Mining Practices

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PREFACE

The variety in conditions and the demands for economical operation as well as pride in thorough accomplishment are the motives that stimulate mining engineers to break away from so-called established practice by improving old practices, by inventing new methods or by adopting appliances and methods in use in other industrial activities.

Mining is peculiarly an industry that demands adaptability and inventiveness. A close study of "going practice" is one of the best means for the miner and engineer to acquire that degree of susceptibility to new ideas that lays the foundation for the successful solution of their problems. There is probably no medium so potent as a good technical journal. A consistent study of such a publication becomes a valuable asset.

In this book a number of examples have been taken from the recent volumes of the *Engineering and Mining Journal* with the objective of bringing out the range in conditions that must be met with in mining operations and to indicate in both a general and also a specific way how some mining practices are developing. These articles have all appeared in the *Engineering and Mining Journal.* It is thought it will serve a useful purpose to reprint them.

The selection of the articles included in this volume has been made by George J. Young, assistant editor of the *Engineering and Mining Journal,* who has revised the several articles for this publication, made the arrangement of them and supervised the printing.

W. R. Ingalls,
Editor.

New York,
Dec. 2, 1918.

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MINING PRACTICES

The Mining Districts of Joplin and Southeast Missouri Part I

BY H. W. KITSON

Comparatively few men in the United States, and especially in the West, have visited the country's greatest zinc and lead districts — Joplin and Southeast Missouri. Until recently there has been a dearth of technical literature from either district, which may be attributed in part to the unconventional practice in mining and milling at Joplin, and to the policy of nonpublicity of the part of the companies operating in the Southeast Missouri area. It is, therefore, not surprising that, with the exception of those engineers whose activities have at some time centered in one or the other of these sections, so little is generally known concerning either. To many, Joplin is but a familiar name associated with lead and zinc somewhere in Missouri, and the Southeast Missouri lead district is either confused with it or is altogether unknown. Those members of the American Institute of Mining Engineers who attended the recent meeting of that body in St. Louis, and who for the first time visited the coal, zinc, lead, oil and gas fields of Missouri, Illinois, Kansas and Oklahoma, and read the technical papers and other literature descriptive of the region presented at the meetings, will agree that the present and future wealth of the lower Mississippi Valley is beyond any previous conception they may have formed in regard to the variety, quantity and extent of the region's mineral resources.

It is the purpose of this series of articles to present first a long-distance view of the entire area of the lower Mississippi Valley, later viewing the Joplin and then the Southeast Missouri districts at closer range and describing each in turn, first as a whole, and then subsequently in its various phases. Technical papers descriptive of the ores, geology, mining and milling of both districts have been published in recent bulletins of the American Institute of Mining Engineers and deal with various technical problems with a thoroughness that need not be undertaken in the present article, which aims only to incorporate all that has been written in a general descriptive manner with information gathered during a recent visit both to Joplin and Southeast Missouri. The miner from the mountainous West will wonder at the smoothness of topography, the absence of "blowouts" and evidence of past mountain-making and ore-making igneous activity. He will marvel that a country can have a cultivated field on the surface and a mine below, and will be puzzled to learn that he cannot stake out a mining claim on land that is more valuable for mining than for agricultural purposes. The millman will note with astonishment the countless number of small mills dotting the landscape so profusely in the vicinity of Joplin and Webb City, each with its conical tailings pile, said to carry from 0.5 to 1% zinc, and collectively appearing like huge sand dunes on a desert, and until he has seen the cost sheets and realized the economic situation as it actually exists, he may think he has found a rare field for betterment of practice both in mining methods and in milling. But in this instance hastily formed conclusions are likely to be misleading.

THE WORLD'S LARGEST ZINC AND LEAD DISTRICTS

The value of the mineral products of the lower Mississippi Valley, a region centering in the State of Missouri and extending into the adjacent parts of Illinois, Indiana, Kentucky, Tennessee, Arkansas, Oklahoma and Kansas, amounted in 1916 to nearly six hundred million dollars.[1] The district is the largest zinc and lead producer in the world, accounting respectively for one-third and one-seventh of the total of each. In oil Oklahoma and Kansas produced 40% of the total in the United States; and from the entire section there was a large production of bituminous coking and non-coking coal, and some semi-bituminous and semi-anthracite of high grade. In addition the field produces barite, bauxite, cement, clay, fluorspar, gypsum, lime, manganese, oilstone, phosphate rock, pyrite and salt, and also limited quantities of cobalt, copper, nickel, gems and precious stones, sand and gravel, stone, tripoli, and tungsten. Iron ores are not produced extensively at present, but there are large known deposits in Kentucky, Tennessee and Missouri. Other valuable deposits known to exist are asphalt, feldspar, fullers earth and pumice. The development of vast mineral resources in metals and fuel, the equally great agricultural yield of wheat, corn, cotton, fruit and other products of the soil, the abundant rainfall, moderate climate and natural facilities for railway construction and river transportation have built up great industrial and distributing centers at St. Louis and Kansas City, smelteries and refineries in Kansas, Oklahoma, Illinois and Missouri, and numerous mining and manufacturing towns throughout the entire region.

The two main sources of lead and zinc in the lower Mississipi Valley are in the Southeast Missouri section

[1] Advance figures published at the St. Louis meeting of the American Institute of Mining Engineers by permission of the U. S. Geological Survey.

and in the Joplin district covering southwest Missouri, northeast Oklahoma and southeast Kansas. Productive areas of lesser importance are in northern Arkansas, southeast Illinois, northwest Kentucky and throughout the central part of Missouri. The disseminated lead deposits of the Southeast Missouri district lie in St. Francois and Madison counties, and to a lesser extent in Washington and St. Genevieve counties, and are concentrated chiefly within a radius of approximately five miles centering at Flat River and encircling Bonne Terre on the north, Leadwood on the west and Fredericktown on the southeast. The principal mining activities are at Bonne Terre, Rivermines, Leadwood, Flat River, Elvins, St. Francois, and Desloge. The zinc-lead district of Joplin in southwest Missouri covers a roughly elliptical area, with the long axis approximately east and west, commencing in the vicinity of Springfield, Mo., and extending westward through Joplin and Webb City, and into Galena and Baxter Springs, Kan., and Picher, Commerce and Miami, Okla.

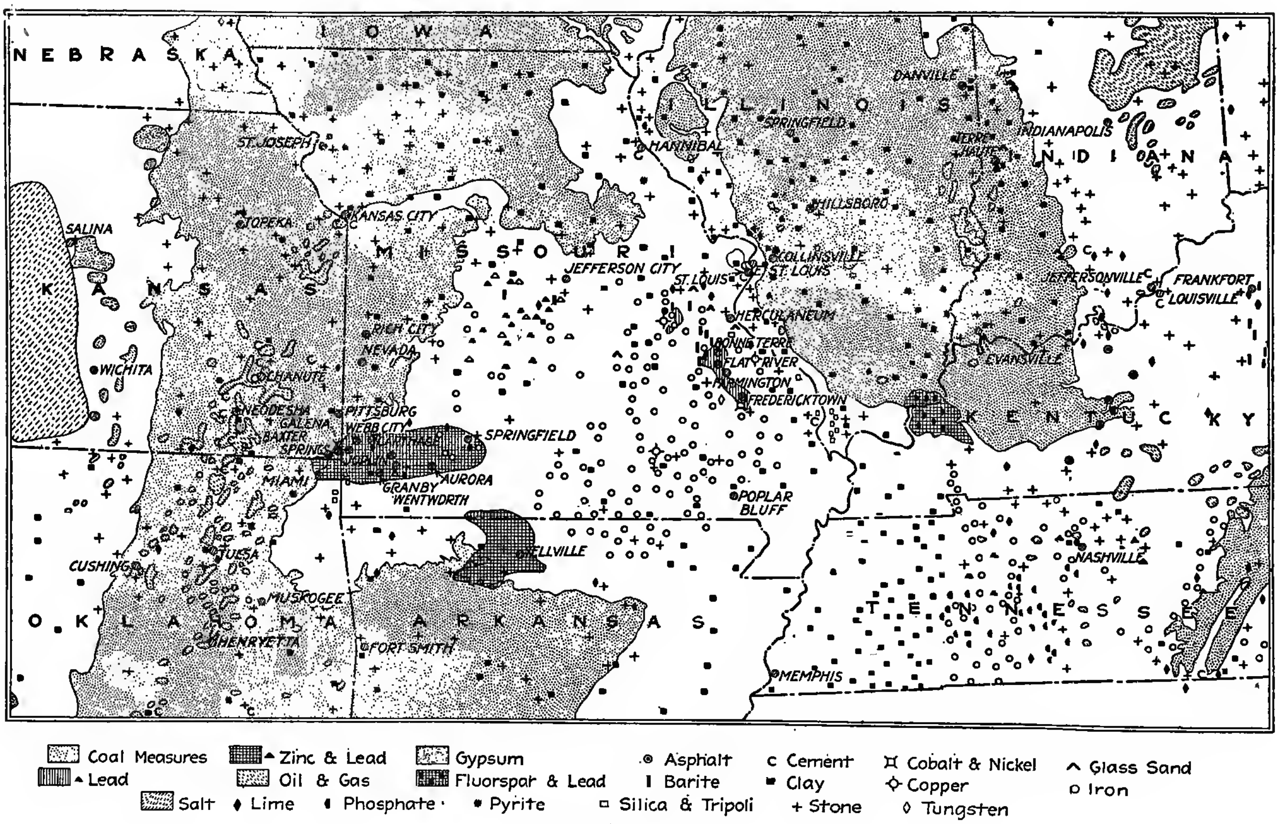


FIG. 1.

The Ozark uplift centers in southern Missouri, rising gently from the plains as an elliptical plateau which extends on the south into northern Arkansas, on the west into the extreme eastern parts of Oklahoma and Kansas and on the northeast into western Illinois. The ascent from the plains on the west, north and northeast is gradual, attaining an average elevation of from 800 to 1500 ft. and rising east of its center in the St. Francois Mountains to 1800 ft. On the south the plateau attains its greatest height at 2000 ft., forming the Boston Mountains, which range along the north side of the Arkansas River, an abrupt southern termination of the plateau region. The topography west of Springfield, Mo., and south of Joplin varies from somewhat broken hilly country to level and occasionally gently rolling prairie. The contrast of this smooth, nearly level country in the Joplin zinc district to the rugged mountainous mining districts of the West is marked. None of the mountain-making eruptives of the Rockies or southwest deserts are evident. Outcrops are rare and gossan is unknown. On the contrary, most of the soil is under cultivation even in the heart of the districts of active mining and in many cases both industries thrive on the same tract. The surface drainage channels of the lower Mississippi Valley are toward that river, but the smaller tributaries on the western slope of the Ozarks course southwestward into the Arkansas.

The St. Francois Mountains of Madison, Iron and St. Francois counties rise from the plateau near its eastern edge, a structural center of outcropping granite and porphyry of pre-Cambrian age — a basement underlying the entire region — sloping with a gradual flattening of dip radially from this eminence under its doomed overburden of later sedimentaries.[2] An east-west section of the Ozarks is shown in Fig. 4. In the Southeast Missouri disseminated lead district, in the vicinity of Bonne Terre, the granite underlies sedimentaries at a depth of 650 ft. below the surface. Northwest, at Sulli-

[2] "Geology and Mineral Deposits of the Ozark Region," by H. A. Buehler, Bulletin, A. I. M. E., October, 1917.

van, Franklin County, the granite lies at a depth of 1200 ft.; at Rolla, Phelps County, at a depth of 1800 ft.; at Carthage, in the Joplin district, at a depth of 1850 ft., and at Kansas City and St. Louis, respectively, at 2350 and 3550 ft. The contact between the granite and the overlying Cambrian sandstone is unconformable and represents a period of sedimentation subsequent to an era of erosion during which the granite surface was deeply carved by drainage systems. This sandstone formation, known as the La Motte, has an average thickness of 200 ft. It is conglomeratic at the base and merges with alternate layers of sandstone and dolomite into the overlying Bonne Terre dolomite which outcrops at the surface in the Southeast Missouri lead district and constitutes the ore-bearing formation. The Bonne Terre dolomite is of Cambrian age, has a thickness of 350 ft. and rests conformably upon the La Motte sandstone. The higher formations of the Cambro-Ordivician period have been removed by erosion from the immediate vicinity of the district and outcrop further out from the granite area in the order shown in the columnar section in Fig. 2. The still higher Silurian, Devonian, Mississippian and Pennsylvanian formations outcrop throughout the Ozark region in roughly concentric succession about the Cambrian, as shown in the areal map in Fig. 3. With two exceptions comparatively little faulting or folding has occurred within the region. The exposures of granite in the St. Francois Mountains are the result of faulting, with a subsidence of the later surrounding formations, some of which faulting is evident in the Bonne Terre-Flat River district. In northern Arkansas faulting is associated with ore deposits and in the Joplin district also some faulting has occurred.

There are two important horizons of zinc and lead mineralization. The ore deposits of the Southeast Missouri disseminated lead district occur in the older of these two — the Bonne Terre dolomite of Cambrian age. In the Joplin zinc-lead district the ore deposits occur in the cherty limestone of the Boone formation, of the later Mississippian age. Both of these formations outcrop in their respective districts, and, although widely separated geographically as well as stratigraphically, the ore deposits bear marks of similarity in mode of occurrence and genetic origin.

Some mineralization has taken place close to the upper contact of the La Motte sandstone near the ore deposits in the Bonne Terre dolomite of the Southeast Missouri district; and in the Joplin district the Chester (sandstones and limestones) of the upper Mississippian is the horizon of the first discovered shallow lead-zinc deposits. An intervening period of erosion removed most of the Chester formation prior to the succeeding unconformable deposition of Cherokee sandstone and shale of early Pennsylvanian age. Isolated remnants of the Chester are to be found scattered throughout the Joplin district and upper Boone area of outcrop, deposited in ancient sink holes in the Boone which protected the later formation during the ensuing period of erosion. A similar condition of deposition in sink holes and depressions of erosion has protected the Cherokee of the Pennsylvanian from the present erosion, and with the exception of these outlying remnants the Chester and Cherokee do not constitute the surface rock except at the extreme western border of the Oklahoma-Kansas portions of the Joplin district. Some mineralization also occurs in the Pennsylvanian in connection with the sink-hole deposits near their contact with orebodies in the Mississippian.

Other ore deposits of lesser importance occur in the intervening formations between the Bonne Terre and the Boone. Bordering the Bonne Terre area in the shale, Derby and Doe Run formations and the Potosi cherty dolomite of Cambrian age. In Washington and Jefferson counties galena associated with barite occurs

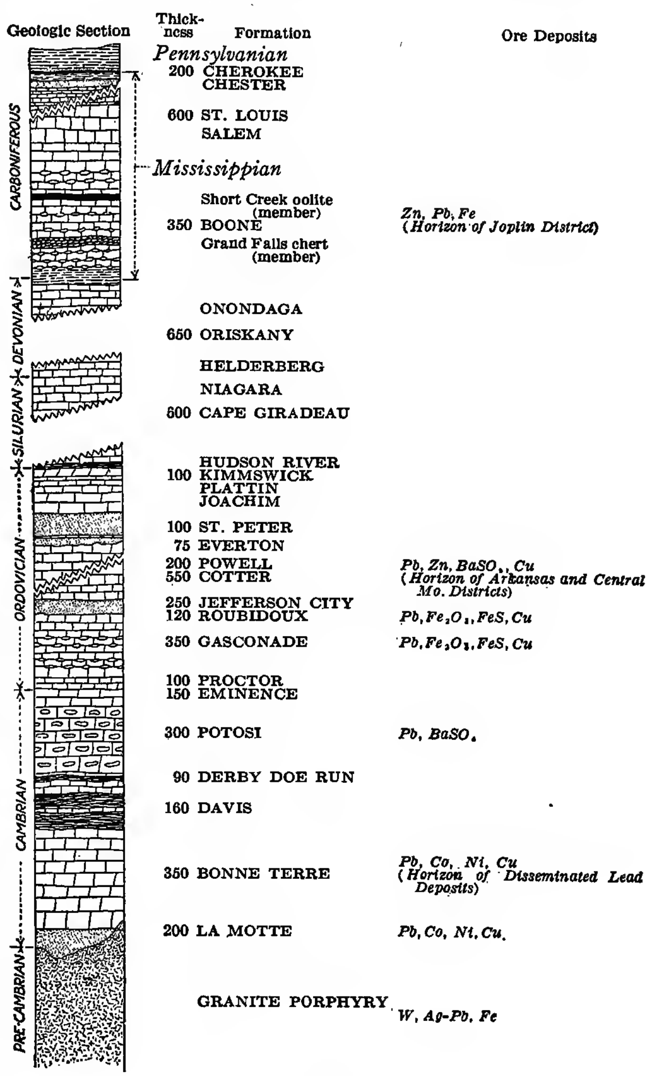


FIG. 2

in joint planes and horizontal seams in this heavy-bedded gray dolomite. The beds of chert interstratified with the Potosi dolomite are the lowest in order of stratification in the Ozark region. The irregularity and shallowness of these ore deposits have not produced any mines of consequence. Shallow veins of the fissure type containing galena and barite occur in Franklin County. Deposits of lead, zinc and barite occur in the Jefferson City dolomite of Ordovician age and have been productive in the north-central part of Missouri. In northern Arkansas lead and zinc have been mined from the Cotter and Powell formations of the same age. In St. Genevieve County these formations have produced some copper. In the vicinity of Fredericktown, southeast of the Flat River district, cobalt, copper and nickel and some disseminated lead occur in the La Motte sandstone and lower Bonne Terre. Carbonates of copper have been mined in Shannon County, occurring on the contact of the sedimentaries and the granite.

Veins of argentiferous galena occur in the pre-Cambrian granite, in which mines were opened up in the early days. These veins are now being worked on a small scale for tungsten, which occurs in association with the galena as huebnerite. Specular-iron ore has been extensively produced from the pre-Cambrian porphyry in the Iron Mountain and Pilot Knob districts. Hematite and limonite ores also occur in Crawford County, east Missouri and Arkansas, and barite is produced from Washington and Jefferson counties in large amounts.

Throughout this vast area of mineralization little or no metamorphism of the sedimentaries has taken place. Igneous activity has been confined to but two known intrusions of an age later than the sedimentaries. Small basic dikes occur as intrusives in the granite and porphyry in the St. Francois Mountains, and in Camden County the sedimentaries are cut by a dike of pegmatitic character. The granite and rhyolite porphyry belong to the oldest series in the region and are thought to have formed part of one magma and to be of Laurentian age. The veins occurring in the pre-Cambrian igneous rocks are undoubtedly depositions resulting from emanation of gases and solutions of an expiring period of volcanism, and analyses of the older rocks show that they contain minute quantities of both lead and zinc.

In the light of later theories on ore deposition, the deposits of lead and zinc in the sedimentaries of the Ozark region are attributed to a deposition in channels from waters descending from the surface containing lead and zinc in solution, metals derived from the eroded portions of the older rocks and precipitated both in open cavities and by metasomatic replacement of the limestone. Calcite, dolomite and chert of a secondary character are the common associated gangue minerals in the Joplin and Southeast Missouri districts, and their presence is evidence of the action of cold descending carbonated solutions of an acid character. The prevalence of an extensive artesian underground circulation accounts for the migration of such solutions over considerable distances governed by the impervious beds and the erosion over a wide area of drainage of all the stratified formations across their dip.[3] Such action is shown diagrammatically in Fig. 5. In the absence of alteration products and the metamorphism of the rocks that invariably accompanies the genesis of orebodies formed from solutions of plutonic origin, the possibility that such a source could account for the deposits of zinc and lead in the Joplin and Southeast Missouri districts seems remote.

Unlike most mining sections of the West, where public land bearing valuable minerals is given prior right in the mining laws of the United States over agricultural claims, the Missouri mining districts have been parceled into tracts of 160 acres and the region was taken up by an agricultural people long before its value as mineral land was discovered. This was the case in the Joplin district of southwest Missouri and southeast Kansas. The mining land in northwest Oklahoma was originally known as Indian Territory and remains in the ownership of the Indians of that section. In southeast Missouri the land is partly surveyed into sections, but the eastern part is cut up into irregular areas, a relic of the old land-grant system.

The earliest records of mining in Missouri date back to the French occupation in the eighteenth century. The Mississippi was discovered by Spaniards under De Soto in 1541, but abandoned by them for the greater riches in gold of Mexico and Peru. In 1682 La Salle came down the Mississippi from the upper valley, at that time being explored and settled by the French, and, arriving at the mouth of this river, took possession of the whole territory drained by the Mississippi and its tributaries in the name of King Louis XIV of France and to it he gave the name of Louisiana. This vast acquisition embraced more than half the North American continent, including, as it did, all the territory north of the Gulf of Mexico from the Rockies on the west to the Alleghanies on the east and extending northward through Canada to Hudson Bay. Toward the end of the series of wars that took place between England and France, beginning in 1689 and ending with the Treaty of Paris in 1763, mining in Missouri according to E. R. Buckley was started in Washington County at Mine à Renault in 1725. Operations at Mine à Gebore were conducted on a small scale between 1742 and 1762. In 1767 Pierre Laclede founded a trading post on the Mississippi River and named it in honor of Louis XV, and in 1775 it had become a well-known fur-trading depot. At the Treaty of Paris, France ceded all her territory in Canadian North America to England, and the territory south of Canada was ceded to Spain. The French who remained in the territory, however, evidently continued their mining operations, as Mine à Layne was discovered in 1795, Mine à Manteo on Big River in 1799, and Mine La Plate about the same period. Winslow estimates that during this century St. Francois County produced about 1000 tons and Washington County 19,000 tons of lead ore. Mine à Burton was discovered about 1763 and "Old Mines" dates as far back as the earliest discovery and has been worked intermittently up to present time. Mine La Motte is among the oldest in the district and is being worked today.

The Napoleonic wars returned the Louisiana province to France in 1801, and two years later this great territory of about 900,000 square miles was acquired by the United States, during the Jefferson administration, for $15,000,000. In 1804 Capt. Stoddard, of the U. S. Army, succeeded the Spanish commandant at St. Louis, and the region was organized into the Territory of Louisiana, and St. Louis made the capital. From this territory the states of Louisiana, Arkansas, Missouri, Iowa, Kansas, Nebraska, Wyoming, Montana, North and South Dakota and parts of Minnesota, Colorado and Oklahoma were formed. In 1812 Louisiana became a state, and the name of the territory was changed to the Missouri Territory. Missouri came into the Union as a slave state under the Missouri Compromise in 1820. The railroad land-grant system originated in 1835, and the transcontinental roads received vast tracts of land aggregating millions of acres along their routes. The

3 C. E. Siebenhal, "Bull. 606," U. S. Geol. Surv.

first railroad in the state was opened with 38 miles of track in 1852. By the Preëmption Act of 1841 any genuine settler could take up 160 acres of public land and make his payments on long and easy terms at a rate fixed in 1862 of $1.25 per acre.

In the Joplin district the first-reported discovery was made by William Foster at Granby in 1850. This, like other succeeding discoveries, was made by accident in the course of well digging for water. Galena was the ore mined from the shallow deposits where shafts were first sunk. Small furnaces were soon after erected and lead bullion was hauled by wagon to Fort Smith, Ark., and transported by boat to New Orleans, New York and Boston. In 1856 Blow & Kennet started operations on a larger scale. Lands were leased and furnaces erected, and railroad construction started. The Civil War put a stop to development in 1861, but the mines were operated alternately by both sides during that period. Right after the war the Granby Mining and Smelting Co. was organized, the various interests combined and operations

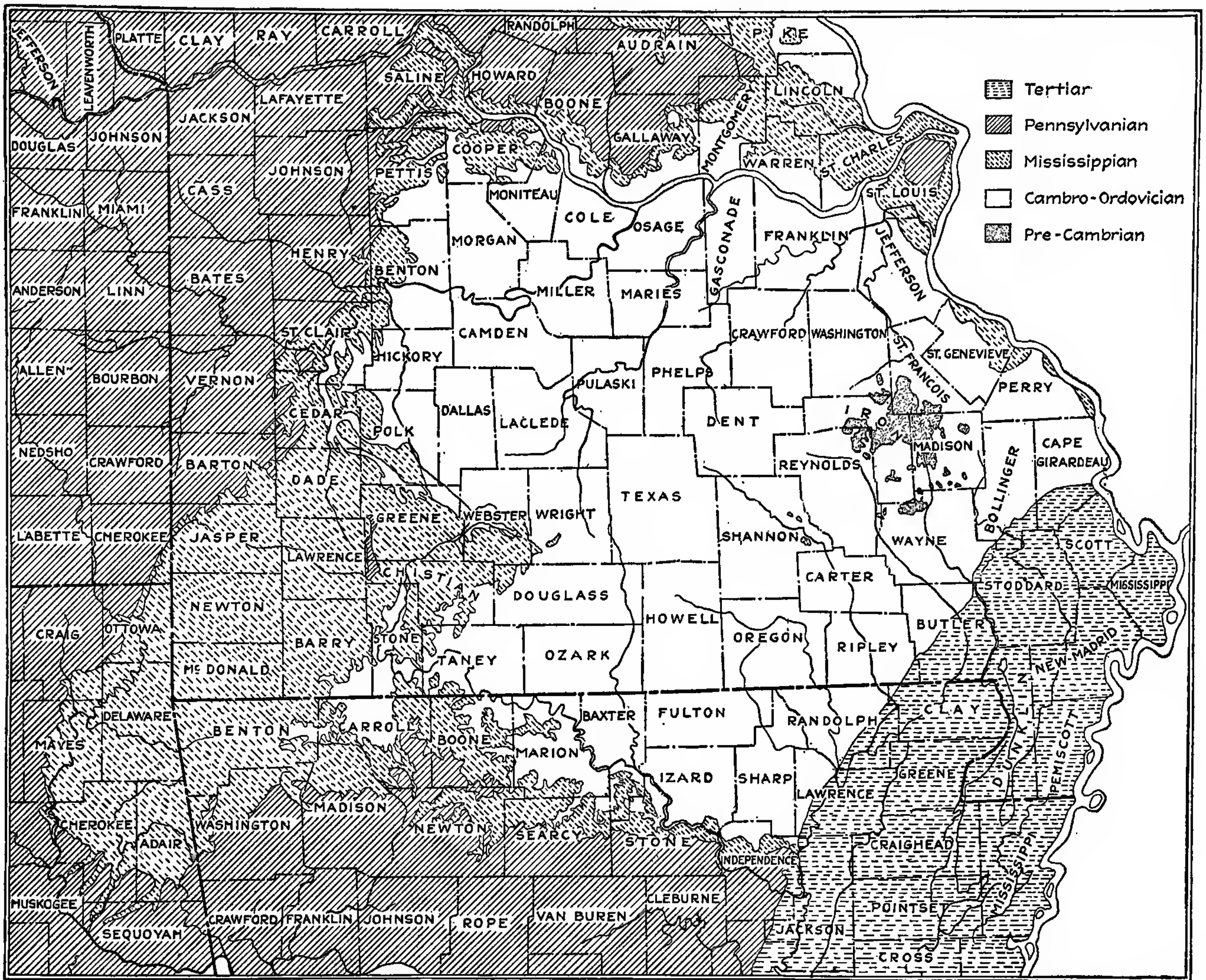


Fig. 3. Geological Map of the Ozark Region — After Buehler

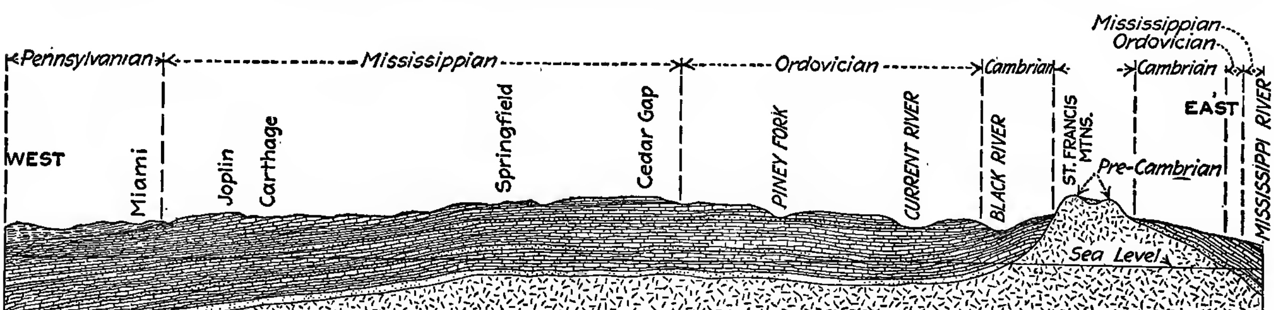


Fig. 4. Geological Section Through Joplin and Southeast Missouri — After Buckley

resumed. Galena and cerrusite were the first ores mined. Calamine and sphalerite were later recognized, and upon completion of the railroad to Granby, zinc ores were shipped.

For the first 30 years galena and calamine were the main ores of the district, but later sphalerite ores took the lead, with calamine second and galena third. The first modern concentrator was built in 1898, the ores up until that time having been concentrated in hand jigs.

The history of the district is one of spasmodic rushes and booms, followed by hasty production on a profitable if wasteful scale. The land, primarily agricultural, was largely settled by a farming people who knew little of mining. For a number of years Granby was the center

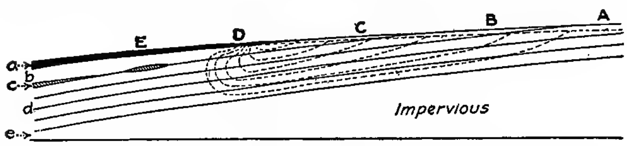


FIG. 5. DIAGRAM SHOWING OZARK ARTESIAN CIRCULATION—AFTER SIEBENTHAL

of prospecting. Later, strikes were made by accident at Joplin and at Oronogo, at which points mining activity became intense. Discoveries followed at Webb City, again by accident, and another rush was made to this section, and such has been the development of the region from the beginning. Strike after strike was made, with intense exploitation in each section, until there became a scattered group of camps with Joplin as a center. Between most of the camps there are wide stretches undeveloped that may prove to be valuable mining ground. There are well-known instances where mining activities have linked together neighboring mining camps by the lateral extension of the orebodies, as from Oronogo on through to Duenweg, a stretch of seven miles, and more recent operations point to a probable repetition of such expansion in other parts of the district. The ground between Duenweg and Granby, under the stimulus of efficient prospecting, may prove to be a continuous run of ore. The undeveloped stretch between Carthage and Neck City forms another possibility for the future.

Early exploitation was by the grubstake plan, prospecting for the shallow lead deposits at first and later the shallow zinc deposits. The mineral had to be free and amenable to hand picking, followed by hand jigging, to be profitable. Problems of concentration came with the exhaustion of the shallow deposits and the discovery of the deeper and lower grade orebodies. Out of all this the present milling practice was evolved, and its low cost of operation has made profitable the mining of ores producing as low as 2% of zinc concentrates. At present, however, many of the lower grade mines have been forced to close owing to the marked increase in wages and cost of supplies and the absence of a corresponding advance in the price of ores. This condition has stimulated prospecting in the newer fields of Oklahoma and Kansas, where ores of higher grade can be made to yield a better profit on the present market.

The earliest mining in the Southeast Missouri district proper was at Mine à Joe in 1801. These were shallow lead deposits in the tract later known as the Bogy Lead and Mining Co. and in 1887 purchased by the Desloge interests. Operations at the Pratt mine, in the vicinity of Bonne Terre, started as early as 1820, but the disseminated ores were not discovered until 1864. Four years later the St. Joseph Lead Co. was organized and diamond drilling started in the Bonne Terre section. The company purchased the land from Anthony La Grave, a property previously owned by the Valle and Aubuchon families. The Doe Run company started in 1887, but has recently been merged with the St. Joseph Lead Co. The Federal Lead Co. began operations in the Flat River district in 1902, later acquiring the properties of the Irondale, Derby, Central and Field companies. The St. Louis Smelting and Refining Co. commenced in 1898 by purchasing the holdings of the Flat River Lead Company.

The region is well served by railroads connecting the productive districts with the smelting and marketing centers. West of the Mississippi the Kansas Southern Ry. runs from Kansas City to Port Arthur, Tex. It is the short line from Kansas City to the southeastern Kansas coal field and the Joplin zinc-lead district and traverses the coal-producing section of eastern Oklahoma and western Arkansas. The Missouri Pacific R.R. connects St. Louis and Kansas City, serving the coal fields of northwestern Missouri, passing through Barton County, Mo., Crawford and Cherokee counties, Kan., the Joplin-Webb City zinc-lead territory, and the oil and smelting districts of Kansas and Oklahoma. It also reaches through subsidiary lines the coal mines of Arkansas and the southeast Missouri lead district. The St. Louis-San Francisco Ry., known as the Frisco Lines, runs from St. Louis and Kansas City to the heart of the Pittsburgh, Kan., coal fields and the Joplin-Miami lead district, passing to the Tulsa and Glenn Poole, Okla., oil and gas country, and with four district lines to Texas handles a large mineral tonnage. The Frisco Lines also serve most of the smelteries.

In 1916 there was produced from the smelteries in the lower Mississippi Valley 512,000 tons of spelter, or 75% of the total production of the United States. Of this amount Kansas and Missouri smelteries produced 34%, Illinois 28%, Oklahoma 31% and Arkansas 7%. Natural gas in Kansas and Oklahoma has made this section desirable for the location of smelting plants of which there are 17 and 14 respectively. In Illinois there are 11 plants, in Missouri three, in Arkansas three and in Indiana one.

The lead smelteries of the region are situated close to the lead-producing districts and in 1916 yielded 236,000 tons of lead. The principal smelteries for the southeast Missouri district are those of the St. Louis Smelting and Refining Co., at Collinsville, Ill.; the Federal Lead Co., at Federal, Ill.; the St. Joseph Lead Co., at Herculaneaum, Mo., and the Desloge Consolidated Lead Co., at Desloge, Mo. The Eagle-Picher Lead Co., one of the largest pigment manufacturers in America, has plants at Joplin and Webb City, Missouri.

The Mining Districts of Joplin and Southeast Missouri Part II

BY H. W. KITSON

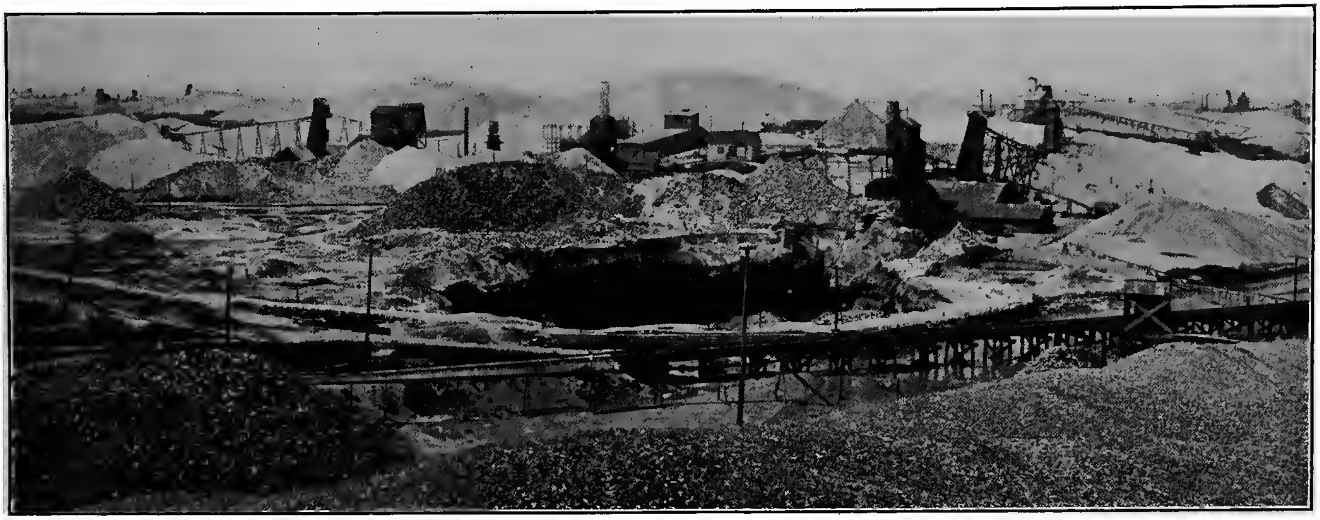


FIG. 6. TYPICAL VIEW IN THE SHEET-GROUND AREA, SHOWING EFFECT OF A CAVE

The Joplin district covers a territory of 3000 square miles that includes parts of Missouri, Oklahoma and Kansas, and consists of numerous scattered camps and groups of camps situated along more or less well-defined belts, shown by dark areas on the map in Fig 7. The main productive section in Missouri lies in Jasper and Newton counties; but the district extends also into Lawrence, Dade, Christian and Greene counties, all of which are in the southwestern part of the state, whence the name Southwest Missouri zinc district by which it is also known. Eastward from the city of Joplin the Missouri section extends to Springfield, a distance of 60 miles, and westward to the Kansas state line. North of Joplin the active camps take in Webb City, Carterville, Oronogo, Waco, Neck City and Alba, and to the south Granby, Spurgeon, Spring City and Saginaw, making a width of 30 miles on a line northward from Granby through Duenweg to Neck City.

In Oklahoma, the Joplin district lies entirely within Ottawa County, in the northeast corner of the state, extending from the vicinity of Miami, 40 miles west of Joplin, to the Kansas state line on the north, taking in Commerce, Quapaw, Sunnyside, Douthat, St. Louis, Century, Tar River, Cardin and Picher. The Kansas section lies in the southeastern part of Cherokee County, and, together with the Oklahoma section, is sometimes called the Miami district. The important communities in the Kansas section are Galena, Treece and Baxter Springs. The main line of the St. Louis and San Francisco R.R. passes through Miami, Baxter Springs, Galena, Joplin, Carthage and Springfield; and the Missouri Pacific, Kansas City Southern, Missouri, Oklahoma & Galena, and Missouri, Kansas and Texas railways connect most of the other camps with Joplin and smelting centers. This city is therefore an important distributing point and an industrial as well as geographical center for the entire district; with the Southwest Missouri Ry. inter-urban electric lines connecting Galena, Webb City, Carterville, Oronogo, Neck City, Carthage and Duenweg, and a population of nearly 50,000, mine workers are able to commute to the various mines and live comfortably in the larger communities.

The panoramic view from the top of any of the innumerable "chat" or tailings piles around Joplin presents the appearance on all sides of a wide level plain, and mining communities can be observed for many miles. The various camps are marked by their mounds of "chat" piles, derricks and dummy elevators, and between them are to be seen cultivated fields and patches of woods.

The country is somewhat broken by the drainage system of the Spring River from its source west of Springfield along its westward course north of Joplin and between Joplin and Baxter Springs, where the surface has a gently rolling relief. The Spring and Neosho rivers join a short distance southeast of Miami, this district lying mostly between the angular convergence of the two, and having a topography similar to that of Joplin, level and unbroken except for one or two low hills, as at Blue Mound, in Kansas, near the state line north of Picher. The region is on the western border of the Ozark Uplift, and elevations range from 1000 ft. at Miami and Joplin to 1250 ft. and more in the eastern district.

The surface conditions of the district offer great facilities for railway construction and highways, although the rapidly growing amount of traffic over the latter has outpaced the rate of upkeep, and the roads are consequently in poor condition. The proposed concrete county road from Joplin to the new Miami fields is therefore a much-needed project. Railroad spurs have recently been extended to Picher, St. Louis and other camps in the Oklahoma district; and in the near future the important producing centers of this district will all be connected to the main lines of one or more of the important railroad systems. On the other hand the extreme flatness of the country offers no opportunity for favorable millsites and consequently mills are all equipped with elevators inside, and "dummy elevators" with launders outside. By means of the latter the "chats" are heaped in conical piles by stages, and at the larger plants accumulate to considerable heights.

Each of the many small mills that dot the landscape is situated over a hoisting shaft, and surface or underground haulage from adjacent properties to a central number of different lessors from the sale of concentrates or ores from an equal number of different 5 to 40-acre tracts.

In the Missouri and Kansas sections the average unit of operations is 20 acres and in Oklahoma 40 acres. The reason for this is that most of the mines are worked under a leasing system. Mining rights are leased on land owned in small tracts by the descendants of the original settlers of Missouri and Kansas, who were altogether an agricultural people, and by the Indians in Oklahoma who own their land in tracts granted by the Government. Many of the larger companies have purchased considerable tracts of land outright and some have purchased the right to mine without royalty, but do not possess the surface rights. The large number of small independent operators obtain leases usually for 10 years and pay royalties on shipments of ore at rates ranging from 2½ to 25%, the average being about 7½% in the Oronogo-Duenweg sheet-ground mines, which are the lowest of all in grade. Since the boom in the newer fields north of Miami premiums are asked in addition when the leases are taken, and royalties average from 10 to 15%. Some of these premiums have run as high as $260,000 on the larger tracts. When a lease is solicited, a one-year's option for drilling and proving up the ground is given, subject to acceptance by the lessor at the end of that time, together with the previously drawn up terms of lease.

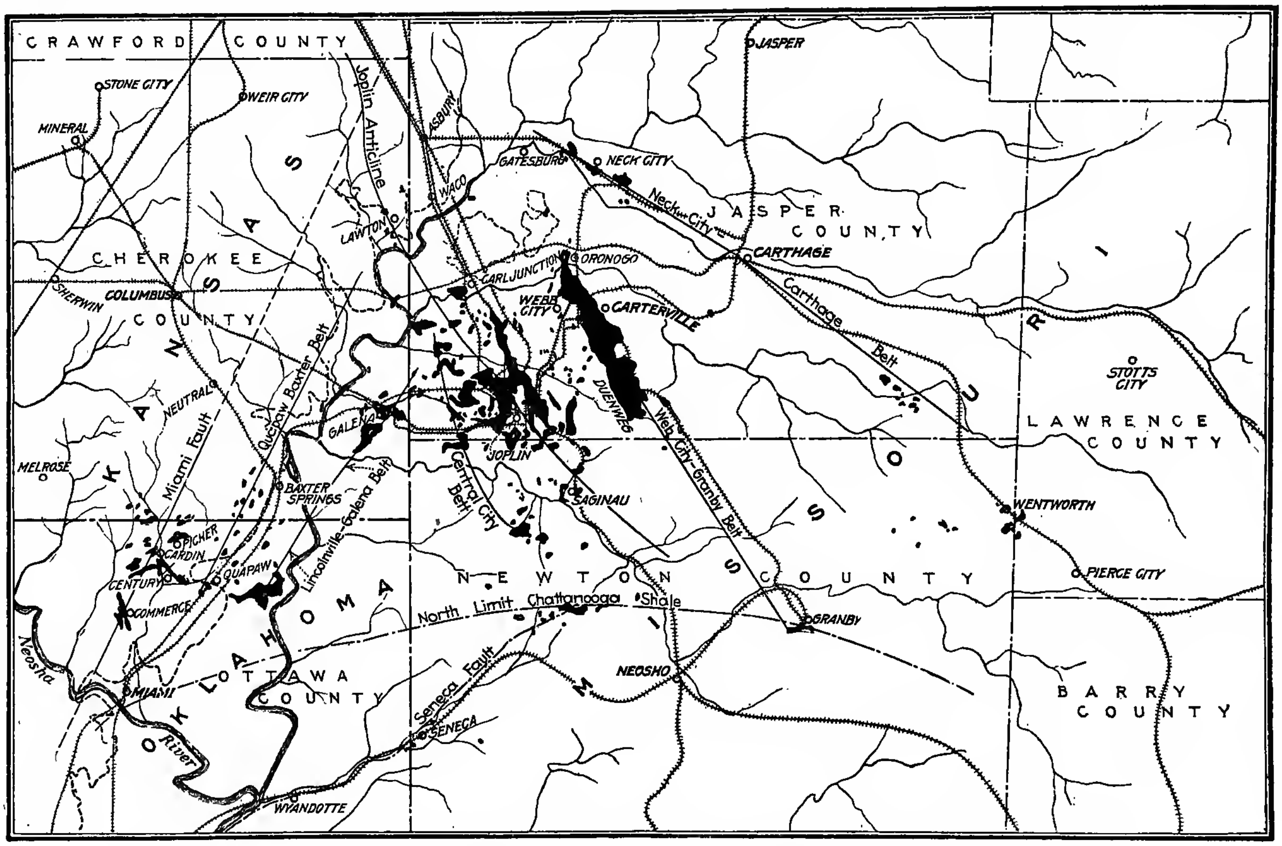


FIG. 7. MAP SHOWING MINED AREAS IN THE JOPLIN DISTRICT, MISSOURI, OKLAHOMA AND KANSAS

mill is rare. The prevailing hoisting practice of the district is by means of buckets, which system enables a light winding engine to be installed at the top of the headframe or "derrick," as it is more properly called. With flat surface conditions this light type of structure has the advantage of giving considerable hoisting height at a low initial cost, which condition is desirable for proper dumping headroom over mill bins or "hoppers" that are most conveniently and customarily built as an integral part of both derrick and mill. Some of the larger and more recent properties, however, are using skips and centralizing hoisting and milling operations. Such practice, however, is only practicable where the operators are not obliged to pay separate royalties to a

Individual orebodies are of varying size, but the range of life is from six months to six years. Under such conditions it is apparent that little capital can be put into plant and development, and the result has been methods both in mining and milling that will yield a marketable product at the lowest possible combined cost of plant and operation, with little regard for loss in pillars underground or in the tailings on top. The wisdom of such policy from a commercial standpoint is evident. More elaborate methods have been attempted, but they proved commercial failures. Exact data on mine and mill extractions in the district are, in the nature of the methods used, almost impossible to obtain. It is estimated, however, that from 10 to 15% of the orebodies remains in pillars and that from 58 to 70% is recovered in the mills.

individual ores. Concentrates of higher or lower grade than 60% are respectively allowed or penalized $1 for each unit above or below this base. An iron content of 1% is accepted, but above this amount a penalty of $1 per unit is imposed. The maximum allowable lead content in zinc concentrates is fixed at 0.3%, and amounts over this are penalized indirectly in calculating base-price quotations. Zinc silicate ores are sold on a base price for 40% zinc metal, with premiums and penalties corresponding to those established for sulphides. Concentrates of lead sulphide are bought in a similar manner, quotations being fixed on a base of 80% lead metal, with the same rate of premium or penalty per unit above or below 80% as for zinc. There are, however, no penalties or premiums for zinc or iron.

TABLE I. PRODUCING COMPANIES OF THE JOPLIN DISTRICT

MISSOURI		
Joplin		
	Blende	Galena
Bumble Bee M. Co.	242,140	......
St. Regis M. Co.	264,830	......
Little Martha M. Co.	149,150	......
Fifteenth Street M. Co.	171,300	......
Gibson M. Co.	120,800	......
Consolidated-Inter-State M. Co.	83,410	......
Swartz M. Co.	92,290	......
Malcona M. Co.	65,450	......
Ethel Gray M. Co.	62,270	......
New Geronimo M. Co.	63,100	......
Eaglewood M. Co.	55,370	......
Wise Guy M. Co.	40,270	......
Paragon M. Co.	62,700	......
Malsbury M. Co.	25,480	......
Rabbits Foot M. Co.	7,690	......
Sundries	241,610	241,210
Totals	1,747,860	241,210
	Calamine	
Falkner M. Co.	63,800	......
Webb City—Carterville		
	Blende	Galena
Bertha A. M. Co.	510,730	......
American Z. L. & S.	458,660	......
Kirkwood M. Co.	161,150	......
Queen Esther M. Co.	170,280	......
Concord M. Co.	156,140	......
Unity M. Co.	86,190	......
Ben Franklin M. Co.	72,280	......
Mahatmah M. Co.	65,820	......
Nearby M. Co.	47,510	......
Lucy Bell M. Co.	35,690	......
Shawgo M. Co.	13,230	......
Sundries	309,340	220,710
Totals	2,087,020	220,710
Duenweg		
	Blende	Galena
Athletic M. Co.	803,790	......
Coahuila M. Co.	545,050	......
St. Regis M. Co.	206,180	......
What Cheer M. Co.	65,660	......
Total	1,620,680	......
Oronogo		
	Blende	Galena
Mutual M. Co.	227,680	175,300
Oronogo Circle	192,000	......
Totals	419,680	175,300
Aurora		
	Blende	Galena
Red Wasp M. Co.	178,000	......
Bonanza M. Co.	159,800	......
Mills & Co.	62,180	......
Murray & Co.	12,660	......
Total	412,640	......
	Calamine	
M. A. & F. R. Co.	80.000	......
Mathews-Phelps M. Co.	70,000	......
Sandridge M. Co.	44.240	......
Consolidated M. Co.	34,000	......
Total	228,240	......
Granby		
	Blende	Galena
American Z. L. & S.	66.800	......
	Calamine	
Perkins M. Co.	1,400,000	......
J. R. Underwood M. Co.	80.000	......
Woodcock M. Co.	80,000	......
Buick M. Co.	70,000	......
Total	1,630,000	......
Belville		
	Blende	Galena
Tennessee M. Co.	130.800	......
Culbert L. & Z. Co.	45,940	......
Kirkpatrick M. Co.	19,330	......
Total	196,070	......
Alba-Neck City-Purcell		
	Blende	Galena
Joplin Concentrating	67.000	......
Sponable M. Co.	66.460	......
Total	133.460	......
KANSAS		
Galena		
	Blende	Galena
Diplomat M. Co.	130,730	......
South Side M. Co.	87,000	17,610
Andrew Bros. M. Co.	85,510	......
Wayland M. Co.	87,520	106,010
C. W. Squires M. Co.	72,000	......
Boor M. Co.	48,510	......
Brenz M. Co.	9,990	......
Galena L. & Z. Co.	......	3,040
Sundries	......	116,660
Totals	521,260	243,320
Waco-Lawton		
	Blende	Galena
Eastern L. & Z. Co.	360,160	......
Smart & Crane M. Co.	254,540	......
Total	614,700	......
OKLAHOMA		
	Blende	Galena
Picher Mines	2,630,400	......
Underwriters Land Co.	936,850	......
Mahutska M. Co.	820,470	......
Montreal M. Co.	659,740	......
U. S. Smelting Co.	600,000	......
Bilharz M. Co.	398,240	......
Admiralty Zinc Co.	322,290	......
Skelton No. 2	300,040	......
Anna Beaver M. Co.	281,600	......
Bethel M. Co.	278,840	......
Ramage M. Co.	191,130	......
Metals M. Co.	130,000	......
C. Dawson, trustee	122.560	......
Skelton No. 1	87,890	......
Lucky Kid M. Co.	85,350	80,880
Central L. & Z. Co.	117,540	......
Lion M. Co.	115,690	......
Piokee L. & Z. Co.	83,960	......
Lucky Bill M. Co.	80,000	......
Dewdrop M. Co.	87,080	......
Haynes M. Co.	78,970	2,860
Woodchuck M. Co.	54,360	......
Croesus M. Co.	......	85,960
Blue Goose M. Co.	......	65,440
Sundries	......	243,220
Totals	8,463,000	478,360

Total production for the week ended Oct. 20, 1917, was: Blende, 16,283,170 lb.; calamine, 1,922,040 lb.; lead, 1,358,900 lb.

Leases usually provide for the removal by the lessee of all machinery and equipment at the expiration of his lease, and as the standard practice of the district in both mining and milling is adaptable in all mines it enables an operator to salvage and transport the machinery of his mill, hoisting and other equipment almost intact to another part of the district and continue operations on a new tract under a new lease.

Ores and concentrates are sold at the mills through ore-buying agencies representing the various smelters, according to the time-honored custom of the district, at so much per dry ton. For zinc sulphides the base price per ton is quoted by the buyers for a 60% zinc metal content and is subject to fluctuation according to the market price for spelter and special characteristics of

In the Joplin district grades of ore mined are considered in terms of base grade concentrates made. Thus a 2% ore is understood to mean one from which 100 tons would produce at the mill 2 tons of combined concentrate of the base grades given. Zinc concentrates vary in grade in different parts of the district from 50 to 65% metal, and lead concentrates from 70 to 83% metal.

Separate description of the many independent operations in the Joplin district is not necessary, as their number is great and the difference slight; but reference to the holdings of some of the larger companies is of interest. The American Zinc, Lead and Smelting Co. holds in fee thousands of acres in groups located north of Joplin, Webb City, Carterville and Oronogo, and south of Joplin in small scattered tracts extending from the Oklahoma line to Granby and including the properties absorbed in 1816 of the Granby Mining and Smelting Co., and recent acquisitions along the western border of present activities in the Kansas field. In the Oklahoma section the most important producer is the Eagle-Picher Lead Co., with rich and extensive tracts leased and sub-leased at Picher, Cardin and south of Tar River.

The Miami Zinc Syndicate has leases aggregating 4000 acres in tracts under development west of Baxter Springs in Kansas; and the American Metal Co. has a vast number of small tracts in the northwestern part of the Missouri section and extending from northeast to southwest across the Kansas section along the so-called Miami fault belt. The properties of the Commerce Mining and Royalty Co. lie mostly in the Miami field to the west of Miami, Century and Picher, extending along the Miami fault belt into the southern part of Kansas. This company was the first in the western fields, having struck ore at Commerce in 1905. The Vinegar Hill Zinc Co. has a comparatively smaller tract west of Baxter Springs and west of the Commerce company's properties, and the U. S. Smelting Co. has developed and completed a large mill on tracts west of the Vinegar Hill ground, and holds some leases northeast of Waco. Other large development enterprises in the district are: The Chanute Spelter Co., with 16,000 acres; the Butte and Superior; the Waco Mining Co., with 71,000 acres; P. B. Butler, with 6500 acres; the Danglade, and the Church & Mabon interests. The most important producers at present are those given in the table, grouped according to their respective camps and showing the relative production in various classes of ores and concentrates as officially reported for the week ended Oct. 20, 1917. The base range of prices for ores and concentrates for the week was: Blende, from $60 to $75, with an average of $67.50 and an average settlement price of $66. Galena was sold in prices ranging from $70 to $80 per ton and calamine $35 to $38 per ton.

Zinc silicate ores are mined in certain parts of the district, as at Granby, Duenweg and Joplin City. Granby is the chief producer, and in 1916 averaged about 50 tons per day. The American Zinc, Lead and Smelting Co. is at present producing from its own and sub-leased properties in Granby about 40 tons of this ore per day. But little lead-oxide ore is found. The average rate of production in silicate ores from all camps is about 75 tons per day.

According to the annual statistics published by the *Journal*, the production in ores and concentrates of the Joplin district in 1917 was 550,000 tons. Of this amount 80% was as blende, 6.5% calamine, 13.5% galena. Missouri produced 58.5% of the total tonnage, Oklahoma 34% and Kansas 7.5%. Considering the proportional production of total blende ores, which was 442,000 tons, Missouri accounted for 56%, Oklahoma 36% and Kansas 8%. Most of the calamine came from Missouri mines and amounted to 35,695 tons, Oklahoma producing only 65 tons and Kansas none. Of the lead ores produced, which altogether amounted to 72,500 tons, Missouri yielded 55%, Oklahoma 39%, and Kansas 6%.

In each of the four years since 1913 production from the Joplin district responded to advancing prices of zinc; and although in 1917 zinc concentrates averaged only $67.70 per ton with zinc at 8.7c. per lb., as compared to $84.72 in 1916 with zinc at 12.5c. and $79.30 per ton in 1915, when zinc was 13c. per lb., Joplin increased its production nearly 26% over that of 1916. The annual increase over the preceding years in 1916 was 23% and in 1915, 12%. Considering the production in 1914 as normal compared with previous years since 1900, when prices of zinc concentrates ranged between $40 and $45 per ton and spelter between 5 and 6c. per lb., the production of 1917 was 60% greater than in normal pre-war years. Lead ores reached the highest average basic price in 1917, which, roughly, was $98 per ton, as compared with $84 in 1916, $55 in 1915, $46.50 in 1914 and $52.50 in 1913. Although an important byproduct of the Joplin district, lead is produced in comparatively small quantities, and the fluctuations of the price of zinc practically govern operating conditions, irrespective of the price of lead.

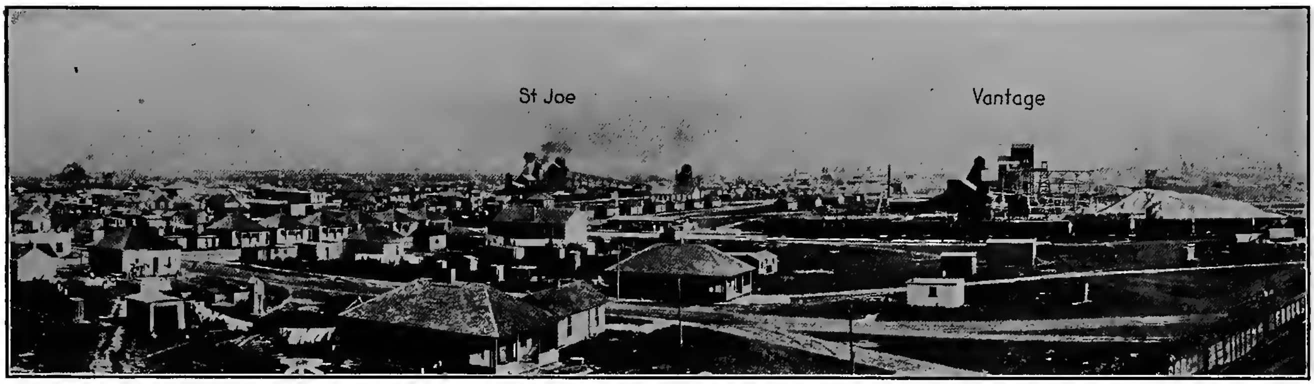
The response to the demands of the war, reflected in the increased prices of zinc and lead, was made possible by the exploration of lower-grade ores in the Missouri sheet-ground mines and the rapid expansion of developments in the Oklahoma and Kansas high-grade fields. The last six months of 1917 witnessed a strong reaction in metal and concentrate prices, and many mills operating on low-grade sheet-ground ores were forced to cease operations. A large percentage were dismantled and taken to the newer fields. As a consequence, this condition decreased the 1917 production of blende from Missouri about 9%. The production of lead and calamine was slightly increased. In Oklahoma and Kansas

Fig. 8. Panoramic View of Picher, Okla., Showing the Most Advanced

many developments started in 1915 and 1916 reached their productive stages early in 1917, and notwithstanding the drop in metal prices later in the year, this section of the district continued its intensive production and was only prevented from making greater gains by shortages of labor and power. The increase in production of blende from Oklahoma and Kansas in 1917 was 175% and of lead 94% of the production of 1916.

The orebodies of Joplin, as mentioned, occur in all Carboniferous sedimentaries of the district, but mainly in the cherty crystalline Boone limestone of Mississippian age. The Boone outcrops throughout all but the extreme western limits of the district, has a total thickness of 350 ft. and a slight regional dip somewhat north of west, disappearing in the Oklahoma and Kansas sections and at the northwest corner of Jasper County, Mo., under the overlying and successively outcropping Chester and Cherokee limestones, sandstones and shales, respectively of Mississippian and Pennsylvanian age. Two distinct classes of orebodies occur in the Missouri section, and the present mines, are much lower in grade, yielding in 1916 mill recoveries, in the orebodies then worked, ranging from 1½ to 3% combined sulphides. At present 2% ores are barely yielding a margin of profit.

In many places the sheet-ground orebodies are continuously connected to the sink-hole deposits above by orebodies occurring in the upper Boone. Such orebodies also are replacements and deposits in and upon the limestone walls of former openings or channels. The mineralization has in some places been followed in mining for varying distances below the sheet-ground deposits in what is known as the "second lime." Camps where the mining of "upper run" orebodies still exists are Duenweg, Joplin and Granby. The largest sheet-ground mines are at Webb City, Carterville, Joplin, Duenweg, Porto Rico and Granby. The "lower run" orebodies have been mined at Orongo, Sarcoxie, Aurora and Cave Springs. At Granby and Springfield ore has been mined from the "gumbo runs" or deposits in the basic shales of the Boone formation.



Operations and the Intensity of Development in a Period of Two Years.

locally these are known as the "upper run" and the "sheet-ground" deposits. The "upper run" deposits were the earliest found, occurring near the surface in the higher horizons of the Mississippian limestones, mainly in the bouldery ground along the roughly semi-spherical contact made by the deposits of the later shales in ancient sink holes in the Chester and Boone. The sheet-ground deposits occur in a cherty stratum of the Boone known as the Grand Falls chert, that lies in nearly horizontal position about 250 ft. below the surface and from 50 to 100 ft. from the lower shale base of the Boone formation.

The Grand Falls chert has a thickness varying from eight to 50 ft. The sulphides occur as replacements of the lime and deposits in openings, disseminated in a cherty breccia and laminated in roughly horizontal seams intercalated between alternate narrow bands, or sheets, as the name implies, of chert, flint and jasperoid. The Grand Falls chert is seldom mineralized for more than 20 or 25 ft. vertically, the orebodies being inclosed above and below by barren zones of flint of varying thickness; but these deposits extend over wide areas in many parts of the western portion of the Missouri section, and most typically and persistently in the zone, shown on the accompanying map, between Oronogo and Duenweg. The sheet-ground deposits, although far more extensive and productive than the "upper run" deposits, especially in

In the Miami district definite geologic relationships are not so apparent and have not as yet presented sufficient evidence for complete and satisfactory conclusions. Two distinct ore horizons or "runs," however, are recognized. The "lower run," which to date is the most important both in grade and size of the orebodies, lies at depths ranging from 250 to 300 ft., and in shape, thickness and general dip and strike presents characteristics similar to the sheet-ground deposits of Missouri, without, however, having the sheets of flint characteristic of the Grand Falls chert. In the Miami orebodies the mineralization is considerably higher in grade than in the sheet-ground mines and generally presents more of a disseminated structure. There is more limestone in the gangue of the orebody, and considerable yellow calcite or "tiff"; and although much of the ore occurs in tabular narrow seams, and considerable flint and chert are present, the formation in the mineralized areas so far mined appears to be more extensively of the brecciated and recemented type, and on the whole presents decided marks of distinction from the structural characteristics of the sheet-ground orebodies. The richest ores have been concentrated along bedding planes and parallel watercourses where a sheeted condition exists, and may represent horizons which, according to Siebenthal, are at the terminus of an artesian circulation controlled by the imprevious shales which begin to outcrop in this district and mark the ever-increasing radius of erosion from the central dome.

At Commerce a well-defined fault was recognized underground, and fissuring occurs in other parts of the Miami district. Apart from this, however, no well-defined evidence of faulting has yet been found and given general recognition. In fact, the so-called Miami fault shown on the map is arbitrary, and merely represents the general trend of present development. The "upper run" deposits of the Miami district are the shallower orebodies, and these occur from 100 to 150 ft. below the surface, mostly in the Chester limestone, and generally are somewhat lower in grade than the ores mined from the "lower run," and contain a considerable amount of bituminous matter that interferes more or less with concentration.

The proportion of lead and zinc in the Joplin sulphide ores varies considerably in different sections of the district, and in each section there is a considerable variation due to relative depths from the surface. If the artesian theory is accepted, the original deposits in the bedding planes and seepage channels would have a primary character varying both in grade and relative lead and zinc content to such deposits as have been either leached or enriched by the later action of surface waters. The primary ores exposed by erosion have been impoverished by the oxidation and solution of the zinc sulphides and relatively enriched by the less soluble residual galena. Part of the zinc taken into solution replaces limestone or combines with silica in solution at or near the surface to form carbonates and silicates in favorable channels, and part migrates in solution to greater depths below the surface, reprecipitating secondary sulphides below the zone of oxidation and thereby producing an enrichment in zinc relative to the lead contained in the primary sulphides. To a much less extent in the Joplin district, owing to the lower solubility of lead, galena has undergone a similar process.

The first orebodies mined in the Joplin district were for the lead content, and for a number of years carbonates of lead only were recognized. As greater depth was attained galena became the predominant ore, but when still deeper zones were reached the relative quantities of blende increased, until at the depths exploited in recent times sphalerite establishes the district as a producer of zinc. Pyrite and marcasite occur as associated minerals in nearly all Joplin ores, but although the percentage of iron sulphides is comparatively small, their relative proportions in ores vary considerably in different localities. The common gangue minerals are dolomite and calcite, deposited contemporaneously from acid solutions with the sulphides, and jasperoid, a metasomatic replacement of silica and limestone which occurs abundantly as a deposit from the same solutions. Excepting in the camps mentioned, most of the mines at present in the Missouri section are producing from the sheet-ground deposits, which are the lowest-grade ores of the district, and but for the large tonnage available could not be made to yield a profit. In the Miami and Kansas sections the ores are uniformly higher in grade and contain a relatively greater percentage of galena than the sheet-ground deposits of Missouri.

Some of the mines in the Picher section are producing ores steadily from which a 15% combined sulphide mill recovery is made and a large number of properties are milling 10% ores, while others are mining varying grades, probably averaging over 5%. The Laclede mine, in the Miami district, recently broke into an open cave the walls of which were solidly lined with cubes of galena, individual crystals ranging up to four inches in size. Such deposits are probably of scientific interest rather than of commercial importance, as the amount of such ore is limited and furthermore presents certain difficulties in handling.

The Mining Districts of Joplin and Southeast Missouri Part III

BY H. W. KITSON

The silicate ores of the Joplin district occur in the weathered zone above ground-water in limestone. They are irregular and pockety orebodies and for the most part mined by hand-steel methods and the ores sorted underground. Mines in the " upper run " sulphide orebodies are practically exhausted, but in the early days of the Joplin district constituted the main source of production. These deposits were for the most part in what are known as " soft-ground " mines, necessitating, as the name suggests, the use of timber, as distinguished from those that were minable by the support only of pillars. The formed the walls and roof. When to this condition was added the considerable amount of moisture and mud, stoping was difficult, and such mines required heavy timbering, close-lagged and often spiled, and shafts frequently had to be cribbed solidly for their entire depth. Where the ground permitted, square sets were used in stoping. These mines were sometimes deep enough for more than one level. Where the ground would stand, the ore was followed from the lowest level to the upper limits with inclined raises. When completed, the raise bottoms were undercut in benches progressively for the length of

Fig. 9. Underground Haulage in the Joplin District with Storage-battery Locomotive

deposits were high grade, but pockety and irregular, and the ground when soft was also wet. Many of the orebodies deposited along the contacts of shale in limestone sink holes had an elongated form, sometimes of considerable length and width; and they often occurred with a circular strike following the peripheral contact and forming the so-called " circle " deposits.

Large pockets containing nearly pure galena and blende were frequently found associated with a dark secondary chert filling cavities and fractures. A form of soft white decomposed chert locally known as " cotton rock," and decomposed shale or " soapstone," often the raise, leaving high walls behind. The ore as broken fell to the bottom, where it was shoveled into " cans " or buckets set on trucks, trammed to the shaft and hoisted in the buckets. Stopes of this character were often over 60 ft. high and sometimes 40 ft. wide.

The ores mined at present occur mainly in the sheet-ground deposits of the Grand Falls chert and the interbedded deposits of the western part of the field. The methods used in mining are similar in both cases, there being from a mining point of view but little difference in shape, depth or self-supporting character of the ground. As the details of *modus operandi* of churn drilling operations and mining methods are already well described in a recent paper [4] by H. I. Young, general discussion and further observations only will be undertaken.

The deeper orebodies are prospected altogether by churn drill. The cuttings, when "shines" (sulphides) have been reached, are sampled and assayed. By operating two or more rigs working radially from an initial central discovery hole, drilling on the corners or squares staked out on the surface 20 ft. or more on a side, the size, grade and depth of orebodies can be closely estimated before putting down a shaft. Previous to the war

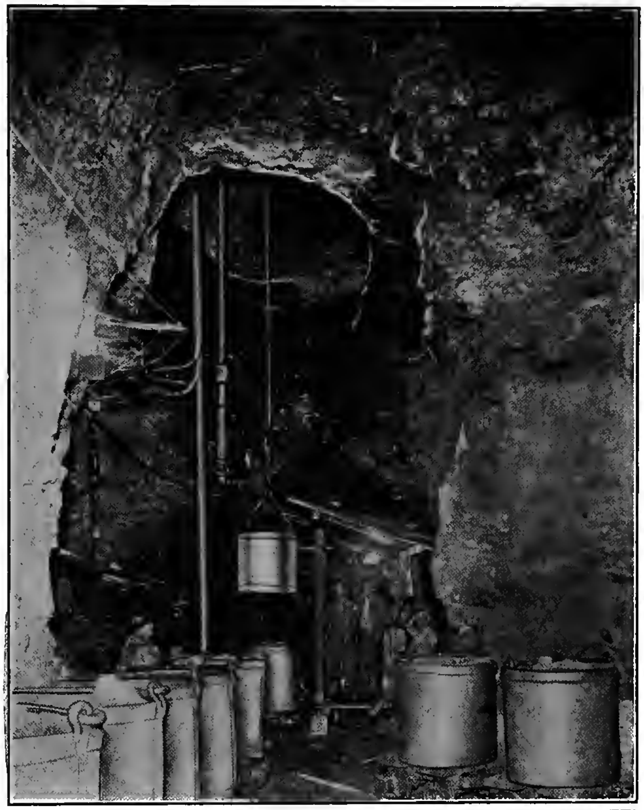


Fig. 10. Hoisting from a Joplin Shaft Station

contractors undertook to prospect a tract for from 75c. to $1 per ft., supplying their own labor, rigs, tools, casing and power. In the fall of 1917 the cost of churn drilling had risen to $1.25 and $1.50 per ft. and over, and concessions in supplies also were required. When a certain number of holes have demonstrated the presence of an orebody, a vertical shaft is started and sunk to ore simultaneously with further drill operations, as described in the reference already given. Shaft sinking costs vary from $25 to $30 per foot.

It has been found in practice that on account of local variations due to the presence of lean or barren portions in the orebodies, the samples from some holes cannot be relied upon. Some holes may pass through good ground without giving evidence of ore in cuttings by striking a lean or barren portion adjacent to high-grade ore. Fissures and vugs or other cavities in the orebody will also often concentrate the heavy sulphides to such an extent that but a small proportion enters the sample bailer, and the results of single holes are therefore often deceptive and consequently not dependable, and drilling data is valuable only after a great number of holes have been put down at closely spaced intervals. This method of ascertaining the size and trend of an orebody gives data, also, invaluable as an aid to later mining. Shaft sites may be selected near the center of gravity of the orebody and the lowest point in the trough of the deposit, thus equalizing future tramming distances and obtaining proper drainage grades. Drill holes also assist materially in ventilation where there is but one shaft in a mine and the working face recedes from it to considerable distances.

With but two or three exceptions all the shafts of the district are vertical. Some of the larger companies have two-compartment shafts, but the majority are single compartment. These shafts are not timbered except where cribbed for short distances below the collar in passing through the soft alluvium soil or soft shale caprock in the Miami section. Buckets are used in tramming and hoisting almost altogether, and these are hoisted with ore, supplies or men, without guides and

Fig. 11. Cutting Around a Pillar from Two Stopes

crosshead. Where shafts have two compartments, a simple plank lining fastened to cross stulls divides each. Over the shaft or each compartment there is a separate hoisting engine, either steam or electric, set in the derrick close enough to the shaft to enable the "hoisterman" to raise and lower without signals by simply looking down and observing the operators.

Shafts in the sheet-ground mines vary in depth from 230 to 260 ft. and in the western part of the district

[4] Development and Underground Mining Practice in the Joplin District, Bull. 129, A. I. M. E.; and Eng. and Min. Jour., Oct. 6, 1917.

from 150 to 250 ft., with but one or two at materially greater depths. The hoisting engine is usually from 50 to 60 ft. above the collar and the hoisting rope passes over a small sheave about 10 ft. higher. The "tubs" or buckets are run to the shaft on trucks by hand or in trains as shown in Fig. 9, according to the size of the mine. The shafts are usually centrally located with respect to the size of the orebody and are sunk for a sump a short distance below the floor of the ore. Stopes are started in two diametrically opposite directions from the shaft for the full height of the orebody, leaving substantial pillars of ore on two sides. A plank platform covering the sump is built, to the level of a bucket truck, and each truck in turn is brought to the edge of this platform prior to affixing the snap or worm hook at the end of the hoisting rope.

A typical shaft station is shown in Fig 10. The stations are electric lighted and the hoistman can easily observe the hooking and unhooking operations from his position. Empty buckets are lowered on the brakes to the platform, the rope is given the necessary slack and the hook transferred by the "tubhooker"— one at each compartment when there are two — from the loop in the bale of the empty to that of the loaded bucket. At an arm signal from the tubhooker the bucket is slowly raised high enough to clear the empty on the platform, where it is momentarily steadied by him, and then jerked to the surface and level of the hoistman at apparently incredible speed. The tubhooker then rolls the empty on its lower rim to the cleared truck, which is run back to a return track and the next loaded truck is pushed up to the platform. When the loaded bucket arrives at the top, a counterbalanced door operated by the hoistman is dropped across the shaft directly under the suspended bucket, inclining toward the bins, and a swinging arm hook, offset toward the bins, is brought to engage the ring fastened to the bottom of the bucket. The hoisting rope is then given the proper slack and the bucket dumps its contents backward toward the shaft and onto the incline. The capacity of buckets used in the district is from 800 to 1500 lb., but outside of sinking operations those most in use are 1000 or 1200 pounds.

The No. 3 shaft of the American Davey mine of the American Zinc, Lead and Smelting Co., in Webb City, may be taken as typical of the most efficient hoisting practice in the district. The shaft has two compartments and is 252 ft. deep. Two vertical duplex steam hoists directly driving 24-in. drums wound with ½-in. cables and hoisting 1200-lb. buckets are in use. The average time of one complete cycle of lowering and hoisting is 35 sec. and the average rate of hoisting from both compartments in one eight-hour shift is 1300 buckets, and records have been as high as 1500. Some electric hoisting records are over 2000 buckets per shift.

In his paper Mr. Young gave the cost of hoisting per ton for 1912, 1913, 1915 and 1916, respectively, as 4.68c, 4.33c., 3.99c., and 4.83c., and for four months of 1917 at 6.4c., including power and repairs, which compares favorably with hoisting costs by cages or skips in mines elsewhere hoisting similar tonnages. In the matter of safety, a recent disastrous accident occurred in the Miami district from lowering by means of the brakes a bucket with four men, a risky practice indeed, unless the brakes are frequently inspected. As a matter of fact, however, accidents in Joplin shafts have been remarkably few. Although no bell signal is used in ordinary hoisting operations, there is a bell signal wire in most shafts. In case of accident to the bucket while hoisting ore, the engineer can pull a "skidoo" bell, at the sound of which everyone at the bottom of the shaft gets away as fast as possible.

From the shaft the orebody is stoped radially in all directions, leaving pillars, as shown in Fig. 11, from 15 to 20 ft. in diameter and 20 to 100 ft. apart, depending upon the character of the ground. In the sheet-ground mines the flint floor and roof are usually quite smooth and level, making an ideal back and floor to break to and obviating the necessity of shoveling plats. Six feet is the minimum height stoped, and the back or height of ore varies up to 25 ft., probably averaging 20 ft. in most sheet-ground mines, but often 30 to 40 ft. or more in the Miami fields. Ore in the sheet-ground deposits is fairly constant throughout the minable areas, and there is little choice of ground when cutting or robbing

FIG. 12. DRILLING AND SHAFT SINKING NEAR PICHER, OKLA.

pillars. In the higher-grade orebodies of the Miami district there is this factor of consideration, and although pillars are generally cut entirely with respect to the standing character of the roof, when they are robbed in the final process after all stoping has been carried to the property or natural limits of the orebody, the richer pillars are taken out entirely or in greater proportion than those in the leaner ground. There is, however, in some parts of the sheet-ground mines a "cotton rock" condition of the roof that cannot be supported, which is entirely barren and has to be shot down with the ore, causing at times a certain dilution.

There is no underground sorting in any of the mines, but a small percentage of waste is sorted at the grizzlies over the bins on top. On account of the friability and local variations in grade, underground sampling is impracticable, and estimation of whether certain ground is ore or waste, based on experienced observation, gives satisfactory results. Within the sheet-ground orebodies there are areas where the ore is disseminated rather than laminated, and this ground, being of a brecciated character, does not support the roof so well as the latter, consequently requiring greater pillar diameter and less open stope between. In the sheet-ground mines the flint roof is usually about 40 ft. thick and may be frozen to the limestone above, or separated from it by a thin layer of clay selvage. The first condition exists north of Webb City, where pillars are spaced as far as 100 ft. apart, but in most parts of the district the second condition prevails and 20- to 25-ft. intervals are most common. Sometimes a simple stull and headboard will assist in steadying the back where it is not in a too bad condition and not too high.

There are two methods of breaking ground in practice. In the sheet-ground mines, where the back is frequently under 15 or 16 ft. high, an 8-ft. high heading is driven into the face for a short distance along the floor of the stope and the brow left overhead is drilled from beneath. The practice where the back is higher than 16 ft. is to advance the entire height of face progressively from top to bottom in a series of two or three benches or steps. The upper benches are not advanced too far with respect to those below, in order that the ground blasted will clear the edge of the bench below and obviate unnecessary shoveling. In both methods the breast of each bench or heading is carried forward with a serrated face presenting one open side to which the next rounds of holes can break.

Headings are advanced at the width established for the distance between pillars cutting around them until two adjacent headings meet behind the ground so left. The next succeeding row of pillars is then staggered or offset with respect to the first; so that more uniform support is obtained and the minimum amount of space is left standing unsupported. Sufficient provision is made in offsetting to permit the main track lines to be extended to the face without undue deviation from a straight line. In this method of mining from 15 to 20% of the orebody is left in pillars during the stoping stage, but these are robbed in the final process and from 25 to 40% of their tonnage is reclaimed.

Regard to surface conditions must be carefully maintained. Every precaution is taken to insure the surface from a cave. This is necessary on account of the retained surface rights agreed in the terms of the lease. Now and then, however, a cave has occurred, the effect of which is shown in an illustration on pg. 7 of Part II. Many of the sheet-ground mines extend under the residential sections of large communities and paved streets and carlines pass overhead regardless, while dull thuds can be heard and felt underfoot in the streets at shooting time. In the vicinity of Carterville and Webb City the sheet-ground mines have been stoped over single areas underlying 160 acres, and developments indicate that eventually these will have an extent of 350 acres or more.

FIG. 13. SHOVELING CREW CLEANING UP A STOPE IN THE JOPLIN DISTRICT

Throughout the district, in drilling underground two men are customarily used to a machine and Ingersoll-Rand mountable water-type machines have been adopted in most of the mines. Drill helpers act as tool nippers, assist in setting up the machine on the drill column or tripod, either of which is used, depending on the height of the back, and fetch sharp and dull steel between the shaft station or underground drill sharpener and the machine drill. But little dust is created in drilling, owing to the adoption of the water-type hammer drills, and at some of the mines in the sheet-ground district as high as 60 tons per machine shift is broken, as compared to 40 tons when the piston machines were in use. Large sizes of steel are used and deep holes drilled. Jackhamers are used to break up the larger boulders. Holes are chambered or "squibbed" before final blasting and considerable powder is required per hole. This is especially the case in the Miami section, where the ground is harder to break, although possibly somewhat easier to drill.

In the Miami ground machinemen can drill as much as 90 ft. of hole per machine shift. Air pressure is maintained at 90- and 100-lb. gage. Drilling costs in the sheet-ground mines before the war were under 20c. per ton and blasting costs about 15.5c. per ton. At present these costs range from 25c. to 30c. per ton for drilling and 15c. to 30c. per ton for blasting, according to the ground and whether the powder is obtained under favorable contracts or purchased in the open market. According to the prevailing custom in the district, the drilling is done on day shift and mucking on both day and night shifts. Special crews chamber and load the holes on the second shift, firing as the shift goes off.

Ventilation in mines connected to more than one shaft is generally good, and the smoke has plenty of time to clear between the night and next day shifts. Air whistles are placed underground at the shaft station in the large sheet-ground mines that can be heard by miners in the remote headings at lunch and "quitting time."

The Joplin district is notable for its shovelers and the large tonnage of mine "dirt," as the ore is called, handled per man. No. 2 scoop shovels are used altogether, and shovelers work in pairs, usually under contract, as shown in Fig. 13. The contract prices are based on a price per "can," according to the bucket tally at the shaft station or motor "lay by" where they leave the loaded trucks, and vary in different parts of the district according to the local wage scale, the height or capacity of the buckets used and the length of tram to the shovelers' "lay by." When labor in the district was abundant and conditions were normal, a shoveler would handle 20 tons per day or more and earned as much as $6 or $7 per shift at a cost of from 20 to 25c. per ton. At present, however, good shovelers are scarce, the efficiency is much lowered, and underground power shovels are again being given consideration at some of the larger properties. Motor, mule, and, where grades are steep, rope-haulage systems are used, depending upon the distances advanced from the hoisting shaft. Shovelers seldom tram their buckets over 200 ft., and where possible these distances are considerably shortened.

The drainage problem in the district, although a factor of considerable moment, is not one offering great difficulties. The gentle grades of levels usually enable the water to be conducted by means of a system of ditches to some central sump at one of the shafts, where steam or electric pumps can handle the flow without much difficulty or undue cost. In the newer mines in the Miami fields, considerable water has to be handled during the early stages of sinking and opening up the ground, but after a time the flow diminishes and the normal quantity necessary to pump at any one station is seldom over 1000 or 1100 gal. per minute. In the sheet-ground mines where adjoining properties are connected underground over considerable distances some workings are forced to drain others, and in such cases central pumping companies have been formed and the cost is distributed on an equitable basis.

In the Picher camp a single drill hole supplies water for the domestic requirements of the population. This hole is 1100 ft. deep and passes through the Netta mine, belonging to the Eagle-Picher Lead Co., a pillar having been left as protection. The mine water in the district is markedly acid and although used in milling throughout the district it is useless for other purposes and has a strong corrosive action on pumps and pipe lines. Centrifugal and triplex pumps are used most commonly.

The wages of the Joplin district are based on a sliding scale that varies with the concentrates market. When concentrates are sold at a basic price from $40 to $45 per ton, machinemen get $2.50 and helpers $2 per day. The wages increase 25% for each increase of $10 per ton in the price of concentrates, and with the market at $75 per ton machinemen receive $3.50 and $3 per day. In the newer fields, where living conditions are less attractive than in Joplin, the wages are higher, and where machinemen, trammers and hoistmen are working on a bonus or contract basis, as they are in some mines, the earnings rise as high as $10 per day in individual cases. As in other parts of the country, cost of mine supplies has increased considerably and the rate of increase for Joplin is shown by the relative pre-war and present prices given in the accompanying table.

TABLE II. PRICE OF MINE SUPPLIES IN THE JOPLIN DISTRICT

	1914	1917	Per Cent. Increase
Coal, per ton	$2.10	$3.60	71
Dynamite, 40% pulp, per 100 lb.	11.00	18.25	66
40% gelatin, per 100 lb.	11.50	20.50	78
80% gelatin, per 100 lb.	15.50	31.75	105
Fuse per 6,000 ft.	24.75	37.86	53
Caps No. 6 per 1,000	7.57	14.58	92
T-rail, No. 8, per ton	35.00	80.00	129
Drill steel, solid, per lb.	07¼	.16	120
Drill steel, hollow, per lb.	.10	.23	130
Rubber wire-wound air hose, 1 in., per ft.	.25	.35	40

Total underground mining costs per ton, as given by Mr. Young for a typical sheet-ground mine hoisting 1000 tons per day, are: In 1912, 68c.; in 1913, 65c.; in 1915, 88c.; and in 1916, 99c. Miami costs are higher, as may be expected in a comparatively new camp.

The Mining Districts of Joplin and Southeast Missouri Part IV

BY H. W. KITSON

The magnitude of mining operations in the Southeast Missouri district is best conceived by a comparison of its mineral production with that of Joplin and other districts in the United States and foreign countries. In 1917, Southeast Missouri produced 204,869 tons of lead, or 35% of the total domestic output of the United States, and; including foreign imports, the relative proportion was 32 per cent.

In point of base metallic tonnage, the production from Southeast Missouri compares favorably with that from the Joplin district, which in 1917 yielded 40,575 tons of lead, or 7% of the domestic output, and 290,945 tons of spelter — a total metallic output of 331,520 tons. Both districts produce a practically pure base metal, the silver content in the ores being commercially negligible, and average prices in 1917 for lead and zinc differed only by 8.721c. and 8.813c. per lb., respectively.

In 1913, the world production of lead amounted to 1,120,000 metric tons.[5] Of this total the United States contributed 395,000 tons, of which 133,000 tons was produced from Southeast Missouri. In the same year, Spain produced 203,000 tons; Germany, 181,000 tons, and Australia, 116,000 tons. In 1917 the rate of production of lead from Southeast Missouri had increased by 54% over the 1913 output, thereby excelling the normal output of Germany before the war.

The productive area in Southeast Missouri, as shown in Fig. 14, is mainly in the vicinity of Flat River, St. Francois County, and is locally known as the " Lead Belt." It is situated 60 miles south of St. Louis and 225 miles in a northeasterly direction from Joplin. This section accounts for about 90% of the total production from the district, the remainder coming mostly from the vicinity of Fredericktown,[6] Madison County, 25 miles further to the southeast, with relatively smaller amounts from Washington and Ste. Genevieve counties.

The Southeast Missouri district lies on the eastern side of the Ozark uplift, about 40 miles west of the Missouri River, toward which the drainage system of the area is directed. Elevations in the Flat River section vary from 700 to 800 ft. above sea level. At Big River the valley has been eroded to an elevation of only 610 ft., but in the southern part of the district the St. Francois Mountains attain elevations of 1800 ft. Over most of the productive area the surface is gently rolling, affording but few good gravity millsites. The rainfall amounts to about 40 in. a year, and water from underground sources is plentiful for milling purposes. Normally, the winters are mild, but the summers are invariably hot and humid.

The stratigraphic features of the district[7] are simple and consist of Cambrian sedimentaries deposited near shore in an ancient shallow sea unconformably upon an eroded pre-Cambrian granite, of which much of the material constitutes the later formations. The granite basement and its overburden of sedimentaries slope downward radically from the St. Francois Mountains, at which point they have been relatively elevated over a comparatively small area by faulting, and stand exposed at the surface by erosion.

The La Motte sandstone directly overlies the granite, with a thickness ranging from 200 to 300 ft. This formation is conglomeratic at its base and dolomitic near its upper contact, and decreases gradually in thickness as the central granite core or ancient shore line is approached.

Overlying the La Motte with conformity, the next formation above is the Bonne Terre, which is economically the most important in the district. Buckley[8] gives the following description:

> The Bonne Terre formation consists chiefly of dolomite with thin laminæ or beds of shale and beds of chlorite, occasionally arenaceous, dolomite. The upper and lower parts of this formation are quite uniformly interstratified with shale, while the middle portion contains only occasional thin leaves of shale between the beds. There is generally an absence of stratification planes, but the bedding planes are well defined and reasonably persistent. The position of most of the bedding planes has been determined by thin films of shale and of abrupt changes in the texture of the dolomite.
>
> The bedding planes are frequently smooth and level, but more often they are rough and wavy. A pivot or pinnacle-like surface is not uncommon. There has evidently been more or less solution and deposition along the bedding, as a result of which some of the adjacent beds are attached and others are free. The coalescing of two bedding planes through the feathering out of an intervening bed occurs frequently in the lower part of the formation. Occasional examples of cross-bedding have been observed in the mines.
>
> Overlying the Bonne Terre conformably in order of deposition are the Davis shale, with a maximum thickness of 160 ft., the Derby and Doe Run dolomites, with a thickness of 100 ft., and the Potosi cherty dolomites, having a thickness of 300 ft.
>
> Within the productive area, faulting of the Block Mountain type has relatively elevated the lower formations, and erosion has either entirely denuded the Bonne Terre near its former upper contact or has left it with but a relatively thin cover of Davis shale. All formations have a slight dip to the southwest, modified locally by faulting and gentle anticlinal and syncline folding. This folding varies often at different horizons of the Bonne Terre within the same areas, and appears to be the result of deposition upon the uneven granite floor and to solution, rather than lateral dynamic stress.

The Bonne Terre dolomite is characterized by extensive jointing, and several definite systems have been recognized, of which some are marked by their persistence and broad openings. Fracture and solution channels have been formed which at some points extend continuously from the surface to the sandstone below, and

5 " The Mineral Industry," Vol. XXIII.
6 " Eng. and Min. Journ., Vol. 105, No. 2, p. 65.

7 " Geology and Mineral Deposits of the Ozark Region." By H. A. Buehler, Bull. 130, A. I. M. E.
8 " Missouri Bureau of Geology and Mines," Vol. IX, Part 1. By E. R. Buckley.

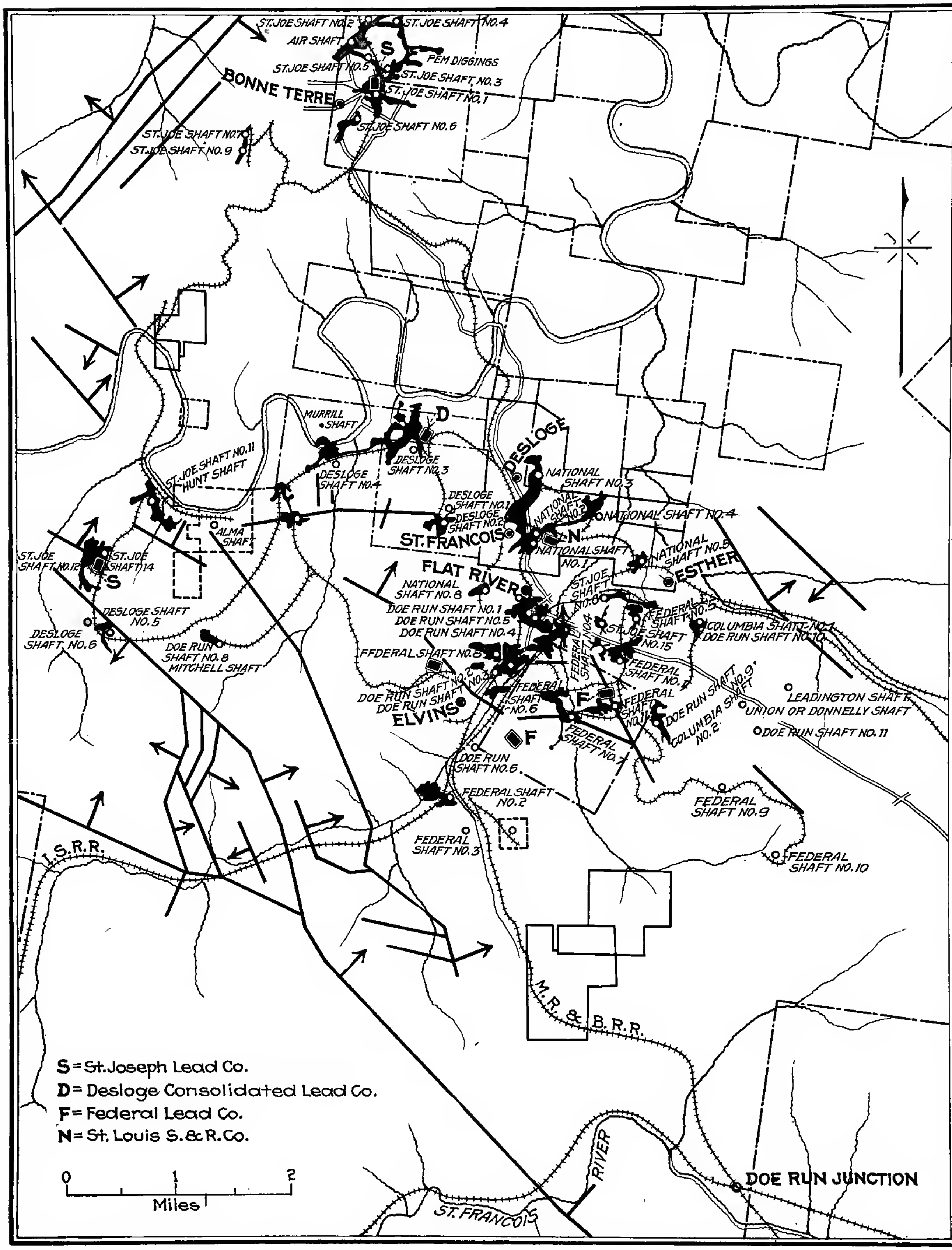


FIG. 14. COMPOSITE MAP OF THE SOUTHEAST MISSOURI DISTRICT SHOWING GEOGRAPHIC DISTRIBUTION OF MINING CAMPS, AREAS MINED, TRANSPORTATION SYSTEMS AND LINES OF MAJOR FAULTING

such undoubtedly had an important influence on deep ore deposition. Joints and fracture planes are found both open and filled with clay, and underground circulating waters have developed caves and sink holes. The surface clays are the alteration product of the eroded Davis shale, and in places these have been productive of galena in the early days of mining. Jointing appears to occur in zones, and evidence has been found in the mines which indicates a series originating at the base of the formation.

Faulting is more in evidence in the Flat River district than at any other mining area of the Ozark region. The faults are of the normal or gravity type, and contiguous blocks, as shown in Fig 15, have been displaced so as to form a step-like series bounded by the fault plane zones. The major system of faulting antedates the genesis of the ore deposits, which orebodies, however, have been subsequently faulted by a minor series.

The main fault of the immediate district has a displacement of 120 ft. and strikes northeast and southwest, passing through Big River north of Bonne Terre, as indicated in Fig. 14; and a subsequent series to the south has a strike northwest and southeast. Minor faults occur with north-south and east-west strikes. As shown in Fig. 15, this Big River fault has lowered the Davis and overlying formations on the downthrow side to the level of the outcropping Bonne Terre dolomite.

Faulting has had an important effect in guiding underground circulation, and in the Flat River district has created a basin-like condition by which seepage from the surface has formed a reservoir of mineral-bearing solution over the previous La Motte sandstone, favorable to deposition in the dolomite channels by slow infiltration.

Geologists favor the theory of ore deposition from solutions of surface origin, as explaining the genesis of all ores in the Ozark region, with the exception of the hematite deposits and argentiferous lead veins in granite. The Davis shale forms an impervious barrier to ascending or descending solutions, and the disseminated-lead deposits have been found only where the Bonne Terre formation outcrops at or lies close to the surface, the shale having been completely or largely eroded over such areas. The marked absence of metamorphism of the sedimentaries, the presence of numerous underground water courses and subterranean caves, and deposits of galena in surface clays have generally been accepted as irrefutable proof of this origin of the ore.

In Southeast Missouri the lead has been deposited by matasomatic replacement of the dolomite. There appears to have been little or no secondary enrichment or formation of oxidized ores. Deposition appears to have favored the darker carbonaceous beds, and little ore is found in the lighter-colored part of the formation. In general, the ore is found in the lower horizons of the Bonne Terre disseminated through the dolomite and shaly portions, and there is little if any marked brecciation, as commonly exists in the Joplin district. Where oxidation has occurred, it has been slight and the rocks have in general retained their dark color. Leaching, when there is any, appears to have been complete, and perfect casts of the original galena are sometimes to be seen. In general, the lateral limits of the orebodies are not sharply defined, but fade gradually into the wall rock, decreasing in grade from the richest portions at the center, except where bounded by joint or fault planes. In many cases, there is a vertical displacement in ore horizons on each side of a joint plane where no movement has occurred. Such a condition is undoubtedly due to preferential replacement in certain favorable portions of the dolomite beds.

The lead deposits of Southeast Missouri are confined mainly to the Bonne Terre dolomite and to independent areas that have been productive in the upper portion of the La Motte sandstone near the contact of the two formations. The lead ores originally mined were found in masses and pockets in the residual surface clays overlying the Bonne Terre and close to the eroded Davis shale. These deposits, though rich, were not extensive, and, although highly productive at one time, are relatively unimportant at present. Such deposits often extended downward into the dolomite, following fissures and joint planes and extending along the bedding planes, but seldom attaining a depth of greater than 50 or 60 ft. At Bonne Terre, however, in the northern part of the district, the lead deposits have been followed from the surface to the La Motte sandstone.

The most important deposits of the district are the low-grade disseminated lenses or sheets in which galena replaced dolomite along certain horizons parallel to stratification planes in the lower beds of the Bonne Terre. The main zone of ore deposition appears to occur at short distances above the sandstone contact, as shown in Fig. 16. Some of these orebodies have widths of 800 ft. and frequently extend irregularly for a half mile or more. The orebodies vary in thickness from a few feet up to over a hundred feet, but the average range is probably from eight to twenty feet. Laterally, these orebodies spread out irregularly, and their positions, owing to local variations in grade, as shown by the outline of areas stoped in Fig 16, indicate the favorable channels of original ore deposition. The trend of the long axis of the Bonne Terre orebody lies with a northeast strike somewhat parallel to the Big River fault. In the southern part of the district the orebodies appear to lie parallel to each other, with a southeast trend.

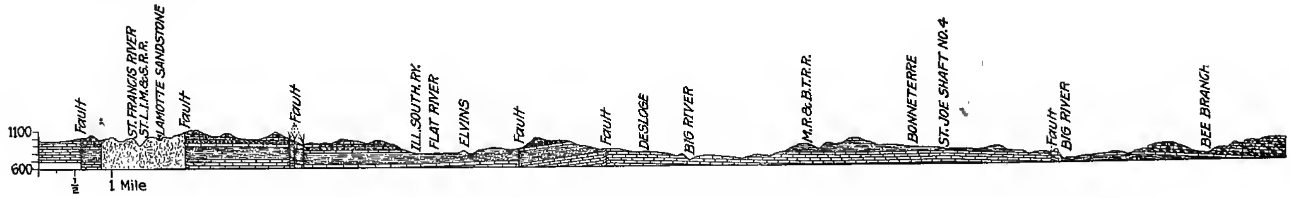


Fig. 15. Geological North-South Section Through Flat River, Southeast Missouri District

parallel to the faulting zones in that area. Minor series of orebodies often overlie or underlie the first series at distances varying from 10 to 80 ft., as shown in Fig. 17, and constitute in such mines second and third levels. A small proportion of the lead ore mined — probably not 1% — comes from deposits in the La Motte sandstone and carbonaceous shaly dolomite along the contact.

In 1917 the Flat River district mined and milled about 20,000 tons of ore per day. The principal companies, in the order of their productive capacities, are the St. Joseph Lead Co., with a 2400-ton mill at Bonne Terre, a 2000-ton mill at Leadwood, and a 4200-ton mill at Rivermines; the Federal Lead Co., which has a 5000-ton concentrator at Flat River and a 3000-ton plant at Elvins; the St. Louis Smelting and Refining Co.; the National Lead Co. with a 2500-ton mill at St. Francois, and the Desloge Consolidated Lead Co., which has a 1500-ton mill at Desloge. The Baker Lead Co. has a 500-ton mill at Leadwood, and the Boston-Elvins Lead Co. mined about 10,000 tons in 1917, which was concentrated at the mill of the St. Louis Smelting and Refining Co.

The mines at Fredericktown are among the oldest in the country, and the famous old Mine La Motte has been a producer for many years. The limestone cap overlying the La Motte orebody is being stripped by steam shovels preparatory to mining the low-grade disseminated-lead deposit by opencut methods. This work is being done by the Missouri Metals Corporation, which is erecting a 1500-ton mill to concentrate this ore. The Federal Lead Co. operates the Catherine mine under option, and a 600-ton mill treats this ore at La Motte. This property was producing in 1917 from disseminated-lead deposits similar to those at Flat River, but late in the year operations were discontinued pending more favorable economic conditions. The Missouri Cobalt Co. has a 300-ton mill at Fredericktown; and the Einstein mine, 12 miles west, has a 25-ton mill treating tungsten and argentiferous lead ore. In 1917 the Fredericktown mines produced from 2000 to 3000 tons of lead ore per day. Concentrators have been constructed and are nearing completion for the treatment of copper-cobalt-nickel ores that have been opened up. Such deposits have been found both in the sandstone and shaly dolomite.

The Flat River district is served by the Mississippi River & Bonne Terre Railroad, the tracks of which extend from Doe Run through to Elvins, Rivermines, Flat River, St. Francois, Desloge, Bonne Terre, Big River, and Herculaneum and terminate at Riverside. This road is owned by the St. Joseph Lead Co., which has a smeltery at Herculaneum, but all the traffic for the district, consisting of freight, ore, and concentrates, is transported over this line. The St. Louis, Iron Mountain & Southern Railway, which has been absorbed by the Missouri Pacific Railway, connects Riverside with St. Louis and Potosi. The Missouri Pacific also connects the district by two lines to southern points, and the Illinois Southern Railway connects St. Francois with the Mississippi River and Illinois coal fields. Perryville and other eastern Missouri points are connected with Farmington, the St. Francois County seat, by the Cape Girardeau Northern Railway. The Federal Lead Co. ships concentrates to its smeltery at Alton, and the St. Louis Smelting and Refining Co. has a smelting plant at Collinsville, Illinois.

Mining conditions and methods throughout the district are similar in essential features and differ but little in detail. Orebodies are found and explored by diamond drilling, and subsequently are developed by vertical shafts sunk through the main ore level, which ranges at depths in various parts of the district from 350 to 500 ft. The orebodies in the main lie from 40 to 50 ft. above the La Motte sandstone, and shafts are sunk at a point in the area most convenient for drainage and tramming grades, as indicated by diamond drill holes. Skip pockets and sumps are cut below the track level, and orebodies that exist either above or below this level are connected by raises or winzes. The ground, whether in ore or country rock, is prevailingly hard and of good standing quality, and little or no timbering is necessary except for chute sets, manways and in the shafts. Timber in the latter is used only for stulls between compartments to support guides and ladderway equipment. Stopes are worked laterally, following the ore for the full height of the breast, leaving columnar pillars for support of the roof at intervals, depending upon the local strength of the back, grade of the ore, and convenience in laying track. The method of drilling and advancing a breast varies according to height of the heading. Where the height is above that of a drill column, an advance heading is carried close to the roof, and the ore below the drift is taken out in a series of stepped benches descending to the main track level. The broken ore is shoveled into cars and trammed by hand or mule to the main haulageway, where it is picked up in trains by a motor and drawn to the shaft.

The St. Joseph Lead smeltery, at Herculaneum, has a total capacity of 100,000 tons a year, and reduces the concentrates from the company's mills at Bonne Terre, Rivermines and Leadwood. The company produced in 1917 from its 20 shafts a larger tonnage than the rest of the district combined. Power is furnished from the new Rivermines turbine plant, which has a capacity of 6000 k.w. and supplies 6600 volts to all the company's mines and mills. A low-cost slack coal fuel is used, and electric power is transmitted over a triangular circuit connecting with the auxiliary power plants in reserve at each of the three mills. This system provides a possible concentration of power at any point from any source along the line.

At Bonne Terre, Shaft No. 1 is located at the 2400-ton mill. The bulk of the tonnage is hoisted through this shaft, though a small tonnage is received on railroad cars from Shaft No. 2. Shaft No. 1 has a depth of 180 ft., has two compartments and is equipped with two 2.4-ton skips, which dump directly into the mill bin.

The main haulage level at Shaft No. 1 is at a depth of 180 ft. The skips are loaded from chute pockets 38 ft. below this level, and these chutes are equipped with air gates of the vertical cut-off type, which operate through a slot in the chute lip below the ore stream. All the ore from this mine is hoisted to the surface from the 180 ft. main haulage level. Trains of 20 one-ton cars, coupled with link chains, are drawn to the shaft by Porter compressed-air locomotives. These locomotives are charged with air at 850 lb. pressure, and one charge will haul 20 tons one-half mile.

Cars are of the solid-box type mounted rigidly on trucks. The bodies are 2 ft. deep and have a square bottom with an area of 11 sq. ft. The wheels are equipped with Whitney Wonder roller bearings, for a 24-in. gage truck of 30 lb. rails. The coupling chains are attached to the end of a drawbar, which is a solid casting with a hook at each end, and is fastened rigidly to the bottom of each car.

The shaft station has a three-track arrangement. The cars are dumped into the skip loading pocket by means of two tipples or dumping cradles, described in the *Engineering and Mining Journal* of Dec. 1, 1917. Trains back in on either of two outside tracks, which at the station have a down grade toward the pocket. The station tenders unhitch the cars, and after unloading one by one at the tipples, return them to the central track, which has a slight down grade away from the pocket. This system is capable of rapid unloading where a small-size car is used. The gateless feature eliminates all trouble and delay incident to jamming of levers and catches, spilling or overturning. The cars and their auxiliary equipment can be made at the company's machine shops and foundry at a reasonable cost, and few repairs have been found to be necessary.

The mine at Bonne Terre is opened up by stopes and workings from the surface to a depth of 375 ft. At 80-ft. intervals sub-haulage levels, connected to the main motor-haulage level by raises or winzes, are equipped with the same type of car drawn by mules. At each sub-level above the motor level, dumping cradles similar to those used at the shaft pocket are installed. These raises in general have a zig-zag course from sub-level to sub-level. Such inclines minimize the tramming distance on each level according to the various relative horizontal projections of the orebodies on the various sub-levels and help to break the fall of the ore.

The mine workings below the 180 ft. level are connected by low inclined winzes equipped with single-drum hoists which raise and lower the cars. A 30° incline from a point near Shaft No. 1 connects to workings 90 ft. vertically below the 180 ft. level. Ore was mined from around this incline, but these workings are now abandoned and act as a sump. One inclined winze has a length of over 200 ft. and an inclination of 30°. A drift from the bottom connects over half a mile distant to another 200 ft. incline at 45° which connects to the 375 ft. level near Shaft No. 2, about one mile north of Shaft No. 1. The output from Shaft No. 2 is about 140 tons per day of one shift. The ore mined at present is mostly of the chloritic type, occurring close to the La Motte sandstone, and operations are confined mainly to old workings. This ore has a thickness of from 9 to 12 ft., and levels have been opened at 400 ft., 450 ft. and 500 feet.

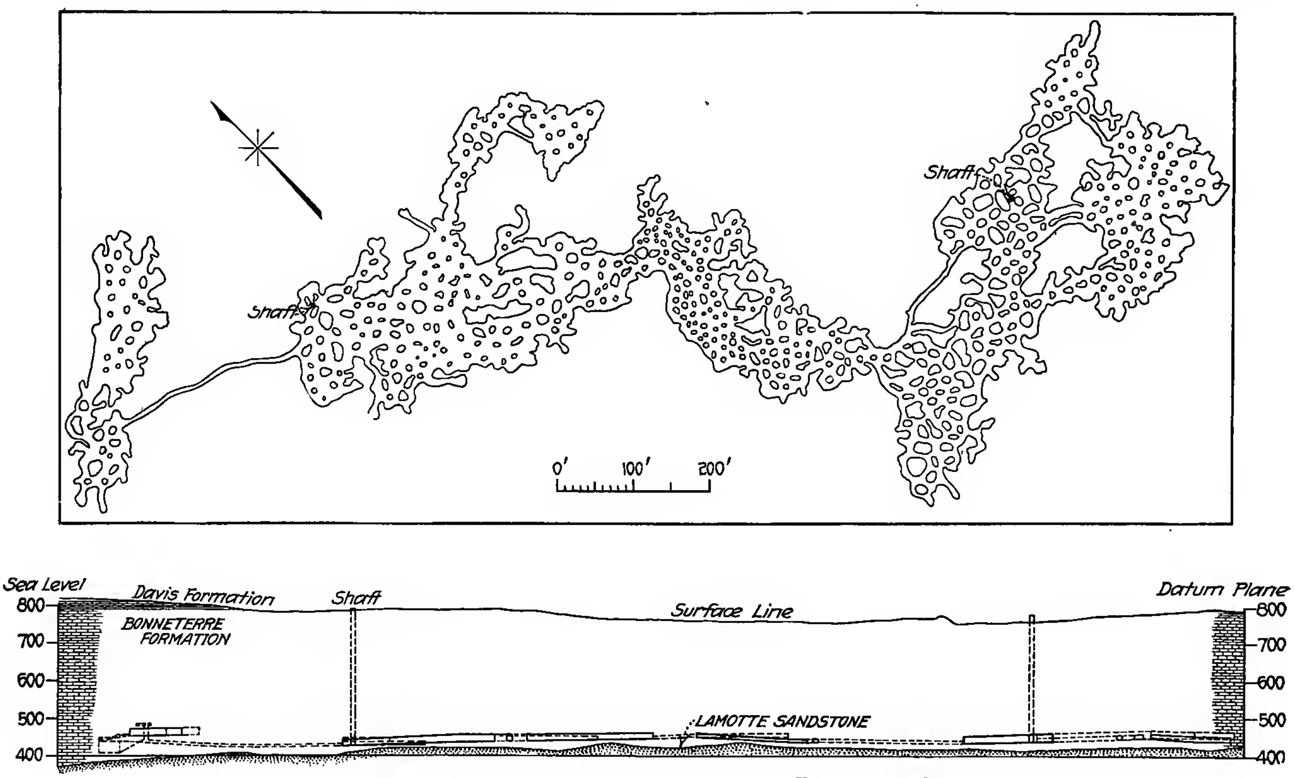


FIG. 16. PLAN AND SECTION OF TYPICAL MINE, FLAT RIVER DISTRICT OF SOUTHEAST MISSOURI

The mule barns at Shaft No. 1 are all on the 180 ft. level, and at the end of each shift the mules from the various sub-levels descend through the old stopes and connecting workings over well-trodden trails resembling those on steep mountain sides. The orebodies at Bonne Terre extended without regularity from the surface to the 375-ft. level or to the contact with the La Motte sandstone. In the early days the ore mined ran as high as 40% lead, and only the highest grade ore was extracted. The St. Joseph Company at present is mining over 75% of the tonnage from Bonne Terre from old workings, taking out pillars, stripping above and below old stopes and extending laterally into the low-grade walls. The ore now mined yields an average of 2% lead, but the grade as broken varies from 1 to 3%. The ore occurs in horizontal layers in the dolomite, with disseminated galena between layers. There are, besides, large bodies of purely disseminated ore. The ore breaks from the solid rather fine. Individual lumps are hard and tough and consist mostly of magnesian limestone. There is some chalcopyrite in the ore, the iron content amounting to about 5%. Concentrates from the Bonne Terre ores contain from two to three ounces of silver per ton, 0.4% copper and 9% iron.

The orebodies are often stepped up or down varying distances, as followed in mining, owing to local faulting. Some stopes are 136 ft. in height from floor to back and some only 25 to 50 ft. wide. Orebodies parallel to the bedding planes generally have considerable latitude, but vary in height from 8 to 25 ft. Individually and collectively the orebodies are irregular, and as the mineralization is erratic, the ore grade is subject to considerable local variation.

In virgin ground, heading and underhand stoping methods have been adopted and are similar in principle to methods to be described in more detail presently. Exceedingly deep orebodies are mined by underhand methods in horizontal slices from top to bottom. Besides many minor faults occurring throughout the orebody, open water courses are frequently tapped. Old pillars are robbed and in many cases are reclaimed altogether by stoping out a horizontal slice at the top and subsequently shooting down the whole pillar and blockholing large fragments.

Ingersoll-Rand and Sullivan plugger type drills have been adopted throughout the district. These drills are used unmounted or mounted with a 24-in. screw feed, and either wet or dry. When mounted in high stopes, extensible 2½-in. columns are used. Extensions are made by means of threaded sleeve joints. The rose bit is used altogether at Bonne Terre, and holes are drilled 10 to 16 ft. deep. Du Pont Red Cross powder of 40% strength is used for blasting. The ground cannot be sprung as at Joplin, and therfore holes are not chambered. Air pressure is supplied at 80 lb. gage at the mains. Brow or back holes are drilled in mining strips or ore left in the backs of old stopes. Such holes are drilled with pluggers mounted on extensible columns, and they are pointed at an angle to the back and are generally 16 ft. deep. Steel is sharpened at a shop on the 180-ft. level.

Ventilation all through the mine is good, although there are no strong air currents. A dozen or more openings to the surface through old shafts create a sufficient natural draft, and no fans are necessary. The men enter the mine through a stairway in a vertical shaft used solely for this purpose, but are hoisted on a cage at a third shaft which handles only men and supplies.

A 500-gal. motor-driven centrifugal pump is stationed at the sump near Shaft No. 1, and a 400-gal. pump at Shaft No. 2. A third pump of 400 gal. capacity is stationed near old Shaft No. 7, and the total water handled at present is only about 1300 gal. per minute.

The mine is worked two 8-hour shifts per day, but hoisting continues for three shifts at Shaft No. 1, which has a capacity of 2750 tons per 24 hours. Owing to the nature of operations about 48 tons of ore is broken per machine shift, which is somewhat higher than at mines working in virgin ground. About one ton of ore is broken per stick of powder.

In the Flat River area the mineralization appears to be confined to the sandstone basin, the limits of which have been rather well defined by contour data obtained from diamond drilling. Within the area of this basin the orebodies appear to have a northwest-southeast trend parallel to each other. The explanation of this condition is to be found in the more or less well-defined fault zones shown in Fig. 14. The shale is not always in the lower part of the basin, and the ore is not always deposited in connection with the shale; and no generally accepted theory as to ore occurrence has been formulated with sufficient exactitude to serve as a guide to exploration. Within the orebodies, the successive precipitation of lead, zinc, copper and iron is often found to occur in the reverse order or without any apparent order of succession. Rich ore is sometimes found at synclines and sometimes at anticlines. In general, the dip is to the west, but local faulting and gentle folding cause variations. Orebodies usually occur near the older faults and are themselves faulted, with displacements that range from 7 to 20 ft. Folding is often strong enough to carry the ore above or below the main level to such an extent as to necessitate working from sub-levels by raises or inclined winzes, which are, however, relatively short.

Six shafts are tributary to the St. Joseph Lead Co.'s Rivermines 4200-ton mill. St. Joseph company's Shaft No. 3 is equipped with cages. Some shafts in the district are equipped with skips and over-head cages and one shaft has an auto-motor hoisting equipment. All the hoisting is from one level. At the Federal Lead Co.'s Shaft No. 12, a pocket of 300-ton capacity has been cut for a depth of 38 ft. below the main level, together with a sump and pump station. This sump receives the drainage from auxiliary pumps installed in the lower stopes, where some ore occurs close to the sandstone. Cars are nearly all of the same type as those used at Bonne Terre. On the cage, these cars rest upon a false track bottom which has a rectangular area equal to the gage and wheel base of the cars. This section is cut out of the center of the main deck. When the cage rests on the station chairs, the false bottom is raised level with the track of the main deck and station, but when the cage is lifted from the chairs, the false bottom drops relatively with the car two or three inches below the main deck, thereby preventing movement of the car along its track during hoisting by effectively blocking the wheel fore and aft. This is another advantageous feature. in connection with this type of car, conducive to rapid hoisting. The cars are hoisted to the top of the loading bins at the surface, where there is a pair of dumping cradles, operated by two tenders. From the bins railroad cars are loaded and the ore is transported by locomotives to the mill at Rivermines.

The haulage level from St. Joseph Shaft No. 3 is 475 ft. below the surface, the horizon of the main orebody; and conditions are altogether fairly well representative for the district as a whole.

The station at this shaft is equipped with three 24-in. tracks which handle the cars as received in trains drawn by gas-motor haulage locomotives. At some of the mines, electric haulage equipment has been installed. The central track has a slight down grade toward the shaft, and from it the loaded cars are trammed by station tenders to the cages. The empties are returned on either of the side tracks, which are sunk below the main track level to give the proper down grade away from the shaft.

From the shaft station a haulage drift has been driven 7 ft. high and 8 ft. wide. The elevation of this level was established from diamond-drill data, and the station track elevation thus predetermined by making proper allowances for an upgrade drift away from the shaft. This elevation was selected as nearly as possible according to the average of the lowest points in the main orebody. The slight effect of anticlines and synclines in the orebody is disregarded for the sake of an efficient haulage system, and any ore that occurs below the track level is stoped subsequent to the completion of its upper portion.

The main haulage drift after reaching the orebody follows its general trend through the longitudinal center. Drifting and stoping operations may be conducted simultaneously, stopes being opened up laterally from the main drift by leaving wall pillars between and a sufficient lag in stoping operations behind the heading in the haulage drift to avoid conflict of operations.

With the exception of a few drifts that are driven through the dolomite, connecting the shaft with the main and outlying orebodies, and raises or winzes connecting upper and lower orebodies at other horizons, there is comparatively little development through barren country rock. Most of the ore is developed in the course of stoping, the general limits of deposits only having been roughly predetermined by diamond drilling. Diamond drills, however, do not give dependable outlines, and the cuttings from rock drills are watched closely as the faces of stopes are advanced.

In drifting, rounds are invariably drilled by pluggers mounted on columns. The wide drift section and the flat stratification of the dolomite have developed an advantageous side-cut system of drilling rounds. Most of this work is done on contract, one shift drilling and the next shoveling. Each round is alternately drilled from a set-up near one corner of the heading, all holes being drilled radically from the column 4-ft. deep. Twenty holes are generally drilled per round and pull about 3½ feet of ground.

There is little or no ore in the sandstone in this section of the district, though there is sometimes an occurrence of ore in the dolomite close to the contact. In drifting, the unevenness of the sandstone floor often necessitates driving through this formation. While driving, such headings are invariably wet, and a veritable shower follows the heading, lasting for several weeks or until the formation in the vicinity is drained. This causes a considerable flow of water along the drifts, which cannot be handled by a ditch, as the amount of loose sand would soon fill it up and render it useless. Such headings are avoided where possible on this account. In drilling, water reservoirs are frequently tapped, and for a time there is often a considerable flow. Such reservoirs eventually become drained, but until such time work is often necessarily suspended.

Where the main heading is driving through ore, lateral crosscuts are driven at intervals and throw switches are put in the main track. The motors draw the cars to and

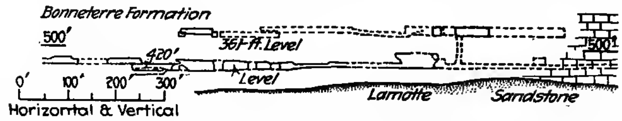


Fig. 17. Section of Typical Flat River Mine, Showing Relation of Main and Upper Ore Levels to the La Motte Sandstone

from the stope headings where convenient, but in most cases mules also have to be used supplementary to the main level haulage, as well as on sub-levels above, where such exist.

The height of a stope breast is 7 to 8 ft. The breast and underhand bench method of advancing faces is used throughout where the face of ore is greater than 8 ft. high. Breast holes are drilled from 8 to 10 ft. deep, depending upon the height of breast. In all cases where the ore is high, the breast heading is driven at the top of the orebody. Stope holes are invariably drilled down, although lifters are sometimes drilled under the bench when the stope is mucked clean, which is seldom the case at the time drilling starts. High stopes are advanced in steps, the breast of each bench lying on a general incline of about 60 degrees.

Each round in a stope breast pulls about 3½ ft. of ground horizontally and is drilled with three holes at an angle to the face. To gain greater breaking efficiency in plan, stope headings are advanced with a serrated outline. Three holes are used and the central hole is drilled a few inches closer to the face than either the upper or lower. In blasting, the center hole is set off first, which relieves the burden from the other two, thereby securing greater breakage than is otherwise possible. Stope holes are drilled down, and the face is advanced in 8 to 10-ft. benches. These benches are drilled and blasted in order from top to bottom as the face advances.

Plugger-type drills with a pneumatic feed, and rose, Carr and bull bits are in general use. Where hollow steel is used, the holes at the bits are sometimes placed in the side instead of in the center. Extension columns are used up to 25 ft. in length, and 60 to 80-lb. air pressures are maintained.

As a rule, the bulk of the ore is shoveled only once, the shots being so placed at the various benches that the ore is thrown to the track floor of the stope. The long-handled round-point shovel is preferred, the ore being shoveled from a rough bottom. One of the companies in the district is using five Meyers-Whaley shoveling machines with success. A power shovel of another make had been tried out a few years ago by one of the large companies without much satisfaction, but the conditions of the trial were not conducive to the best results. Power shovels at present are solving or will solve to a great extent the labor problem, and stoping conditions are especially favorable to their use.

As at Bonne Terre, the upper ore horizon is connected to the main level by raises equipped with dumping cradles at the top and chutes at the bottom, and lower orebodies by winzes equipped with hoists and single track inclines over which cars are raised and lowered.

The ore occurs in layers parallel with the stratification. Some layers are cut by small faults, but the edges of the orebody in general taper out in grade from the center. Large bodies of waste are found to occur within the limits of the orebody. Ore is often found disseminated between the stratified layers, and often there is nothing but waste between such layers. Bands of shale frequently occur in the dolomite and make bad roofs. When careless machinemen drill their holes too high and break into such bands, the roof is thereby greatly weakened. Such loosened strata or shells in the back have to be either barred down or shot out.

The ore varies from 7 to 50 or 60 ft. high in different parts of the mines. A stope in one mine was over 90 ft. high, consisting of superimposed layers of ore about 300 ft. in diameter. This stope appeared to lie along one of the older fault fissures and yielded an average of 4% lead.

Pillars are left where possible in the low-grade ground, but in ore in moderately high stopes they are from 16 to 18 ft. in diameter and from 20 to 30 ft. apart in the clear. In high ground, pillars are often 20 to 30 ft. in diameter and are spaced from 10 to 12 ft. part. In high ground it is better to have small pillars close together than large ones far apart, as a better extraction is obtained when robbing later. Big pillars when robbed leave too wide a roof unsupported between. The pillars at some of the mines are left with a triangular spacing in the clear of 22 ft. and are from 20 to 25 ft. in diameter.

Pillars are usually mined from top to bottom from scaffolds set up on ladders until a bench has been cut around the top high enough to set up a drill column. From this bench, holes are drilled down around the pillars, and a big tonnage is thus broken.

Work is guided largely by local conditions, and barren limestone often has to be taken to gave a stoping height of 7 ft. Ore is mined as low as 2% lead, and the

FIG. 18. STOPE HEADINGS ARE ADVANCED DRILL COLUMN HIGH CLOSE TO THE BACK, AND ORE BELOW THE HEADING, IF ANY, IS UNDERSTOPED IN BENCHES PROGRESSIVELY, LEAVING COLUMNAR PILLARS IN THE OREBODY, SPACED AND PROPORTIONED ACCORDING TO HEIGHT AND STRENGTH OF THE ROOF OR GRADE OF THE ORE

average for this type of stope must be estimated accordingly. Low-grade ground is often mined to develop higher-grade places not discovered by diamond drilling and in order to supply a constant workable tonnage to the mill.

Upper ore horizons are frequently 50 or 60 ft. above the main level stopes, floor to floor, and have from 35 to 40 ft. of barren ground left between. In mining such upper orebodies, where they are directly over a lower ore horizon, the latter is mined first and pillars are left with some degree of regularity. In subsequently stoping in the upper horizon, the pillars are spotted by the engineer, so as to be left superimposed with respect to those below.

If less than 10 ft. of barren ground lies between the orebodies, the whole is taken out.

The percentage or ore left in pillars varies with the height of the ground, but averages in the district from 15 to 20%. One of the greatest difficulties to overcome among the miners is the tendency to leave large pillars. Pillars are later robbed, however, as at Joplin, but the total extraction is probably higher in Southeast Missouri mines on account of different surface conditions, pillars being robbed to the extent of about 50 per cent.

Incline raises to upper levels are usually flat and consist of two compartments. The manway compartment is lined off from the ore chute and is frequently equipped with steps. Ore chutes are equipped in general with arc gates, and the aprons are lined with track rails. Two-compartment inclines are sometimes made with the manway over the chuteway, instead of alongside, the object being to save the timbering in the lining from wear. In the fall of 1917 wages in the district were, according to the sliding scale in use, 75% above the base wage, which for machinemen is $2.75 per day and for shovelers $2.40 per day. Thus machinemen receive $4.80 and shovelers $4.40 per day.

In general, throughout the district, the average broken per man underground varies from seven to eight tons. The average amount of ore broken, except at Bonne Terre, is about 40 tons per machine shift. Shoveling varies in different parts of the district, largely according to the method of tally, and about 17 cars, or 20 to 21 tons, is shoveled per man shift at present although formerly 23 tons was common.

A comparison of present mining costs in the Southeast Missouri district, although necessarily of economic importance locally, can have under the abnormal conditions that now prevail throughout the country but little significant value elsewhere. For this and other reasons best known to the various companies operating in the district, no cost data could be obtained. H. A. Guess, however, in a paper [9] presented at the New York meeting of the American Institute of Mining Engineers, February, 1914, gives the following costs, which may be taken for an average in the district at that time. The costs are broadly segregated into breaking, shoveling, motor haulage and drainage in the first table given by Mr. Guess. In the second table which is somewhat more comprehensive, although representative for one company alone, prospecting, development and general expense are given.

TABLE III. UNDERGROUND COSTS PER TON OF ORE IN FLAT RIVER MINES IN 1914

Breaking, labor	$0.14
power	.05
explosives	.08
Shoveling	.13
Motor haulage	.05
Drainage	.08
Total	$0.53

In 1915, Skinner & Plate published [10] the data in Table IV given by Mr. Guess for mining costs of the Federal Lead Company.

TABLE IV. MINING AND MILLING COSTS PER TON IN SOUTHEAST MISSOURI

Prospecting	$0.12
Development	.04
Ore breaking	.46
Mine to mine bins	.23
Mine bins to bill	.04
Milling	.22
General expense	.14
Total	$1.25

The St. Joseph Lead Co. costs from the annual report also are given, as in Table V, for the year ended Apr. 30, 1914:

TABLE V. COSTS PER TON OF ORE MINED AT THE ST. JOSEPH LEAD CO.

	Bonne Terre District	Leadwood District
Mining	$0.87	$0.78
Milling	.35	.30
Railroad and freight	.06	.04
Total	$1.28	$1.12

Interesting cost data were published [11] by J. R. Finlay in 1909. Mr. Finlay made a thorough study and analysis of local costs, and estimated the following ranges as an average for the district, as shown in Table IV.

TABLE VI. TOTAL COSTS PER TON FOR MINING, MILLING AND SMELTING IN THE SOUTHEAST MISSOURI DISTRICT

Mining and hoisting	$1.00	to	$1.50
Transfer to mills	.05	to	.10
Milling	.30	to	.50
General expense	.10	to	.20
Freight to St. Louis	.097		.097
Smelting	.378		.378
Total operating	$1.925	to	$2.775
Adding depreciation	.312		.312
Dividend costs	$2.237	to	$3.087
Adding amortization	.576		.576
Total net	$2.813	to	$3.663

Mining methods as a whole in the Joplin and Southeast Missouri districts are similar, except in details of auxiliary operations, and appear to be well adapted to the class of deposits mined. The flat nature of these bedded deposits, the relative low height of ore, compared to lateral extent, combined with the depth, hardness of ground and necessity for preserving surface rights, present certain conditions for roof support that preclude caving, and the low grade of the ore not only prohibits the use of timber but makes it far more economical to sacrifice a small percentage of the ore in the form of pillars. The wholesale manner in which the ore is stoped and milled does not, from a cost standpoint, permit the practice of underground waste sorting, and even where the grade hoisted might be materially improved thereby the economy of so doing is questionable, as the gain would undoubtedly be more than offset by the added cost per ton.

9 "Mining and Mining Methods in the Southeast Missouri Disseminated Lead District."
10 "Mining Costs of the World."
11 "Cost of Mining."

Recovering Caved Stopes in Narrow Veins
Part I

BY C. T. RICE

In the most carefully stoped mines, caving will occur at times if the ground suddenly becomes heavy. If the ore widens in the vein, and it becomes necessary to stope greater widths than previously, a much greater width is thrown upon the timbers than before, as the weight of ore or back, supported either by timbering or self-supported by the cohesion of the ground itself, increases rapidly in proportion to the length of span over which the back must arch itself. The sub-arch, or zone of ground below the natural arch or dome to which the ground would slough in order to support itself, represents the weight that is coming upon the timbers of the stope, as shown in Fig. 19.

Often, in wide orebodies, stopes cave below a flat fault, the presence of which is wholly unsuspected, for no special precautions are taken in order to hold the working. So long as the block of unshattered ground between the stope and the fault is thick enough to allow the back to arch itself across the opening, the load upon the timbers is little greater than usual. But just as soon as the arch encroaches upon the fault, there is a collapse, and a great mass of rock is released, the weight coming so suddenly upon the timbers that the pressure exceeds their power of resistance. Usually the weight comes so suddenly that it is impossible to rush in cribs and reinforcing timbers quickly enough to save the stope. But occasionally, if an abundant supply of timbers is kept on hand upon the different levels, it is possible even then to prevent a cave.

In mining narrow steeply dipping veins that require timbering it is not usually a weak back that causes the trouble, except as a consequence of weak walls. When stopes cave along narrow veins, it is almost always the effect of top weight coming suddenly from the back that crushes and collapses the timber sets. Occasionally, and then only when the walls are highly fractured and filling has been permitted to lag behind the back so that five or six floors are left open, does the unsupported arch which extends into the side walls, as shown in Fig. 19, become too great for the strength of the wall rock. Still, even in narrow veins that have never been faulted, and which are characterized by fairly strong walls, the ore itself is often so greatly shattered by subsequent movement along the plane of the vein, and frequently as a result is so broken up by slickensides, talc seams and clay gouges, that a cave results. For in such ground the weight is likely to come suddenly and erratically upon the timbering, as the talc and gouge, especially if damp, act as excellent lubricants to cause the ore to slip upon itself when the back tries to arch itself across the stope. Especially is this somewhat peculiar condition of ground to be met when the ore occurrence, as in the Cœur d'Alenes, is the result of the mineralization of a fractured zone rather a single open fissure.

Occasionally when a fault cuts through the vein or when the ore itself is much shattered, the timbers in a narrow stope will hold even with several floors of the stope unfilled. But in order to minimize the occurrence of caves the filling should be kept as close to the back as is consistent with efficient mining and economical handling of the ore in the stopes. In the past there was much greater inclination to let filing lag far behind mining than at present, but even now it is not amiss to remind mining men of the great importance which waste filling has in preventing the weight that is thrown upon the timbering of a stope from becoming excessive.

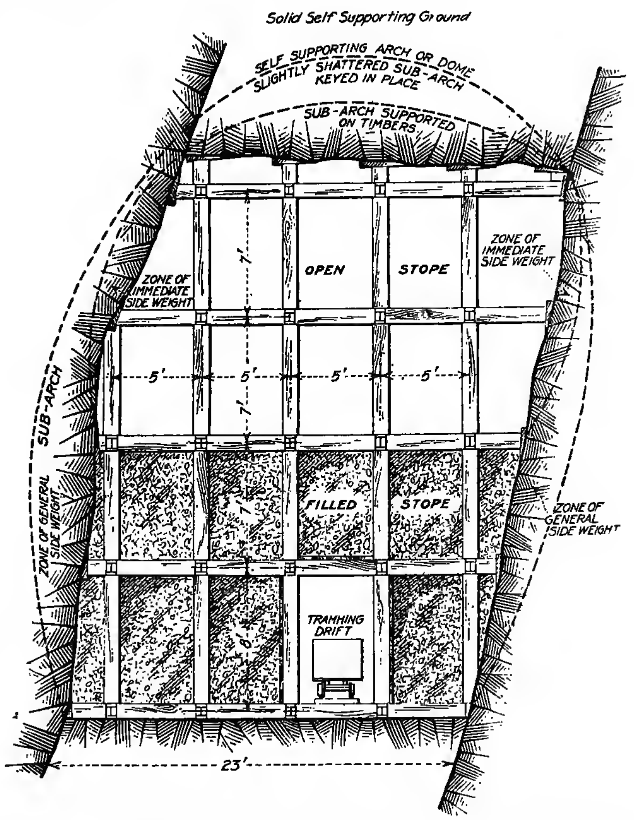


FIG. 19. CROSS SECTION OF TIMBERED AND PARTLY FILLED STOPE, SHOWING ZONES OF WALL AND BACK ARCHING

When narrow stopes cave it is frequently the result of a peculiar combination of conditions not previously understood. Often the ore along comparatively narrow veins "makes out" into the walls in the form of wide kidneys or lenses, enlarged to two or three times the average width of the vein, and such kidneys of ore are usually accompanied by considerably fractured walls. These lenses usually are found either where the vein crosses an earlier fissure that permitted the ore solutions to mineralize both it and the broken-up ground between, or at points where brittleness caused the wall rocks to shatter more than usual. In either instance the walls of the stope will be much weaker than elsewhere on the vein, and, unfortunately, this condition occurs at stoping widths where it is most essential that the walls be strong.

Often wide kidneys along narrow veins can be stoped without great difficulty. Little trouble is experienced usually in working out the lower part; but in stoping the upper portion, when the width of the stope is rapidly diminishing, the conditions, as shown in Fig. 20, are such as to make a cave extremely probable. In the lower part of such a kidney or lens the width of the stope increases rapidly, but fortunately the shape of the walls is such as to prevent the stope from caving, for the back arches itself from the walls much as if they were the buttresses of a bridge, while whatever top weight may come upon the timbers tends simply to tighten them more securely in the blocking. Unfortunately, in the upper part of the kidney, mining conditions are reversed.

Between the converging hanging walls the full effect of wall and back fracturing combines to destroy the arch, and the back becomes a bridge from which the buttresses have collapsed, with resulting caving of the unsupported arch. The consequence of this condition is that, instead of the back being able to arch itself across the stope from the walls at points immediately above the timbers, it has to find a footing higher up in the walls, throwing a greater weight of sub-arch upon the timbers, and the stope begins to collapse. When only one or two floors are open, frequently the timbers will stand long enough to permit cribs and reinforcing sets to be rushed in, and the stope to be saved. But usually the weight comes so suddenly that nothing can be done to prevent caving of the stope. Once the ground is really in motion, it is useless to try to save a stope, although even then it may be possible to rush cribs and doubling-up sets in fast enough to hold the bottom floor open when several floors of the stope have been left unfilled.

In a narrow vein, it is seldom that ground will continue to cave until the stope fills itself clear to the back. Generally the ore caves just enough to give sufficient support to the walls for the back to find footings strong enough to enable it to form an arch across the top of the cave. But usually the cave will have by then eaten up into a narrower part of the vein or else to a point where both the walls and the ore itself are much less fractured. Typically considerable open space will remain between the pile of caved ore and the back.

Sufficient time for establishing this state of equilibrium must be given before attempting to reënter the stope. When the back and walls are naturally strong, and the orebodies narrow, though occasionally even in orebodies so wide as to require square setting, it is possible to begin this in a few weeks after the cave occurred. If the ore is weak, however, and the orebody wide, it may be necessary to wait several years before it is safe or economical to reopen the stope.

The time-honored method of reopening caved stopes, whether they be in narrow or wide ore occurrences, is to begin at the bottom and work up through them. This is true largely because practically all our methods of reopening caved workings are based upon methods developed in the course of recovering caved stopes in wide orebodies. But this practice is fundamentally wrong. It is wrong no matter whether the caved working is wide or narrow, as I hope to prove in this and another series of articles dealing with the problem. For not only is it safer, but, taking everything into consideration, it is also cheaper, to work down through the caved ore than it is to try to come up through it from below, with runs of ore and serious accidents always imminent. Moreover, this is true whether the ore has arched itself over the cave, as is generally the case when the cave has occurred along a vein characterized by narrow stoping widths, or the stope has been so wide and the walls so weak that the back could not arch itself over the opening, and so both walls and back have closed in completely. This discussion will not be theoretical, but it will be based upon mining practice under widely differing conditions that have come under my own personal observation. Therefore while most mining men will no doubt think it rather absurd to argue that ore can be obtained not only much more safely but also fully as

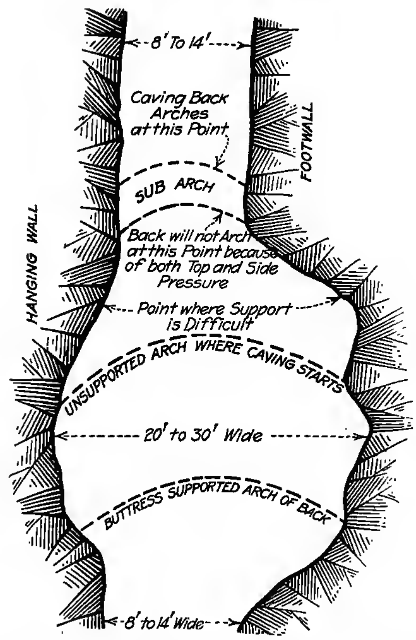


FIG. 20. CAVING STARTS WHEN THE ARCH OF THE BACK FAILS TO RECEIVE PROPER WALL SUPPORT

cheaply by working down through a caved stope in such a camp as Bisbee or Butte, I believe that my observations will go a long way toward making many readers resolve to try at the earliest opportunity the methods that I outline. As the reopening of a caved stope in a narrow ore occurrence is much less difficult than the recovering of a caved working in a wide deposit, I will discuss that problem in this series, and leave the other for the next.

The procedure to follow in recovering narrow caved stopes is to work in to the top of the caved ore pile from one end, usually by stoping to the necessary height in the undisturbed ore next the cave.

With ore that is strong enough to arch itself securely over the caved area, the back can be caught up with temporary timber supports. By this means miners are protected from a sloughing back, and the timbers will give ample warning if the back again begins to take weight. The problem thereafter is to support the walls as the height of the "muck pile" is lowered. The more usual procedure is to begin mucking out the ore from the edge, drawing it out from below at one end of the pile and securing the walls with timber sets as fast as room is made. If the walls can be supported promptly and at regular intervals during removal of the fill, so that the span of the arch is not permitted to increase, the back will stand indefinitely.

In the case of caved stopes in veins of stulling widths, there is far more latitude in the method used in getting out the ore and catching up the back than in the case of a caved stope in a naturally wide ore occurrence; for when a square-set stope has caved, it is either a case of booming out the ore by means of suspended square sets, working downward from the top in sections, or of waiting for several months until disintegration of the gangue minerals and the weight that comes upon the pile from the walls and back consolidate the broken ore sufficiently for it to stand over an opening a set or two in area while room is being made for timbers.

As the methods of recovering caved stopes in narrow veins are based upon the practice that has been developed at the Hecla mine at Burke, Idaho, in the Cœur d'Alene district, I will outline the ore conditions which characterize that mine as well as the method used in mining the ore. In the western part of the Coeur d'Alene the lodes dip close to 80° and show a tendency to "make out" into the walls at certain points, thereby forming large kidneys of ore ranging from 20 to 30 ft. or more in width, along vein-like deposits usually only from 8 to 10 ft. wide. The walls are quartzite, and generally stand fairly well, although often showing tendencies in places to slab off as they take weight.

The orebodies which have made the Hecla mine famous occur along fracture zones that generally follow a 2-ft. porphyry dike. Some ore occurs independently of this dike, but the main ore occurrence follows it closely. The mineralization makes out laterally into the crushed quartzite walls, with indefinite limitations. Generally the ore zone is from 8 to 14 ft. wide, but frequently it expands laterally to form rich lenses. When such lodes widen out into large kidneys of ore, and there is a clay selvage or gouge at the boundaries of the minable width, it is almost impossible to carry up a stope without having it cave. A regular system of handling caves has been evolved at the Hecla by working downward and using spliced stull sets up to widths of 30 ft. and for distances as great as 100 ft, along a vein caved to a height of 60 feet.

Caved stopes occur more frequently at the Hecla than at most of the mines in the Cœur d'Alenes. The reason for this is the system of mining. The endeavor is to mine the ore at as low cost as possible, with due regard to the safety of the stopes, so as to get a maximum tonnage at a minimum cost, rather than a clean product with no waste.

But the method has not been developed merely to get the ore out of the stope cheaply. Owing to the manner in which the ore has been formed, it is not unusual to break into a rich seam of galena on an upper floor which comes into the stope from the side and below, and so has not been found in mining the ore on the lower floors. Unless a considerable height of stope were kept open, it would be expensive to go back to get this ore. In fact the expense would be prohibitive, and either a great deal of money would have to be spent in prospecting the walls of the stopes or much rich ore would be left in them. Indeed, with all the prospecting much ore would be lost, so I believe that the method of mining which has been developed at the Hecla mine has had more to do with making it the important producer that the property has been in the last few years than any other one thing, for in all my mining experience I have never seen a method so admirably adapted to the ore occurrence as the Hecla stull-set and waste fill method.

Briefly, the method consists in mining the ore overhand in horizontal slices three floors high, with subsidiary stope tramming for each slice. As soon as a slice has been mined across the block, the stope track is raised three sets, and the floors below are filled with waste. This waste filling is obtained either from the surface or from development work, as all the rock broken in a stope is sent to the surface and sorted, the waste being sent back into the mine for use as filling in the stopes through a waste raise. As there is a tramming floor always open, and as the floor above is always left open so as to aid in getting the ore through the cross-boards, there are five and sometimes six floors open by the time that three floors have been mined across the stope.

Instead of putting in simple stulls to hold the stopes open, stull sets are used; that is, the stulls are carried by posts and placed sideways by girts from one another. This is done for two purposes: First, in order keep the floors level in the stopes, for otherwise the stulls would have to be carried at right angles to the dip of the vein; and, second, so that when a seam of ore is found on an upper floor to be making down into a wall, that wall can be shot out on the lower floors without stulls dropping out, as would be the case were they not carried by these posts.

Round timbers from 10 to 16 in. in diameter are used for the stull caps in these sets, and stulls are put in up to 16 ft. in length. Above that width of stope the stulls have to be spliced, as it is difficult to get longer timbers through the manways. The stull caps are put in at 5-ft. centers horizontally, and at 9-ft. centers vertically. They have headboards both along the foot and the hanging wall. Always two and usually three 3-in. planks, 2½ ft. long, are used at each end in forming these "headings," so as to provide a cross-grain cushion from 6 to 9 in. thick at each wall to protect the stull from being broken by the initial creep of the ground, which is heavy, as 45 ft. is left open vertically along the vein, and occasionally 54 ft., by the time that the tramming tracks are raised to take out another slice along a stope.

The posts of the stull sets fit into 1-in. daps cut into the stulls in the stopes with hand saw and adze. These posts are generally about 10 in. in diameter, as they do not have to carry much top weight except when it is necessary to shoot out one of the walls or a stope begins to take weight owing to bad ground. As the stulls are much larger than the posts, separate girts or collar braces are used to brace the caps and posts of the different stull sets one from one another.

The level interval varies throughout the mine, but is usually 250 or 300 ft. Stopes vary greatly in length, being usually several hundred feet long. Raises for sending waste filling down into the stopes are generally from 250 to 300 ft. apart, while manways and chutes are carried up at 50-ft. intervals. These are arranged rather peculiarly. They are made three sets wide, and the manway is placed on one side, the timber slide on the other, and the chute — a box chute of special design — in the center.

An effort is made at the Hecla to minimize the handling of ore in the stopes. Therefore, instead of adhering to the older practice of the district, of carrying the filling close to the back and of keeping a mucking floor immediately below the mining floor, the present method is to drop the ore down several floors to a cross-board system of lagging immediately above the tramming floor, and to run most of it into a car and thence to the nearest chute, with little, if any, shoveling.

The floors in the stopes are laid with single 3-in. planks of random widths. The cross-board floor is of peculiar construction. It consists simply of ordinary 3-in. planks with a plank 12 in. wide taken out directly over the center of the track on the tramming floor. The two planks forming the side of this opening are nailed to the stull caps to prevent them from slipping. Then this opening is closed with a series of 3-in. planks, 12 in. wide and about 18 in. long, placed crossways with the opening, and therefore side by side. Pieces of 3 x 10 in. plank 10 in. long are then nailed to the under side of the cross-board pieces, so as to keep them from being kicked out of position. The ore is worked through to the car below by removing the cross-boards one at a time, at the edge of the pile. This is a mucking development somewhat similar to the Australian "Chinaman" chute system that originated independently to meet the requirements of the method of mining at the Hecla.

As not only the wall rock but also the ore is strong, being a mineralized quartzite, no trouble is usually experienced in holding a stope open even for five floors, and for considerable periods, when, owing to shortage of waste, delayed filling is unavoidable. Occasionally, however, the stopes will cave, and often this occurs soon after the tramming tracks have been raised and the lower three floors filled. Now that it has been found that these caved stopes can be mined at little, if any, greater cost than those which do not cave, and that often the lowest-cost ore comes from caved stopes, no great precaution is taken, as there is no especial danger.

Usually caving will continue until the ground has stoped itself up to narrow widths in the lode, as shown in Fig. 20 and after the caving ceases there is usually an opening 4 to 10 ft. high between the caved ore and the arched back. While this back must be securely caught up by timbering before any of the caved material can be safely drawn, it is generally found strong enough to require blasting after the stope has been filled with waste and normal stoping operations have been resumed.

At the Hecla, caves do not come without considerable warning, and generally ample opportunity is afforded in which to reinforce the timbers sufficiently at least to hold the tramming floor open, and often there will be time left also to reinforce the cross-board floor above.

As it is top weight oftener than side weight that gives the trouble, more posts are required. Reinforcing posts are stood on stringers which are laid crosswise to the stulls between the original posts, and directly support the stulls above. Sufficient floor lagging is removed to make room for the stringers. These stringers do not tie together the stull caps above, for if a stringer were interposed under the stull caps and on top of the helper posts, it would interfere with intermediate stull sets, should such become necessary to hold the ground. If time still permits, and there is enough old timber lying along the level, bulkheads may also be built on the tramming floor. Stringers and helper posts, also intermediate sets if they seem necessary, may also be placed on the cross-board floor. If time remains after this much has been done, attention is directed to strengthening the rest of the stope.

When it is found impossible to hold the stope open, it is abandoned for several days, or until the first period of caving has ceased. Then, if still open, whatever additional reinforcements that may seem necessary on the tramming and cross-board floors will be put in, and an attempt made, as soon as serious sloughing of the back has stopped, to get in on top of the pile. Sometimes the caves are not so high but that it is possible to get into the top of the pile from the top floors of that part of the stope which did not cave. Generally, however, it will have extended to such height that this is not possible.

Success in recovering caved stopes by catching up the back and then working down on the muck pile depends mainly upon proper placing of the stulls. In catching up the back, the stulls must be put in with long headboards and as closely spaced as required. These stulls must hold the top weight that will first come upon them until the nip of the wall sinks them deep into their headboards. The headboards used for this purpose are 3-in. planks 5 ft. long. The reason for using such long headboards on these catching-up stulls is to provide grip on the walls. Generally the stulls are of such diameter that two planks have to be put in side by side to fully cover the end area. The stull is usually cut in length so as to take between it and the walls at each end a thickness of three planks and the tightening wedges.

Any irregularities in the walls are, of course, filled in tightly with blocking, although it is best to get the headboards themselves against the ground for as much of their length as possible. The headboard scheme A, as shown in Fig. 21, is put in so that about three feet of plank extends below the stull; then the wedges are driven so that the lower part of the headboard is keyed out appreciably to allow for possible top weight, before the side pressure has compressed the ends firmly into the headboards. Such top weight will then tighten the stull in its headboard, blocking instead of kicking it out. In Fig. 21, B, C and D, three typical examples of stulls put in to catch up the back over caved stopes are shown. It will be noticed that their purpose is not so much to hold the walls apart as to keep up the back. Blocking, short sprags, or even small false sets are used to prevent possible slabbing of the rock from above, and, occasionally, when the stulls are some distance apart, the back between them is securely laced up with plank.

The permanent stull caps of the regular stope timbering are put in below the catching-up stulls, just as fast as sufficient wall is exposed to permit of their being placed at 5-ft. centers horizontally and 9-ft. centers vertically with respect to the timbers in the part of the stope alongside which has not caved.

Occasionally the stope is so open at the top, when it is first entered, that two stulls, one above the other, can be spaced at the vertical interval forthwith, after which posts are placed between them. After all open spaces along the stope have been caught up, the permanent stulls of the stopes are worked in singly, no more wall being left unsupported at any one time than is absolutely necessary. Then, as soon as a stull has been worked in under one above, the posts are entered in the 1-in. daps between, as in regular stope sets. Longitudinal braces are also put between the stulls and also between the posts. When the stope gets wider than 16 ft. in the caved workings, spliced stulls are used. These are put in by butting two together, either over a single or double post, using one of the stulls full length. Two posts are always used when the shorter stull or butt block is longer than about three feet, for, with stulls of differing lengths, the top weight is unequally distributed, and a single post would tend to split as a consequence.

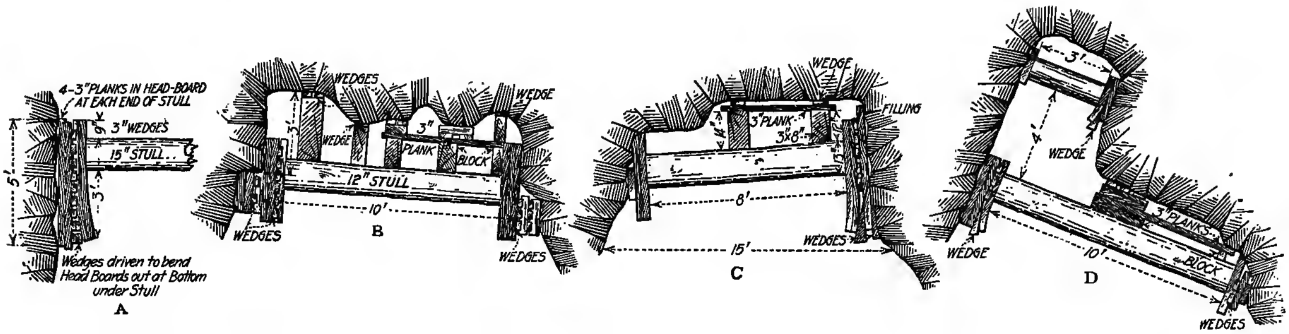


FIG. 21. DETAILS OF CATCHING-UP STULLS AND BLOCKING UNDER BACK IN STOPES WHERE CAVING HAS CEASED

Moreover, spliced stulls are put in with the spliced ends a few inches higher than the wall ends, so that with side pressure they will yield upward rather than downward. To resist this movement, the butted ends are firmly blocked by posts to the upper timbers and then to the back. Thus, any top weight transmitted from above through the posts tends to settle the wall end more securely into the headboards. For the same reasons, all stulls put in singly or spliced are placed with any existing warp or bend curving upward.

The stulls for catching up the back are put in wherever ground conditions seem to demand them and at whatever angle the shape of the walls at these points requires them to be placed in order to hold, for the ground is in such ticklish condition that little picking can be done to get them in, but the permanent stulls are always placed horizontally, except for the slight trussing effects mentioned. This is possible because the walls are nearly vertical, and it has the advantage of floors level across, as well as along the stope, the posts dispensing with the necessity for footwall hitches and usual angle of underlie. No trouble is experienced maintaining stulls, even when they are far from being perpendicular to the walls.

The "headings" for permanent stulls consist of a thickness of three, and generally four, 3 x 12-in. planks, but in this case the planks are only 2½ ft. long, as the main function of these headboards is to provide cross-grain cushions as protection against excessive wall pressures. As the permanent stulls have to be placed according to definite position, no matter what the wall conditions may be, considerable blocking must often be worked in back of the headboards, as indicated in Fig. 22.

Whether the ore pile is to be removed by drawing it down on the stope of the caved ore from one end of the tramming floor, or the material removed in vertical slices from wall to wall, starting at the top and coming down, the back must be caught up well. If the stope has caved to a considerable height compared to its length, and the muck is coarse, so that it will not have a tendency to run, the probability is that it will prove less expensive if sliced vertically from top to bottom. If the cave has caught itself up with a long arch from end to end along the vein, and the ore is broken comparatively fine, so that it will run without difficulty, the best method is probably to draw the material on a long receding slope,

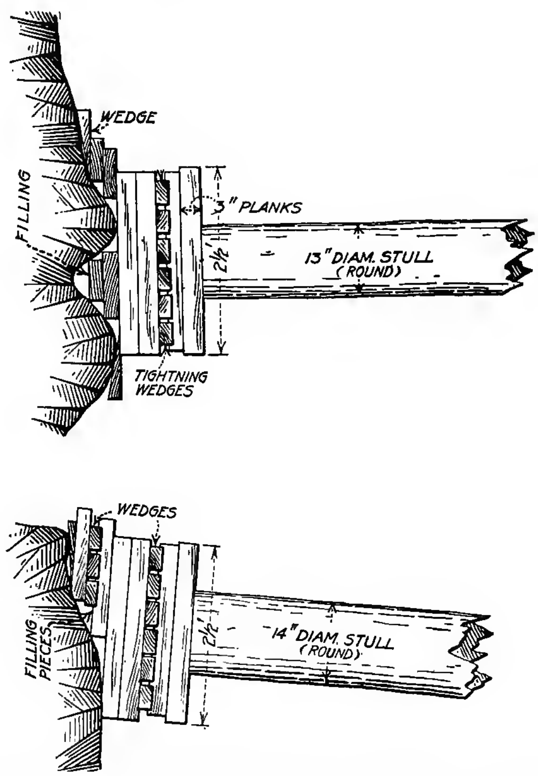


FIG. 22. STULLS ARE CUSHIONED AT BOTH ENDS BY 3-IN. PLANK TWO TO FOUR LAYERS DEEP

putting in the stull sets above as fast as proper wall space becomes exposed.

In order to gain access to one of these caves it is often necessary to stope up alongside one end until a shot breaks into the opening at the top. Usually it is better to do this only at one end, and to leave the mining ore overhanging the other end until all the caved ore has been removed and the cave has been filled completely. Then, after the filling has settled, the sloping back at the far end of the caved stope is worked out at the time that part of the lode is stoped. This, however, is a matter depending largely upon local conditions in the stope itself, and not infrequently the back at the far end of the caved stope is taken out at the same time that the arch at the other end of the cave is squared up.

Generally, once work has started in the caved stope, blasting in and near the cave is avoided as much as possible until after the stope has been refilled with waste. Whenever it becomes necessary to blast a boulder, it is holed half way through with a plugger drill, and then loaded with only sufficient dynamite to split it open, shaking the pile as little as possible. For, as the caved ore is quite likely to have many open spaces in it of considerable size, especially if many large slabs are scattered through it, a heavy blast might cause the pile to shift considerably, and that might easily prove disastrous. The work of recovering these caved stopes is not as dangerous, however, as might at first be thought for the ground is securely timbered and the muck pile kept under close control from sudden runs. In fact, during the whole life of the Hecla mine, only one man has been killed in a caved stope, and comparatively few men have been injured. Moreover, this man was killed years ago, before the present method of reopening caved workings had been systematized.

There is usually some ore left in the walls of caved stopes. Generally this ore is not mined until the cave has been completely caught up. Then, when everything has been made secure, the ore is blasted out, drilling it either with stoper drills or plugger, and working in new timbers, if necessary, when the width between walls requires it. Sometimes, however, the ore that is found in the walls is shot out in the ordinary course of working the cave. But this is done only when the amount of ore in the wall is so small that it would not pay to come back after it. In a cave everything depends upon conditions. If the walls are strong and the back is giving little if any trouble, considerable blasting can be done in the cave without any danger. But if the walls and back are weak, no chances are taken, and little blasting is done either while the ore is being got out or after the cave has been completely retimbered, for it is much cheaper to leave a little ore in the walls than to run risk of the stope caving again.

Recovering Caved Stopes in Narrow Veins
Part II

BY C. T. RICE

FIG. 23. TWO 16-FT. SPLICED STULLS FOR THE NEXT SET AHEAD IN THE CAVED PORTION OF A STOPE ARE TEMPORARILY SUPPORTED ON STRINGERS EXTENDED FROM UNDER THE CAPS OF THE SETS IN PLACE

At the Hecla mine a year or two ago a stope, approximately 600 ft. long and five floors, or 45 ft., high, in one of the veins on the 1200 ft. level, suddenly began to take weight and to give trouble. This was not surprising, for that part of the stope had been open to a height of five floors for several months, as the mine had been short of waste filling for some time. The mining of the top floor of this stope had been permitted to lag greatly, owing to the fact that this working was in ground that appeared to be well able to wait indefinitely for filling.

The back of ore was strong enough; it was the walls that had begun to give trouble, for the span of five floors, or 45 ft., as shown in Fig. 24, had proved too much for them, and they were suddenly showing signs of great weight. The stulls of the lower part of the stope began to snap and pop; especially those upon the middle floor. Something had to be done immediately, or the stope would surely be lost.

With 600 ft. of stope threatening to cave, it was a formidable problem. Yet the loss of the stope was averted in a simple way, as the men who had the condition to contend with understood what was really happening. Large stulls, many of them 24 in. in diameter, were put in at 5-ft. centers on the middle floor of the stope, which varied in width from 7 to 14 ft. Nothing else was done; yet these stulls prevented the stope from caving and held the working open until it could be filled.

The reason that this line of stulls held the stope open, even after the weight from the walls had begun to break fairly large stulls, is indicated by the fact that these new stulls were almost immediately driven into their headboards several inches. In other words, it was the initial creep of the walls, as they tried to arch themselves across this wide span, that was causing the timbers to fail. The walls, therefore, were prevented from coming in not so much by the size of the stulls that were used as by the especially thick headboards — five 3-in. planks at each end — that were put in between the stulls and the ground to act as cushions. With 30 in. of crossgrain timber, exclusive of wedges and filling pieces, between the walls and the stulls, sufficient cushioning timber had been provided to take care of the creep of the ground until arches of equilibrium had been established, in both walls, across the span of five floors.

Providing for the initial creep of the ground is a most important point in the timbering of any working, and it is because many mining men do not have a clear conception of what really happens when ground begins to take weight that hundreds of posts, stulls and caps are broken in mines each year that should not be broken and would not have been if put in with sufficient provision for taking care of the initial creep of the ground. It therefore may not be amiss to digress slightly in order to explain just what occurs when ground takes weight, and precisely what did happen in this large Hecla stope when the walls began to cave in, as the whole subject is rather intimately entwined with the matter of caved stopes and their occurrence.

It seems extraordinary, especially after the weight of the walls had begun to break stulls 18 in. in diameter and only from 7 to 14 ft. long, that a stope 45 ft. high and 600 ft. long which was threatening to cave could be saved simply by putting in a series of large stulls at 5-ft. centers along the middle floor. Yet to one who understands just what happens when ground takes weight it is not surprising that such a simple reinforcement was sufficient to hold the stope. The reason why this line of stulls, which was put in with exceedingly thick headboards on the middle floor of this stope, prevented the working from caving, even after the weight had become so great that it was breaking stulls 18 in. in diameter (and ground is pretty heavy that will break such stulls when they are only 7 to 14 ft. long), was simply that the 30 in. of crossgrain timber in the headboards of the new stulls was amply thick enough to continue to yield until the initial creep of the walls had stopped. When that was over, these new stulls, 2 ft. in diameter and placed at 5-ft. centers, were strong enough to hold in place the ground that had not keyed itself up.

If these stulls had failed, the ground that had not been able to key itself up into position in the walls would have come in. The footings from which the "dry" arches or sub-arches of self-support were springing would have failed. The ground that had keyed itself in place would then have come in. The failure of the footings of the arches would have increased the space of the major arch, and that would have caused the solid rock of the walls again to become beams, and they would have had to slab off under tension and shearing stresses until they had again assumed the arched shape proper for carrying the weight entirely in compression across the increased span. By that time the working would have been filled with broken material from the walls, and probably the span across the top of the opening would have been increased so greatly that the back itself would have begun to cave. The stope would have been filled with broken material from the back and walls, and the working would finally have been lost in spite of everything that could be done. That is the mechanics of caves.

Sometimes it is walls, at other times it is the back, occasionally it is both the walls and the back that give the trouble. It all depends upon the conditions and the strength of the ground. But practically all caves, if the truth were known, could be traced back to the fact that when the working was first opened up, and also when it was subsequently retimbered, the sets or posts or stulls were put in without any provision for keeping the weight of the ground from becoming excessive on the main members of the sets during initial creep. The timbers have consequently failed much sooner than they should have, and so the working has had to be retimbered much oftener than it should have been, ground has had to be taken out at each retimbering, and this has made the working larger and larger, and therefore even harder to hold. Finally, as the result of this improper handling of the timbering of the opening, the ground in the vicinity has got in motion, and the working has probably caved.

The first caved stope the reopening of which will be discussed in detail is that which occurred between No. 4 and 6 chutes on the 1200-ft. level in the main Hecla orebody. As shown in Fig. 25, this cave was 60 ft. long and did not catch itself up until it was about 38 ft. above the tramming track of the slice that was being mined. Throughout most of its length this cave had a width of 24 ft. In caving, the back had come down mainly in large slabs, so that the pile of caved ore showed little tendency to run, even in rather steep faces. Consequently the ore was taken out in vertical slices from the top down, supporting the walls with spliced timbers as fast as sufficient room was made for a stull to be positioned properly with respect to the sets in the uncaved part of the stope.

At the time that the cave occurred, the back had been stoped up 10 floors, or 90 ft., above the 1200-ft. level. The customary horizontal slice three floors high had been taken out across the top of the stope, the stope tramming tracks had been raised from the sixth to the ninth floor, crossboard lagging had been put in on the floor above, and the stope tightly filled with waste below the ninth floor in preparation for resumption of stoping, when suddenly the back for 60 ft. along the vein began to take weight. So little time remained for putting in reinforcing timbers that only the tramming floor could be attended to before the stope caved. But fortunately the stringers put in on both sides of the track and the helper posts stood on them to catch up the stull caps of the crossboard floor had proved sufficient reinforcement to hold the tramming floor open, although the rest of the stope closed in completely.

As soon as the back had stopped caving and the stope had become quiet, reopening of the working began. Beginning on the crossboard floor in the uncaved part of the stope, and two sets back from the end of the cave that was nearest to a chute, the back was stoped up a floor and advanced to the edge of the cave, timbering the new floor securely with stull sets as the face was carried ahead. As the end of the cave was still going up almost vertically, the stope was again raised a floor two sets back from the face, advanced to the edge of the cave, and timbered securely. On this floor the stope broke into the open space above the pile of caved ore, and as the back over the cave was only three or four feet higher than the back over the stope, it was decided to begin to work in from this floor in catching up the cave.

As the caved stope was 24 ft. wide, even right under where the back had caught itself up, spliced stulls had to be used in making the back secure. In order to facilitate the handling of these catching-up stulls, stringers 16 ft. long and extending 6 ft. out into the cave were put in on the top floor of the stope. These stringers were securely carried at the stope end by being blocked down tightly upon false posts from the stull cap above, being held in position the while by means of collar cleats from the stull cap, as shown in Fig. 23. At their outer ends the stringers were blocked up to the proper height from the pile of caved ore. Four stringers were put in, so that there was a stringer under each end of the two stulls that were to be butted together. One of these stulls was 16 ft. long — the greatest length of timber that could be got into the stope conveniently. The other was 8 ft. long.

As soon as they had been lifted up on the stringers, the two stulls were toenailed together with 120-penny spikes. This having been done, the spliced stull was rolled out into proper position at 5-ft. centers from the last stull cap in the stope. Headboards of 3-in. planks, three deep and 5 ft. long, were put in at each end, and the spliced stull was then securely blocked in place. In blocking all the stulls in these caves, the bottom wedges were driven in much further than the upper ones, so as to bend the headboards out considerably at the bottom. Then, in case the top weight should come again on the stope before side weight had nipped the stulls securely into their headboards, the timbers would be much less likely to give way.

As soon as the first catching-up stull had been got in, and the back caught up securely from it, the miners began to carry down an approximately vertical slice across the end of the pile of caved ore, so as to make room for a line of stulls from top to bottom of the cave. The ore from the slice was allowed to run down to the crossboard floor, where it was drawn into a car, with a minimum of shoveling, and then run to the nearest chute.

In carrying down the slice, little trouble was had in keeping the face almost vertical, owing to the boulders that were scattered through the pile of caved ore. Occasionally one of these boulders would give trouble, and then it would be drilled half-way through with a Jackhamer drill and blasted with just enough dynamite to split it open, shaking the pile as little as possible. But if the builder could be skidded down to the crossboard floor without much difficulty, this would be done before the boulder was block-holed.

As soon as enough wall was exposed to permit of positioning a spliced stull properly at 5-ft. centers horizontally and 9-ft. centers vertically from the other timbering in the stope, four stringers would be put in on that floor, with their outer ends carried from the pile of caved ore. Then the spliced stull, after having been toenailed together, would be rolled into position on these stringers and securely blocked in place, just as in the case of the catching-up stull, except that the headboard planks of the lower stulls were only 2½ ft. long — the length of headboard used in the stull caps of the stope. As soon as a stull had been got in below one above, not only would the two be braced from the stull caps adjoining, but posts would also be put in between the lower and the upper stull, so as to tie them together into a stull set and make the timbering in the cave more secure.

In this manner the vertical slice was worked down until the crossboard floor was reached. Then, as the place of the ore had been taken by a series of stull sets which were tied to the timbering of the uncaved part of the stope adjoining, and the walls were securely supported, the miners went again to the top of the cave, moved the stringers ahead a set, again carrying their outer ends on blocks from the pile of caved ore, and put in another catching-up stull, just as before. As soon as the back had been securely blocked up from this catching-up stull, which, of course, had 5-ft. headboards, as did the first catching-up stull, the miners would begin to carry down another vertical slice across the end of the cave, timbering it as fast as they took out the ore. In this way the pile of caved ore was worked out in a series of vertical slices.

As soon as all the ore had been got out of the cave, the uncaved part of the stope on the end at which the reopening of the cave had begun was stoped across to the nearest waste raise, the 13th floor mined across the

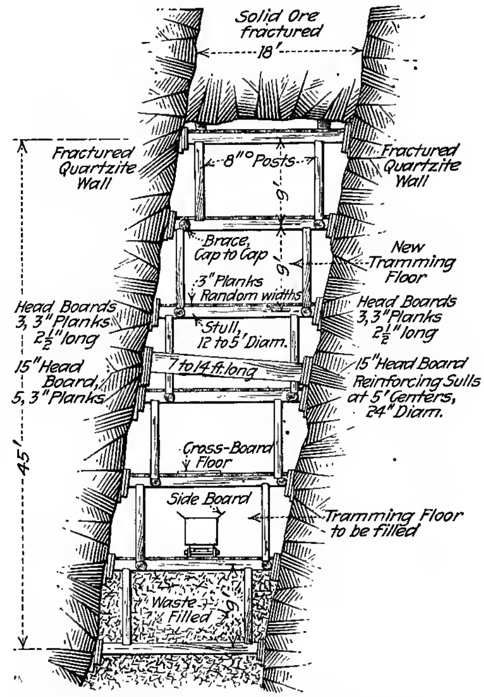


FIG. 24. CAVING WAS AVERTED BY QUICK ACTION IN PUTTING IN HEAVY STULLS WITH THICK HEADBOARDS ALONG THE CENTER OF THE STOPE

stope, and the back over the far end of the cave squared up. Then, after the far end of the cave had been lagged so as to keep the waste filling from becoming mixed with the ore when mining of the block next that end of the stope began, the tramming track of the stope was raised to the 12th floor, the crossboard lagging was put in on the 13th, the stope filled tightly with waste up to the new tramming floor, and everything made ready for the resumption of stoping, just as though there never had been a cave.

The stope that caved between Nos. 10 and 12 chutes of the 1400-ft. level had to be reopened in a somewhat different manner, as the ore had broken small in caving, and so ran easily. The vein had been stoped to a width of about 20 ft. between these chutes, and the working carried up seven sets above the level. The tramming tracks of the stope had just been raised to the sixth floor, the crossboard lagging put in on the floor above, and the stope below filled, when the back began to cave for a length of almost 100 ft. along the vein. Stringers and helper posts were rushed in on the tramming floor, and these proved sufficient to hold that floor open. But most of the stope above was stripped of timbers, and the stull sets at the edge of the cave were smashed down at the center, even on the crossboard floor.

This cave, as shown in Fig. 26, was more than 90 ft. long and had eaten up for more than 50 ft. above the tramming track of the stope before it stopped. Throughout most of this height the cave was 20 ft. wide, but at the top the vein was only 7 or 8 ft. wide. The ore had broken fine in caving and had run down at the ends at a fairly low angle of repose. As the stope had arched itself in the plane of the vein, and an open space from 5 to 7 ft. high had been left between the back and the top of the caved pile, it was easy to enter the cave from either end after the ground had become quiet.

Stulls with 5-ft. headboards were put in at the top to catch up the back, starting at one end and working ahead and up the cave. Generally the headboards of these catching-up stulls were put in three 3-in. planks deep at each end and planks 5 ft. long were always used as headboards on the catching-up stulls. When these catching-up stulls could not be got in near enough to the back to block the ground up directly from them, sprags, and even small false sets, were stood on top of them to catch up the ground. In places where the stulls, owing to the shape of the walls, had to be placed rather far apart, bad ground between was also caught up by lacing old planks across from stull to stull.

stope. Headboards $2\frac{1}{2}$ ft. long and of the usual thickness were put in at each end between the spliced stull and the ground, and, in order to enable these stulls to carry top weight better, they were put in a little high at the splices.

As fast as the stulls were got in they were braced sideways from the nearest stull on the same floor both at their ends and at the splice, and as soon as a stull had been put in under one above, posts were stood under the splices as well as under the ends of the stulls, so as to tie the timbering together securely into stull sets, and these posts, in their turn, were also braced sideways from one another, so that if a boulder should roll down the stope it would not knock out a post.

As rapidly as the pile of caved ore was worked back at the bottom, and stull caps that had been crushed down at the center were uncovered on the crossboard floor, new spliced stulls were put in over them, blocking the splice up from the crushed-down stull at the same time that posts were put in both at the ends and over the splice to tie the new stulls to the stull cap above so as to form new stull sets. By the time five or six of these new spliced

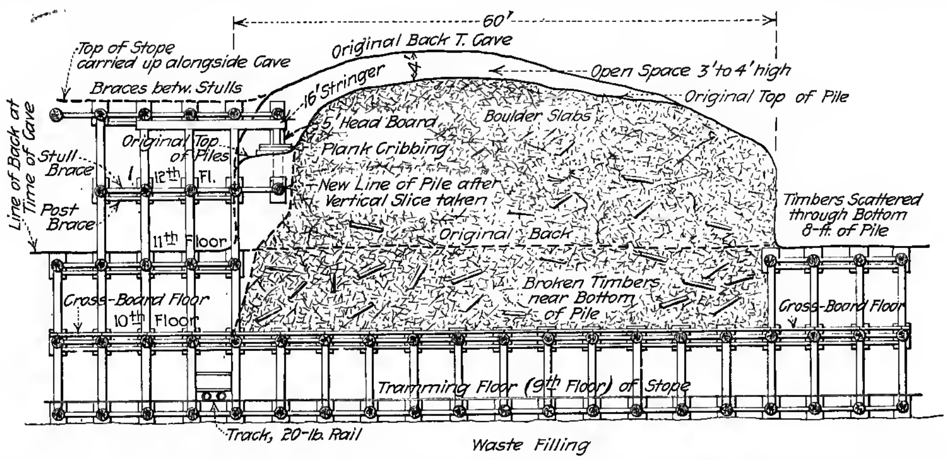


Fig. 25. Longitudinal Section of Stope Above 1200-ft. Level That Caved Between Chutes Nos. 4 and 6, 60 Ft. Long, 38 Ft. High and 24 Ft. Wide

When the top of this cave was reached, an Ingersoll-Rand tugger hoist was mounted on a 3-in. cross-bar and used for "snaking" stulls and other timbers up the pile, for as soon as the near side had been caught up securely, catching-up stulls were also put in down the other side of the cave and the back at that end also made safe.

As soon as the back had been securely caught up and the sides trimmed down carefully, the miners began at the end nearest a chute to run the ore in the pile down at its angle of repose through the crossboards into a car, and then trammed it to the nearest chute practically without any shoveling. In this way the pile of caved ore was removed, and the walls were supported with spliced stulls as soon as enough wall was exposed for placing these permanent stulls at 5-ft. centers horizontally, and 9-ft. centers vertically, from the timbering in the uncaved part of the stope adjoining.

The stope being 20 ft. wide, these stulls had to be spliced. First a stull 16 ft. long was blocked up into proper position on the pile of caved ore, and then a butt-block stull 4 ft. long was put in to splice it out across the

stull caps had been got in on the crossboard floor, the walls had pinced the stull caps on the different floors above sufficiently into their headboards for the old crushed-down stull caps of the crossboard floor to be taken out with perfect safety. Of course, as soon as the crushed-down stull caps, some of which were 18 in. low at the center, were taken out, posts were stood under the new caps, both at their ends and under the splices. This leaving of the crushed-down stull caps in until the ground had nipped the spliced stull sets above securely into their headboards was much better practice than to have taken out the crushed-down stull caps as fast as they were uncovered.

In this way, as fast as the ore was drawn down at its angle of repose into cars on the tramming track of the stope, and taken to the nearest chute to be sent to the level, new stull caps were put in to support the walls, and then as soon as they could be, these stull caps were tied to those above, and to the side, with posts and braces to form a series of stull sets.

When all the ore had been got out of the cave, and the working had been timbered securely, stoping was resumed at the end that was nearest to a waste raise, and the different floors were carried through to this waste raise and manway. Often, when that had been done, the tramming track of the stope was raised to next to the top floor, the crossboard lagging put in on the top floor, and the floors below the tramming track were filled with waste in preparation for resumption of stoping.

Before filling the stope, the little ore that had been exposed in the walls by the slabbing resulting from the caving of the working was drilled, either with a stoper of a Jackhamer, and shot out carefully with a minimum of dynamite. Generally so little ground had to be taken out in mining this ore that blocking could be used to fill in behind the headboards of the stull sets, but occasionally so much ground had to be shot out in getting the ore that a new butt stull had to be put in. Fortunately, owing to the way in which both the stull caps and the posts are braced from one another in the Hecla stull-set system of timbering, there was little possibility of the timbers collapsing when the ground was shot out, and it is just to prevent that possibility that both the posts and the stull caps are so securely tied to one another by collar braces. Of course, the ore could have been shot out of the walls while working the ore out of the cave, but waiting until the stope is all caught up and everything is ready for filling, before the ore is shot out of the walls, is much better practice, as there is then much less danger of starting the stope to caving again.

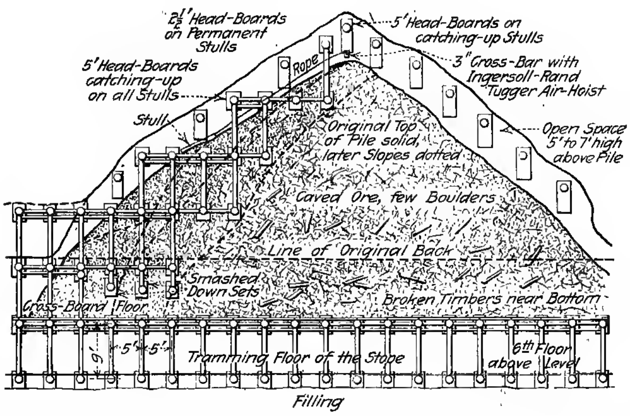


Fig. 26. Longitudinal Section of Stope Above the 1400-ft. Level That Caved for a Length of 90 Ft., Between Chutes Nos. 10 and 12. This Cave was 55 Ft. High and 20 Ft. Wide

Another cave, if such it may be called, that presented a somewhat different problem, occurred in the stope between Nos. 30 and 32 chutes on the 1400-ft. level. The stope had been carried up six floors, but as the mine was temporarily short of waste filling, the seventh floor was being worked out. At that time only drilling was being done by the miners, and the blasting of the holes was left for a special crew to do after the night shift had left the mine. As a result, often 100 holes would have to be blasted at a time in one stope. In this particular stope about 60 holes 7 ft. deep had been drilled in the back, and were ready for blasting. The ground, although breaking well, required considerable dynamite to break the holes to bottom, and the blasting crew loaded the holes a little heavier than usual. As a consequence the holes not only broke to bottom, but, going all at one time, they shook the back so badly that a block of ground about 6 ft. deeper than had been drilled fell out of the top of the stope. The back was thus several feet higher than the seventh floor, as can be seen by referring to Fig. 27, though, as there had been considerable muck on the different floors at the time the holes were blasted, the pile of broken ore in the stope came almost up to the fifth floor. This pile of broken ore gave excellent support to the walls and probably prevented a more serious cave, as this great mass of ore, falling all at one time, had stripped the stope of timbering down to the crossboard floor.

When the shift came on in the morning the bosses found a stope, 32 ft. wide, stripped of timbers to a height of about 55 ft. and for a distance of 30 ft. along the vein. Both back and walls appeared to be strong.

The plan adopted for catching up the back was to build a trussed structure of spliced stulls 16 ft. long across the top of the stope. Then, after the back had been caught up by bulkheads from this truss, the ore was to be drawn out of the stope, and waste filling run in without putting in additional timbers to hold the walls. In Figs. 28 and 29, which are sections through the top of this stope, the method that was used in building the truss is shown.

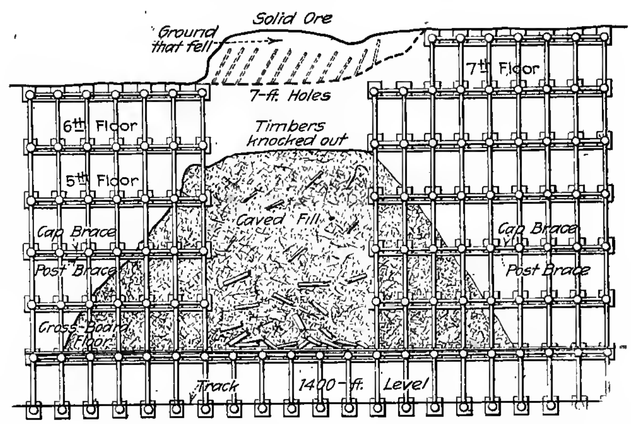


Fig. 27. Longitudinal Section of Stope Above 1400-ft. Level That Caved for a Length of 30 Ft., a Width of 32 Ft. and a Height of 50 Ft. Between Chutes Nos. 30 and 32

Putting in the truss to catch up the back was not such a simple operation as might appear from the drawings. Stringers were laid lengthwise with the stope. At the cave end they rested on the pile, but at the stope end they were carried by the top timbers. The stringers, which were 16 ft. long, were round timbers slabbed off on their top and under sides, put in to carry the stulls that were to be trussed together to support the roof. To enable the stulls to carry a considerable top weight, they were put in about 18 in. higher at the center than at the walls.

The four stringers which were to carry the bottom trusses of the structure were therefore put in so that the outer end of the stulls of the trusses came level with the stull caps of the stope, and the inner ends where the stulls butted against one another were raised about 18 in. higher than the outer or wall ends. This required that the stope ends of the stringers which were next the walls should be carried from false posts, so that they came just under the stull caps, and in order to keep them from moving while the stulls of the trusses were being rolled out on them, these stringers were wedged down securely on the posts from the stull cap above. The stope ends of the two stringers that were to carry the center or spliced ends of the trussed stulls were blocked up from the top of the stull cap, and posts were put in on top of them, so as also to wedge them securely in place, in order to prevent their moving while the trusses were being positioned upon them.

As soon as the stringers had been blocked securely into position at their stope ends, and their outer ends had been blocked up from the pile of caved ore to the proper heights, three stulls were rolled out upon each pair of stringers and positioned at 5-ft. centers from one another and the stull caps of the stope. These stulls, 16 ft. long, were butted against one another and then made ready for blocking in place by putting headboards made up of 3-in planks, three deep and 5 ft. long, between them and the walls. But until the whole structure was in place, the wedges of the headboards were driven just tight enough to keep the timbers from moving out of position, as if they were driven tighter the stulls would ride on one another at the center.

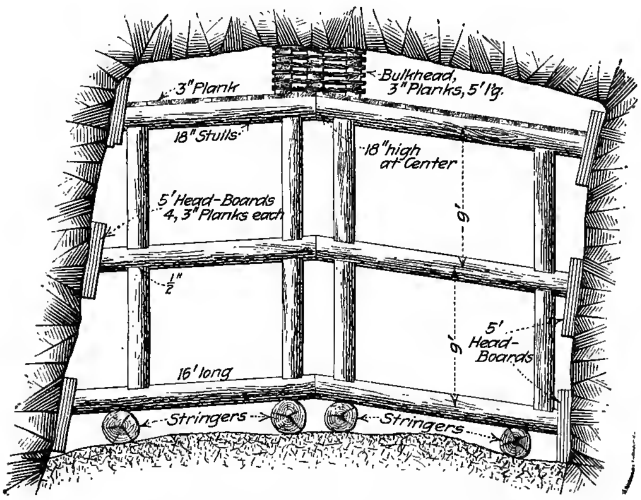


Fig. 28. The Back Is Caught Up by a Trusswork of Spliced Stulls Supported on Stringers Below and by Deep Banks at Headboards at Their Ends

As soon as these three pairs of trussed stulls had been put in, posts were stood on them in daps cut about ½ in. deep, a post being stood at each end of each of the stulls of the three trusses. Then, on top of these posts, stulls with daps cut in them at the proper points were placed so as to form a second series of trusses across the stope, with headboards 5 ft. long put in between them and the walls, and the wedges again driven just tight enough to hold the trusses in position. As soon as these upper trusses were in, both the stulls and the posts were braced girtways from one another and from the timbering of the stope, so as to prevent the trusses from swinging sideways. This having been done, posts were again stood upon these stulls in properly positioned daps, and stulls for another series of trusses lifted on top of them. Headboards, 5-ft. long and made up of 3-in. planks three deep, were put in between them and the walls, with the wedges again driven just tight enough to hold the trusses in position. In this way, as shown in Fig. 28 and 29, a trussed structure three sets long and three sets high had been erected across the top of the stope. But before the structure could be blocked into place, a cribbed bulkhead had to be built along the top of the structure to catch up the back and keep the stulls from riding on top of one another at the splices when the structure was blocked from the walls.

The top trusses were therefore lagged with 3-in. planks, and a cribbed bulkhead was built along the ridge of the truss to catch up the back as well as to hold the trusses down and prevent the spliced stulls from riding on one another when they were being blocked in

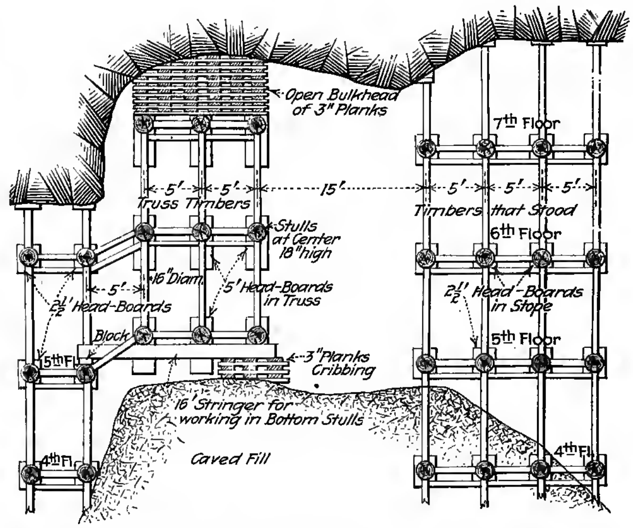


Fig. 29. At the Top of the Caved Fill the Trusswork of Spliced Stulls Is Supported on Cribbed Plank, Which Also Catch up the Back Along the Truss Ridge

place. Three-inch planks were used in building this bulkhead, because they could be put in place much more easily and cheaply than larger timbers, even if many more of them had to be used, and because, when stoping was resumed, the planks could be used for flooring in the stope. These planks were 5 ft. long, and three planks were used to each course of the bulkhead, one at each end and one at the center. One of these cribbed bulkheads was built at each end of the structure, butting the planks of each bulkhead against the planks of the other over the center truss.

When everything was ready, men were stationed at different points in the trussed structure so that the wedges in the bulkheads and the various headboards could all be driven home at the same time, as otherwise the stulls of the trusses would begin to ride on one another while the wedges in the different headboards were being driven, and the whole structure would become distorted, and would then have lost much of its strength. A man was stationed on each side of the two bulkheads that had been erected on top of the trusses, and a man placed at each of the headboards of each of the three trusses upon the three floors of the structures, so that it required 11 men to drive wedges and 1 man to watch while the timbers were being blocked in place. Then while the wedges in the bulkheads and in the various headboards were slowly and carefully driven home, the man in charge of the work watched the various trusses closely so as to keep any of the trusses from being tightened faster than the others. In this way the several trusses were seated firmly against their headboards without developing tendency to ride at the splices, and the back over the structure was keyed into position.

As soon as this first trussed structure had been completed, another was built, two sets wide, to catch up the rest of the back over the top of the cave, and to tie the first structure to the timbering of the uncaved part of the stope on the other side. Then, when a bulkhead had been built on top of the second truss, and the structure securely wedged in place by driving the wedges of the different headboards and the bulkhead home at the same time, the back was carefully picked down and caught up with sprags and blocking from these trusses wherever it was weak.

When the back had been made secure in this manner drawing of the ore from under these trusses through the crossboards into cars on the stope track below began. As the ore was withdrawn, close watch was kept of the trusses to see how they were taking the weight, and the walls were carefully picked down as fast as they were exposed. In this way the ore was drawn out of the stope without accident and with comparatively little shoveling.

As soon as all the ore had been got out of the stope, the tramming track was raised to the fifth floor of the stope, waste brought in from the nearest waste raise, and the stope below the fifth floor tightly filled. Then, when crossboards had been put in on the sixth floor, mining of the seventh floor was resumed.

Recovering Caved Stopes in Narrow Veins
Part III

BY C. T. RICE

FIG. 30. VIEW OF A WORKING OPPOSITE CAVED STOPE IN THE HECLA MINE

The most difficult problem in handling caved ground which has as yet presented itself at the Hecla mine was that of reopening the stope which caved several years ago between Nos. 4 and 6 chutes of the 900-ft. level. This stope had been carried up a few sets above the tramming tracks on the 14th floor, when the back for a distance of about 90 ft. along the vein suddenly gave way and took out everything before it down to the tramming floor. The stope was 30 ft. wide, although at the time practically all the ore was confined to a 10-ft. strip next to the hanging wall. The ground, furthermore, was broken up by a number of minor slickensides. As a result, the stope continued to cave until the back had reached a point about 75 ft. above the 14th floor, as shown in Fig. 31.

A trench cut across the caved material from wall to wall showed that in this ground, before the accident occurred, the ore had been confined almost entirely to the 10 ft. next the hanging wall, as was the case lower down, and that, in caving, little of it had become mixed with barren vein filling. In no place throughout the other 20 ft. of the pile was it thought that there was enough ore to make its recovery worth while. The real problem, however, in reopening this stope was not to get the ore, but to recover, at a reasonable cost, No. 5 cribbed chute, which was buried under the center of the pile of barren vein filling, which was needed in order to mine the ore above the cave economically.

The plan adopted in reopening the working was to catch the back up securely over the caved area, and then, beginning at the top, to remove the ore by trenching along the hanging wall, timbering this opening with drifts sets and using spiling to hold back the barren quartzite on the foot-wall side of the cave. Then, when this hanging-wall trench was even with the top of No. 5 chute, a trench 15 ft. wide was cut from wall to wall across the pile. Spliced stulls, slightly trussed and tied together into stull sets, were used to hold this central trench open, the broken vein matter along the sides being kept from moving by driving spiling down until new stull sets could be put in below. Thus proceeding carefully, a floor at a time, these two trenches were carried down through the caved vein, filling a distance of eight floors, or approximately 70 ft., without setting the pile in motion and without serious accident.

A large open space had been left at the top of the cave, and as the stope had been worked two floors higher at the west than at the east end, it was possible to get into the cave at the former end, to study conditions. It was seen that the back was much flatter at the east than at the west end, and so, although it necessitated carrying the stope two floors higher in order to bring it even with the top of the cave, it was decided to reopen the working by trenching in from the east end. Consequently, the ore between No. 6 chute and the cave, and the much longer block of ground between it and the waste raise at the east end of the working, were stoped out even with the top of the cave. Stull sets were used to hold the ground open. But as many large slabs and boulders were scattered through the caved material, the end of the stope next to the cave did not have to be lagged to hold back the loose filling as the stope was carried up.

It was necessary to keep open the ground between No. 6 chute and the cave, in order to use that chute in reopening the cave, and fortunately this opening was only a few sets long. But the last part of the stope between No. 6 chute and the waste raise was lagged off and then filled tightly with waste to within two floors of the back, so that no more ground would be open in the vicinity of the cave than was absolutely necessary. The waste were being worked into position and blocked, for the stulls which were spliced together to form the trusses were green timbers 15 ft. long and from 14 to 18 in. in diameter, and so were heavy. Moreover, by carrying the stulls in this way, it was certain that they would be put in properly and at an even height — an advantage when stoping was resumed in the ground above.

The full importance of trussing spliced stulls at a greater angle was not understood at the Hecla mine at that time, and so the center stringers were put in only about 12 in. higher than those along the walls. These stringers were round timbers, 16 ft. long and slabbed off on their upper and under sides to 8-in. faces. Four of them were placed in the stope, one near each end of the stulls that were to be trussed together. These catching-up stulls were put in beginning at the east end

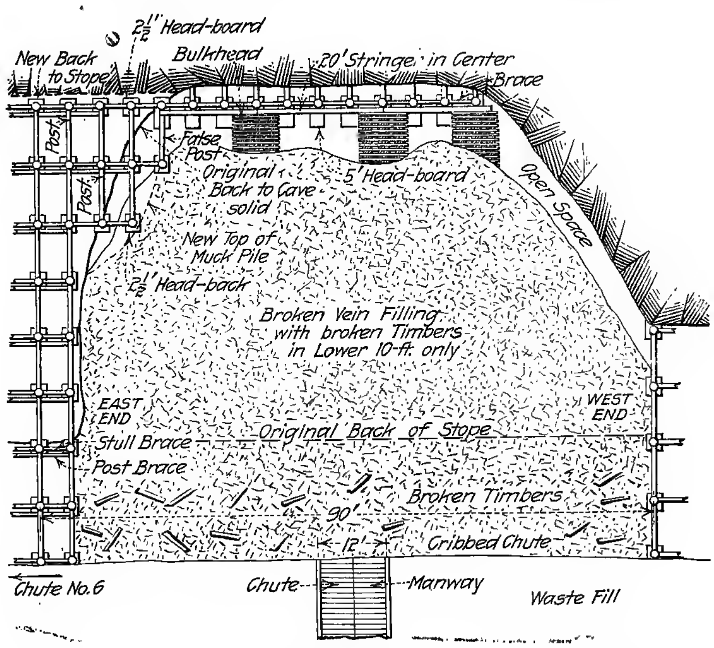


FIG. 31. LONGITUDINAL SECTION OF STOPE THAT CAVED ABOVE THE 900-FT. LEVEL, BETWEEN CAVES NOS. 4 AND 6, FOR A HEIGHT OF 75 FEET

for filling this part of the stope was trammed from the waste raise at its east end, and later the tracks were extended into the cave when it became necessary to fill the central and hanging-wall trenches.

The first thing to be done in the cave proper was to catch up the back temporarily with sprags and posts, until the rather heavy flat part at the east end could be timbered securely with trussed stulls. These sprags and posts were stood on top of the pile with head- and foot-boards. Then, when the back had been picked down carefully, the work of placing the spliced stulls under the flatter part of the back began. As the west end of the cave had arched itself up rather securely, these stulls were necessary only a little more than half way.

Even under the flatter part of the back, the open space over the pile of caved material varied greatly in height, sometimes 10 to 12 ft. and again only 3 or 4 ft., according to the way in which the boulders and large slabs were scattered. Because of this irregularity, it was thought best to carry the stulls on stringers while they of the stope. In working ahead, the temporary sprags and posts were removed when necessary to get the stulls in properly.

The eastern ends of the first stringers were carried on false posts from the stull caps of the top floor of the east stope, and were wedged down on these posts securely from the stull cap above, so that they would not be displaced while the new stulls were being worked into position on them. The outer or cave ends of these four stringers were carried at their proper height by cribs from the pile. These cribs were built of 3-in. planks 5 ft. long, with three planks put in to a course, and were positioned, as shown in Figs. 31 and 32, so that they would catch the inner ends of the next stringers as well as the outer ends of the first ones. When the center of the cave was reached, care was taken to place the cribs so that they would not interfere with the sinking of the trench that was to be carried down to uncover No. 5 cribbed chute.

As soon as the first four stringers were ready, three pairs of stulls were positioned on them at 5-ft. centers from the stull sets of the stope. Headboards three deep, of 3-in. planks, 5 ft. long, were then placed between their ends and the walls, and sufficient blocking was put in over the splices to catch up the back. Spreaders were put in to keep the stulls from swinging sideways, and the several trusses finally wedged tightly in place by driving home the wedges in the headboards.

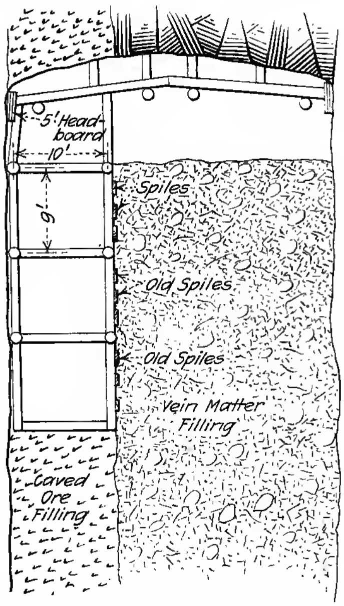


Fig. 32. Cross Section of Caved Stope Showing Stull Support Under Back and Three Floors of Spiled Drift

As soon as the back had been caught up securely from these first three trusses, cribs were built to carry the next four stringers, three other pairs of trussed stulls were put in, and the back was again securely caught up from them. Thus, working ahead three stulls at a time, the back was caught up for a distance of about 50 ft. over the east end of the cave. At that point the top of the cave began to arch itself both crossways and longitudinally along the vein, so that only an occasional post or sprag was required to make it perfectly safe. In putting in the cribs on the hanging-wall side of the cave, care was taken to place them in such a way that they could be caught up easily from below when the ore along that side was being trenched out, for it was highly important to keep these cribs in until the top weight had seated the stulls and their headboards securely against the rough walls of the cave, as until then most of the top weight coming on the trusses had to be carried by the stringers.

Occasionally the back ran up so high that short posts, or even false sets, had to be placed on top of these catching-up trusses to make the back secure. But generally that was unnecessary, as the back of the cave, although 30 ft. wide and rather flat at the east end, had caved until it was in fairly strong ground.

Starting at the top and east end of the pile, a trench was cut along the hanging wall, wide enough to get out all the ore. This was then timbered with drift sets having 10-ft. caps and 8-ft. posts, put in at 5-ft. centers. Mud sills were placed under these sets, which later became the caps of the drift sets on the next floor below. In cutting this trench, spiling had to be driven on the cave side, as already stated, so as to hold back the barren vein filling. But as many large slabs were scattered through the pile, and the quartzite of the vein had, as a whole, broken into rather coarse pieces, the splices, which were nothing but 3-in. planks, 9 ft. long, that had been beveled somewhat at one end, did not have to be placed close together, and generally were driven from 12 to 18 in. apart. Bridge pieces, therefore, were unnecessary for catching the ends of the spiles, when the set ahead was put in, as sufficient room was always left between them for starting new spiles, even when the weight had forced the spiling in tightly against the posts.

As soon as the hanging-wall trench, which was 9 ft., or a floor, deep, had advanced far enough, the trench 15 ft. wide, previously mentioned, for uncovering No. 5 cribbed chute, was started across the pile and carried 9 ft. deep from wall to wall. The drift sets were placed in the hanging-wall trench as fast as it was advanced. In making room for the sets, no trouble was caused by ore running in the face, and breast-boards were found unnecessary, as the ore had broken rather coarsely, and would not run enough to undermine more than one mud sill at a time. As the sills of the different sets were securely tied together by means of braces when they were put in, the catching up of the timbers on the floor above was comparatively easy, the only thing needful being to put in two angle braces from the last mud sill on the lower floor to carry the mud sill of the set ahead on the floor above before it was undermined. These angle braces were placed so as to come just inside the posts that were to be put under the sill, converting it into a cap. Then, as soon as the ground had been worked out from under the set above, two posts and a mud sill would be positioned properly under it, braces put in to tie this mud sill to the mud sill of the last set, and the set finally blocked up tightly in place from beneath so as not to permit the timbers above to settle when the slice below was carried further ahead.

In the central trench, the caved vein filling had to be held back, as well as the walls kept open. This was done, as shown in Figs. 33 and 34 by means of stull sets and spiling driven down along their sides. As soon as the central trench had been dug deep enough to get in two floors of stull sets below the catching-up stulls above the trench, driving of the side spiling in the central trench was begun.

As the spliced stulls were put in at 9-ft. centers vertically from one another, longer and stronger spiling had to be used than in the hanging-wall trench. So they were made by beveling off at one end round timbers about 8 in. in diameter that had been ripped in half. These did not have to be placed closely together, owing to the general coarseness of the rock in the pile. Furthermore, as the caved vein filling had little tendency to run, they were not driven down level with the stulls of the floor below. Neither was it so hard to work these spiles in as might be thought, owing to the coarseness of the caved material, for round spiling drives more easily than flat spiling.

Some trouble was caused by boulders in driving the spiling, both in the hanging wall and in the central trench, but not much, except when one came in the way of the next set or stull, as the boulder, caught securely in the pile, would hold the loose ground back as well as spiling would. When the boulder could not be worked out of the pile without too much danger of causing the muck to run, or when so much of it was in the way of a timber that the projecting side or corner could not be sledged off (care always being taken to brace the boulder securely up into position before beginning to sledge it), recourse to explosives became necessary. A hole would be drilled into the boulder (or slab) so as to break off enough, with a minimum amount of powder, to make room for the timber. Great care was taken in such blasting not to use charges heavy enough to shake the pile badly, which might result in disaster. Dynamite, therefore, was never used in this cave unless it was absolutely unavoidable, as there were large holes or voids in the pile, due to the way the ground had broken. Heavy blasting would undoubtedly have caused serious shifting and possibly have crushed important timbers.

Nothing about the timbering of the central trench is specially to be noted. As soon as there was room enough for stulls at 9-ft. centers directly under those above, four pairs of spliced stulls were put in across the trench.

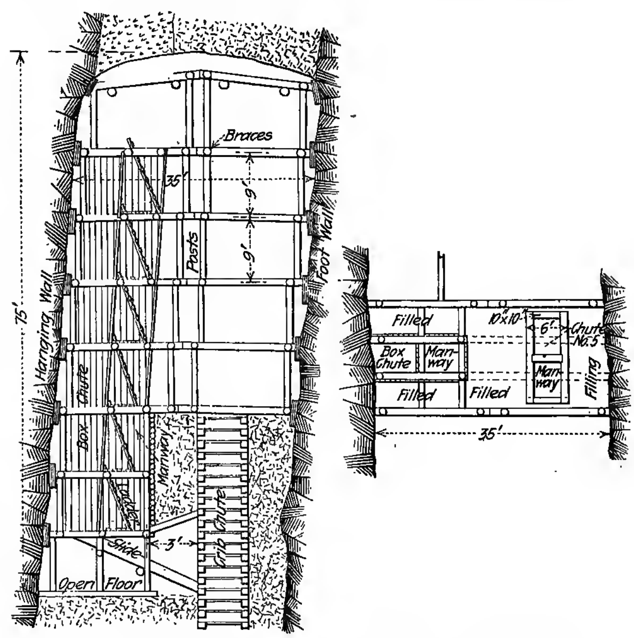


FIG. 33. SECTION AND PLAN OF CENTRAL TRENCH SHOWING METHOD OF CARRYING UP CRIBBED CHUTE AND FILLING

These were placed with their centers slightly higher than their wall ends, just as in the case of the catching-up stulls. But though 3-in. planks, three deep, were put in as headboards, the planks themselves were only $2\frac{1}{2}$ ft. long, as their main function was to act as cushioning rather than as friction pieces for tying the stulls to the walls, as in the case of the catching-up stulls. Posts fitting into shallow daps, both at top and bottom, were put in to tie the different spliced stulls together into stull sets, and the stulls and posts were braced sideways from one another on the different floors to take care of the side pressure of the filling, and to prevent "jack-knifing" of the timbering in the central trench. The spliced stulls in the trench were, of course, blocked into place by tightening the wedges in their headboards after the posts, one at each end of each stull, had been put in, for in that way the stulls of the lower truss, through these posts, were brought securely home against the truss above.

Thus, by first working the ore out along the hanging-wall, and then trenching across the pile, the workings being timbered as they were carried down, the ore was

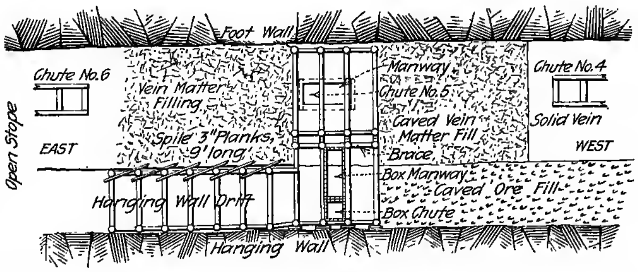


FIG. 34. PLAN OF CAVED STOPE SHOWING SPILED DRIFT AND CENTRAL TRENCH

worked out a floor at a time, and the central trench carried down until the bottom of the cave was reached. In this way the ore in the east end of the cave was removed and No. 5 chute finally uncovered.

A timbered working 10 ft. wide and 8 floors, or approximately 70 ft., high had now been opened along the hanging wall from No. 6 to No. 5 chute, a distance of about 50 ft., and through the center of the pile of caved vein matter was a trench 15 ft. wide, 30 ft. from wall to wall, and 70 ft. high, held open by stull sets put in slightly trussed. The cribbed chute that had been uncovered had to be carried up, and the opening along the hanging wall and the central trench had to be filled, before the ore along the hanging wall in the west part of the cave could be mined. For if an attempt were to be made to mine the ore in the west half of the cave without first filling the openings in the east half, so much ground would be open that the timbering along the hanging wall would probably begin to swing, and the chute which had taken so long to uncover would again be lost.

Raising No. 5 chute and filling the central trench was not such an easy undertaking as it might at first seem, for this chute, which measured 12 x 6 ft. overall, and was timbered with 10 x 10-in. cribbing, ran lengthwise with the vein, so that the two central stulls of the trench had to be taken out in order to raise the chute. Moreover, in order to mine the ore in the west end of the cave without first carrying a stope up at that end, it was necessary to carry a box chute and a manway through the filling along the west side of the trench near the hanging wall, as shown in Fig. 33. This box chute had to connect at the bottom through a slide with the chute compartment of the cribbed chute. Furthermore, this chute compartment was on the east or far end of the cribbed chute, and the manway and timber slide were on the near or west end. To carry such a slide from the box chute to the chute compartment of No. 5 cribbed chute necessitated that practically the entire hanging wall half of the bottom floor of the central trench be left open.

Obviously it would not do to take the two middle stulls out of this central trench, from top to botoom, in order to raise the chute all at one time. So after the slide had been put in to connect with the cribbed chute, and the stulls on the hanging-wall side on the bottom floor of the trench had been posted up and braced at the posts, the chute was raised, and the central trench filled, a floor at a time. At the same time that No. 5 chute was raised a floor, the whole cribbing was shifted over bodily a little toward the hanging wall, so that, when the top of the cave was reached, the chute was again under the ore, as it was highly desirable that, in mining the ground above the cave, the stope should not be carried any wider than the width of the ore along the hanging, unless drill holes showed that ore was again making toward the foot wall.

As soon as the slide on the bottom floor had been put in from the bottom of the box chute to the chute compartment of No. 5 chute, the work of preparing the central trench for filling began. Posts were placed under the two central stulls, about 3 ft. back from the line of No. 5 chute. Then braces were put in to tie these posts and the stulls to the side stulls and posts. After that had been done, the stulls were sawed off about 2½ ft. back from the line of the chute. By that time the two central stulls on the foot-wall side of the chute had also been taken out.

Then, while No. 5 chute was being raised to the stull caps of the floor above, with a hole 3-ft. square left in the side of the cribbing where the slide from the box chute joined it, a manway-and-chute compartment was started next the hanging wall and on the west side of the trench by putting in, even with the outer edge of the ore, braces from the outside to the central stull, and lagging them. Then, when the other side of this comparement had been lagged off, and the bottom floor had been covered over, everything was ready for dumping in the filling. Lagging the bottom floor of the trench was easy, except for the space left between No. 5 chute and the ends of the sawed-off stulls. This opening was covered over without possibility of future damage to the chute by putting in short lagging, leanto fashion, against the chute from the regular lagging carried by the stulls.

To aid in filling both the hanging wall and the central trench, the stope tracks were extended into the cave and carried across to the foot wall over the center of the chute trench. Then waste filling was run into the cave by car from the raise to the east.

In filling the cave the sole precaution taken was to dump only the finest filling around the cribbed chute, and that part of the trench was always filled first, so that no boulders would roll over against the cribbing. For when large boulders or timbers get against the side of a cribbed chute, there is danger that the pressure of the filling will finally drive them through the cribbing. Moreover, it was to prevent the stulls on the hanging-wall side of the trench from getting against the side of the chute later in the life of the stope that they were cut off from 2½ to 3 ft. back from the line of this chute.

As soon as the central trench and the opening at the east end along the hanging wall had been filled even with the top of No. 5 chute, the central stulls on the next floor would be taken out, the chute raised up to the next stulls, fine filling dumped in around the chute, and coarser filling further away, the hanging wall trench then filled, and the process repeated.

When the central and the hanging-wall trenches had been filled up to within two floors of the back, the mining of the ore along the hanging wall in the west part of the cave began. Starting at the top and from the box chute, this ore was trenched out, and the opening held open with drift sets having 10-ft. caps and 8-ft. legs, put in at 5-ft. centers, and with the barren vein filling on the cave side held back with spiling, just as had been done in getting out the ore at the other end of the cave, except that this time the ore was taken out through the box chute and slide to No. 5, instead of to No. 6 chute. When the ore had been worked down to within a floor of the old gob at the bottom of the cave, the slide was torn out and the ore wheeled directly to No. 5 chute.

When all the ore had in this way been sliced out along the hanging wall, the hole in the side of No. 5 chute was closed, the opening at the bottom of the cave as well as the trench along the hanging wall filled, and stoping began in the block of ore to the west of the cave. As that stope rose, the different floors were carried over to the edge of the cave. In that way the sloping back over the west end was squared up for resumption of normal stoping as soon as these floors and the west stope had been filled to within two floors of the back, tramming tracks for the new slice raised to that floor through the stope, and the crossboard lagging put it on the floor above. Thus, without a serious accident of any kind, the ore along the hanging wall was mined out, and the cribbed chute carried up through the center of a pile of caved vein filling 75 ft. high.

In this series of papers I have described methods of combating practically every condition encountered in reopening caved stopes in veins having walls strong enough to remain open when the back caves. I have shown how such caved stopes can be worked out with spliced stulls, even when the cave is 30 ft. or so wide — by putting in the stulls so as to obtain a trussing effect. I have also shown that when the walls are fairly strong the ore can be removed by catching up the back of the cave securely with a truss structure, and drawing the ore out from under this truss without other support. I have endeavored to demonstrate that if care is taken to keep the mass of caved vein matter from getting in motion, it is possible to trench down through the cave and take out only that part of the vein filing which is rich enough to pay for its recovery.

All the methods detailed were devised by John F. Allen, assistant foreman of the Hecla mine, and are therefore especially adapted to wall conditions as strong as those that generally characterize the Cœur d'Alene veins. But they have a wider application, and no doubt by working out small sections at a time ore can be recovered with equal safety from caved stopes in veins where the wall are considerably weaker, although still strong enough to hold when the back caves.

In another series of articles, to appear later, I shall show in detail how the ore can be removed cheaply and safely, working down from the top, when not only the back, but also the walls, of the caved stope have closed in completely. Moreover, in the projected series of articles I hope to prove that the ore in any caved stope is not only more safely but also more easily recovered by working down through the cave than by coming up through it.

Stoping Methods at the Franklin Mines

BY H. H. HODGKINSON

The orebody at Franklin, N. J., consists principally of a granular mixture of willemite, franklinite and zincite, the gangue being a highly crystallized white limestone. It has a maximum vertical depth of 1150 ft., dipping southeast 40 to 60 degrees.

The greater part of the ore is mined by means of shrinkage stopes 17 ft. wide, the orebody being so mapped out that there is a pillar of solid ore 35 ft. in width left in place between each stope slice. The stope slices are carried transversely across the orebody according to line (each stope is laid out on a coördinate and numbered, starting with a line through the old Parker Shaft as 00). Each stope varies in length according to the variation in the horizontal distance from foot to hanging wall, while the height is equal to the distance between levels, which is about 50 feet.

Formerly the ore was mined entirely by means of large drifts on each level driven longitudinally with respect to the orebody, there being as many as five of these drifts on a single level, the pillars between varying from 10 to 25 ft. in thickness or more. Crosscuts at frequent intervals were also driven connecting these drifts. The ore was all shoveled by hand into cars. Owing to the equipment and capacity of the mill, only the richest ore was mined, the leaner material being left in the pillars. The ore chutes delivered the ore to the main level, and it was hauled by mules to the shaft or slope to be hoisted.

When the method of mining was changed, a new shaft was sunk and haulage drifts were driven in the foot wall on the main levels 300, 750, 950 and 1150; thus they are outside and away from the orebody a sufficient distance to be protected from any injury due to mining operations, it being unnecessary to have large pillars of ore in place to support them. From these haulage drifts, which are equipped with 6-ton electric locomotives hauling cars of five tons capacity, a number of raises were driven to serve as ore chutes connecting the foot-wall drifts of each level. These ore chutes are placed in the pillars between the stopes and hence receive no interference from stoping operations.

Besides these raises, others were driven for the purpose of bringing down rock mined in the quarries and mill rejections, with which to fill the stopes after the removal of the broken ore.

In beginning stoping operations a crosscut 5½ x 7 ft. is driven on the lowest level of the ore from foot to hanging wall, which is widened later to 17 ft. with a height of 12 ft. A temporary track is laid in the small drift, consisting of 20-lb. rails of 2-ft. gage, the ore being loaded into 1½-ton cars and trammed to the ore chutes by two men.

After the ore is removed, sills are placed and the track graded to 1½% from the foot wall, so that the grade will be in favor of the load in tramming the ore. Set timbers, which consist of 4-ft. 3-in. caps inside and 8-ft. legs, are next erected and loaded with rock filling packed in behind to support them; about 18 in. of 6-in. lagging is placed on the caps of the set timbers, which are braced and equipped with plank chutes by means of which the ore is loaded in cars instead of being shoveled. From on top of the set timbers a raise 4 x 4 ft. is driven in the foot wall up to the level above, which serves as an entrance and exit to the stope after the chutes in the set timbers are full of broken ore and no longer available for that purpose.

At the intersection of the stopes with the numerous drifts on each level it is necessary, before beginning stoping operations, to build cribbings of 8-in. oak timber on each side of the stope to give additional support to the pillars and also to hold in the stope the broken ore which would otherwise flow out on the level. The cribbings are filled with broken ore, obtained from the stope to which they belong, to make them more substantial.

The work of mining the ore in the stope slices is done from on top of the set timbers. Holes 8 ft. in depth are drilled in the back or roof at an inclination of from 45° to 60° by means of hammer drills using ⅞-in. quarter-octagon drill steel sharpened with a cross-bit. These holes are loaded with six sticks of 50% 1 x 8-in. low-freezing gelatin and blasted at the close of each shift. The miners begin mining operations at the hanging-wall end of the stope, retreating toward the foot wall, standing on the broken ore as the operations proceed upward. The work of stoping is performed by two men in each stope, who not only do their own drilling and blasting, but also clean down their loose ground, place props and blockhole the chunks that are too large to pass through the chutes. These men break from 100 to 250 tons per drill shift, drilling from ten to eighteen 8-ft. holes, the amount of ore broken per foot of hole drilled varying from one to two tons.

TABLE VII. MINE OPERATING SUMMARY

Results for Active Shrinkage Stopes (Average)

Tons broken per man in stope	48.5	Tons trammed per man; tramming, handling and electric haulage	15.3
Tons broken per drill shift	145.3	Tramming explosives cost, per ton	$0.001
Tons broken per foot of hole drilled	1.34	Fill trammed per man tramming, cu. yd.	14.1
Breaking explosives, cost per ton	$0.036	Fill trammed per man, electric haulage, cu. yd.	96.0
Timber charges, cost per ton	$0.15	Fill placed per man; tramming, handling and electric haulage, cu. yd.	10.1
Tons trammed per man tramming	20.4		
Tons trammed per man electric haulage	69.0		

The above results are based on a shift of 8 hours.

The surplus ore in the stope is drawn out through the chutes in the set timbers so as to permit a space of about 6 ft. from the surface of the broken ore to the solid back of the stope to permit good working space. In this manner the stope is mined 12 ft. above the next higher level, thus permitting space to erect set timbers again in order to continue stoping operations from this level.

The stope, which is now filled with broken ore, is ready to be drawn as soon as the back is timbered. It is usually found necessary to protect the miners working in the stope and engaged in cleaning the sides and pulling the ore down into the chutes, from any falls of ground; for, as the ore is drawn down, these men are continually getting farther away from the ground over their heads.

The operation of drawing the broken ore out of a stope is begun at the hanging-wall end of the stope, the miners working on a sloping pile. As the chutes become empty, the car is moved back to the next chute, filled with waste rock. The fill is dumped from the foot-wall end of the stope by means of the regular tram-cars, the car advancing out into the stope on a track laid on the fill. When the stope becomes filled, set timbers are again erected and the whole operation is repeated.

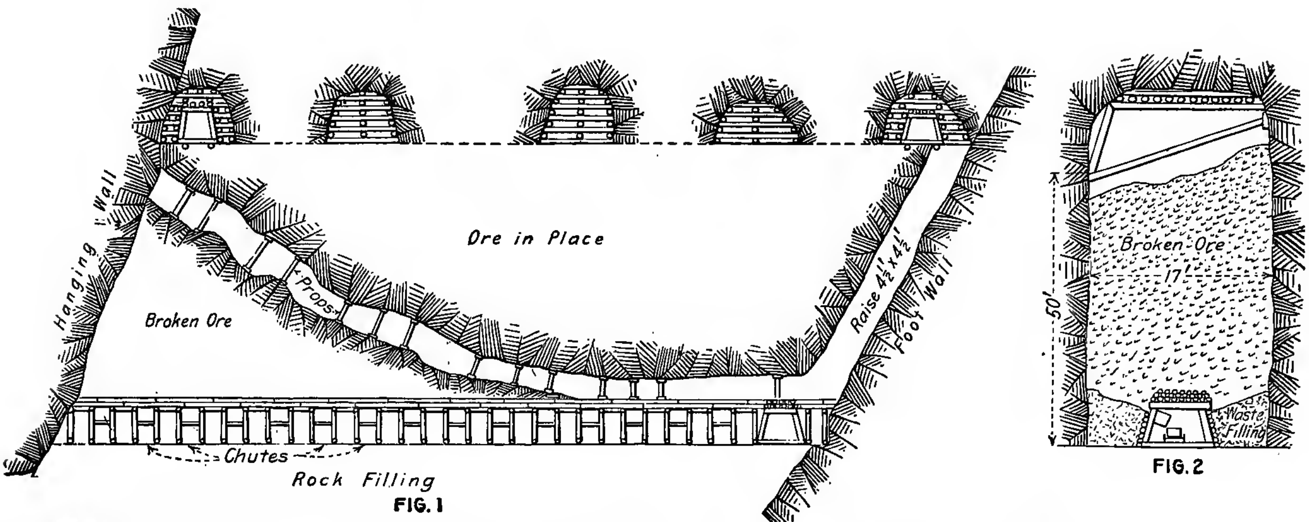


FIG. 35. LONGITUDINAL AND CROSS SECTIONS OF STOPE IN FRANKLIN, N. J., MINE

retreating toward the foot wall and protected by the broken ore on the set timbers from any falls of ground which would otherwise crash through the exposed set timbers.

After the stope is drawn, the set timbers and lagging are removed to be used again elsewhere and the stope

The largest amount mined in a single stope for one month was 7310 tons of ore. The largest amount of ore trammed from a single stope in one month was 5250 tons, while 4490 cu.yd. of rock filling is the maximum amount of rock placed in a single stope with hand tram-cars in a similar period.

The Gilman Cut-and-Fill System of Mining

BY ROBERT H. DICKSON

The Gilman cut-and-fill system is used quite extensively on the large sulphide orebodies of the Junction and Briggs shafts of the Calumet & Arizona Mining Co., Bisbee, Ariz. Only orebodies that are firm and will stand without timber are mined by this method. This cut-and-fill system was devised by Oscar Gilman, mine foreman, and Fred Sandtner, shift foreman, to take the place of a shrinkage-stope method, which was considered less safely under the circumstances. This system is used with various adaptations, so as to be applicable to very differently shaped orebodies. Essentially, it consists of blocking the ore out into sections 40 ft. wide and mining each section by alternately taking 10-ft. incline cuts from the back and then filling with waste. In this way the back of the stope is never over 10 ft. above the working floor, allowing any slabs in the back to be easily caught ly with stulls. Alternate or consecutive 40-ft. sections are mined, so that the width of the back that is unsupported will never exceed 40 ft. at one time.

The exact method of blocking out the ground preparatory to stoping depends on the shape of the orebody. With the larger bodies the ground is blocked out by running, from a main crosscut, parallel crosscuts at 40-ft. centers to the limits of the orebody. Vertical raises are driven off these crosscuts, near the intersections of the main crosscut or at some multiple of 50 ft, from it, usually. These crosscuts and raises serve as the first step in stoping. The raises are run up to the level above, where they hole into a crosscut and provide a means of filling the stope later on. Fig. 1 is a plan of the orebody showing crosscuts and raises. With narrow deposits only one crosscut, passing through near the center of the orebody, is necessary.

After crosscutting, the next step in stoping consists in shooting a little ground off the side of the crosscut so as to accommodate the temporary timbers (shown in Fig. 3) and to make headroom enough for men to work above. These sets are placed at 5-ft. intervals along the drift. These temporary timbers are set in place merely to catch the ore that is broken in placing the regular stringer sets and thus lessen the amount of shoveling. Regular 4 x 10-in. lagging, 5 ft. long, which later serves as flooring, is used as temporary posts. One of the regular posts 10 x 10 in., 8 ft. 10 in. long, is used as a cap. The temporary sets are easily and quickly set up and taken down. When in place they will allow a 30-cu.ft. car to pass under them. Lagging, 4 x 10 in., is placed across the sets, except for a space 12 in. wide running lengthwise with the track, which serves as a chute mouth. This opening is covered with loose 2 x 10-in. plank 2 ft. long, placed crosswise over the lagging. A 2 x 6-in. strip is nailed lengthwise with lagging, at both ends of the row of plank, to keep them in place while blasting. As the ore is broken, it falls to the floor of the temporary set. Enough ore is shoveled off one of the short planks, so that it can be taken up, allowing the ore to fall through the opening into a car below. Then each successive short plank is removed by prying is up with a pick or bar, for the same purpose.

When sufficient ground is broken, stringer sets (Fig. 2) are placed in position above the temporary sets, which are then removed. A stringer set consists of a 10 x 10-in. stringer 18 ft. long, set up over two 10 x 10-in. posts, 8 ft. 10 in. long. A 6 x 10-in. block, 2 ft. long, is placed on top of posts under the stringer. To block the stringer to the next one in place 4 x 6-in. spreaders are used. A floor of 4 x 10-in. plank, 5 ft. long, is laid on top of stringers. Chutes are placed in every other set on each side of the crosscut.

Starting at the end of the stringers, the ore is then broken on both sides of the stope to make room for the incline sills. These are 10 x 10-in. stringers, 14 ft. 2 in. long, set on a little over a 30-deg. incline. A 4-in. plank floor is laid on these and serves as a slide for the broken ore to run into the chutes. It also acts as a mat between the gob (which is later introduced) and the ore below, which is mined from the lower level. As the bottom of the whole width of a 40-ft. section is timbered, mining is started with the cut-and-fill system. A raise has already been run to the level above, which is used while filling the stope. The first 10-ft. cut is taken from the back across the whole width of the section, from 30 to 40 ft. on both sides of the raise. As it is possible to drill a large number of holes, the tonnage per man is very high. A few larger boulders are broken, which must be drilled with a plugger machine and blasted. All the ore broken runs from the slides into the chutes. The back of the room, made by taking out the first slice, is usually 10 to 12 ft. above the stringers.

When the ore is all withdrawn after the first cut, waste is run in from above through the raise until it forms a cone reaching to the back of the stope. The gob pile is then evened off on the sides so as to have a triangular cross-section instead of a cone, as shown in Fig. 5. Sills of ordinary 2 x 10-in. plank are laid horizontally across both sides of this pile, and a 2-in. floor is laid on this, making two slides sloping on a 40-deg. incline from the center of the raise to the floor on the stringers. All subsequent cuts are taken off the back, parallel to these slides.

As the waste tends to get higher than the ends of the incline sills, 10 x 10-in. stulls, usually 12 to 15 ft. long, are stood up vertically at the ends of the inclined sills, as shown in Fig. 2. Gob lagging is nailed to the inside of these, which serves to keep the gob away from the ore, thus facilitating mining the next section without mixing with waste.

After the slides are ready, a second cut is taken off the back. Work is started above the bottom of the slide on both sides of the raise, and a 10-ft. cut is carried upward to the raise, parallel to the slide, across the whole width of the section. All the ore is not removed as soon as broken, and hence a little tends to accumulate on the slide, which gives the miners a footing while breaking the ground higher up. The ore is then all withdrawn, and the floor, together with the sills, is taken up, preparatory to the second fill. Waste is introduced as before, forming a cone reaching to the highest part of the section. This pile is made into two flat inclined surfaces on which 2 x 10-in. sills and a 2-in. floor are laid, as before. Enough room is always left between the waste and back of stope to permit this floor being laid. This section is now ready for a third cut.

When the waste cone extends to the back of the ore, of the orebody is reached. Stulls with headboards, or a bulkhead if necessary, can be thrown against the back to catch up any bad slabs. Long bars are used to bar down any loose ground. Vertical stulls and gob lagging are carried up on the sides of the section as necessary.

As the ore is removed up to the waste and the gob will no longer run in to fill the open spaces between the top of the pile and the back, it is shoveled in by hand until it is snug against the back, for which it provides a substantial support. In special cases, where it is impracticable to use gob, bulkheads made of old timber are placed between the top of the gob and the back.

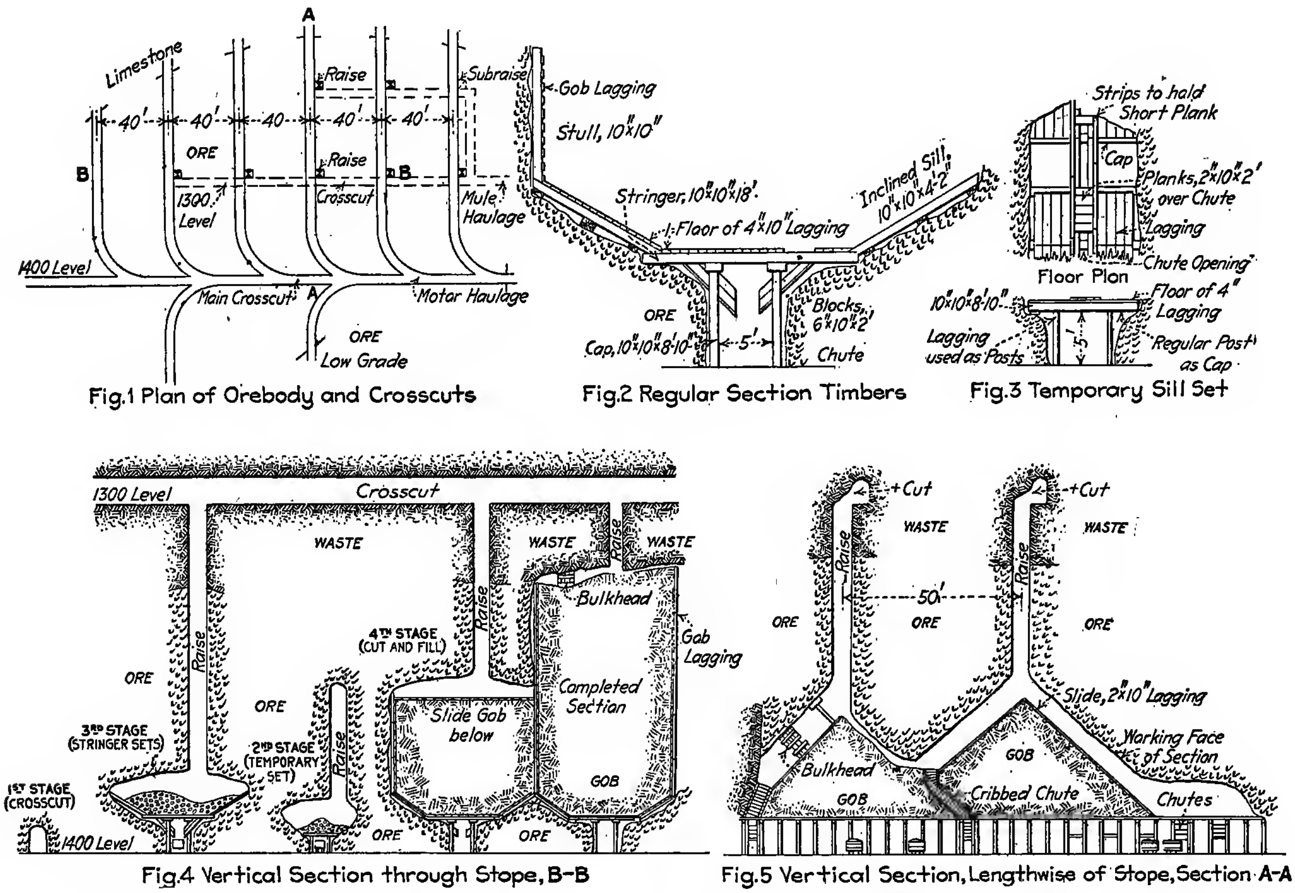


FIG. 36. DETAILS OF THE GILMAN CUT-AND-FILL MINING SYSTEM

a second raise is necessary to fill the stope. In case another raise is not already there, a subraise is run from the back of the stope, about 50 ft. horizontally from the first one, to the level above. This raise is then the center of cut-and-fill operations. In mining a long section, several raises in succession are the center of cut-and-fill operations. Where the ground will permit, cuts may be taken off two raises simultaneously, as in Fig. 5. As the lower ends of the slides get above the stringer floor, cribbed chutes of 4 x 10-in. lagging are carried up through the gob to the bottom of the inclined slides. A grizzly consisting of two 8 x 8-in. timbers is usually placed over the top of the chute to prevent extra-large boulders from rolling into it.

In this manner a whole section is mined by taking out 10-ft. inclined cuts and filling with waste, until the back

In mining a large orebody, alternate sections are usually taken out and afterward the section between is mined. Sometimes sections are worked progressing across the orebody, starting at one end and working toward the other. The sections vary in length up to 200 ft. Where they would be much longer than this, mining is carried on at both ends, as if they were two separate sections, the pillar between being subsequently removed.

The ore is drawn direct from the chutes into 30-cu. ft. cars and thence taken by electric motor to the station pocket. Skips are used to hoist the ore. The even-numbered levels (1,400-1,600) are the main haulage levels and have electric-motor haulage, while the odd levels are more the prospecting levels and have only mule haulage. The gob used to fill the stopes consists of limestone, broken in development work.

The Gilman cut-and-fill system is a very cheap method and affords a degree of security that the old shrinkage stope and glory-hole methods of mining lacked. The back of the stope is always within reach of the miner, where he can bar down loose ground or throw up a stull against the back if necessary. As the ore is taken out in successive cuts, the back does not have time to slack or become slabby. This method can be used on almost any shape of deposit. Besides being used as herein described, it is applied, with as good results, to an orebody 30 to 40 ft. thick with a limestone foot wall dipping about 45 deg., where the length of the section is made parallel to the strike of the lime. All drilling in the sections is done by stoping drills with water attachment, except on starting, where timber is being placed. Here plugger and chippy drills are used as well. The various slides eliminate practically all the mucking. The gob requires no handling except in laying the slides and in keeping it snug against waste in back, after the ore has been taken out.

Mitchell Top-Slice and Caving System

BY ROBERT H. DICKSON

The orebodies at the Cole shaft of the Calumet & Arizona Mining Co., Bisbee, Ariz., consist of a series of roughly tabular lenses that are found in breaks cutting through the limestone. The material in these breaks consists of leached limestone, broken limestone and a limonitic clayey leached material, parts of which are impregnated with sufficient oxides and carbonates of copper ore. These lenses of ore in the leached material vary in size; some are several hundred feet long, 50 ft. wide and 50 ft. high. They vary in copper contents from 10% to as low as 4%. In order to mine some of these lenses that assayed 3 to 4% copper, it became necessary to devise a method that would be cheaper than square-setting, and yet be safe for the heavy ground. After much experimenting M. W. Mitchell, mine foreman, developed the system known as the Mitchell top-slice and caving system, patterned after the top-slice and caving system as used in the Lake Superior mines and elsewhere, but so modified as to suit the local conditions.

After the extent of the orebody is determined by crosscuts and raises, it is cut into a series of sections by rows of square sets extending across the orebody at 45-ft. centers. Each row of sets is a single set in width and as long as the width of the orebody. Vertically this timbering extends, when completed, from the top of the ore to the bottom and is known as the gangway sets. The top of each section is then mined by the square-set method, one set high or more, so that there will be an even floor under which to start slicing. The ore below is then mined out in slices 8 to 10 ft. thick by running a series of continuous drifts about 6 ft. wide across the section from one gangway to the other. As each slice is mined out, the back is caved and another slice is started below. While extracting the ore in these contiguous drifts, the roof is temporarily supported by props or bulkheads. The main haulageways are generally driven parallel to, and a short distance from, the lenses — usually paralleling the longer dimension. Crosscuts are driven at right angles to these across the ore-bodies at 45-ft. centers. Raises are then driven off these crosscuts, about 40 ft. apart up to the top of the orebody. These serve as manways and chutes to the top of the stope and as a starter for the gangway sets. A row of lead sets, or gangway sets, is run from one raise to another at the top of the orebody so that it will be vertically over the crosscut on the level. In the meantime the top of the orebody is being mined by the square-set method until there is an even

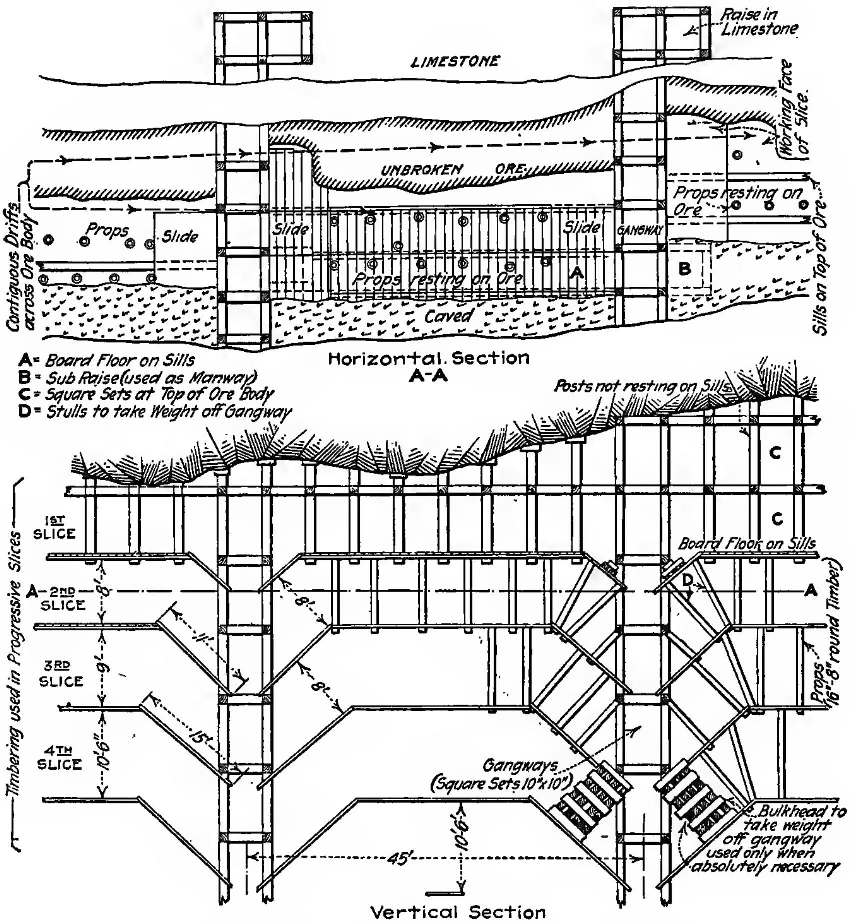


FIG. 37. MITCHELL TOP-SLICE SYSTEM AS APPLIED IN COLE MINE

floor under which to start slicing. Usually one floor of square sets is sufficient. In slicing, each 45-ft. section is considered one unit. No sills are placed under the posts of the square sets; they are placed directly on the ground. After the ore has been taken out of the square sets, long sills of 4 x 10-in. timber are laid across the stope from one gangway to the other between each row of posts. A 2-in. plank floor is laid on these sills and serves to keep the fine waste (capping) from mixing with the ore during slicing. Then, if possible, the sets are filled with waste on top of this floor. This is to serve as a cushion for any ground to fall on in case the back does not immediately follow the caved timber after slicing has started below.

While the top of the orebody is being mined by the square-set system, a second row of gangway sets is run just below the first row, and then another row is started below this. After the square sets have been properly filled, or the timbers shot down in case there is no filling, a drift is started from one gangway to another at the edge of the orebody just below the caved or square-set ground. The first 11 ft, of this drift is driven as an inclined raise on a 45° pitch (see Fig. 1), then driven horizontally until it reaches a point halfway across the 45-ft. section. A similar drift is driven at the same time from the opposite gangway, and these two hold together in the center of the section. This 11 ft. of incline at the start of the drift serves as a slide for ore to run into the gangway sets. A contiguous parallel drift is then run across the orebody in the same manner. All the ore in the slice is finally taken out by this series of parallel contiguous drifts, each starting on a 45° incline, or slide. The sills above and any loose ground are caught up by props consisting of 8- or 10-in. round timber. These props rest on the ground, so that should they take weight, they will not crush, but be driven into the ground.

The gangway sets are kept several floors below the slicing floor. They serve as the working ways of the stope, affording both manways and chutes. Slides of 2-in. timber are placed in the gangway sets so that all the ore of the slice from any part of the slides, will run into a chute without any mucking. Usually the gangway sets are connected with a permanent raise, on waste, which is used as a manway and timbering into the slice.

As the ore is mined out in slices by contiguous drifts, 4 x 10-in. sills are laid across the section, about 5 ft. center to center between the props, in the same manner as with the sills at the top of the stope. A 2-in. board floor is likewise laid on these. The ground is allowed to cave to within 15 to 20 ft. of the working face as it moves across the slice from one side to the other. Just enough props are used to keep the back safe for this distance from the face. When the whole slice has been taken out across one section, the props are shot down, in case they have not already come down, and the capping is allowed to cave. Often it is hard to make the capping fall on starting a slice, but once it has started to cave, the ground often caves, following the working face at 15 to 20 ft., as fast as the slice can be worked.

After the ground above has settled, a slice is taken out below in the same manner as before, except that the slice slide in this case is 15 ft. long and the slice is 9 ft. high. With this longer slide more of the ore will fall into the chutes without mucking. On the completion of each slice the back is caved and a new one is started until the bottom of the ore is reached. In case the bottom of the ore extends below one haulage level, raises are run up from the level below and slicing is continued off the gangway sets as before. The distinguishing features of this system are the slides and gangways. The slides (slice slides) extend the whole length of the section and eliminate the largest part of the mucking. It is rarely necessary for a machine to be set up in a slide, as it can be set up in the gangway or in the horizontal part of the extraction drift. The ore delivers from these slice slides to a grizzly in the gangway and thence to chute slides and into the chutes, from which it is loaded into cars. As the gangway sets are the working ways of the stopes, it is necessary to protect them from moving and crushing as the sections on both sides of it are being caved. Stulls placed on the stope slides are used to take the weight of the gangway timbers. In case the weight of the capping tends to disturb the gangway sets, the posts of the upper sets are chopped, or dislodged, in order to take the weight off the sets below. The gangway sets are kept intact during the crushing, because the top set, in position, does not bear any of the weight of the capping, and is usually 6 to 10 ft. below the horizontal floor, on which most of the weight falls. Bulkheads (see Fig. 38) in case of very heavy ground have proved to be very satisfactory in preserving the gangways, in the place of stulls. Two of them set up over a gangway act as an arch, which

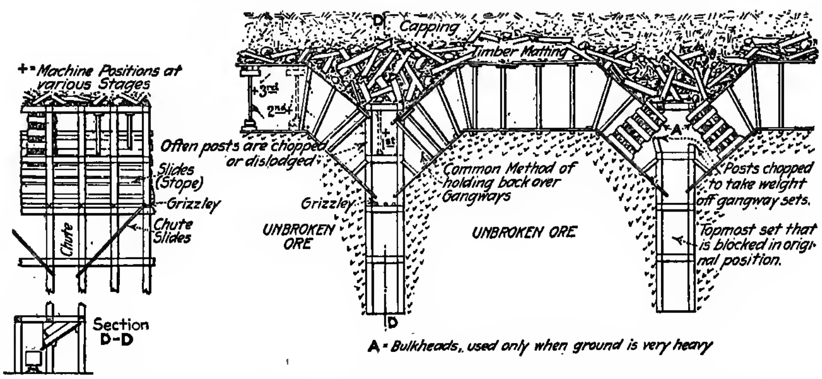


Fig. 38. Section Showing Timber Mat and Bulkheads for Heavy Ground

takes the weight of the gangway timbers onto the solid ground.

In starting this system, as is true in starting all new systems of mining, the men had to become familiar with their work before the maximum efficiency could be realized. The first slice was only 8 ft. high, the same as a square set. Finding this to be safe, the men next took out a 9-ft. slice. By this time they understood enough about the ground and how to use the props to the best advantage, so that they then took out a 10½-ft. slice, which is the regulation height. The higher the slice the longer the slide and the less the mucking. At the start a great many more props were used than were actually needed, but with experience the men found out they could get along with fewer.

All drilling was done with 2¼-in. piston machines on a vertical column. The weight of the capping tended to crush the ore, so that a few long holes would break a large round. Several sections touching each other were worked at the same time. In working each 45-ft. section, it was divided into two equal parts, one-half being worked from the gangway on one side and the other from the gangway on the opposite side. The day shift worked the north half and the night shift the south half. In this way each crew was made directly responsible for its part of the stope. The production from a single section can be varied at will. If necessary, several contiguous drifts can be driven in the slice at the same time, one advancing ahead of the other. Usually a round can easily be put in in half a shift.

In this particular character of soft, heavy ground with rather small lenticular-shaped ore-bodies, this system of slicing and caving proved to be not only satisfactory, but also cheaper than square-setting. It permits a saving in labor, timber, powder and air. The slides minimize the amount of shoveling in the stope. Cheaper timber is used and, wherever possible, timber is taken out and used again. There is a small saving in powder and air, due to the crushing action of the capping. There will undoubtedly be a larger saving in timber and labor as the men become more familiar with the system, now new. So far this method has proved to be as safe as, if not safer than, ordinary square-set mining. Practically no ore is lost and any waste encountered can be thrown on the part of the floor about to be caved.

The orebodies of the Briggs mine consist of a series of lenticular masses of iron and copper sulphides, averaging from 3 to 15% copper. One of these orebodies is 400 ft. long, 70 ft. wide and extends 50 ft. above and 40 ft. below the main level. One side and the bottom of the orebody are limestone, while the other side is leached material and pyrite with a capping of siliceous breccia. The physical character of the ore in the lenses will not permit open stopes, but parts of them are favorable for the Mitchell top-slice system. The ore is of a fairly uniform grade, containing no horses of waste, and the back of the stope will stand for a short time with the timbers used.

Usually, before beginning mining operations, the orebody has been cut by several crosscuts at right angles to each other and several raises to the top of the orebody. The Mitchell top-slice then is used in four distinct steps:

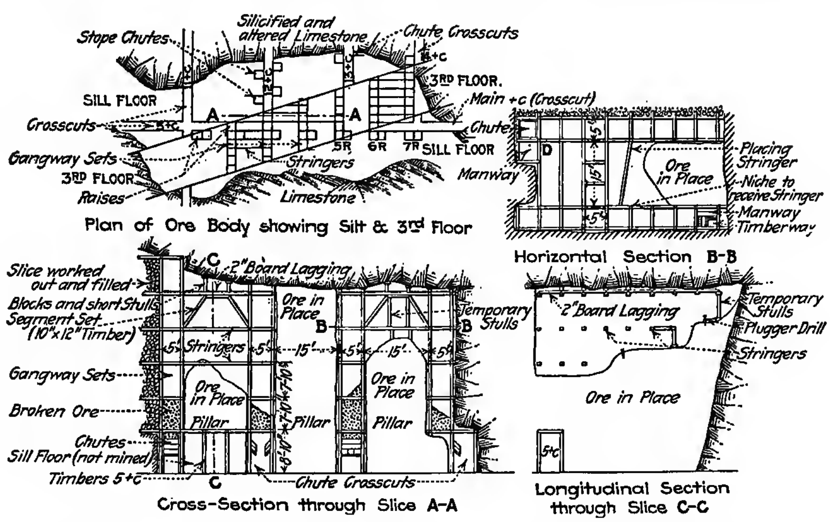


FIG. 39. MITCHELL TOP-SLICE SYSTEM AS APPLIED IN BRIGGS MINE

1. The orebody is blocked out by chute crosscuts driven at right angles to a main crosscut and spaced at an average interval of 40 ft. Raises are driven off the main crosscut at 20-ft. intervals to the top of the ore.

2. The orebody is cut into a series of pillars 15 ft. thick, by running single rows of lead, or gangway, sets from the first floor of the raises to the edge of the orebody. These are carried upward to the top of the orebody.

3. These pillars are mined by overhand stoping, and long stringers are placed between the two adjacent rows of gangway sets to keep up the back and hold the square sets in position.

4. The space occupied by the pillar is filled with waste, and the adjoining section is then mined out in the same manner.

Figure 3 is a section taken through one of the lenses, showing the sill and third floor. The ore was first cut by the prospecting crosscuts $1 + C$ and $5 + C$. One prospect raise was then put up at the intersection of $1 + C$ and $5 + C$ and another 60 ft. beyond it, off $5 + C$, to determine the height of the orebody.

As the first stage, the chute crosscuts are driven at right angles to 5 + at an average of 40 ft. center to center, starting at 1 + C. These are driven to the edge of the orebody and later serve as haulageways from the stopes. Two-compartment timbered raises are put up, off 5 + C at 20-ft. intervals, starting at No. 1 raise, to the top of the orebody. Usually every second raise is carried up to the level above, to permit timbers being lowered into the stope, and also so that waste can be introduced as filling after the pillars have been mined out. In the case of the orebody in Fig. 39 the ore lying to the east of 5 + is only about 25 ft. wide, hence no crosscuts were driven off 5 + on this side. In most cases the crosscuts in ore require timbering.

This method contemplates mining only the ore from the first floor (8 ft. above the sill) to the top of the orebody at this period; the ore below will be mined at a later time. After the ore has been blocked out by raises and crosscuts, a single row of lead gangways sets is run at right angles to 5 + off the first floor of the chute compartment of each raise to the edge of the orebody. Every other row of gangway sets will be directly over, or at the side of, the top of the chute crosscuts. These sets, if over the crosscut, are taken out in the same manner as in square-set stoping — usually being drilled with a stoper machine. The row of lead sets not touching the chute crosscut are taken out by driving a crosscut with a machine on a vertical column and timbering it with regular square-set timbers laid on sills. Sets are carried upward from these single rows of lead sets to the waste, as in regular square-set stoping. In this manner the orebody is cut into a series of pillars 15 ft. thick extending at right angles to the main crosscut across the whole width of the orebody. As the first row of lead sets, touching the chute crosscuts, is being taken out, a series of chutes is placed at the side of the crosscut by shooting out a small amount of ground. The ore mined above falls into these without mucking. In the odd line of gangway sets, which do not touch the chute crosscut, chutes are placed in front on the main crosscut. The ore is allowed to accumulate in these rows of sets until the material forms a slide, above which it will roll by gravity into the chute. The length of these rows of gangway sets does not usually exceed 40 ft. from the main crosscut. In the case of friable ores the sides of the sets are sometimes lagged.

The next step consists in mining the pillars by first catching up the back with long stringers and then removing the ore in horizontal sections, beginning at the top of the pillar. In starting to mine the pillar, a piston machine is set up in the top set in front of the raise, and an extraction drift is driven on about a 30° pitch upward to the center of the pillar, where it meets a similar drift driven from the opposite gangway set. The broken ore runs down from the face of these inclined drifts into the gangway sets and thence to the chutes. A 10 x 10-in. stringer 14½ ft. long is then placed next to the back, between the posts of the two opposite gangway sets. Sometimes the top of the pillar at the center is only 2 ft. below the stringer. A drift contiguous to the first one is run in the same manner, making room for a second stringer. Usually the stringer is blocked temporarily to the back and pillar in case there is any danger of breaking it while shooting. A slice is thus taken across the whole top of each pillar, from the raise to the end of the orebody, by running this series of contiguous extraction drifts. The back is caught up with 15-ft. stringers, which extend from one row of gangway sets to the opposite. Two-inch lagging is placed across the stringers. No mucking is necessary, as the ore runs by gravity into the gangway sets and thence into the chutes.

While the last of the upper stringers are being placed in position, a second slice is started on the floor below, by taking a 5-ft. vertical cut (using a plugger machine) across the front of the pillar, sloping upward to the center of the pillar, as described. A 15-ft, stringer is then placed, butting up against the posts of two opposite gangway sets, as before, making two stringers in place, one 8 ft. above the other. A segment set consisting of two angles and a spreader is then placed between the upper and lower stringers, and the upper stringer is blocked to the back. As the second slice is being carried across the pillar, stringers and segment sets are placed

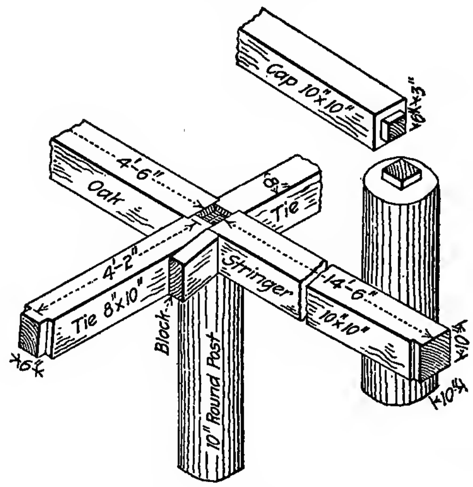


FIG. 40. DETAILS OF TIMBER FRAMING

between the gangway sets as before. Segment sets are used only at the top of the stope and are for the purpose of keeping up the back while the ore is being taken out of the pillar below. As the second slice is progressing from the front to the back of the stope, a third one is started below, in the same manner as before. The slices are progressively started at the front, so the ore will run toward the front part of the stope (over + C), as the only chute in one gangway is in front. All the ore runs into the chutes without mucking until near the bottom; the ore in one corner of the stope must be shoveled, as it will no longer run into the chutes. The slices usually average 5 ft. in thickness, and the slope on the side nearest the chute crosscut is longer than the opposite side. Plugger machines are used exclusively except in starting the slice.

Often the top of the orebody extends above the top stringers a little distance. In this case the ore is usually mined out after the segment sets are in place, and short stulls are used to catch up the back. If the ore is only 2 to 3 ft. above the stringer, it is usually taken out before the stringer is placed in position.

Each stope has several outlets. The manways of the raises serve as ladderways and also for the purpose of bringing timber into the stope. Usually there is a manway at the end of the gangway sets from the chute crosscut. A timberway is partitioned off one corner of the manway with 1-in. boards and is used to bring tools and timber into the stope. The ladders are staggered and inclined, having a platform every 8 ft. The long stringers are usually lowered or hoisted (if necessary) through the manway in front, which has vertical ladders. When bringing a stringer into the stope from below, it is sometimes necessary to remove the cap in front of the raise, in order to get the stringer into the manway. Usually the stringer can be landed into the stope with little trouble.

Fig. 40 shows the details of the timbers. The cap tie and post are regular square-set timbers. The stringer is 14 ft. 6 in. long and squared off at both ends to fit into the gangway set. The stringers are placed in position by placing one end against the gangway sets as an ordinary cap is placed; and then the other end is swung to butt up against the set in the opposite gangway. To allow this being done, a niche (Fig. 40) has been previously chopped in the tie, making the stringer fit snugly against the horn of the posts. After the stringer is in place, a 2-in. block is nailed to the tie on this niche to hold the stringer in position.

In mining the orebody in Fig. 37, chute crosscuts were driven only on the west side of 5 +, as the ore on the east side extended only 20 ft. from 5 +. Gangway sets were driven on the same side, from the raises, and the pillars were mined as before. All of the broken ore rolled to the chute in front of the stope.

Method of Mining Pillars at the Fortuna Mine Of the Braden Copper Co., Chile

BY CHARLES HOLLISTER[1]

The method of mining at the Fortuna mine of the Braden Copper Co., Chile, is adapted from that used at the mines of the Ray Consolidated Copper Co., Ray, Ariz., namely, shrinkage stopes separated by pillars at regular intervals transverse to the orebody. At the Fortuna, the orebody presents an elliptical horizontal area 3000 ft, on the long axis and 260 ft. across at the widest part. The dip of the orebody is steep, and it has been developed by tunnels at four levels from the surface to a depth of 1425 feet. Shrinkage stopes extended across the orebody from wall to wall and ranged from 15 to 25 ft. in width; pillars left between stopes were usually from 15 to 24 ft. wide. The flexibility of this system permitted a widening or narrowing of the stopes or pillars, according to changes in the character of ground — wide stopes and narrow pillars for hard ground and *vice versa* for soft ground. At Ray, the earlier 25-ft. wide pillars, alternately separated by 25-ft. wide shrinkage stopes, were mined first by taking a 10-ft. wide shrinkage stope up through the center of each pillar, thus stoping out the flexibility possessed by the old method, and though increasing the proportion of initial breaking necessary by reason of the additional amount of shrinkage stoping required, the saving in chute blasting and the extraction of cleaner ore compensated for the additional work. Accordingly, the stopes were all completely drawn and waste filled in, under and alongside the partly caved pillars and close to the stope backs.

To mine the uncaved block and recover the arched pillars, it was decided to raise in the foot wall at a distance sufficient for a good margin of safety and to develop a sublevel system for stoping and caving. The block was divided into two parts, an upper and a lower. The upper section was that portion of the ore above the ground that had been stoped and the lower section included the pillars of the stoped-out and filled portions below. The upper section was developed, first, by a drift in the foot wall from raise to raise; then drifts 24 ft. center to center from the foot-wall drift to the hanging wall of the orebody, driven parallel to the original line

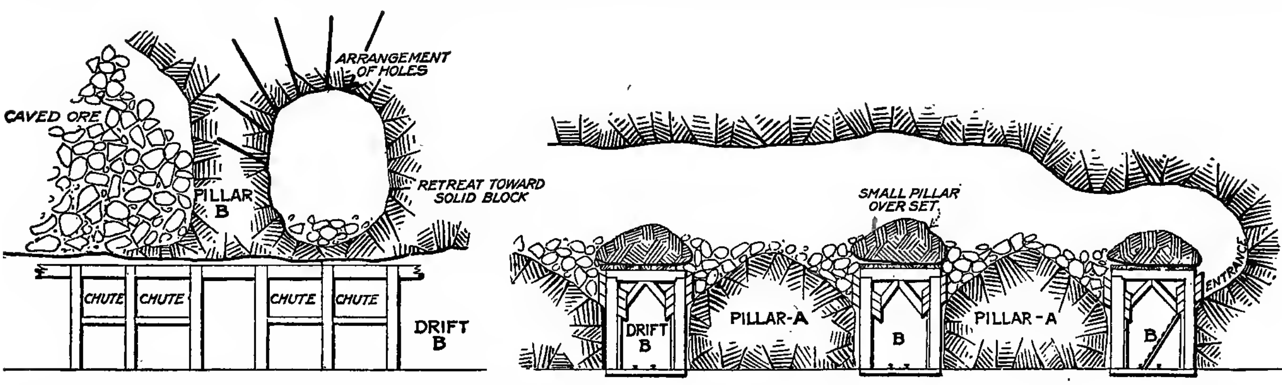


FIG. 41. LONGITUDINAL AND END SECTIONS SHOWING METHOD OF OPENING UP AND STOPING UPPER SUB-LEVEL AT FORTUNA MINE

core, and then undercutting the pillar base and caving the whole over large areas by simultaneously drawing the ore from several stopes and pillars. Several methods of pillar attack were originally tried at Braden. One method was to excavate under the pillars by a series of drifts and crosscuts, undermined by drilling and blasting out the solid sub-pillars so left between. Though this plan seemed feasible at the time, and probably would have been successful under different conditions, results at the Fortuna were not satisfactory.

In drawing the broken and caved stope and pillar ore after undermining the pillars, the action was a gradual sloughing of the ore from the back of the pillars until the back appexed at the inclined hanging wall, forming an arch that prevented further caving, and leaving a large block of ore on the foot wall.

Horace Graham, who became superintendent about the time that the operations detailed were in progress, made some decidedly advantageous and well-devised changes in the mining method, eliminating entirely the wide pillar and stope systems in favor of narrow stope and pillar units. This change retained the advantages of of pillars. Incidentally, in driving the foot-wall drifts, a large orebody was developed which had not figured in former calculations. Preliminary work was practically completed on this level when I took charge of the Fortuna division. Some stoping had been started, though this did not appear to promise entire success.

Eventually, the following method was adopted: Chutes, of which only a few had been previously constructed, were placed staggered for two sets, then a set missed, etc., to the end of these transverse drifts. As shown in Fig. 42, short inclined raises were driven at about 35° to the horizontal, connecting with similar raises through the pillars between adjacent drifts. This left a pillar 6 ft. thick over each chuteless set between raises. The connecting raises were then stoped up from 7 to 8 ft. high, and the muck was drawn down, leaving only working head room. The intervening 6-ft. pillars were then drilled and blasted successively. The men were always safe, as the retreat was away from the undercut and caving area, as shown in Fig. 42. The result of this procedure was completely to undermine the entire block,

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and though it was thought at first advisable to put up a cut-off shrinkage stope to free the block from the foot wall, the rapid and complete caving of the ground made that procedure unnecessary. The block yielded 110,435 tons out of a possible expectancy of 120,000 tons. Timbers in this section were 8 x 8-in. sets with 8-ft. posts and placed at 5-ft. centers along the drifts.

The second or lower section above the Fortuna No. 3 sublevel, with the exception of a part of the foot-wall drift, was developed during the drawing stage of the upper section. The development plant was practically identical, central pillar drifts being 24 ft, center to center, and timbering and chute building following at a working distance behind each heading. Owing to the heavy character of ground expected in this section,

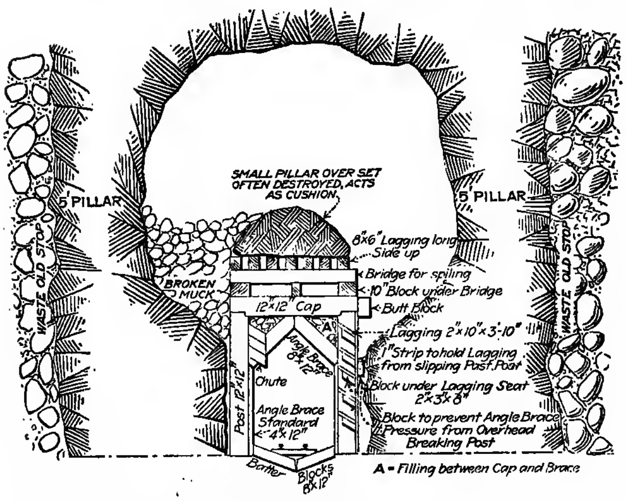

FIG. 42. DETAILS OF TIMBER SET AND SECTION OF STOPE OPENED UP IN LOWER PILLAR LEVELS

12 x 12-in. timber sets were used, supplemented with bridges for spiling purposes.

The conditions that existed here were entirely different from those of the first section. The ore in the shrinkage stopes on either side of the pillars had been completely withdrawn, and the stopes were filled with waste. The problem that presented itself was to prevent pillar stopes from breaking into the old filled stopes and allowing the waste to mix and dilute the pillar ore. Chutes as shown in Fig. 42 were placed in each timber drift set and raises driven up to 6 ft. at an angle of 45° into the pillar, along each side of the drifts. The raises were then continued by inclining backward to connect over the top of the drift. After this was done for the total length of a drift, the small pillars between openings were drilled and shot. The result was the removal of a core which, with occasional holes to widen the excavated area, caused the pillars to cave freely, and the caved ore was drawn from the chutes. The extraction up to Mar. 1, 1917, from pillars No. 1, 2, 3, 4, 5, and 6 was, respectively, 100, 77.3, 64.2, 78.6, 37.7, 80.7, 80.4, and 58 per cent. Upon my return to the United States at this time, the chutes, with the exception of those in Pillar No. 1, were all still producing ore.

It is of interest to note that at Ray the top weight on timber sets manifested itself first, resulting in a firm setting of the timber sets before side pillar pressure took effect. This, however, was not the condition at Braden. The primary pressure was exerted by side squeezing, causing the posts to yield by bending inward. To relieve this strain, the angle braces, as shown in Fig. 42, were changed from 50° to 40° from the horizontal, and 8 x 12-in. batter blocks were used in place of sills. A crew was kept picking away the ground back of posts to relieve the pressure. Conclusions were:

1. In heavy sections of wide area it is imperative to start caving operations as soon as possible after drift development and timbering have fair lead. This to permit chute construction and raising without interfering with the other work.

2. Timber must be well blocked all the way around, or the first weight of the subsiding ground will push the sets over, thereby losing an entire drift, with resultant high cost and frequently with fatal delay. Poorly fitting collar braces seem to be the main contributory cause of leaning timber sets, owing to their failure properly to withstand excessive pressure.

3. A timber set, to give the best service, should subside gradually and vertically under pressure. No timber can hold up the enormous weight imposed by caving ground, and the best to be expected is a maximum life of resistance. Caps 12 x 14 in. in section have been crushed to a thickness of four inches, yet the posts showed no sign of strain except a tendency to splinter with the grain. It was noticeable, both at Ray and Braden, that the point of least resistance along the posts, provided the set stood squarely, was about 18 in. above the bottom. This is undoubtedly caused by the expansion of the pillar base through the crushing weight of the irresistible mountain overhead.

Notes on Shaft Relining With Concrete

BY G. G. STONEMARK

In American mining methods, permanent lining of shafts with concrete has established a procedure of maintaining the alignment with permanent stability and for fire-resisting qualities, and the development of concrete shaft lining has been rapid and progressive. In Europe, brick, steel and cast-iron linings are common, and there is a scattering of concrete linings of unit construction. The influence which concrete lining has exerted upon the design and upon the problem of choosing a shape has made it unnecessary for the engineer to adopt the cross-section to limitations imposed by rigidity of lining material, and the thickness of the lining and reinforcement can be varied to meet local conditions of changeable stresses.

In most shafts constructed of timber, unless the conditions are unfavorable to the fungus growth which causes rot, the sustaining power is so diminished as to force the original alignment out of plumb. To reëstablish this, some means of determining with approximate accuracy the position of the shaft must be resorted to, and the most efficient way I know of is by the use of line and plumb bob, similar to the method used in plumbing a shaft to establish lines underground. Four lines are essential, one in each corner, or as nearly so as possible, that they may hang free and be placed in reference to a proposed or actual axis of the shaft. The next step will be to determine the profile of the shaft timber by measuring the distance down from the collar of the shaft as a base vertically, and the distance from each wire to each end piece and wall plate. These points, plotted in four planes, will determine the inclination and an accurate condition as to the alignment of the existing timbers. Where a careful alignment was made when the sinking operations were in progress, it may be a simple matter to determine the original alignment, because of the fact that part of the old timber still remains in its original position or a portion of some section may be in such condition as to hold its original alignment.

The plotted points representing these timbers will show themselves on the profile by their parallel to the plumb line, but, should this not be the case, an average can be plotted so as to conform with the headframe, so that a small movement either way from the original axis of the shaft is negligible, both from a standpoint of excavating new ground or that of moving or shifting the sheave wheel to conform to the new alignment. Of course, judgment is necessary as to what amount of shifting is possible.

Having determined the alignment by placing the plumb lines with respect to the proposed or original alignment, it becomes necessary to make the lines permanent, so that the placing of the concrete forms can be done with accuracy. For retaining the lines in the shaft diagonals, sprags can be nailed across the end piece and wall plate, and a nail or spud driven directly in the plumb line and at distances about 40 to 50 ft. apart. Then when one spud is destroyed by the removal of the old timbers in the lining operation, it will only be necessary to go to the one higher up and hang the plumb line. These lines should always be referenced on surface in some manner, because when the operation of lining approaches the surface, somewhere near the top there will always be more or less settlement of the timber remaining, and one can therefore reset new lines with little trouble.

The following methods were used in lining with concrete the Tener shafts "A" and "C" of the Oliver Iron Mining Co., at Chisholm, Minn. The headframe at "C" shaft, Tener mine, is higher than usual above the collar and higher also than is general practice on the Mesabi range, and has an ore pocket of ample dimensions. Inside of the pocket was built a temporary partition, dividing it so that sand could be placed in one-third of the space and the remainder allowed for rock. The cement was stored in a temporary shed constructed in the framework beneath the pocket, but was placed at an elevation over the mixer. The loading track was next moved as near as possible to the headframe, an opening between the rails excavated, and a launder used as a chute was built to extend down to the nearest compartment of the shaft. This compartment was bulkheaded and boarded up to form a skip pit, having an opening over the wall plate and with the end of the launder chute extending through.

All the aggregate was loaded in ore cars, so that it was an easy matter to dump the contents into the launder, from which it could be drawn off into the skip and hoisted in the same manner as ore. A temporary hoist was installed for this purpose, because the other hoisting engine consisted of a single drum and was used to hoist the cage in the other compartments. If sand was to be hoisted, it would be dumped directly from the skip into the pocket. An inclined chute, built on an ordinary stock pile trestle car truck and wide enough to accommodate the full width of the skip, was used when rock was hoisted, and a skip loaded with rock dumped on this chute would simply slide across the sand compartment and into the rock compartment.

The quarter pans in the pockets were taken out, the chute was reversed, and a temporary covered chute built to hold the different aggregates which led down to the measuring hopper. This hopper was situated over the mixer and on the same level as the shed for the storage of cement. Each chute was provided with a quarter pan, so that the workmen could run out the required amount of the different aggregates. Two workmen were stationed here to fill the measuring hopper and another workman whose function it was to open the cement sacks, pour them into the measuring hopper, trip the outlet and chute it down to the mixer. Two men were stationed on the mixer floor to look after the water and dump the mixer into a specially made car, which was placed on the cage and lowered to the forms below. When the pouring was completed, the car was taken off and the cage used to hoist all material taken out of the shaft, such as old timbers and lagging, and also for lowering and hoisting the men.

Before the work was started, I thought probably that the method of dropping the concrete through a pipe would be speedier. If the depth had been greater, this might have been so, but not in this instance, because, comparing the working time with that consumed at "A" shaft, in which all concrete was dropped through a pipe for a depth of 250 to 300 ft., the time taken to lower 45 to 50 batches at "C" averaged $3\frac{1}{2}$ to 4 hr., the same as at "A." On timing several batches sent down 250 ft., it took on an average of $2\frac{1}{2}$ to 3 min., with less time for shorter distances.

Comparing the two methods of conveying the concrete from the collar to the forms below, certain conditions will have to be met successfully, such as stopping the concrete when it is dropped through the pipe. At "A" shaft, where all the concrete was dropped through a 4-in. pipe, a hopper built at the top of the pipe at the collar conveyed the concrete to the pipe from the surface car. At the bottom of the pipe, in order to convey the concrete to the car on the cage, and regulate the flow, a long sweep elbow was screwed to the column pipe. This elbow reduced the force of the rush of concrete to a medium flow, but so quickly did the concrete wear a hole through the elbow that it had to be replaced at every pouring, and finally an extra heavy cast-steel elbow, with a short piece of pipe screwed on, was tried, with better success.

In depositing the concrete in the forms, essentially the same methods were used in both shafts. The door at the bottom of the hopper-shaped car on the cage was opened, and the concrete flowed through a hole in the cage floor into a section of a funnel-shaped pipe. This pipe had an opening about 18 in. in diameter at the top, tapered with a slight curve, and was reduced in section to about 8 in. at the other end, so as to fit a light steel pipe having lengths of 1, 2 and 4 ft., and so made as to telescope into each other by means of a pair of eyes riveted at one end and a pair of hooks at the other. This arrangement made a flexible pipe, that would reach any part of the forms and in almost any position desired for depositing concrete.

Among the difficulties encountered in dropping concrete through a pipe is the liability to clog, for care is not always taken to see that all large stones are removed. Several times during the pouring in "A" shaft, the column pipes had to be disconnected to determine where the clog was situated, and it was necessary to hammer the pipe until the concrete was loosened, in which case all the material remaining in the pipe was wasted.

At "C" shaft all the concrete was lowered with the cage, using the same car as employed at "A," down to a sill placed across the shaft on the wall plates for each consecutive pouring. The same flexible pipe was used to distribute the concrete to any part of the forms. A shaftman, stationed on a platform built level with the sills, opened the door of the hopper-shaped car as soon as the cage came to rest on the sills, and it was a matter of a few seconds for the contents of the car to flow out. The cage was then hoisted to the surface, where another batch was in the mixer and ready to be dumped into the car. The process continued until a form was completely poured. The men knew their respective duties, and became more proficient as the work continued, so that extra help was not necessary. Of the six men employed, including the foreman for each shift, three were generally required in the shaft and the other three at the surface, although an additional man was always at the shaft to look after the odds and ends and to run the mixer when concreting was being done.

If the lining is to start at the bottom of the shaft and the lines, surface plant, and all else is in readiness, the shaftmen first clean the bottom of the shaft of all dirt and remove all the old lining sets for 10 ft. or more, depending on conditions. If the bottom is in rock, no bearers will be required, but if not, hitches must be cut in the sides of the shaft to permit the placing of steel bearers, the length of which depends on the size of the shaft. These hitches should be at least from 1 to $1\frac{1}{2}$ ft. deeper than the bottom of the shaft, so that a spread footing can be secured. In order to get the full benefit of the steel used for the bearers, concrete is slushed in and reinforced with short steel rails or other rods that can always be found in the scrap pile, and placed so as to form a grillage which is built to the level required for the bearers. The first form in the shaft is built as high as the top of the bearers. This is made of plank, set carefully to line and leveled up and the bottom of the form made to fit the profile of the shaft bottom so that the concrete will not flow out. After the form is poured, it is allowed to stand for a few days, and during this time the sides of the shaft are trimmed out to the next line required for the thickness of the lining.

Until the regular forms arrived, a few sets were placed by means of wooden forms in "C" shaft, but these were found to be uneconomical. The steel forms used by the Oliver Mining Co. were panel sections of such dimensions as to be easily handled by rope and block, and made of $\frac{3}{16}$-in. plate stiffened at the edges by $2\frac{1}{2}$-in. x $2\frac{1}{2}$-in. x $\frac{3}{16}$in. angles. The panels varied in size, the smallest being $6\frac{1}{2}$ in. x 12 in. and the largest 2 ft. 11 in. x 5 ft. $11\frac{1}{2}$ in. In all, 46 panels were required for one complete 6-ft. set, and 32 for a 4-ft. set. In forming an end lining, six panels, and for a wall plate lining eight panels, were used above the dividers and end pieces, with nine small filling panels in between the end pieces and dividers. In between the panels were placed insert keys of $\frac{1}{2}$ in. x 4 in. flat steel that were made a little short of the required height of a full panel. If these keys had not been so placed, it would have been next to impossible to remove the panels. So the keys were first removed, and this gave the panel an additional space, so that all panels could readily be removed, and whenever one panel was removed in a tier, the others were simply unbolted in consecutive order.

To give the shaft line a greater stability and a means by which the skip guides could be fastened, cast concrete dividers and end pieces, reinforced by steel rods and of sufficient length to be well imbedded in the lining, were placed at intervals and the reinforcement was allowed to extend beyond the end. These end pieces and dividers also supported the steel forms in rigid alignment by means of cord holes so spaced that a bolt of short steel rod could be inserted through the angles of the forms and into the end pieces or dividers. A set of end pieces and dividers was first placed on the bearers previously set and leveled up, and the first set of forms placed and bolted together, the slab reinforcement having been previously fabricated by wiring together vertical and horizontal reinforcing rods.

Inside the shaft forms, braces made of 2 or $2\frac{1}{2}$-in. pipe were placed transversely. These braces were flattened and had holes made in each end, so that when they were fastened at an angle the correct dimensions necessary to hold the forms rigid could be secured. The braces were also used to support staging planks. Another set of spacers was required to keep the end pieces and dividers in alignment, and these were made of 3-in. x 6-in. plank, armored at each end, with strap steel securely fastened to the top of the forms. The lower end of end pieces and dividers. The forms were then shifted to conform with the plumb lines, and temporarily wedged to keep them in place during the pouring of the concrete. The reinforcing was held in place by a steel template fastened to the top of the forms. The lower end of the forms was tied to the rods of the preceding sets, which were cut long enough to extend 3 or 4 in. out of the concrete.

Everything being ready for the pouring of the concrete, temporary sills were laid across the shaft on a wall plate 8 to 12 ft. above and used to support a funnel-shaped spout having an 18-in. opening and slightly curved and reduced at the other end to telescope into the short section of the light steel pipe, which has been previously described. A number of shaftmen were then sent to surface to handle the mixing, and other work. The hopper-shaped car was placed on the cage, the first batch of concrete started in the mixer, and when the desired consistence of the concrete was obtained it was sent on its way to form an integral part of a lasting structural unit. Immediately after the forms were placed, the concrete was covered with planks, and the work of removing the old timber and trimming for another set of forms was begun. Removing the old timber was dangerous, and care and watchfulness were absolutely essential, for paint rock and the other materials which have little or no support are generally loose, and it seemed at times that little was required to bring down a mass of the ground.

All unnecessary material was hoisted to the surface in a bucket hung underneath the cage. In case too great a space between the form and the wall was left, a back form was built, and the space filled with the excavated material. When the sides were trimmed to the required dimensions, the work for the next form was placed and lined up the same as the prceding one.

Six complete sets of forms were used in both shafts, making a section of 24 ft. of lining, and this gave sufficient time for the concrete to set and become strong enough to support the superimposed load.

During the progress of the work some unfavorable ground was encountered, and room for a 6-ft. set was impossible. In these cases, a 4-ft. set was poured, and the same method as for the 6-ft. set used. Several such cases were encountered in the "A" and "C" shafts when working the surface above the ore. At every third set, when the sets were spaced 6 ft. apart, an extra divider was inserted to support a steel solar deck, which had an opening for a ladderway. Steel ladders long enough to reach from one solar to another and bolted at the foot to the deck and stiffened by struts fastened to the wall were used. These ladders are made of $1\frac{1}{2}$-in. x $1\frac{1}{2}$-in. x $\frac{1}{4}$-in. angles, using $\frac{3}{4}$-in. pipe spaced 12 in. on centers for rungs.

The table shows the comparison in progress made at "A" and "C" shafts:

TABLE VIII. PROGRESS MADE AT "A" AND "C" SHAFTS, TENER MINE, CHISHOLM, MINN.

	"A" Shaft	"C" Shaft
Number of weeks worked	18	19
Average progress per week, ft.	$16\frac{1}{2}$	$15\frac{1}{2}$
Average progress per month, ft.	75	68
Greatest progress per week, ft.	30	48
Greatest progress per month, ft.	112	132
Depth of concrete, ft.	304	292

At "A" shaft, column openings were made at the pump station and at the level. At "C," a skiptender's drift and small pump station were concreted; also column openings for two levels. In both shafts, concrete covered I-beams were used, 15-in. beams for columns and 12-in. beams for caps.

Shaft Sinking at the Seneca Mine

BY W. V. FEATHERLY[1]

The Seneca Mining Co., at Mohawk, Mich., is sinking a compound shaft, which is to be vertical to a point 1450 ft. from the surface and will then follow a 400-ft. radius curve to meet the Kearsarge lode, which dips at an angle of 34° from the horizontal. The equipment and methods adopted in sinking are in accordance with modern practice, and the progress made has established a record for shaft sinking in the Lake Superior district.

The shaft is to be of four compartments, having two skipways, one ladderway and one pipe compartment. Over-all dimensions of the excavation are 11 ft. 4 in. x 21 ft. 4 in. and steel sets of 5-in. H-beam construction are placed at from 6 to 8-ft. centers, depending upon the nature of the ground. These sets are put in so that they are placed 4 ft. from the bottom when the ground is soft, up to 30 ft. from the bottom when the ground is hard. The lagging is of 2-in. hard wood.

A double-drum steam hoist operates two 36-cu.ft. buckets. Air is furnished by one Sullivan tandem-compound, Corliss steam-driven compressor, and one class WB-2 Sullivan straight line, simple steam, two-stage air compressor is installed as a reserve. At the beginning of operations eight Sullivan DP-33 air tube rotators and eight hand hammer-drills of another make were in use. It was soon obvious that the Sullivan rotators were securing better results. Four more Sullivan rotators were purchased in April, and since that time 11 have been in constant operation, and two kept as spares. The header, shown in an accompanying cut, is connected to the main air line by a 3-in. hose and is supported in the center of the shaft while drilling by means of a cable leading from the steel framework. There are 11 hose lines, each 14 ft. long and leading from the bottom of the header to the drills; also one connection for a blowpipe.

SHAFT CREW WITH HAND HAMMER-TYPE DRILLS USED IN SINKING SENECA SHAFT

Excavation was begun on Feb. 13 during severe weather conditions as well as other handicaps. Water froze in the buckets, there were no steam lines installed in the shafthouse, and the rock had to be trammed by hand. Three shifts were worked, with only four or five men on a shift, and under these conditions progress was slow. During February, 51 ft. was sunk, and in March 113 ft., with conditions about the same. Early in April the rock-disposal hauling outfit was completed, the headframe inclosed, and steam heating coils were installed, so that 154 ft. of shaft was sunk during that month. By increasing wages 50c. per day the management was able to secure sufficient men by May 1, and since that time the work has been carried on at full capacity, the crew averaging ten men and one shiftboss per shift, and working three shifts.

During May 208 ft. was sunk in 27 working days, and this rate was maintained in June, when 195 ft. was sunk in 25 working days. This figure would have been exceeded had not the men encountered loose, soft ground, which caused considerable delay, as it was necessary to keep the steel work close to the bottom of the shaft to guard against loose rock falling on the men.

The bonus system which has been adopted consists of the following: The men receive a flat rate per day up to the first 100 ft. per month; 2c. per foot per day for all over 100 ft. up to 150 ft. per month; 3c. per foot per day for all over 150 ft. up to 200 ft. per month, and 4c. per foot per day for all over 200 ft. per month. This provides an incentive for the men to work hard, and each shift tries to send up more rock than the previous one. The shifts are well organized, and the same men have their regular work to do each time they blast. Two men attend to the raising and lowering of the header, which is hooked on the bottom of one bucket. The drills are taken up in the other bucket. One man looks after the main wires while the others are charging the holes and getting the tools up. The last two men connect the main wires. One of them inspects everything at the last minute and carries the key to the blasting switch.

[1] Walker Bank Bldg., Salt Lake City, Utah.

In blasting, electric igniters and blasting caps are used. The bottom of the deeper holes is charged with 60% powder, 45% being used in the remainder. The average depth of hole is 8 ft., and the round of 45 holes is drilled in from two and a half to three hours. The "sink" is made with a double V-cut, and is shot in two separate blasts, the cut holes being fired first. In the diagram, cut holes numbered 1, 2, 3, and 4, are shot in

AIR HEADER USED IN SINKING SENECA SHAFT, MOHAWK, MICHIGAN

that respective order with instantaneous and delay caps. If all of these holes break satisfactorily, the squaring holes numbered 5, 6, and 7 are set off. This plan insures the breaking of the sink holes before the ends of the shaft are squared. Thus far, two pounds of explosive per cubic yard of rock has been the average.

Eight construction men are employed in placing sets while the men are drilling and not using the buckets. In drilling, 7/8-in. hollow drill steel is used, and this is sharpened and shanked by machine.

Ventilation is secured by means of a 7½ hp. blower, which forces fresh air through a 16-in. Flexoid tube. With this equipment it is possible to return to the bottom of the shaft in less than ten minutes after blasting. Light is furnished at the bottom of the shaft by two factory domes, with four 100-watt lamps in each dome. The men are required to wear special hard hats, made

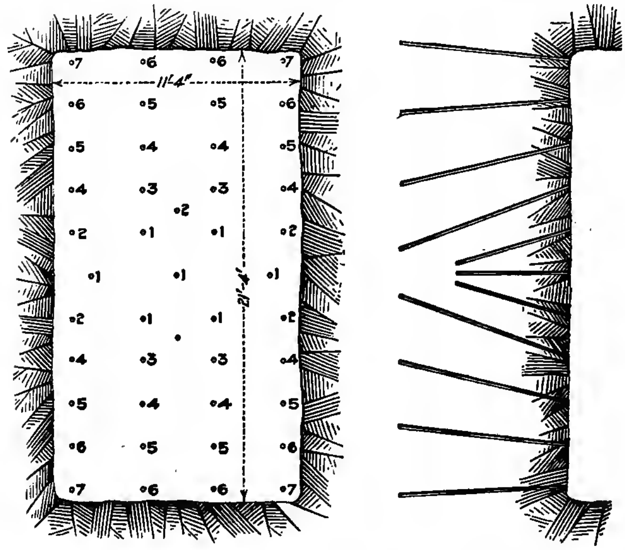

DIAGRAM SHOWING ORDER OF BLASTING AT SENECA SHAFT

of felt, treated with resin and shellac, which will resist a severe blow. Movable sollars, made of steel plates and operated by levers, are placed over the two compartments used for loading supplies in buckets. When the bucket hangs at the brace, the lever is thrown and the plates close in around the bucket, making practically a complete cover over the compartment. Electric signals are employed throughout.

Cement Gun in Mining Work Part I[1]

BY GEORGE S. RICE

The use underground of hydraulic cement and the scope of its uses increased almost as rapidly as in surface construction. One of the early underground applications was for the filling of the cavities behind the linings of tunnels, this being accomplished by pumping cement grout into the open spaces. In France in later years the "cementation process" of impregnating broken water-bearing ground by forcing in cement under high pressure was successfully employed, revolutionizing the method of shaft sinking through water-bearing chalks and marls in the north of France and Belgium. The employment of concrete for making massive linings of shafts and tunnels in place of timber framing and brick arching has been very extensive. The development of thinner, reinforced lining came into use later. In 1906 I water-proofed the air shaft of a mine in central Illinois by placing a thin but reinforced concrete lining, rectangular in section, inside a wood cribbing. The shaft passed through water-bearing ground.

About 1907 one of the large colliery companies of northern France, Mines de Béthune, began, in a new colliery, the use of a light, reinforced-concrete lining for all its cross-strata tunnels. When I visited this colliery in 1908, two miles of this lining had been constructed. It was only 5 or 6 in. thick, and the mixture was lean. Moreover the reinforcement was so very light that the cement work seemed to be hardly more than self-supporting and probably bore little of the weight of the encompassing strata. In other words, it apparently served merely to protect the strata from weathering.

The lining served its purpose satisfactorily, as I found out when I again inspected it three years later (1911), by which time more lining had been built. On the other hand, timbered tunnels in the same formation had given much trouble. This demonstration convinced me that the important factor was to protect the strata from the action of the air. The cost of the lining in the Béthune mine was relatively low under the conditions then prevailing in France, only $5 per lineal yard, which figure would have to be doubled if an estimate were to be made of the cost of such lining in this country, even prior to the war. The general method of lining seems an admirable one, although the first cost was considerable, but, in the long run, it should be economical because it eliminates the cost of timbering and of cleaning up of falls of roofs inseparable from the maintenance of a roadway and it also serves as a protection for the men against injury from roof and rib falls.

Concluding that the protection of the natural roof and ribs against weathering, a process for which oxidation is principally responsible, was probably the important factor in preventing subsequent falls, I studied the problem carefully in the hope of determining what sort of inexpensive coating might be applied to a shale roof to prevent its coming in contact with the air currents, with the hope of thereby saving expensive work later on, and, furthermore, of lessening accidents from falls.

In many mines, when the mine entries are first driven, the top or bony coal roof seems admirable. It appears as if it would stand forever; but in a year or even less the weathering action begins, falls occur and timbering and retimbering follow. Finally you find the roof has dropped until the entry is twice as high as it was originally, and more or less filled with timber. When this condition has to be met it involves a great annual expense for the maintenance of the roads, and should a fire start there is plenty of fuel to feed it. Also the timbers provide places for the lodgment of the dangerous coal dust which floats in the air. In this manner timbering increases the explosion hazard.

In 1910, when I saw the cement gun exhibited at the convention of the National Association of Cement Users, held in New York, I thought that here was an apparatus that would furnish the agency I sought for the placing of a more or less impervious coating on the mine walls. I made reference to its availability for that purpose in an address on concrete in mining (which appeared in the transactions of the 7th convention in 1911). After showing how much safer was a smooth, concrete lining as compared with timbering, I made the following statement, which appears to be as true now as then:

"The apparatus for applying cement mortar by means of compressed air, commonly known as the 'Cement Gun,' offers great possibility for the lining of passageways, etc., with cement. By protecting from weathering the roof and walls of a passageway with a thin coating of cement, it is possible that the heavy expense of timbering in many cases may be avoided. The machine also offers possibilities of use in the fireproofing of timber and board stoppings and in the erection of firewalls in places difficult of access, since the material can be pumped for a considerable distance."

For many years grout has been pumped into cavities behind tunnel and shaft linings. However, G. L. Prentiss, in a paper given at the meeting of the cement users above referred to, on the "Use of Compressed Air in Handling Mortars and Concrete," stated that compressed air was first used for the transporting agent in placing concrete and mortar in France, for the repairing of tunnels on the Paris-Lyons-Mediterranean Ry., France. This was in 1906. The linings of the tunnels were leaking and the arches were thereby becoming weakened. The engineers attacked the problem by using a machine consisting of a charging hopper connected with a pressure tank, into which air was forced by a compressor at a pressure of 40 to 50 lb. per sq.in. The tank was first charged with a grout composed of cement, sand

[1] Paper read before the Coal Mining Institute of America at its Pittsburgh, Penn., meeting, Dec. 6, 1917, and entitled "Weatherproofing Mine Roof and Walls and Making Tight Stoppings with Cement Gun." Reprinted from "Coal Age."

and water, and after the pressure had been put on the material was forced out through a line of flexible hose to a discharge nozzle, which was applied to holes drilled through the arch of the tunnel. It is stated the results from an engineering standpoint were satisfactory, but the machine gave trouble, as it and the hose became clogged with grout, and the process was therefore tedious and expensive.

Various other attempts were made to use compressed air for blowing cement or concrete mixtures through pipes, and according to Mr. Prentiss, J. W. Buzzell and W. H. Larkin, in 1909, undertook work in this country of that nature. They found that they could force a mixed concrete through a 4-in. pipe a distance of 400 ft. with a pressure of 40 to 50 lb. per sq.in., a higher pressure than this not being found advantageous.

The predecessor of the cement gun appears to be the long-used spraying device employed for the painting and whitewashing of rough work, like railroad freight cars. Here the paint or whitewash is ejected from a nozzle by means of compressed air. Nevertheless, this method, like those previously described, used the compressed air to transport the grouting, concrete, or whitewash in liquid mixture, whereas the cement gun operates on a different principle; namely, to transport the cement and sand dry, wetting it only at the nozzle. It is not, of course, possible to employ crushed stone in the mixture, which must be composed of cement and sand only.

As the cement gun is a patented medicine, I feel it is necessary to explain that I have never had any financial interest in it, and that I am concerned in it only as a means of obtaining a certain practical result in the mines.

Mr. Prentiss described in the paper, to which I have just referred, a variety of applications of compressed air to the placing of cement mixtures, some as early as 1910. The method was used for applying stucco and cement-sand mixtures; thus it was used for coating a frame building with stucco. Brick walls and fences were by this means covered with stucco or cement. He also instanced its use in the construction of the General Cement Products Co.'s buildings; the coating of structural steel, with and without the addition of reinforcing wire mesh; the fireproofing of the interior of wood-lined buildings; the building up of sections of cement pipe from a wire-mesh skeleton; the use for tree surgery; and the very important application of lining the iron syphons in the water supply system of New York. These syphons were 11 ft. 3 in. in diameter, and the total length to be treated was nearly 14,000 feet.

In that same 1910 meeting there was some talk of using the cement gun in mines, but it was with the idea of pumping concrete or cement into an excavation and filling it solid. In this manner it was suggested that a deposit of cement mortar might be placed which would later serve as a pillar. Men not familiar with mining thought that as concrete was so much stronger in compression than coal, no coal pillars would be required, if much smaller concrete pillars were substituted. Manifestly with the ordinary roof or floor the concrete blocks would be pushed into the roof or floor before the full strength of the concrete was developed, or else the roof spanning the excavated area would fail. So far as known, this futile project was never tried.

After seeing the demonstration of the gun in the exhibition of 1910, and the way the cement coating, sometimes termed "gunite," would stick to steel and stone, I concluded that here was a machine which offered a means of applying a thin cement coating to the roof and ribs of an entry or tunnel before weathering took place.

When the experimental mine of the United States Bureau of Mines, located near Bruceton, Penn., had been fully developed for its purposes, a cement gun was loaned in 1914 to the bureau by the Cement Gun Co., for trial applications. The manufacturers at that time, not having turned their attention to its mining possibilities, could not be induced to lend the gun without considerable negotiation. It was found, after experience had been gained in handling the gun, that a cement-sand covering, varying from a thin coating to one several inches thick, as desired, could be placed without difficulty on the coal ribs. It would even stick to the "draw slate" or clay band. Moreover, a thin to a ¾-in. coating could be made to stick to the roof without the use of wire mesh.

It was also found to be of great advantage in making stoppings practically air-tight. In October, 1914, when the American Institute of Mining Engineers met at Pittsburgh, an exhibit was given at the mine of the use of the cement gun for these purposes. The cement-sand lining has been extended in the mine entries from time to time until now nearly 1000 lin.ft. of entry has been thus lined, the roof coating being from ¼ to ¾ in. thick and the rib covering from 1 to 2 in. thick.

These coatings have been found very successful in the prevention of weathering on the sides and roof. It is true that from time to time repairs have had to be made on the roof coating, but this, in part at least, seems to have been due to the effect of the violent explosions produced in our experiments, of which over three hundred have been conducted. After many of these the coating would shell off the roof in patches, particularly where a void had developed behind the coating. These voids, or spaces, are caused by the concrete shell pulling away from the main roof where it is shaly, taking some scale with it. The concussion or rush of the explosive blast later knocks off the shell.

In offsets, where the work is protected from the explosions, the coating has rarely come off. As a result of these experiments it was thought that if the coating was well put on, particularly when the roof was fresh and sound, it would stick tight for a long period under ordinary mine conditions. Since the conditions at the experimental mine were abnormal, the failures in patches of the roof cannot be considered to condemn the system and there has been no trouble whatsoever from the coating on the ribs and draw-slate. This is most important, as those who have operated in the Pittsburgh beds well know. Usually disintegration which leads to falls in the roof comes about through draw-slate over the coal softening, thereby widening the roof span. Then the roof falls and the widening and cutting back goes on, making the timbering more and more expensive.

These tests at the experimental mine were brought to the attention of leading mining men, and this caused some of them to try experiments of their own. Trial of the method was made slowly at first, but the representatives of the bureau constantly urged further experiments as the change seemed to promise increased safety as a result of better lighting, the reduction of falls and the danger of coal dust explosion propagation. Now the development is proceeding more rapidly.

Fortunately the cement gun can be used for other purposes than to prevent the weathering of the mine roof. For example, it can be used in the fireproofing of wooden stoppings, the sealing off tightly of fires, something which much appealed to the metal mining companies, operating large bodies of rich sulphide ores, which are prone to spontaneous combustion in stopes. The cement gun is also most useful wherever iron or steel beams or posts are employed underground or in shafts, as it prevents the metal from rusting.

An excellent description of the cement gun was presented by Carl Weber in a paper before the Western Society of Engineers on Mar. 9, 1914. It described the gun as it appeared in earlier stages of its development. A good description also appeared in *Engineering* (London) for June, 1916. These articles show the remarkably wide adaptability of the cement gun to other uses than those already mentioned, such as the repairing of concrete walls in subways, the rehabilitation of retaining walls, the lining of reservoirs, the repairing or street-car tunnels and the lining of coal bunkers.

The cement gun (see outline drawing) consists of two hoppers, an upper and a lower one, the upper one being the receiving hopper for the mixture of sand and cement. This mixture is made on the ground and shoveled into the hopper in a dry state. The receiver had a cone stopper in the top and one for the discharge of material from the bottom, thus the feeding material is "air locked" into the lower hopper which is under air pressure of from 20 to 50 lb. In the bottom of this hopper there is a, feeding wheel, which is turned mechanically. As it revolves, it delivers the right quantity of cement material opposite the outlet. Here the pressure of the air in the lower hopper forces the measured material out. As the dry mixture is forced into the flexible hose, additional air is injected and this aids in carrying the material onward to the nozzle. The hose may be of considerable length — 50 or 100 ft. or even more. At the end of the discharge tube there is a nozzle where water is supplied at a pressure about 20 lb. per sq.in. higher than the pressure of the air. This is delivered in such a manner that it immediately mixes with the sand and cement. The nozzle is handled like a water hose and pointed approximately at right angles to the wall or surface to be covered. The sand is mixed with the cement in proportion of 3 to 1, but about 20 to 25 per cent. of the sand, after impinging on the wall, drops to the floor, so that the coating put on is about 2½ to 1. The surplus sand can be shoveled up and used again. It is surprisingly clean.

The wheel is turned by a worm which is driven by a small air motor. The compressed air and the water under pressure are usually furnished by the mine plant. but if the mine is not equipped it would be necessary to have a portable compressor and also a pump with a supply of water for the operation of the cement gun.

A. cross section of the machine is shown in Fig. 43. *A* and *B* are cone valves operated by control level *C* and *D*. *E* is the air-supply valve. The air-supply hose is connected at *F*. *G* is the supply cock for the air motor; *H*, an air pipe, controlled by cock *A*. The main air-supply pipe to the lower main hopper is controlled by the cock *K*. *L* is discharge tube of the

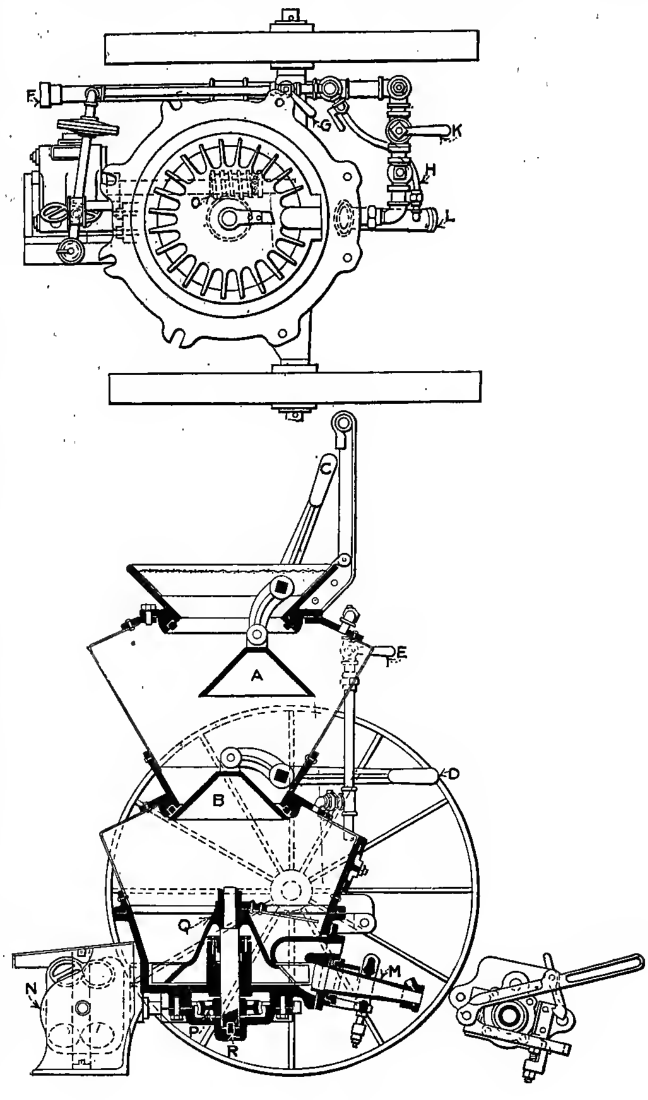

Fig. 43

machine and it is shown at *M*. The tube is flexible and may be throttled by the pinching action of a level shown in the small detail drawing. *N* is the air motor and *O*, a worm, driving through gear *P*, the feed wheel *Q*. *R* is the footstep of the vertical shaft of the rotating wheel. It is to be noted that the discharge tube *L* is of rubber and the hose is rubber lined, there being no metal exposed to the sand blast except a little at the nozzle.

According to Mr. Weber, the sand-cement material leaves the nozzle with a velocity of about 300 ft. per second. In striking a hard surface the sand rebounds and falls down, only the neat cement adhering until a coat is formed thick enough to hold the sand. Then the rebound lessens and the coat is built up until the desired thickness has been obtained. The tightness by which the sand-cement coating is placed by this method is of great value. A report was made by Westinghouse, Church, Kerr & Co., testing engineers, which was quoted at length in the paper just referred to. It bears testimony to the quality of the cement-sand coating produced by the cement gun. Their conclusions were:

In all of the tests made the products of the cement gun were shown to be superior to good hand-made products of the same kind. The degree of superiority varied between wide limits.

In tensile strength the gun work excelled hand work in every case by amounts ranging from 20 to 260 per cent.

In compressive strength the excellence of the gun work was even more marked, ranging from 20 to 720 per cent. better than hand work.

In the matter of surface permeability the gun work absorbed from 7-10 down to 1-20 as much water per hour, per unit of area as the similar hand-made surfaces.

As regards absorption of water, the hand-made mortars took up from 1.4 to 5.3 times as much as the gun-made mortars.

The percentage of voids of the gun-made product ranged from 52 to 75 per cent. of that in the hand-made product.

The adhesion of the gun-applied mortars was on an average of 27 per cent. better than that of the hand work.

I will not attempt to take up at length the many uses other than those already cited for which the method may be employed, except to quote from Mr. Weber's paper that a large reinforced-concrete power house with a chimney 150 ft. high was coated with a thin gunite coating, the chimney receiving the same coating as that used on the building, thus providing a uniform finish for the entire structure. There was no difficulty experienced in shooting up the sand-cement mixture through the hose to the nozzle in working on the high chimney, with the gun standing on the ground level.

The foregoing description refers to a specific machine. There are other machines which are said to accomplish similar results, one called the "Concrete Atomizer." These machines use a different principle, forcing out a wet or liquid mixture with steam or compressed air. A concrete atomizer was employed in 1914, on the Delaware, Lackawanna, & Western R.R., in the repair of honey-combed and cracked concrete piers, girders, beams and slabs. No doubt there are other workable machines, but I have only had experience with the cement gun. It is presumable that there be a great variety of forms which might meet the needs of the mining industry, but the essential requirement for that service is the delivery of the mixture under considerable pressure through a nozzle with a flexible hose which may be over 10 ft. long if desired.

Cement Gun in Mining Work Part II

BY GEORGE S. RICE

Following the first tests made in 1914 of cement coating with the gun at the experimental mine, I and my able assistant, the late L. M. Jones, began a campaign among mine operators to urge the trial of the method in mining operations. We advocated the application of a coating to the roof and sides to prevent weathering and also to render ventilating and fire stoppings air-tight. In regard to the latter, it frequently happens that while the stopping itself is tight the ground surrounding it is permeable. Often the air passes freely through the coal strata around the end of the stopping. This is largely prevented by the use of a closely adhering cement coating such as can be put on by the cement gun, not only adjacent to the stopping but for a number of feet back along the sides of the opening.

The matter progressed rather slowly for a time, but within the past year much testing has been done, and I have brief statements from a number of concerns which show the present widespread application. Among other things, the cement gun has been used for fire-proofing shafts which had wood linings, wire mesh being placed over the wood, such fireproofing or other alternative being required in Illinois.

In the fall of 1916, the writer, in visiting the Leonard mine of the Anaconda group at Butte, Mont., was struck with the difficulty in keeping the extensive fire areas sealed off. These fires were fed by timber and rich sulphide ores. I suggested to the superintendent who accompanied me the important advantages that might accrue from the use of the cement gun for the repeated coating of the stoppings. These had to be renewed from time to time because the heat would crack them. I suggested also that the cement gun could be used to advantage to seal the cracks in the adjacent broken ground. The patching work had been done by hand and was very laborious and difficult on account of the heat. The cement-gun method was tried and found to be of such an advantage that now nine guns have been purchased by the Anaconda company and, according to an article in the *Mining and Scientific Press* of July, 1917, the work is accomplished in one-fifth of the time required by hand work. The article states:

The finished work shows that the mixture of sand and cement has been shot into every crack and fold of the rock until it can hardly be distinguished from the rock itself. In some instances, on account of fumes, it was necessary for the nozzle-man to wear a compressed-air hood. . . .

In preparing the ground for these bulkheads, no other work than picking out the loose rock on the bottom, sides, and back was done. The loose rock in the bottom extended from 16 to 24 in. below the track level.

In some cases there were large loose boulders in the back which it was not feasible to remove. They were held in place by stulls, and cement was shot in the crevices between them. When this had set, the stulls were removed and the bulkheads built from the floor up to meet them.

In some cases, after the completion of these emergency-bulkheads, cement was shot on the entire back and sides of the drift on either side of the bulkhead for a distance of 16 to 18 ft. in length. This sealed all seams that were likely to bypass fumes around the bulkhead. Heavy blows will not cause this coating to sliver off nor to show any line of cleavage. The only result is a powdering of the cement directly under the hammer-head. Immediately above the passageway in these bulkheads, pipes were cemented in to allow of passing air, water, and electric wire through the bulkhead.

It is learned, through the courtesy of W. J. Richards, president, and Charles Enzian, mining engineer, that the Philadelphia & Reading Coal and Iron Co. now has eight guns distributed through the mines of that company. It is stated that

The use of the machines so far has been confined to lining tunnels, pointing and facing retaining walls and gob stoppings, lining reservoirs, etc. We hope to try fireproofing shafts, etc., as labor becomes available for such work.

Douglas Bunting, chief engineer of the Lehigh & Wilkes-Barre Coal Co., states that his company has used the gun for lining a gangway driven in top fireclay, also for coating a short tunnel in fireclay rock and for covering the underside of Hy-Rib used in connection with steel timbering. As yet, the experience has been too recent to determine how serviceable the gunite will be.

John G. Smyth, chief engineer of the Consolidation Coal Co., says that his company has used the cement gun in coatings headings at its mine No. 155, Van Lear, Ky., and at mine No. 125 at Somerset, Penn. In the latter mine there was a rock slope driven across the measures. It was first expected to use massive concrete archings to line these slopes, but later it was thought that if the action of the atmosphere could be kept away from the strata it might stand. In the main slope 675 lin.ft. were gunited, and in the manway slope, 300 lin.ft. Similar treatment was given to 1786 lin.ft. of air course driven through the coal bed, where 12 to 30 in. of slate of uncertain character were taken down. This work was begun in May, 1917, and continued at intervals to September, 1917.

Mr. Smyth states:

From our experience with gunite underground the serious question is the action of the gunite under atmospheric changes during winter and summer, and of course the job in question has not been in place long enough to pass any judgment on this feature. It may be found necessary to use a reinforcing wire mesh in the gunite to take care of the expansion and contraction. On account of the irregular nature of the side walls and roof the application of reinforcing will be tedious and the increase in labor cost will be quite an item.

The work in question was done in a very thorough, workmanlike manner. The conditions were such that we feel that this job will settle the question in our minds as to the permanency and value of gunite in underground work, particularly to protect the roof and side walls from disintegration.

The Consolidation Coal Co.'s first work of this character was done in November, 1916, in the coating of the roof and ribs of mine No. 155 at Van Lear, Ky. It is stated by Mr. Collier, of the Cement Gun Co., that, "the thin coating put on stood through the first year, then dropped off in patches, but the roof had not come down, while in the parallel entry without treatment the roof had slacked off to the amount of many carloads." He added that though some repairing of the patches in the cemented entry would need to be done, this would cost less by far than what was needed to repair the other entry.

The Clearfield Bituminous Coal Corporation, according to J. William Welter, chief engineer, purchased a cement gun, but have only just begun to use it. It is interesting to state the condition; namely, that the purpose was to place a coating on the exposed surface of a fire-clay roof. Mr. Welter writes:

> As is characteristic of fire clay, this material slacks when the air comes in contact with it. . . . It was necessary for us to make considerable preparation, such as removing all loose particles from the roof. . . .

The Rock Island Coal Mining Co. employed a cement gun in their No. 10 mine, Hartshorne, Okla. It is understood that 350 lin.ft. of entry was covered as an experiment, and that it has withstood the heavy slacking of the past summer.

As an offset to some of the promising results, one of the earlier tests, which followed the testing at the experimental mine, has recently been reported as unfavorable, although at first it looked as if it would be a success. This test was made by the H. C. Frick Coke Co. in 1914. The then chief engineer, J. P. K. Miller, wrote the author on Jan. 15, 1915, as follows:

> The cement gun has been used at two points near the shaft bottom and the superintendent is well pleased with the character of the work it is doing, which consists of coating the roof, after it has been properly treated. The work is being done with a view of preventing any further falling of roof. We intended at one time to arch the roof, but hope that now it will not be necessary.

Thomas W. Dawson, writing Dec. 1, 1917, says:

> The results [of the cement gun] were not satisfactory, but, in our opinion, this was due to the fact that in a number of places the covering of the cement applied was very thin; the whole covering being of variable thickness.

Mr. Dawson quotes from a report from Mr. S. Mack, the mine superintendent, that:

> For about two years it [the gunite] looked as if it was going to be very satisfactory. Shortly after that it began to crack off in small pieces, and I find that there are a number of places where it has spalled off where before it looked solid. It seems that the spalling starts where the cement is thin and then extends to the thicker parts.
>
> Of course this work is all experimental, but I believe that if all the roof that is loose is taken down and the space thoroughly washed and a coat not less than ¾-in. thick applied it will last for quite a long time, but it must be done right.

Mr. Dawson reported that the mine foreman advised that:

> After an interval of about one year it began to show signs of failure by some cracks appearing. He did not know whether these cracks were due to movement in the roof or sides or due to the slacking of the rock and coal, but when this covering was tested it sounded hollow, showing that the bond between the cement and the rock and coal had been broken so that it would seem that the failure was due to the slacking of the rock and coal behind the cement covering.

In spite of this unfavorable evidence, I do not feel that this testimony should be regarded as condemning such a coating. I do not know fully the conditions of this particular test, for instance whether the ribs were also covered. If they were not, then with the usual tendency of the coal of the Pittsburgh district to spall off and crack so that air is admitted, weathering might continue upward and over the cement coating. The test was a difficult one, as it is understood the entries were wide. One must also weigh the first cost plus the cost of annual repairs against the annual cost of timbering and cleaning up falls over a number of years.

Undoubtedly, if a roof is of a nature that it can be "brushed" so that its cross-section will be in the form of an arch it will assist in making the coating self-supporting, and as a final resort a light wire mesh resting in a shallow groove in either rib at the springing line of the arch and a heavier coating about 2 in. thick could be employed. This would make the coating self-supporting and it would not then have to be held up by the roof to which it is caused to adhere. The rib coating does not need reinforcement, so the final cost should not be great. In the long run it is believed that this additional cost would be justified, and certainly such work would be much cheaper than massive concrete lining or brick arching.

The Consolidation Coal Co. of Iowa, the Valley Camp Coal Co. of Pittsburgh, Penn., the Cambria Steel Co. of Johnstown, Penn., the Cameron Coal Co. of Marion, Ill., the New River Co. of MacDonald, W. Va., and many other companies, are all experimenting with the use of the gun in underground work. Therefore in the near future there will be available a large amount of information on the success or failure of weatherproofing roof by cement coating.

The cost of applying the cement-sand coating varies widely with conditions. It has been rather difficult to get figures. Some of the companies do not care to disclose them and others have not kept them in precise shape. In the first work done at the experimental mine the figures were as follows:

For coating 378 ft. of entry averaging 5.9 ft. in height and 9.15 ft. wide, the cement averaging about 2 in. in thickness on the ribs and ½ in. thick on the roof, the costs were as follows:

Labor and Repairs:		
Labor	$138.86	
Supervision	23.04	
Cleaning and repairing gun	10.74	$172.64
Material:		
Cement 280 sacks, or 70 bbl., at $1.10 per bbl.	77.00	
Sand 42 tons at 89c. per ton	37.38	114.38
Total		$287.02

The cost per lineal foot of entry averaged 76c., and the cost per square yard of surface averaged 32c. This job took ten days and the speed was 4.7 lin.ft. per hour, or 11¼ sq.yd. per working hour. In a subsequent job the total cost per lineal foot or entry was 93c. and the average cost per square yard was 40c.

Tested by the explosions in the experimental mine the coating on the ribs remained intact, but the roof coating shelled off from time to time. It did not always separate from the shaly material of the roof, for the two occasionally came down together. In these places and in other parts of the mine we attempted to get a stronger job by drilling holes in the roof as keys for the cement. At the same time we put in reinforcing wires. Also where the roof was naturally very weak some rails were put in for support. The total cost of repairing 6600 sq.ft. of roof and sides, including the railing reinforcement, was $617.95, or 85c. per sq.yd. The cost of applying the gunite averaged alone 42c. per square yard.

In a certain western mine the cost of the coating put on by the cement gun is given as follows: Labor, $21.50; and material, $17.80 for a total of 190 sq.yd., or 20.6c per sq.yd. This cost is very low and the figures cover apparently only the application, the coating being ½ in. thick.

One of the companies in Pennsylvania reports that the actual cost was $3 per lineal foot of heading. The section of heading averaged 22 sq.ft., making the cost about 13.6c. per sq.ft., or $1.22 per sq.yd., of gunite deposited. This cost is based on cement at about $2.20 per bbl. and sand at $2 per ton at the site. The thickness was about 1 in., no reinforcement being used, but it was most carefully done and the estimate includes all costs.

At a mine in the Connellsville district, the cost of cementing 5229 sq.ft., which was accomplished in 12 days, was as follows per day:

One demonstrator	$2.86
One nozzle man	2.85
One machine tender	2.15
Two laborers at $2.15	4.30
One-half time teamster at $2.00	1.00
Total	$13.16

The total labor cost was $154.02.

Material:	
325 sacks cement	$125.12
600 bu. sand	36.00
50 ft. mining machine hose	17.50
Total	$178.62
Grand total	$332.64

The coal face as shown on the left, like all natural faces, is provided with ledges on which coal dust can accumulate. On the extreme right is the gunited coal, without ledges, with the insulated wire covered and so protected from injury. Note also how the gunite aids to lighten the heading.

In this instance the roof and the floor near a massive concrete stopping has been carefully gunited. This prevents the air from leaking through the rib or the roof to the area beyond. In case of a mine fire this is a most valuable feature and assures a more rapid extinction of the conflagration.

The cost per square foot was therefore 6.38c., or per square yard 57c. It will be observed that these figures vary widely, but it is thought that under average conditions, with men fully trained in the use of the cement gun, the work can be done for at least 50c. per sq.yd. on the basis of wages prevailing prior to 1917.

It may be pointed out that the gunite not only protects the roof and sides from weathering, but also assists in the illumination of the passageways by reason of its light color. It also lessens the hazard of coal-dust explosions.

The ordinary timbered entry has innumerable projections and there are many recesses on the coal ribs which serve as places on which coal dust may collect in dangerous quantities. Comparatively little dust can collect on the surface of a cement-coated heading and what does collect can be readily be washed down without damage to the roof or ribs.

The conclusions are:

1. By compressed air a cement-sand coating can be successfully applied to the weak roofs of entries and tunnels in coal mines.

2. By the use of the above means the ribs and a weak roof in coal mines promise to be satisfactorily weatherproofed.

3. Steel timbers in entries and shafts can be protected from rusting and wood timbers fireproofed.

4. The method is of the greatest advantage in fireproofing wood overcasts and stoppings and in making them air-tight. By facing gob stoppings with this material they can be made tight.

5. Fire stoppings can be made gas-tight and are quickly built and repaired.

The last four purposes alone justify the employment of the cement gun method, regardless of what machine is used with the accomplishment.

Use of the Cement Gun in the Cœur d'Alene Mining District

The Hercules Mining Co., at both the Burke and Wallace mines, has made use of the cement gun in recent improvements, with a saving of time and money. The canyon at Burke is very narrow, and in order to provide ample tailing and waste dumps, the management has found it necessary to build high cribbing to impound the waste. The machine shops, sorting plants and other structures have been built on top of the spoil banks, so that if the cribbing should be burned these buildings would be destroyed and there would be a greater loss in the closing of the canyon. It was therefore determined to fireproof the cribbing by covering it with mortar. On the face of the cribbing small furring blocks were first placed to insure that the wire mesh would clear the face of the timbers, and to these blocks the wire mesh was attached, care being taken to cut an opening at the end of each tie log, in order to secure comparatively true planes. To the wires, projecting at these logs, were fastened small cap pieces of reinforcement. All of the mesh was firmly stapled to the logs and furring blocks. Between the logs, the voids in the back fill were of considerable extent, but it was deemed advisable to attempt to fill them with the cement gun, using a mixture of four parts sand to one part cement, and thus obtain practically a retaining wall with the face timbers fully incased. The face was finished off with about ¾ in. of mortar.

At Burke the gun was next used in the construction of a reservoir. In a gap about 250 ft. above the level of the portal of the lower tunnel an excavation about 26 ft. in diameter was made in the rock. Across the outlet end a concrete wall 26 ft. high was built, after which the upstream face of this wall, as well as the entire surface of the bowl, was covered with about 3 in. of gunite. The mixture used was one of cement to four of sand, with an addition of about 10% hydrated lime.

No attempt was made to obtain a smooth surface, the rock contour being followed. The reservoir, holding about 190,000 gal., has never shown any leak except a small one where the outlet pipe passed through the wall, and this was probably due to the fact that no flange was shot around the end of the pipe. The roof of a rope house was next covered. This roof was 22 ft. wide and 102 ft. long, and was slightly arched. Sheeting was nailed to rafters having 18 in. centers and light tar paper placed on top. The ⅜-in. furring strips were tacked on and on these 1-in. mesh chicken netting. A one to four mixture was shot on to a depth of 1¼ in. The work was completed in two days. Heavy rains and 14 in. of snow have developed no leaks.

The walls and the roof of a three-story hall and clubhouse was next treated. This building is 70 x 88 ft. and 55 ft. high. The studding consisted of 2 x 6 in. planks spaced 16 in. centers. On the outer face of these was tacked building paper and on top of this a light tar paper. Three-eighths inch furring strips were then nailed on, and on these expanded metal lath. The walls, the coping and the tower (amounting in all to about 20,000 sq.ft.), were shot to a thickness of 1¼ in., and the concrete basement walls given a stucco coat, in eleven days. No hand finishing whatever was done on the walls, the entire slab being shot on at one application. The finished surface approximates a true plane. These walls have been completed three months, and only one small crack has occurred. The roof of the building, an area of over 6000 sq.ft., was shot to a thickness of 1¼ in. in three days. The roof was concave. The sections which slope to a central drainage point were arched 5 in. to take care of expansion. The construction consists of 2 x 8-in. planks spaced 16-in. centers and covered with shiplap, on top of which tar paper was placed. Three-eighths-inch furring strips were then nailed on and 1-in. chicken netting stretched tight and fastened to them. One and one-quarter inches of gunite was shot on, a one-to-four mixture being used with an addition of 10% hydrated lime. After this had set, a coat of Flotine paint was applied. Melting snow has shown one small leak around a vent pipe, but this was easily stopped. This leak was probably caused by a heavy rain falling the night after the roof had been shot. The wooden lath having been purchased before the gun was decided on, the inside of the building was hand plastered. Two of the best hand-plasterers obtainable, with a crew of one mud man and three helpers, or a total of six men, placing plaster ⅜ in. thick, did not cover as much as was done on the outside walls by a crew of five men with the cement gun, placing material 1¼ in. thick.

Studding was placed between the posts, in front of a large ore bin, and on this old boards were nailed with 3-in. furring strips, to which was nailed diamond-mesh expanded metal lath. Gunite, 1¼ in. thick, was shot on at one application, no hand work being done. This surface is 77 x 36 ft., and was completed in 1½ days at considerably less cost than galvanized iron, and presented a neater and superior wall. The same company completed at the Hercules Mine, at Wallace, last fall, a tank 60 ft. in diameter for the Franz thickener. The bottom of this tank was lightly concave in shape, and was built of hand-placed concrete. A tunnel, through which the thickened product is drawn off, extends underneath the bottom from the center of the tank to a point outside. On the radius of the walls of the tank ⅝ in. square twisted rods spaced about 4 ft. on centers were placed upright in the foundation concrete, extending about 8 ft. above the foundation. The upper three inches of these rods were bent at right angles. Three-quarter inch square twisted rods, bent on a roller to a radius of 30 ft., were then wired to the vertical rods, the spacing ranging from 3 in. at the bottom to 10 in. at the top. The walls of the tank had to be strong enough to bear the weight and to withstand the vibration of the mechanism, which consisted of a 16-in. worm rotating at 12 r.p.m., the actuating mechanism being a 5 hp. motor and the necessary gears mounted on a truck which runs on rails on top of the walls. It was therefore deemed advisable to make the walls 6 in. thick.

On top of the rods two layers of reinforcing mesh were placed. Templates, cut from planks to a radius of 30 ft., were attached to the reinforcing 10 in. below the top, and held in place by brackets made of 1½ x ¼-in. strap iron. The brackets were fastened to the boards with stove bolts and were wired to the rods. The lower end of the brackets and the outer edge of the new boards were afterward imbedded in the wall, and to the inner edge the wooden overflow lip was nailed. In shooting the walls of this tank, a canvas form was used. The canvas was first stretched tightly over the outside of the reinforcement, and wired in place, after which it was thoroughly wet to further tighten it. The gunite, consisting of one-to-three mortar with 10% hydrated lime, was then shot from the inside to a thickness of about 1½ in. After this had set, the canvas was removed and the wall completed by shooting layers from both sides, thereby producing a wall 6 in. thick, with reinforcement in the center. After the sidewalls were completed a layer of triangle-mesh reinforcement was placed on the bottom, and a coat of gunite shot over it, neatly finishing the work at the walls. Water was turned into the tank within two weeks after completion; it has been kept full ever since, and as yet no sign of a leak has developed.

J. T. Torkleson, construction superintendent of the Hecla Mining Co., where the cement gun was also used, stated that the work cost about two-thirds as much as hand plaster and was superior to the latter product. A comparison was made between hand plasterers covering the poured walls and the cement gun on the same work. A wall 105 x 17 ft. was covered ¼ in thick by hand and cost $75. The same amount applied with the cement gun cost $25, or 0.042c. per sq.ft. by hand against .014c. per sq.ft. by the gun.

Sampling of Mine Floors

BY ALBERT G. WOLF

Sampling mine floors is hard and unpleasant work, and because of inaccuracies involved, is to be avoided where the necessary samples can be obtained from the walls or roof. In floor sampling it is difficult to catch all the fines and keep the sample free from mud or other floor material, to which is added the impossibility of inspecting the vein carefully before sampling without, in some cases, a prohibitive amount of preliminary work. Further difficulties not involved in sampling walls or roof are: Diversion of water flow, removal of loose floor material, tearing up and relaying of track, and sometimes the additional hardships attendant on the necessity of doing the work without interference with regular mine operations.

Sometimes, however, it is absolutely necessary to sample mine floors; for example, on a level where the ore is stoped out above; where the drift is tightly timbered; in the floors of underhand stopes; in the bottoms of winzes and shafts, and on outcrops. As to the probability of inaccurate results, this disadvantage is counterbalanced somewhat by the fact that generally the samples are taken from places in the mine where only "probable ore" could be estimated, and not "proved ore," "positive ore," or "ore blocked out," and due allowance must be made in basing any estimate upon such samples.

Workings to be sampled may be classified under two heads, dry and wet. The difficulties and inaccuracies involved in the latter are much greater than in dry mines. By the term "sampling," is meant cutting the ore by hammer and moil.

In dry mines, the chief difficulties are those arising from catching flying sample particles, removing track and loose floor material and, in a mine in operation, conducting the floor sampling without interfering with regular work. The only chances for error involved in sampling a dry floor are in catching all the flying particles, gathering the fines from the bottom of the cut, and preventing extraneous floor material from rolling into it. Catching flying particles is not nearly so difficult in roof or wall sampling, as the particles fly in a general downward direction and can be caught in a box or canvas basket held close to the point of cutting, or by a large canvas spread on the floor; although with the best precautions many pieces fly astray. In sampling floors, the chips fly upward and outward in every direction. To catch these, it is best to place one canvas across the line of cut, just ahead of the point of cutting, and one on each side of the cut. If a deep preparatory trench has to be cut through loose muck on the floor, it should be made wide enough to keep extraneous material from rolling into the sample groove, and the canvas should be pressed down into the trench ahead of the point of cutting. A box held a foot or so above the moil and just ahead of that point, will act as a baffle, preventing the particles from flying too far and causing them to fall back upon the canvas. When one-half the sample has been taken, the canvas should be laid on the other side, and the second half of the cut made, always keeping the canvas close up to the point of cutting. A stiff brush will be necessary to aid in gathering up all the fines.

In addition to the difficulties already enumerated, in wet mines mud and water must be overcome. In the first place, the main flow of water must be bypassed, so as to make sampling at all possible, and secondly, seepage must be minimized in order not to contaminate the sample with mud.

Obviously, in sampling the floor of a wet mine, handling the water is the chief problem, and the accuracy of the work will depend upon the precautions taken. In the preliminary sampling of a small property, it would be too costly to take all the precautions that might be enumerated. In such a case, where the vein is narrower than the drift, the water can be run through a ditch at one side of the drift, or if the vein is too wide to permit this, the water can be passed across each trench in the loose floor muck by means of a launder or piece of air pipe a few feet in length. In this way, by bailing out the seepage a few times while moiling is in progress, it is possible to get a sample with the least expenditure of time and effort. The sample, it is true, will be somewhat subject to inaccuracies, either on account of mud carried into the cuttings, or, if the cuttings are washed free from mud, by the loss of fines. This is a makeshift method, and justifiable only on a preliminary examination where the necessity for greatest accuracy does not warrant expenditure of more time and money.

In more important examinations, greater care must be taken. If the water in a drift comes from one or a few chief sources, down certain raises or chutes, for example, it can be carried past the working point in launders or pipes. If this is done a day or so before the sampling is started, the loose floor material will have drained to such an extent that seepage into the trenches will be little. On the other hand, if the water comes from a general percolation through the vein, the use of dams must be resorted to in order to collect it, and launders or pipes used to carry off the water from behind the dams. If a level is extremely wet, a dam will have to be built for each short length of drift sampled. For example, beginning at the lowest point of a level, which would be (or should be) where it connects with a shaft, crosscut, or other main working, a dam should be built across the drift about 100 ft. distant. This dam can be easily made with a couple of 2 x 12-in. planks, superimposed edgewise, and fitting snugly against the floor and walls. It should be made as nearly watertight as possible by calking, or tamped with clay, cement, talc or any fine-grained impervious material. The launder or pipe should lead from the upper half. The dam should be only high enough to give a slight grade to the launder with enough elevation to carry it over irregularities in the floor. If the dam is too high, the hydraulic pressure will be such as to cause water to leak through so crude a structure.

With the dam in place, the main flow of water is cut off, and trench digging and sample cutting can proceed. If the inflow of water along the vein is large, it may be necessary to have the dam quite close to the point of cutting, for otherwise the trench may fill with water so fast as to require constant bailing. The seepage into a trench may sometimes be greatly lessened by digging auxiliary trenches above the sample trench, and bailing from them. In some cases, the removal of all loose material, instead of just digging trenches, may be necessary to prevent too great an infilteration of mud into the sample cut.

The methods of conducting water flow are by pipe and flume. Each system possesses advantages not to be found in the other, the greatest advantage being in favor of the flume, especially under certain conditions, which will be described. Light, galvanized-iron air pipe is easier to handle and put together, and in a straight drift with no branch openings conducting tributary water, is to be preferred to a flume, if the material is immediately available. The flume, while heavier than the pipe, and more trouble to put together, has the following advantages: It can be cut up into short lengths in order to make turns; it is open at the top and can be bailed into, which is a great advantage when more than one cut is being made at the same time, as the bailed water is conducted away instead of running down the next lower cut; and if a crosscut is adding water to the flow in the drift, a second cut can be built across it at the intersection, and a short flume used to carry the water collected into the main flume.

Catching the cuttings is the chief difficulty in sampling wet floors. A canvas cannot be spread over the floor and down into the sample trench as with dry floors, because the seepage would carry upon it a great deal of mud. Some engineers suggest washing the sample to eliminate this infiltrated mud, but that would cause a loss of fines from the cuttings as well. The best way to catch the cuttings is to hold a canvas sampling bucket, or a conical "butterfly-net" sample catcher, close to the bottom of the trench and a few inches in advance of the moil point, with the opening of the receptacle in a vertical position. The circular shape of the bucket opening permits either to get closer to the floor in the bottom of the trench than is possible with a box.

Two other errors are involved in sampling wet floors. One, the obscuring of the work by dirty seepage, making it impossible to be sure that a uniform cut is being taken; the other, the concentration of rich fines into floor cracks by the action of running water, causing high assay samples. The first cause of trouble has been dealt with; the second can be guarded against by making a preliminary cut across the vein and rejecting the material before making the regular sample cut. If error from this source is expected, the rejects from a few preliminary cuts should be assayed with the regular sample cuts as a check or confirmation, before going to the trouble of making further double cuts.

Although floor sampling, where it is possible to take roof or wall samples, is not to be recommended, frequently it is advisable at least to examine the floors as a precaution against estimating ore where there may have been underhand stoping, especially if it is an old mine that has been badly gouged. For example, at a certain mine that was examined, the maps showed no stoping below the lower tunnel level. An oreshoot several hundred feet long had been opened on this level a number of years previously. Before any appreciable amount of stoping had been done, a report was made on the property by an engineer of prominence. Subsequent to that examination, considerable ore was stoped above the level, all of which was shown on the maps submitted by the owners. Most of the stopes were accessible, and the backs were sampled. Now, as the drift had been sampled by an engineer of unquestioned ability and reputation before stoping had started, it would appear safe to take his results in estimating the grade of probable ore below the drift. However, it was decided to take some floor samples at rather wide but regular intervals, as a check, even though the difficulties were great on account of a heavy flow of water along the vein and much loose material on the floor.

At one part of the drift the depth of muck grew steadily greater up to 3 ft., an extraordinary amount of filling. At this point, when no bottom was struck at a depth of 3 ft., a long drill was driven down into the loose material without touching bottom. Evidently there had been an underhand stope, the existence of which had not been mentioned by the owners or suspected by the examining engineer. The length of this stope, which proved to be of considerable extent, was determined by digging pits followed by driving down long drills. The depth could not be determined without hoisting a large quantity of material, which was out of the question in an examination of that nature. From either end of the drift the samples showed increasing assay values up to the point where the stope started, so that it was evident a high-grade shoot had been gouged. The uncertainty of the depth of this stope and the secrecy maintained regarding its existence manifestly reduced the value of the property, its value becoming dependent largely upon the possibility of opening other orebodies below the existing level by a new adit.

Ore Car Designed at Hecla Mine

BY C. T. RICE

The best car for train haulage at shaft mines that I have ever seen is the one designed by J. B. Sloan, master mechanic, and C. H. Foreman, engineer, of the Hecla Mining Co., which has now been in use at the Hecla mine, Burke, Idaho, for several months. This car not only has a capacity of 82 cu.ft., and holds approximately 5½ tons of Hecla ore, but is so built that it can readily be taken down a 4 x 5-ft. shaft compartment, and yet will easily go around a 28-ft. curve on a 24-in. track.

The car is of entirely new design. The body, instead of being carried on longitudinal sills with the trucks fastened to the latter, is mounted, so as to lessen the height, directly upon the wheel trucks by means of truck plates. Extraordinary flexibility, though the car is about 10 ft. long and 24-in. gage, is obtained by using a bolster and turntable truck at each end. This, together with the use of 14-in. wheels, makes it possible to keep the top of the car 4 ft. 8 in. from the rails, and prevents the car from being top-heavy. By a slight modification of the truck and the use of 12-in. wheels, the same body is used on an 18-in. track and works equally well.

Owing to the absence of longitudinal sills, the car body is reinforced in several ways to prevent telescoping in case of wreck. This reinforcement comes partly from a 2 x 2 x ⅜-in. angle iron top rim of the body, partly from 1¼-in. bumper rods and 2½ x 2½ x ⅜-in. angle irons riveted to the side plates of the body to carry the hinges of the bottom doors, but mainly from the 3-in. lapping of the top and bottom side plates upon each other about one-third way up the car body.

In lowering the car down a shaft, the body is dismounted from the trucks, which are placed on the cage. Suspended from a yoke by chains, it is then swung under the cage and slowly lowered in that manner. The dismounting of the car body is simple. The cotters are removed from the king pins that hold the trucks and body together, and the body is lifted off. The assembling is almost as easy. When the level is reached, the trucks are properly positioned on the turn sheets, and the car body is swung into place on them as the cage is slowly lowered. By doing away with sills, full advantage is taken of free space in the shaft to obtain car capacity.

The bottom doors of the car, as shown in Fig. 1, are closed by means of chains passing around a ratchet shaft. A quick discharge of ore is obtained by pulling the locking arm and releasing the ratchet wheel. A keep on the chains stops the doors when they are almost flush with the inside faces of the rails. In this way the ore, as it discharges, is kept from getting on the track and causing subsequent derailment.

By using four wheels on each truck and cottering the king pins so that the bolsters have a half-inch vertical play on the turntables of the trucks, enough flexibility is provided between body and truck to make the car ride well even when the track is very uneven. Consequently the car is not easily derailed. This is important, as the car weighs, when loaded, about eight tons, and when empty, about 2½ tons. But compared to its capacity the dead weight is not large for a mine car intended for train haulage.

In building the car no machining of parts is required, as trucks and bolsters are rough-cast and used in that form, being bolted together with two-ply belting between to insure a tight fit. Moreover, except for the wood blocks that are put between the truck plates and the bumping blocks to act as shock absorbers, all work on the car is ordinary blacksmithing. True, the end plates and the bottom side plates of the car body must be bent to shape, but this is easily accomplished in any shop having power shears, such as the 26-in. Improved Doty, with the aid of a few special tools that I will describe later.

The body of the car is made up mainly of ¼-in. plates. Each end plate is a single piece bent to shape. Each side consists of a bottom and top side plate lapped and riveted together so as to obtain a double thickness for stiffening the car body lengthwise and to enable ordinary sizes of plate to be used. The side and end plates are brought together by means of 3 x 3 x ⅜-in. angle irons. Ninety-degree angle irons are used for fastening the end and top side plates together. Special angles are needed to join the end plates and the bottom-side plates. All rivets are driven hot.

The hinges for carrying the bottom doors are bolted to angle irons riveted to the side plates of the car body near their bottom edge. The doors consist of 5/16-in. plates reinforced by riveting pieces of angle irons along both edges. The outer piece carries the hinges, and the inner one, with the aid of short lengths of angle iron, as shown in Fig. 1, carries the wheels over which the door chains run. Straps over the wheels hold the chains in position. At one end the chains bolt to the truck plate. At the other they pass through holes in the truck plate and bolt to the chain shaft, which is carried on the end plate.

The only thing to be noted in the design of the turntable truck is that the outer ribs for catching the bolster plate, as well as the king-pin boss, are made ¼ in. lower than the turntable ring. This enables the bolster to rock on the turntable, the king pin being cottered to give ½-in. vertical play between the two plates, just enough to give flexibility between car body and truck, and yet keep the turntable ring of the bolster from jumping out of its seat on the truck.

The bolsters of the car shown in the drawing, which is made for a 24-in. track, are bolted crossways to the truck plates. But when the car is to run on an 18-in. track, the bolsters are bolted lengthwise to the truck plates with 2-in. plates between the wheel housings and the turntable plates, and 12-in. wheels are used, their tops coming flush with the top of the trucks. This enables the trucks to swing under the bolsters on sharp turns. Slight recesses for receiving the wheels are also ground in each truck.

The bending of the end and bottom-side plates is done in a 26-in. improved Doty power shears by means of specially designed forming tools that are bolted to its jaws. The fuller tool used is shown in Fig. 45.

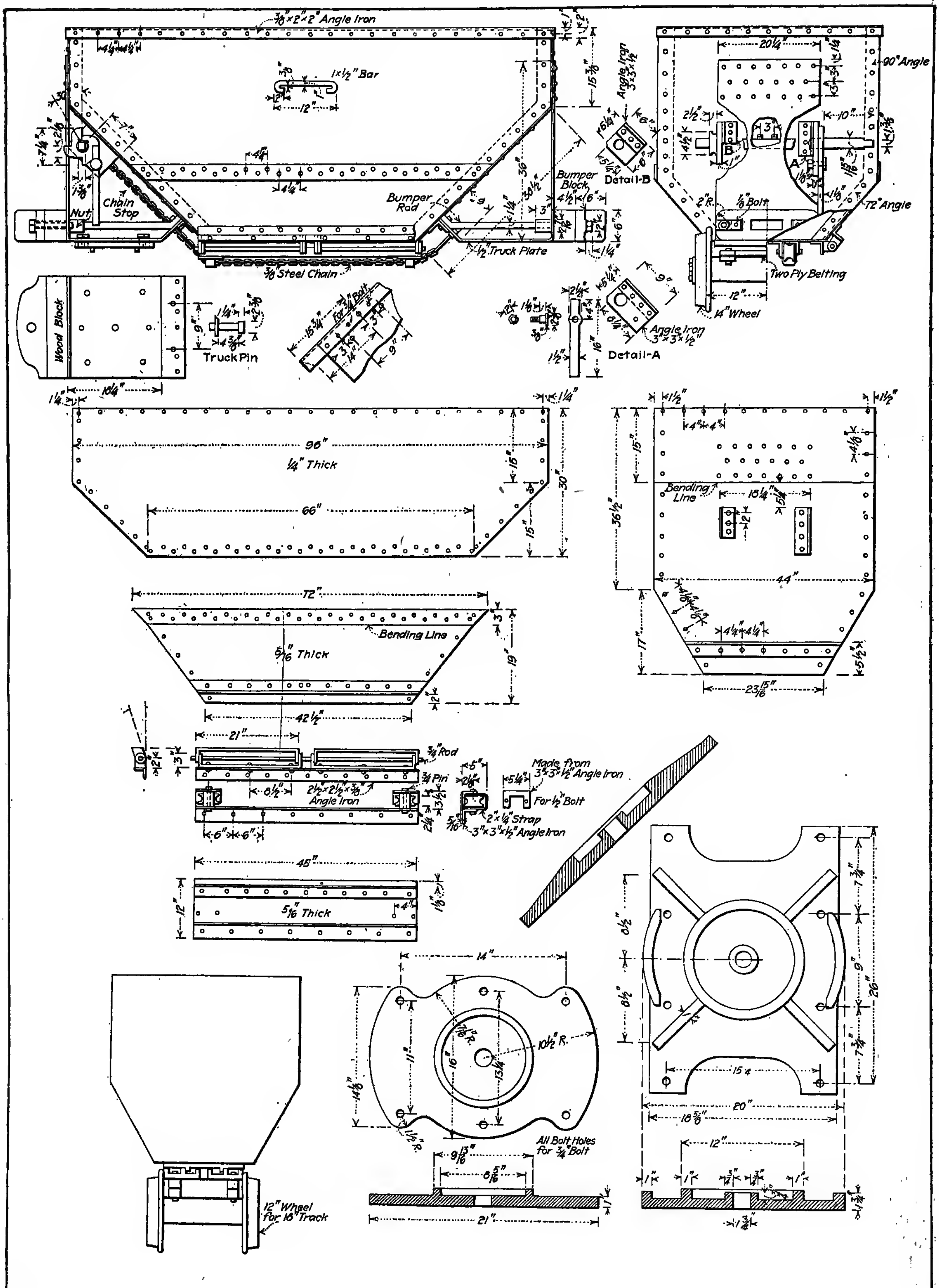

FIG. 44. DETAILS OF ORE CAR DEVELOPED AT THE HECLA MINE, BURKE, IDAHO
The car has a capacity of 82 cu. ft., but can be taken down a 4 x 5-ft. shaft compartment. It will take a 28-ft. curve on a 24-in. track

When bending the plate, this fuller is bolted lengthwise to the upper jaw of shears; on the other hand, when taking out the fluting that develops in the plates while the bend is being formed, it is fastened at right angles to the jaw. In the same cut are shown the bending and straightening blocks, which are used by bolting them to the lower jaw of the shears. As the jaws of the 26-in. Doty shears that are used have a maximum movement of only 1¼ in., the bending block used is made in two parts, *A* and *B*, to make it possible to get sufficient bend in the plates. First the plate is bent as much as it can be, using block *A*, the bending block proper. Then block *B*, the shimming block, is put under block *A* to raise it higher with respect to the jaws, and the desired bend in the plate is finally developed. The first bend in the plate is, of course, carried clear across from side to side before the shimming block is put under block *A*.

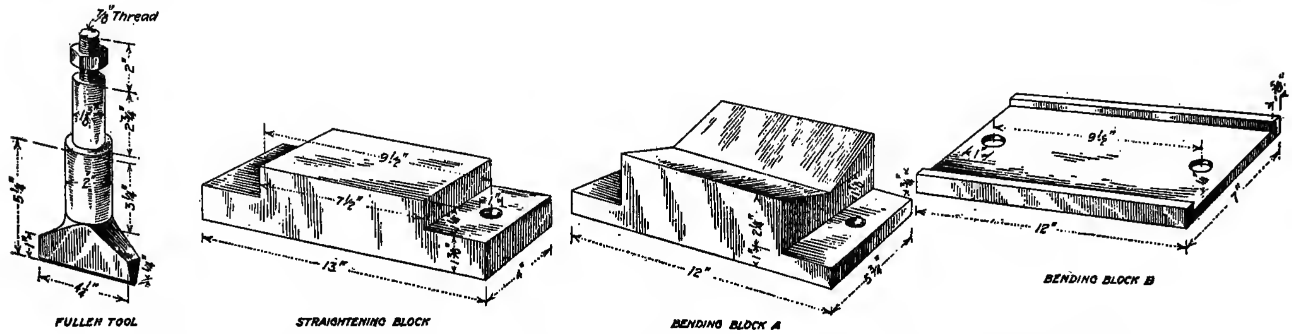

Fig. 45. Tools Used for Bending End and Bottom-Side Plates with Power Shears Employed

In such a makeshift method of bending the plates with the machinery available, a fluting tends to develop in them. This is taken out by replacing the bending block with the straightening block and turning the fuller tool at right angles to its former position. Then the plate is worked back and forth under the jaw, and the fluting worked out. The wings often tend to warp or bend at the point where they overhang the bending block, and this kinking is also worked out of the plate at the same time that the fluting is removed.

The cars described are run in trains of seven or eight at the Hecla mine. The last car of the train is therefore provided with a brake at its rear end, which is of rather clever design. It consists of a carrier plate *A*, Fig. 46, that bolts under the rear truck with a piece of two-ply belting between. One arm of this carrier plate is bored to receive the 2-in. brake shaft. The other wing is bored to receive the 3-in. collar of the foot lever *B*, which is keyed to the brake shaft *C* in such a way as to be positioned properly for being carried by the carrier plate with minimum overhang. The brake shaft is turned down at each end to form a 1-in. arm, having an eccentricity of ½ in., for carrying the brake rods. As the foot lever is 18 in. long, a strong braking action is obtained. The brake rods *D* are cottered on the eccentric arms of the brake shaft Each rod screws into a nut carried in a square recess cast in the brake block *E*. These blocks, which are made of soft cast iron to permit a good braking action on the manganese wheels of the car, have two brake faces, so that all four wheels of the truck are braked at the same time. By screwing up the

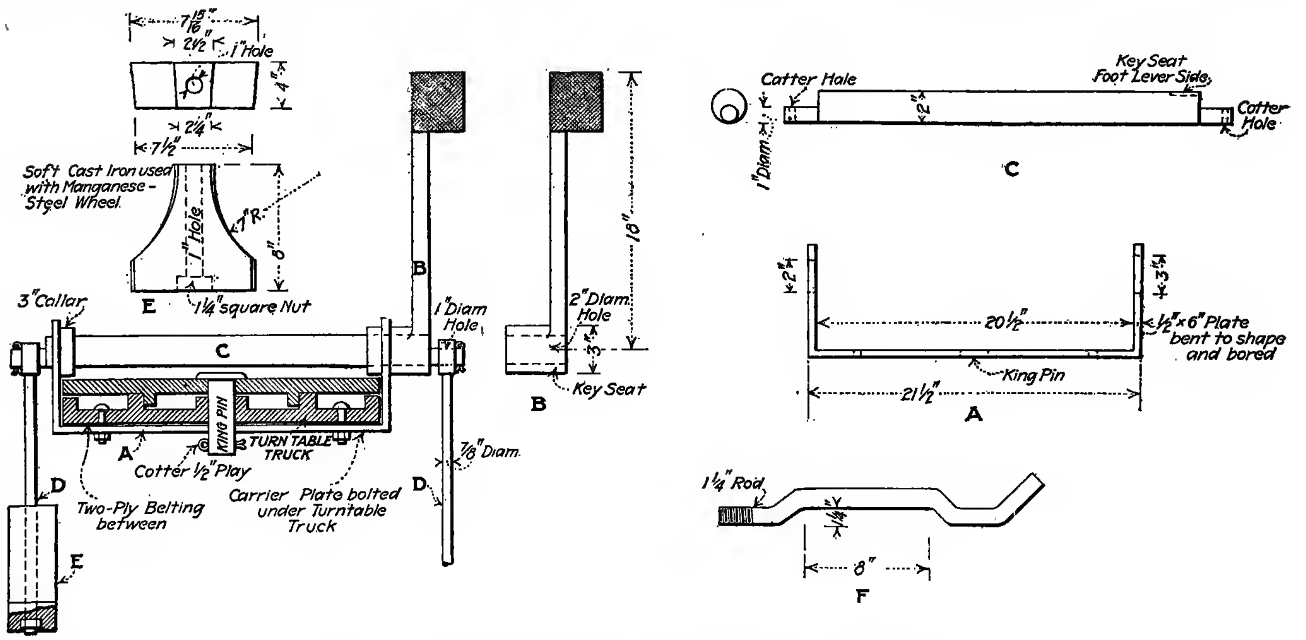

Fig. 46. Details of Specially Designed Brake for Hecla Ore Car
A — Carrier plate. B — Foot lever. C — Brake shaft. D — Brake rod. E — Brake block. F — Bumper rod on brake car as bent to clear brake shaft

brake rods, wear is taken care of in the brake blocks.

In order to permit the brake truck to swing sufficiently in making turns, the bolster at that end of the car is machined down to 20 in., and the carrier arms are kept as far apart as possible; for as the carrier plate is made of ½-in. iron, little play would be left to the truck on a 24-in. track if the bolster remained full width.

The carrier arms are bored so as to carry the brake shaft just clear of the top face of the truck plate of the car. This necessitates a bending of the bumper rods at that end, as shown at *F* in Fig. 46, so that they will clear the brake shaft. In order to give the truck ample play, this offset in the bumper rods is made 8 in. long.

The advantages of a bottom-dump car, having a capacity of 82 cu.ft., for underground train haulage at shaft mines are so many and so obvious that it is unnecessary to enumerate them. When the car is so simple in its design and construction and is so carried on its trucks that it can go around curves of 28-ft. radius, as is possible with the Hecla car, it will be apparent that an almost ideal car for train haulage has been developed. Moreover, the car runs easily and smoothly, as it is fitted with Taylor-Hyatt roller-bearing axles, and it should find favor at larger shaft mines where electric haulage is employed, as its use will reduce the man-power required to handle the ore, as well as the cost of haulage. At a time of labor shortage such as now confronts the mining industry, this is no small recommendation.

Mine Management

BY HENRY M. ADKINSON

It is a truism dangerously close to a bromide to say that never in our country's history was there a time when the need for efficiency of effort and the necessity of conservation of time and material were so impelling as now. All must by this time appreciate the need for the highest degree of personal effort and community coöperation, but when, in a desire to "do his bit," one tries to lend a hand in his particular field he seems fairly to be groping in the dark and to be accomplishing nothing more than before this urgency existed. When one was doing all that he possibly could, now that conditions require added effort it is not apparent just how exertion may be intensified and the results increased.

The proper direction of a business — its management — is the most important factor in any industry. If this one factor can be stimulated through conscience excitation of the mental qualities which go to make up the successful manager, the whole business must necessarily respond to the spur. The importance of effective management cannot well be overestimated. This is not a theoretical statement but is intensely and essentially practical — how much so can be realized from the two quotations following, from competent authorities on the subject. Nothing could be more matter-of-fact nor have a more direct bearing on our day-to-day problems than the *Monthly Bulletin* issued by the National City Bank of New York. In the July number there appears this comment on "Industrial Efficiency":

> A well-known engineer who is an expert in industrial practice has recently gone on record with the opinion that the industries of the United States on an average are not realizing over 20% of their possible capacity. . . . He says that we have scarcely begun to appreciate the gains that are possible under more effective organization.

If the country is not getting more than 20% effective results out of possible capacities, who or what is responsible for the loss of the 80%? On this point A. J. Hemphill, chairman of the board of the Guaranty Trust Company, one of the largest trust companies in the world, throws a high-power spot-light when he says in the July issue of the *World's Work:*

> In considering employment of funds in any concern, the greatest emphasis should be placed on the ability of the management. I would say that the most important consideration in any investment is management, management, management, and again management. It is not enough to know that it is honest; its ability in that particular business should be established. Ninety-five per cent. of the worth of an undertaking, I should say, is in the management.

These statements indicate the source of a large part of the troubles and ineffective results encumbering modern business. If 95% of the worth of an undertaking lies in the ability and effectiveness of its managment, and if only 20% efficiency is the measure of present possibilities, then of the lost 80% of effective results it is fair to attribute 76% to inefficient management. So management looms up as an important and indispensable factor.

Is it possible to take up any part of the slack in this business tackle? Are there any general principles of successful management that may be applied and amplified by executives through self-analysis and intensive training? Probably no ready-made formulas exist for success in business as each business presents its own problems, and even where the problems seem similar the same solution is not always applicable. But certain general principles hold the germ of success in all fields. A prominent financier makes certain mental qualities the basis of his selection of executives, and lays especial emphasis on imagination, judgment and initiative. Discussion of these qualities is bound to be philosophical. It cannot be scientifically exact. The essentially practical man will say that it is all very well to talk about imagination being necessary to the effective executive, but will want to know just how is imagination going to meet the problem of his payroll this week; just how is judgment going to help him on actual matters of fact where the answer is hard and fast, with only one way out, and how is initiative going to get him a supply of needed raw materials when there is none to be had.

Consider for the moment this point of view and grant that the objections are well taken, and that these qualities are altogether too philosophical and idealistic to be of value in meeting the problems of practical business. The alternative is to follow the customary business path, letting present methods direct each succeeding step, and following no plan imagined at the outset, no project preconceived and definitely outlined in the beginning. Such acceptance of conventional conditions inevitably creates the situation so well described by H. G. Wells in his analysis of the governmental attitude in "Mr. Britling Sees It Through," where Mr. Britling says:

> We had a government that seemed guided by the principles of Mr. Micawber, and adopted for its watchword, "Wait and see. . . ." We were too lazy, we were too negligent. We passed our indolent days leaving everything to somebody else. Was this the incurable British, just as it was the incurable Britling, quality? . . . Was the whole prosperity of the British, the far-flung empire, the securities, the busy order, just their good luck? It was a question he had asked a hundred times of his national as of his personal self.

Of course it isn't just "good luck." It may now plainly be seen that the answer lies in the fact that in a moment of national emergency, when everything seemed headed toward destruction, some imagination-judgment-initiative called Lord Northcliffe gets to work. It analyzes the situation and puts a Lloyd George on the job as manager of Britain's war. Then it wonders, through its imaginative quality, if the same "in-the-rut" type of mind which had formerly been directing the military end might not also exist in the civil branch, the "wait-and-see" type, and finds it does. And the initiative quality of mind, in the face of apparently overwhelming opposition, shows the facts to the country — to the stockholders of the British Empire, so to say — and the board of directors is changed and a new Cabinet installed. The intensely practical management in charge of the nation's business couldn't analyze itself — thought it was doing the best possible under the circumstances — and the best results were obtained only when the executive mind trained in imagination-judgment-initiative supplanted the practical manager. The same thing is happening today in industries running below par in efficiency. Velvet rugs and mahogany furniture cannot substitute for the fundamental qualities, though for a time they may conceal their absence.

The observation concerns general principles, but concrete facts are necessary to drive them home. Therefore to illustrate these principles wherein my applications would lie, because of my work as a mining engineer, I am drawing on personal experiences in the mining field. It is not possible to draw a line of demarcation where one mental quality begins and another leaves off, but these examples will serve to show what the combined qualities that make up the proper executive may accomplish. For several years a certain mine had been operated on a profitable basis, and returned a good income to its shareholders. Neither the president nor the general manager was a man of imagination, and while matters ran smoothly they ran on a level much below the plane of high efficiency. A change of management was urged by some of the directors, and after much persuasion and pressure this was brought about. Within 30 days after the new manager was installed the output increased 260%, dividends doubled and the surplus mounted by leaps and bounds. In the first month the net profits increased $40,000, and soon they were mounting at the rate of three-quarters of a million a year. Now, the new management worked with no equipment other than at the service of the old management. It merely brought vision — imagination, if you will — to the problem and saw what could be done through simple and seemingly slight readjustments all through the organization. Of the changes made no single one seemed of particular moment, but all in combination put the entire business on a higher plane, and gave the immensely profitable result. It was judgment, knowing what rearrangements were feasible and proper; and it was initiative, getting out of the rut of the obvious routine. These qualities in 30 days returned an actual cash payment of $40,000 — well worth while even to the "practical" mind.

Again, a certain mine is owned by men having at their command all the money that is needed to accomplish a splendid piece of constructive work. Yet year after year, for 20 years, the mine has remained idle because they see it only as a small property and cannot imagine it as a large and highly profitable business, run as an independent industrial unit. They also lack the initiative to investigate the best metallurgical methods and do not understand how to engage the best brains and bring them to bear on the problem. This really doesn't seem a difficult thing, but it is. The best brains go with broad-gage minds, and the man whose mind leans toward doing things in a small way will get small men to help him, and he will achieve a small result. In this particular instance the product of the mine is silver and copper, both of which are now at supernormal prices in the market. It would seem that even a narrow judgment would appreciate that the obvious thing is to market these products now. But this particular judgment does not apparently realize that to a degree as never before has time become the essence of action and that all effort should be concentrated now without delay. Failure to grasp the situation is due solely to executive narrowmindedness, and not to lack of money or labor.

One more instance: A certain large mining corporation retains in an important executive position a man who does not get nor hold the goodwill of its customers. A competitor told me that even if this corporation felt that its manager was too expensive to be retained, it would be good business for the competitor to pay the salary of the corporation official because of the business which was diverted to the competitor through the unpleasant personality of the corporation manager. Needless to say the competitor is affluent, and has a broad-minded, broad-gage management in its highest executives, but the corporation with the unpleasant manager held a place in the industrial procession two jumps ahead of the sheriff before the war prices of metals came and saved the day. This corporation represents millions of invested capital, and the wonder is that the executives at the top can remain either ignorant of, or ignore, the conditions of personnel which so profoundly affect its fortunes.

I sat last month with a group of mine directors who had recently bought a property, and who were continuing as manager the man who sold them the mine. (Parenthetically I may say that mining engineers recognize this procedure as one of the most fruitful causes of mine failure. It is done on the theory that the man who has been in charge of the mine knows most about it, but this is a fallacy that has cost many men hundreds of thousands to unlearn.) This manager was the only "mining man" on the directorate. The suggestion was made to this general manager, a relic of the metallurgical days of the '70's, that he would greatly increase his profits by installing a flotation method to which his ore was peculiarly susceptible. His reply was that "flotation is yet in its experimental stages, and I propose to crush my ore with rolls and concentrate it with jigs." So he chose the method that would be certain to show the greatest losses, calculated at a minimum of $15,000 a month. The manager's judgment was faulty; it failed to recognize, and the initiative was lacking to learn, that on the basis of tonnage treated flotation is today the leading single metallurgical process. About 30,000,000 tons are annually treated by flotation, no other method equalling it in tonnage handled. The directors had thought their manager the best they could get for their business! Why did they not have enough imagination and initiative to appreciate that many hundreds of men are trained in that profession? Why did they not try to get the best manager possible, even though their choice might happen to fall on one who did not possess the happy qualification of being able to induce them to buy the mine?

Not all examples that might be cited are along the line of carping criticism, and to maintain the proper ratio one good example will offset the four horrible ones. A certain company is operating its mine on a leasing system. This is commonly held to be the last resort and is used only when the orebodies are approaching exhaustion and the mine is *in extremis*. As a matter of fact the general manager is broadminded, and he believes so thoroughly in the efficiency of accurate accounting that he is able to select the most profitable method with no uncertainty. His operations on both the system of company operation and leasing operation gave exact figures, and by far the most money is made by the company, and larger dividends are paid, under the leasing system. So in the face of natural opposition he installed that system, with the result that his company is earning more money, the mine is in a better physical condition, the holdings of the company have been constantly enlarged, and the profits of the leasers are greater than they would be under the day's pay method. Therefore the workers are more contented, and the net result is an excellent industrial organization. Imagination-judgment-initiative have had their place in determining the attitude of the management, and the outcome is a highly profitable one from any viewpoint. These are elementary qualities, but backing them up and making them effective in mental make-up must be self-confidence, the confidence that, convinced of the accuracy of its judgment, goes ahead to develop along the lines aimed at. But self-satisfaction is a sure antidote to progress, and there, in all probability, lies the answer to the lost 80% in efficient capacity. Satisfied with present progress, no effort at self-analysis is made, and a chosen field of work develops its possibilities only when a new mind is brought to bear on the problem, a mind which has the qualities, even in a small degree, which make for better organization and more intensive results.

There are many volumes dealing with the development and training of mental qualities, but the first step is to recognize one's deficiency. As soon as this is taken, judgment and initiative begin to leaven the whole lump, and ways to reach the goal take form from within.

The Bonus System Applied to Mining

BY W. V. DeCAMP

Several articles have appeared lately in various technical publications, dealing with the bonus or contract system as applied to underground labor, but in all this mass of information and opinions there is one important consideration that has not been dwelt upon. In mining work the Blue Bell mine is trying to adopt a bonus system, which works out with excellent results in a shop where the piece-work system is successful and where it is possible to limit the operations and movements necessary to do a certain job; but when attempt is made to adopt the method underground, it often fails miserably, which is due to the many variations in the character of the work.

Occasionally there are certain classes of work which can be placed on a definite bonus system and which will work out satisfactorily both for the employee and the employer — such work is shoveling in a large stope where there is a constant supply of broken ore, or in tramming ore or waste on a level where the operation is definite and the supply more or less regular. Also certain classes of work, such as drilling in a large stope where the conditions from day to day do not vary too greatly, or in driving a drift or crosscut where the ground is uniform in character.

When such conditions are constant enough one is able to estimate a certain task for a day's work, and any unit of work above this task can be used as a unit for a bonus to the workman concerned. The amount to be paid is generally determined at the end of two weeks or a month, as the case may be, and at the end of that period the company engineer measures up the particular working place and calculates what the daily output has been; after which a bonus notice is posted to the effect that John Doe has earned a bonus for the period of a certain amount per day. John Doe, on seeing this notice, if he happens to think he has earned more than is stated, is inclined to be peeved, and decides in his own mind that the system is a fake and designed for the company alone; that he will get paid only a certain amount anyhow. As a result he ceases to put forth any effort toward earning a bonus, and since he waits 15 or 30 days to find out what his earnings were, he is likely to slow down to a leisurely pace and forget about the bonus.

The above statement is made from observation during many years of effort to institute a bonus system for general mining work that will be satisfactory both to the employee and the employer, and carries a full realization of the many difficulties to be expected. As a result of this experience the conclusions I have reached are that the principal fault lies with the employer, in that he does not give his bonus system sufficient supervision; and both close supervision and a daily check on results are necessary to success.

For the average miner or laborer working under a bonus system it is not sufficient to know the results of his labor every 15 or 30 days. He should know positively at the end of every working day, or before he goes to work the next shift, what he has earned. This is not a new idea by any means, but it is a psychological fact that whatever one succeeds in impressing most forcibly on an individual's mind will eventually become his most marked point of view. To apply this to mining close supervision is essential, and it becomes necessary to reduce the number of men working under each boss or, what I consider much more satisfactory, to employ a separate checker to handle bonus details for from 30 to 40 men. This man can be directly under the shift boss, but gets his bonus rates from the mine office, as determined for the various working faces, and is directly responsible to the office for the results of his work. He should be supplied with record cards showing the working place of every man and should detail on these the class of work and the amount of bonus earned by each. The card records can then be summarized daily, and a notice posted to the effect that certain men have or have not made a bonus. It is just as essential that the bonus earners be posted as it is for those that have earned none. By posting the notices daily every man is made acquainted with the earnings of himself and others.

The bonus idea should thus be kept alive in the minds of the men, and if any question arises it can be brought up and settled at once. The miner working in a drift should know just how much ground he is breaking per round, which makes daily measurements of every heading necessary. The cards for each working place should state the time lost due to causes not under the control of the men, and if any question arises it can be brought parts. Thus if a miner is held up all the morning by being forced to bar down a dangerous back, and in the afternoon does an unusual amount of work, he should receive credit for the full shift at the same rate of earning he has made during the afternoon. Bonus rates should be posted on each level or at the entrance to each working, and every new man starting to work should be made fully acquainted with the methods in use and rates of pay and bonus.

This may appear complicated to the manager, and the idea of increasing the number of bosses and clerks will probably meet objections, but once the confidence of the men is obtained and the system well started it will work out nicely and will tend greatly toward maintaining or increasing the average rate of the work performed.

As to the method of determining the bonus to be paid in a particular case, I have found that one cannot be too careful in establishing this figure, and, once established, the rate should not be changed except in the advent of a change in wage scale or the adoption of some new condition or mining method that would make the old rates impossible. Careful records of past performance and a study of actual conditions should be made as the basis of the rate, and I have found that where a certain amount per man has obtained for a certain period it is well to accept this as the base rate and begin the bonus from this point, increasing at a regular rate as the work increases. In the event, however, that the employer feels that the average has been entirely too low, it is well to begin at a low rate and gradually increase the unit rate as greater output is obtained.

As an illustration of the first case, assume a carman tramming 30 cars per day from a certain chute. The cost to the company on a base rate of $5 per day will be 16.67c. per car. With a bonus based on 50% of the cost the carman would therefore receive, say, 8c. per car for every car over 30. If he can save a little time at the beginning and end of the shift and not "take five" too often, it would be quite possible for him to increase his output by four cars, thereby increasing his wages 6.4% and decreasing the cost per car to the company by 6.06%. This means a direct saving of approximately 34c. per carman per day for the company. If the same amount could be saved on the labor of 40 men, it would be sufficient to pay the wages of the extra boss and leave a good margin for the company.

In a drift where the average footage advance has been four feet per round at a total cost of $20 for miners and muckers, or $5 per foot, the bonus to each man working, assuming the employment of two carmen and two miners, might be placed at 50% of the total saving made. Thus: 4½-ft. in advance at $21.25 costs $4.72 per ft., and 5-ft. advance at $22.50 costs $4.50 per ft. A direct return to each man at the 4.5-ft. rate of 31c. per day and of 62c. in the last case.

One thing that should never be attempted is to so arrange the bonus so that it will always be constant. I have often heard mine superintendents say, "We want our better class of men to earn at least 25c. per day more than the scale." This is a grave mistake and one that will result in the slowing up of the men, as the incentive is then taken away, since this constant bonus, in the eyes of the men, will mean nothing more after a certain period than a straight increase in wages, and they will lose sight entirely of the bonus idea. This was aptly illustrated a few years ago in the copper mines of Arizona. When copper went up the companies raised wages a certain amount per unit increase in copper, with the result that every man, from the mine superintendent down, felt, after the lapse of a few months at the increased wage, that his value to the company was the total wage paid, and lost sight of his base rate entirely. If the companies at this time had taken the trouble to issue two different checks for each period worked, one check for the number of days due at the base rate and the other check of different color for the copper bonus paid, it would have resulted in every man's looking upon that bonus check in an entirely different light and would have kept constantly in mind his actual base rate and the amount he was earning because of the increase in the price of copper.

The bonus or task system, to be entirely successful, should be handled in a similar manner, and if rules were to be established for a bonus system they should be grouped as follows:

1. Never change a bonus rate.
2. Notify men daily of amounts earned.
3. Notify every man when he starts of the exact conditions under which he is working, bonus rates, etc.
4. Keep the bonus idea constantly in mind.
5. Pay all bonus by separate check.

Economic Importance of Wood Preservation

BY KURT C. BARTH

The practice of wood preservation in the mining industry is not only desirable economy, but a requisite to the proper utilization of structural wood, which for many years has been the most popular material used in mine structures, due to its availability, lower cost, and ease of employment. These factors are now important, since the demand for finished-steel products has increased inordinately, and skilled labor is not always available. The average increase in prices of 22 metals since 1914 is said to be 124%, as compared with an increase during the same period of 20 to 30% in the prices of lumber products.[1] Furthermore the national need for steel products for war purposes compels the use of wood and the conservation of steel, wherever possible, as a patriotic duty. These circumstances combine to create conditions under which the practice of wood preservation is not only of greater material advantage than heretofore, but an actual necessity in many cases. Structural steel latterly has been widely used in mine structures because of its permanent character, while the decay of mine timbers, both above and below the ground, makes necessary continual replacements and repairs, although this can be obviated to a marked degree by the proper preservative treatment. It is obvious that in order to protect wood from decay and insects it must be neutralized, or chemically treated, so that the development of destructive agencies is arrested and made impossible. Therefore, paint, although it naturally gives a certain amount of protection to wood, not being a toxic agent, is not a preservative for the purposes under discussion.

The average life of untreated surface timber structures must be placed at from 10 to 15 years to provide a conservative basis for figuring depreciation. Wood used underground is destroyed by decay in a much shorter period, the average natural life being from four to six years. This period does not take into consideration timber the usefulness of which is, previous to the expiration of its natural life, destroyed by mechanical abrasion. Wooden surface structures, shaft timber and "lath," sawed timbers used underground in permanent sets and for other purposes, are frequently destroyed by decay rather than by wear, so that preservative treatment of the timber is not only advisable but necessary. In mine shafts that have been lined with concrete and steel and in which timber has been used for cross-ties, longitudinal ties, or cushions, etc., the replacement of the timber involves considerable expense and interruption of traffic and preservative treatment is an important factor in prolonging the life of this material.

In general, underground timber and structural wood used in surface structures are classified alike, and the question as to whether creosoting is advisable may be decided by the following rule: Timber that is permanent in character; that is, which is not exposed to destruction by mechanical wear before the expiration of its natural life, or the usefulness of which does not cease before the advantages of preservative treatment can be realized should be creosoted.

Among surface structures may be classed tipples, elevators, trestles, shafthouses, headframes, tramways, ore bins, coal docks, coak bunkers, chutes, etc. Where wood or semi-wood mine cars and skips are used the parts not likely to be destroyed by wear may also be treated to advantage. Another field in which the opportunities for the use of creosoted wood are numerous is that of miscellaneous equipment used in concentrating mills, stamp mills, smelteries, etc., which are usually included in the plants of zinc and copper mines. Continuous replacements and repairs are necessary, and probably 90% of the wood requiring replacement has been destroyed by decay. The period of service of concentrating tables, launders, jigs, flotation machines, sand boxes, wooden screens, tanks, foundation timbers, floor planking, etc., could be doubled by the use of creosote.

The method of preservation most available to mines at considerable distances from commercial creosoting plants is the nonpressure process, known as the open-tank system, which consists of hot and cold treatment of wood in refined coal-tar creosote oil, or the brush method of applying two or three coats of refined coal-tar creosote oil to points of contact and exposed surfaces. Creosoting is advisable under conditions where the mechanical abrasion is slight or entirely absent and a heavy impregnation of the wood is, therefore, not required. The open-tank system is sufficient to preserve structural timbers and lumber used for miscellaneous purposes.

The brush method should be used when a temporary increase in life is sought; that is, when an average increase in durability of five years is sufficient, and when the open-tank system of treatment is not practicable. To obtain the best results the oil should be heated to a temperature of 150° F. However, a properly refined coal-tar creosote oil is liquid at what would be considered normally a low working temperature, which permits its use without heating during seasons of the year when climatic conditions are favorable to outside construction work. The open-tank process consists of alternate hot and cold applications of refined coal-tar creosote oil by immersion and continuous soaking in open tanks without artificial pressure. This requires no mechanical apparatus other than tanks, hoists (in some cases) and means of heating the oil.

The procedure in the open-tank process is as follows: After the lumber to be creosoted has been framed, bored and cut to size it is given the hot treatment, which consists of immersion in a bath of oil having a temperature of between 150° and 200° F., and this is continued for varying periods, depending upon the species of wood and the size of the timber. Immediately afterward the timber is immersed in a cold bath of refined coal-tar creosote oil, which is kept at a temperature not in excess of 100° F. for periods equal in dura-

[1] L. C. Boyle in the "Lumber World Review," Sept. 25, 1917.

tion to the hot bath. The theory of the open-tank process is that in the hot treatment the heat of the preservative expands and expels a portion of the air and water contained in the wood cells. Upon immersion in the cold bath, or when subjected to a change in temperature caused by the cooling of the creosote, there is a partial contraction and condensation of the air and water that remains, which results in a slight vacuum within the wood. This vacuum is aided by atmospheric pressure, combined with capillary attraction between the wood cells and the preservative, which achieves the actual impregnation.

Modifications conforming to special requirements and local operating conditions are inevitable on all large jobs, and may be made without affecting the efficiency of the treatment, and this is one of the most important advantages of the open-tank system. Lumber of small cross-section often can be effectively treated by immersion for short periods in the hot bath alone, especially when of a species not particularly resistant to impregnation. Heavy timbers are most economically treated by continuous immersion in one tank during the entire process. Upon expiration of the hot treatment the timbers are not removed to a cold tank, but the stream is shut off and the bath allowed to cool. If an open fire is used to heat the oil it is quenched and both the oil and the wood are permitted to cool to about 100° F. If loblolly pine timbers or any species that readily absorbs creosote is to be treated, the cold-tank bath may sometimes be dispensed with.

The dipping method is a modification of the open-tank system and is used when lumber of small cross-section and of a species easily impregnated is to be treated. It becomes a modification of the brush method, also, when timbers of a large cross-section are to be given a short immersion in order to eliminate the double handling required by the brush method. The oil should be maintained at a temperature of 150° Fahrenheit.

In general the period of immersion in the preservative should be as follows: Close-grained woods naturally resistant to impregnation, one hour in the hot and one hour in the cooling bath for each inch of the largest cross-section. Species more easily impregnated require a quarter of an hour for each inch of the largest cross-section, and dimension lumber should remain from 10 to 30 min. in each bath, although in many cases the dipping method is sufficient for boards.

The proper preparation of wood is probably more important than the treatment itself, the effectiveness of the latter depending largely upon whether or not the timber has been satisfactorily seasoned. It is difficult to obtain satisfactory results by nonpressure methods in the treatment of green timber with the exception of sap and loblolly pine. If the wood cells are filled with moisture the creosote cannot penetrate unless the water is first expelled by artificial means. This is accomplished either by steaming, as in the pressure processes, or by continuous soaking in the hot bath until the wood has been heated throughout to the temperature of the oil, which must be between 200° F. and 220° F. The former process is not often available, as mines seldom have the necessary apparatus for creosoting by pressure, which requires more time than it is practicable to devote to the work. It is therefore self-evident that wood which is to be treated must be seasoned until air-dry.

Numerous cases have been observed where the attempt to treat practically green timber by short immersions in hot creosote oil resulted in failures for which the preservative was held responsible, or that were used as a basis for charging nonpressure methods of treatment with inefficiency, whereas the real trouble may be attributed to negligence or ignorance.

With few exceptions, such as a breakdown, etc., it is usually possible to purchase timber requirements sufficiently in advance to permit of proper seasoning; that is, seasoning for about six months previous to treatment. Seasoned lumber is more expensive than green wood or timbers direct from the saw, but the increased charge made for wood properly conditioned, or the interest on the investment resulting from storage of timbers for three to six months to allow proper seasoning, is more than offset by the increased value of the dry timbers and by the greater efficiency of the preservative treatment. If it is considered a burden to carry such extra charges in any other manner, they may be added to the cost of creosoting without materially decreasing the profit derived from the proper practice of wood preservation.

Although a number of copper and iron mines have to some extent adopted the practice of wood preservation they could widen their activities in this direction with profit. Some iron mines are creosoting large quantities of timber and lumber for surface structures, such as headframes, shaft houses, trestles, coaling plants, and material for shaft timbers, and "lath," and certain copper mines have creosoted surface structures such as wooden pulley stands, ore bins, trestles, coal bunkers, cross-country launders, etc., and also concentrating tables, launders, sand boxes, tank screens, tanks, flooring, etc., in concentrating mills, stamp mills and leaching plants. One mine, in particular, recently built over 100 concentrating tables, the lumber for which was creosoted by the open-tank system previous to assembling.

The zinc and lead fields offer many opportunities for the practice of wood preservation. A recent inspection showed that wood was used almost exclusively for surface structures and equipment for mills. Rock and ore hoppers, exterior and interior launders, jigs, sludge tables and catch boxes, elevator shafts, floor planking, foundation timbers, beams and girders, etc., are exposed to more or less severe conditions and require continual repairs and early replacement due to decay. Practically all of the timber used in this construction should be properly creosoted before erection. In addition to prolonging the life of the lumber, creosoting will eliminate the necessity of painting, thereby saving nearly enough to cover the entire cost of preservative treatment.

Underground conditions in zinc mines do not warrant the use of creosoted timber, excepting in special cases where the period of usefulness of the timbers is greater than their natural life. Operations in the zinc and lead fields often require removal of the entire plant, and it would seem under such conditions that it might not be economical to creosote the various materials referred to. However, in erecting them in a new location permanency is a desirable factor, and if the life of the lumber used in the construction of this apparatus is prolonged by chemical treatment the salvage value would be high and would leave a handsome margin of profit over the cost of creosoting.

An important feature is the selection of the preservative to be used in nonpressure treatments. Pure coal-tar creosote oil, which conforms to standard specifications and is refined to meet the special requirements of nonpressure treatments, is most efficient. Wood-tar creosote or adulterated coal-tar products are not permitted by any standard specification in the wood-preserving industry today. In the case of surface structures the preservative must be effective for many years and it is necessary that the product selected have a clear record as to its efficiency. Experiments are no longer necessary and success from proper treatment with the right preservative is assured.

Refined creosote oil may also be used as a paint. Not only is it more economical than the cheapest of oil paints but it lasts longer on account of its hue and non-volatile nature. After application the color is an attractive dark brown.

Other building materials than wood are treated in accordance with the respective specifications or directions furnished, because it is fully realized that unless care is taken in their utilization a monetary loss and sometimes disaster may follow. Structural steel is given frequent attention to prevent deterioration, and it seems unreasonable that when structural wood is used an effort is rarely made to protect it against destructive influence. An ounce of prevention is worth a pound of cure. Creosoted timber is most economical for general building purposes, and is practically permanent.

Manganiferous Iron Mining in the Cuyuna District, Minnesota

BY P. M. OSTRAND

The restriction on the export of manganese ores from India, the cutting off of the Russian supplies caused by the blockade of the Black Sea, and the increased demands for manganese in steel manufacture have contributed within the last year or two toward the development of the Cuyuna range, the manganiferous iron ores of which constitute one of the important deposits in this country. Should access to the Brazilian deposits be cut off, the steel industry of the United States would face a serious situation and for immediate relief would probably have to rely largely on the production of the mines operating on the Cuyuna range.

The Cuyuna iron range of Minnesota embraces an area approximately 65 miles in length, and from one to 10 miles in width and is near the geographical center of the state. This area has locally been divided into two ranges, known as the North range and the South range. A strip of territory three miles wide, in which no iron-bearing formation has yet been found, separates the two ranges. It so happens that the main line of the Duluth-Brainerd branch of the Northern Pacific R.R. follows this strip through the district, and the track has been popularly regarded as the line of division between the ranges. The two ranges, while structurally and geologically similar, differ somewhat in the character of the ore. The North range, comprising an area of approximately 50 square miles, contains nearly all of the productive mines of the district and it is in the Northern part of this area that the important manganiferous iron ore deposits are found.

The district has no marked topographic features. The surface is level and is covered by a heavy glacial mantle of sand from 50 to 100 ft. thick, deeply dented in places by lakes, swamps and marshes. No outcrop indicates the mineral-bearing formation, so that prospecting is difficult, although magnetic surveys with dip needle and sun dial have been of assistance, especially on the South range. The orebodies in the district and the majority of the enclosing rocks are not in themselves magnetic, but the association of certain magnetic slates with the iron ore has made magnetic surveys valuable in new ground. In fact, it was the deflection of the compass that caused the engineers of the Northern Pacific Ry., as early as 1883, to classify this area as being underlain by iron-bearing rocks, and this fact perhaps influenced the building of the railroad through the barren strip separating the two ranges. Preliminary drilling in new ground has been and still is largely directed by the results of magnetic surveys. However, the rule has exception, particularly as applied to the deposits on the North range, and no royal road to finding ore by means of magnetic surveys alone has as yet been discovered.

The prevailing opinion among geologists seems to be that the area comprising this district was originally a part of an inland sea, and that the iron formation was deposited as a sediment with other associated rocks. Subsequent to the precipitation and deposition of the iron sediments, the rocks of this region were subjected to intense pressure and folding and the various sediments deposited were altered to the slates, schists, ferruginous cherts and other rocks found today by heat and pressure. Following this period of folding came the period of glacial erosion, in which the tops of the anticlines were eroded and carried away, thus exposing the iron formation to weathering and the action of atmospheric waters, which resulted in local concentration of the iron ore. The manganese was originally deposited at the bottom of the inland sea along with the iron. During and subsequent to the folding processes, the manganese salts (through the action of descending atmospheric waters), were dissolved and carried downward through the exposed limbs of the iron formation; other constituents were dissolved and in their place manganese was deposited. The fissures and cleavage planes in the formation and the pore space developed by the leaching out of the silica and other constituents, all of which controlled the circulation of the descending solutions, developed different phases of the manganese replacement that have been fully discussed [1] by E. C. Harder.

The main structural features of this range consist of a series of more or less parallel folds extending in a northeast-southwest direction, in general the same as that of the Lake Superior synclinorium, and probably contemporaneous with and produced by the same forces which caused the distortion of the whole Lake Superior region. The folding and subsequent erosion and concentration resulted in a series of more or less parallel, lense-like and tabular orebodies with their longer dimension parallel to the bedding and dipping at an angle usually of from 60° to 70°. The formation usually dips to the southeast, probably due to overthrow folding, and is surrounded by barren rocks. The orebodies on the North range average over 100 ft. in thickness, and in places are as much as 500 ft., while those on the South range are narrower and do not average over 50 ft. Some of the orebodies extend unbroken for more than a mile along the strike of the iron formation. As a rule the highest degree of concentration is found on the hanging wall side and near the top of the formation, although drill holes have encountered merchantable ore 700 ft. below the base of the glacial drift.

Seven or eight main belts of iron- and manganese-bearing formation follow an approximate northeast direction through the district. The main producing belts, as defined by the location of operating mines, beginning

[1] Bull. of A. I. M. E., Sept., 1917.

with the most northerly belt are five. The first includes the Ferro, Algoma, McKenzie and Ida May mines, all mined for their manganese content only; the second includes Merritt (manganese) and Kennedy (iron) mines. There are breaks in the continuity of this belt. The third belt includes Cuyuna-Mille Lacs, Sultana, Mangan No. 1, Hopkins and the Joan mines, all of which are mined for their manganese content excepting the Hopkins, which is partly iron. In the fourth belt are Mahnomen, Mangan No. 2, Evergreen, North Thompson, Armour No. 1, Armour No. 2, Pennington, Feigh, Hill Crest and Rowe mines. This belt contains both iron and manganese in commercial quantities and most of the ore is sold on a combined basis. The metal-bearing formation continues unbroken along the same strike for more than eight miles and contains the most important ore deposits in the district. Cuyuna-Duluth, Armour No. 2, South Thompson, Meacham and the Croft mines are in the fifth belt, all being mined solely for their iron content. These five belts contain all the producing mines of the North range. In addition, other belts, which in a measure complete the successive limbs of the folds described above, have been partly explored by drilling, but their continuity for any great distance has not been proved. The rocks between the limbs are barren and do not have uniform characteristics. It is possible that the relation between the limbs may be determined from a study of the associated rocks and ore that has thus far escaped the drillmen.

The unit of exploration is usually 40 acres of land. Previous to exploration, the corners and property lines are established and a dip-needle survey is usually made in the following manner: Beginning at any corner of the property, dip-needle readings are taken at 100-ft. intervals along the property lines surrounding the tract. The needle is held in the same manner as a compass until it comes to rest and then turned so that the axis of the bearings of the needle is perpendicular to a vertical plane passing through the magnetic meridian. The points of greatest attraction around the traverse are noted and, bearing in mind that the strike is in a northeast-southwest direction, a line connecting the two points of highest attraction is projected across the tract and stakes are set along this line at 100-ft. intervals. Readings are then taken at 50-ft. intervals on lines passing through the stakes at right angles to the projected line. From the data thus obtained, deductions as to strike, the direction and approximate inclination of the dip may be made, and the drilling operations directed accordingly. A sun-dial compass has been used in connection with the dip needle, but generally the dip needle alone is used. The interpretation of the results of a magnetic survey and the direction of the subsequent drilling call for technical skill as well as considerable local experience.

Drill holes on the Cuyuna are usually placed on a line at right angles to the strike and across the formation. Inclined drill holes are sometimes made, although vertical bores have been found to be more dependable. Both churn and diamond drills are used and average charges by local drill contractors are $2.75 per ft. for churn drilling and $4.25 per ft. for diamond drilling. Drill holes are numbered and, upon completion, their position in the field is substantially marked. During the drilling, samples of the formation are taken at 5-ft. intervals. The situation of the holes as well as the results of the drilling are carefully platted and preserved as permanent records. Probably 5000 holes, averaging 300 ft. in depth, have been drilled on the Cuyuna, and exploration is still active. The deepest drill hole in the district was bottomed at a vertical depth of 1037 ft. in the formation. This hole is situated on the northwest quarter of the northwest quarter, Sec. 17-46-29, and is a part of the Rowe lease held by the Pittsburgh Steel Ore Company.

Openpit, milling and underground mining systems are used in the district; the method decided upon being a question of cost and adaptability. In each case, cost of the plant, stripping and other steam-shovel operations, shaft sinking, underground cost of mining, etc., are estimated and the figures compared. In many cases the narrowness of the orebody and the irregularity in the composition of the ore make the use of a steam shovel impracticable, while some of the wider orebodies, originally operated as openpits, will eventually use all three methods. The steam-shovel operations are followed by milling, and this in turn will be succeeded by the regular underground methods. Of the six openpits operating on the Cuyuna range, but one — the Thompson, at Crosby — has as yet reached the milling stage and that operation was begun in the early part of 1917, when the grade of the pit approach became too steep for the economical employment of a locomotive.

Mining an orebody having a width of not more than 100 ft. and a dip of 70° by steam shovel has its disadvantages. The pit is narrow and track grades to or from the approach have to be developed by switchbacks. The shovel must continue ahead in a straight line until the limits of the mining operations are reached, and all material, rock as well as ore, must be taken as it comes. This makes the problem of grading the ore a difficult one, so that some of the pits have established and are shipping as many as four or five grades of ore.

The depth to which ore may be mined by steam shovels is limited largely by the track grades developed on the approaches, and the depth of milling operations is controlled largely by the proximity of the overburden, the pitch of the orebody and by the nature of the hanging wall, which also affects steam-shovel operations. Under usual conditions, 80 ft. below the base of glacial drift is considered a maximum depth for steam-shovel work, with an average of perhaps 60 ft. Milling will further reduce this ore level an average of 60 feet.

Milling is a combination of openpit and underground mining. A shaft, usually well off the orebody, is sunk to a depth below the limit of the proposed milling operations, and a main haulage drift is then run from the shaft to a point at about the center of the orebody, which in this district, owing to the dip, will be near the foot-wall side. A drift cross-cutting the main drift at an angle and parallel to the strike of the orebody is then driven to the limits of the desired operations and raises to the top of the ore, now the surface, are driven at intervals of from 50 to 70 ft. along this crosscut. Well-timbered chutes are constructed at the bottom of these raises and the ore is blasted and broken into the raises or mills from the surface and drawn from the chutes into tram cars, hauled to the shaft and hoisted. Inverted cones of ore will be left between the raises, as the ore is mined into the chutes, and are tapped by another set of raises situated halfway between the first set, and so on until all the ore to a point about 10 ft. above the roof of the main haulageway has been mined. When conditions limiting the depth of milling operations have been reached, the surface of the pit will probably be leveled off and covered with a layer of poles and boards. The surrounding overburden will then be blasted in on top of this to a depth that will insure safety to subsequent mining operations below. The usual method of top-slicing and caving will then be started below the boards.

The underground mining method generally used in the Cuyuna district is the top-slicing and caving system, which is so well known that a description is scarcely warranted here. However, some features peculiar to this district may be mentioned. Some difficulty has been experienced with sand runs. The sand overburden is usually fine, and when saturated with water forms quicksand, which, having once found an outlet through the caves to the rooms below, is an exceedingly difficult thing to stop. Often a layer of ore four to five feet thick, left in the back of the top sublevels in wet mines, will reduce if not eliminate trouble from sand runs and in the end result in a larger recovery. A mattress of carefully laid poles and lagging on the top sublevels will also help considerably.

Hydraulic stripping has been successfully used on two openpit properties on this range, namely at the Rowe and the Hill Crest mines. The method was initiated at the Rowe mine in 1913, under the direction of J. Carroll Barr, general manager of the Pittsburgh Steel Ore Co. The light sandy overburden at this property being comparatively free from boulders and hardpan, the proximity of favorable dumping ground and the availability of water suggested the hydraulic methods finally adopted. Two hydraulic giants were used at the Rowe, each requiring one 10-in. clear-water pipe, one 10-in. two-stage centrifugal pump for the clear water and driven by a 200-hp. motor, one 12-in, discharge-pipe line and two 12-in. centrifugal pumps, used in series as a sand pump, and driven by 250-hp. motors. The clear-water pump was situated on the lake shore near the inlet of the clear-water pipe, and the sand pump, with its motor, was mounted on a standard flat car so that it could be moved to different points in the pit. Wherever it was desirable to establish a sump, six 4-in. casings were driven from surface to the top of the ore, around the platform of the car and in 10-ft. lengths. The platform was attached to these pipes by clamps. By releasing the clamps the car could be lowered and the distance from the suction to the sump regulated as required, a minimum grade of 4% being maintained between the working face and the sump. The innovation of using a portable flatcar and lowering same by means of casings and clamps, as far as known originated at this mine.

Previous to the installation of the regular hydraulic machinery a unique method of moving dirt was used. A 12-in. pipe line with a 75-hp. steam pump was available at the time, and the pump was installed near the lake and about 1200 ft. of the pipe attached and laid on the rising ground back from the lake shore, the grade of which was slightly over 6%. A furrow was plowed in a straight line from the lake to the outlet of the pipe and the pump started. Water flowing from the pipe into this furrow eroded and moved 80,000 yd. in a month at a cost of 1.8c. per cubic yard.

Approximately 2,000,000 cu.yd. of overburden was moved at the Rowe mine in three years, at a cost of 6.7c. per cu.yd. The distance between the center of the excavation and the center of the dump was about 1200 ft. and nearly level. The best stripping record was made in June, 1914, when dirt was moved at the rate of 205.2 cu.yd. per hour. The plant was in operation 74% of the 720 hours in the month, the remaining 26% representing the time taken out for Sundays, repairs, etc., and at times the presence of heavy gravel cut down the rate to an average of 64.7 yd. per hour. The total cost of plant was $34,000, which includes material for repairs bought from time to time. Electric power was purchased at the rate of 1¼c. per kilowatt-hour.

The manganiferous orebodies of the district, owing to adverse market conditions in the past and consequently a restricted output, have naturally had a high unit cost per ton; and the irregularity and narrowness of the deposits have contributed toward high exploration and development charges. However, the larger iron mines have a fairly uniform range of costs and the following figures represent an average of amounts paid per ton for underground mining in the district, based on a daily production of 700 to 800 tons of ore and assuming that the orebody contains a million tons or more. A hoisting depth of 250 ft. and a pumping capacity of 1000 gal. of water per minute have been assumed.

TABLE IX. COSTS PER TON OF UNDERGROUND MINING ON CUYUNA RANGE

Exploration	$0.07
Slicing or stoping	.90
Tramming	.08
Hoisting	.10
Pumping	.15
Supervision and office	.10
Insurance	.04
Total	$1.44

To the above cost must be added the royalty charge, usually about 50c. per ton removed, and an amortization charge of 15c. per ton. The slicing cost includes timber, powder, tools, compressor charges and labor. The cost of operating the surface plant has been prorated among the items of pumping, hoisting and compressor charges.

In so-called "all-manganese" mines the costs, for the reasons mentioned, are considerably higher. It is probable that the cost in stockpile or on cars at the mine, including all charges will range from $3 to $4 per ton at a property producing 150 tons or more per day.

The cost of stripping an openpit property by steam shovel will range from 15 to 20c. per cu.yd., depending on nature of material encountered, proximity of the dump and grades necessary. Hydraulic stripping when applicable will reduce the cost perhaps 50%. The cost of mining ore by steam shovel, including necessary subdrainage, maintenance of tracks and all supervision and office charges, will average from 25 to 35c. per ton.

Milling costs range between the amounts paid for open-pit and underground mining.

Over 100,000,000 tons of merchantable ore have been developed in the district to date. The greater part of this reserve is iron ore sold for iron content only, while about 20 million tons is manganiferous iron ore running 4 to 30% in manganese. Two million tons will perhaps average 20% and the remainder 10 to 12% manganese. These figures are conservative. Should the manganese situation require it, the know reserves of manganiferous ore could be grouped with respect to iron phosphorus and manganese content, so that not only will present metallurgical requirements be met but a much larger amount of manganese than indicated by the above figures would be available for steel manufacture.

The following figures show the growth of the district since 1911, when the first shipment was made from the Kennedy mine, to date:

TABLE X. CUYUNA SHIPMENTS OF IRON AND MANGANESE ORES, 1911—1917

Year	Tons shipped
1911	147,000
1912	305,111
1913	733,021
1914	859,000
1915	1,136,113
1916	1,716,218
1917 (estimated)	2,400,000

The district in 1918 will, in all probability, be a very active one, for the seriousness of the present situation on the Atlantic seems to justify the belief that fewer ships will be used in the manganese trade. Furthermore, should the Government guarantee that the price of manganese will not be regulated downward during the period of the war, a natural rise will result and an unprecedented boom in the exploration and the development of the manganiferous ores of this district may be looked for.

Observations on Sinking Pumps

BY J. F. KELLOCK BROWN

The following notes are the outcome of considerable practical experience with many sinking pumps operated under a variety of conditions. They are written from the viewpoint of the man who has had to handle this class of plant on the ground, and for that reason they will in all probability not be gratifying to the pump maker. There may be all sort of mechanical disadvantages in connection with some of the ideas brought forward, but I have hardly ever handled a sinking pump without wishing that I had a pump designer standing in the pouring wet shaft beside me.

There are undoubtedly many sinking pumps that are excellent machines mechanically; that is, under ideal conditions, they do their work with satisfaction and

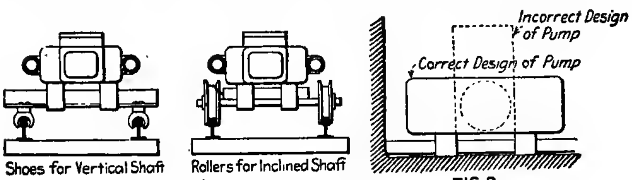

FIG. 1 FIG. 2

FIG. 47. ARRANGEMENTS FOR PUMPS IN SHAFTS

some perfection, but put to the test under adverse conditions, which usually appear in shaft-sinking work, where water is troublesome and the going bad, there nearly always appears about the pump some point that is capable of improvement. In other words, it would seem that designers have paid sufficient attention to the proper design of the pump itself, but not enough to the general conditions under which it has to work.

First, consider the handling of the pump in vertical shaft-sinking, in which heavy water is encountered, and which may or may not require heavy timber. Getting a small sinking pump down a shaft of this type is an operation that absorbs the energies of the entire force. The pump has to be hauled over the ground on skids, lifted by main force upon the platform at the shaft collar and the hoisting rope attached. If the hoist is too small for the work, trouble begins with the failure to lift the pump. Perhaps the hoistman thinks he will take a running jump at it, so he gets a good head of steam, uses a little slack, and does enough to jerk one end of the pump off the platform. This more than likely causes it to swing round suddenly and erratically; everybody gets out of the way to avoid smashed feet, and the pump swings with a jerk and a jar on the rope, into the shaft, while congratulations are rife that no further damage is done, which would be a miracle. Should the pump, however, prove too heavy for the hoist, the work is at a standstill until someone takes a 40-mile journey out to the nearest station and waits there a week for a single block of sufficient size to take the hoisting rope without damaging it through bending round too small a radius. Makers of course contend that the provision of such a block for an emergency of this nature has nothing to do with the design of the pump. To this contention I agree, but I might add that it would aid their sales if they would confer with their customers and find out the conditions under which the pump is to be used, the size of the hoist, and if too small, recommend and provide such a block ready made to assist in the work, thus ensuring good results.

Having raised the pump either by sheer lift or by dividing the load between the headframe and the hoist, it is swung clear in the shaft, swaying about from side to side. All parties make a grab at it to steady it, and one man is saved from falling down the shaft by the slack of his pants. Down it goes slowly, while a few of the more venturesome follow it cautiously down the ladderway. If the shaft is sunk at a slight inclination, one corner of the pump sooner or later catches on the walls, and the pump rolls over and smashes part of the ladderway, if it does nothing else. The damage having been repaired, pinch bars are brought down to roll the pump over again. Some engineer who passed his examinations in brilliant style, in order to get what he thinks is a good leverage, inserts the end of his bar between the pump casing, or waist piece, and the piston rod. Not quite satisfied with the power he is exerting, he sits down on the ladderway and shoves with his feet against the end of the bar. United energy turns the pump, and the aforesaid engineer sustains an unexpected descent and is only rescued by quick work on the part of someone else. In the resultant effort the pump rolls completely over the other way, and the outstanding pinch bar takes another workman below the knee, laming him for a week. On perhaps the third attempt in the confined space, the pump rights itself and an examination in the half-light available discloses the fact that the piston rod looks somewhat the worse for usage, and the pump has to go to the surface for overhauling, which cannot be done in the shaft. And so the work goes on with a great waste of time and expenditure of effort, partly through ignorant labor and lack of proper directive knowledge, but mainly because there is no simple, easy system of getting sinking pumps up and down a shaft. These pumps have to be taken up and sent down many times because there is something wrong with them, and although the labor force improves with practice, the operation of hoisting and lowering is always troublesome and often dangerous.

Some makers think they provide against the cause of these difficulties by supplying skids beneath the pump. They are made of wood, however, and the effort expended in trying to overcome the friction of wet wood-slides against wet wooden barring or close timbering that is not exactly flush is greater than any advantage that is gained. There is also little benefit, except that it protects the under side of the pump where there are rock walls whose projections would otherwise catch first one side and then the other, causing the pump to tilt easily and roll over. In a vertical shaft these troubles are not so great, but the dangers from swinging and twisting are increased.

To overcome these difficulties, is it not possible for pump makers to provide a simple runway of small rails, built in short sections capable of being extended and driven into position by a wedge? On this would run a rectangular carriage bolted to the pump, the gage being wider than the pump. Before taking down the pump, this runway would be spiked to the timbers and a length left below, projecting into the water. On being raised, the carriage and the pump attached to it would be placed on this runway, and it could then be slowly and steadily slid into position. As the carriage would be broader than the pump, there would be no tendency to overturn. In this manner a sinking pump could be raised and lowered with a maximum of ease and without a great waste of time. In the vertical, or nearly vertical, shaft the small wheels of the carriage could be replaced by guide shoes, as shown in Fig. 47. The slight sacrifice made in space, which need not be more than a few inches, is more than offset by the greater ease in handling. There is no disputing the fact that a sinking pump has to be raised and lowered in shaft work from a variety of

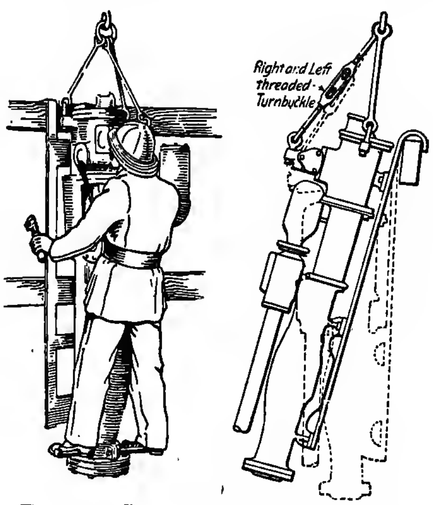

FIG. 48. SINKING-PUMP INSTALLATION

causes — in firing shots, for breakdowns, changing suction hose, changing pipes, timbering — and in a confined space with a constant stream of water pouring down, the slowness of the work is exasperating. Pump makers have eliminated a great many of the troubles of faulty design of the past, and if they would now turn their attention to the provision of some easy means of raising and lowering, many mining men would be thankful.

Perhaps the next point that calls for consideration is the shape and balance of the pump. By shape, is meant the relation of the width of the base and the depth through the pump. To get easy handling, the base should be greater than the depth at any section of the pump, as shown in Fig. 47. If this is not the case, the pump is easily tilted and rolled over. There are a few sinking pumps on the market that are irregular in this respect, some part or other jutting out beyond the line of the center of gravity, causing the pump to be more susceptible to twisting and rolling over to that side. A pump with a broad base means stability and is not so dangerous to handle.

This brings us to other points in the design of sinking pumps. Every door, and every set of bolts on the pump, holding the door of the water chest and of the steam and valve chest, should be on the side of the pump toward the center of the shaft. A pump so built that the water door faces one way and the steam chest another way, at right angles to it, is storing up trouble for itself as soon as conditions require the pump to go into a corner of the shaft. A pump of this awkward design must be shifted from its position every time something has to be examined inside or the valves cleaned. Moreover, it is easier to have a man who has to stand on awkward footing, work at bolt heads in front of him rather than stretch around corners after them.

The next point concerns the pipe connections for inlets and outlets. These commonly are placed at right angles to the length of the pump. This arrangement not only causes an abrupt right angle bend, but also gives rise to troubles from another source. In order to minimize time lost in the shaft and to do as much work as possible on the surface where conditions are more favorable, short lengths of pipe or nipples of the proper size are inserted in the exhaust, steam, and discharge outlets, before the pump goes down. But in lowering the pump these nipples are often broken off within the thread of the casing, which causes much trouble in taking the broken parts out. The alternative of putting together these nipples, elbows, and unions (all from awkward position in the shaft) is a slow and troublesome task. Unless the men are familiar with the work, it necessitates much running up and down to get the proper sizes, loss of others in the sump below, perhaps accompanied by some wrenches that can ill be spared. There is surely no mechanical objections to designing a pump in such a manner that the pipes enter and leave parallel with the length of the pump. This would not only make the pump easier to handle in itself, but it would also facilitate the attachment of the pipe lengths. Again, is there not such a thing as a good reliable tight swivel joint that could be profitably used on all these pipe lines and placed on the pump on the top of the short entrance piece? As already stated, the pump and the piping cannot always be maintained in a parallel line, and it is always difficult to make the pipes come near to their proper positions on the pump. A joint that would permit play at this point would tend to make the handling of the sinking pump quicker. Unless one has stood in a shaft in which the water is rising steadily and fast, he cannot appreciate how anything that would aid speedy work is to be so desired.

A sinking pump is usually placed in a corner out of the way of the ladder and bucket run. This may require a staging over to the pump on which a man can stand while working. Usually when a breakdown does occur, it is found that this has yet to be done, and all the time the staging is being built the water is rising and driving the men away from their work. Is it not possible to have two folding steps attached to the foot of the pump, on which a timberman or pumpman could stand, as shown in Fig. 48. If in addition he had a strap passed round his waist and hooked to some part of the pump or to some special hook provided on the pump, he would be in the most natural position to work at the most troublesome part of the pump (the water chest), would have both his hands free and would be independent of a staging that would have to be put in every time the pump was lowered.

The rods by which a sinking pump is suspended should be adjustable, so that the pump may hang vertically or be made to lean either in or out at the foot as required, as shown in Fig. 48. There are many occasions when a pump that hangs away from the shaft wall at the bottom is much easier to get into position than one that is apt to dig its nose into every spike and rock projection on the way down. There should be little difficulty about this.

Then there arises the question of stud-bolts versus common bolts and nuts, and despite difficulties of design and the additional weight that has to be added to get this, the common bolts are preferable. Everyone knows the condition of the pump which has its water-chest cover or steam-chest cover held in position by two stud bolts while the rest have worked loose or been broken off short. A pumpman working with such a machine faces disaster every day, because there is no way of remedying the trouble except by sending the whole pump, or half of it, to the nearest machine shop, often many miles away. If common bolts had been used, it would have been easy to stock a few sizes and replace them when broken or lost down the shaft. The use of stud and common bolts are shown on the right of Fig. 48. These same reasons make it advisable to have the nuts, single or double, holding the plunger to the continuation of the connecting-rod and the piston to the piston rod, replaced by keys. Nuts strip under hard usage, get lost and cannot be replaced or made on the ground, while at a pinch a key can be made that will temporarily answer the purpose.

A plunger should not be threaded to the piston-rod continuation, but it should pass through the plunger. This has to be a water-tight connection or some day the plunger will be found split from a "freeze." The piston, piston rod and plunger must pull through the pump when the top cover is removed, and if designers would set their wits to devising means whereby some hold could be got on this piston and the rod, at the top, so that it could be hoisted from the pump if need be, they would be doing a favor to pumpmen. Such an arrangement would be a great deal better than the present practice of having a man driving the ram end with a billet of wood and a sledge hammer.

Pumps that are all arms and legs, levers and projecting arms are not suited for shaft work. These things get smashed, bent and broken too easily, but at the same time some arrangement of a bypass from discharge column to suction column is certainly handy. On the discharge column a valve immediately above the pump — a clack valve which will hold up the column of water — is useful, but it would be more useful still if some of the water in the discharge column could be utilized for flushing the water chamber and its valves under proper regulation.

One sometimes wonders why it is that pump makers make the entrance to the water chest and the steam chest, or the steam-valve chest, so small that a man's hand can't get properly inside. This is especially true of the valve chamber in the water chest, which is apparently designed to admit a small, ladylike hand and not that of an average burly miner, who is lucky if he can get two fingers through with which to extract a piece of rock or other foreign matter that is interfering with the proper working of the pump.

The question of screwed-in valve seats compared with pressed-in and keyed seats is perhaps more for the designer to settle than the pump user, but the viewpoint of the man who has to run the pump should not be ignored. In general, seats that are pressed in are more easily removed, replaced, and in case of great necessity temporarily repaired by crude appliances on the site of the actual work, than is the screwed seat. Owing to rusting and other water effects, a threaded seat has most frequently to be cut out with hammer and chisel, an operation which under careless hands may destroy the corresponding thread in the casing and result in the whole pump being sent to the shop, to get rebored and rethreaded and finally ending up with valve seats of assorted sizes.

No doubt pump makers will despise these suggestions as being crude and at variance with good mechanical design and as being the cry of a crank on sinking pumps. But before they are finally dismissed one way or the other, let it be said that they are the results of some bitterly gained experience, of much standing in rubber boots on the water chest of the pumps, in oilskins and hat, with a hot steam exhaust enveloping the pump, and operating under conditions where to make any headway on the water, the change from an upper lift to a lower one had to be made at lightning speed.

Ideal Shop for Sharpening Drill Steel

BY GEORGE H. GILMAN

During the last decade the machine drill as applied to the excavation of rock has been in a state of transition from the cumbersome reciprocating engine to the light pneumatic hammer drill, with the result that machine efficiency has been raised to a point hoped for by only the most optimistic miners of 10 years ago. This great change, largely the result of coöperation between the miner and the rock-drill manufacturer, has been consummated, and for the next few years it is reasonable to assume that, with the object of raising the drilling efficiency of the mine or quarry to correspond, the main lines of development will be in the direction of means and methods for taking full advantage of the hammer drill in its present state — instead of mechanical changes applied to the actuating engine.

The rock-drill manufacturer is constantly striving not only to improve the metallurgy of the drill materials in the effort to secure maximum machine effectiveness and efficiency with minimum weight but also to take full advantage of all improvements in manufacturing methods, thus insuring absolute duplication of parts in regard to their chemical and physical properties, interchangeability and, in addition, the field service that he must render users.

The question that confronts the mine operators is: "Can we stand this new type of progress with its incidental increase in the cost of machine equipment unless we likewise enter upon the program of this new development and take full advantage of it?" The answer is obviously in evidence, for no progressive, right-minded man is harboring the thought of reverting back to hand-drilling methods of the good old days, but instead he is striving to develop all the divisions of his mine to conform with present-day facilities.

The rock-drill bit has been justly termed "the business end of rock excavation" and it is therefore the logical starting point in the campaign to improve drilling efficiency. The importance of the size and quality of the drill steel and of the shape and temper of its bit and shank ends has been emphasized. The purpose, therefore, of this paper is to advance suggestions relative to an efficient method and satisfactory means for the production and upkeep of rock-drill steel.

Fortunately in the mining industry the old-time practice of maintaining secrecy regarding methods and means of production has been broken down, and today a stronger spirit of coöperation exists among mining men than among many operators of kindred industries. This has resulted in knowledge and resources being pooled for the common good. The suggestions advanced in this paper have been gained only as a result of a personal investigation of a great variety of conditions in different sections of the country, made possible by the coöperation of a great many mining companies.

The selection of the best form of equipment to meet a given set of conditions is dependent upon a number of factors, and the extent to which it is profitable to go, in providing facilities for a drill-steel equipment, is largely governed by the volume of material to be handled, which may be determined by the number of rock-drilling machines employed and the character of the ground. In general, it may be assumed that for a mine or other permanent rock-excavating plant of sufficient capacity to warrant the installation of a mechanical drill-sharpening machine a drill-steel plant equipped with the facilities herein described and possessing the following qualifications is warranted.

Primarily a drill-steel plant having one mechanical sharpener should be designed so that the output of the smith and his helper may be doubled by the addition of two men, without any increase in the original shop equipment being necessary, and to provide for possible growth, it should be so constructed that by adding space and certain features of equipment to one end only of the building the capacity may be increased further at slight expense without making any change in the original equipment; that the various operations required to make the drill steel and to reforge and temper used steel are performed with the fewest possible movements and with the least amount of exertion on the part of the operator; and that the character of the work when executed conforms with the established standard for the existing conditions.

To secure these results the building should be of such size that the various machines and their appurtenances may be placed in the most advantageous position. It should be built of materials that are accessible and in conformity with the standard surface equipment of the mine. It should be lighted in such a manner that the piece being worked upon and the operating end of the machine employed are clearly visible to the operator, so that his vision of the parts is not obstructed by shadows; in addition, to meet the requirements of a shop of the size and type herein described, certain features of equipment are recommended in Table I.

TABLE XI.

EQUIPMENT FOR DRILL-STEEL SHOP

Transportation

106 Ft. of 24-in. gage track (rail 35 lb. per yd.) comprising 36 wood ties 4 x 6 x 36 in.
4 Mine-car turntables — 24-in. gage
1 Turn plate (if required)
3 Compartment cars with revolving body (Copper Queen type)
3 Division cars with detachable body (Copper Queen type)

Furnace

1 Forging furnace (oil or gas burner)
1 Tempering furnace (oil or gas burner)
2 Drill-steel supports for furnaces
2 Ventilators with hoods for furnaces
1 Oil storage tank
1-in. pipe with fittings for low-pressure air supply
½-in. pipe with fittings for fuel supply.
2 Pyrometers, equipped with base metal couples having indicating voltmeter and cabinet
¾-in. electrical-conduit piping (for pyrometers)
2 Pyrene fire extinguishers

Sharpener

1 Drill sharpener with full equipment of dies and dollies including supplementary gaging dies and pneumatic-hammer punch
1¼-in. pipe with fittings for compressed-air supply
1 Storage cabinet for drill-sharpener appurtenances
2 Suspended drill-steel supports (for sharpener and grinder)
1 Floor grinder (direct electric-driven) with two grinding wheels 18 x 3 in.
1 Electric switchboard for motor

Tempering
- 1 Rotary quenching and tempering machine (Homestake type), with 1-hp. motor
- 1 Electric switchboard for tempering-machine motor
- 1 Oil-quenching tank with cyanide receptacle
- 1 Ventilator with hood for cyanide receptacle

Racks, Tables and Accessories
- 1 Inspection and bundling rack with incline, equipped with one gage cabinet
- 1 Storage rack for drill-steel bar stock
- 1 Storage rack for finished drill steel
- 1 Used steel table

Sanitary Equipment
- 1 Sanitary drinking fountain
- 4 Steel double-deck lockers
- 2 Wash basins
- 1 Sanitary towel rack
- 1 Urinal
- 1 Flush bowl
- 1 Emergency cabinet

General Equipment
- 1 Work bench equipped with 5-in. machinist's vise
- 1 Desk with drawers and hinged top
- 1 300-lb. anvil
- 1 Set of hand chisels and sledges
- 2 500-watt Mazda lamps with reflectors
- 1 60-watt Mazda lamp with reflector
- Water piping
- Compressed-air piping and hose equipped with Davies blowgun
- Waste piping
- Oil piping
- Electric wiring
- Concrete floor

In order to obtain maximum efficiency in a drill-steel plant it is essential that the transportation of the drill steel be accomplished expeditiously and with a minimum effort, for, regardless of how suitable the mechanical equipment may be, it is impossible to secure and maintain maximum production from the machines and furnaces if a suitable means of transportation conforming to the requirements of the plant is not provided.

To meet the requirements of the type of plant described in this paper it is recommended that the shop, Fig. 49, be equipped with a permanent track and turntable the gage of which corresponds to the standard gage of the mine or quarry tracks, provided such gage does not exceed 36 in. For transporting the drill steel to and from the shaft, or loading platform, a division car of the Copper Queen type, Fig. 51, described in the *Engineering and Mining Journal,* June 24, 1916, will be found to meet the requirements satisfactorily. This car may be equipped with a body detachable from the truck to facilitate the work of transferring the drill steel to the mine skip, cage or wagon without removing the bars from the receptacle, or it may be employed for the purpose intact, and in the case of the mine the whole may be lowered to the different levels where the distribution of the steel may be made in conformity with standard practice. Of the many forms of carriages or trucks employed for the transportation of the drill steel while it is in the transition stage during the process of working, that developed by the Copper Queen Consolidated Mining Co., and described in the article referred to, is perhaps best adapted for a drill-steel plant of this general type.

The compartment car, Fig. 51, consists of a truck having the wheel gage corresponding to the track standard, on which is mounted a revolvable compartment body supported by a ball bearing and equipped with a latch for locking the body with relation to the truck in any desired position. This car is employed for transporting the steel from the sorting table and the oil-quenching tank to the heating forges. In it the various lengths of steel are segregated and by virtue of the revolvable body the insertion and removal of the drill steel is accomplished advantageously. Subsequent to the final heating of the bit the steel is transported mechanically by the tempering machine and deposited upon the inspection rack, Fig. 51, whence it is loaded into the division car and transported from the shop.

The four turntables with which the track system is equipped provide means for the cars to be bypassed at any point except the dead end and, by having the main line extend through the building longitudinally, a convenient means is provided whereby the track system of the shop may be in intercommunication with the yard system and the other departments of the mine plant. At the outside of the building adjacent to the main entry a turn plate is provided, composed of boiler plate 3/8 in. thick, to facilitate the switching of the cars to and from a series of tracks from the yard system which may terminate at the outer edge of the plate.

The heating of rock-drill steel preparatory to forging and tempering the bit and shank end is an engineering problem and it must be considered as such if maximum efficiency is to be obtained. The success of the furnaces employed depends upon the type and the method adopted for their operation. In order to perform their function in the most uniform and economical manner they must be constructed according to scientific principles in which the following considerations serve as a basis:

1. Size and shape of the drill steel to be treated.
2. Quantity of drill steel to be treated during a specified time.
3. Temperature required to insure proper results, and means for its regulation and maintenance.
4. Local conditions.
5. Most convenient and economical cycle of operations.

Gas as fuel is recommended where it is available at low cost but as oil is of necessity adopted in a large majority of mining camps in this country, the employment of either as a heating agent will apply to the type of furnace described herein. For present-day requirements of the average mine or quarry, situated in many instances in remote and inaccessible places, where the carrying of a varied stock of refractory material is not only difficult but also expensive, simplicity of furnace design with special regard to repairs should be considered primarily. With this in view the refractory material used in the construction of a furnace should be composed solely of standard-size firebrick, procurable in almost every local market, and it should be assembled in such a manner as to enable any blacksmith to repair or rebuild it with ease and success.

The heating chamber of the furnace should preferably be of such size and shape, and its burner so positioned, that the steel is heated by reflection or radiation rather than by direct impingement of the flame on the metal. This will prevent blistering and reduce the formation of excessive scale which is likely to result if the flame is caused to play directly on the metal. The heating chamber should be of such size as to absorb economically the heat of the products of combustion.

A certain amount of flexibility that may be taken advantage of by an untrained operator is desirable, inasmuch as the character of the flame required for heating, to the best advantage, standard types of 7/8-in. hexagon and $1\frac{1}{4}$-in. round hollow steel will vary with the size of the bit and the shape of the drill shank; and, in the case of the lugs or collars on the shank, will require only the heating of that portion to be upset, without affecting the extreme end of the bar, and thus obviate the necessity of cooling the tip before it is inserted in the dies of the drill sharpener. This flexibility may be secured by employing a burner of such construction that a perfectly round or fan-shaped flame is produced at will by the operator by merely adjusting a thumbscrew. Another desirable feature of such a furnace is to have at the end of the chamber from which the drill steel is withdrawn a maximum heat zone of constant and uniform temperature, from which the temperature is sure of approximately 15 lb, below the lowest fluctuation of the main-line pressure. This reducing valve may be placed in the wall adjacent to the point at which air enters the building from which a supplementary low-pressure air duct is provided for the two furnaces, as shown in Fig. 50.

To obtain the best results from any type of burner in point of economy and ease of operation it is of prime importance to have both the oil and the atomizing agent as constant and steady as possible. After passing through the pressure-reducing valve it is desirable to have the air for the furnace supply pass through a pipe mounted directly over and running lengthwise with the mouth of the furnace. In this pipe provision may be

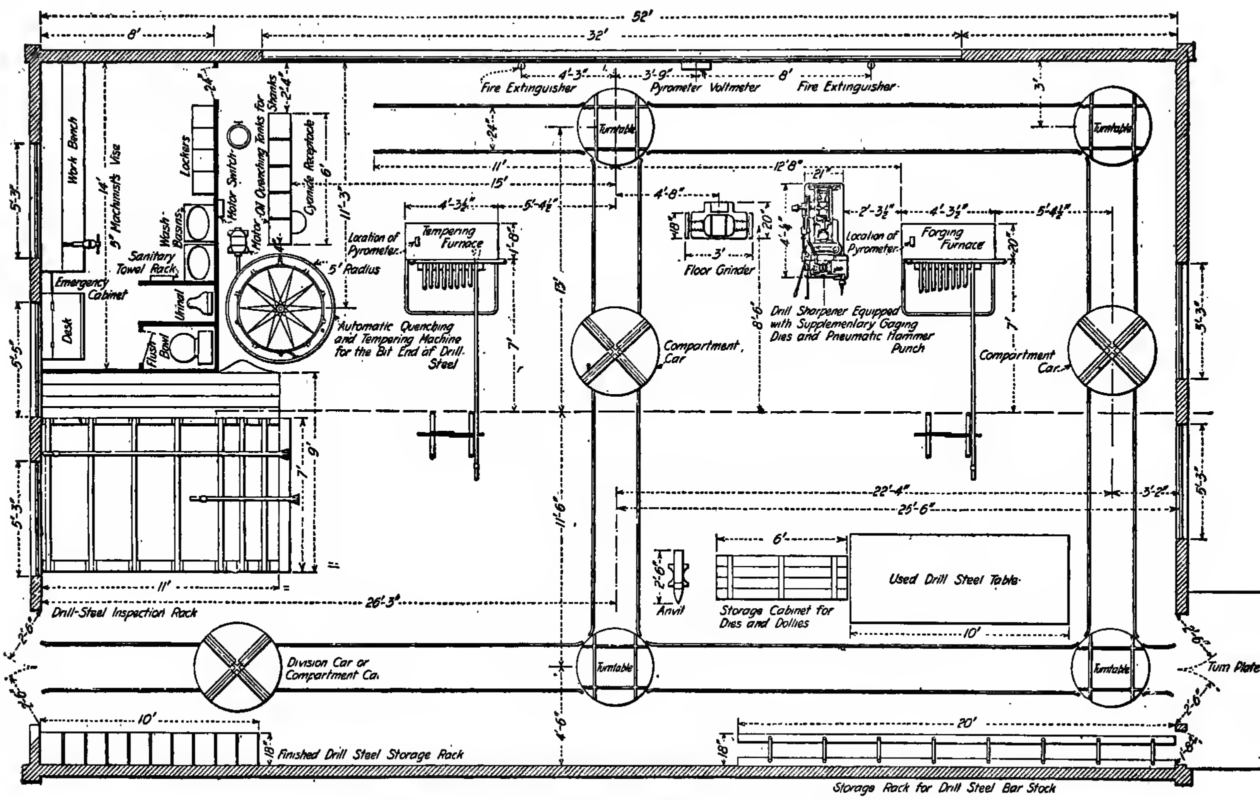

FIG. 49. FLOOR PLAN OF IDEAL SHOP FOR SHARPENING STEEL

lowered gradually but to an appreciable degree toward the end opposite where the steel is inserted. This will give the steel time to absorb the heat gradually, and will secure uniformity in the opening of the grain as the steel bars are handled in rotation.

It has been determined by experiment that the time period of 4 min. is desirable for heating the larger sizes of bits formed on the $1\frac{1}{4}$-in. round hollow drill steel and in order to insure an output of one steel every 30 sec. from the forging furnace, the mouth or opening for the insertion of the steel should be 2 ft. 6 in. long with a height of $3\frac{1}{2}$ in. Pyrometer readings should be taken from that part of the furnace directly adjacent to the drill steel at the point of withdrawal.

If desirable, the low-pressure air supply for the furnace may be secured from the main high-pressure air-supply of the mine or quarry, in which case a pressure-reducing valve should be used to secure a terminal pressure made by drilling a series of small holes directed downward — for the escape of a part of the air which will serve the double function of preventing the waste gases from flowing out in the face of the operator — and assisting in keeping cool that section of the drill-steel bar that projects from the furnace. After passing along the mouth of the furnace the air should preferably make two three-quarter circuits on the top of the brickwork, but inside of the cast-iron casing, for pre-heating it, preparatory to admission to the burner. Fig. 51 shows a furnace constructed along these lines.

The drill-steel support shown in Fig. 51 provides a simple, rugged and convenient means for supporting the drill steel during the heating operation. Constructed from flat bars of $\frac{1}{2}$ x 2-in. low-carbon machinery steel or wrought iron it may be made by any blacksmith with an ordinary forge-shop equipment.

By far the most essential requisite of a drill-steel plant is the machine for forging the bit and shank ends of the drill steel. There are few apparently unimportant items of expense that are likely to assume more serious proportions than the wasting of time by men, caused by failure of the machine to produce the finished product in a condition that meets the requirements of the plant. Correct sizing and shaping of rock-drill steel are equally as essential to efficient mining as the efficiency of the drill itself and, therefore, a machine sharpener that conforms with these requirements and in addition produces the bit and shank end in the best possible condition to be tempered should be adopted.

It is a well-known fact that the "standing-up" quality of a piece of steel when forged to a finished shape while the wheel opposite, which should be preferably of somewhat finer grain, may be employed for refacing dies and dollies and other general grinding. The speed of the grinder when driven by a 5-hp. motor should be 1150 r.p.m. When grinding drill shanks the steel may be supported by a hook suspended from an overhead trolley having its track mounted in alignment with the grinder wheel and at right angles to the grinder shaft. Such a support is described in the *Journal* of Feb. 10, 1917, by James E. O'Rourke.

The heat treatment of rock-drill steel is a factor which determines success, as applied to the excavation of rock, to as great a degree as the heat treatment that is accorded the various parts that enter into the construction

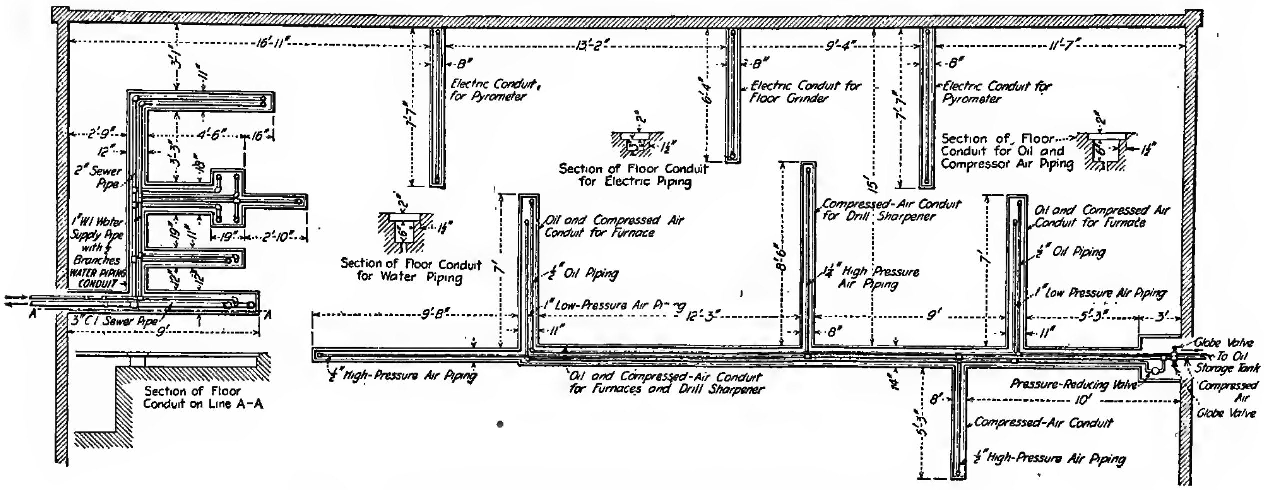

FIG. 50. PIPING SYSTEM UNDER FLOOR OF DRILL SHARPENING SHOP

by hammering, under light rapid percussive blows, is superior to that secured by any other known method. For this reason a drill sharpening machine answering this requirement is desirable. It should be equipped with means supplementary to the main forging dies for gaging the various sizes of bits mechanically, within close limits, and for punching out the axial hole of the steel at both the bit and shank ends. In construction it should be strong enough to withstand without undue wear or breakage the severe service to which it is subjected and all control levels should be interlocked in order to insure the various operations being performed in proper sequence and with a minimum danger both with respect to the operator and the piece being worked upon.

For the storage of dies, dollies and other drill-sharpener appurtenances the cabinet shown by Fig. 52 will answer the requirements satisfactorily. Placed at a convenient distance with respect to the drill sharpener and between the used-drill-steel table and the anvil, the heights of which correspond, it serves also as a support for the drill-steel bar stock when being cut to proper length preparatory to forging and also as a straightening bed for bent steel by virtue of the three bars of iron rail with which the top of the cabinet is equipped.

A floor grinder, preferably direct-motor-driven and equipped with two 18 x 3-in. corundum or carborundum wheels, should be placed at the left of the drill sharpener where it will be readily accessible to the operator. The function of the wheel adjacent to the sharpener is that of squaring off the ends and redressing drill shanks, of the drill itself and, when the requirements of the work are once fixed, it is of the utmost importance that the temper of each piece be the same. This result may only be secured by reducing to a minimum the personal equation. While many expert steel workers are able to determine by experiment the required hardening heat and the cooling method for steel of various chemical compositions, the human potential is influenced by the physical condition of the man, and in order to carry the operation of tempering beyond the point of human influence, machiery must be employed as a substitute.

For the requirements of a drill-steel plant of the size and type described herein, the tempering machine for the bit end of the steel as developed[1] by the Homestake Mining Co. is perhaps the most satisfactory. This machine, preferably motor driven as shown in the plan view, Fig. 44, is placed at a point adjacent both to the tempering furnace and to the main water supply, and is so positioned and equipped with guide hangers that the drill steel, when discharged, falls on the inspection rack where it is accessible for inspection. Back of the tempering machine and adjacent to the heating furnace, where it is accessible to the compartment car when placed near the dead end of the track, is the oil-quenching compartment tank shown in Fig. 52 with its supplementary receptacle for cyanide of potassium. This is employed only for tempering the shank end of the bar in the manner described in my article on "Drill Steel and Drill Bits for Metal Mining," published in

[1] "Eng. and Min. Journ.," May 12, 1917.

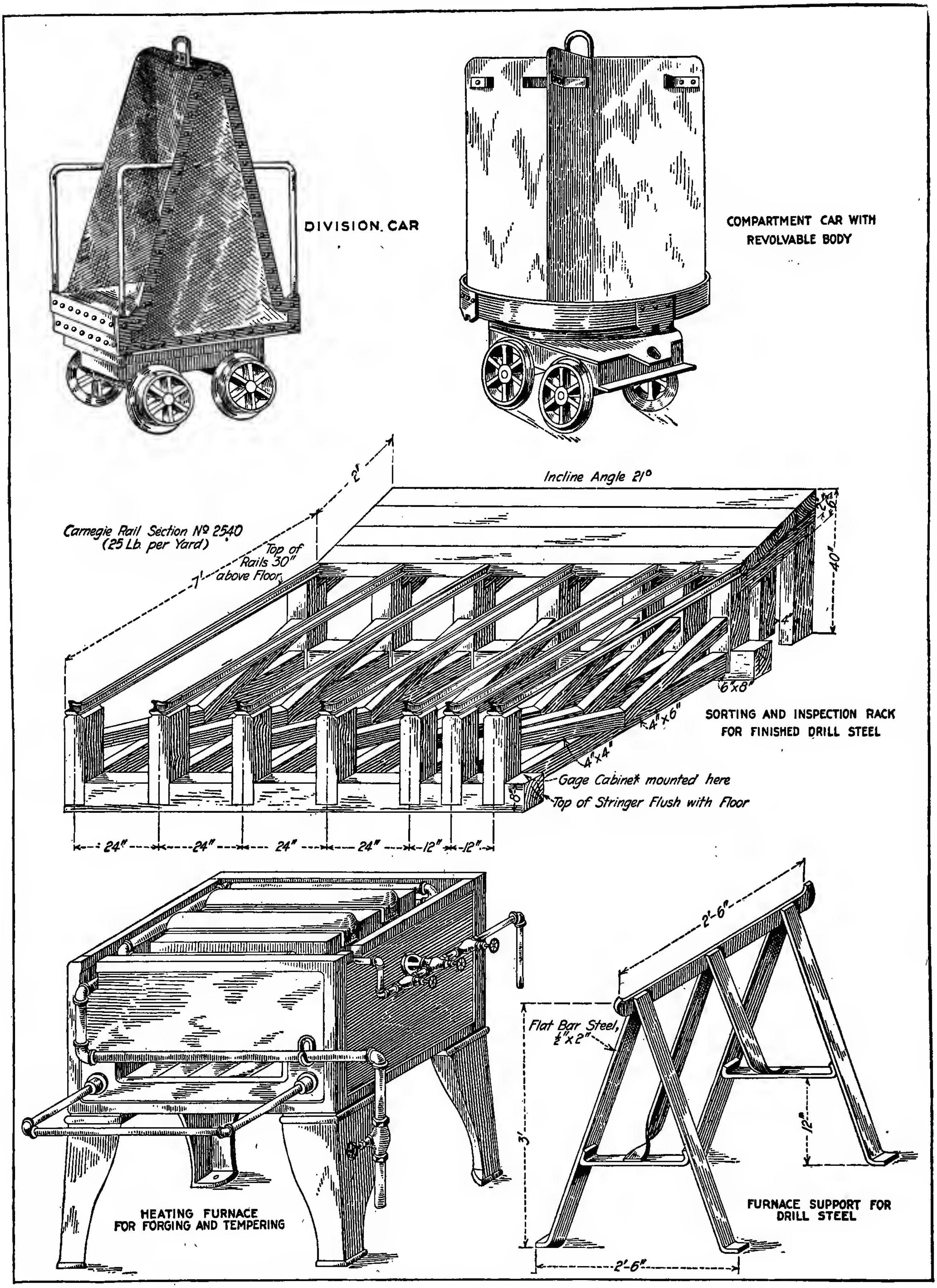

FIG. 51. ACCESSORY EQUIPMENT FOR IDEAL SHOP FOR SHARPENING DRILL STEEL

the *Engineering and Mining Journal* of May 12, 1917. Both tanks are preferably of steel construction and it is recommended that the inner surfaces be coated with an acid-resisting paint as insurance against oxidation.

The used-drill-steel table of the type shown by Fig. 52, on which the used drill steel is deposited from the division car, provides a means for facilitating preliminary inspection. If bent or plugged steel is found, the defects are corrected by using the straightening and cleaning facilities, provided on the top of the drill-sharpener cabinet before the steel is loaded into the compartment car preparatory to heating for forging. In the case of broken steel, it is cut to length, piled on the floor at some convenient point until a sufficient quantity has accumulated, when, as regards the necessary forging and tempering operations, it is treated as new steel.

For the storage of finished drill steel a rack, shown by Fig. 52, and placed adjacent to the inspection sorting rack is recommended. This rack should have compartment preferably made of boiler plate, and should be equipped with a base plank of hard wood to protect the floor and to insure the bit ends of the finished steel against chipping. The various compartments serve to accommodate the different lengths and types of drill steel that may be employed. From its position, as shown by Fig. 49, the finished drill steel may, when required, be transported directly to the division car or to the inspection rack for bundling.

A convenient and inexpensive storage rack for drill-steel bar stock is shown by Fig. 52. This may be constructed by any experienced mechanic or blacksmith from standard materials usually sold by hardware stores in every mine and quarry district. Preferably the base timbers are embedded in the concrete floor while the top stringers may be fastened by any means convenient to the girders of the building. From its position opposite the supplementary doorway in the east end of the building (Fig. 46), where it is conveniently situated for receiving the material, the various sections and sizes of drill-steel bar stock may be readily removed and carried to the anvil and straightening bed for cutting to the desired length for forging.

For inspecting the finished drill steel a rack, the details of which are shown by Fig. 51, will be found convenient, as it permits the inspector to walk between the rails and thereby facilitates the work of inspecting and sorting. After inspection the steel bars may be easily rolled to the edge near the track, from which they can be loaded directly into the division car, or transferred to the finished drill-steel rack as desired. As shown by the general plan, the inspector's gage cabinet, Fig. 52, is mounted on the timbers at the operating end of the rack where it is conveniently accessible. In this cabinet, which is equipped with sliding doors, wood pins and shelves, means are provided for storing the necessary gages and measuring devices.

Applications of the principles of scientific management have demonstrated the importance of providing sanitary facilities, not only as applied to the surface equipment of the mine but to the underground workings. The wide-awake mine manager is striving to narrow the gulf between himself and his workmen by the giving of happiness and the meting out of justice. Health is the foundation of happiness and it is on this theory that industrial manufacturers have for years recognized the economic advantage of substituting clean, light and well ventilated buildings for the unsanitary shops of the past.

A workman who finds pleasure in his work and his environment usually prides himself in producing a satisfactory product. It is on this theory that toilet facilities are recommended as applied to the mine drill-steel shop. The toilet room illustrated on the plan view is in a corner of the building where the space it occupies is not required for other equipment. It is adjacent to that part of the plant where running water is required for the tempering machine so that the branch length of water piping necessary for the flush bowl, urinal and wash basin is reduced to a minimum. These accessories are placed near the inner wall of the toilet room in preference to the outer wall of the building, to minimize the liability of freezing in cold weather. Furthermore, this arrangement permits the placing of the work bench and the foreman's desk, Fig. 47, where the light from outside may be used to the best advantage. Duoble-deck lockers, one for each workman, an emergency Red Cross cabinet and a sanitary paper-towel rack, in addition to the sanitary drinking fountain, placed in the main shop adjacent to the toilet room, complete the sanitary equipment.

Compressed air is necessarily used for operating the drill sharpener and, in the absence of a blower, for operating the furnaces. Another desirable use for the air is to facilitate the work of inspecting the hole in hollow drill steel and to assist in cleaning out plugged steel. For this latter purpose the air is conducted by a pipe line to a point adjacent to both the used drill-steel table and the inspection rack, Fig. 50, from which it may be conducted to the work by a 12-ft. length of 7/16-in plain, pneumatic, air hose equipped with a Davies blowgun [2] (the invention of Thomas Davies, head blacksmith of the United Verde Copper Co.). In the use of this feature the operator, by merely pressing the nozzle of the gun against the mouth of the hole in the drill steel, causes a valve to be automatically opened which allows a blast of air under line pressure to be blown through the opening. The act of removing the nozzle of the gun from the end of the steel causes the push throttle to be closed automatically. If preferable for the final inspection of the drill steel, water under pressure instead of compressed air may be employed at the inspection rack, in which case the nozzle of the gun is preferably equipped with a backing flange and a soft rubber gasket, which, when compressed against the end of the steel shank, provides a seal and prevents undue leakage. In Fig. 50 subfloor conduits for the compressed-air piping are shown.

For heating the building in cold weather a four-coil radiator composed of 2-in. wrought pipe, extending from the west entrance and along the west, north and east walls to the east entrance, will answer requirements satisfactorily except for extremely cold climates in which which additional coils may be required. Live or exhaust steam may be employed for the purpose but in either

[2] "Drill Sharpening Methods at the United Verde Mine, Arizona," by Frank Richards, "Eng. News," Feb. 1, 1917.

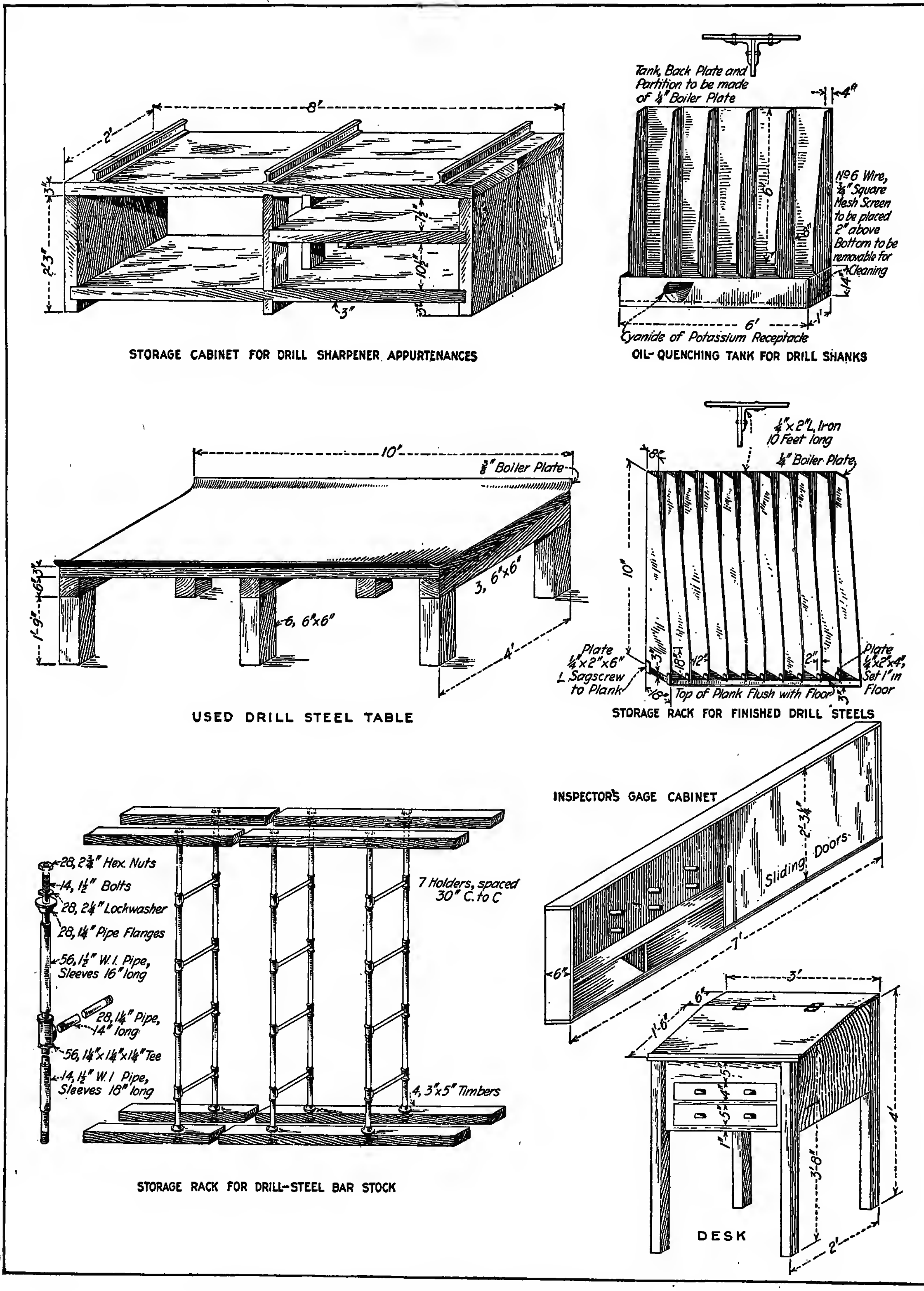

FIG. 52. ACCESSORY EQUIPMENT FOR IDEAL SHOP FOR SHARPENING STEEL

case the precaution of providing suitable drains in the system should be taken to insure against freezing.

The lighting and ventilating facilities of the plant are items that should receive due consideration. In Fig. 49 is shown the location and size of the windows recommended for a plant of the type herein described, on the assumption that the building is placed on a line running east and west and the normal position of the operator faces in a northerly direction to avoid the direct rays of the sun falling upon the front side of the furnace and drill sharpener. A steel sash pivoted at the sides is recommended for the windows and may be operated by means of a rocker shaft procurable from the manufacturers of such apparatus. The windows at the ends of the building may, if desired, be composed of four rows of sash, while the sill of the window in the north side of the building should preferably be mounted at least 7 ft. above the floor and composed of eight panels, in a single row, operated by one gear from the rocker shaft. From such construction, a fair degree of natural light and ventilation is secured. For exhausting the gases from the heating furnaces, suspended hoods, adjustable vertically and equipped with telescope flues extending through the roof of the building, should be provided.

Artificial light sufficient to meet the requirements of the main shop may be secured from two 500-watt Mazda lamps with reflectors, supported from the roof on a center line through the building, one of which should be placed at a point slightly to the left of the drill sharpener, while the other should be slightly to the left of the tempering furnace. One 60-watt Mazda lamp suspended by an extension cord directly over the desk in the toilet room will be found ample for the purpose.

The floor of the shop should preferably be of concrete, poured after the wood base plates of the various fixtures, including ties for the track system, are placed in position. Conduits for all subfloor piping of such size and shape that ample clearance for the pipe and its fittings, with provision for a covering with wood plank, should be provided (see Fig. 50). The advantage of this construction is obvious as it provides a means for making the system readily accessible in case the pipe should become plugged or broken through accident. If desirable, a wood floor pad made of 2-in. planks may be embedded in the concrete at various points repeatedly covered by the operators. This provides for a certain amount of resiliency which will be found advantageous.

By adding 24 ft. to the east end of the building and the installation of an extra drill-sharpening machine, heating furnace, grinder, cabinet for drill sharpener appurtenances, and track with one turntable, the output of the plant may again be increased approximately 50% by the addition of two operators. The ultimate capacity of the plant will then correspond to the requirements of all but a few of the largest users of rock drills in this country having a central drill-steel plant.

Remove the drill-steel bar from the bar-stock rack and place it on the top of the drill-sharpener cabinet with the end of the bar to be cut off supported by the anvil. Nick the bar at the desired point, using a chisel and sledge and break to length, leaving temporarily the pieces to be worked upon on the drill-steel table. When a sufficient quantity of pieces are available, place in the heating furnace, handling the steel in rotation by inserting a new piece when one is removed, after having attained the required degree of heat for forging. Form the shank in the drill sharpener, using the grinder and pneumatic punch, if necessary, after which place in compartment car, shank end down. When the forging operation is completed, turn the compartment car one-half way around, place the shank end of the bar in the tempering furnace, handling the steel in rotation as before and, when the shank has been heated to the required degree for tempering, remove from the furnace, dip the tip end in the cyanide of potassium and then plunge the heated end in the oil vat, allowing it to remain in the oil until cool. Now remove the steel from the oil vat and place it in the compartment car shank-end down. Run the car to its position adjacent to the forging furnace, remove the bar and insert the blank end in the furnace, handling it in rotation as before. Form the bit to the required gage diameter for the particular length of the piece being worked, after which place it in the compartment car at the left of the furnace. When the forging operation of the bit end is completed, turn the compartment car halfway around, place the steel in the tempering furnace, handling it in rotation as before and, when the required degree of heat is attained, quickly insert it in the bath of the tempering machine. After the steel has reached the inspection rack, inspect each piece carefully, place the pieces that pass inspection in the finished-drill-steel storage rack or in the division car and rework the defective pieces if any are found.

After the used drill steel is transported by means of the division car from the shaft, or receiving platform outside of the building, it is deposited on the used-drill-steel table, where it is subjected to a preliminary inspection. If bent or plugged drill steels are found they are straightened and cleaned out by the use of the anvil in conjunction with the straightening bed on top of the sharpening-machine cabinet, after which they are sorted and placed in the compartment car. If a shank is found broken to such an extent that reforging and tempering are warranted, the steel is discarded temporarily and placed in position convenient for reforging when a sufficient quantity has accumulated to warrant the operation, in which case the procedure will be the same as that required for shanking new drill steel.

In the case of shanks that merely require regrinding to put them into proper working condition the steels are placed in the compartment car (shank end down), which, after being loaded, is pushed to a point adjacent to the grinder, which such steels are removed and redressed, after which they are placed in the car (bit end down) and the whole is transported to a point adjacent to the forging furnace where the procedure for sharpening the bit is the same as that required for the bit end of new drill steel. After the final inspection they are again bundled, if such be the practice, loaded into the division car and transported to the shaft, or loading platform, for distribution.

If desirable, a lessening of the distance covered by the operator, while removing the steels from the furnace, forging and grinding prior to depositing them in the compartment car, can be made. The grinder, drill sharpener, and forging furnace may be grouped more closely together and the sharpener and furnace set at an angle of 70° with relation to the longitudinal line of the building by swinging the working end of the sharpener and forging furnace in a westerly direction from the position shown in Figs. 49 and 50. The oil-quenching tank for drill shanks may also be set at an angle of 60° with relation to the tempéring furnace by swinging the end of the tank nearest the north wall in an easterly direction.

Salt Mining and Dressing

BY J. B. CALKINS [1]

The production of salt is of economic importance and a sure gage of industrial and social progress. Everything eaten has its portion of salt. The stock raiser must have salt for his cattle and sheep; the packer for his meat; the chemical manufacturer for sodium hydroxide, chloride of lime, other chemicals and asphyxiating gases; the textile manufacturer requires salt for his dyes.

Some of the earliest developments in the mining industry were connected with the production of salt. History records that the Romans found the barbarians mining salt in central Europe before the Christian era. For hundreds of years the evaporation of salt from brine was unknown. It was then introduced into England and mine salt was dissolved in fresh water and then evaporated by the solar process.

Until the eighteenth century practically all the salt used in this country was imported from England. Nevertheless, as the frontier pushed westward in New York State, the settlers found that the Indians knew of a source of salt, and finally discovered that they were boiling down the water that issued from salt springs near where Syracuse is now situated. Until after the Civil War this district was practically the only salt-producing locality on the North American continent, and even then the greater proportion of the salt consumed was imported from Europe. With the introduction of the process by means of which fresh water is forced down through drill holes into the crystalline salt bed between the casing and the tubing, the dissolved salt in the form of brine returning through the tubing for evaporation, many new fields were opened, and salt plants erected. The first operated in New York State were just south of Rochester, in Wyoming and Genesee counties, and later in Tompkins, Schuyler and Oneida counties. This development was followed by the erection of plants in central Kansas, Ohio, lower Michigan and Louisiana; and the discovery of the salt deserts in the Southwest.

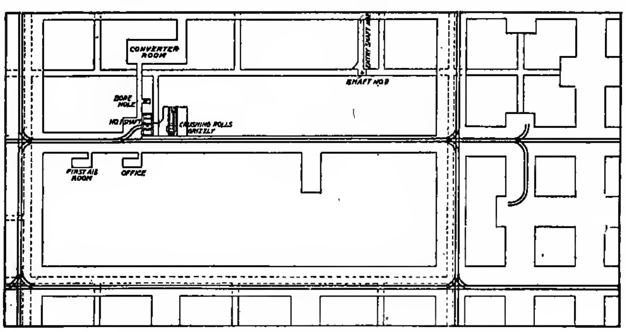

FIG. 53. LAYOUT AT FOOT OF SHAFT

With low initial cost and cheap fuel, the evaporation process has supplied domestic needs and a large part of the industrial business up to the present time; but with the ever-increasing costs and difficulty in getting coal (and it takes nearly a ton of coal to produce two tons of evaporated salt), mine salt is rapidly assuming a commercial status that will enable it to meet industrial needs and bids fair to encroach on the domestic trade. With this new and enlarging demand, the mine-salt industry has received a great impetus. One of the results is the opening of a new mine by the Rock Salt Corporation, near Ithaca, N. Y.

Salt beds in the East are generally fairly regular in their geological features and consist of horizontal strata from 10 to 50 ft. thick, with possibly a slight pitch to the southward. Nevertheless, where this corporation has sunk its shaft rather unusual conditions are found. In exploration, drillings were made about a mile on either side of the shaft site, and salt was found about 1400 and 1500 ft. from the surface, respectively; also water at about 1000 and 1100 ft. The

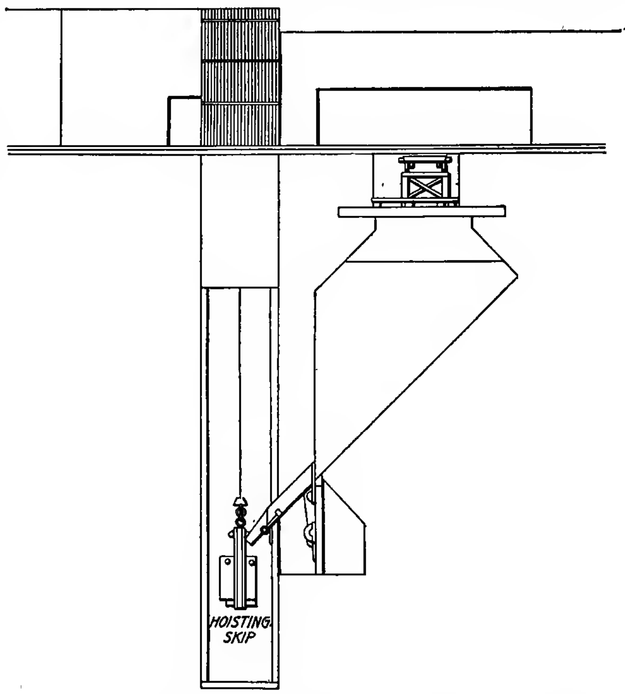

FIG. 54. ARRANGEMENT OF CRUSHER, BIN, SKIP AND GRIZZLY

shaft itself is situated between these two drill holes, and, although only 100 ft. from the shore of Cayuga Lake, is practically dry. What is more remarkable, instead of striking the usual 50 ft. of salt at the average depth of 1450 ft., it was found at 1365 ft. and consisted of an upper bed of 12 ft., then 41 ft. of mixed shale and salt in the middle, and 97 ft. of clear crystalline salt at the bottom, comprising one of the finest salt beds ever developed. The company contemplates working the lower 97-ft. bed.

After a thorough exploration, the company purchased title or mining rights on about 650 acres of land, including a mile front on the lake; and in July, 1916, began work on the shaft and breaker. The shaft is 9 x 21 ft. inside the timbers, and is divided into five compartments, namely, two skip roads 9 ft. x 5 ft. 8 in. each, one-man cage road 5 x 5 ft., an air way 4 x 5 ft., and a ladder way 3 x 5 ft., partitioned off between the

[1] Assistant general manager and chief engineer for the Rock Salt Corporation, Ithaca, New York.

skip roads and the remainder of the shaft to give up- and down-cast ventilation. The shaft is timbered throughout with 10 x 12-in. white oak sets placed on 6-ft. centers and lagged with 2-in. oak plank.

The actual shaft sinking was started early in September, 1916, with a 65-ft. wood headframe, a 12-in. x 18-in. single 5-ft.-drum Lambert hoist, and a 1100 cu.ft. per min. Ingersoll-Rand imperial-type compressor. Three shifts, each of 13 men, worked below ground. Bed rock was encountered at about 12 ft., but the concrete shaft collar was extended to a depth of 36 ft. and the work carried forward at the rate of about 100 ft. per month through comparatively easy-working shale rock to a depth of 840 ft., where a seam of carboniferous shale was reached, and with it a big flow of methane gas. The natural ventilation was not sufficient to take care of the large volume of gas encountered, so work at the bottom had to be suspended in July, 1917, while a 20-in. steel pipe was lowered down the shaft and a gas pump installed. In the meantime the wood headframe was replaced with the first half of a permanent steel tower; and the final double drum, single-clutched, Vulcan electric hoist set up in the newly erected drum house.

Even with the improved ventilation system, it was impossible to take care of the gas, so grouting had to be adopted. This was accomplished in the following manner: First a 7-in. churn-drill hole was put down about 12 ft. from the shaft to relieve the pressure of the gas, and then 2-in. holes were drilled every 2 ft. around the outside of the shaft (at the bottom) into the gas fissure, and successively grouted with a thin paste of cement and water. The grouting was done with a grouting gun made of a length of extra heavy 10-in. pipe with caps at both ends, bored and tapped for a 2-in. pipe. First a short length of 2-in. pipe, tapered at one end, was driven tightly into the drill hole, and the gun was connected to this by suspending it with rope and tackle in the shaft with a valve between the gun and the grout tube. Next the gun was filled with water by opening it at the top, and a bag of cement dumped in. To get a good mixture, compressed air was forced in at the bottom of the gun and allowed to bubble up. When everything was ready to force the charge, the cap at the top was replaced and the 2-in. pipe connected to the water pipe leading down the shaft, and the gate valves in both the grout and the water line were opened. In this way the grout was forced in with the 300-lb. pump pressure at the surface, plus the static head of the water column down the shaft, giving a total pressure of about 700 lb. The valve at the bottom of the gun used for admitting compressed air for mixing the paste served a double purpose, for, while the grout was being forced in, this valve was left slightly open to indicate, by the presence of clear water, when the whole charge had been delivered. When completed, the valve in the grout tube was closed, so as to hold the charge in, and the gun moved over to the next tube.

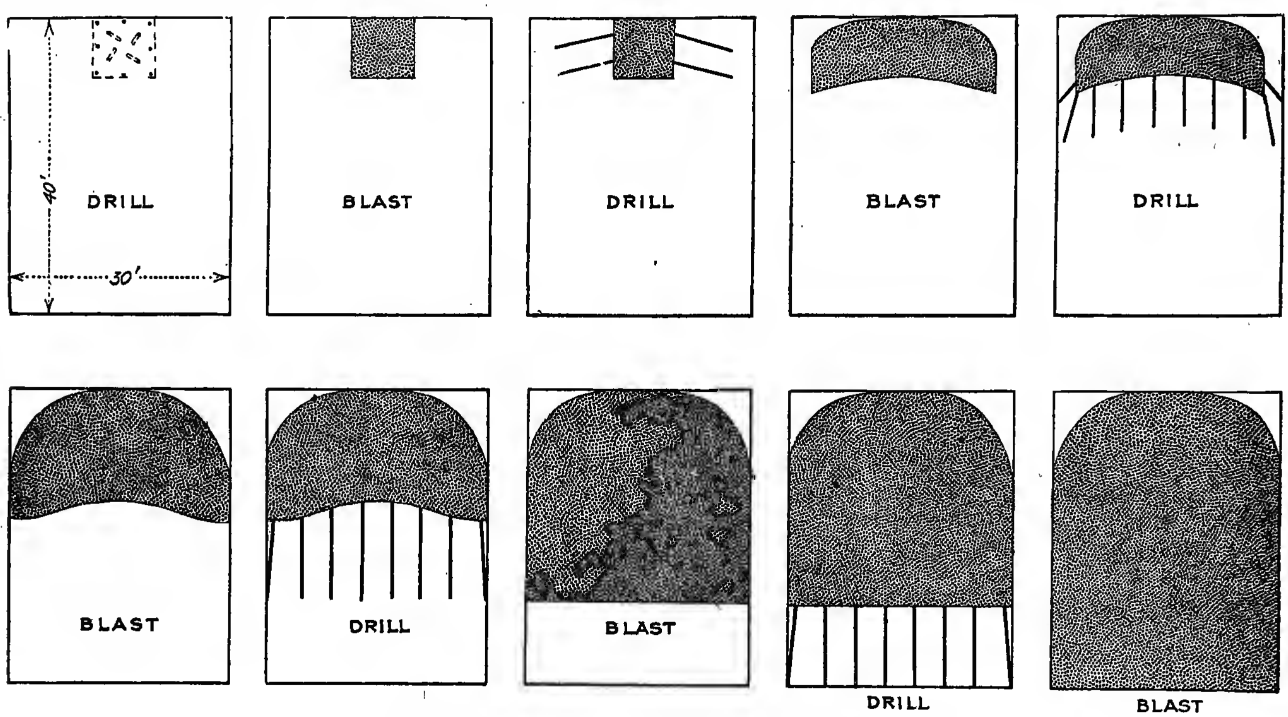

Fig. 55. System for Working Out Gangways

By grouting about forty holes the gas was effectively isolated and shaft sinking continued without further difficulty. The effectiveness of the operation was admirably illustrated when the shaft was cut through the gas-bearing fissure, which was found completely filled with concrete across the entire section. The cement must have penetrated at least 12 ft. beyond the shaft, for it came out into the churn-drill hole and set so hard that the hole had to be reamed out before it could be carried down. Following about 80 ft. of carboniferous shale, which was really a low grade of coal (about 50% carbon), hard limestone was encountered, and the jack-hammers for drilling had to be replaced with Leyner drills. On Jan. 1, 1918, the shaft had reached a depth of 1150 feet.

With regard to outside developments, it may be added that in the autumn of 1916 the company erected a 40 x 60 ft. stucco-tile repair shop, and equipped it with lathe, shaper, pipe machine, drill presses, grinders, and drill sharpener. This was followed by the erection of a 30-room boarding house and a well-equipped "dry" for the miners; also a large, double tenant house. The following spring a drum house was erected similar in size and construction to the repair shop, and grading was started for railroad tracks and the reinforced breaker building. During the latter part of last summer the foundations for the breaker building were laid, and at the first of the year it was nearly half finished. The breaker building will be of fire-proof, reinforced-concrete construction throughout, and will cover an area 90 x 120 ft., the average height being about 100 ft. The steel headframe is incorporated with the breaker and has a total height of 150 ft. Three loading switches will be carried under the breaker for loading bulk salt and one switch along the east side for package salt. With this arrangement the company hopes to be able to load from 60 to 100 cars per day. The breaker building incorporates not only the crushing and screening machinery but also storage for 20,000 tons of prepared sizes of salt.

The character and nature of the salt deposit offer unusually favorable features for large-scale mining; for it is not only homogeneous, but can be mined in large chambers without timbering. With this in view, the company has purchased a large electrically operated Thew shovel, with a 1½-yd. dipper for loading the salt into cars after it has been blasted from the face. There is also a smaller shovel of the same type, which is being used on the surface at present, but which will be sent underground when the mine is opened. Mechanical-shovel operation has led to the development of a rather original system of mining, so as to work out large gangways and at the same time get adequate ventilation, especially after blasts. The entire output of the mine depends upon one or two large shovels, which will necessarily have to stand idle while powder gases are clearing from the workings. The system contemplated calls for taking out 40% of the salt in gangways 30 ft. wide and 40 ft. high, the remainder being left in the form of square columns 40 x 40 ft. for roof support.

To open the working face, an 8 x 8 ft. tunnel is driven ahead of the gangway, and later enlarged crescent shaped so as to open a regular bench system of quarry blasting, and also form an arched roof which can be readily trimmed without the use of stagings. Jack-hammers can be employed to drill vertical holes 4 or 5 ft. from the face, and the salt blasted down directly in front of the shovel. This 8 x 8 ft. drift or tunnel serves a secondary purpose as a means of ventilation. The gangways are started 140 ft. apart, with an 8 x 8 ft. ventilation drift half way between them, which is connected with the down-cast part of the shaft. Every 70 ft., cross drifts are carried out at right angles to this ventilation tunnel, until they meet the drifts ahead of the gangways. Thus fresh air is carried from the ventilation fan down the shaft, through the center tunnel between the gangways, across to the drifts ahead of the gangways, and thence back directly into the working face. The foul air ahead is driven through the mine workings and up the shaft. As each succeeding cross drift is completed, the previously used one is bratticed up so as to force the air around through the new drift, thus always keeping it coming in from the front of the working face, instead of along the side.

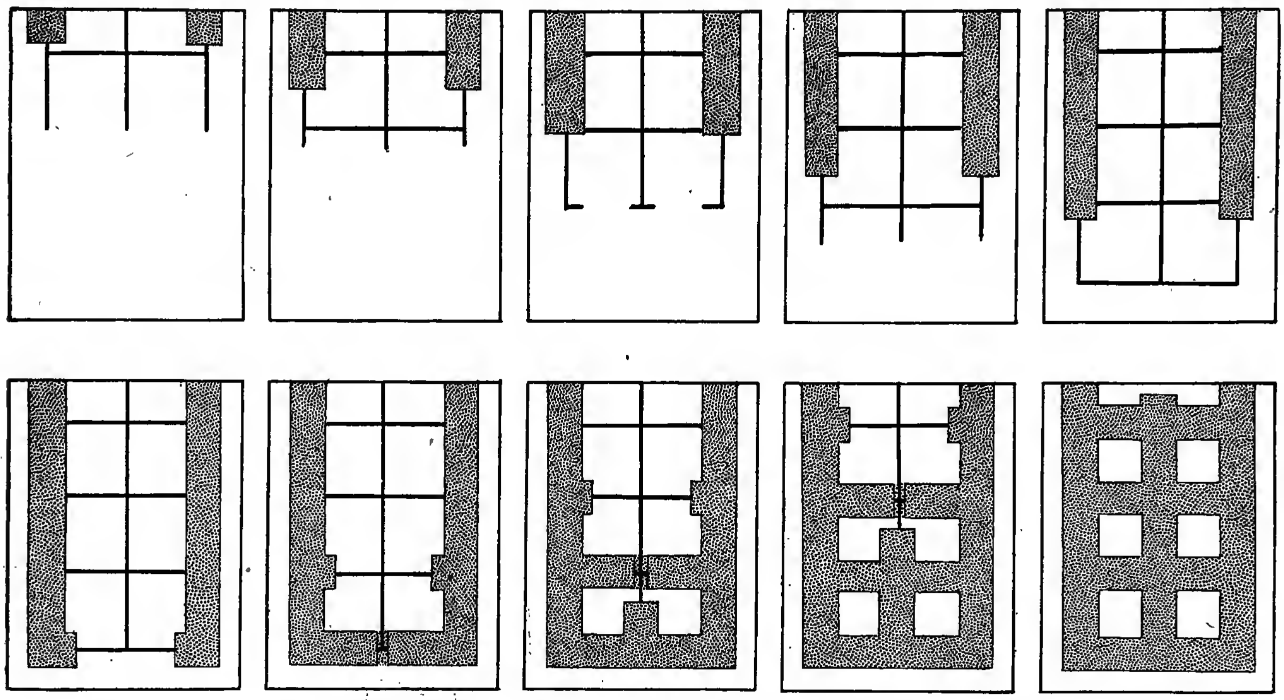

FIG. 56. PILLAR AND BLOCK SYSTEM OF MINING SALT

When a gangway is completed as far as the property line, work is started on the last cross drift, and, when this is completed, back on the center air tunnel. This does not affect the ventilation system, for the fresh air is still coming directly into the face of the working; but it does reclaim the drifts, which have thus far been used only as a means of ventilation. It also possesses the advantage of a robbing-back system in which the men are always working under the heavily supported portions, instead of where the full amount of salt has been worked out. When completed, all the airways have been used as a means of working out a quarry face, and nothing is left but 40 x 40 ft. salt columns to support the roof.

The salt, after being blasted down from the working face, is shoveled into 6-ton dump-bottom cars, and hauled by electric locomotives to the foot of the shaft, where it is dumped on a grizzly. The oversize passes through a pair of 40 x 36-in. crushing rolls, which reduce the product to 4 in., and is stored in a 5000-ton skip pocket beneath. Thence it is spouted into 5-ton self-dumping skips and hoisted to the surface, where it goes through the breaker. Here it is reduced to 1 in. and screened for the four commercial grades — 3/8 in., 1/4 in., 1/8 in., and common fine. The oversize passes through another set of crushing rolls, which reduce it to 3/8 in., and over a second set of shaking screens. The entire breaker equipment is laid out in duplicate, and, with the exception of the distribution over the bins, the material is carried through by gravity.

The hoisting equipment consists of a double drum 7 ft. diam. 4 ft. face single-clutched Vulcan hoist, equipped for shaft sinking, with double reduction gearing, and driven by a 100 hp. 550-volt alternating-current motor. With the completion of the shaft, the second set of reduction gears will be removed and the 100 hp. motor replaced with a magnetically controlled 500 hp. 2200-volt alternating-current motor. The hoist is equipped with automatic air brakes and all the latest safety devices for the prevention of overwind, overspeed, and reversal in case of failure of power. The 100 hp. motor taken from the Vulcan hoist will be used to convert the Lambert steam hoist, on the man cage, to an electric hoist; but the steam rigging will be kept intact and boilers will be kept ready, so that the men can be taken out of the mine in case of failure of the electric current.

Power is purchased from the Cayuga Power Corporation, received under a potential of 11,000 volts and transformed to 2200 for the large hoist and 550 for the smaller motors. Electric service for the mine will be carried down the bore hole as 550-volt, alternating current, and converted into 250-volt, direct current, at a substation below ground. All electrical equipment has been furnished by the General Electric Company.

The ultimate capacity of the plant will be 2000 tons of prepared sizes of salt per 10-hr. day, and it will have shipping capacity for twice that amount. Shipments may be made either in bulk or package, and by freight or water via New York State barge canal. The erection of the plant is under the general supervision of M. E. Calkins, A. S. M. E.; engineering design and purchasing, J. B. Calkins, M. E.; concrete construction Alexander Shumway & Utz Co.; and steel work, Seneca Engineering Company.