whose "lost time" accident rate is less than 1.25 per 10,000 hours worked. To encourage the bosses to concentrate their attention on the sources of serious accidents, which cause considerable loss of time, a further annual prize of $20 was offered to each underground boss whose men average, for the year, less than 4 hours lost by injuries per 1000 hours worked. Since the number of men in a gang may range from 20 to 80, it is evident that the opportunities for a monthly bonus, requiring a perfect record, are best for those bosses having small gangs; but it is hoped that the semiannual and annual prize rates, as well as the larger amounts offered, will equalize the monthly advantages of a small gang, for the greater number of hours worked in the larger gangs permits some "lost time" accidents in every month without destroying prize chances. It is interesting to note that the boss who supervised the largest gang (and that in a territory where the natural hazard was comparatively great) was so unfortunate as to have "lost time" accidents in every month during the first half-year and yet participated in the midyear prize, while no bosses achieved more than four monthly prizes and one of these did not gain the semiannual prize. The fact that more monthly prizes were paid in the first half of 1915 than would have been paid in any six months of 1914, on the same basis of "lost time" accidents seems to indicate an improvement over the former scheme. For the year 1916, 92 monthly prizes were paid to the underground shift bosses out of a maximum possibility of 176 (therefore 52% of perfection without disabilities), 22 semi-annual prizes out of 28 possibilities (or 79% perfect on a disability rate under 1.25 per 10,000 hr. of labor worked), and 2 annual prizes of a possible 13 (or 15% with an accident lost time record under $\frac{4}{10}$% of the hours of labor worked).

For the year 1915, 72 monthly prizes, 15 semiannual prizes, and 1 annual prize were won. The greatest amount won by any boss was 8 monthly prizes, 2 semi-annual and 1 annual prize; this was the same boss who won the lump-sum prize in 1913, although his territory and duties were entirely different in the former year. During 1915 there were 87 underground gang-months (totaling 692,644 hours) with no lost time, out of a possible 188 gang-months and 1,566,631 hours worked. This record of 46% of gang-months and 45% of hours worked free from disabilities is about 30% better than the corresponding figures for 1914. In the opencuts, 32 out of 43, or 74%, of the gang-months were without disabilities and they included 141,368 hours of the 203,518, or 69%, of the time worked. These figures were about 50% better than the corresponding ones for 1914.

A big improvement was shown for 1915 over 1914, and the latter over 1913 total, as well as the last eight months and first four months respectively of 1913, as follows:

PERCENTAGE OF TOTAL GANG-MONTHS FREE FROM DISABILITIES

1916	1915	1914	1913	First Half 1913	Second Half 1913
60	52	41	29	32	22

PERCENTAGE OF TOTAL HOURS FREE FROM DISABILITY RATING

1916	1915	1914	1913	First Half 1913	Second Half 1913
52	45	37	25	26	22

As a result of this personal stimulus, there has been a conservation of the laborer's earning powers, with profit to his home, and at the same time the employer has greatly benefited because there have been fewer changes in the personnel of his plant, with consequent improvement in production and efficiency.

In conjunction with the bonus schemes described, other means were taken to educate the men in safety and first-aid work. A first-aid corps of eight members was formed by the appointment, by the mine foreman, of two new members biweekly to succeed two retiring members. This corps met once a week to receive instruction from the company doctor, and its members were paid their regular hourly rate for such time. At the end of the eighth week, if the doctor considers a retiring member proficient, the latter receives a first-aid badge as a mark of his eligibility to serve in the workmen's safety committee. The apprentice course does not fit a man for the handling of accidents in the mine, and he is not given access to the emergency kits, but it is intended to interest him in the proper care of himself and others, and not only furnishes an index of the men who will prove most helpful on the workmen's safety committee, but also forms a basis for the organization of first-aid and rescue teams in the different territories, who can assist the shift bosses in the care of accidents. On Jan. 1, 1917, the total number of graduates was 195, and the formation of 12 rescue teams had been completed and they have been so thoroughly trained as to warrant their having access to the emergency kits and caring for injured.

Methods Facilitating First-Aid Training

In order to facilitate the training of first-aid and rescue teams and to permit the drilling of squads of men in the proper handling of the injured, the company has erected a building having an area of 22 x 56 ft. A room 13 x 22 ft. is equipped with six closets for storing the Fleuss oxygen apparatus, with sink, drainboard and water supply, and a closet for the doctor's supplies. The large room, 22 x 43 ft., has a dirt floor and is supplied with much timber for drift sets, cribbings, slopes, chutes, etc., and a mine car and track, with turn sheets. This is all arranged to simulate the various mining conditions with which a rescue team might have to cope, and the intention is to train the squads in this chamber when filled with smoke and noxious gas.

To permit the inspection of those training in smoke, without requiring the attendant constantly to wear an oxygen helmet or to be with those practicing, a 4 x 7-ft. observation tunnel extends 24 ft. into the practice room with a clear height of 10 ft. above the floor. It is double-sheathed to exclude smoke and is generously provided with observation windows. An electrically operated release will permit the inspector within the observation tunnel to open the doors and windows in the smokeroom so as to clear the gases quickly by natural ventilation if he observes that one of the men is in danger of being overcome or is in other trouble. This scheme for observation has proved of greatest value.

The members of the workmen's safety committee are appointed, two biweekly, by the head of the mining department, and serve four weeks, holding weekly meetings with the department head or his representative, to discuss the nature of the accidents that occurred the previous week and to take a tour of inspection through some portion of the mine, thus devoting an entire shift. Suggestions are encouraged from the men, but probably the most important results are found in their education and the prompting of their interest in the protection of themselves and others; the bosses also are stimulated to more active supervision by the inspection and possible criticism of their territories by a workmen's committee. Each member is active for four weeks and then exchanges his active badge for an exempt one, but receives his usual pay for time devoted to active service. He is then privileged to attend the scene of an accident, when his absence from regular work will not cause injury to company property or working operations. When any serious accident takes place, the safety committee visits the scene to discuss causes and remedies, and the committeemen are asked to spread the news as to how the accident took place, so that they may profit by the example. The exempt members of the workmen's safety committee are requested to take an active interest in safety work and to contribute suggestions to the safety suggestion box; also to caution any workman endangering his own safety or that of anyone else.

Although the scheme has served admirably it has now seemed wise to change it in view of the fact that so much time is required in training or rehearsing each of the first-aid and rescue teams respectively, one-half day each month. Also it was found that not many suggestions were forthcoming from the safety committee, so the plan for the coming year is to arrange for a safety committee to make two tours of inspection in successive weeks with the object of being particularly instructed in the rules and regulations as illustrated by the practical means taken to guard against hazards in the workings, at the same time having pointed out to them any improper practices existing.

The heads of departments are expected to make quarterly tours of inspection throughout the entire plant, accompanied, if possible, by a safety inspector from some other plant, and then hold a meeting to discuss any improvements and receive suggestions of practices elsewhere as noted by the department heads or visiting safety inspector.

COMPARISON OF MINE ACCIDENT RECORDS
NEW JERSEY ZINC CO., FRANKLIN, N.J.
GRAND TOTAL MINING

MONTH	NUMBER GANG MONTHS	TOTAL HOURS LABOR	GANGS WITH DISABILITIES RATING PER 10,000 HRS. WORKED UNDER 1.5		GANGS WITH DISABILITIES RATING PER 10,000 HRS. WORKED 0.0		PERCENTAGE TOTAL GANG MONTHS WITH DISABILITY RATING UNDER 1.5	PERCENTAGE TOTAL HOURS WITH DISABILITY RATING UNDER 1.5	PERCENTAGE TOTAL GANG MONTHS WITH DISABILITY RATING 0.0	PERCENTAGE TOTAL HOURS WITH DISABILITY RATING 0.0
			GANGS	HOURS LABOR	GANGS	HOURS LABOR				
JAN. 1913	10	87245	6	56619	2	19074	60	65	20	22
FEB. "	10	79473	4	32698	2	16516	40	41	20	21
MAR. "	10	85451	4	34747	2	14888	40	41	20	17
APR. "	10	86837	4	34498	3	23486	40	40	30	27
TOTAL & AV. 4 MOS. 1913.	40	339006	18	158562	9	73941	45	47	22	22
MAY 1913	11	89473	4	24712	4	24712	36	28	36	28
JUNE "	11	84547	5	37027	4	28792	45	44	36	34
JULY "	11	69777	3	17122	2	9986	27	24	18	14
AUG. "	11	75695	7	48273	4	21135	63	64	36	28
SEPT. "	11	72673	4	22890	4	22690	36	32	36	31
OCT. "	11	85863	7	55406	4	27137	63	85	36	32
NOV. "	11	79798	6	43628	3	17017	55	55	27	21
DEC. "	11	83403	7	51910	3	15729	63	62	27	19
TOTAL & AV. 8 MOS. 1913.	88	641229	43	300968	28	167198	49	47	32	26
YEAR 1913	128	980235	61	459530	37	241139	48	47	29	25
JAN. 1914	11	85208	7	54408	4	24896	64	64	36	29
FEB. "	11	73582	3	17428	3	17428	27	24	27	24
MAR. "	11	80022	3	27033	1	8322	27	34	9	10
APR. "	11	80337	6	45049	5	36053	54	56	45	45
MAY. "	11	85317	9	44628	4	27744	82	52	36	32
JUNE "	13	89123	8	65012	6	40791	62	73	46	46
JULY "	13	95190	7	55597	6	43522	54	57	46	46
AUG. "	13	99041	10	82222	7	46959	77	83	54	47
SEPT. "	13	98003	8	67324	6	40353	62	89	46	41
OCT. "	13	109861	8	75057	5	36029	62	53	36	33
NOV. "	13	99792	11	88315	6	40941	85	88	46	41
DEC. "	13	101701	8	68502	7	44011	62	58	54	43
YEAR 1914	146	1097171	88	680475	60	407048	60	62	41	37
JAN. 1915	13	105743	8	73980	2	12252	62	70	15	12
FEB. "	13	100230	8	52728	5	25608	62	53	38	26
MAR. "	18	131387	14	102517	10	57556	78	78	55	44
APR. "	19	136526	13	91510	12	74999	68	67	63	55
MAY "	19	143178	12	92132	8	51547	63	64	42	36
JUNE "	20	157084	16	124479	9	53151	80	79	55	34
JULY "	23	174315	14	106428	13	90746	61	61	57	56
AUG. "	23	181324	20	154222	13	81040	87	85	57	45
SEPT. "	23	180284	17	134546	9	61850	74	75	39	34
OCT. "	24	196236	21	172981	14	97923	87	88	58	49
NOV. "	24	188748	18	137288	14	99494	75	73	58	53
DEC. "	24	199866	22	178812	17	140888			71	70
YEAR 1915	243	1894921	183	1408623	126	847135	75 92	74 86	52	45
JAN. 1916	24	185600	17	130276	13	92318	71	70	54	50
FEB. "	24	184210	18	141501	15	107989	75	77	63	64
MAR. "	22	207208	16	149107	11	82332	73	72	50	40
APR. "	22	175770	19	142152	16	107073	85	81	73	61
MAY "	22	202468	19	168121	13	109153	85	83	59	54
JUNE "	22	198913	19	167781	13	99738	85	84	59	50
JULY "	20	185008	14	132052	8	54253	70	71	40	29
AUG. "	20	198218	13	128436	11	98970	65	65	55	50
SEPT. "	20	176191	16	133779	12	87146	80	76	60	49
OCT. "	20	181067	18	147396	13	95029	90	81	65	52
NOV. "	19	162357	12	98537	11	91837	63	61	58	57
DEC. "	19	148977	17	125961	16	116842				79
YEAR 1916	234	2205987	198	1665099	152	1142680	78 90	78 84	84 60	52

We have supplemented this safety campaign by stereopticon and motion-picture lectures showing safe and dangerous practices; and since we did not find a sufficient supply of such material pertinent to metal mining to insure the proper development of a series of lectures, we composed a scenario depicting improper and dangerous practices and the fatal or serious consequences that were likely to follow such trangressions; it also showed the safe and proper methods of working as well as the way serious emergencies should be met. The 90 motion-picture scenes were mainly staged underground in various operating places in the mine and required about 7000 ft. of film.

It is noticeable that our safety organization does not require the formation of a special department, but places the responsibility upon the heads of the ordinary departments, with the expectation that they will exhibit the same zeal for safety that they do for production. In this way are avoided many occasions for friction between operating and safety control. The work of the company is also accomplished more efficiently because there is no division of authority in any department and because the head of operations is best qualified to select the proper time for taking safety measures, without undue increase of floating gangs and without waste of time. The success of this scheme really depends upon the earnestness of "the man higher up." In order to develop safety work intelligently it is highly important that each department head have sufficient clerical aid to attend to the routine of bulletins, the abstracting of safety literature, and most especially the recording of all accidents, with classification of their causes, their distribution among the various gangs, and the amount of time lost.

System of Recording Accidents

Our system of recording accidents is as follows: Blanks are furnished to the shift bosses on which to record the following information concerning injuries sustained by any workmen in their gangs, whether while engaged in company work or off shift, stating the latter case particularly, so that the company will have information to disprove fraudulent claims for injuries in their employment: Name, payroll number, address, age, occupation, place where accident occurred, date and hour of accident, where taken after accident, supervisor of the work, witnesses, description of the injury, manner in which the accident happened, classification according to the Bureau of Mines system, and suggestions as to how it might have been avoided. These reports are kept by the mine foreman, who investigates the matter and makes a copy of the report. This is forwarded to the time office, where information is entered as to weekly wages, weekly rate of half-time, nationality, length of time man had been employed at the work in which he was injured, how long he had been in the service of the company and whether married or single. The report then goes to the main office of the plant, where the information is typewritten in triplicate on large report sheets, to which is added information as to the probable length of disability (shown by the surgeon's prognosis), the actual time of disability shown on the payroll, the dates that claims and settlements are made and their amounts, the dates and amounts of "surgeon's charges," "hospital charges" and "compensation," and finally the "total burial expense" and "total expenditures." One of these reports is kept on file in the main plant office, one copy is sent to the time office, and one copy to the main offices of the company. The State of New Jersey requires the immediate reporting of accidents, in prescribed form, to the Department of Labor at Trenton, after a man has been absent from work two weeks or more, due to the injury sustained. All fatal accidents are reported to the Department of Labor in the same manner.

The shift bosses are also supplied with forms for admitting men to the hospital; these they fill out with the date, name, payroll number and occupation of any man injured in company work, no matter how slight the injury. In case of serious injuries first aid is given, stretchers being employed if necessary, and the injured men are sent immediately to the hospital in the company's automobile ambulance. If injuries are so slight as not to incapacitate the men, bandaging is performed by the shift boss (who carries sterile rolled bandages and gauze pads in a container on his person) and the men receive their hospital slips at the end of the shift. Since the hospital has dispensary hours at night as well as by day, it is possible for men to report there for the inspection and dressing of wounds without interfering with their work. The shift bosses permit no men to return to work until they bring hospital discharge slips from the company surgeon, bearing the date when they are considered fit to work. Thus responsibility for the cleanliness of wounds, however slight, is placed directly upon the surgeon; he is able to control the injured men because they cannot get work until he has had an opportunity for a final examination of their injuries and has formally discharged them. Supervision by the shift boss is also aided, as they can note whether men have stayed away from work longer than their injuries warranted and can fairly exercise such discipline as seems necessary. This system of reporting accidents seems quite efficient. The shift bosses do not rely solely upon the voluntary reports of their men, but question and report any man who exhibits an injury; if a man fails to report his injury, it usually becomes known ultimately and results in a severe disciplining by temporary laying-off from work or by dismissal.

In classifying accidents as to their causes we have adopted the headings employed by the United States Bureau of Mines, with the addition of a group of "Trivially Injured," including all such accidents as did not prevent the men from returning to work the next day. In 1915 a further subdivision was made by listing "Permanent Disabilities" and separating "Slight Injuries" into those causing a loss from 1 to 14 days and from 15 to 20 days.

It should be noted that three systems of safety work have been tried since 1912 and the results have been very gratifying. The following percentages exhibit the decrease in accident rates, as based on 1000 employees for 300 days of 10 hours, from Dec. 31, 1912, to Dec. 31, 1914:

	Fatal %	Serious %	Slight %
Underground mining	77.4	36.9	70.5
Surface mining	100.0	(increase)	56.8
Total mining	87.2	54.9	72.4

It is possible that some credit for the reduction in accident rates is due to the change, in July, 1913, from a 10-hour to an 8-hour working shift. But, on the other hand, the system of reporting accidents now is so improved over the old methods that the actual decrease in serious and slight accidents must have been even greater than the figures given indicate.

A first glance at the accident record for 1915 would lead to the conclusion that safety work was not producing

COMPARISON OF MINE ACCIDENT RECORDS
THE NEW JERSEY ZINC CO., FRANKLIN, N.J.

Old Numbers	Causes of Injuries	1912 Killed	1912 Injured Seriously	1912 Injured Slightly	1913 Killed	1913 Injured Seriously	1913 Injured Slightly	1914 Killed	1914 Injured Seriously	1914 Injured Slightly	1914 Injured Trivially
1	FALL OF ROCK OR ORE FROM ROOF OR WALL	3 75%	8 30.8%	57 17.5%	3 100%	4 22.2%	30 16.1%		4 32.2%	12 11.3%	49 8.4%
2	BY ROCK OR ORE WHILE LOADING AT WORKING FACE						7 3.8%			2 1.9%	3 .5%
3	TIMBER OR HAND TOOLS		4 15.4%	75 23.0%		3 16.7%	39 21.0%		3 16.7%	27 25.5%	181 30.9%
4	EXPLOSIVES		1 3.9%	2 0.6%		2 11.1%	1 0.5%		2 11.1%	1 .9%	2 .3%
5	HAULAGE ACCIDENTS		5 19.2%	47 14.4%		5 27.8%	29 15.6%		3 16.7%	17 16.0%	88 15.0%
6	FALLING DOWN CHUTE, WINZE, RAISE, STOPE	1 25%	3 11.5%	10 3.1%		1 5.6%	6 3.2%	1 100%	2 11.1%	13 12.3%	18 3.0%
7	RUN OF ORE FROM CHUTE OR POCKET		5 19.2%	98 30.1%		1 5.6%	49 23.6%		1 5.6%	15 14.2%	123 21.0%
8	DRILLING, MACHINE OR HAND			28 8.6%			15 8.1%		1 5.5%	13 12.3%	49 8.4%
9	ELECTRICITY										
10	MACHINERY - PUMPS, HOISTING ENGINES, ETC.			4 1.2%		1 5.6%	7 3.8%				
14	NAILS						1 0.5%			1 0.9%	8 1.4%
15A	SLEDGING CHUNKS AND MUCKING			5 1.5%		1 5.5%	2 1.1%		2 11.1%	5 4.7%	65 11.1%
15	OTHER CAUSES										
16	FALLING DOWN SHAFTS										
17	OBJECTS FALLING DOWN SHAFTS										
20	SKIP, CAGE OR BUCKET										
21	OTHER CAUSES										
	TOTAL UNDERGROUND	4	26	326	3	18	186	1	18	106	586
	RATE PER 3,000,000 EMPLOYES HOURS	14.55	94.6	11.88	10.1	60.6	626	3.31	59.7	351	1944
	PERCENT DECREASE IN UNDERGR. ACC. RATES				31.7%	36.9%	47.3%	61.1%	1.48%	43.9%	
	MINING METHODS AND TONNAGE										
	No. AND PERCENT OF SURFACEMEN KILLED - INJURED										
23	RAILWAY CARS AND LOCOMOTIVES										
26	FALLS OF PERSONS										
27	HAND TOOLS, AXES, BARS ETC.										
29	MACHINERY										
30	OTHER CAUSES										
31	FALLS OR SLIDES OF ROCK OR ORE	1 50%		2 33%			3 50%			3 23.1%	5 8.5%
33	HAULAGE ACCIDENTS, CARS, LOCOM. ETC.						1 17%			2 15.4%	8 13.6%
35	FALLS OF PERSONS	1 50%				2 100%					10 17.0%
38	MACHINERY									1 7.7%	3 5.0%
40	HAND TOOLS			4 67%			2 33%			1 7.7%	25 42.4%
41A	SLEDGING CHUNKS AND MUCKING								1 100%	6 46.1%	8 13.5%
41	OTHER CAUSES										
	TOTAL - OPEN CUT AND SURFACE	2		6		2	6		1	13	59
	RATE PER 3,000,000 EMPLOYES HOURS	133.3		400		46	138.1		13.25	173.0	785.0
	PERCENT DECREASE IN SURFACE & O.C. ACC. RATE						65.5%		71.2%	25.3% inc.	
	MINING METHODS AND TONNAGE										
	GRAND TOTAL MINING	6	28	332	3	20	192	1	19	119	645
	TOTAL RATE PER 3,000,000 EMPLOYES HOURS	20.70	89.70	1146	8.81	58.7	564	2.65	50.5	316.2	1711
	PERCENT DECREASE IN TOTAL MINE ACC. RATE				55.8%	34.8%	50.1%	69.9%	14.0%	43.9%	62.1% inc.
	No. ACCT. WITHOUT DISABILITIES & TOTAL ACCTS.	131		495	359		574	645		784	
	RATE OF TOTAL ACCS. PER 3,000,000 EMP. HOURS	1707			1686			2125			
	NUMBER OF DAYS MINE WAS WORKED	300			307			308			
	AVERAGE No. OF MEN UNDERGROUND PER DAY	275			324			367			
	AV. No. OF MEN PER DAY - SURFACE WORKING	15			44			91			
	LENGTH OF WORKING SHIFT IN HOURS	10			10 JAN.-JUNE, 8 JULY-DEC.			8			
	TONS OF ORE AND ROCK BROKEN	480,320			762,830			763,440			
	ACCIDENTS RATED - PER MILLION NET TONS BROKEN	12.45	54.2	690.0	3.98	26.6	255.0	1.31	24.9	156.0	847.0
	ACCIDENT TONNAGE RATING WITHOUT DISABILITIES - TOTAL ACCIDENTS	273		1030	476		761	847		1030	

Old Numbers	Causes of Injuries	1915 Killed	1915 Permanent Disability	1915 Injured Seriously Over 20	1915 Slightly 15-20	1915 Slightly 1-14	1915 Slightly Total	1915 Trivially	1916 Killed	1916 Permanent Disability	1916 Injured Seriously Over 20	1916 Slightly 15-20	1916 Slightly 1-14	1916 Slightly Total	1916 Trivially
1	FALL OF ROCK OR ORE FROM ROOF OR WALL	3 75%		2 20%	1 11.1%	15 10.1%	16	71 11.7%				4 57.%	22 16.8%	26	76 14.1%
2	BY ROCK OR ORE WHILE LOADING AT WORKING FACE				2 22.2%	26 17.6%	28	99 16.4%			1 33.3%		29 22.1%	29	149 27.6%
3	TIMBER OR HAND TOOLS			1 10%		31 21.3%	31	148 24.4%				1 14.3%	14 10.8%	15	106 19.6%
4	EXPLOSIVES	1 25%	1 100%			5 3.4%	5	2 .3%		2 100%			6 4.6%	6	3 .5%
5	HAULAGE ACCIDENTS			4 40%	4 44.4%	15 10.1%	19	43 7.1%			1 33.3%		21 16.%	21	25 4.6%
6	FALLING DOWN CHUTE, WINZE, RAISE, STOPE			3 30%		6 4.1%	6	28 4.6%			1 33.3%		9 6.9%	9	27 5.1%
7	RUN OF ORE FROM CHUTE OR POCKET					11 7.6%	11	35 5.8%				1 14.3%	4 3.%	5	16 2.8%
8	DRILLING, MACHINE OR HAND				1 11.1%	19 13.1%	20	85 14.7%					10 7.7%	10	66 12.0%
9	ELECTRICITY					1 .7%	1	4 .7%					2 1.5%	2	1 .2%
10	MACHINERY - PUMPS, HOISTING ENGINES, ETC.							1 .1%							
14	NAILS							3 .5%							2 .4%
15A	SLEDGING CHUNKS AND MUCKING					11 2.6%	11	42 6.9%							
15	OTHER CAUSES				1 11.1%	6 4.1%	7	44 7.2%					13 9.9%	13	63 11.6%
16	FALLING DOWN SHAFTS														1 .2%
17	OBJECTS FALLING DOWN SHAFTS							1 .1%							
20	SKIP, CAGE OR BUCKET							1 .1%							
21	OTHER CAUSES							1 .1%				1 14.3%		1	7 1.3%
	TOTAL UNDERGROUND	4	1	10	9	148	155	608		2	3	7	130	137	541
	RATE PER 3,000,000 EMPLOYES HOURS	7.31	1.83	18.3 / 20.1	16.5	271 / 284		1114		3.35	5.02 / 8.37	11.7	217 / 229		905
	PERCENT DECREASE IN UNDERGR. ACC. RATES	121% inc.	65.4%		19.1%			42.7%		58.5%		19.4%			18.7%
	MINING METHODS AND TONNAGE														
	No. AND PERCENT OF SURFACEMEN KILLED - INJURED														
23	RAILWAY CARS AND LOCOMOTIVES														
26	FALLS OF PERSONS					1 5.5%	1	1 2.0%							3 3.8%
27	HAND TOOLS, AXES, BARS ETC.					2 11.1%	2	1 2.0%					3 11.4%	3	3 3.8%
29	MACHINERY					1 5.6%	1								
30	OTHER CAUSES					2 11.1%	2	11 20.3%					3 11.4%	3	14 18.1%
31	FALLS OR SLIDES OF ROCK OR ORE			1 33.%		2 11.1%	2	13 24.0%					7 27.0%	7	15 19.5%
33	HAULAGE ACCIDENTS, CARS, LOCOM. ETC.							2 3.7%	1 100%				1 3.9%	1	3 3.8%
35	FALLS OF PERSONS	1 100%		2 67.%		4 22.2%	4	12 22.0%					3 11.4%	3	9 11.7%
38	MACHINERY					1 5.6%	1	2 3.7%					1 3.9%	1	4 5.2%
40	HAND TOOLS					1 5.6%	1								6 7.8%
41A	SLEDGING CHUNKS AND MUCKING														1 1.3%
41	OTHER CAUSES					4 22.2%	4	11 20.3%					8 31.0%	8	19 25.0%
								1 2.%							
	TOTAL - OPEN CUT AND SURFACE	1		3		18	18	54	1				26	26	77
	RATE PER 3,000,000 EMPLOYES HOURS	9.9	29.8			179		755	7.29				189		561
	PERCENT DECREASE IN SURFACE & O.C. ACC. RATE		125% inc.			3.5% inc.		3.8% dec.	26.3%				5.6%		25.7%
	MINING METHODS AND TONNAGE														
	GRAND TOTAL MINING	5	1	13	9	164		662	1	2	3	7	156		618
	TOTAL RATE PER 3,000,000 EMPLOYES HOURS	7.92	22.2		274			1050	1.36	6.79		221			840
	PERCENT DECREASE IN TOTAL MINE ACC. RATE	199% inc.	56% decr.		13.3% decr.			38.6%	82.8%	69.5%		19.3%			20%
	No. ACCT. WITHOUT DISABILITIES & TOTAL ACCTS.	662				854			618				.787		
	RATE OF TOTAL ACCS. PER 3,000,000 EMP. HOURS	1350							1069						
	NUMBER OF DAYS MINE WAS WORKED	309							308						
	AVERAGE No. OF MEN UNDERGROUND PER DAY	645							727						
	AV. No. OF MEN PER DAY - SURFACE WORKING	122							135						
	LENGTH OF WORKING SHIFT IN HOURS	8							8						
	TONS OF ORE AND ROCK BROKEN	1036,750							935,080						
	ACCIDENTS RATED - PER MILLION NET TONS BROKEN	4.82	.954	12.5 / 13.5	8.68	158 / 167		640	1.07	2.14	3.21 / 5.35	7.50	167 / 174		662
	ACCIDENT TONNAGE RATING WITHOUT DISABILITIES - TOTAL ACCIDENTS	638			824				662			844			

the satisfactory results that should have been expected in view of the results obtained in preceding years; for there were four more fatal accidents than in 1914, an increase in rating of 199%, and the open-cuts and surface accident rates also increased. This introduces a most important point in the analysis of accident statistics, for it exemplifies how easy it is to be deceived by laying too much stress upon small numerical quantities rated by small divisors. It is obvious that gross carelessness, disobedience or suicidal tendency of a few may account for a percentage increase that will appear large but will not represent any change in actual hazards. For this reason it seems a great mistake to put so much stress on the rating of fatal accidents (most people seem to devote their entire attention to them) for the element of luck may so disguise the tendency toward reduced hazards as to make the benefits of organized safety work seem conspicuous by their absence. The same fall of ground which resulted in a fatality because it struck a man on the head might have produced only a slight injury if it had only grazed his foot or might not enter into the records at all if it missed him. It would therefore seem advisable to devote greatest attention to the serious, slight and trivial accidents, which occur in greater numbers and best illustrate the hazards involved. An analysis of these and a comparison of their rates, as based upon units of hourly labor exposed to the hazards, will more truly show the benefits of safety work; and such figures show that the safety-work methods employed during the year 1915 bettered the already excellent results of 1914, as shown graphically on sheets Nos. 1 and 2.

In explanation of the different types of graphic lines on sheet No. 2 it may be stated that the heavy lines indicate the rated hours lost in any month owing to accidents in that month, thin lines show the rated hours lost in a certain month because of an injury sustained in some previous month in that year, and dotted lines indicate a rated arbitrary value of hours lost because of a fatality or total permanent disability. Because there are about 300 working days in a year and it is only a matter of luck whether the accident occurred the first or the last day of the year, an arbitrary average of 150 days lost is applied in one month to any accidents of this type and these days are rated accordingly.

The managements of many coal mines and of some metal mines like to rate their accidents against tonnage. This appears to be of but little value, since the production of a large or a small tonnage per man depends upon such physical features as the size, position, configuration and constitution of the deposit. These do not, as a rule, affect the total hazard, which depends most largely upon the length of time spent in jeopardy.

Foreman's Chart for Showing Accidents

In order that the mine foremen and assistants may be conversant with the causes of accidents which need the most attention, they keep a chart showing a list of the fatal, serious, slight and trivial accidents against the record of each boss. Classification of accidents by causes shows "falls of rock or ore" to be the most serious, "haulage or tramming" next in importance, and "falls of persons down chutes, raises and stopes" third. "Handling of timber and hand tools" has sometimes been a source of serious accident, but more commonly causes only slight injuries, where it is rivaled by "runs of ore from chutes and pockets" and by "drilling operations." We are gratified that our fatality rate in 1916 fell to 1.36, a point below the average of either coal- or metal-mining industries, the rates for which were quoted as 4.44 and 3.89 respectively for 1915.

Our charting of the causes of injuries has been based upon the classification used by the United States Bureau of Mines, to which some few items were added. But realizing the difficulty of properly classifying many accidents, because of the meagerness and ambiguity of the headings, we prepared a more detailed classification which we submitted to A. H. Fay, the accident statistician of the Bureau of Mines. We further suggested that if forms suitable for the entering of accident statistics were issued by the bureau to the different operators at the beginning of the year and were tabulated so as to be of use as a daily journal, a summary of which would be the report desired by the Bureau of Mines at the end of the year, it would be far easier for the different companies to make intelligent reports, and the statistics would be much more accurate and valuable than those collected in the past. This suggestion was adopted, and a form (No. 6-4791) was issued to assist the mining industries in reporting for the year 1916; it is arranged so as to apply to both coal and metal mining on a standard basis of hours of payroll labor. A corresponding sheet was also issued to cover milling and smelting operations. These sheets do not provide for trivial accidents (from which no loss of time has resulted). It would seem desirable to do so, as such accidents also indicate the hazards, for the same causes might have produced slight, serious or fatal injuries. An important entry on this sheet is the one relating to the "principal mining methods used."

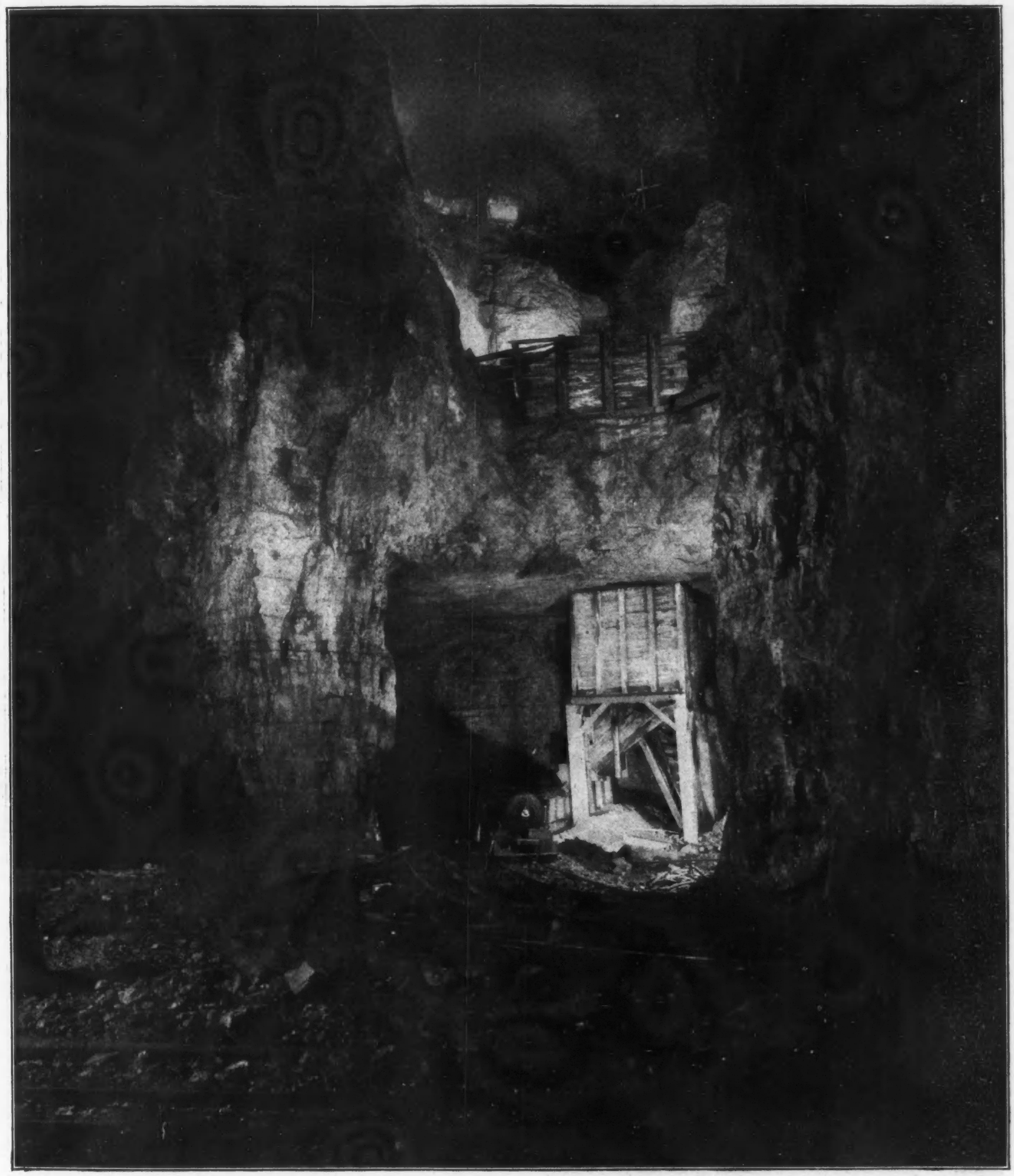
Medical examination of applicants for employment has been practiced since February, 1915, but as yet there has been no systematic canvass of old employees. Of the approximately 1500 applicants, none has refused medical examination. Men suffering from hernia are not permitted to go to work, where they may receive further injury, until they have been operated upon or wear a truss to remedy the trouble. Many other serious physical ailments such as trachoma (or other infectious or contagious diseases), varicose veins, deafness, partial blindness, serious lung or heart conditions, have been detected and the applicant refused if he could not be placed without hazard to the company or himself; or else he receives medical attention and employment for occupations for which he is fitted.

It is only fair to our record of fatalities and serious injuries to call attention to the fact that the Franklin mine is in an orebody that possesses the unusual condition of being in a state of internal stress. Ground that appears and sounds solid to all tests will suddenly crack with an explosion and will fall in masses from a fraction of a pound to many tons in weight; therefore, accidents from "falls of ore and rock" are extremely hard to control. Furthermore, during 1915 and 1916 labor and production were abnormal because of the European War, and resulted in the employment of large numbers of unskilled and deficient foreign workmen, with greatly increased production and lessened efficiency. The increased consumption of alcoholics and the greater disorder in the community during this period both would exert strong tendencies toward increased accident rates. The operation of portions of the mine on two and three shifts per day, where it had formerly been worked on a single day shift, required the employment of bosses who did not have so much experience as the staff of bosses prior to that period. It is therefore all the more remarkable that real progress has been made in accident prevention.

British Tin Output in 1915

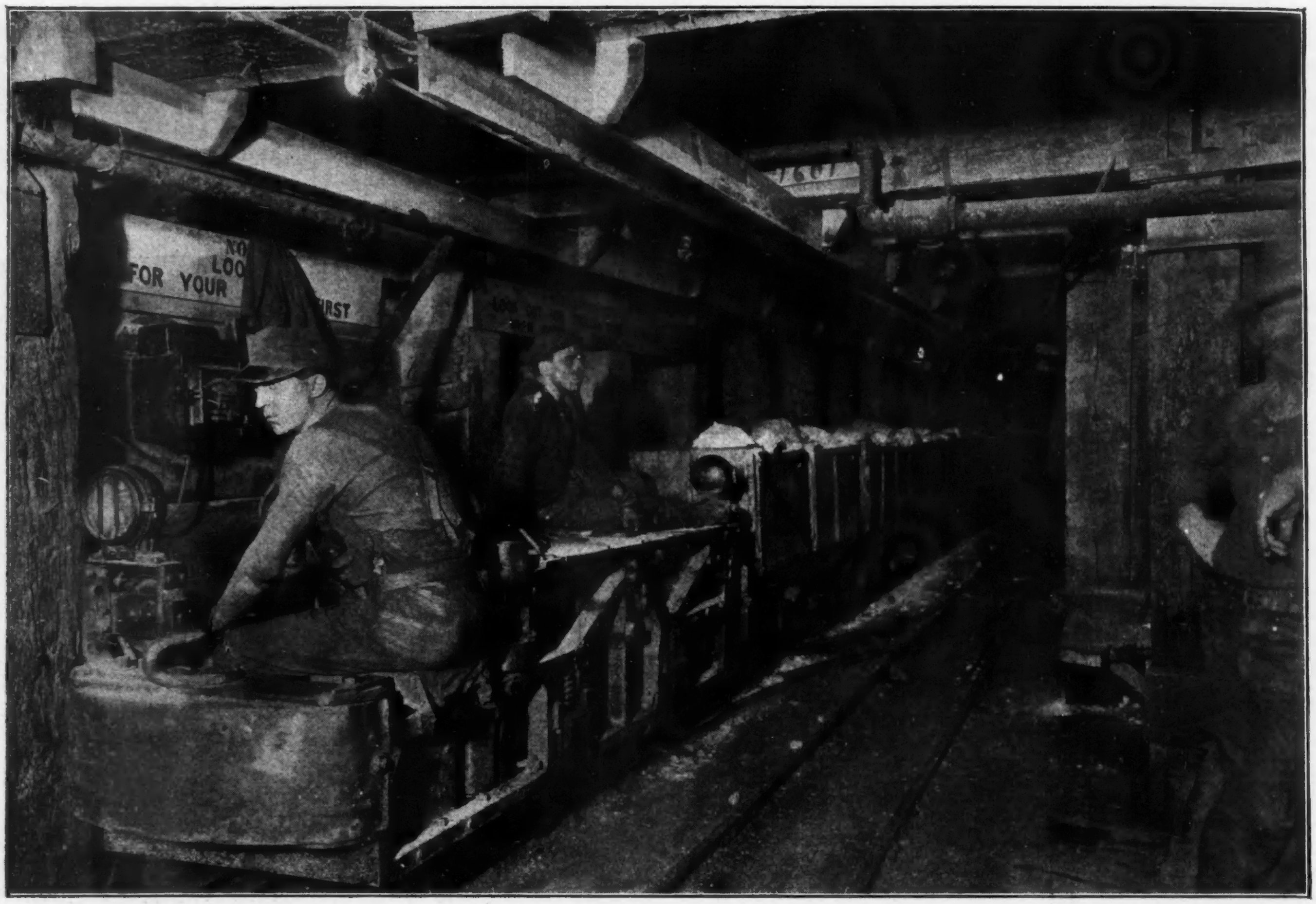
The total output of dressed tin ore according to press accounts, in the United Kingdom in 1915 amounted to 8144 tons 11 cwt. (8085 tons 8 cwt. in 1914); the total amount of tin obtainable by smelting was 4967 tons 17 cwt. (5056 tons 5 cwt. in 1914). The returns supplied by the owners of mines and quarries show that the average percentage of metallic tin obtainable was 66 in the case of ordinary ores. In the case of the product of "stream works," the average obtainable was 35.1%. There was imported into the United Kingdom in 1915 44,748 tons of tin ore and 38,896 tons of block, ingot and bar tin.

Photographs from the Field

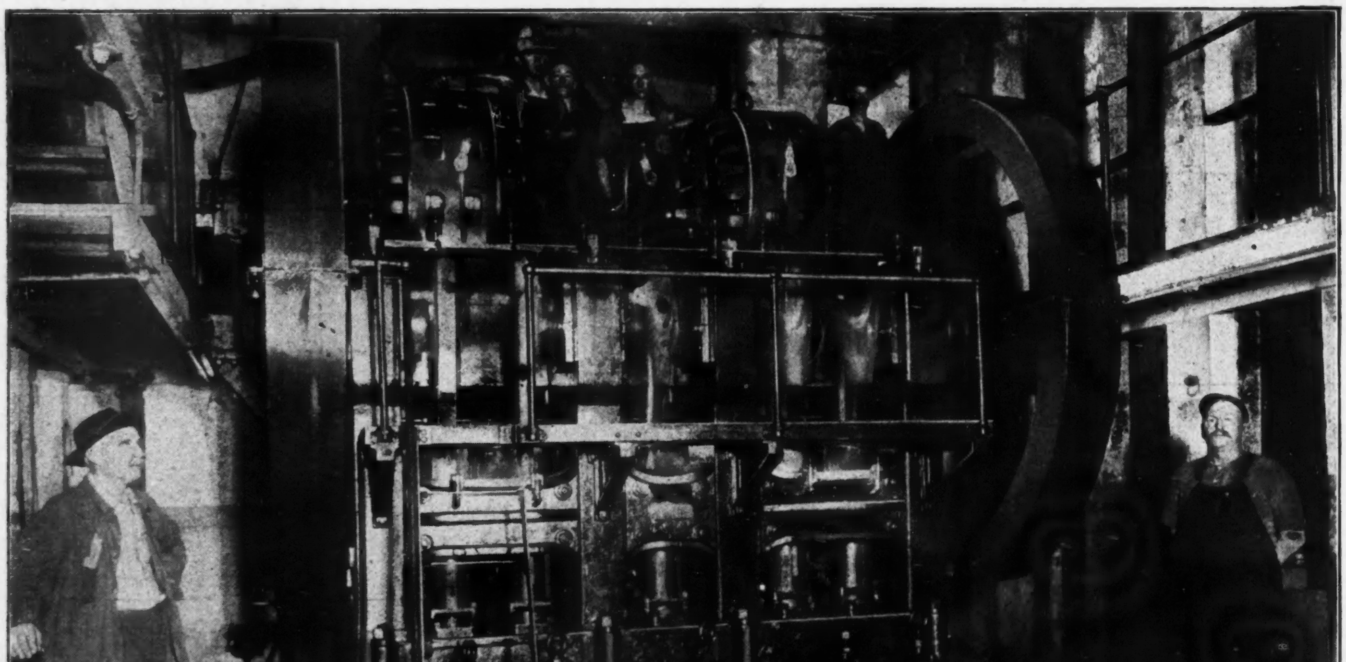


UNDERGROUND VIEW IN ONE OF ST. JOSEPH LEAD CO.'S MINES AT BONNE TERRE, MO.

In this high stope, work is proceeding at three levels; at the bottom a compressed-air locomotive is about to pull a train of ore from the loading pocket. The stope is an unusually high one, even for southeastern Missouri; from the main haulage level to the track on which the mule can be seen is 100 ft.; the total height of the stope is 117 ft.



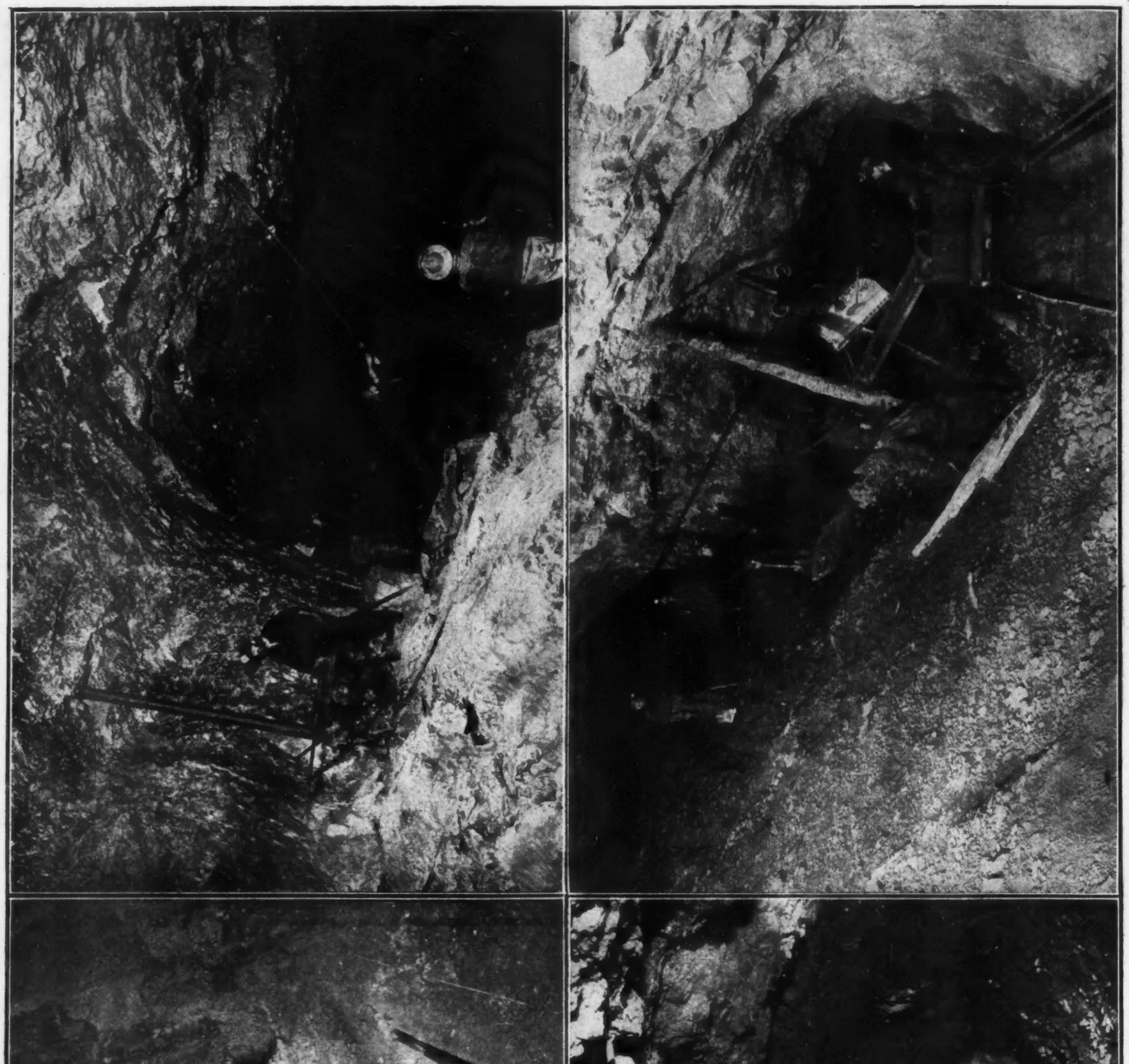
UNDERGROUND ELECTRIC LOCOMOTIVE AND ORE TRAIN IN ONE OF THE ANACONDA COPPER MINES



UNDERGROUND ELECTRIC PUMP IN THE LEONARD MINE OF THE ANACONDA COMPANY AT BUTTE, MONT.

CALUMET & HECLA WORKINGS IN THE CONGLOMERATE LODE, UPPER PENINSULA OF MICHIGAN

Upper view at left shows mancar about to take the men to the surface; car holds 30 men. View at the right shows first set of stull timbers in level; timbers 10 to 30 in. in diameter and 16 to 26 ft. in length. Lower views show the beginning of a stope in a vein about 20 ft. wide



CALUMET & HECLA WORKINGS—MAINLY IN THE AMYGDALOID LODE—IN THE LAKE SUPERIOR COPPER REGION OF MICHIGAN

Upper views show stoping in the amygdaloid lode, and barring down loose rock in a conglomerate stope. Lower views show the loading of a round of holes in the amygdaloid lode, and the mechanical loading from the stope of a tram car on one of the haulage levels

ADDITIONAL UNDERGROUND VIEWS OF WORKINGS OF THE CALUMET & HECLA MINING CO.

Loading tram cars from chute in the conglomerate lode; dumping car into skip at plat. Lower left-hand view was taken on the 72nd level in the Red Jacket shaft and shows the dumping of a car into a skip pocket; view at the right shows a transfer truck in a subshaft

Editorials

The United States About To Enter The War

President Wilson in an address, embodying the loftiest ideas and phrased in wonderful language, told Congress, Apr. 2, that the United States is in a state of war with Germany, owing to German aggression; that there should be a declaration of this and that the United States should coöperate with the Allies to the fullest extent to bring the world war to an end. Up to the time of our going to press Congress had not declared war, but it is expected promptly to do so.

The United States is going to war for the sake of its rights, of its honor, and of the principles of democracy and humanity. The violation of these by Germany became so outrageous that the sentiment of America crystallized regretfully but promptly in favor of war, no matter how terrible, no matter what the consequences, no matter what the cost. The condition was reached where the price of peace was war. We enter it as an undivided nation, loyal to our principles and our country. The time of discussion has passed. The time of action has come.

Industrial Concerns Aiding the Nation

Following the generous offer of Mr. Ford in placing his plant at the disposal of the Government to make things at cost, and the contract of the copper producers to supply copper at a greatly reduced price, it is reported from Washington that more than 5000 industrial concerns in the United States have offered their services to the Government in time of war. Almost every commodity which would be required for maintenance of the army and the navy, these departments announce, can now be obtained in virtually unlimited quantities at near cost, or at only a fair profit to the seller.

The War and Navy departments highly commend the almost universal response of American firms to the threat of war, and the "self-sacrificing attitude" of the industries of the nation. Some of the concerns have offered to produce at actual cost; others have offered to place their plants at the disposal of the Government for whatever use may be required of them, and still others have named moderate profit arrangements whereby they may serve the Government and still keep corporate interests intact. For iron, steel, zinc and lead arrangements similar to those made with the copper producers are now being negotiated.

Only here and there is there a discordant note with respect to this policy of self-sacrifice. But whatever be the academic questions that are involved, it is perfectly obvious that the money for carrying on war must be raised largely by taxation and the more that can be saved to the nation the less will be the taxation. The important thing to show the country, however, is that America is entering the war out of principles of honor and humanity, not to make money for anybody. Therefore the policy of self-sacrifice having been agreed upon by general consent, it will appear selfish if any producer fails to participate in what is intended to be a joint action of his industry.

It is opportune that the demand of the American Government for supplies comes at the time when the great bulk of the contracts of the Allied Governments with American makers of munitions are about to expire. This leaves the way clear for filling our own requirements and also for paying more attention to the sadly neglected needs for peaceful purposes. This does not mean that there is any slackening in the demand for raw materials, but simply that Great Britain, France, Russia and Italy are now able to do their own manufacturing.

David H. Browne

The entire brotherhood of metallurgists will be shocked and grieved to learn of the untimely death of David H. Browne. Among their ranks there was none who was more beloved. He was in truth one of the anointed. The major part of his professional career was passed at Sudbury, in the wilds of Ontario. Although everybody had heard of him and the remarkable work he was doing there, it was not until two or three years ago, when he was promoted to the head office of his company and came to New York, that we began really to get acquainted with the man. We felt sorry then that we had not had his company for many years previously, but we rejoiced in the prospect of it in the future, for he was still a young man—only 52 when he died. The entire profession grieves and pays tribute to his memory.

David H. Browne was a great man—a great metallurgist and a great citizen. As a metallurgist he is unreservedly credited with one of the major improvements of all time in the metallurgy of copper. In publishing his portrait a few months ago in a group of eight distinguished metallurgists, we described him thus: "If not the father of coal-dust firing, he was distinctly the doctor who made the idea live and grow into a major improvement in the art of metallurgy especially copper smelting."

The coal-dust firing of reverberatory smelting furnaces is something that in metallurgic and commercial importance ranks in the metallurgy of copper with the introduction of electrolytic refining and of basic converting. It was one of those fundamental innovations that reduced the cost of smelting not by a little nibble, like the minor improvements, but by a great chunk. Although Browne was not the man in whose brain this conception originated, he was distinctly the man who first did the thing successfully and made it a part of the art of smelting where other good men had failed. Browne's achievement in this was acclaimed by the entire profession of metallurgists without dissent. And it was characteristic of the man that, having done this great thing, he freely and generously communicated his knowledge to others so that they might do it too.

When a man is really great, his powers cannot be confined to any narrow channel. He may begin by being an engineer or a metallurgist, or what else may be his profession, but the very qualities that cause him to become distinguished in his profession make it inevitable that he should overflow into the broader affairs of life and become great as a citizen. Thus Hoover, and Ricketts, and Mathewson, and men like them. Brown was of their class, but it was only a few years ago, when he came to New York, that he began to have his opportunity. If anybody wants to study the man and see what we have lost, let him read two of his notable addresses, both before the Mining and Metallurgical Society of America, one on the occasion of the presentation of the gold medal to Robert H. Richards, the other last January on the subject of Hoover and the Belgian Relief.

Bread for the Belgian Children

In the death of David H. Browne, Belgian Kiddies, Ltd., has lost its trustee and the Belgian children have lost a loyal, hard-working friend.

Mining men, you who hammer the drill, push the car, handle the compass, the crucible or the pen, it is up to you to carry on David H. Browne's work. Remember that each child is asking for only 3c. a day; it is very little to you, but so very much to him. He is not asking that you and yours be stinted; indeed, he is not asking for anything, but he is pathetically grateful for the few morsels that you can well spare.

Fellow engineers, can we not do a little more to make good; to deserve the praises that those poor creatures are daily bestowing upon us? This is men's works, but we are now allowing our women folk to take it on their shoulders. Our wives are hearing the call that we men have heard but faintly. The Relief Committee of the Women's Auxiliary of the A. I. M. E. is now asking the coöperation of all women whose husbands are interested in mining, and from the progress already reported we predict a result that will both surprise and shame some of us.

To you good women who read our columns, and we know that you are many, to you we extend our encouragement, and we feel confident that your united efforts will bring results equaling if not surpassing those attained by your husbands, too busy to think of this but as a petty charity.

As an example of what the women are doing we cite only one instance. The wife of a member of the A. I. M. E., who felt that one share of Belgian Kiddies, Ltd., was all that he could buy, attended the meetings of the Institute in February, met Hoover and Honnold and the others, heard their stories and read the appeals sent to her husband. "I must do my bit," she said, and, like our friend Hoover, started to work, leaving the formalities to be carried out later. She started with a card party with a prize for every player, the privilege of contributing to the charity of which she told them. For working material she needed no more than the address of David H. Browne, delivered before the Mining and Metallurgical Society of America, and printed in our issue of Mar. 24, 1917. Her friends who heard the story invited their friends to small functions, the village churches and the school children did their bit. The result of the efforts of this one little woman and her 50c. campaign was a square meal for 20,000 children and still more meals to come.

Fellow mining men we will have to look to our laurels; we have spent too much time looking for high-grade pockets as the only source of gold. Our wives may prove to be the better miners; they are proving that large, low-grade deposits, carefully worked, are remunerative.

Mrs. Axel O. Ihlseng, secretary of the Women's Auxiliary of the A. I. M. E., 29 West 39th St., New York, will gladly furnish information respecting the relief work of the Auxiliary.

The Commision for Relief in Belgium has received a cable announcing the arrival of Hoover in Europe and he reports that the reorganization of the Commission's membership in Belgium is being effected rapidly and without hitch, and that there will be no interference with the food supply.

Our Special Mining Number

We call this issue a special mining number. Note the qualifying adjective "special." Every issue of the *Engineering and Mining Journal* is a mining number as well as a metallurgical. But now and then we draw the line between the miners and the metallurgists and publish a number devoted exclusively to their special interests. Thus six months ago we issued a special number for the smelters and three months before that one for the mill-men.

This special mining number comprises a noteworthy collection of contributions upon subjects of major interest to the mining engineer, mine manager and mine superintendent. The entire list of contributors will be recognized as consisting of men who know what they are talking about. Underground methods, including drilling, stoping, transportation, pumping and hoisting are all treated in an authoritative way, and the reader may feel sure that he is presented with an exposition of the latest practice. This issue of the *Journal* is one to be preserved in a separate file.

Mineral and Metal Statistics

It has been brought to our attention that the work of the Division of Mineral Statistics of the United States Geological Survey in reporting the present status of mineral production and of the mineral industry is being hindered by the failure of some producers to make returns. We ourselves are experiencing the same trouble in certain of our own statistical investigations, more limited and more special in nature than those of the Survey. The experience and condition is like this: There are 40 producers of zinc, let us say. The majority of them return the blanks properly filled in a few days after they are issued. The remainder dribble in after prolonged delay and after two or three or four stirring-up letters. Finally there may be only one or two reports missing. The summing up and reporting of results has to wait until they can be secured by importunity and at considerable expense for telegraph tolls and even for personal interviews.

Such delinquency is unfair to everybody—to the self-sacrificing statistician who is simply trying to render a service, to the majority of fellow-producers who have promptly made their returns, and to the whole industry which needs the report. We shall not venture to indicate all the reasons for it. In some cases it is attributable to the inattention of clerks and subordinates who do not appreciate the importance of the matter, probably regarding it as being merely one other example of a pestering nuisance. Our own annual report on zinc production, etc., has always been highly valued in the industry. In former times—a long while ago—we used to be bothered by the chronic tardiness of one important concern. One year we received a letter from its president, saying:

"When in thunder are you going to publish those spelter statistics of yours?"

It happening that his company's was the only missing report, we replied:

"Just as soon as you communicate your figures."

They came by return mail, and that company has ever since been among the promptest.

At the present time the Geological Survey is being called upon continually by the Army and Navy Departments to furnish accurate information as to the mineral resources of the United States. It is especially desirable to have exact information in regard to the supplies of minerals that are in unusual demand. It is very important to secure prompt information in regard to certain materials that are known not to be plentiful as domestic supplies. It is believed, moreover, that the possession of full information will be of importance in the readjustment of the mineral industry after the war.

We think we do not overstate the case when we say that it is the view in Washington that compliance with requests for statistical information may be construed as a patriotic duty.

Sappers and Miners

We urge mining men not to be too quick in volunteering for the army. Wait until there is a special call for recruits for mining and tunnelling companies. The same advice may be extended to all men engaged in engineering pursuits, but we address the miners particularly. We should profit from the experience in Great Britain. When it was found there that sappers and miners were needed it was found also that many of the experienced men were already at the front with the infantry and cavalry and it was hard to get them out. To our own people, therefore, we reiterate the advice to wait until sappers and miners are called for.

Up to last Tuesday evening the subscriptions to Belgian Kiddies amounted to $83,000. Mining engineers have yet a lot to do to complete the fund in honor of Hoover.

BY THE WAY

The price of copper to the investor in copper securities is an enigma, remarks the New York *Commercial* some weeks ago. He sees the Wall Street newspapers reporting copper as selling at 35c. and 36c., and on Saturday at 37c., all successively new record highs since 1873. He begins to figure profits on the basis of at least 35c. copper. He is fortified in his figures by so-called market letters of more of less irresponsible Curb brokerage houses who, assuming that they ever get to Heaven, would exaggerate the delights of Kingdom Come. The habit of exaggeration gets to be part of the grain of Wall Street life. It so happens that the bulk of the copper metal now being sold is fetching between 32@33c. per pound cash f.o.b. New York. On Thursday the Shannon Copper Co. reported sales of 400,000 lb. March-to-June delivery at 33¼c. terms and destination not stated. All the copper now being delivered and which will be delivered to July 1, at least, will average under 30c. per pound. A very large part of the output was contracted for at lower prices in 1916. One must not assume that reported sales at 35 @37c. are false. Such sales have doubtless been made, but evidently in small lots, so that they must not be taken as any criterion on the average price of copper. These sales, as also the so-called "quotations" (the asked price of the producers), are considerably above the average price received "net cash f.o.b. New York." The prices received by the copper companies usually correspond very closely with the averages reported by the *Engineering and Mining Journal* and are on the basis of net cash, New York delivery. The figures thus reported are based on actual sales of the metal, and not merely on quotations or asked prices, and still less on sporadic or exceptional sales.

A comprehensive survey of the Jerome copper district, in Arizona, is shown in "The Secret Spring," a motion picture recently made by the Yorke-Metro Co., at Jerome. The scenes made are the first to be taken of this district, and also are the first to show the copper-smelting process. The scenes around the mines and at the Clarkdale smeltery were taken by permission of the officials of the United Verde mines. The scenes around the mining district which are to be included in the picture consist of panoramic views in the vicinity of the mines and in the smeltery. The picture should give to those interested in this district a good survey of the various Jerome properties.

There are many dangers in sampling old mines besides that of getting an inaccurate sample, and the necessity of the mining engineer practicing "safety first" is emphasized by the experience of Albert G. Wolf, of Denver, while engaged in sampling the Hoosier mine in the Cripple Creek district. With one assistant, Samuel McAllister, he had proceeded into the mine and descended the ladder just above the sixth level. McAllister reached the level safely and stepped to one side of the ladderway. Wolf reached a point about 10 ft. above the level, when the section of ladder on which he stood pulled loose and fell. He crashed through the platform at the sixth level, dropped 25 ft., crashed through the next bulkhead in the manway and fell another 25 ft. to the next bulkhead, which did not give way. The bulkheads were old and dry. McAllister could not descend because the timbering was unsafe and the ladders were all swept out during the fall, which was a total distance of about 60 ft. After a brief conversation with Wolf, who was unconscious for only a brief interval, McAllister went for help having to climb up 600 ft. to get out of the mine, the hoist being out of commission. About two hours after the fall, a doctor was lowered on a rope, gave Wolf first aid and made him as comfortable as possible pending the arrival of a rescue party from the Portland mine under the leadership of Fred Jones. Both Frank Willis and Fred Jones rendered efficient help in getting the injured man out of the shaft, which operation was performed under most inconvenient and trying circumstances. He was raised to the sixth level by a rope about his body, and from there to the surface on a stretcher, one level at a time the stretcher being steadied in the shaft while the men above pulled by hand. The patient was landed in the hospital at Victor just eight hours after the accident. An examination showed a compound fracture of the left tibia, a simple fracture of the left fibula and a simple fracture of the right clavicle. Wolf will be confined to the hospital for about four weeks longer, though he is recovering at maximum speed.

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A correspondent wrote us recently:

I shall appreciate it if you will advise me about what percentage of the copper produced in the world since the beginning of the war has been used directly or indirectly in the manufacture of munitions of war. I have heard somewhere that the percentage is 90, and that only 10% is being used for industrial purposes. This refers to the total copper produced in the world, not in the United States alone.

We answered as follows:

Replying to your letter of Feb. 26, we have no statistics which would enable anybody to determine what percentage of copper produced in the world has been used in the manufacture of war munitions. Also, we do not know where such information can be obtained.

Our correspondent came back with:

I am in receipt of your favor of Mar. 6, in which you state that you have no statistics which would enable you to estimate what percentage of the copper produced in the world has been used for the manufacture of munitions of war since the war begun. It seems strange to me that the "Journal" should not be able to give an estimate of the percentage of the copper produced in the world since the war began that has been used in the manufacture of munitions, which of course includes all sorts of warships. It does not speak well for your paper if you cannot give a rough estimate of what this proportion is. I have written some other papers for this information.

We, ourselves, would like to have the information that our correspondent wants, and if he can get it from "some other papers," we shall be truly pleased. We do not know how the statistics are to be had except from the munitions departments of the ministries of Europe. We have been unable to get *any* statistics from them—even on what look to us like perfectly innocent subjects.

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Some months ago it was suggested that any hegira on the part of mining companies from the warring countries to avoid taxation would probably be met by measures forbidding such a change or altering its advantages. This has come about in Great Britain. Aramayo Francke Mines, Ltd., which had decided to remove its headquarters from London to Geneva, Switzerland, has altered its plans It will remain a British company, but the business in Bolivia will be entirely under the control of a local board, to the exclusion of the directors in London. This will not be a marked change in the actual transaction of the company's business in Bolivia, which has long been in the hands of Malcolm Roberts and staff. The former chairman of the company, F. Avelino Aramayo, under the new régime will act as chairman of the local board, his colleagues being Malcolm Roberts, Carlos Victor Aramayo and Carlos Navarro. The directors in England will retain no power of interfering with the business in Bolivia "either by way of supervision, direction or participation," that retention of interference rendering the business liable to be charged with being carried on partly in England, in which case the whole of the profits made in Bolivia would be liable to taxation, and not simply that part remitted to England for British shareholders. In the proceedings brought by the British Board of Trade against permitting the change of the company's headquarters to Switzerland, it was argued that such a change at this time might permit the products of the company to fall into the hands of enemies of the empire. The reason for the original plan was to escape the double taxation of shareholders residing outside of England, who under the old régime had to pay taxes in Great Britain and were also liable to taxation in the place of their abode. The Bolivian board will in the future remit to England only such sums as shall be necessary to pay dividends to stockholders in the United Kingdom and the expenses incurred by the London board

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March Mining Dividends

Dividends paid in March, 1917, by 45 United States mining and metallurgical companies making public reports amount to $36,484,228, as compared with $27,109,-803 paid by 44 companies in March, 1916. Holding companies paid $943,433, as compared with $197,485 in 1916. Canadian, Mexican and South American mining companies paid $2,416,887 in March, 1917, and $1,922,-642 in March, 1916.

Total dividends in the first three months of the year paid by United States mining and metallurgical companies, amount to $62,756,962; by holding companies, $2,283,433; by Canadian, Mexican, South and Central American companies, $6,243,086.

United States Mining and Metallurgical Companies	Situation	Per Share	Total
Am. Sm. and Ref., com	U. S.-Mex.	$1.50	$814,485
Am. Sm. and Ref., pfd	U. S.-Mex.	1.75	875,000
Argonaut Cons., g	Calif.	.10	20,000
Bunker Hill & Sullivan, l.s	Ida.	.50	163,500
Butte & Superior, z	Mont.	2.50	725,492
Caledonia, l.s	Ida.	.03	78,150
Calumet & Ariz., c	Ariz.	3.00	1,927,386
Calumet & Hecla, c	Mich.	25.00	2,500,000
Centennial, c	Mich.	1.00	90,000
Champion, c	Mich.	6.40	640,000
Chino, c	N. M.	2.50	2,174,950
Cons. Interstate-Callahan, l.s.z	Ida.	1.00	464,990
Copper Range, c	Mich.	2.50	987,500
Cresson, g.s	Colo.	.10	122,000
Electric Point, l	Wash.	.01	7,938
Empire Cop	Ida.	.05	50,000
Federal Min. & Sm., pfd	U. S.	1.75	210,000
Gemini, s.l	Utah	5.00	25,000
Golden Cycle, g	Colo.	.03	45,000
Hamburg, g. s. l	Nev.	.01	8,000
Hecla, l.s	Ida.	.15	150,000
Homestake, g	S. D.	.65	163,254
Internat. Nickel	U. S.-Can.	1.50	2,510,076
Kennecott, c	......	1.50	4,170,874
Magma, c	Ariz.	.50	120,000
Mammoth, g.s.c	Utah	.10	40,000
Mogollon Mines, g.s	N. M.	.05	17,784
Nevada Con., c	Nev.	1.00	1,999,457
New Idria, q	Calif.	1.00	100,000
New Jersey Zinc	U. S.	4.00	1,400,000
North Star, g	Calif.	.50	125,000
Oaks Co., pfd., g.s	N. M.	....	169
Old Dominion, c	Ariz.	3.00	880,059
Oroville Dredging	Calif.	.12	82,385
Phelps, Dodge & Co	U. S.-Mex.	6.00	2,700,000
Quincy, c	Mich.	5.00	550,000
Ray Con., c	Ariz.	1.00	1,577,179
St. Joseph Lead	Mo.	.75	1,057,098
Silver King Con., s.l	Utah	.15	103,287
United Cop. Min	Wash.	.01	10,000
United Verde, c	Ariz.	1.50	450,000
Utah Con., c	Utah	1.00	300,000
Utah Copper	Utah	3.50	5,685,715
Yak., s.l	Colo.	.07	70,000
Yellow Pine, z.l	Nev.	.03	30,000
Yukon Gold, g	Alas.	.07½	262,500

Canadian, Mexican and So. American Companies	Situation	Per Share	Total
Can. Min. Corpn., s	Ont.	$0.37½	$622,520
Cerro de Pasco, c	Peru	1.50	999,999
Dome Mines, s	Ont.	.50	200,000
Hedley, g	B. C.	.50	60,000
Kerr Lake, s	Ont.	.25	150,000
Le Roi No. 2, g.s.c	B. C.	.24	29,160
Lucky Tiger-Combination, g	Mex.	.09	64,380
Mexico Mines of El Oro	Mex.	1.68	282,400
Right of Way, s	Ont.	.005	8,428

Holding Companies	Situation	Per Share	Total
Exploration Co	Mex.	$0.24	$180,000
General Dev	U. S.	2.00	240,000
St. Mary's Min. Land, c	Mich.	2.00	320,000
Yukon-Alaska Trust	......	1.00	203,433

The first quarter of 1917 showed a considerable increase over the corresponding period last year.

Personals

D. Ernest Marquardt is in Wyoming.

C. T. Griswold is in southern Oklahoma.

S. F. Shaw has returned to San Antonio, Tex., from Monterey, Mexico.

W. H. Wright, of The Malm-Wolf Co., is making mine examinations near Bonanza, Colo.

F. G. Clapp, managing geologist of the Associated Geological Engineers, has gone to Arkansas.

A. T. Thomson, of the Phelps-Dodge New York office, visited the company's plant at Morenci last week.

A. V. Dye, secretary to Walter Douglas, visited the plant of Phelps, Dodge & Co. at Morenci last week.

O. F. Heizer is manager for the Louisiana Consolidated Mining Co., at Tonopah, Nev., reopening the old Tybo mine.

J. M. Hill, of the Geological Survey, has returned from a four weeks' inspection trip of the bauxite fields of Georgia and Alabama.

Frank McLean, superintendent of the Detroit Copper Mining Co., visited the plant of Phelps, Dodge & Co. at Nacozari recently.

J. P. Bickell, president of the Schumacher mine, with a party of officials and others recently paid a visit of inspection to the property.

George T. Fletcher, for a long time manager for the United Zinc Co., Joplin, Mo., has resigned, and is succeeded by Fred L. Crouch.

V. F. Marsters is investigating a section in eastern Kansas for oil. His temporary address is 1218 Colcord Building, Oklahoma City, care of Chas. N. Gould.

Professor C. K. Leith, of the University of Wisconsin, has recently completed a six weeks' course of lectures on metamorphic geology at the University of Chicago.

Karl G. Roebling, treasurer of John A Roebling's Sons Co., Trenton, N. J., has been elected a director of the Otis Elevator Co. to succeed F. W. Roebling, deceased.

Capt. Stewart Thorne, formerly manager of the Trethewey mine, Cobalt, who was fighting in France, has been decorated by Gen. Neville with the French Croix de Guerre.

Harry E. Dennie, formerly representing the Imperial Belting Co., at Salt Lake City, has become Western manager for the company, with headquarters at 525 Market St., San Francisco.

John B. Stewart, in company with Jesse C. Porter, is now in charge of the Havana branch of the C. L. Constant Co., Calle Cuba, No. 74. The Havana branch is a permanent establishment.

R. S. Rainsford is to undertake the management of the Senorito Copper Corporation Senorito, N. M., in collaboration with W. E. Greenawalt and John T. McLaughlin, according to a recent press report.

C. H. Scheuer has resigned as chief engineer of Fayal district of the Oliver Iron Mining Co., a position he filled for eight years, and will undertake general consulting practice with headquarters at Crosby, Minn.

Dr. W. A. Lynott, mine surgeon of the Bureau of Mines, is in Washington giving first-aid instruction to the female employees of the Bureau of Mines and to the wives and daughters of the male members of the Bureau's staff.

Simon Guggenheim of New York, in company with E. L. Newhouse, first vice-president of the American Smelting and Refining Co., and Frank H. Brownell, president of the Federal Mining and Smelting Co., passed through Butte on Mar. 25, while on an inspection trip to the smelter plants of the companies, in which they are interested. After a visit to the East Helena smeltery, the party left on the morning of Mar. 26, for Salt Lake.

Obituary

Joe P. Sullivan, a miner at the Anaconda Copper Mining Co.'s Bell mine, in Butte, fell 70 ft. to his death down a chute on the 1800 level on the afternoon of Mar. 22.

John Balatino and Joe Rose, leasers at the old Hecla mine in Beaverhead County, 16 miles above Melrose, were killed on Mar. 27, by being caught in a cave-in at that mine. How the accident happened is unknown as the men were both dead when dug out by a rescue party that started for them when they failed to show up at the boarding house at the accustomed time.

Fred R. W. Thomas, age 37, resident of Seattle for three years, died Mar. 22, at the Providence Hospital, Seattle, Wash. Mr. Thomas lived formerly in Roslyn, where for seven years he was engineer for the Northwest Improvement Co. He had offices in the Alaska building. Prior to his residence in Roslyn, Mr. Thomas was employed by the same company as superintendent of its coal mines at Ravensdale.

David H. Browne, chief metallurgist for the International Nickel Co., died on Friday, Mar. 30, at his home, 104 Gates Ave., Montclair, N. J. Mr. Browne was one of the best known and loved men in the mining profession. His activities in metallurgy, in addition to his interest in the affairs of the American Institute of Mining Engineers and the Mining and Metallurgical Society of America, made him a host of friends, to whom his personal qualities greatly endeared him. His funeral was held on Monday, Apr. 2 Mr. Browne is survived by his wife and three sons. He was born at Hollymount, County Mayo, Ireland, 53 years ago. A more extended notice will be published later.

Societies

Mining and Metallurgical Society of America—A meeting of the Society has been called for Apr. 19, at the Engineers' Club, preceded by the usual informal dinner at 6:30 p.m., to consider what it can and should do to assist the government of the United States in the present national crisis. Every member is urged to attend if he can possibly do so.

Engineers' Society of Western Pennsylvania will hold a meeting of its mechanical section at the Society Rooms, in Pittsburgh, on Tuesday, Apr. 3. The principal subject will be "Cotton Rope for Power Transmission," a paper upon which subject will be read by J. M. Allison, of William Kenyon & Sons, Dukinfield, England. The meeting will be preceded by a dinner at the William Penn hotel.

American Gear Manufacturers Association was formed at Lakewood, N. J., at a meeting held there Mar. 25 to 27. Its purposes are to advance and improve the gear industry, and to standardize gear design, manufacture and application. Officers elected were: President, F. W. Sinram; Vice-president, H. E. Eberhardt, Secretary, F. D. Hamlin; Treasurer, Frank Horsburgh. These with Biddle Arthur, Geo. L. Markland, and Milton Rupert form the executive committee.

Southwestern Society of Engineers was organized at a convention held in El Paso, Tex., on Mar. 8, 9 and 10, with more than 100 charter members. Membership is open to civil, mechanical mining, electrical, or chemical engineers, or architects or other persons belonging to a technical profession, who are not less than twenty-seven years of age, and who have been in active practice of their profession for at least six years. Provision is also made for Associated, Honorary, and Affiliated Members. The great distance from centers of population makes it difficult for southwestern engineers to attend meetings of the national engineering organizatons, so it is believed that the new Society will fill a real need. It is planned to hold at least two conventions of the Society each year for the reading and discussion of professional papers and for social intercourse. At the first convention the following papers were read and discussed: "The Purpose of Engineering Education," by Dean G. M. Butler, College of Mines and Engineering, University of Arizona; "Some Lessons Taught the Mining Industry of the Southwest by Present Activities and Prices," by Gerald Sherman, Mine Supt. of the Copper Queen Consolidated Mining Co., Bisbee, Ariz.; "Engineering and National Defense," by Lieut.-Col. M. L. Walker, Corps of Engineers, U. S. Army, Chief Engineer of the recent Punitive Expedition into Mexico; "Modern Highways and the Dividends They Pay," by J. L. Campbell, Chief Engineer of the E. P. & S. W. R.R. The officers of the Society are: President, A. F. Barnes, dean of the School of Engineering, New Mexico College of Agriculture and Mechanic Arts; vice-president for 2-year term, G. M. Butler, dean of the College of Mines and Engineering, University of Arizona; vice-president for 1-year term, S. H. Worrell, dean of the Texas College of Mines; secretary, Forrest E. Baker, El Paso, Tex.; treasurer, R. W. Goddard, professor electrical engineering, New Mexico College of Agriculture and Mechanic Arts.

Industrial News

Deister Concentrator Co. has just closed a contract with the American Zinc Co., Mascot, Tennessee, for 56 Deister-Overstrom tables.

Astra Electric Novelty Works has been incorporated to manufacture dry batteries for domestic and foreign trade. The address is 45 E. 17th St., New York.

The Lunkenheimer Co., Cincinnati, Ohio, made a large number of valves for special purposes last year, such as iron valves for cyanides and other solutions attacking copper and brass, and nickel-iron composition valves for acid solutions.

U. S. Manganese Corporation, 74 Broadway, New York, is mining manganese ore at Elkton, Va., and shipping it to the Temple furnace near Reading, Pa. The ore is reported to average over 45% manganese. This furnace, which is controlled by the Seaboard Steel & Manganese Corporation, started to produce spiegeleisen about Feb. 15. Interests affiliated with these companies are also mining an ore averaging about 30 to 35% manganese, at Vesuvius, Va., 60 mi. north of Elkton. This ore is being sent to the same blast furnace. These ores, with others, will be first converted into spiegeleisen and later ferromanganese will be the product. The company has already a large quantity of ore on the ground.

Trade Catalogs

Electric Fans. Sprague Electric Works, 527-532 W. 34th St., New York. Form No. B-3409. Pp. 22; 3½ x 6 in.; illustrated.

Wetherill Magnetic Separator. The Stearns-Roger Mfg. Co., Denver, Colo. Catalog 100, Section 110. Pp. 10; 7½ x 9 in.; illustrated.

Excavator Buckets. The Hayward Co., 50 Church St., New York. Catalog No. 42. Pp. 80; 6 x 9 in.; illustrated. Describes the various types of excavator buckets and shows the class of work for which they are especially adapted.

Imperial Duplex Dry Vacuum Pumps. Ingersoll-Rand Co., 11 Broadway, New York. Form No. 3038. Pp. 24; 6 x 9 in.; illustrated. Complete details of design and construction with dimensions, horsepower required and other data.

Ingersoll-Rogler Straight Line Dry Vacuum Pumps. Ingersoll-Rand Co., 11 Broadway, New York. Form No. 3037. Pp. 24; 6 x 9 in.; illustrated. Full particulars of construction and sizes together with dimension drawings indicative of economy in floor space with this type.

Single-Phase Motors, Varying Speed. Sprague Electric Works, 527-531 W. 34th St., New York. Bulletin No. 41514. Pp. 12; 8 x 10½ in.; illustrated. Variation in speed and alteration of direction of rotation, water push button control, of motors of ⅙ to 7½ hp., are points described.

Apparatus for Chemical and Allied Industries. J. P. Devine Co., Buffalo, N. Y. Bulletin No. 105. Pp. 32; 4 x 5½ in.; illustrated. Details of construction of autoclaves; reduction, fusion, and tilting kettles; vacuum pans and evaporators; dyesters and steam-jacketed valves and pipes are given.

New Patents

United States patent specifications listed below may be obtained from "The Engineering and Mining Journal" at 25c. each. British patents are supplied at 40c. each.

Aluminum—Production of Metallic Compounds from Carbides. Paul R. Hershman, Chicago, Ill., assignor to The Mineral Products Co., New York, N. Y., (U. S. No. 1,220,843; Mar. 27, 1917.)

Cement—Water and Acid Proof Cement and Process of Making Same. Utley Wedge, Ardmore, Penn., assignor to Electro-Chemical Supply and Engineering Co., Philadelphia, Penn. (U. S. No. 1,220,575; Mar. 27, 1917.)

Flotation—Apparatus for Separating Ores by Flotation. Herbert E. T. Haultain, Toronto, Ont. (U. S. No. 1,218,400; Mar. 6, 1917.)

Gold Dredge. John T. Cowles, Chicago, Ill. (U. S. No. 1,220,197); Mar. 27, 1917.)

Haulage—Truck Blocking Device. Peter J. Cunningham, Battle Mountain, Nev. (U. S. No. 1,220,322; Mar. 27, 1917.)

Hydrochloric Acid—Manufacture of Hydrochloric Acid. James B. Garner and Howard D. Clayton, Pittsburgh, Penn., assignors to Metals Research Co., New York. (U. S. No. 1,220,411; Mar. 27, 1917.)

Hydro-Metallurgy—Process of Extracting Metals from Their Ores. William E. Greenawalt, Denver, Colo. (U. S. No. 1,218,996; Mar. 13. 1917.)

Iron Metallurgy—Puddling Iron. Joseph Ernst Fletcher, Dudley, and James Harrison, Tipton, England. (U. S. No. 1,220,081; Mar. 20, 1917.)

Lamp. John T. Jones, Ernest, Penn. (U. S. No. 1,218,185; Mar. 6, 1917.)

Leaching Flue Dust. Frederick W. Huber and Frank F. Reath, Riverside, Calif., assignors to William G. Henshaw, San Francisco, Calif. (U. S. No. 1,220,989; Mar. 27, 1917.)

Oil Refining—Process and Apparatus for Continuously Distilling Mineral Oils and the like. Carl Heinrich Borrmann, Essen-on-the-Ruhr, Germany. (U. S. No. 1,220,067; Mar. 20, 1917.)

Ore Pulverizer. Alfred Molander, Minneapolis, Minn. (U. S. No. 1,220,257; Mar. 27, 1917.)

Roasting Furnace—Gilbert Rigg, Palmerton, Penn., and Walter L. Coursen, New Rochelle, N. Y., assignors to The New Jersey Zinc Co., New York. (U. S. No. 1,220,789; Mar. 27, 1917.)

Sheave—Mine Sheave. John N. Derrick and Millard M. Vann, Rockwood, Tenn. (U. S. No. 1,220,963; Mar. 27, 1917.)

Sulphuric Acid—Manufacture of Sulphuric Acid. William J. Kee, Jr., Kansas City, Kan., and Utley Wedge, Ardmore, Penn. (U. S. No. 1,220,752; Mar. 27, 1917.)

Editorial Correspondence

SAN FRANCISCO—Mar. 28

Another Double-Stacker Dredge is to be added to the Natomas fleet in the American River district, by rebuilding Natoma No. 5. The old dredge is still digging, but will finish its ground in the early part of April. Meantime the designing and building of new machinery and other parts necessary are in progress and a large part of the hull has been framed. It is to be a composite dredge made up of wooden hull and steel trusses. The tailing equipment will be composed of two belt conveyors and two tail sluices, similar to Natoma No. 4 and embracing the improvements of Natoma No. 1, which has four stackers. There is some slight variation from the designs of No. 4 and No. 1, which contemplates a further advance in the building of a satisfactory resoiling dredge. Both the other dredges are doing the work of resoiling satisfactorily, but both of them have undergone slight changes in methods of disposing of the tailing. For the No. 5 dredge, there will be a considerable area of ground adjoining that already dug. This dredge was installed in December, 1905, equipped with 9-cu.ft. buckets, to dig at Rebel Hill, in the hardest part of the American River district. On June 8, 1913, the dredge was sunk and was righted and ready for repair on July 15, as described in the "Journal" of Oct. 25, 1913. The present work of rebuilding and construction is being done by the shops and crews of the Natomas Co. of California, under the direction of L. D. Hopfield, district manager.

Mining Advancement in Southern Inyo is largely due to the progress of the Cerro Gordo Mines Co. during the last two years. The demands of that company upon the railroad facilities terminating at Keeler have encouraged other mine owners and operators to make similar demands for the shipment of ores and concentrates. The fact that the requirements of the mines have exceeded the facilities of the railroad has not deterred the miners from proceeding with development and production. They are acting upon the theory that the surest inducement for the Nevada & California and the Southern Pacific railroads to increase the present equipment of rolling stock is to continue to increase the demand. The metal mines in this region are contiguous only to the station at Keeler and many of the shipments go out of the state through Nevada to the smelteries at Salt Lake and Oklahoma. And the initial point for shipment either way, to the north and east or to the south and west, is Keeler; so that if the Nevada & California road does not provide the rolling stock to move the products of the mines from Keeler to the junction at Owenyo, the open way to Los Angeles and San Francisco might just as well be closed. But the Southern Pacific could remedy this by the extension of a branch line for a distance of about 20 miles around the southeast side of Owens Lake from Olanche to Keeler, over an easy grade. The Cerro Gordo mine has the advantage of being equipped with a bucket tram and the shipments of ore are limited only by the number of cars available. Santa Rosa mine, situated between the Cerro Gordo and the Darwin, has a wagon or motor truck haul of 25 miles or more and the Darwin about the same distance. The former ships sacked ore, while from Darwin the shipments are chiefly concentrates and the wagon or truck movement is always ahead of the car supply. Add to these shippers the smaller mines in the Darwin and Santa Rosa districts, the Troeger Tunnel and other producers in the Cerro Gordo district, the two soda and one salt works, the talc plant and the new soda works under construction, with the new development in progress all through the Owens Lake region, and there is evidently enough railroad shipping business in sight at Keeler to urge at least a greater effort to move the accumulating freight. The shipments are constantly increasing, but the demand is still in excess of the available cars. The mine operators are employing animal teams and motor trucks and caterpillar tractors. Just now, however, the roads are not well adapted to motor-truck hauling, so the animals and the tractors are handling most of the loads. There are about 100 head of stock and five 10-ton caterpillars and one larger one on the roads. Keeler is a busy town—a large spot on the map of southern Inyo.

BUTTE—Mar. 29

Sanitary Conditions in Butte Mines are commended by Inspectors D. J. McGrath and W. D. Orem, working under the industrial accident board of Montana. The inspectors have been on a tour investigating working conditions in the Butte mines for the last three weeks. In their report to the board they refer to the many improvements introduced in these mines tending to make conditions not only more comfortable but far more sanitary than they were in years past. As a particular instance they mentioned the ventilation system installed at the Speculator mine of the North Butte Mining Co., consisting of canvas tubing through which fresh air is pumped to all working places. In the Mountain View, Pennsylvania, Anaconda and Berkeley they likewise noted great improvement in ventilation by increasing the number of air shafts and mechanically driven fans. Drinking water for the men is kept cool in many of the mines by various devices in which the water does not come into direct contact with the ice and is therefore more sanitary and healthful. This week the inspectors are visiting the mines known as the Anaconda group of the Anaconda company.

DENVER—Mar. 29

Flotation Oil from De Beque Shales is a possibility now being investigated in a research conducted at the Colorado School of Mines. James M. Hyde says he has proved that such oil is well adapted to flotation and the question is whether or not it can be produced commercially at suitable cost. These shales produce, when distilled, kerosene and gasoline as well as numerous heavy oils and some ammonium sulphate. Estimates place the expense of quarrying and treating this material at about $2 per ton.

Welfare of Employees goes beyond "safety first." Large employers are making their employees feel contented in their occupations by extending the comforts of pensions and life insurance. The recent announcement of the American Smelting and Refining Co.'s new system of life insurance is highly appreciated by the many employees in its Colorado plants. The policies were issued without solicitation and without expense to the men, and without medical or physical examinations. The Colorado Fuel and Iron Co. has come forward with a system whereby any man of 65 or any woman of 55 who has served the company for 20 years is automatically retired upon a pension amounting to 30% of the average salary or wage paid to the individual during the preceding 10 years. Continuity of service is not required in reckoning the 20-years' employment, a feature distinguishing this from other industrial pension schemes. With the approval of the president, same benefits may be applied to men of 60, or women of 50, while special provisions are being arranged for employees with records of but 15 years' service.

SALT LAKE CITY—Mar. 29

Abandonment of Park City Sampler was announced on Mar. 26 by the Utah Ore Sampling Co. The company's newly remodeled plant at Murray—capacity 1000 tons a day—together with the plant at Silver City, will take care of the ore to be sampled in this section. The Park City plant, which had a daily capacity of about 300 tons, will be dismantled, and much of the equipment used at other points. The Utah Ore Sampling Co. recently completed a plant at Jean, Nev., to serve the Goodsprings section.

Utah Copper's Tailings Pond at Garfield is increasing rapidly in extent, and there is danger of its encroaching upon the tracks of the San Pedro and the Western Pacific railroads, so that it may be necessary for these lines to move the tracks to higher ground. The railroad companies have recently filed condemnation proceedings in the third district court against land owned by J. L. Wilson, so that the roadbeds may be changed. The rights of way desired are south of their present course, along the benches, where the Arthur and Magna mills are situated.

Fire in Utah Apex Mine at Bingham, starting Mar. 24 on the 1300 level, has caused almost complete suspension of operations at this property. The fire was due to decomposing ores, which later started the timbers burning. At present most of the ground affected lies between the 1100 and 1300 levels. No fatal accidents occurred, although several employees were overcome by fumes. When it was found on Mar. 27 that the fire was beyond control, efforts were made to remove pumps and hoisting machinery. Later the management began to pump water into the workings from Carr Fork Creek, and with this water as well as the mine water coming in, the mine is being rapidly flooded. There are 500 ft. of workings below the 1200 or tunnel level. It is estimated that from three to ten weeks may elapse before the mine can be put in condition for resuming work, as it will take some time to unwater the mine after the fires are put out. Also it may be necessary—after the fire in the ore has been stopped and the water pumped out—to bulkhead that part of the mine, which was the source of the fire in order to prevent a recurrence from spontaneous combustion. About 450 men were laid off temporarily; and lessees working above the 1200 were driven out.

WALLACE, IDAHO—Mar. 26

Mining Bureau Bill Vetoed by Governor Alexander. The proposed bureau of mines and geology was to have been established in connection with the mining department of the state university and conducted in coöperation with the U. S. Bureau of Mines. Much resentment is felt by the mining interests over the action of the governor, who justifies his veto by saying that Idaho is not ripe for such a bureau. On the contrary, the opinion is held by mining men generally, and has been voiced repeatedly in the annual reports of State Mine Inspector Bell, that a state bureau is the most pressing need of the mining industry in Idaho. The state has a vast mineral area covering rugged and remote sections, much of which would no doubt prove attractive to capital if the facts were properly presented. Another case in which the mining interests suffer through the narrow-mindedness of a chief executive, who makes the demagogic plea of economy to curry favor with his constituents in the agricultural sections.

Restraining Order Against the Federal, as actually issued by Judge Deitrich, differs somewhat from the first published report. The Federal company is forbidden to work in the Star ground from the five shaft levels, but may continue on and above the 800, which is identical with the main working tunnel. All ore taken out in pursuance of this order must be milled separately in what is known as Morning mill No. 2 at Wallace and an accurate account rendered of all ore extracted and of the operating costs. The concentrates must be shipped under the Federal contract with the American Smelting and Refining Co., but the Federal company is to deposit the difference between the returns received under its contract and what would have been received had the ore been shipped under the contract held by the Hecla Mining Co. with the International smeltery. Federal may file a bond covering this difference in lieu of the cash. Each of the mining companies is given the privilege of working in the ground of the other for the purpose of proving their respective contentions when the case comes on for trial.

TONOPAH, NEV.—Mar. 23

Tonopah and Manhattan were never in better shape than they are today. In Tonopah scarcely a week has passed lately that has not marked the opening of a new orebody. Recently the Monarch Pittsburgh encountered a vein that has been looking better after every round; this property will now probably enter the producing class, after a long period of assessments, uphill work and discouraging results. Development on the Cash Boy strike indicates that it will soon be in the class of steady shippers. Lately the Belmont cut an orebody that outclasses anything since the Belmont vein was discovered; this orebody has been opened for 20 ft., with the sides, bottom and top in ore, and the hanging wall has not yet been cut. Belmont is operating two diamond drills away out on the east end, and activity in the western section of the district is likewise extending the proven ore-bearing area. In the southern part of the district the Jim Butler is also doing diamond drilling. Tonopah breathes, radiates, and perspires prosperity. New stores are opening up, and the stock markets are doing almost a boom-camp business. People are making improvements on their houses as if they intended to settle down here for life, and nearly every "Bohunk" owns a flivver. While the increased price of silver has heartened everyone, the prosperity of the district rests on the splendid condition of the mines. In Manhattan, the White Caps Mining Co. is doing excellently under the new management, and following the recent discoveries the stock is going up "to beat the band" and taking all other Manhattan stocks with it. There is real mining activity in the district, however, such as has not existed there for a number of years.

MORENCI, ARIZ.—Mar. 30

Many Small Operators are at work in the surrounding country, this condition being something of a novelty for this district. Approximately 300 leasers and claim owners are exploring vigorously and shipping smelting ore, via burro backs, with all the burros that can be commandeered in the neighborhood.

Weekly Conferences of employees of the Detroit Copper Mining Co. are being held regularly. These are of a business and social nature and promise good results. Addresses are usually made by the superintendent, heads of departments and other engineers on different mine, mill and smelting practices, as well as on other topics relative to the industry. The meetings are well attended and much interest and enthusiasm are being shown. Employees of one department are getting acquainted with those of the other departments and they are mutually learning something about each other's work.

PHOENIX, ARIZ.—Mar. 28

Quicksilver District, north of Phoenix, has become active. Though discovered only three months ago, production already has been started on the Hughes, one of the active cinnabar mines in the Phoenix Mountains, 9 to 14 miles north of Phoenix. A crude distilling furnace has been erected by the owners, Sam Hughes, L. H. Larson and F. E. Jetter and quicksilver is being made and brought to Phoenix for sale. The Hughes mine is less than a mile north of the Arizona canal, near what is known as the "slate quarry." The material is found in a broad ledge, apparently schist, with greenish copper stain, but inspection under a glass brings out the presence of the typical vermilion red nodules. A number of other ledges have been located in the hills, following the original discovery, made by Joe Porterie, a Phoenix assayer, who sampled some rock brought in by his young son, with no idea that anything save copper would be found. Rich cinnabar deposits are being worked in the northern Mazatzal Mountains, about 70 miles northeast of Phoenix, but they lie in a rocky wilderness, without even a wagon road for 20 miles.

WASHINGTON—Mar. 30

Low-Grade Lead-Zinc-Copper Ores in Beaver and Iron Counties, Utah, are the subject of a special study being conducted by Roy R. Hörner, who is connected with the Salt Lake City station of the U. S. Bureau of Mines.

Each Mineral Affected by War conditions is to be the subject of a concise review by the U. S. Geological Survey. The object is to show the present status of mineral production and of the mineral industries from the standpoint of the development witnessed and the experience gained since the outbreak of the war. Special attention is being given to conditions and to the demand that will confront the United States in case it should become engaged in war. The work has been assigned to the regular geologists having charge of each of the minerals to be reviewed. They will show what has been accomplished under the extraordinary stimulus of foreign war markets, on the one hand, and of restricted or even totally obstructed importations on the other. An effort will be made to show the ability and the extent to which the mineral industries can meet a possible war demand. The result of this work will be published as a bulletin which will be the logical successor of the brief summary, published a few weeks after the beginning of the war, as "Bulletin 599." That publication outlined what might be expected of the United States in developing industrial independence. The new bulletin will state what has been accomplished.

JOPLIN, MO.—Mar. 31

Cut in Geologic Work to be done in Joplin district by the Missouri State Geological Survey seems probable. While the legislature appropriated $67,000 for the two-year period for the survey, Governor Gardner has signified his intention of cutting the appropriation to $25,000 or $30,000. This will necessitate the abolishment of the Joplin local office, according to State Geologist H. A. Buehler. The office was established here three years ago and, while always handicapped through lack of funds, has done much valuable mapping and drill-log recording. An effort may be made by the mine operators to retain the office.

TORONTO—Mar. 31

Domestic Refining of Ontario Ores is a requirement of a bill introduced in the Legislature by Hon. G. Howard Ferguson, Minister of Mines, on Mar. 30. It provides that hereafter all leases or grants from the Crown of mineral-bearing lands shall contain a condition that all ore mined therefrom shall be refined in Canada unless otherwise directed by the lieutenant governor in council.

Wage Discussion at the miners convention, held at Cobalt last week resulted in a decision to ask the mine managers for an increase in wages amounting to about 50c. per day all around; this is subject to the sanction of the different camps and it is not expected that a formal demand will be made before Apr. 20. Meantime a number of the Cobalt mine owners have agreed on the following scale: Base wages to remain as at present; bonus of 25c. per shift when silver is over 60c. per oz.; double bonus when over 70c.; and treble bonus when over 80c.; should silver go below 70c. and remain above 60c. per oz., a bonus of 50c. to be paid just the same so long as the cost of living as determined by Canadian government statistics remains at its present level. The proposition is not favored by the miners who regard the bonus system as too unstable. The situation at Porcupine is regarded as serious as managers are not disposed to concede any increase and some have signified their intention of curtailing operations or closing down rather than yield to the demand.

Increased Mine Tax is proposed in another bill introduced on Mar. 28 in the provincial legislature of Ontario by Hon. G. Howard Ferguson, Minister of Mines. In accordance with the recommendations of the Nickel Commission it provides that the principle of taxation shall remain unchanged, but that the rate shall be raised from 3% to 5% and upwards according to the following sliding scale: Upon annual profits in excess of $10,000 and up to $5,000,000, 5%; on profits in excess of $5,000,000 and up to $10,000,000, 6%; on profits in excess of $10,000,000 and up to $15,000,000, 7%; on profits in excess of $15,000,000 a percentage increasing with each additional $5,000,000 in the same proportion. On nickel and nickel-copper mines the tax is to be made retroactive as from Jan. 1, 1915. In estimating profits, the companies will be allowed to deduct 15% instead of 10% for depreciation, and will also be allowed to deduct all taxes on profits payable under acts of Great Britain or of Canada. The provisions of the sliding scale are aimed more especially at the International Nickel Co., which has hitherto, through its subsidiary, the Canadian Copper Co., paid a flat rate of $40,000 per year. Under the new system of taxation it is roughly estimated that the tax levied on the company will be $1,000,000 a year. The acreage tax on mining claims is increased to 10c. per acre and is made general for the whole province instead of being limited to unorganized territory. The action of the Provincial government in initiating increased taxation on mining companies has created much dissatisfaction among mining men, especially those interested in gold mines. It is pointed out that while silver and the base metals fluctuate in price and have greatly advanced latterly, gold remains stationary. The increase in the price of materials, the shortage and higher cost of labor and the imposition of a Dominion war tax have already heavily handicapped the gold mining industry making this a specially inopportune time for placing additional burdens upon it. It is regarded as grossly unfair that the Provincial government, because it considers that the International Nickel Co. has been too lightly taxed in the past, should press with undue severity upon other branches of the mining industry. The fear is widely entertained that the increased tax, following so closely upon the Dominion tax on excess profits, will deter American investors from putting capital into mining development.

The Mining News

ALABAMA

HILL-GRIFFITH GRAPHITE (Birmingham)—Contract let to Evans Brothers Construction Co. for graphite-refining plant, Seventh Ave. and Thirtieth St., near "Frisco" tracks.

GULF STATES STEEL CO. (Shannon)—Slope lined with concrete to depth of 700 ft. Permanent buildings such as tipple, power and boiler house being constructed; will house three 350-hp. Stirling boilers, 32 x 72-in. double-drum Nordberg hoist, generators, air compressors, crushers, etc.

ALASKA

ALASKA TREADWELL (Treadwell)—In February, 225 stamps in 300-stamp mill crushed 34,790 tons yielding $62,243 or $1.81 per ton; operating expense $38,100; operating profit $24,142; construction expense, $13,543; sundry income, $11,838; net profit, $22,437.

ALASKA MEXICAN (Treadwell)—Crushed 6755 tons in February from own mine; also 6552 tons from Ready Bullion mine; yield from Mexican ore, $9929; operating expense, $8796; operating profit $1132; construction expense, $3127; sundry income, $4919.

ALASKA UNITED (Treadwell)—In February, Ready Bullion mine supplied 28,469 tons and 700-Ft. Claim, 20,090 tons, yielding $53,867 and $37,129 respectively or $1.91 and $1.86 per ton. Ready Bullion operating expense was $42,950; operating profit $10,916; construction expense, $8459; net profit, $2456; sundry income $2040. The 700-Ft. Claim operating expense was $66,087; operating loss, $28,957; construction expense, $7843; net loss, $36,800; sundry income, $2040.

ARIZONA

Greenlee County

DETROIT COPPER (Morenci)—Churn drilling has been finished on Colorado Hill and drills now working on the Ryerson mine ground.

ARIZONA COPPER (Clifton)—New electric hoist installed at Yavapai mine. New surface-haulage road completed to Clay mine.

Maricopa County

ARIZONA CACTILONE COPPER CO. (Glendale)—Organized by S. C. Kingsbury and other Phoenix men; raising funds for development of about 320 acres about 13 miles from Glendale. The 250-ft. shaft on Slocum claim to be unwatered and stoping to begin on 160-ft. level. New double-compartment shaft to be sunk.

Mohave County

DISTAFF (Chloride)—This old property is being unwatered. A. Lambert in charge.

DIANA (Chloride)—George F. Beveridge has purchased hoist and old 40-ft. shaft, about 500 ft. east of present prospecting shaft, will be sunk to 300 ft.

CHLORIDE X-RAY MINING (Chloride)—Larger power plant, including 100-hp. engine and five-drill compressor, ordered. Tunnel being driven on Marietta claim by contract.

Pinal County

MAGMA (Superior)—Orebody, carrying from 5 to 10% copper, reported encountered between 1700 and 1800 levels.

RAY CONSOLIDATED (Ray)—February copper output 7,177,898 lb., as compared with 5,767,087 lb. in February, 1916.

RAY HERCULES COPPER CO. (Ray)—General Manager C. E. Addams states that No. 1 main shaft has been sunk to depth of 862 ft. and timbered to 850 ft.; cost of sinking was $47.35 per ft., and for timbering $20.24 per ft. Off this shaft, first level was started at 472 ft. and tunnels are being driven to connect No. 1 and No. 2 shafts; this level will be the first hauling level. No. 2 shaft is completed at a depth of 543 ft.; cost of sinking was $28.50 per ft. and for timbering $17.96 per ft.; station being cut at 522 ft. In No. 1 main shaft, encountered carbonates and native copper and in No. 2 shaft considerable sulphide ore. Carbonate ore being shipped to smeltery at El Paso. Excavations completed at the mine for coarse-crushing plant, engine and compressor room, engine and compressor foundations, blacksmith and machine shops, 5000-ton ore bin; concrete being poured. At new millsite, excavations are well under way for mill and machine shop; concrete is being poured for foundations for McIntosh & Seymour 1000-hp. Diesel engines, of which three are ordered. Steel will be arriving within next fortnight and all other equipment has been ordered.

Santa Cruz County

R.R.R. (Patagonia)—Mill, lately started, has shut down for some rearrangement of machinery.

ROYAL BLUE (Alto)—Now worked by A. J. Hooks solely; sending good ore to smeltery up to contract limit.

ALTO (Alto)—Formerly a large producer, and credited with valuable dump; just examined by Julius Kruttschnitt, Jr., and other Guggenheim engineers.

PINAL (Patagonia)—John A. McDonald, superintending work, financed by Narragansett Mining Co. New shaft, 100 ft. south of old shaft, will be sunk to 500 ft.

DUQUESNE (Duquesne)—Affected by the late smelting congestion; last month shipped 2000 tons, though having contract only for 600 tons; the smelters called a halt; consequently mine obliged to curtail production of lessees.

Yavapai County

McKAY GOLD MINES CO. (Congress)—This Phoenix company preparing to install 50- to 100-ton cyanide plant. C. M. Cooper, secretary.

JEROME VERDE (Jerome)—Shaft has reached 1050-ft. level, where station is being cut preparatory to lateral exploration.

JEROME-DEL MONTE COPPER CO. (Jerome)—New company formed to develop 711 acres in Jerome district; part of tract adjoins the Green Monster. Directors are Former Governor T. L. Oddie of Nevada, Frederick Sutter, of Phoenix, Ariz., A. B. Ewing, Thomas S. Wood, E. P. Thompson and Francis E. Young, of Boston, and E. S. Armstrong, of Los Angeles, under whose direction a shaft has been started.

CALIFORNIA

Amador County

PLYMOUTH CONSOLIDATED (Plymouth)—Preliminary report of James F. Parks, superintendent, for February. Ore milled, 9600 tons; value, $56,200; working expense, $27,927; development, $6633; surplus, $21,640. Other capital expenditures, $399.

ARGONAUT (Jackson)—Simonds & Latham cyanide plant operating under contract with Argonaut Mining Co. at east end of mine on old tailing dam. Three shifts of five men each employed. Number will be increased when full capacity of 100 tons is in operation. Similar to plant operated by the same men at the Melones mine in Calaveras County.

Calaveras County

ISABEL GOLD DREDGING (Jenny Lind)—No. 2 dredge under construction; wooden hull, to be equipped with modern machinery. F. J. Estep, superintendent.

GARIBALDI (Mokelumne Hill)—New shaft down 80 ft. disclosed high-grade ore; stockpiling for later shipment. Old producer. John and Louis Gardella, owners.

PENN MINING (Campo Seco)—Bonus paid on Mar. 15 of $\frac{1}{12}$ of total amount earned to all 1916 employees who are still working. Company intends paying bonus in July, 1917, and January, 1918.

Inyo County

DARWIN DEVELOPMENT (Darwin)—This company, which is treating ore from its own mine by the Murex process, is negotiating for purchase of the Defiance mine, an old lead-silver producer.

LOW-GRADE GOLD GROUP in southwestern Inyo, near Brown Valley, reported optioned by Raymond A. Gardiner, of Boston, and California associates. Owners, scattered in Inyo, Kern and Los Angeles Counties, represented by George Brown, of Brown postoffice.

Nevada County

BITNER (Spenceville)—Reported that mine is being unwatered by Harry Johnson, of San Francisco to prospect copper vein.

EASTERN & MAIN (Grass Valley)—Situated between You Bet and Quaker Hill gravel camps; reported optioned by Isaac Ostrom, at price of $15,000.

UNION HILL (Grass Valley)—Shaft will be deepened 400 ft. and raises driven to develop tungsten ore. No difficulty in handling water with an 18-in. Cornish pump. Crew of 30 men will be increased soon. Erroll MacBoyle, manager.

Plumas County

CATE-CAULBACK-LEAVITT (Quincy) — Will extend tunnel 200 ft. to tap rich gravel; now in 580 ft. most of which was driven in winter season. Property at Newton Flat.

CRESCENT (Crescent Mills)—Great Western's transmission line from Las Plumas completed to this property; unwatering 400-ft. shaft and drifts with two Cameron electric pumps and bailing tank having capacity of 1000 gal. per min. Pumps started Mar. 25. Under option to Philadelphia Exploration Co.

ENGELS COPPER (Paxton)—The 500-ton flotation plant being increased by addition of one 750-ton unit; will be further increased by addition of two more units to make a total of 3000 tons capacity. Indian Valley R.R. now within 3 mi. of mines, and new electric line of Great Western Power Co., 8 mi. to the southwest, will soon furnish all additional power.

FIVE BEARS (Genesee)—The U. S. Smelting, Refining and Mining Exploration Co., operating the Five Bears copper mine, has extended main tunnel to 1600 ft.; crosscutting at 1600-ft. point cut vein 18 ft. wide, averaging 5% copper. Mr. Greenwood, formerly at Gold Road mine, Oatman, Ariz., is superintendent; employing about 30 men. On Calnan group, adjoining on east, same company has opened high-grade gold-copper ore in Little Joe claim.

San Bernardino County

MOLYBDENITE reported in Lytle Creek Cañon, short distance from San Bernardino.

DRUM-CLANCY-GOLDSTONE (Barstow)—Reports two strikes of high-grade ore, in 3-ft. and 2-ft. veins.

GOLDSTONE (Barstow)—Crew increased and lumber for power plant and other surface building now on ground.

CALZONA MINES CO. (Vidal)—Sinking shaft, now down 240 ft., to 500 ft. Installing 15-ton amalgamation mill for the straight gold ore. H. E. Olund, resident engineer.

Shasta County

BULLY HILL (Winthrop)—Anchor shaft being unwatered by San Francisco men, with M. E. Dittmar of Redding as manager. Hoist and bailer have been replaced by electric pump. Michigan group adjoining is also being reopened by the same men.

AFTERTHOUGHT (Ingot)—Reported that contract has been let for 300-ton flotation plant; also that J. L. (?) Milliken of Colorado has become interested with S. E. Bretherton and others in operating the mine. Twelve men employed in reopening.

DREDGING GROUND on the Hilborn and Neal Patterson ranches on the east side of the Sacramento River opposite Redding being prospected with Keystone drills by El Oro Dredging Co. of Oroville.

SHASTA NATIONAL COPPER (Redding)—Company, now organized under laws of Nevada, has taken over assets of Shasta National Copper Co. of California. Property situated in Big Backbone district and includes Big Backbone group of 20 claims, Elsie group of 18 and Keystone group of 11 claims.

Sierra County

ORO (Downieville)—Former rich producer on Alleghany-Downieville serpentine belt, being reopened under option to Dr. Reynolds of Alameda and associates; a 4-ft. vein struck by crosscut, both gouge and rock showing much free gold. Lower tunnel to be continued, giving 500 ft. backs. John Costa in charge.

NORTH FORK (Forest)—Under development by North Fork Co. of San Francisco. Vein of ribbon quartz struck at depth, showing abundance of rich sulphides. Gouge prospects well in free gold. Drift to be continued northerly to determine extent of payshoot and to reach rich section of upper workings. George F. Stone, manager.

Tuolumne County

MONARCH (Sonora)—Situated near Middle Camp on Stanislaus River. Tunnel now in 1300 ft. from portal. Gravel tapped by three raises, the last disclosing gravel running $3.10 per ton. Leyner water drill installed for driving adit. Will install water-driven mill for handling gravel.

C. E. RIVES CO. (Sonora)—Developing high-grade gold property on Lawrence ranch on Sonora-Jamestown road. Also shows small percentage of copper. Another property nearby on Felippo Cavalero ground is being developed; hoist and pump installed; shaft down 90 ft. A third property being opened by tunnel on Hales & Symons land.

COLORADO

Boulder County

U. S. GOLD CORPORATION (Sugar Loaf)—This company, owning roasting and cyaniding plant, will add custom sampling department and purchase gold and silver ores.

Dolores County

SILVER SWAN (Rico)—This group, owned by Hay brothers and Swan, recently shipped two cars of lead-zinc ore to Midvale, Utah.

IRON (Rico)—Lessees Christopher Wold and Emil Baer recently shipped car of copper ore taken from 2-ft. vein in this property.

Lake County

MANY SMALL OPERATIONS in Leadville district are attracting attention. At the Ontario in Iowa Gulch, a rich shoot of gold-silver ore was recently disclosed in the adit; regular shipments have been made during winter from the Bartlett development on Sugar Loaf Mountain; the Dinero, in same section, also producing silver ore both on company and leasers' account; the Denver City on Yankee Hill is being reopened by a leasing company under management of Isaac Jones.

Ouray County

SAN ANTONIO (Ouray)—Working force increased recently.

MOUNTAIN KING (Ironton)—County Sheriff E. A. Krisher, who resigned to lease this property, is shipping six tons daily of high-grade zinc ore.

GENESSEE (Ironton)—Active development in progress in this Red Mountain property; considerable copper ore has been opened. New flotation mill will be ready early in April. Under lease to J. M. Hyde. William Rith is superintendent.

VERNON (Ironton)—Shaft sinking resumed; interrupted about 10 days by cutting of ore pocket at 300 level; local mill will probably be leased this summer, pending determination of capacity suitable for company's proposed new mill. A. G. de Golyer, manager.

Rio Grande County

EL MONTE MINING AND MILLING (Jasper)—Replaced dry process with wet concentration, using Card tables. E. V. Busch, superintendent.

San Juan County

SHIPMENTS FROM SILVERTON in February were: To smelters from Iowa-Tiger, 26 cars; Sunnyside, 23; Silver Lake Custom Mill, 22; S. D. & G. Leasing Co., 11; Kittimac, 7; Silver Ledge and St. Paul, each 6; Mayflower, 5; Congress, 4; New Green Mountain and Hamlet, each 1; total, 115 cars. Shipments from properties of district to Silver Lake Custom Mill were: Champion, 22 cars; Old Green Mountain, 15; Champion, 4; New Green Mountain, 2; miscellaneous, 7; total, 43 cars.

GOLD KING (Gladstone)—Repairing mill preparatory to resumption in April. J. H. Slattery is manager.

CONGRESS (Silverton)—Contract awarded Joe Garrett and associates to sink shaft 50 ft. deeper on the K. P. & G. lease.

HAMLET (Silverton)—Development during last few months opened shoot of payable lead and copper ore at about 1400 ft. from portal of fifth-level adit.

HENRIETTA (Gladstone)—New company to be known as Henrietta Copper Mining Co. being organized to take over and operate this property.

ELK MINING AND MILLING (Silverton)—This property of 325 acres developed by crosscut tunnel to depth of 1400 ft. below surface, will be reopened in the near future. J. J. Cusick, manager.

COLUMBUS (Animas Forks)—Development in progress under A. S. Sturgeon. Contemplates installation of new air compressor. W. T. Johnson is manager.

IOWA-TIGER (Silverton)—Extensive development during winter opened considerable bodies of lead, copper, and silver ore. Mill in continuous operation, producing about 25 cars of concentrates per month.

BIG GIANT (Silverton)—Peter Orella and associates developing this property; mill completely remodeled and flotation equipment installed. Repairs are in progress on tramway. Probable that the Esmeralda and Mountain Quail properties will be worked in conjunction with the Big Giant.

Georgia

CROWN MOUNTAIN MINING AND POWER (Dahlonega)—At recent meeting in Birmingham, finances arranged for the 100-ton cyanide plant, based on A. H. Head's experiments indicating 85% extraction.

IDAHO

Shoshone County

NATIONAL (Mullan)—After shutdown of several months, 20 men are now employed on development at this low-grade copper mine. Shaft down 200 ft. below main tunnel, will be sunk 400 ft. further, with lateral work at each 100 ft.

JACK WAITE (Union)—Rush J. White, manager, states that four teams are now hauling 50% lead ore from mine to railroad, 10 miles. At mine, 35 men are employed; shipments paying all operating expenses and providing surplus which may be used to build mill next summer.

CALEDONIA (Wallace)—Report for 1916 shows 46,177 tons mined at cost of $2.61 per ton; 38,929 tons milled at cost of 59c.; 18,117 tons ore and concentrates shipped at cost of 22c. per ton. Total operating cost, $147,350; net realization, $1,302,113; operating profit, $1,154,762. Lead produced, 10,412,640 lb.; silver, 1,297,193 oz.; copper, 741,225 lb.

MICHIGAN

Copper

NORTH LAKE (Lake Mine)—Drift on new lode for 100 ft. in fair rock; will be continued.

WINONA (Winona)—Boilers repaired and mill in full operation again.

ISLE ROYALE (Houghton)—Output now back to normal; shipping about 3800 tons daily to mill.

FRANKLIN (Demmon)—Every machine drill, but one, stoping in the conglomerate lode; one machine working in amygdaloid.

ADVENTURE (Greenland)—Mine operating 13 machine drills in Butler lode, ore averaging from 15 to 16 lb. refined copper to the ton.

NEW ARCADIAN (Houghton)—The two drifts on 1500-ft. level now in about 140 ft. Sinking will be begun after new compressor is installed.

NEW BALTIC (Baltic)—Down about 90 ft. in rock; shaft will be timbered as soon as frost is out of ground. Rock now hoisted with small "puffer" and derrick.

KEWEENAW (Phœnix)—Four new Wilfley tables received. At present ten drills are at work stoping. Still short of men and more cottages and boarding house may be built.

SENECA (Mohawk)—New shaft, west of the No. 2 Mohawk, will be sunk at angle of about 80° for approximately 2000 ft. and then make an easy angle or curve to the lode, which at this depth has dip of about 33°. There is still a question as to exact location of shaft, and whether it will be in the lode, or in the hanging or foot wall. Shaft in foot wall would give advantage of using raises as rock chutes.

MISSOURI

Joplin District

UNITED ZINC (Joplin)—Drilling on lease recently obtained in Waco, Mo., field. Fred L. Crouch, manager.

BLOUNT & CO. (Picher, Okla.)—Excellent strike of lead at shallow level in new shaft just south of Picher.

ROBERTA (Webb City)—Has taken over Schoolhouse mine, at Carterville, formerly operated by J. F. Dexter. G. M. Burke, Joplin, interested.

WAMPLER (Webb City)—New mine, just south of former rich producers at Duenweg, is proving bonanza. Ore being taken out at 130-ft. level, is as rich as any ever found in district. Working in three drifts.

DEFENDER (Picher, Okla.) — Church and Wright, Joplin, owners, sold their 40-acre lease just south of state line, northwest of Picher, to Van Laningham and associates, Kansas City, for $50,000. One shaft partly down, and numerous drill holes showing ore. New owners will sink two shafts and build mill.

MONTANA

Broadwater County

BLACK FRIDAY (Radersburg) — Breitung & Co., Ltd., of New York, which has option on two-thirds of capital stock of company, has been doing extensive development work for last nine months. In years past, mine produced rich gold ore from upper levels. At depth lead was encountered to considerable extent; this will probably be mined separately.

Deer Lodge County

IN GEORGETOWN DISTRICT, due to the collective activity of the recently organized Georgetown Mining Association, the mineral resources of the district will be more fully developed than was possible by individual efforts. Most important undertaking will be the unwatering of the Montana mine. The Venezuela properties, controlled by Arthur Fortier, are being developed and shaft will be sunk an additional 50 ft. On the Glencoe, J. J. King has opened ore in shaft at 150 ft. From Holdfast property, seven cars of ore were shipped in February and shaft will be sunk additional 100 ft. In Red Lion section, contract has been let for driving 200 ft. of tunnel on Badger-Montana property.

Mineral County

RICHMOND (Saltese)—Constructing 1½-mile aërial tramway from mine to Adair; will be in operation in April. Mine on divide between Idaho and Montana, and ore now being hauled 6 miles to Saltese. Company reports 41 carloads shipped in 1916 having gross value of $59,431. Ore averages about 14% copper and $4 in gold. Control of company recently acquired by New York men.

Silver Bow County

DAVIS-DALY (Butte)—Shipments of copper ore now averaging 170 tons per day and of zinc ore 50 tons; latter runs from 15 to 18% zinc. Profits for February were $35,000; expected to be exceeded in March, as shipments of copper ore will exceed 5000 tons. Two new air compressors just started and production will soon be increased.

BUTTE-DETROIT (Butte)—While company's Ophir mill is not running at capacity, outlook for increased supply of zinc ores is so promising that management is arranging to increase capacity to 400 tons a day to care for growing supply of Davis-Daly properties, but also for output of this company's Ophir mine and of adjoining properties owned by the company. Crosscutting on 1000-ft. level of Ophir mine to reach Ophir-Travona vein which showed 4 ft. of rich zinc ore on the 500 level.

MINES OPERATING CO (Butte)—On Mar. 23, made second payment of $30,000 on option on Bullwhacker stock held by East Side Mining Co.; about 45 men working and from 125 to 150 tons being shipped to Tacoma and Garfield smelteries; copper production from Bullwhacker for March is estimated at 300,000 lb. At Butte-Duluth, 55 men are working and from 180 to 210 tons of 1¼% ore are mined and treated daily in leaching plant; March output from Butte-Duluth estimated at over 100,000 lb; additional equipment being installed to increase treatment capacity to 500 tons a day.

ANACONDA (Butte)—Due, evidently to the settling of the ground in the fire zone, by which the company's Tramway and West Colusa mines are chiefly affected, the power cables in the Tramway shaft broke Mar. 25, and started a blaze in the shaft timbers near the 1100 level; required some time to extinguish and necessitated laying off men in both mines; ground in the Tramway is still settling and work of bulkheading to confine the old fire will be going on for some time and prevent actual mining in that property. At the West Colusa, men have returned to work. Decrease in output made up by resumption in the St. Lawrence and by an increase in the working force at other mines. Over 16,000 tons of ore shipped daily to the reduction works at Great Falls and Anaconda.

NEVADA

Churchill County

WESTERN ORE PURCHASING CO. (Hazen)—The 300-ton sampling mill will be doubled or tripled in capacity. F. M. Manson, manager.

Esmeralda County

LOUISIANA CONSOLIDATED (Goldfield and 120 Broadway, New York)—Has force engaged in active development of old Tybo mine, 55 miles northeast of Tonopah. The mine has been opened to 400 ft.; though much ore still remains on upper levels, present plans are for thorough development to the 800-ft. point. Diesel-engine power plant, with electric generators, compressor, new hoist and two sets of electric pumps, now being installed. Transmission line of Nevada-California Power Co. will later be extended from Belmont, 30 miles, after which present power plant will be held in reserve. Building of a 500-ton smelting plant at the mine and railroad 175 miles long to connect Tybo with Ely or Eureka on the north and with Tonopah on the south under consideration; franchise for railroad granted recently.

REORGANIZED DIAMONDFIELD TRIANGLE MINING CO. (Goldfield)—Organized to take over old Diamondfield Triangle group of seven claims in Diamondfield section of district; J. K. Turner, consulting engineer. Period for exchange of old stock expires May 21.

GOLDFIELD CONSOLIDATED (Goldfield)—Annual report shows cash on hand, Jan. 1, 1917, as $1,021,086, comparing with $573,370. Milled 338,680 tons, yielding $6.53 per ton. Net profit, $428,619. Ore reserves at end of year, 85,000 tons, but General Manager J. W. Hutchinson states that it is safe to say that "at least 300,000 tons will be milled before profitable operations cease." Retreatment of tailing is expected to begin in June. No new properties acquired, but some exploration was done on property of Lampa Mining Co., near Santa Lucia, Peru.

Humboldt County

NENZEL CROWN POINT (Rochester)—Construction on first unit of 200-ton mill to start about May 1. Flotation and cyanidation will both be used.

ROCHESTER MINES CO. (Rochester)—New tramway nearly ready for operation. Orebody on 800-ft. level developing satisfactorily.

Mineral County

AURORA CONSOLIDATED (Aurora) — This Goldfield Consolidated subsidiary in 1916 milled 173,270 tons of an average grade of $3.32 per ton; gross profit $71,006. Paid $31,300 on its $400,000 indebtedness to parent company, leaving $358,700 still due. Ore reserves 336,978 tons.

Nye County

TONOPAH ORE PRODUCTION for week ended Mar. 24, was 8920 tons, valued at $160,500, as compared with 9614 tons the week previous. Shippers were Tonopah Belmont, 3011 tons; Tonopah Mining, 1500; Tonopah Extension, 2380; Jim Butler, 700; West End, 462; Rescue, 189; Halifax, 208; Montana, 290; North Star, 55; Cash Boy, 50; miscellaneous, 75 tons.

WHITE CAPS MINING (Manhattan)—Regarding rich gold strike on 425-ft. level, General Manager John G. Kirchen of Tonopah wrote, on Mar. 17, that east orebody had been followed for 125-ft.; average width, 30 ft.; average sampling for fortnight, $36 per ton; expects oreshoot to extend about 80 ft. farther. From 425-ft. to 310-ft. level, above is 178 ft. on pitch of vein, all virgin ground; also, the 138 ft. on dip to next level above. Installing seven-hearth 22½-ft. Wedge roaster. Milling to be resumed in June. Ore will be crushed to 8 mesh, roasted and cyanided by percolation. Driving to west orebody, richest one worked in former operations. (Later—An orebody reported cut by southwest crosscut from east vein.)

Storey County

COMSTOCK PUMPING ASSOCIATION (Virginia)—Repairing 2900 winze station.

ANDES (Virginia) — Saved 67 cars of ore averaging $9.30; raise for ventilation started in stopes.

MEXICAN (Virginia)—Sampling 2300 level and making repairs to 2700-2900-winze; 4 bars bullion shipped from Mexican mill.

SIERRA NEVADA (Virginia)—Advancing work on 2400, 2500, and 2600 levels, last level showing porphyry and quartz.

OPHIR (Virginia)—Draining 2300 level preparing for exploration. Crosscut 2700 level through porphyry and quartz seams.

CON. VIRGINIA (Virginia)—Southwest drift 2700 level in wide vein of quartz, face exposing low-grade ore; 80 cars extracted averaging $7.

UNION CON. (Virginia)—Shipped 564 tons ore to Mexican mill averaging $18.67; extracted from 2400, 2500, 2600 and 2700 levels; high-grade streak on 2600 produced eight tons sampling $144.92 per ton.

White Pine County

NEVADA CONSOLIDATED (McGill)—February copper output 5,708,214 lb., as compared with 6,553,412 lb. in February, 1916.

CONSOLIDATED COPPERMINES (Kimberly)—Four cars of concentrate being shipped weekly to Nevada Consolidated plant at McGill; some high-grade carbonate also being shipped to Garfield, Utah. Some of equipment for second unit of flotation mill now on ground. About 250 men employed. Churn drilling to be resumed this month.

UTAH

Box Elder County

STIBNITE MINING (Brigham City)—Incorporated Mar. 26, with 1,000,000 shares, par value 10c. Owns 686 acres, a few miles east of main Oregon Short Line track. Recently car of 29 tons brought $2647.13 gross. One deposit opened by 225 ft. of tunneling and crosscutting. Also a shaft 155 ft. deep. Property little developed. H. C. Baker, president and general manager.

Juab County

TINTIC ORE SHIPMENTS for week ended Mar. 23, were 212 cars.

DESERET MOUNTAIN (Tintic)—Car of copper ore loaded for shipment, on return trip of teams hauling machinery to property.

TINTIC STANDARD (Eureka)—This property, which has recently become one of regular shippers of camp, is hampered by 1½-mile haul to railroad. Negotiating for railroad spur; or may install tractors.

TINTIC CENTRAL (Silver City)—Drifting to be started on Iron Blossom 1700 level, with double purpose of opening this property at 1500 level—500 ft. below present workings—and of prospecting in Iron Blossom ground, undeveloped, except by diamond drilling several years ago.

EMPIRE MINES CO. (Eureka)—Filed articles of incorporation Mar. 27. Controls 600 acres, representing consolidation of eight mining properties in Tintic and West Tintic districts. Examination will be made to determine where work will be started. Probably at Lower Mammoth, one of the properties merged.

Salt Lake County

ALTA CONSOLIDATED (Alta)—Shipments resumed. Body of low-grade ore also being developed.

UTAH COPPER (Garfield)—February copper output 13,459,829 lb., as compared with 11,849,972 lb. in February, 1916.

COTTONWOOD KING (Salt Lake)—Assessment of ¼c. a share, levied on this Big Cottonwood property to sink winze in low-grade pyrite ore.

OHIO COPPER (Lark)—Purchased all stock and majority of bonds of Bingham Central Ry. Co., which owns the Mascotte tunnel through which Ohio ore is transported to mill at Lark.

GARFIELD CHEMICAL & MANUFACTURING (Garfield)—Plant, which began operation in December has been gradually brought up to capacity of 100 tons daily. Second 100-ton unit to be added.

MICHIGAN-UTAH (Alta)—Sixty-five cars of ore shipped first quarter 1917. Aërial tramway operated satisfactorily during winter. New ore showing in Copper Prince tunnel—500 ft. from portal and 350 ft. from surface.

Summit County

PARK CITY SHIPMENTS for the week ended Mar. 23, were 212 cars.

CALIFORNIA-COMSTOCK (Park City)—Shipments to be started with more favorable weather conditions.

DALY (Park City)—Dividend of 10c. a share, amounting to $15,000, to be paid Apr. 10. Some high-grade ore being shipped. The 800 and 500 levels being developed. Dividends were paid by the mine during early days of the camp. Present dividend is first in a long time; property idle until recently.

Tooele County

OPHIR HILL (Ophir)—Shipping two cars of concentrates and one of high-grade daily. New body of shipping ore opened on lower levels; carries silver, lead and gold. Net proceeds filed for taxation purposes for 1916 were $429,000.

Washington County

SILVER REEF (St. George)—Reported that tailings dump in this old camp, west of Toquerville, will be reworked.

SOUTH DAKOTA

Lawrence County

GOLDEN CREST (Deadwood)—Being unwatered, and will be thoroughly sampled before mining is resumed.

HOMESTAKE (Lead)—Camps are being established along Spearfish Creek, preparatory to construction of 8-mile ditch for new power plant; included in the rock work is one tunnel more than a mile long. Will be driven by contract.

MOGUL (Terry)—By arangement with the Ofer company, working tunnel is being driven in formation below the orebodies. This is now in 330 ft. and will be continued to 2200 ft. Raises will be driven to the stopes and the ore dropped to the transportation tunnel from portal of which it will be conveyed by aërial tramway to the Mogul mill. New method will eliminate railroad transportation and winter interruptions due to snowfall.

WISCONSIN

Zinc-Lead District

RODHAM (Shullsburg)—New two-jig mill in operation.

DOMESTIC (Benton)—Will build single-jig mill on McCabe-Buchan lease.

FRONTIER (Galena, Ill.)—Sinking shaft and constructing two-jig mill at Mird No. 3, Benton.

BERRYMAN (Dodgeville)—Single-jig mill will be built at once by H. F. Roberts, T. A. Metcalfe and A. T. Cretney.

NEW EMPIRE (Platteville)—Contract let to Galena Iron Works to move Grant County mill to Powder Mill property of Fritz Hoppe et al.—christened the New Empire.

CANADA

Ontario

ADANAC (Cobalt)—Strong vein encountered in crosscutting on 400-ft. level.

HOLLINGER CONSOLIDATED—Report for four-week period ended Feb. 25 shows gross profits of $210,868, from 48,252 tons; average value, $8.54; working cost, $3.96 per ton.

The Market Report

Metal Markets

SILVER AND STERLING EXCHANGE

Mar.	Sterling Exchange	Silver: New York, Cents	Silver: London, Pence	Apr.	Sterling, Exchange	Silver: New York, Cents	Silver: London, Pence
29	4.7556	72⅜	35 9/16	2	4.7556	74⅛	36 13/16
30	4.7556	72⅜	36	3	4.7556	74⅛	36 13/16
31	4.7556	73⅜	36½	4	4.7556	74	36¾

New York quotations are as reported by Handy & Harman and are in cents per troy ounce of bar silver, 999 fine. London quotations are in pence per troy ounce of sterling silver, 925 fine.

DAILY PRICES OF METALS IN NEW YORK

Mar. Apr.	Copper: Electrolytic	Tin: Spot.	Lead: N. Y.	Lead: St. L.	Zinc: St. L.
29	30¼ @31¼	54½	9 @9⅛	9 @9⅛	9¾ @10¼
30	30 @31	54¼	9 @9⅛	9 @9⅛	9¾ @10¼
31	30 @31	54¼	9 @9⅛	9 @9⅛	9¾ @10¼
2	30 @31	54¼	9 @9⅛	9 @9⅛	9¾ @10¼
3	29¾ @30¾	54¼	9 @9⅛	8.80 @9	9¾ @10¼
4	29¾ @30¾	54¾	9 @9⅛	8.80 @9	9¾ @10¼

The above quotations are our appraisal of the average of the major markets based generally on sales as made and reported by producers and agencies, and represent to the best of our judgment the prevailing values of the metals for the deliveries constituting the major markets, reduced to basis of New York, cash, except where St. Louis is the normal basing point.

The quotations for electrolytic copper are for cakes; ingots and wirebars. Electrolytic copper is commonly sold on "regular terms" (r.t.), including freight to the buyer's works and is subject to a discount for cash. The difference between the price delivered and the New York cash equivalent is at present about 0.25c. on domestic business. The price of electrolytic cathodes is 0.05 to 0.10c. below that of electrolytic. Quotations for spelter are for ordinary Prime Western brands. We quote New York price at 17c. per 100 lb. above St. Louis.

Some current freight rates on metals per 100 lb. are: St. Louis-New York 17c.; St. Louis-Chicago, 6.3c.; St. Louis-Pittsburgh, 13.1 cents.

LONDON

Mar. Apr.	Copper: Standard Spot	Copper: Standard 3 Mos.	Copper: Electrolytic	Tin: Spot	Tin: 3 Mos.	Lead: Spot	Zinc: Spot
29	136	135½	151	215¼	214¾	30½	47
30	136	135½	151	214¾	214¾	30½	47
31	....	....	...	....	....	....	...
2	136	135½	151	215½	215½	30½	55
3	136	135½	151	214	214	30½	55
4	136	135½	151	216	216	30½	55

The above table gives the closing quotations on London Metal Exchange. All prices are in pounds sterling per ton of 2,240 lb. For convenience in comparison of London prices, in pounds sterling per 2,240 lb., with American prices in cents per pound the following approximate ratios are given, reckoning exchange at 4.80. £15 = 3.21c.; £20 = 4.29c.; £30 = 6.43c.; £40 = 8.57c.; £60 = 12.85c. Variations, £1 = 0.21⅜c.

NEW YORK—Apr. 4, 1917

The markets were all dull and rather confused during the last week, but there was a distinctly softer tendency in copper. Zinc and lead were also easier in tone. Freight delays are still playing a part in the markets, but conditions are improving.

Copper, Tin, Lead and Zinc

Copper—The business of the last week was very light indeed, several producers reporting no transactions at all. There was no doubt, however, about there being a softer tendency, for sellers who were anxious to place third-quarter copper gradually reduced their prices, and at the close were offering this delivery freely at 30c., r.t., equivalent to about 29¾c., net cash, New York. In early copper, on reselling by consumers, there was exhibited a similar softness, second-quarter delivery being sold at 33@34c. Prompt and April copper was to be had at 34c., May, at 33½c., and June, at 33c., r.t.

Copper Sheets—No change has been reported in the situation as described last week. We continue to quote hot rolled at 42@44c. per lb., cold rolled 1c. per lb. higher. Wire is now quoted 38@40c., f.o.b. mill.

Tin—Business was extremely quiet without there being much change in prices.

Lead—Transactions were in fair volume, but were lighter than in the previous week, about 2400 tons being reported, against 3000. The bulk of the business was done at 9c., April lead being sold at that price, and also May and June. The buyer needing prompt lead, i.e., delivery within two or three days, was compelled to pay a little premium, but this is gradually diminishing as the situation becomes easier, although it is expected that there will be a certain tension throughout April. The lead market was regarded last week as being rather baffling by several sellers who were able to realize 9¼ and even 9½c. for April lead, while they knew that sales were being made at lower prices. The explanation is that such fancy prices were realized only on carload business, in the main, done with buyers who did not know the market. Fancy prices were also realized by some sellers who could ship from favored points and obtained comparatively high prices when figured up to New York basis or back to St. Louis basis. There were inquiries in the market for lead for Canada, South America, France, Archangel and Vladivostock.

Zinc—Business was again light. Second-quarter spelter was sold at 9¾c., while business in prompt was done at 10@10¼c. As in the case of lead, fancy prices were sometimes explained by sales for shipment from special points, but reckoned on St. Louis basis. In St. Louis itself there are believed to be unsold stocks of spelter. In view of this, the manner in which the premium on prompt spelter is maintained, while second-quarter spelter is offered at half a cent less and exhibits signs of pressure, is rather mysterious. It may be inferred that prompt metal is being held back to cover the marketing of contracts for the quarter. Some sales running beyond the second quarter were made during the week, including some export business. Sales of brass special were made at ¼@⅝c. premium over prime Western. High-grade spelter continues to be quoted at 16@17c.

Zinc Sheets—Price of zinc sheets has not been changed. Market remains at $21 per 100 lb., f.o.b. Peru, Ill., less 8% discount.

Other Metals

Aluminum—The aluminum market remains in about the same condition as reported last week. No great demand has developed. We report no change from last week's figures. Nominal quotation is 58@60c. per lb. for No. 1 ingots, New York.

Antimony—Market firm and quiet at 35@37c., business being in small lots only. April-May shipment from China is quoted 13½@16c., c.i.f., New York, duty unpaid.

Nickel—The market remains steady at 50@55c. per lb. for nickel. Electrolytic commands an additional 5c. per lb.

Quicksilver—Market shows little change; is steady at $118@120. San Francisco reports by telegraph $115, steady.

Gold, Silver and Platinum

Gold—Gold coin to the amount of $395,000 has been withdrawn from the New York subtreasury for shipment to Spain, and $300,000 for shipment to Cuba. Gold to the amount of $10,000,000 has arrived at the Philadelphia mint from Canada, bringing total imports from all sources to date this year, to $250,700,000, and since January, 1915, to $1,388,200,000. A gold shipment of $10,000,000 arrived late last week for account of J. P. Morgan & Co., from Canada. Half was deposited at the assay office and half with the Federal Reserve Bank. This adds to the totals given above.

Two transfers amounting to $1,274,000 have been made through subtreasury to San Francisco, against deposits of gold, presumably for export, of an equivalent amount of gold bars from San Francisco to Japan.

Silver—Silver has experienced a sharp advance this week due to a strengthening of China exchanges, small offerings in London and a continuance of buying in London for coinage.

The U. S. Government has recently purchased some 800,000 oz. for subsidiary coinage, which has lessened the amount available for London shipment. London stock has been reduced to some 3 million ounzes. Market closes steady at 30¾ pence in London and 74c. in New York.

Mexican dollars at New York, Mar. 29, 56c.; 30, 55⅝; 31, 55¾; Apr. 2, 56⅝; 3, 57¼c.; 4, 57¼c.

Platinum—Sales were reported at $102½, which we quote as the market.

Palladium—Sales were made at $90@95, which we quote as the market.

Zinc and Lead Ore Markets

Platteville, Wis., Mar. 31—Blende, basis 60% Zn, $85 for premium ore down to $80 for medium grade. Lead ore, basis 80% Pb, $115 down to $110 per ton at the week end. Shipments for the week are 3112 tons of zinc ore, 131 tons of lead ore, and 808 tons of sulphur ore. For the year to date the figures are: 33,303 tons of zinc ore, 1048 tons of lead ore, and 5645 tons of sulphur ore. Shipped during the week to separating plants, 3240 tons of zinc ore.

Joplin, Mar. 31—Blende per ton, high $88.40; basis 60% Zn, premium $85, medium to low $82@75. Calamine, per ton, basis 40% Zn, $50@45; average selling price, all grades of zinc, $78.94.

Lead, per ton, high $118.10; basis 80% Pb, $115@110; average selling price, all grades of lead, $116.27.

Shipments the week: Blende 9107 tons, calamine 1642 tons, lead 1877 tons. Value, all ores the week, $1,063,740; three months, $11,636,850.

Buying strengthened at the week-end, following the recession in base prices, marking a strong demand on the lower level.

The Miami-Commerce field in Oklahoma continues to be badly handicapped in shipment on account of switching facilities. It is reported the Frisco has begun an extensive system of spurs to cover the entire field.

Other Ores

Antimony Ore—This ore is in demand, but supplies are scarce.

Molybdenum Ore—There is not much change in this market.

Tungsten Ore—We quote $17@18. The terms for this ore are now showing considerable variation according to quality.

Under date of Apr. 3, Charles Hardy reports as follows: "The past week has shown further great activity in the tungsten market, which, however, becomes more and more difficult on account of the many demands by buyers for guarantees against impurities, and only well-known marks that have reached this market continuously in approximately the same quality can be readily sold at the top prices. Other marks can only be sold at concessions and therefore, it will be found that the prices for tungsten ore ranging according to degree, from $18 per unit for high grade wolframite and scheelite, down to $16.50 for material, which on account of impurities has only a limited market. A very large amount of business has again been done for home consumption. France and Italy have been in the market. Besides, forward material has had ready buyers and it was impossible to fill an order for 100 tons of spot delivery wolframite, as such a quantity was not available for prompt delivery. The prospects for the market are quite good and present prices should be fully maintained."

Iron Trade Review

NEW YORK—Apr. 4

The stirring events of the week have put ordinary trade considerations in the background, says "Iron Age." But leaders in the steel industry, the Government's main war resource, have been busy finding ways to put their product and the best service of their plants at the country's disposal.

There was further conference in New York Tuesday between the steel manufacturers' committee and the metals commissioner of the Council of National Defense. It is probable that the Government will buy in the near future for its own wants and those of shipyards which are to do its work, 500,000 tons of plates, shapes and bars. Substantial concessions are to be made from current prices, but the proposal to take a 10-year market average was not adopted, as that would represent less than cost to companies not owning their own ore and coal.

The problem is not so simple as it looks to be. A steel-plate price of 3c., instead of 6c., would mean a fraction of a cent above cost to a mill paying $35 a ton for basic pig iron. The integrated companies could only stand it by telescoping the profits now made on coke, pig iron and ingots.

PITTSBURGH—Apr. 3

Pig iron, unfinished steel and finished steel products all show a decided advancing tendency. Bessemer iron is up $2 and foundry iron at least $1. Billets and sheet bars are not quotably higher but are still harder to negotiate. An advance of five points or about $9 per net ton in standard steel pipe, begun by one independent a week ago, has now become general and wrought iron pipe is up six points. The National Tube Co. has issued a new discount card on welded steel boiler tubes, effecting an advance of about five points, but 12 points above the previous list, dated Nov. 20, and withdrawn shortly afterward. The American Sheet and Tin Plate Co. has announced prices for third quarter to jobbers and second half to manufacturing consumers as follows: Tin plate, $7.50; terne plate, $7.15; black sheets 5.50c.; blue annealed sheets, 5.00c.; galvanized sheets, 7.00c. The tin plate price is lower than was generally expected while the sheet prices are a trifle under the prices ruling in the past fortnight for deliveries in the next few months. The great bulk of the mill tonnage will no doubt be disposed of in the next two or three weeks.

Deliveries of all steel products have become very uncertain on account of the war. All steel that can be used for government account will readily be diverted by the manufacturers, but ordinary consumers will have their deliveries set back. Plans thus far matured cannot possibly involve more than a very few per cent. of the steel now being made, but eventually may run to between 10 and 20% provided shell making for the Entente Allies is resumed on the former scale. Such production has lately become very light.

Transportation conditions continue to improve, as evidenced, for one thing, by lower prices for spot coal and coke, and iron and steel production is now very nearly at capacity.

Pig Iron—The strength in basic iron is shown by the fact that a large merchant producer has bought a large tonnage at $35, valley. Several lots of bessemer have sold at $40, valley, now the minimum and representing an advance of $2 in the week. Inquiries for round lots are being turned down and the market may easily advance farther. Foundry iron is bringing higher prices. There is no definite knowledge whether pig iron is likely to become scarcer or more plentiful, there being a delicate balance between production and consumption. We quote: Bessemer, $40; basic, $35; foundry, prompt, $38@40; second half, $37@38, f.o.b. valley furnaces, 95c. higher delivered Pittsburgh.

Iron Ore

Washington—Without waiting for a supplemental decision by the Interstate Commerce Commission in the ore-rate case, the railroads have prepared tariffs proposing a uniform increase of 15c. a ton, on iron ore from lower Lake Erie ports, to furnaces in western Pennsylvania, Ohio, West Virginia and Kentucky. All tariffs are intended to become effective May 1. It is also planned by the railroads to increase in coal and coke rates.

The commission's supplemental decision in the ore case has been expected before this, with the new rates under it to be made effective Apr. 1. For some unexplained reason, the decision has been held up, but may be handed down soon. This decision would prevent the ore tariffs now prepared by the railroads from becoming effective on the proposed date, if at all. If insisted upon by the carriers, it is believed that furnace interests would protest vigorously, as the new tariffs would mean a big advance—about 30% in the case of the Youngstown rate. These tariffs would continue present rules and regulations governing ore transportation, while the commission has proposed the segregation of charges for different services, such as handling from rail of vessel, spotting and storing.

Coke

Connellsville—The spot market is a trifle easier, inquiry being limited while offerings are slightly increased owing to better car supplies. Neither buyers nor sellers show any disposition to contract for second half. We quote: Spot furnace, $8@8.50; contract, nominal, $7@8.50; spot foundry, $10@10.50; contract, $8.50@9, per net ton at ovens.

Chemicals

Arsenic—White, powdered, quoted at 16@17c. for general market. Steady demand.

Copper Sulphate—In car lots quotation is 9½@10c.; powdered, 13½@14c.

Lead Acetate—Crystals, white, quoted at 14c.; powdered, 15@17c. Brown quoted at 12¾c.

Sulphur—Commercial, $1.60@1.70 per 100 lb. Flowers, $2.75@2.95.

STOCK QUOTATIONS

N. Y. EXCH.†	Apr. 3
Alaska Gold M	9
Alaska Juneau	7½
Am.Sm.&Ref.,com.	103¾
Am. Sm. & Ref., pf.	113½
Am. Sm. Sec., pf. A	102
Am. Sm. Sec., pf. B.	96½
Am. Zinc	36½
Am. Zinc, pf	68
Anaconda	83¾
Batopilas Min	1¼
Bethlehem Steel	148
Bethlehem Steel, pf.	120½
Butte & Superior	45½
Cerro de Pasco	39¾
Chile Cop	24½
Chino	59
Colo.Fuel & Iron	52
Crucible Steel	70⅝
Dome Mines	18
Federal M. & S.	18
Federal M. & S., pf.	42
Great Nor., ore ctf.	33⅜
Greene Cananea	42½
Homestake	‡126
Inspiration Con	61
International Nickel	44
Kennecott	46
Lackawanna Steel	86½
Miami Copper	42
Nat'l Lead, com	58
National Lead, pf.	111
Nev. Consol	24
Ontario Min	5¾
Quicksilver	2
Quicksilver, pf	2⅝
Ray Con	31½
RepublicI.&S.,com.	83½
Republic I. & S. pf.	103
Sloss-Sheffield	68½
Tennessee C. & C.	16
U. S. Steel, com	116⅜
U. S. Steel, pf	118⅜
Utah Copper	111¾
Va. Iron C. & C.	70½

N. Y. CURB†	Apr. 3
Butte & N. Y.	1⅜
Butte C. & Z.	11½
Caledonia	.64
Calumet & Jerome	1¼
Can. Cop. Corpn	2
Carlisle	6
Con. Ariz. Sm	2
Con. Coppermines	4
Con. Nev.-Utah	⅜
Emma Con	1¼
First Nat. Cop	2 1/16
Goldfield Con	.62
Goldfield Merger	.08
Granite	‡.25
Greenmonster	1⅜
Hecla Min	7⅜
Howe Sound	6¼
Jerome Verde	2 1/16
Joplin Ore & Spel.	.29
Kerr Lake	4⅜
Magma	54½
Majestic	5/16
McKinley-Dar-Sa	.49
Mother Lode	.36
N. Y. & Hond	14
Nipissing Mines	7⅜
Ohio Cop	1 7/16
Ray Hercules	3⅜
Rochester Mines	.58
St. Joseph Lead	‡18½
Seneca	12½
Standard S. L.	.75
Stewart	5/16
Success	.40
Tonopah	6½
Tonopah Ex	3½
Tribullion	.50
Troy Arizona	.51
United Verde Ext.	39
United Zinc	5½
White Knob, pf	1½
White Oaks	2
Yukon Gold	1⅜

SAN FRAN.*	Apr. 3
Alta	.04
Andes	.12
Best & Belcher	.05
Bullion	.02
Caledonia	.14
Challenge Con	.05
Confidence	.10
Con. Virginia	.15
Gould & Curry	.02
Hale & Norcross	.04
Jacket-Cr. Pt	.03
Mexican	.05
Occidental	.65
Ophir	.11
Overman	.04
Savage	.05
Seg Belcher	.02
Sierra Nevada	.11
Union Con	.33
Utah Con	.01
Belmont	4.25
Jim Butler	.74
MacNamara	.07
Midway	.11
Mont.-Tonopah	.20
North Star	.15
Rescue Eula	.27
West End Con	.65
Atlanta	.12
Booth	.10
Comb. Frac	.05
D'field B.B.	‡.02
D'field Daisy	.04
Florence	.21
Jumbo Extension	.42
Kewanas	.17
Nevada Packard	.45
Round Mountain	.43
Silver Pick	.11
Argo	.05
Ariz. Central	.10
Big Jim	.93¾
Lazy Boy	.06
Nellie	.18
Tom Reed	2.18¾
United Eastern	4.37½

BOSTON EXCH.*	Apr. 3
Adventure	3
Ahmeek	100
Algomah	.85
Allouez	65
Ariz. Com., ctfs	13¾
Bonanza	.25
Butte-Ballaklava	1⅜
Calumet & Ariz	80
Calumet & Hecla	552
Centennial	20
Copper Range	63½
Daly West	2¼
Davis-Daly	5⅜
East Butte	14½
Franklin	8⅝
Granby	85
Hancock	16½
Hedley	16
Helvetia	.55
Indiana	3¾
Island Cr'k, com	67½
Isle Royale	32¾
Keweenaw	2⅝
Lake	13½
La Salle	3¼
Mason Valley	6½
Mass	13¼
Mayflower	2½
Michigan	3¼
Mohawk	85¾
New Arcadian	4¼
New Idria	17½
North Butte	22¾
North Lake	1½
Ojibway	2⅛
Old Colony	2½
Old Dominion	63
Osceola	87
Quincy	90
St. Mary's M. L.	84½
Santa Fe	1½
Shannon	8⅝
Shattuck-Ariz	27
So. Lake	4¼
So. Utah	.25
Superior	14½
Superior & Bost	6¼
Tamarack	‡59½
Trinity	4
Tuolumne	1½
U. S. Smelting	62½
U. S. Smelt'g, pf	51½
Utah Apex	2
Utah Con	15½
Utah Metal	4½
Victoria	4½
Winona	4
Wolverine	45
Wyandot	1

BOSTON CURB*	Apr. 3
Alaska Mines Corp.	1 1/16
Bingham Mines	9⅜
Boston Ely	.70
Boston & Mont	.64
Butte & Lon'n Dev.	.25
Calaveras	4
Calumet-Corbin	.01
Chief Con	2⅜
Cortez	.20
Crown Reserve	.30
Crystal Cop	.82
Eagle & Blue Bell	1⅜
Houghton Copper	1¼
Intermountain	1 3/16
Iron Cap Cop., pf.	15
Mexican Metals	.15
Mines of America	1¾
Mojave Tungsten	.40
Nat. Zinc & Lead	.50
Nevada-Douglas	2
New Baltic	1½
New Cornelia	17¼
Oneco	.50
Pacific Mines	.25
Rex Cons	.40

SALT LAKE*	Apr. 3
Big Four	.86
Black Jack	.05
Cardiff	5.00
Colorado Mining	.16½
Crown Point	.04
Daly-Judge	8.50
Empire Copper	1.50
Gold Chain	.15
Grand Central	.51
Iron Blossom	1.17½
Lower Mammoth	.18
May Day	.09¾
Moscow	.12
Prince Con	1.20
Seven Troughs	‡.02¼
Silver-King Coal'n	3.00
Silver King Con	4.12½
Sioux Con	.14
So. Hecla	1.37½
Uncle Sam	.03
Wilbert	.13½
Yankee	.10½

TORONTO*	Apr. 3
Bailey	.06
Beaver Con	.39
Buffalo Mines	1.15
Chambers Ferland	.13
Coniagas	3.80
La Rose	.54
Peterson Lake	.11
Right of Way	.05
T. & Hudson Bay	43.00
Temiskaming	.51
Wettlaufer-Lor	.07
Dome Exten	.23
Dome Lake	.18½
Foley O'Brien	.60
Hollinger	5.20
Jupiter	.30
McIntyre	1.86
Newray	1.33
Porcu. Crown	.64
Schumacher	.58
Vipond	.43
West Dome	.27

STOCK QUOTATIONS—Continued

COLO. SPRINGS	Apr. 3
Cresson Con	5.81¼
Doctor Jack Pot	.07
Elkton Con	.09⅞
El Paso	.25
Gold Sovereign	‡.04
Golden Cycle	2.30
Granite	.55
Isabella	.16¾
Mary McKinney	.26
Portland	1.76
United Gold M	.16
Vindicator	.70

LONDON	Mar. 15		
Alaska Mexican	£0	12s	6d
Alaska Tre'dwell	2	12	6
Burma Corp	3	15	0
Cam & Motor	0	8	0
Camp Bird	0	5	0
El Oro	0	7	0
Esperanza	0	7	9
Mexico Mines	4	2	6
Nechi, pfd	0	12	6
Oroville	0	15	0
Santa Gert'dis	0	8	9
Tomboy	0	19	6

* Bid prices. † Closing prices. ‡ Last Quotations.

MONTHLY AVERAGE PRICES OF METALS

Silver	New York			London		
	1915	1916	1917	1915	1916	1917
January	48.855	56.775	75.630	22.731	26.960	36.682
February	48.477	56.755	77.585	22.753	26.975	37.742
March	50.241	57.935	73.861	23.708	27.597	36.410
April	50.250	64.415	......	23.709	30.662	......
May	49.915	74.269	......	23.570	35.477	......
June	49.034	65.024	......	23.267	31.060	......
July	47.519	62.940	......	22.597	30.000	......
August	47.163	66.083	......	22.780	31.498	......
September	48.680	68.515	......	23.591	32.584	......
October	49.385	67.855	......	23.925	32.361	......
November	51.714	71.604	......	25.094	34.192	......
December	54.971	75.765	......	26.373	36.410	......
Year	49.684	65.661	......	23.675	31.315	......

New York quotations cents per ounce troy, fine silver; London, pence per ounce, sterling silver, 0.925 fine.

Copper	New York		London			
	Electrolytic		Standard		Electrolytic	
	1916	1917	1916	1917	1916	1917
Jan.	24.008	28.673	88.083	131.921	116.167	142.895
Feb.	26.440	31.750	102.667	137.895	133.167	148.100
Mar.	26.310	31.481	107.714	136.750	136.000	151.000
April	27.895	......	124.319	......	137.389	......
May	28.625	......	135.457	......	152.522	......
June	26.601	......	112.432	......	137.455	......
July	23.865	......	95.119	......	125.500	......
Aug.	26.120	......	110.283	......	126.304	......
Sept.	26.855	......	113.905	......	134.071	......
Oct.	27.193	......	122.750	......	142.523	......
Nov.	30.625	......	134.659	......	155.432	......
Dec.	31.890	......	145.316	......	162.842	......
Year	27.202	......	116.059	......	138.281	......

Tin	New York		London	
	1916	1917	1916	1917
January	41.825	44.175	175.548	185.813
February	42.717	51.420	181.107	198.974
March	50.741	54.388	193.609	207.443
March	51.230	......	199.736	......
May	49.125	......	196.511	......
June	42.231	......	179.466	......
July	38.510	......	168.357	......
August	38.565	......	169.870	......
September	38.830	......	171.345	......
October	41.241	......	179.307	......
November	44.109	......	186.932	......
December	42.635	......	183.368	......
Av. year	43.480	......	182.096	......

Lead	New York		St. Louis		London	
	1916	1917	1916	1917	1916	1917
January	5.921	7.626	5.826	7.530	31.167	30.500
February	6.246	8.636	6.164	8.595	31.988	30.500
March	7.136	9.199	7.375	9.120	34.440	30.500
April	7.630	......	7.655	......	34.368	......
May	7.463	......	7.332	......	32.967	......
June	6.936	......	6.749	......	31.011	......
July	6.352	......	6.185	......	28.137	......
August	6.244	......	6.088	......	29.734	......
September	6.810	......	6.699	......	30.786	......
October	7.000	......	6.898	......	30.716	......
November	7.042	......	6.945	......	30.500	......
December	7.513	......	7.405	......	30.500	......
Year	6.858	......	6.777	......	31.359	......

Spelter	New York		St. Louis		London	
	1916	1917	1916	1917	1916	1917
Jan.	16.915	9.619	16.745	9.449	89.810	48.329
Feb.	18.420	10.045	18.260	9.875	97.762	47.000
Mar.	16.846	10.300	16.676	10.130	95.048	47.000
April	16.695	......	16.525	......	99.056	......
May	14.276	......	14.106	......	94.217	......
June	11.752	......	11.582	......	68.591	......
July	8.925	......	8.755	......	50.750	......
Aug.	8.730	......	8.560	......	51.587	......
Sept.	8.990	......	8.820	......	52.095	......
Oct.	9.829	......	9.659	......	54.159	......
Nov.	11.592	......	11.422	......	56.023	......
Dec.	10.665	......	10.495	......	55.842	......
Year	12.804	......	12.634	......	72.071	......

New York and St. Louis quotations, cents per pound, London, pounds sterling per long ton. ¼ Not reported.

Pig Iron, Pitts.	Bessemer‡		Basic‡		No. 2 Foundry	
	1916	1917	1916	1917	1916	1917
January	$21.60	$35.95	$18.78	$30.95	$19.70	$30.95
February	21.16	36.37	18.93	30.95	19.51	30.95
March	21.81	37.37	19.20	33.49	19.45	35.91
April	21.65	......	18.95	......	19.45	......
May	21.78	......	19.11	......	19.58	......
June	21.95	......	18.95	......	19.34	......
July	21.95	......	18.95	......	19.20	......
August	21.95	......	18.95	......	19.22	......
September	22.88	......	19.58	......	19.53	......
October	24.61	......	21.26	......	21.51	......
November	30.07	......	28.18	......	26.55	......
December	35.16	......	30.95	......	30.79	......
Year	$23.88	......	$20.98	......	$21.15	......

‡ As reported by W. P. Snyder & Co.

The Mining Index

This index is a convenient reference to the current literature of mining and metallurgy published in all of the important periodicals of the world. We will furnish a copy of any article (if in print) in the original language for the price quoted. Where no price is quoted the cost is unknown. Inasmuch as the papers must be ordered from the publishers, there will be some delay for the foreign papers. Remittance must be sent with order. Coupons are furnished at the following prices: 20c. each, six for $1, 33 for $5, and 100 for $15. When remittances are made in even dollars, we will return the excess over an order in coupons if so requested.

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