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A
MANUAL OF MINING.

BASED ON THE COURSE OF LECTURES ON MINING
DELIVERED AT THE SCHOOL OF MINES
OF THE STATE OF COLORADO.

agnus olbjörn
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TO

Ogden N. Rood, A.M.,

PROFESSOR OF PHYSICS, COLUMBIA COLLEGE, NEW YORK CITY,

WITH RESPECT AND ADMIRATION, THIS

VOLUME IS DEDICATED

BY

THE AUTHOR.

PREFACE.

THIS treatise is an abbreviation of a course of lectures upon mining, delivered at the School of Mines of the State of Colorado, and is issued with the advice of its Board of Trustees, which recognizes the importance of having, within a moderate compass, the best information obtainable upon this subject. In its presentation, the writer has followed what his own experience has taught him to be the natural sequence, and has endeavored to introduce such matter as sixteen years of lecturing and field work have suggested as requisite. Part I contains a brief geological review and a discussion of such points as the engineer must include in his report, i.e., the preparatory and development work, systems of mining and the plant for power, hoisting, pumping, and ventilation. Part II embraces the practice of prospecting, drilling, blasting, shafting, tunnelling, and timbering, in addition to some remarks upon the examination of mines.

The work is designed as an elementary treatise for the use of those desiring a reference-book. The complexity of the subject, its extent, and the variety of machines to be described and represented, demand an elaborate discussion that would fill several quartos. Descriptions of obsolete and expensive systems or machinery are relegated to the historical works on mining. American and foreign practice is described, and suggestions for lines of future progress are offered herein. The principles of the construction and operation of machines used in mining are explained with a perspicuity and conciseness compatible with the field in which this publication is to be sown—among students and mining men, to whom a knowledge of the fundamenta of their work is valuable, but whose acquaintance with the theory is slight.

The wants of the latter class have been kept in mind, and the writer hopes that the manual may prove of some benefit to the intelligent reader, of whom it presupposes an elementary knowledge of the sciences and of the simple machines.

The author regrets his inability to deal with the subject of "electricity in mining" as it deserves. Two reasons account for this: insufficient data, as yet; and the large space which a satisfactory explanation of the principles would demand.

The writer would also beg leave to say that the literature of mining and its cognate branches has supplied much of the material contained herein. References could not be made for each hint obtained, but obligations are acknowledged to the authors of the publications mentioned, to which the reader is referred for further details. The information has been garnered from the best available sources and condensed. The *Engineering and Mining Journal*, the *Colliery Engineer*, and the transactions of the *American Institute of Mining Engineers* have been copiously drawn from, as also the experience of the practical men, to a long list of whom the Author is indebted for many courtesies. Finally, to the manufacturers and engineers thanks are rendered for the use of the electrotypes, which have so largely contributed to make the work attractive.

MAGNUS C. IHLSENG.

GOLDEN, COLORADO, NOV., 1891.

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“ “ “ “ “ Colorado.
“ “ Bureau of Labor Statistics, Illinois.
“ “ “ “ “ “ Missouri.
“ “ Commissioner of Mineral Statistics of Michigan.

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MANUAL OF MINING.

PART I.

MINING ENGINEERING.

CHAPTER I.

GEOGNOSY.

1. Bird's-eye view of the subject; native metals, minerals, ores, and their occurrences, definitions. 2. Vein matter, gangue, and gouge; geognosy of ore-deposits; gash and fissure veins, beds and blankets; geological theories and miners' rules; prejudices and fallacies regarding ore-deposits. 3. Prospecting; searching for veins; indications, float, shode, and slide rock; examining new districts; divining-rods, spiritual mediums, and the drill as miners. 4. Remarks upon the chaotic state of the U. S. mining laws; apex *vs.* side lines; safety in the side-line law; advice to locators; insecurity of locations on the apex; patenting claims.

1. The search for the useful and precious minerals has been diligently prosecuted since the early days of civilization; their discovery and application have made nations powerful exponents in the world's history. And nowhere is this fact better exemplified than in our own land, in the wonderful opening and rapid settlement of the Western mining States.

No subject is more entrancing, no occupation more exhilarating, than mining, with its wonderful kaleidoscopic changes. In early times excavations were made and mines worked only to a small depth and in easy rock, and that, too, only for substances of high intrinsic value, notwithstanding the myriads of slaves to furnish the labor. The attempts at systematic mining were few and far between; but since the advent of the steam-engine, mining has been acknowledged an important profession, requiring technical education. Competition with the whole world, brought about by the improved means of communication, the paucity of bonanzas and their rapid exhaustion, compel a skilful utilization of all the aids to a cheap extraction of our immense wealth.

The accessibility of the mine and the vendibility of its product are the ever-ameliorating features in the mining history of nations, districts, camps, and individuals, gradually divesting mining of its risks and rendering it more and more akin to manufacturing. Each new camp, untrammelled by tradition to keep it in the rut of prejudice, displays its genius for organization and absorbs the latest devices, tried and true. Nevertheless, it must be admitted that in each camp an adequate solution of the problem involves intricate questions of environment. The economy of mining is a function of many variables, as geological stratigraphy, subterraneous uncertainties, wages, water, timber, transportation, and treatment. The constants are few. The proper relation of these it is our province herein to discuss.

Hitherto a gambling spirit has frequently controlled investments in metal mines. Speculative tendencies, not technical economies, have dominated some of our operators; their heavy aggregate outlay may have proven unprofitable, for the present, because of salted mines, attractive prospectuses, or incompetent management. It must be remembered, however, that they have contributed to the prosperity of the country, and at some later date their abandoned exploitations will be pursued to profit, when the potential investment of to-day will have been resolved into future kinetic dividends, the cost of production being continually on the decrease.

The occurrence of the useful or precious minerals in the state of native purity is rare. Still less often are they found superficially: they must be delved for. In the extraction of this subterraneous material, and its delivery to the surface, consists the art of mining. The legal definition of a mine includes such "workings as must be artificially lighted."

Gold and platinum are found native in the placer accumulations of ancient and modern river-beds, which furnish fully 75 per cent of the total output of these metals. Gold occurs in segregated veins, alloyed with tellurium, and always associated with pyrites and titaniferous iron; also intercalated between the sheets of slate or shale, or finely disseminated in eruptive rocks. The only extensive native copper deposit is the remarkable product of the Lake Superior region, where the irregular masses are mined out of the amygdaloid trap and sandstone. Singular masses of metallic iron ore are found in several localities, but they are curiosities and casual, if not meteoric.

With these few exceptions the metals are encountered in chemical union with non-metallic substances, more or less completely segregated to constitute mineral. Any accumulation of mineral of good quality and in sufficient concentration to warrant the expenditure of energy for its extraction is an ore. Manifestly this is a fickle term, since it depends for its stability upon the casual conditions of the market as well as the mineralogical features. The quality of the ore is a matter of greater importance than its quantity.

The most common substance is iron, entering as it does into almost all rocks and veins. Its most frequent, and valueless, combination is with sulphur. Magnetic and specular oxide and the carbonate constitute the entire supply. These occur as irregular masses in the rocks of every geological age, or in veins mixed with other minerals, but are chiefly in the metamorphic crystalline, Archæan rocks. Zinc is obtained from calamine, franklinite, and blende, which are quite extensively distributed in the Carboniferous strata. With very few exceptions, galena is exclusively the ore of lead. The carbonate and the sulphide, in the lower Silurian and Carboniferous strata, mostly occur in irregular shoots and pockets, and rarely argentiferous. In the older metamorphic rocks the galena is confined in fissure veins carrying silver and gold. The main supply of silver is from its minerals, more or less intimately associated with other ores. Similarly with them, it has a wide geological distribution, and is also found "dry" in fissures. Copper, as chalcopyrite, bornite, and cuprite, is disseminated in and along slates and sandstones, rarely above the Triassic. Many galena veins in the metamorphic rocks change with depth to copper. Mercury comes from cinnabar, which is found in true veins and in contacts. It is not commonly encountered. Tin has a characteristic occurrence in but one form, as an oxide, and only in gash or segregated veins, or "stockwerke" of the older rocks.

Tin lodes are of the segregated type, and gold or silver bearing, pyrites and cassiterite being the common form.

Millerite and pyrrhotite are nickeliferous and occur in gash and segregated veins, rarely deeper than 500 feet. Rich films of genthite in talc veins often constitute a commercial supply.

Manganese ores (standard contains 44 per cent of the metal) are generally associated with limonite and occur in pockets usually embedded in clay as contacts or beds or permeating slates. Films of manganese appearing in moss-like forms on the face of rock give it the name of "landscape" rock.

Mica is generally in bedded veins, instances of contacts and true lodes being rare. They are simply and always dikes in coarse granite. Hitherto only large slabs were sought, but now the fine, clean mica has a ready sale for lubrication and other purposes.

Phosphate rocks for fertilizers, the practical value of which is determined by the amount of phosphoric acid contained, are found as beds of irregular thickness; veins or lodes transversely to the strike of the strata; or superficial deposits. Apatite occurs concretionary in a clay matrix between limestone and clay. These are more frequent in the Miocene.

Many of the metals are incidentally obtained from their mineral compounds while smelting for other metals with which they are associated.

The metalliferous portion of a lode usually comprises only a small portion of its contents. The argentiferous galena, bornite, blende, or their oxidized derivatives in grains, pockets, or streaks, more or less connected, are associated with a "gangue" of clay, quartz, fluor, calc, or heavy spar. These earthy materials sometimes are intimately mixed with the mineral, and again lie in layers contiguous with it, or the different constituents may even manifest a ribbon-banded structure.

The entire mass, metalliferous and earthy, constitutes a deposit which is known as a bed or a vein, and may exist under such circumstances as to render it workable. The term *vein* is intended to describe a regular unstratified deposit in a fissure that traverses the country for a considerable distance, longitudinally and vertically. The Supreme Court has defined it as "any zone or belt of mineralized rock lying within boundaries clearly separating it from the surrounding rock." This demands a well-defined crevice of ready identification, and two solid walls to give it individuality. Its lead must be metalliferous. A vein is the filling of a pre-existing fissure. The term has lost the significance it once had. The mineral system was originally supposed to have a resemblance to the human circulatory system. True, the fissures have originated during periods of great dynamic movement, producing folds and fissures which are supposed to have extended deep into the earth's crust, but the main artery has yet to be located. Though argentiferous lead veins are quite persistent, no evidence exists for the dogma, so tenaciously held, that they increase in richness with depth. They may or may not become richer, or change, in constituents. Examples can be cited for either side of the argument. In folded strata the deposit inclines to be thicker at the ridges, or troughs, and thinner at the sides of the folds. But this is not generally the case in massive rocks.

Usually the vein matter is crystalline. It is commonly separated on either or both walls from the surrounding rock by a sheet of clay (called "selvage" or "gouge"), or by other quite distinct lines of demarcation. The surface of contact of the deposit with the adjacent rock is called a wall, roof, or floor, according to its relative position to the miner. Not infrequently the walls are polished surfaces ("slickensides"), due to grinding caused by the slips during nature's contortions. Sometimes portions of the vein have slid on one another, causing "false walls"; therefore the miner is advised to occasionally break into the walls to assure himself as to the fact. On the other hand, a vein may have only one or even no wall. In the process of mineralization, the original face or faces of the fissure may have become disintegrated, and all evidences of the looked-for wall obliterated. In such cases, economic, not geologic, or legal conditions define the vein.

2. Fissures belong to regions of metamorphic action, and are the principal repositories of the precious metals. And it is a striking fact that they are rarely found singly, rather in groups of parallel veins, often in congeries. *Stockwerke* is a term used to describe a condition of affairs in which the country rock is creviced in all directions, so that the whole mass must be mined out. Some are filled with eruptive matter, others with vein matter, still others were subsequently closed without any deposition. The mineral components are markedly dissimilar, and indicate different sources. Those filled with the same variety of mineral were doubtless produced by contemporaneous forces. Those fissures which interrupt the continuity of the older veins are called cross-courses. The manner in which the intersections occur determines their relative age. Their absolute age is not ascertained, unless in stratified rock. Drags are more common than is supposed, and should not be confused with intersections. The former are usually richer, the latter not necessarily so, at the point of juncture. Many of the older veins are broken and displaced by faults. Not only do veins "pinch and shoot," but the pay streak will vary in thickness, plunge from wall to wall, or split up into numerous feeders and ramifications, and even disappear in a thread.

Gash veins hold a subordinate position to fissures. But they are of small extent, and are usually confined to a single member of the formation in which they occur. Their habitat is unmetamorphosed sedimentary rock. They have no distinct walls or gouge, and are unreliable.

The most important sources of the mineral wealth are the metalliferous deposits which occur in the sedimentary strata, and are termed beds. While the geologists may classify them, the group is sufficiently identified by this term for mining purposes. It includes deposits, somewhat irregular in dimensions, occurring in the transverse joints of the rocks; as cementing material to the remnants of shattered or insoluble rock; as layers conformable with the strata; as isolated impregnations of grains or bunches in porous rock; or as a metasomatic replacement of porous rock. They may be found similar to fissures in a certain formation, then as a blanket contact parallel to the stratification, to again plunge into a lower series of rocks like a fissure, or branch out into a chamber. They are more easily mined, but are less persistent in depth, than veins. Their mineral contents are very compact, seldom crystalline, and the gangue hardly distinguishable from the country rock. The mineral is more or less concentrated along certain lines called "ore-shoots," which probably constituted the channels of communication with the ultimate source. The same is also true of veins.

Irrespective of any theory, one requisite condition for deposition is a crevice, a porous or soluble rock conduit for the fluid from which local action has precipitated the mineral. Open cavities were not necessarily pre-existing, for a vesicular rock would allow of an easy flow to the magma, or it might be equally well secured by dissolving action on the rock and a subsequent replacement. This is independent of its geologic position. In every age are rocks which will satisfy this condition. Besides this, a long train of circumstances has preceded the vein-formation involving dynamic agencies, heat and metamorphism, and even eruptive action, as important factors. These disturbances have been repeated through the different ages, the older rocks having been more frequently shaken up. Beyond this no reason exists for the prejudice which favors certain geological formations as ore-bearing.

The geognostical relations between veins and their contents are of importance to the mining engineer, but our limited space will not admit of any discussion here. The various works on geology will supply the information as to the vagaries manifested by ore occurrences and the numerous theories held. Some isolated examples exist under such circumstances as to suggest the same origin for the ore as for the adjoining rock-formations. Many of the beds and veins have been impregnated by percolating waters, perhaps at high pressure and temperature, contemporaneously with the country rock. Their metallic contents may have been carried in solution or they may have been in a molten or a gaseous state when the way for their passage was opened.

This is a matter for conjecture, as is also the ultimate source of the mineral. Certainly the evidences point to its deposition as a sulphide, the oxidized forms being accounted for by long-continued action of atmospheric agencies. The presence of coal and bitumen in many lead and zinc veins and beds in a measure suggests a theory of cause. The "water-line" theory has served its day and is no longer tenable. The current theories have, for hundreds of years, afforded satisfactory explanation of the genesis of some of our ore-deposits. But when we find contiguous depositions contrasting widely in point of density; narrower parts of fissures filled by larger deposits or richer ores; superior minerals higher up than the more volatile or lighter ones, even alternating with them, we must admit that since the day Job declared that "silver is in veins," little material progress has been made by our geologists beyond the slow garnering of facts which, ultimately, are hypothecated. Our knowledge upon this branch is cumulative and in expression conservative. However, any theory explains some, but none all, of the capricious examples of lodes or their anomalous fillings. The veins we find, but not always the silver; and this inability to formulate a general law by which to locate the hidden bonanzas has led to the compounding of the numerous witcheries, and divining-rods of every conceivable form, for imposing upon the credulity of the prospector who seeks a quicker means of acquirement than is afforded by the use of the pick, shovel, and patience.

There is no particular angle of dip or bearing of trend that is universally favorable to rich veins. Rules based upon such observations are local only. The same may be said as to the supposed "live"-ness of certain rocks to mineral. Attempts to formulate indications of "quickening" mineral by associations with certain gangue matter or minerals have failed of generalization. The mineral is where you find it. The Cornishman's adage, "riding a zinc horse to fortune," has no verity in this country. Each locality has its own peculiarities of mineralization, which the careful and systematic engineer will observe and regard.

3. With the two classes of rocks, stratified and massive, are coexistent the two classes of mineral deposits, beds and veins. Though many occurrences are of a nature that admits of question as to classification, for mining purposes a sharp line of distinction is not sought. Legal technicalities have so confused the definitions of deposits and veins as to obliterate all semblance to the original intent of geologists and mining men. Of this, more later. At present we shall consider some rules to assist the prospector in his search for mineral. And while it must be admitted that many a find has been made through accident, the existence of the ore would be found not to be at variance with the cumulative rules of geologic science.

Accordingly, the prospector will seek within geological confines. In regions of stratified rock the matter is simple. Coal is found in two horizons only, and the presence or absence of the rocks belonging thereto is indicative of the prospects.

The metals and their minerals are distributed, geologically and geographically, over a large extent. The zinc ores in this country occur in the Carboniferous and along the Mississippi valley. The Archæan and Silurian are most prolific of the other ores. The precious metals are chiefly found in the mountainous districts, because the phenomena attendant upon their formation were conducive to the filling of veins, and the forces which gave character to the mountain also impressed themselves upon the vein, which is exposed to view and subject to location. Without some such providential occurrences to change the monotonous topography of the preadamic surface, bedded veins of the stratified districts would have been revealed only by boring, while those in massive rocks might never have been formed.

Surface prospecting is confined, therefore, to the seeking for an outcrop. In igneous rock the outcrop is easily found. For, unless the hill is covered with slide rock, it is indicated by a jutting ledge (if the vein matter is harder than the country rock), or by a sag (if it is decomposable). In heavy timber this may go unnoticed. At high altitudes snow in the sags calls attention to the leads.

The same is true of coal, which is located by the terraces which mark the outcrop. The trend of the terrace, relative to the topography of the hill, gives a good idea of the slope of the coal. The bench itself may give the desired information, but usually it will be found that the coal dips with the hill, when the terrace or depression deflects outward toward the bottom of the hill, and the reverse for a coal dipping inward, when the outcrop will be concaved toward its top.

Substances foreign to the rock deserve notice. Alternations in the color of the slide rock covering the hill are good indications of the presence of oxidizable minerals above. So, too, vegetation is a guide. Iron springs often accompany the outcrop of coal; the ochreous covering of the rocks and soil is noticeable near some of the anthracite seams, and is common in the semi-bituminous districts. Masses of highly oxidized matter, broken from the veins, compose what are called "blow-outs," and are common in galena regions.

If no evidences of outcrop are thus found, "booming" may disclose it. During winter or a wet season, snow or water is collected in a reservoir upon the hill, and, at a convenient time, turned loose to plough its way over the soil in its fall. Many a vein has been thus discovered without great expense.

In stratified regions the order of the geological series may be observed, and certain fossils furnish the guide. Or, if the prospector is examining new ground, he has but to look for mineral in the float on the surface or in creek-bed. The appearance of material derived from erosion is indicative of the character of the rock from regions higher up. Therefore the bed of the stream, or the hill slope, is minutely examined for fragments of ore, or blossom, and followed as long as mineral is found. If the float or shode boulders are pebbly or rounded, or in vegetable soil, they have come from afar and the lode is not at hand. If the shode is large and angular, it has not come very far, and the discovery of a point beyond which no float or blossom is detected is presumptive evidence of approach to the vein. The lode will be found above the point of discovery, and the prospector will go in the direction of the drainage and thoroughly search the ground.

In high altitudes the oxidation of the minerals in, and the electric manifestations of, the vein outcrops have assisted the prospector by the light playing over them. This is of continued occurrence in Colorado above timber line, and particularly in regions of arsenical veins.

When found, the vein should be examined, and its value confirmed at several points ; most monstrous disappointments have ensued from testing of the lode at one point only. If the country is stratified, care is taken to ascertain all the data of thickness, etc. Frequently the ore oxidizes and rots away, to be crushed by the overlying strata showing only in a small streak. Or the outcrop may fold back, " tail out," and give false impressions of great thickness.

Maps are serviceable as showing the important features, and a systematic plotting of all data, geological and otherwise, gives a good basis for conclusions. Dr. H. M. Chance, in the Second Geological Survey of Pennsylvania, has an admirable discussion on the construction of geological cross-sections, to which the reader is referred. Prospecting for oil or gas is speculative, and the sole guide is the geologist's facts. Reports I, and J of the Geological Survey of Pennsylvania, and the Treatise on Petroleum by Benj. J. Crew, are monographs on the subject.

If surface examination fail to give a trace of mineral sought, and there remains reasonable expectation of finding it, tunnel, shafting, or boring may be resorted to. The two former are more expensive but safer guides than that offered by boring. Shafting is slower and costlier than tunnelling, but is shorter for flat seam, while the latter strikes the steep vein quicker; in flat seams its time may even compensate for the increased expense. The choice between them depends upon local conditions. Both are advisable for shallow explorations, while drilling may be employed for deep work. The latter is very commonly employed on account of its cheapness. But even when it has determined the data, previously doubtful, the shaft or tunnel has to be subsequently driven. So drilling has its limitation of use. It is rarely employed as a seeker for mineral, but merely to give confirmation to, and assist in a rational estimate of, the value of the undertaking. Many properties owe their rehabilitation to the results of the diamond-drill exploitation, and none should be abandoned until after a liberal use of it.

A shaft or tunnel is too expensive to undertake without misgivings, except after satisfactory surface examinations, followed by numerous bore-holes. Notable instances may be cited in which the results of the diamond-drill explorations have entirely refuted the geological indications of the surface.

Either the punch (**72**) or the diamond-drill (**88**) method may be used for the boring. The former is cheaper, but the pulverulent material brought up by the sludger is unsatisfactory; it may indicate the constituents of the rocks pierced at different depths, but can give little of its physical character or dip. The diamond-drill core gives a litttle more information, but even its indications are hardly trustworthy. It affords an opportunity to identify the rock, but some of the soft strata is worn away and the core may be turned in its tube, so its revelations are not much better than those of the sand of the punch-drill, which is faster. The smaller diameter of the hole of either renders its results doubtful; for it may have just missed the mineral, or have struck a solitary, small, soft chunk of ore, which would supply cuttings to discolor the sands for a long distance and give amazing report. Very important deductions cannot be based solely upon the indications of the borings. Only after numerous holes and a satisfactory surface examination can a conclusion be reached.

Good, hard common-sense, observation and pluck win, and they alone. There is no mystery about the finding of mineral. Nature is bountifully supplied with precious metals and valuable minerals, but her secrets are hid. Only the cumulative information of geological experience gives any clue as to the habitat. There is no witchery or magic charm that can hasten the knowledge of the whereabouts of an ore body or deposit.

The wizard with the hazel wand, or the spirit medium who is controlled by some disembodied Comanche chief, is an impostor. No sooner is he thus equipped than he affects a versatility and occcult power that transcends combined scientific knowledge. Nevertheless, to a paltry amount of " filthy lucre" he is not averse, when he plays upon the credulity of natures which are duped to making extensive explorations upon the purported previsions. This would be ludicrous, were it not also painful, to see the number of misguided men who have squandered hopes and possessions in their search for a short-cut to wealth.

4. The discovery of mineral at the surface must be followed up to prove the existence of a lode or vein. The existance of an ore-deposit is a stratigraphical fact which is demonstrable, and the granting of mining rights under the law is accorded upon this proof. The General Land Office of the United States and the courts decreed that a single shaft does not necessarily carry evidence upon this point. Besides the exposure of an outcrop or an apex on the surface, the existance of a mineralized vein, or of rock in place underneath, is an essential feature. If the ore-body underneath is not a vein, then the concurrence of mineral at the surface is not a part of a vein. The vein may have become disintegrated; but if the general features still prevail—a crevice carrying mineral matter between rock of a nature and origin different from it—a valid location may be made thereon. If, however, the vein matter has been transported by the elements and become mingled with other rock, it has lost all identity with its lode.

The number of legal definitions of veins is equal to the number of judges who have passed upon the cases. But as the U. S. statutes divide mineral ground into veins and placers only, the presumption would be that any well-defined metalliferous crevice, capable of ready identification by the miner, is a vein, whether fissure or not,—only it cannot be a placer.

The difference in the grants under the two cases, besides a question of acreage, is that the mining of ore within placer ground is confined to the vertical planes through the boundaries (sec. 2329, U. S. Rev. Statutes), while vein deposits may be pursued along their dip, "throughout the entire depth," even if they "so far depart from the perpendicular" "as to extend outside of the vertical side lines of the claim;" and the extent of the miner's right is determined only by the vertical planes through the end lines, which should therefore be properly drawn.

Locations 1500 feet in length are permitted upon the public domain to the discoverer of the lode. But for access thereto, and for convenience of working, the U.S. grants, as incident to the principal feature, surface ground which, measured from the middle of the vein, shall not exceed 300 feet on either side. Some States have reduced this to 150 feet on each side, while in some Colorado counties only 25 feet was, and 75 is, the outside limit. The claim must be essentially a parallelogram. It may be 1500 feet, or less, in length, located substantially along the middle of the apex, across which are drawn two parallel end lines and side boundaries, within the limit prescribed, parallel in pairs following the contortions of the outcrop. However else the Act may be vague, it certainly is not upon the fact of the parallelism of the exterior boundaries. Excessive locations are valid as to the legal limit and void as to the excess.

It is incumbent upon the locator to define the boundaries of his claim, by placing stakes at all corners and intersections, to notify others that the ground is entered upon and being exploited. These, with the filing of a location certificate in the county, maintain possessory right from the moment of posting a location notice of discovery upon the lode. Within a reasonable time thereafter, sixty days usually, the locator is required to sink a "discovery" shaft at least 10 feet into the vein. This satisfies the regulations regarding discovery, and maintains a mining right against all comers until the expiration of the calendar year.

From that time on, an "assessment" of $100 must be expended annually as evidence of mining intent. A failure to expend such sum constitutes a forfeiture by which the claim reverts to the public domain, and is subject to relocation. As the value of assessment work is a matter of opinion and not easily proven, it is safer to each year file an "affidavit of labor," certifying to the assessment work for that year having been performed.

A prospector is not confined to a single entry upon a discovered lode. He may appropriate as many claims as he chooses, contiguous or otherwise, with that of the first discovery. Upon each 1500 feet, or less, of length he must show the intent to mine, by a discovery shaft and the assessment work.

For the development of the mine, the annual assessment work may be done upon the surface or upon the vein, and all efforts outside of the limits of the location with a bona-fide intent to work the claim are justly considered as if upon the claim—as, for instance, development by tunnel instead of shaft.

This concession is further extended by the U. S. Supreme Court; for where one person owns several contiguous claims, capable of being advantageously worked together, one general system of development may be adopted, after the discovery shafts are driven. This encourages more economic work and subserves the best interests of all concerned.

The principle having been fixed, it is not remarkable that further concession was granted. "Where many claims are consolidated in the hands of one company, there is no impropriety in calling it one mining claim." This rule, adopted by the U. S. General Land Office, solves many harassing questions, but is more prodigal with the public mineral lands than was contemplated by the framers of the mining code.

When, therefore, a vein or rock in place is discovered on the public domain, it may be located on and operated. When the locator has demonstrated his ability to develop the mineral resources of his claim by the expenditure of at least $500, he may proceed to the purchasing of the land from the United States, i.e., "patenting" his claim. Certain preliminaries are necessary: a survey approved by the U. S. Surveyor-General for the State; notification to the public, by descriptive notices posted on the claim, in the U. S. Land Office, and published in the nearest newspaper for a period of sixty days; affidavits of citizenship and of the execution of the preliminaries; abstracts of title, and the payment for the land at $5 per acre or fraction.

An exclusive right of enjoyment "of all veins" cropping inside of the boundaries is given with the claim. These ancillary veins and their contents, to any depth whatsoever, cannot become the property of another, even if they are discovered and entered upon in adjacent territory. The subsequent locator, according to the laws of the State of Colorado, may have right of way through the cross-vein to his ground on the other side of the prior claim, but none of the mineral. In every case it is intended that priority shall govern. Sec. 2326 grants to the senior locator the mineral at the intersection, and to the junior the right of way through it.

By the interpretation of the U. S. Statutes, easement and title were clearly intended to be conveyed for all forms of metalliferous deposits, in the use of the terms "veins, lodes, or rock in place." The Act recognizes any mineralized rock in place, enclosed in the general mass of the mountain, as a vein. The arbitrary classification by geologists into veins, beds, and irregular deposits is unimportant in relation to this matter. Whatever the theory of vein-formation may be, it is positive that crevices were formed during certain convulsions of nature; in these ore-deposition may have occurred simultaneously with, or subsequently to, the fissuring, giving rise to various forms of veins. The dynamic disturbance and the atmospheric agencies that followed still further modified the geological and topographical features of the country. The processes were more or less similar, but the results are distinguished by geologists by the terms of beds, blankets, fissures, veins of impregnation, of infiltration and contacts. The legal expert has confused these terms.

The Statutes favored the miner and assumed to cover all lodes whose indications were sufficiently marked for the miner to continue explorations thereon. A crevice, crevice matter, a fair wall, and mineral are the essential conditions.

The discovery in Leadville of an outcropping bed, to which a few lucky prospectors were entitled, was followed by the promulgation of a "side-line" theory, the common law of Leadville, Colo.

It has been seen that a lode claim, whether patented or not, carries with it all that is beneath the surface-ground claimed, with a servitude upon the adjoining territory obtaining the right of following the dip of the vein, and subject to a like easement granted to the locator on adjacent ground to pursue his vein wherever it may go. This obtains until some one can show a better right. The common law as to realty is modified when applied to mining property.

It has, however, happened that rulings were so made and construed that a party may locate vacant ground and maintain ownership to the mineral covered by it, unless *it is shown that the mineral body belongs to a lode cropping elsewhere within legally claimed ground.* The proprietor who calmly continued work upon his discovery found himself breaking into the subterraneous workings of others who had stolen a march on him. To secure his right he must bring action to eject. To vindicate title the lode must be proven to be in place and continuous from the point of his discovery *to, into, and through* the ground of the trespasser. Failing to do which, his claim is defeated and all incidents thereto attached fail.

Naturally the train of reasoning led farther and farther away from the original intent of the law to reward the discoverer of an apex, until the accepted idea is that, although the " defendant's location may appear to you to be along the line of the top, apex, or outcrop of the vein, it cannot prevail against a senior location on the dip" of the lode.

Again, Judge Hallet makes this observation, which, unfortunately, has not been passed upon by the Supreme Court:

" I will say to the counsel in that case [a location made on the middle part of a lode, or otherwise than at the top or apex], which is not for the consideration of the jury, that it has always been a question in my mind whether a location on the dip of a vein would not be valid as against one of later date higher up. That is to say, whether if a location be made upon the dip of a vein, the locator may not pursue it in a downward course, although he may not in the upward course, and may not hold the whole which lies within his location and below it, as against any one locating subsequently at a higher point on the same vein."

A lack of development at the time of hearing in court, a lack of other proof of the "perfect continuity" of the vein from apex to side line, or one imperfect wall, invalidates the title of apex claimant to a lode claim, and the deposit is condemned to be a placer, on the doctrine that the law recognizes no presumption in favor of the existence of a vein, but treats each local aggregation of ore as a separate lode. The decree of "no lode" cuts off the privilege, nay, right, to pursuit along the dip, and permits extraction only within the boundaries of the claim.

To what absurdities the law has led us, by reason of the vagarious interpretations, the reader may learn by referring to Dr. R. W. R. Raymond's articles in the Trans. of American Institute of Mining Engineers, or to those of the author in the Annual Reports of the Colorado State School of Mines.

The only remedy is to repeal the present enactment, or else so prescribe and define the subjects of the U. S. grant, that a purchaser shall have a warranty title to the entry. At present he has possessory right only, and this state of affairs will continue just so long as the present system attempts to convey a right to mineral apart from that to the soil.

It has come to be accepted that litigation is one of the regular and inevitable stages in the development of a mine—the fruit of a strike. Justice W. E. Church, in a concluding and conclusive sentence of a decision said: "The present laws are a hot-bed of litigation and a fruitful source of error." Judge Bradley declared them "imperfect," and those who have had any experience will say "Amen."

CHAPTER II.

PREPARATORY AND EXPLORATORY WORK.

5. Discussion of the means of reaching veins by shafts, slopes, tunnels, and adits; conditions and comparative advantages; dimensions of the entries. 6. Levels, drifts, and gangways; necessity for, and positions of, reserves; size of lifts and stopes; ratio of dead work to stoping ground; dimensions and extent of gangways; cleats and their influence; mode of finding the continuation of a vein beyond a cross-course or fault; mill-holes. 7. Quarrying and "getting" of salt; hydraulic mining; exploitation of peat and phosphate beds.

5. ASSUMING that the question, "Can the deposit be worked with profit," has been answered in the affirmative, the following features are next considered:

1st. Preparatory works—shafts, tunnels, and drifts.

2d. Exploitation.

3d. Plant organized for hoisting, pumping, ventilation, and treatment.

The means of reaching veins are by shafts or slopes, or by adit or cross-cut, the determinative factors in the choice being local or casual conditions. A blanket vein, without outcrop, is reached only by vertical shaft. But, as most veins crop to daylight, the choice of a mode of access is governed by the engineer's geological knowledge and the system of mining to be selected. Metalliferous veins present the greatest difficulties, because of their uncertainties and irregularities, and at the outset the problem of selecting the site is not simple, demanding as it does the best judgment of the engineer.

The entry should be centrally located near the rich ore-body, and in the best position for drainage and underground haulage. Often several entries are operated when the danger of caving may require a hasty removal of mineral. Generally, however, a mine is planned for a long run, hence its treatment differs from that of a short lease. Much of the success of a mine depends upon the location of the entry-mouth. Concentration of the plant and ample dumping room must be obtained, and boggy ground shunned for foundations.

Wherever practicable, drifting on the vein by adit is favored. The first cost may be, the running expenses certainly are, less than by slope or shaft, and the cost of equipment is *nil*. Occasionally the outcrop of the vein may extend along the hill-slope under such conditions that a series of adits may be driven at convenient distances to explore the vein, and at the same time develop it. But such cases are rare. Each adit then serves for haulage and drainage of its own block of ground. It is then of the customary dimensions and grade.

Cross-cut tunnels have some of the advantages of adits, but more disadvantages. They are run from the steepest part of the hill and the lowest available point, through the country rock, to the vein. They favorably attract capitalists because they serve to prospect and to drain a considerable field and furnish a cheap, secure permanent way. Instances of successful development by this means are few, while the many failures or disappointments are not very encouraging. The vein may be cut where it is split up and poor, its character changed with depth or fault may have dislocated it, and thus the lode is not disclosed or recognized. And this discovery made after several years of dead work, has discouraged many operators and frustrated the development of many promising veins. After a vein has been opened and its value demonstrated, a cross-cut is justifiable, and may yield large profit on the outlay. Obviously, the size will depend upon the service. Ample double trackway is obtained from a 7′ × 10′. Many large tunnels serve as haulage canals. Undoubtedly they are a commercial success, but they involve a scheme too elaborate for the individual. Dozens are over two miles in length.

A vertical shaft may be sunk in the country rock to intersect the vein at a certain depth. But the irregularities of lodes and their eccentricities of pitch make this method as uncertain as the tunnel. If the shaft fails to disclose the vein at the expected depth, considerable prospecting is entailed to find the lode. Even when the vein is pierced, cross-cuts at stated intervals in depth must be drifted from the shaft to the vein. This variety of dead work is very expensive; and should the vein have reversed its pitch, the expense may become a serious item, and the length of the cross-cut required to reach the vein may become unprofitable and deter most operators at the outset. And well it may. It is slower and more laborious than tunnelling, but develops a more economic system and promises surer results unless the mine is very wet. The shaft is safer on the foot-wall than on the hanging side of the lode, but may not always be so advised, for each lower cross-cut is longer than its predecessor, and in hard rock and a vein not steep its cost may soon be prohibitory. Of course these cross-cuts lengthen as the shaft deepens, but the matter of driving is now so quickly done as to cost comparatively little. So this is of minor importance to what it once was. On the other hand, instances of recent wrecks and abandonment from the caving in of hanging-wall shafts are common.

In conjunction with this great outlay is the uncertainty of the continuity of the lode. This may be obviated by sinking on the vein. Inclines are in favor for many reasons. Following its contortions, they explore the vein, and more or less pay their way. Though the cost of maintenance is much higher than shafts, these slopes are preferred in coal regions where the dip is over 10° and the depth not over 500 feet.

When the vein has frequent enlargements, becomes tortuous and even knuckles, or if a fault is encountered, the pursuit becomes awkward. The question then arises as to whether it is advisable to continue the dip, follow the sinuosities of the vein, or begin anew; but the conditions under which it is not advisable to sink a new shaft entirely are very few.

The author favors the plan of sinking on the vein until its value has been demonstrated, after which the slope may be relegated to sudsidiary purposes as a second outlet, for escape or ventilation. If, however, the operators prefer to risk the outlay upon a cross-cut tunnel or shaft at once, they will have the most conservative form of attack if the vein proves good to this level. But considerable development work might have been done from the incline, on the interest upon the investment, while trying to reach the vein. No good exploitation can be effected until the conditions of the vein are developed. For attacking beds which are less freaky this preliminary incline may not be justified, but with the vagarious veins this plan seems indispensable. A subsidiary slope-entry, partially prospects the vein; and so long as two outlets are advantageous it seems rational to first disclose its value before venturing on the tunnel, or the sump shaft and its succession of cross-cuts to the deposit.

There is a diversity of practice as to the dimensions of a tunnel, drift, or adit, varying with the demands upon it. The dimensions of the main level should always be as great as convenient, because of its service. It may be driven in the country rock, as more advisable and safer for a permanent way, and it not unfrequently happens that the country is softer than the lode. For a single stope-lift one compartment suffices for an adit. A tunnel is generally double-tracked, and frequently has an additional compartment for ventilation. The plan of laying two or even three rails in a narrow tunnel, which is only widened at turnouts for four rails, is of doubtful economy. The inconvenience of a crowded gangway is undeniable; the relatively low initial cost is its sole recommendatory feature. Yet the difference in cost is not so great as might at first be imagined. In a large tunnel greater advantage can be taken of the face in drilling, and but little more powder is required per lineal foot; the difference in cost of timbering is little, if indeed it is anything, and the cost per cubic yard, broken, is much less than in small headings. Not even an approximate estimate can be given of the progress and cost of driving tunnels. They vary from $3 to $15 per cu. yd. of material removed. In granite it cost 90 c. per cu. yd., using 40 per cent Giant and percussion drills; progress, 750 cu. ft. per day. In porphyry 200 cu. ft. can be removed, costing $1.90 per cu. yd.

The upper bench is driven first, after which the bottom is easily lifted. See Chap. 8, Part II.

Slopes and shafts are of such dimensions as the hoisting, pumping, and ventilating appliances require. Slopes of two compartments are generally 12 to 16 feet wide, increasing to 18 or 20 feet if three are provided for. The height is fixed by the dip and the conveyance employed. Nine feet is not uncommon in a 35° dip, where the car is elevated by carriage. In driving, the lower bench is kept in advance of the rest.

6. The preparatory workings are far from complete when the ore-body has been struck. Permanent gangways for haulage must be run and securely supported. In coal-mines two parallel ways are driven with a rib 20 feet between them, one from each entry. In thick and steep veins the haulage-way is built near the floor, to facilitate loading of the cars. The airway is smaller, and above. For the lower lifts of the mine only one airway need be driven—the intake—if the main level of the exhausted lift, or lifts, be connected and employed as a return-airway. When the vein is reached, or penetrated some distance, it is then divided into blocks, according to the system of exploitation. Gangways pitching slightly towards the outlet are drifted right and left in the ore-body—from 60 to 100 feet apart vertically, in veins, and from 200 to 600 feet "on the rise," in beds. They divide the deposit into "lifts," or "stopes." Adits serve as gangways, as well as entries.

As many of these levels or drifts are run as the necessity for reserves, or the exploitation, may demand or the means of the operators will permit. It is undoubtedly advisable to open numerous and large spaces for attack, thus ensuring steady output without "picking its (the mine's) eyes out." They are extended to a natural boundary. Though the relation between the cost of maintenance and haulage, and that of sinking a new entry, may prescribe the limits. For example, a thin, deep bed in good ore, having a strong roof, is worked 2 miles from the downcast. Ordinarily, 3000 feet is far enough. In mines working on ore as uniform as coal, or those in bodies of known extent, only a sufficient number of lifts need be maintained to control the output. If the ore or adjoining rock is soft or decrepitates, the supports deteriorate rapidly, and induce a continual fear of danger from caves or the evolution of gas, so but few lifts are kept open, and each is worked out as rapidly as may be. The height of the stopes, or the length of the lifts, and the ratio to the thickness of the deposit, depends more upon the ore value than on the method of mining. The distance between the levels is increased with the hardness of the rock, the smallness of the deposit, and the cheapness of the ore. The lifts are smaller, as the intended output is larger.

How and where to place the level in the lode is of great importance. In the middle or on either side? With a lode of uniformly low-grade mineral it makes no difference. Generally it is safer to keep it in the foot-wall, or along it, if the country is softer. Injury by subsidence is less, and seepage of water is more readily taken care of. In thin veins the foot-wall is cut away to secure height for the car, and in thin beds the roof or floor, whichever is the softer. In thick beds the gangway is in the lower bench. If the mineral is in a small streak, it is followed as it jumps from wall to wall, unless the divergences from a straight line are too great. Otherwise the "level" is continued straight, without regard to minor deviations or rolls, on a grade of 1 in 100. The dimensions depend much upon the nature of the ground and the length of time it is to be maintained.

This class of work is very expensive, compared with ore-extraction, and for this reason is called "dead work." But it is indispensable, as exploratory. Though primarily unproductive, its location bears vital relation to the mine economy. Besides careful timbering, heavy stump and chain pillars of ore are left for support, the mineral of which is only incompletely recovered when the lift is abandoned. Indeed, all permanent ways should be so protected as not to jeopardize lives or the mine. Shafts should be surrounded by from 30 to 60 feet of unworked vein; haulage-ways in beds, by pillars 60 feet wide on either side; stopes, by arches of 10 or 20 feet thick.

A fair ratio of total dead work to stoping-ground opened is 1 to 8. In beds the unworked matter for support nearly equals the amount designed to be mined in the rooms.

All rocks are more or less uniformly creviced. Stratified rocks, for example, have horizontal planes of growth and vertical planes called joints, caused by shrinkage. Some coal-beds, besides the horizontal planes of cleavage, are cut by one set of parallel planes only, others by two sets, producing rhombohedral coal.

As crevices facilitate the breaking of rock, so do these "cleats" the mining of coal. In fact, in soft coals of small pitch the direction of the cleat alone may determine the direction of the gangways. In order that the working faces may be against the cleat, the most important drifts are with the cleat. This is not so true in anthracite veins because strong explosives are used. In steep-pitching coal-seams cleat is of less importance than the grade of the haulage-ways. Here the main galleries are with the strike, or slightly diagonal to the rise, the butt headings (see Fig. 10) being nearly perpendicular. They should not be driven far before breaking off the face-entries.

Deviations in the course, or changes of rock, occur in the lode, often so imperceptible as to lead the miners away from the vein. The freaks, horses, "jumping" of the streak, pinches, or faults may have gone unnoticed. A temptation to follow the softer country rock often accounts for "losing the vein." It is of common occurrence. In such event, fresh exposures of the sides of the level should be carefully examined for some distance back, to ascertain the point of departure and its cause. Cross-cuts in the lode may even be necessary.

If a faulting dike or cross-course is encountered, its strike and pitch are noted. After cutting through to the other side, the character of the rock is examined. In stratified country the rock encountered should be identified, and its geological position, relative to the ore-bearing stratum, known, thus guiding the engineer. But if the operations are in massive rock, the problem assumes a serious aspect when he attempts to follow the prolongation of the vein beyond the plane of the fracture. It is a matter of record that fully 80 per cent of the intersected veins were heaved, apparently, to the right or left. Those to the right are twice as many as those to the left. Henwood also discovered that the heaving to the side of the greater angle is five times as common as to the smaller angle. In every district may be found a rule for finding the other end of the vein. But it is purely of local application and unreliable. To formulate a general rule out of these numerous and apparently eccentric displacements would seem well-nigh impossible; but Herr Schmidt, in 1810, offered a solution to the problem, which, though not infallible, is the best extant and has done valuable service. "When the cross-course dips away, after going through it, the drift is run along its far wall in a direction opposite to that in which the vein pitches. If it dips toward the mouth, the drift is carried along the far wall, to the right or the left, as the vein dips to right or left." The amount of the displacement, i.e. the distance to be drifted for the continuation of the vein, cannot be premised. It varies between very wide limits, and is thousands of feet in many localities. Henwood averages the throw of veins at 16 feet.

The vein is still further divided into parallelopipeds, by mill-holes 50 to 150 feet apart. Through these the mineral descends to the level, from which they are upraised. Winzes, or secondary shafts, do similar service, but until connection with the lower level is made the mineral is hoisted from the stopes they work.

7. Quarrying is the simplest means of extraction. It disengages large masses, and admits of operations on a large face. It may be employed for all deposits near the surface, when the removal of the alluvial and friable rock is cheaper than timbering them up. Slate, building-stone, iron, lead and zinc ores, peat, coal, graphite, and mica are thus mined. The overlying loose material is stripped, and pays better than mining, so long as not over 4 yards of soil must be removed for each yard of coal. Practically all the deposit is recovered, and to a moderate depth is quite profitable. The point selected for the beginning of the work, and the discharge of the output, is the lowest convenient spot for transportation. Hoisting is accomplished by derrick and buckets; drainage, by bore-holes and wells sunk deep enough to drain the pit. The influx and accumulation of surface-waters give trouble, which is somewhat relieved by ditches and drains dug alongside the quarry. But the limit is soon reached with the difficulty of propping the unsupported sides of the cave. In the Tilly Foster mine this problem is solved by the blasting away of 200,000 tons of the hanging wall. Such work is dangerous and uneconomical, though it is systematically employed for iron ores all over the world. Nevertheless, the critical moment must come when a more rational method will be necessary. It is difficult, however, to induce a change to the more expensive underground work, and where it must ultimately be adopted the previous quarry is deprecated. Increased pumping-machinery and more timbering will be required than if the mine had not been previously injured. Several properties might be mentioned, in Leadville and Lake Superior, in which vast quantities of ore were lost in the caves primarily caused by originally working as an open pit.

The above strictures placed upon quarrying do not, of course, apply to the extraction of structural materials, which always occur superficially and flat over extensive areas. Building and mill stones are best recovered by open work, and easily mined in blocks by trenches and channellers (see Chap. VIII, Part II).

The getting of salt is generally by a special process. It is always found in old river-beds, and quite liberally distributed over the world. In England and in Germany the thick beds are mined systematically. Elsewhere, the heavy investment of capital involved would militate against the mining of impure beds, especially if cheap fuel is to be had. Then holes are drilled to the bed, water poured down, and the rock-salt leached out. A pump-pipe is carried to the floor, and the strongest brine thus drawn. It will be seen that the capital required is thus reduced to a minimum, and the output may be increased at a moment's notice. There will be no expense for storage, and no deterioration. This solvent process is also used in mines which have collapsed. The brine is evaporated by solar heat, or boiled in drying-pans. One ton of coal will evaporate 1600 gallons of brine, carrying 77 bushels of salt.

Hydraulic mining is a species of open work, in which water is the agent for removal. The main objection to it is the damage done by the sediment and waste in inhabited regions. An ore of 20 per cent gold per cubic yard pays. A. Bowie's "Hydraulic Mining" is a complete work on the subject.

The exploitation of peat and phosphate beds is by dredging. In heavy bogs of the former, canals are run for drainage, and for the navigation of a scow, which cuts away the peat. Afterwards it is pulped, pressed into blocks, and dried. This furnishes a clean, cheap fuel. Phosphate rock, for fertilizers, is dredged and grappled for, in rivers and deep water, by machines. Peat, or fertilizer above water-level, is quarried in steps.

Materials which occur in large bodies, and regular, require systematic exploitation. Short fissures, feeders, gash-veins, and pockets can hardly be classed as other than special deposits, for which local conditions determine the means of mining. Beds and veins of clay, salt, coal, gypsum, and the metals have a continuity and a consistence sufficiently uniform to admit of classification as minable masses. They are found in all manner of positions, with varying boundaries and variable admixtures of foreign substances. There is always a right and a wrong way of doing things, so it rests with the operator to select the best method of husbanding the resources of his mine.

The cost per ton is by no means the sole consideration. A speedy and complete removal is of utmost importance. Differences in dip and thickness, the relative amount of barren rock in the seam, the amount of gas, and the character of the bounding walls, are the factors determining the choice. Other elements of perplexity are added to the problem, as the friability of the ore, the disintegration of the vein matter, and its value, but these are of minor import. It should be borne in mind, also, that each method has its special adaptability. Numerous instances of failures may be quoted resulting from the error in the adaptation of a good method to wrong conditions, and it is earnestly hoped that a careful perusal of the following brief conditions may be of assistance to mine operators.

Whatever the method, first, facilitate the breaking of mineral by making the working places large, with ample, free, face; second, concentrate the workmen as much as possible; third, reduce the length and cost of gangways to a minimum, keeping them open, only so long as needed, for stoping and the robbing of the supports.

CHAPTER III.

METHODS OF MINING.

8. Analysis, discussion of the general applicability of mining, "retreating," and of caving; differences between coal and metal mining; the least minable thickness of deposits; handling of the ore from the breast to the tramway by runs, chutes, batteries, and mill-holes. 9. Overhand and underhand methods; description, comparison, and applicability of; account of the long-wall system; details of the plan; gob roads and their care; anthracite and bituminous coals. 10. Modes of mining thick, steep veins by traverses, filling, or caving; descriptions of the systems; relative merits of mining retreating, timbering, filling, and caving; mining loss and waste; Lake Superior iron mines; method of pillar and galleries. 11. Pillar and stall method of mining; description of the plan; dimensions of the rooms and of pillars; creep, cave, crush, or squeeze, and their definitions; manner and order of winning pillars. 12. Modifications of the pillar and room system; "County of Durham" method; barrier pillars; "Wasmuth" for hard, thick coals; "Steubenville"; panel method; order of mining out the panels and pillars; square work, for gypsum, puzzolana, salt, etc.; Monongahela block system; the American "square set" system, as applied to veins or beds; order of mining the sets; cost and consumption of timber; modes of mining thick seams in slices.

8. DEPOSITS containing organic, earthy, or metallic mineral may be flat or steep, thick or thin, and accordingly the systems for their winning are:

1. OVERHAND: for lodes of a thickness limited by the unsolidity of the ore; not advisable in wide veins of friable ore without modifications; universal.

2. UNDERHAND: only for narrow veins, and may replace above in those of rich ore, or where the entire lode is salable; never for coal.

3. TRAVERSE WITH FILLING: wide veins, mineral friable, and may or may not be all marketable.

4. SQUARE SETTS: with or without pillars; thick seams, flat or steep; mineral soft, roof medium, pressure considerable, no movement.

5. TRAVERSE WITH CAVING: thick, steep or flat, veins of friable mineral, and roof not solid.

6. LONG-WALL: flat thin beds, roof poor or medium, mineral not too soft; for coal hard or long grained, not for anthracite; also for thick beds having partings.

7. PILLAR AND STALL: thin or thick seams pitching less than 40°; requires a good roof; several modifications.

8. SQUARES: very thick seams quite flat; roof may or may not be good.

9. PANEL: thin coal-beds, flatter than 25°; coal free; top rotten; mine gaseous.

The most simple and natural method would appear to be to open series of working breasts as wide as the nature of the walls would admit, leaving unworked masses of mineral to sustain the walls. The main headings are carried to the extreme end, and the vein left intact, except for the necessary roads. The mineral is gained, mining homeward, toward the shaft, robbing the pillars as soon as it is safe. This plan of "retreating" presents a maximum of safety and economy, the roads being kept in solid mineral, and practically all of the deposit is won. Pecuniary circumstances may oppose this plan, for it calls for large capital and great patience from the operators; and if the area, or the royalty, be large, it cannot unconditionally be advised.

The mining of organic or earthy beds is not dissimilar to metalliferous exploitation, except that, because of their greater regularity, tentative explorations are dispensed with, and systematic development may be proceeded with at once. Coal-seams are less complex than are other beds. But certain extrahazardous perplexities make special attention to ventilation, imperative. Again, the brittleness of the contents, and the weakness of the floor and roof, demand great care and skill, to reduce the loss in culm and dust. Coal-beds less than 28 inches thick are unexploitable. The thinnest iron-ore bed mined, is

43 inches; of other ores, the local conditions fix the minimum limit.

Due regard must be had to the means of handling, tramming, and transporting the product from breast to entry. This question of accessory haulage does, in many instances, decide the plan of work, and it increases in importance with the low value of the ore. The rooms, breasts, or stopes from which the ore is mined are to "the rise" of the gangway, that gravity may assist the loading of the cars. In veins nearly vertical the ore drops through mill-holes, or is hoisted through winzes. Where the pitch of the bed is greater than the natural slope of the ore (26° to 30°), a shute is built, 6×6 to 4×3, behind the miner, to deliver the ore. If the vein is thick and the breast wide, two shutes are carried at the sides, with waste or timber between. Gates (Fig. 1) for loading are easily manipulated at the bottom, where a level platform receives the mineral for the cars.

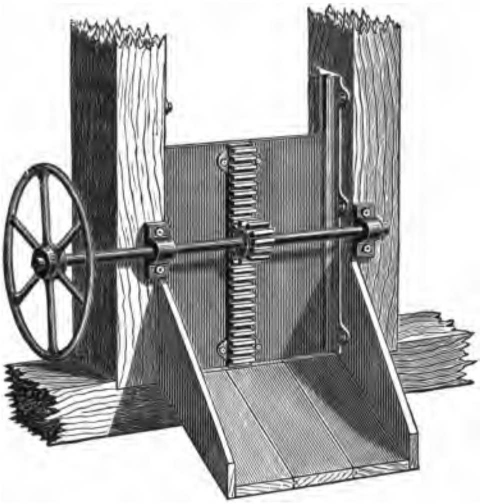


Fig. 1.

In thin beds, having a poor roof, the shute is carried in the centre of the room. Soft ore will not slide or roll, without shovelling, down a flatter grade than 26°, unless the shute be lined with sheet-iron, which brings the minimum limit to 18°. In beds between an 18° and an 8° dip, shutes are built of 3″

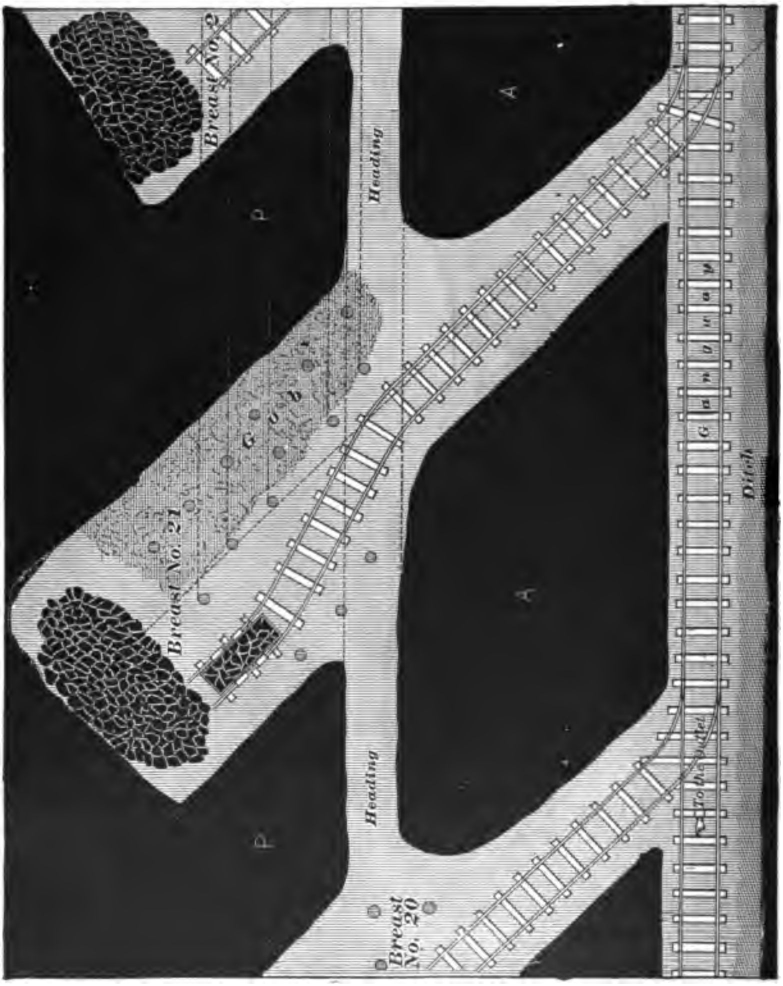


FIG. 2.

plank, supported from props 6′ or 8′ apart, long enough to reach the top rock, or buggy roads are laid in the same way. The rails rest on ties across longitudinal stringers, laid on a pair of 2″ planks, spiked, ends up, across the road, connecting each pair of props. Flatter than 8° shutes are out of question.

Then the breasts are said to be flat; mules can haul cars over the grade. Tramways are laid, following the progress of the rooms, diagonally to the "rise" (see Fig. 2), aiming to secure the most advantageous haulage grade. Generally they are backwards, towards the outlet, and are called "runs." With a bad roof, the track is laid near to the pillars. Shovelling should be avoided as expensive and injurious, especially to friable ores. The loss by attrition may be diminished by keeping the mill-hole, or shute, full. Interference with the ventilation is the sole objection to this. In some mines a gravity plane, similar to that described in Fig. 54, is applied to use in stopes. It saves shovelling. Rooms as flat as 3° are di-

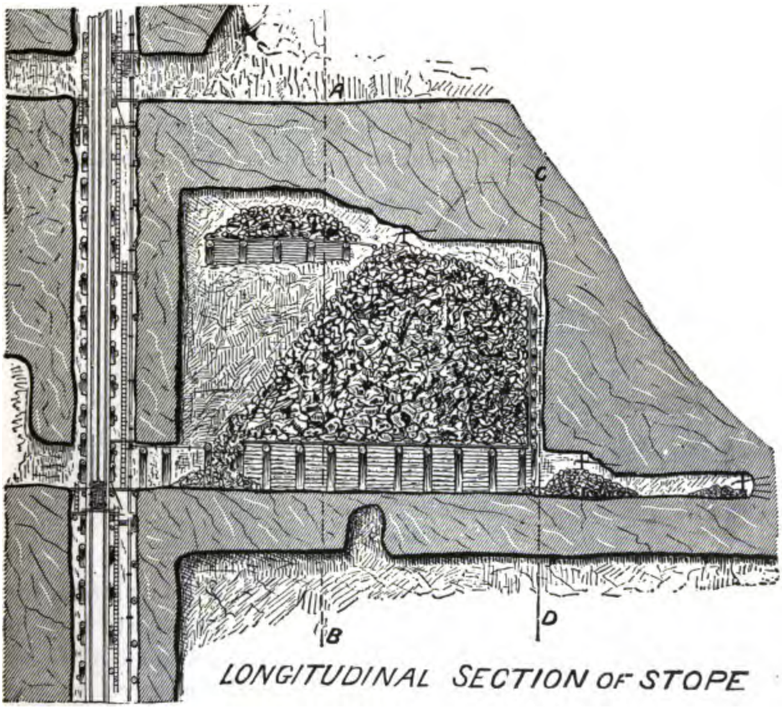


FIG. 3.

rectly up the slope. Little sorting is done underground, the coal being free, while metallic ores are better culled at the surface. It is preferable to hoist an excess of waste, and to save all the useful material, than to sort below, to economize in hoisting.

9. After the narrow, steep lode has been blocked out, such blocks as appear workable are mined, and in any order, by underhand or overhand stoping, the difference being only one of direction of working.

In *overhand*, the attack begins at the lower angles of the blocks next the mill-hole, Figs. 3, 4, and 5. The miner, standing on the caps of the drift-timbers, removes a horizontal slice, 5 or 6 feet high, one half the length of the block or stope, for the full width of the streak or vein. Gravity assists him in loosening the rock, and at the same time endangers his life by threatening a fall. The ore is picked and delivered to the shute or mill-hole, which is sometimes carried up, keeping pace with the progress of the stoping, instead of being in advance of it. The waste rock is piled on the gangway timbers. The next upper slice proceeds in the same direction, the miner standing on the "gangue" of the previous slice; or, if there is not enough of it, upon a temporary staging placed at a convenient height. The stoping continues, in slices, to the upper level. Another practice consists in mining all the slices of a stope simultaneously, each layer having a gang of miners work-

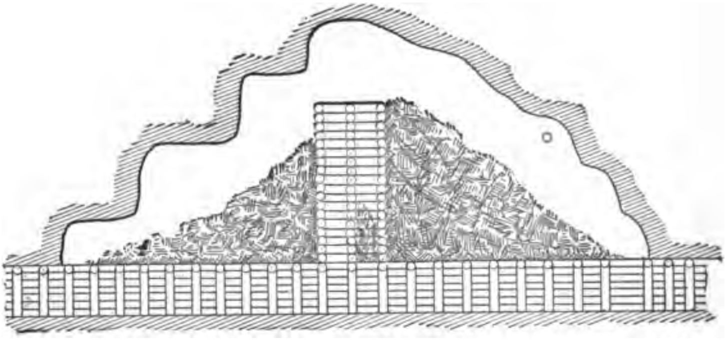


FIG. 4.

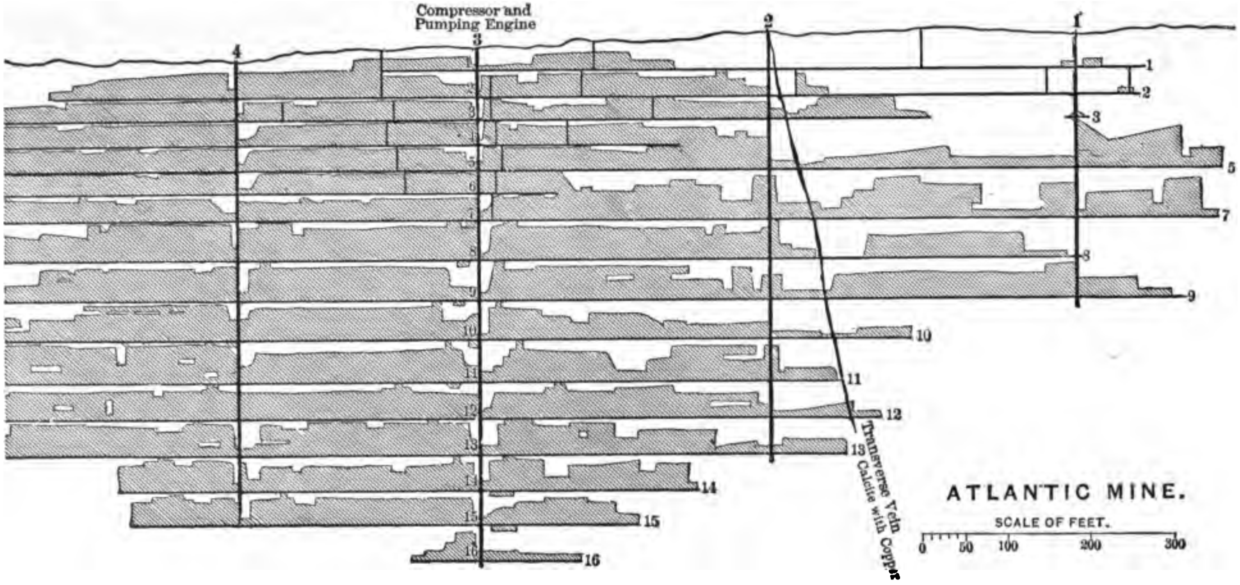

Compressor and Pumping Engine
4
3
2
1
1
2
3
5
7
8
9
10
11
12
13
14
15
16
Transverse Vein Calcite with Copper
ATLANTIC MINE.
SCALE OF FEET.
0
50
100
200
300

FIG. 5.

ing 10 or 15 feet in advance of the next upper gang. Fig. 5 shows the mode of working the Atlantic Copper Mine,—our "crack" mine,—total cost of mining, transportation, and treatment being $1.19 per ton.

In *underhand* stoping the blocks are worked away in horizontal slices, beginning at a winze (Fig. 6), but they are taken descending. The miner always stands on the vein matter, takes off the floor of the level, replacing it with stulls, on which planks and rails are laid for the cars. The waste is thrown on a platform built behind him, while the ore is tumbled down to the winze and windlassed to the cars above. If the entire vein is pay dirt, the staging is dispensed with.

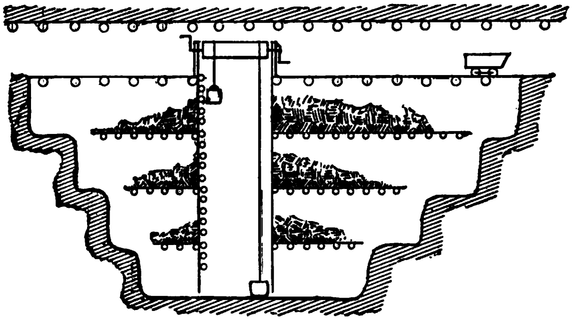


FIG. 6.

Taking the amount of timber as a criterion, the overhand has the preference, for it requires no timbering other than the gangway support, while underhand requires as many stall platforms as there are stopes. Moreover, the former may be used in a wide lode, according to the solidity of the mineral, while the absolute limit for the latter is determined by the length and size of the stulls necessary and available. Overhand affords greater facilities for breaking down rock, and delivering it to the cars. Underhand is accounted best for veins containing valuable ores, and when the lode is rotten, but not for crushable ores or coal. All the rock must be handled: therefore there is no excuse for throwing good ore among the rubbish. In the overhand the vein matter is shot down on the

PLAN OF LONG WALL.

Dotted Lines, Progress by Months.

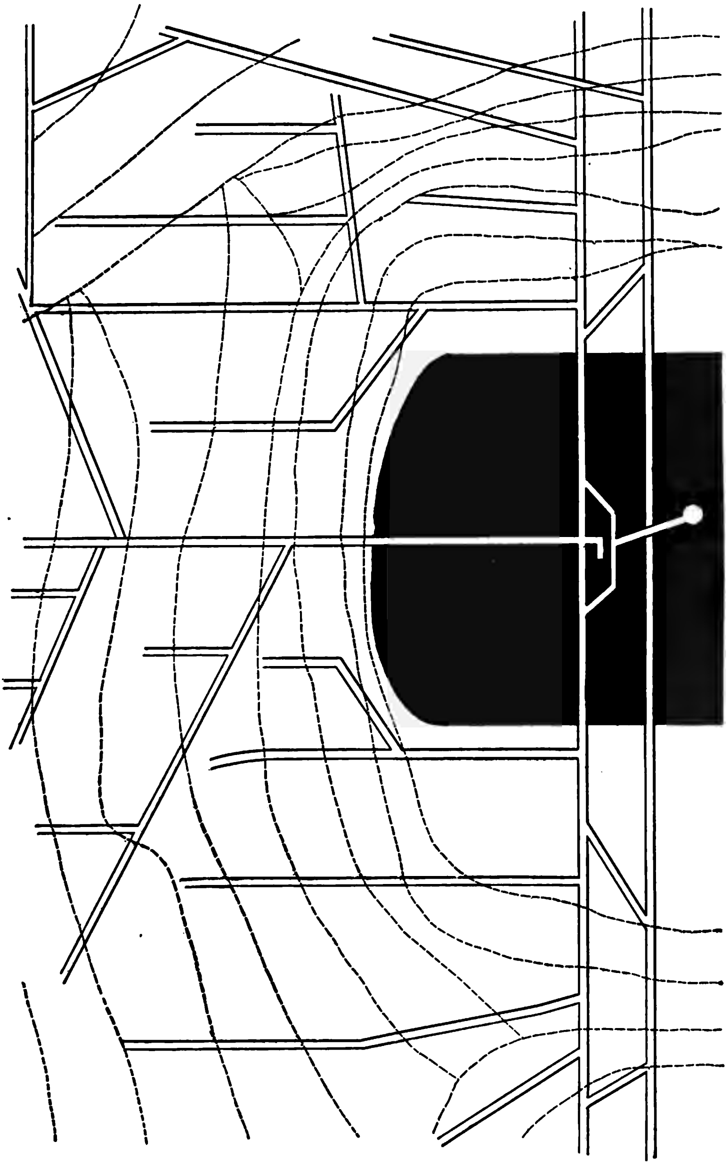


FIG. 7.

stull dirt, and unless the latter is covered by canvas, bull-hide, or boards, much fine material will be lost in the waste. Some of the lignite veins along the foothills of Colorado are nearly vertical, and worked overhand.

Long-wall is in general use in thin beds whose pitch is less than 35°, flat seams thinner than the minimum height for working places, and thick veins having partings. It contemplates a complete removal of the deposit, no attempt being made to support the roof. See Fig. 7.

Beyond the solid portion left intact for security of the hoistways the attack upon the deposit begins. Main headings are made for the haulage; from these the gob roads lead to the workings, advancing in every direction, more or less elliptically. The whole wall of the deposit is simultaneously attacked at several faces. The miner by day underholes the coal which over night breaks down from the pressure of the overlying strata. Two lines of props supporting the roof for ten feet back protect the miner (Fig. 8). They are set diagonally, and moved forward, one row at a time, with the progress. The culled waste is thrown behind the miner, and more or less completely fills the empty space or "gob," which is soon replaced by the fallen roof. It is necessary that the roof be poor or medium, that its caving supply the filling for the empty space. A thick unyielding roof which is liable to sudden heavy falls will not do; nor will a wet one. A tough broad stratum that will bend, rather than break, is the safest.

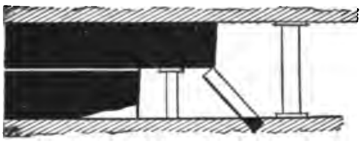


FIG. 8.

The face of attack may be of the horseshoe form, from slopes, and ellipse or flat stopes from shafts. The latter makes more cutting, and gives greater friction to the ventilating current; besides, the pressure of the superincumbent strata crushes and breaks off much mineral which may be lost. "Right-angled longwall" is applicable to anthracite beds whose dip is not too great for mules. When the pitch increases till the mineral will slide on the floor, level stopes are opened out successively, each following a level, the roadways being driven to the rise. Chutes then replace the gob roads. The walls are parallel with the pitch, their length decreasing inversely with it.

Each gang has a length of face 60 to 90 feet to work away with its own roadway laid behind from the middle of the face. This road is somewhat protected by pack-walls of stone on either side 5 to 10 feet thick (Fig. 9). The building of these roads and the setting of the props is done by men whose sole duty it is. The continual settling of the roof and the swelling of the waste crowd out the walls, and necessitate hacking the roof or cutting the floor. These gob roads therefore require constant attention. With the progress of the work their number increases, the length also, and their care is so serious an item of expense as often to open the question of the desirability of continuing the plan. An attempt to leave a thin rib of unworked mineral on each side failed, because the unequal resistance offered by the waste and the ribs threw an excess of pressure on the mineral. In collieries there is one great danger in employing waste—i.e., its liability to the evolution of gas, the swelling of the goaf, and the spontaneous combustion which increases with the sluggishness of the ventilation. This is recognized by the "stink." While plastering the gob will delay the combustion, the slightest chink admitting air undoes the work. The only remedy is to remove the heated mass and replace by clean rock.

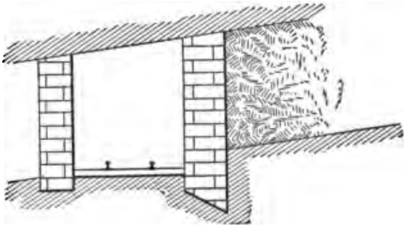


FIG. 9.

A good ventilation is easily obtained along the face, little roof has to be contended with, and there is money in the method if the distance to the boundary is not very great, or if the surface is not injured by the subsidence. Occasionally operators, who exercise the courage of their convictions, mine "retreating." The pecuniary gain is unquestioned.

10. Thick metalliferous veins of high inclination are mined by some modification of traverse with filling or with caving, the latter being in vogue where the vein matter is friable. The shafts and cross-cuts are in country rock; the levels are in the footwall or made secure along it; and timbered traverses are driven across the vein to the hanging-wall. Beyond this the two methods differ. And one need not long remain in hesitation as to the choice.

Whenever the entire deposit is meant to be removed, either the roof must be allowed to fall and fill the space occupied by the mined-out ore, or its subsidence prevented by extraneous means—timber or rock waste. The plan first suggested, of progressively exhausting the deposit, and allowing the overlying material to cave behind the miners, is very convenient and economic, provided the surface land be not valuable or its subsidence imperceptible; and if the *rationale* of "mining retreating" be accepted by the operators, it may be advantageously introduced. Long-wall is a species of caving for beds. The only conditions justifying the selection of any caving method are absence of filling material, scarcity of timber, softness and depth of the deposit. Otherwise the question will be as to the relative merits and cost of rock waste or timbering. Filling offers many advantages, is universally applicable, and is specially commended for thick veins. It admits of complete removal of the ore, which is replaced by waste from the cullings of the ore, from mill tailings, or that quarried from some convenient ledge. It is independent of the character of the roof, which may be poor, while caving demands pretty safe covering. As to the application of "square setts," timbering is expensive: its increasing scarcity and short durability in mines account for its replacement by masonry and iron in Europe. If substantial pillars are to be left for temporary support, as is the case in the methods of "pillar and stall" and its modifications, the per cent of ore extracted is comparatively reduced, because of the great loss accompanying the recovery of mineral—for one does not want to abandon the pillars. Dr. H. M. Chance (Report A2, Second Geological Survey of Pennsylvania) estimates a loss of 55 per cent in mining thick, deep, soft coal, and at least 15 per cent with thin, shallow, hard coal, under a good roof. These increase with the dip. A 28-foot gypsum bed sells but 29 per cent of the amount of the area mined.

For *caving*, the timbered traverses are 6 or 8 feet wide, with pillars of ore 9 feet wide between them. The timbering is cautiously removed, beginning at the hanging-wall, when the vein matter for 10 or 20 feet above creeps down into the cavity, the miners removing the mineral and retreating towards the main level. When these traverses are exhausted the adjoining pillars of the same level are likewise mined. The levels 20 feet apart are similarly worked and closed in descending order. Drawing the pillars, sinking the shaft, and driving a fresh set of levels and cross-cuts all go on simultaneously, insuring a regular output. Lodes of moderate width and firmness are taken, retreating from the boundary by removing each slice of a full width of vein, allowing the roof to cave. After quiet reigns (from a week to two months), work begins afresh on the next slice below. While this progresses the upper slice of the next nearer stope of that level is proceeded with. Each stope toward the shaft is one slice behind its neighbor toward the boundary. The dangers and the disturbances at the surface, together with the trouble with seepage, renders this system obsolete.

For *filling*, the traverses are permanent drifts 60 to 80 feet apart, connected at the foot-wall by winze, and along the hanging wall by mill-hole, with the drifts 50 or 60 feet above. If the ore is dry and stands well, and if the vein is not too wide, the men on each side of a pillar advance from the traverses toward each other, taking up a slice 7 to 8 feet high, replacing its volume by gangue, or, if necessary, filling winzed from the surface. When a slice is exhausted the men stope, from on top of the filling, another rise of 7 feet, carrying up with them a mill-hole for delivery of the ore to the cars, replacing the ore by filling. The slices are removed in ascending series to the level above, unless it may be desired to leave an arch as a support to the filling of the upper block. Several contiguous blocks on the same level are being simultaneously mined and filled. The blocks on the next lower story are then proceeded with, working upward to the level last referred to.

Any available material will do for the filling, but the best is waste mixed with clay, as it packs well. In the Lake Superior region it is mined from a neighboring quarry at 15 to 30 cents per ton. Formerly it was believed that several years should elapse before touching the next lower block. But it has since been learned that the waste will settle quite rapidly, and in a short while become compact enough to be safe. Any squeezing of the walls further increases its solidity. The filling and spreading of waste is done by others than miners, who simply shoot ore. This method is indispensable where the entirety of a thick vein is to be removed; it is recommended all the more as the thickness increases, because of the fewer chances of accident, the small expense of timbering, and the increased output. It has gained a permanent foothold even where the ore is soft.

In veins over 30 feet wide the attack is by traverses, instead of longitudinally, as described. A longitudinal drift along the hanging-wall connects the cross-drifts or traverses, and from this the advance proceeds toward the foot-wall, the face of attack being the length of the pillar. The filling is received as mentioned, but the ore is chuted through a mill-hole carried up with the filling, midway between the traverses. Some of the thick, highly inclined coal-seams of France are divided into lifts 37½ feet apart, worked descending; and these into five slices of 7½ feet high, ascending. The traverses are driven from hanging to foot, and consecutively.

Some firm lodes are worked in longitudinal slices, parallel to the dip, each slice being treated as if it were a separate lode of ordinary dimensions, filling as usual.

There is no peculiarity about the method preventing its adaptation to flat beds, even to those of coal. Long-wall requires a bad roof; but if the roof be good it may be used, the filling replacing the material expected from the caving of the roof. This plan is sometimes used.

If the filling be inadequate or expensive, and the mineral of low value, a method of *pillars and galleries* is in use. This method proceeds, in any given block from below, upwards, is the only one for thick, low-grade beds, but is imperfect in that the inevitable loss of unworked ore is too great. Each block is divided into slices, galleries being driven at the bottom, leaving a sill overhead as a floor to the upper gallery. The entire exploitation is in mining out these galleries, which must be vertically above each other, and as wide as the roof will allow.

11. The system known as the *pillar and stall* consists of long, narrow, open working breasts (stalls or rooms), between solid pillars of mineral (Figs. 10 and 2). It is the most common and perhaps the oldest system for flat beds of all varieties. It especially demands a firm roof, is cheaper than long-wall, but produces less per acre, and is harder to ventilate. Though it is employed with beds 50 feet thick, the writer believes that better results are possible from long-wall or filling. It is less useful in steep than in flat deposits.

After the gangways have been carried a safe distance beyond the entries, the rooms *B* are turned as fast as possible to the rise, or backward up the slope, according to the mode of conveyance of ore, from breast to gangways. The jaws *a* of the rooms are but 8 or 9 feet wide for a distance equal to that allowed for the stump pillars, *A*, when the width is suddenly enlarged to a working face of 20 or 30 feet, as time and the roof will allow. The rooms are mined to the rise, being carried regular and uniform to a length of from 8 to 10 times their width, leaving a chain pillar above for the upper level. The rooms of each lift are mined progressively towards the boundary as rapidly as circumstances permit. Fig. 11 illustrates the mining of breasts, "runs," diagonally to the rise. (See p. 34.)

The pillars *P*, left between the rooms, are unbroken, except for three or four small connecting passages. Through these the return ventilating air-current passes from one room to another, on its way to the upcast. The relative thickness to be given to the pillars is determined by an answer to this ques-

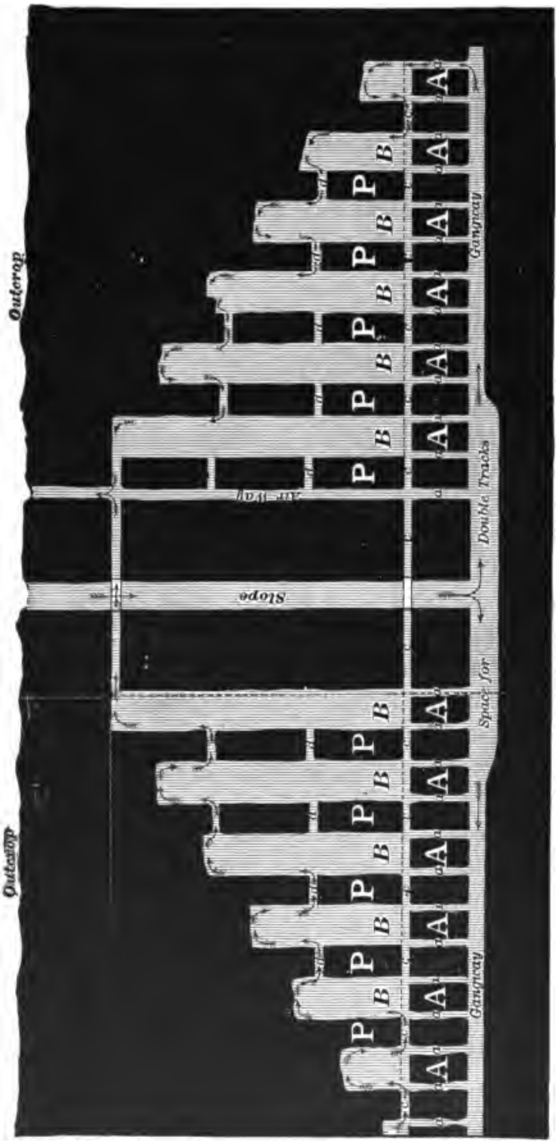

Outcrop
Outcrop
Air-Way
Slope
Space for Double Tracks
Gangway
Gangway
P
B
A

FIG. 10.

tion: Are the pillars intended only for the support of the roof, or are they to be regarded as reserves, to be mined at the proper time? Thinner pillars are conceded for the former purpose than for those to be maintained for future supply. In any event, the pillars constitute a support to the roof while

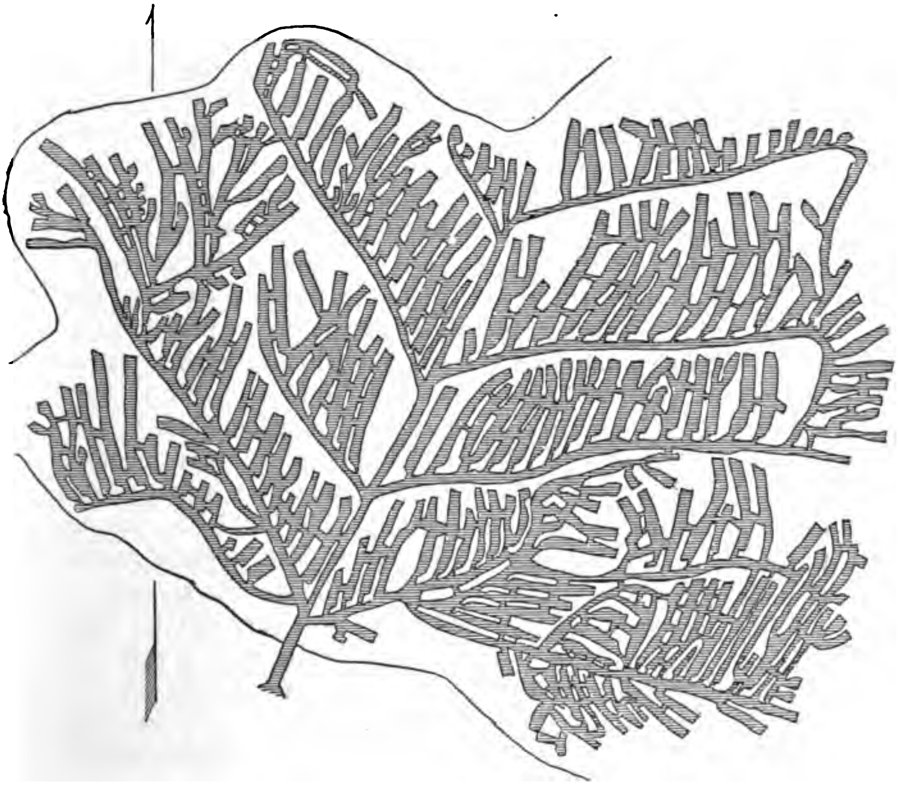


Fig. 11.

mining is in process in the adjoining stalls. Their size must be sufficient to resist the crushing due not only to the mass above the pillar, but also to that transmitted to it by the roof over the room. The sides ought to be respectively parallel and perpendicular to the dip. The pressure is about 8 tons on each square foot, for every 100 feet of overlying strata. At

700 feet depth the pillars are 30 feet, and the rooms 20 feet wide, in a coal seam: so each square foot of pillars sustains a weight of 93 tons—a big load, even for stone pillars. As the crushing weight of the different minerals is not known, no formula can be offered for determining the size of pillars, and each mine will have to be guided by tests in its own locality. The resilience of the various layers may, however, relieve the roof of much of this weight. Nevertheless, the pillars should increase, and the rooms decrease in size as mining deepens, until a limit of workable depth is reached, when the rooms allowed are an unprofitable portion of the deposit. At 1600 feet they contain less than 25 per cent of the total ore. A good roof threatens a thrust, which tends to slab off the pillars till they become too slight to resist the pressure. If the roof is weak, less pressure is transmitted to the pillars than when the roof is firm. Stronger supports are therefore required in the latter case to confine the limits of a crush, and narrow rooms to prevent the yielding of the roof (squeeze). With a soft floor its "creep" (reverse of squeeze) is induced, choking the air-ways, unless the pillars have a large base. This is the most insidious and serious kind of destruction, which is first recognized by the trouble with the tram-rails. A dull hollow sound heard when treading the pavement is also a good alarm. Once begun, it is amazing how rapidly the trouble spreads to neighboring rooms. Larger pillars are necessary in thick beds than in thin seams; in free, broken, or brittle than in compact or tenacious coal; or where partings exist in the seam.

It is manifest, therefore, that large pillars are indispensable, under whatever condition of roof or mineral. Moreover, the exposure to alternations of humidity, temperature, or pressure injuriously affects the quality of the coal. Its rapid deterioration reduces its resistance, and shortens the life of the pillars. Not only does this danger threaten, but also the evolution of gas incident to a crush. Hence, for reasons of safety, the rooms should be mined, the pillars recovered, and the lift abandoned as quickly as possible. Mineral admitting of the expense may be partly replaced by timbering or masonry, and the size of the pillars correspondingly reduced, and particularly so in thick beds, where pillars are less efficient.

After all the rooms have been mined to the end of the lift, the pillars are robbed, retreating. This is the most dangerous part of mining. In a coal mine the amount of large, good quality coal recovered is never very great, unless large pillars and small rooms were planned. Including "smalls,"not over 30 per cent of excellent coal is obtained. Much of the remainder is left in the mine, and some is sold as slack or mine run. Some day, perhaps, this coal will be recovered from roads cut below the strata; otherwise it is all lost, for the lift is abandoned after the stumps are removed. The roof coal, if any is left, is also obtained at this time. The pillars are mined by removing scales as thick as safety permits from the sides, by slicing the end off the pillar, by driving a heading through the centre and removing some of the middle portion of its mineral. The face of attack must be kept small, to retain the mastery of the subsidence. A regular line of retreat is maintained, to keep the pressure uniform on the neighboring pillars. Robbing is continued to within a short distance of the entry.

The same order of mining is established in the next lower lift.

12. Local modifications of this general method are numerous, particularly in collieries. For example, the breasts, 20 feet wide, are worked with 52-foot pillars between them. When each contiguous pair has advanced 50 feet in length the pillar is pierced, and a 20-foot room advances, leaving ribs 16 feet wide on each side. Again, the empty rooms are filled with waste, and the pillars robbed. In the "County of Durham" variation the plan of the mine is much the same. It is a connecting link between "pillar and stall" and the "panel." The breasts, with their pillars, are laid in groups of 8 or 10 from the gangway. Each such section is mined separately and systematically towards the boundary. Between the sections large blocks of coal are left, 150 feet thick, as barriers, not broken, even for airways. If possible, the barriers are placed in poor quality coal, and of a liberal size, to reduce the breast

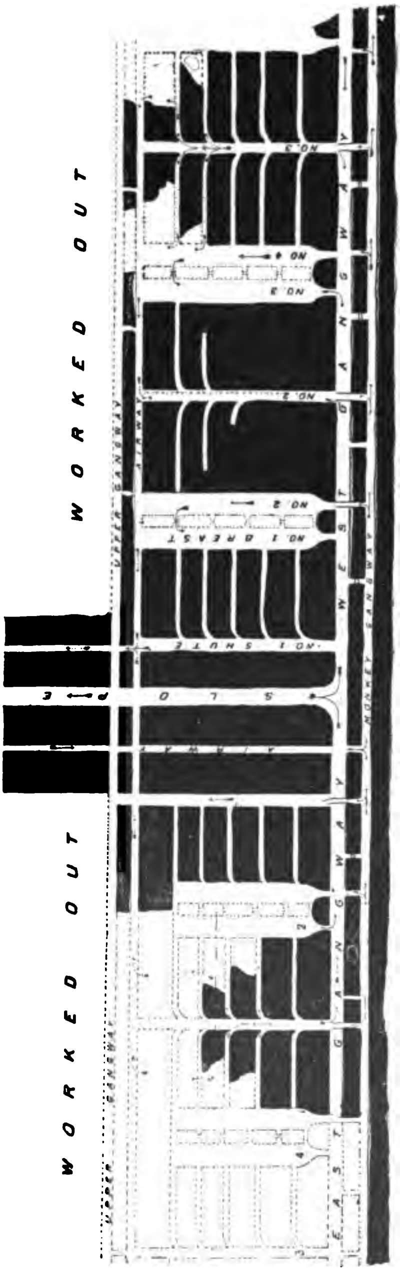


FIG. 12.

pillars and give complete isolation to any section. They localize any movement of the roof. The plan is commendable in gaseous mines, as safer than pillar and room, and more economical than panel. The order of the mining is the same as mentioned. No barrier is robbed until after its adjacent panels are exhausted. The "Wasmuth" is another variation for hard thick coal. But whatever the system, it is certain that the culm loss is too great. Nor will larger pillars entirely remedy it. It is, however, fast giving way to long-wall. When the deposit pitches between 20° and 40° the working faces must be on the dip, and mining progress with the strike. This gives a species of flat stope, or "Steubenville."

The "Panel" system is much the same method, but development is slower (Fig. 12). After the two gangways are driven on the strike, driveways are carried to the upper level. From these, right and left, horizontal roads and breasts are run, leaving small pillars between them. These workings constitute a district, or panel. Barrier pillars confine the accidents, and each panel is separately ventilated, and in fact capable of complete isolation from the remaining panels of the mine. The pillars are drawn after the breasts are mined, the roof is allowed to settle, most of the pillar coal having been saved. The adjoining panels are mined out, advancing. Subsequently, the coal in the heavy rib barrier is gained, after which the stump pillars are robbed, and the gangway closed.

Square work is a variety of pillar and stall, by which very thick beds of salt, gypsum, coal, puzzolana, etc., are mined. Square rooms are opened, of about 150 feet on a side, from the gangway, the pillars between them being about 30 feet wide. The rooms are not mined over the entire face at once, but are driven in chess-board chambers, divided by galleries 20 feet apart, one set to the rise and another with the strike, leaving from 12 to 16 pillars. The galleries are as high as the bed allows. If the roof or floor is bad, a layer of mineral is left for security. Though much in vogue, the method involves great risk to life, since the ear, not the eye, must be relied upon to detect a threatening roof. Systematic ventilation is difficult, and very little of the pillars is recovered.

The "block system" is a relic of unsystematic days, and notwithstanding its extensive use in the Monongahela region and elsewhere, has nothing to commend it.

A distinctively American method of mining thick soft veins is known as the "*square sett*" system; and the peculiarity is its equal adaptability to flat and to steep deposits, and it succeeds equally well with soft and firm ore (Fig. 13). It is a species of pillar and stall. The rooms and pillars alternate, and are of the same width—20 to 30 feet. The breast of each room is divided into working faces of about 7 feet wide: as fast as the miner has advanced his face 7 feet the square sett of timbers is built to uphold the roof, and is framed to those on either side of and behind him. When this horizontal slice of the room has advanced to the hanging-wall or the boundary of the ore-shoot, the next slice above is removed for a height of 7 feet, square setts being placed directly above the lower ones. The mining of slices proceeds thus, ascending to the roof or to the sill left as a floor for the upper level. The robbing of the pillars is accomplished in the same manner, by advancing sett by sett laterally from the rooms. They are rarely taken until the level is just about to be abandoned. The timber is left to preserve the stability in the open ground; little if any is recovered unless the rooms are subsequently filled with waste or the ground frozen. As nearly 1 cu. ft. is consumed per ton of ore, the expense of timbering is high—37 c. per ton, not uncommonly. Care in placing and close inspection afterwards is necessary, or the results will be discouraging. Inevitable collapses and stupendous caves, burying much of the ore, follow the weakening of the props or the softening, decrepitation, or movement of the

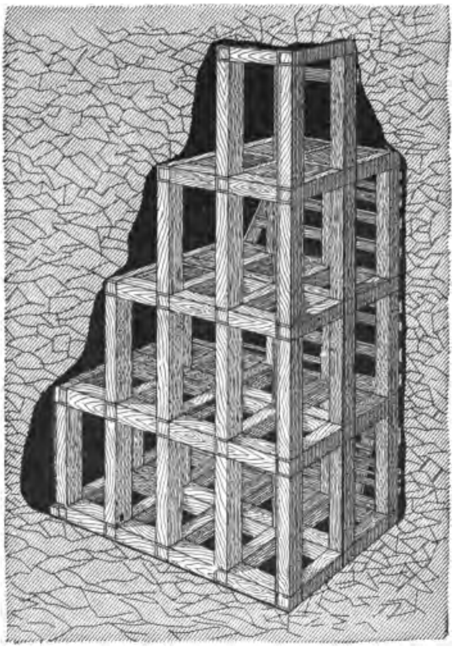


FIG. 13.

walls, and constitute the main argument against this method. In many localities the scarcity of timber necessitates its abandonment. (See II, **68,** and Figs. 152 and 173.)

It is noticeable that for seams of ordinary thickness several methods are in use, and but few suggested for thick beds. It may be possible, and even desirable, to mine a thick seam in slices, and with the same economy as in medium veins. Advantage is taken of partings, cleavages, or layers of slate or clay that subdivide the mass, and thus permit the working of each streak separately. If the partings are thin, the lower bench is first worked advancing. After the subsidence has ceased, and the mass consolidated, the others may be treated likewise, the top bench being mined retreating. The author is aware that in England, Poland, Southern France, and Northern Bohemia the reverse plan is practised. The terrible and needless loss of life, and the small proportion of coal, iron, or lignite recovered, remain as arguments against these operators. In such cases longwall and "gobbing up," offer greater inducements, in that all the ore is removed, and the land will yield a regular profit for a longer period of time than by some other method which may be two or three cents cheaper per ton, and waste considerable in its hasty extraction.

When several seams are somewhat apart they are worked separately, and in a descending order. The influence of the subsidences and disturbances must be guarded against, and the roof well stowed up by waste. As the distance between the seams decreases and the pitch becomes less, the liability of movements of the roof is lessened.

CHAPTER IV.

HOISTING MACHINERY.

13. Manual labor; description of windlass and winches; the work of man; examples; modes of increasing the efficiency of a windlass, double and conical barrels. 14. Hoisting by horse and whim; the work of a horse; examples; descriptions of whims, derricks, pulleys, etc.; double and conical drums. 15. Engine hoisting; conditions, etc., for selecting a machine plant; sectional and tubular boilers and their care; consumption of fuel and water; anti-incrustators and economizers; importance of the concentration of machinery; distribution of power; location of hoisters; description of the engine; cut-offs and condensers. 16. Descriptions of types of hoisting-engines; first- and second-motion engines; gearing and friction hoisters. 17. Description of the various types of friction-clutches; drums, their sizes and construction; the Calumet and Hecla leviathan; modes of equalizing the work of the engine; conical drums, reels, and counterpoises.

13. ADIT or tunnel workings are growing more rare, and it becomes an absolute necessity to deliver to shafts all products of the mine (ore, water, or waste), and this traffic is an important matter.

For shafts of moderate depth, and for winzes underground, manual labor on a windlass is employed, though the amount that can be raised by two men 100 feet in eight hours cannot possibly exceed 4 tons, allowing for delays, etc.

The windlass has a cylinder 6 to 10 inches diameter, long enough to reach across the shaft, resting by its axle on uprights, and operated at each end by cranks 15 inches long, and set at right angles to each other. Iron crabs or winches are to be had in every possible combination. But the simpler the machine the less is the friction, and the more acceptable it is.

Each additional gearing involves a large percentage of loss, and there is little to spare from the average man-power of 5300 ft.-lbs. per minute. They are, however, useful for incidental purposes in handling heavy pieces of timber, machinery, and pump pipes. A given power can only equal a certain product of weight and velocity; for an increase of speed, the weight must be proportionately diminished. With a 15-inch crank-arm, 12 revolutions per minute, coil 25 feet of rope on an 8-inch barrel, and with this speed of hoist the greatest load that may be moved under the circumstances by an average laborer is 214 lbs. Friction and stiffness of the rope will reduce this. A 150-lb. load can be raised at a speed of only 35 feet per minute.

To increase the efficiency of the windlass, the barrel may be lengthened and two buckets be suspended from the ends of the rope; one descending balances the other ascending. The length of rope is a few coils only greater than the depth of the shaft. At the start the weight to be hoisted is only the contents of the tub plus the rope. This weight diminishes in rising till at the top it is the contents minus the rope. This does not, however, obviate the great stress from inertia which arises at the moment of starting. For this reason single or double conical barrels are used, on which rope is coiled in such manner that the empty bucket is hung from the larger diameter, while the rope from the loaded tub at the bottom is wrapped around the smaller end. The tubs balance each other, but the empty acts with a greater leverage than the loaded tub, and thus assists the power in overcoming the inertia. After the hoisting is under way, the empty and its lengthening rope uncoils with diminishing leverage, while the load with its shortening rope gradually winds on a larger diameter of the cone. The buckets do not meet in the middle of the shaft, but in the middle of the number of revolutions of the barrel. In any event the pitch of the cone must be calculated for the given conditions of depth and load, otherwise its advantage is manifest only at the start. When properly constructed, the conical drum is to be recommended. Not many attempts are made to apply this mechanism to hand hoisting, but it is rapidly coming into favor with horse and engine power, notwithstanding that it is dearer than cylindrical drums.

14. Manual labor is manifestly too expensive to be regarded as any but a temporary expedient for hoisting.

When the height of the hoist is over 60 feet or the output more than 5 tons per shift, horse-power is employed to advantage. The average horse develops an effort of 135 lbs. when walking at a speed of 180 feet per minute. This is much below the theoretical value fixed for a horse-power, yet it represents the results of many tests upon the energy of the animal, which will raise 8 tons 200 feet per day.

For greater depths or quantity two horses are used. But when a still larger quantity is to be handled, a more efficient and economic power is employed. Nevertheless it is an intermediate step in the history of many mines, where it serves long as a simple, cheap, and tolerably satisfactory hoister where water is scarce, fuel dear, and the transportation of machinery difficult.

The invariable arrangement is a wheel-and-axle machine, consisting of a drum and driving-beam to which the horse is harnessed. Two sticks 6×6 and 9 feet long are mortised together at right angles to each other, with four 4-inch planks trimmed to the quadrant of a circle. These are held a foot or two apart by studs, and to them 3-inch plank staves are spiked to form the barrel, which, though upheld on the axle, turns freely about it. The axle is a round 2-inch rod stepped in a block of stone, and held at the top in an iron socket on the span-beam. The latter is 10 inches square, 36 feet long, supported on legs mortised and strapped to it. A square iron axle-rod fastened to the drum and turning with it is often seen, but is not so good as the free axle. The entire frame can be built for $100. The drum may be above or below the driving-beam.

A derrick frame is necessary over the shaft at a height sufficient for convenience of handling the hoisted tub. The hoist-rope passes over a sheave at the top—direct to the drum if set up high over the driving-pole, or under another pulley at its base if the drum is close to the ground. The latter arrangement is cheaper to build, but is wasteful of power, particularly so with a wire rope. For lowering the tub, a lever is in reach of the driver, by which the driving-beam may be disengaged from the drum and the tub lowered by its own weight, uncoiling the rope from the drum. A band-brake 3×½ regulates the speed. The brake must be set so as to work with the motion, not opposite to it; and the brake force exerted to produce larger tension in the driving, not in the slack, portion of the band. The lengths of the driving-beam and the diameter of the drum may be altered at will, but the ratio between them is also the ratio of the speed of the horse to that of the hoist.

If circumstances permit, two ropes may be operated from the same drum, one ascending, and the other descending. Conical drums may also be employed. Iron-framed whims are on sale, whereby a drum is horizontal and turned by a bevel-gear on its axle, fitting to another at the central end of the drive beam. While convenient and easy to erect, the introduction of the bevel gear involves additional friction.

The plane of the derrick pulleys should be tangent to the drum, and the latter far enough away that the rope may coil and unwind freely without chafing on its adjoining coil. This is accomplished by making the point of departure of the rope from the pulley at the same height as the central coil. Where the full and empty tubs are simultaneously operated, this can only approximately be attained. If greater nicety is desired, the pulley slides on an inclined plane as the coiling or uncoiling proceeds, or else the drum shifts its position by turning on a screw-thread. No lateral motion is allowed to the sheave, since the rope must occupy a central position in its own hoisting compartment.

15. When the engineer is compelled to plan for greater output, he is met with many difficult questions. The probable conditions, as well as the actual output, must be the guide, and common prudence suggests that the machinery be in excess of the immediate wants, which are more urgent with the cheapness of the mineral. Undoubtedly, permanent hoisting machinery must be placed sooner or later; though the sooner the better, it may momentarily be a question of available capital. With rich mineral and small product, the use of the horse-whim may be justified; yet money is wasted with each day. A 28-horse-power engine can do as much work as, at less cost than, 300 men on a windlass, or 35 horses on whims.

In selecting boilers, calorific economy, the amount of repairs, and the duration are to be compared. The records of the several types are easily had, and the facts gathered. The return tubular, or portable, boiler is being superseded rapidly by the multitubular, which in turn has a most active rival in the safety boiler. The efficiencies of the types average about 50, 60, and 90%. The relative initial cost at Denver, in place, is not far from 1–1.4–2.5. The last-named boiler consists of a series of connected tubes carrying the water, and whose outer surfaces are in contact with the fire. They are also called sectional boilers, are handily transported, safe, quick to clean, require little repairs, give dry steam and a high saving in fuel; particularly is this the case with high-pressure steam.

The tubular connections of the Babcock & Wilcox and the Root sectional boilers are illustrated in Figs. 14 and 15. Boilers should be set up as high as is consistent with the fireman's duties. An excess of boiler power is advisable for emergency.

The standard boiler horse-power, as adopted by the American Society of Civil Engineers, is the hourly evaporation of 3.55 gallons of feed-water at 100° Fahrenheit to 70 lbs. gauge pressure (33305 British thermal units per hour.) A storage of an ample supply of the purest obtainable water is urged. Incrustations of solid matter deposited from the water are frequent causes of explosion, in consequence of the destruction caused by the contact of the scale, as well as by the overheating necessary to evaporate the water. The presence of a scale one-sixteenth of an inch thick on the tube causes a loss of 13% of the fuel heat. Filtration through hay in the bottom of a tank will remove the muddiness; but the solids in solution are rendered innocuous only by the use of alkalies. Common washing-soda is as effective as any, and much more so than the majority, of the nostrums sold as "anti-incrustators." A little glycerine—one pound to every 400 gallons of water—is beneficial. An excessive use of any of these causes "foaming." Tannates or organic compounds are injurious. Frequent "blowing off" and cleaning every week is necessary with the common tubular boilers.

The consumption of fuel per hour averages 3.5 lbs. per horse-power with the common boiler and ordinary engine. A cord of well-dried spruce is capable of running a common slide-valve hoister 320 hourly horse-powers; a ton of lignite, 470; a ton of anthracite, 650; and a barrel of petroleum, 170. Thus, a barrel of oil is worth more than 0.5 cord of wood or 720 lbs. of lignite. Firing with a thin bed of coals, and a proper lubrication of the engine, will reduce the quantity of coal consumed. A steaming-coal should kindle readily, and burn steadily without clinkers. A superior coal is that high in fixed carbon; a little volatile matter will assist the firing. Slack is nearly equal to coal in calorific value, but it has, usually, too much refuse to be acceptable, except where the transportation is cheap. On account of the variant "personal equations" of coals, experiment alone will determine the best coal for a given grate.

A great saving in fuel can be had by heating the feed-water before injection. A braced tank of matched, dovetailed boards receives the exhaust steam and well serves the purpose of a heater. The open end of the exhaust-pipe blows steam over the surface of the water, and is placed higher than the waste pipe which maintains the level. The exhaust-pipe may also be laid as a coil, with its end discharging into the air. Of course "economizers," or combination heaters and purifiers, may be purchased. In these, some of the heat from the chimney gases is employed to heat the feed-water (Fig. 14). The benefit from them increases with the wastefulness of the boiler. There is little gain in adding them to a condenser. Inspirators are more troublesome than pumps for feeding boilers.

The concentration of the machinery within as small a compass as possible should be aimed at. A large boiler and engine is more efficient in the development of power, and with less attendance, than a number of small plants. A distribution of its power to remote points by electricity, compressed air, wire-rope, or string-rods, is possible with little loss; and few arguments favor the establishment of numerous small outfits, so often encountered in extensive mines, except those which also excuse the lack of a system. Fireproof housing is indispensable; not only because of the weather protection given the plant, but also because injury to the motors threatens the dependent machinery, affecting the safety of the men and the security of the mine. Many causes of fire above and below ground contribute to make this essential. The buildings should be as far away from the shaft as is consistent with

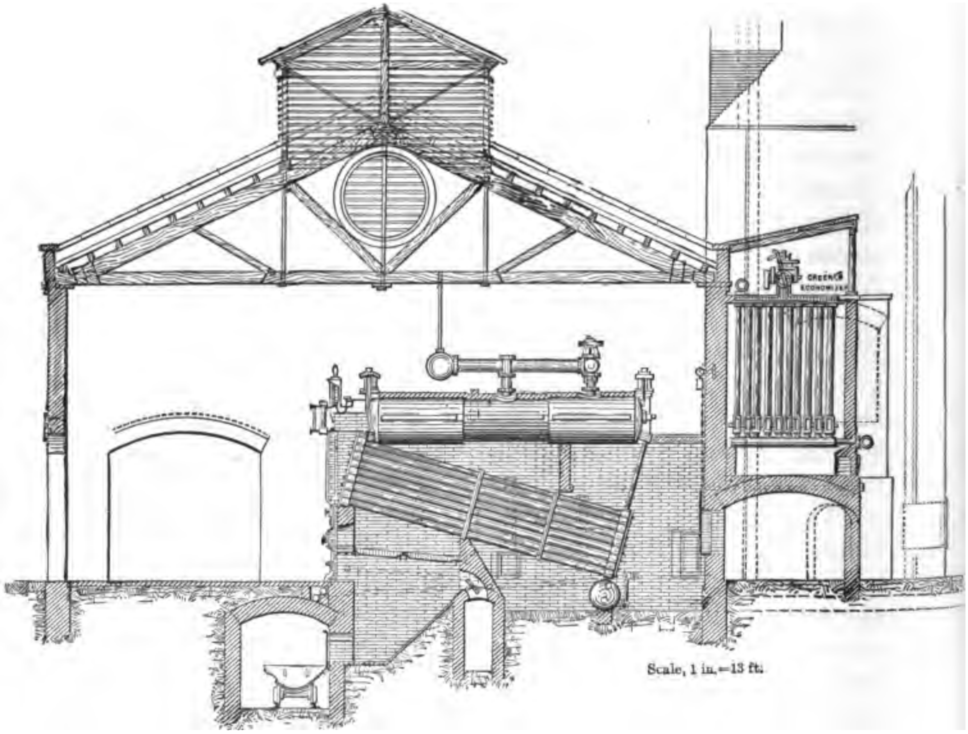


FIG. 14.

economic work. But two reasons exist for having hoisting-appliances close to the shaft,—quick landing of the cage, and dumping facilities. The former is secured by isolation of the hoister-man, who has a free view of the shaft-mouth from a position high enough up and far enough away to give a good lead without allowing oscillation of the rope (Fig. 16). A complete system of signals gives him control. Where the output is small, the tubman, who attends to the dumping, can also give the signals, or, by having brakes, levers, and throttles at hand, he may manage the engine from a distance. A gravity plane from the shaft to the mill or dump, under the charge of the surface unloader, will dispose of the ore and waste.

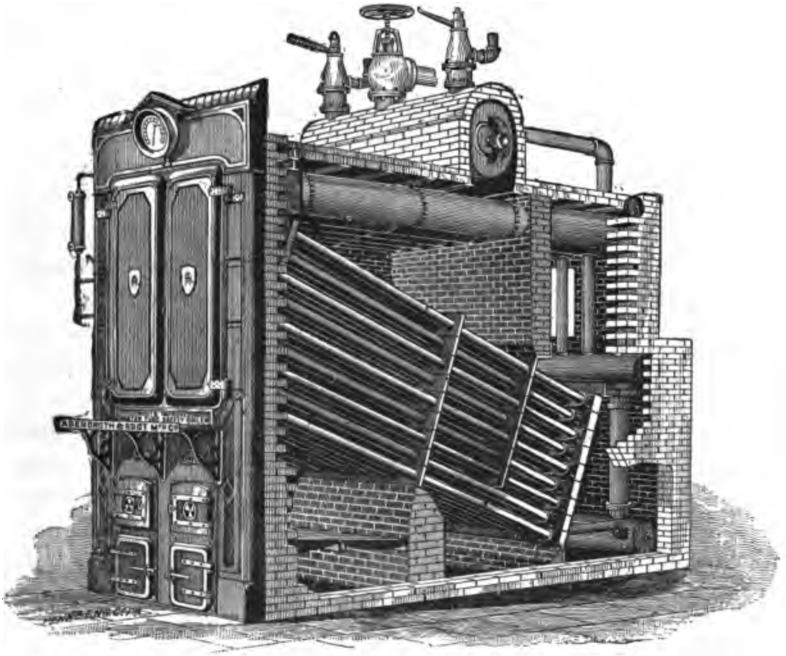


FIG. 15.

The hoister is best located opposite to one end of the shaft, with the boilers as close as possible, and the fan connected to its compartment, containing also the pump-pipes and ladder-way.

The foundation of the engine is prepared before its arrival by laying an accurately constructed template in the proper position for the engine, with the bolts standing erect from it, long enough to reach through the masonry to the bed-plate. The foundation of concrete and brick, gives a perfect bearing throughout for the bed-plate. Timber seats are often used, but are too elastic for rapid hoisting, which demands rigidity. Only second- and third-motion haulage-engines may have timber foundations.

A hoister may be described as a simple steam-engine attached to a drum for receiving the rope. The combinations are numerous, from that of the upright boiler and its engine on the same base, to the vertical beam condensing-engine connected with the drums on separate foundations. The latter is of European preference; the former for prospecting-service only. As a rule, however, the cylinders are horizontal.

The intermittent operation of hoisting an un-uniform load at high velocity is not inherently economic. The regulating fly-wheel had long since to be abandoned as dangerous for reversing engines, so all the means of conserving power must be by the use of condensers, high-pressure and expansive working. The comparatively high initial cost of these improvements constitutes the objection to their more extensive use, yet this bears a small ratio to the saving in fuel. Ordinarily, 33 per cent. is quoted as the fuel gain in favor of condensers where plenty of water is procurable. The exhaust steam is condensed in a water-jacket reservoir, and pure distilled water obtained for return to the boiler. The best remedy for overloaded engines is to add a condenser which, for ordinary pressures, will shorten the cut-off. It does not usually effect any saving if attached to a lightly loaded engine at high pressure. It is better to reduce the pressure.

16. The designs of hoister are numerous, but the types few (tail-rope and other engines are embraced in this discussion), divided into the slide-valve and cut-off classes. The slide-valve allows of the use of the steam at full pressure throughout the entire length of stroke, and simply acts to

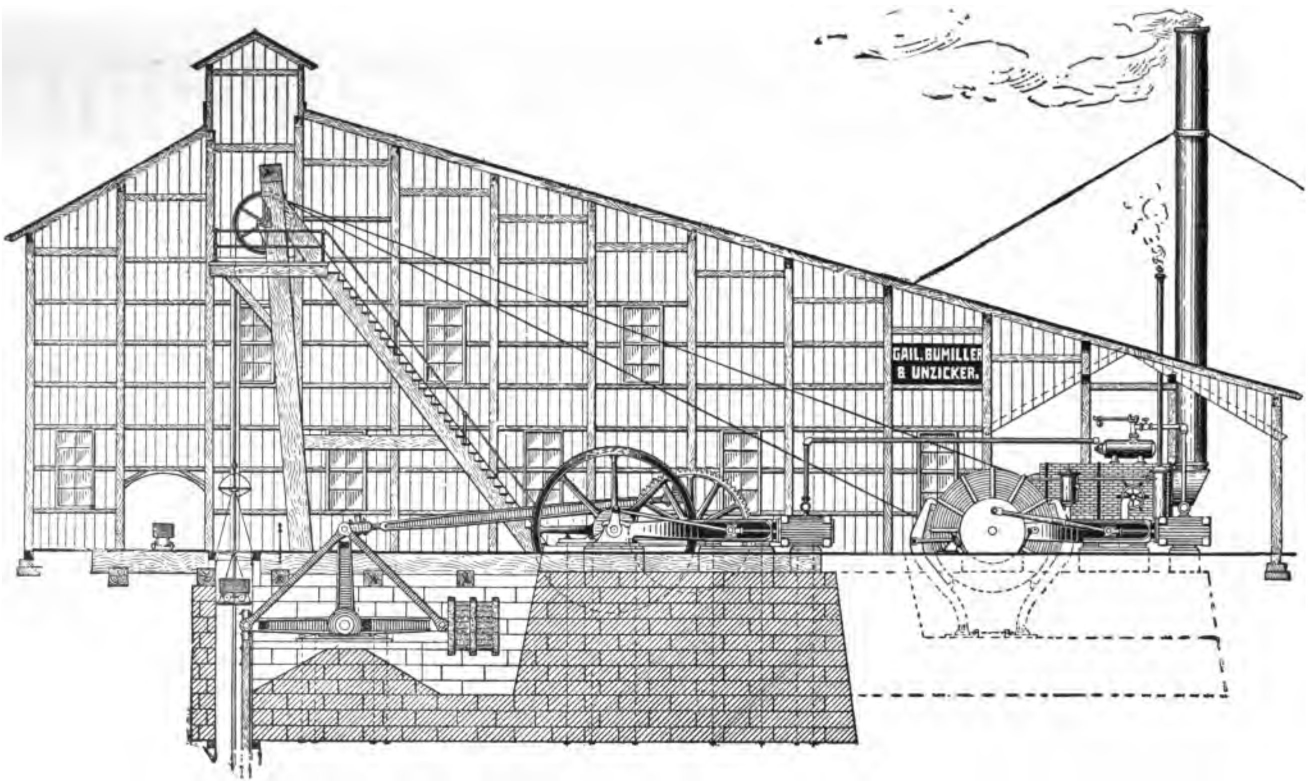


FIG. 16.

reverse the direction of the induction. Balanced slide-valves are much admired. Portable and pony hoisters are of this class. The larger and more perfect types of engines work the steam expansively, by cut-off or by compound cylinder. The mechanism for shutting off the steam supply before the piston reaches the end of its stroke may be fixed or variable, automatic or adjustable.

A fixed cut-off is intended for a constant duty, as for marine purposes. In it the induction-valves are set in some such fixed position that the gearing, or governor, will always close them when the piston has reached a given point of its stroke; after which the piston is urged by the expanding steam. In the variable cut-off, only a single valve is used, but it operates on the exhaust as well as on the induction, and steam is caught on the exhaust side, and capacity is lost. The Meyer's cut-off is adjustable and can be set by hand, at any time, to regulate the point of the cutting off. The exhaust is not interfered with, because an auxiliary valve closes the induction. In the automatic cut-off, the governor operates a valve for the induction and one for the exhaust, each with a motion independent of the other. In one class of the automatic, the valves are inoperative until the engine has attained sufficient speed, when they are brought into play by the governor, and the speed becomes constant.

The Corliss is a fixed type in principle whereby a trip-gear, regulated by the governor manipulates the valve at each stroke. It is practically instantaneous, and is increasing in popularity among mining men. Its best results are at 0.2 cut-off with an average load, and at this rate generates about 40 per cent more power than is attained from an equal amount of fuel in the plain slide-valve. A 100-horse-power Corliss will save fully 300 tons of coal annually. (Figs. 17 and 23.)

Still greater economy in generating power is had by a high degree of expansion and the tendency of the times is toward this goal. The final pressure in a non-condensing cylinder cannot be below 19 pounds, as it requires three or four pounds to expel the exhaust and to drive the engine. A short cut-off,

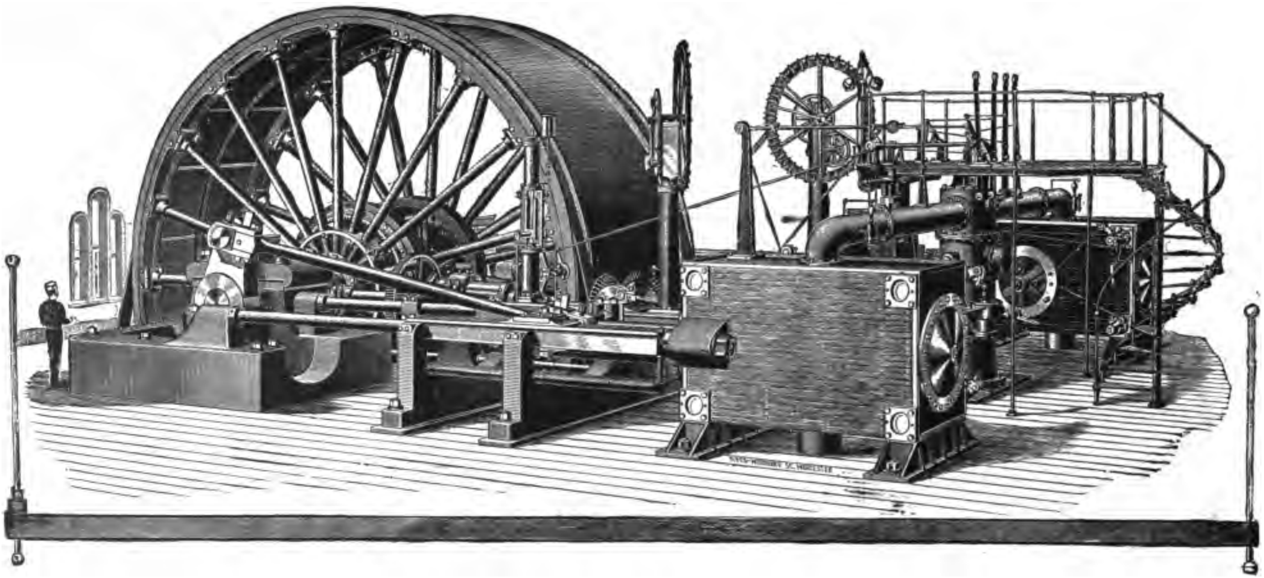


FIG. 17.

therefore, necessitates a heavy boiler pressure. Again, the rate of expansion is limited because of the condensation accompanying steam highly expanded. The ratio, or the minimum pressure, is also prescribed by the conditions of the hoister. It is expected to be able to start its load from any point in the shaft and at any point of its stroke. With a low terminal pressure a fast hoist may not be possible unless the cylinder be large. So, valve-gear cannot be set permanently for a short cut-off without a high initial pressure. Moreover, a high rate of expansion means a low average pressure throughout the stroke, and this involves the use of a large cylinder for a given power. So that, while hoisting is medium slow, the ordinary pressure is permissible; but, for reasons suggested, only high boiler-pressure and considerable expansion can satisfy the tendency toward rapid winding. This is creating a demand for high-class machinery, the which is being rapidly installed, heavier and larger than formerly.

The evils attendant upon high single expansion are best remedied by the use of compound cylinders. The steam, after doing duty in one cylinder, is discharged into a second larger one (sometimes from there into a third), where it is expanded before being exhausted into the atmosphere or a condenser. By this means each cylinder has a range of temperature and pressure which is not large, but previously fixed. Triple—and even quadruple—cylinders, and expansion, are employed; but their advantage is developed only by a high pressure, a comparatively steady load, and heavy work; hence not for hoisters. The cylinders are placed tandem or alongside, the pistons being on the same rod, or united by cross-head; each piston furnishes an independent force, and the power of the engine is equal to the sum of the pressures operating. The steam-cylinders should always be enclosed in a poor-conducting envelope or else by a steam-jacket. This prevents condensation and allows of a higher rate of expansion. Fully 8 per cent of fuel is saved thereby.

It more frequently happens, however, that economy of fuel is the last consideration, the other requisites being more urgent,—simplicity and safety, for example. The engine must be under complete control, and capable of quickly attaining full speed.

The communication of motion from engine to drum may be direct or secondary, and the mechanism reversible or not. In the former class the load is hoisted, held, and lowered by steam, the reversible movement being easily imparted at will. It is perfectly safe and speedy, but does not admit of expansion-gearing. The piston is directly coupled to the drum, and this is the only form by which high speed can be quickly attained (Figs. 17, 18, 19). Pedestals for supporting the drum-shaft are upright or only slightly inclined.

Second-motion engines include all those by which the power is transmitted to the drum through a driving-shaft carrying some form of gearing or friction wheels, or through some variety of friction surface. Its introduction is intended for the regulation of the speed; but gearing involves a loss of power, so that, where its intervention could be dispensed with, it would be highly desirable so to do. See illustrations, Figs. 20, 23, 24, and 25.

The driving-shaft may carry 1 or 2 pinions which mesh with an equal number of spur-wheels keyed with the drum to the shaft. The ratio of this gear is between 4 to 1 and 7 to 1. Some quarry engines are provided with a double set of gear for rapid and slow motion (Fig. 20). The ratio must be the same as that of the weight or speed of the load to the capacity of the cylinders. Power and speed are convertible terms. Occasionally, instead of spur-wheels, V friction wheels are encountered. The small wheels or pinions are in continual revolution, a lever throws the drum against them, and motion is effected. Great friction is thus obtained without excessive pressure; but they soon fail to work properly, because of the irregular wear of the grooves.

For pony hoisters, friction wheels are used instead of the V-grooves. The revolution of the drum is obtained by bearing it against a *papier-maché* wheel on the driving-shaft (Fig. 26). Only for prospecting-work are they advised. The sur-

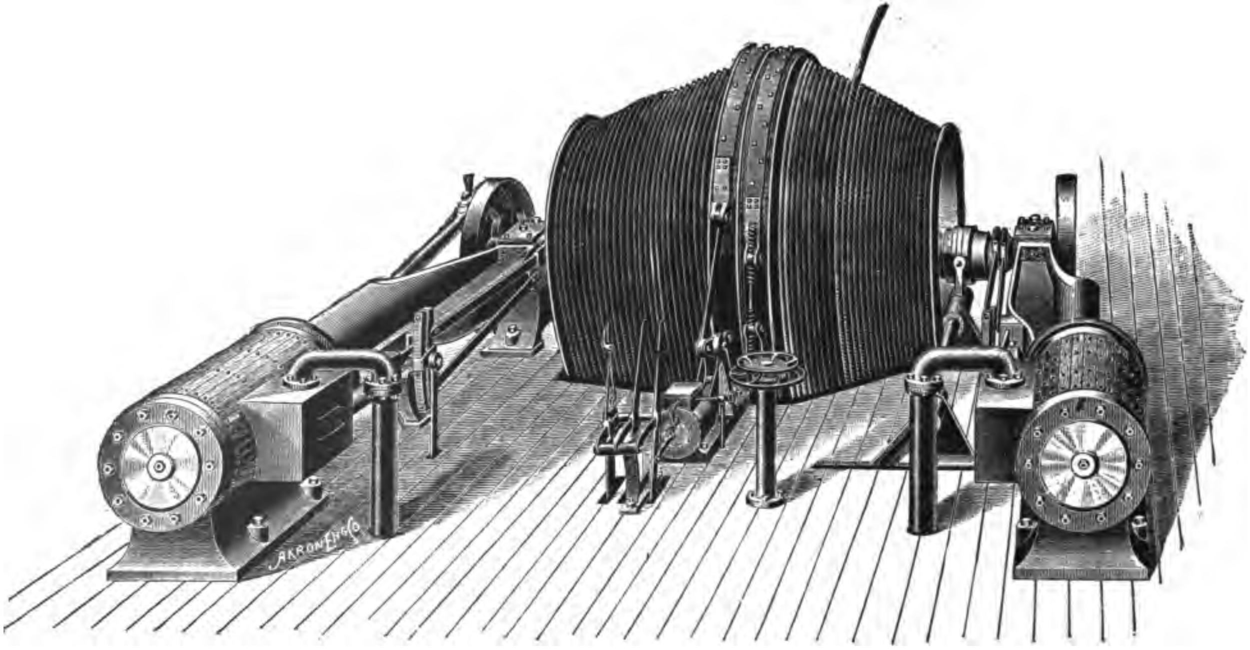


FIG. 18.

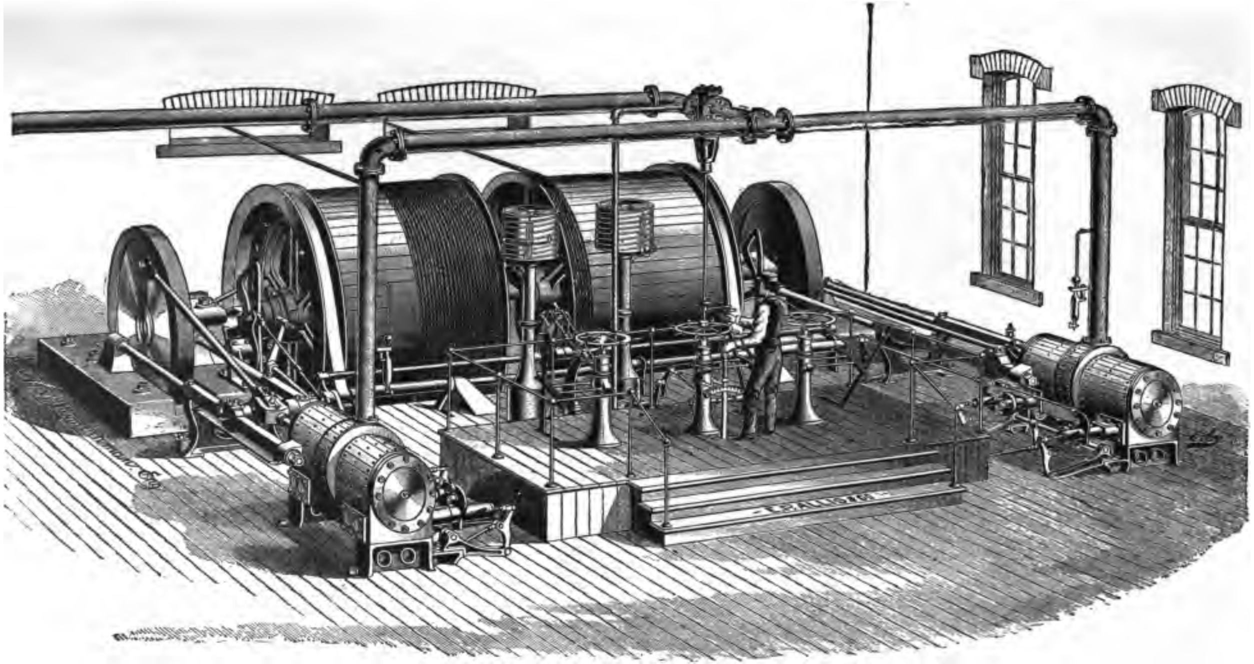


FIG. 19.

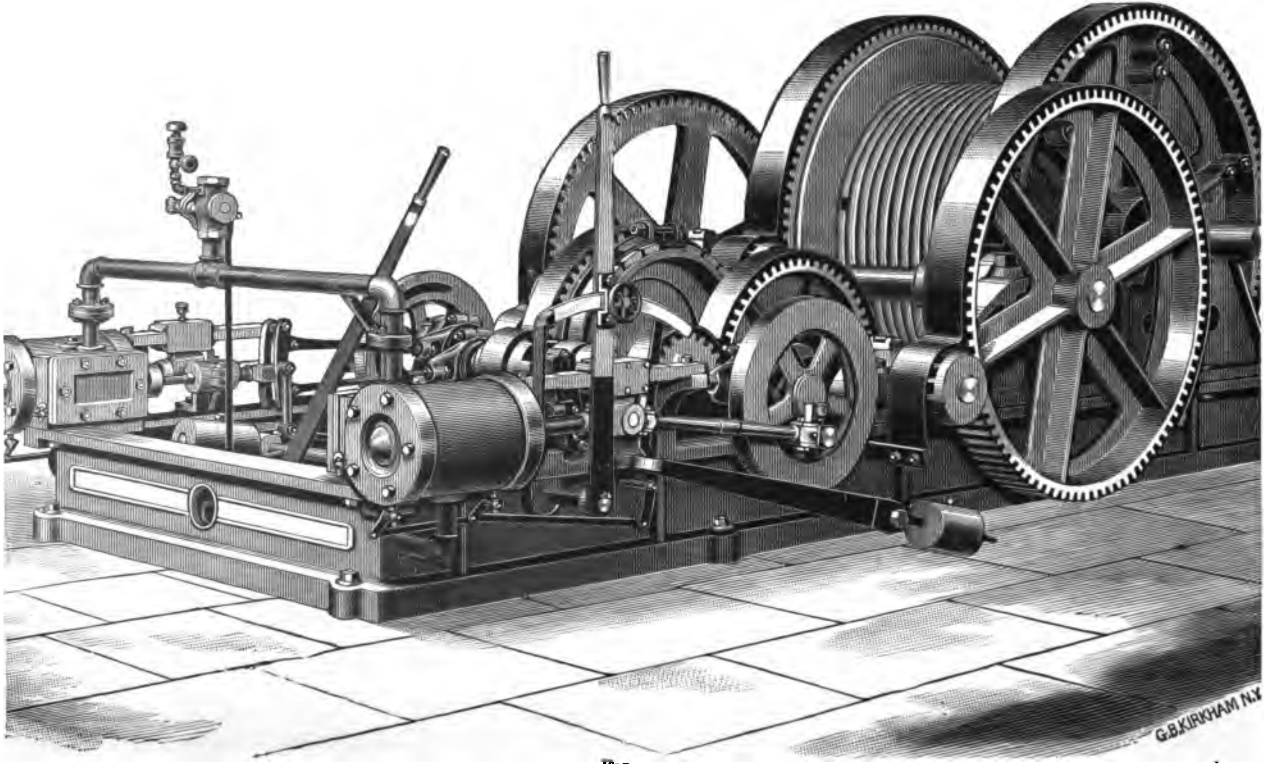


FIG. 20.

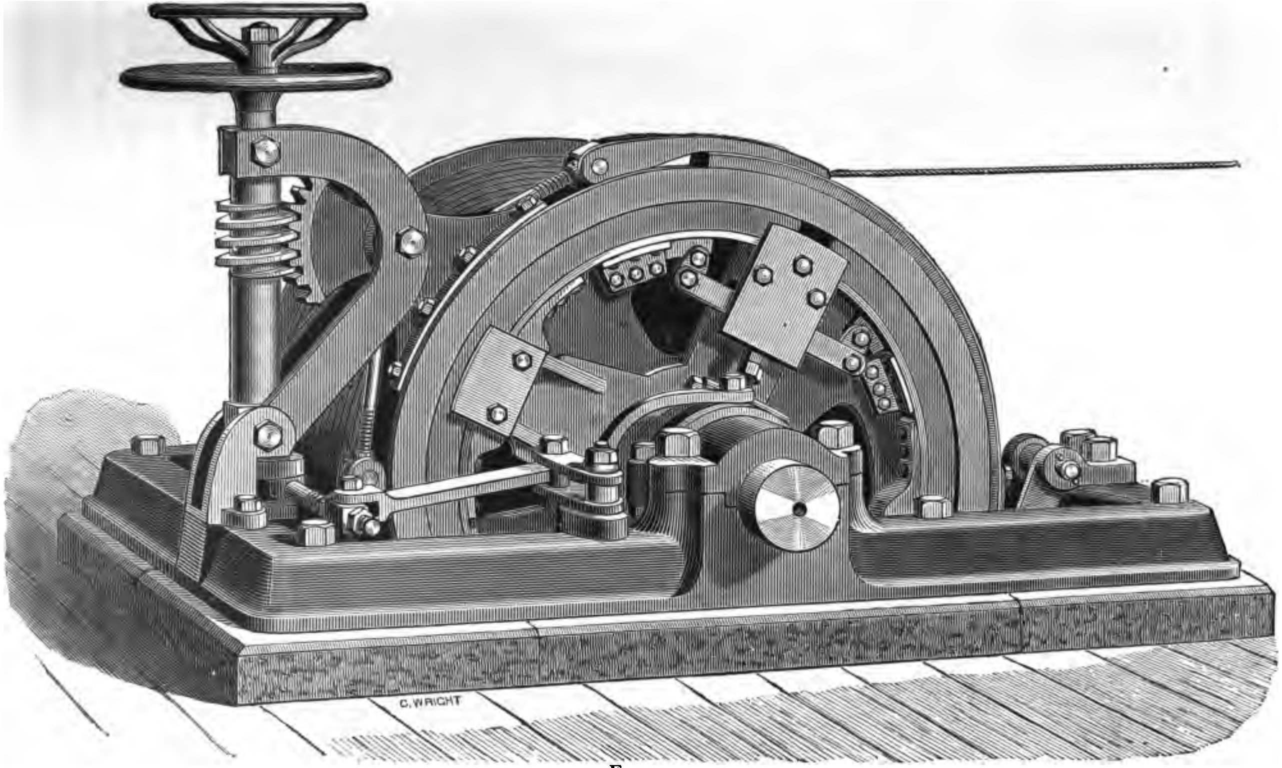


FIG. 21.

faces must be kept dry and free from water, steam, or rosin. A word of caution against the use of rosin, coal-tar, etc., on friction surfaces. Undoubtedly they increase the friction momentarily, and the men are prone to use it, but the heat developed soon melts or gums it, and the surface has become too slickery for remedy by other means than a cold chisel and elbow-grease. The author has posted, about his machinery, notices of prohibition against the use of these materials.

17. The commonest class of hoisters, after the direct-connection engines, is represented in the types of friction-clutches engaging the drum in various ways. They are quick, powerful, noiseless, and give little shock. One form of internal friction consists of driving-band and four arms keyed to the drum-shaft (Fig. 21), the gear-wheel of which receives the motion from its pinion on the driving-shaft. The drum is well bushed, free on its shaft, and turns as soon as the band is expanded to contact with its inside rim. This the engineer manipulates by a lever. In another variety the contrivance is a set of curved friction-blocks on spokes fast to the shaft. These bear against a similar surface inside of the drum, when the latter is slid up to the engagement. The reverse is also in use, whereby the friction-blocks are forced into the drum. The device resembles a dished-wheel. Internal gearing regulates the hoisting-speed without altering the engine-speed, and is quite successful. Of the external frictions we have a sectional driving-band attached to a two-armed driver keyed to the drum-shaft and operated by sliding-blocks and bell-crank levers. When the sliding-collar is moved towards the drum, the band clasps it and imparts the motion (Fig. 22).

For variable lengths of hoist, drums have an inside pinion gearing into a rack on the drum. This can be slipped out at will and the free drum pays out rope until the proper level is reached. Then the pinion is returned, and the drum is ready.

These appliances are keyed, by the troublesome feather, to round shafts. Octagonal shafts are preferable, though more costly. Some hoisters are provided with both link-motion for men, and frictional-clutch for other purposes. The best type

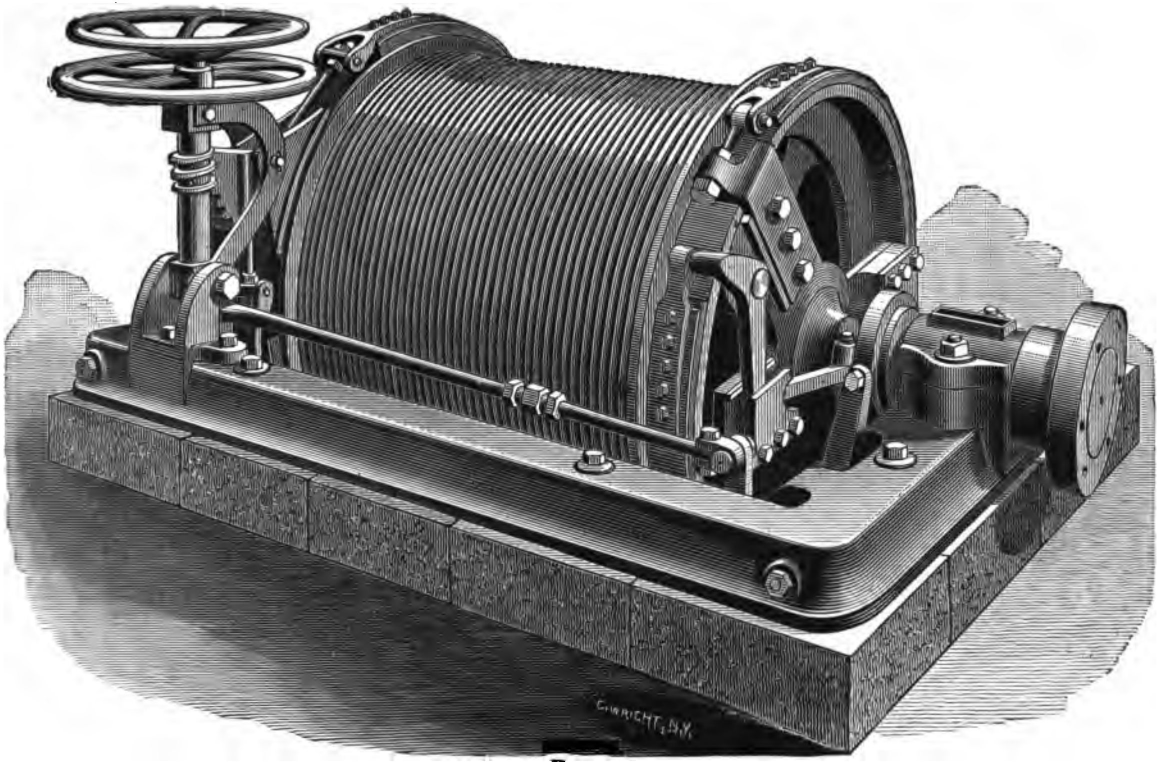


FIG. 22.

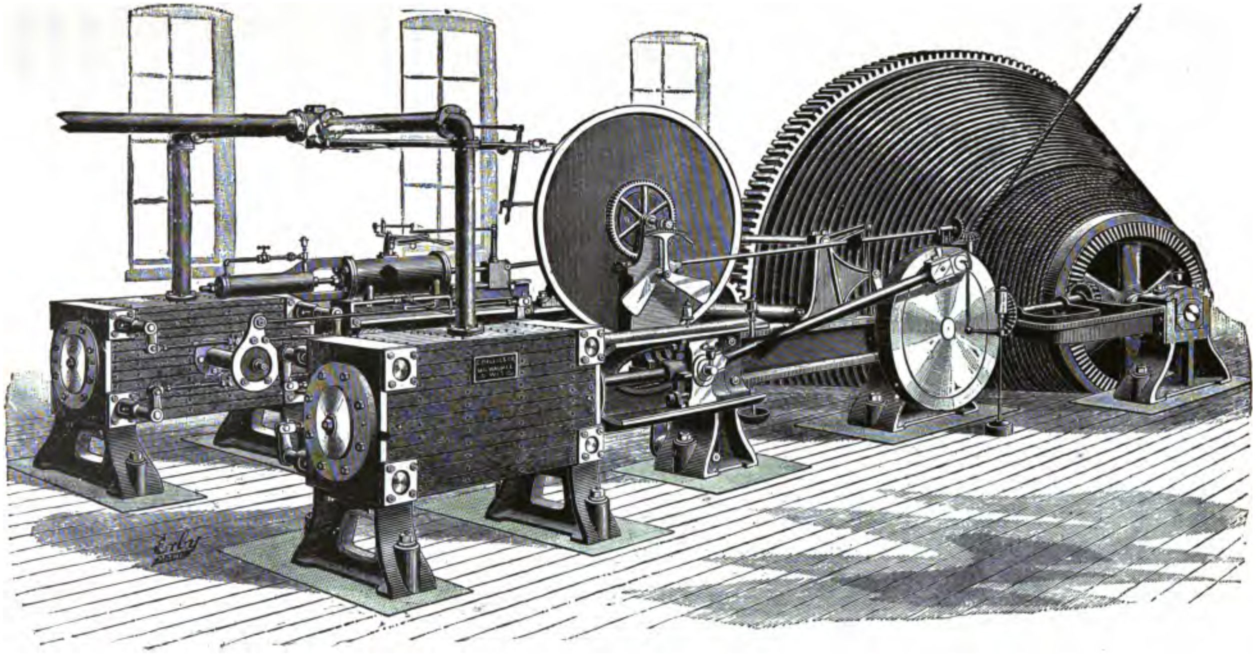
is a double-cylinder, high-pressure engine fitted with variable expansion and reversing-gear, having pistons directly connected on the fly-wheel shaft, on which are the drums, with steam-brakes. In some localities, the drum, on a separate foundation, receives its motion by wire rope, electricity, or otherwise. To avoid dead-points, the drum, or its driving-shaft, is coupled to two independent engines by cranks at right angles (Figs. 20 and 23). This arrangement is more equable and safer than the single cylinder.

The direct-connection hoister and the reversible-link hoister, Lidgerwood pattern, Figs. 20 and 24, are always in gear, have reversible valve-motion, enabling them to turn "over" or "under." They employ steam, whether for hoisting or for lowering, and are perfectly safe for carrying men. By varying the position of the cut-off, the speed is controlled. With the other classes, hoisting cannot begin until the drum be thrown into gear after the engine has "got up" speed. Moreover, their motion is positive and continuous, non-reversible. To lower the load, the drum is disengaged from its gear or friction, the speed of the uncoiling rope regulated by a brake.

The brake is either an iron band clasping the rim of the drum, blocks of hard wood on end against it (Fig. 25), or V-grooved wheels. The first is the least troublesome and is safer, though having less friction than wooden blocks. The band-brakes are represented in Figs. 18, 19, and 21.

The brake is suitably applied by a simple lever, a band-wheel and worm-screw (Fig. 21), or an auxiliary steam-piston, according to the size of the hoister (Fig. 18). The last-named acts powerfully,—indeed, too suddenly,—causing injurious shocks, against which are numerous devices. The leviathan at the "Calumet and Hecla" mine requires a small auxiliary engine to stop and start it.

Each compartment of a hoist- or tram-way has a rope and a drum. Sometimes, for double-compartment shafts, two ropes are wound—one over, the other under—on the same drum, which is directly connected to a reversing-engine. Ordinarily, however, for multiple-hoisting, each rope has its own drum in-



FIG, 23.

dependently controlled. The drums are all geared to the same driving-shaft and placed centrally between the steam-cylinders. The two cylinders may or may not be fed by the same throttle.

Drums up to 8 feet diameter are cast whole; above that, in segments, cylindrical or conical for round ropes, or in reels for flat, and lined with plank staves to reduce the wear of the coils. The considerations favoring one or other of these remain the same as mentioned previously (**14**), only more so. Their size is proportioned to the diameter of the rope and the depth of the shaft. On large cylindrical or conical drums a groove is turned spirally over their face for the reception of the round rope (Figs. 23 and 24). Its diminishing pendent weight, as the hoist progresses, throws a markedly unequal work on the engine, and, without a variable expansion, regular speed cannot be maintained. Some approach to equality is established with two cages or tubs going in opposite directions, but that is only for certain moments (Fig. 16 and 18). Nor is the inequality of motion effectually remedied by flat ropes winding on reels (Fig. 25): the start is eased, that is all; the decrease in the load, due to the shortening rope, is not compensated for by the increased leverage. Equalization during each revolution is what is aimed at. Conical or tapering drums partially attain the desired end; tapering ropes diminish the difference between the initial and final loads; and some compensation is had from the use of the flat rope and reel; but as yet no more satisfactory solution has been reached than that by counterpoises.

The conical drum, or fusee, is built for a given condition of load, speed, and depth. It may be single or double (Figs. 23 and 18). Its minimum diameter is determined by the size of the rope (see **26**), beyond which the winding is on a surface described by a curve such that the moments of the ascending and descending load are constant, and give perfect balance for all positions. One danger with tapering drums is the tendency of the rope to slip from its place, where the angle is over 30°, but the spiral groove largely prevents this. In a

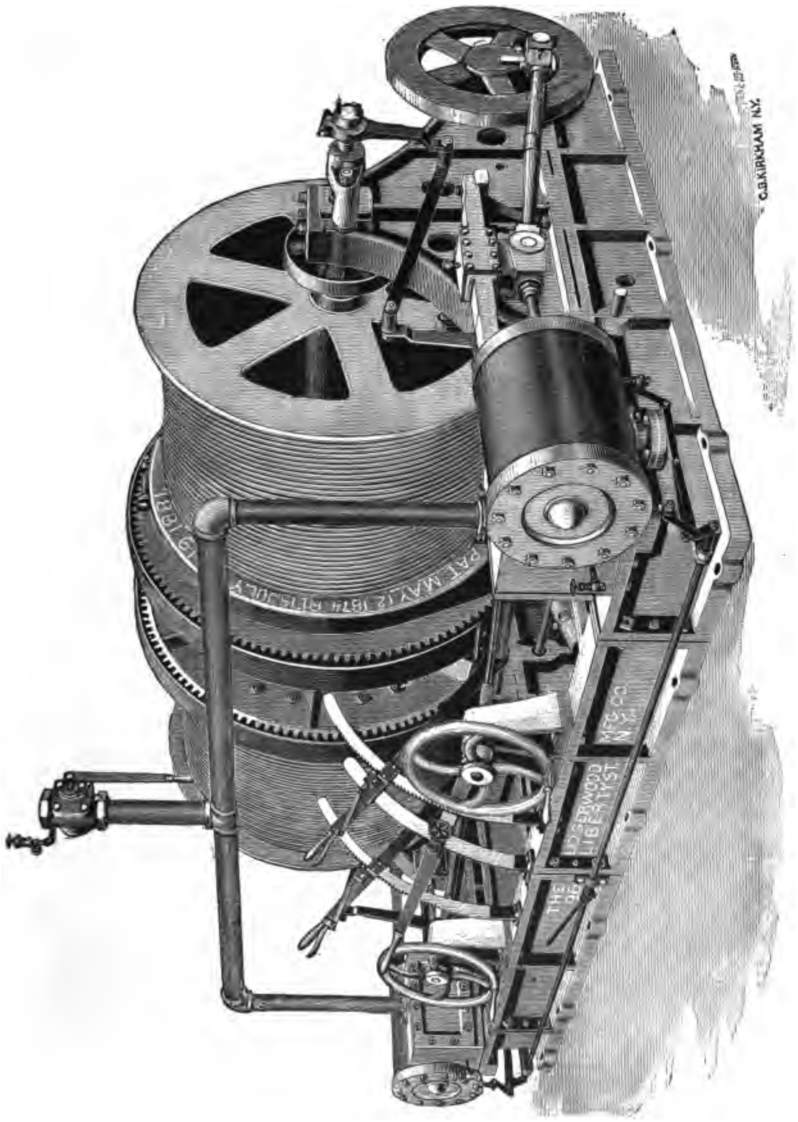


FIG. 24.

shaft whose depth or point of hoist is fixed, the equality may be maintained: evidently this is impossible in vein-mining, and counterpoises are resorted to; beside which there are no better

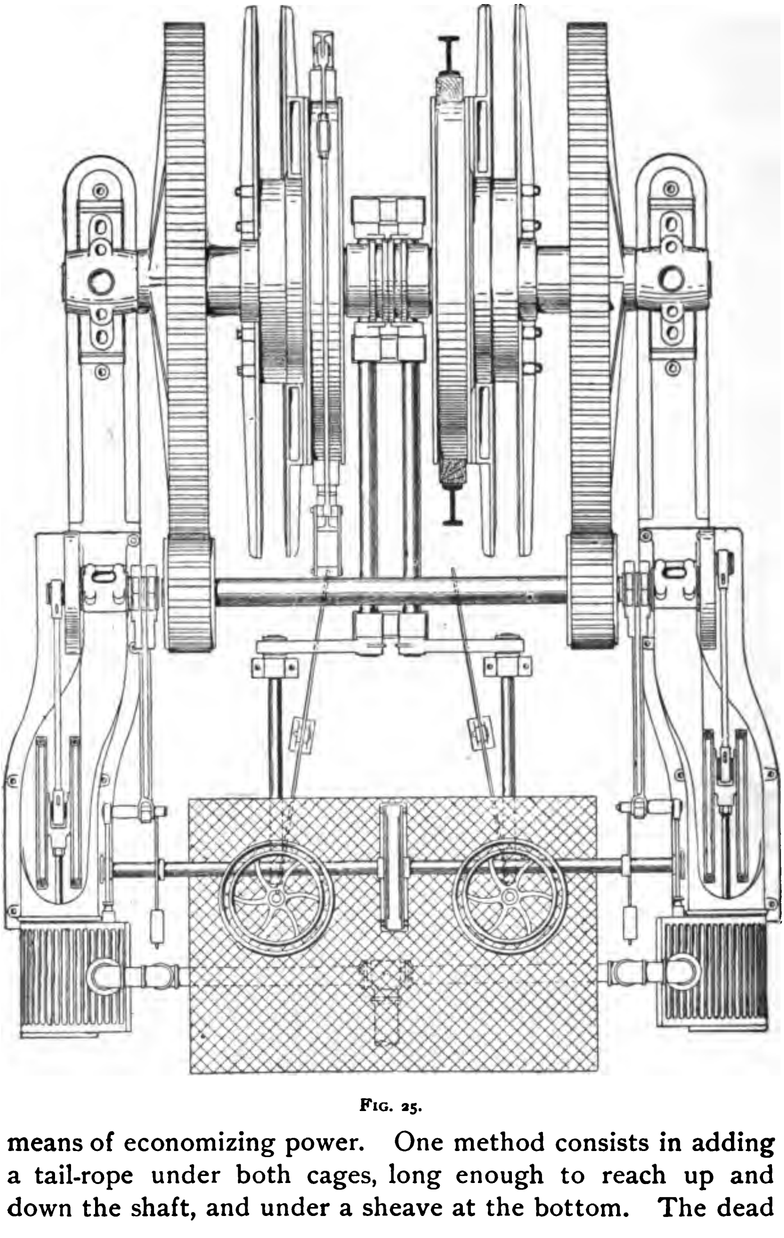


FIG. 25.

means of economizing power. One method consists in adding a tail-rope under both cages, long enough to reach up and down the shaft, and under a sheave at the bottom. The dead weight on each rope is constant; the oscillation of the cage is reduced, a regular speed quickly attained; friction, also the size of the rope, is increased. A shaft free from impediments is necessary, as also a dry pit. Another plan employs a heavy weight, and a chain wound on an auxiliary drum in such a manner as to assist the engine at the start, reducing its effect as hoisting progresses. The Koepe system of winding meets almost all the requirements of a perfect equalization, and is highly efficient. It ensures against overwinding, decreases wear, and dispenses with the enormously heavy drum, using a sheave instead. The two cages are connected by tail-rope below, and by main rope above. Then the engine does a steady, uniform work of lifting the net load only. Hoisting is possible only when the friction caused by the loads on both ends of the main rope is greater than the weight of the net load carried on the rising cage; and in several localities it has been abandoned, because, immediately after oiling, the rope would slip and the work was unsatisfactory. By counterbalancing, the work is only that of raising the live load + friction. Single hoists (unbalanced) are excessively wasteful in power, fuel, and hard on the brakes. The fuel value of hoisting 1000 feet of rope and a heavy cage 30 times an hour is no small quantity.

It may be well here to mention a most useful piece of apparatus, and by no means superfluous about a mine, for handling heavy articles,—a portable tripod and a Weston differential-pulley block. It is simple enough to be manipulated by any one without fear of injury by rough treatment, and is exceedingly powerful (Fig. 26).

FIG. 26.

CHAPTER V.

ELECTRICITY AND WATER-POWER.

18. Application of electricity and water-power to long-distance transmission; comparison with mechanical means; universality, to all operations of mining. 19. Conducting wires, size, etc.; two-wire and three-wire systems; safe voltage; explanation of the electric units, and formulæ; conversion of electric into kinetic energy by motors; efficiency of motors; storage batteries. 20. Mode of obtaining water-power by the use of Leffel, Knight, and Pelton wheels; description, efficiency, and application of the plants and machines.

18. THE most valuable acquisition made to any branch of industry during the past few years was electricity, and with phenomenal rapidity it has gained favor. Not more than sixteen years ago electricity was a mystical force that was not suspected as capable of operating even a telephone. To-day the installation of a plant ceases to be a novelty; and its utility as an illuminator, and a power capable of long-distance transmission, is unquestioned. It is true, it has not yet realized all the hopes and anticipations of its zealous advocates. Serious objections have been raised against it, and many plants have proven failures, yet it has so demonstrated its merits that, with better understanding, it cannot fail to work an entire revolution in the industries.

Electricity may be carried to any moderate distance, in any desired quantity, through a small light conduit on inexpensive supports and with slight loss. It offers great assistance to engineers in utilizing remote cheap sources of power, and is destined to supersede all known methods of power transmission. It is preferable *per se*, and because the difficulties in the actual transference of matter by mechanical means over the intervening distance are great. The difficulties increase with the distance in any of the systems, but those with electric are less than with mechanical methods, appliances whose great initial cost and low efficiency have hitherto restricted our work. The first cost of a moderate-sized plant is considerable, compared with other modes; but, once installed, it is easily capable of great extension. Its efficiency is high: whereas a fine steam-engine pays out to the recipient belt only 14 per cent of the fuel energy consumed in the boiler, a dynamo will convert fully 90 per cent of the total water-power into electric energy. The former will consume, perhaps, 2.5 lbs. of fuel per hourly horse-power, which is saved to the latter. The first cost may, in certain cases, favor the latter.

The losses from condensation, friction, etc., in the conduction of steam cannot but be great. If the engine operates an air-compressor, the efficiency is reduced from 14 to 10 per cent at least. As a matter of fact, neither steam nor compressed air can be converted into power with a loss of less than 50 per cent of the energy received. Hence from 5 to 7 per cent is the best that can be expected from the use of these expansive fluids, which can never be regarded as serious competitors, except within a very limited scope. Then, compare the cost and inconvenience of large thick pipes required for the conduction of air or steam with the ease and rapidity of laying, supporting, and insulating a mile of wire. Wire rope gives better results; but for distances greater than half a mile it is superseded by electricity, because of the losses by friction; besides, it can transport power to a certain class of appliances only.

On the other hand, electricity subserves practically all the operations of mining: signalling by annunciators or indicators, lighting, blasting, drilling, hoisting, haulage, etc. It neither vitiates the air, as do engines; nor fog and chill it, as compressed air. There is no leakage of power when the motor is not in use, as with other means, and is especially commended when the power is to be intermittently required. A copper wire $\frac{1}{8}$ of an inch in diameter is equivalent to a $3\frac{1}{4}$-inch air-pipe, or $\frac{5}{16}$ wire rope, for conveying power at average pressures; cost, 1 : 27 : 19 for equal lengths.

19. The transmission is by wire, the size of which is commensurate with the quantity of energy to be transported. This is realized, irrespective of distance, with only a slight loss due to the heating of the conductors and poor insulation. This heat-loss is directly as the length and inversely as the cross-section of the wire employed. It consumes power, and is independent of the pressure employed. Practically the size of the wire to be employed depends upon the current, and its cost varies inversely with the square of the pressure. Now, as theoretically the economy of transmission increases with the pressure, a high potential may be employed without increasing the cost of the conductors, as is the case with pneumatic or water-power. If the Ferranti alternating current scheme is a success, the field of application for electricity will be enormously extended. But there are drawbacks to the use of high voltage; greater care must be given to insulation, else broken insulators and hot wires give rise to injuries and fires that have interrupted its success. By proper safeguards and careful construction these dangers are eliminated, though a pressure of 450 volts seems to be a safe mining limit. As a matter of fact a high potential current may be generated, and, at the points of distribution, transformers may, as it were, convert a part of the intensity into a greater volume of low tension, which is not only safer but also more economical. This, however, is not yet efficiently accomplished with positive currents.

Two wires may be used, or even three, the middle being a neutral wire. This latter allows of a higher voltage being carried, and at the same time reduces the size of the wire to one-fourth, and its cost to one-third of those of the two-wire plan.

At first the greatest impediment was the large cost of conductors. This is now partially overcome; at present the expense of placing is far greater than the cost of the wire. Tesla's discovery of high potential alternating currents, creating induction, promises to dispense with wires altogether.

The electrical units are, *Ampere*, *Volt*, and *Ohm*, respectively, measuring the quantity C, pressure E, and the resistance

R, of a current. *Ampere* is the unit of current strength, measured by the deposition of metal from a solution (0.017253 grains of silver per second, or 0.005084 grains of copper). The unit of resistance is the *Ohm*, which equals the resistance of a column of pure mercury 1 square millimeter section and 1060 long. A *Volt* is the unit of electromotive force (usually written E. M. F.) and expresses the difference of potential, or of electric pressure. Its value is arbitrary, but fixed. One volt will force one ampere through one ohm of resistance. The energy, *P*, of a current, is measured by the product, *CE*, in Watts (the unit), 746 of which equal a horse-power. The loss of horse-power in a conductor equals *CE* divided by 746.

$$P = CE \qquad E = CR.$$

A Joule $= W$, is the work done or the heat generated by 1 Watt in a second.

$$W = QE = 0.7373 \text{ ft.-lbs.}$$

Manufacturers' tables furnish the data for wires of various sizes, by which their resistances may be known. For example, 1000 feet of No. 1 gauge copper-wire (0.2893 inches in diameter), offers a resistance of 0.1212 ohms and loses 9.5 volts with a given current of 77.7 amperes.

The conversion of electric into kinetic energy is accomplished by a motor directly connected with fixed or movable appliances (Fig. 27), which may be operated by rotary motion; for the reciprocating motion of pumps and percussion drills, it has signally failed. Rotary drills, fans, hoisters, and coal-cutters are in successful operation, with an efficiency of from 60 to 80 per cent of the energy received. Neither the generator nor the motor is a large or a complicated piece of machinery, being easily transported and run. It therefore admits of introduction within the prescribed limits of the stope or gallery. No power is consumed, and none transmitted to machines which are idle, and the power is always proportional to the work doing. The commercial efficiency of the motor is nearly the same, whether working at full capacity or not, and it quickly responds to recurrent demands upon it without excessive loss.

If the three-wire system is used, then all motors not requiring frequent handling should be connected to outside wires; drills, and the like, to the neutral wire. This plan lessens the pressure on the motors. For lighting, continuous or alternating currents may be used with equal efficiency; but for motors, I do not believe the alternating current can be advantageously used. When one recalls that the current which furnishes the power likewise gives brilliant illumination at the work, one must confess the superiority of the entire discovery.

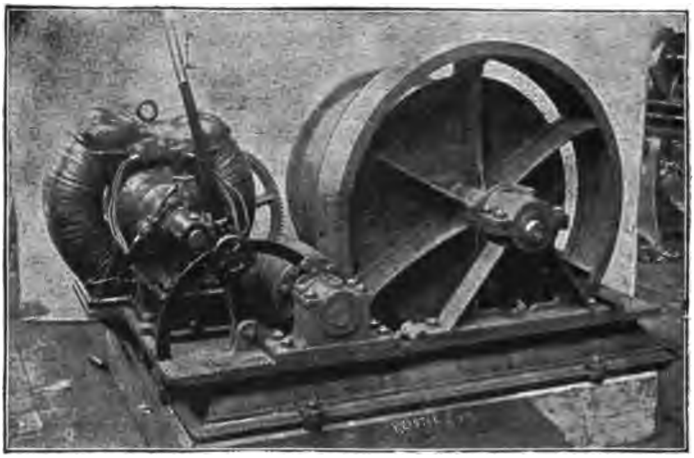


FIG. 27.

When the evolution of the storage battery has reached an efficient stage, an important adjunct to mine appliances will have been attained. As yet the storage battery is tentative.

For a proper knowledge of this subject, which is of too extensive a scope for introduction here, the reader is referred to Kapp's "Electrical Transmission of Power," and H. W. Leonard's articles in *Electrical Engineer*, Sept. 2, 1891, and in *Engineering News*, May 1891.

20. Water-power has long been employed for operations in the immediate vicinity of the wheel. No cheap and efficient means had been discovered for its transmission to great distances above the wheel, until the successful enchainment of electricity to man's use. The installation of electricity has opened up the possibilities of water-power to a marvellous degree, with but one disadvantage—the limitations of seasons in drought or cold. The gross power of water is the product of the weight discharged, by the height (h) of its fall. $W = 62.5Q$. The net power is from 40 to 90 per cent of this, according to the kind of wheel used, whether breast, overshot, turbine, or hurdy-gurdy. $Q =$ cubic feet, delivered per minute. Then horse-power $= 0.00161Qh$.

In the early days of the undershot and overshot wheels, enormous volumes of water were consumed by large slowly turning wheels, in developing small power. The Leffel and other forms of turbines were next in order. These are quite small, and, revolving at high speed, give a good duty with large volumes of water, under moderate heads up to 300 ft. They may be placed with an axis horizontal or vertical, the largest size being 48″, submerged at the bottom of a penstock, or encased in a globe, or cylindrical casing, connected to the bulkhead or piping, by which the water enters centrally and discharges circumferentially. The globe casing with a horizontal axis is the preferable form for mining purposes.

In our mountainous districts the numerous creeks are not large; but their fall, and hence their velocity, is great, and it is rare that water-power can not be found within a moderate distance of the mines. This, the "hurdy-gurdy" wheel has been designed to utilize; and most effectually is it done, giving, as it does, a guaranteed duty of 85 per cent (Fig. 28).

A wheel 18″ to 90″ diameter, the plane of which may be in any convenient position, carrying a number of small cup-shaped vanes, receives the impingement of one or more jets of water at high velocity, and tangentially. This principle is entirely at variance with the previous methods of generating power, and most nearly conforms to hydraulic laws. Its execution is simple, and a pronounced success, the entire absence of machinery leaves nothing to get out of repair. Placed at the lowest practicable point, to obtain all the head available, the high velocity of even a small volume of water delivered through the nozzles, will develop an enormous power. Though essentially a high-pressure machine, it is almost as efficient with a moderate head. Thirty feet is regarded as the inferior limit of head, while theoretically there is no maximum limit. It is actually in use with a head of 1,700 feet, and a measured velocity of

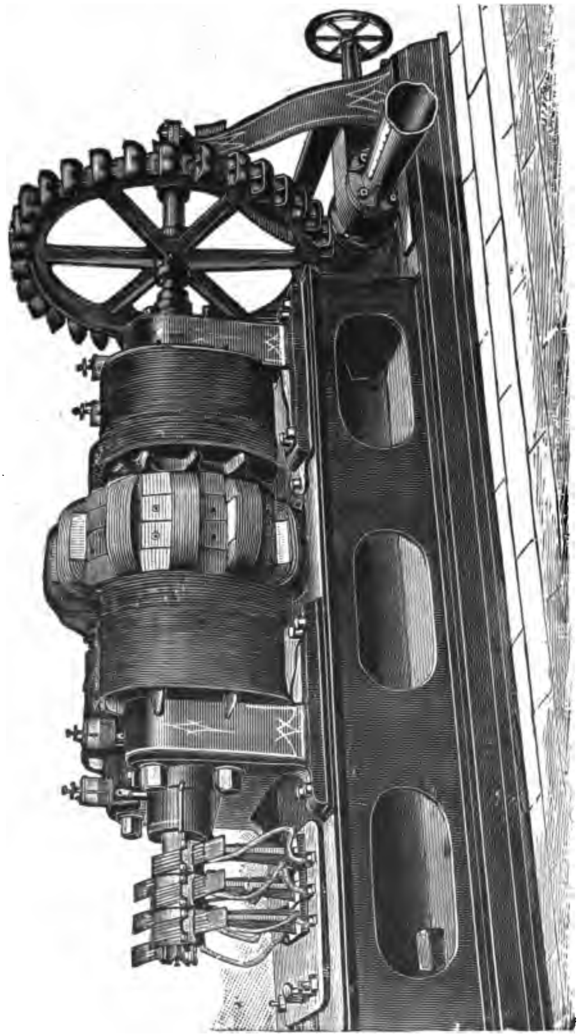


FIG. 28.

revolution of over 7000 feet per minute. The size of the wheel may be proportioned to the rate of revolution desired for the main shaft. A small wheel at high head has a very rapid revolution, and a pulley on the shaft may be directly connected with a dynamo, while, for slow motion of pumps, air-compressors, etc., a large wheel is desirable.

The Pelton and Knight patterns are of this type of tangential wheel. Several European forms are noticed on the market, notably the Girard; but a hasty glance at them suggests that their parts are not as accessible as the American models, which are durable, reliable, efficient, and easily accommodated to wide variations of power. By altering the tips,—this is easily done,—the varying conditions of the stream may be provided for; or, if the head should, from any cause, be reduced, the speed of the driven belt is proportionately diminished by lagging the pulleys. The high efficiency is maintained as long as the wheel velocity is one half that of the spouting discharge.

These wheels are capable of greater transposition than the turbines, are less liable to clog or become obstructed, and cost less to place. The regulation of the speed of the wheels is accomplished by governors, varying the flow of water. These are of different designs, some being planned to give uniformity to the revolution of the connected machine.

The Leffel receives its power from the pressure of a head of water in a ditched penstock, or piping. The tangential wheels take the impact of the water carried from the source through a continuous line of pipe, which should be as large as admissible; and this is specially urged where the amount or head is small. The pipes are dipped in tar or asphalt, and laid on the ground, or steep slopes, cabled to stumps. The slip-joint connection is better than the ring-joint (Fig. 62).

CHAPTER VI.

HOISTING OPERATIONS.

21. Hoisting-derricks, construction of; essentials for strength and safety; overwinding, and the devices for preventing the same; indicators, and the modes of communication with the mine. 22. Calculation of the strains in hoisting-frames; constructions in iron and wood; sheaves and their importance. 23. Calculation of the hoisting-capacity of a mine or shaft; hoisting-velocities under different conditions of timbering; loading and unloading conveniences; formulæ and examples; work of the engine in hoisting; definitions of horse-power, indicated, theoretical, and calculated; formulæ; examples.

21. THE most important surface feature is the frame, " head gear," or " derrick," which affords the skilful constructor excellent opportunities to satisfy the two necessary conditions, height and strength; the first for security against overwinding, the second is fundamental.

It is obviously essential that the sheaves on the frame should be placed at considerable height above the ground, to allow sufficient margin within which the engineer may stop the hoist. With the present high speed and large drums, the allowance should not be less than the length of one drum coil of rope, for in a moment's hesitation, or error in the interpretation of the signals, carelessness in signalling, or a derangement of any appliance, the tub or cage may be dashed through the roof before the engine could be stopped. Ordinarily, a brakeman at the mouth of the shaft, having charge of the delivery and receipt of the cars to the mill or dump, may, as the cage approaches the top, signal to the engineer, or the latter may have to depend upon his own watchfulness. So, devices for preventing overwinding are more or less adopted. But, while desirable, they are not satisfactory. The number of casualties are reduced by their use, but they are not wholly prevented. The principle consists in automatically detaching (Figs. 30, 31, and 32) (ice-hooks open and free the cage) or in cutting (a pair of shears cut the rope) the cage from the run-

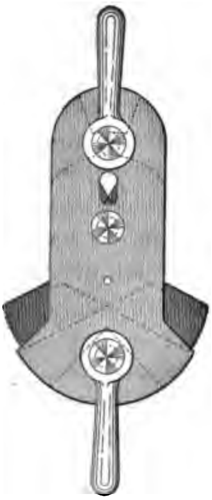


FIG. 29

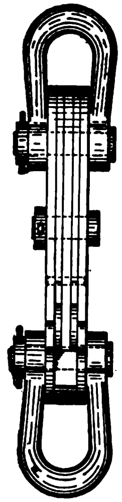


FIG. 30.

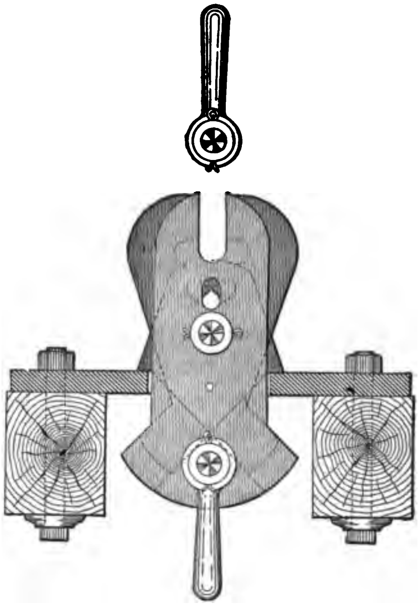


FIG. 31.

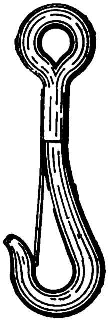


FIG. 32.

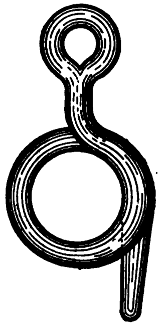


FIG. 33.

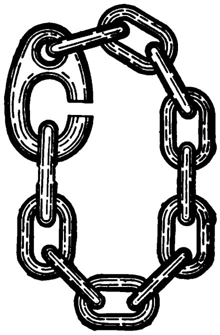


FIG. 34.

away rope, when it has reached a dangerous height, and simultaneously throwing out landing-dogs, to catch the falling cage or tub. Or, another plan is to have the guides incline slightly inward. Then the cage, in its ascent, gradually wedges tighter and tighter, and this acts as a brake to the engine. Again, self-acting steam brakes on the engine are constructed so as to operate when the cage reaches a certain point in the shaft. Eternal vigilance is the price of safety, and the only safeguards, after all, are a competent and sober engineer, with machinery in order, a good indicator, and an unobstructed view of the shaft. If the last-named is not possible, a cool, competent brakeman at the platform is indispensable. One device suggested appears to be eminently worthy of introduction,—a 10-ton weight hanging, like Damocles' sword, by a thread, over the engineer in charge, to be dropped when the overwinding has reached the fatal limit.

The same may be said of other forms of safety appliances, even those required by law. They may remedy evils aimed at, but introduce others. First, too great a feeling of security is induced, and negligence results. Second, when the emergency arises, they are rusted or out of order and fail.

The position of the cages in a shaft or slope is ascertained by indicators. An index, operated from the drum shaft by gearing, rope, or worm and screw-wheel, moves around a dial or along a miniature representation of the shaft, at a speed commensurate with that of the cage or tub (Fig. 17). A cylinder has 15 or 20 turns of a spiral thread cut on its face; a pointer moves vertically in the thread as the cylinder revolves, accurately indicating the position of the cage (Fig. 19). The more trustworthy ones are so geared that the index moves faster as the cage approaches a landing stage. A glance suffices to inform the engineer, who need not fear overwinding if brake and throttle are in order. White marks or rags tied on the rope are useless, as also the attempts to make the cage automatically signal its warning to the engine-room.

The safest and most natural means which suggests itself for communication between the engineer and the miner is the voice, with or without the intervention of the speaking-tube. The telephone or annunciator is more convenient. Then misinterpretations can be excused only by sudden death or criminality. The clumsiest and most unreliable signaling arrangement is the gong-bell or triangle, which is struck by a weighted lever, operated from below by a rope or wire. Its simplicity commends it, while its crudeness condemns it. Mistakes do so readily occur. A stroke of the bell may be lost by too light a pull; or an engineer, anticipating two bells to lower, may not await the completion of the signal, and lower before he has heard the third bell, meaning, perhaps, "hoist with a man on." If a simple uniform code of signals could be agreed upon and adopted by mining men, a great advance would be made. A man formerly accustomed to hoist for one bell, will do considerable damage in his new job, where one bell means to lower.

22. The strains to which a derrick is subject are those arising from the weight of the conveyance and its contents, the rope and friction operating vertically, and the pull of the engine on the inclined rope, which is greater than the weight, by the amount of frictional resistances at the sheave. This latter may be taken as 4 per cent. of the weight. These combine to produce a resultant operating in a direction nearly bisecting the angle between the two ropes, and equal in intensity to about twice the inclined strain, multiplied by the cosine of half the angle between the ropes. A single stick of sufficient size may be placed in this direction, and used as a gin-pole for hoisting; but it is not stable, and, instead, should be a frame, the base of which shall embrace within its parallelogram the line of the resultant. The more nearly central this line falls, the more stable the structure, but the brace then becomes long, and its section large. Still the engineer will prefer an increased stability to a slight saving in material, and hence give an angle of about 60° between the ropes. The patterns for tying the parts of the frame, and reducing the size of the brace, are numerous.

The form of the frame is essentially two right-angled triangles (Fig. 16), the brace and upright being nearly parallel to the rope. They are set into cast-iron shoes, bolted to the sill; sometimes the posts are tenoned or dovetailed into it. The top frame is slightly narrower than the base, which consists of a sill on each side, connected by three cross-sills mortised and dovetailed to them, the whole bolted and anchored to heavy "dead men" buried in the ground. Their risk of fire, the exposure to weather, the working of the joints, and the difficulty of securing sound, long, large sticks, render the adoption of other material than wood highly necessary. Wrought-iron is much used. Height is the essential feature of derricks, but this, with stability, is difficult of attainment without a rigid frame, perfectly made. This can better be secured by the use of Phœnix or Kellogg columns, set in cast-iron shoes bolted to heavy masonary pillars.

Greater stability can be secured for the sheave, by building, of wood or iron, a framework of four vertical posts at the four corners of the hoisting compartments, suitably braced and tied with struts from the top, inclined slightly less than the resultant force, the lower ends being stepped into masonry pillars, or joined to a substantial base frame. Fig. 36 is an example.

Upon the top of the uprights is mortised a frame supporting the sheave with its axle horizontal, and its unsupported length as short as possible. The diameter of the sheave should be 100 times that of the wire rope (48 at least), owing to the rigidity of the rope, which resists bending. To minimize this resistance, the wires of the rope are as fine, and the angle of the bend as small, as obtainable.

This is more imperative as the speed of hoist increases, and not uncommonly sheaves are seen 12 feet in diameter. All pulleys over which the rope bends more than 30° become, to all intents, sheaves. The hubs are double, connected to the cast-iron rim by wrought-iron rods let into sockets, and they should be as light as practicable at the rim, because, by reason of the impetus they acquire, they continue to run long after the hoisting has ceased. Often the sheaves are cast in sectors, afterwards bolted together. The grooves are lined with wood, on end if possible, and tarred hemp to prevent slip; for the transmission of power, with rubber.

Never house the derrick, especially about collieries, for, in the event of a fire it becomes a draught-chimney (Fig. 36).

FIG. 36.

23. As the mining engineer may find himself compelled to calculate the plant required, we will briefly consider the conditions and the process. The output of the mine depends upon the time occupied in each hoisting-trip and the load carried, assuming that the conveniences for delivering to the shaft at the bottom, and the facilities for the disposal of the ore and its carrier, are equal to, if not greater than, the hoisting capacity. The speed of hoist is limited by the equipments of the shaft, which must be timbered very substantially to permit rapid hoisting. Cages are being hoisted in vertical shafts, at rates up to 2500 feet per minute; skips and slope-carriages at 1,000 feet; and buckets at not over 300 feet per minute. The time allowed per trip must also include the arrangements for loading and unloading. The time lost in filling an attached bucket at the bottom, and dumping it at the top, is from three to five minutes; if the empty buckets are immediately replaced by full ones, much less at both ends. A car can be run on and taken off a cage or slope carriage in twenty-five seconds. A skip occupies from two to three minutes to side-track, unload, and return. The influence of this loss of time can readily be calculated.

Let t = the minutes to load and unload;
D = the depth of the shaft in feet;
v = velocity of the hoist per minute;
v' = " " lowering " " ;
n = number of trips per hour;
T = minutes per round trip;
Q = output tons per hour;
q = load tons per trip;

$$T = \frac{D}{v} + \frac{D}{v'} + 2t;$$

$$n = \frac{60\,vv'}{Dv' + Dv + 2\,tvv'};$$

$$q\,n = Q.$$

Various transformations may now be made according to the known conditions. Usually q is given, and it is desired to ascertain Q.

Thus, in a poorly timbered shaft, if only one bucket be run up and down without detaching, the output from a 300-feet shaft is about 3.6 tons per hour. With 3 buckets in constant use, each holding 600 pounds, the hourly product cannot exceed 6 tons. With excellent timbering, double the speed may be permitted, in which event 3 buckets will deliver 7.2 tons per hour at the surface.

So it is evident what a large proportion of the time may be lost at the landings, when even the doubling of the speed only increases the quantity one fifth. Skips loaded from shutes are almost as wasteful of time. Hence, for large mines, cage and cars are resorted to. Then, from the 3000-feet level, 36 tons may be hoisted per hour, by three cars in constant use, holding each 3,000 pounds, and assuming a not uncommon rate of 1,800 feet per minute.

The size of the engine must necessarily depend upon the velocity of the hoist, the load, the dead-weight of the rope, cage, car, etc., and the various resistances. Unless there is a counterpoise (see p. 79), the maximum work of the engine is at starting, when the inertia of the load, M, is to be overcome.

R = the weight of rope per foot;
B = weight of bucket, car, cage, etc.
Then $M = 2000q + RD + B$.

The resistance of friction, etc., is about 12 per cent with cage; 4 per cent with buckets; and 20 per cent with skip. Therefore the value of M is greater than that given by 4, 12, or 20 per cent, as seen; and equals

$$1.04,\ 1.12 \text{ or } 1.20 \text{ times } (B + RD + 2{,}000q).$$

Though the load, M, is not operating throughout the hoist, it is necessary to have sufficient power to start as quickly as possible, without jar. Moreover, a force nearly twice M is required to overcome the inertia of the load. So the allowance is made as indicated, though it is indeed too small for the initial stages of the hoist, and too great during the final.

The work done is always a product of the resistance M, (the strain on the rope) and the velocity per minute, the horse-power, H, being found by dividing by 33,000.

Thus we require theoretically 754 horse power to hourly raise 36 tons from the 3,000-feet level by a cage and car weighing 2,300 pounds and rope. With a tapering rope, 150 horse-power may be saved. With double cylinder drum, two cages and cars, the same product may be raised at 1,300 feet per minute, and only 430 horse-power needed.

The term horse-power of an engine has a three-fold interpretation:

1. The *indicated* horse-power, wherein the actual work done is measured by an indicator, the friction by a dynamometer.

2. The *theoretical*, which is the product of the boiler pressure, the area of the cylinders and the piston speed per minute;

the losses from friction of valves, imperfect delivery of steam, or friction of parts, are not considered.

3. The *calculated* horse-power is a certain fractional part of the theoretical, found by multiplying by a modulus which allows for the losses mentioned and the intermittence in running.

In ascertaining the size of the cylinders, we have:

s = length of the stroke, in inches;
k = the diameter of the cylinders;
n = revolutions per minute, 30 to 60;
p = steam pressure (about 80 pounds);
c = the number of cylinders;
f = the factor to cover emergency, etc.

$$H = \frac{cpsnk^2}{252{,}000f}.$$

Equating, we have three unknown quantities, of which one may be determined when the other two are given. Thus, if $s = 48$, $n = 40$, $p = 60$, and $c = 2$, the engine of the last example becomes one of a double cylinder 28×48, not allowing for clearances, etc.

Required, the size of an engine to hoist 1,000 tons per 10 hours' shift, from a shaft 1,200 feet deep, the cage load being 4 tons, the mean effective pressure, 50 pounds, $n = 40$, allowing for 20 per cent. excess.

$Q = 100$. $t = 0.4$. $D = 1{,}200$. Let $v = 1{,}500$, then $T = 1.2$, $n = 50$ trips, and $q = 2$ tons. $M = 8{,}000$ plus the rope, $= 11{,}600$, which with friction $= 13{,}000$ pounds. Finally $H = 592$, and, allowing for 20 per cent. excess, 710 horse-power. If the stroke is 5 feet, then the diameter of the cylinder is 27". As 1,500 feet of rope is to be coiled on the drum each minute, with 40 revolutions, the drum must be 12 feet in diameter.

If it be required to find the depth of shaft to which a single 16×30 cylinder with 20 per cent. allowance will do service, and from which the output is to be 60 tons per shift, the order of procedure is about the same. Let the maximum steam-pressure be 70 pounds, and piston speed be 300 feet per minute. Then $H = 107$, $Q = 6$. Equating, we find $v = 438$ feet, and for $t = 0.5$, that $D = 3{,}963$; while if $t = 2$ minutes, $D = 3{,}504$. Should the mean pressure be 40 pounds, then the maximum velocity is 250 feet, $D = 2{,}375$ and 2,000 respectively. For an output of 300 tons daily, and $q = 1.5$ tons, then $v = 284$, and the limiting depths become 960 and 384 respectively.

It may be necessary to state here that the formulæ given assume that the maximum load is already in motion and is to be maintained; to obtain the load which the engine *will start*, the principle of moments expresses the relation:

$3.141\, cfk^2pm = rM +$ friction—for a direct-connection drum, whose diameter is r'' and crank arm m''. For a second-motion engine, we have $3.141\, cfk^2pmy = Mrx +$ friction, in which y and x, respectively, are the number of teeth in drum and pinion wheels.

The friction is taken as a certain per cent. of the total work done (Mrx), or it may be calculated, knowing the sizes of journals, stiffness of rope, etc.

The data here given serve as a guide by which to shape the engineer's opinion as to capacity required, and the approximate depth to which the shaft may be sunk without calling for additional machinery. Expansion engines and counterpoises cannot be here considered. Reference may be had to books listed in the Appendix.

In slopes the resistance equals the component of the weight parallel to the incline, plus friction due to the normal pressure of the weight.

CHAPTER VII.

HOISTING-CONVEYANCES.

24. Kibbles and buckets, their sizes, etc.; objections to buckets in hoisting; guides, etc., for rapid hoisting; skips and gunboats for slopes; automatic dumps and brakes. 25. Slope-carriages compared with skips; cages for vertical and inclined shafts; single- and double-deckers; safety appliances and clutches discussed; landing-doors, dogs, etc., for cages; ropes of hemp, iron and steel wire, round and flat; locked wire ropes; tapering ropes for equalizing the work of the engine. 27. The life of a rope, its care and preservation; splicing and testing; cost of ropes.

24. THE ore is conveyed from the workings to the shaft in kibbles, buckets, or tubs, on small platform cars, or in small box cars. Tubs are of stout barrels, heavily coopered, and supplied with a bale of ¾-inch round iron, hooked into eyes on a strap which is bolted on both sides and under the bottom. A liberal supply of these, equal to the capacity of the hoister, is required for constant use and for emergency. A snap-hook, fastened on the hoisting-rope, catches into a ring of the bale for hoisting (Fig. 32). If, however, the tubs are frequently detached from the rope, serpentine hooks are used instead (Fig. 33). A length of ¾-inch round Norway iron is turned, at one end, into a small ring (in which the rope is to be fastened), the rod is softened and bent to make one spiral turn of 3″ diameter, leaving the end free and open.

Buckets are used where the developments are not of an extent to warrant more elaborate arrangement, and also during the sinking of the shaft. Two features are observed in the designing of kibbles: lightness, to keep the size and weight within limit of easy handling by one man, and to reduce the dead-weight in hoisting; and capacity. They are plain or bellied

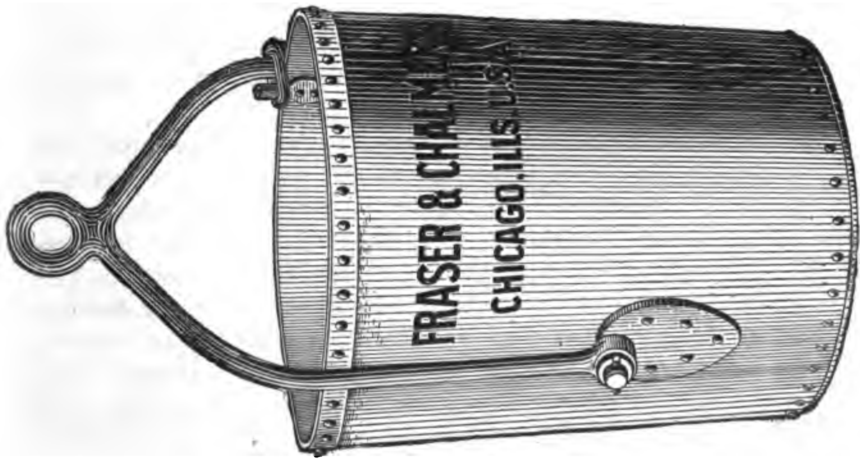

FRASER & CHALMERS
CHICAGO, ILLS. U.S.A.

Fig. 37.


FRASER & CHALMERS
CHICAGO, ILLS. U.S.A.

Fig. 38.

cylinders of boiler iron, 18″ to 33″ diameter, 30″ to 54″ deep, and weigh from 150 to 900 pounds. They carry from 600 to 3,000 pounds of mineral, and are hung on the rope by an easily-disengaged hook. If wire rope is employed, a short length of heavy chain intervenes between the hook and the rope socket; without this flexible attachment, the wires break at the socket in a very short while. Where a large number of buckets are in use, the empty bucket is replaced by a loaded one, which, on reaching the surface, is immediately unhooked and dumped while another is being lowered. The expense for such an outfit would be high for a mine having several stoping gangs, in which case a very large bucket is permanently attached to the rope, being filled from cars or a shute below, and emptied above by various means. The bucket (Fig. 38) may have a becket and eye underneath, by which the surface man swings it to and fro till he can catch into the eye a short length of stationary chain. The tub, with its hoisting-rope, is lowered until it is upside down; then it spills its contents into a car or over a grizzly. Another device (Fig. 37), not so safe, consists in having the bale pivoted a little below the centre of gravity of the bucket, which is held in position by a loose ring on the bale, slipping over a stout pin at the upper rim of the bucket. To dump, the brakesman merely slides the ring up a little, and the bucket turns over automatically. It is easily righted and fastened again.

Hoisting by buckets is slow and insecure, besides producing and losing much fine material with each handling. This is a loss in coal and even in ore, for, if the fine stuff reaches the surface, it never arrives at the smelter; much value is therefore lost, for the silver minerals are usually the softer. The speed of the hoist can never be large, because of the liability of collisions, unless each tub compartment of the shaft is smooth-lined. Even then the lining must be watched. The end of a plank which has become loose will project and often trip the tub, resulting in great damage and frightful accidents. In an incline the tub slides between a pair of "skids,"—planks laid on the floor.

In the endeavor to employ buckets for quicker hoisting, and to prevent collisions, rails or wire rope were laid up the shaft. The bucket was suspended from a yoke that slides up and down the guides. Iron rails were abandoned because too "clattery." Wire rope kept taut by screws is feasible, but it has no advantage over wooden guides, besides, as will be noted in previous lecture, the better line of improvement is in the conveniences for loading and unloading. No simple safety appliances are applicable to tubs. The wooden guides are of 4×6 scantling, spiked end to end on the shaft timbers. For cages they must be narrower, and trimmed at the landing-stages so that the safety clutches do not take hold.

For hoisting in inclined shafts, skips, or gunboats, are commonly seen. A strong iron box weighing 900 to 1500 pounds is set on four wheels, held by bosses riveted to the sides. Its cross-section is rectangular, but its side view is a trapezoid (Fig. 39). The inclined end is uppermost, with automatic dumpers, while in those discharging by door it is below. The hoist-rope is attached to the bale which rotates on a pin passing through the side of the skip, often back of the centre of gravity, so as to dump automatically. The charge of one or two car-loads is shovelled or shuted into the skip, which empties at the surface into a bin or on grizzly. In one variety the contents are discharged by the mouth at which it is loaded, while the other form has a swinging door opening at the upper side. A vertical safety skip is shown in Figs 40 and 41.

The automatic dump is simple, the rear or lower wheels are of wider face than the fore wheels. As the dumping plat is reached, the guide-rails on which the wheels have been travelling, gradually bend to horizontality, and these the front wheels follow. As the hoisting continues, the wider rear wheels catch and roll on a pair of outer guides, and continue up the slope. By this means the lower end is elevated and the skip emptied.

The brake is generally a drag, consisting of a bar about 4 feet long, trailing on the floor, and only catching if the skip breaks away on its up trip. Often the wheels are confined

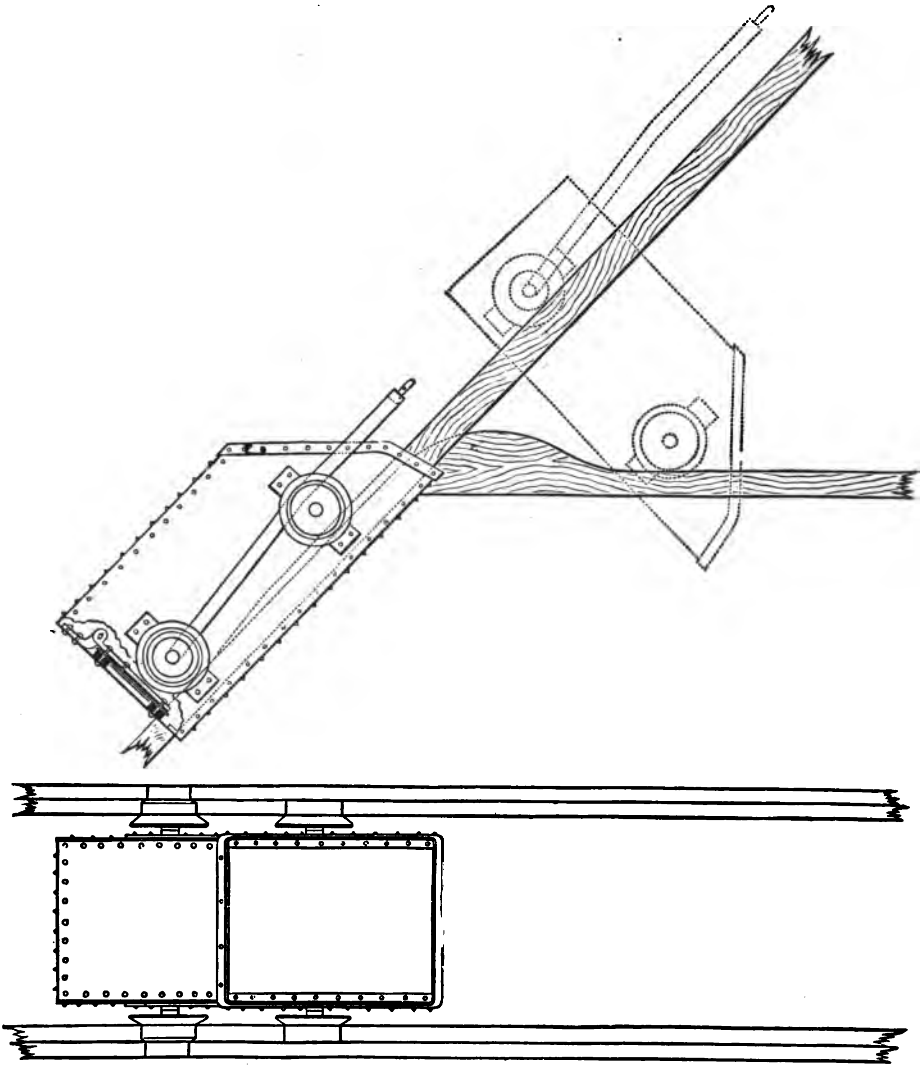


FIG. 39.

between two guides on each side as, for instance, where the direction of the slope changes. The sole objection to these skips is the double and treble handling involved. The car from the mill-hole empties into the chute, whence the skip is loaded, and at the surface the reverse operation takes place.

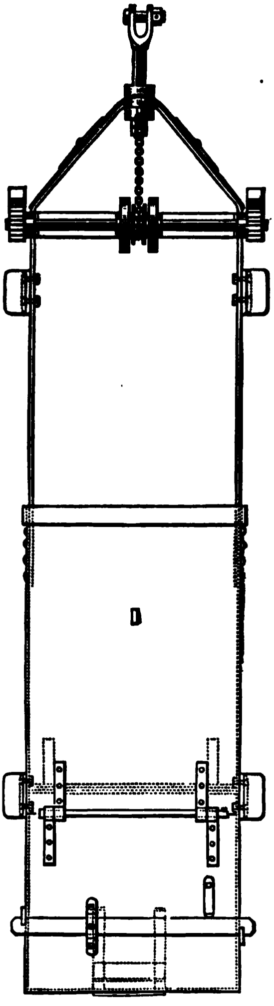


Fig. 40.

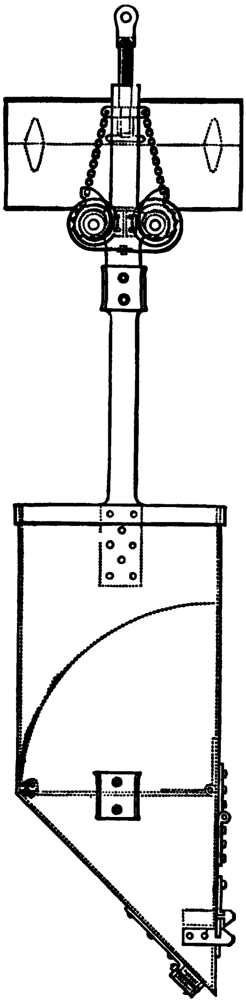


Fig. 41.

25. A slope-carriage dispenses with two handlings, taking the car at once to the surface. This is simply a double triangular frame, large enough to accommodate a car with two rails on its horizontal top, and two wheels on each hypothenuse. A hook or lock holds the car while riding. For convenience, the loading and unloading gangways are not on the same level, the track for "empties" being 6 feet higher than

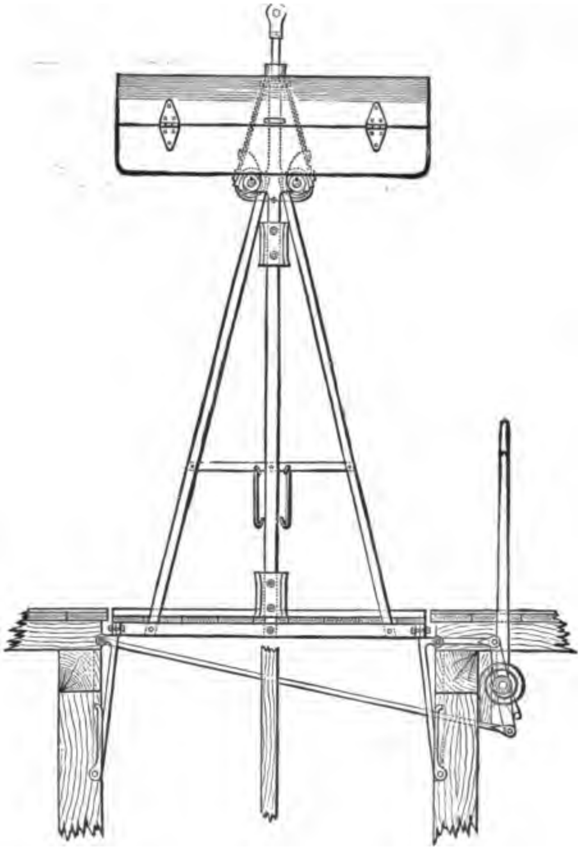


FIG. 42.

the "full" track. If the seam is too thin to allow of this, each track has a curved roadway connection with the gangway.

The head-room required for the slope, having this means of haulage, implies expensive timbering and large area, particularly as double trackway is necessary for a considerable output. There is not much necessity for a carriage, except in slopes over 30°.

The cage is now almost exclusively used for raising large output from great depths. It is a simple elevator platform, accommodating one or more cars, held on the sides next to the guides by two stout iron frames, united at the top by a cross-bar, to which the hoist-rope is attached (Figs. 42 and 43). Two iron ears at the top and bottom on each side confine the cage to the guides, and, generally, an iron roof or "bonnet" over the cross-bar shields the passengers from falling rocks. The size of the car governs that of the cage, which just fits the compartment. A latch-lock holds the car in place on its journey. Cages are preferably of iron, because the required strength is obtained with less increased dead-weight, though they are not so easy to repair as wood, and accidents arising from "jambing" in the shaft are frequent.

This is the safer, quicker means of hoisting, consumes less time in loading and unloading, and involves less handling of mineral, but demands a well-timbered shaft, otherwise the swelling of walls or bulging of timbers interrupt a steady hoist. A small plain cage

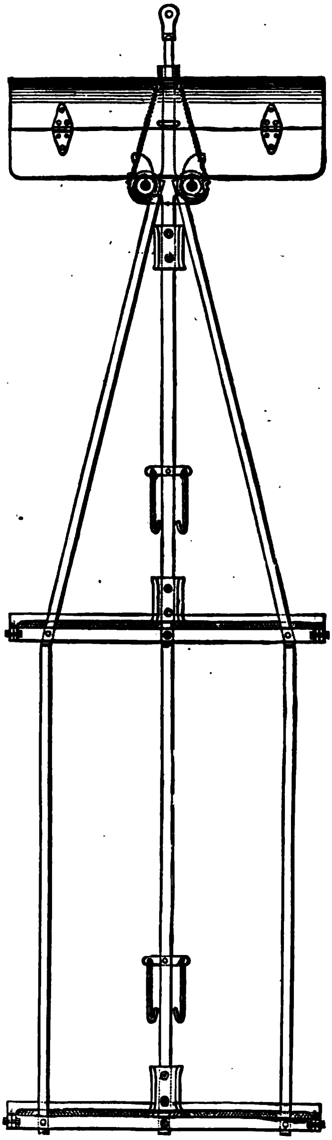


FIG. 43.

costs about $150; the Nevada safety pattern, $400. The self-dumpers are cumbersome, complicated, and not a success.

Cages are built with one, two (Fig. 43), or three stories. Single-deckers are almost exclusively used in America, and are sufficient except for narrow, deep shafts, when heavier loads are necessary for large output. The landing-stages are then arranged in the same number of tiers, from and upon which cars are simultaneously run, without moving the cage. With ample facilities for "decking" the cars, the saving in the trip-time, per car, improves the capacity of the plant. An objection prevails against multiple-deck cages, in the necessarily complicated underground stations. The single-platform two-car cage is relatively heavier than the single- or double-decker.

For inclines of uniform slope, as well as for vertical shafts, cages may be had, the platform being hung on an adjustable lever; but the carriage is better.

About fifty years back, various contrivances began to be proposed for guarding against accidents resulting from the breaking of the rope. The sudden starting and stopping of the hoist shock and strain the cable more severely than does the mere weight. This inevitably shortens its life. Without careful hoisting and a heavily timbered shaft the rope cannot be insured against the jars which soon rupture it. To avoid the accident sure to follow, some form of safety appliance must be thrown into action, and arrest the fall of the cage or skip. The safety catches differ in design and efficiency, but depend generally upon a spring, so held between the rope and cage as to be compressed, while the weight of the cage strains the rope, but acts on a clutch that grasps the guides and stops the fall, if relaxed by rupture of the rope or otherwise. The clutch is either a pair of sharp-pointed steel levers, which are thrust outward into the timbers, or a serrated cam, the wider part of which will be turned against the guides, and clutch them on either side (Figs. 40 to 43). The heavier the weight the better the bite, after they once take hold. Many a life has been saved by them, but in many instances also they have failed to operate. A momentary check, or any sudden change in the speed often unnecessarily throws them into action. On the other hand, they are rarely, if ever, in order when the emergency arises, or the guides are so wet and dirty that the clutches fail to catch, if the momentum of the falling cage is great; besides, they are costly and troublesome. Though useful adjuncts, which the law requires, yet it is not surprising that the distrust of them is strong; they breed other and more serious causes of alarm. The tendency following their use is a lack of attention to the condition of the rope and an undue extension of its working life. Fortunately, the rope more often breaks at the moment of starting at the bottom than at any other time, and the point of rupture is where the rope enters the socket. If, however, the rope snaps as it turns the sheave, and this is of common occurrence, there is nothing to prevent the inevitable and frightful calamity that follows—the entire length of the rope falls and crushes cage and contents. A simple appliance might be introduced at the top. After all, the employment of the best materials and men, the careful supervision and repairing of the plant, are the only safeguards.

At the several levels in metalliferous shafts the landing of the buckets is effected on a hinged door, of double 2″ planks, lined on top with iron. This is swung against the far side of the compartment, closes it completely, and standing as it does at 45°, the bucket slides into the drift. When not in use, it is hooked upright, closing the mouth of the drift, and leaving the hoistway clear. That miners may go from one drift to the opposite, an escapement roadway is provided around the shaft in the rock. At the mouth of the shaft are two heavy doors, opened for or by the rising bucket, and closed at all other times. These are convenient, and prevent accidents arising from stones dropping down the shaft, especially while dumping the bucket. A similar arrangement in slopes—a movable drawbridge, lowered at pleasure—receives the car for attachment to, or after detachment from, the rope, when up, it closes the gangway.

At the surface the cage is held by dogs or chairs (Fig. 42).

These are merely steel levers, projecting slightly over the shaft, which are lifted out of place as the cage rises, but immediately return to position to support it while in use. When placed at the various stations down the shaft, they are manipulated by lever, and thrown into the shaft only as desired. They are simple and convenient, if not indispensable..

26. Ropes used for hoisting are of hemp, aloe fibre, iron or steel wire. The essentials are flexibility and strength, combined with a small weight. Hempen ropes have until recently been entirely used for hoisting, but great caution is necessary in the selection. The practice is, and has been, to work up into new rope fibres and strands of worn-out ropes. This is a serious mistake, for long service cannot be expected of short fibre. Another objection is the increase of weight, while working, from the absorption of moisture. Tallow is its only preservative. Again, the size of the rope for big work is so large, that several round ropes are sewed together into an unwieldy mass, to obtain a sufficiently strong cable. The coiling and uncoiling of one so thick entails great wear, and is expensive.

So wire rope was substituted. It is as pliable as hemp, is much stronger for the same weight and same money, and now has survived the early objections to its introduction, i.e., the great damage predicted from its use in shafts not properly timbered, and that it would not give as early warning of its breakage as does hemp.

The usual number of wires to the strand is 19, twisted about a centre of hemp, which gives a better wearing, a more flexible rope, and one less liable to snap than those of 7 wires, which are used only for guy-ropes, etc. Several twisted strands compose a rope. The twisting gives elasticity to it, and a short twist is advisable for those liable to shock. If the twisting has been conscientiously done, the individual wires of the rope will be equally strained while in use. Its tensile strength, equal to a sum of the strengths of the individual wires, is greater than that of the rod of equal cross-section and material, the extra manipulation of drawing the wires having increased their strength.

Locked wire ropes are giving good results a great flexibility is imparted to them without in the least impairing their ability to withstand friction.

Steel has a great advantage over iron by reason of its lightness and high elasticity ; besides, it has a slower wear, but it corrodes faster than iron in wet shafts, and if tempered highly breaks quickly. Plough-steel, the highest grade of steel made, is the strongest ; but soft steel is more flexible.

The constant winding and unwinding of the rope over sheave and on drum diminishes its strength more than mere wear of friction or rupture of fibres : it is a loss of molecular elasticity, and indicates the importance of slight bends, and some form of flexible elastic connection between the socket and the cage or tub. The larger the wheel the greater the durability of the rope. The ratio of their diameters should be as 100 to 1, the minimum drum 48 times that of the rope. A 7-wire rope requires a larger drum than a 19-wire rope of the same diameter. A 1″ rope, at 140 lbs. to the 100 feet, good for a working load of 9000 lbs., requires at least a 4-foot drum. It takes a 3″ hemp rope to give an equal strength. The weight of a wire rope in pounds per foot is ascertained by multiplying the square of its diameter in inches by 1.48.

As the depth of the shafts increased, it was found that the use of the round rope was attended with many difficulties. Its size and weight increased enormously, the inequality of the work of the engine became more marked, and its drum cumbersome. To obviate these troubles, and the whirling of the bucket as it travels in the shaft, flat ropes and reels were introduced. They are formed by placing side by side and uniting by wire several round wire ropes. The adjoining strands have their wires wound in contrary directions, to counteract any tendency to untwist. The running weight of flat ropes is greater than that of round ropes of like material and equal strength.

There is a limit beyond which a rope of uniform section cannot safely carry its own weight. It will break with 12,000 feet, exclusive of any extraneous load. The additional weight of cage, car, and mineral (say 5000 lbs.), and the energy exerted in starting it, reduce this limit materially: the safe weight is from ¼ to ⅛ of this. By the use of the tapering rope there is no limit within mining possibilities. The lower portion is of an area sufficient for lifting the cage and its contents above it; the rope is graded in size, proportionate to the increasing strain of its own pendent weight. The taper helps to equalize the work of the engine. Practically, a tapered round rope is preferred to a flat, which on occasion slips off the top coils and becomes wedged between the reel-side and the lower coils.

The taper is not well adapted to long slopes, because the lowest portion is subjected to the greatest wear, since it passes over more pulleys than any other portion of the rope. This is the reverse of the situation in a shaft, where the thickest part of the rope has the most wear.

The cables are preserved by smearing with dope, or hot coal-tar. A bushel of lime is added to each barrel of tar, to neutralize the acid and prevent corrosion. Hoist ropes are never galvanized; the zinc soon rubs off, and then electric action is set up, rendering it worse than useless.

27. The life of a rope will depend upon the good condition and adjustment of the wheels and drum, and the care taken against corrosion and undue friction. The coiling and uncoiling of wire rope in the same manner as hemp injures its flexibility and displaces the strands. It is safe for only eighteen months of continuous service. The ropes of tramways do service for a longer period where precautionary measures are not so urgent as with vertical lines. The flat wears out more rapidly than the round, because of the winding on itself in reels. Indeed it is this rapid wear which counterbalances the only advantage that flat ropes have—that of equalizing the engine work,—that militates against its more extensive use. Where a spiral groove is not turned on the drum for the reception of the round rope, or where the drum is near the sheave, the rope is rapidly destroyed by chafing. The most dangerous element of destruction is vibration set up in the wire by jars, caused by careless starting and stopping, or by jerks from coiling on a drum too small. The paying out of an excess of rope is injurious, because while hauling in the slack the engine gets up speed, which jars the rope when taut. A short chain between the rope-socket and cage forms an elastic connection, and partially corrects this (Fig. 44). As the links wear rapidly they should have frequent examination. Every few weeks they should be annealed by heating to a red and cooling in the air.

Rapid hoisting tends to shorten its life, since the strain imparted to the rope is proportional to the square of the velocity. For a given output, it is better to decrease the velocity and increase the load. This is not always possible; for the live load is fixed by the size of the cars, and that by the hoisting-compartment size, which cannot be altered. The remedy is a large shaft at the outset (see II, 61).

The rope is inspected weekly by feeling for broken wires as it passes through the hands while hoisting. If any short length of it has numerous ruptures, the defective part should be removed. Along a long rope many of the wires may be frayed and broken without condemning it.

Fig. 44.

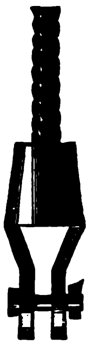


Fig. 45.

There are two types of sockets for round ropes—the conical and the double-pin, the former being stronger. The conical socket (Fig. 45) is slipped on to the rope, the wires are untwisted, hemp centres cut out, the wires bent back and forth into a tangled mat to fill, as nearly as possible, the conical socket, which is then slipped into place. This is slightly heated, and soft lead poured in to solidify the mass. The socket and rope are surrounded with wet clay to prevent heating of the wires beyond. The double-pin is treated in the same manner, but its connection with the chain is by a pair of pins through the links, instead of a ring for hooking, as in the former case. A "goose-neck" socket consists of a pair of trough-shaped tongs, bent to a loop, and riveted to the rope by three or four rivets, driven cold. Flat ropes have riveted to them shackles with eyes, which receive the first link of the chain. Six inches of the end are untwisted and doubled back, bound with wire, the shackles slipped on, riveted through the rope, and the hoops finally slipped on and driven tight.

In securing the rope on the drum it is only necessary to continue several extra coils of the rope, insert the end through the wooden lagging, and fasten it on the hub or shaft; or, instead, the end may be bolted to the arm of the casting. If the fastening have but a 10-lb. grip on the rope, it will resist a weight of 90 lbs. if there is only one coil around the drum; if there are two extra coils 800 lbs. will not budge the 10-lb. grip; with three extra coils it requires 7300 lbs.; while with four it has a 65,000-lbs. resistance.

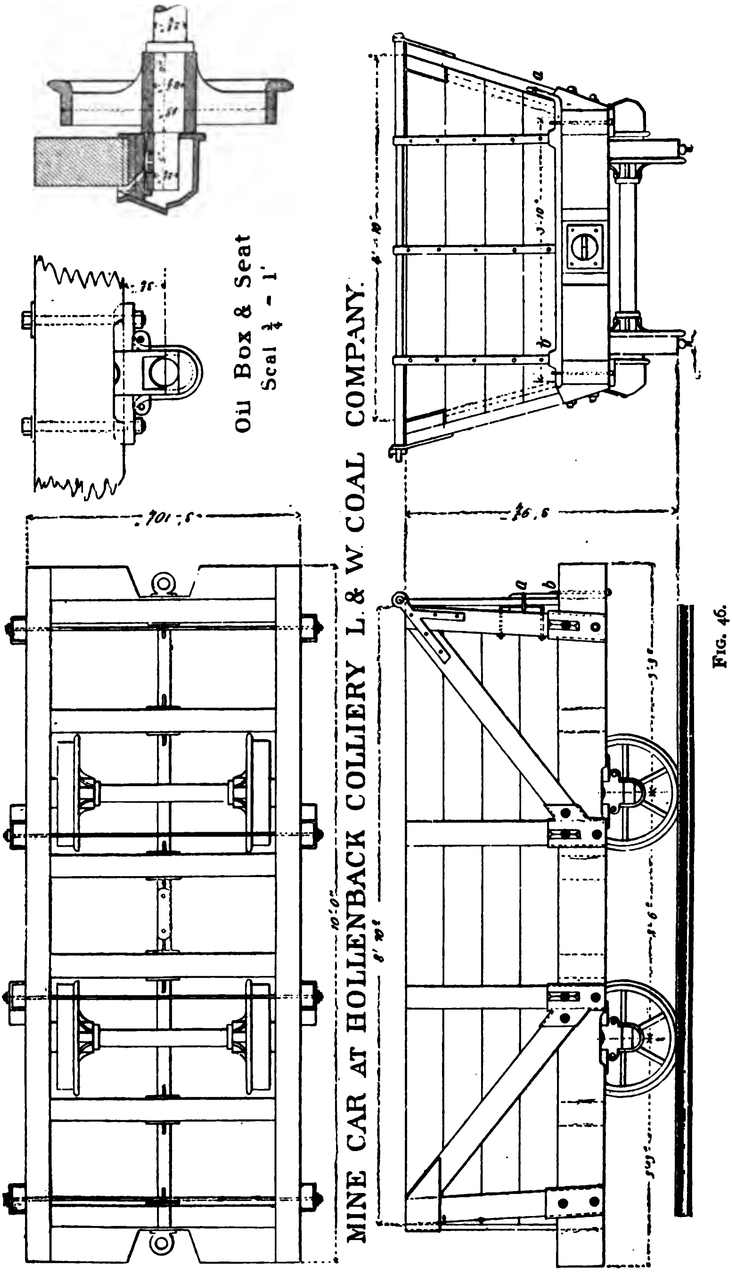
Wire rope is spliced in the same manner as hemp. The strands are unlaid for 3 feet, and each passed over one and under another of its corresponding strands on the opposite rope, for a like distance; the free ends are then trimmed off close. Short twisted rope is more easily spliced than one of long twist. The average cost for rope per ton raised 100 feet of shaft is 0.053c., and 0.069c. per 100 feet of slope. From a report (1884 of the German Government), the cost of winding is as follows: charcoal iron, round, 0.04c.; steel, round, 0.06c.; iron, flat, 0.08c.; steel, flat, 0.14c.; aloe, flat, 0.13c. The relative economy in the use of hemp and wire rope is still in question among engineers of Continental Europe, many having abandoned iron for hemp, claiming the former to be 80 per cent dearer. Their conclusions of practical tests are evinced in the almost universal adoption of aloe and hemp ropes. Their estimates show that these ropes carry equal loads for equal weights with wire.

CHAPTER VIII.

UNDERGROUND TRAFFIC.

28. Description of cars, low *vs.* high; investigation into the minutiæ of rolling-stock; wheels and self-oilers; gauge and grade; spragging; automatic devices against runaways. 29. Life of a car, dumping cradles, etc.; rails and turnplates; economy of rolling ways; consideration of friction, grade, consumption of power, etc.; tramming by hand; work of a man or animal in haulage; mules and horses, their cost and efficiency, compared with mechanical appliances; grades and the various limitations to haulage powers; objections to underground engines. 30. Locomotives for underground haulage; their sizes, speed, cost, and efficiency; smokeless, pneumatic, and electric engines; details of gravity roads, self-acting inclined planes, engine planes; clips, wheels, brakes. 31. Tail-rope systems; details, size, and cost of plant; mode of passing around curves. 32. Endless cable systems; descriptions of the four varieties; comparison of their advantages and adaptability; report of the tail-rope committee; example.

28. FOR the transportation of mineral along nearly level roads, from the working face to the entry, and on cages of carriages up the shaft or slope to the surface, various forms or cars are employed. Their size is determined by the dimensions of the gangway, and the demands of cheap haulage. Evidently the car should be as light as compatible with strength, and such that at rammer can easily manage it. Tightness is an equal requisite. For this reason a boiler-iron body is preferred, though not so much in collieries, where the wooden box is used (Fig. 46). The weights of cars vary widely—usually about half that of their cubic contents. In metal mines they weigh from 600 to 1500 lbs., and their cost is from $50 to $200. Some colliery cars carry as much as 120 cu ft. of mineral, those in metal mines, not over 30 cu. ft. Undoubtedly haulage in



MINE CAR AT HOLLENBACK COLLIERY L. & W. COAL COMPANY.

Oil Box & Seat
Scal ¼ - 1'

FIG. 46.

large cars is cheaper, so the tendency is toward a great capacity, though the natural conditions of underground work restrict the dimensions. The small cars used in buggy roads (see p. 34) have a capacity of 26 cu. ft., and are made for a 3′ or 4′ gauge. The length of the cars is limited by the sharpness of the curves. In collieries it is a maximum of 9 feet; in metal mines they are 40″ to 70″ long. Their height depends upon the conditions of loading. In thin seams and steep veins cars only run in main haulways, and are filled from chutes, provided with a spout and gate, easily manipulated at the bottom; if also hoisted on a cage, their height is a matter of indifference. When the seam is thick and the roof good, they are carried up to the face of the work, in which case they are filled by hand. If so, or if raised on carriage, the difficulties and expense are greater with a high car. To shovel one ton into a 3-foot car requires over 7300 ft.-lbs.; into a car 4 feet high, 9500. The average man can exert a continuous shovelling effect of 28,100 ft.-lbs. per hour. Allowing for the weight of the shovel, delays, throwing the mineral forward, a shoveller may load about 20 and 14 tons, respectively, in the cars per shift. Even for a medium output the economy is manifest. In metalliferous mines this is observed, but in collieries cars of 4′ 9″ and over are common. For stability, too, a low car is desirable. The width of the car depends upon the gauge and its "set." Broad cars are preferable, but may not be advantageous because of the wide gauge. Nor are they desirable if set up on a narrow gauge. A compromise is frequently taken, by which a low, wide car on a narrow gauge is employed. The axles are elbowed for large wheels, and set down on them is a narrow body, which bellies out wider over the wheels. All mine cars should be provided with bumpers, to keep the bodies of said cars at least 12″ apart.

The gauge varies from 2 feet to 4 feet, with good and sufficient reasons for the choice of any intermediate. Broad gauge gives greater stability, and a reduction of haulage-expenses. The minimum gauge of 2 feet is advantageous for easy haulage and sharp curves, cheaper track and rolling-stock, but tends to reduce stability and capacity. It reduces the length of the car, but allows of the use of inside wheels.

The wheels are as large as circumstances will permit (the larger the wheels and the smaller the axles, the less is the friction). The wheels may revolve loosely on the round or the square axle, or they may be fixed to the axle and revolve with it. Some are capped with a recess in the hub, to receive the collar on the axle, and thus prevent admission of grit (Fig. 46). They may be "inside" (below), or "outside" (beyond), the body of the car. As to the relative merits of the inside and outside, or loose, wheels, it must be admitted that engineers are not united in the opinion, though the former has the larger number of adherents. Outside wheels are more easily oiled, are cheaper and admit of the body of the car being set lower down; they do not run so smooth, or last as long as those fixed under the body of the car. Loose-wheel cars may be better for short roads with sharp curves, but they are harder to pull. With fixed wheels, one of the mutually dependent wheels, in travelling about curves, must slide. For this reason, and the ease of lubrication, loose wheels, or cone-fixed wheels, are preferred by many. The U. P. R. R., 3 feet gauge, abandoned loose wheels after careful trial. At the Drifton anthracite mine, a compromise is effected by using a pair of fixed and a pair of loose wheels.

The coal-car wheels are of cast-iron, between 16 inches and 18 inches diameter, and those of ore-cars about one half that, and solid; while the former have hub and arms to allow of "spragging."

Coal-cars are fixed on two trucks, and dump from the end, being provided with a swinging door (Fig. 46). Iron-frame cars are more commonly provided with a swivel and a lever, which hooks or unhooks the body from the trucks (Fig. 47). Dumping is easily effected by opening at the side or end a swinging door hung on an iron rod across the top by two hinges (Figs. 48, 49, and 50). Another variety of car in use in tunnels consists of a double iron-framed car, pivoted together at the centre top of each side, on two trucks. In dumping, the latch lock on each side is raised, the car opens in the middle and empties between the trucks.

Often the dumping is accomplished at the surface by some automatic device, consisting of a balanced frame, or pivoted cradle, upon which the loaded car is run and held. The moment its centre of gravity is beyond the point of support, it tips and empties the car. In another type of the dumping cradle, the entire combination is inverted; the car, which is held, dis-

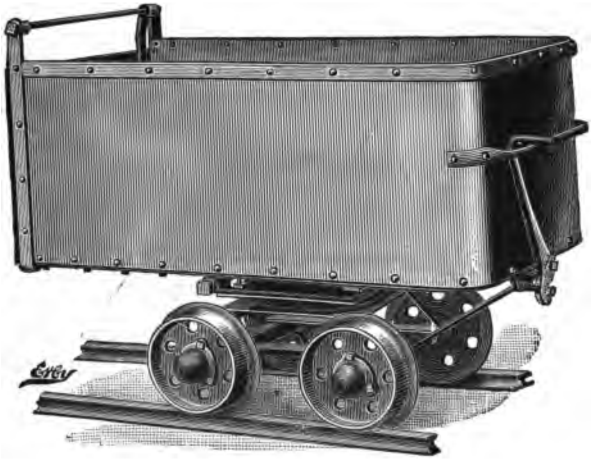


FIG. 47.

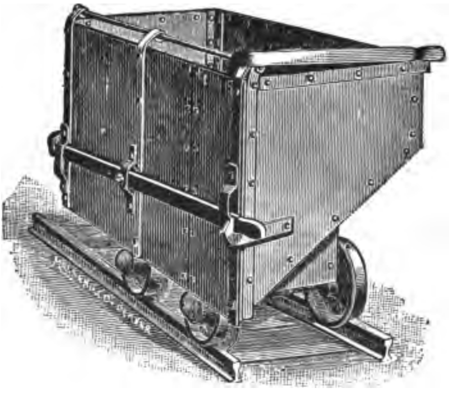


FIG. 48.

charging its load at once. Behr's device is of this character and is represented in Fig. 50. This permits the use of a stronger and lighter car, and dispenses with a dumping device on the car. A similar idea is adopted in the construction of the cars, without the use of the cradle. The body of the car is hung on a horizontal axle over the truck, so that the centre of gravity of its contents is very near to the fulcrum; a very small effort by the trammer will tip the car after it is unlocked.

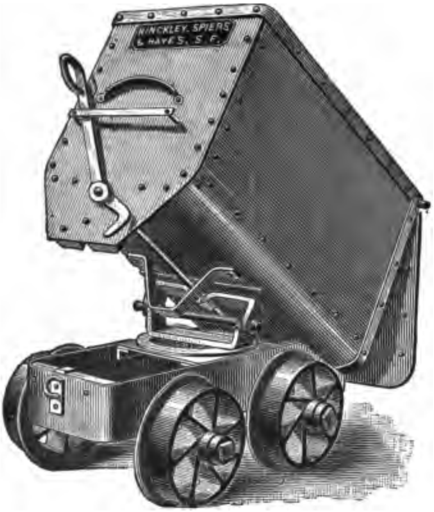


Fig. 49.

As much attention should be given to even the most insignificant details as on surface roads, to secure maximum economy; rigorous examination into anti-friction method is advised. Bearings and axles should be readily accessible. Self-oilers can be had, simple and cheap. They save power and reduce the wear. By their use, one-third of a pint of oil will last two to four months, according to the quality and the distance traversed. Their addition, in one mine, saved $7 per year per car, in oil and grease. Another example, as a corroboration, taken from an ordinary trip, showed that a locomotive can haul 20 cars with plain wheels, with a loss of 12 pounds steam-pressure, against 28 cars with Bowden self-oilers and a loss of only 3 pounds. The oils used for lubrication and illumination, underground, in the Lake Superior mines, are let down into the mine through a small pipe into tanks, instead of lowering the barrels.

The haulage inclinations and velocity being slight, brakes are not needed, nor are they much used, unless the stout stick with its fulcrum under the body of the car, and pressed on the wheel face by the weight of the trammer at the other end, may be dignified by the name. In coal mines, "spragging" is resorted to, and is more effective. A sprag is a billet of wood 12″ long and 2″ thick, which is deftly thrown between the arms

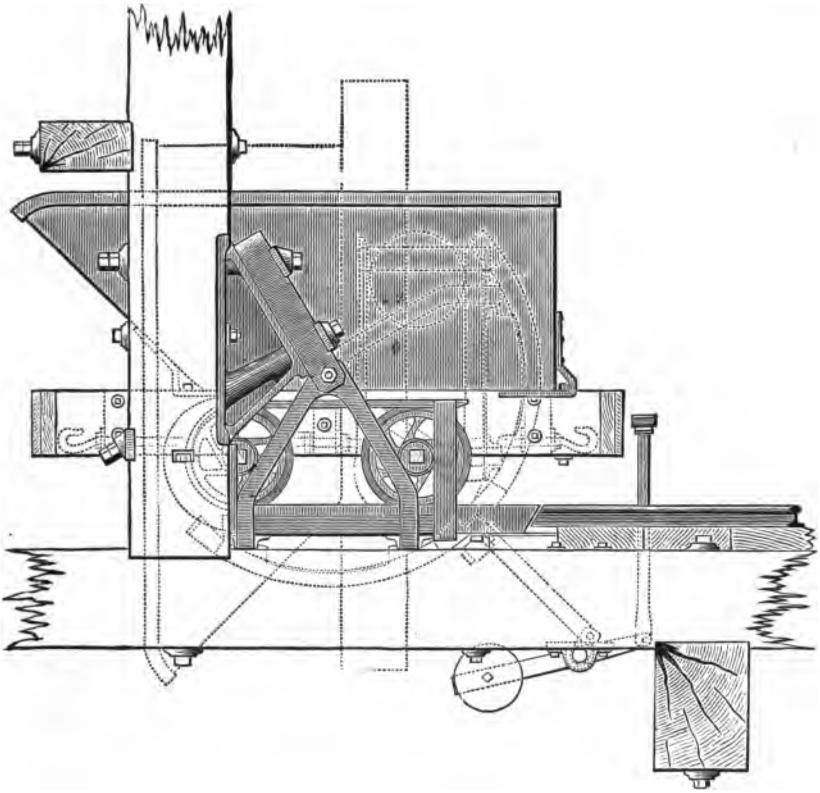


FIG. 50.

of the wheels and prevents them from turning, and converts the car, more or less, into a sled. On a slight grade only one sprag may be required, while on a slope of 1 in 6 it takes 4 sprags to check the speed of the car. They are of no use on an incline of over 1 in 5, the angle of sliding friction. Roads are often designated as 1-, 2-, 3- or 4-sprag roads, by which is meant the number of wheels spragged.

To prevent accidents from runaway cars on a grade, or from the mill back into the shaft, automatic devices are used. They are usually a balanced timber block automatically thrown across the track before the car reaches it, temporarily blocking the way.

The life of a car depends upon conditions too varied to state here. Wooden cars become loose, shaky, and larger with age. Iron ones are battered and bulged, particularly if the mineral is in large lumps.

29. The tramway is of T rails, weighing in rooms, 12 pounds, in gangways and levels 16, and in slopes as high as 35, per yard. Very light rails are not economical. A broad rail favors the wheels; depth and weight give stability. Only in petty mines does the strap rail survive, and in some steep slopes the wooden strap. The cost of 1 mile of 16-pound rail, laid, is $1600 (steel at $33). If the floor of the gangway is uneven, the sills or sleepers are laid on the knolls or ridges. For heavy duty the gangway is double tracked, or, sometimes, a single track is laid with suitable turnouts and plain frogs. At junctions, a simple iron turn-plate laid on stout planks is used, instead of the more elaborate frogs and crossings. The men drag the end of the car around, and shift it to the desired track, when it is run off. Self-acting switches are not in favor.

The opportunities for economy in underground work are not many, for the conditions are necessarily proscribed. Mechanical appliances are difficult of application, and particularly so in veins of high pitch, where the inconveniences increase, because of the narrowness of the gangway. Other lines of economy are easily obtained, and the virtue of well-laid track specially commended. On level surface roads, the tractive force required to overcome friction is about 1 per cent of the load. Underground it is rarely less than 3 per cent. R. Van A. Norris, Wilkesbarre, conducted an elaborate series of experiments, with the result that the coefficient of friction was rarely less than 60 pounds per ton, occasionally 100 pounds is reached. With self-oilers this was reduced to 1½ per cent. With any mode of rope haul, the frictional loss of power may and often does amount to 40 per cent of the weight. There is no reason why the same care should not be employed below as above ground.

The difference, in the cost and time of laying, between a substantial track and one poorly laid is trifling, but in efficiency is astounding. One car running two-thirds of the year consumes, per ton of mineral it carries, on the latter, an equivalent of 30 tons of fuel more than it would on a decently laid road, and the amount that an animal or a locomotive can pull is reduced to a quarter of that on a good road. Perhaps it is not essential, but it is certainly advisable, that good workmanship be expended upon the trackway and rolling-stock. This is more urgent as the output is large, or its value small.

For a small turnout, or on a short level haul, man power may be employed; but, as any of these increase, power must be invoked. For steady tramming, the average man is capable of exerting 27 pounds push, at 2 feet per second, moving a 2-ton car on a level, and making, according to the condition of the roads and the running-gear, from 3 to 12 ton round-trips of one mile.

Under like conditions a horse or mule makes three single trips with from 4 to 10 tons gross load per hour; though the delays, friction and bad air reduce the average to 40 or 50 gross ton-miles per day at 4 or 5 cents. The utility of the animals is confined to haulage in the secondary ways, the rooms, and for switching-places where economy in height is practised. This gives the mule a superiority over horses, because it is not so tall, though equally strong. Even as it is, the roof must be ripped off from beds below the standard thickness, to admit of a gangway high enough for mules; often, dogs or pushers are employed instead.

Usually mules are driven in teams of from two to six, according to the length of the trip and of the train, averaging two cars to the mule. Their ordinary speed is about two miles per hour. Some small mines employ them for haulage from working breasts, or only from the secondary ways, to daylight. In larger properties the mule never again sees daylight, and travels between the rooms and the general parting, or even only in the branch roads. Except where there are numerous ventilating doors to be opened and closed, or many sharp turns to be rounded, there are few cases in which the mule has the advantage over mechanical means, and where the railway may not be extended throughout the entire workings without great additional expense. It is the privilege of the operator to replace animal power by machinery, and just in proportion as he avails himself of the latter, so will his profits be.

The average cost of mules is about the same as with horses. It takes thirteen animals to supply ten workers, the balance being on the retired list for various causes. This excludes the allowances for death by accident. The cost of their keep is 90 cents per day. The accommodations for stabling need not be expensive, but attention must be given to ventilation and cleanliness. The expense of keeping up the path between the rails is an important item.

About 9000 gross ton-miles per year represents the work of one animal, which, however, varies with the grade and condition of the track. The limit is determined by the tonnage that an animal can return with uphill. Only a small grade is admissible, and that, too, from the breast. The maximum is somewhere near 3 feet per 100. A gradient of equal resistance (that on which the work on a loaded car down equals that on the returning car) should be provided where possible. If the empty returns with stowage and supplies, the grade of its track should flatten as their weight approaches that of the mineral, as, otherwise, the duty of the animal is lowered appreciably. A power which will pull 100 tons on a level can take only 47 up so slight a grade as 20 feet per mile, and 25 and 13.5 tons respectively on a 1 per cent and 2 per cent incline.

It is easily understood that the development of an extensive property by numerous shafts is expensive, so it is likely preferable to transport the mineral underground even a considerable distance to a shaft centrally located, if this can be done quickly and cheaply. This involves the elaboration of a system of haulage depending upon conditions of grade and the method of mining, as follows:

1. Where the tramway is horizontal, power is required both ways by man, horse, locomotive, stationary engine or ropeways.

2. If its grade is towards the shaft, the full train down pulls an empty up, on a self-acting plane or tramway.

3. When the grade is reversed, the loaded and empty cars are moved by a tail-rope or endless-rope arrangement.

The method adopted should not be complex, all the details carefully proportioned from direct calculations, the positions of branches and the location of machinery comprehensively planned; often the opportunity for obtaining cheap haulage fixes the entire plan of the mine.

The advantages offered by either of these systems cannot be generally stated. Any plan allowing of a single track in a narrow gangway usually has the preference, though on a double way no extra power is required to overcome the friction and the dead-weight of the conveyance. Of course the most favorable plan for tramming is that working by down-grade both ways. But as this is not feasible, it behooves the engineer to diminish the resistances of friction, and avail himself, as much as possible, of the acceleration of gravity, which assists motion on the down-grade and opposes the force operating uphill. The employment of other power than gravity becomes indispensable, when the grade is low or reversed. But the dangers and inconveniences from sparks and exhaust-steam are often prohibitory to the use of underground engines, when some form of continuous traction is necessary, the engine being at the surface, or connected with a special ventilating shaft.

30. The locomotive furnishes a cheap haul for great distances and large output (Fig. 51). Its simplicity and convenience recommend it to favor; it is much cheaper than animal power, and has the advantage over it in times of strikes and lockouts (it has not to be fed); it is easily accommodated to varying demands on it. As it is usually coal-burning, the gaseous products turned off into the mine justify the outcry against it. It befouls the ventilation, introduces a risk from fire, and also elements favorable to the decay of the roof and timbers,—heat and moisture. Its passage up and down interferes markedly with the volume of air traversing the entry; when the locomotive travels with the current, 20 per cent more air passes through than while travelling against the current. On this account the ventilating current should have a greater speed than the locomotive. However, its gangway is usually cut off from the general ventilation of the mine, the inlet current being introduced beyond the inside terminus of the locomotive run. The cost of these improvements, preparatory to this introduction, is no small item ($7000 in one mine).

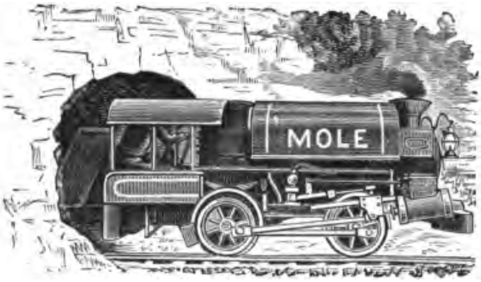


FIG. 51.

Still, to a large extent, much the same objections obtain to any underground steam-engine, compared with which its greater haulage velocity results in less cars for equal tonnage, and less cost (2 cents per ton-mile). On the other hand, it is useless on grades of over 3.5 per cent.

Compressed air may be used for the power, instead of steam, and afterwards for ventilation. If this is done, stationary engines will have the advantage over locomotives; otherwise, all things considered, the latter will ordinarily be preferred; and it is remarkable what dry steam they furnish, and what work they accomplish considering that their draught height is limited, the rails wet, and the curves sharp.

The locomotives are made of a shape to suit the mine opening, for narrow gauge (36″ to 40″) rarely over 78″ high, have four to six wheels (for curves of 50 to 75 feet radius), weigh 4 to 13 tons, and carry 125 to 350 gallons of water. Their cylinders are from 5×10 to 10×14, on 22″ to 28″ drivers, running over 16 to 28 pound rails, and costing $2,600 to $4,000. They have a traction of from 150 to 600 tons on a level. There is no difference in the price between the wide and narrow gauge locomotive of the same design and size of cylinders.

FIG. 52.

The hauling-capacity (the total weight of train guaranteed to be hauled on a level, straight track) is limited by the adhesion of the drivers (about $\frac{1}{8}$ the weight of the locomotive). A 6×10 on a 105-foot grade hauls 28 tons of train 20 miles daily on 600 pounds of coal. One of 10×14, on a 52-foot grade, and 50° curves, has an actual duty of 46 tons, 28 miles per day, with 1000 pounds of fuel. The average grade in mines is about 2 per cent, on which the capacity of the locomotive is 13 per cent of that on level. Grades are usually reduced on curves 0.02″ per 100 feet for each degree of curvature. The daily running expense of a locomotive is $4.50. Locomotives are made with inside cylinders and crank axle, but they are expensive, and advised only for very narrow tunnels.

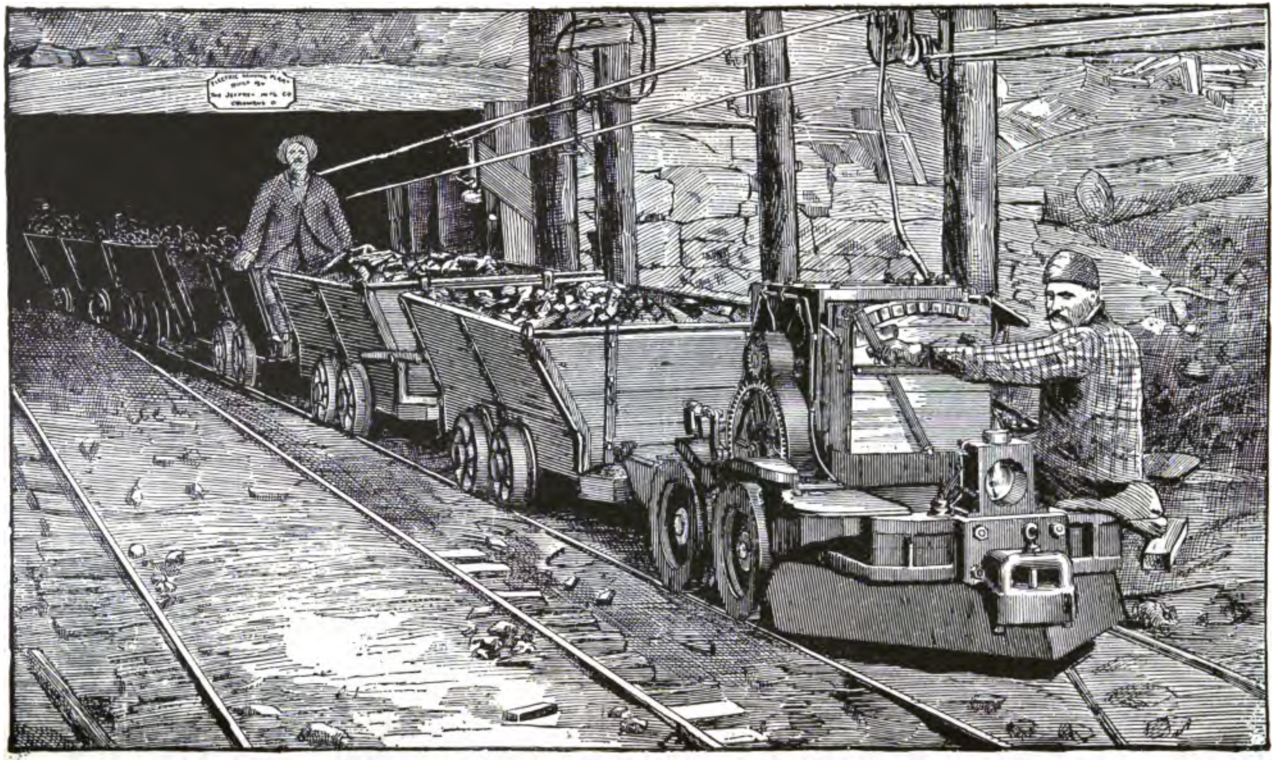

ELECTRIC MINING PLANT
BUILT BY
THE JEFFREY MFG. CO
COLUMBUS O

FIG. 53.

The tractive force, T, of a locomotive is measured by the formula $DT = 0.0654k^2ps$, wherein D is the diameter of the driver in feet, and k, p, and s as in 23. The traction of a locomotive, a static force, expressed in pounds, must not be confused with its horse-power, which is a unit of dynamic force, embracing the elements of weight, distance, and time. (Compare with 23.) The traction, T, must be equal to or greater than the sum of the train resistances; 1, the frictional, which in mines is not less than 50 lbs. per ton of train, and equals $0.025\,Y$; Y is the number of tons weight of train and load; 2, due to grade, which is $20gY$, g being in feet per 100; and 3, due to the curve, $\frac{1}{8}wzD^\circ$; this is $\frac{1}{8}$ lb. per ton per foot width of gauge, z, per 1° curvature; w is the weight of the number of cars which are on the curve at the same time; for substitution, when the radius of curve, instead of its degree, is known, we have $D^\circ r = 5{,}730$.

Pneumatic locomotives (Fig. 52) are not yet successful, nor the various fireless and smokeless engines constructed to be operated by volatile chemicals, leaving electricity and the wire-rope systems as the only real competitors of steam-locomotives, which are confessedly not as economic or as safe machines as stationary engines.

The signal success of electric installation has led to improvements in haulage methods by the use of motors and storage-battery (Fig. 53). Aside from other considerations, the rotary movement of electric appliances admits of a better running balance than can be given to ordinary reciprocating engines, and they are therefore less liable to jump the track. An 80-ampere, 450 volts Compton series wound machine, operating haulage engines, hauls from three parts of the mine, by cable 3200 yards long, 100 tons per day, and replaces 27 mules, besides several helpers, etc. As yet electric propulsion is more expensive than steam, and gives more trouble, because of the liability to indefinite delay from frequent groundings and other mishaps peculiar to electricity when carried by trolley-wire as in the surface roads. Judging from the Census Reports, the very large item in the cost of electric propulsion is the interest on the big plant. "If a car-mile be taken as a unit of traffic, the first cost, relatively to business done, is: horse, 1.00; electric, 1.23; and cable, 1.91." "The sum of operating-expenses and interest per passenger carried is 4.77, 5.08, and 4.39, respectively, as against 2.85 by steam." The electricity is carried by No. 0 wire, T-or L-irons for the trolley.

Storage batteries obviate the necessity for wires, but are as yet too expensive. About 25 lbs. of battery will carry one horse-power per hour. A space of 40 square feet will accommodate 250 elements, which will furnish 11 horse-power for a 10-hour shift. The weight, and hence the adhesion, of this engine is over twice that of the steam locomotive of equal horse-power, even if the coefficient of adhesion is not increased.

Where the conditions are favorable to the use of gravity as the sole power, on a self-acting plane, the principle employed is to let a loaded car going down pull an empty car up. A rope connecting the two cars, passing around a sheave, or a drum, at the head of the incline, is the only mechanism required. With the former a single rope is used, while with the latter two ropes, wound in opposite directions, connect the cars. The axis of the drum is horizontal; of the sheave, vertical. The ropes are of a length equal to that of the plane. Often on a drum a single rope is used, which receives four or five extra wraps to prevent slipping, the ends being attached to the empty and loaded cars, respectively. This is not recommended.

The sheave or drum is in a recess fitted for it on or above the counter gangway, where are received and connected the cars from the breast, or sometimes this arrangement delivers direct from the breast; the terminus is on the lower gangway or at the foot of the shaft. Swinging platforms connect the drifts or gangways with the slope. Usually two lines of rails (35 lbs.) are laid the whole length; single tracks with turnouts are false economy.

The smallest gradient at which these are operating is 6°; the best, 1 in 5; while on one over 1 in 3 there is an excess of tractive force above resistance. At an angle exceeding 35° the method of mining is such as would not afford an opportunity for this means of delivery. Occasionally, in steep veins, such a scheme is in operation for delivery down an "auxiliary" from the several levels to the main gangway. The surplus force must be counteracted by a strong brake, which regulates the speed to a nicety. The mean velocity is about 400 feet per minute, and to diminish the momentum toward the bottom the plane is flattened. The theoretical curve of the slope is a cycloid, concave upwards.

The brake is usually an iron strap with wooden blocks, actuated by a lever. India-rubber has many advantages over wood, though hitherto the trouble has been to fix the rubber on the shoe, because it disintegrates so readily where the bolts pass through. This has recently been overcome; the rubber blocks have a dovetail at the back, which is inserted and fits easily into its shoe. The tightening of the blocks on the wheel while "braking" crowds the dovetail into the shoe.

Means must be devised for preventing the rope from slipping off the sheave, and provision also made to protect it from undue strains caused by shocks. A V friction-clip wheel, designed so that its friction will equal that on the road, is the simplest plan. The clip bites the rope, and there is no slip or wear unless a car jumps the track, when the slipping of the clutch will notify the brakeman. This clip-pulley is also very effective on endless-rope systems for transmitting power.

The limit of length to which these planes may be used is fixed only by the friction of the dragging rope. Anti-friction rollers, 6 × 20 inches long, on 1-inch axles, are therefore necessary at intermediate points, about 18 inches apart, to give it support. The amount handled is limited only by the facilities and conveniences affecting the trip time. The system is inexpensive, requires strict discipline and an ample signalling code. The cars may be connected singly or in trains, but equally on each branch. Trailing-forks behind the cars prevent catastrophe if the rope breaks—usually it is the up-rope.

These tramways are equally good underground—as on the surface—where the cost does not exceed 10 cents per mile-ton, and is often as low as 3 cents (Fig. 54). The cost of construction is about $3 per foot of length. Smaller sized ropes are needed than for an equal length and weight of vertical hoist. Two-car trips on 10° planes require $\frac{5}{8}$ rope, and $\frac{7}{8}$ on 45° slope.

When the slope cannot be self-acting, "engine-planes" are used. A stationary engine is located at the head, and has a

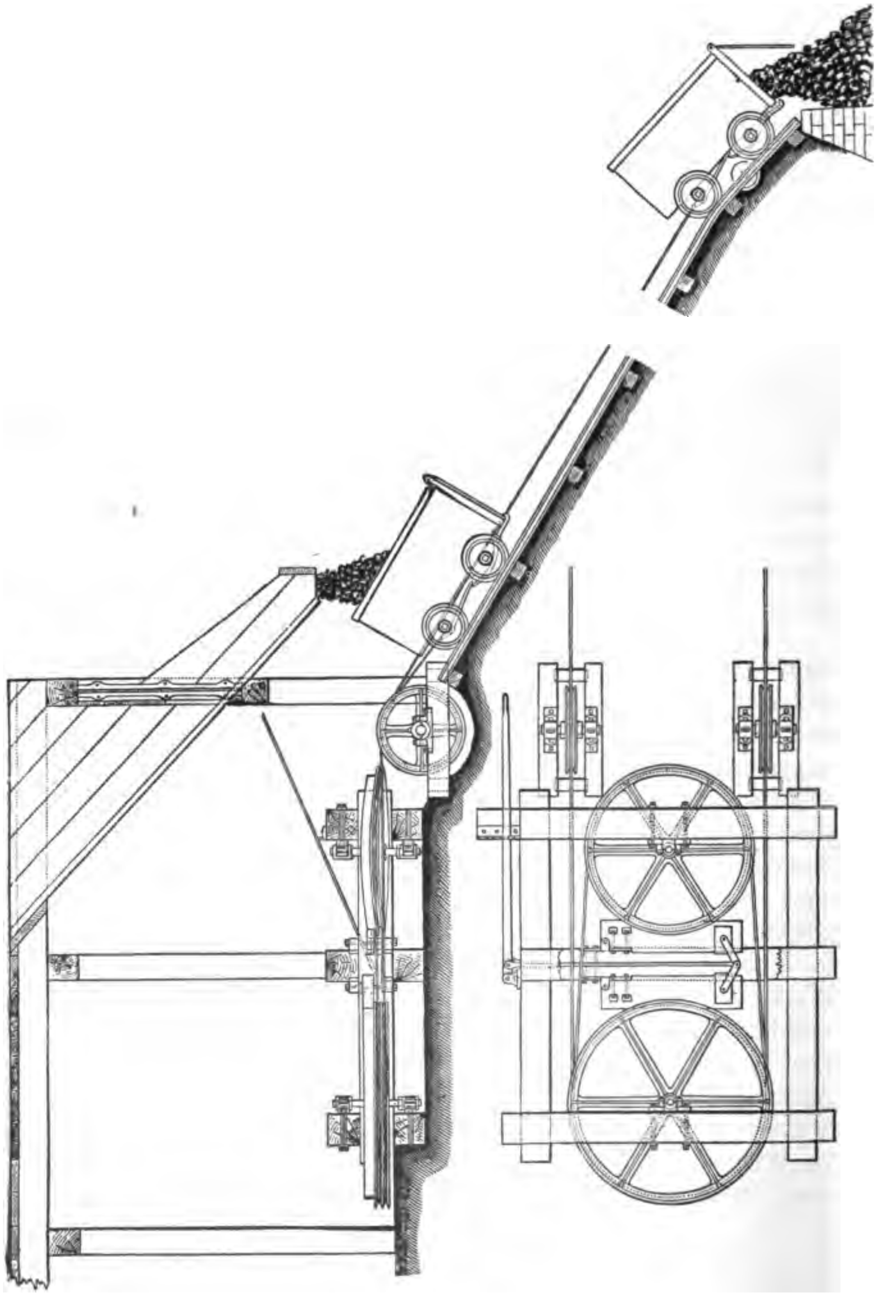


FIG. 54.—INCLINE GRAVITY TRAMWAY.

drum which may freely turn and pay out the rope for the descending cars, and be geared to pull them up returning on the same or a parallel track. On single-track planes the engine is non-reversing. On a grade of 1.7 per 100, gravity will take the loaded cars down with a reasonable velocity (empties on a 2.25 grade) and pull the rope after them. On a 10 per cent grade a break will be necessary.

The cars usually travel in trains, of 10 to 30 in number, in charge of a conductor who operates a dead-fall timber-block to hold the train while the cars are being shunted. So the system is well adapted for delivery from side-entries at different levels, and may be used on slight curves by curving the rope on iron guide-wheels. Ordinarily the rope will last four years. A 14 × 30 cylinder, 3-ton fly-wheel, $\frac{9}{16}$ steel rope, on a plane 4600 feet long by 80 high, has a daily output of 950 tons of mineral in 25- to 30-car trains.

31. Of rope-ways there are two classes, the tail-rope and the endless cable. Each system has its advocates; both are extensively used in beds, the former for a limiting gradient of 3 in 100, either with or against the loaded cars.

In the tail-rope system, haulage is effected by a stationary engine, two ropes on drums which are thrown alternately in and out of gear. The main rope, having a length equal to that of the road, is hitched to the front end of a train, the tail-rope, of double this length, passes from its drum around the sheave at the bottom, and thence to the rear end of the cars; a short chain at each end of the train couples the cars to the two ropes. In operating, the main-rope drum is thrown into gear, the other out of gear, engine started, and the loaded cars are drawn from the mine to the outlet, dragging the tail-rope after them. Then the main drum is thrown off, the tail drum in gear, the empties replace the full ones and return, pulling the main rope off its drum. See Fig. 55.

The lower sheave is a clip-clutch. The main rope is rarely over $\frac{7}{8}$ inch diameter. It is generally replaced every year by a new one, doing two years of additional service as a tail-rope, after which it is discarded.

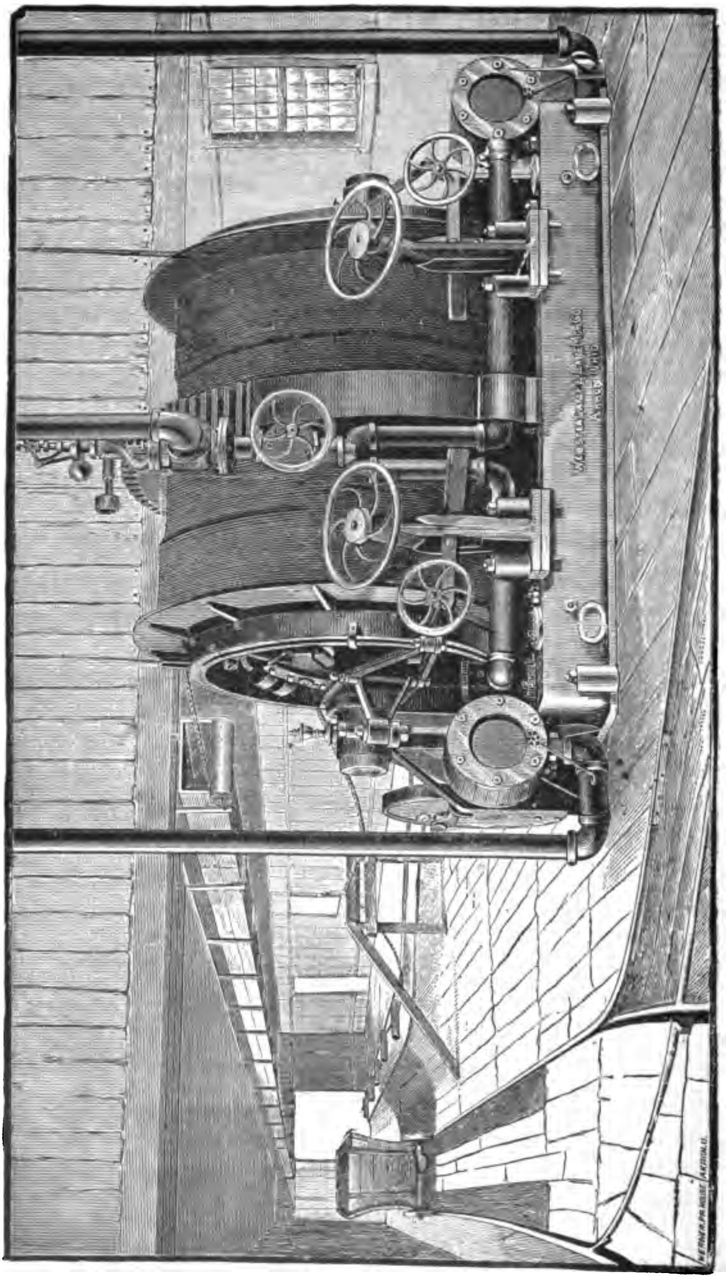


FIG. 55.

The length of haul is limited only by the engine-power and the resistances. While the dip may be anything less than 3 in 100, yet its greatest advantage is manifest on a level or very slightly falling dip. The velocity of haul may reach as high as 10 miles an hour. Each trip takes a train of 10 to 100 cars accompanied by a conductor, whose duty it is to look out for accidents, the train being hard to control when the tail-rope loses its hold. The system is preferred in American collieries, and is the best plan by which branch-ways may be operated. Each branch has its own rope passing over sheaves at the ends, the principal ropes are opened at the proper points for connection with the branch ropes, the train engaged and hauled to the end of its journey.

This is an inexpensive plant to build and repair; it does not require a double track, though it demands a double length of rope. With a single roadway the sheave is vertical, and the tail-rope moves along the roof to its drum vertically above the other. In this form it is encountered in iron mines. In extensive workings it has dispensed with animal power, and is advantageously used for slight grades, and even for undulating roadways. An 18 × 30 cylinder, with 75 lbs. pressure, on a 2800-foot slope, grade 1 in 200, 4½-foot drums, ¾ main- and ½ tail-rope, 30 trips of 17 cars each are made with a velocity of 8 to 11 ft. per second.

For passing around curves, 24-inch wheels are laid horizontally and beyond the inside rails. These carry the rope till the car reaches the turn. On reverse curves they are laid nearer the inside rail and slightly inclined to the horizontal toward the centre.

32. The endless-cable systems are much in vogue, and require less rope than the above-mentioned plan. They are very suitable for a double-track line of communication with frequent stoppages and no branches. A continuous motion in one direction is imparted to the rope by a single wheel or drum, and the tension produced by artificial means—a friction-grip or clip-wheel, or else several tnrns of the rope on the drum. The clip-wheel keeps the chain or rope tight, being on a carriage frame, to the far end of which a rope is attached. This rope passes over a pulley suspending a weight, which maintains the tension desired in the tram rope. On a single track a reversing engine works the rope alternately forward and backward, the return rope being supported overhead or at one side of the gangway. The position of the rope varies in the four different methods, being above, on top, or below the cars:

1. Is the cheapest, most universal, and effective method; the chain rests in forks riveted on top of the cars, which are singly attached at intervals of 25 to 100 ft.

2. For heavy grades, with stations; the chain may run on rollers underneath the cars, a short length of chain being used for connection.

3. For uniform grade and sharp curves; the cars are attached by chain to an endless rope above them.

4. For varying grades, curves, and without branches; they are operated like our surface roads.

(1) This system supplies a continuous power which may be taken off at any point. The cars, readily connected and disconnected, are distributed singly along the line, from 20 to 100 feet apart, and as the velocity does not exceed 3 or 4 miles an hour, the boys have ample time to hitch them on the rope. The capacity is independent of the length, being determined only by the number of cars delivered. The power engines have heavy fly-wheels for regularity and compactness. A sprocket-wheel keeps the desired tension and prevents slipping. It is a driving-wheel of 3 feet or more diameter, carrying forks set radially, and capable of being screwed out and in; these are turned a little to seize the chain as it lengthens and drags, until they are paid out to the limit, when a few links are removed and the forks adjusted. Extensions are easily made when required; this is not possible with tail-rope.

This system costs less for power than tail-rope, and admits of sharper curves and steeper grades, but requires two lines of rails.

As an example of a plant: a 5773-foot road with 70° curves, cars (0.5 ton each) 50 feet apart, at 2.8 miles per hour, delivers 1250 tons per shift. The chain shows a tension of 4000 lbs. The engine is 17 horse-power, has a drum of 5 feet diameter, lagged with wood, lasting a year (costs $25); the sprocket ($80) wears out in four months.

A "tail-rope committee" of the North of England Institute of Engineers reported (vol. xvii. of its Transactions) that, "as far as the cost of maintenance and working expenses are concerned, this endless-chain system can be applied, with few exceptions, to every condition of wagon-way with greater economy than any of the other systems." It is not restricted by grade, nor by any irregularities or crookedness in the roadways.

(2) The friction, and of course the wear, is much greater than in (1).

(3) The mechanism of the endless-rope system differs little from that of (1) except in the method of connection. The cars at intervals are hooked to the short chains pendent from the rope, or a small chain is wrapped twice around the rope or into loops along it. The speed is lower and the cost is higher than in (1). The first cost of one of 900 tons capacity, 3200 ft. long, is $4000, and its yearly repairs amount to $200.

(4) Each car is connected by a hand-clamp, somewhat like that in use on the surface cable-roads, similar in action to a pipe-tongs, and is also provided with a device to keep the rope on the rollers. The clamp is detachable. At each station or branch a man knocks off the grip and switches the cars. Instead, often a train of 20 to 50 cars is hauled from a guide-car, which is under control of the gripman who rides it.

Fuel, labor, etc., of a 450-ton endless ropeway is about $7.50 per day. The initial cost is about $6700 for a plant of 4200 ft. double track (with a grade of 40 feet in 1500, the rest level), a ⅞ rope at 260 feet per minute, and a 12 × 20 cylinder at 56 revolutions, geared to 6-ft. drums at 14 revolutions.

For cable-roads the ropes run on grooved rollers, 6 inches diameter, 30 feet apart, resting on spindles supported between longitudinal oak stringers across the ties, as nearly central as possible. Around curves the wheels are 12 inches diameter, bolted flat to each tie, and have the upper rim smaller than the lower one, to let the grip pass easily.

The safest way of connecting ropes is, of course, by splicing, but sockets are convenient where shortening may be required.

An electric system along the tramways is requisite for safety as well as for signalling, though the malicious destruction of insulation, etc., has caused its abandonment by many operators.

EXAMPLE.—A one-mile endless rope, travelling at 2 miles an hour over an average grade of 3 in 100, delivers 50 tons per hour. Required, the total resistance on the line and the size of the cylinders under 50 lbs. boiler-pressure and 160 ft. piston-speed. Each car weighs 800 lbs., and carries 2000 lbs. Along the line are distributed 25 loaded and 25 empty cars, about 100 feet apart, and a ton is delivered every 36 seconds; with a coefficient of friction of 0.02, the frictional resistances on the halves of the line are 1400 and 400 lbs. respectively. To raise 25 tons up the plane requires a force of 3 per cent of 50,000, or 1500 lbs.; the gravity component of the cars is 600 lbs. The gravity components of the cars and the rope, up and down, balance each other, leaving the work of the engine to be that of overcoming 1500 + 1800 (the drag of the rope is assumed at 3000); the which, carried at a rate of 176 feet per minute, requires 1,109,800 ft.-lbs., or 33.6 indicated horse-power. 6.283 fk^2psn is the dynamometric power doing by the steam. Substituting, k is about 7⅛ inches, and for 130 strokes, s nearly 15 inches. If the driving-sheave is 6 feet diameter, it makes 9.3 revolutions per minute, and is therefore geared 1 to 7.

CHAPTER IX.

SURFACE TRANSPORTATION.

33. The pioneer burro; aërial tramways; description of the Bleichert, Hallidie, and Huson types; capacity, cost, etc.; regulation of the tension of the rope. 34. Wire-rope transmission of power; pulleys, sheaves, rope, etc.; formulæ.

33. MINING in the mountainous regions encounters difficulties in the transportation of the product and supplies, which are not readily overcome. Often a mine is inaccessible to wagons, and burros constitute the only means of transportation. The ore is carefully sorted, sewed up into sacks containing 90 lbs. each, and packed, one on each side of the jack, to market. They travel in trains of 20 to each driver, averaging about a mile an hour, and return at the same pace with the supplies for the mine. The cost of filling and sewing the sacks and their repair is high; and as it takes 11 jacks to "pack" a ton away and fetch enough fuel to run a 26 horse-power engine 24 hours, it may readily be understood why the much-abused, patient brutes remain only in isolated camps as companions to the pioneer prospector, for whom they continue to do service between mine and wagon or mill.

For larger output they are replaced by an aërial tramway, which is a sort of an endless rope-way that can be run night or day in all seasons, without road or expensive machinery, and furnish a cheap, convenient conveyance for ore and supplies, down a declivity, around bluffs, over intervening hills, and around flat curves for a mile or more. When the grade to the point of delivery is about 14 in 100, the tramway is self-acting, the speed being regulated by a brake; below this, auxiliary power is applied to the rope at the upper end.

There are two varieties, one represented by the Bleichert, and the other by the Hallidie and the Huson patents. In the former, one or two ropes are stretched tightly and supported by standards, 10 or more feet high, to give a continuous slope from the mine to the discharging point, where they are well anchored by screw-rods and buckles. The cables constitute the roadway for the trolleys, from which the tubs are suspended. The trolleys are operated by a single or an endless rope which passes around clutch-sheaves at the top and bottom. The tubs carry as much as a ton, and dump automatically into a bin or wagon. The cost of this system is quite high, but it is handling 1000 tons per day. The carrying cable (Fig. 56) is stationary and about 1½ inch diameter, though it is locally strengthened according to the strain to be carried. A line of steel rods may replace it for short spans and light loads, but cable is better, as tending to convert the otherwise transverse strains into tension.

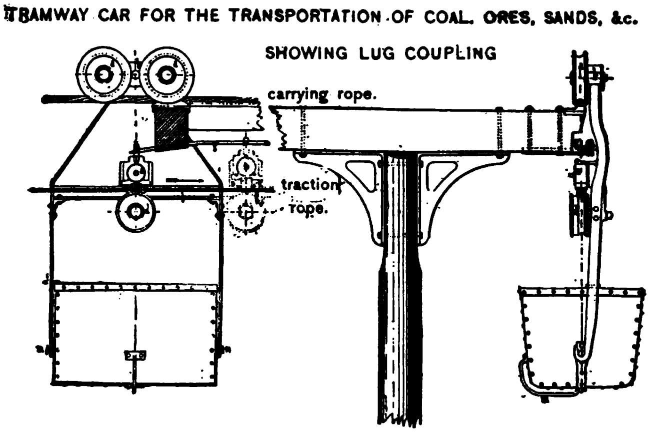


FIG. 56.

In a somewhat different style of aerial tramway, one or two ropes stretch the entire length to constitute the guides for one or two large skips holding a ton or so, and attached at the end of a rope. They are operated like the gravity planes, p. 129, and may be self-acting or not. See Fig. 54.

The principle of the Hallidie or Huson design is as follows: An endless wire rope is supported, at intervals of 150 to 300 feet, on suitable sheaves, which are mounted vertically on the ends of cross-arms fixed to the necessary posts or frames, and at sufficient height to clear all surface obstructions. At both ends of the line the rope passes around clip-pulleys set horizontally. The upper wheel is placed on a frame below the level of the tunnel or shaft mouth (Fig. 57). At the lower end may be a plain or a grip wheel on a carriage tower frame, which assumes a position such that a constant tension may be main-

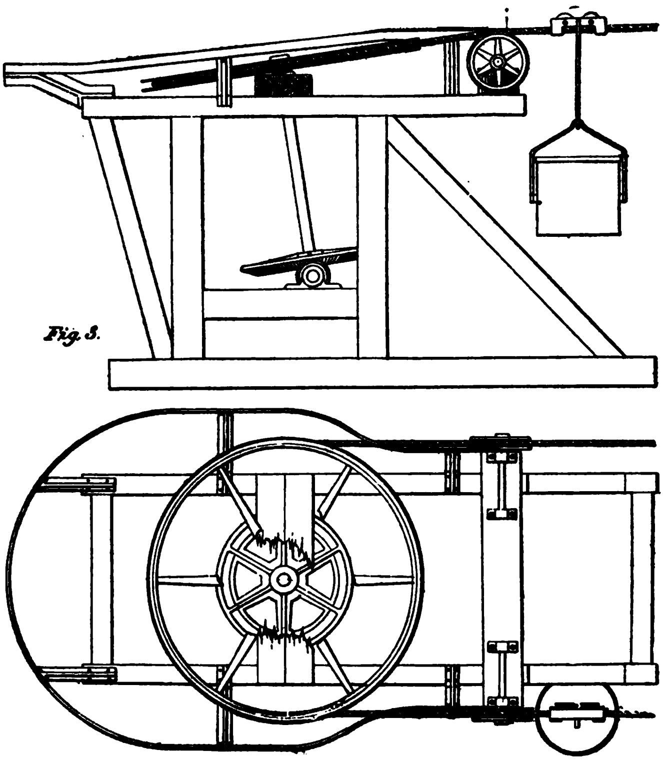


FIG. 57.

tained in the rope. Precaution should be taken to provide a hold-down rail on top of the wheels to prevent the carriage from upsetting. The distinguishing feature is that the load is at once supported and moved by the same rope, which has a continuous motion in one direction, at a velocity of about 200 feet per minute. A higher velocity is inadmissible because of the tendency to fly off the sheaves.

Buckets of various designs, according to the character of the material to be handled, are suspended by hangers or clips, which are either inserted into the rope or clinched around the outside of it, and attached at intervals determined by the amount of material to be delivered. Usually they are wrought-iron rectangular buckets holding about 100 lbs. each (Fig. 58).

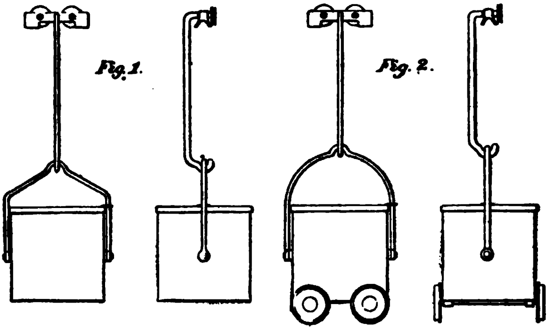


FIG. 58.

For the transport of very large outputs the buckets may be nearer together than the average 200 feet, or larger, and the rope may be heavier than the ordinary size of $\frac{5}{8}$. The buckets may be loaded at any point along the line, automatically or by hand, and are unloaded at the lower end by some automatic device. The carrier strikes a lever, which opens a catch (holding the bottom in place), and discharges the ore; a counterpoise on the bottom closes it again. The hangers are so made that they may pass uninterruptedly over the rims of the supporting sheaves and around the terminal pulleys, their consumption is large, and amounts to $100 per year on a medium line.

The strongest form of intermediate supports are stout rectangular frame standards of four sills, from each end of which is built an X transversely, from on top of which are heavily bolted cross-arms. As these X's lean towards each other at the top, they are not liable to get out of line, nor does the weighted side of the rope pull the cross-arm out of level. To the ends of the cross-arms are boxed the carrying-sheaves, rubber-lined and loose on the axle. To round a curve, the standards are nearer together, and the rope is slightly deflected with each wheel.

A rope-way running 200 feet per minute, carrying 100 lbs. per bucket every 100 feet, will deliver 60 tons per shift. With a descent sufficient for gravity to supply the power, three men can manage all of its operations. It requires some supervision, and delivers ore at 20 to 35 cents per ton-mile (inclusive of all allowances), and about 60 cents per cord-mile for wood. The line can be completed for $1.30 per foot, and $2000 for the machinery at the terminals. Curves and long stretches increase the cost; grade does not.

34. For power transference from, instead of ore transportation to, remote points, a similar arrangement is widely applied, efficient and cheap. For moderate distances, up to a mile, its efficiency is greater than by any other system; at half a mile it is 90 per cent. The inevitable concomitants—which accumulate so rapidly that for distances of over a mile electricity gives much better results—are the losses of energy due to friction of bearings, air-resistances from centrifugal action, stiffness of ropes, and elasticity, due to the spiral winding.

The rope is of the seven-wire pattern, of from $\frac{3}{8}$ to $\frac{7}{8}$ diameter, passing around sheaves, and supported over the intervening spaces by wheels. The size of the rope increases with the tension, and that, in turn, depends on the sag allowed, which fixes the distance between the stations (60 to 300 feet). The rope runs in cushioned-grooves on leather or rubber without slip, noise, or swaying, if the wheels are well-balanced and carefully aligned. Those on the driving-side are nearly of the same size as the sheaves, those on the slack-side one half smaller, the tension being less there. Where it can be arranged, the upper side should be the slack side, and the lower, the pulling side. Large wheels are advised also, because they keep the two ropes apart.

Evidently, the power that can be transmitted depends upon the adhesion of the rope to its driving and driven sheaves. Grip-pulleys, or clutches, increase this adhesion, and through it the velocity limit. The product of the velocity and the force at the sheave-rim measures the work done. The force available at the sheave is the assumed maximum tension, T, less the loss due to centrifugal force. Not all of it can be used, because some of it is absorbed in giving adhesion, and this is an uncertain quantity.

Let v = velocity in feet per second;
d = diameter of the rope in inches;
w = weight on the journal;
W = weight of rope between stations;
j = loss, in ft.-lbs., due to journal-friction,
F = " " " " " centrifugal force;
N = the horse-power transmitted;
n = number of revolutions of the wheel per minute.
Then $F = 1.4d^2v^3$;
j = for each end-sheave, $249N + 0.185wv$;
j = for intermediate stations, $0.1856(w + W)v$;
$N = 1.7d^3n$, when the diameter of the wheel is 165 times that of d;
and $N = 3d^3n$, when it exceeds $200d$.

These latter are approximate values. Six- or seven-foot wheels, with $\frac{1}{2}''$ rope, at 80 to 140 revolutions per minute, will transmit 10.7 to 29.6 horse-power, while ten- to eleven-foot wheels, with $\frac{5}{8}$ to $1\frac{1}{8}$ rope, give 58 to 135 horse-power.

Tension is adjusted and maintained, as in aërial trams, by tightening-sheaves on carriages; for the rope cannot be so nicely spliced as to get the proper sag, which for spans of 150 to 250 feet should be from 1.3 feet to 3.6 feet when the rope is at rest. Every two or three months the stretch of the rope is taken up by shortening and resplicing. With inclined lines the proper deflections cannot be obtained without tighteners.

Often, instead of a continuous line and an endless rope, a series of closed ropes and double pulleys in sections do fair service, are easily repaired or renewed, and less influenced by changes of temperature.

CHAPTER X.

PUMPING.

35. Exclusion of water by cribbing and tubbing shafts; building dams and plastering cross-courses in levels; the use of advance bore-holes in approaching abandoned workings; drainage by tunnels; co-operative drainage; hydraulic rams and the Hungarian system of pumping; bailing by self-filling buckets, skips, and tanks. 36. Single-acting lift-pumps; details of sizes, of rods, pipes, valves, gaskets, etc.; spiral weld *vs.* riveted pipes; formulæ for calculating the dimensions of parts; cost of surface plant; descriptions of the Cook, Wormer, and Bull pumps; working by steam or water pressure; formulæ. 37. Single-acting force-pumps; method of altering lift- to force-pump; description of the mechanism and operation of the Cornish pump; size of pipe, length of lifts, and dimensions of pump-rods; tapering rods, catches, V-bobs, and balance-bobs; formulæ for the thickness of pipes, discharge, etc.; account of the Ontario, Friedensville, and other mammoth plants. 38. Regulation of the speed of pumping; churning of the plunger, vibration of the rod, and its prevention. 39. Double-acting pumps, sinking pumps, Cushier system; steam-pumps; their construction and operation; formulæ for sizes of cylinders, discharge, etc. 40. Comparison with the Cornish pump; relative advantages of the steam plants; pumping-engines; compound and condensing pumps, duty and calculation of; rotary pumps; water-pressure engines; California and Nevada systems; electric pumps; the windmill for power.

35. TURNING to the subject of raising water from the mines, we must not forget that water gains its entrance by many and untraceable ways. In some workings it is an incessant flow from some watery stratum, in others it is an intermittent seepage. The subterranean current is easily excluded from the mine by the use of a cement lining, or an iron tubing to the shaft (see II, **63**), but the seepage accumulates and must be pumped off, unless the workings possess a natural drainage or an easy effluence by adit or tunnel for the upper ground. A gutter at the side of the track, or under the tramway path, with a slope of 1 in 500, readily carries off the water, and not uncommonly delivers it to a small wheel to drive a ventilating-fan. Generally the seepage, following the hydrodynamic law, increases with the depth of the opening, and a very liberal sump is provided for its accumulation. Often one shaft and its workings become, naturally, a sump for the entire district, and drain all the neighboring properties above its level, and this suggests a simple means of keeping one's mine dry. Otherwise, as the amount of water to be encountered is uncertain, provision must be made for the handling of a large volume, according to the history of similar properties. In some coal-mines of Pennsylvania as much as 4000 gallons of water are raised per ton of coal; in Colorado often 40 tons of water per ton of ore. The Ontario and Friedensville mines raise many times larger volumes. The magnitude of such work demands the employment of powerful machinery, and often on a plan too elaborate for the means of the average operator. In some localities the drainage of the district is accomplished by a co-operative scheme with extremely beneficial results. A long tunnel penetrating the country at a level much below the lowest point of exploration drains considerable territory, dispensing with the heavy individual plants, and extends the exploration and the productiveness of the mines. Numerous examples of tunnels ten miles or more in length may be quoted, some even carrying so much water as to become canals for transportation. Several such drainage tunnels are driven in the coal regions of Pennsylvania.

Upon cutting a wet cross-course to the vein, it is a common practise to plaster it up; or, in encountering old workings, to build a brick or stone bulkhead, arched convex towards the water (Fig. 171). To provide means for the escape of the accumulated water which might otherwise do injury, a cast-iron pipe is built into the dam near its top, and another near the bottom. Either, or both, may be plugged as required. Similarly, in approaching abandoned works, it is required by law in some States that a bore-hole be kept 30 to 50 feet in advance of the drift, and flank-holes on each side, to guard against dangers from the sudden breaking into the reservoir.

Under certain conditions, in stratified regions, a hole is drilled from the sump down to some permeable stratum, into which the water is discharged.

When the surroundings are such that a tunnel may not be used for the unwatering of the mine, pumping arrangements are indispensable. The earlier forms were crude, the engine being of recent date. Surface water-falls were employed to operate wheels, which raised bucketfuls from below; or, the surface water was arranged to compress air in a reservoir at the surface, from which pipes to the sump conveyed the compressed air, the elastic force of which, in turn, forced the water up to the surface through another pipe. This is a wasteful system and intermittent, but doubtless was cheaper than any other means then available.

At the Comstock mines a sort of hydraulic ram is used, by which 1800 gallons are pumped from the 2600-foot level to the Sutro tunnel at 1600 feet. The air-pressure in the accumulator is 960 pounds per square inch, and the pipes at the bottom sustain a pressure of 2000 pounds. The engine-pressure is 80 pounds, and the actual duty given, 35 horse-power per ton of coal. This has just been introduced at Eureka.

The efficiency of the ram diminishes with the ratio between the quantity of water raised and that used. With a fall of 1 and a lift of 4, the efficiency is 86 per cent; if the lift is ten times the fall, it is 53 per cent; at 1 to 20, it is 17 per cent; and with 1 to 26, it is 0.

Small volumes of water are handled by buckets, obtainable of any size, and with a capacity up to 200 gallons (Fig. 59). At the bottom is an inlet valve by which the tub is quickly filled as it sinks into the sump; it is then hauled up, its valve closes, and at the surface it is

FIG. 59.

discharged by being brought down on a pin which again opens the valve. In some mines the water-bucket is attached underneath the cage, and travels continually with it. Bailing-tanks (Figs. 60 and 61) holding 450 to 900 gallons, with balanced valves and a discharging-peg at the bottom, built to fit a compartment of the shaft, and manipulated by an individual drum, give great satisfaction in many properties. Slopes are equipped with a similarly valved skip, the emptying being done from the mouth, as with ore. But if the mine makes more water than can be handled by these means at

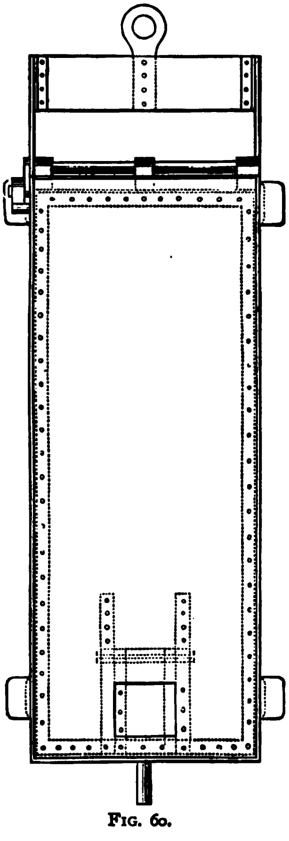


Fig. 60.

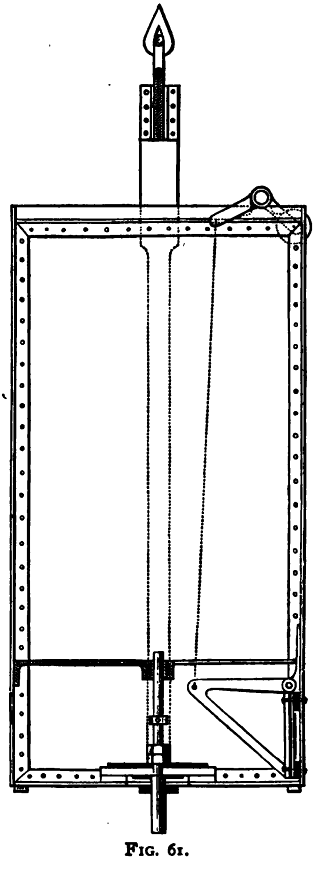


Fig. 61.

spare hoisting moments, special machinery is added, and of one of two kinds: the single-acting lift-pump, or the force-pump, single- or double-acting.

36. Pumps of the first class are much in favor because of their simplicity. Their use is restricted to vertical shafts and a lift of less than 300 feet. A plunger-rod terminates in a piston in the bottom length of a pipe, where it "sucks" up from the sump water which, with the next up-stroke, is lifted into a stand-pipe, from which it is ultimately discharged at the surface.

Fig. 62.

The stand-pipe, of a diameter commonly 10 inches, often as much as 20, extends from bottom to top. It is of cast-iron, lap-weld, wrought-iron, spiral riveted seam, or weld-steel, procurable in lengths of 5 to 20 feet. The cast-iron pipe, having a smooth interior and uniform diameter throughout, is preferable and more convenient than the riveted pipe (Fig. 62) or the lap-weld iron (Fig. 63); but as it represents too much dead-weight for the strength, its days of utility are nearing an end. The ideal pipe is of steel, which gives the lightest, strongest, and most durable tubing; this may be had in four grades, light to extra heavy. It is made of spirally-laid sheet steel riveted at the overlapping-joints or cold-hammer welded. The pipes are united by bolting together at the flanges, which are riveted, screwed, or locked on the pipe (Fig. 63); or, preferably, they are coupled on the hub-and-spigot plan of sleeve (Fig. 64). This is a double socket, into which the pipe is slipped, "oakumed," and leaded from each side, as shown. For this joint the pipes have expanded ends.

A water-tight joint is secured by placing rubber, leather, lead, or, best of all, corrugated copper gaskets between the flanges, which are then bolted together while lowering. Spence's metal, used as a calker, offers an excellent joint, is cheaper than lead, and ought to be better known.

The pipes last fifteen or twenty years unless the water is corrosive, in which case gun-metal is used. If the water is very bad, wooden pipes are made by hollowing the trees, fitting the joints, tarring them, and strengthening by wrought-iron bands at every three to six feet. In many mines recourse has been had to these as the only stand-pipe that will last over six weeks.

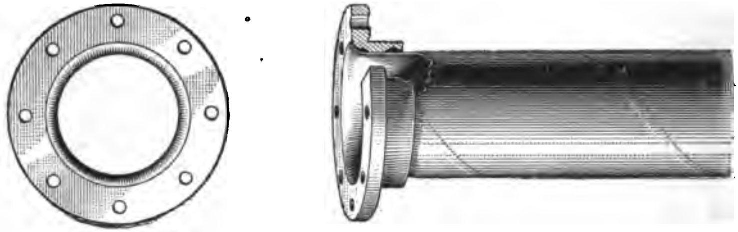
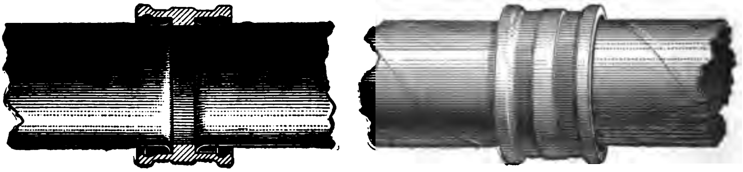


Fig. 63.



Fig, 64.

At the lower end of the stand-pipe a 12-foot length of cast-iron constitutes the working-barrel, in which oscillates a piston carrying an upward-opening valve, similar to that at the lower end of the barrel (Figs. 65 and 69). For acidulous waters the barrel is bushed with gun-metal. It should be thick, to admit of being bored out several times, as it is rapidly cut away by the gritty waters during sinking.

The valves are made of several thicknesses of oak-tanned leather cut into discs, tacked together, and slipping easily on a grid at the top of a cast-iron cellular ring-bucket. A perforated cast-iron guard on the grid limits the rising of the valve as the water passes through the bucket. These lifting-clacks are raised clear of their seats by the rising water, and open as

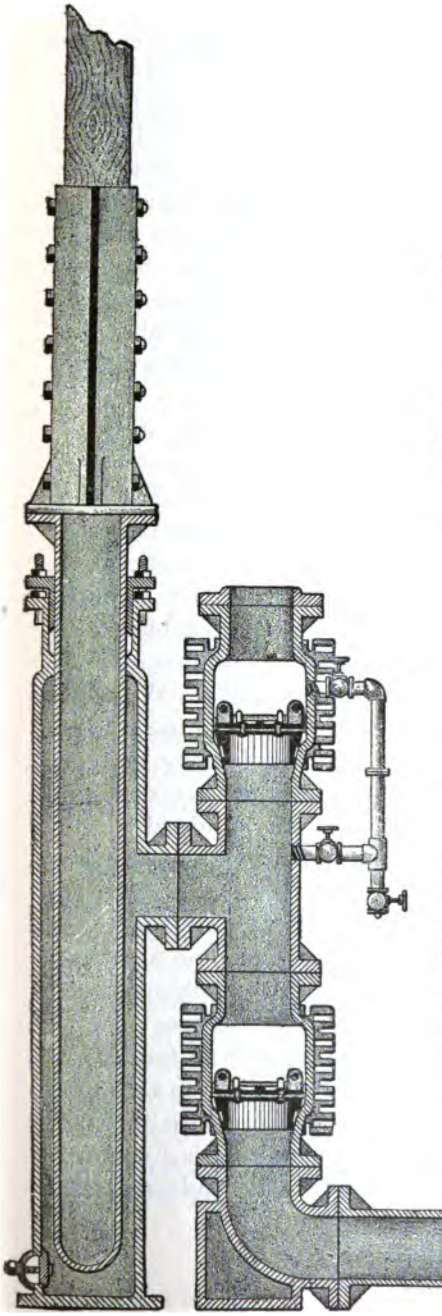


Fig. 67.

Fig. 66.

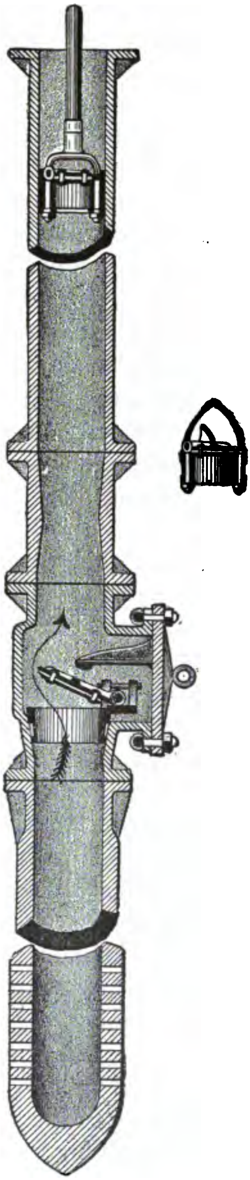


Fig. 65.

widely and shut as quickly as possible. The cellular-ring bucket casting is all there is of the piston, which fits fairly well in the barrel, and has no other packing than that offered by the leather discs forming the valve, and which are cut larger than the cylinder. The rapid movement, the wear, particularly during sinking, and the heavy pressure upon these valves, consume a set once every two weeks, or oftener. Substitutes have been suggested, amongst them flexible brass or gutta-percha plates, but they have not proven good, nor have the brass balls or conical poppets had any marked success. The valves are repaired or replaced by raising the entire pump-rod, opening the standpipe or opening a bolted door-plate in the barrel opposite the valves (Fig. 65).

Below the barrel is a length of pipe or flexible hose dipping into the sump and receiving the water through a perforated strainer. During sinking this suction-pipe must follow the lowering of the sump, and while blasting it is raised for each shot or boarded over. The flexible hose is preferable, because it can bend and adjust itself to lie on the bottom of the shaft, or hang vertically in the sump. It is of wire-wound rubber and canvas hose, which will endure considerable hard usage, and cost, for a 14-ft. length of 10 inches diameter, $65. Without this the only way to keep up with the sinking is to use a telescopic joint on the working-barrel, allowing for say 10 feet play (Figs. 66 and 69). When the water-level has receded beyond the mouth of the suction-pipe, a length is added to the stand-pipe at the surface. The working-barrel can never be more than 28 feet from the sump-level; in mountainous districts still less; at 5600 feet altitude, 23 feet; and at 10,000 feet, 18 feet. Usually the working-barrel and suction-pipe are suspended by chains from two stulls resting in the cribbing, and the stand-pipe supported at intervals by stout reachers.

The piston, or "bucket," is attached by an iron fork (Fig. 68) to a wooden rod 4 or 5 inches square, extending up through the pipe to the surface where it is connected either with one end of a walking-beam or to the piston of a single-acting engine. As it receives a tensile strain, the joints are scarfed and strapped, or, if the ends are flushed, two continuous lines of strap-iron breaking-joints are bolted together through the rod. The latter plan reduces the breakage and the number of stoppages for repairs. (Drill the bolt-holes in the iron; never punch them; and keep a good set of taps, dies, and drills for this work, also a good iron crab or winch.) A 4-inch rod is large enough for a 12-inch pipe; and a 5-inch, properly spliced and strapped, for a 13-inch to 16-inch delivery. The size of the straps is easily calculated. The area of each one, *a*, should be $d^2D \div 40{,}000$. A 200-foot pump-rod requires two straps $4 \times \frac{1}{8}$ or $3 \times \frac{1}{6}$ for a 10-inch pipe.

At the surface the column-pipe terminates in an elbow discharge or in a laundry-box and trough, the pump-rod continuing up to the framing. The mechanism by which the motion is communicated to it is simple. A stout frame, with two samson posts, supports a working-beam receiving its oscillatory motion from a pitman actuated by a crank-arm, adjustable to a 1-, 2-, or 3-feet radius, giving strokes of double this length, at the opposite end, to the pump-rod, which requires little force besides its own weight. The arm is on a shaft turned from the engine by cog, geared 1 to 6 or 7, giving 12 to 20 strokes per minute to the rod. The iron-work of this frame, inclusive of cogs, pulley, and castings, will cost about $250. The wood-work, including a 24-ft. × 15 inches square walking-beam, about $125. Whereever cog-gearing is required for heavy work, the author insists upon a solid hub if the wheel is not too large for a single casting.

FIG. 68.

To save the cost of this cumbrous framework and the loss of power, a steam-cylinder is placed over the shaft, standing vertically, its piston being bolted to a fork on the rod. This arrangement requires no framing beyond a solid foundation for the engine, and involves the purchase merely of a steam-cylinder. The piston receives steam on both sides, though, theoretically, it need only be single-acting.

FIG. 69.

It is not certain that this form gives a higher duty per bushel of coal than the drive-rod pump, for, while the friction is less, the steam consumed in the down-stroke is unnecessary, except for increasing the speed. The main objection preventing its more general adoption is the large portion of the shaft-mouth it covers. Besides, to lengthen or repair the rod or column-pipe, the cylinder must be displaced, or the additions are made below; either is slow. This pump cannot be used in slopes; the irregular wear of the cylinder on one side cannot be compensated for, nor the friction of the rod in the pipe counteracted. These cylinders are easily set, not very expensive, and work to a charm. A 12×36 cylinder, with fittings, cost $325; larger ones may also be had at moderate prices. They are known as the "Cook" (Fig. 70), or the "Wormer" pump, from the name of the manufacturers. By the name of "Bull" pump, first introduced by Wm. Bull in 1798, they are better known in collieries, where their size is greater than, and their piston-speed about the same as, the former varieties, which run best at 24 double strokes of 3 feet each, while the latter makes 6 or 8 of 10 feet each, in a cylinder as large even as 55 inches.

Where water under considerable head is obtainable at the surface, its property of incompressibility may be utilized by admitting it under the piston to raise it, after which it flows out. Unless kept well under control, it causes shocks and blows. A pressure of 57 pounds was obtained in a 50×120″ cylinder from a head of 132 feet, and 5000 gallons raised per minute 132 feet, by a 42″ plunger.

On the down stroke the rod falls through the column of water, while the valve in its piston opens and the clack of the working barrel closes. Returning, the valve's action is reversed, water rises from the sump into the working barrel, and all that above the piston is lifted a distance equal to the stroke, and a column of water simultaneously discharged at the surface.

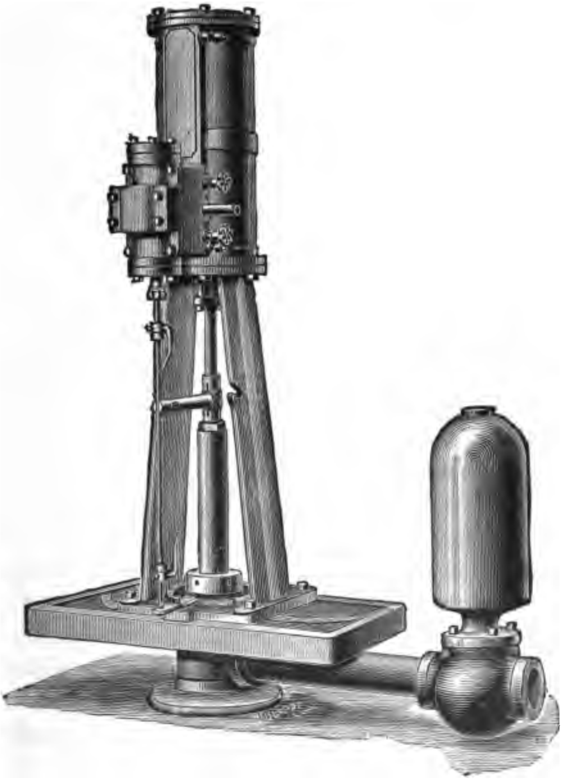


FIG. 70.

If d be the internal diameter of the pipe in inches, and L the stroke in feet, the discharge in gallons with each up stroke is $0.0408d^2L$; and the work done per minute, in foot-pounds, exclusive of resistance in the cog-gear and the mechanism for transmitting the power, is $0.34272d^2LnD$, where n is the number of double strokes per minute, and D the height of the water-column in feet. The direct-connection lift-pump wastes less power in friction, and has an efficiency of about 85 per cent. Its least working steam-pressure is represented by this equation : $k^2p = 0.345d^2D$.

It will be seen that with a moderate steam-pressure of, say, 80 lbs. the ratio between k and d must be large if the shaft is deep. Moreover, the size of the wooden rods for a long lift would have to be so large as to nearly, if not quite, close up the pipe. Hence, when the depth of the shaft has reached 250 feet the lift-pump is no longer practicable, and must be altered to a single discharge-force, or replaced by the more economic continuous flow steam-pump.

36. In converting the lift to a force-pump, the pump-rod is removed from and works outside of the stand-pipe, the lower end of which is bolted to one end of a cast-iron H-chamber, the other stem of which carries a long working-barrel, into which plays a solid plunger-piston (Fig. 67), instead of the bucket-valve of the lift-pump. Below the stem carrying the stand-pipe is the suction-pipe, and in it are two clacks opening upward. The H-piece is set on a rigid foundation. Then during the up stroke the lower clack opens, while water rises through it into the working-barrel; as the plunger falls it drives the water through the upper valve against the column of water in the stand-pipe. As sinking progresses and the sump lowers the suction distance is exceeded, and a suction-barrel is added at the bottom and carried with the lowering of the sump, while additional lengths of pipe are attached between it and the H-piece until a lift of 300 feet has been reached, when the suction-barrel is removed, to do service similarly for the succeeding lifts, and is replaced by another H-piece.

Another mode of converting the lift into a force consists in retaining the working barrel and suction length formerly used, and disconnecting the stand-pipe, which is rested on a water-tight tank, into which the water, lifted from below, pours through a goose-neck at the top of the barrel, and from which it is forced to the next station above. During sinking the lengthening of the pipe between the working-barrel and tank, keeps pace with the lowering of the sump until another full lift of 300 feet has been attained, when the goose-neck and all below it are replaced by the H, or tank (Fig. 73.)

An H-piece and a plunger working-barrel are placed at every station, beginning a lift of 300 feet (Fig. 73). The H-piece has a door-plate opening into it opposite each valve. The valves are poppet or hinged, and on account of the heavy pressure on them—the upper one especially—they are of iron with leather washers below. The H-piece depicted in Fig. 71 allows of the heel of the valve raising whenever chips or pebbles are caught under it. The working-barrel is about 15′ long and at the top has a stuffing-box, through which works a cast-iron piston-plunger carefully turned, and of a length greater than the stroke. Lifts are rarely over 350 or less than 150 feet in height. Their number increases the first cost, friction, and repairs, but permits of greater speed and more strokes, hence smaller pumps.

Excepting the short suction lift-pump at the bottom, all the column-pipe is in one continuous line, broken only for the introduction of H-pieces or tanks at the stations. The discharge at the surface is into a laundry-box and trough. The thickness of cast-iron pipe, in inches, must be

$0.00009Dd + 0.34$; and of wrought-iron, $0.000025Dd + 0.12$. These give, for a 10″ pipe of 300 feet depth, 0″.61 and 0″.2, respectively; their weight, 17.3 and 6.3 pounds per foot.

The pump-rod extends in one continuous line down the shaft, outside of the pipe, terminating at the bottom in the piston of the bucket-lift. At intervals are offsets, to which are bolted the rods carrying the plungers. Wings, i.e., strong timbers, clamped by iron collars, are also attached to the sides of the rod. On either side of the rod, in close proximity to it, reaching across the shaft, are pairs of stout stulls, to catch the wings in case of accident, and to serve as stays to prevent buckling of the rod. The size of the rod is only a question of the strength of materials. It is in tension due to the weight of the rod below, and subject to the compression of the weight

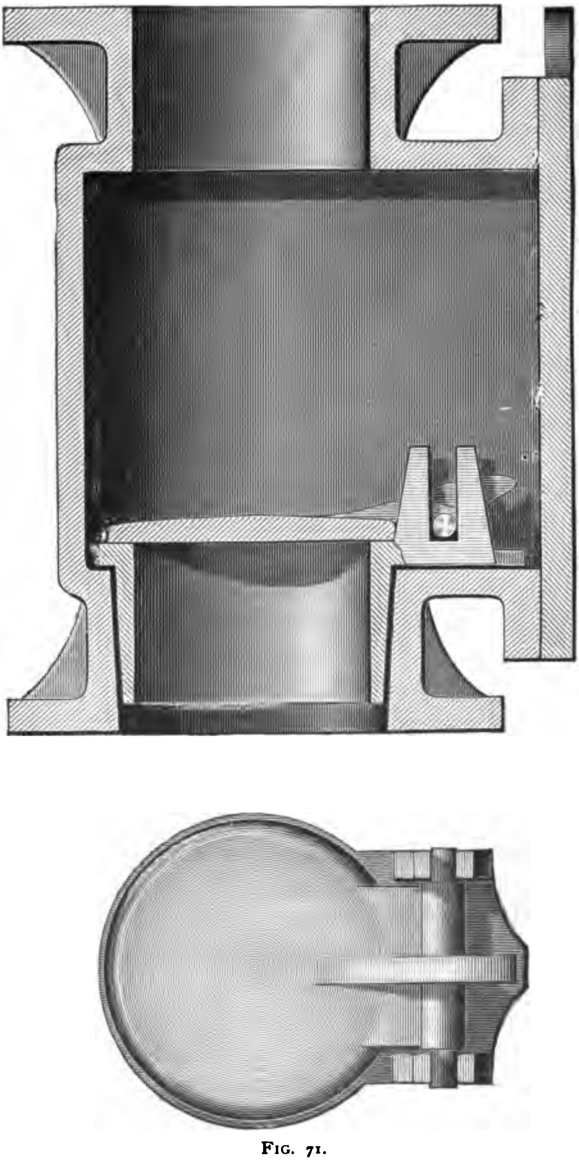


FIG. 71.

of the water to be forced up. Its cross-section is al...
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plus that of its own lift. Hence it tapers to
an instance of the size of a rod,—that of
deep,—we find the first 780 feet down was
to 12″ X 24″; at 864 feet it was
square at 964 feet: thence

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38. Such is the mechanism and ... force pump known as the Cornish ... and its speed regulated, it gives ... reliable and also the most ex... numerous advocates as again... planting the system to ... deeming feature—the ... ing to the worst—th... cal direct-acting en... great improvem... bob; but it wa... was quickly r... plete, with t... for a 600-f... gallons, a...

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...ways pro- ... that point ... bottom. As ... 2300 feet ... tapering ... tapered to 14 ... inches to the bot- ... rod is to ... 13 and 12 ... timbers ... been tried ... rated air rots the timber rods having ... wrought-iron rods ... shafts the rods fall freely by their own weight; ... friction-rollers placed about thirty feet ... wooden rods. When iron ropes are used instead of ... slope may be provided ... support them. Changes in the slope ... A chamber is cut in the shaft ... the use of a rocking-arm. A frame, on which swings a ... angle in which is firmly set a ... the two arms of the ... bob by a hinge-pin at the apex. To the ... the inclined rods are attached. While this arrangement ... not desirable because of the expense and the loss of power ... it is the best to be had when slopes are sunk on contorted ... veins.

The rod is connected at the surface to one apex of a king-post truss balance-bob. The horizontal beam is about 25 feet long, with a saddle and axle underneath near the centre. At the upper end of the king post, which is 8 feet high, is the connecting-rod to the engine. Besides the braces on each side down to the beam, are a pair of tie-rods, taking with the braces on each side alternately tension and compression. All the members of the frame are of wood or iron, in iron shoe castings at ends. The frame stands vertically, in a pit dug alongside the shaft, 8 or 10 feet deep (Figs. 10 and 72). The ... motion is communicated to the bob by a connecting-rod ... crank from a wheel geared to the fly ... of which, during the up-and-down ... the bob and its counter... ...ied with a ...

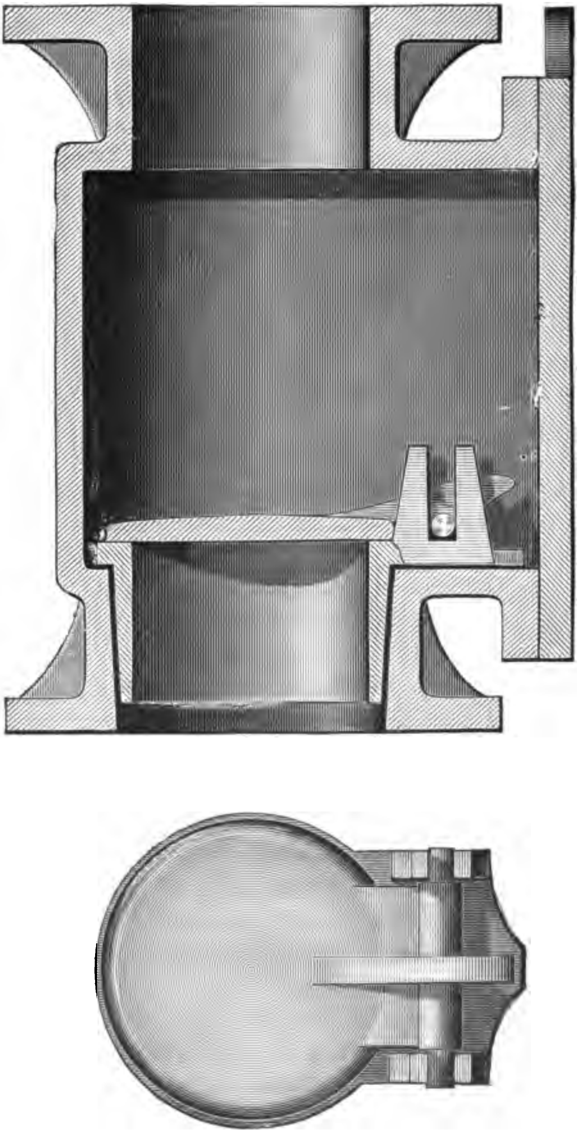


Fig. 71.

of the water to be forced up. Its cross-section is always proportional to the weight of the number of lifts below that point plus that of its own lift. Hence it tapers to the bottom. As an instance of the size of a rod,—that of the Maira, 2300 feet deep,—we find the first 780 feet down was 16″ × 32″, tapering to 12″ × 24″; at 864 feet it was 16″ square; it tapered to 14″ square at 964 feet: thence it was 13 and 12 inches to the bottom. Except where vitiated air rots the timbers fast, oak is to be preferred as material, wrought-iron rods having been tried without success.

In vertical shafts the rods fall freely by their own weight; in slopes they rest on friction-rollers, placed about thirty feet apart. When iron ropes are used instead of wooden rods, sheaves support them. Changes in the slope may be provided for by the use of a rocking-arm. A chamber is cut in the shaft at the angle in which is firmly set a frame, on which swings a V bob by a hinge-pin at the apex. To the two arms of the angle the inclined rods are attached. While this arrangement is not desirable because of the expense and the loss of power, still it is the best to be had when slopes are sunk on contorted veins.

The rod is connected at the surface to one apex of a king-post truss balance-bob. The horizontal beam is about 25 feet long, with a saddle and axle underneath near the centre. At the upper end of the king-post, which is 8 feet high, is the connecting-rod to the engine. Besides the braces on each side down to the beam, are a pair of tie-rods, taking with the braces on each side alternately tension and compression. All the members of the frame are of wood or iron, in iron shoe-castings at the ends. The frame stands vertically, in a pit dug alongside of the shaft, 8 or 10 feet deep (Figs. 16 and 72). The rocking motion is communicated to the bob by a connecting-rod, operated as a crank from a wheel geared to the fly-wheel shaft of the engine, the work of which, during the up and down strokes, is somewhat equalized by the bob and its counterpoise. The third apex of the triangle is occupied with a box full of iron and bowlders, to counterbalance the excessive weight of the rod; for it will be found that the weight of the long column-rod of a strength requisite to force the water up is much greater

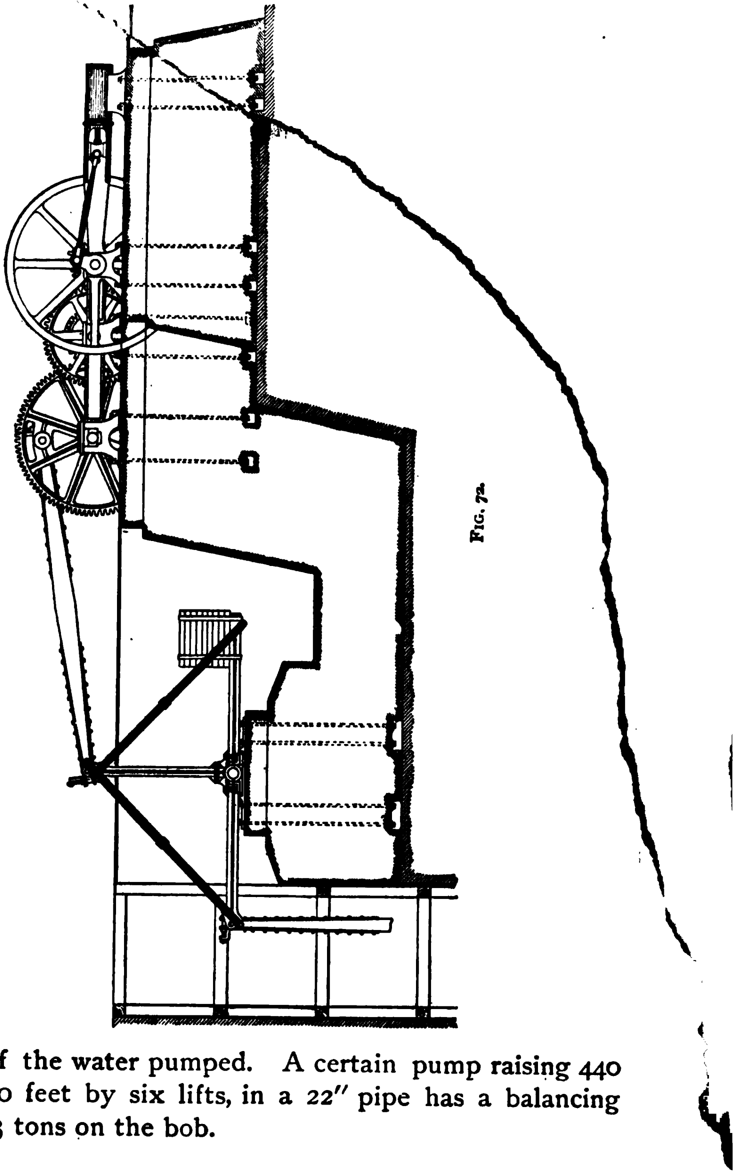


FIG. 72.

than that of the water pumped. A certain pump raising 440 gallons 1690 feet by six lifts, in a 22″ pipe has a balancing weight of 33 tons on the bob.

In mines utilizing the pump-rod for a man-engine additional counterbalance-weights are connected at intervals down the shaft (Fig. 73). Sometimes two lines of rods are used in a shaft, working two pumps from the same bob, in which case no counterbalance is needed. All the foundations about the shaft should be carefully laid: condensed steam and pump-water soon makes the ground yielding. In stable ground, heavy beams buried in the stone will suffice; in ground at all soft, a secure foundation can only be obtained by concreting a considerable area, for 6 feet deep, and erecting a rigid timber- or brick-work base several feet high (Fig. 81).

38. Such is the mechanism and construction of the lift and force pump known as the Cornish pump. When once placed and its speed regulated, it gives little trouble. It is the most reliable and also the most expensive pump in use. It has numerous advocates as against the steam-pumps; but in transplanting the system to America we discarded its sole redeeming feature—the cataract-engine—while persistingly clinging to the worst—the cumbrous bob and rod. When the vertical direct-acting engine was introduced it was thought to be a great improvement, because of the suppression of the heavy bob; but it was soon discovered to be a mistake, and the beam was quickly re-established. An engine, boiler, and fittings complete, with three 15″ plungers and one 16″ lift-pipe, etc., etc., for a 600-foot shaft, weighed 650,000 lbs., had a capacity of 800 gallons, and cost in place $54,000.

The behavior of the pump may now be easily understood. Consisting of a suction-length and a series of force-pumps, one above the other, the water is driven by stages to the surface (Fig. 73). The engine raises the rod, and water is sucked up into the working-barrel at the bottom while that above the bucket is lifting to the first station above. Here the plunger is drawing water up through the lower clack into its barrel; likewise all the other plungers. At the end of the stroke (6 to 10 feet) occurs a slight halt, incidental to the change of direction. The rod falls by reason of its own weight, and each plunger closes the lower clack in its H-piece, opens the upper one, and forces

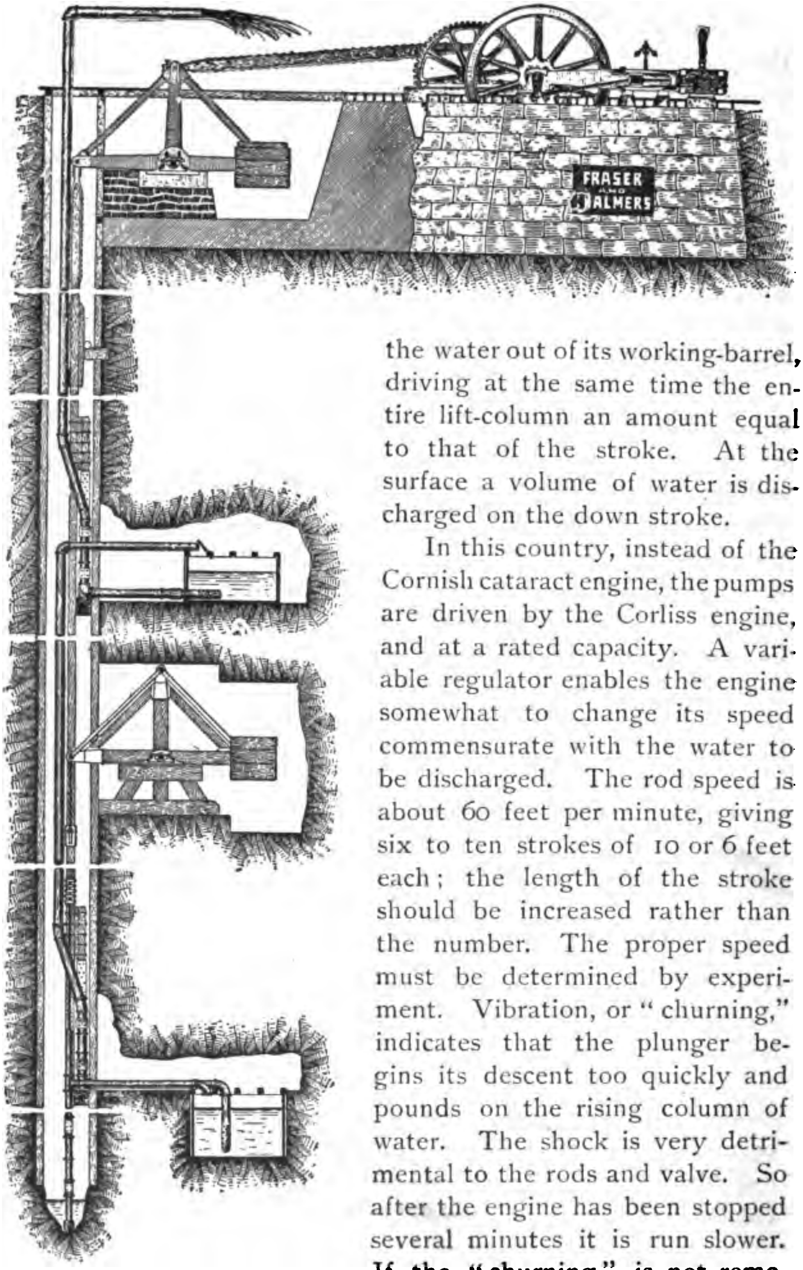


Fig. 73.

the water out of its working-barrel, driving at the same time the entire lift-column an amount equal to that of the stroke. At the surface a volume of water is discharged on the down stroke.

In this country, instead of the Cornish cataract engine, the pumps are driven by the Corliss engine, and at a rated capacity. A variable regulator enables the engine somewhat to change its speed commensurate with the water to be discharged. The rod speed is about 60 feet per minute, giving six to ten strokes of 10 or 6 feet each; the length of the stroke should be increased rather than the number. The proper speed must be determined by experiment. Vibration, or "churning," indicates that the plunger begins its descent too quickly and pounds on the rising column of water. The shock is very detrimental to the rods and valve. So after the engine has been stopped several minutes it is run slower. If the "churning" is not remedied, look to the valves—they may not be giving a free flow. The valves should afford unobstructed passage to the water in one direction, and close perfectly in the other,—two antagonistic conditions, which can be attained only partially. The strain on the valves is enormous, and if tried too hard they become weak, do not work properly at either stroke, and lose water by their "slip." If the valves play all right and the vibration still continues, it may be because the joints of the rod have a backlash, or the bob requires resetting. This vibration must be particularly guarded against if the rod is also to be utilized for a man-engine. Arranging the rod so as to be always in tension may remedy the difficulty. For convenience in repairing, the pumping compartment should be large enough for a ladder-way, with plats and chambers.

39. There have been a number of attempts to make a double-acting pump, retaining therewith the advantages of the Cornish. Its use would save space in the shaft, the pipes for a continuous discharge occupying less than one fourth the area of a single discharge-pipe and its rod.

Cushier's system of pumps for deep mines consists in having sets of two pumps, each working in concert, one above the other, the suction and discharge pipes being common to both pumps.

The pumps are placed at intervals of about 200 feet in the shaft, the power being transmitted directly through the centre of the plungers. The connection with each other and to the motive power is effected by means of a steel-wire cable, encased in wood, preventing it from external wear as well as from rust. This cable is fastened to, and its length regulated by, shackle-bolts.

The plunger of the lower pump, in a set, is double in area that of the upper one, so that in working on the upper stroke one half the water raised fills the chamber of the upper pump, the other half being forced out through the discharge-pipe on the down stroke; the upper pump-plunger forces out, in its turn, the water in the chamber, thereby causing a continuous delivery.

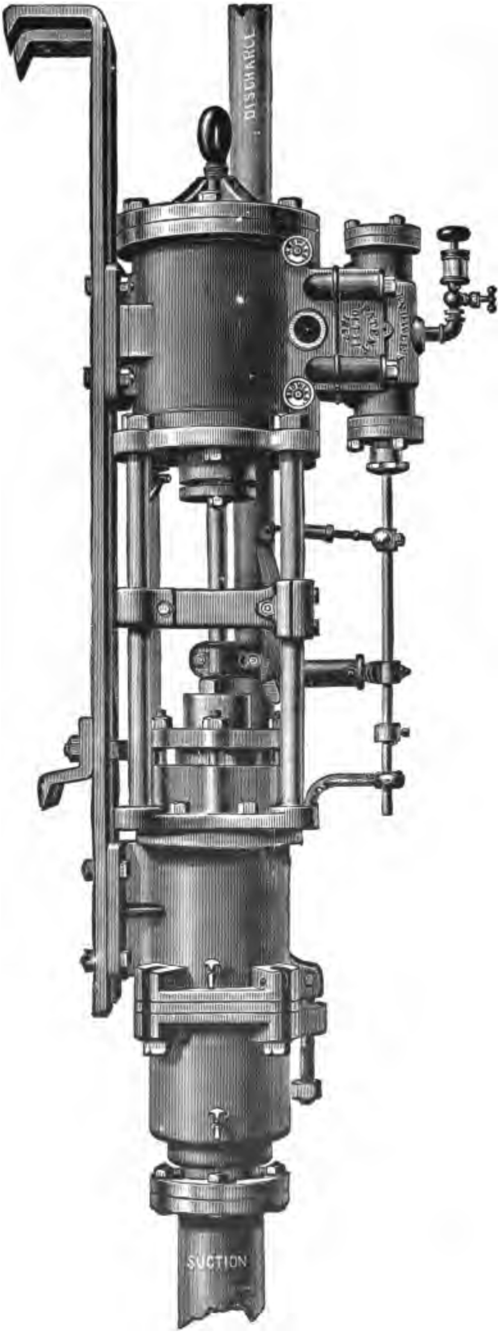


Fig. 74.

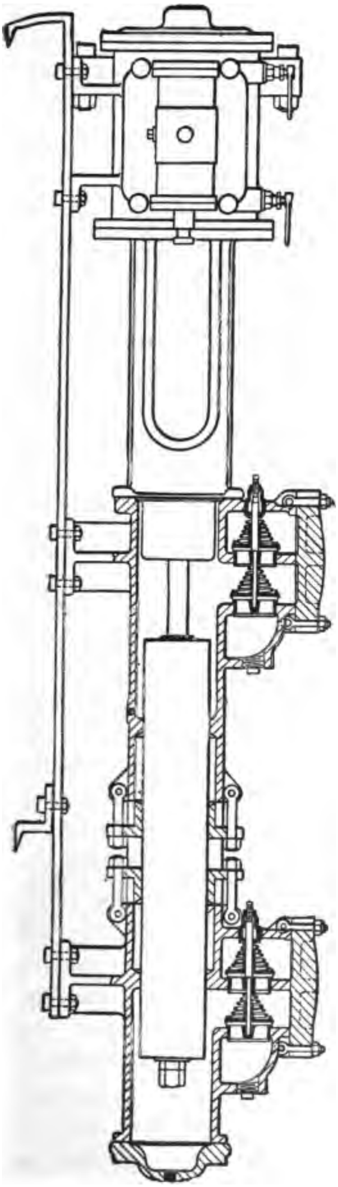


FIG. 75.

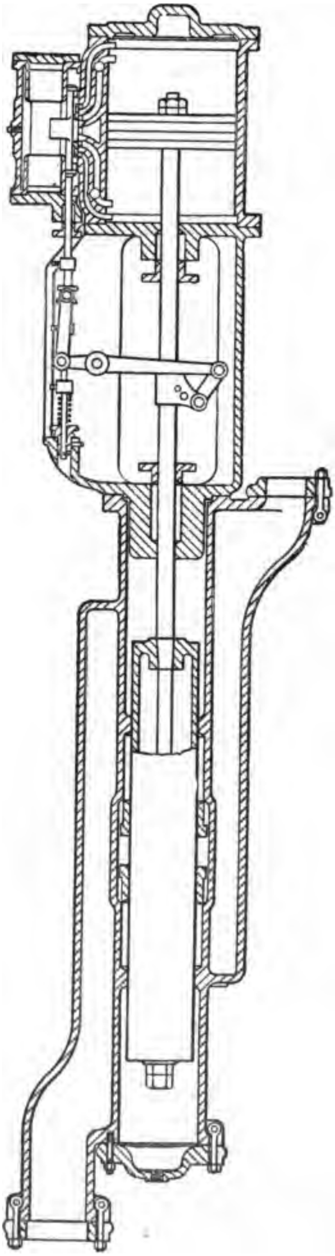


FIG. 76.

This form of pump can be worked at any angle, to any depth, and is almost perfectly balanced. The last-named advantage enables it to be connected with, and worked by, a direct-acting steam-cylinder, and thus do away with the complicated gear and bob of the Cornish.

The sinking-pump is on this plan, but not so complex for sinking or recovering mines. It is operated, hanging vertically from some support, by a chain to the bale attached to the steam-cylinder, and taking the suction in at the bottom of the pumps. By this suspension it is able to be accommodated to a varying water-level, and is used while sinking. A double-acting, centre-packed plunger is directly connected with the steam-piston, from which it receives its power, as in the horizontal pump (see Figs. 74 to 76). Cameron, Knowles, and Deane have sinking-pumps of similar pattern. The valves are absolutely positive, and are protected by a cast-iron shield serving as a yoke between the steam and water ends, while those in the steam end are cushioned to regulate the strokes. Hand-hole plates, with hinged bolts, allow of easy repair of the valves and shoes and dogs of easy handling.

The steam-pump dispenses with the cumbrous gear, bob and rod having instead a small, well-lagged steam-pipe, conveying the power down from a surface boiler. It is essentially on the same principle as an air-compressor (Fig. 77), consisting of a steam-cylinder in which a piston oscillates, and by rockers moves its steam-valves without the aid of any rotary appliances; at the same time it reciprocates a solid plunger centrally in a water-cylinder, at each end of which is a set of double-beat valves of appropriate construction, open only so long as the water is being forced, and closed at once without the aid of springs. One valve is removed in the figure. In large pumps double sets of inlet and discharge, brass-covered, single-beat, rubber valves are used, closing with a spring. Where great pressure or the gritty nature of the water renders the use of the single piston undesirable, the water-cylinder is divided in the centre, and a pair of plungers, discharging alternately, work in the opposite ends, and are connected with

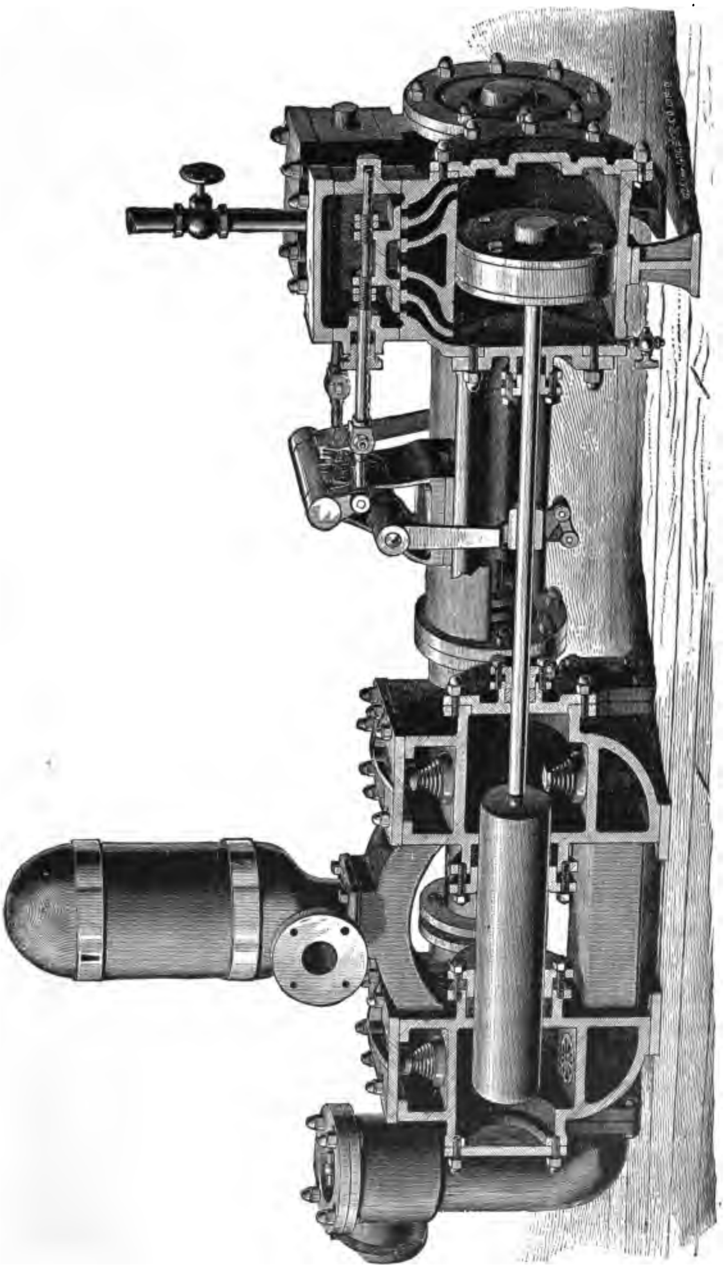


FIG. 77.

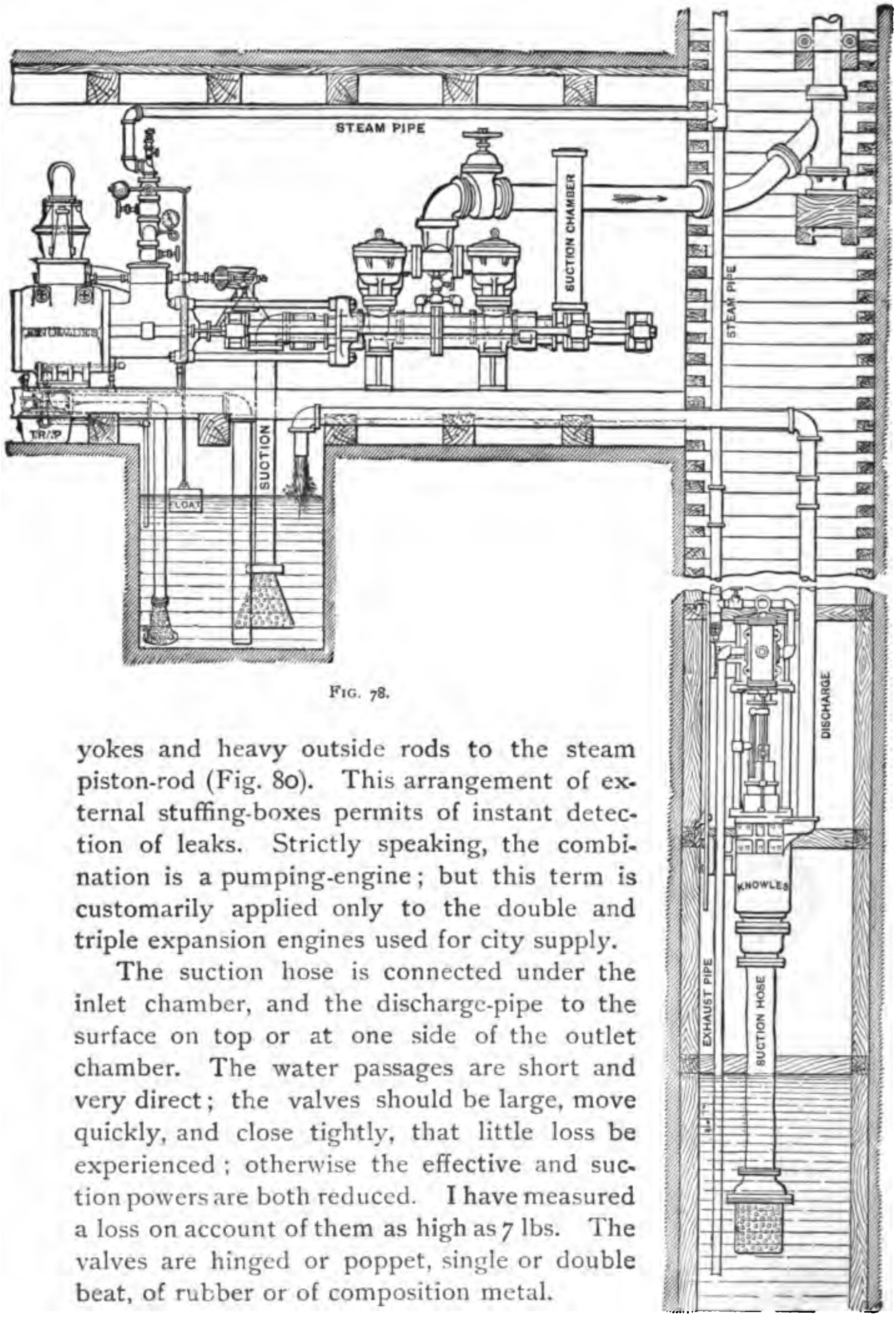


FIG. 78.

yokes and heavy outside rods to the steam piston-rod (Fig. 80). This arrangement of external stuffing-boxes permits of instant detection of leaks. Strictly speaking, the combination is a pumping-engine; but this term is customarily applied only to the double and triple expansion engines used for city supply.

The suction hose is connected under the inlet chamber, and the discharge-pipe to the surface on top or at one side of the outlet chamber. The water passages are short and very direct; the valves should be large, move quickly, and close tightly, that little loss be experienced; otherwise the effective and suction powers are both reduced. I have measured a loss on account of them as high as 7 lbs. The valves are hinged or poppet, single or double beat, of rubber or of composition metal.

These pumps operate under the direct action of the steam, the pressure of which may be within limits increased. The ratio between the areas of the steam and the water cylinders somewhat exceeds that between the height of the water-column and the steam-pressure; but when the lift becomes so great as to require a boiler-pressure of over 100 lbs., the pumps are employed in relays, each pump delivering to the tank above. Fig. 78 shows the disposition and arrangement.

Where k and c are respectively the steam and water cylinder diameters, $k^2p = 0.651Dc^2$ represents the equilibrium of the pump.

The piston-speed may be altered as desired, though the standard running rate is 100 feet per minute. It certainly would be desirable to operate at a higher rate, if it could be obtained without "thug." In the Lake Superior region it is higher, a speed of 200 feet being reported, in isolated cases, against heavy pressure.

The number of gallons, G, discharged by the pump per minute is $0.0408c^2s$ (s being the piston-speed in feet). The indicated horse-power is measured by $0.000253GL$, where L is the height of the lift.

The commoner makes encountered in the United States are the Knowles, Blake, Rogers, Cameron, and Worthington, differing in the form and solidity of construction. The Knowles is the most popular. The finest in the Lake Superior districts is an 18×96 compound duplex for heavy service. A 12×20×24 is equal to 1,000,000 gallons daily.

40. For general purposes, these direct-acting force-pumps are coming into almost universal use. Their chief feature is their equal efficiency slow or fast; they are capable of quick adjustment in speed and discharge, as emergency demands; but they require close watching, especially where the water is "quick" or they may be drowned. The Cornish pump does not admit of variations in its rate: during summer and winter it is run only a few hours during the shift to empty the sump which has been filling overnight. The small cost, great simplicity, and ease of repairs give the steam-pump an important advantage. A plant with boiler, pipe, and fittings, complete, can be installed for less than one fifth that of a Cornish outfit. One for 850 gallons per minute, 400 feet, cost $15,000 in place.

As their maximum suction length is 28 feet at the sea-level, these pumps are placed on timber seats, in a large, well-timbered excavation alongside of the shaft and near the sump level, which must practically be invariable. They are useless during sinking without a sinking-pump to deliver to their tanks. In coal-mines, and where the machinery can be established for a permanent bed, these pumps have no rival (especially if compounded); whereas in vein-mining the pumping apparatus, and indeed all of the machinery, is continually being planned and arranged with a view to further continuation. For this reason metalliferous miners are compelled to choose between a set of relays of direct-acting pumps at each 200 or 300 feet, with a sinking-pump at the bottom, and the Cornish pump with its several force-stations and its bottom-lift.

The principal difficulty is with the disposal of the exhaust. If turned off into the sump (Fig. 78) or air-way, the temperature of the mine is raised, ventilation is injured, and the timbers ruined; if carried to the surface, the condensation in the pipe gives trouble. The best remedy is to use a condenser, which reduces the back-pressure and increases the efficiency. Jacketed compound or condensing steam ends can easily be connected to these pumps.

But whatever the fuel-saving appliances, the direct-acting steam-pump does not equal the Cornish cataract pumping-engine, though the introduction of the fly-wheel and compound cylinder gives a good approach to it. A fly-wheel is really necessary in order to secure the full benefits of a high degree of expansion, which, as has been seen on page 66, is not feasible in one cylinder, because the resistance (the weight of water forced up) is always the same throughout each stroke. A fly-wheel distributes the steam-power excess of the first part of the stroke to the latter. By the use of the compound cylinders high pressure and expansion are carried on simultaneously throughout the entire stroke, and the saving of steam-power may be fully 30 per cent. See Fig. 79 for compound

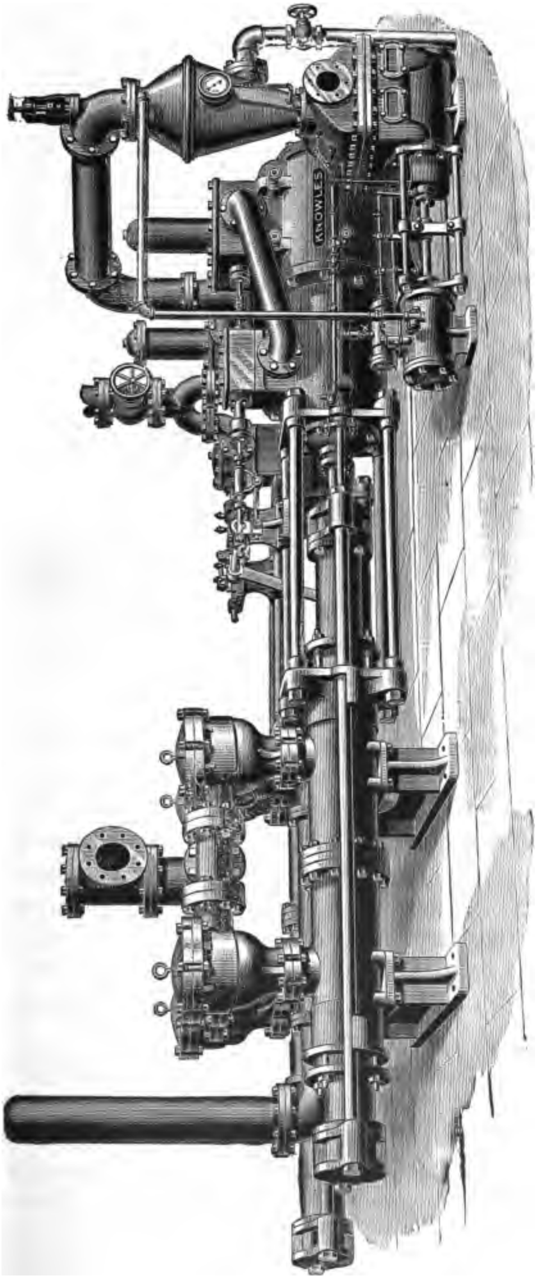


Fig. 79.

steam-pump. Two duplex are illustrated—the Deane (Fig 80) and the Knowles (Fig. 79).

The calculating of the steam and fuel economy is easily made; the necessary elements are few innumber, no assumptions are involved. The standard of comparison of the work of a pump is the number of million foot-pounds of work actually performed per bushel (80 lbs.) or per 100 lbs. of coal. The combustion of one pound of anthracite gives sufficient heat to, theoretically, do 12 million foot-pounds of work. The ratio between this and the work actually done measures the efficiency or "duty," to the consideration of which in and about mines insufficient attention has been given, notwithstanding its pecuniary importance. The duty of a small pump is from 7 to 15 million foot-pounds per 100 lbs. coal; a compound gives from 15 to 30 millions; while the higher types of pumping-engines furnish from 30 to 100 million dynamic units, corresponding to the consumption per hourly horse-power of 28 to 13, 13 to 6.6, and 6.6 to 2 lbs. coal, respectively. To find the consumption of coal per hourly horse-power, divide 198 by the duty (in millions). A recent report of a Worthington engine having a capacity of over 1000 gallons per minute, against an equivalent of 2000 feet head of water, showed a duty of 184 foot-pounds per thermal unit, or 158,000,000 dynamic units per 100 lbs. of coal.

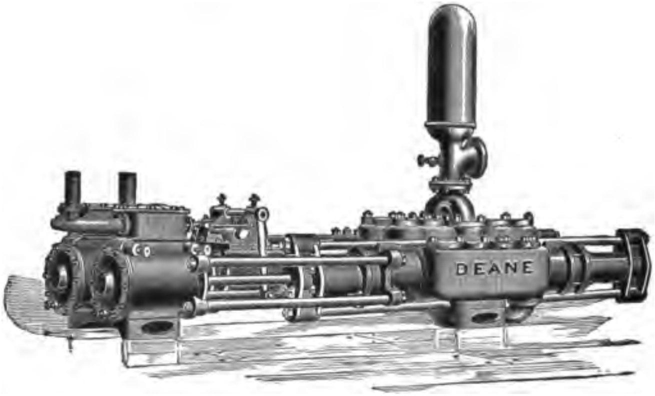


FIG. 80.

To illustrate the influence of compounding and jacketing the steam-cylinder, and of condensing the exhaust, upon the coal bills, two examples will suffice. As has been stated, many of the collieries pump 4000 gallons of water per ton of coal hoisted. To raise this only 300 feet requires the consumption, theoretically, of 336 lbs. anthracite for a daily output of 400 tons. If the duty be 90 million, as in Cornwall or by a Knowles duplex compound, or 20 million, as with our average pumps, the aggregate yearly consumption is 675 and 3005 tons, respectively, or of 900 and 4000 tons of lignite.

But duty is not the sole feature of a piece of machinery: the repairs and lubricant accounts and the durability of the plant are not to be overlooked; for the indicator-card is a less valuable guide than are the coal, oil, and packing bills. Moreover, the question of the comparative value of the inconveniences in the use of steam underground with those of the occupation of a shaft compartment by rods, catches, etc., has some monetary value. The cost of pumping a million foot-pounds is about 1.6 cents with the direct-acting pumps, and 2.5 to 2.9 with high-pressure rotative engines.

Rotary pumps, or those of the crank and fly-wheel type, are not popular among mining men, though a uniform velocity is maintained and the gain in power compensates for the increased initial cost in the proportioning of the engine.

Water-pressure engines are coming much into vogue for utilizing the force from some neighboring stream of water. Reservoirs are established at some distance—400 or 500 feet—above the shaft mouth, and connected by pipes to the engine, the piston of which receives the pressure and communicates its power to the rods of a Cornish pump (Fig. 81); or the engine may be direct acting. In the latter form the hydraulic engine does a fair duty, and as it works equally well under water, cannot be drowned out as can the steam appliance. 280 miners' inches of water falling 100 feet will raise 80 inches 200 feet, or 40 inches 400 feet. Mines having both shaft and tunnel communicating with the surface are admirably convenient to the employment of the hydraulic engines, which may be placed in

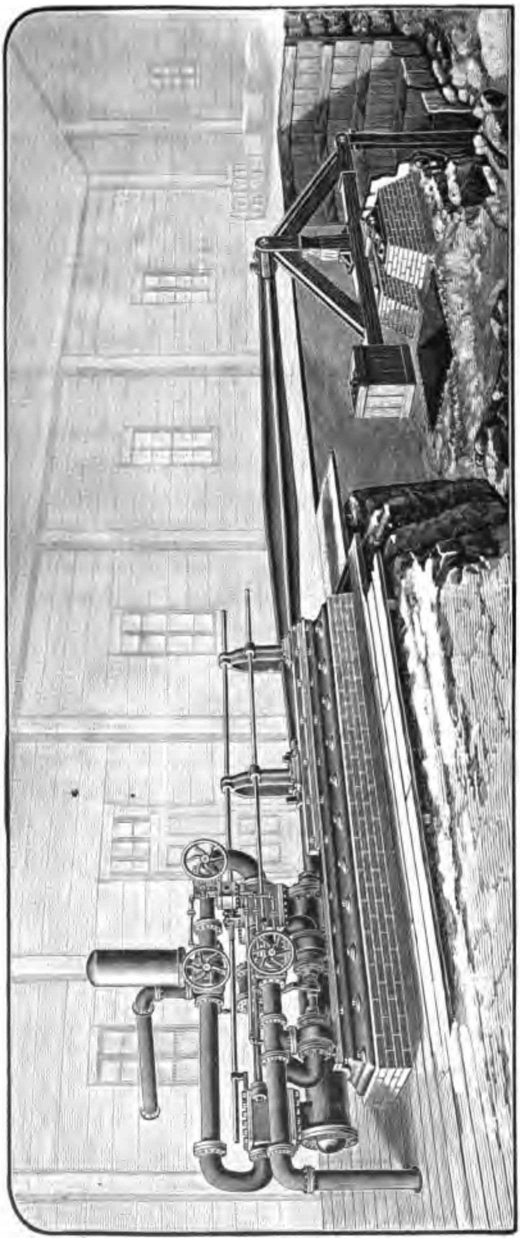


FIG. 81.

the tunnel-level and receive from the surface at the shaft mouth the water-pressure under a great head and discharge it and the pumped water through the tunnel. This is the method at the Comstock mines. The great trouble with these engines is in the valve-gear, which is more complex, and must be carefully balanced, for an incompressible fluid like water, than is necessary for an elastic fluid like steam or air. The cutting off of the valves produces a not inconsiderable concussion in the inelastic fluid which is entering the cylinder at a high velocity, and unless delicately manipulated they fail to operate.

The difficulty in converting electric-motor rotation to reciprocating motion militates against electric pumps for the present.

The windmill has been suggested as a motor for unwatering mines. An experiment on Musquito Pass resulted, three months after erection, in a collection of bric-a-brac for three miles down the gulch.

For limited supply the siphon has done service. One need not dwell upon the principle, except to give the rule. The level of entry must be above the level of delivery, and cannot be over 34 feet below the highest point of the bend; the distance between the delivery and entry may be anything. This takes no account of friction, etc.

CHAPTER XI.

VENTILATION.

41. Laws regarding the ventilation of mines; output dependent upon the hygienic conditions; division of the subject into three branches; the gases encountered in mines, carbonic acid, sulphuretted hydrogen, carbonic oxide, and firedamp; their physiological effects; how evolved, where accumulated, and how removed. 42. Treatment of asphyxiated persons; effect of the gases upon lamps; modes of testing for firedamp; Hepplewite-Gray tester; Shaw's apparatus; explosions; after-damp; influence of the barometric changes upon the evolution of gas; the sole means of obtaining security. 43. Consumption of air by combustion, blasting, etc.; dilution of the products of combustion; volume of air required in a mine for man, light, beast, powder, and extent of working-face exposed; allowance necessary for drag and friction; physical laws of the movement of air. 44. Relation of the volumes and pressures of gases; effect of temperature and the release of pressure upon the movement of air; calculation of the velocity of wind, of motive column; use of the water-gauge and interpretation of the different modes of measuring air; the ventilation paradox.

41. FROM an economic as well as a humanitarian point of view, the ventilation of a mine is a matter of very great concern. Neither shaft nor tunnel can be carried more than 200 feet beyond an opening without some special means of stirring and freshening the stagnant air. The men should not be compelled to work in the hot atmosphere of a stope or room vitiated by the variety of gases given off from and by coal, powder, lamps, respiration, rotting timbers, and decomposing ore. These cannot support combustion, or be inhaled with impunity; and any air conveying them is impoverished because of the absorption of its free oxygen, as well as by reason of the presence of these gases. In coal-mines. additional peril accompanies some of these gases, which with the air form explosive mixtures, the which, bursting into flame, destroy everything in their path and emit dense volumes of poisonous fumes that are fatal to all who have escaped the shock.

Only during recent years does the life and energy of a miner seem to have been acknowledged as having a pecuniary value. The statutes are becoming more and more rigorous in the insistence of safety and hygienic measures. They specify the methods by which the noxious gases may be made harmless, and ordain inspectors with sufficient authority to suggest needed improvements and to punish non-compliance. Not only are the miners benefited by the diminishment of risk, but also the operators, who profit in the increased energy of men working under favorable circumstances. The illumination is better, smoke clears away quicker, and the men are invigorated.

This subject naturally divides itself into three branches: the various gases encountered, the means of producing an air-current for their dilution, and the mode of its distribution. There is a marked contrast between the requirements of vein-mines and coal or other mines in flat beds, which usually have two shafts connected by a labyrinth of workings nearly on one level; for the former trust to natural means, and obtain it by the winze connections of the different levels. This is inadequate, as the British Commission declares that the mortality in well-ventilated coal-mines is less than that of metal-mines, for which it recommends artificial ventilation. Metal-miners should remember that the inflammable gas is not the only one to be guarded against. The cloud of dust in the air and the carbonic-acid gas are insidious causes that destroy the health of the miner, and cause as many deaths (miner's consumption) as do the fearful explosions in an equal number of coal-mines.

In ore-mines carbonic acid and sulphuretted hydrogen are the only gases met with. Both are heavier than air, and will be found near the floor. For this reason all box-pipe supply ventilators should be carried along the roof.

Carbonic acid, CO_2, "black-damp," or "choke-damp," at normal pressure, weighs 124 pounds per 1000 cubic feet, as against 81 for an equal volume of air. It is the common exhalation from animals, timber, and coal, and is indicative of combustion. Air containing 2 per cent of this gas produces an overwhelming depression upon those breathing it; with 6 per cent lights will not burn; and 10 per cent is positively fatal. As it accumulates in the lower part of idle workings, or wherever the air is stagnant, no man should venture without previously testing, by a candle, the condition of the atmosphere. If the light is extinguished, the poison must be swept out by a strong current through the place; or in shafts the circulation is started by violently raising and lowering a bundle of hay in it, which then should be ignited. The introduction of absorbents, like ammonia or quick-lime, may rid the place of the gas. The amount of carbonic acid may be judged by the milkiness produced in baryta hydrate by the passage of a given volume of air.

Sulphuretted hydrogen weighs 96 pounds per 1000 cubic feet. It is said to be extremely poisonous; a breath of it certainly is debilitating. As it is the result of the decomposition of pyrites, always accompanied by heat, a pyritous coal must be watched against any tendency to spontaneous combustion. The rotten-egg odor is depended upon for the detection of the gas, as a candle will burn in a mixture of it with air.

In addition to those named, coal-mines are troubled with emanations, sudden or continual, of carbonic-oxide and marsh gas, CO and CH_8, known as "white-damp" and "fire-damp." Being lighter than air, they are to be looked for in the upper portions of workings, and when present in quantities cause suffocation.

The existence of the former is doubted: first, because it is an unstable compound; secondly, because it only results from secondary combustion. Free air over ignited carbon produces CO_2; this in the presence of an excess of carbon is converted to CO. Only a high temperature during the formation of coal or its combustion could produce this gas. Thus in the ultimate analysis of coal CO is obtained; in natural gas, the distillate from organic matter, there is 0.5 per cent; and in the gas of a Siemens generator there is 20 per cent. It is only barely possible that it might be found in the goaf, where the oxidation of the pyrites, or absorption of oxygen by the coal or fine dust, gives rise to a sufficiently high temperature. The author refuses to admit its presence in mines.

Fire-damp has long been recognized as a constituent of the gases ejected from below. It escapes at various springs and salt-mines; it has fed the sacred fires of Baku and the mud volcanoes of Bulganak; in the Silver Islet mine, in iron-mines of Alsace, in lead-mines in Tuscany, and at Darley explosions and fires have occurred from fire-damp; inflammable and explosive gas was met with in driving the lake tunnel at Chicago; it constitutes from 40 to 90 per cent of the natural gas, and is obtained among the volatile and combustible constituents in the ultimate analysis of coal. While sinking shafts through porous strata fire-damp has been encountered, and precautions are therefore necessary in regions of natural gas.

This gas is a stable, never-failing constituent among the products of dry distillation of organic matter. Strictly speaking, it is not a single gas, but is intermixed with others. 1000 cubic feet of it weigh 45.22 pounds. It is occluded in the coal, diffused through its pores, collected in crevices or cavities, or stored up in reservoirs. From these it continually exudes, with more or less violence; falls of roof, low barometer, squeezes, or creeps liberate it in volumes. An approach to old workings, which are reservoirs for damp, always inspires an uneasiness, for when tapped the sudden outburst of gas is often disastrous. From the report of the Royal Commissioners on accidents in mines, 1886, it will be seen that pressures of 200 to 450 pounds per square inch were observed. This gas is the most dreaded enemy in coal-mines, not only because of the large or uncertain volume that may be encountered, but because of its deadly and inflammable nature. Even if it occurred pure, without any CO_2 or N mixed with it, it could not support combustion; neither light nor man could survive in its atmosphere. This insidious gas gives off no warning of its presence, as it accumulates in a nook or under a platform until some untoward circumstance brings it into contact with a naked light. The slight hiss accompanying its exudation is hardly enough to be distinguishable. These "blowers," of all sizes, up to the outbursts that for a time overpower the ordinary ventilating current, contain 90 per cent of marsh-gas, and may be liberated anywhere and at any time. To escape the consequences of its inflammable and explosive nature a variety of devices, chemical and mechanical, have been tried; but no methods have yet been found feasible other than a thorough ventilation to "drown out" the enemy, and the use of safety-lamps.

42. Persons asphyxiated by these gases may be revived by blowing oxygen into one of the nostrils, the other being closed, and by inducing artificial breathing; Epsom salts, and water acidulated with vinegar, are better than alcoholic stimulants. The warmth of the body should be kept up, and mustard-plasters applied over the heart and around the ankles. If these produce no effect, recourse must be had to bloodletting from the foot or jugular vein, and, as a last resort, an opening into the trachea, by which pure air is forced into the lungs.

In minute quantity the behavior of the lamps do not indicate the presence of fire-damp. As the amount increases to 3 per cent of the air, the flame of the candle is surmounted by a large blue nimbus, which shades into brown when choke-damp is present; with 6 per cent, the cap becomes large and the flame elongates; with 7 per cent, the flame is invisible; with 8 per cent, ignition takes place; at 10 per cent, the propagation of the ignition is quite rapid; at about 12 per cent, the propagation is instantaneous, and the mixture is at its maximum explosive point. Beyond this the force of the explosion decreases with the addition of gas; at 20 per cent it is the same as at 8 per cent; and in a still greater amount a light is extinguished.

These facts are the guides to the fire-boss, who daily visits and tests the workings before the men are admitted to them. This method, old as the hills, requires a skilful, steady hand. Shading the candle with one hand, and raising it up from the floor, it shows the phases as mentioned above. When the flame elongates and becomes smoky, the test ceases, the candle is lowered, and the boss withdraws. A face giving these symptoms of danger requires more air. If the heading does not, it is safe, its entry is chalked, and its safety certified to.

The ordinary testing-lamps do not reveal the presence of a quantity of gas less than 2 per cent. The Hepplewite-Gray detects smaller quantities than does the unbonneted Davy or Clanny. It burns benzoline, and shows a cap $\frac{1}{2}''$ high in air of 1 per cent of CH_4. The Pieler spirit-lamp is also a good tester; air containing $\frac{1}{2}$ per cent of gas will give a cap 1″ high. A very sensitive gas-detector, much used in England, is the one described in the Transactions of the Mining Institute of Scotland, viii., p. 3. The great success of Shaw's testing apparatus leaves no room for plea. This machine recognizes the presence of a minute quantity of gas in the air, occupies only four feet of room, and weighs 100 pounds. A rubber bag, held in the return air-way, collects sufficient mixture for analysis. The only test by the author showed 2.15 per cent CO_2 in the air: this is unpardonably large. Various other fire-damp detectors have been offered, but their results are irreconcilable with the facts determined chemically. Those depending upon the difference in density of the gases are unreliable, because changes of temperature will produce similar results. Aitken's indicator is ingenious, but not much more reliable. Its thermometer is coated with platinum-black and plaster of Paris, and when exposed to fire-damp becomes heated. If the difference of temperature between it and the normal air always bears a comparable ratio to the percentage of fire-damp contained, it would work well. One objection to the special forms of gas-detectors is, that they do not serve for illumination, and a lamp must also be carried. The problem is somewhat solved by the change of burning fluid in the lamps from oil to benzine.

When the fire-damp mixture takes fire from any cause whatever and explodes, it is rare that doors withstand the shock, and if no safety-traps are sprung down (see **53**) in their stead, the air-current is deranged, and the men have no chance for their lives in the "after-damp." This is the chemical result of the ignition of the fire-damp, and is found to contain seven parts of nitrogen, N, two parts of steam, and one of CO_2, without any air—a mixture wholly incapable of supporting combustion, certainly irrespirable, as the condition of the unfortunate victims verify.

In its production two volumes of marsh-gas (CH_4) combine with 19 vols. of air and develop 23,550 heat-units, giving a temperature of 6064° F. (6471° absolute), and a pressure of 179.3 pounds per square inch. If m be the weight and c the specific heat of a gas, the heat required to raise it $t°$ is expressed by mtc. To raise the 2.75 pounds of CO_2, 14 pounds of N, and 2.25 pounds of water from 52° F. to $t°$ F. requires

$$2.75 \times 0.1711t = 0.470t;$$
$$14 \quad \times 0.1727t = 2.418t,$$

and $$2.25\,(160 + 990) + 2.25 \times 0.2675t = 0.602t + 2587;$$

whence, $$0.470t + 2.418t + 0.602t + 2587 = 23550.$$

The occlusion of these gases from fissures, crevices, and pores of the coal is sometimes without warning; sometimes accompanied by a heaving of the floor or a trembling of the roof, preceded by a slight hissing. At one mine this movement closed a drift for some distance; at others the gas burst through the floor, heaving quantities of stone with it; at Ryhope collieries, in the North of England, it rent a fissure at a pitch of 25°; and in Belgium, while sinking a shaft for coal, an immense volume of the natural gas broke out from below, that opened a large chamber, from which gas exuded for months. An element of danger, additional to the gas magazines, is that engendered by the bore-holes of the oil and pipe lines, which leak and seep gas into the coal-seams, and when they have become matters of history subsequent miners will encounter these percolations, as many now do on shallow diggings.

An attempt has been made to hypothecate a relation between the periods of gas outbursts and the movements or seasons of low barometer, but the author fails to find any simultaneity in the phenomena. A falling barometer has not invariably been followed by a heavy discharge of gas, nor does a study of the tables show its unfailing precedence to the evolution. Instead of laying stress on the acknowledged fact that December is the worst month, rather would the author call attention to the frequency of the Monday and early morning explosions as more suggestive. An excessively low barometer at the sea-level is 28.3″,—a fall of only 6 per cent of the total pressure, and of but 1 per cent, or less, of the pressure of the magazine gas. Upon the emissions from the pores of the coal and from goaves the effect is more noticeable. But even here an acre of ground of standard thickness will, with a barometric fall of 0.1 inch, exude only 18 cubic feet of mixture for every 25 yards of length of face exposed.

There seems to be no means of preventing the evolution of the gas, or the numerous causes resulting in disaster from its ignition. The wilfulness of the miner who endangers life and property to steal the luxury of a pipe, or a better light, is beyond correction after the explosion. Nor is there any unanimity of opinion concerning the utility and adaptability of the different methods suggested for rendering the gas innocuous. But one plan has universal application—a strong ventilating current, to drown out the dreaded scourge and carry it off at once. The accumulation of gas in proscribed workings can be confined therein, or it may be carried off by a pipe leading to the return, or to a safe place, where it is burned off. One novel expedient, adopted in Saxon mines, is to continually consume the gas by candles and lamps distributed throughout the workings, and thus prevent an accumulation.

43. In attempting to specify the amount of air required for proper ventilation of a mine, we are treading upon uncertain ground. Within close limits we may ascertain the amount required for the vital chemical purposes of horse, light, and man—constant sources of vitiation. A pound of carbon requires for complete combustion $1\frac{2}{3}$ pounds of oxygen, and produces $2\frac{2}{3}$ pounds of CO_2. Hence the ordinary-sized mining candle burns up 11.8 cubic feet of air, and discharges 3 cubic feet of CO_2. Eminent medical authorities state that a man consumes about 1 cubic foot of air per minute, converting the life-giving principle into 2.1 cubic feet of CO_2 per hour. The respiration of a horse is about 13 cubic feet CO_2 per hour. The deflagration of a pound of explosive produces about 2.6 cubic feet. According to Angus Smith, two hewers using a ½-pound candle and 12 ounces of powder produce 25½ cubic feet CO_2 in a shift.

The amounts of air sufficient to satisfy the conditions of combustion during the generation of the respective amounts of CO_2 are small, and if the exhalations were instantly removed, the theoretical chemical supply would suffice. But the air in the confined spaces of mine-workings is somewhat stagnant, and the atmosphere is further deteriorated by the effluvia from man and beast. Some of these are not easily detected chemically, but are more deleterious than CO_2, which is not the sole test of vitiation.

To sweep away the hot, noisome emanations, the poisonous exhalations, the unconsumed azotic gases, and, finally, the exuding pent-up gases from the coal, and to render them comparatively harmless, require a very large volume of air for their dilution and renewal. Pure dry air contains, by volume, 21 per cent of oxygen, O, and 79 per cent of nitrogen, N; and every 1000 cubic feet of it, weighing nearly 81 pounds, contains only about 18.7 pounds of the life-supporting constituent, the remainder being matter inert in its physiological effects.

Judging by the rough and unsatisfactory test afforded by the sense of smell, the air of a room ceases to be good when it contains 8 parts of CO_2 in 10,000. And to preserve the lowest standard tolerated by sanitarians, 1 in 10,000, the supply will be proportioned as follows: 59 cubic feet per hour per light; 4585 per horse; 9192 per pound of powder; and 1500 per man employed. Competent authorities vary in this matter, and the statutes of the various States differ in their requirements (55 to 300 cubic feet per man per minute). But the allowance for a mine cannot be based on the single per capita element, for it will be seen later that the friction or "drag" of air, in moving through headings and along faces which increase with the developments, diminishes the volume of air actually allowed to move. Moreover, the emission of gas from the strata, proportional to the area exposed and the character of the coal, constitutes another and constant source of pollution. Experiments in vacuo on the emission of occluded gases from the pores of coal show it to vary from 9 to 100 cubic feet per ton. From anthracite it was most; bituminous, least; and steam-coal, medium. Cognizance must be taken of this unfailing source to the extent of an hourly allowance of 0.3 cubic foot of air per square foot of working face, in a dry, dusty, fiery mine. For a non-gaseous seam 0.1 cubic foot will suffice. Some property-owners allow also 200 cubic feet air per hour for every acre of goaf. For the eruptions from the magazines no provision can be made, except vigilance and a well-regulated system.

The dilution and disposal of the gases produced is dependent upon the principle of diffusion. Gases which do not mix chemically will upon agitation mix mechanically, and after a time become intimately blended, no matter what their respective densities. A stagnant gas diffuses into a moving volume slowly,—less so as the velocity increases. If an air-current sweeps slowly, three or four miles an hour through workings, the fire-damp is diffused readily; the CO_2 less so, and hence less quickly carried away.

The creation of an air-current is governed by the physical property possessed by air and every gaseous fluid,—i.e., indefinite expansion,—by reason of which it may, if unrestricted by cold or extraneous force, continue to expand to the tenuity of interstellar space. Conversely, to overcome this natural repulsion between its particles, and to confine it within limits, requires the exercise of force. It is the attraction of the earth for the matter composing the air that confines the atmosphere here for our uses. Either heat or the removal of the extraneous force will expand the gas, until the resistance encountered is equal to or greater than the repulsion among its molecules. It is this readiness with which air or any gas tends to adjust itself to varying conditions of temperature and pressure that plays so important a part in mine-ventilation. This property is called its "tension."

44. When a volume of gas expands, some of its potential energy is converted into motion, and the tension of its final state is less than that of its initial state in the same ratio, but inversely, as that of the change in volume; that is, if the temperatures are the same throughout, the tensions are inversely as the volume. We measure a resistance by the force required to overcome it: so the tensions are in the same ratio as the pressures confining the gases.

The volumes, u, assumed by a given weight of gas are inversely as the corresponding pressures per unit of surface :

$$u : u' : : p' : p.$$

If the temperatures change while the pressures are constant, the volumes, reduced to absolute zero (— 459° F.), will be found to increase proportionately.

$$u : u' : : 459 + t : 459 + T;$$

t and T being, respectively, for up and $u'p'$, Fahrenheit readings.

The weight of a cubic foot of air at a temperature t, and a barometric pressure B, in inches of mercury, is obtained by the following formula:

$$w = \frac{1.3253B}{459 + t}. \qquad W = \frac{1.3253B}{459 + T}.$$

When the tension of air is altered by a change of temperature, or of pressure, either a rarefaction or a compression ensues; if unrestricted, the volume rises or falls, communicating the motion to the surrounding air, and causes "wind," the velocity of which depends upon the difference in tension, and equals, in feet per minute,

$$v = 481.2\sqrt{\frac{P}{W}},$$

in which P is the difference in tensions, or the hydrostatic pressure, in lbs. per square foot. This may be expressed also, in feet of head.

Let a column of air, at t° F. and D feet high, with a base of one square foot, be heated to T° F.; the new height of the column would be greater by some quantity we may call M; or if two connected columns of air, of the same length be of different temperatures, the hotter column would be lighter than the cold one, by a quantity, $w - W$; the difference of pressure, P, would drive the hot air up, producing a draught with the velocity, v, due to the hydrostatic head, M. It would require a resistance of $w - W$ lbs. per square foot to hold P in equilibrium, or a column of warm air W lbs. per cubic foot and M feet high.

$$M = \frac{w - W}{W} = \frac{P}{W} = D\frac{T - t}{459 + t}.$$

This quantity M is called the motive column and is represented by OT (Fig. 82), which will equalize the pressures of the unequally heated columns of air from O down.

Suppose the depths of two shafts be 300 feet, T of one 160°, t of the other 60°; then at 5000 feet altitude, where B is 25″, $P = w - W = 3.08$ lbs. per square foot of area of the base, $M = 57.8$ and $v = 3657$ feet per minute.

For purposes of mine-ventilation, there will be required a motive column, much larger than that here obtained by substitution, because of the enormous friction of the air in rubbing along the rough surfaces of the workings, turning sharp corners, and squeezing through small openings. The resistances due to these causes amount often to as much as 90 per cent. of the power. In other words, the actual motive column is fully ten times that of the theoretical. This principle is equally true of wind or chimney draught as mine-ventilation,

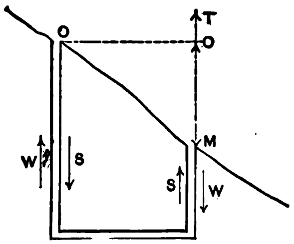


FIG. 82.

FIG. 83.

excluding, of course, frictional allowances. In chimneys, the draught is best when W is $0.5w$, and a good fire is maintained, without much stirring or smoke, when the water-gauge is not less than 0.5 of an inch.

Another mode of expressing the ventilating force is by the reading of the water-gauge (Fig. 83)—a U-tube, whose arms contain water, and are provided with measuring-scales. One arm receives the pressure of the cold incoming air, the other is connected with the upcast hot air. The difference in level measures the difference in pressures existing between the incoming and outgoing air, or, in other words, the pressure required to force the air through the mine, i.e., the friction losses, or the "drag," in the mine. It does not give any idea of the *amount* of ventilation, but, paradoxical though it may seem, only the force of the exhaustion or the power of the ventilator. The water-gauge may show high, and yet only a trifling amount of air may be circulating. The momentum of the air is acquired at the expense of the tension, for, properly speaking, air in motion cannot be said to be under pressure. Whatever pressure does exist is due to the dynamic friction. Where g is the water-gauge reading in inches, $P = 5.184g$. This is not a very reliable instrument. Eugene B. Wilson's Treatise on Mine Ventilation and J. Atkinson's Ventilation of Mines are recommended to the reader.

CHAPTER XII.

METHODS OF VENTILATION.

45. Methods of ventilation of a tunnel or advancing gangway; by conduit or brattice; single-, and double-entry, and outlet; diagonal, or adjacent, systems for double-entry; increase of temperature with depth; limit of the depth of mining; natural method of ventilation by two outlets at different levels; limitations of the method by season and depth; ventilation of railroad tunnels; account of the different experiments and that finally adopted. 46. Furnace ventilation; cost and construction of the furnace; temperature and volume of the air produced; dangers and limitations in its employment; dumb channels in fiery mines; exhaust-steam as a ventilator. 47. Mechanical ventilators; description of hand-fans and their adaptability; blowers; Root fans; Champion blowers; use of compressed air as ventilator; exhaust-fans; details in the construction, arrangement, efficiency, and cost of the same; Guibal fans; lines of improvement; method of housing; outlets and connection. 48. Description of the Waddle, Schiele, Lemielle, Cooke, and Fabry fans; comparison of them. 49. Effect of a low barometer and high temperature on the volume of the exhaust; fan *vs.* furnace.

45. To secure ventilation in the confined workings of a mine a conduit must be furnished, by which the warmer and lighter air may ascend, to be supplanted by cold or compressed air entering by a different compartment. The fresh air should always be carried to the deepest point in the mine, and then the circuit arranged to constantly ascend. This ascensional method is especially advisable in steep seams carrying fire-damp. Mines having a single entry only are ventilated by a wooden air-tight box-pipe, or, if the liability to corrosion is small, a galvanized pipe. A compartment partitioned off from the main tunnel or shaft will serve the purpose better, though no proper relation between the intake and upcast can be kept up because of the constant interference by the other uses to which the entry is put; nor can a bracing current be carried to the working face. An explosion may destroy the brattice box or pipe, and also the current, at the very critical moment. Double entry is not only always precautionary, but often imperative; and as the depth and extent of workings increase, the insufficiency of single entry becomes more and more manifest.

This is why mining ordinances exact two distinct outlets, separated by a safe distance of unbroken rock. The upcast, advisably, should terminate in a large chimney, high enough that its draught be not influenced by changes of wind or the surrounding buildings. The location of the two entries, in reference to each other, varies within wide limits. One plan consists in having them near together, thus concentrating the plant; but both air-ways are carried with the development, and the current passes through double the length of the mine. On the other hand, as the next lower lift progresses it may be readily connected with the air-ways of the upper lift, and thus receive ventilation from the start. The other plan is the "diagonal system," the shafts being at the extremities of the workings. This is well enough for long-wall method, but the ventilation must suffer until the connection has been made.

Double entries may be easily obtained in coal furnishing sufficient rock from the roof or from partings, by driving a wide gallery and walling it up centrally from the waste. If there is not enough rock for this, the two entries are carried with the usual pillar between them, having connecting "throughs" at intervals, but always closing one as fast as the next is ready. To ventilate that part of each entry between the last connection of the entries and the face, it is subdivided by a duck brattice (see **53**), fastened at the near side of the "through" and leading up to the work. Pipes will do, carried to the faces, through the door closing the intake entry. They need not interfere with the haulage. The practice of relying upon diffusion to do the work of ventilation is injudicious. These remarks also hold true regarding the "throughs" connecting the rooms in pillar and stall working. Diffusion is usually relied upon there. Though not perfectly safe, it proves better than the very costly plan of having them nearer than 100 feet.

Before proceeding to the consideration of methods, there is an important item of interest—the downward increase of temperature. At a depth of about 50 feet below the surface the temperature of the ground will be found to be invariable, and equal to the mean annual temperature of the locality; at greater depths the ground becomes hotter. In the deep bore-hole at Spernberg the temperature rose 1° F. for every 50 feet depth; at Swinderly, for each 70 feet; Prof. J. Phillips reports 59.3 feet; and in the Bogg's Run hole, Mr. W. Hallack of the Geodetic Survey found the increase to be 1° for every 78 feet. At the Adelbert shaft, Prussia, observations five times a month, in different levels, for a year could deduce no regular law of increase; at the 30th level, 3200 feet, the temperature was 98° F.; at Gilly, Belgium, 3489 feet deep, 126° F.; at Rosebridge colliery, Wigan, 2418 feet, 113° F.; at the Comstock 2000-foot level, 108° F. Certain it is that there is a marked increase of heat with depth, though observations fail of uniform results. Copper-bearing lodes are the hottest; those in slate being cooler than those in granite. Unless some means be discovered for rendering mines like the Comstock habitable, the limit of mining depth is soon reached; in fact it is questionable if the 2800-foot level is not near the workable limit; there it takes five or six men to do a day's work, for the miner must retire to a cooling station after a five-minutes' spurt.

This increase of temperature is availed of for a natural system of ventilation, by driving to the mine two openings from points at different elevations on the surface. In winter the subterranean air is more rarefied, and in summer less so, than the external atmosphere, the greater weight of which therefore establishes a current,—down the lower, or shorter, opening in winter, and up it in summer, as the arrows (Fig. 82) marked *S* indicate; in winter the direction of the current follows the arrows *W*. Similarly, while driving a shaft or tunnel, or for ventilating a railroad tunnel, a box-pipe carried some distance above the mouth of the opening serves the same purpose. The above is true of workings at moderate depth but if the mine is very hot or deep, say over 800 feet, the subterranean air is always hotter and lighter, and unless the two outlets have excessive difference in level, the current will continue uninterruptedly, without fear of reversal, down the lower opening.

While this method may be universally applied under favorable conditions, the danger in collieries is the reversal of the current. At one time—say during winter—the current distributes fresh air along the working faces first; but in the summer-time the air moves in the opposite direction, and first passes through the goaves, whence it may carry noxious gases to the workings, and thus spread calamity: while during the other two seasons of the year the difference of temperature above and below ground is so slight that the air moves sluggishly for lack of motive column, and neither ore nor coal mines will derive sufficient ventilation without resorting to supplementary means.

Somewhat similar conclusions have been arrived at regarding this complicated and unsafe natural method for railroad and similar tunnels, whose capacity is limited by the activity of ventilation. The conviction prevails now that natural ventilation is inadequate; that a direct current produced by the passage of the trains through the tunnel is better. This is compatible with the investigations of the previous lecture; *D* is small: therefore the motive column and its corresponding pressure are small. From Lecture **51** it will be found that the frictional resistance is too large for *M* to overcome, unless the difference of temperature continues very large. This subject is still more complex, because, usually, long tunnels are driven from several shafts, each of which would react with the other to confuse the main circuit, by counter-currents and eddies, that vary with every change in the external atmosphere. Unless they can be forced to act together, the shafts should be cut off from connection with the tunnel, and an exhaust-fan, attached to one connecting-shaft or to the bratticed or boxed compartment of the tunnel, may be effectively employed.

46. It has long been a favorite plan to warm the air which has traversed the workings, by passing all or part of it over a fire before its expulsion from the mine. Close to the bottom of the shaft is constructed a fire-place, walled and roofed by a fire-brick or common-brick arch (Figs. 84 and 85). When special care is taken, a second wall is built outside and over, with an air-space between, to isolate it from the coal and prevent fire. If the roof is wet, a double arch must surmount the furnace, as otherwise the steam generated will burst the arch. If the mine is fiery, or considerable dust is floating, care must be taken that the gases are well diffused, or else the current must not be brought into close proximity with the fire. In such cases the current is split, a small portion being heated over the fire, the remainder passing through a "dumb-channel," entering the upcast 50 feet or so above. A still safer plan passes all of the fiery current through the channel, and feeds the furnace by a split current of fresh air direct from the intake.

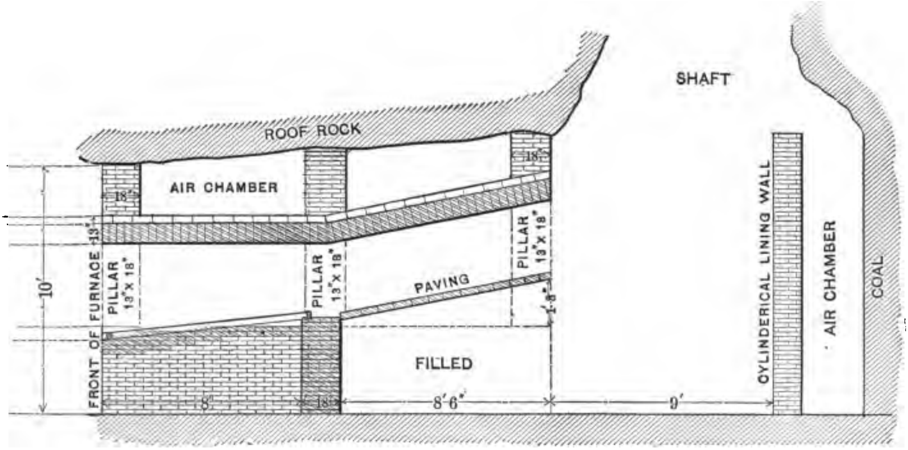


FIG. 84.

The size of the grate depends upon the work to be done.

Its bars are 3 feet from the floor, slanting upward toward the shaft 1 to 6, distance to the roof 4 or 5 feet. The width, wall to wall, is 6 feet and its length from 4 to 12 feet, according to the volume of air to be moved, which is about 5000 cu. ft. per foot breadth of fire on a properly constructed furnace.

The area of the grate-surface in sq. ft. (F) may be known by the relation $FAvp = 1{,}716{,}000 \sqrt{D}$.

An ordinary furnace of 34 sq. ft. heating-surface, costing $130, will heat a column of air such as will furnish 29,000 cu. ft. per minute. A large number of furnaces 10×12 furnish 200,000 cu. ft. The cross-sectional area must be 50 per cent. greater than the upcast air-way, and the shape capable of regulation by double sliding iron doors, to produce varying degrees of contraction and of combustion. The fire is spread over its entire width, and over only as much of its length as is necessary to furnish an adequate motive column, at a temperature of 140° to 160° F. Emergencies, as low barometer and high thermometer, and the cleaning of the grates, require other and more heating-surface. The coal consumed is 2 to 5 tons per day, spread thin and evenly over the bars, and fed from both ends, on a long furnace. This rate is 40 to 70 lbs. per hourly h. p. of work done on the air. Attendance, etc., is $5 per day.

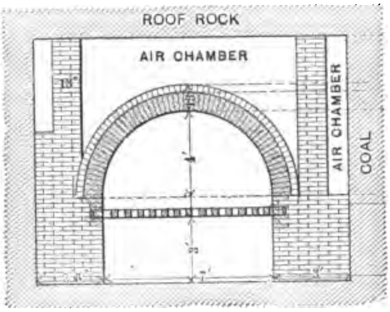


FIG. 85.

An arrangement so simple and cheap in construction and management presents advantages which have long commended it. But the difficulties in its use are manifest. 1st. The serious reduction in its efficacy by atmospheric changes of the seasons. With a low barometer, or in hot weather, the natural forces oppose the furnace less as the workings deepen. 2d. The state of dryness and the area of the upcast play an important part in the action of the furnace. In a wet or large shaft there is what may be called an invisible brattice setting up strong counter-currents, which check its efficacy. This also happens if the furnace is of much smaller area than the shaft, or if it is placed too far back. So there is a proper ratio between the areas of furnace and shaft, for maximum economy. The power of the furnace upcast increases arithmetically with the temperature, and that with combustion. Only a limited quantity of coal can be burned by the furnace. The resistances of the mine (see **50**) increase geometrically with the square of the velocity of the current. It is therefore manifest that the latter soon must reach the furnace-limit. This has been confirmed experimentally. 3d. Numerous calamities, involving hundreds of lives, are traced to the furnace, which has fired either the solid coal surrounding it, the timbers of the shaft, or the surface plant. 4th. The ignition, in gaseous mines, of the inflammable constituents not sufficiently diluted.

One would suppose that these dangers would prohibit its use; but such is not the case—probably because nothing surpasses it as regards the mere production of current. In the shallow workings of Ohio, during the past five years, over 250 furnaces were built, and only 70 fans. In Illinois the majority of mines, averaging 5 colliers, use furnaces; of large mines, 50 hewers each, fans. In Pennsylvania the ratio is about 15 fans to one furnace.

The steam-jet, once a rival of the furnace, is now obsolete, except for small metalliferous mines. A jet of exhaust-steam from the engine or pump is turned upward into a chimney, serving the purpose of an upcast. Close investigation has satisfied the Commission that it can be used advantageously only under peculiar circumstances. It is very likely that a fiery mine may be well ventilated by the jet, but not more so than by fan. At an actual steam-pressure of 47 lbs. it is capable of moving 217 times its own bulk of air.

47. Mechanical ventilators include a variety of devices which have served their day, fans remaining as our main reliance at present. These are of two varieties—blowers and exhaust. The blower is connected with and forces air through the intake compartment into the mine. The latter draws the mine air through the upcast entry, to which it is attached.

In small workings, like breasts of stopes and drifts, where a small volume of air is required, force-fans are much in vogue, turned by hand or by the force of the drainage-water on a small wheel. The construction of the blower is simple, consisting of a radial wheel provided with blades, in a casing having a central opening on either side by which the air is admitted to be acted on by the fans. This casing has an increasing space outside of the fans, into which the air is driven before discharging. The two varieties of blowers, pressure and fan, differ only slightly in construction; each has a positive blast-delivery. The Root, Baker, and Champion machines are favorably commented upon, and do excellent service in large ore-mines, but are inadequate for extensive collieries. One of 7 feet diameter, with 3-foot blades, supplied 16,000 cu. ft. These blowers may be had at from 1 to 15 h. p., and furnish 250 to 12,000 feet per minute. For coal-mines this plenum system is not acceptable, because the main haulage must be by the return- or upcast-way, whose atmosphere is thus too much befouled. Still, the ready alteration of the Champion from blower to blast, or the reverse, recommend it, and particularly for wet shafts which freeze in winter.

The exhaust-fans are of the windmill or the centrifugal patterns, the difference being in the mode of taking the air; the former cutting off a definite volume of air, while the latter throws off at the circumference the air it received at the centre from the return air-way. They are not easily deranged, have high efficiency, may be made of large size, and are supplanting all other appliances.

Inside of a fireproof housing revolves a horizontal shaft connected with the engine or dynamo and carrying a hexagonal or octagonal frame on which are built six or eight blades (Fig. 86). They are for the greater part straight, but at their outer ends curve. In some machines the vanes are radial, but many engineers prefer them inclined to the radius about 40° backwards. Their width is from 0.5 to 0.6 their length. For over a quarter of the circumference, fitting closely to the tips of the blades, the housing is circular; and for the remainder of the periphery, at either extremity, the curves change. The long curve, an arithmetical spiral, flattens to the orifice and continues beyond; the other continues, with little clearance, to the opening, from which a vertical partition rises 15 to 25 feet. This shroud, with its funnel outlet, is walled on either side, having but one opening, opposite the centre of the fan, through which the mine air enters. The spiral amount of the housing should be such as to prevent the loss of velocity and consequent jumping of the discharge from one segment to another, and may embrace 130° from the throat.

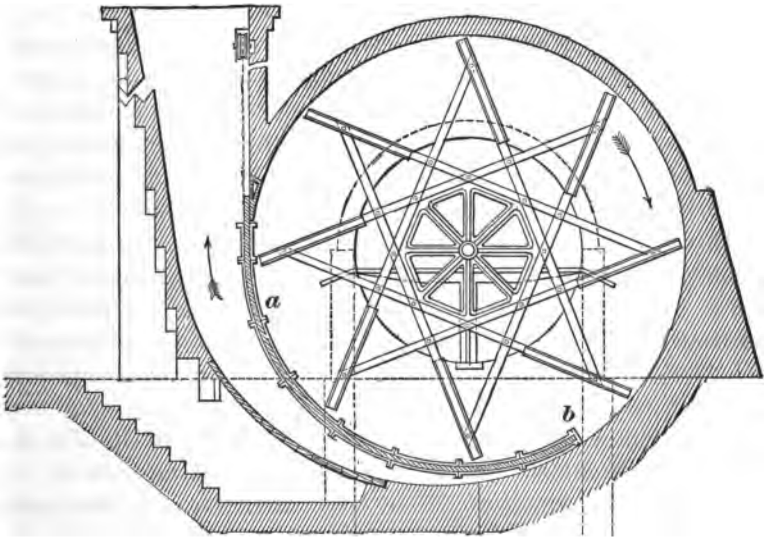


FIG. 86.

The curve or the inclination of the blades has less influence upon the discharge than does the shape of the ajutage, or the ratio of outlet area to the volume of the fan. A very marked advance in fan construction was made when the outlet was extended, by a divergent pipe to an area of four times that of the orifice. If well calculated, and built accordingly, the efflux from it is greater than from cylindrical mouthpieces, and the loss is less than when the air is thrown directly into the atmosphere from the thin orifice, without any pipe. The actual discharge is about 80 per cent of that of the volume generated by the vanes. Thus a 40×12 Guibal at 43 rev. theoretically should give 648,000 cu. ft. of air. In the Severn subaqueous tunnel its efficiency was only 70 per cent, for it gave 450,000 cu. ft. The maximum velocity commonly adopted for the tips of the blades is 3000 feet (less than 50 revolutions for a 20-foot fan). The theory of the fan rests upon that of the mechanical work of centrifugal force, which is 0.0000340 $(v^2 - w^2)$. In this, v is the circumferential velocity and w is the absolute velocity at expulsion, due to compression from centrifugal force. As w increases, so the work on the departing air, and proportionately the effective work, decreases. The use of the funnel-chamber reduces this quantity to $\frac{1}{4}$ or $\frac{1}{8}$, and the work lost to 4 or 5 per cent.

The following are the conclusions reached by R. Van A. Norris, Wilkesbarre, Pa., after elaborate experiments on 25 fans, as the influence of: "1st. The diameter on their performance seems *nil;* the only advantage of large fans being in greater width and a lower speed required of the engines. 2d. Width upon efficiency is, as a rule, small. 3d. Shape of blades shows that the back curvature is better, and diminishes the vibration. 4th. Shape of casing is considerable. The proper shape would be one of such form that the air between each pair of blades would constantly and freely discharge into the space between the fan and casing, the whole being swept to the *évasée* chimney. A large spiral, beginning at or near the point of cut-off, gives in every case a large efficiency. 5th. Shutter on the fan is an advantage. The exit area can be regulated to suit the varying quantity of air, and prevent re-entries. 6th. Speed at which the fan is run. The efficiency is high if the peripheral velocity is large."

48. In the event of an emergency arising, not requiring the full capacity of the fan, it is a great advantage to have a sliding shutter (*ab*, Fig. 86) for temporarily diminishing the opening. Fans always run at full blast do not require it, so, now, few are thus supplied. Its use permits high speed and efficiency, and is anti-vibratory. Its correct position is always known when at a given speed throbbing ceases. The most efficient position is ascertained by measuring the volumes, and corresponding manometric depressions, for different openings.

This mechanism, as described, is the Guibal fan, the simplest and cheapest ventilator, on which, fortunately for us all, no patent exists in this country. It is deservedly popular, and may receive its power directly from engine or dynamo, or by belting or wire rope. The common sizes are 16 to 20 feet diameter. Collieries build them as large as 30×13. The Calumet runs a 30-foot fan at 50 revolutions. A 20×6 fan can be run by a 16×30 engine, and deliver 160,000 feet per minute. Their cost is small. Sixteen and eighteen foot fans can be placed on cars, complete, with engine, etc., for $2200. A double-inlet spiral casing 15×7 Guibal fan will deliver as much air as the ordinary 50×12.

Though the Guibal takes precedence over the other mechanical forms of ventilators, there are several excellent machines. The Waddle is an open-running, and the Schiele a closed-periphery, quick-running, centrifugal. They cost less than the Guibal, and do equally good work. The latter is rarely over 16 feet and the former often 50 feet diameter. The Waddle has both its shell and blades of iron plate, in one piece. The air from entry to delivery moves at a constant velocity. In the Guibal the movement of the air is retarded until it reaches the shutter, through which its motion is accelerated. The Capell is said to have a good record, and capable of a 9″ water-gauge and a high useful effect.

Of displacement machines, those which intermittently discharge a certain quantity of air, Struve has built the best, upon the air-pump principle. A pair of reciprocating air-chambers, provided with valves, alternately rise and fall in water, at 6 strokes of 7 to 10 feet per minute.

Cooke's is a positive machine. An eccentric-drum revolves inside of a 12-foot circular case. A swinging shutter is held very close to this, and cuts off the entering from the discharge current. The inlet and outlet portion occupies 235° of a revolution. At Lofthouse iron-mines are seen two of these side by side, the drums being placed opposite each other on the shaft, so that the revolving mass is balanced, the discharge equalized, and the efficiency raised.

Lemielle has a species of rotary air-pump, complicated and leaky, producing large volumes under great rarefaction. It consists of a vertical cylinder, within which a second revolves eccentrically; on this latter are two or more vanes, which, in one part of the revolution lie close to the shutter, and in another open and expel the air.

The Fabry is much used in the North of France and Belgium. Two fans, each having three broad blades, arranged radially, are hung in a chamber. They revolve with equal velocities in opposite directions, the blades come in contact, isolate a quantity of air, and expel it into the atmosphere. To the fact that there are no joints in it is attributed its success. It resembles the Root blower.

49. Fan ventilation is affected by the atmospheric changes in a manner similar to furnace currents. Low barometer and high temperature require an increased degree of rarefaction for equal results. And it is desirable that the two outlets be on the same surface level. In designing a fan, it is well to recall that, notwithstanding the elasticity of the air, its frictional resistance is high; that the gauge-pressure, for some reason, is one ninth greater than that due to circumferential velocity; and that the pressure varies with the square of the velocity, and the h. p. required, as its cube: hence it is preferable to decrease the rate of the fan and increase its volume.

As the depth of the mine increases, the work devolving upon the fan proportionately increases, because normally the air becomes denser. Every additional 1000 feet of depth requires increased rarefaction of 0.4″ water-gauge. On the other hand, the efficiency of the furnace increases with the depth of the upcast, since a slower fire is more necessary in a deep than in a shallow shaft to obtain an equal depression g and column M. When a certain depth is reached, it becomes an open question of the relative merits and demerits. For shallow workings, the exhaust-fan undoubtedly takes precedence; at 1000 feet a furnace will equal a very imperfect fan, consuming 20 lbs. per hourly h. p.; but a good fan and a condensing engine will be cheaper than a furnace, down to about 4300 feet. This takes no cognizance of the objections to the furnace. On the other hand, machine ventilators are liable to serious objections and an unventilated condition of the mine during repairs, while a considerable circulation will continue for some time after the furnace fires are extinguished. Auxiliary apparatus will provide against this emergency.

The French Fire-damp Commission considers the vacuum method to be less dangerous, though Haize insists that only blowing pipe should be put into workings to the rise, because of the ever-dangerous gas at the roof of the face.

CHAPTER XIII.

DISTRIBUTION OF THE AIR.

50. Calculation of the work done in ventilating a mine; losses by friction; coefficient of friction; formulæ; examples; similarity between the formulæ for frictional resistances of water, air, and electricity; examples and illustrations. 51. Interpretation of water-gauge readings; formulæ; examples; Buddle's system of splitting air-currents; advantages and economy of the plan; principles of dividing air-currents into panels; formulæ; laws governing the area of airways; dangers of goaves, and the necessity for their isolation. 52. Velocity of the air and the modes of measuring it, by candle, smoke, or anemometer; place for observation; calculation of the ventilating power.

50. It has been assumed thus far that the work done upon the air is totally effective in the mine; that with a given *M* and *P* the calculated quantity of air is obtained without any loss; that the momentum, once imparted to the air, would carry it through the mine and out. This is not so. Friction instantly overcomes the momentum; the velocities given by the formulæ (Lecture **44**) are never realized in practice. The rough sides of the galleries and rooms, their sharp corners, and the diminished areas offer resistances to the passage of the current that consume often 90 per cent of the power. Moreover, the subtle air under pressure seeks to escape at every opportunity, and some portion of the precious fluid is lost into the goaf, through doors and at crossings. A certain mine theoretically required a pressure of but 1.2 lbs. per foot to give rise to its current, yet the friction was such that 11.8 lbs. were actually necessary to create the velocity. Not infrequently the ratio between *M* (to which the generation of the final velocity at the top is due), and *M'*, the head actually necessary to overcome resistances, is as low as 1 : 18. In other words, only 5.5 per cent of the work done upon the air is usefully expended. Any means of reducing this loss is to be welcomed.

Let us examine into the laws governing the movement of fluids and their applications to the conditions, that we may learn to reduce this friction to a minimum, and obtain salubrity, safety, and economy with the least outlay. The air which enters the mine from the downcast is distributed to the rooms and chambers in proportions varying with their several needs; or the current as one mass sweeps through the main way, along working faces, thence by return air-way over the furnace or to the fan. The resistances encountered depend upon the ratio of the area of the surface rubbed to the area of the conduit, and upon the coefficient of air-friction against rock. A satisfactory value for the coefficient has not been obtained: the records of experiments show it to vary as in water, according to the conduit and the velocity. For a velocity of 1000 feet per minute, air experiences a loss of head of 0.269 foot, corresponding to a pressure of 0.0219 lb. per sq. ft. With a smaller velocity this is sensibly increased; but accepting this value and reducing it to a velocity of 1 foot per second, we have $f = 0.000,000,021,9$ lb., or $f' = 0.000,000,269$ ft.

Let l be the length, m the perimeter, and a the area of the gangway, through which the air is coursing at v feet per minute, and the rubbing friction is found experimentally to be $flmv^2$. Imagine a piston, fitting air-tight in the passage; to just move it against the resistance requires the expenditure of a force pa, in units of lbs. and sq. ft. Therefore the loss of power due to friction is $pa = flmv^2$, and the loss of head in feet, $p = \frac{flmv^2}{a}$, or in lbs., $p' = \frac{flmQ^2}{a^3}$.

This cannot be ignored, for, other things being equal, the quantity of air received at any face is inversely as the resistances encountered on the way. In the "splitting" of the air it is of special import. It will be observed that the frictional loss is directly as the perimeter and inversely as the area. This would suggest the desirability of selecting such a shape for the air-way as will make it as spacious as circumstances will permit, consistent with a diminution of the exposed surface. The circular form most nearly meets this requirement; but, as a rule, we are restricted to the rectangular or the more advantageous trapezoidal cross-sections. Two galleries, 5 × 5, require one third more power to carry the same amount of air, as a 5 × 10 gallery. Another important matter—in galleries of equal section the volumes of air passing will be inversely as the square roots of their lengths. A 1600-foot gallery, carrying 6000 cu. ft., offers the same resistance and consumes the same amount of power as one of equal area 711 feet long, delivering 9000 cu. ft.

The friction increases with the square of the velocity. So it would be far better, desiring a given quantity per minute ($Q = va$), to increase the area rather than the velocity. Conversely, a local contraction of the air-passage, by the use of a partly opened door, a pile of waste or of gangue, will materially diminish the air passing through it.

A comparison of the above formula for air with those for electricity and water will show an identity of loss, though in different units: for electricity it is $P = C^2R = f\frac{L}{a}C^2$; and for water, Kutter's formula for pipes is $a = clmv^2$.

51. It has already been remarked (**44**) that the water-gauge measures the drag of the air in the mine, and thus serves to indicate the pressure and head corresponding to the motive column M. The pressure varies from ¾″ for easy to 4″ for difficult ventilation (from 3.9 to 20.7 lbs. per sq. ft.). In anthracite mines it is about 2″. The motive column, which is to just maintain this pressure against resistances, should also be sufficient to create a final or exit velocity in the shaft. If the entire current traverses the mine unbroken, the resistance in the shaft or entry is only a fractional part of the mine friction indicated by the water-gauge, and the following formulæ apply with sufficient accuracy:

$$V^2 = \frac{pa}{flm}, \qquad Q^2 = \frac{a^3p}{flm};$$

f to be taken always in the same terms as p. If the air-ways are all of the same dimensions, the calculation of the lost pressure is in one operation. The p thus obtained, added to that P requisite for the generation of a final velocity, determines the total difference in barometric pressures, or the head, to be given the moving column by the motor. Often P is a very small fractional part of p, and may even be neglected without sensible error. Again, at each sharp turn, at each change in the size of the conduit, at each contraction of the air-passage, which becomes to all intents and purposes an orifice, a loss of head is experienced, though it is most marked at regulator doors or at "break-throughs." Ascertaining thus the value of p for each division, and remembering that at the end of one the velocity and its head are the same as at the beginning of the contiguous one, we have but to add the separate fractional losses to the head, or pressure, generating the final velocity at the mouth of the mine.

Formerly it was the practice to meander the air through all the galleries of its lift before expelling it (Fig. 10). This involved heavy pressures, enormous air-ways, or a velocity dangerously fast. The last gang, fed by the departing current, would receive an irrespirable atmosphere, vitiated by the emanations from all previous sources. There was nothing to commend this pernicious system, and it is certainly a matter of congratulation that it is becoming obsolete.

Many years ago Mr. J. Buddle introduced a system of ventilation for fiery mines that has everything in its favor: the aggregate quantity of air is increased, the dangers of explosion are lessened by confining its train of evils to one portion of the mine, and power for ventilation and haulage is saved, since it goes hand in hand with the method of panel-working (Fig. 12). Each panel of the mine is completely isolated from the contiguous districts by barrier pillars, and is ventilated separately by delivery to it of a portion of the volume of the intake which does service in that panel, afterwards discharging it into the return air-way, where it rejoins the exhaust from the other districts. The electrical distribution for illumination and the water-supply of a town are on identically the same principle.

The principle is easily stated: a fluid seeks the easiest escape from confinement, and air is no exception to the rule, with a tendency inversely proportional to the resistances offered by the course taken. If it has the selection of several escapes, it will split up into portions, whose volumes are inversely as the resistances, assuming the initial velocity to be constant. The ventilating requirements of different portions of the mine vary with the development and many local conditions. These should be well studied while planning, so that as much independence as possible is secured. The quantity and the resistances are known; from these factors, *a*, the area of each passage connecting the panel with the intake is calculated in accordance with the formula

$$a^3 = \frac{flmQ^2}{p},$$

which when so constructed or regulated permits of the disposition of the air as intended.

The measurements of p and v are made in the intake, the splits, as near as possible to the downcast and their reunion, to the upcast; otherwise the resistances of the intermediate ways and of the entries determine the maximum number of splits possible, especially as the air-ways increase in length and decrease in area. Then it is more and more urgent, for accuracy, to produce a fan rarefaction P, greater than the panel friction p by M, necessary for the generation of the final velocity. For economical purposes it is highly desirable though generally impracticable to so manage that the pressure and temperature of the exhaust be reduced to that of the external atmosphere at the point of exit from the mine. This is the theory of chimney construction.

Simple as the theory is, satisfactory as the well-developed plan proves, it is not so easy to execute it. The success of the plan involves a nice manipulation and great skill in taking the precautions to balance up the various factors, determine the equilibrium designed, and prevent one panel from receiving too brisk a current at the expense of others. All leakage points must be taken care of. In shallow workings it costs less to sink a shaft than to furnish an elaborate system of ventilation; so splitting is not much resorted to in many of our coal districts.

It hardly requires an example to show that subdividing the current gives greater economy than increasing the area or pressure. Nor can it be doubted that attention should be paid to the air-ways, intake, and return. The latter should be at least as large as (preferably twice) the area of the intake, because of the condition of the air, highly charged with all the products of combustion evolved in the workings. Some collieries are provided with two upcasts to insure sufficient aspiration. There is, of course, a certain best area for the ventilating-shafts, but the casual conditions vary so much, that an inflexible rule cannot be formulated.

J. J. Atkinson, government inspector of mines, England, shows that the power required for 16,200 cu. ft. in one column will produce 70,884 feet in 5 splits, 94,850 in 10 similar parts, and 99,722 feet in 15 splits.

Spacious air-drifts, in the long-run, are a saving in money, and during exploration their driving should be conducted with care, for the escape of gas is strongest from freshly cut coal. The use of powder should be restricted as much as possible. German authorities prohibit blasting altogether when the presence of any fire-damp is noticed.

Goaves are the most dangerous places. It is estimated that the air-space in a goaf is one sixth of the volume of coal extracted, and in it most likely will breed a great deal of gas, of which the sweating of the roof is an infallible sign. Spontaneous combustion once begun therein, nothing will stop it. For this reason, though, aëration is possible. Fiery coals should only be worked by a method involving complete removal of the coal, or its replacement by clean waste.

52. To maintain a distribution of the air through the workings requires a thorough organization and rigid supervision. While the salubrity of the air is preserved by an immediate removal of the mephitic gases, a velocity exceeding 500 feet per minute is not desirable; it is neither comfortable nor safe.

A speed much over that is equally injurious with stagnation. Several mining commissions of Europe have experimentally determined that, aside from the chilling effect of walking against so rapid and cool a breeze, lamps are not safe: a rapid current incites explosion by driving the gases through, or the flame against, the screens. Many lamps can resist higher-speed currents, but none are safe in over 900 feet per minute. In the old country the air-currents at different parts of the mines vary in velocity—at the coal face often as fast as 900 feet; but here our Davy, Stephenson, or Clanny lamps require protection in such a velocity, which exceeds that of American practice.

A miner can approximately estimate the speed of the current by knowing the rate at which he must walk to keep the flame erect. One may also note the time required for some volatile fluid or smoke to travel a certain distance. But the anemometer is the simplest instrument. In a case is a series of vanes which are moved by the current, and these by proper gearing turn indices over their respective dials at such rate that the velocity may be at once read. It does not give accurate results on account of the friction of its mechanism. Each instrument has its own factor, which is not even constant. Biram's and Costello's patterns (Figs. 87 and 88), are most used


FIG. 87.


FIG. 88.

in America. Their factors are ascertained by occasional test. The anemometer is revolved on a whirling table, and its reading compared with the actual velocity of revolution.

The point selected for observing the velocity should be in a straight gallery, whose sides and roof are a fair average in roughness, and where there is neither a sudden bulge nor a contraction. The average of several one-minute readings are taken at the place of measurement, near the roof, sides, and centrally. Then the cross-section of the conduit at the observing station is taken. The product is the volume circulating.

The power U to move the air is vaP, which, divided by 33,000 ft.-lbs., gives the actual h. p.; and the ratio between these observations, simultaneous with the indicator-diagrams of the engine, gives the efficiency. The ultimate comparison is with the coal consumed, which approximates 11 lbs. per useful h. p. by fan, and 40 to 70 by furnace. In designing a motor, a good margin should be granted for emergencies.

CHAPTER XIV.

REGULATION OF THE AIR-CURRENT.

53. Doors, regulators, etc.; safety doors, and extras, to be dropped after explosion; air-crossings, overcasts, brattices, and their use; mineralized brattice. 54. Complete example for the ventilation of a mine, with two outlets and five splits; furnace, fan, and natural ventilation methods compared; example and calculation for a railroad tunnel.

53. THE third division of this subject is the manner of regulating the distribution of the air, of which doors are the main dependence, placed in such positions as to temporarily or permanently check or deflect the current. The most important of all, upon which the entire security of the mine depends, are those in the drifts connecting the two outlets. Wherever their location, they must be built with great care, of matched lumber, closely fitting in a frame, and even provided with weather-strips, and maintained only so long as the drift is to be kept open. After that they are either removed, or replaced by masonry dams.

In main ways they open with the haulage or against the current, and generally are paired far enough apart that one may be closed before the other is opened. Each ought to have an attendant, or its frame be so inclined that the door will close of itself. Automatic mechanical appliances are not much in favor, except some arrangement of pulleys and rope to open and shut the door without stopping the mules. In wide galleries the door swings on a vertical axis in the centre opening two track-ways; or two doors are used, swinging in opposite directions. Doors ought to be dispensed with entirely: it is better to provide overcasts. They are leaky, and, if there is no trapper, offer opportunities for negligent drivers to divert the current from its proper course, or stop it altogether. The worst feature is their liability to destruction by explosion, and a consequent annihilation of the current, which is swamped by after-damp. In French collieries a very good device consists of sheet-iron doors, hinged and suspended from the roof at proper places; when an explosion occurs, these are released by the shock, drop, and close the opening, replacing those destroyed by the accident. This meets the emergency, even if all the original doors were thrown down. Tampering with or propping open of doors or overcasts should be punishable with discharge.

An overcast or crossing is a kind of bridge, carrying the main return current over the intake of the district, having a direction more or less at right angles. There are three classes of stoppings, the main ones, masonry dams; temporary ones, of boards; and the bearing-up stops, of brattice. One air-way is stopped at both sides of the intersection by side walls of 12″ to 24″ thick, 3 or 4 feet high, and an arch thrown over to permit a small drift to continue the return-way from the opposite sides of the stoppings. For seams liable to suffer damage, and to become leaky by pressure from strata, a wood top is more resilient. The second class of crossings is of a common beam, 12 × 6, supporting a box cover of 3″ tongue and grooved planks. Occasionally, crossings are required in workings near the face, to last only a short time, while the pillars are working. These are on a small scale as compared with the others—1″ plank brattice. Natural air-crossings are more rare and costly, but safer. The returns are made in the adjacent strata. Air-courses are a necessary accompaniment of "splitting," and crossings evade the cost of maintenance of doors and obviate the ever-present dread of a derangement of the current through carelessness with the doors (see Fig. 12).

Regulators serve a different purpose—as a gate-valve to a pipe. They are sliding-doors in the air-ways of each district, adjustable to various-sized openings, and are employed to prevent excessive supply to one panel at the expense of the other districts. The securing of these is left to some responsible person, who padlocks them into place. The regulation is in accordance with the calculations of previous lectures.

It is not enough that a large volume of fresh air is delivered into the mine-entries, it must be brought up to the working face.

Brattices are used for small currents of air, or for temporarily carrying it to the breast—farther than it would otherwise range. Planks are nailed on props at suitable distances apart, the cracks lathed, and the whole tarred and calked with oakum, to constitute a partition, which divides the shaft or drift into in take and return compartments. As it cannot be made even approximately tight, much must not be expected from it. A new brattice was found to leak 27 per cent, whereas an old one showed a loss of 61 per cent, with a depression of 0.6″ and a velocity of 670 feet. Another plan is to unroll a quantity of canvas, and suspend it from the drift roof. This can be easily adjusted along the working faces, its cheapness counterbalancing the inconveniences and risks of its use. To make it impermeable to air, it is coated with tar, though the stench of it has resulted in its substitution by an incombustible material, as soluble silicate, or asbestos, which resists fire.

54. EXAMPLE.—A downcast 6 × 11, an upcast 6 × 12, 300 feet deep, supply air to a mine having a gangway 3000 feet long, 50 feet sectional area (5 × 10); five splits receive each a portion of the 40,000 cubic feet moving. Required the several amounts delivered to the panels through the resistance of 6 × 8, 700 feet long; 5 × 7, 1000 feet; 5 × 8, 1200 feet; 6 × 6, 750 feet; and 4 × 7, 800 feet long.

By substitution it will be found that the pressure required to overcome the downcast resistance is 1.23 pounds per square foot. In the mine the pressure will be the same throughout; hence for each split

$$q^2=\frac{a^3p}{flm},$$

which, solved for each division, gives volumes $q = 16,124\sqrt{p}$; $9072\sqrt{p}$; $9723\sqrt{p}$; $16,490\sqrt{p}$; and $7600\sqrt{p}$, respectively. The sum of these equals the total quantity Q, from which we get p, equal to 0.82 pound. Assuming that the splits are made as near as possible to the downcast entry and reunion to upcast, which by the way is advised, there need be no further allowance for resistances other than those of sudden turns or contractions. The distribution of the air q for each district is 10,930, 6150, 6590, 11,180, and 5150 cubic feet.

The upcast offers a resistance to the exhaust-air, the volume of which is greater than the 40,000 cubic feet, because of accretions from "blowers," moisture, etc. Disregarding these increments, the volume to be exhausted is carried up at a velocity of 555 feet per minute, whence p is 1.00 pound.

The total pressure to be imparted by increasing the downcast barometric or rarefying the upcast is therefore 3.05 pounds, requiring M of 57 feet, or a difference in temperature of about 100° at $B=25''$. If the splits are made at stated distances along the main gangway, an allowance must be made for each of the several losses of friction in the various lengths thereof, remembering that each branch split reduces the volume passing through the remaining portion of the gallery, and correspondingly the friction therein. It would require 3.7 h. p. to do the work upon the air; a fan of 43 per cent efficiency would necessitate a 9 h. p. engine.

The calculation for the ventilation of a railroad tunnel is similar. Assume a tunnel 4961 feet long; sectional area, 336 square feet. Then 4961 × 336 = 1,666,900 cubic feet of air to be changed every ten minutes. Velocity of current, 496 feet.

$$p = \frac{flmv^2}{a} = 4.8 \text{ pounds per square foot.}$$

A fan 20 × 6 at forty revolutions easily meets this demand. The fan may be applied, with or without brattice, at either end of the tunnel; but this is a delicate matter. It would be better to place the fan at the mouth of, or a furnace at the bottom of, one of the connecting shafts used during construction, and block the others off.

CHAPTER XV.

ILLUMINATION.

55. Use and consumption of candles, etc.; Davy's discovery and invention; description of the safety-lamp; remarks regarding later forms; Stephenson, Mueseler, Hepplewite-Gray, and Marsaut. 56. Requirements of a safe lamp; modes of rendering them secure; candle-power of the different types; electric illumination.

55. IN an atmosphere containing gas sufficiently diluted to render it harmless, naked lights may be used. Candles—sixes—are the commonest means of illuminating. Of these, the stearic acid, Proctor and Gamble's, is the most uniform, and will best withstand the temperature of a heated atmosphere. The consumption averages three per man per shift.

In many districts they have been replaced by oil-lamps of various kinds, the simplest being the tin lamp, with a hinge lid on top and a hook and spout at either side. The wicking projects from the spout, and gives a moderate light of four candle-power, but with considerable smoke. It burns white lard winter-strained oil, one half-gallon per month. The best burning fluid is petroleum, but its unsafety, requires the admixture of a less volatile oil. Rape-seed is much used, and lard-oil quite satisfactory; but a half-and-half seal and petroleum best meets the case. In a mine using 260 duplex-wick lamps the annual expense for oil, repairs, interest, etc., is $504. In metal mines candles are cheaper than lamps.

In the long-wall system, with ample current, the use of naked lights should be restricted to the narrow workings; in the pillar and stall system, to the rooms ventilated by splits—not while robbing the pillars. Many engineers are inclined to the opinion that naked lights should only be excluded from development work, and while testing for gas. Though this practice would compel a good ventilation thrown up along the working face, it would be tempting Providence.

As the operators attempted more and more to work in places infested with fire-damp, various illuminating substitutes were tried. Great ingenuity has been expended in their endeavors to invent a safer means of lighting than that offered by the naked flame. In 1815 Davy discovered that a sheet of iron wire-gauze was so good a conductor of heat that a flame in contact with it could not readily pass through. Further experiments indicated that for mining purposes a mesh of 784 holes to the square inch was the safest, and was therefore adopted as the standard. A cylinder of this mesh, surrounding the light, surmounting an oil-lamp and capped by a perforated top, is the form, which has been little changed since Davy's time (Fig. 89). After the lamp is filled with oil and lighted, it is locked, to bar the miner against access to the flame, the wick of which is trimmed by a wire passing up through a close-fitting tube from the bottom. The combustion is supported by air penetrating the gauze at all sides.

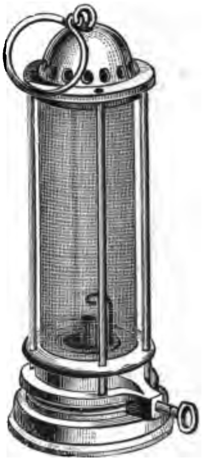


FIG. 89.

The lamp has done and continues to do great service; but it has two defects. The first is the liability of the gauze to become red-hot, and pass the flame through to the inflammable mixture outside. The second objection is its low illuminating power. The open spaces occupy only one fourth of the area of the gauze, through which the light escapes horizontally; still less light gets out at the top, to illumine the roof. Miners require light thrown in every direction, especially upward; and in a certain investigation, while giving evidence, confessed that they would rather unmask the flame and risk explosion, than not to watch and see distinctly the roof, the ever-present imminence of which can scarcely be denied. These defects have been partially remedied in the subsequent patterns by the use of glass, the only impermeable, strong, though brittle, transparent substance.

The Clanny is the first alteration of the Davy, a lower portion of the wire-cloth of which, if replaced by a short cylinder of glass, gives somewhat better illumination (Fig. 90). The simple expedient of enclosing it or the Davy in a tin can or shield is also quite an improvement.

FIG. 90.

Stephenson's, almost as popular in this country as those above, has a long cylinder of glass surrounded by a wire-gauze, and bonneted above by perforated copper. The feed is also through the gauze, going underneath and into the cylinder to the flame, thence out at the top, as usual. This plan keeps both cylinder and gauze cool, and its relative security rests essentially on the regularity of the draught, for if the inside air becomes overheated the light goes out; so it must be suspended properly. This is an English favorite.

The Marsaut is an improvement upon this form, and stands a fair amount of tilting safely. With care, its glass cylinder will last three years before breaking. Of 370 in use, the average consumption of rape-seed oil was a gallon in six months.

The Mueseler, a Belgium lamp, is like Dr. Clanny's, having in addition a conical chimney centrally above the flame. It is highly recommended in Europe, but must be carefully handled. It does not burn well in "dampy" or slow currents. The bonneted Mueseler, an English improvement, is receiving the highest encomium for use in fiery mines and high velocity.

The Hepplewite-Gray lamp admits air at the top, down four tubes, and through an annular chamber above the oil vessel. The only gauze employed is that covering the outlet and annular inner chamber. A serious defect is that, if suddenly lowered, the light goes out.

Notwithstanding the various modifications, there is yet no really safe lamp—one that *cannot* ignite in an explosive mixture outside of it. Generally, the elongated appearance of the flame gives warning of danger to the man carrying it into a fiery atmosphere; but it would be the better part of valor to smother the light or to withdraw from the spot before the heating of the gauze begins.

56. But the danger lies in other directions. First, the velocity should be such that the air-current cannot be blown into the lamp, or the flame against the gauze. Although very few accidents are really traceable to these causes, and many alleged cases are doubtful, and the various investigators and mining commissions are not in accord, it is certain that there is a limiting safe coarseness of mesh and speed of current. Besides, the point of union between the glass and gauze is leaky. The Hepplewite-Gray and the bonneted Mueseler have the best resistance to explosive currents of high velocity, and the South Side Committee report the following relative speeds at which the respective lamps and the air-current can safely pass: Davy, 360 feet per minute; Clanny, 600 feet; Stephenson, 780; Mueseler, naked, 1200; Mueseler, bonneted, 2400; Marsaut, in a can, 2440; and the Davy, in a shield, 2400. The North of England Institute of M. E. gives the safe velocities at 720, 540, and the others higher. The British Royal Commissioners of Accidents approved the Gray, Marsaut, and the bonneted varieties as safe at high speeds. The common Davy or Geordie lamps are unreliable.

Next, the reprehensible behavior in opening a lamp to light a pipe or to illumine the roof is to be devised against. All manner of permutation locks and magnetized plates are offered on the market, and serve fairly well. The magnetic locking device of the Wolf lamp has resisted all efforts of the miner to open it. In some lamps, the lower parts are riveted by lead plugs. Another favorite method is to arrange it so that the light is extinguished on opening the lamp.

The illumination from any of these lamps is very feeble—best horizontally, and less in any other direction. Of all the lamps the Gray sends the best light upward. The candle-power, horizontally, of the Roberts is highest—about 18, and of the Clanny the lowest—nearly 6. On this account a lamp must be able to be held tilted without extinguishment, and be unaffected by violent oscillations. The conditions dictated by safety circumscribe the lines of attempted improvement in the degree of illumination. The brass lamp is found to be 70 per cent as bright as an iron lamp, preferred by the Germans, of the same pattern. Photometrically speaking, seal-oil is better than rape-seed, and a broad, flat wick than a round one. The insufficiency of the light of a safety-lamp, combined with the difficult and trying conditions of the bonneted forms, is proving injurious to the eyesight of miners, which serious evil is growing. Photophobia is rare where candles are used, or where the lamp is hung behind the miner.

At Zwickau, Saxony, a novel and bold plan is in use, owing to the difficulties with all safety-lamps ; an innumerable quantity of naked lights are burned constantly, which ignite the gas as fast as it reaches the candles. No explosions have been recorded.

Whatever the means of illumination, the lamp must be self-contained, be strong, portable and not heavy, require little attention from the miner during twelve hours of sustained light, and be capable of placing in any position, besides giving perfect insulation from the fiery gas. In an incandescent lamp, wire-bound, and with flexible connection, electricity fulfils many of these requirements, besides requiring no oxygen, and it seems reasonable to expect it to supersede the present form of lamp. Its success in metal mines makes the proposition for collieries not so absurd as would at first sight appear. Large chambers would thus be safely and so thoroughly lighted as to render every part of the roof visible, affording greater security to the hewer. A greater number of lights would be required than of oil, as the former cannot be continually carried about beyond the limit of the flexible connection. Again, along the entire galleries numerous lights would have to be placed, except in the haulage-ways, where lamps in the hats may be permitted. Though the electric system is not sufficiently perfected, many mines employing this force for other purposes find it better and not much dearer than oil. The cost of a plant for 100 lamps, exclusive of the generating machinery, is $500; and for coal, renewals, interest, etc., the annual expenses are $518. One h. p. will run ten 16 c. p. lamps at 75 to 150 feet apart. The life of a lamp (60 cents) is fully 100 shifts. A serious detriment is the fracture of, or the injury to, the wires. A portable, self-contained secondary battery lamp may obviate this, but it is both heavy and wasteful of power.

Lamps are not safe unless kept in thorough repair, and infractions of rules regarding their use severely punished. The gauze should be steeped in a hot alkaline solution, to free it of soot, etc. Lamps burning benzine are not clogged with carbonaceous deposit as are those burning oil. There is therefore no occasion for poking at the flame.

Animal and vegetable oils, adulterated by the manufacturers with mineral oil, are objectionable compounds. The incomplete combustion, dense smoke, and almost unendurable odor are detrimental to good air. There is little saving in their employment, as they are photometrically worse and burn away quicker than the pure oil.

Benzine burns with a clear, strong, uniform flame, shows an easily-perceptible "cap" in the presence of a small amount of gas, and is free from danger in a well-constructed lamp even in the hands of an unskilled miner. The Pieler, Hepplewite, Gray and Wolf lamps are of this type—all photometrically good.

To avoid waste, manufacturers furnish automatic fillers, holding just enough for a lamp. Lamps should be occasionally tested for leakage and other sources of danger.

CHAPTER XVI.

HYGIENIC CONDITIONS.

57. Laws upon ingress and egress; accidents in mines; ladders, their arrangement and cost; loss of time and energy; use of cages for men; conclusions of the Cornwall Society. 58. Movable ladders or man-engines, single or double; utilization of the pump-rods for the purpose; comparison of the safety of the man-engines with other means; cost of the machinery and plant. 59. Accident laws for the protection of life and limb; are equally effective for the security of the mine; statistics; accident-rate decreasing; tables; lessons drawn from their inspection; causes and prevention of accidents; fall of roof; lack of timbers; explosions; premature blasts; necessity for a rigorous enforcement of the rules and laws. 60. General remarks concerning fires in mines, their causes, prevention, and treatment; entering old mines; aërophones.

57. FOR purposes of ingress and egress, mines are provided with ladders or man-engines, where the cage or bucket is not used. The statutes of many States present varied ideas, theories, and requirements for the accommodation of the men. Some require the maintenance of substantial ladders in a separate compartment, as the sole means to be used by the men for entry and exit. In other States operators are relieved of the necessity of keeping up a ladder-way, if safety carriages are employed. The laws of many States forbid the use of buckets by the miners, while the general tendency in all regions is to insist upon two well-equipped escapement ways.

If the angle of entry is below 30°, no special provision is necessary. The mud-sills of the timbering break the descent into sufficiently convenient steps. Steeper than this, and up to about 60°, some variety of treads is necessary. When the pitch exceeds this, the compartment must be provided with ladders, isolated from the hoistway. They should be inclined, uniform in direction, at an angle of not less than 10° from the vertical, to diminish the fatigue of climbing, and enable the men to carry tools with them. At equal distances down the ladder-way (20 to 40 feet down a vertical shaft, and at greater distances on an incline), platforms are built of 2 × 6 beams and 2-inch planks, closing it, except for a man-hole, at the foot-wall end. The ladders extend up through the man-hole, and are fastened by staples or toe-nailed to the shaft-timbers, and rest on the far side of the plats. They are made of 2 × 6 standards, 18 inches apart, with iron or wooden rounds or rectangular slats 12 inches apart. The last-named are cheaper, last longer, and give better toe hold than wooden rounds, which, in turn, are easier to use than the more durable iron. Wooden ladders cost from 6 to 10 cents per running foot; iron, 20 cents.

Though used in Europe for 1200 to 1500 feet depth, and in this country in deep mines, they are certainly not advisable. According to the Cornwall Society, the use of ladders deranges the respiration, and shortens life by ten years. The miners reach the workings more or less exhausted, and the operators have lost the benefit of a proportionate amount of energy. Unquestionably, an element of success worthy of attention by mine managers—a pecuniary as much as a humanitarian question—is the proper treatment of and the conveniences for the men, who unconsciously reciprocate in an equivalent of work. Besides, time is lost. It takes 15 minutes to go down 300 feet, and the ascent is twice as slow. A shift of forty men, following one another at intervals of 8 feet, entails a loss to the company of 31 minutes each shift. With buckets and cages the loss is not so great; eight men at a time, lowered 1200 feet, consume 40 minutes for every shift of 100 men. An additional loss occurs at tally-time from the reduction of the hoisting capacity, which, with the impatience of the men, leads to the crowding of the cage; but in most States the limiting number of men permitted on the cage is named. A serious form of accident, peculiar to deep mines like the Comstock, is the fainting and falling, which occurs when the heated miner, while being hoisted, comes into contact with the air near the surface. There is no safeguard against it, and owing to its frequency men never go up alone.

58. Movable ladders or man-engines, invented by D'Orrell, of Clausthal, were instantly adopted as acceptable substitutes to the methods previously used, and now are very popular in deep mines. Mr. Lorn, who introduced the engine in Cornwall, was handsomely rewarded by the Royal Polytechnic Society, which declared it a "great boon to miners." Its introduction involved the addition of some machinery, but it was easy to operate.

Two rods, of decreasing cross-section from top down, receive at the surface an oscillatory motion from balanced bobs, operated by an engine having a fly-wheel and other regulators. The dimensions of each rod at any point must be such that it will have the requisite tensile strength to support the weight of the part below it, loaded with men. They play between roller-guides 50 feet apart, and are provided with wings and catches, after the manner of the Cornish pump-rods, which may, in fact, be utilized as "Fahrkunst" rods without much extra power.


FIG. 91.

Each rod has a small platform, Fig. 91, about 12″ × 12″ or 18″ at every 12 feet —double the length of the stroke. A handle four feet above the platform gives support to the miner, who is carried up 6 feet on one rod, which brings him opposite a platform on the companion rod; upon this he steps, to be lifted 6 feet more, to meet a plat on the first rod, which has been coming down to receive him. A miner stepping from one to the other is carried up or down at a rate of from 48 to 96 feet per minute (each rod makes 4 to 8 double strokes, delivering one man each time, those at the Calumet and Hecla make five strokes). As there is no limit to the depth at which these may be carried, and as they are capable of working alike in slopes as in shafts, it is not surprising that they "take" so well. They replace hoisters, and require little additional power or space. Tools and supplies cannot be carried by the miner, but may be delivered by the cage or bucket.

A single rod is also used, its companion being replaced by stationary platforms attached, 6 feet apart, to the shaft timbers. Upon these the ascending men wait during the down stroke of the rod. The single-acting man-engine requires chains and counterpoises at intervals to balance it, and to prevent the shock incurred at the end of the stroke.

From the fact that a misstep would be fatal, it would seem as though man-engines were extra-hazardous, yet the accident record does not confirm this fear. Some confusion is caused by a man missing his plat and riding on, to the annoyance of those following him ; but this is of rare occurrence unless his light goes out, for there is a halt of several seconds at each change of motion. Out of an average of 100,000 men employed for ten years (in Prussia), only 57 were injured on the man-engines; in Cornwall, 17. This is more than compensated for by the increased length of life of the miners using them.

The cost of machinery, etc., for a 1200-foot man-engine is $18,000, upon which interest and depreciation may be figured at $2500,—amounting to 10 cents per man daily, on a gang of 100 men. The running expenses at the Dolcoath mine are 4 cents per man, 2400 feet.

59. We now arrive at the consideration of a theme which, sad as it is, should suggest the lines of improvement. Deplore as we may the immolation attendant upon mining, there seems no way, by legislation, threats, or punishment, of impressing the necessity of vigilance upon the miners, who by long inurement to peril that is imminent have become oblivious of the unavoidable sources of danger.

The statutes make stringent requirements of the operators and of the employees, enforce frequent thorough inspection by competent men, impose fines and penalties for negligence or non-compliance, and our appliances are useful, durable, and modern; yet the benefits that should accrue are not realized—the death-rate continues deplorably high. The percentage of accidents in steep vein-mines is less than that of iron mines, and only half that in coal-mines, where 3 out of every 1000 employees are injured annually. Bituminous collieries are more dangerous than anthracite or lignite mines. The rate has been decreasing somewhat, as might be expected, though the increased depth of working tends to make mining more hazardous; and, assuming equal conscientious announcements by the authorities reporting the casualties, it will be found that the safety of life in our mines bears satisfactory comparison with that in European mines. Generally, the accident statistics are compared with the output tonnage, and it may be said that for every 200,000 tons of coal mined one life is sacrificed and two men injured. That this proportion is diminishing is patent to any one inspecting the reports of inspectors. Though it is difficult to get a trustworthy comparison of the number and class of accidents, the following table is given, showing in percentage the fatalities and casualties. The miscellaneous accidents vary from 2 to 27 per cent of the total number.

	Falls of Roof and Coal.		In Haulage Ways.		Fire-damp.		Powder.		Tons per Accident.	Tons per Life Lost.	Employees per Death.
Pennsylvania, 1889:	*f**	*s**	*f*	*s*	*f*	*s*	*f*	*s*			
Anthracite	45	32	25	13	6	16	7	11	34,817	105,764	342
Bituminous	65	60	22	25	5	2	3	1	102,414	397,612	585
Illinois, 5 years......	63	60	11	11	4	3	5	14	66,200	215,549	716
Ohio, 1874............	61	58	14	18	11	9	4	3	4,844	108,919	412
" 1889............	76	61	15	21	..	2	6	5	128,322	330,529	619
Iowa, 1889............	64	63	17	15	8	6	5	7	52,400	98,620	321
Missouri, 1889........	50	..	..	6	..	10	..	13	111,173	222,347	514
Nova Scotia, 5 years..	62	64	18	16	9	7	3	4	167,083	238,697	618
Comstock.............	..	12	..	21	..	..	..	8	52,140	......	...
Missouri, zinc........	..	45	..	19	..	..	.	5	8,376	......	...
Colorado, ore	..	60	..	21	..	..	..	9	19,460	......	...
Illinois, 1889.........	70	67	13	20	7	2	8	5	42,988	238,450	539
Italy, 1889...........	54	61	24	19	8	11	9	2	$42,400	$152,993	752

* Columns headed *f* are fatal accidents; those headed *s*, serious.

PERCENTAGE OF FATALITIES.

	Falls.	Ladders.	Fire-damp.	Powder	In Shafts.	Tons per Life Lost.	Employees per Death.
England, 1851........	34.3	1.7	30.0	....	21.2	63,562	222
" 1888........	53.0	2.3	5.4	4.0	8.5	194,430	600
Prussia, 1852.........	50.0	....	13.5	....	20.4	83,051	600
" 1889.........	40.0	2.0	12.8	5.0	10.0	109,528	365
France, 1853.........	40.0	....	20.1	3.6	21.4	37,346	260
" 1889.........	30.0	1.5	28.6	1.0	14.3	117,105	538
Belgium, 1861........	36.7	2.1	10.3	3.1	13.4	51,840	422
" 1888........	35.3	1.5	16.0	2.8	8.2	106,110	571

But the tales which these figures tell must be noted. First, that notwithstanding the frequent holocausts, with the reports of which we are shocked, the loss of life by explosions and fire is not by any means as great as by the more numerous unpublished accidents to individuals resulting from the caving of roof by reason of insufficient timbering. Fully one third of the deaths are from this cause,—and the percentage was the same in the '50's as now—and neither the operators nor the bosses are responsible always for them, as subsequent investigation reveals. The crushing of men by the fall of coal upon them is an equally common accident. Many casualties are caused by the indifferent miner, anxious to make a big turn-in, neglecting to support the roof of coal with the timbers right at hand; in fact, I have seen instances where a crush had caught victims who were compelled to crawl over a supply of props in order to reach their work. It is an incontestable fact that the miner will take too many risks, and an accident ensues solely from his own carelessness. It would be unjust to attribute all accidents to wilful neglect, for mining is precarious; but surely many calamities might be avoided if the miner would exercise precaution. It is not sufficient that he is the victim of his own wilfulness,—for the evasion of the law carries its own penalty,—but he endangers the lives of his co-laborers, and the property of the employers who have invested heavily in measures for his protection. The sudden dislodgment of the roof or sides of a breast or stope, or the unnoticed yielding of the pillars, is due to so many causes, that it is impossible to prescribe rules for its prevention. Horses, sigillaria, balls of ironstone, rock creviced naturally or by excessive blasting, are threatening conditions that demand a liberal supply of precautionary timbers or filling placed before the movement begins; otherwise, once begun, no amount of subsequent support will save it: the ensuing damage is out of all comparison with the insignificant item of props judiciously used. Moreover, without a better system of illumination of the underground workings the miner cannot discern the condition of the overhanging rock, and props, to be opportune, must be placed at once. The substitutes of iron, steel, and masonry for wood must conduce to a greater safety, as also the increased facilities for the more expeditious removal of mineral. The miner should be provided with a "timber-bar" and chain where props are to be drawn—as, for instance, to avoid a squeeze.

Second, the causes of explosion, and the nature and extent of some of the most prevalent conditions preceding it, have long been understood. Nothing but eternal vigilance and anticipatory action could decrease the magnitude of the fatalities from this source, and the disaster following it can only be guarded against by an active air-current, and the exercise of precaution by stout primary and secondary doors, properly hung. A liberal interpretation of, and a willing compliance with, the section requiring two outlets will bring its own reward. But co-operation must be had from the men, as neither laws nor improved appliances can counteract the effects of their recklessness.

Accidents in and about the traffic-ways are being reduced by the use of safety appliances, previously referred to; by gates and doors at the mouth of shaft and level; by a small drift cut in the hanging-wall, for miners to pass around instead of across the shaft; by whitewashed safety niches at every 100 feet in a gangway; by care in signalling; and by having a space 2 feet wide between the "loaded" cars and the side of the heading.

The effects of sudden changes of temperature experienced by those coming from a hot portion of the workings to the surface may be remedied by prudence on the part of the miner, and by railings around the cages. The new Pennsylvania law requires hand-rails on cages. Dr. G. C. Swallow, Mine Inspector of Montana, suggests an excellent idea to prevent the mutilation of men riding on the cage. A coiled wire screen, which may be drawn down at the sides of the cage, is fastened below, and prevents the contact of men with the timbers. Except in fainting, men caught between cage and timbers have only themselves to blame for accidents on cages.

Accidents caused by premature blasts are more frequently the result of carelessness, though many unaccountable explosions have occurred. Electric firing of cartridges, the prohibition of loose powder, and the avoidance of firing in collieries while much fine dust is afloat materially diminish the casualties.

After all, neither legislation nor appliances will avail if the men do not have an ever-present realization of impending danger, and a corresponding caution. Doubtless many of the charges of carelessness are unjust, for only at the critical moment may have come an instant of absentmindedness, when the fatal act was committed. The only hope is for a change in human nature, for until men willingly obey the laws, and on occasion deny themselves of some slight fancy, accidents cannot entirely be prevented. All precautionary measures should be announced, rigorously enforced, and the offender, in no matter how slight a particular or what the plea, discharged. It is not the visitors who are the victims: it is the old hands, in the pockets of whom pipes and matches are presumptive evidence. These, with fuse, tobacco, etc., should be contraband goods, and subject the miner introducing them to fine. (See Abel's Mine Accidents.)

60. The causes of fire are quite numerous, and cannot be always avoided. If the surface plant is not placed so precariously as to imperil the shaft, the causes, primary and secondary, are careless blasting, insecure safety-lamps, inadequate ventilation, and floating dust. The first three are most frequently responsible for much of the danger. Explosives whose temperature of detonation is less than 1000° F. are incapable of igniting fire-damp, unless the holes are badly stemmed. Unfortunately very few available compounds realize this condition, unless it may be ammonite. So the only security lies in an almost instantaneous mixture of the deflagrated gases with an ample supply of air, which after all is the only preventive of fires. One of the consequences of the replacement of the hewers by machine is that blasting, as the other operations of mining, falls into the hands of a specialist. This diminishes the accident rate from "shooting." A fuse burning without flame is essential; a means of lighting it, without the fear of sparks which are first thrown off, coming into contact with the air, is obtained in the Heath & Frost lamp. The powder flame cannot be entirely suppressed, even by tamping with water. So the substitution of electric firing, at tally times, for the practice of single shots, is about the only other means of lessening the risks. It is in the driving of the main levels, winzes, and upraises (preparatory works) that the dangers of the fire-damp are the greatest, because the escape of gases is strongest from freshly cut coal; capillary fire-damp is not difficult to manage, but that under pressure at great depths is serious. Often, near clay veins, the danger of igniting blowers by shots may be avoided by drill-holes, kept in advance of the drift.

It is now conclusively established that soot is a provocative of fire. The effect of the presence of coal-dust has been a subject of trial and many experiments; the most recent—those of Wm. Hall at the British Home Office agreeing with those of the Prussian Fire-damp Commission—show that, without a fierce flame from a blown-out shot, coal-dust in absence of fire-damp might not explode: that it could not alone originate an explosion, though, if initiated by fire-damp, soot may aggravate its effect. This will depend upon the degree of its fineness, the readiness of its diffusion, and its chemical composition.

Coal dust presents conditions but little less perilous than gaseous mines, but precautionary measures for it are simpler than for gas. It intensifies and extends an explosion originated by gas. Without any floating dust, the flame from a blown-out shot does not travel more than 25 feet. Soot may convey the flame even 200 feet. A means of laying the dust, developed in some diggings, is by a spray continually delivered to the air-current from 1″ or 2″ pipes under 50 pounds pressure, and a stand-pipe 3 feet high at every 50 or 70 feet apart. The spray is delivered through lead plugs, slit as desired. No other jet or fibrous material gives a fine spray.

Spontaneous combustion can be obviated only by an active air-current or by change in the method of mining, to one involving complete removal of the coal and a substitution of clean waste.

Some mine fires are started in the stables, pump-room, or at oiling stations. The prohibition of naked lights, a care in handling the oil and waste, and a liberal use and renewal of clean sand and gravel absorbent is recommended.

Fire-damp requires for its ignition a temperature of 1400° F.,—higher than its composition would indicate. At a temperature higher than this, and in a mixture with air, sparks will fire it. Without odor, invisible and only occasionally audible, there is great difficulty in defending a mine from explosions and fires. Blue jet flashes give warning of the emanation of gas. Adequate ventilation, strong overcasts, a minimum of doors, safe lamps, a moderate use of powder, and the deposition of the coal-dust are the only preventatives of the violent explosions that wreck underground structures, decapitate the unfortunate miners, coke the coal-exposures, and fire the mine.

Sometimes when blowers of marsh-gas ignite a mine wet clothes will beat out the fire. But when it has attained such headway as not to be overcome by ordinary means, it may be effectually confined by cutting off the air supply and building masonry dams, completely stopped up, if the superincumbent strata are not porous, or the mine not so shallow that air is admitted or the gas escapes. This failing, the burning portion is hermetically sealed, and then drowned with water, or better, CO_2. For extinguishing the fire at the Calumet and Hecla (supposed to have been communicated to the shaft timbers by the friction due to the binding of the rollers on which the hoist-rope rested) all sorts of plans were resorted to: among others, the surface was kept frozen to stop leaks; finally, the shafts were sealed and CO_2 injected. For the manufacture of 350 cu. ft. of CO_2 there were used 1200 gallons of sulphuric acid and 4500 lbs. of limestone. Water is the simplest quencher, but it has happened that the water could not reach certain portions of the mines above the foot of the shaft, because of the compression of the air which could not escape. Until it was consumed, the fire continued to rage above the water-level, perhaps for a long time. On pumping out the water the conflagration might break out again. A pipe leading from the face of the burning portion, up the shaft, would release the air and permit quenching.

At the Anaconda mine, Montana, steam was injected into the burning stope, but it failed to quench the fire.

For penetrating a very impure atmosphere aerophones of different makes are to be had. They consist of a portable bag or cylinder carrying enough compressed air or oxygen for the respiration of a miner and his lamp while making repairs or exploring. The oxygen is inhaled by one tube, while through an exhaler is ejected the CO_2, which is absorbed by caustic soda, leaving the N only to return to the bag. Fleuss' apparatus looks like a knapsack, weighs 28 pounds, contains a 4-hours' supply of oxygen, and has besides a self-contained illuminator—a lamp burning methylated spirit, heats a plug of lime and renders it incandescent.

In entering an old mine to be prepared for development, the mine is divided into sections and each split in turn, cleared out, all the others meanwhile being cut off and worked in connection with the dumb drift.

PART II.

PRACTICAL MINING.

CHAPTER I.

SHAFTS.

61. Shafts: their location, dimensions, and shape; round *vs.* square; sump and subsidiary shafts; equipment, number, and size of compartments; single and double entry shafts or slopes; shafts for railroad tunnels; mode of sinking, progress, and cost. 62. Timbering shafts; various modes of cribbing by wood, masonry, and iron; shaft pillars; slope timbering; Hollenback shaft; walling of circular shafts.

61. SHAFTS may be sunk for permanent or temporary objects, and they may be intended for one especial purpose only —of hoisting, travelling, or ventilation; or their size may be sufficiently large to warrant division into a number of compartments, one each for the pumping and ladder way, the remainder for hoists, according to the output. Collieries require additional communication with the surface for ventilation. The large area required for, and the foulness of, the return air demand a separate outlet for upcast, as also for the intake, which should never be interfered with by hoisting.

The numerous drawbacks to single-entry compartment shaft or slope are so fully recited in I, **5**, that only in vein-mines should the development be thus risked. Certainly the ventilating ways should not be in adjoining compartments, because the bratticing could never be kept tight enough to prevent a leakage of fresh air into the upcast. Only the expense of sinking in hard, or the difficulties in soft or watery, ground preclude double entry. Where a prospecting drill-hole has tested the ground the shaft should not be carried down along on it, but near it, for the drill-hole will eventually be of greater service than it is capable of during sinking.

When it is desired to remove the mineral quickly, several shafts are sunk, their positions being a matter of indifference. Ordinarily, however, the location of a shaft and its equipment is a matter of vital import. The configuration or nature of the surface affecting transportation may govern the selection of a site; but, *cæteris paribus*, the principal shaft should be so located as to reach the lowest point of the workings. This is not at the outset always possible to do, so we are accustomed to see one shaft after another abandoned or relegated to secondary uses. Instance the numerous illustrations from the Lake Superior region. The Calumet and Hecla has eight shafts, each over 3000 feet deep, and four of 1000 feet with a complete plant over each one. Nor is it the exceptional case. Theoretically, there is no limit to the depth of workings. At the beginning of the present century there were few of over 1000 feet deep; now thousands exceed 2000 feet.

Shafts sunk to facilitate the execution of long tunnels for railroad and other purposes are best located with their axes in the same plane. Alignment is better, and only because of the difficulty of supporting the shafts at the tunnel level is it the common practice of placing them to the side. Shafts are, however, losing their importance for this work, since the introduction of the rapid, ventilating, drilling-machines.

As regards form, the rectangular is the most common (Fig. 92). Its timbering is easily accomplished, and the best adapted to loose ground. Where brick or stone is used instead of wood for lining, the sides are arched to give great strength, and this perhaps led to the round or elliptical shapes, which are such favorites in Europe on account of their greater resistance, and particularly because of the loose soils and watery strata encountered. That their entire area cannot be utilized is, however, an objection (Fig. 93). The timbering of the polygonal (12 to 16 sides), used in Belgium and the North of France, is

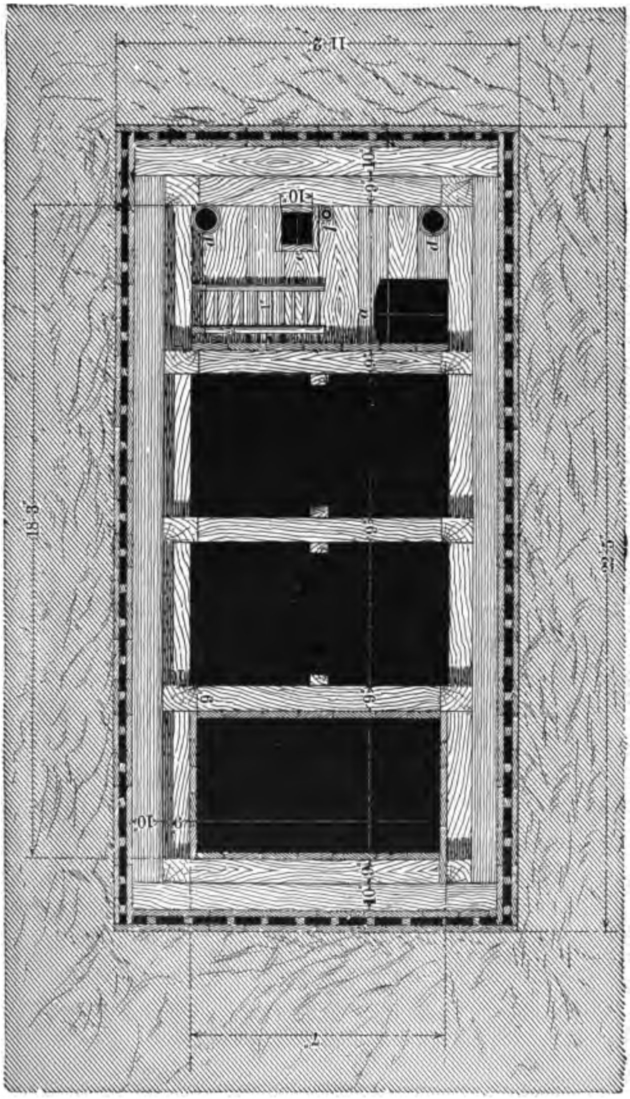


FIG. 92.

not so easy to fit as is that in the hexagonal or octagonal shafts.

The dimensions of the shafts, governed by the number of compartments, should be carefully studied to meet all requirements of strength, output, and escapement for a prolonged period. The scale increases as the depth and output is large. Outputs of 100 tons were regarded as large not so long ago; but now many hundreds of shafts have a capacity of 1000 tons daily. They are larger on the Continent than in Britain, and colliery shafts demand a greater area than do those in metal mines, which have less traffic, besides being restricted generally by the distance between the walls. The size of the compartment is determined by that of the bucket, skip, or cage, its length being the width of the shaft, the length of which is governed by the number of divisions (see I, **23** and **27**). Compartments placed side by side make a stronger shape than if arranged in a more compact form (Figs. 94 and 95). The compartments for metalliferous cars are about 4 × 5, those for coal-cars, from 6 to 8 feet wide, by from 10 to 12 feet long. 10 × 38, 12 × 24, 11 × 20, measured inside of the timbers, are common coal-shafts sizes 13 × 52, 12 × 47, 11 × 54, are

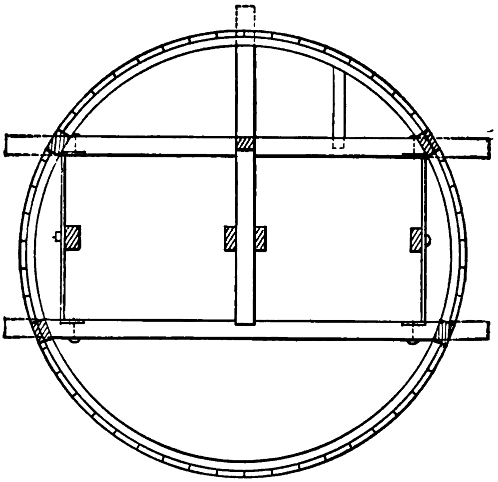


FIG. 93.

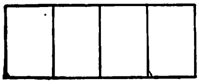


FIG. 94.

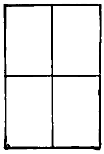


FIG. 95.

some of the very large shafts. In the Lake Superior iron region 9 × 16, 8 × 12, and 9 × 20 are usual. In Montana and Nevada smaller sizes prevail, as 10 × 10, 9 × 20, and 10 × 16. In Colorado 5 × 9 to 6 × 15 suffice for the small outputs of high-grade mineral. The largest yet begun is a nine-compartment shaft 38 × 42 in the clear. Circular shafts for buckets holding about 1500 lbs. are 8 feet diameter; for 1½ tons, 10 to 12 feet and for cages 13 feet. The sizes of the ventilating shafts are not a matter of indifference. They depend upon the volume of air to be moved, and therefore upon the extent of the development; they should be large enough that the current velocity does not exceed 1000 feet per minute. The upcast is usually round, and the downcast a walled rectangular. Neither should be housed, though the former for a furnace ventilator may be provided with a chimney high enough to prevent the distraction of the current by surrounding buildings; or by traps closing tightly and quickly if a fan is used. An area of one square foot for every eight men employed is a good basis for the upcast of a moderate-sized mine.

The features governing the selection of site have already been examined on p. 20; so there remains to consider the process of sinking. In a soft-ore lode the shaft section should reach from wall to wall, and massive shaft pillars be maintained, else it is sure to succumb. In hard-rock lodes the shaft should preferably be on the foot-wall; on the hanging-wall heavy supports are necessary, especially if the country rock is porphyry.

The sinking of shafts is laborious, because of the difficulty of putting long angling shot-holes. Small shafts are sunk by hand cheaper than by power drills, and almost as expeditious, unless perhaps the continuous system (see No. **91**) is used; and the loss of time in removing all the implements for each shot bears a large ratio to the total. Even in drifting, the actual drilling heat is not more than half of the whole time. The number of men depends upon the size of the shaft opened; only two miners can drill to advantage on an area of 20 sq. ft. A larger size gives more room proportionately to each miner, and permits faster work per cubic yard removed. Though two ten-hour shifts are customary, three of eight hours each are much more rapid. Two machine-drills can work conveniently in an 11 × 10 shaft, and in ordinary rock will sink 5 feet per day; in a 10½-foot circular shaft the progress was 4.6 feet daily. A shaft long in proportion to its width, sunk by two or four machines, has two centre-cut ranges of holes (see **90**), which are independently fired. The cost of sinking is from $5 to $18 per cu. yd. Below 100 feet the rate increases each 100 feet almost as the square root of the depth. Rziha says that in Europe the cost of excavating shafts is from 50 to 100 per cent higher in wages, and the cost of putting in timber 15 to 30 per cent higher in wages than the estimate for the same amount of tunnel-work. In the Lake Superior region one lineal foot of average shaft costs as much as a lineal yard of gangway and a cubic fathom (216 cu. ft.) of stoping. There is nothing but a local criterion for the means of calculating the cost of any kind of rock-work.

Through the first score of feet the progress is quite rapid; the dirt is thrown up to the surface from platforms; beyond this, small shafts can be carried quite satisfactorily for 90 feet or so by windlass, but as an engine must ultimately be used, it were better to place it at the start. The entire section is attacked at once, a small corner sump being carried in advance for drainage and for "bearing in" while shooting. Often a hood is provided for the protection of the miners against falling of small rocks, and trap-doors at the surface too, unless the ventilation is poor. If a shaft is to be prolonged while the upper part is still in use, safety is obtained by opening only that portion of the shaft area not under the hoistway for a distance of 12 or 15 feet, and then widening it out to the entire size of the main shaft. This leaves a roof of rock ("pentice"), Fig. 96, that shields the men. When another lift has been sunk, the pentice is cut away, and another started for the next drop. Hoisting is by underground engine or bucket and windlass. A box-pipe, projecting some distance into the air, from over a stove or burning torch, will furnish almost as good air as a small fan or the air from the power-drills. Except in the neighborhood of oil and gas lines, no especial precautions are necessary against fire-damp.

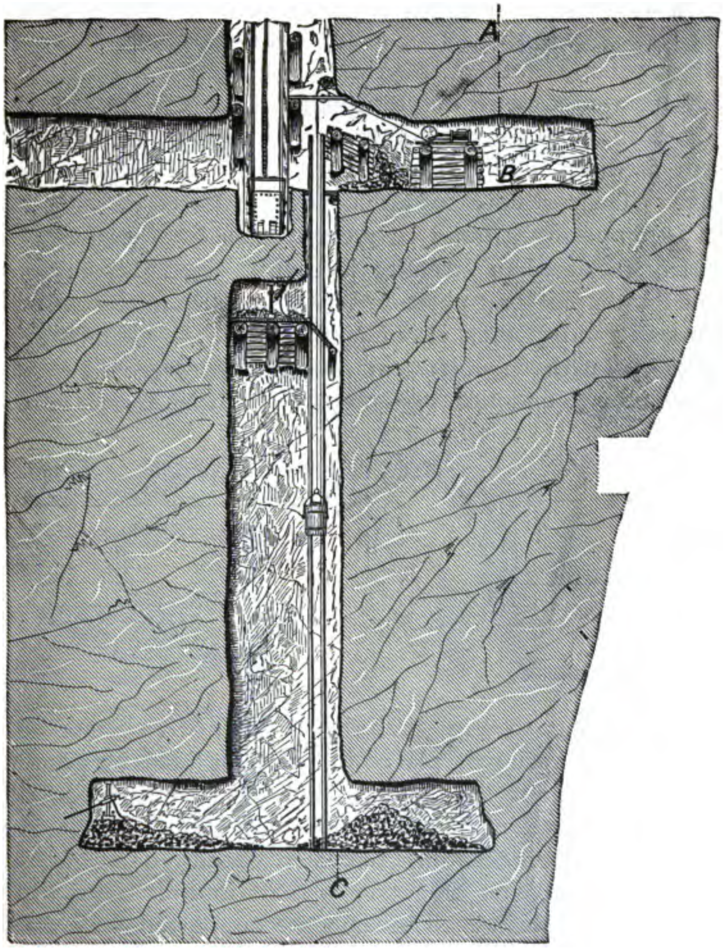


FIG. 96.

62. There is neither safety nor economy in the practice of leaving the shaft untimbered, even if the two walls are hard and self-sustaining, and shoot clean. To resist the thrust of the country, timbering or lining is urgent (see also notes on hoisting). This may be done simultaneously with or subsequent to the sinking, according to the firmness of the ground. Each timber set is supported on its stulls, resting in notches ("hitches") in the rock; or sections rest on heavy reachers at every 25 or 30 feet of depth. Sometimes the timbers are hung from an upper frame by spiking one set to the other.

The timbers are preferably dressed, though hewn logs are much used for solid crib-work where plenty of help and room is had. Their size is not a matter for calculation, as in firm, non-decomposing ground they experience little pressure, and stability rather than strength is sought, the latter being secured by ample shaft-pillars. Under such conditions a lining with stiff guide-planks is sufficient. This may consist of 3″ planks cut with shoulders in sets of four pieces, two wall-plates and two end-pieces (Fig. 97). If cut by template or by machine similar to Fig. 172, they need not be matched in height. This casing is placed in position, lined by plummet at the corners, not spiked or joined, but simply held up to bearings by waste rock packed close between it and the rock. Each 30′ section is held on a pair of 10″ stulls. Two men can complete one section of a 5 by 9 shaft in four days, with a helper at the packing. The men are supported on a cradle, suspended by a rope from the upper stulls. In very good ground this casing will suffice for three compartments, but not for cage use, unless perfectly backed. In bad ground a casing of larger timber, say 8″, is not infrequent (Figs. 78, 98, 99). These are laid "skin to skin," with their ends shouldered.

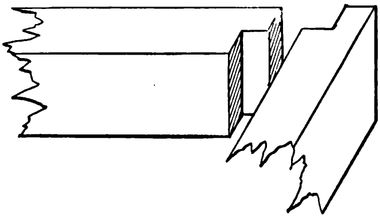


FIG. 97.

FIG. 98.

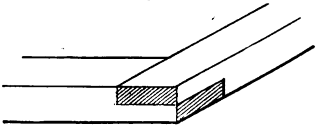


FIG. 99.

The wall-plates are stayed by "buntons" (Fig. 100) bolted or gained into them. The longer the wall-plates the stouter the buntons, the interior side of which and of the end-plates carry the cage-guides. The travelling and pumping ways are partitioned off by planks nailed vertically to the buntons. Another method, which is still better, is to have the wall-plates break joint with the end-pieces instead of arranging the four in a horizontal set. The reachers are hitched into the floor and forced down against the hanging-wall. None of these plans are practicable with inclines, which will require framing.

In framing vertical shafts the stulls are inserted into both walls horizontally. On the reachers four sticks are placed and framed to the studdles or struts at the corners and at compartment portions. On these struts a similar set is framed 6′ above, to in turn support another parallelopiped, and so on up. Planks ("lagging") are driven in around these frames, and the spaces to the rock filled with broken waste. The joints of each timber are of the pattern shown, Fig. 101. Fig. 102 is better carpentry, and quite standard. The end-pieces and struts are usually square, 8″, while the wall-plates are laid 8″ vertically and 10″ or 12″ horizontally. The buntons are stouter as the wall-plates increase in length (Fig. 103). Fig. 16

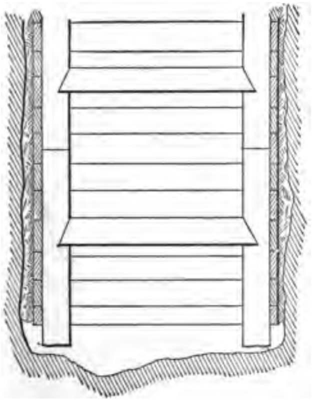


FIG. 100.

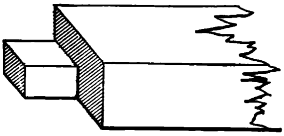


FIG. 101.

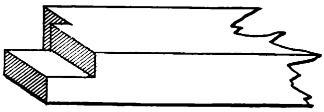


FIG. 102.

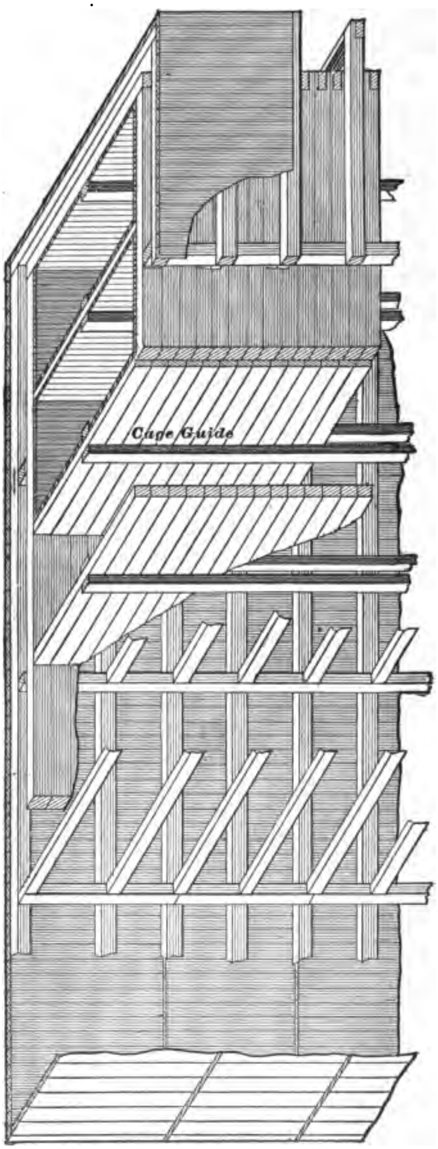


FIG. 103.

illustrates another form of timbering rectangular shafts with vertical corner-plates and horizontal lagging.

Shafts such as the Comstock, 6 by 24, for prolonged, rapid, heavy hoisting are fitted with timber as much as 14″ square, and lagged with 3″ plank. Where friable rock is penetrated, the frames are braced by inclined struts that prevent settlement. When the ground is friable, marly, or wet the methods approach a caisson character. Another plan comprises a stout framing as described, inside of which is another strong planked cribbing, between which clay is puddled to exclude surface water. The B. & O. shaft at Taylorsville, Ind., was thus successfully carried through quicksand; the outside crib was of 12″, the inside of 10″, timbers, with a 4″ puddled wall. The famous Hollenback shaft, 45′ 4″ × 11′ 6″ inside, has a 12″ clay wall for 31 feet deep (Fig. 103). It was designed for a daily output of 2500 tons of coal.

If the timber shows signs of giving way, other means of securing the shaft must be invoked. With expert timber-men the joints may be strengthened or the frame replaced, but it is preferable to reinforce them by sets closer together. Where the expense would warrant it, and the diminished area is not objected to, the insertion of a second lining may secure the works. In the Lake Superior region, after futile experiments with other accessory modes, iron caissons were invoked. In stiff ground they were forced down inside or outside of the old timbers; in soft, they sank by their own weight with the undermining. The cylinders were in segments and sections, bolted at the surface, keeping pace with the progress, averaging a foot a day. A cast-iron cylinder 15′ diameter, 1⅛″ thick, was forced down 84 feet at a rate of 2 feet per day in morainal matter.

Forepoling, a form of sheeting see (Fig. 182), is also quite successful, but requires much timber. When the ground is treacherous there is a constant contention against the rising of the bottom. In the event of this happening, the simplest plan is to floor and brace the bottom, continuing only a small shaft by forepoling (Fig. 104).

For circular shafts the framework descends with the shaft in sections, which, however, are built upward from reachers, bedded whenever suitable foundation offers; or the "curbs" rest on a properly dressed ledge of the rock, and are firmly wedged against the sides. The timbers, assuming the character of voussoirs, are hooped with iron and called "curbs." The timbers composing the curbs may be mere wedge-blocks, or are long enough to form a regular polygon, when they are held by dogs. In ordinary ground the sets are held apart by props, and the solid-packed lining backs them. Otherwise they may be formed into a solid walling, often suspended from a heavy frame at the surface by iron rods. In any event the joints and fitting receive the greatest care, and many of the old shafts are high types of the carpenter's art.

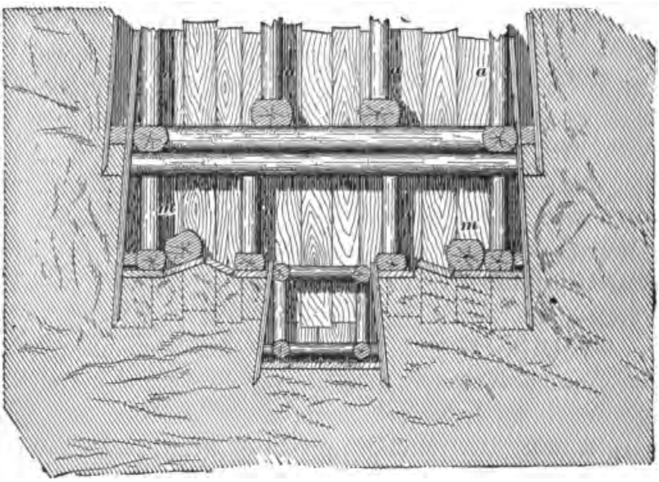


FIG. 104.

The increasing scarcity and cost of large timbers, the expense of fitting and maintenance, their short life, and, finally, the corrosion of spikes and splice-plates, with the consequent leakages, have caused the abandonment of wood tubbing, and the adoption of iron and masonry for all permanent ways. The effect of the heavy, hot atmosphere of the mine upon timbers is a decomposition, that is not always detected on the surface, and once begun, only better ventilation can delay ultimate destruction. Dry timbers should be frequently probed; alternations of wet and dry are exceedingly destructive; wet timber will last longer than dry. Preservatives have been attempted, with much success. In salt-mines steeping in brine gives great endurance. The sulphates and chlorides of zinc have proven excellent antiseptics; and a grand opening offers to the discoverer of a means of freeing the lead ores of the Western States of the obnoxious zinc, and at the same time utilizing it as a preservative.

The use of masonry for the walling of shafts involves but one disadvantage: it presupposes ground that will stand safely for a couple of weeks without much support. Before the permanent structure can be introduced, a considerable depth must be reached, to obtain a sure foundation upon reachers, or upon a ledge from which the masonry is erected, the temporary timbering and bracing being gradually removed as the construction proceeds. When a very secure ledge or base cannot be had, a conical chamber and a solid crib are built to support the walling.

If the pressure from the walls is not great, the brick or the lode-rock is built up in plane walls, packed behind by waste. Often the mine water carries matter in solution that cements the whole into one solid, mass. When great pressure is expected, the sides are arched toward it; and in very bad ground all four sides are curved, or the circular form is adopted. The arc should be such that its chord is perpendicular to the direction of pressure. In such event, the foundations for the sections are curbs of overlapping timbers patterned to the curve, or of late years of cast-iron, with slabs of wood at the joints. The packing behind is carried up with the brick or masonry until the ledge of the upper section is reached, when it is removed gradually and the two sections united. In some instances the masonry compartments are built at the surface and lowered into place. Brick is well adapted for quick arch-work. A 13′ cylinder is four half-bricks thick; the labor of laying from a staging is one and a half days per M. The rods *b* support the masonry (Fig. 105) from buried beams (*a*, *a*, Fig. 106).

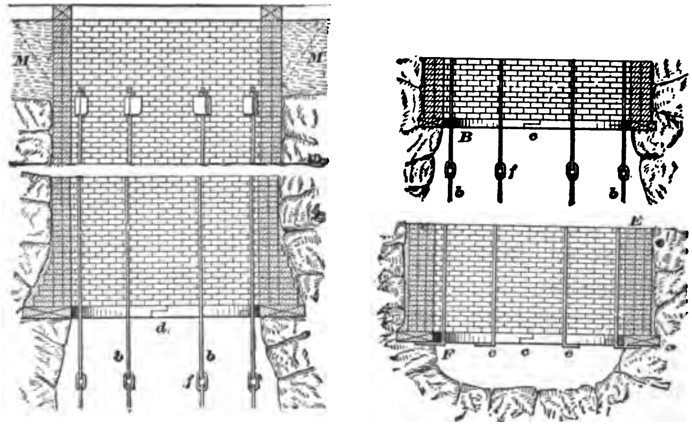


FIG. 105.

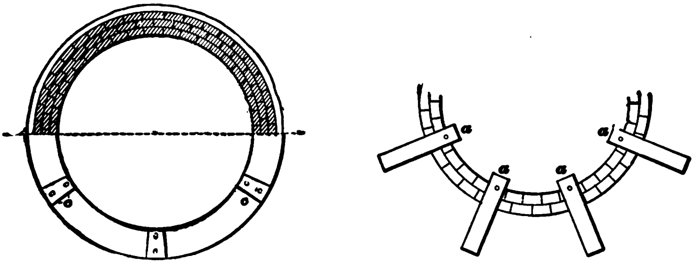


FIG. 106.

Masonry is heavy to support, and not any cheaper now than iron, with which many shafts are successfully curbed. Rings of I beams or channel-bars form the curbs, upheld at proper distances apart, by struts of wood or iron, and backed by heavy planks or $\frac{3}{16}$ sheeting (Fig. 93). English engineers use old railroad iron similarly. Prepared at the surface, the curbs may be lowered into place and quickly set, with little labor. A packing of concrete is used at Saarbruck, giving increased strength and durability. It is estimated that the initial cost, placed, is 2.2 times that of wood and 1.2 that of masonry, but maintenance is one-third that of wood and nearly the same as with masonry, if the shaft is dry.

The sinking and timbering of a slope are similar to gangway-driving, except that the sill is indispensable to the set. It must be well bedded, and let into the rocks on each side to prevent the roadway from slipping down hill. Ofttimes it is stayed by plugs driven into the floor. The sets are also braced against one another by longitudinal studs, between the posts, at their head.

CHAPTER II.

SINKING IN RUNNING GROUND.

63. Precautions taken to exclude water; tubbing; description of and estimates for Triger's method. 64. Kind and Chaudron process of tubbing and sinking through watery strata; description of the tools; estimate of cost; applicability and advantages; examples; Haase's system; J. Mill's Californian method; Poetsch's freezing process.

63. WITH the increasing avarice, the search for minerals is penetrating more and more treacherous ground, and to greater depths. The almost insuperable obstacles are one by one overcome. The works become of a most delicate nature, as the material to be traversed is a water-bearing stratum, and the underground currents are encountered. To pass through the strata without letting the water into the mine is an undertaking the success of which is phenomenal. Often the expense of pumping while sinking is the largest item of deadwork, and frequently compels the abandonment of the works. The author has occasion to regret investment where 10″, 12″, and 14″ pipes were delivering water from an intended shaft, which at 150 feet was abandoned for another mode of procedure.

Two varieties of cases present themselves—one in which the ground penetrated is quite firm but watery, and the other includes running soil, marl, quicksand, etc. In the first case the pumping facilities must be ample, or the water kept back during mining; in the second, the excavation is more rapid than the facilities for removal to the surface permit, and there is the danger of overwhelming the laborers with soil. In either case the ground traversed must be insulated by a tubbing, hermetically sealed above as well as below the soft measures.

This not only renders sinking possible, but it excludes water and stuff from the mine, and permanently dispenses with much of the pumping arrangements. It very seldom comes into play in vein-mines; with the verticality of the lodes the water cannot be prevented from working down somewhere into the mine. It is in the stratified regions that the use of the crib is of the highest importance. Beds of gravel, sand, and clay, or porous strata, percolating large quantities of water, are not easily traversed or held up without a strong water-tight lining, for the pressure of the moving material tends to make the bottom rise, as well as threaten the sides. A deep shaft in such a region may encounter several occasions for such tubs, which under suitable conditions may be introduced in lengths as required, and only to the extent of the soft ground. Still, it would give more substantiality to the work to form one continuous length of tubbing, even across the good ground. It is not uncommon to find in Germany shafts with three sections of iron tubbing, united by lengths of brick or wood lining.

This process of tubbing consists in confining the seepage area to that of the bottom only, by building a water-tight cylinder lining to the shaft, and carrying it down with the sinking beyond the wet stratum. In England, a bed of sand called the Lower Red Sandstone, which is almost fluid, has several shafts tubbed through it. In Britain, Belgium, and the North of France several mines are reached by tubbing through the fissured chalks and marls of the Cretaceous. The Thonmergel of Germany is frequently tubbed to the Bohn Erz, below, dry enough for work. While sinking the Murton pits 4000 gallons were pumped per minute, and the " come in" of water for the Exhall shaft was 1650 gallons. Still, the inefficiency of this plan, sometimes called the English system, is recognized, and several methods better applicable to loose and watery beds have been applied with more or less success. Excepting the Poetsch method of freezing the ground to be penetrated, they are modifications of the diving-bell, or pneumatic pile.

Wrought-iron tubes in segments are bolted on at the surface as fast as the lowering proceeds, until the secure, impermeable bed is reached. Here a smooth base is prepared for one or more wedged curbings, behind which moss or concrete is rammed. The tubbing is backed with rock or concrete all the way up, and connected with the next upper section. The holes in the segments, for convenience in handling, and to relieve the tubbing of pressure till the work is completed, are plugged up. The early practice of bolting the segments together through the inside flanges was soon abandoned, and now the flanges are outside, wedging, pressure, and friction keeping them. On account of the curious accidents occurring from the pressure of air locked behind the tubes, it is advisable to lay a pipe to the surface for the gas to escape, and, similarly, to relieve the water. A shaft of 16′ diameter was sunk at a monthly average of 104 feet with four shifts of 8 and 10 men each. Several Canadian salt-mines, having shafts 10′ 6″ diameter and reached at a depth of 1150 feet, are tubbed through 260′ of water-bearing strata, in sections 2 feet high and $\frac{5}{8}''$ to $\frac{13}{16}''$ thick. The columns rested on iron curbs with firm base. The joints are calked.

Tubbing of masonry or of timber, once much employed, is cheaper than iron. With good hoisting-machinery, three masons in four-hour shifts finish 10 feet per day of a 16-foot shaft. Towers of masonry, resting on an iron curb with a cutting edge, were built on at the surface; while, to facilitate the sinking, digging was being carried on below, or, if the material was wet, a process of "bagging" was employed. When abundant in size and quality, wood gives great satisfaction, as being elastic, easily laid and repaired—qualifications not possessed by masonry. Iron offers the advantages of strength, combined with a facility of handling, which recommend it for large shafts and enormous flow. Though it is not possible to presage or measure the pressure, and thus determine the kind and strength of tubs, a thickness, of 12″ wood, 7″ masonry, and $\frac{7}{16}''$ iron may be suggested as common. As a matter of fact, the tubbing should taper off toward the top. In many cases, however, the use of 12″ staves hooped with iron did not prove adequate, nor did the backing of 12″ more of concrete help matters; where the shafts were not abandoned, $\frac{5}{16}''$ sheeting met the emergency.

When a bed is encountered of so soft a material as to behave like a fluid and be pumpable, not even as small an area as that of the shaft can be opened with safety: in this event some variation of the "spilling" processes may be employed (**71**); or, failing of them, Triger's plan or the Kind and Chaudron process will succeed.

M. Triger employed, 1839, the principle of the pneumatic pile, in which the iron tubbing, an indispensable complement to boring through watery strata, extends down to an air-lock communicating with a diving-bell at the bottom. The atmosphere of the caisson is maintained at a pressure of not over 60 lbs. per square inch, and checks the influx of the sand, which the miners shovel to a sump, whence it is aspirated to the surface. Meanwhile the tubbing is being rammed at the surface. In the air-lock, which connects the diving-bell with the surface, the men are prepared for the change in pressure. A windlass in the air-lock and one at the surface raises the coarse stuff in two stages. The physiological difficulties prescribe the applicability of this method to a depth of not to exceed 150 feet.

64. Over a very extensive tract of country in France and Germany the loose, watery marls presented difficulties which the methods described failed to overcome. What with pebbles and fine rock interfering with aspiration, water completely inundating the shafts, and the difficulty in establishing water-tight joints, the operators were routed. In 1850 Herr Kind devised a scheme for mechanically sinking shafts, just as one does a bore-hole, and still further conquered difficulties hitherto insurmountable by a variation in the mode of lowering the tubbing, and by a device for regulating the influx of water. When M. Chaudron added the sliding bottom-piece to form a perfect joint, after the Kind boring-tool had prepared the base, the acme of shaft-sinking was reached. Since 1862, when the first shaft was sunk, 6 feet wide, 480 feet deep, at a cost of $450 per foot, not a single fatal accident is recorded against the process, which owes much of its success to the fact that the sinking and lining are completed before a soul enters the shaft. Two abandoned shafts, through soil feeding 11,000 gallons per minute, were carried down 267 and 216 feet, respectively, in 23 and 20 months, with a cost of $280 and $340 per foot. In the latest application 569 feet were sunk, 16 feet diameter, at an average cost of $143 per foot for both shaft and lining, which alone cost $70 per foot. With a guaranteed success at so low a rate, it is surprising that American engineers, usually so progressive, have not employed this method before acknowledging failures; but no attempt to introduce this plan here is as yet recorded.

The mode of procedure consists in drilling a hole, generally about 4.5 feet wide, in which accumulates the débris; a larger tool 50 feet behind widens the shaft to the required diameter. The drill, called a trepan, is suspended by rods from a walking-beam operated by an engine.

The small trepan (Fig. 107) consists of a blade of forged iron, into the lower side of which are keyed a number of pointed steel teeth, and a stem connecting the blade to the suspension-rods by means of a sliding-box. This last partially corresponds to the "jars" (Fig. 186) of oil-well outfits, takes up the jar, and is an essential element of the tool. The trepans are massive;—for hard rock, weigh from 8 tons up,—and are raised 6″ or so turned slightly for each blow, and dropped; their concussion disintegrates the rock along a diameter of the circle. The progress is from 3″ in flint to 3 feet in chalk per day; 1 foot in sandstone and 16″ in coal measures.

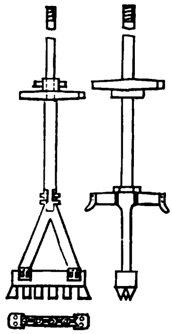


FIG. 107.

Most of the material is broken quite fine, though 2″ and 3″ stuff is not unusual. When the hole has advanced some distance a larger trepan is attached (Fig. 108). This is similar to the smaller one, but the blade is deeper at the centre than at the ends, so that its teeth cut a base sloping to the centre. The central, toothless portion of the tool has a U-shaped guide that fits the smaller hole. This tool, often weighing as much as 15 tons, cuts the shaft to full width, or it may be succeeded by another similar reamer, the detritus falling into the smaller hole, from which it is hoisted by the sludger (Fig. 109). In alternate stages the drilling and widening progresses, while simultaneously the tubbing is lowered by a separate engine. All these operations being conducted under water, the trepan requires to be automatically kept vertical. Two guides, carrying at the extremities horizontal and vertical cutters, accomplish this marvellously well. A record of the preliminary shaft, 4½ feet in diameter, showed for 508 feet an average progress of 3.3 feet per 24 hours, divided up as follows: 51 per cent of the time was occupied in drilling, 19 per cent in raising and lowering the tools, 20 per cent in dredging, and 10 per cent in repairs and delays. Widening the shaft to 14 feet and down 460 feet took ten months; reaming, 46 per cent of the time; altering and operating the tools, etc., 14 per cent; dredging, 22 per cent; delays and repairs, 18 per cent.

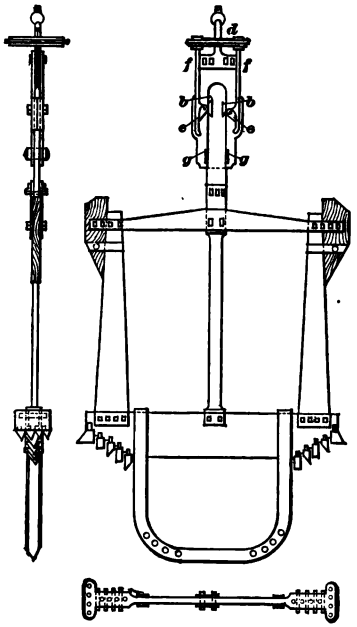


Fig. 108.

Fig. 109.

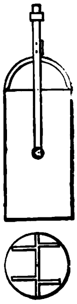


Fig. 111.

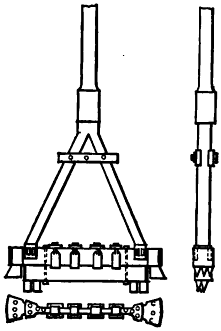


Fig. 110.

The small trepan is illustrated in Fig. 107, a widening trepan with double blade in Fig. 110, a reamer in Fig. 108, and the sand bucket-dredger in Fig. 111.

There are no screws or nuts to loose; all the parts are keyed to place; but special tools are provided for grappling broken rods, stems, trepans, teeth, etc.

At the surface, the operations of boring the pits, building and lowering the tubbing, puddling and sealing the base, are conducted with engines and capstans from a tall derrick, at which extra lengths of rods may be attached with the progress of the drilling. Nine men are employed about the works, only three of whom are skilled laborers. The cost of the installation of the machines, tools, etc., all of which are portable, is about $13,000 to $20,000.

The tubbing, which is indispensable to operations of this character, is of iron sheeting, built on in 6-foot sections, with leaded joints, and suspended by rods (Fig. 112). The flanges come on the inside of the tubing *bb*, leaving a perfectly smooth exterior, the joints true and bolted together. Two sections are lowered daily. As an example is quoted a tubbing 12′ 7″ internal diameter, 280 feet high; it was 1″ thick at the top, $1\frac{5}{8}''$ at the bottom, and weighed 400 tons. The sections were 5′ high, the flanges $3\frac{1}{2}''$ wide, 2″ thick, having leaden wedges between, $4\frac{7}{8}''$ wide and $\frac{1}{8}''$ thick, and 20 bolts $1\frac{3}{16}''$ diameter.

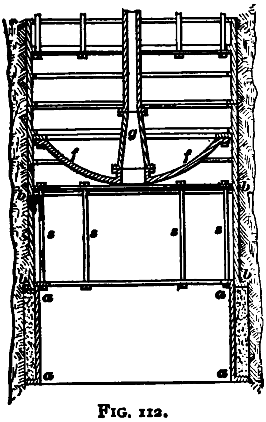


FIG. 112.

At the bottom of the iron cylinder are attached two very ingenious appliances, which, operating automatically, have established the process as a success beyond all cavil; the first is a moss-box *a*, for hermetically sealing the lower end of the tubbing against any influx of water; and the second is the introduction of a false bottom, *f*, by which the sinking of the tubbing is cleverly controlled. These are both adapted to the bottom of the tubbing, as is illustrated in Fig. 112. All the flanges of the tubbing turn inward except the lower one of the bottom section, *bb*, which is outside, and may act as an annular piston to a lower section, *aa*, of smaller diameter, the upper flange of which turns inward, and the lower one outward. Between these flanges, the moss-box, and the rock, the annular space is filled with moss, which is not, however, under compression so long as the screw-bolts *ss* support it from the tubbing. It operates like the seed-bag of oil-well diggings.

The false bottom *ff* is attached to the tubbing, with the lower sections of which it forms a diving-bell, that floats the whole system. The greater the head of water encountered, the more complete the balance, and the greater is the relief to the rods *dd*, supporting the hundreds of tons of iron. The safety-pipe *g*, with cocks and plugs operated from above, is an equilibrium column that permits sinking, or rather regulates its speed. Opened at the top, sinking proceeds rapidly, as the compressed air and water find vent; closed, the whole structure is upheld against gravity. When the plugs are opened they discharge into the tubbing, weight it with water, and at the same time release the pressure below. By proper manipulation, therefore, perfect control is had over the lowering of the casing.

When the tubbing has traversed the water-bearing measures, a seat having been scraped for the moss-box, the entire weight is allowed to fall on the annular piston *b*, by opening *g* at the surface. The moss is compressed to a hard, water-tight mass, the rods *s* gliding in their bearings (Fig. 113). Up to this time the shaft is more or less full of water (the process is independent of the amount of water encountered), which may now be pumped, but usually is not until a cement backing has been inserted and hardened to insure solidity; after that, if the joints of the metallic column are well made, the shaft is perfectly tight, and the mine is insulated from the subterranean current. The introduction of the cement is effected by a closed spoon holding a barrel or or so, curved to suit the space. Three sets of six men each do this work, burying 400 cu. ft. per day, at a cost of

FIG. 113.

about forty cents per square foot area of lining. A solid foundation of wedged iron curbing is subsequently built on a stout ledge, to take the weight of the cribbing after the other work has been completed.

This method is generally applicable to conditions of soft ground, and especially in watery ground, which can be pierced without recourse to ponderous pumping machinery. Though the pressure of the water is not essential to success, it materially facilitates operations. In a few cases, where the ground was merely wet, not running, tubbing was cheaper than this method. But the facts should not be lost sight of, that none of the delays, perils and discomfitures of the ordinary methods are here experienced. Its progress is greater, and initially its maintenance is cheaper than other schemes for wet ground, besides never having had a failure. True, the size of its shaft is limited to about 14 feet.

Herr Lippmans is using a drill of a double V-shape, instead of a straight trepan. It does faster work, as it cuts equally at the periphery with the centre of the circle. With Kind's trepan the blows fell too far apart at the outer edge of the shaft, and too near together at the centre.

An objection to the Kind and Chaudron method is, that there are no means of knowing when the water-bearing stratum has been penetrated and when the tubbing should cease, unless the preliminary geological examination has revealed it.

A short while ago a pair of shafts were sunk in Samlund, Eastern Prussia, for amber, through 147 feet of clay and sand, by a variation on this method. The drill-tools, weighing 1700 lbs., cut a 4′ 6″ space, though they had little to do except in the shale-beds. No moss was necessary, as the ground was not wet. Four-feet lengths of tubbing were forced down by jack-screws, each shift with 27 men. The total weight of tubbing in each shaft was 45 tons, and the total cost $17,500.

Haase's system is a modification of sheeting-piles,—small round iron cylinders driven close together to form a cribbing for the intended shaft. The tubes were about 15 feet long, ¼″ thick, and 4″ diameter, and enclosed an area 10 × 7, which was then timbered. These were driven through 90 feet of quicksand in five months' time, at a cost of $135 per foot. While standing, they gave good drainage, and did not yield when the excavation of the shaft began.

Jas. E. Mills of California successfully carried a shaft 55″ wide for a distance of 210 feet, through water-bearing, loose material, driving a sharp-toed caisson by screws, and later by hydraulic pressure. Excavation was effected by a modification of the sand-pump of artesian-well boring. It was small, and hung from a movable clamp, which received the reciprocating motion of the engine. The weight of the tool was over a ton; 32 lifts of 2 feet each per minute gave an average daily rate of sinking of 1.43 feet with the employment of five men.

Another method was applied at mines near Harhausen and Nachtenstedt. The floor was paved with small inverted wooden funnels about a foot square. These were forced down by rams, and the loose soil squeezed up through the cones, and cleared off. It resembles somewhat the method of wedge II, **70.**

The long-hole method of sinking is described in II, **91.**

For dealing with loose saturated alluvium, Mr. Poetsch has originated the novel idea of freezing the mass to a solid, by boring a series of holes by the diamond-drill, about three feet apart, lined with copper tubes, inside which are smaller tubes. A very concentrated solution of the chlorides of magnesium and calcium circulates through the tubes and freezes the ground, after which the pit can be excavated in the centre of the mass in the ordinary manner, and the tubbing put in. This refrigeration is continued till solid rock is reached. It has been applied successfully in the anthracite and Lake Superior regions, with 23 holes of 12″ diameter to the shaft.

No safe criterion can be given by which to estimate the cost and progress in sinking shafts. The nature of the ground, the inconveniences and delays to working, hoisting, and pumping are features peculiar to each individual enterprise. The cost per lineal foot increases with depth. The Hollenback shaft, to which reference has been made, averaged $106 per foot.

CHAPTER III.

TIMBERING.

65. The use and preservation of timbers; for jointy rock, horses, and disintegrating rock; consumption of timbers in mines; selection of timbers. 66. Props, sprags, stulls, and their plates; formulæ for strength and the calculation of their dimensions; variety of joints. 67. The construction of setts, frames, etc., for various conditions of roof, walls, etc.; timbering for levels, gangways, gob-roads, and for support of vein, gangue, etc.; in salt mines; lagging; wood, iron, and masonry for levels. 68. Square setts, joints, and sizes of parts; full account of the American method; cribs for rooms; timbering of mill-holes, underground chambers, plats, and winzes; timber-man's tools; framing-machines.

65. WHEN one examines the story that the accident-tables of page 222 tell, it becomes manifest how the neglect of a few simple rules endangers life and property; and in no respect is this more painfully impressed than by the mortality record of unpropped rock. Excavations, even in the "rock of ages," cannot be left open any great length of time without support, which, if introduced in time, will prevent disastrous results. Successful superintendents personally watch the timbering and the face-rock diligently, and guard against any springing of the walls. All the effects of pressure are intensified by neglect, and the secret of success is to place timbers before movement begins. Supports are not for bad roofs only; while "awaiting a weak spot, the good roof, so called, catches him," and his stope or room is lost. The eagerness to quickly win the face, while pardonable, promotes avarice, parsimony, want, and then provokes collapse.

Though the conditions underground are such that very simple timbering is required compared with that on the surface, the tendency of the time is toward the employment of special timber-men to make and place the supports. In rooms, driving in soft ground, and the like, the miners must at once prop the excavations. For this reason—and the proscribed space—the character of the timbering should be simple and the sticks light. Fortunately, the pressure of the country rock is inward and toward the openings, and compression, not tension, as on the surface, is to be combated with. This tends to hold the sets together. Tenons and framing may therefore be dispensed with, except in loose ground, where they are beneficial.

The relative merits of the different varieties of wood need not be discussed here. Oak is undoubtedly the most preferable, but the mines take what can be had in the vicinity. Above "timber-line" we are content with "scrubs." Sawn timber is better than hewn, on account of its better resistance to decay; and durability is of importance prime to strength. Again, green-wood is heavy; the ordinary 10″ stick, say 6′ long, is as much as three men can well handle. Lightness is an essential feature in this most onerous of underground work.

The life of timber varies with the conditions of the atmosphere and care in dressing. It is rarely as great as that of railroad-ties (twelve years). In many mines head-pieces crumble after two years' standing. Wood rots faster, and shows it less on the surface in dry, vitiated air than in moist air. Alternations of temperature or moisture are very destructive. A cotton-fungus mould is a sure indication of bad air, and, being contagious, requires attention at once.

The decay results from the fermenting of the albuminoids of the sap, the admission of water, and the attack of insects, to which several causes contribute,—bad air, damp air, standing water, and oxidation,—causes all of which are mitigated by an active circulation, and materially remedied by saturation of the pores with some antiseptic. Creosote, Kyanizing, or Burnetizing will give greater life to timber. Wood creosote is better than tar creosote, because less volatile. Ten gallons are necessary for 1 cubic foot of coniferous wood. Sulphate of copper is the best preservative for oak; tar for fir, pine, poplar, etc. Zinc chloride or sulphate does fairly well.

It is this destructibility and the increasing scarcity of supply that is bringing about the employment of iron and masonry as the certainties of future support. Fortunately, the metal-mines above "timber-line" require but a moderate supply. In others, however, the consumption is alarming. The annual estimated timber construction in anthracite mines is 1 cubic foot per ton product; in the L. S. copper-mines, 1¾; in Leadville and L. S. iron-mines, 3; and in Nevada, 4½.

Every 100 cubic feet of coal extracted consumes 3.4 cubic feet of timber; every ton of excavation in running ground requires about 5 cubic feet to support the balance. In the Anaconda mines, Mont., 80,000 cubic feet of timber are used daily; 2000 of such mines would consume the entire forest-area growth. The West Vulcan iron-mines, in L. S., annually consume 2,000,000 feet of lumber and 60,000 pieces of lagging, at a cost of 37 cents per ton of ore mined. In the copper-mines of L. S. this item amounts to from 15 to 31 cents per ton of rock hoisted. So important in the economy of mining and to the safety of lives, the selection and placing of timbers should therefore receive skilled attention; adequate ventilation is equally urgent for their preservation.

The most important question is as to whether the timbering is to be done in a substantial manner at once, or to be considered as provisional. This is only answered according to the importance of the gangway. Rooms and stopes are only temporary, and treated as such. The excavations may be filled with waste or timbered up during the period of their mining. Gangways may or may not be subsequently cribbed or masonried; pump and machine rooms and stables are very substantially lined.

In any event, each piece should be placed conformably to the principles of the strength of materials, and employed in such a manner that its resistance to crushing, rather than its resistance to bending, be brought into play. Joints should bear the pressure uniformly and their planes be perpendicular to its direction. Sticks should be placed in the line of the pressure wherever practicable, or in such manner as to act like or take part of an arch; then, when any movement takes place, its effect will be to tighten the timber in place. Every mortise or joint impairs the strength of the stick.

66. Single sticks are much used as props to hold the roof back of the men. In long-wall working a large number is used, resting on the floor or on a plate, and hammered into place with a wedge-plate at the roof. They are 6″ or 8″ diameter, and stand 3′ apart, in two or three rows, beyond which the roof caves in on the gob. They remain only a few days, are removed by rows to let the roof cave, and are replaced nearer the face (Fig. 8). An average of 70 per cent are recovered; some of the balance cannot be removed; others would endanger the timber-men. Flat caps on top, 20″ × 10″ × 2″, are ample for most bad roofs. Slate requires a large plate. It is a poor roof, because it crumbles from the presence of pyrites; sandstone or conglomerate makes a good roof; soapstone is bad; but the most dangerous is fire-clay, which melts upon contact with the air. Props come into play (Fig. 8), 12″ long, 3″ or 4″ in diameter, for holding up holed coal. The props have to support not all the strata above the coal,—this the pillars do,—but a portion of the immediately overlying seams which constitute the roof; and its condition determines the number of props, which is greater for a brittle stratum, and less for a flexible roof.

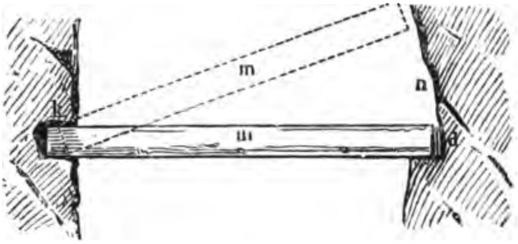
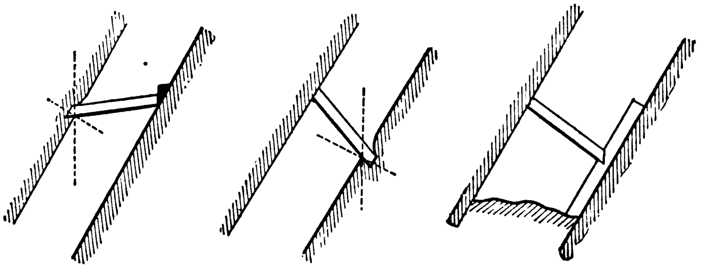


FIG. 114.

States vary in their requirements as to the nearness of the props to the face, but 15′ is the farthest allowed in any coal region. A distance of 5′ or 6′ allows ample mining-space.

In metal-mines the prop is used as a stull (Figs. 114, 115, 116), resting in a notch ("hitch"), generally on the foot-wall, unless the hanging-wall is much softer, and driven into place with a wedge-piece, by mallets. In veins of small inclination the stulls are normal; otherwise they set between a normal and the vertical. They are round or dressed, and of a size and distance apart dependent upon the weight of waste stope-rock to be upheld. It is better to increase the number than the size, though their strength is directly as the cube of their diameter. $d^3 = 0.05hw^2m$ is the formula for calculating the size of any round stull held at two ends. h is the height of rock along the vein; m the distance between the stulls; w the width of the vein; d the diameter of the stull. For 60′ of stull



FIGS. 115, 116, 117.

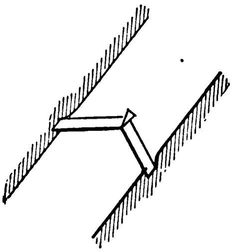


FIG. 118.

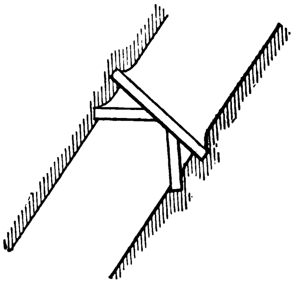
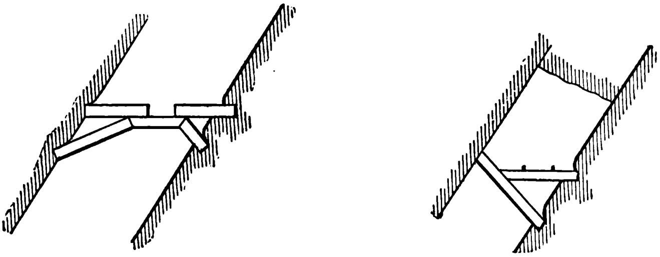


FIG. 119.

dirt, the timbers, 7′ long, 30″ apart, are 12″ through, neglecting the friction of the mass on the wall.

If either wall is soft, a broad slab or post laid against it takes the thrust (Fig. 117).



FIGS. 120, 121.

Should the distance between the walls be too great for a single convenient-sized stull-piece to be used, the use of the smaller sticks, in a manner indicated in Fig. 118, is common. A wedge or plank is braced againt the walls and extends longitudinally with the drift to be covered. This is not so good as

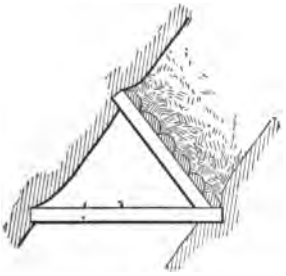


FIG. 122.

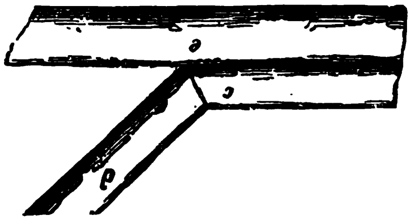


FIG. 123.

Figs. 119 and 120, wherein one or two struts relieve the stull; or Figs. 121 and 122, for underground work for the floor. In Fig. 123 *c* acts like a straining-beam to *d*. Not infrequently the caps may be supported by struts in flat seams, like Fig. 124, or a single centre-prop in double-track gangways and slopes (Figs. 125 and 126); but besides taking room, they are the cause of too many accidents. In fact, much depends upon the cleverness of the men in setting the timbers to the best advantage. For example, a curved stick is beneficially placed if used as shown in Fig. 127. It then becomes an arch. The temptation to cut and notch and spike should be restrained, or the tenacity of the fibres will be destroyed. The use of wedges should be avoided where possible, for the weight may fall on one corner and detract from the strength of the post.

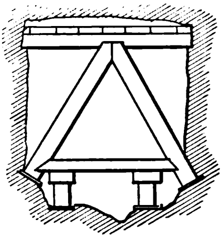


FIG. 124.

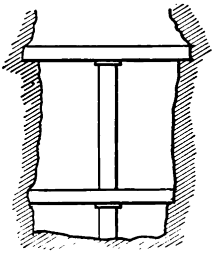


FIG. 125.

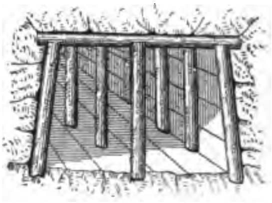


FIG. 126.

The presence of seams and cleavages traversing one another in the rock materially affects the selection of the modes of timbering. The parallel joints are not troublesome. Horses in the vein usually require attention, as do evidences in coal seams of sigillariæ. The latter occur like truncated cones, base down, and the circular layer in the roof should be propped as soon as observed.

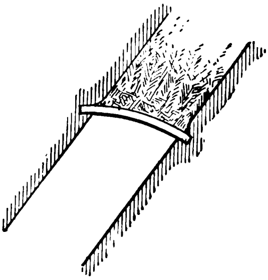


FIG. 127.

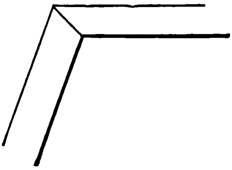


FIG. 128.

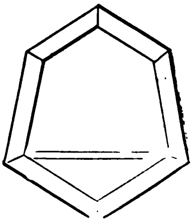


FIG. 129.

In the cases thus far illustrated, the object aimed at was simply the support of the roof or of the stull-dirt—vertical pressure only. When the stress is also from the sides, frames are made in sets composed of a cap or collar resting on two posts or legs studded on a sill or sleeper. The trapezoidal form is stronger than the rectangular, and is equally serviceable for car-way. This form is susceptible of many varied modifications of shape, frame, and joints.

The only joints desirable from every point of view are the flush or butt (Figs. 128 and 129), provided they are cut with precision. Whether the sticks be round or square, the joints should be flat. Never should a round cap be made to rest in the hollow of the post (Fig. 130), for the fit cannot be made perfect nor the splitting of the post prevented. The cap should be shouldered to bear flat on the leg (Figs. 131 and 132).

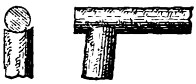


FIG. 130.

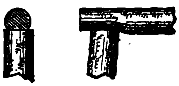


FIG. 131.


FIG. 132.

When the cap receives vertical pressure only, its entire width bears on the legs, as in Figs. 133 and 134; if the pressure is partly from the sides, the joint is dressed to (Fig. 135); for Fig. 136 the lagging and backing must be firm. The prop and collar joint (Figs. 137, 138, 139, and 140) is simple and effective; the bevel-joint is not uncommon in mining-work (Fig. 141);

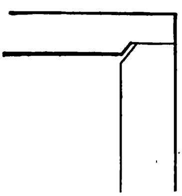


FIG. 133.

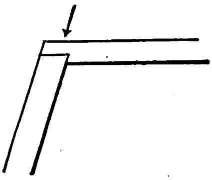


FIG. 134.

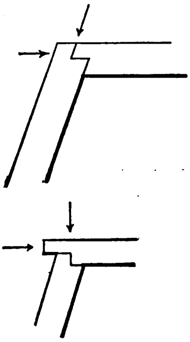


FIG. 135.

Fig. 179 is an elaboration of it, seen in large tunnel-work, but is a very injudicious concentration of pressure at one point, the avoidance of which is the very design of framing. The mortise and tenon is very rare in underground work, except in pump-rooms and the like. So, also, there is little use for the scarf-joints, unless perhaps in building beams, for arch centres. Wedges and head-blocks are essentials in the tightening of frames and to lengthen timbers. Their removal eases the work of reclaiming the sticks whole. The joints ought to be tarred for effective preservation.

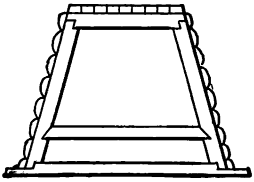


FIG. 136.

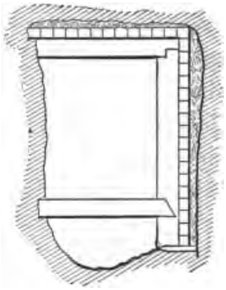


FIG. 137.

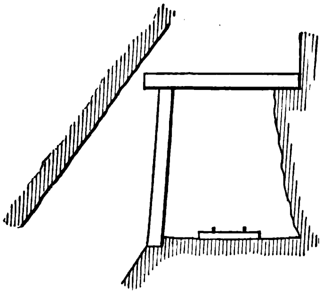


FIG. 138.

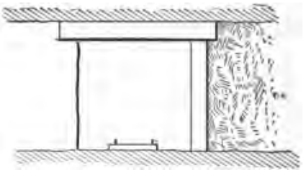


FIG. 139.

Except as clamps, dogs, staples, bands, and spikes, little iron is used in underground work—perhaps 1 lb. for every 100 cu. ft. of lumber placed. Timbers near blasting are often fastened, and bands are used around pieces tending to split. Iron props and screw-jacks may be advantageously employed ; otherwise the use of iron is not commended, except as auxiliary fasteners, for centres, etc. Iron rusts, particularly in contact with ligneous matter.

The dimensions of the sets are a clear height of 5′ 6″ or 6′ 6″, and a width dependent upon the number of compartments. A single way is about 4 feet, though this leaves little spare room. The least dimension of a heading in which miners can work conveniently is 3′ wide and 4′ high. A width of 5′ at the bottom, and at the top of 4′, is ample for all purposes of a single way. A double way should be 9′ wide for men and cars, and 7′ high; and a three-compartment way is not much wider, because no provision is made in the haulage-ways for the men who travel in the third compartment. Attention to table on page 222, where it will be seen that fully 12 per cent of the mine fatalities are from crushing by cars in haulage-ways, urges ample room for passing cars, and numerous niches for retreat.

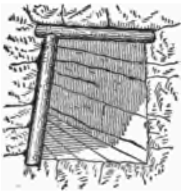


FIG. 140.

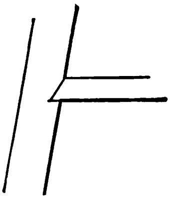


FIG. 141.


FIG. 142.


FIG. 143.


FIG. 144.

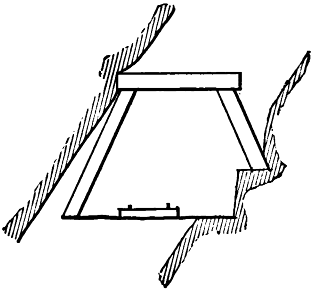


FIG. 145.

Though the total cost per lineal foot may be somewhat greater for a wider drift, than for one smaller, the cost per cubic yard of broken rock is less, and the difference in the cost of timbering is slight, but the gain in rapidity markedly favors large tunnels. Compare the quoted results, Rziha's table, No. **70.**

Careful observation in the Hoosac Tunnel showed that in a heading 16′ wide a man drilled 42″ of holes, removed 0.22 cu. yd. of rock, and advanced 0.058 lineal foot per day, when he could only do 30.119″, 0.134, and 0.047, respectively, in an 11′ heading.

In the standard coal-seams a great height is not admitted, though the roof is not infrequently ripped to secure mule height. When the gangway serves as a ventilator or gutter more height is required, and the timbering for the purpose is illustrated in Figs. 142, 143, and 144. The dimensions of the sticks vary; though 14″ is the average used. Large timbers are often needed, but not placed because inconvenient to handle. Instead of building of thicker pieces, the sets are placed nearer than the average—3′ 6″; skin to skin is not unusual in shattered ground. The height of slopes depends upon the mode of haulage. The use of carriage requires great height; with a dip of less than 40°, the height is about 7′. For skips, the ordinary height of 6′ will do.

Though four pieces constitute the frame, the sill may be dispensed with where the floor is not bad. In slopes they are essential, being wedged into place to secure the trackway. In laying the sills the trench is dug lower in the centre than at the ends, and they will not break. Near their ends the posts rest to support a cap, which is wedged into bearing.

Where a flat road is driven through firm material, the roof of which only needs support, the post and collar form of timbering (Fig. 139 or 140) will suffice. In pitching seams a variety is employed, as Fig. 138, when the vein and country rock are sound and the hanging-wall soft. With a bad roof and good vein one leg is floored (Fig. 145); the other, sometimes longer, rests on the vein. This is also seen in coal-regions. Fig. 146 is a strong form for pressure from sides and top. Again, instead of long sticks in sets, short timbers may be utilized in framing arches. Sets may be laid close together, or the distance between them may be lagged.

Should the vein matter be too loose to stand up, the gangway is lagged, as is shown, against a long brace (Fig. 146). In wide, soft veins a similar idea employs the elementary form of strength, as in Fig. 147. Gob roads are timbered only at the roof, by the lagging of caps, resting on dry pack-walls. Occasional chocks will give greater consistency to the whole (Fig. 148). But as the subsidence of the roof cannot be prevented, the road cannot be kept open long at a time until the gob is pressed solid. Meanwhile the maintenance is a serious item. The plan of keeping a road open along the face of coal through waste is worse yet. The mode of timbering alluvial gold-mines, called "blocking," is said to be perfectly safe by the inspector of mines at Sandhurst. The prop and collar system is used while passing the chain pillar on either side of the gangway; beyond, a modification, in which each cap is, in turn,

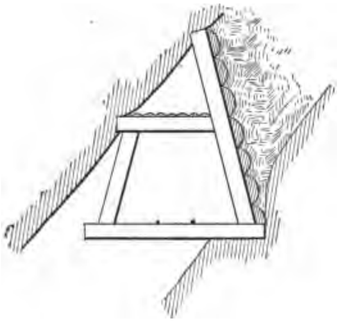


FIG. 146.

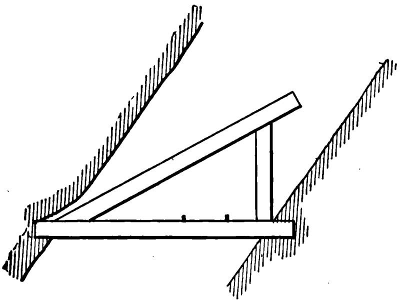


FIG. 147.

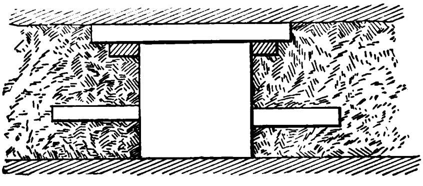


FIG. 148.

made to rest upon the caps of the preceding sets, already built, and upon the collar of the next. The timbers are 8″ diameter.

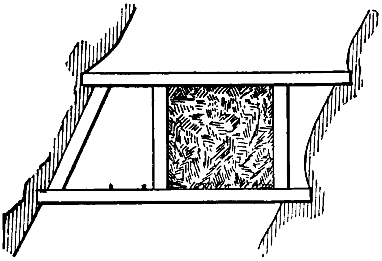


FIG. 149.

In very soft ground each cap has two posts for its support.

In timbering wide galleries a line of props form the compartment, like buntons in shaft-work. They relieve the caps of pressure, and are much smaller than the other members of the set, Figs. 125 and 126.

Large lodes are difficult to timber. The gangway takes only a small portion of the width of the vein ; the rest, if firm, is left to stand, or if loose, packed with waste. In the Great Devon Consols mine the compartments are used, one for "attle" (waste), and the other for travelling (Fig. 149). The vein runs 22′ wide the stulls were 20″ and 18″. In a 24′ vein a sole-piece of 24″ timber and 18″ struts, with a longitudinal piece at the apex, made a very strong frame (Fig. 147), which with 3″ plank lagging carried from 10 to 50 fathoms of attle. The hitches were cut 18″ in. In Southern France, with great pressure from the roof and for supporting heavy waste, Figs. 150 and 151 are much seen. In many cases an arch of vein matter 10 feet thick remains untouched from the stope below. With a

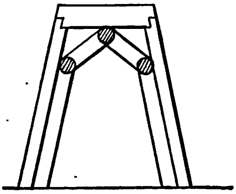


FIG. 150.

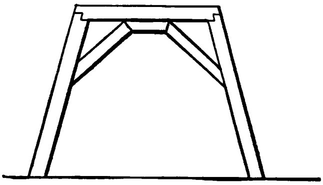


FIG. 151.

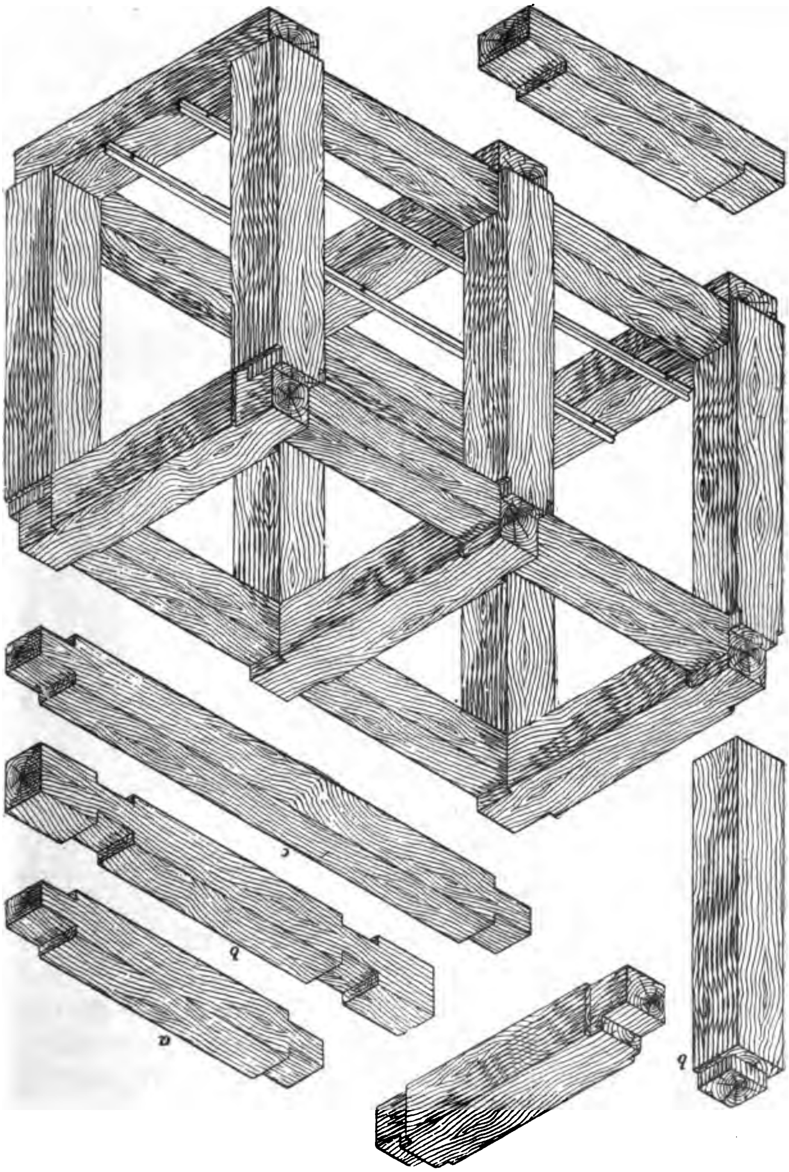


FIG. 152.

system of filling this arch is subsequently recovered. In fact, without filling, no large deep mine can be held by timbers. If, besides, the vein matter is soft, the character of the framing must be entirely altered. In the Austrian salt-mines the problem is very difficult, because the material assumes the nature of a fluid. Also, in rock that decomposes upon exposure to the air the timbering tends to crush, unless an elaborate form of framing is adopted. The gangways in lode or bed may then be a component part of the square-set system (Fig. 152). Where ground has a tendency to swell, the only way to save the timbers is to ease up the ground behind the timbers from time to time until the ground settles to its natural state. The swelling can neither be prevented nor resisted. The accompanying cut (Fig. 153) represents the style of Sutro-tunnel timbering, preferable to that illustrated in Fig. 180, which is insecure in shifting ground. Another admirable plan also employed in driving

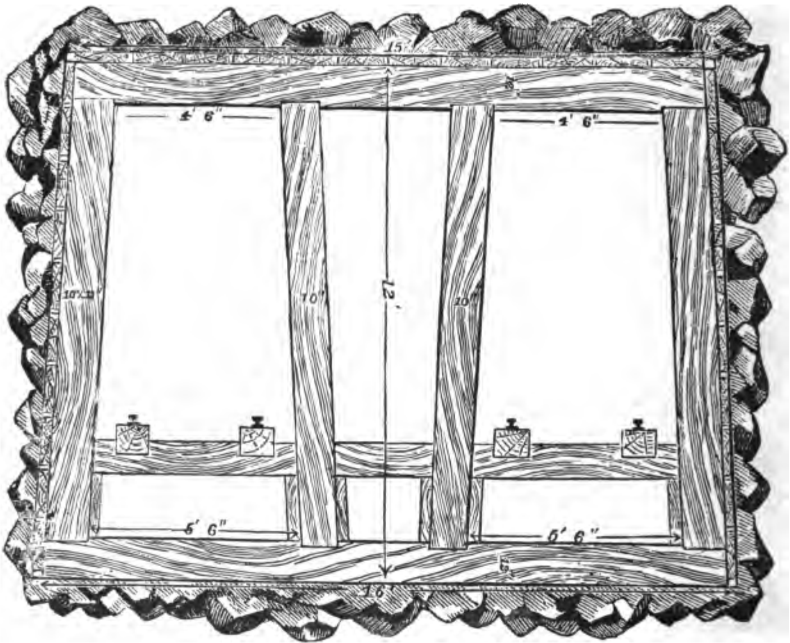


FIG. 153.

tunnels is to put a lateral tunnel, or two, very heavily timbered, with an open face to serve as a safety-valve.

Few cross-cut tunnels require timbering further than 50 feet or so from the mouth, for the ground stands well. In pitching rock and porphyry, the roof should be heavily braced.

67. If the spaces between the sets cannot be left open, then the sides and top are lagged with plank or "slabs" from the saw-mills. These are driven close—more frequently flat side to; but reversed they are better. Lagging should never be very strong; the slabs are always weaker than the members of the frames, and serve merely to prevent the fine rock from sifting into the gallery and leaving an open space that gives opportunity for movement, which if once begun can never be resisted. A very slight movement produces sufficient pressure to break the lagging, and thus relieve the costlier work. These are readily replaced on occasion. In many cases round poles are used, 3″ or 4″ in diameter, overlapping two sets. If the roof does not run, a few logs suffice (Fig. 142). In soft ground or under poor roof the men are protected, and advance made by "poling" ahead of the face (Fig. 181). The open space between the lagging and the rock is packed with waste or wedged perfectly. Brush piled back of the lagging holds up the "smalls" well.

When the capping over the firm ore of a flat-bed is not good, timbering may be saved by not stripping the entire height of the vein, but leaving a layer of it for roof to be removed later.

Of the relative merits of iron, wood, and masonry for underground work, much has been heard. Suffice it to say, that in Europe, where the utmost care is taken to preserve the timber, —by replanting for each tree cut down,—iron and masonry are put to extensive service. As to props of metal or of wood, the number would be the same—more for a brittle, less for a flexible roof; and whatever the condition of the roof, the size of the metal props would be nearly the same. Iron props are of the + or **O** cross-section, set on the thill or upon a foot-block, to be drawn by lever or bar and chain. Jack-screws have been used, but their expense is too great for their general use. Cast-iron props, auxiliary to pack-walls, 9′ apart, 5′ long, 4″ outside diameter, weighing 150 lbs., have been employed in collieries; their use is emphatically stated to be cheaper than wood.

Levels are not infrequently lined with iron tubbing similar to that adopted for shafts. An illustration (Fig. 154) is given.

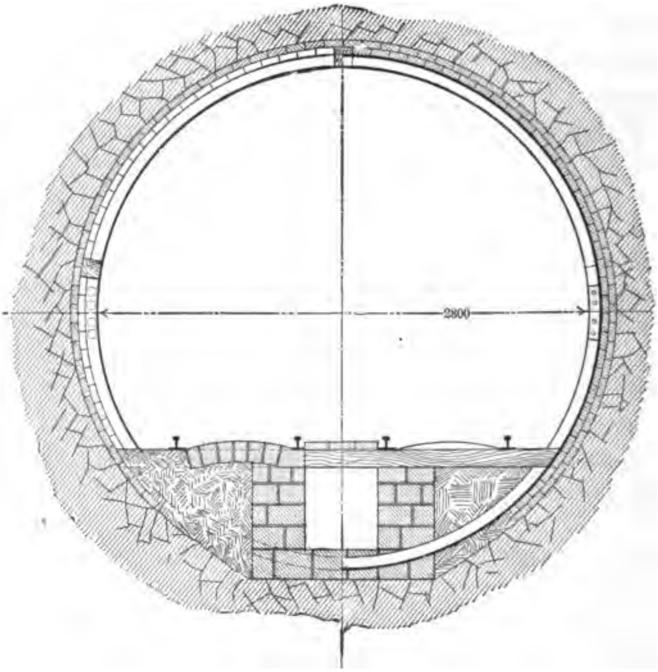


FIG. 154.

No system of timbering can be made sufficiently strong to prevent a general subsidence of the roof until the waste has been squeezed solid. Masonry for drifts and tunnels in this and the old country is very common. Where timber is rapidly destroyed, or where the pressure is too great for rectangular frames to be economically employed, the arch is pressed into service. On the other hand, good stone must be plenty and cheap. It is laid dry or cemented, with the walls straight or curved and the top arched. Some mines dispense with timber altogether, and in others the masonry leisurely replaces the temporary framing. In the latter case, whether the masonry is

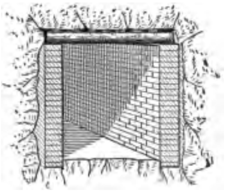


FIG. 155.

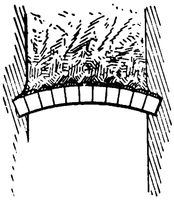


FIG. 156.

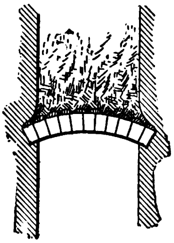


FIG. 157.

inside or outside, the timbers must be removed, and no spaces are to be left open, or to contain decomposable material.

The side walls are sunk into the floor 2′ or 4′ to a good bed, and with a batir if there is pressure from the sides. In New Almaden, Cal., mercury-mines 70,000 cu. ft. of masonry walling is built annually in the drifts. It costs four times as much

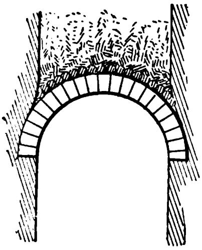


FIG. 158.

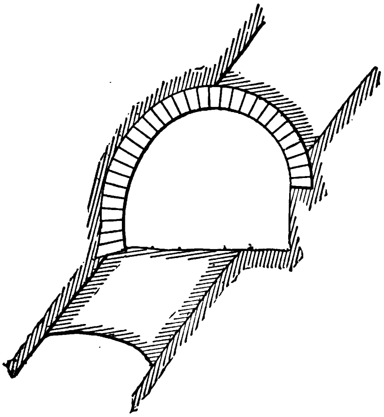


FIG. 159.

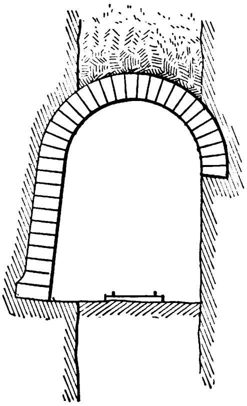


FIG. 160.

as timbering, but requires no repairs. The roof may be natural roof (Fig. 9), a timber cap (Fig. 155), or a pair of struts; but the arch is preferable for a permanent way. The arch may be segmental, or, for heavy pressures from above, full centre.

Arches are used similarly with stulls, seldom over two bricks thick, and the centre in the axis of the lode. (See Figs. 156, 157, and 158). If one wall requires special support, a parting arch is frequently put in. When properly laid, arches will withstand the pressure which future emergencies may require. As it is not always necessary to use complete sets of timber, so only portions of the arch need be used. Where the vein or hanging wall is too weak for a cushion, Figs. 159 and 160 represent the arrangement. Imposts are always made for the arch if firm seats cannot be had (Fig. 161). With a very bad hanging-wall some German mines use Fig. 162. The arches, of course, are built on centres, made easily of four pieces (Fig. 163).

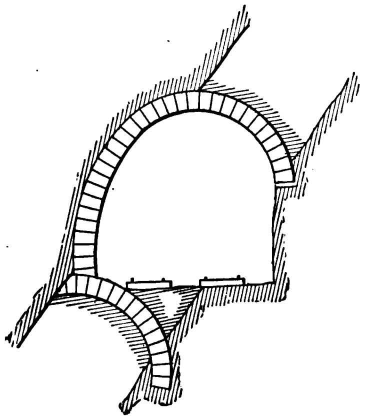


FIG. 161.

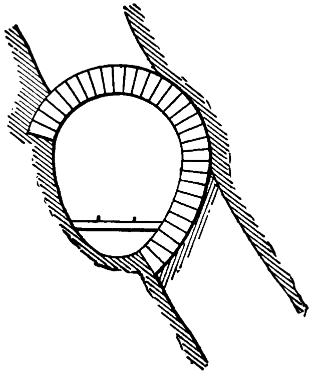


FIG. 162.

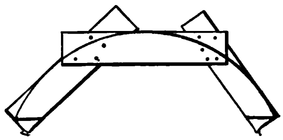


FIG. 163.

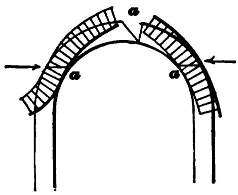


FIG. 164.

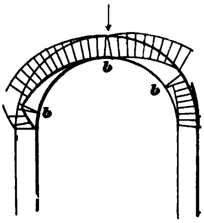


FIG. 165.

The two accompanying figures, 164 and 165, may be interesting as suggesting the places of requisite strength for given conditions of pressure. If the pressure is from the top and any opportunity for bulging is given, the collapse will take the form of 165. If the side pressure is very great and the roof resistance small, the break is shown in Fig. 164.

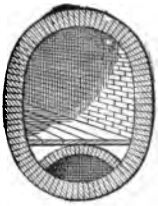


Fig. 166.

Finally, in German mines will be seen the various examples of tubular walling of the elliptical (Figs. 166 and 167) or inverted oval form, the choice between them for greatest strength being still an unsettled matter. The latter gives greater width at the bottom and smaller area for pressure at the top. Masonry cannot be built unless the ground is previously timbered, or firm enough to stand while the mortar is drying. In soft ground the level is driven by spilling, which can only be replaced by the masonry retreating toward the shaft. When the temporary service of the timber has been accomplished, and masonry is to be substituted, the uprights are cut at the foot, the sills and spilling laths (Fig. 182) removed, the bottom arch is made first, side walls next replace the posts, the caps being temporarily supported on props, the centre set up, and the top laid. Figs. 168 and 169 represent a masonry walling employed where the floor is sound.

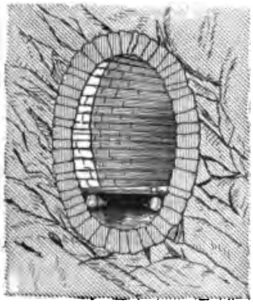


Fig. 167.

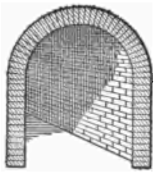


Fig. 168.


Fig. 169.

Dams for keeping back water, formerly straight-backed, (Fig. 170), are now arched, as illustrated (Fig. 171).

The use of iron is advocated and receiving ready acceptance in mining as well as tunnel work. The life of timber is short, its resistance low, and the component parts of the frame must be rigidly connected. The ordinary constructive forms of iron are applied in the ordinary way for columns, caps, or arches.

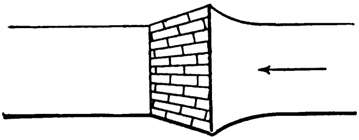


FIG. 170.

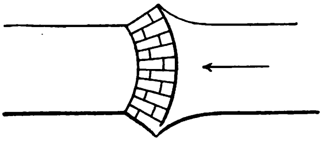


FIG. 171.

Their merits and methods are fully considered by R. Gottgetreu, in his " Baumaterielien."

68. In soft ground, which is liable to run, some form of stout framing is indispensable to prevent separation of the members. In the creviced matter of the Comstock mines, in the very poor ground of the Lake Superior region, in iron-mines where pillars cannot be trusted, in the rotten lead-ores of the Leadville beds, a most extensive yet simple system of framing has been introduced, which has found ready acceptance in various sections of the world. As a distinctive method of mining it was referred to in I, **12,** and though many properties are substituting a filling method, it still retains a hold on the mining public that qualifies it for a place here. Still, several causes hasten the decline of its popularity.

The plan is devised for soft ground, though it gives security in rock with a small tendency to cave, slide, or swell. The members are generally dressed square, and all to template for a faultless fit. For this work Hendey's or a similar machine (Fig. 172) is recommended. It is absolutely essential that the joints fit perfectly; then there is no swaying or buckling, and only an immense crushing can affect the frame. A slight play to each stick multiplies the opportunity for movement, and consequent destruction. The dimension of the members depends upon the quality of the ground. In the average soft material they are 6′ long, giving a working face of that size, which having advanced an equal distance, is built up to. All

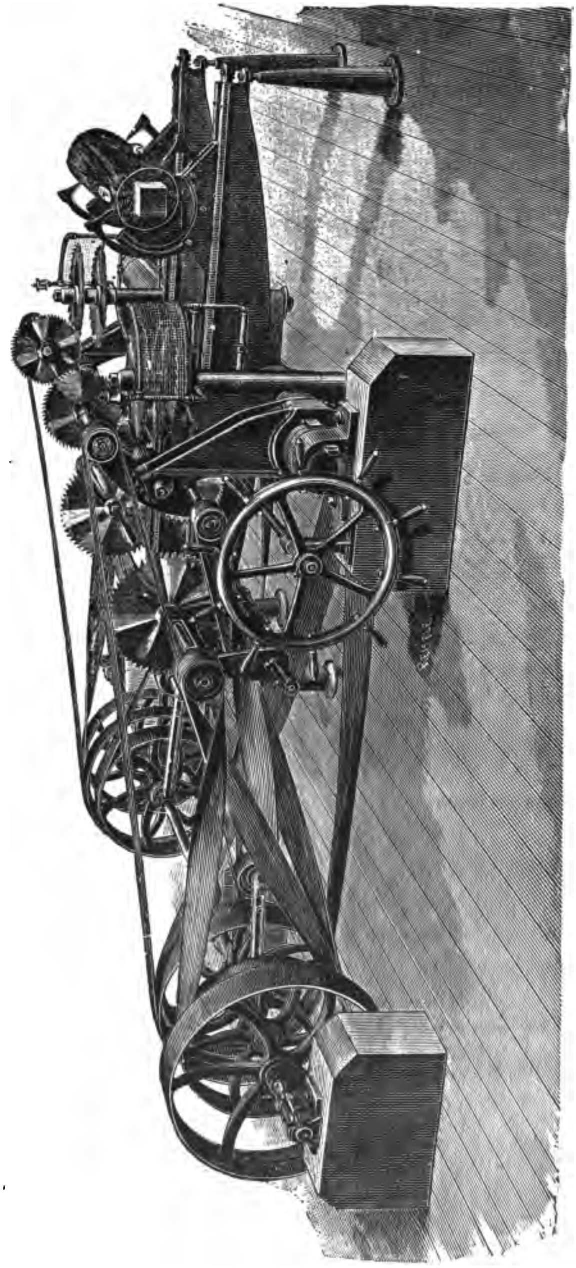


FIG. 172.

the sets are joined, and when properly cut the members act together to transmit the pressure throughout, and likewise to relieve one another.

Each set encloses a rectangular volume or cell (see Fig. 13), made up of sills, *s* (Fig. 173), cross-sills, posts, *p*, and caps. The posts are laid in the line of the greatest pressure, or else vertical, while the caps are at right angles to them. Flat mud-sills are laid on the floor, forming a square base, on the corners of which rest the posts carrying the caps. In the centre of each end of the posts square tenons $\frac{1}{9}$ the area of the top separate the caps and sills, each of which rests on $\frac{2}{9}$ the area, and are of a length of $\frac{1}{2}$ the thickness of the caps. Thus a post supports two cross-sills and two caps, and rests on four sills (Fig. 13).

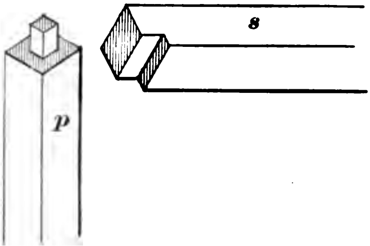


FIG. 173.

The frames are built up as fast as the work is opened, unless it happen that the ground will stand a while, until the timbermen can attend to the chamber. In ore that will not remain in place long enough to advance a set, a false set of cap and uprights, put in half-way, supports lagging overhead until the men reach the full length of a cut. If the ground will not allow of this advance, it runs, and only caving or filling is admissible.

The method of procedure varies in different regions, or perhaps with the nature of the ore. The lower floor is worked out first, the bents being added on at the right, left, and ahead; after which the next tier is set in the same manner directly over the first. Or, a species of overhand stoping is employed (Fig. 13), whereby the floor tier is progressed only a set or two before the next upper is advanced, to be followed later by another set above, and so on up. The order is a matter of indifference provided a perfect alignment is secured. Where this method is used in steep veins, very careful surveying is necessary to carry up the tiers of the lower level to a line with those in the upper stope. Rooms in the Cambria iron-mine, Lake Superior, are timbered 250 feet high. Sound, plumb, square sets are still to be seen in the East Vulcan mine and elsewhere in the Menominee range, that were put in in 1881. Sticks 10″ diameter or 14″ square are used for 6′ sets, and as large as 18″ square are not uncommon, with 9′ posts.

For platform, or under a scaling roof, temporary lagging is placed. Under the permanent roof or hanging-wall lagging is laid, and the space packed as close as possible with waste after the framing has been wedged tight.

Where the posts cannot be placed in the direction of the greatest pressure, diagonal braces are trimmed for, and driven into, the bents to take the inclined pressure.

Wedges are made of waste lumber in blocks 18″ long, ripped into pieces 4″ square. Then each piece is sawed diagonally from one edge to the other. Stulls cost in Leadville 6 cents per running foot; lagging, 1½ cents per lineal foot; wedges, about 1 cent each; and head-blocks, 10 cents. The cost of a square set of 14″ sticks is about $4.50 entire; lining and placing, $1.50 more.

In swelling ground it is customary to have some weak spots or frames that can be easily replaced. The swelling of felsite and trachyte will carry everything before it; roof is crushed, timbers are driven into the floor. Should movement commence, and any sill or range of sills squeeze or crack, the men should be called out at once, or the series attended to without delay. The engineer should "plumb" the framing occasionally in watching for any movement. The rupture of a single stick causes an increase of thrust upon the remainder, and ruin spreads rapidly. A chamber 90 feet high, with 13 tiers, was in ruins thirty minutes after the first warning.

Rooms and abandoned large stopes are often supported by a massive column of heavy timbers carried to the roof and filled with waste. These "cribs" are also built as an abutment in a chamber having the wall as a back. A good foundation is prepared, of the proper size and shape; two logs are laid parallel, and upon them, in the notches at the ends and at the middle, three cross-sills are laid; upon them again rest a pair of sills slightly inside and above the others; upon these, in turn, another layer, etc., etc. Inside of this space, as fast as building, waste is piled. The logs are 10′ and 14′ long, and 12″ to 20″ diameter at the butt. Cribbing may also be built with the removal of the ore, stoping overhand, and utilizing the waste to fill the crib (Fig. 174). In abutment cribs the cross-sills rest on the waste at one end, and are held in place by the weight above them. It is questionable if there is any choice between the square set and cribbing in large rooms. The former is well suited to turning off into small cells at the ends of the room, but it is dangerous if side-pressure exists. Cribbing will never do in very soft ore under brittle roof. This arrangement constitutes a very strong "made" dump, where the rock is not permitted to roll away freely,—as for instance, on account of contiguous buildings down the hill.

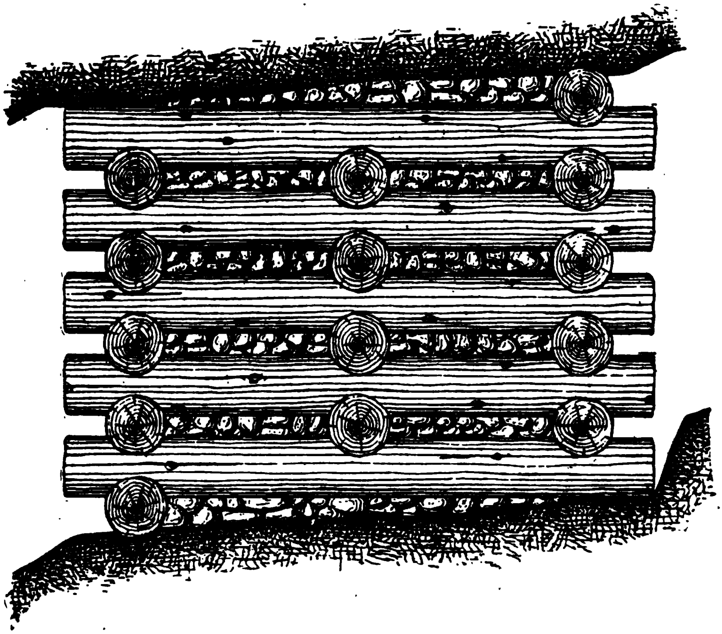


Fig. 174.

The securing of underground chambers intended for steam-pumps, hoisters, stable, etc., is not an easy matter. They may be built in any suitable shape that provides sufficient room. Being large, great skill is required to utilize framing or walling materials to the best advantage. Undoubtedly an arched roof will give the greatest resistance and strength, and masonry is therefore suggested. Besides, the hot, damp atmosphere of steam would rapidly destroy timber. Still, the latter is more convenient than masonry, in heavy sets, butted against each other, and spliced, with a key-block introduced. They are laid close together, lagged over, and packed to prevent wedging apart. The ventilation of these rooms should receive special attention. For air-compressors, engines, and coal-cutters an enlarged level will do, with some stouter caps on the wall-posts. Railroad rails and **I**-beams, log-lagged, are also used.

The styles for timbering plats and landings have already been suggested. The main difference at these points from that throughout the level is due to the intersection of two prisms or cylinders.

The styles of the timbering must vary with the character of the intersections. The frames must support one another as well as the country rock, and give firm fastenings for the plats and doors used for the landing of the cars and buckets. The level or gallery should be widened near the shaft, to give room for sidings, storage closet for powder, and steel. The sets of the shaft timbering opposite the level opening will be lacking one plate each, or two, if the level crosses the shaft. In this event, two very heavy stulls should be hitched at the bottom of the level, and two at the top of gallery, between which are heavy corner posts carrying the end-plates and the rest of the sets.

The masonry lining of a shaft is supported by an arch to, or by lintel on, two posts or walls at the sides of the gallery. The former gives a high opening for landing the buckets.

Mill-holes carried up with the waste are solid cribs of 30″ square for a man-hole, and may also have a compartment for sliding ore (Fig. 4). Either cordwood or sawed blocks are used for the lining. The latter plan gives them greater durability, for the abrasion is along their cut faces.

The timbering of slopes is similar to that of galleries, except that greater care is taken in cutting.

The timbermen's tools are few. Generally speaking, they should be only hatchet, hammer, and wedge, with a bar and chain, for casual work. The wood should arrive below ground ready for insertion. A saw-mill at the surface is now as much a component of the surface improvements as is the boiler. The Hendey machine is an excellent framer for round or square logs (Fig. 172). The saws being adjustable in every direction, it may be set to cut the required size. The ends are cut simultaneously. With three movements by the laborer and three positions of the log the tenons and shoulders are cut accurately, and at a cost of but 18 cents per set, as against $1.70 by hand-work.

CHAPTER IV.

DRIFTS, TUNNELS, AND ADITS.

69. Utility, dimensions, and location; mode of driving, progress, and cost. 70. Tunnelling through hard and soft ground; dimensions for various purposes; difficulties in soft rock; description, and comparison of the English, Belgian, German, and Austrian methods; the American method; examples of long tunnels; auxiliary shafts. 71. In treacherous ground; method of spilling by laths; by wedges; poling; Durieux's method; iron shield and pneumatic processes; masonry for permanent security; principles in the construction of arches and centres.

69. DRIFTS, levels, galleries, or gangways, according as they are driven for temporary or permanent ends, receive a corresponding amount of care. The manner of driving does not differ much from that of shafts, except that, as their area is less, and the water encountered can flow away freely, the difficulties are not so great as for a permanent hoistway which traverses a variety of strata. The locations of these drivages are determined by the considerations affecting the method of mining (see I, **6**).

Before locating a tunnel of any importance a careful study of the ground is requisite. All the data obtainable from geological reports, borings, etc., should be availed of. The character of the strata, their pitch, and the direction of the subterranean drainage should be known. They are of decided service, though not absolutely decisive. When the strata are recognized, the kind and amount of timber required may be ascertained.

Cross-cuts, connecting the shaft with the vein, are rarely timbered, for the country rock will generally stand during the period of their utility. Their size is the same as that of the drifts, or, for a busy level, wider, for siding and storage. If they require protection at all it must be by masonry or iron. Adits and levels are in the lode, and secured as shown in the previous chapter. Usually the stull suffices for the average length of time the drift is kept open. In underhand stoping only the trackway requires support, below which nothing but reachers, 5 feet apart, are needed for the work. Gangways and galleries have a larger area, and, being exposed to more treacherous conditions, possess better examples of the timbermen's art. The securing of them by masonry or iron is optional with the owners. Tunnels for railway or drainage purposes must be walled. In all cases the drifts should be driven to a normal profile at once without requiring subsequent trimming; the corners of intersecting levels should be rounded off, to ease haulage and aid ventilation.

Uncreviced hard rock may be penetrated without serious drawback. Granite, dolomite, and gneiss are hard drilling, but

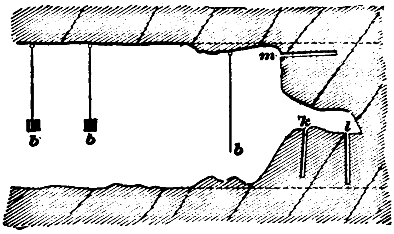


Fig. 175.

offer good roof, and are dry. Slates and shales are bad, and require arching or block timbering. Porphyry is treacherous, because, though hard when first opened into, it soon decrepitates on exposure. Some of the limestones and sandstones are porous and wet. Clay seams are bad primarily, besides being watercourses for the upper porous strata. The most favorable material is that which is sound and durable without being very hard, though hardness presents no special difficulty beyond an increased time and cost.

The alignment is maintained (Fig. 175) by three plumb-bobs hung from the roof. They pursue an almost dead level, though 2 feet per 100 is an accepted grade in metal mines. In collieries, the question of grade is involved with the proposed system of mining. The attack of the heading is over the entire face, unless the size of the way or the softness of the material may require its attack in sections.

In uncreviced rock, the order of the breaking is immaterial, so long as a good bench is had to shoot from and a good "leave" is obtained for the next shot. Even this is a matter of indifference, since the introduction of machine-drills and simultaneous firing, with a system of drilling, somewhat as explained in **90.** A drift of ordinary size will easily accommodate one drill, while two can advantageously work in a heading 10 feet wide, obtain more angling holes, and advance more rapidly. In railroad tunnels four are simultaneously drilling. Hand-work is much more depended upon for the driving of gangways, but machine-drills are becoming popular. Undoubtedly there is gain in time; the progress by machine is greater than by hand, even if the actual cost be the same per lineal foot—which it is not always (see **87**). Where the drills and air-compressors can be put to use after the mine is opened, it certainly would be advantageous to employ them in development.

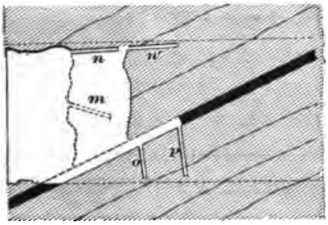


FIG. 176.

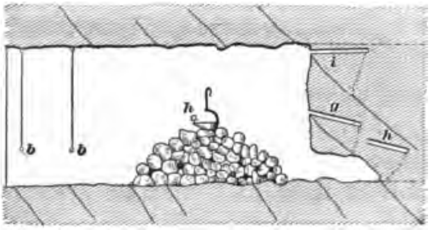


FIG. 177.

Drifting into pitching and stratified ground is simplified by the assistance afforded to shooting. If the seams pitch toward the face (Fig. 176), the upper holes are fired first; or, in high galleries, the central holes will make a good bench to shoot to from top to bottom. If the cleavage is not marked, or away from the face (Fig. 177), the bottom holes are fired before the upper ones. With electric firing, the sequence of shots is of little importance; besides, as the driving is usually done by contract, the mode of working varies with the individual. Three men form a gang in driving, and can manage the ordinary-sized drift. In the gangway of 7 × 7 two machines or two pairs of cutters have ample room. Their progress cannot generally be stated. In hard rock a foot a shift is fair, while soft can be penetrated at the rate of three or more feet a day. In shaly ground greater progress might be made did it not require good timbering. In soft ground the advance depends upon the skill of the timberman. The consumption of steel in medium ground is about 25 cents per cubic yard removed, though pink quartz will dull 150 bits to the hole, and the blacksmith will consume more steel than the rock. The consumption of powder is about $1 per cubic yard of medium-tough rock removed. This amount will vary with the area of the face, and whether the breaking is done by simultaneous or single shots.

70. Levels with a greater height than 8 feet must be broken in benches. Railroad tunnels are of this description, and in hard rock may be driven without any temporary timbering, the benches being attacked like stopes (Fig. 178), where the numbers express the sequence of openings. Frequently, in driving long tunnels, the drift No. 1 is pushed as fast as possible in order to make connection with a shaft or a similar drift approaching from the opposite direction. This communication is for ventilation and haulage purposes.

Numerous tunnels have been driven more than five miles for mining purposes. The Freiberg is 24 miles long; at Clausthal is one nearly 11; the Joseph II. is 9½; the Ernest August, 6½ miles; and the Sutro, 5. These have, moreover, lateral branches that enable them to subserve a great territory. The Gwennap adit, in Cornwall, is said, with its branches, to attain a length of 30 miles. The greatest depth below the surface is not over 900 feet for any one of these.

Progress is facilitated and ventilation obtained by the sinking of shafts at convenient points along the line to the tunnel level; and from them the excavations are begun contemporaneously with those at the mouths of the tunnel. Sometimes the shafts are to one side of the tunnel line, to keep them free and clear of the tunnel-work. The amount of time allowed for the completion of the tunnel determine the number of points of attack and the number of shafts to be sunk. The latter is also dependent upon the cost of sinking and the hoisting through them relative to that of the long haul in the tunnel. These relations can be mathematically expressed when the several aspects of the case are given. From Foster's Callon's "Lectures on Mining" the following formula is taken:

$$Q = xq + (PS + P'S')\frac{l^2}{4x};$$

wherein $q =$ the total known cost of shaft, $x =$ number of shafts to be opened over a length l, $S =$ the area excavated, $S' =$ the section of the lining, $D =$ the distance between the two adjoining shafts, P and $P' =$ the cost of haulage per lineal yard for each cubic yard of rubbish and of walling material.

$$D^2(PS + P'S') = 4q.$$

The St. Gothard Tunnel, 48,840 feet long, was run from the two ends only; so the Mont Cenis Tunnel, 39,840; the Hoosac and the Sutro tunnels with the assistance of two shafts. The Washington Tunnel, 20,715 feet, had four working shafts. The Rothschenberger mining adit had 18 points of attack along its 24 miles of length. Rziha estimates that the "additional cost of running headings from a shaft is from 5 to 10 per cent higher than running from portals." He also gives a unique table to show that the rate of progress per month is apt to be greater in long than in short tunnels. Of those less than 100 m. long, 29 feet advance was the average of 3; 13 between 400 and 600 m. showed average of 57; 8, up to 1000 m., 114 feet; while 6 between 3000 and 4000 m. had an average of 219 feet, and 4 over 4000 m. progressed 259 feet per month.

Provision for traffic is not a simple matter when it is recalled that, besides the actual mining of an area (say, for example, that of the St. Gothard Tunnel), 26 × 20, at a rate of 18 feet per day, and the placing of 1000 cubic feet of timber and 300 tons of masonry per day, 750 tons of broken rock, 15 tons of lumber, and 300 tons of masonry must be handled, loaded, transported, and unloaded at the same time.

As the hoisting through shafts is about half as fast as haulage through a heading, the use of auxiliary shafts is not so economical as it was before the present development of machine drilling and blasting methods.

There are other methods of attacking the tunnel face besides the one depicted in Fig. 178. The two side drifts at the bottom may precede the main excavation; in these are built the walls for the roof arch which ultimately lines the tunnel. While the small drift, 1, is being driven to the connection, the sides 2, 2 are following, and in firm ground they may precede 5, 5 at the bottom by 150 feet. At this point the tunnel is being lined with timber or masonry while gangs are also breaking ground on benches 3 and 4.

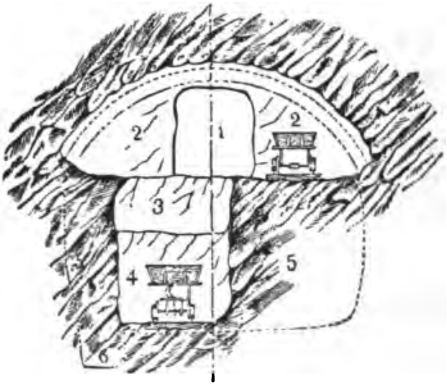


FIG. 178.

If the roof needs support, the plan of work contemplates one similar to that of Fig. 179, with the exception that the central core, 3, 4, remains as a support to the timbers arranged as shown below. Inside of this frame the masonry may be completed, after which the inverted arch, if required.

In soft ground, and especially in inclined strata, the order of driving the benches and the amount and character of timbering varies according to the dip. Circumstances and difficulties are so diversified that no uniform infallible rule has been established for the guidance of the engineer—probably because no system has a superiority over all others for any and all conditions. Safety is an element as important as time and money, and often is a determinant.

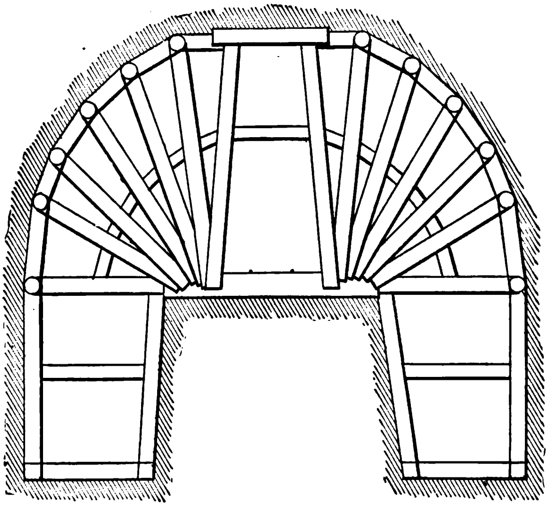


FIG. 179.

We have four systems of operating, known as the English, Belgium, German, and Austrian. A brief review of them is taken from Drinker's " Tunnelling."

The English, developed from the Thames Tunnel difficulties, consists in taking out the full area at once, after the preliminary top-heading has been made, and in supporting the roof by longitudinal top bars while removing the lower section. This gives a full clear area for putting in the masonry. After which, in material having any tendency to swell, the space behind the side walls is securely grouted. Though built quicker and applicable in 90 per cent of cases, it is unsatisfactory in very bad ground; in heavy ground it requires more timber than does the Austrian, its strongest competitor.

The Belgian method, introduced after the iron shield had been tried ineffectually in quicksand, builds the tunnel as an open cut down to the springing-line of the arch. The arch is then laid, recovered, and underpinned, the bottom removed in benches, working downwards, and the abutments built. Sometimes the centre-core is left, though that is only done in the French and German modifications. The entire area is not attacked at once, but divided into several benches, each being worked separately. The underpinning of the arch may be safe enough in hard ground, but it certainly affords a doubtful security in loose material.

The German system gave rise to what may be called the centre-core system. The work of excavation begins at the foot of each abutment of the arch, where a small heading (Fig. 179) with timber sets is first driven. In this the foundation is laid; above it a second heading, large enough to build another height of wall; above this another, in which the masonry is carried up. The top is then excavated across, and a connection effected between the two sides. In this the arch is completed without the use of centres, while the roof is being supported by stulls or props. The core is then removed.

In hard ground it gives cheap working, from the fact that the core has several faces of attack. In soft ground it is safer, because of the small exposure of roof and face; and the centre-core saves timber. Its ventilation is bad, and the cost of laying masonry is larger than where the masons have elbow-room; it is hard to securely timber, and several prominent engineers have decided against it. Certainly, in soft, treacherous ground, like shales and clays, its defects disqualify it.

The Austrian distributes the mining over the whole area in small sections. First a bottom heading is driven and afterwards connected to a top heading which is finally widened to full width for a bar-timbering which is carried down to the springing-line. Sides are excavated for the walls, after which foundations are prepared and the side walls built; finally, the invert arch. The cross-rafter timbering (Figs. 180 and 227) used for support admits of the transfer of pressure, and there is no such undue concentration, as in Fig. 179. This disposition of the timber affords a greater strength, and is the leading feature. Additional braces are sometimes added, and the sets connected every which way; but the design is to arrange the timber of each section in such manner as to form an integral part of the completed system. After the roof-timbers are in place, plenty of space is left below for masons, and good ventilation is had.

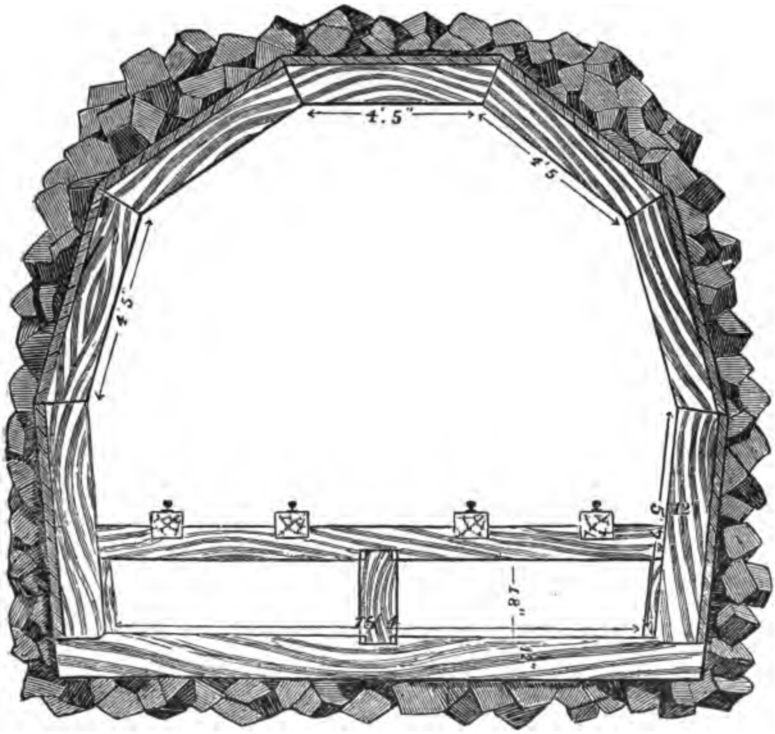


FIG. 180.

We cannot claim any system of tunnelling as our own, for neither the number of tunnels nor the difficulties encountered are as great as in the Old World. The Austrian method is the nearest approach to ours, or, rather, is the one which our engineers have adopted with a modified framing. The mode of driving and timbering is illustrated in Fig. 178. The upper heading, 1, with its enlargement, 2, 2, precedes the work upon the "bench." With a tunnel area of 21 × 27 the heading is about 8 feet high, and the bench, the remainder, attacked in one or two sections. Fig. 226 shows the order and direction of the drill-holes, which are in accordance with the "centre-cut" system (**90**). The upper heading is timbered, rafter style, three or nine circular bents form a block-arching, on heavy permanent pole standards, or on the side rock if the latter is sound. In loose rock the number of bents increase and the timbering is heavier. Inside of this frame arch the masonry is built. A segmental arch 21 feet high by 18 wide in the clear had nine voussoir-blocks 25 inches long and 10 × 10 inches section; two wall-plates, 6 feet 4 inches long, 12 inches square; two posts, 14 feet 1 inch × 10 × 12 inches; and 45 lagging, 6 feet 6 inches × 6 inches.

Fig. 180 illustrates the style of bar-timbering, which, however, will not permanently withstand much lateral or unequal pressure. A distortion is the result, as in this case. Note the comparison of the style with Fig. 153, also used in the Sutro Tunnel.

The Cascade Tunnel, just finished, had 5 segments of 12-inch timbers in the arch, covered with 4″ × 6″ pieces and cordwood lagging. Its timbering cost $104 per foot.

71. In running ground or decomposed rock the area attacked is limited by the rapidity with which permanent support can follow the excavation, during which provision must also be made against the pressure coming from all sides. In no region, not even in the Rocky Mountains, is the engineer free from the liability of striking ground that may overwhelm the miner before timbers can be inserted; so that the ground has to be restrained up to the face as well as behind the frames, with which close lagging may suffice to prevent movement. If it is not in new ground, the consistency of which will determine the details of its penetration, it may be where timbers have rotted or given way that a cave or a run may occur.

In alluvial or sand the method of spilling or a pneumatic system is indispensable. Varieties of the latter were employed under the waters of the Thames, 11,880 feet long; Severn, 22,992; and the Mersey, 23,760 feet, where the inrush of water amounted to 20,000 gallons per minute.

Spilling by laths is a method applicable to shafts or drifts equally. In front of one stick of a set, and behind its mate in the next advancing set, pointed heavy planks are driven, one set in advance of the face, close together on one or all of the sides from which the pressure is exerted. The fore end of the plank is forced down upon its set, while the rear end is held against the lower side of its cap, being protected from pressure by the previously driven upper lath. This is the method of forepoling illustrated in Fig. 181. The progress in soft, heavy timbered rock is three times as fast as in very solid rock.

FIG. 181.

The "spilling" protects the fore-breast by horizontal laths, as long as the breast is wide, held against it and braced to the nearest set. Each lath, *a* (Fig. 182), is removed in descending order to permit some ground to run off; it is advanced a short distance and braced again, *b*. The progress depends upon the speed with which the spaces may be opened and closed; as these are small, the movement is controllable, and there need be no fear of sudden shock to the timbers. The method is simple, and has been eminently successful under many circum-

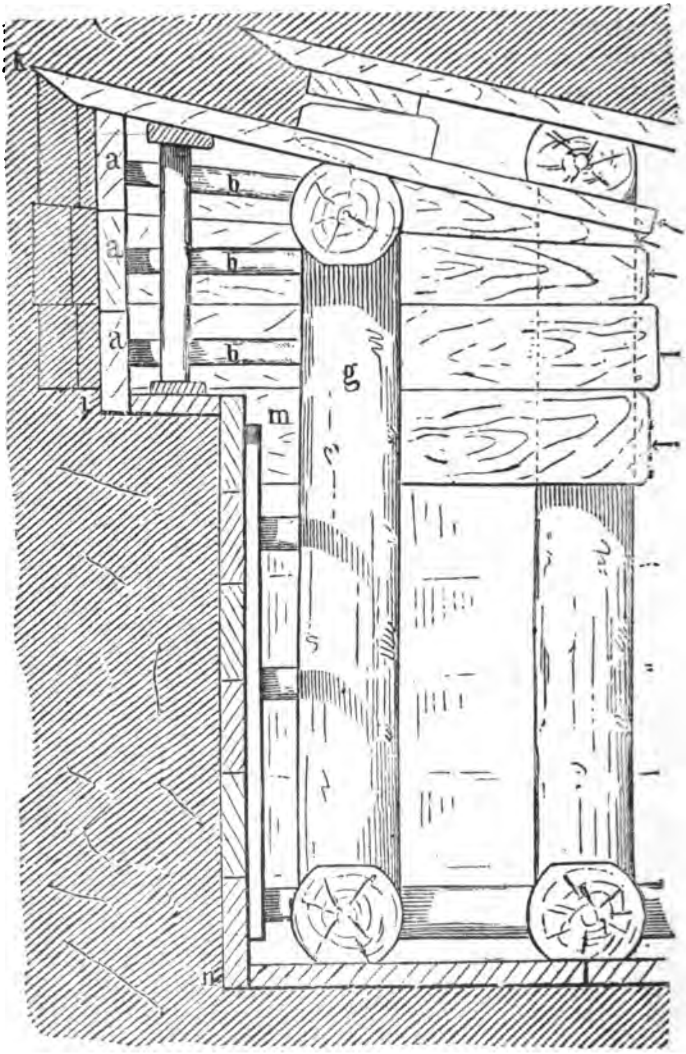


FIG. 182.

stances; once, with masonry lining following the forepole, a tunnel was executed within 18 feet of a river bed.

An entirely different principle is that employed by Durieux and others in Westphalia, whereby the ground was forced ahead of the work, and was not removed at all (Fig. 183). The walls and roof were forepoled, but the breast and floor were checkered by pyramidal pickets, completely covering the exposure. Those on the face were square faced, and larger than the floor-wedges, which were 12 inches long and 4 inches diameter, driven by mallets. The floor-pickets remain in permanently, while those at the face force the soft material ahead and advance in this manner. The battering they receive renders them useless in a very short while, when they are replaced. There is no material to be hauled except that used in the construction, and rather bad ground has been thus traversed. On occasion a short lateral drift is similarly pushed to relieve the main work. Four men in a drift 5 feet × 6 feet in morainal matter will advance 4 feet a shift.

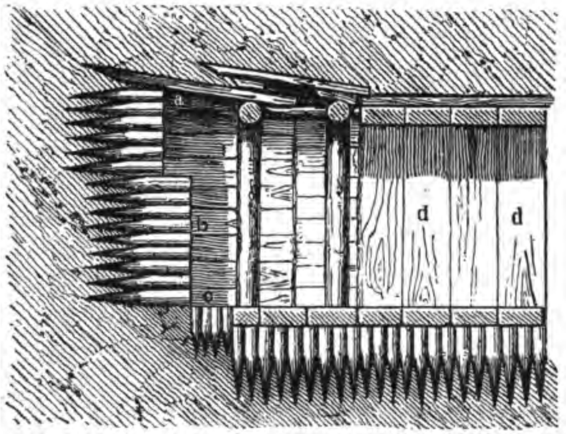


FIG. 183.

The finest piece of tunnelling is the construction of the new Croton Aqueduct, where water, mud, quicksand, and all varieties of loose soil were penetrated in an area of 676 square feet. Forepoling for the roof and sides, picket-spilling for the floor, and the American system with block-arching were adopted at various points. The ground in extended places was so bad that 24-inch timbers were crushed by the pressure.

Iron naturally suggests itself as a safer constructive material for loose soil; and in 1825 Brunel put it to use for excavating an area 38 × 22 under the Thames River. As in the spilling method, the face was covered by laths about 3 × 3 × 6 inches, and for rapidity of work was subdivided into three tiers of 11 breasts each (Fig. 184), each being protected by a cast-iron shield, from which struts hold the laths *d* in place. One man to a cell operates by taking away a lath and replacing it 3 inches in advance. The laths are removed successively downwards until 3 inches of progress has been made, after which the alternate frames are carried ahead 6 inches and the performance repeated. In each cell is a similar performance. The masons follow the miners very closely at *G*.

The second Thames Tunnel was completed by the use of a shield, which with pointed shoes was forced into a stiff clay at the rate of 9 feet per day by jack-screws exerting a pressure of 60 tons on it. Three men crawled through a door in its face, and excavated some earth preparatory to the next move.

The practice at the present time, by which long subaqueous tunnels are executed, is a variety of the pneumatic system. A pilot-tube or caisson penetrates the soil, which is held back by compressed air. The masonry, perforce, is built as fast as the shield, tube, or caisson advances. In the Greathead system a cylindrical iron shield, 21 feet in diameter, is thrust from the masonry by hydraulic pumps under a pressure of 3000 lbs. per square inch. The front end of the shield has a heavy ring-shoe, while the rear end encloses the masonry. The silt squashes through the doors opened in the face of the shield. Only about one half of the silt is trammed out; the remainder, mixed with one fifth water, is taken out by aspirator.

The Anderson system has a pilot-tube only 6′ diameter, with timbers inside resembling the spokes of a wheel, and pre-

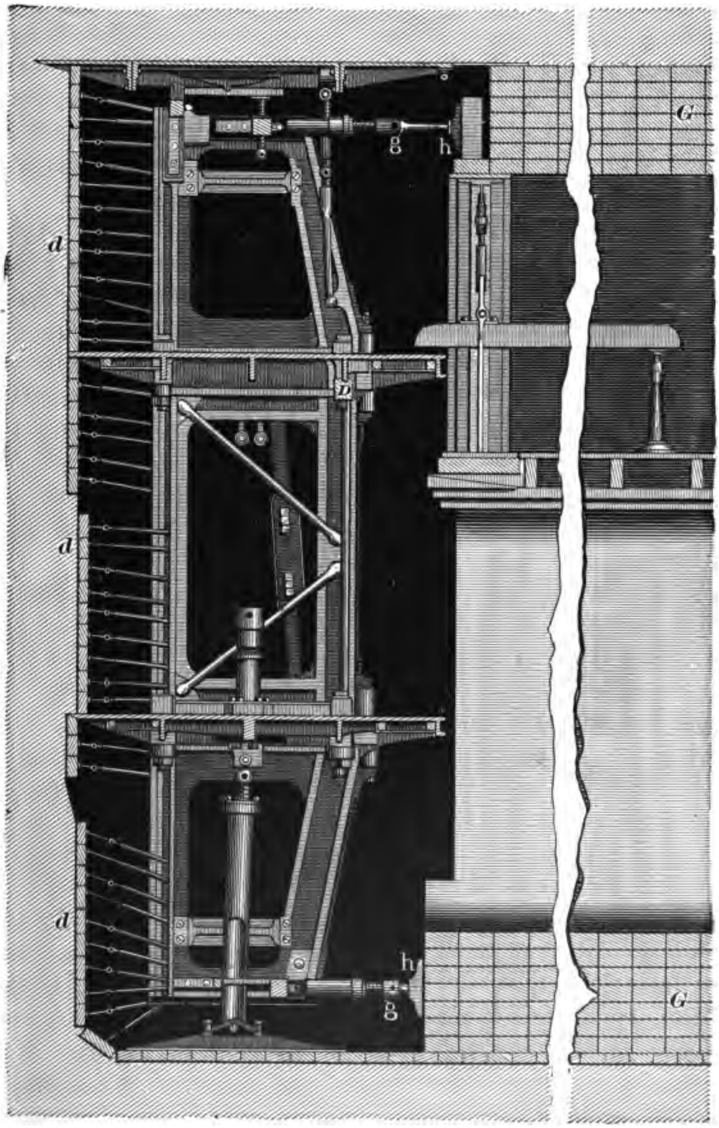


FIG. 184.

ceding the main work. It is of ¼″ plates, 12″ × 24″, riveted together by means of flanges; and when a cut has been excavated into the heading large enough, one of the plates is placed and held by props (often the plates are held by compressed air during the work), on each side other cuts are made for two more plates, which are riveted to it. Rings of the pilot are thus successively completed.

Around this, in small terraces, and considerably behind the pilot, the main shell, 17′ in diameter, is finishing in a similar manner, the plates being propped from the pilot-tube, which is always braced from the masonry that lines the shell. With its progress the rear rings of the pilot tube are removed and their plates shifted to the front end. The masonry consists of six courses of brick laid in cement. To reduce the volume of the tunnel that is kept under the compressed air, brick bulkheads, 4′ thick, provided with two air-locks, are built every 400 or 500 feet. Only the two nearest the work are maintained.

Whatever the procedure, the masonry is built on centres and by template, for invert and walls. The centres should be made of light, small, easily-framed sticks, that are not so close as to interfere with work, yet strong enough to support the thrust that may fall on them when the tunnel-timbering is removed. Its shape may be whatever is the most convenient for the traffic. The elliptical linear arch is, however, the form most commonly adopted, the side and roof comprising the upper part of the ellipse, which is closed below by a segmental invert arch, except at the two ends, under the walls, which are horizontal. In stratified rocks, the strongest form for the roof is that of a pointed arch. Sometimes in solid rock the horseshoe form is used for the top and sides, the floor being level.

In preparing to tunnel silt, both the weight and the vertical pressure of the overlying material and the lateral movement of the loose paste are to be resisted. The first is a matter of determination, and the ability of the completed structure to withstand this is also a matter of mathematical calculation; but the second is the difficulty to be apprehended. The

Hudson River Tunnel engineers, however, satisfied themselves that the tendency of the gravel to pour over the top of the tube would reduce the lateral stress to the resistance of the tube.

Before locating a tunnel the ground should be well studied from geological reports, borings, and such other information as may be procured. Maps are serviceable in showing the important features; and a systematic plotting of all data, geological and otherwise, gives a good basis for conclusions. In Drinker's "Tunnelling" will be found a discussion of the geological conditions affecting tunnel locations.

CHAPTER V.

BORING.

72. Punch-drills for artesian and oil wells; history of its advancement; accounts of deep bore-holes; Fabier, Kind, and Degousee tools; Mather and Platt system; description of an oil-well plant. 73. Spudding, cost, progress, accidents, etc.; tools, rods, torpedoes, tubbing, and their recovery; where used in preference to the diamond-drill; novel Colorado method.

72. BORE-HOLES for testing the nature of the strata below may be drilled by one of several methods. They may be driven to prospect for gas, to afford an outlet for water (p. 145), for pumping brine (p. 28), or to subserve many of the precautionary measures of mining, and they may subsequently be utilized for ventilation or the tail haulage-rope. Many holes of over 2000 feet have been bored for commercial or scientific purposes, the largest being that of Sperenberg, 4151′ of 14″ diameter. In these was observed the increase of temperature with depth (p. 189).

The evolution of the punch-drill, as it is called, may be traced through several stages, as the difficulties of weight, vibration, breaking of the tools, and keeping out water were one by one overcome. The primary idea of a cam operating an oscillating beam, suspending the tools, has not been much altered. In primitive days the tool was manipulated by a spring-pole and hand labor, or by a foot stirrup,—giving rise to the expression of "kicking down a hole,"—and served for moderate depths.

The drills were originally hung by rods; but their use gave rise to various troubles that were not easily remedied. Every 600 feet of 1″ rod weighs a ton; and the rapidly accumulating weight for deep holes became serious. The jars produced concussion that injured the material, and loosened the joints; and the repeated breakages of rigid rods proved fatal. If iron rods are used, they should be long, so as to reduce the time occupied in screwing and unscrewing; or else the derrick must be tall (Fig. 185). In cross-section the rods should be square. All other forms have failed to give satisfaction. It is the simplest and the most easily handled; 1″ square will do for ordinary depths.

Both the screw and the sword-blade joints are used. Care should be taken that the socket is on the lower end of each joint, or the sand will give trouble by jamming into it. The introduction of the hollow rods instead of solid ones gave great progress with the same power, and tapered wood rods, substituted for iron, were an advantage; but rope is the lightest connection between engine and drill with which our artesian and oil wells were dug. The Chinese were ages ago ahead of us in its use.

The Mather and Platt system of connecting the borer with the motor by a flat hempen rope is still used in Europe for deep holes. An ingenious device was added to rotate the tool, since this could not be accomplished by the twist of the rope. A movable collar, cut with inclined teeth at both ends, played two or three inches vertically in an iron bow, which had above the collar and below it sets of corresponding teeth, one half tooth apart. With each drop the collar engages the teeth of the lower set, and turns it and the chisel below one half a tooth; rising, the collar is turned one half a tooth by the upper set.

The evils consequent upon the blow were remedied by appliances similar to the "jars" now used (Fig. 186). Oennhausen's chisel had at the top a four-armed projection which played in corresponding slots of a cylinder, which terminated the rope. The rope was lowered and turned slightly; the ledges of the cylinder thus slipped under the cross-arms of the chisel, which was caught by raising the rope and its cylinder; a side-jerk freed the tool, whose cross-arms slid in the slots, and dropped it a distance equal to the length of the slots.

A pair of bent levers acting like pincers, grasping the tool and raising it to a proper height for a free fall, is the form of Kind's invention.

The tool to the development of which the oil and gas discoveries gave impetus is called the jar. The chisel, with its auger-stem (Fig. 186), is connected to a yoke which can slide inside of a large, flat chain-link. These fall freely to execute the blow; but on the rising of the rope and link the yoke is jarred, and the chisel loosened and raised for the next stroke. The chisel is a straight or an X bit of the best iron, and tipped with steel. An auger-borer is used in clay; a V cutting-edge in hard rock. The set of drill-tools is made as long as 50 or 70 feet, to more readily keep the hole vertical.

A temper-screw regulates the feed to the rope and tool as the hole deepens. It clamps the rope to the walking-beam of the engine, and allows of a play of 4 feet. The débris is cleaned out; and samples of the rocks traversed are secured by a sludger, which is a plain cylindrical iron tank with a stem-valve at the bottom. (See Figs. 59 and 111.) To save time in changing, it is usually operated by a separate windlass from that winding the tools. This is also called a sand-pump, from the fact that the tank must be run up and down several times in the hole to fill it with sludge. The débris sampled should be vialed and labelled or poured into a glass tube for reference.

73. The operations of drilling imitate the jumper. By the Mather and Platt system the rope with its pendent tools were raised by a single-acting piston operating on a pulley, which on the up stroke raised the tool, that was allowed to fall on the descent of the piston. This is the feature of the plan, which is distinctive; its progress compared favorably with the other methods applicable to 14″ holes. Only three men are employed. By removing the inside cutters a solid core could be obtained.

The spring-pole is operated by hand- or foot-power, though its rebound lifts the tool after each stroke. Holes over 3″ diameter or 300 feet deep cannot be drilled with this crude machinery. Usually, spliced or strapped rods connect the chisel with the pole. To start the hole with this or the portable derrick-machine, a shaft is sunk through the surface-soil to bed-rock, and a 10″ plank-box leader, held vertically, guides the tool.

The more complete outfit, with which 6″ holes may be carried a thousand feet or more, consists of a tall derrick (Fig. 185) having a sheave at its crown and a 10-horse-power engine winding the rope on a "bull-wheel" drum, and operating the tools by walking-beam through a pitman. The height of the derrick should be sufficient to suspend the full "string" of tools. The rope is of hawser-laid cable about 1½-inch diameter.

Under certain conditions, "spudding" must be resorted to until the rocking-beam connection is made. While the bull-wheel is revolving, the rope holding the tools is tightened a little by hand and wound up somewhat; it is then suddenly relaxed, and the tool falls. Another pull again winds the loose coils, which drop the tool when slackened; and so on for 50 or 60 feet of hole. After each blow of 2′ or so, the chisel is turned by a lever in the temper-screw. When the screw has been paid out, the tools are hoisted and the chisel sharpened, if necessary, or replaced. Meanwhile the screw is raised and the sludge pumped out, and another run is begun.

A uniform rotation of the drill and a constant watchfulness are the only means of avoiding flat or crooked holes, which are so prone to occur in conglomerate and inclined strata. Again, some of the shattered rock caves in and wedges the tool fast, from which only vigorous jarring dislodges it. If this does not do it, the accumulation around the tool may be broken or cut loose by a spear; else each piece is unscrewed, a spring-socket clamped on, and the tool raised. The experience of the drill-man and the "feel" of the jar are the only guides to the working of the drill.

If the sand-pump or reamer breaks loose it may be fished out. If grapnel does not free it, it must be chopped up or the well abandoned. A "winged" tool is often brought into play to straighten out a hole. The wings are mere projections that extend up and down some distance on the four sides of the chisel, and nearly fit the hole. A hollow cylindrical reamer is also resorted to on occasion for the same purpose. With a vigilant, competent drill-man these should not be needed.

The progress is 10 feet a shift in magnesian limestones, more in sandstones, and less in metamorphic regions, and as much as 70 feet a day with heavy tools in soft ground. Two men constitute a shift, and about 200 lbs. of coal are consumed. The rig complete cost at St. Louis $1500 to $1800. The high derrick pattern is preferable to the portable plant which has everything concentrated on a truck. Many a deep hole can be drilled by the machinery, and the uses for them are numerous. There are many circumstances under which no other system can be employed than that here described. In the cement flint-rock of the Missouri zinc-fields the diamond-drill, its main competitor, was an absolute failure.

The cut on p. 303 illustrates the tools of an outfit, in order from the left: sinker-bar, wrench-bar and wrench, jars, temper-screw and gauge, small bit, large bit and rope-socket, auger-stem, and floor-circle (Fig. 186).

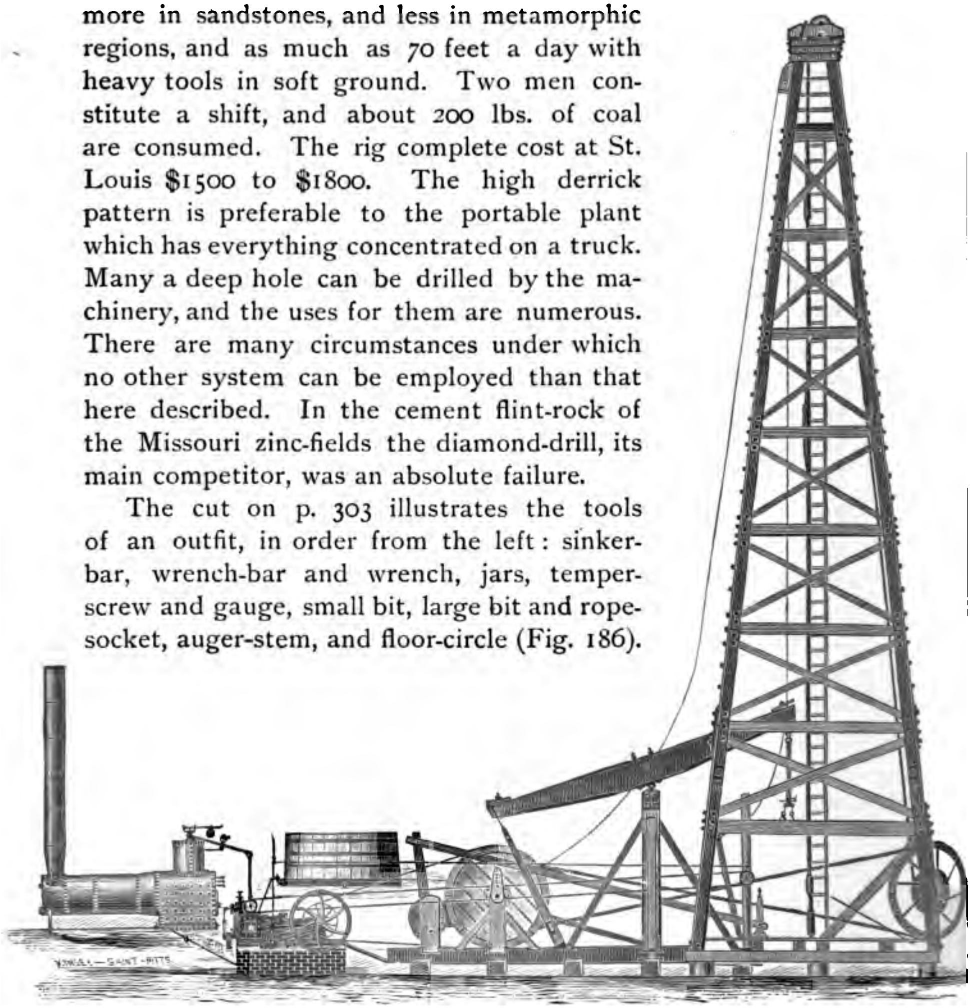


FIG. 185.

The average cost of an oil-well in Pennsylvania was $2175. The cost of drilling wells in 1888, was, in Allegheny Co., 60

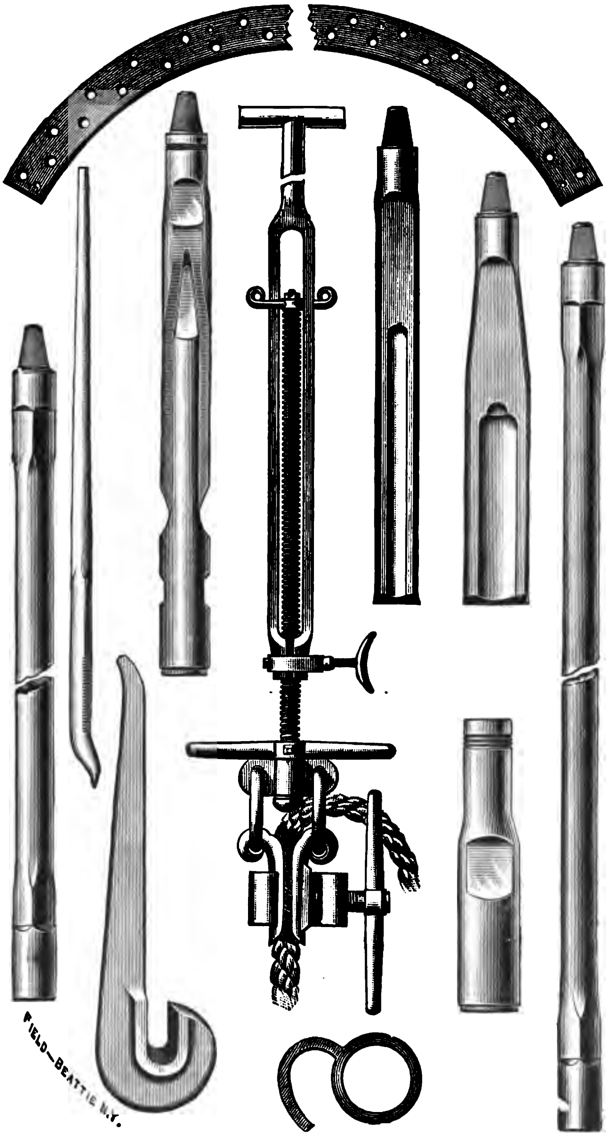


FIG. 186.

cents per foot; in Warren Co., 55 cents; and at Mt. Morris (very deep), $1.14. Reaming is done, when required to shut off salt water, at 40 cents; use of the machine, 10 cents per foot.

Loose or fragile matter must be held up and prevented from interfering with the progress of, or jamming up, the bore by a tubing which is forced down a few feet behind the drilling. A guide-tube at the surface maintains the verticality of the tubing, and a block of wood receives the blow. The steady pressure from jack-screws is also a common mode of driving. Wood is regarded as the best tubing material, but the objections to it are apparent, so cast or wrought-iron pipe is used. In certain chemical waters zinc and bronze are preferred. The pipes may be screwed together, joined by wrought-iron bands, welded and shrunk on the shoulders, or telescoped. The first length is sharpened, driven by percussion until its full length is in the guide-tube, when another length is screwed on and driven. The hole is usually smaller than the tube. When the depth has so far increased that the pipe cannot be rammed, the bore-hole is enlarged for the same pipe to be lowered, or the same drill is used with a smaller pipe. Casing remedies another difficulty which was experienced in the old plan of boring without tubing. The water would fill up the hole and exert such a pressure (14.7 lbs. per sq. in. for every 34 ft. in depth) as would prevent oil and gas from escaping, and even flood neighboring wells.

If the drilling has been for oil, pumping for a day or two may be necessary after the proper depth has been reached. Sometimes the pressure of the confined gas which accompanies the oil is so great as to blow out the tools, and a flow follows. On other occasions communication with an oil-chamber in the sands may be opened by exploding a torpedo at the bottom of the well. Except in dry holes, this heroic treatment should not be resorted to in shaly ground, as it usually results disastrously. Formerly the shell had 6 to 30 lbs.; now 100 to 300 lbs. are used to open a cave. The torpedo is fired by dropping a cylindrical weight or a drill on it.

When the object for which the hole was drilled or lined has been attained, the tubing may be recovered by the use of an ovoid screw-plug of oak, which is attached to the tool-rod. The plug is lowered to the bottom and a little sand is poured in to wedge it into the tube, which is now hoisted without trouble. To loosen the hold of the plug, it is only necessary to lower it and let the sand run off. A three-pronged expansible hook lowered down under the tubing may raise it. If the full column of tubing is not to be or cannot be removed, an expanding pliers is let down to cut the iron as it revolves. As a measure of courtesy to neighbors, abandoned wells should be plugged below the underground watercourse.

Often circumstances arise in which it may be desirable to isolate the several flows encountered at the different depths. For example, to shut off water from the lower petroleum, or a salt current from an artesian flow. This is readily obtained by the use of the seed-bag: outside of the tubes, at a point which is expected to dam off the flow, is a stout leather tube 6 feet long, the space between them being filled with flax-seed and both ends lashed. When the tube reaches its place the water soaks the seed, which swells and makes a tight joint, somewhat like that of the moss-box (Fig. 113).

A unique mode of drilling artesian wells is in vogue in the San Luis Valley, Colo., where 300 feet of 5-inch hole can be made in 24 hours. The soil of that valley is very porous, and for 900 feet down will hardly hold water except in the clay seams. Ditches have failed of their irrigating purposes, and each ranch is provided with one or more of these spouting wells. From a tripod a 5-inch tube is held vertically, and by a pair of blacksmith's tongs is turned by hand. At the foot of the pipe, outwards ½ inch or so, projects a bit, which, like a carpenter's borer on rotation, will work spirally, cutting a 6-inch hole into the gravel. As fast as tubing disappears in the hole lengths are added; and without exaggeration it may be said that in 24 hours 300 feet are run. A barrel or two of water is poured in to wash the detritus out through the ring, or a small pump is used until the ground-current is encountered a short distance down. Beyond 300 feet, horse- or engine-power will be required to turn the borer against the great friction of the pipe with the soil.

CHAPTER VI.

BREAKING GROUND.

74. Notes of cost and progress; fire-setting method; description of. 75. Description of miners' tools; the pick and varieties; underholing; shovels and spades; sledges; hammers; plug and feather; lewising; gads and moils. 76. Hand-borers; single and double hand-work; tools for the same; hammers, drills, and steel; jumpers; consumption of steel. 77. Blacksmith's work; kind of coal to be used; brief account of the materials employed in miners' tools; their selection and preparation for use; welding, hardening, and tempering, and how accomplished. 78. Varieties of bits and points for different rocks; sharpening and steeling picks, drills, etc.; making handles and helves.

74. THE character of the rocks which are to be excavated and the difficulties of their removal present such varied and delicate questions that it would seem useless to attempt a systematic account of the principles of and the means for breaking ground. Materials are "hard" or "soft," gauged by their resistance to abrasion as they affect drilling operations ; and they are tough or brittle according to their resistance to concussion in shooting. These qualities determine the cost of breaking ground. Live, dead, or pink quartz have different degrees of hardness and toughness that experience alone can gauge. They are all hard to drill through, but the latter is so tough that only heavy charges of strong rupturing agents will make any progress. Porphyry is too variable in its constituency to have the same treatment by drill and powder over an extended area. Stratified rocks which have been metamorphosed are not affected by the same mining agencies employed in those not disturbed. The effect of a subterranean current of water, the influence of the degree of dip, and the character of the cleavage are all elements pertinent to the problem of mining, and affect the progress in and the cost of excavation. Igneous rocks have fewer variables to affect the estimate than stratified rocks.

Estimates are easily had in any camp, but close observations of the results of a few experiments upon the given rock will supply the practical coefficients which, with the theoretical fundamenta to be discussed, should enable the engineer to determine the cost of working in the rock under examination. No criterion for general application can be offered herein; the estimate is largely speculative: yet the efficient superintendent should master this branch of practical work by which the miner may be rated.

Elaborate formulæ for estimating the cost of extraction are given in Drinker's "Tunnelling" and Foster's-Callon's "Lectures on Mining," and they are available as approximate guides; but there "cannot as yet be any rigid limits with reference to cost and progress."

A method prevailed among the ancients for conquering the hard rock which resisted their efforts of wedge and hand tools, and did great service for Hannibal in his campaign across the Alps. The miners of the Middle Ages, the Aztecs and the Japanese applied the "fire-setting" system, which is still employed on occasion as a cheap, effective method of softening the rock.

It was nothing more or less than exposing the face of the rock to fire and suddenly cooling it with water. The rock dilated, split, or was even changed in composition by this agency; and the unequal contraction on cooling further disintegrated the rock, which became amenable to the commoner tools. When we recall the instances where explosives availed little, where blacksmithing was the heavest item of mining, one can appreciate the advantage of such a cheap, simple substitute for working the tough pink quartz of our metalliferous districts.

As practised in mines, a portable grate, inclined toward the face, on four legs, had billets of wood piled upon it, the flame and heat from which were directed against the breast by a shield overhead. In some cases the grate was of the basket form, suspended near the roof. The wood was fired on a special day, say Saturday, and when burned out and cooled off the men attacked the calcined surface by ordinary means.

As the atmosphere is fearfully vitiated, full, free ventilation is necessary. The heat, the steam, the products of combustion of the wood, and the vapors generated by the roasted rock were such as to be intolerable, and prohibited mining during the incineration. With a powerful air-current they could be swept out rapidly, and this objection to the method overcome. But another remains to offset the economy of the work—the shattering not only of the face but of the wall rock, and especially of the roof, which would be rendered dangerous to an uncertain depth, and thus endanger lives and require prolonged re-forming of levels injured by the scaling of the material. Again, while there may be no objection to a disintegration of the rock from the action of the heat, a decomposition of the ore or mineral is another matter. It is said that in the mines of St. George the silver actually melted out of the veins, and roasted ores came from the vein for 100 feet back. So ores injured by fire cannot be subjected to this treatment. Otherwise, where fuel is cheap, ventilation good, and the rock capable of standing in large excavations this method, may be profitably applied.

75. In loose material and shattered rock shovels and spades are the only tools needed; clays and soft rock require picks, crowbars, and shovels; ground that is scaly, brittle, and seamy is managed by wedges in different ways; and massive rock cannot be broken without the aid of the hammer and drill. Where expedition is desired, these tools are replaced by machines, which duplicate their motion, with, however, greater motor power than the muscular effort of the laborer.

The ordinary shovels are made of iron plate rolled under a welding heat with an edge of steel, and ears drawn out for the handle. A concavity to the blade imparts stiffness and carrying capacity to the tool. The end is square or pointed according to uses, and set on a long or short handle. The long-handled pointed blade is the form in which they are most used, and the "diamond-labelled" ones are the most popular. The width of the blade in inches designates the size of the shovel. The life of a shovel is greater if it is used only to lift and convey stuff. Those which are used as a pry break at the helve instead of wearing out on the bowl.

From the very nature of mineral worked, the pick is essentially the collier's tool, the hammer and drill the metal-miner's. It is variously known as a pike, mandril, slitter, hack, and mattock. It comes in different shapes and weights. There is the straight or the curved, the anchor-pick or the poll-pick; each has favor for certain work. Indeed, in the same mine several forms are to be seen; perhaps it is a matter of individual prejudice. The straight pick assists the reach, the curved enables a fairer blow. So, for underhand work, for underholing, and for getting into corners, a straight-head pick is used. In downward work the curved will strike fairer than one without any "sweep." Its curve should be, properly, one of a radius equal to the combined lengths of the arm and handle.

The length of the iron is about 22″, the largest used, to the author's knowledge, being 29″ long, which in the hands of the "box" cutters are doing remarkable "jadding" (cutting the top). The weight of picks varies between 2 and 9 lbs., with 3½ or 4 lbs. as a common weight. The heavier weights are for downward cutting.

Picks may be made at the mine or purchased ready-made, of any weight, in all steel or in iron "eyes" with or without steel ends. They may be double pointed or single, with a hammer head, called a "poll," at the other end if they are to be used for driving gads or breaking rock. The taper or chisel-pointed ends of the pike slit the rock and the handle gives leverage to pry it off. So the three desirable points of a pick, aside from the material of its manufacture, are strong cutter tips, stout eye, and a tight handle (commonly called helve by Cornishmen).

The wearing parts of the pick are the tips, which should therefore be replaceable. On this account the all-steel pick is not advisable, for when its two points are blunted a few times they are worn so short and bluff as to be useless without having done much work. On the other hand the iron-pick eye, a 14″ length of best iron, gives long service by welding on tip ends whenever desired. Kendrick's make is in favor hereabouts. In some collieries the men have picks for removable points (5 to 7 oz.), of which they carry several to work with them. The picks are sharpened to form on an anvil, and commonly drawn to a four-sided pyramidal point for hard rock, a slim taper for fissured rock, a bluff taper to cut crisp ground, and to a chisel end for chipping the ground.

Poll-picks are forged out of 1¼″ iron, and have a stem 12″ from end to eye, an eye of about 2″ long, and a stump at the other end 3″ long, to form a poll for striking blows. The stem is pointed with steel, the poll faced, and its corners chamfered. The stem is long and slim for soft ground.

The eye is oval, and should be well surrounded with metal. It is formed by gashing the bar of iron in the middle, swelling it by working a drift into the clift, and hammering out stout cheeks. All the strain of the prying falls on the eye, which must be true and stout. Many of the drifting picks have the eye raised so as to give firmer hold for the handle.

The handle is of hickory or ash, and should be selected with care. Only straight-grained, firm, well-trimmed sticks are to be accepted. The handle is trimmed to the shape of the eye into which it is hammered, and then wedged tight by a pair of iron feathers. There should not be the least wincing, nor should the feather be so keen that its driving will split the eye. Wincing could be prevented by driving a T-piece into the handle at the eye, and bending its arms under the pick. The handle should be at right angles to the pick. This may be tested by drawing arcs with the tips, and using the handle end as a centre. The radii of both arcs should be equal. The length of American handles is about 28″, those of the English are 30″ to 35″.

This tool, properly built, dressed, and tempered, is a most effective excavator in soft and shattered rock. It is the sole tool of the coal-hewer.

The operation of coal-mining is called undercutting, underholing, bearing-in, or kirving, and consists in cutting a groove underneath a mass of coal in the soft floor or in the coal itself to a depth of, say, four feet, after which the block is sheared by passing grooves on each side as deep as may be necessary to assist in felling the mineral held only at the back and top. There is necessarily some waste in cutting the groove, which is often 9″ high at the face and 2″ at the back. In some mines, for example, those working long-wall, the holing extends some distance underneath a long face of coal, which is propped up by sprags (Fig. 8), and left for a day until the pressure of the superincumbent strata cracks the coal off or breaks it down. In other mines a hole or two must be drilled into the coal a few feet above the floor, and the mass blasted down. The presence of cleavage-joints materially assists the getting of the mineral; if they are close together, or if the seam parts readily at the roof, the underholing is not deep. Anthracite is rather too firm and brittle for kirving. It is blasted "off the solid."

In the operation of bearing-in the miner stands and cuts the groove a few inches deep; then he sits on the floor and carries the holing further, but ultimately lies on his side and picks the hole as deep as he can reach, meanwhile propping the mass of coal over him at every few feet. This is the most hazardous operation connected with mining. Generally the men work in pairs, and two men can underhole 25 feet in four or five hours.

The hammer and wedge has become a thing of the past for most mining work. The primitive wedges, used dry and wet, were of wood; now they are of iron, steeled iron, or solid steel. They had a hole in the side for convenience in carrying, and to insert a holder during use. Now, the wedge is used for mining out large, sound blocks of stones from rock which have a tendency to split in certain directions. The Egyptian obelisks were obtained in that way.

The plug-and-feather arrangement is much more efficient than the wedge which it has superseded, and consists of two iron wedges, with their outer surfaces arched, inverted in the hole, while their inner surfaces are flat. They consist of a number of lengths connected by hinges. They are put together at their flat surfaces, and are introduced into the hole so that they nearly fill its deepest part, while there is some play between them at the top. A flat plug is then driven between the two pieces, acting most strongly in the deeper parts of the hole. The driving of the wedge may be done by hand or by the drill employed for drilling the hole.

Coal, slate, and ornamental stones are extracted by this means. Powder would shatter them, and start incipient fractures that would facilitate their subsequent disintegration.

Of late an hydraulic wedge has been used with good success at the collieries near Saarbruecken, the position and the effect of the driving-wedge being reversed, the thicker end being in the bottom of the hole and the thinner end near its top. It is driven from below upward, between two half-round cheeks, by the hydraulic pressure. In an extension of the idea, black powder replaces the water, and its explosion forces the wedge up and snaps the rock in the direction of the thin edge. This process gets out dimension stones in some quarries, but the risk of huffing off the rock at the top of the hole is great.

Gads or moils are very useful accompaniments to a miner's equipment, differing from the early wedges only in that they have pointed tips, not chisel edges. Now they are made of a piece of steel that has done service for drilling and been dressed and smithed until it is less than 10″ long, and could no longer be used for starting a hole, and converted into a moil by tapering off the bit point to a pyramidal tip. It is used for chipping the rock, to give a smooth bearing to. timbers, etc. Though the process in hard rock is slow, the moil is practically indispensable. It will do service as well to remove scaly fragments of rock. A number of them, sharpened and annealed, should always be on hand for the use of timbermen and shaftmen. Many are lost in the mining waste and débris. The moil is used in the Lake Superior copper-mines for "blockholing" or splitting the large masses of copper.

76. The simple appliances described in the previous lecture constituted the only reliance of the early miners. With the improvement in iron and steel manufacture the methods have changed from that of slitting along the lines of cleavage to that of the actual penetration of the rock by several forms of machines. According to the character of the rock and the quality of the tool, the soft or the hard material is entered by a rotary borer or by a percussion-drill. These tools are operated by hand, or, with greater speed, by other motor power.

The variety of hand-borers presents a diversity of form, patterned after the jumper or the auger. In either case a crank is turned by hand, the motion being imparted to the borer by a more or less complex mechanism affecting the patent rather than the efficiency of the machine. Indeed, simplicity of parts is highly desirable as less wasteful of power.

Time is lost if the machine is awkward to place; besides, the work of turning a crank is not as agreeable or as efficacious as churning a drill. Moreover, the weak spot in the appliance is in the torsion of the bit, where the strain is maximum, and of a character more difficult to resist than in any other part of the machine. Finally, the same machine cannot indiscriminately be used in rock, shale, slate, hard or bony coal. Different grades of machines would therefore be required in a mine. In consequence, the machine is heavy, wasteful in time and power, and offers no kinetic advantage over the pick or jumper, in the use of which all the muscular effort expended upon it is paid out in the intended blow. Where they were delivered to the men, the results were unsatisfactory; for all refused to buy them, the inexperienced hand injured them, and the benefit from their introduction fell upon a few only.

Still, Howell's and McMurtrie's drills are extensively used in large properties. The McMurtrie is more quickly set than the Howell, while, on the other hand, only one hole can be drilled from setting. It is shifted after each hole.

The percussion on the rock is accomplished on one of two plans. The first is known as the "jumper," to distinguish it from the drill, which is the second and more common plan.

The jumper is a heavy 5′ or 6′ iron bar, swelling near the middle, and sharpened to an edge at each end. It churns a hole down into the rock, by being lifted at the bulge and allowed a free fall, after which it is turned slightly, raised, and dropped 1′ or so. When one bit is dulled, the jumper is reversed, and the other end, a mite smaller, continues the operation. In this way a hole is "jumped down." The débris is cleaned out at intervals by a scraper or a "spoon." Only holes which are vertical can be churned, friction reducing the momentum of the drill. The holes are apt to be triangular.

Coal and limestone is churned at a rate of 40 or 50 feet of hole in a shift; granite, 15′ or so.

The jumper, with one cutter and a head for hammering, has lost the significance of its name, and is properly a drill.

The cutting edge may be straight, concave or convex, and acute (slim) or "bluff." The shape is somewhat a matter of individual preference and skill. A slim bit cannot stand in hard ground, for which the obtuse convex edge is designed. The convex bit is stronger than the straight, and does not "stick" in the hole as does the concave or straight, but it is not so good a cutter.

The drill is a bar, which has one cutter edge and one hammer end. It is of round or of octagonal steel. Steel transmits the blow better than iron, and saves time and metal. The solid octagonal bars are almost universally adopted in preference to the round, for they are better turned with each blow.

The diameter of the steel depends upon how it is to be used. Where one man holds the bar and hammers it with the other hand, the size is ¾″ to 1″; for "double-hand" work, where one strikes while his companion holds and turns the drill, the diameter is from 1″ to 1½″; for two strikers and one holding, steel as large as 2″ is used. Generally speaking, it is better to have as wide a hole as convenient; but of course the single worker can only handle a small drill. In rare cases he is found working a ⅝″ drill, but that is rather light and springy.

Holes over 1¼″ are also exceptional, even with a triple gang. It has been found by experience that it is cheaper to drill a narrow hole and to increase the strength of the powder than to have a hole of large diameter with a larger quantity of weak powder. The work of drilling a 1¼″ hole is nearly three times that of putting a ¾″ hole of the same length; the relative volumes of the rock pulverized are as the square of the diameters; or, in other words, all else being equal, 25 lineal feet of small holes can be put while 9 feet of the 1¼″ hole are being drilled.

Such being the facts, it is very important for the engineer to determine whether the men shall work single, double, or triple. For many reasons, mainly social, double-hand work is almost universal; yet there are not only objections to this practice, but benefits favoring the single-hand work.

In double-hand work the men alternate the work of striking and holding, and to be able to work in any sort of a position, should be capable of keeping " either hand fore,"—that is, strike right or left handed.

In narrow places, working stringers and stopes, it is highly desirable to work the men single. The miner who is proud of his work prefers single-hand, and he can be relied upon for conscientious service. It is very tedious, however, for it does not give the relief of alternating, as does double-hand. Briefly, it may be said that if the single-hand work can be enforced under capable supervision, it should be introduced, except in very hard ground. Drinker says that in "point of economy of time and money one-hand drilling is from 30 per cent in soft schists to 20 per cent in soft sandstones cheaper than two-hand drilling." In hard rock " one-hand drilling gives the more rapid advance." Dr. A. Serlo in his " Leitfaden zur Bergbaukunde," believes that, " except in shaft-work, all the other forms of drilling may be executed more quickly by single than double hand," and " perhaps more cheaply."

A hole is drilled by chipping the rock from the concussion of the bit which receives the blow of the hammer. After each stroke, the drill is turned about ⅛th revolution. A little water is poured into the hole to preserve the temper of the tool and to mud the drillings, which are scraped out frequently by a spoon, or, if the hole is very wet, by a gun. A swab-stick is also much used. The hole is started wider than it is intended to be at the bottom, where it is about $\frac{1}{8}''$ wider than the cartridge. Upper holes are dry, and need no cleaning.

A spoon is a round $\frac{5}{8}''$ iron bar 40″ long, with a handle at one end and the other flattened out and curved slightly for 5″ or 6″ of its length, then bent to form a small cup that will scoop out the débris. A gun is a syringe, made of a 4′ length of gas-piping, with a suction piston and handle.

The depth of holes varies with the work, but for hand-drilling rarely exceeds 36″; single-hand work averages 25″. With nitroglycerine the holes may be one third deeper than those to be fired with powder. Hard, brittle rock requires long, narrow holes; while tough or fissured material is best broken by the short, wide holes. They are also shallow, and multiplied in jointy or "vuggy" ground. While deep holes are expedient, expert miners refrain from very deep ones, except on occasion, because they may leave the ground in bad shape; and it is often as much his endeavor to secure a good bench for subsequent shots as to break ground with the present one. Drilling in hard rock is preferred to that in variable rock, which does not give round holes or a uniform wear to the steel.

Holes are drilled wet or dry, but generally the uppers are only half as fast in putting as those pointing downward, which can be kept wet; for this reason the overhand holes are paid more for than those underhand. The relative direction of holes is of no significance. There is no reason for the prejudice favoring horizontal holes, except in shaly ground, where vertical holes cave badly. Sometimes a gas-pipe or drive-tube enclosing the drill will hold up the ground till primed or even fired. About 30″ of holes can be drilled single-hand, in medium rock, per shift. In quarrying limestone the holes are 7″ for plug-and-feather work; cost, 29 cents per foot. Single-hand miners in Swedish iron-ores make 5′ to 8′ per day; 12′ is very easy working. An 8 × 10 drift can be driven with four holes in the bottom, and three to blow down the top before squaring up. A shaft 10½′ diameter, in medium rock, is lifted with 8 holes 30″ deep and 3½ sticks of giant per hole.

The consumption of steel varies; in a porphyry heading 4½ × 7 an average of 25 lbs. per 100 lineal feet may be estimated. A double-track gangway consumes about 1.4 lbs. per foot.

Hammers are carefully selected for weight, varying from 3 to 10 lbs., according to use and preference. Single-hand men prefer the light weight, 4 or 5 lbs., and short handle; double-hand hammer-heads are provided with 20″ to 24″ helves. A good striking sledge is short and its weight concentrated to a large diameter. The expert prefers the round face, though the orthodox is flat. The hammer-heads used for wedge-driving are long and slender.

Different hammers should be provided for the several operations. Striking hammers should never be permitted for breaking rock. Their face is soon injured, and no miner would think of striking with a cobbing hammer.

77. The duties and work of the blacksmith may not seem relevant to the engineer, but as a matter of fact it is highly essential that he be capable of judging of the performance of the blacksmith, who comes as the intermediate between the engineer's complaints of some miner's laxity and that man's retaliation in pleading bad tools. On the other hand, a smith who can sharpen well for hard ground is held in high esteem by miners. No manager can afford to be ignorant of any element connected with the working of his diggings.

The shop should be supplied with a full kit of tools that would not cost over $20, good bellows and tuyeres, Peter's anvil, vise, taps and dies, twist-drills, round and square ½″ to 1¼″ bar-iron, strap and hoop iron, an assortment of carriage and machine bolts, screws, spikes, nails, a few horse-shoeing tools, benches, etc., in a space of about 14′ × 12′, with hinge-door openings near or over the fire, in the two walls, for working long bars.

One important element of success to the blacksmith is the fuel. This may be a slightly caking coal that gives flame and a high heat. Coke is hotter, but harder to keep fire in. The fuel should be as free of sulphur as possible; white-ash coal is better than red ash; the sulphur makes the iron hot short, and tends to produce scales; the coal should be clear of shale and slate, for they fuse, and make a pasty cinder that is annoying.

A few remarks regarding the materials used may not be inappropriate. It is a positive cruelty to furnish the men bad metal, or to compel them to work with an incompetent blacksmith's product, especially if they pay for the wasted steel or for the sharpening. Wrought-iron is the most variedly useful. It is so easily worked to any purpose. Its greatest strength lies in its resistance to tension, hence it is used for straps to tie frames together; and in whatever form, each square inch of cross section is capable of a five-ton strain, or each pound weight per foot of length of bar will resist 1.5 tons of tension.

A very useful property is its capability of welding, by which two short lengths of iron may be united, to form a useful bar. The process consists of wedge-tapering an end of each bar, heating them to red, and subsequently hammering the softened parts together. A more difficult joint, known as the split, is described further on in the steeling of picks. If the welding has been well done, the point of union is as strong as any other part of the bar. The main precaution in the process is to keep the surfaces clean and free from scales, which are so apt to form in a thin fire of the forge. Scales are due to the oxidation of the iron, which while red-hot is not sufficiently surrounded by ignited carbon to consume the free oxygen of the air. When the layer of fuel is thin, or where too much blast is given, the nascent iron takes the oxygen. Once formed, the scales cannot be melted or fused off, and would interfere with perfect welding contact. The remedy against the formation, then, is to keep the iron well covered. One plan—acting as a preventive rather than as a cure—is to sprinkle borax over the surface to be fused. This slags off the iron and keeps the surfaces bright. Sand does very much the same thing, only at a very much higher temperature. The presence of sulphur in the coal injures the welding by forming sulphide scales.

Steel is a compound of carbon with iron in varying proportions, and, though pages are written on "What is steel?" and "What steel is," all that one can say is that it occupies a chemical position between wrought and cast iron. H. M. Howe, in his "Metallurgy of Steel," says that steel, in its specific sense, is "a compound of iron possessing or capable of possessing decided hardness simultaneously with a valuable degree of toughness when hot or when cold, or both. It includes, primarily, compounds of iron combined with from, say, 0.3 to 2 per cent carbon, which can be rendered decidedly soft and tough or intensely hard by slow and rapid cooling, respectively; and, secondarily, compounds of iron with chromium, tungsten, manganese, titanium, and other elementary compounds, which, like carbon-steel, possess intense hardness with decided toughness." "This specific sense was formerly the sole one in all lands." "'Iron' and 'steel' are employed so ambiguously and inconsiderately, that it is to-day impossible to arrange all varieties under a simple classification." The various adjectives qualifying the term "spring," "shear," etc., apply to the uses to which the steel is put, and imply a certain percentage of carbon constituency.

The homogeneity of steel and the presence of carbon imparts to it a capability of hardening and tempering to a degree depending on the temperatures of the heating and the subsequent cooling. As the amount of carbon increases, the melting-point of the iron decreases; and this greater fusibility reduces its welding quality.

A steel is called "hardened" when it has been suddenly cooled, and thereby become as hard as possible. The reason for this change is not readily understood. Manifestly, it is owing to the presence of the carbon; for pure malleable iron is not in the least affected by the operation, while both steel and cast-iron are to a marked degree.

The operation consists in forging the steel to a certain temperature and then plunging it into some fluid which abstracts the heat from the tools. The quicker it is done and the greater the difference of temperature, the harder is the tool.

Either water or oil is used. Both volatilize or decompose at a temperature much below that of the immersed tools; so that the hardening takes place in a vapor formed on the principle of Leidenfrost's phenomenon of the spheroidal condition. It is supposed that perhaps decomposition takes place, whereby the hydrogen takes fire and the oxygen scales the iron. At any rate, oil contains less oxygen and hydrogen, than water, and has 77 per cent of carbon, which at the hardening temperature becomes charred. The specific heat of the oil is less than that of water, and its chilling effect is less rapid. So on the first plunge the metal is chilled and coated with soot, after which a slow process of cooling—almost an annealing—takes place. Again, instead of the iron being scaled by the oxygen of the water hardening, it is carburized by the carbon of the oil process. Finally, tests show that the tenacity of the steel is not affected in the oil as in water. Mercury and molten lead are also used for the immersion, but they are admissible only in large establishments.

Tempering is a process which follows hardening, whereby the steel is subjected to a subsequent lower heat, which softens it and removes its brittleness. To obtain the proper degree of tempering requires skill; and to attempt it without previously hardening is an exceedingly delicate performance, to be intrusted to an adept only. The risk is in overheating and scorching the metal, i.e., burning out its carbon. Iron is very easily decarburized.

When the hardened iron is slowly reheated, its surface gradually assumes phases of color, beginning with a light straw, passing through the shades of yellow, brown, purple, blue, and red. At a red heat—the original color before hardening—the effects of the chilling are practically removed.

Now, the operation of tempering consists in carrying the second heat to one of the above-mentioned colors, according to the amount of brittleness to be annealed. This depends upon the use to which the article is to be put. As, however, it is not possible to stop the forging at exactly the temperature desired, a second stage of the operation finishes the job. The aforementioned reheat goes on a little way beyond the desired color; the article is carefully plunged part way into the water or oil, till the disappearance of the steam or fog indicates that it is cold, when another portion of the distance is further immersed for a moment. The article is withdrawn, the scales rubbed off, and the heat of the remaining portion draws to the edge, until it has assumed the proper tempering color. It is then thoroughly cooled. The impression that the steel is cooler at a blue than at a yellow, in final drawing, is erroneous; for more of the heat is conducted from the red portion to the point than it radiates to the air, and the first heat to the edge only gives a yellow. With more it becomes purple; and so on. Hardened drill and pick points are treated in this way, 4″ of the end being heated to a yellow; and in thirds the tempering is proceeded with as above.

Caution is urged that the plunged tool, while tempering, be not held too long a time at a certain color-line, for it has a strong tendency to break there when in use. The tool should be slightly waved in the water.

Pieces which are to be tempered throughout must be allowed to "soak;" i.e., become uniformly hot before plunging.

The proper color for a given ground is only ascertained by experience. Generally speaking, the picks and drills are stopped at a straw if intended for hard rock, and carried up to a blue for mild ground. It is always desirable to preserve the toughness of the steel as far as possible; therefore select the lowest color compatible with the service to be performed. A high-carbon steel is given a lighter color than steel of low-carbon.

Metal-working tools are given a pale straw-yellow; wood-working tools, a brownish tint; hatchets, saws, etc., a light purple; picks, to a rose; cold chisels, to an orange-rose; key-drifts, orange; rock-drills, yellow-orange; screw-cutting dies, light yellow; and hammer-faces, a pale straw. A blue color would make the tools too soft for any of the above purposes.

78. A pick is made of a square iron bar 14″ × 1¼″, heated at the middle, and then struck endwise till about 1½″ across. This spot is softened in the fire at a red heat, cut open, and swelled by a drift, to form the eye. This—or the purchased pick-eye—is then slit at the ends and softened, while a 6″ length of pick-steel is being heated. When ready, the steel is tongued into the iron and hammered. A reheating with borax, and a hammering, complete the weld, after which the picks are sharpened and tempered. When the job has been properly done, no signs of the weld should be visible.

"Pick-steel" is a special steel that can be had in bars 1¼″ or 1½″ × ⅝″ or ¾″, and used only for tips.

Never harden a crowbar; for, its tenacity having been destroyed, it will "fly" on the application of some severe strain.

Steel bars for drills come in lengths of about 14′ each, and from ⅝″ to 2″ diameter. The cross-section is either round or octagonal. Two brands have now equal favor—the American "Black Diamond," and the English "Jessup," which has for a long time "had the call." Our American brand is equally good, is tempered a little brighter than the Jessup, and costs 3 cents less per foot.

The bars are cut up as desired,—more economically if cut into pieces as long as can be conveniently used; 30″ and 36″ are the best sizes. Never cut the "starters" or short drills, for they are obtained soon enough as the long ones wear out. To save delays and be armed against emergencies, a very liberal supply of drill steel should be provided and ready for use. The bits are wider than the tool, to save weight, and also to prevent it "sticking" in the hole. They are widened according to pattern, so they can "follow" well. The first drill

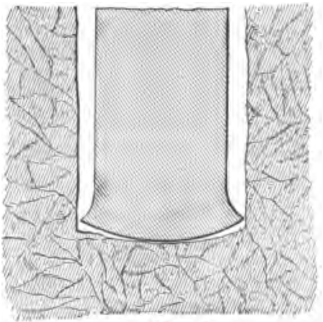


FIG. 187.

has the widest bit; the followers have narrower edges to the last one. In hard or jointy rock, where the stress increases the liability to fracture, the flare is small compared with that in soft rock, where a ¾″ drill is forged to 1½″ wide, and to 1⅛″ for conglomerate. The curved convex bit (Fig. 187) is best for ordinary hand-drilling. It is stronger, and for the increased work to be done at the circumference is more properly proportioned, than the straight.

The temper is a lighter color for hard than for soft rock, and for Jessup than for Black Diamond steel. Experience alone can tell of the proper heat. If the edges of the returned drills are cracked or broken off, the steel is too brittle, and should be softer, or other coal should be used. If the edges blunt much by wearing round, they are all right, though a harder temperature may give them longer life. Cast-steel borers are never heated above a cherry. They are annealed at the striking end.

Generally the men are supplied with steel and tools as called for; but for many reasons it is judicious to weigh or measure the steel given out at the beginning and returned at the end of a contract, lease, or month, and to charge for it accordingly. The latter arrangement is more advantageous for the mine, if all the tools are marked privately. No kind of supervision will prevent the carelessness which buries tools in the waste or breaks handles. Loaning out the tools and holding the men accountable for their return is the only possible check.

For forging and dressing machine-bits a special set of "dollys" and "swages" are used to give the X-shape to the bits. An outfit costs $20.

An elastic, tough wood is required for handles; and these qualities hickory and ash have. The former may be objected to on account of its weight, but it gives perfect satisfaction where used. An oval shape to the handle gives a more perfect guide to the blow than the round. They will last two years with a moderate care, but most of them meet an untimely end by breakage.

For a small mine employing 20 men in all, one blacksmith will suffice, though, of course, it depends upon what he must do. A good sharpener can dress tools for 8 or 10 gangs on medium rock or swage the I or X-bits for 7 machine-drills. Excepting the pointing of picks, the cutting of steel, and the handling of large pieces, he will need no striker. With this help he can make 12 heavy picks, 20 light ones, or weld 40 pick-stems in a shift; or he can finish 2 sets of colliers' tools of 5 coal-picks, 2 wedges, a hammer, and 2 bottom-picks. Alone, he can dress 40 bits an hour; with help, he can forge 25 double hand-bits, or draw out and temper 50 pick-points per hour.

Where the blacksmith does custom work for the miners, they pay $1 per month in the coal-mines and 5 cents per bit in metalliferous mines. Usually, however, the blacksmith does the sharpening, even for the contractors, at the expense of the mine.

CHAPTER VII.

BLASTING.

79. Principles in rupturing soft mineral or rock; substitutes for powder; lime, compressed air, and wedges; theory of explosion; tables of comparative force of explosion. 80. Gunpowder, its composition; "barrel" and "needle" methods of firing; use of, and care with, powder; tools, fuse, caps; lewising; consumption of powder. 81. High explosives; nitro-glycerine, its mode of manufacture; precautions. 82. Dynamite and its modifications; composition, etc.; relative explosive effects of the nitro-glycerine compounds; their storage and care; comparative safety; tools, fuse, and caps. 83. Simultaneous firing; electricity from battery and magneto machines; difference in the caps, fuses, and care; manufacture of fulminates; relative advantage as compared with single shots; cost of electric outfit; consumption of materials; precautions. 84. Principles; direction of holes; line of least resistance; formulæ for calculating the effects of shots; influence of seams, cleats, etc.; expanding bits.

79. THE principle employed in rupturing rock consists in subjecting the surface of a sub-facial cavity, regular or irregular, to a sudden increase of pressure acting radially outward. When the agent is sufficiently powerful to produce a high degree of compression upon the surrounding rock, it either fractures the material by the formation of a congeries of crevices, or it shatters it. The extent of the destruction depends upon the intensity of the pressure, and the cohesion or toughness of the material.

A sudden moderate explosion may be as effective a force as a slowly applied intense pressure. In this is constituted the difference of explosives. The slow, steady hydraulic pressure obtained from the ram; that of pistons actuated by compressed air; the expansion of freezing water; the swelling of slaking lime; and the spreading produced by wedges are illustrations of the second class of agents, all of which are, under appropriate conditions, advantageously employed. For driving preparatory workings in fiery mines, these are safer and less expensive than blasting. In foreign lands, the Coal Mine Acts prohibit the use of powder unless in conjunction with water cartridges. But they are no safeguard against the flame of explosives. Substitutes are sought that powder may be abolished.

Compressed air is used. An empty reservoir is inserted into the hole and connected with a powerful air-pump, that will exert a pressure of, say, 10,000 lbs. per sq. in., and breaks the coal. This is used in Wigan, is compact, handy, and, beyond the initial outlay, costs nil. Hydraulic pressure is used similarly.

Another scheme comprises a pair of iron cheeks between which a piston under hydraulic pressure forces a strong wedge.

Lime as a blasting agent is effective in coal. A three-foot hole is charged with unslaked lime and tamped. Water is pumped into it, and in 20 minutes the coal will break away.

The ignition of explosives is our second aid to mining which has had rapid development, as the previous agents fail to make sufficient headway in the tough massive rock. But the several means mentioned above are in vogue where the use of strong explosives may injure the material sought to be removed, where the gases from the combustion of explosives would injure the ventilation, or where the mineral is soft or seamy. Otherwise, where time is the important element sought to be gained, strong irruptive agents are employed.

An explosive, according to Andre, is a mixture "capable of being suddenly transformed into gases by the application of heat." In this sudden evolution of gas, in a space formerly occupied by a solid, a pressure is produced upon the confining surface in proportion to the volume of the evolved gas to that of the explosive. The expansive force of the gas is greater as the temperature of ignition increases. Finally, the rapidity with which the decomposition takes place is important as determining the value of the explosive. If the evolution is instantaneous, the maximum pressure is imparted at the moment of explosion; if the combustion is transmitted from grain to grain, its strength is dissipated over a longer period of time, and the pressure is less. Thus, the strength of an explosive is measured by its specific volume, the amount of gas it produces, the temperature and the rapidity of evolution.

Without attempting to follow the history of blasting, for which the student is referred to Rziha's "Lehrbuch der Gesammten Tunnelbaukunst," an enumeration of the several simple and compound substances used or suggested, at various times, to produce concussion, may be here given in chronological order: Common black powder, picric acid, gun-cotton, terchloride of nitrogen, nitro-glycerine, and ammonite. This list may seem brief, but a longer list would be merely an enumeration of the varieties obtained by the substitution of a single constituent, We have the Artillery, Sporting, and Blasting powders composed of charcoal, sulphur, and saltpetre in varying proportions; picric acid and picrates with saltpetre or chlorate of potash; gun-cotton, combined with other explosives; nitro-glycerine, with admixture of absorbents and dilutants. The result is that we have various grades of explosive compounds, from those which may be ignited by heating to a temperature of about 300° C., to the nitro-glycerine, which requires a shock. In other words, we have igniting or detonating compounds.

When an explosive is fired, the heat it develops is communicated from grain to grain with utmost rapidity and the particles decompose with the liberation of gases. According as these two phenomena follow each other slowly or quickly, we have rending or shattering powders. In the first, the gas is evolved so slowly as to give time for a concentration of pressure along a line or lines of least resistance. This is the quality desired for a sporting-powder,—ability to *project*. The slow combustion operates upon the small mass of the bullet, which can instantly take a very high velocity, and thereby give a rapidly increasing space for the evolved gases to escape. A tight bullet or plug would burst the breech or muzzle.

The miner desires to *break*, and this property is obtained from such agents as rapidly produce gases at high initial temperature and pressure. Very little plugging is needed, for the concussion produced by the gases of the quick powder is practically instantaneous, and a wave of pressure extends in all directions, which, being resisted by the rock, spends its force there and shatters it. The more sudden the action, the more local the effect.

Powders which produce gases that afterwards either dissociate or unite, or those whose combustion is incomplete, lose some of their initial pressure. This fact is indicated by the smoke which is given off. Some of the powders give a flame which not only wastes some of the explosive force, but also is dangerous, particularly in gaseous places of fiery mines. It is estimated that fully 68 per cent of the explosive force of black powder is lost in flame and smoke.

Accordingly, the miner has the choice between slow and quick powders for brittle ground or hard and creviced rock. For tunnel-headings and sinking he needs the strongest kind of powder, the weak ones being economical in enlargements and for stoping.

The relative volumes of gas, their temperatures and pressures in the table appended below, are based on the assumption of a perfect combustion. The ordinary conditions of mining reduce the effectiveness of black powder more than of the detonators.

	Volume of Gas.	Heat-units.	Relative Force—	
			By Single Explosion.	By Detonation.
Blasting-powder	2.38 cu. ft.	900,000	1.00	4.34
Chloride of nitrogen	5.09 "	570,000	1.08	3.61
Gun-cotton	11.01 "	1,050,000	3.06	6.46
Picric acid	10.72 "	1,240,000	3.15	6.00
Nitro-glycerine	9.76 "	2,375,000	4.55	10.00

80. The term gunpowder embraces mechanical mixtures of carbon, sulphur, and salpetre, varying from 8, 12, and 80 per cent, respectively, for sporting purposes, to 20, 16, and 64 per cent for open-air blasting, and 11.5, 17.5, and 71 for hard rock under ground.

History attributes its invention to Berthold Schwartz, a St. Augustine monk, during 1320; although it is said to have been employed in Rammelsberg 200 years before. Certainly, fire-arms, fire-balls, and fiery projectiles were spoken of earlier than the 14th century.

At the present time the above-mentioned ingredients are pulverized, compressed into cake, granulated, sieved, glazed, and dried, the size of the grains depending upon the use to which the powder is to be put. It may be exploded by a fulminating powder, as well as by impact with a red-hot substance, producing, in a given case, gases in the following percentage of total volume: CO_2, 43; N, 35; CO, 12; H, 6; carbohydrogens, 4,—all of which injure the ventilation.

When the hole has been put to the required depth, the powder is either poured in from a can or inserted as a cartridge, made outside. Fiery mines prohibit the use of loose powder. The powder should not fill over one third of the hole and all its irregularities, so "tamping" is necessary. In badly creviced rock or wet ground, a "clay-iron" or a "bulling-bar" often accomplishes what quick powder will not. It is an iron bar, with a ring handle or an eye at the end to permit of its withdrawal, that is used to pound clay into the crevices before loading.

Two methods of loading black powder are practised: one known as the "barrel" system, the other as the "needle." In the ordinary method the cartridge of powder is inserted in the bottom of the hole, with a needle projecting from it to daylight. Above the powder, around the needle and filling up the hole, a very soft clayey material, called tamping, is rammed gas-tight by a copper-tipped "tamping" bar. The bar may be of hard wood, but never should be of iron throughout. In many States and countries iron is forbidden by statute. The needle is replaced by a fuse, which, when ignited, fires down into the powder. The needle is a round copper bar, pointed at one end, with a handle at the other.

In the barrel method the powder cartridge is pierced by a wire which leads up through a half-inch copper tubing or "barrel" that extends the entire length of the hole. Around the barrel tamping is rammed, after which the wire is pulled out and replaced by a fuse. A comparison of the methods shows a preference in favor of the barrel, because less and a poorer quality of tamping may be used; it is twice as fast; the cartridge has less opportunity to soak water; and the cheap barrels are recovered after shooting.

Care should be taken that the tamping be free from quartz or other material that may produce sparks during the ramming. Tamping-bags of strong paper can now be had of any size, to order, at $3.20 to $6.00 per 1000. They may be filled with tamping of a poorer quality, are easily inserted into the hole and, by preventing contact between the tamping and the rock, obviate many risks of premature explosions from sparks.

Herewith is appended the notice issued by the Franklin (iron) Mining Co., Lake Superior:

1. That black powder and high explosives of any kind are not safe to use together in the same hole.

2. The cap is the only exploder that is safe to use in firing off high explosives, which does away entirely with tamping.

3. The only tamping necessary on high explosives is a small piece of miner's clay, which is easily put down on the charge with a wooden bar.

4. The practice of picking and boring out missed holes that have been charged with either black powder or high explosives is strictly prohibited.

5. Any miner on the mine known to use black powder and high explosives together, thereby necessitating tamping, will be discharged from the employ of this company.

The fuse, also called squib, is a thread of powder wrapped in tarred hemp, or in cotton, and waterproofed outside. The Connecticut make of fuse has the greatest favor in Colorado, and burns very uniformly. This is an important feature, for the miner may learn how short a length will give him safe escape. It burns about 20″ per minute. The fuse is supplied in rolls of 24 to 40 feet. If the tamping has been carelessly done, or contains sharp particles, the fuse may be cut and thus fail to ignite the powder. This not only delays the men, waiting ten minutes or so for the shot—if a number are fired, all the men are out, and the misfire cannot be located—but there is danger in removing the charge. Generally, it is advisable to leave the misfire alone and fire succeeding shots near it.

Though not essential, many mines have the end of the fuse in the cartridge fitted with a fulminating cap which, being a high explosive, fires the powder by detonation. This is a $\frac{5}{8}''$ copper cap, $\frac{1}{8}''$ diameter, having a small quantity of fulminate of mercury, which explodes with heat, or even by pricking, as the mutilated hands of many careless miners attest. The point of entry of the fuse into the cap is greased with a little "cartridge" soap. The California cap is popular. The XXXX is stronger than the XXX. The table in previous lecture shows emphatically the great gain of force by detonating the explosive instead of igniting it. The effect of powder is increased fourfold; that of nitro-glycerine twice.

In bituminous mines it is highly desirable and essential to break coal without adding deleterious substances to the ventilation, or elements of danger to the gases, and yet many of the primitive methods are still adhered to. Care in the handling and use of explosives is a matter of prime importance in coal mines, where the safety of the entire property of the employers and the life of the employees are mutually dependent upon one another. In metal mines the injury done by overloading or careless tamping is confined to the immediate vicinity of the victim.

For heavy galena veins, in serpentine and similar rock, and wherever the quicker shattering powders would pulverize the mineral too much, black powder is used. Granite is quarried by a procedure called "lewising." Several holes are drilled close together, and the partitions between them broken down with a flat steel bar, or broach-bit (Fig. 206). This extensive hole fixes the direction of the fracture, which is usually selected as parallel to the "rift," or cleavage. Three drill-holes make a "complex" lewis hole. The benefits of this lewising may also be secured by the Knox system, which is meeting with favor for dimension work. The hole having been drilled, a reamer cuts V-shaped grooves in its opposite sides, to determine the line of break. The tamping is not driven down on the powder, but an air space is left between them. This scheme permits expansion of the gases and gives time to effect rupture along the plane desired.

One foot of a 1″ hole can hold 5 oz. of powder, or 38″, a pound. In the anthracite mines a keg of powder (25 pounds) is consumed for every 40 tons of coal mined; the bituminous miner breaks 300 tons with a keg. In Illinois the tonnage varies from 51 to 110 per keg, with a very slight difference between hand or machine work. Long-wall mining consumes comparatively no powder, which, for pillar and stall work, averages 18 per cent of the gross cost of mining.

81. Manuel Eissler, in his "Modern High Explosives," calls nitro-glycerine the "ideal of portable force," being "the most powerful known to man." Sobrero, its discoverer, called it "pyroglycerine," which, however, was so extremely dangerous as to prevent its extensive adoption. It is an effective pharmaceutical preparation for congestion of the cerebrum. Mr. Nobel, seventeen years later, in 1864, discovered a means of making it safe to handle while retaining its explosive qualities. This gained for it universal adoption by imitation in every conceivable form, so that powder is fast falling into desuetude.

Nitro-glycerine, chemically known as tri-nitro-cellulose, is glonoin oil, $C_6H_5N_8O_{18}$, made by treating glycerine to nitric and sulphuric acids at a low temperature. The resulting liquid is less shattering than terchloride of nitrogen, and more explosive than powder. The temperature of its firing is 360°. It does not take fire when touched by a red-hot body, or, if it does, it burns quietly without smoke. If it is confined, it breaks the case easily. A thin layer will explode at the point it is struck. A large volume is almost certainly exploded by detonation of a neighboring body: and when explosion does take place, the combustion is instantaneous and complete.

The concussion is so rapid, that if it is laid on the face of a boulder in open air, the surrounding air cannot move aside as rapidly as the undulation of the shock, and the rock underneath is badly creviced. The air is almost as good a tamping-material as clay in powder work. The impression that nitro-glycerine works downward has, perhaps, gained ground from the fact aforesaid.

If the combustion is complete, N and CO_2 are the resulting gases, innocuous and inoffensive; if not, the lower forms of oxides are evolved, and they give rise to the objections. There should be no smoke, such as is noticed coming from the absorbents of the lower grades of nitro-glycerine. Schoen, in "der Tunnelbau," says "the inoffensive nature of gases of combustion has been demonstrated by experience in mining and tunnelling, even in cases where the means of ventilation are very inferior."

The liability to explosion, even by influence, is the obstacle to its more general use in the pure state. Transportation is refused it, so it must be made on the spot. For those who must use it, the following detailed account of its manufacture by Thomas Withers, of Denver, Colo., is invaluable:

"We made the nitro-glycerine in the cool stream along which our work lies—about ten three-gallon jars. The nitric and sulphuric acids are first mixed, the pure, colorless glycerine poured in, in a fine stream, the acids stirred, while they boil and send out thick red vapors of gas. If the glycerine is poured in too fast, or the water in which the jars set gets too warm, the mixture will blaze with a blowing noise, and if it is not stirred fast the blaze will shoot up some three or four feet. Keep stirring with a glass rod until action becomes less intense. Stir each jar in turn as glycerine is added, with a good current of cool water running around the jars, until the ten jars have each had some glycerine. Then commence over again at the first jar, and keep on until no more action takes place and no more fumes are given off. When all is quiet, and as much glycerine taken as the acids will convert, the nitro-glycerine will be in the bottom of the jars like a milky, heavy, oily-looking fluid.

Pour off the acids and wash with water, for nitro-glycerine is insoluble in water. An old-fashioned wooden churn with a dasher is good. After most of the acid is washed out and poured out of the jars, pour the nitro-glycerine and acids left into the churn ; dip in water, churn it up hard ; wash very clean so that litmus-paper will show no acid reaction. Keep putting in more water and churning and pouring off, for on the freedom from all acid depends the safety and keeping quality of your nitro-glycerine. When the churning is done, pour into a wooden bucket, and it is ready for use, looking milky, with a little clear water on top. After a day or two the milkiness will disappear, and the nitro-glycerine will look clear. If it grows yellowish, it shows free acid, and churn some more with cool water. If it gets orange-colored, put into the churn some lime or soda. If it begins to look deep orange and cloudy, explode it at once, or pour it out on the ground where it will not be dangerous."

82. Dynamite is a generic term that was coined by Nobel to include the mixture of nitro-glycerine with an inert or a chemical absorbent material which renders it perfectly harmless so long as it is in a state of absorption without any exudation. The compound is like moist brown sugar in color, and freezes at 46° F., when it hardens into a white mass. In this state it cannot be fired, and requires thawing out by exposure to a non-radiant heat cooler than the vaporization-point of nitro-glycerine. Immersion in a double kettle, around the jacket of which lukewarm water circulates, is the safest plan. If the kettle is kept away from direct fire or heat, no superheating can take place.

Dynamite is almost as sensitive as nitro-glycerine to sudden rise in temperature or pressure. Atlas, dualine, forcite, etc., are but names synonymous with dynamite, differing among themselves in the material used for absorbent. Infusorial earth, sawdust, wood-pulp, and magnesia are used. Besides these inert substances, explosive bases are added to increase the strength. Nitro-glycerine is, however, the active principle, and its percentage determines the power of the explosive, excluding consideration of the redundant chemicals. As the amount of absorbed nitro increases, so will the disruption of the rock.

So the infusorial earth, which absorbs three times its bulk of nitro, excels all other bases, and furnishes the strongest dynamite, called No. 1. The weaker grades are designated as Nos. 2, 3, etc.; No. 3c being about the lowest stock grade. No. 1 has an explosive strength of three fourths that of the pure article, and six times that of black powder, than which it is unquestionably safer to transport and store. Experiments upon the relative efficiencies of the various explosives under water have been made, and are recorded in General Abbot's "Submarine Mines;" but no formula can be prepared from the apparently conflicting results, because of the foreign substances added to the main explosive. Dynamite No. 1 showed, however, a greater intensity of action than does the "pure stuff." So do explosive gelatinc, dualine, and Hercules No. 1.

The following will suggest the constituents of some explosives:

Tonite is macerated gun-cotton 52.5, and baryta nitrate 47.5.
Gelatine is soluble gun-cotton 2.5, and nitro 97.5.
Dualine is nitro 50, sawdust 30, and nitre 20.
Rendrock is nitro 40, paraffine 7, nitre 40, and wood fibre 13.
Atlas A is nitro 75, fibreless wood 21, nitre 2, and magnesia 2.
Hercules No. 1 is nitro 75, chlorate of potash 1, nitre 2, sugar 2, magnesia 20.
Giant No. 2 is nitro 40, rosin 6, sulphur 6, absorbent 8, nitre 40.
Rackarock is nitro-benzol 22.3, chlorate potash 77.7.
Vulcanite is mealed gunpowder and nitro in different proportions.

Dynamite with ammonia nitrate is a strong combination, particularly if another material be added to combine with the oxygen excess of the salt and the nitro. Its hygroscopic property militates against its storage.

Ammonite, composed of 81.5 ammonia nitrate and 18.5 nitro-naphthaline, is claiming attention. The two innocents are made up into cartridges and may be exploded by fulminator. It is said to be unaffected by changes in temperature. Certainly there is no exudation to endanger it; percussion cannot explode it; a three-months freezing does not injure it; and its combustion gives neither flame nor fumes.

The storage of nitro and its compounds should be in large, dry, airy caves or earth-covered sheds provided with ventilating flues and open gratings, but no solid doors. It is dangerous to permit dynamite to become wet, for water will replace the nitro by capillarity. Some of its forms are very sensitive to a sudden rise of temperature, and all to a slow heating which decomposes them. When it shows an acid reaction, or the least signs of deterioration, it is liable to spontaneous explosion, especially if strongly confined. But if it has been properly made, combustion cannot ensue, at least not for several years. The French Government reports only four accidents, with no deaths, during eleven years, per 100 tons of stored giant as against 7.5 with an equal storage of powder.

Dynamite should never be stored with caps; nor should cartridges be laid away with the caps attached.

Not only are the nitro compounds less dangerous than powder to handle, but the work of the miner is lightened. The tamping is less, and, indeed, No. 1 requires none whatever; the mineral is broken finer, requiring less dressing at the surface; smaller holes can be drilled; the consumption of steel and supplies is less; and the mining time is materially shortened. Another important advantage is, that water does not affect it, hence it is incomparable for wet ground. As the harder rocks are encountered, the advantages become more manifest. If a rock is so hard that small holes and dynamite cannot make much headway in it, the only hope is to "fire-set" it first.

Holes are loaded with nitro by pouring with a tin cup over and upon the fuse and cap, and covering with water. In creviced rock a bottom is made of sand or dry earth before the charge is made. In a narrow crack the explosive is allowed to soak in and be fired by the influence of others.

The loading with dynamite is identical with black powder. The giant, coming in cartridges to fit the hole, is placed, as many as desired, in the hole, and with a safety fuse or electric wire and cap slightly tamped.

Miners sometimes use black powder with giant; "it starts the hole;" the fact that the entire ground to the very bottom of the hole is broken,—i.e., no "collar" or "bull-ring" is left,—is sufficient evidence of its advantage. This is practised everywhere in the proportion of $\frac{1}{2}$ lb. of black to three sticks of giant.

83. The fullest benefit of these high explosives can only be obtained by the use of strong detonators, and by arranging a number of shots in such positions as to be fired simultaneously and mutually assist one another. As has been noted, the explosive effect of any powder progresses radially in every direction, but the break is toward the free face, and it follows that power is lost at the back or in the solid ground. The amount lost increases with the quickness of the explosive. If, however, several contiguous holes be so placed that their backward rupturing tendencies may superpose they will shatter the rock which would, by single shooting, have been only fractured.

The zone of explosive effect may be divided into three sections: a sphere of pulverization immediately surrounding the bottom of the hole; a cone of rupture with its apex near the centre of gravity of the cartridge, and its base of an area dependent upon the relative explosive and resisting powers; and lines of fracture that extend into the rock.

The effect of synchronous firing is about 1.4 times as great as that obtained for the same amount of powder fired in consecutive shots. Benjamin Frost says in his report on the Hoosac Tunnel that "greater depths of holes are admissible," and greater advance attained.

The sole means of obtaining perfect simultaneity is by electricity; fuses will not burn with sufficient regularity to rely upon them. The holes are charged, the cap is inserted, two wires take the place of fuse, and the tamping is done. The wires are separated, each one being connected with its neighbor in such manner (Fig. 188) that a continuous wire circuit *E*, extends throughout the series of holes. *A* is the fuse, *B* the cartridge, and *C* the cap.

The men drill the holes till nearly "tally" time, the loading and connecting of them being effected by the foreman, who is the last man to retire. When he has assurance that all hands are out, he makes the final connections, raises the handle, *F*, and with one movement fires all the shots.

It may be well to state here that electric firing is equally applicable to black powder, indeed, a greater gain in force is had from its detonation than from the impact upon the higher grades (see table, p. 328).

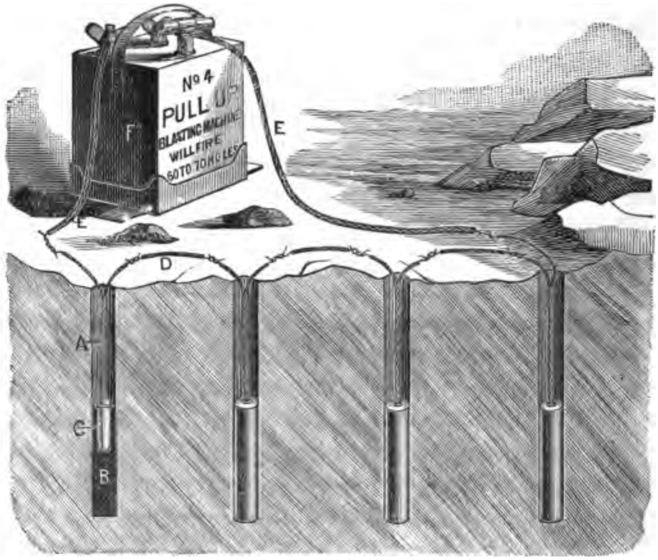


FIG. 188.

In addition to the increased efficiency, other important advantages are secured. The intervals of long waits of men who stop to fire off a couple of shots at any time during mid-shift, and drive all of their neighboring comrades to shelter, are obviated by firing just before quitting-time. The smoke can then clear off between the shifts instead of vitiating the atmosphere while the men are at work. It fires with certainty and eliminates the numerous causes to which premature and slow explosions are due, and it enables firing at a definite, safe distance. On the whole it should be encouraged wherever practicable.

There are two classes of machines used: a dynamo or magneto-electric machine (Figs. 189 and 188), producing medium tension electricity, and employing platinum fuses (Fig. 190); or a frictional machine which gives very high tension electricity to ignite gold-leaf fuses. The magneto machine is safer, cheaper, and more reliable. It can be had of a size to fire from 10 to 70 holes, irrespective of their distances apart—within the moderate working distances that a mine affords.

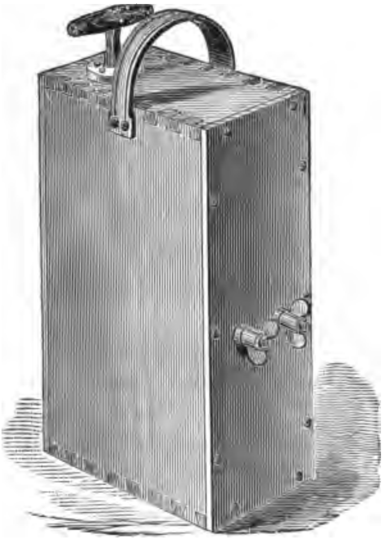


FIG. 189.

FIG. 190.

The fuses cannot be indiscrimately used by either machine. The wires *C*, Fig. 190, terminate at *E*, in a fine thread of platinum, which, upon the passage of the electric current, becomes incandescent, and ignites the fulminate, *B*, of the cap. High-tension fuses are waterproof, and must have perfect insulation. The low-tension fuse may even be connected to a bare wire. Beside this advantage, its state can always be tested by a weak current. The early defects in the priming of electric fuses, which rendered them uncertain, have been remedied, and now they are as reliable as the magneto.

The fuse should be used with wires attached to it long enough to protrude from the surface after the hole is charged. A fuse with 6-foot wires costs only $3.50 per 100; with 14-foot wires $5.60. Common safety-fuse costs 40 cents per 100 feet.

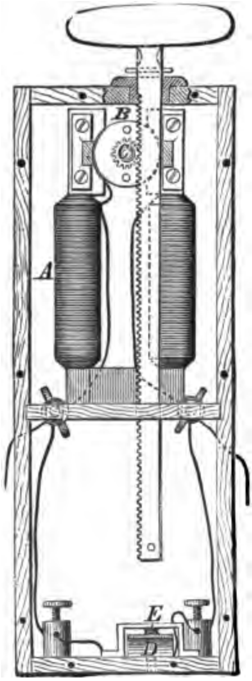


Fig. 191.

Fig. 191 shows a section of the magneto machine, which consists of a weak principal magnet, *A*, with an armature, *B*, revolving between its poles; said armature is rotated by the rack-and-pinion, *C*, operated from the handle on the down stroke. As the rack-bar strikes the spring, *D*, at the bottom, it breaks the continuous current between the two platinum bearings, *E*, and causes it to pass through the outside circuit, the fuses, etc. A machine which will fire 16 shots at once sells at $25; one good for 70 (Fig. 188) can be bought for $55.

Between the machine and the fuses is a pair of well-insulated "leading" wires, *E*, Fig. 188. They are usually wound on a pair of reels, to pay out with the progress of the work, and cost $1.25 for the reel, and a cent a foot for wire.

At *D*, Fig. 188, is represented "connecting-wire," a common wire for uniting the fuses, bought for 10 cents per lb. Some of this is lost, though much of the connecting- and fuse-wire can be regained after shooting. In a mine firing 80 shots at each "tally," the consumption of connecting-wire averages nearly 2.4 lbs. per 24 hours; in a small, double-track mine tunnel 2.6 oz. of connecting-wire, and 7 exploders per lineal foot; in the Musconetcong Tunnel, 0.32 foot of connecting-, 0.066 foot leading-wire, and 0.7 exploder per cubic yard of rock removed.

The mine may be divided into shooting-districts, and each successively fired by the same man and machine at the level mouth; or the several districts may be coupled by permanent lengths of leading-wire, and placed on the same circuit.

The precautions to be observed are: (1) that the wires are not broken or in contact with wet rock; (2) that the boss or blaster shall touch his hands to the wet rock before working with the wires: (3) that the wires at the point of union present a fresh, clean surface; (4) that the wires are tightly twisted together (Fig. 188). This is *essential* with the magneto, and expedient with frictional machine; (5) that the advertised limit of the machine be not exceeded; (6) that connections with *E* (Fig. 188) be not made until the moment of firing; (7) that, where the battery is used, all its latent or engendered electricity be let off by touching the wires together before connecting.

84. In the manifold operations of mining, tunnelling, etc., the circumstances of drilling and blasting are so varied that empirical rules cannot be formulated for general use. The character of the material to be excavated, the area and openness of the face of attack, the prescribed limits within which the miner works, and the difference in treatment to be given vein or country rock, are a few of the contingencies that influence the efficiency of this item of underground expenses. Only experience can premise the effect of the explosive, for it becomes almost a matter of instinct instead of rule. Nevertheless, certain fundamental principles may be cited as guides taken from the experience of military engineers, who are our only source of systematic information upon this point.

When an explosive is fired, the tension of its gases acts in all directions upon the confining rock. Where the resistance is least, a tendency to rupture takes place. With powder the gases find time to concentrate their pressure upon the line to the nearest external point, and, perhaps, may break off a cone of rock. The high explosive, on the other hand, is so instantaneous that a concentration of force is not effected, and the rock will break anywhere as soon as at the weakest line.

The placing of holes should have due regard to the structure of the rock. Whether of igneous or aqueous origin, it is traversed by a congeries or by systems of planes which rive the rock into more or less regular blocks. These rifts, cleats, or seams constitute the lines of slight resistance which are advantageously employed as lines of rupture. The quarrymen endeavor to select them on which to split the stratified rock by lewising or wedging. In the work of removing the more massive rocks, even in a fractured condition, the crevices or free faces may avail.

Under no condition can the miner arbitrarily select the lines of fracture, except it be in "tight" ground and massive rock. He then depends upon the shortest vent from the explosive chamber to the surface, and is guided thereby. Any clay seam, gouge, or fault is hailed as a welcome accessory. By "tight" ground is understood that which presents only one free face,—containing no protuberances or cavities.

In each case the line of least resistance is the objective. The shorter it is, or the more brittle the material, the less or the weaker may be the disruptive agent. To secure this with the minimum of drilling requires the instinct which experience imparts, but a few rules may assist the judgment.

A hole should not be on the line of least resistance, though it may be in the plane. If a known gouge, fault, or crevice traverse the rock, the direction of the hole should be normal to that plane, and for the reason that the explosive will find vent along it the hole need not be carried down to the seam. The hole will break to *a*, Fig. 192, if the powder is disposed wholly within the bottom layer. Sandstones and limestones are better split by weak powder; a strong one may pulverize a large portion of it without breaking stone. The roof holes are usually fired first in driving a face through strata pitching toward the men (Fig. 176); the bottom holes precede the uppers in a reverse pitch. With sufficient explosive the amount of material removed is measured by the entire block of stone from the face to the plane of the holes (Fig. 193).

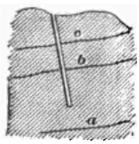


FIG. 192.

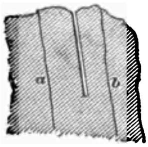


FIG. 193.

Colliers avail themselves of the cleat in coal, which comes away with great freedom if properly attacked. Generally it has one direction only, sometimes two, producing cubical or rhombohedral coal. In flat seams the trend of the main cleat determines the direction of the attacking breasts and, correspondingly, the order of mining. The most important galleries are run with the cleat, and the headings perpendicular to it are called butts. In pitching seams the dip is of more importance than the cleat.

Homogeneous rock, and particularly massive rock, has no crevice or seam to assist the miner to its displacement, and an additional element is added to his work. He must consider not only the volume of the rock to be removed, but also the state in which the shot will leave the face. In other words, each blast is a "bearing-in" shot for the next succeeding (Fig. 194). In stratified rock or seamy ground the shooting is to the joints, and the stone breaks well, just as if the seams are open faces. Porphyry and quartz is always "tight" ground, i.e., there is no seam to shoot to; there is only one natural face for attack. The careful and experienced miner will see to break to benches which will offer favorable opportunity to displace large masses with little powder.

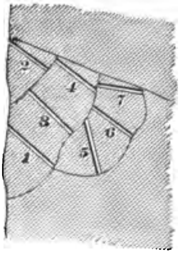


FIG. 194.

Where simultaneous firing is practised upon several neighboring holes, less heed is paid to this matter, for one cannot foretell the shape or volume of the cavities opened. Generally, several holes looking toward one another are fired merely as bearing-in holes to facilitate blasting. For this same reason, the procedure by machine is entirely different from that by hand; all the required number of holes decided upon are drilled in one heat before any blasting is attempted, and it does not signify if a hole or two too many is drilled; so the cost of dynamite is naturally higher than by hand, of steel consumed, more, and of labor, much less.

Manifestly, with single holes, the miner must drill the hole with some reservation as to future needs, and so place it as to accomplish as much as possible with the explosive. The two forces to be considered—the strength of the powder and the resistance of the rock—may be known, the first, accurately, the second, varying with the cohesion of the rock approximately.

The drilling resistance is not the same as the shooting resistance. Trap, granite, and syenite are firm and brittle; they are hard drilling but easy shooting. Pink quartz neither drills nor shoots well. Dolomite, amygdaloid, limestone, and porphyry drill easy, but break short. In other words, the components of the rock may be hard, but if the grain is open it is not difficult to work. Drinker's "Explosive Compounds" gives a table of relative resistances of different materials and the coefficients of their toughness. Having, besides, the coefficient of the rupturing effect of the explosive upon a certain material, the excavation may be ascertained from the formula,

$$W = CL^3,$$

in which W is the weight in ounces of the disrupter; L, the distance to the face in feet; and C, the charging coefficient dependent upon the rock. In a given mine, the value of C may be experimentally evaluated by repeated trial. And the rational loads in any other case are thus fixed with a moderate degree of accuracy. Thus, if a 27-inch hole shows an average of 0.5 oz. of dynamite No. 1, C is 0.38. A subsequent 40-inch hole, under like conditions of rock and agent, will require 3.3 oz. The volume of rock thrown is estimated to be approximately equal to the cube of the line of least resistance, though it will be greater with several open faces. Against one face only a shot breaks out a funnel approximately conical.

The relative position of the line of least resistance varies somewhat with the position of the hole and the condition of the face. It is the line of general throw and rupture, and extends from slightly below the centre of the explosive to the nearest external point (line *ab*, Figs. 195 to 198), measured perpendicular to the free face or to the direction of the hole.

In soft rock, with a moderately slow explosive, the line may be quite long comparatively, but the same powder in tough rock cannot get far away from the face, and *ab* is small; while with medium rock it may be three-quarters the depth of the hole.

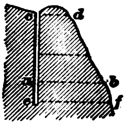


FIG. 195.

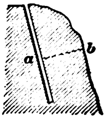


FIG. 196.

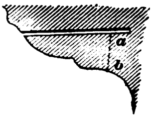


FIG. 197.

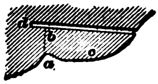


FIG. 198.

If the hole be placed as shown in Fig. 199, the hole becomes the line of least resistance, and a "pop" shot results, no matter what the rock. So too, *d*, Fig. 200, fails to break; *e* has a very short line to break, while *f* is about right.

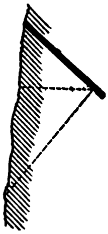


FIG. 199.

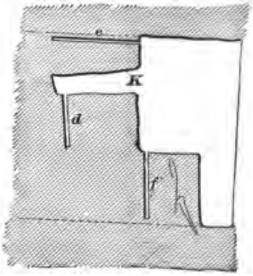


FIG. 200.

With common powder the holes cannot exceed an angle of 45° with the flush face. With dynamite, 60° is a limiting angle for almost any variety of rock. A larger angle is advised

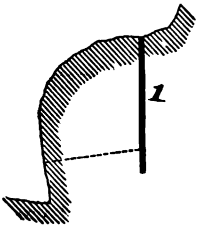


FIG. 201.

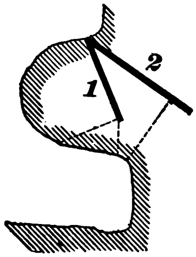


FIG. 202.

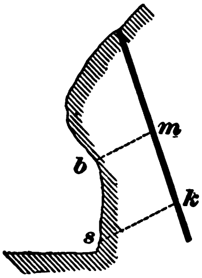


FIG. 203.

only when a free face offers a hollow or bunch (Figs. 201 and 202). Such an exigency, while it may require a deeper or a shallower hole than that of the average hand work, increases the efficiency of the blast. In Fig. 202, hole 2 will displace more than hole 1 with equal powder and work. Fig. 203 illustrates an unfavorable hole. If very deep it will blast out to *ks*. A hole, *om*, will do proportionately better for the same weight of powder and much less drilling; but then the subsequent removal of the block of ground *bmsk* will require nearly as much powder as for the original hole, *ok*, the line of resistance being the same.

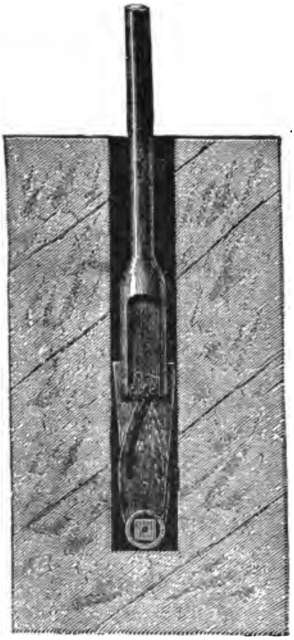


FIG. 204.

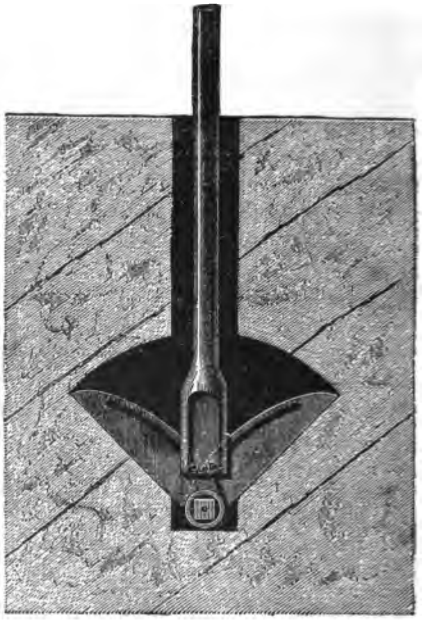


FIG. 205.

Blasting in homogeneous material is more satisfactory than in short fissured rock, which can only be worked with shallow holes. It is also true that drilling uniform rock, even if hard, is preferable to putting holes in variable rock. In large galena veins, deep, narrow holes do great execution; so "squibbed" holes in the flint-zinc-lead beds of Missouri. The direction of the holes relatively to the earth has no influence pro or con. Vertical or horizontal holes are equally effective, other things being equal, except that in shelly ground horizontal holes are preferred because of smaller liability to caving in.

Occasion arises when it is desirable to have deep holes—the ground may be brittle and coarse-fissured; or, occasionally, a deep hole may be desired in hard rock, and sufficient powder to do the work cannot be crowded down into the hole. In such event a chamber is prepared by exploding a light charge of giant under heavy tamping. Into the cavity thus created is tamped ample explosive for the purpose. This process is called "squibbing."

Expanding bits are also used to accomplish the same purpose (Fig. 204), When the desired depth has been reached, a pair of cutter-wings are forced out (Fig. 205), and in rotating cut out a hemispherical chamber. In soluble rock, acid poured into the hole will eat away a space for the powder.

When a streak of rich and brittle, or soft, mineral is to be recovered from an extensive exposure, it is blasted separately from the rock, which has a different degree of tenacity. Long lightly loaded holes are drilled in or alongside of the ore. On this account hand-work is more economical than machines in mines of high-grade thin ore streaks. In shafting it is seldom that any attention is paid to the reservation of ore, so that the methods more commonly adopted are similar to those used in driving. In a shaft long in proportion to its width, two centre-cut craters are separately fired.

CHAPTER VIII.

DRILLS AND DRILLING.

85. Channellers and quarrying machines; cost, economy, and use; tools needed; steam and pneumatic power. 86. Percussion drills; requisites for a good drill; construction; valves and improvements; descriptions of the different drills in the market, Rand, Sergeant, Ingersoll, Burleigh, Schram, and Darlington. 87. Rate and length of stroke in hard and soft rock; drifting, sinking, and stoping by machine; relative cost and progress by machine and hand labor; shapes of bits, tools, connections; column *vs.* tripod. 88. Diamond-drill; description of machine; operation; gear and hydraulic feed; solid and annular bits; consumption of stones. 89. Rate of progress; economy, cost; its function as a prospector; mode of keeping its record; Brandt's drill; electric drills; perforators and entry machines. 90. Size and depth of holes; system of arranging holes; Mt. Cenis and St. Gothard system; the American "centre-cut" system. 91. Brain's radial system; progress, cost, and ratio of cubic foot broken to the foot of hole; Gen. Pleasant's method of long hole or continuous drilling by diamond drill. 92. Coal-cutting machines; discussion of the types; comparison of the work done, with hand-labor; account of the Harrison, Jeffry, Sergeant, Lincke, Winstanley, Marshall, and Frith's machines; electric cutters.

85. THE successful substitution of machinery for hand-labor has proved a most important advance in engineering. The extraction of fuel, ore, and rock is more economically and rapidly accomplished with greater comfort and safety to laborers; hard rock is no longer an obstacle, and very long and large tunnels are rendered possible. The time spent on preparatory workings is shortened, and this element of time is an important consideration in the rapid opening of, and quick returns from, mines. As machinery never "strikes for wages or time," irregularities and "shut-downs" are less frequent than formerly.

Every form of hand-labor tool has been successfully imitated and extensively introduced. The quarry methods of lewising (p. 331), "jumper" (p. 314), saw, chisel, pick and auger, find their counterparts in the channeller, percussion-drill, coal-cutter, and diamond-drill.

In days of yore the quarrying of dimension-stone was accomplished by the trenching along lines decided upon. Carried often to 10′ depth and wide enough for a man to operate his pick, these trenches wasted much good material. The channellers and gadders now used dig these trenches as deep as desired, but only 2″ or 3″ wide. These machines are mounted in different styles, and cut perfectly true lines at any angle with or across the strata.

For extensive quarries these machines are mounted on a portable sliding carriage, with boiler, rails, etc., and a feed which automatically moves it with the progress of its channel. A set (gang) of five cutters receives a reciprocating motion from a steam-piston, through a connecting-rod, or through some yielding contrivance from the crosshead of the engine. The latter gives an elastic blow to the cutters. Automatic contrivances keep the cutters to their work. Machines are also supplied for cutting two channels at a desired distance apart; these are known as "double-gang machines," and cost from $1200 to $2000 complete. With 3 men and 400 lbs. of coal, at 150 strokes per minute, they cut from 75 to 400 sq. ft. of stone, —the former in marble, the latter in soft lime,—and replace 50 men.

Many quarries employ, instead, a steam or air drill, mounted on and traversing longitudinally a long stout bar, which lines up the work. This frame is comparatively light, and is adjustable to a high or low position and for vertical or horizontal holes (Fig. 206). With this a channel is cut to the length and depth desired; or an X-bit drills round holes, at certain distances apart, to full depth, the partitions between to be broken down by a broaching-bit (Fig. 206), or shallow holes are drilled for plug and feathers. 300 linear feet of 2′ holes are "put" in 10 hours, or 70 sq. ft. of channel; in granite, 28 sq. ft. of channelling is done. The U. S. Census Reports show the cost and progress in quarrying to be very varied, with a marked improvement over hand-labor in both respects. Moreover, the value of the stone is enhanced, being less shattered, as also the value of the quarry, because all the stone is saved. In Vermont the Ingersoll percussion and the Sullivan diamond-drill are used.

A tripod can be had arranged with a slot movement to the drill body, so that, with one setting, three parallel holes can be drilled for "complex lewising."

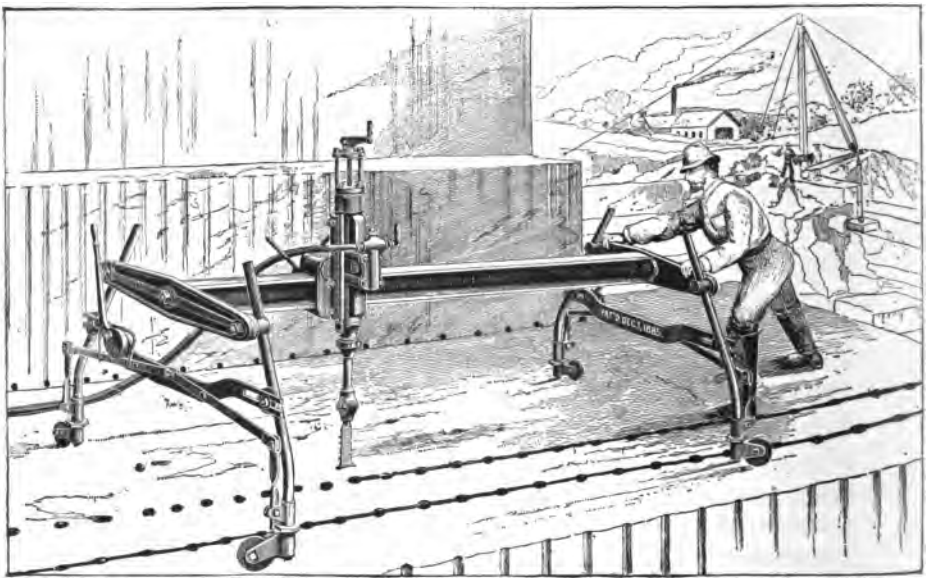


FIG. 206.

In limestone quarrying 5-foot beds, 23 holes of 7″ depth can be done per day by hand, and 400 by machine. In blue-lime, steam-power drills are seven times as rapid and one-fifth as cheap as hand-work.

86. Power-drills depend upon percussion for penetration of the rock. A steam-cylinder, sliding in a guide bed-plate, mounted on a tripod or column, and a cutting-tool clamped as an extension of the piston-rod, comprises the mechanism, which has attained a simplicity of parts that has made it the "chief element of mining success." C. D. Lawton, Commissioner of Mineral Statistics of Michigan, says that "in the progress of Lake Superior mining two forces must be allowed to have the precedence before all others—the air-drill and giant-powder."

Its comparatively small weight, 200 to 350 lbs., makes it portable, and yet it has enough metal to withstand the extremely hard usage it must receive. It occupies a small space, and can be set up in a stope or room without greatly interfering with the removal of broken rock, and will drill holes in any position or direction.

Steam is the motor fluid above ground, and compressed air below, with an ordinary pressure of 50 to 80 lbs. per sq. in. The horse-power of the drill is estimated as a simple steam-engine, with the important difference that the ratio of the area of piston-rod to piston is larger. Again, the steam-engine does its work throughout the entire stroke, but the drill-engine only at the end of its stroke. Hence it can never work expansively. The air enters the cylinder and propels the piston to the end of its stroke, and the attached drill strikes the rock. At that moment the piston reverses the valve, which admits air at the lower end of the cylinder, while a ratchet and spiral device slightly turns the tool, which is being drawn back for the next blow. As the work to be done on the return-stroke is merely to lift the tool, the annular area of the piston is but half that on the other side, and little power is consumed. At the proper, point in the up-stroke the valves are again reversed and the operation repeated.

The rapidity of the blow varies with the ability of the machine, and is altered to suit the hardness of the rock. The speed averages 200 blows per minute. A short stroke, light blow and rapid rate give the best progress in hard rock, and a hard blow is best in soft rock, provided the drill does not "stick" in the hole. High speed may be desirable to attain rapid penetration, but kinematic difficulties place a limit to the speed. A maximum of effectiveness is obtained when the full air-pressure is exerted at the moment of the blow. So the valve should not reverse until that instant, and then instantly, without "dancing"; nor should there be any back-pressure on the lower side of the piston. To do this rapidly and accurately was the problem.

It is in the solution of this, the predominant feature of drill mechanism, that the Sergeant, Rand, Ingersoll, and Burleigh types have survived the active competition, in this country; the Darlington, the English favorite, accomplished it in a different manner; while on the Continent the successful native machine is the Schram.

There are two systems of moving the valves,—the tappet, or two-lever system, and the duplex, requiring a fluid. The first has long retained its place. In all the early forms of drills, except the Wood, the valve was operated by means of an external rod from an exposed three-arm tappet, moved by a projection on the piston-rod. This is the principle of the steam-pump, but its slow speed does not give rise to the trouble that was found with power-drills, in which the numerous and violent shocks caused the breakage of the moving parts, particularly in a cold atmosphere. During the progress of the Hoosac Tunnel, so continual were the repairs, that a perpetual stream of men was passing, carrying some piece of the machine.

These repairs, the loss of head and of power, because the valve is reversing before the piston has completed its stroke, the danger of knocking out the cylinder-head if the tappet fails, and other early objections to the tappet, were gradually overcome. The tappets were concealed, the arc of their motion was reduced, and the form of the machine was rendered more compact. Many of these disadvantages were inseparable from the form, but the fact that a positive valve movement is obtained and that it is safer in the hands of unskilled labor, explains its retention.

In the Burleigh, the piston operates two rockers, which in turn oscillate the valve. This requires more dead-space and consumes more steam than the improvements adopted in the

"Little Giant," Fig. 207. Its valve is thrown by a centrally located three-arm rocker, that insures a positive motion. The durability was increased by separating the spindles from the valves and tappets which they connect. The Sergeant tappet has the valve and the rocker in one three-armed piece. In both, the movement is effected by contact with the inclined planes on the piston.

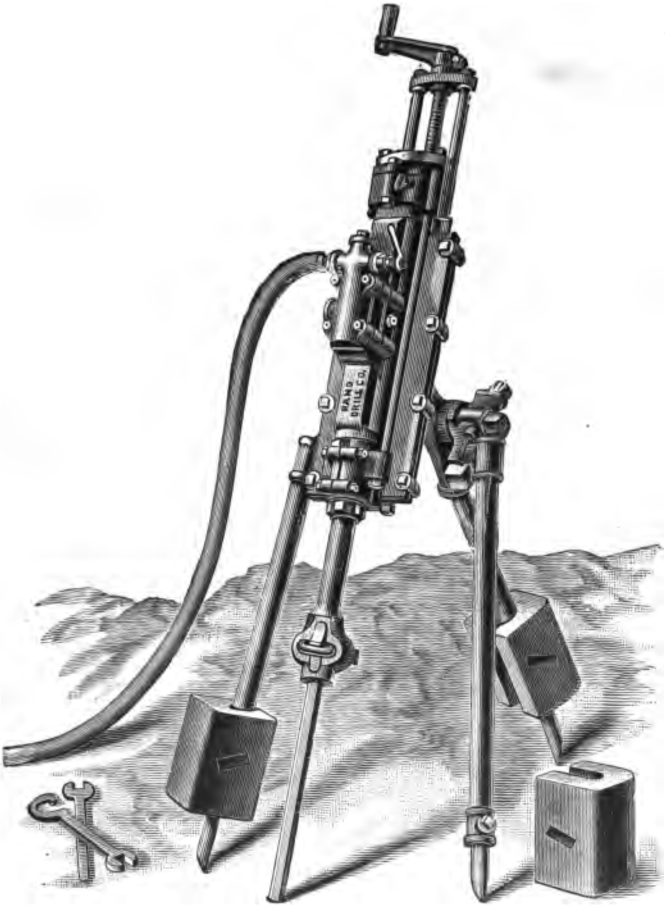


FIG. 207.

The other valve mechanism is that adopted in the "Slugger," Sergeant, Ingersoll, and Schram. It embraces a steam-moving valve, which admits of higher rate of speed. The "Eclipse," Ingersoll (Fig. 209), and the Rand "Slugger" (Fig. 208), are of similar action. Two port-holes connect the annular groove in the piston with each opposite end of the valve-chest, and are opened or closed by the piston passing over them; the supply for one end, and the exhaust to the other end of the valve-chest, are simultaneously opened. The annular groove, therefore, is a general exhaust-outlet for the valve steam, while the motor steam is exhausted by the valve connecting the inlet passage with the exhaust-pipe. The sectional views (Figs. 208 to 210) show the connections clearly. When the piston and cylinder wear away slightly, the steam-pressure works to the wrong end, the exhaust becomes imperfect, and the valves fail to act properly.

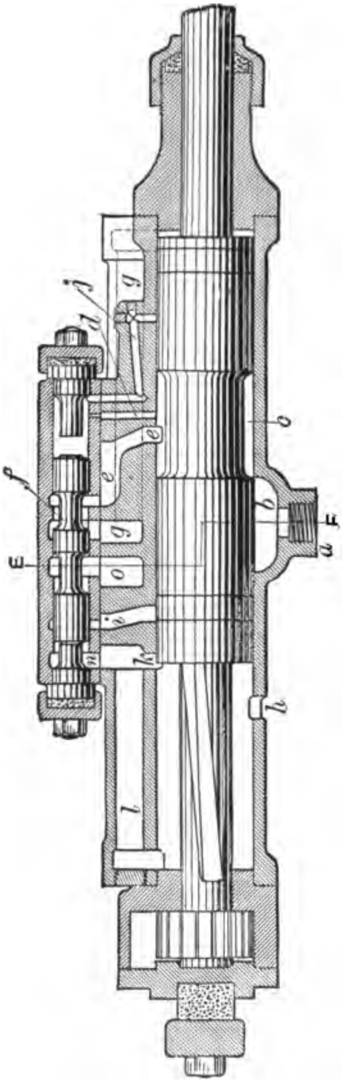


FIG. 208.

In the Sergeant (Fig. 210), the piston-valve is moved by exhaust steam from the opposite ends. An auxiliary slide-valve moves over the arc of a circle by shoulders on the piston, opens and closes the ports, and is a trigger regulating the movement of the main valve. There are no openings in

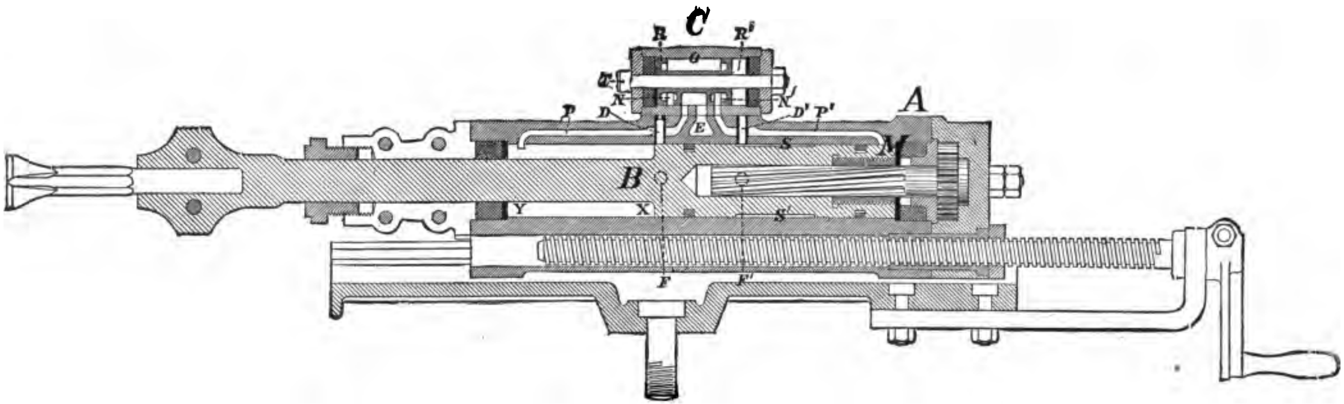


FIG. 209.

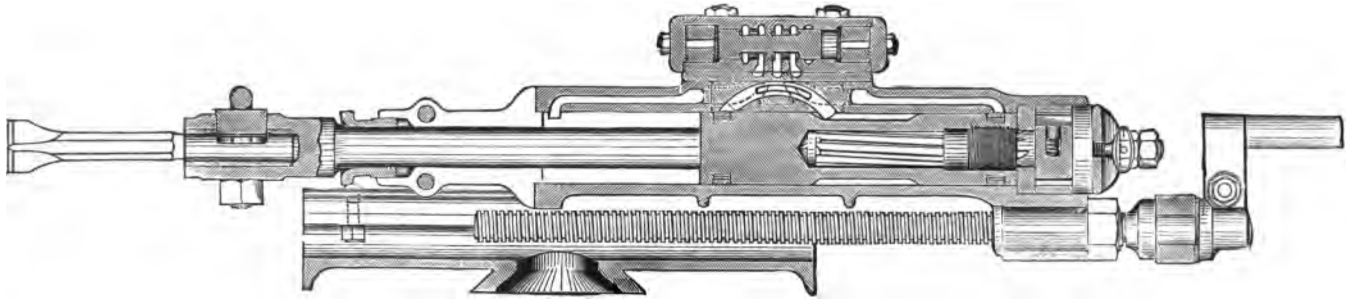


FIG. 210.

the side of the cylinder, and no ports for the piston to close; the exhaust remains open at one end till the blow is struck, when the valve reverses immediately.

An account of the Schram and the Darlington drills is to be found in "André's Mining Machinery," from which the following is taken: "Schram's consists of a slide-valve and a slide-rod that admits steam to the cylinder for raising the piston and drill. When the piston passes a certain front port-hole, steam enters through it into the back of the valve-chest, at the same time that the front valve-chest, through the other port and the hollow circular groove of the piston, communicates with the exhaust-pipe. Steam then works full pressure on the slide cylindrical rod, which, with the slide-valve, is forced towards the front valve-chest, so that the back steam-passage is open to the cylinder, and the front steam-passage connects with the exhaust pipe. The piston moves forward, and, when it passes the back port, allows the steam to enter the front valve-chest at the same time that the back valve-chest, through its back port and the circular groove of the piston, communicates with the exhaust. The slide-rod is forced back, the front steam-passage opens, and the back passage communicates with the exhaust. The slide is in the form of two spindle-valves, so that it remains in position without recoil, and the annular groove of the piston is always in communication with the exhaust.

"The Darlington has only two working parts,—an extreme of simplicity: a cylinder and its cover, and a piston and its rod. The piston is made to operate as a valve. The inlet pipe, having open connection with the cylinder, *always* furnishes the pressure to lift the drill, which rises whenever there is no pressure on the back. On its way up, the piston first covers the exhaust (above the inlet), and then uncovers an equilibrium-passage, by means of which communication is established between the front and back ends of the cylinder. Then air or steam enters and operates over the greater area, at the back, and first checks the upward movement, soon overcomes it, and finally produces a forward motion. The propelling force, now, is dependent upon the difference of area between the back and front of the piston. On its way down it soon cuts off the equilibrium-passage and the air can only enter at the inlet; the steam operates by expansion for a short space, till the piston has passed and uncovered the exhaust-port, when a discharge takes place as the blow is being struck. One fact is noticeable, that the amount of steam used is only that necessary for the down stroke; for that used to raise the drill escapes by the equilibrium-passage to the top."

87. The drill-tool is of steel, with an X, I, Z or S cutter, the first three forms being more common, because more rapidly dressed to a shape; but they must be very regularly turned, or the hole will be "rifled" (cut triangularly instead of circularly). The S is the surest for a round hole. If there should be a tendency to rifling, try a change in the form of the bit. Each bit is of specific value. The flat cuts homogeneous rock well, but will not stand long. In sandstone, the bit should be bluff, and in some silicious rocks even have a slightly flattened edge, a "stub." For rocks that do not crush, but chip, a sharp edge will be needed. The steel used is from ¾" to 1½" diameter, according to the percussion to be imparted to the rock. The smallest is for a 2", and the largest mentioned for a 5" piston, corresponding to a blow of about 200 lbs. and 1200 lbs. respectively. The drill steel is obtained in sets graded according to the amount of the feed of the drill with which they are to be used. Each bit has a life of about 275 feet of holes, of moderate depth each, provided the machine is not too powerful to handle long steel. The average mining size is the 3" piston, with a 1¼" to 1" steel, feeding 20" to 24", and having 8 pieces to the set; the longest for a hole of about 10 feet. Ordinarily, a bit will drill 3" before requiring sharpening; and in changing the tool care should be taken that the follower has an edge narrower by $\frac{1}{16}$" to ⅛" than the one withdrawn.

The shank of the drill steel is inserted into the enlarged end of the piston-rod and clasped by a split-chuck lock-ring (Fig. 207), or it is keyed or bolted.

The rotation of the drill through a small arc, each stroke, is accomplished by about the same appliance in all patterns,—a fluted bar and nut constituting a rachet. The rotation must be perfectly regular, to prevent rifling. The Burleigh has a spiral feather on the piston-rod, recessed into a groove-piece in the cylinder-head. It is toothed and held by a detent, which permits it to turn on the forward stroke, but prevents turning during the up stroke of the engine. In the Ingersoll, a grooved bar fitting into the back of the piston turns it on the back stroke, and is itself allowed to rotate on the down stroke (Fig. 211). The Darlington device is like the Burleigh. It turns the piston and drill on the up stroke, and itself turns during the down stroke. In the Schram, an auxiliary piston turns the drill.

FIG. 211.

In whatever the pattern, the cylinder and its tool slides in a guide-way which, being rigidly mounted, carries the drill-point forward more or less rapidly as the cutting is fast or slow. This must be done simply, and may be by hand or automatically. For mining purposes, and whenever the small sizes of drills are employed, an automatic feed is of little value, and a man is employed instead. Irregularity in the nature of the rock implies a varying rate of penetration, and, hence, a variable feed. If a fissured rock or cavity is encountered, the drill would suddenly give way, and the uncompromising regularity of the blow would result disastrously. Only a prescient feed would obviate this liability to excessive stroke. In the larger sizes of drills the piston strikes a knuckle-joint at the bottom of the cylinder and revolves a nut that feeds the drill to its work. This saves one man as each machine would otherwise require two.

The percussive effect improves as the full steam pressure is obtained at the moment of the blow; and what is called a perfectly "dead blow" is highly desirable; but it is inadvisable, on account of the shock to the machine and the consequent repairs. The piston is, therefore, caused to terminate its stroke on elastic buffers, or against an air-cushion in the clearance-space. The latter consumes motor fluid, but is less expensive than the repairs due to a dead blow. In several forms of drills the piston is cushioned by the exhaust, instead of live air, as is the case with the plain slide-valve patterns.

A rigid support is an essential adjunct to the drill, and several types of mountings are provided, each having a special end in view, though a machine can be shifted from one style to another. In tunnels and shafts where the ranges of holes have approximately parallel directions, it is clamped to a stout hollow cylindrical column (Fig. 212), or upon a projecting arm, as in Fig. 224. Some columns have two arms for a pair of drills (Fig. 224). The arm on the bar gives an eccentric range to the drill. Jack-screws at one end clamp the column, which terminates in claws that bear into blocks resting on the rock. These can be had 6, 8, or 10 feet long, weighing about 30 lbs. per foot, at $60 to $90.

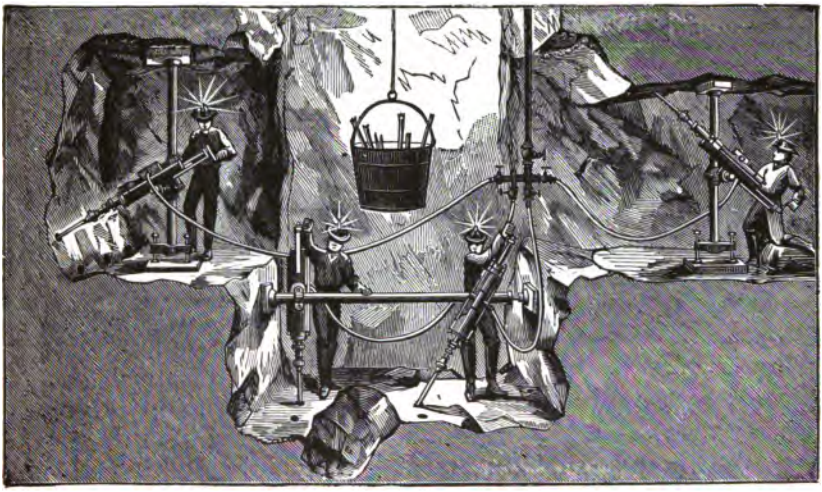


FIG. 212.

The tripod form is the more advantageous support for surface or for stope work, where it is expected to be an acrobat

(Fig. 213). It should have a universal joint, be strong, and easily set. Each leg rests in a moiled-out hole.

This machine, as described, has no intricate mechanism to watch and manipulate, and should operate from the "go." Steam gives a little trouble in starting, because of the unequal heating of the parts, but, by proper throttling, injury is avoided. The drill should always be started on a square face. Glancing blows are ruinous. Holes should be started at short, light strokes; the short stroke is obtained by feeding the cylinder close toward the rock.

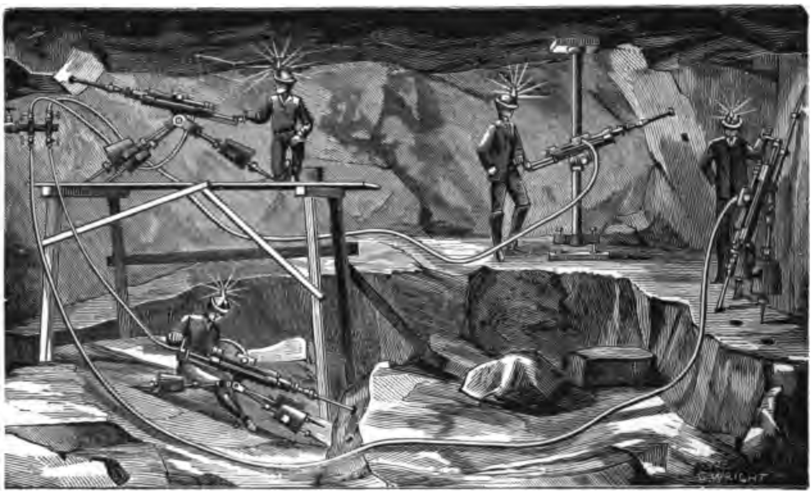


FIG. 213.

It is admitted on all hands now that the power-drill has passed its tentative stage, and can do more work with the consumption of less powder, steel, and smithing than can hand-work, and in any place that can accommodate a "double-hand" gang. One would not undertake to discuss the comparative excellences of the different drills on the market. There are several styles, doing all manner of work at shafting, tunnelling, and stoping; Figs. 212 and 213 illustrate the manner of their use. Personal observation among, and discussion with, operators in various districts fail to reveal any formula by which the makes may be gauged. In one camp the Rand, in another the Rand, Waring, and National, are indiscriminately used; still another prefers the Burleigh; while in others out here the Ingersoll excludes all others. One region prefers the "Little Giant," and another mine will discard it for the "Slugger;" in like manner preferences are displayed for the "Eclipse" or the "Sergeant." They are all highly commended, and their employment in a particular locality may be a matter of accident or of natural selection, the rock happening to be most suitable to the given form which has then survived the periods of test.

Certain it is that the author's experience favors the fluid-moved valve-drill for hard rock, the Slugger and Sergeant being adapted to our Rocky Mountain material; but whether or not they are under all circumstances the best, one would not dare to aver. Each miner must determine, from the nature of his rock, the proper air-pressure, rate of speed, and proportion of rotary motion required for the most effect. The manufacturers can give great assistance in this regard.

The comparative tests announced by different makers are of too short a duration, and are conducted under conditions too limited to avail the engineer. As a matter of fact, it becomes a question of the survival of the fittest, and that is determined by the success with which the essential attributes are supplied. A stated air-pressure will accomplish a certain penetration in an ideal drill, but the various patterns will approach this amount more or less satisfactorily as the frictional resistances are less, if the blow is uncushioned, and if the reversing-valve is perfectly accurate.

Of course, the heavier the impact, the greater the effect; but the blow is dependent upon the pressure and the drill weight. For hard rock, therefore, either the pressure should be high or the moving mass large. The former is inexpedient for economical reasons explained in No. **93,** so a heavy striking mass is imperative.

FIG. 214.

On the other hand, power-drills should be portable, necessitating a light frame and guide. A high piston-speed may be desirable and advantageous, but the kinematic difficulties render it unadvisable.

Besides these qualities, however, are those which never figure in the comparative tests, so called, the convenience in handling, and true automatic rotary and feed appliances. If a machine is capable of a variable stroke, so as to start the hole on a light, short stroke, and will "mud" well, it meets two very important features that are not always possessed. A long stroke conduces to quick mudding.

In remote camps the dominant attribute is a simplicity of parts to assure a "lasting-capacity" as well as a "boring-capacity." The early pattern is said to have had 80 pieces in it, and its repairs were so numerous that each drill was built over every two years, and it required five machines to keep one going. Now continuous work is maintained with one drill in the shop while two are working, its average life being 8 shifts, corresponding to about 400 lineal feet of holes. In a certain Lake Superior copper-mine the cost of blacksmithing is about 64 cents per drill per 24 hours. The amount and cost of breakages are too variable for any precise estimate. A mine employing 22 drills constantly allows for $60 annual repairs per drill. These two items, amounting daily to 85 cents per machine, may seem an unfavorable comparison with hand labor, where 55 cents was the allowance per daily gang; but a reduction to the relative progress will prove more equable.

A recital of a few of the comparative tests may be of interest. About Silverton, where 7 inches of hole will dull 14 to 20 drills, a machine cut 2 feet, the length of its lead, in 12 minutes. Three men will drill three 30-inch holes in 10 hours, while a machine does seven holes of 5½ feet each. An average of nine neighboring mines, in the conglomerate, showed machine drifting and stoping to be, respectively, 22 and 36 per cent cheaper than hand, and sinking 4 per cent dearer, with a progress 60, 54, and 38 per cent more rapid, the latter gain in sinking compensating for its increased expense. In the iron-mines, machine labor is one fourth as expensive as manual. Three men on a 6 × 16 shaft did 0.37 feet daily, while two machines advanced 3.4. H. S. Drinker, "Explosive Compounds," quotes an average daily progress by hand and black powder, in 21 tunnels driven in solid hard rock, of 1.441′ in heading, and 1.96′ in the bench; and of 58 tunnels in easier rock, 2.55′ and 2.62′, respectively. With machines and nitro-glycerine the progress was five to seven times as fast. The recently completed Cascade Tunnel made 2 lineal feet per 24 hours with 17 men on the heading, and 6.9 feet with 5 machines.

An eleven-months comparison of hand, Schram percussion-drill, and Brandt's rotary drill, gives an efficiency as to speed of 1 : 4.73 : and 5.26, relatively; and as to cost, 1 : 0.62 : and 0.60.

The consumption of fuel and air per drill may be calculated as in any ordinary steam-engine. The cost of the ordinary mining-drill is about $325, and of a complete plant of 6 drills, with a 16 × 24 compressor, etc., is $7000. A smaller outfit for 3 drills was recently delivered in Denver for $3700.

There are several patterns of percussion-drills operated by electricity, but the results give as yet insufficient proof of its value for reciprocating machinery. Fig. 214 shows the Edison drill.

Twelve to fourteen feet is the average depth of hole; greater than this is rarely put by the ordinary machine. A very deep penetration cannot be obtained; the impact of the blow would be destructive to a long line of rods, and the draw-back power of the piston is small.

88. M. Leschot has the credit of the first application to the miners' art of rotary diamond-drills, which have since steadily gained in favor and increased in range of utility. Several diamonds are forced into sockets on the end of a steel tube, and on a rapid rotation abrade the rock. The cutter-face is entirely covered with diamonds in such manner that no concentric circle fails to touch one, and one or more projects transversely beyond the tube. The bit may be annular (Fig. 215) or solid convex or concave face (Figs. 216 and 217). The first is more commonly used, as by that means a central core of rock is uncut, and may subsequently be withdrawn for inspection. The débris is carried away by means of a stream of water passing down inside of the tubes, washing the drill-face and carrying the cuttings up outside. The solid-head bits are preferred for mere drilling, except for large holes, the concave surface being better than the convex. The wash water escapes through the holes in the face.


FIG. 215.


FIG. 216.

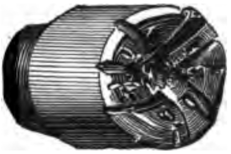


FIG. 217.

The diamonds used on the face are of the black or deep red variety; on the outer edges, borts (imperfect diamonds). "Theoretically, too many carbons cannot be put in; there should be never less than 12," and as many as 20 may be mounted on a bit. Recesses are accurately prepared for them, into which they are set and secured by metal hammered up around them. In some cases a firm setting is obtained by forcing the stones forward through small holes in the metal by means of a screw, or by hydraulic pressure. A later method consists in forcing the stones nearly through the metal, and subsequently grinding the steel down until the stones are exposed. The bit is coupled to the tube, which is added in 8-foot lengths as the hole deepens. The diameter of the hole is a matter of indifference where prospecting or the long-hole drilling is intended. Those of ordinary depth are up to 3 inches diameter, and those of great depth taper from 5 inches down. The tubes are of slightly smaller diameter, 1⅝ inch tube is used in 1¾ hole, and weighs 3.4 lbs. per foot. Figs. 218 and 220 show the guide, which is just the size of the hole, and maintains the bit in the direction in which it started; the spiral grooves allow the water to escape.


FIG. 218.

At the upper end of the drill rod is a joint or swivel, through which the supply of water is forced by means of a pump. Above is the connection with a rotary and feed motor operated by a steam-engine, the capacity of which varies with the amount and size of drill-tube to be manipulated. An 8-horse-power engine is suitable for a 1000-foot bore-hole. The running-gear should be firmly framed and supported, that the weight of a great line of rods may be easily handled; 1000 feet will weigh from 4500 to 6000 lbs. A very light temporary shed will suffice for cover.

Two methods avoiding a positive feed are in vogue for driving: one, a spur-wheel feed; the other, the hydraulic. The former is so adjusted by differential gear that its friction shall equal a desired resistance; and when this is exceeded, because of undue strain below, a regulation is obtained. Stratified rock changes so much and so rapidly in structure that a uniform feed is impracticable in deep holes, and inferior to the hydraulic feed, of which Fig. 219 is a section. It is a simple motor, which by means of hydraulic pressure on the piston produces a pressure which is maintained constant. Both ends of the cylinder are connected with the pump, and suitable cocks admit of a perfect control by the operator, who gives any variation or reversal of speed within the limit of the pump and piston-area. Gauges indicate the

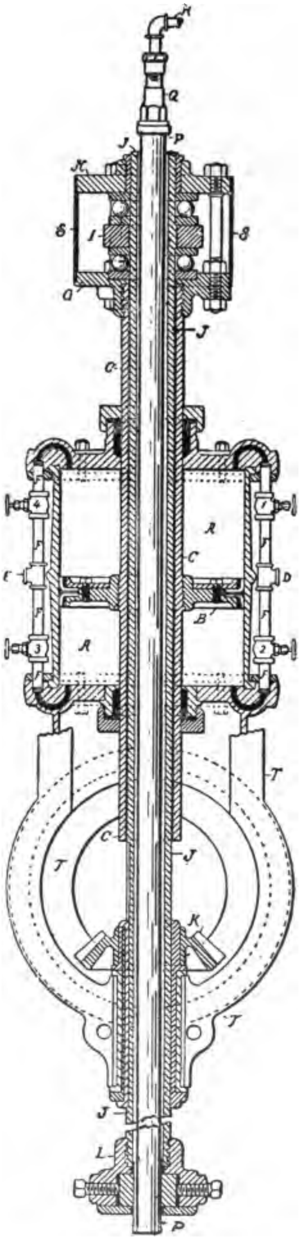


Fig. 219.

pressure. Only the hardness of the rock determines the rate of feed, and this rational system saves all parts of the machine from danger of breakage. Fig. 221 shows the feed-cylinder as the extension of the drill-tubes. The connection between the tube and the feed is by some form of chuck, which may be loosened at the end of the feed-stroke and run up to the top for a new grip.

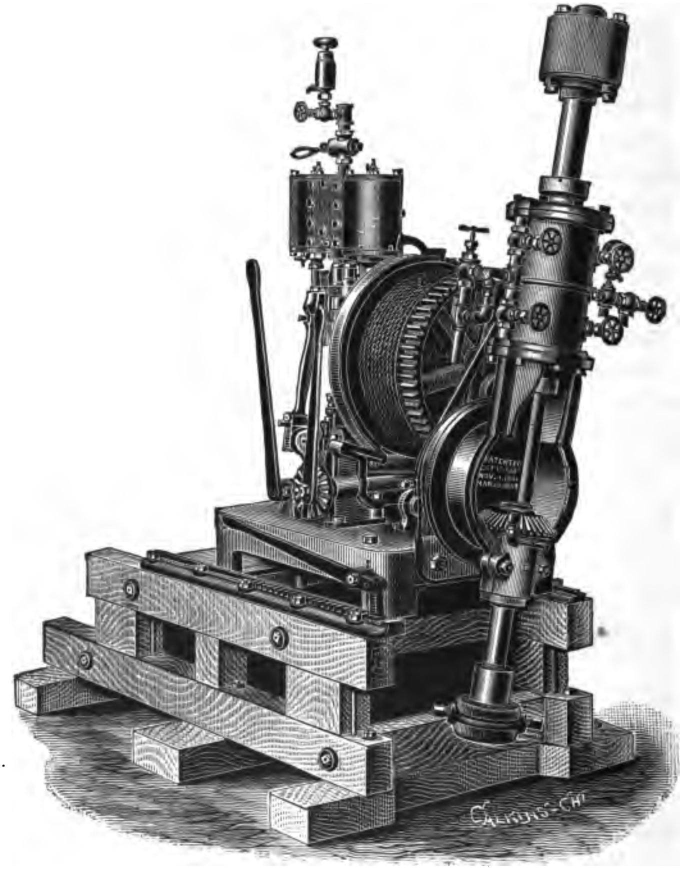


FIG. 221.

The pressure exerted by the feed is just sufficient to produce abrasion, not to cut the rock. The tube is partially suspended by friction-rollers at the surface, so that it is subjected to very little tension. The power producing rotation must be less than the torsional strength of the rods. This would place a limit on the possible depth of explorations, while the regulating power of the feed limits the capacity of the machine.

In addition to the integral parts mentioned, a steam-engine, gear, and hoisting-drum are compacted upon a rigid mounting, varied with the purpose of the borer. It may be bolted to a heavy frame bed placed on wheels, with portable boiler, or mounted as in Fig. 221 for underground work. One form is of gun-metal and steel, and weighs only 400 lbs., yet can bore 150 feet with ease. The drum is added for hoisting the drill-tube without altering the position of the machine, which remains in place till the bore is completed. A high derrick facilitates the addition or disjointing of tubes.

FIG. 220.

The rate of revolution of the tube and its bit is from 400 to 800 per minute, and the progress is remarkably fast, averaging a penetration of 13 inches to 2 feet per hour, stops inclusive. The drill bores only about one half the time. The use of the annular bit does not increase the speed, for the rods must be raised every 10 to 15 feet of advance to examine the core, which is broken from its place by the core-lifter (Fig. 222), and raised with the tube. In uniform rock the tool need not be raised as frequently as in strata of varying texture. Should the hole have penetrated a soft layer between two hard ones, the core would twist off and grind it away, and its existence would not be made known in the core. Again, the tendency of the core to turn in its tube would give false information as to the dip of the strata. For this reason, also, a flat, not round, hoisting-rope should be used. At best the core is only a partial guide. A slime-box receiving the cuttings would indicate the presence of the soft rock, but many causes combine to make even this examination unreliable. A careful measurement, an

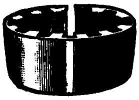


FIG. 222.

allowance for wear, and frequent raisings are the only checks. Shales and clay slates give smooth sailing, but fire-clay chokes the barrel. In such cases the full pressure of the pump will usually wash it out; if not, the tube must be lifted. With holes of a moderate diameter there is no necessity for tubing the hole.

Accidents are rare. A diamond may fall out, and, if it cannot be recovered, must be chopped up at once, or the water supply must be reversed to wash the stone up the tube. A chopping-bit is used to break up hard nodules or boulders.

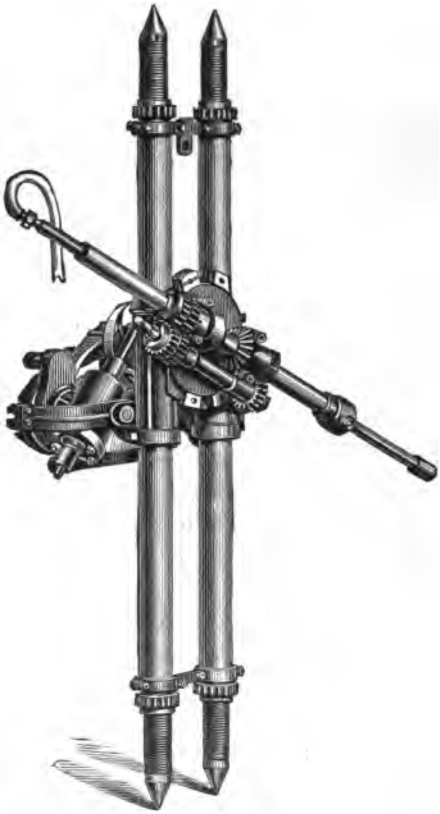


FIG. 223.

89. Holes may be bored in any direction, though the machine is best adapted to vertical ones. Fig. 221 shows the machine drilling at an angle; the "Little Beauty" (Fig. 223) drills 70 feet horizontally without trouble. The friction of the tube on the rock limits the length of flat hole that may be drilled.

In Fig. 224 are exhibited explorations in the Silver Islet mine by the use of an underground machine. For prospecting territory, for drilling a deep sump-hole to drain a mine, for rapidly sinking a connection through which to pump out a drowned mine, to sink a tail-rope bore-hole, etc., the utility of the diamond drill is generally recognized. The Poetsch method (p. 253) depends upon it, and the long-hole process is possible only by the use of it. It is suitable in hard or the hardest

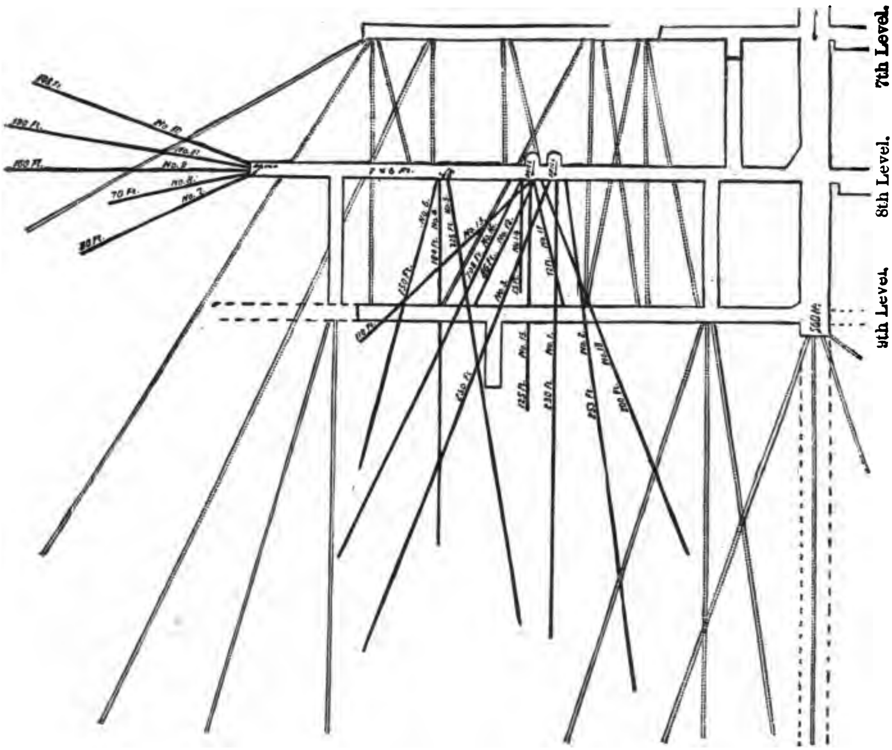


FIG. 224.

rocks, and, remarkably enough, will perform in granite better than in soft stone, according to the report of the Superintendent of the Hope Mining Co. Doubtless many properties owe their existence to the result of diamond-drill discoveries, and its use has frequently saved expense in various ways. But it is not considered infallible in its indications as to the presence or absence of the ore body sought. Though it is true that tunnels have been carried by the long-hole process at home and abroad, the percussion-drill is cheaper in tunnel and for short holes. The cost of drilling varies materially. An average of 29 2-inch holes, 400 feet each, was $2.35 per foot in a Lake Superior iron-mine; 16 holes, aggregating 5877 feet, cost $1.97 in the Pennsylvania coal measures; and 24 holes, averaging 18.9 feet per shift, with a total of 9902 feet, cost $2.22 per foot. In the Mariposa estate, the cost of prospecting holes 74 to 231 feet deep, in 34 to 146 hours, averaged $1.10 per foot, including diamonds ($0.32). The actual drilling time was about one half the total.

There is a great difference in the item charged to wear and tear of the diamonds, varying from 21 cents to 56 cents per foot. Experience has determined that the diamond is practically useless after 6 settings. Manufacturers say that there is a remarkable difference in the quality, hence in the wear of the stones. The borts and black stones are tougher than the vitreous. The item does not refer so much to the wear of the stones—as that has been found to be inappreciable after drilling 400 feet, but rather to the loss due to the falling of the stones out of their sockets. Ground charged with pyrites is especially bad, causing the stones to crumble.

Comparing it with other methods, the diamond drill is rarely cheaper in deep soft rock than the Mather and Platt system (p. 300); in hard rock it supersedes all others, except where water is very scarce. Tubing in conjunction with it is troublesome, if not out of question, for deep holes, and reaming is not easily done.

Cost of drill and outfit for 1000 feet of 2″ rods, $3872. Two drills require 5 men.

For underground work a 3-horse-power electric motor is mounted on a truck, with drum, drill, and pump, and permits core-drilling to advantage in small spaces. In many mines 1″ cores in sections of 5″ to 20″ are cut for 80 feet depth, and a great deal of prospecting has been prosecuted with this compact machine, which makes 1.60 feet per hour at a cost of 68 cents to $1.03 a foot. Fred. G. Bulkley, of Aspen, Colo., has devised a graphic representation of the results of borings by plotting them to scale on a cross-section paper, which pictorially conveys the information as to seams, faults, etc.

Rotary perforators for tunneling-out the full area of headings and entries are offered on the market. At one operation a series of cutters on a rotating boring-head grinds away the whole face for a core from the heading some 7 feet in diameter. One was used in the Mersey subaqueous tunnel. It travelled at the rate of 39″ per hour, and executed its work satisfactorily in the argillaceous chalk.

Brandt's borer, which is highly esteemed in Prussia, is a hollow cylindrical steel bar, on the end of which are formed five teeth. Rotated by a pair of small hydraulic engines, it is forced against the face of the rock, and cuts a hole the core of which is cleared away by the continuous stream of water escaping from the driving-cylinders.

90. Since the advances made in the manufacture and use of the machine-drill, the systems of drilling and of blasting have had to undergo corresponding changes. In hand-work, the object sought is as much to secure a good bench for the next shot as to break ground with the present. With simultaneous shooting, and particularly in tight ground (on faces of drifts or shafts), all of the holes are drilled more or less axially, and the blasting operations are conducted differently, because the inconvenience of handling machines supersedes the gain from attention to the lines of least resistance, and it is not always possible to drill holes with the machine in such a way as to conform to the fundamental principles.

According to the mode of arranging holes, we have three systems. The first was employed with the earliest experimental work on the pioneer machines at Mont Cenis and St. Gothard tunnels. Eight perforators were mounted on a carriage, and bored holes at different angles covering an area of 250 square feet. When the requisite number of holes was drilled, the machine was shifted to another space, where it

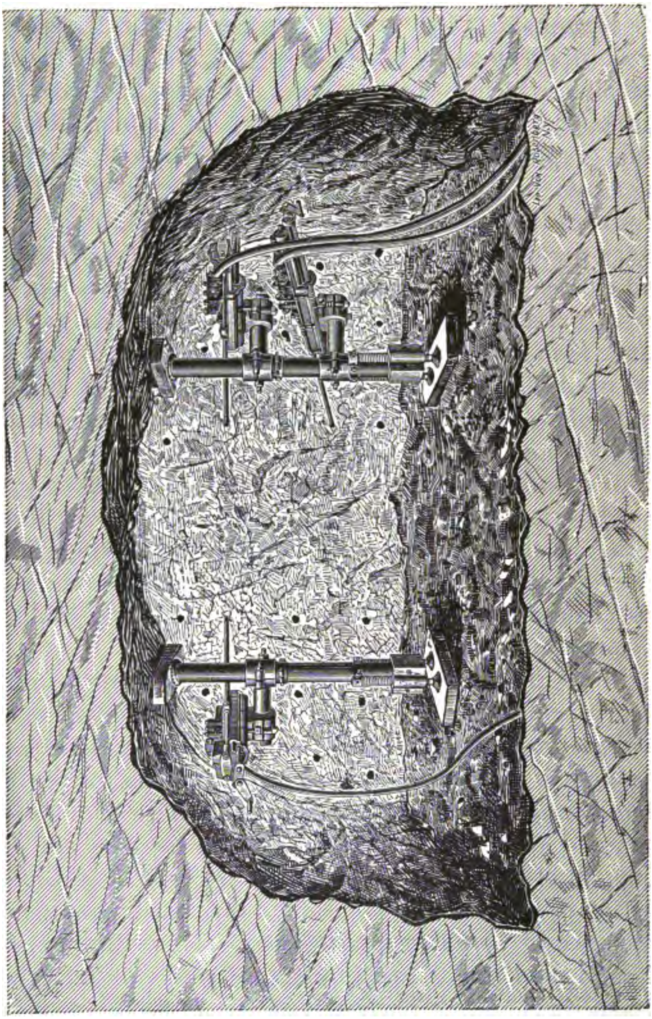


FIG. 225.

repeated the performance. It was run away when the firing was to be done. A centre hole was surrounded by a ring of eight rupturing-holes, outside of which were 3 full and 2 segmental concentric rings of holes. These were fired in volleys after the first central set. With 18 holes of 3 feet to 5 feet each, charged with 1¾ lbs. dynamite, the progress averaged 18 feet a day through schists and gneiss.

The second system is very popular, and known as the "centre-cut," which was introduced in the Musconetcong Tunnel, increasing the progress from 89 feet to 116 feet per month. The American method of tunnelling was in process (see Figs. 178 and 227, and p. 290). The face of the heading was 8 feet high by 26 feet wide, and had six machines operating on it, drilling 36 holes of 1½″ to 2¾″ diameter. The holes are drilled in vertical rows of four each, and a depth according to the location. The two central rows "look toward" each other, and meet at the bottom (Fig. 225). The next two rows on each side of the axis also point inward, but less so than the central or cutting rows; while the outside rows are parallel to the axis, or incline slightly outward. Roof holes and corner squaring-up holes complete the drilling, and should trim up the profile of the tunnel at once. The positions of these holes are variable. In very hard rock the holes of the two central rows are in pairs close together; sometimes they are single, but large, 4″ diameter. In firing the two central rows (1, 1, Fig. 226), first break out an entering wedge,—not to the bottom of the holes,—which facilitates the work of the next two rows (2, 2), which shoot toward the walls, after which the advance is squared up. The breaking-in is done with electricity, but the enlargement and squaring-up is done by fuse and a lower grade of explosive.

The depth of the holes and their distance apart depend upon the rock and the advance desired. Advances of 14 feet have been made, but there is a limit to the capacity even of nitroglycerine, and 10 feet is quite sufficient. To secure this, clean, the two central rows of holes are 10½ feet deep, the remainder 12 feet, except the six roof-holes of 8 feet each. In a narrower tunnel of say 11 to 16 feet wide, an 8-foot advance will suffice. Four machines can easily operate in a double-track (27 feet) tunnel—six can be arranged by placing two on each of the two central columns. Three machines in a single-track tunnel, and two in an 11-foot heading, will give progress as rapid as the shovellers can handle the dirt. Out of an average 8-hour shift the actual drilling heat is about 5½ hours; the shifting of tools, etc., takes ½ hour; loading and blasting and removing rock, about an hour each.

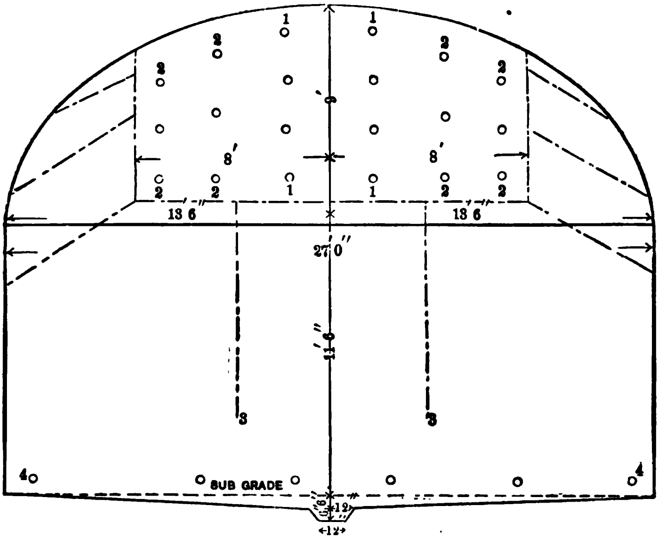


FIG. 226.

As illustrations of progress we have: the heading of the Haverstraw Tunnel, 9×16, requiring 20 holes of about 8.4 feet each, was completed in 20 hours; weekly progress in 8×27 South Penn Tunnel, 74 feet of sandstone; two machines in Washington Tunnel, 7½×11, progressed 8.26 feet per day in solid rock, with 26 holes of about 10 feet; the Cascade Tunnel, 16 feet wide by 22 feet high, progress 200 feet per month, with two faces of attack, 20 to 23 holes 12 feet deep, by 5 machines; in medium hard basaltic rock, average 6.9 feet per 24 hours; four machines in the Vosburg heading, 8×27, made the advance in 10 hours, with 26 holes (8 centre and 18 sides); in the D. & R. G. R. R. Tunnel, two machines made a complete drilling round of 20 holes, 9 feet deep, in 7 hours; the aggregate depth of the 36 holes in the Musconetcong Tunnel was 408 lineal feet, the firing of which gave nearly 10 feet advance; one shift drilled and broke a cut or a side round with six machines.

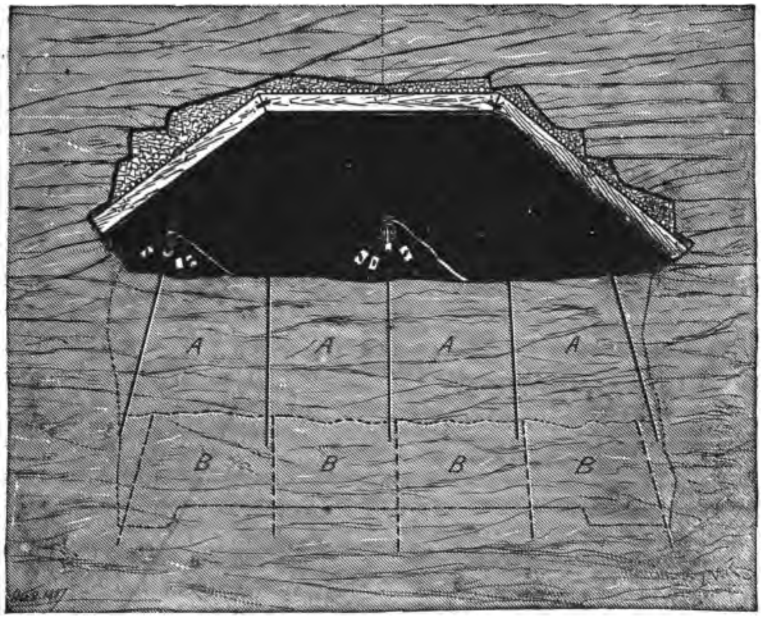


FIG. 227.

The consumption of powder varies. 7 lbs. of Giant No. 2 was used in the centre cut of the Washington Tunnel, 5½ for each side round hole; the Musconetcong consumed 0.4 lb. of nitro-glycerine, and 4 lbs. of Giant No. 2 per cu. yd. broken; on the Mariposa estate, in very tough rock, 7 lbs. of Hercules No. 1 and 10 lbs. of No. 2 per lineal foot of drift; in the Vosburg, 100 to 120 lbs. of Rackarock per advance.

The bench of the tunnels is attached in one (Fig. 226) or two (Fig. 227) sections, *A* and *B*; two wall holes, one or two transverse rows of 4 top holes downward, and half a dozen bottom holes, lift each bench with every other shift. Fifty-four feet a week is the record on a very hard sandstone bench, 14 × 27. This work is not only more rapidly accomplished, but also with a powder consumption per cu. yd. of rock of about one half that in the heading.

91. Brain's radial system is employed in headings too small for more than one machine, and, like the "centre cut," is equally applicable to shafts. The design is to drill all the holes from one position of the machine, and thus minimize the time lost in shifting. The holes are shallow and vary greatly in length, those making the smallest angle with the face being the longest. Four ranges of holes are drilled, and in a certain case the machine, from a position 4′ 8″ from the bottom, 2′ from the top, and 2′ 6″ back from the face, put 29 holes with a total length 70′, advancing 3′ with an average of 2.4 cu. ft. broken rock per lineal foot of hole. Sometimes a few extra squaring-up and lifting-holes are necessary to trim the periphery of the drift, but, ordinarily, the firing of the most angling holes first breaks out the rock to daylight and opens a face for the other successive rounds. The advance cannot be large, for neither deep nor angling holes are possible in a narrow drift. In a drift 8′ wide, two settings of the machine are sometimes made drilling from near each wall, and thus forming a modified centre-cut plan. In some mines a practice prevails of cutting a horizontal range of bottom holes, two ranges of holes looking downward, and a top row to break out horizontal instead of vertical wedges; this plan requires a bar-mounting for the drill, and a drift say 7 × 8 feet.

Gen. Henry Pleasant's method of shaft-sinking is a novel and eminently successful application of the diamond drill. One or more diamond drilling-machines are set up over the site of the shaft, and bore vertical holes as deep as the shaft is to be carried. The machines are moved to new positions and additional long holes bored. The operation is continued until the entire area of the shaft is pierced by holes at suitable distances apart. The St. Clair shaft of the Reading Coal Co.

had 35 holes drilled to a depth of 200 feet; 25 holes covered the space 13′ 10″ × 16′ of the Norwegian Colliery shaft. An average of three machines in six weeks bored 35 holes through 300 feet of hard rock over an area of 25′ 8″ × 13′ 10″.

When the "continuous process" is completed, the machines are removed for the blasting. The holes are filled with sand or water for the full length, except in the upper 3 or 4 feet, which are treated like short holes, charged with dualin and fired,—the central ones first. When the débris has been cleared away, the shaft will have advanced 3 or 4 feet. A few feet more of each hole are cleaned out (sometimes the bottom plugged with clay), loaded and fired. Thus each section advances with an alternation of shooting and hoisting. Herein lies the secret of the success of the method. The operation of boring is continuous to the end, and the other operations may be uninterruptedly prosecuted. Though it is not always cheaper per cubic feet, it effects a great saving in time, and quick access underground may prove the element essential to the success of the undertaking.

92. The undermining of coal is accomplished by machinery of two general types: one depending upon abrasion produced by a saw or chisel cutter, the other upon percussion. With electricity or air as the motor powers, these extensively replace hand labor, with beneficial results.

A machine must occupy little room, be low and light; when these essentials are fulfilled, the most imminent danger of the miner's occupation is avoided. Over 1800 machines are mining coal in this country, in conjunction with 15,000 men, and doing the work of about 35,000 miners, without supplanting as many as the Knights of Labor would have us believe. The machine does not dispense with the labor of the miner, it only more efficiently accomplishes the most arduous part of his work. The chief value of the change lies in the subdivision of the labor formerly imposed upon one man, and the consequent celerity and safety resulting from the attention of each man to his own branch of the work.

In Illinois the Legg machine is used in driving the rooms;

elsewhere, the Harrison, Jeffrey, Yock, Lechner, and Sergeant. The Lincke is used to some extent in the western country; the Marshall and Frith is the old-style machine still in vogue across the water.

The rotary or chisel-cutters are always accompanied by a positive feed which advances the machine; in this class are the Lechner, Jeffrey, Marshall, Hurd and Simpson, Baird and Lincke. In the percussion class are included the Frith, Harrison, and Sergeant, each of which, except the Frith, must be moved after the bearing under hole has been drilled. The Harrison machine is the most popular in Ohio and Illinois, and is illustrated in the accompanying cut (Fig. 228), which requires no explanation.

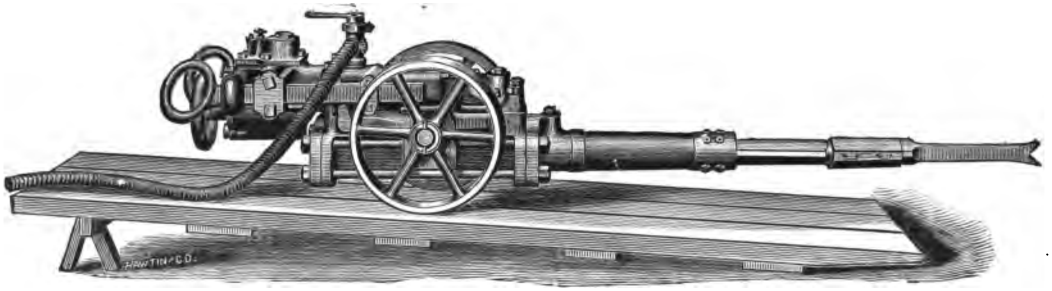
The valve-motor is a single-cam rotary device. It is compact, light, and will "bear in" about 80 lineal feet of 3½-foot holes in ten hours, allowing two hours lost in changing bits and positions. In twenty minutes it will cut along the face to the width of its board. 10 cu. ft. of 70 lbs.-air, at a rate of 200 blows per minute, is the average consumption. The several sizes of machines differ only in power and depth of groove.

The kit of tools (three pairs of augers 2′, 4′, and 6′ long, and one pair of 18″ extenders) is dulled every day, and refaced, by the blacksmith. Each bit is refiled by the blaster after use. With two machine-men, it employs five loaders and a blaster. The Sergeant rock-drill has been adapted also to coal-mining, and gives eminent satisfaction in the South.

Frith's machine imitates the miner, working a 75-lb. pick by bell-crank lever. At a rate of 70 blows per minute, 11 square yards of a 2″ groove are cut 42″ deep per hour. The simplicity of these patterns enables them to be readily handled in thin seams.

The other class of machine is represented by the Jeffrey air and electric, of which the latter style is shown (Fig. 229).

In 6 minutes it will cut a groove 39″ wide to full depth; can be reset to position in 9 minutes, and moved into an adjoining room in 20 or 30. It therefore undercuts a room in



Outside view.

FIG. 228.

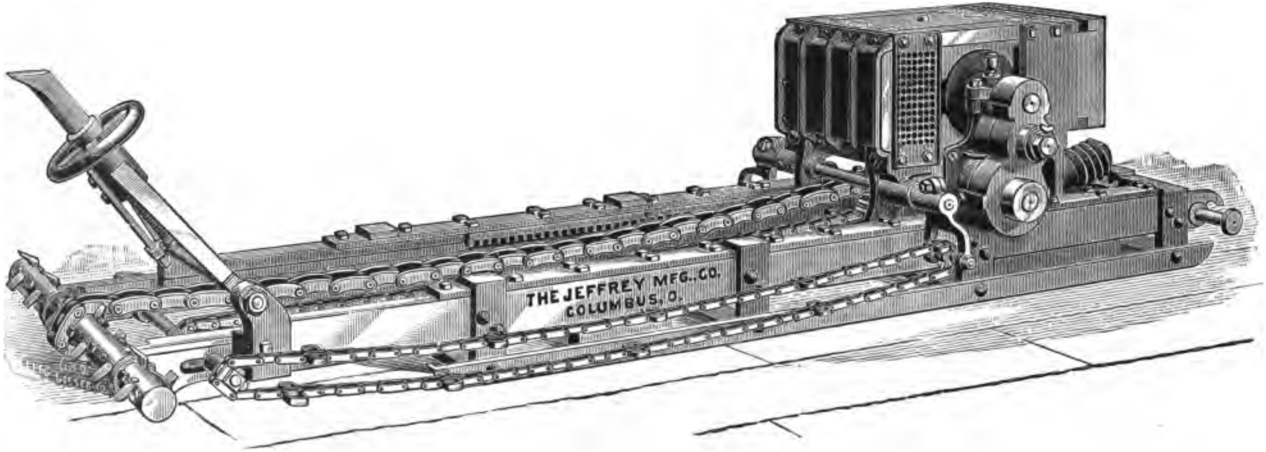


FIG. 229.

about 2 h. 10 min.; 60 amperes at 250 volts will operate it. It occupies an area of 2′ × 7′ 6″ and weighs a ton.

The Lincke cutter is a revolving axle 3′ long, like the Jeffrey, and gives nearly equal satisfaction. The Lechner is similar.

The Winstanley is a rotary toothed disc capable of being turned under the carriage or out against the face, revolved by two oscillating cylinders working at a pressure of 30 lbs., and cuts 70 square feet per hour. It is mounted on a carriage moving along a track longitudinal with the coal-face; and weighs 1500 lbs.

A chain carrying several chisel cutters is the device in Marshall and Carret's machine on wheels. It is braced to the roof, and the cutters are so set that they carry the scrapings outwards. This has the advantage of keeping the machine to the coal. Another Marshall design, also hydraulic, "cuts into coal like a scoop into cheese."

Whatever the design, it should be capable of being handled by two men, and in size be small enough to admit of working around and between the props. In this respect the cumbrous Yock machine is lacking. It should also be capable of starting in the corner of a pillar or loose end, and of cutting clean to the walls of the rooms, right-handed or left-handed, and at any height.

Hard fireclay bottoms are cut into more conveniently and deeper than by hand. Firm and even steep coal can be better hewed by a 7′ kirving. The danger of falling coal is eliminated, for the machine underholes before the roof can "fell." Rooms and long-wall, thin or thick seams, are holed with equal facility. Pillars, as yet, are more economically robbed by hand. Pyritiferous and bony nodules cannot be cut with the same ease as coal; and in these the machine should have a feed-appliance capable of slipping when the resistance exceeds a given amount. The percussion has a marked advantage over the rotary tool, in that it can avoid much of the pyrites. The rotary cutter-bar does not find favor with some operators in a highly pyritiferous coal.

The introduction of a properly selected cutter always results favorably, though some mines have abandoned them for causes not pertinent to their economy. The men do not take kindly to them; but their efficiency is undoubted. Any machine will cut 13 square feet in, say, 10 minutes, or a 20′ room in less than 2 hours. Making a liberal allowance of 30 minutes for shifting, 4 rooms are underholed in a shift with the employment of 2 men. This in a standard vein (4′ thick) corresponds to 45 long tons, or at least 7 men's kirving. As a matter of fact, the output is more nearly 70 tons per 10 hours. The regular work of a Legg machine shows 86 tons per 10 hours on a vein 3′ 7″ thick; and 272 machines produced 2,243,210 tons in Illinois (1888). On the output of 70 tons, with 2 men to a face, 11 rooms would have to be kept open to equal the supply of one machine. A mine producing 1000 tons of screened coal (1300 tons of "run of mine") could obtain it from 28 machine-rooms, or 77 hand-rooms. The operations are therefore more concentrated,—less territory has to be kept open than in hand-work; and this is an important feature, recommending the "iron man." Again, the price for cutting is 20 cents, which, with 32 cents for timbering, makes the cost of machine-coal 52 cents, against 72 cents formerly paid. The quality of the work is in every way superior to that by hand. Its groove is only 2½″, as against an average of 6″ for even the skilful miner, whose kirv is 9″ high at the face and 2″ at the rear. In 4′ holing, over 3 cubic feet of coal is gained per yard of face. There is therefore less waste, more large coal; and the ratio of slack by machine and hand is $\frac{1}{18}$ to $\frac{1}{4}$.

To keep 7 at work uninterruptedly, 3 to 5 additional machines are necessary.

CHAPTER IX.

THE COMPRESSION OF AIR.

93. Theory and principles; heating during compression; influence of altitude; losses in the compression; equalizers and compound cylinders; construction of the machine and its requirements; means for rendering the resistance of the piston uniform. 94. Calculation of the work done upon the air; tables; formulæ; discussion of the valves and forms of the principal air-compressors on the market; air-receivers and their form and utility. 95. Conduction of the air; air as a motor; pipes, expanders, etc.; theory in the operation of the motor; tables of losses by friction; discussion of the economy of working with or without expansion.

93. ANY of the machines described in the previous chapter may be run by air or steam; but though steam is the cheaper motor, air has the advantage of giving cool, dry, ventilated rooms.

When air is subjected to pressure, its volume is proportionately diminished (see page 184), and, transported, its expansion is capable of being applied as is steam. To secure 100 lbs. of absolute pressure from "free air," its volume must be reduced to 0.147 the original bulk; for 200 lbs., 0.074. To obtain the same pressure from steam, it must be superheated to 338° F. and 388° F., respectively. By "free air" is understood air at the atmospheric pressure of 14.7 lbs. per square inch, absolute. By absolute pressure is meant the pressure above a vacuum, distinguished from gauge-pressure, which is measured above the atmosphere (14.7 lbs. absolute).

Owing to the molecular repulsion in gases, a compression of volume cannot take place without a corresponding development of heat, the increment varying with the initial temperature, as the accompanying table shows. The last column gives the factor of ratio between the initial and final absolute temperatures, t and T, of the air. $(t + 459) f = T + 459$.

Gauge.	Volume isothermic.	Volume adiabatic.	Temperature, Deg. Fahr.	Temperature, Deg. Fahr.	Factor.
0.0	1.000	1.000	61.	90.	
14.7	.500	0.612	175.6	211.9	1.222
29.4	.333	0.459	255.1	294.2	1.375
44.1	.250	0.374	317.4	362.0	1.495
58.8	.200	0.319	369.4	417.0	1.595
73.5	.167	0.281	414.5	464.8	1.681
88.2	.143	0.251	454.5	506.8	1.758

If this compressed air be immediately used in an engine, it will return to its initial stage of temperature and pressure. This expansion is said to take place adiabatically,—freely, without receiving heat. At 461.2° F. there is no pressure.

The above table is based on a sea-level pressure. At different altitudes the absolute pressure and density vary as below.

Altitude.	Pressure.	Density.
Sea-level.....	14.7	1.00
¼ mile above	14.0	0.96
½ “ “	13.3	0.91
¾ “ “	12.7	0.86
1 “ “	12.0	0.82
1¼ “ “	11.4	0.78
1½ “ “	10.9	0.74
2 “ “	9.9	0.67

The increase in temperature is an obstacle to rapid running; it tends to expand the volume so heated, as indicated above. If, however, the expansion is resisted, the heat reacts on the compressed air and increases its tension and, consequently, its pressure. The increase of resistance to compression is 0.00204 of the pressure for each degree Fahrenheit. A volume of air which has been compressed to 73.5 lbs. has a temperature of 414.5° F., by reason of which it would expand to 1.78 times its original bulk, in accordance with the following formula: $491u' = u\ (491 + T - t)$, in which the volumes u' and u correspond to the temperatures T and t. Since, also, it is impossible to contain the heat, spite of every precaution, and there is no necessity for its retention, it is extracted as soon as possible. The loss of heat begins with the instant of its development by radiation from the conducting-surface of the cylinder, receivers, and pipes. The cooling attachments added to the cylinder more or less perfectly complete the dissipation of heat and permit an isothermic compression. It may be noted that the increment of heat is greater in the early stages of compression than toward the final; so the cooling is best done at the beginning of the stroke.

Fig. 230 is a graphic representation of the adiabatic and isothermal curves of air. The volumes of air at various times

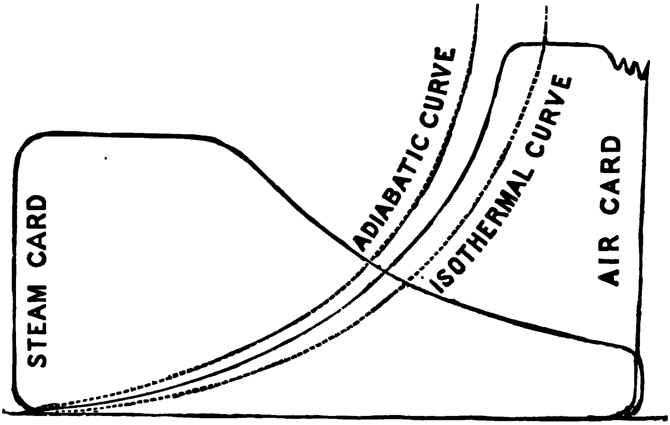


Fig. 230.

are laid off on the horizontal line and their corresponding pressures may be measured by the verticals. It will be seen that the adiatic curve rises more rapidly than the isothermal or than the intermediate condition of cooling attained in the com-

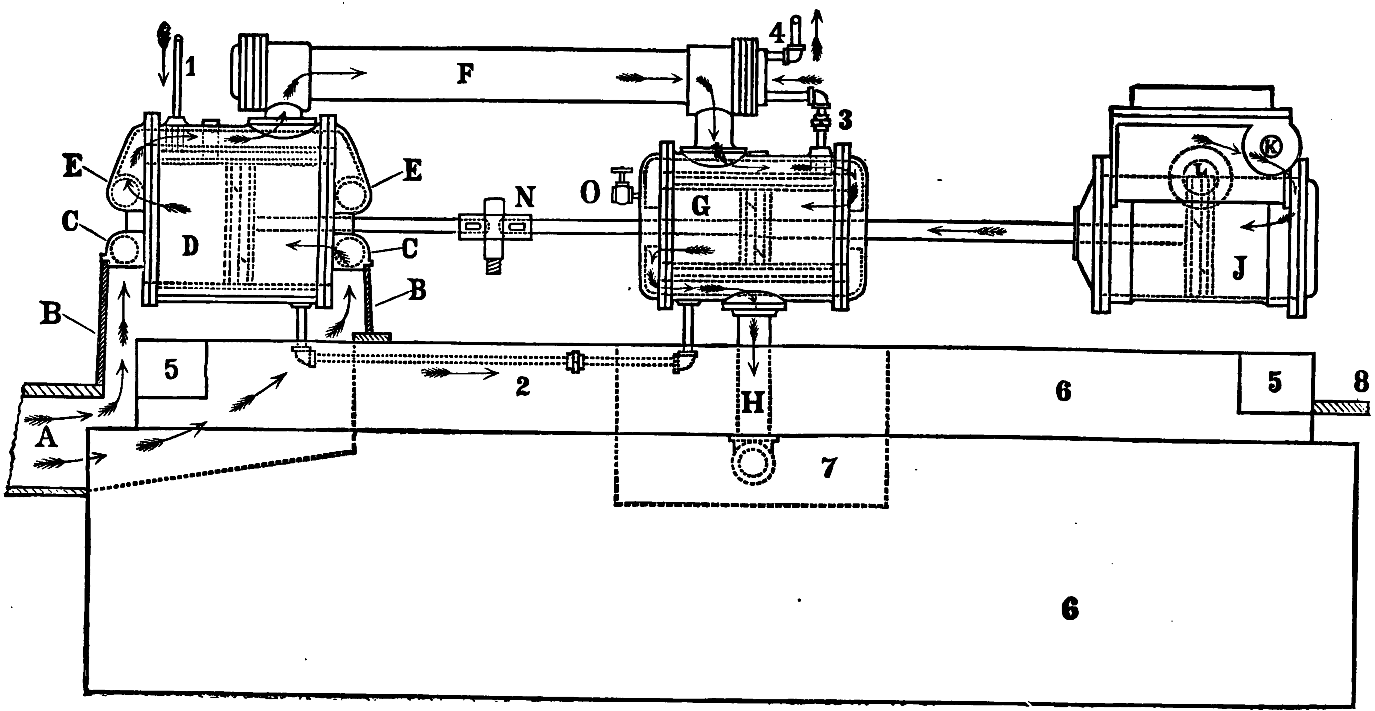


FIG. 231.

pressor. The indicator card also shows the behavior of steam when expanding from 58 lbs. pressure at 0.3 cut-off.

If the air cools, while its volume remains constant, a fall in pressure ensues, and the capacity for work in re-expansion is reduced. This is a serious loss, and is greater with the degree of compression. Rankine showed that the loss is rarely less than 65 per cent of the work performed by the motor. For 1, 2, 3, 4, 5, and 6 atmospheres, gauge-reading, the losses are 28, 37, 46, 50, 53, and 56 per cent of the original power. That is, the higher the pressure, the less is the efficiency of the expanding-engine. It is also greater if the cooling be not effected in the cylinder. Although the heat has been extracted from the air, it is still under pressure, and its unrestricted (adiabatic) expansion is yet capable of producing work; but by no means to the extent otherwise possible. There remains only the potential energy of the comparatively cool air, which is discharged at a temperature of about 180 F., amounting to somewhat over 6000 ft. lbs. per cu. ft.

Economic work is best obtained, then, by operating at as low pressure as consistent with the work. Again, if the air could be cooled before compression, so that after compression it will have the temperature of the surrounding air, better work would be done. The storage, too, of high-pressure air is difficult. The loss would be less if the air was heated, during its use, to an isothermic condition. This is impracticable, ordinarily.

Thus it becomes essential that the engine have its greatest power during the early part of its stroke, and yet drive the air-piston, with its maximum resistance at the end. For high pressures the difference is very marked. Single-acting conical cylinders have been used, but the compound-air cylinder has proven more effective for this purpose. In Fig. 231, the air, after a partial compression in *D*, is forced into the small cylinder *G*, where the operation is finished. The arrows show the directions followed by the air. This renders the resistance more uniform than where the compression is effected in one cylinder. A fly-wheel, and heavy parts, are partial equalizers,

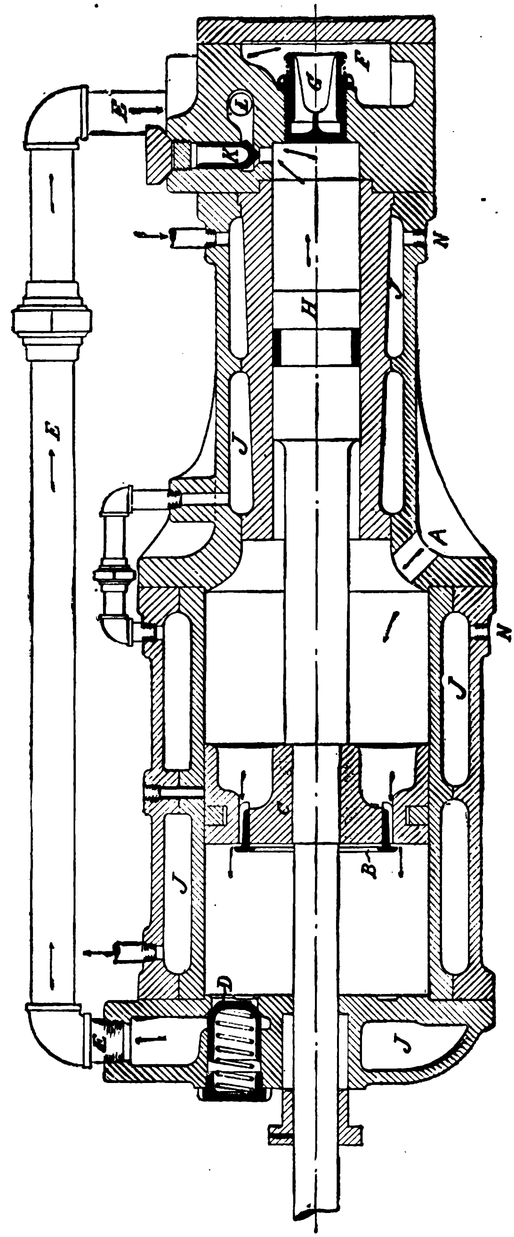


FIG. 232.

adding to the power when the steam is weak by expansion. The ratio between the air-resistance and the steam-pressure is fixed by the relation between the areas of the steam- and air-cylinders.

The air-cylinder is connected, directly or through intermediate gear, with the steam-cylinder, or geared to a water-wheel. It is simple or compound, and single or duplex. The air-cylinder is said to be "tandem" to the steam-cylinder when their pistons are extensions on the same rod; and "crossed," when alongside and joined to a cross-head. Its operation is identical with that of the steam-pump, and, substituting air for water, and dispensing with the air-chamber above, Fig. 77 may well represent the air-compressor.

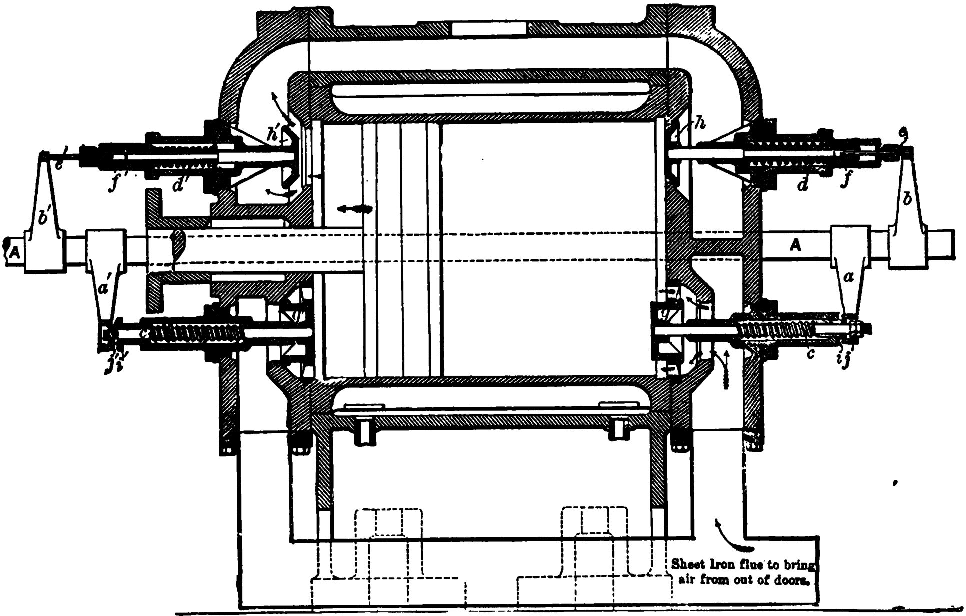


FIG. 233.

94. The absorption of the heat of compression is accomplished by a cold-water jacket surrounding the cylinder. Formerly, a spray of water, injected into it, extracted the heat; but owing to the obstruction of the machines by the formation of

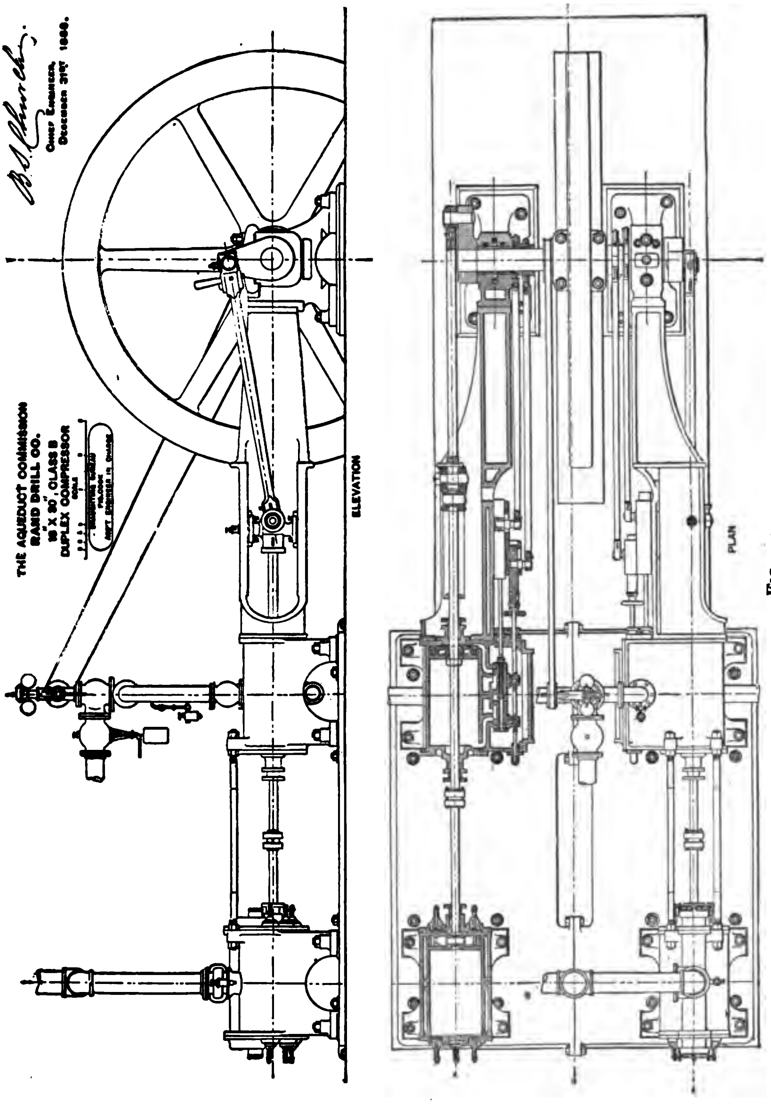

THE AQUEDUCT COMMISSION
RAND DRILL CO.
18" X 30", CLASS B
DUPLEX COMPRESSOR
SCALE
B. S. Church
CHIEF ENGINEER,
DECEMBER 31ST 1886.
ELEVATION
PLAN

FIG. 234.

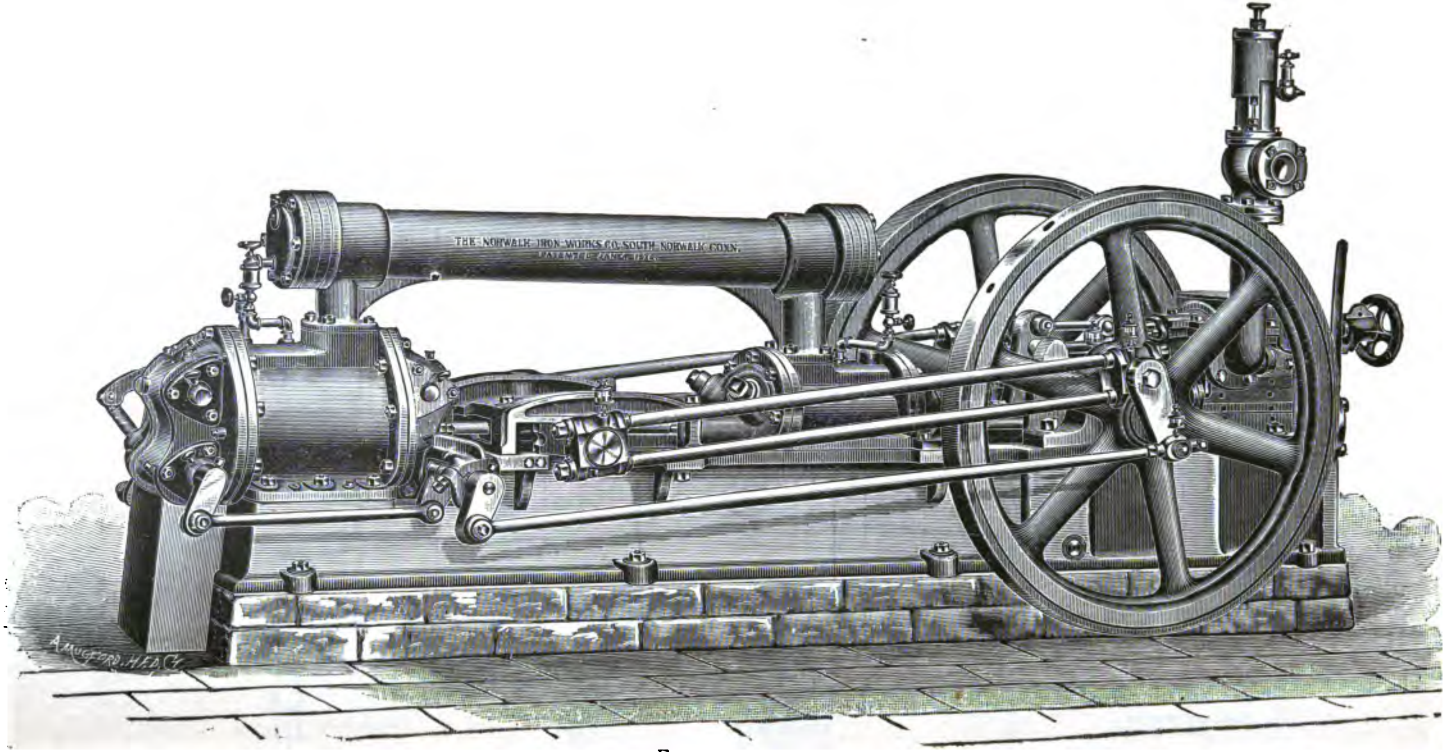

THE NORWALK IRON WORKS CO. SOUTH NORWALK CONN.
A. MUGFORD. H.F.D. Co.

FIG. 235.

snow, in use, the plan was soon abandoned. The accompanying figure (233) is a section of the Rand cylinder, showing the cooling-jacket space, outside of which is the air-inlet. Fig. 234 illustrates the direct-acting duplex compressor, one half of the plan being in section. Fig. 235 is a view of the Norwalk pattern, of which Fig. 231 is a section.

The cooling-water should be taken very cold, and may be used for feeding the boiler afterwards. As its time of contact with the air is very short, the cylinder should be a perfect conductor of heat, to extract an appreciable amount. In compound cylinders the air meets two currents; but even they cannot perfectly cool the air. So, often, an additional cooler is used, *E*, Fig. 232, in the Ingersoll, and *F*, Fig. 231, in the Norwalk. A reservoir filled with thin brass pipes circulating cold water offers a very efficient cooling-attachment, which saves as much as 10 per cent of the power, and about counterbalances the friction of the machine. Perfect cooling of 60 lbs.-air would save at least 20 per cent of power.

The work during one stroke of isothermic compression is measured by the formula,

$$W = P(V + v) \text{ hyp. } \log \frac{V + v}{V' + v} - (P' - P)\,v;$$

in which V and V' are the volumes of gas at P and P' lbs. tension per square foot, and v is the volume of the clearance. The work of compressing air varies with the initial and final temperatures, t and T, as $183.45(T - t)$ per pound of air. It requires about 0.19 per cent more of work for every additional 1° F. of warmer air. One pound of 90° F. air, raised to 88.2 lbs. absolute (464.8° F.), requires 68,757 ft.-lbs. of work; 60° F. air, to the same pressure, requires only 67,171 ft.-lbs. For this reason, and that its density is greater, the air should be taken from outside of the engine-room.

Moreover, the air should be dry and free from dust, to prevent clogging of the machines, though the work of compressing a pound of dry air is somewhat more than that for moist air, as the accompanying table shows; so, too, the temperature of the dry air rises more rapidly than that of the moist.

Absolute Pressure.	Temperature.		Work on One Pound.	
	Dry.	Moist.	Dry.	Moist.
14.7	68° F.	68° F.	—	—
22.	133.8	94	13,300	13,200
29.4	185.9	111	23,500	22,500
36.7	229.5	124	30,500	29,000
44.1	266.7	135	37,000	35,000
51.4	300.2	145	43,200	40,600
58.8	330.1	153	48,500	45,000
73.5	383.5	167	58,500	52,500
88.2	428.9	179	67,160	60,000

One inevitable source of reduction of cylinder-capacity is the clearance-space between the piston and the cylinder-head at the end of a stroke. The warm air filling it is never discharged, but, on the return stroke, expands and fills a volume that should have been occupied by the fresh atmospheric air. An increased length of stroke, or compounding the cylinders, are the only remedies. At 75 lbs. pressure, the single cylinder must be over three times as long as would the compound for equal clearance loss. Manufacturers dare not plan for a smaller than $\frac{1}{16}''$ clearance. A German device for obviating the effect of clearance consists in effecting a communication between it and the other parts of the cylinder. Indicator-cards show an increased efficiency of 5 per cent by the use of the contrivance in connection with a dry slide-valve.

The valves of the compressor are of the poppet, spindle, or ring patterns. Whatever their form, they should open quickly, have a full lift, and be ample in size. The inlet valves offer little difficulty, though for a short time they are subject to full reservoir pressure. An unrestricted entry for the air is obtained easily by the use of poppets held by springs. The Ingersoll-Sergeant compressor admits air through a hollow piston and rod (Fig. 232), and thus leaves more room for the discharge valves and the cooling-surface. The Norwalk-employs a Corliss valve. The inlet valves of the Rand are shown at *g*, *g′*,

Fig. 233. They are provided with guards that prevent falling into the cylinder.

The valves should be positive, and this the poppet obtains. Its objection, however, is the tendency to "chatter," and for the discharge valves this is serious. This arises from the two opposing efforts—one, of the air, to open, and the other, the spring, to close the valve. The Norwalk-Corliss pattern does away with this trouble in the high-pressure engines; as also does the valve-gear shown in Fig. 233. The arms, *a*, *b*, relax the spring-pressure and allow of the valve rising full lift without dancing. Poppet-valves can hardly be improved upon for low pressures, though their springs lose elasticity and open too soon. This reduces their efficiency, as also does any slip of the valves. In the Norwalk pattern (Fig. 235), a positive discharge is obtained by moving the valve by cams, such that it remains at rest till the pressure is sufficient to open it quickly. A difficulty about this, it would seem, is that, as the reservoir pressure constantly varies (unless perfectly regulated), the valves must be able to open at different points in the stroke, and this it would be difficult to arrange mechanically.

The discharge valves require careful construction, for their leakage is equal to a large clearance-space. In order to reduce the friction of the air passing they are made large, and, to prevent inordinate loss and wear, as numerous as possible. An excess of engine-pressure over receiver-pressure is necessary to open the valves and expel the air. This unavoidable loss has an important bearing upon the uniformity of speed. An automatic regulator assists this to a certain extent, though, as a matter of fact, a hand regulator is found to be equally satisfactory for a long line of pipe. There have been devised plans for unloading the engine, and maintaining a uniform pressure, even under a heavy draft upon the receiver, but of their performance we have yet no returns.

Thus the simple principle of the air-compressor becomes exceedingly difficult of execution in an efficient manner. To obtain a compact, high-speed, uniform, rapidly cooling, efficient engine is not easy. These essentials are secured in various ways in the Burleigh, Clayton, Delamater, Ingersoll, Norwalk, and Rand patterns. The engines are "straight-line," direct-acting, or duplex, supplied with fly-wheels, and run by plain slide-valve or Corliss engine, water- or electric motor. The direct-acting and horizontal form is preferable, for many reasons, though one objection to the straight-line form is its liability to centering. Fly-wheels—and some are weighted at certain points—remedy this somewhat.

High piston-speed is advantageous for economy of steam and capacity; but rapid wear, the difficulties of large valve-area, and the inordinate resistance developed, forbid a greater velocity than 300 or 400 feet per minute, except in the larger sizes. Automatic valve-gear, on the Rider system, are also added now; they entail no loss of steam, and, with the variable cut-off, regulate the engine speed. High-test cylinder-oil is required for lubrication; graphite is the best lubricator.

Duplex engine connections are by cranks at quarters. The frames are solid and well founded. For purposes of transportation to remote camps, they may also be had sectional. Water-power and wheels are admirably adapted for this work. Two 66″ Swain turbines, with 16′ fall, run four compressors 24×60, furnishing air for 20 drills, 8 hoisters, and 17 pumps, at the Republic mine. At the Anaconda, the 30×60 duplex, with Corliss valves, is the largest compressor that the author knows of.

The Burleigh is upright, and its peculiarity is in the comparative sizes of the steam- and air-cylinders, and in putting the steam-entry $\frac{1}{8}$ behind the air. Its air-cylinder is single-acting. The Clayton has the usual poppet-valve, and is a compact machine, with its fly-wheel centrally located. The Delamater has an important contrivance for dropping the discharge valve from its seat. This form is very heavy. The Ingersoll, Norwalk, and Rand are the popular pneumatic machines. The Waring has a bonnet, or conical valve, like that in Fig. 233. Its pistons are moved by a rocker on the fly-wheel, the steam-cylinder being set at an angle to the horizontal air-cylinder.

The discharged air is stored in a receiver, whence it is withdrawn as required. It is simply a strong iron reservoir, of any convenient shape, and commodious enough to meet any draught upon it.

95. The air is conveyed to drill, coal-cutter, hoister, etc., by pipe, and in its transmission a great loss is experienced by reason of the friction.

Recurring to the formula in I, **50,** it will be recalled that the frictional resistance is directly proportional to the square of the velocity of the flow, directly as the length of the conduit, directly as its periphery, and inversely as its area, or the square of its diameter. As the periphery is proportional to the diameter the resistance becomes an inverse function of the diameter. Tables are given by manufacturers of the loss of pressure by flow in pipes, and it will be found therein that air at 32.8 feet per second loses 8.26 lbs. pressure in a mile of 10-inch pipe, 10.04 in an 8-inch, and 20.08 lbs. in a 4-inch pipe. Below is appended a table giving the loss in pressure, p, for various velocities, v, in feet per second, and different diameters of pipe, length being 1000 feet; q is the volume of "free air" passing per minute (air-cylinder capacity), corresponding to an assumed gauge-compression of 60 lbs.; q', the volume at 80 lbs. This is copied from the Norwalk Iron Co.'s book.

	1"			3"			4"			6"			10"		
v	p	q	q'	p	q	q'	p	q	q'	p	q	q'	p	q'	q'
3.28	.143	6	7	.046	48	60	.034	86	109	.023	193	244	.014	537	680
6.56	.640	12	15	.209	96	121	.152	172	217	.104	386	488	.064	1073	1359
9.84	1.454	18	22	.488	144	182	.360	258	326	.244	579	633	.145	1610	2039
13.12	2.562	24	29	.838	193	243	.628	343	436	.419	772	977	.256	2146	2719
16.40	3.934	29	37	1.317	241	304	.982	429	544	.658	965	1221	.393	2683	3399
19.68	5.422	35	44	1.808	289	364	1.356	515	653	.904	1158	1466	.542	3220	4079
26.24	10.248	47	59	3.352	386	486	2.513	687	871	1.676	1544	1954	1.024	4293	5438
32.80	15.738	59	74	5.270	480	607	3.928	859	1088	2.635	1931	2443	1.573	5367	6798

Elbows and short turns will reduce this pressure. For any compression m than that given above, the bulk of the air *after* compression will be different from that at 60 by $74.7 : m + 14.7$, taken inversely; or from that at 80 lbs. by

94.7 : $m + 14.7$. To carry this the velocity of the flow is different, also its friction. Suppose 300 cu. ft. of air at 60 lbs. per minute was required at 5000 feet from the compressor, the corresponding volume of free air is 1530 cu. ft. A 10-inch pipe would carry this with a loss of but 0.657 lb., a 6-inch losing 8.362 lbs. This would require a receiver pressure of 61 and 69, respectively. The 10-inch pipe would save 9.5 horse-power.

The power to be transmitted varies as the product of the air-pressure and its volume, which varies with the product of the velocity and the square of the diameter of the tube. Hence we may increase the power and diminish the resistance by enlarging the tube. The size most expedient is determined by the relative and absolute costs of power production and the pipe laid. The motor pressure must exceed that of the desired pressure at the expanding machinery by the amount of the friction loss. As the friction increases so does the most economical pressure. With a pipe friction of 5.8 lbs., 30 lbs. at the compressor will give an efficiency of 63 per cent; with a loss of 8.8 lbs., the compressor gauge-reading should not be below 38 lbs.; with 11.7 lbs., an efficiency of 56 per cent is obtained from a 47-lb. gauge-reading; while, if the friction should reach 26 lbs., the lowest admissible gauge-pressure is 82 lbs., when an efficiency of 50 per cent is to be obtained. This efficiency may be increased by diminishing the friction, but it can never be 100 per cent. The friction loss may be reduced by increasing the pressure of the air, thereby reducing the volume to be transmitted, and hence its velocity. But in doing this we have not increased the efficiency. So there may, or may not, be a gain by employing high pressure; this is a matter of local circumstances.

In utilizing air-power, pipes as large as convenient should be employed: 2 inches is as small as practicable, and this size is only for rooms and stopes; they should be united to not less than 4 inches in medium-length entries and levels.

All bends should be gradual; with leakage they reduce the pressure, which can be maintained only by care and judgment.

Unless purposed for ventilation, leaks should be plugged. Compressed air escapes at a velocity of over 20,000 feet per minute, and the consequent loss is apparent.

The pipes used are steel-riveted or lap-weld, as illustrated in Figs. 60 and 64. The joints should be carefully tightened. Means must not be neglected for providing for changes in length due to alternations of temperature. Iron expands 0.000007 its length per 1° F. This allowance is more essential above than below ground; and in shafts where the temperature is inconstant. Compensation-joints are necessary; at every 300 or 400 feet a copper U-tube is attached. Its straightening out will allow for contraction, while expansion of the pipes will double it up. At the Republic mine brass-lined expansion-joints every 500 feet allow for 12-inch movement. They rest on gas-pipe rollers. The Chapin iron-mine has $500-expansion-joints at every 680 feet of the 24-inch pipe.

Irrespective of the friction losses, the power possessed by the compressed air would equal the work done upon it if the extremes of volume, temperature, and pressure are the same, for the cycles of changes which the gas experiences are duplicates. But friction and cooling absorb so much power that the "duty" of the pneumatic engine is small. Besides, as the compressed air is at a temperature not much above that of the surrounding atmosphere, it must fall far below the normal when used. Being colder, its volume is less than that occupied by it in the compressing engine; hence, even between the same extremes of pressure a return cannot be expected equal to the work expended.

In the maintenance of the isothermic conditions during compression there lies the greatest loss of power. If the heat there extracted could be returned to the gas while in use an economy of power is experienced, as the gas would expand isothermally. The cylinder (in drill or coal-cutter) encased in a hot-water jacket might accomplish the same result. But this is impracticable under normal conditions and the air must therefore be used adiabatically.

On the next page is Mallard's table showing the merits of working the compressed air at full pressure, P', or a complete expansion, taken from H. S. Drinker's "Tunnelling." The initial temperature, t, is 68° F., and the final, T.

Pressure Ratios.	Fall of Temperature after Expansion.	Theoretical Efficiency.	Fall after Working at P'.	Theoretical Efficiency.
	Degrees F.	Per cent.	Degrees F.	Per cent.
2	102.	85.5	72.3	82
3	140.	80.6	98.4	72
4	170.6	78.2	109.7	67
5	192.2	76.8	118.4	63
6	208.4	75.8	123.8	60
7	212.8	75.1	127.4	57
8	234.9	74.6	130.1	55
9	244.6	74.2	132.1	53
10	253.2	73.9	133.7	51
11	260.4	73.6	135.3	50
12	266.9	73.4	136.4	49

From this we see that, working without expansion, the refrigeration is not so excessive, but the efficiency is low.

Air at 60° F. and 60 lbs. pressure falls to 26° F., and if used expansively to 100° below 0° F. A saving in power would be had if air could be used expansively, but on account of the intense refrigeration (even the dry air of the Colorado climate at 60° F. contains sufficient moisture to saturate it at 10° F., below which point the ice will form) this is impossible. So, usually, full pressure is maintained during the stroke. There is a slight gain from the presence of vapor in the air and also in the work of compression (see table on page 392), and were it not for the formation of ice we would have an economic condition.

As the efficiency of the driven machine is not much over 60 or 70 per cent, the indicated horse-power of the entire combination does not exceed 25 or 30 per cent of the original motor power.

These facts account for the limited use of this physical agent. Yet it answers well for many purposes, notwithstanding those unavoidable losses inherent to the gas, and it will continue to retain its place as a valuable accession to mining. Its ventilating power enhances its value, and improvements in mechanism will increase its efficiency. A simple compressor with a simple motor has about 40 per cent efficiency; a compound compressing cylinder, 46 per cent; with a compound air-cylinder and a compound motor, 51 per cent. Its sole motive competitor is electricity, which may surpass it in energy, but which cannot furnish ventilation without additional appliances.

As examples of compressor-plants and their price we may quote the following: An 18 × 30 duplex, at 50 revolutions, and 8-inch pipe, will run 26 drills, the compressor alone costing $4800; an 8 × 12 (costing $1100) will drive 10 coal-cutters; a complete plant for operating three ordinary-sized mining drills would cost $3800—for two extra drills, steel, etc., add about $800; an 8-drill plant complete cost, f.o.b. at St. Louis, $8400, and weighed 42,000 lbs. The Chapin iron-mine, L. S., has a plant 3700 horse-power, giving an efficiency of 27 per cent, that cost $500,000.

EXAMPLES.—Required the loss of air in 4400 feet of pipe, 10 inches diameter, if 1960 cubic feet of 50 lbs. air is to be conveyed per minute. 9.9 lbs.

Required the sizes of pipes from three connected pairs of air-compressors, at 60.8 lbs. gauge, 29.4 revolutions, producing 1,826,700 cubic feet air per 24 hours, for 15 drills, at a distance of about 2100 feet, 14 at 2600, 21 at 3100, and 5 at 3600 feet. 12″, 10″, 8″, and 4″; total loss, 6.89 lbs. pressure.

Required the coal consumption of the above system of compressors having an efficiency of 40 per cent.

An 18 × 30 duplex at 75 revolutions gives 205.7 cubic feet of air at 80 lbs. absolute. How many $3\frac{1}{8} \times 6\frac{1}{2}$ drills at 220 blows will it drive, allowing for 12 per cent pipe loss? 18.

At the Hoosac Tunnel 7150 feet of 8-inch pipe lost only 2 lbs. of pressure of a 68-lbs. (absolute) air. What was the velocity and the volume of air passing?

11′ 6″ per second, and 269 cubic feet per minute.

Required the volume of 60-lb. air to drive a pump with 14-inch air and 8-inch water cylinder against 200 feet head. Piston-speed 100 feet. 430 cubic feet free air.

CHAPTER X.

MINE EXAMINATION.

96. Examination and evaluation of mines; sampling and measuring the deposit; features to be noted; capitalization; "ore in sight." 97. General remarks regarding the treatment of ores; factors determining their value; deleterious substances; various milling processes; cost of mining; formulæ for mine valuation. 98. The mining-labor problem; variety of skilled labor employed; selection of men; necessity for regulations and their enforcement; conveniences, hygienic and otherwise; number of shifts and their length; mode of paying; necessity for reciprocity; day's pay *vs.* tribute system; contracts and the mode of letting; pay by the output or progress; dead work; leasing mines. 99. Retrospective.

96. THE examination of a property with a view to purchase or exploitation will not be a difficult matter to one who is cognizant of all the local economic conditions, and is familiar with the geognesy of ore-deposits. It calls for the exercise of sound judgment and technical instinct which we endeavor to cultivate: these faculties are self-taught; observation and caution will strengthen them. Vigilance and prudence prevent many of the fatal mistakes to which hasty conclusions of the human mind are prone. You cannot foresee or predict in detail the future of a mine. Your only basis is the general canons. To apply these to the selection of a system and plant you must collect and investigate all data in any degree suggestive.

Proceed to a careful study of the survey maps of the district, for they will assist in your determination of the trend, frequency, extent, and size of the ore-shoots. Close investigation of any idle mill or abandoned works in the vicinity, and the commonly received opinions regarding the causes of their failure, may prove valuable as a guide. Converse with the miners, as they are in possession of valuable details, and may be able to furnish the history of similar enterprises. Consult all convenient sources before visiting the mine, then "sort" out the prejudices, "screen" the ignorance, "jig" the balance by technical knowledge, and examine the "concentrates" for probabilities.

On visiting the property proceed systematically with the geological examination of the enclosing rocks, the character of the vein matter, and its irregularities of thickness or pitch. Look closely for faults or slides. Collate the data thus obtained, and determine if the deposit be gash or fissure, pocket or bed. Note the toughness and hardness of the vein matter, the security of its walls, and the width and softness of the ore-streak with a view to calculating the cost of mining. The dimensions of horses, partings, or barren ground and the cost of their removal should be considered; as also the quality and quantity of the other available materials for support.

Wherever the mineral is exposed, samples are to be taken for assay to ascertain its value and quality. Every variation in the dimensions or character of the ore body should be sampled in a line across the entire exposure. Each streak or bench may differ and require sampling. In every case describe in the note-book the appearance of the face, its location, and the width of the streak. Take the precaution to arbitrarily select the spots for tests, and collect a sufficient quantity of each constituent mineral of the ore for its separate assay. Obtain freshly blasted or picked material. Examine closely for any evidences of concealment by plastering or timbering up of poor faces. Take "grabs" from the cars or stock piles. It would not be amiss to sample the dump. A sheet of paper, cut into snips, is thrown to the winds. Grabs are taken where they alight on the dump. The size of the dump will convey some idea of the amount of the work done. Seal all bags immediately upon filling, and leave no opportunity for injection into them of strong solutions of mineral.

Take all measurements that will serve as guides in estimating the area and extent of the vein or bed, the amount or value of mineral capable of extraction, and the amount and value of that to be left for filling, mined or unmined. Be cautious, and secure results as accurate as can be measured. In estimating the "ore in sight," it is fallacious to be too liberal. Include only the mineral in the blocks exposed on at least two if not three faces by the connected drifts, shafts, or stopes. Be careful, then, in the use of this term. It is often reactive. In computing the body of mineral remember that all is not sold. A portion is not mined out, some is buried in the waste, some discarded in sorting, and some lost in the concentration. The ratio of the volume sold, as per smelter and mill returns, and that of the work done, will prove a reliable guide. Likewise with coal: of its actual volume every acre will sell about 1000 tons for each foot of thickness of the bed. Mines only slightly developed are not readily estimated; but by comparison with the records of neighboring properties a rough estimate may be made, bearing in mind that in veins, a proximity to profitable mines is of less monetary value than in beds. From the calculation of the volumes of mineral and their several assay values, the property may be evaluated. While enormous reserves and untold wealth may thus have been found, a great deal of hard systematic work must be done before the mineral has been mined, shipped, and drawn against. Several mines may be quoted carrying large bodies of 1½ ounces of gold, which are absolutely worthless because the ore is rusty. So with a 55 per cent sulphurous iron ore, a 46-inch coal vein with 5 partings, or a 20-foot zinc bed with pyrites. For this reason the term "ore in sight" should only represent its net value. The quantity of coal in an acre of bed is found by the continued product of 100, the number of inches thickness, and the specific gravity of the coal. For bituminous coal it is 1964 tons, and for anthracite 1775 tons per foot of vein-thickness per acre of area.

The cash value of a mine is that which will net a given annuity to the investors. The amount of this dividend should increase with the risks run. At best mining is somewhat precarious because of the variability of its dependent elements, as ore value, market, wages, floods, and fire, and a greater return is expected than, for example, from United States bonds. So the annuity should be 10 to 30 per cent in addition to the legal rate of interest. Moreover, the deposit not being inexhaustible, the life of the mine becomes a matter of mathematical calculation, which must needs enter our estimate. Against this depletion, and the wear and tear of the machinery, the sinking-fund must provide. The yearly contributions to it must create a new capital within the period of its life. Frequently the impatience of the operators to extract the ore in the shortest possible time fixes the limit of this period. Allow for this proportionately.

On this account a partly opened mine is depreciated by the amount it has shipped, but, on the other hand, this development work has more or less demonstrated its value. Prospects are mere speculations, and their values are inversely as their chances, which must be carefully weighed. The probable continuity of the ore-shoot to a reasonable extent may be quoted, but only as a speculative guide. Every mine, till its value has been assured by a steady output, is a business speculation, and must be considered as such. It may only be compensated for by the high prospective rate of interest. True, the security is not invariably as good as under some other investments, but safety and income are complements—in fact, reciprocals of each other. Those demanding security must content themselves with a moderate interest. So be explicit and frank in your estimate, that your client may be assisted in judging of the risks as well as the returns. Better recommend it for the cultivation of mushrooms than indulge in any imaginative representations, or juggle with figures.

Investigate fully the condition and security of timbering, filling, pillars, and other supports of an abandoned mine with a view to the cost of their reinforcement. The cost of reopening may overbalance the estimated value of the deposit. The causes for its abandonment are numerous. It may have been due merely to the inflation of its stock beyond a fair dividing basis, or to a lack of ready means of communication which time has improved. The exhaustion of means with which to prosecute work may have been the cause of shutting down. Discord among the co-owners is a very common cause of failure. Be careful in accepting the testimony of their books. The permanent-improvement account is often delusive, because it is frequently made to cover awkward expenditures. Take no cognizance of those entries dating subsequent to the moment of actual production.

If the results of the examination do not prove sufficiently positive, or the information regarding the geology or extent of the deposit be insufficient to satisfy the expectancy of a good annuity, duty to your clients requires a full, frank statement, overrating neither the difficulties nor the value of the prospect. A six-months' working bond and option may be advised, or the diamond-drill may be resorted to. Before recommending purchase or exploitation, be sure that the enterprise has a *raison d'être*. Then proceed to a consideration of the capital required.

A justifiable capitalization is one necessary for the proper equipment of the property commensurate with the prospective annuity. This depends upon the scale upon which the property is to be operated. The amount of exposure and the number of faces for attack fix the number of men that may be employed, and either they or the hoisting capacity, the output. For reasons of economy it is advisable to open large areas for attack. This regulates the quality and quantity of the product, and reduces the per-ton working expenses; but maintaining much open ground is expensive, and large capital is needed for the extensive equipment. The choice between this plan and conservative work depends upon the commercial results anticipated. Having figured the amount of capital requisite to properly complete the plans laid out, do not accept any compromise with the parsimonious efforts of the directors, and attempt to work on less than what is deemed neeessary for a successful issue. It is simply inviting failure and disappointment. One canon must be distinctly laid down—that some time must elapse before returns can be realized. Be not over-sanguine in prophecy.

97. All the material coming from the vein must be either valuable, worthless, or injurious, and a metallurgist's skill is requisite in figuring upon the disposal of the ore constituents. The ore is either milling, concentrating, or smelting. A simple ore may be hand-dressed to advantage, though often the gangue is acceptable as a flux, and is not removed with the deleterious matter. The average smelting ores are silicious, and a basic gangue is acceptable if it is not a sulphide. A bonus is paid for an ore with a basic excess. Pyrites injure the quality of coal, iron ore, galena, and gold ores. Coarse auriferous pyrites may be cheaply stall-roasted to advantage. Otherwise washing is the only means of elimination. Blende is objectionable in smelting, and if argentiferous, in roasting. It interferes with amalgamation in pans. Jigging will separate it, and that, too, clean enough to be salable. The roasting of blende and pyrites at the mine for the manufacture of sulphuric acid is suggested as a means of rendering all the constituents marketable without great additional expense. Do not forget, however, that some of the valuable mineral is lost in the operation, and in order that the proper system be adopted to minimize the loss, the assay value of each constituent mineral should be ascertained. Very frequently the blende of an argentiferous ore has all the silver. Concentration under these circumstances would be useless.

Gold and dry ores may be treated by lixiviation, or smelted, and a few with little zinc and lead are better treated by pan amalgamation. These ores are rarely concentrated. Iron oxides with from 25 to 60 per cent of metal are treated as ores. Those with titanium, phosphorus, or sulphur are rejected. If intermixed with clay or loam, they are washed and picked at slight expense. Coal is everywhere acceptable, and is mined with profit if the bed is over 30 inches thick. Whether for gas, coke, or fuel, purity and calorific intensity are prerequisites. Coals are classified by the fuel ratios of their fixed carbon to volatile and combustible matter. Clay is frequently washed and dried.

Hence it follows that every new mining company has this question to solve,—how and where to treat the ore. A smelter may be built, when the ore is unfailing in quantity, but a variety must be at hand. On a big scale, and in close proximity to fuel, it is a most successful method. A concentration-works may be built. Such a mechanical treatment is applicable to almost every variety of ore. An amalgamation or a leaching mill may be erected, but fuel must be cheap and smelting charges and transportation high before competition with smelters will pay. The unchangeable character of the ore is essential to the success of any mill. Before laying the foundation the mine must be fully opened to at least a two-years' reserve, as otherwise another sad mistake may be added to the long list of monuments to similar folly that grace our gulches. Nothing is so certain as the uncertainties of vein continuance. Instances are not rare of sudden changes—a lixiviating ore to a heavy lead-zinc mineral; galena to bornite; free-milling to smelting ores; etc., etc. Proceed cautiously in this matter, minimize the risks, and limit the capitalization. Many a company has been brought to an untimely end by the bane of mining—ill-advised surface improvements. Extensive plank-roads may be dispensed with till there is something to ship.

Several investigators have prepared algebraic formulæ which, after the substitution of the values for the variable factors in a particular instance, will give the price to be paid, or the capital required. Amadee Bruat, Miller, R. W. Raymond and Prof. P. H. van Diest have contributed to this line of mathematical research. While in the hands of a proper manipulator these equations may be satisfactory, it is not safe to entrust the novice with the solution of so intricate a problem as the evaluation of mines by an inflexible rule, into which cannot enter fully all the varied local conditions.

It is unfortunate that in this recital of facts the cost of mining cannot be quoted. The reasons are easily understood. In the Lake Superior region it varies from 94 cents to $3.49

per ton of copper rock hoisted. Gold is mined and milled for from $2 up. All the expenses of iron-ore extraction are not over $2.10 per ton in some localities. The cost of coal-mining varies from 62 to 90 cents per ton. In one section of Colorado a $25 lead ore pays handsomely, while a similar $40 ore near by is unprofitable. No constant proportion exists between the labor and the other items. The ratio of dead-work to ground opened varies considerably with the anxiety of the operators and the proportion of gangue. Ordinarily, the cost of stoping per cubic foot will be one tenth that of drifting, a thirtieth of the sinking, and a fifteenth of the upraises.

98. Besides a familiarity with the practice in vogue among his neighbors, the engineer must have an intimate knowledge of human nature, that the mine-labor problem may be successfully coped with. The large number and variety of men employed, the selection of men and their treatment, are intricate questions, delicate of adjustment.

Besides the foreman, boss, or "viewer," who is the chief officer, parallel with the superintendent of the metal-mines, we have captains, "butties" or contractors, timbermen, shaftmen, masons, hewers or miners, trappers to look after the doors, trammers or "putters," drivers, engineers, etc. The adjustment of their pay and hours is a difficult matter, and is the primary cause of strikes and lockouts. The design should be to secure a mutual interest of miner and employer, in the getting of the maximum of ore in the minimum of time. Good miners are essential to the success of the property, and an ability to judge of their competency is a trait which only long experience can form. A good miner can strike right or left handed, knows the mode of carrying on the work without further notice, and a single-handed man is worth the best wages going. An inexperienced miner is of no earthly account. He will drift off from the vein into the country because it happens to be softer, or from ignorance he will shoot mineral and gangue together and necessitate extra sorting; his consumption of steel and powder will be excessive, or his shots will "pop;" he cannot hold well for the striker, or he will bruise his mate; and he is likely to leave a bald face for the next shift to break from. In all, the greeny costs more than he brings.

The men being scattered in little gangs throughout the mine, the whole force cannot be under the superintendent's eye, and the grossest dereliction of duty may escape his notice.

For these and other reasons a uniform per-diem wages is an unsatisfactory solution to the problem of pay. The old hand certainly should command more than the tenderfoot. The quality and quantity of his work deserves better remuneration, and yet trouble is engendered by attempting to grade the day's pay of laborers on the same class of work. No one but a saint would undertake this.

Some form of contract system remains as a solution to this problem. Dead-work can readily be arranged for at a measured rate of pay. Contracts by the foot or fathom can easily be regulated to mutual benefit; and the plan of fixing a certain minimum and maximum of earning has universal favor in the Lake Superior region. The mode of letting contracts by "shift option" is common. Shift or gang No. 1 has the first bid, No. 2 the next, etc.; after all have given their bid, a chance is offered to any one to underbid the lowest. Besides dead-work, the mining of rooms or stopes in ore of uniform grade and quality is also contracted out in this manner on short times. During the winter, when shipments are slow or impossible, the contracts for dead-work are best let.

In the case of contracts there is an incentive to labor, but the interests of the two parties are not identical. The employé has no more interest in the ore than has the day's-pay miner, and is liable to waste mineral. His sole object is to show as large a measurement as he can.

In coal, iron, or other bedded mines the pay is by the car or ton extracted, and the tramming may or may not be paid for separately. A certain face is let out to a butty and his men, who may work it in one or two shifts. This plan requires constant supervision to prevent the admixture of slate. A certain reasonable percentage of slate or clay is allowable, but an excess over that forfeits the entire car. What with this trouble and the disputes over weights, the life of the "tipple boss" is not a happy one. The practice of offering a bonus to men who exceed a certain given output meets with happy results.

One difficulty with this class of contracts is that the timbering and track-laying are very apt to be inefficiently done. This is effectually obviated by active supervision, a subdivision of the different departments of labor, and their assignment to specialists. While no system can be devised to perfectly meet all cases arising from and under it, contracting, in one form or other, offers a stimulus to intelligent work and fosters habits of observation. This is particularly true in vein-mining, where feeders would otherwise be ignored, while their pursuit might lead to valuable finds.

In the Missouri flat zinc-beds is maintained a system of dividing the ground into plots of 200 feet square and leasing them to operators, who for a definite period of time extract the mineral therefrom and dispose of the ore to the highest bidder, through the owners, who retain a certain royalty and have supervision over the workings. If the work has been done properly, this plan conduces to the benefit of all concerned. Without this, adverse results may be expected. It is a profitable system where a large territory is to be operated, or where capital is scarce and immediate profits dubious. Theoretically, leasing is wrong if applied to mines owned by parties with sufficient capital for developing; for if its operation will pay the lessees, it ought to give similar return with company work. That it frequently does not, demonstrates that there is "a nigger in the wood-pile." It is true the lessees will gut the mine of all the ore and will do no exploratory work; but undeveloped mines, or tracts entirely in virgin ground, have only probabilities on which to base the terms of the lease, which should be liberal as to time and area. In developed mines these probabilities can be approximately computed.

This plan is not confined to the region mentioned. The duration of the lease may be a few months or years. In England it continues with the life of three named persons, and terminates with the decease of the third. The period in any event should be commensurate with the amount of preparatory work to be done. A short term would seem to be unjust to the lessees, particularly if a rich strike is made; but this is fully equalized by the custom that justifies the abandonment of an unprofitable plot, without any forfeit. On the other hand, the articles of agreement often allow the company to order a stoppage of work whenever it may desire so to do. Such leases are recognized by their nature as speculative, and are given under terms that might be a hardship were it not that the corporation is constrained to act with great honesty.

The great difficulty with this system, like to the others, is in the adjustment of dues, company and miner rarely agreeing in estimate. Such differences are inevitable with any form, and a rigid policy must be enforced.

A similar form of leasing, known as "tributing," is adopted in its most characteristic form in many of our American mines, with beneficial results. After the preparatory works are run and the mine has been blocked out, the stopes are leased for a month or so on a stipulated royalty. Each gang is expected to mine with reasonable diligence, to stope up only, to maintain good timbering, and to deliver the ore at the level mouth. The company hoists and markets the product and keeps the mine dry, retaining a certain percentage of the gross values for the privileges. Other conditions, of timbering, smithing, and supplies, are imposed, according to locality. This plan works admirably, except for the trouble over settlements, and is acquiring universal favor.

The men are required to work over-hand, because then the timbering is within easy inspection. Under-hand is not permitted at all.

In beds and pockety mines, with ores variable in quantity and quality, this method is profitably pursued, if the manager is vigilant. Towards the end of the term of the lease, miners not infrequently plaster up the face to deceive him and obtain a renewal on a smaller royalty. Again, much trouble is experienced in the miners on a poor stope or "pitch" helping themselves from a neighboring richer tract.

Mines working on a high-grade mineral usually fit up a room for the change of clothes. The amount of pilfering is thus reduced.

The length of shift varies from 8 to 12 hours. The latter is too long, and even a 10-hour shift accomplishes less than an 8-hour. This is so well recognized, that urgent work is divided up into three 8-hour shifts per day. Men engaged in sinking or in wet ground have either better pay or shorter hours. Large mines, delivering the men below in buckets or cages, lose too much time for short shifts, and they are run under the 10-hour rule. Of those supplied with man engines and ladders, the tally is taken below and some use long, some short shifts.

As to the number of shifts, this very important point is not easily settled. For a given output, two shifts require an area opened and a roadway maintained, of only one half that of a one-shift mine. In metalliferous districts, somehow, night-shifts are not in favor. Certainly, day's-pay mines require very active, conscientious oversight to accomplish as much at night as by day-shift.

99. Attention to these economic details is highly important. No mine can succeed without good miners and conscientious labor, yet this does not constitute the sole element of success. The vital point of the laborer's concern is wages, the proper adjustment of which requires skill, tact, and judgment. Then ability to judge of the quality and efficiency of their work is only acquired by experience and observation. Dereliction or incompetency in this latter respect may undo all the economy and care in the planning and execution of the engineering details. Under these circumstances, the larger the mine and the number of employés the shorter the time necessary to bankrupt the owners.

The designing and selection of the machinery is not by any means the most intricate or even the most important of the multifarious duties of the mining superintendent. Combined with the matters requiring technical knowledge are the endless details, including the supervision of ore sales, the management of men, and the deciding of disputes. A manager incapable of combating these emergencies simply tempts ill-fortune and invites disaster.

The very nature of ore occurrences is such that the element of chance must needs figure in mining as it does in other business, but with the employment of an equal judgment and discretion the result should be equally satisfactory.

The selection and choice of a competent manager is not to be made hastily. The blunder, so frequently committed, of sending a clerk or relative from the counting-room to "run the mine" is responsible for the inevitable failure. Not only is he ignorant of the principles of mining, but he lacks sympathy with his surroundings; nothing recommends him for the position, except, perhaps, his consanguinity or his integrity. The selection of an excellent foreman may counterbalance some of the error in the management, but there is usually nothing in common between the foreman and the manager except the question of salary, which in the case of the former is meagre compared with that of the superior officer, who has little work or experience. The indifference that ensues soon becomes manifest in all branches of work, and the manager has no remedy until the funds become low, when he disappears from the camp, leaving odium upon himself and his class.

Frequently the same superintendent has launched out into that bane of mining work—premature surface improvements, palatial residence, ill-advised mill or process for treatment, and such monuments of folly as should stand out as warnings to succeeding corporations; but the same old mistakes follow one another closely, and striking examples of contrasting extremes are easily quoted.

Until the value of the lode has been demonstrated, neither mill nor elaborate improvements should be erected, for up to that time the prospect is merely a business speculation, and may or may not prove a successful venture. Grass-root bonanzas are rare.

When the plans are being laid, the educated engineer who exercises the business sense required for any other manufacturing pursuit will adopt the tried and true processes: not necessarily those of the camp,—for custom is time-honored to men, and innovations are looked upon with suspicion, and resisted, not having the seal of local usage,—but the most improved methods of successful camps. To such careful, observing management the many mines of Europe owe their continued prosperity, after three hundred years of working. Of shrewd business methods, the Atlantic mine (Fig. 5), with its heavy dividends, 1890, from an ore yielding but 13.27 lbs. of refined copper per ton of rock stamped, is a notable example.

By actual and costly experience, the "practical man" learns what the "theoretical man," the graduate, has been taught—that to profit by the experience of others is wisdom. System will replace obsolete, crude hand-to-mouth methods of yore, and many an idle mine may be quoted that one keenly alive to the improvements in mining appliances might convert into a prosperous property.

Such knowledge comes through the study of the ephemeral conditions through which mining has passed. A compromise between or a union of theory and practice, and in such manner as to inculcate the fundamenta of technical knowledge that will enable the engineer to bring the fancy of expectation to the level of the facts of experience, is the purpose of the School of Mines.

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